ISSN 1454-9174

ANNALS
OF THE
UNIVERSITY OF PETROSANI

MINING ENGINEERING

vol. 11 (XXXVIII)

UNIVERSITAS PUBLISHING HOUSE
Petroşani – ROMÂNIA, 2010

ISSN 1454-9174

Editorial office address: University of Petroşani, 20 University Street, 332006, Petroşani, Romania, Phone: (40) 254/54.29.94, 54.25.80, 54.25.81, 54.25.82, 54.97.49, Fax: (40) 254/54.34.91, 54.62.38; www.upet.ro, E-mail: ibol@upet.ro

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THE ROMANIAN SALT MINE

TUDOR GOLDAN*
EUGEN COZMA**
ILIE ONICA**

Abstract: *As a consequence of salt mining activities huge cavities were created, which by shape and aspect are representing real points of touristic attraction and a new kind of services was linked with the valorisation of these cavities in the rock salt massifs, representing real "saline palaces". Considering the admission of the saline treatment efficiency, mostly in pulmonary diseases, the development of speleo-therapy in Romania is presently in continuous development. If, initially, the saline microclimate did not exceeded the volume of a surgery room, resorting to speleo-therapy procedures the characteristic ecosystem elements are increased, the microorganisms concentration grows and the microflora is modified.*

Keywords: *salt mine, attraction, cavities.*

1. INTRODUCTION

From a geographic point of view, the presently active salt mines in Romania are nearly evenly distributed on the national territory (figure 1).

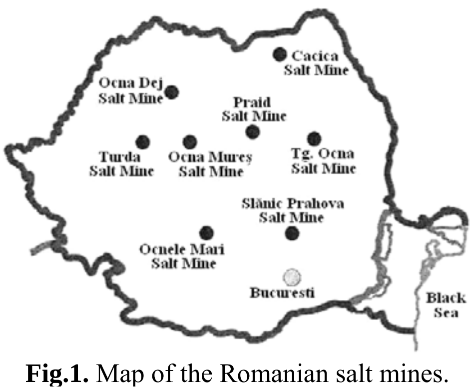

Fig.1. Map of the Romanian salt mines.

During the time, the rock salt mining in Romania was especially carried out employing dry mining methods (deposits located at Slănic Prahova, Praid, Ocna Dej, Tg. Ocna, Ocnele Mari), but at the end of the 18th century there were also applied, simultaneously, mining methods based on salt dissolution (Ocna Mureş, Ocnele Mari, Cacica, Tg. Ocna).

The brine quantity extracted at Ocna Mureş, for example, had increased with time from 45 m^3/day to 500 m^3/day.

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* *Assoc .Prof. PhD. Eng. - University of Petroşani*
** *Prof. Ph.D. Eng. - University of Petrosani*

Afterwards, the salt solution started to be extracted at Cacica salt mine too, after opening the first wells field. The rock salt deposit from Ocnele Mari was mined out through dissolution wells in several well fields, the wells being drilled starting from the surface, the extracted solutions being processed then as raw material for the soda-based and other chemical products in Govora industrial platform.

Due to the fact that the salt exploitation through dissolution lead to floods, as a consequence of soil erosion and caving processes, nowadays this mining method was reduced as frequency, the dry mining being almost generalized.

2. THE OPENING OF MINE DEPOSITS

Initially, the salt mines were developed in a bell shape, but the flooding and caving hazards related to this shape lead to the extension of the mining method with big trapezoid form rooms. This mining method was firstly put in practice in year 1846 at Ocnele Mari salt mine, and afterwards at Slănic Prahova and Tg. Ocna salt mines.

The opening of these deposits was achieved using vertical shafts, or cross adits (figure 2 and figure 3).

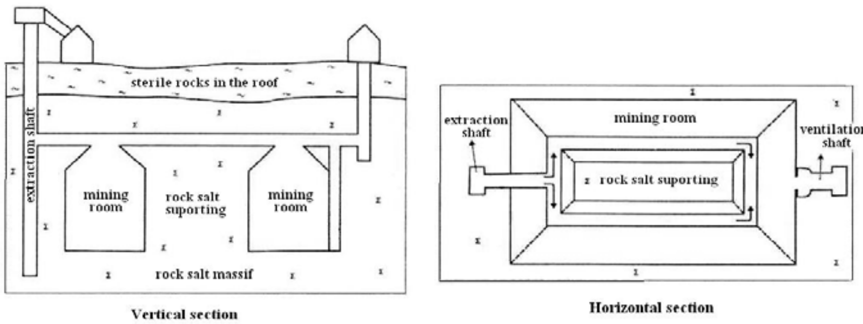

Fig.2. Deposit opening with vertical shaft.

These kinds of structure's sizes are given bellow:

- room width at the roof: 12m;
- room width at the bottom: 35 m;
- room height: 36 m;
- inclination of room walls: 60º;
- pillars dip between rooms: 50 m;
- pillars width: 24 m;
- pillars height: 36 m;

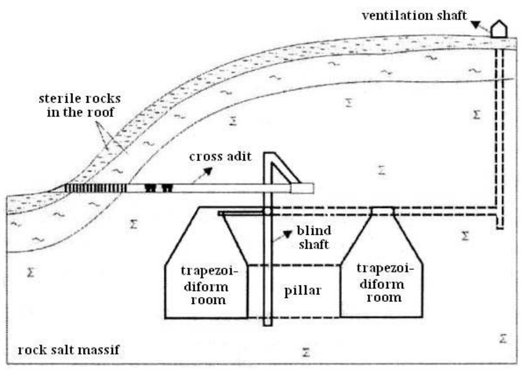

Fig.3. Deposit opening with cross adits.

- protection floor beam thickness: 65 m.

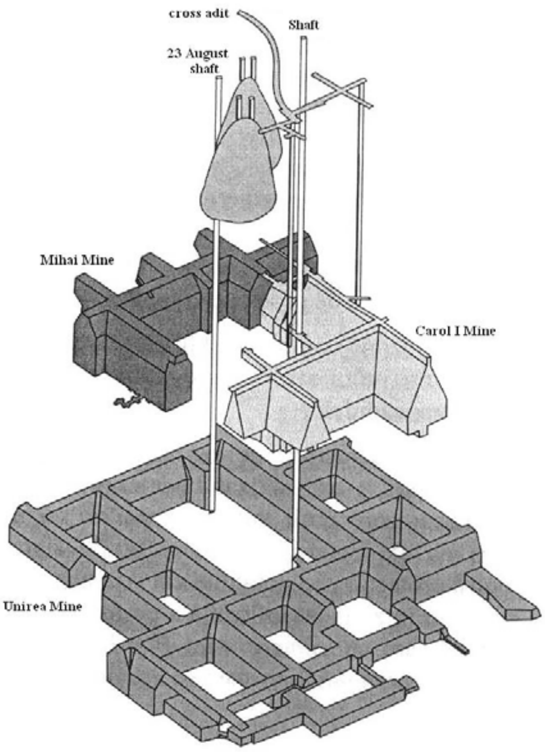

Fig.4. The "Minele vechi" complex – Slanic Prahova.

After the deposit's opening workings were finished, the next stage followed, namely the development of compartments of the mining field in levels/horizons, sub-levels, mining blocks, slices, according to the selected mining method.

Figure 5 illustrates the preparative workings for dry salt mining at Târgu Ocna salt mine, where the main ventilation circuit is based on loop 1-2, developed starting from the main opening working.

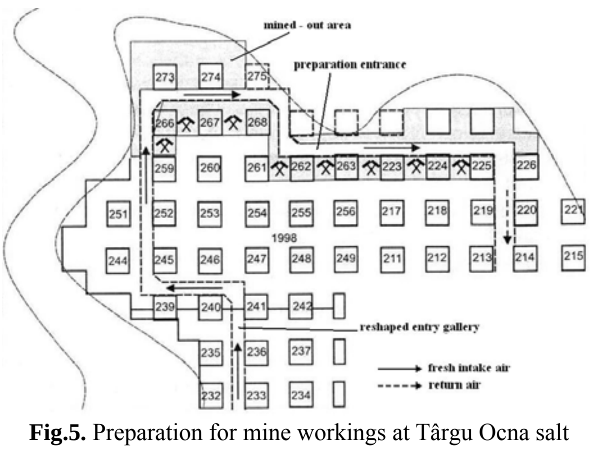

Fig.5. Preparation for mine workings at Târgu Ocna salt mine.

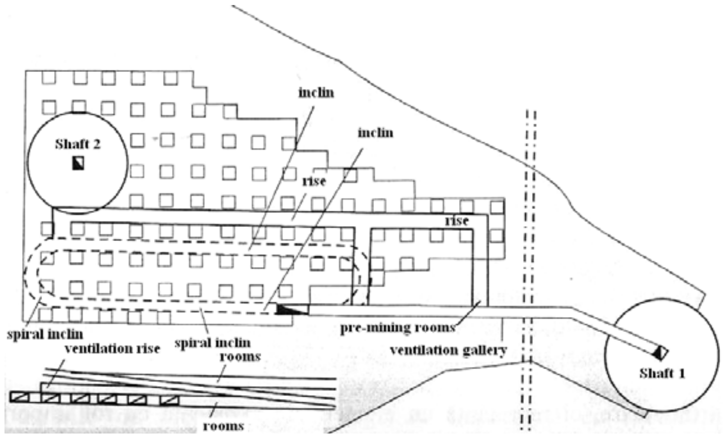

Fig.6. Preparation for mine workings at Slanic Prahova salt mine.

In figure 6 there is diagrammatically represented another preparation method for mining, employed at Cantacuzino-Slănic Prahova salt mine, solution based on ventilation shafts.

3. ATTRACTION IN SALT MINE

While the Romanian salt mines are generally located between hills having low altitudes, of 500-700 m, with a moderate continental climate, with less pronounced values of the climate parameters if compared to those registered in planes and mountain areas, with a sparing, sedative, relaxing bioclimate and having a high number of days with thermal comfort, they can serve for touristic purposes.

Consecutively to the underground salt mining processes, huge cavities have developed, which through their aspect and shape are real touristic grip points, and a quite new kind of services was connected to the valorisation of these cavities, mined in the rock salt massif, which are representing real „saline palace courts”.

There should be noted the very specific microclimate parameters in the old mining rooms, respectively the constant temperature of about 12ºC, the air moisture content of about 50% and an underground atmosphere charged with saline aerosols, exerting a well-known therapeutically effect on the human body.

The salt mine’s microclimate is characterized by constant thermal, humidity, pressure and air velocity values lower than 1 m/s inside the rooms and 0.3-0.4 m/s nearby the ventilation shafts; also, there can be present a slight cooling effect induced by the thermal discomfort, a low stress level due to relatively low temperatures and dehydration effect induced by the diminished water vapour content. The aero-ionization in the small ions field is average, the positive ions are prevailing. The concentration of ions is higher and the negative ions are prevailing in the other field. The aerosols particles concentration is high, with an 80-95% percentage of particles under 3 microns, so with access into the lung alveolus in the lungs.

Because it is widely known and recognized the efficiency of treatment in salt mines, especially in pulmonary diseases, the development of speleo-therapy is extremely actual nowadays in Romania. The medical Romanian research carried out in the last few decades of the 20th century are allowing the more and more efficient use of the natural therapeutic factor – saline microclimate – for the the prophylaxis, treatment and recovery in respiratory, dermatological, immunological and other diseases.

It was considered that the saline environment is therapeutically through his constant thermo-hygro pressure climate, without atmospheric air streams and pollutant agents, with a minimum concentration of microorganisms, so being characterized by antibacterial properties and having a treatment specific mechanism. This one has a determined self-purification period, but with an important quantity of sodium chlorate aerosols, in the conditions of an average to high aero-ionization.

The results of complex researches allow allocating to a speleo-therapeutical treatment anti-inflammatory, hipo-sensibilization, activation of homeostatic mechanisms effect, which in turn provides a higher resistance to microorganisms, different allergic agents and other positive health effects.

If, first time, the saline microclimate did not exceed the size of a surgery room, through the speleo-therapeutic procedures are allowing increasing the microorganisms' concentration, also changing the microflora composition. These remarks are requiring a proper exploitation of these locations and suitable timely monitoring, in order to prevent their pollution.

The above – mentionned properties have lead to development with time, in the old abandoned mining rooms, of spaces aiming for different purposes, such as: tourism; medical care, particularly for respiratory diseases; training facilities for athletes; churches and chapels, rooms for galas and other ancillary activities, museums.

Fig.7. View of the church from Ocna Dej salt mine

An ancient mining tradition required, long time ago, that when a new mine was opened, to build in the underground a chapel or a small church. So, at Ocna Dej salt mine such a church was designed and built at about 80 m under the surface level, at mine's level +188,5 m. The church was opened for public access in 2000 (figure 7).

At Târgu Ocna salt mine operates a sanitarium, having 10 962 m^2 in surface area and 200 available places. It is built in the mined – out rooms from IInd level of Pilot mine, located at a depth of 130 m and dedicated to touristic activities and spa climatic – therapeutic treatment. There are, also, provided spaces for table–tennis, bowling games, resting and lecture rooms, playing spaces for children, etc. There also provides excellent conditions for accommodation and lounge.

Fig.8. View inside the Slănic Prahova salt mine.

The museum of salt, having a 45 m^2, is located at the I^{st} level of Pilot mine, before the main access in the sanitarium. The visitors can here admire beautiful salt samples and gather information's about salt deposit's genesis and evolution of mining.

Slănic Prahova salt mine is located at about 100 km from Bucharest, in the central-northern side of Prahova region, in a beautiful landscape, a hilly area covered by broadleaf forests. The access in the area is done both by the railway track Bucureşti-Ploieşti-Slănic, and by the national route DN 1 Bucureşti-Ploieşti.

Within the saline, the Unirea salt mine, with a 78,360 m^2 of surface area, is provided with a sanitarium for asthma sick people treatment, facilities for ping pong, volleyball, handball courts and playing courts for children.

The salt mine was opened in 1912 and consists in 14 trapezoidiform rooms, having the following characteristics: opening at the floor – 12m; wall dip – 60 degrees; opening at the bottom – 37 m and height – 66 m.

The salt mine of Turda is opened for public access, since 1992, as touristic and curative objective. The temperature inside is comprised between 10-12ºC, the relative humidity between 75-80%, and the maximum air velocity is 0,2 m/s. Inside there exists a spa facility. Here can be visited the Franz Josef gallery, the Rudolf mine (80 m

Fig.9. View inside the Praid saline.

length, 50 m width and 40 m height), the Terezia mine (112 m height) and Ghizela mine. Inside the salt mine, minigolf, bowling or boating can be practiced.

Ocnele Mari salt mine is located at 225 meters above the sea level and has a surface area of 10,000 m^2. Inside, there is a church, a museum, pubs, football, basket-ball courts and playing courts for children. Different respiratory diseases are treated inside the salt mine.

The Praid salt mine (figure 9) is located in Praid basin, in the eastern area of Transilvania, in Ghurghiu mountains, having a triangle shape, oriented towards the south direction in Corund village, on the salty structure.

CONCLUSIONS

The Romanian salt mines are disposing of adequate treatment conditions in the underground, providing sick people's protection through the facilities existing in different treatment locations, access or recreational spaces. Adding to this therapeutical factor the external environment, wealthy in salt lakes, mud deposits and salty clays, sparing bioclimate, the hilly landscape with rich broadleaf vegetation, natural or historical monuments, the nearby loated spa's, then the values expressed in therapy, rest and entertainment of salt mines are significantly increased.

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Scientific Reviewers:
Prof. PhD. Eng. Victor Arad

THE DYNAMICS OF PRODUCTION CAPACITY FOR THE JIU VALLEY MINES

EUGEN COZMA[*]
ILIE ONICA*
TUDOR GOLDAN[**]
ADRIAN STARK[***]

Abstract: *The paper proposes the coal cost analysis of the coal faces from the Jiu Valley coal mines, according to the production capacity. We will also determine the production capacity per coal-face, mine and mining methods that bring coal extraction on the brink of profitability. We will analyze the evolution of production capacities in the Jiu Valley coal basin in the last 10 years. In the end, we present the practical solutions for revitalizing the coal mines in the Jiu Valley basin (Romania).*

Key Words: *coal, mining, dynamics, production, mining filed*

1. PRESENTATION OF THE JIU VALLEY COAL BASIN

The Jiu Valley basin is situated in the inner mountain low lands of Petroşani, geographically placed in the central part of the Meridional Carpathians, between the Retezat and Sebeş mountains in the north, the Parâng Mountains in the east and the Vâlcan mountains in the west.

The basin has the shape of an asymmetrical triangular synclinal of 48 km long, 2 km western width, 9 km eastern width and a 155,5 km^2 surface (fig. 1).

Morphologically, the region is characterized by hills reaching 700-800 in height, with irregular slopes, terraces and diluvial planes, spread along the two main valleys from the region: the West Jiu Valley and the East Jiu Valley.

1.1. Mining field description

1.1.1. Lonea mining field

It's formed by the by the Lonea Pillar sector and the old mining perimeter Lonea. The mining workings have pointed out that this mining perimeter is divided in ten tectonic blocks, framed by major faults whose number and amplitude decrease from the frame to the east, causing a reduction in dip and block numbers.

* *Professor PhD. Eng. - University of Petrosani*
** *Assoc. Prof. PhD. Eng. - University of Petroşani*
*** *Student PhD. Eng. - University of Petroşani*

In the Lonea Pillar sector, the northern flank of the synclinal shows dips ranging between 25° and 40°, and the southern flank, between 10° and 35°. The major faults (out of which the northern marginal fault stands out as amplitude) divide the Lonea Pillar sector into 5 tectonic blocks, numbered from east to west, and corresponding to the blocks in the Lonea perimeter.

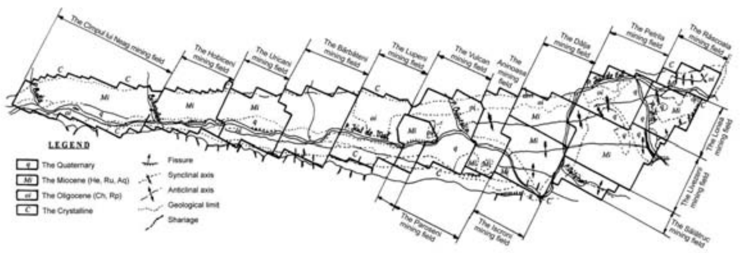

Fig.1. The geology of the Jiu Valley basin divided into mining fields

1.1.2. Petrila mining field

It belongs to the northern flank of the synclinal and as a geo-mining field presents the characteristics of an asymmetrical synclinal with its axes oriented SW-NE, having flanks with different dips. In the northern flank the coal seams have a dip of 60°-70° and in the southern flank the dip is 10°-20°. The disjunctive dislocations are longitudinal and transversal faults. The transversal faults form a conjugated system, oriented NW-SE, with dip towards south-east or north-east, that divide the perimeter in 6 mining blocks, that at their turn are fragmented by different sized lenses, through diagonal and longitudinal faults.

Within the perimeter we find the entire range of coal seams, out of which coal seam no.3, 4, 5, 6, 7, 8/9, 12 and no.13, are important from an economical point of view (fig. 2.).

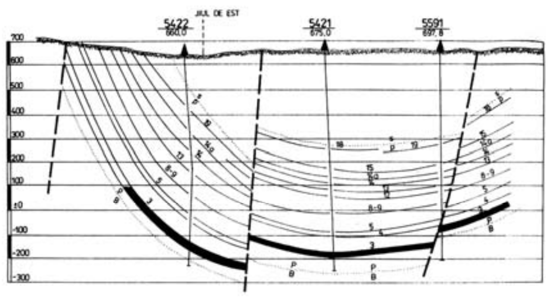

Fig.2. Petrila mining filed – transversal profile

1.1.3. Livezeni mining field

It's made out of the Livezeni and Iscroni mining fields. From a structural point of view, the Livezeni perimeter is a perianticlinal, framed by peripheral faults, screened by faults corresponding to the Maleia, Slătinioara and Sălătruc streams that divide the deposit into five tectonic blocks that gradually descend toward the north.

The dip of productive formations varies between 10° and 15°. From the coal seams of the perimeter, coal seam no. 3, 4, 5, 7, 12 and no. 13 present sedimentation continuity on the entire surface, the rest of the seams having a lens - like character with big thickness variations. Economically, important are the coal seams no. 3 and 5 that have thicknesses between 10-22 m, respectively 2-6 m, as in fig. 3.

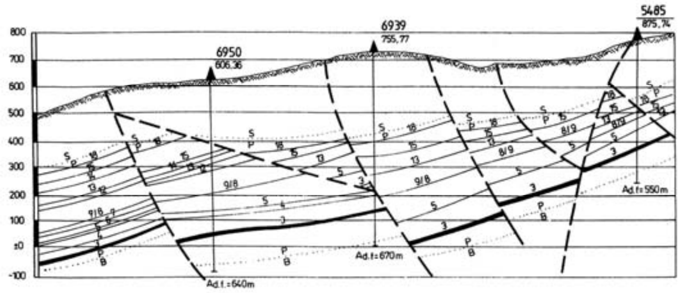

Fig.3. Livezeni mining perimeter – transversal profile.

1.1.4. Vulcan mining field

The Vulcan perimeter has an asymmetrical flank synclinal character, the northern flank being much more inclined (30°-55°), and the southern having smaller dip, with a stronger seam strike towards east-north-east and west-south-west, the characteristic of this perimeter being the gradual coal seam balking.

1.1.5. Paroşeni mining field

It's the extension towards south of the Vulcan mining field. Structurally, the northern flank has the characteristics of a perisynclinal, fractured by transversal faults, oriented north-south, screened by the West Jiu fault, that divides this perimeter into 6 tectonic blocks numbered from west-north-east with 5°-25° inclines west-south-west. The southern flank has the characteristics of a monocline, having a certain mining economical importance because of the coal seam balking phenomenon. The coal seams with economical importance in this perimeter are the no. 3, 4, 5, 8/9, 13, 15 and no.18, according to the fig. 4.

The thickness of the main coal seams varies between the following limits: 2-16m on the coal seam no.3; 1-2.5m on the seam no.4; 1.5-5.5m on the seam no.5, respectively 1.8-3m on the seam no.13, their dips being of 5°-10°.

1.1.6. Lupeni mining field

It consists of the Lupeni North perimeter, which structurally belongs to the northern flank and the axis zone of the synclinal, respectively the Lupeni South perimeter, situated in the northern flank of the synclinal. In the northern flank of the synclinal, the coal seam dip registers values between 30°- 40° (block II) up to vertical (block VII), gradually decreasing towards south, down to 4°- 8°. In the southern flank the coal seams dip reaches 50°-70°, gradually decreasing, towards the West Jiu River, down to 5°-15°.

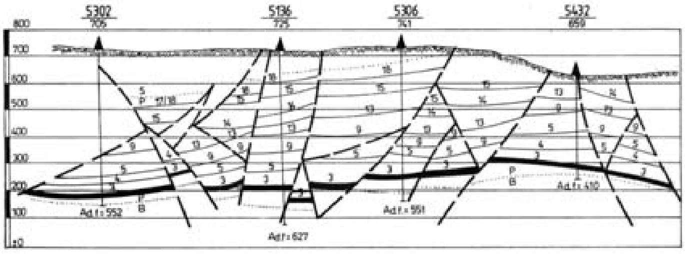

Fig.4. Paroseni Mining perimeter – transversal profile.

Structurally, the northern flank is a perisynclinal opening towards south-south-west, being affected by oriented faults, generally, in the north-north-west direction, respectively south-south-east, dividing the deposit in 7 tectonic blocks. The southern flank has the characteristics of a suspended monocline, oriented east-west, with dips towards north, screen by the reverse fault of the West Jiu river.

The exploitable coal seams are no. 3, 4, 5, 8/9, 13, 15 and no.18. The coal seam no. 3 has thicknesses ranging between 1m and 40m, and coal seam no 5, between 1.5 and 7m, like in fig.5.

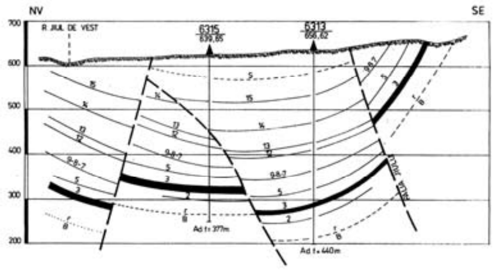

Fig.5. Lupeni mining perimeter – transversal profile.

1.1.7. Uricani mining field

It's framed to the north and south by the crystalline masses of the Meridional Carpathians. Structurally it's similar to the Bărbăteni mining perimeter. The main tectonic elements are The Northern Marginal Fault and The West Jiu Fault. The southern flank of this perimeter intercepts the coal seams no.3, 4, 5, 8/9, 13 and no.17/18, and in the northern flank, coal seams no. 14, 15 and no.17/18.

The coal seams no. 3, 5, 8/9, 13, 14, 15 and no.17/18 are being mined and the rest of the coal seams are sedimented in a lens-shape. The main coal seam no.3 has thicknesses between 2m and 19m; 1.8-5m for coal seam no.5 and 1-1.7m for coal seam no.8/9, with dips of 10°-15°.

2. ACTUAL STATE OF MINES IN THE JIU VALLEY COAL BASIN FROM THE PRODUCTION CAPACITY POINT OF VIEW

2.1. Specialized frame and scientific method applied within the research process

Contrary to the fact that at this time there are no correct and objective data statistics that could lead to an appropriate analysis of production capacities for the mines in the Jiu Valley, the research activity undertaken in this direction was based upon the actual situation of the elements characteristic to the applied mining methods, respectively the cutting technology system and the face support.

Related to these arguments, for the analysis of the annual production capacity of mines we applied the “According to Possibilities” method, using the following equation:

$$A = L_a \cdot v \cdot m \cdot \rho_a \cdot c \text{, [t/an]} \qquad (1)$$

where:

A - represents the annual production capacity, [t/an];
L_a – face length [m];
v – average face advancement speed, [m/an];
m – thickness of the extracted coal seam or slice, [m];
ρ_a – apparent specific density of coal, [t/m^3];
c – extraction coefficient.
The evolution of production in Jiu Valley mines is shown in fig. 6.

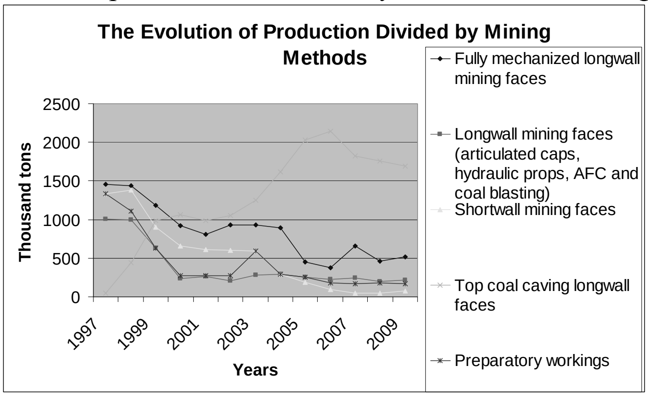

Fig.6. The evolution of production in Jiu Valley mines

The situation of mining methods presently used in Jiu Valley coal mines is shown in table.1 and the evolution of production, divided by mining methods is shown in fig.7.

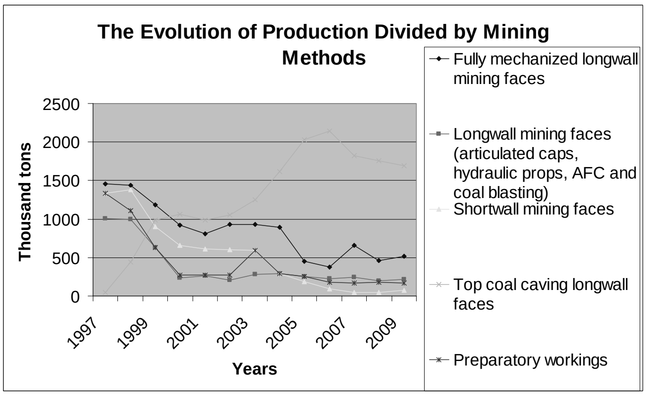

Fig.7. The Evolution of Production Divided by Mining Methods

Table 1. *The situation of mining methods used in the Jiu Valley coal basin*

Mine	**Mining method**	**Number of faces**	**Thick coal seam**
Lonea	Shortwall mining, in horizontal slices, with roof control by caving	1	**no.3**
	Middlewall mining, in horizontal slices, with roof control by caving	2	
	Middlewall mining, in horizontal slices, with top coal caving	2	
Petrila	Middlewall mining, in horizontal slices, with top coal caving	2	**no.3**
Livezeni	Longwall mining, in inclined slices, with top coal caving	1	**no.3**
	Longwall mining, in inclined slices, with roof control by caving	1	
Vulcan	Middlewall mining, in horizontal slices, with top coal caving	4	**no.3**
Paroşeni	Longwall mining, in inclined slices, with roof control by caving	1	**no.3**
	Shortwall mining, in inclined slices, with roof control by caving	2	
Lupeni	Middlewall mining, in horizontal slices, with top coal caving	2	**no.3**
	Longwall mining, in inclined slices, with roof control by caving	1	
	Middlewall mining, in horizontal slices, with top coal caving	1	
Uricani	Middlewall mining, in horizontal slices, with top coal caving	2	**no. 3, no.5**
	Longwall mining with roof control by caving, for gentle inclined coal seam	1	

3. ANNUAL PRODUCTION CAPACITY OF JIU VALLEY COAL BASIN MINES

Initially, the production capacity of mines, in the Jiu Valley coal basin, was determined by economical criteria (fig.8), meaning that the structure of cost per unit is expressed as a function of annual production capacity that is conditioned by a minimum, thus:

$$C = f(A),\ [\text{lei/t}] \qquad (2)$$

where: C is the total cost per coal tone, [lei/t]; A – production capacity of the mine, [t/year].

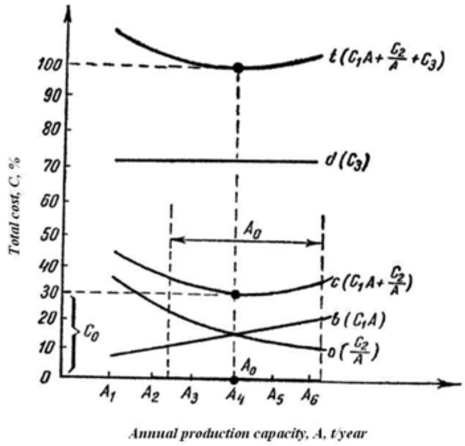

Fig.8. Graphical determination of optimal annual production capacity
a – curve of exploitation expenses; *b* – curve of break-even expenses; *c* – curve of expenses depending on the variation of annual production (*A*);); C_1, C_2, C_3 – constant expenses; *d* – constant expenses; *t* – total cost curve (*a*+*b*+*d*).

Based upon this criterion the production capacities of the Jiu Valley mines are the followings:

Table.2 The production capacities of the Jiu Valley mines

Mine	**Production capacity, in10^6 t/year**
Lonea Mine	1.20
Petrila Mine	1.80
Livezeni Mine	2.00
Vulcan Mine	1.50
Paroşeni Mine	1.00
Lupeni Mine	3.00
Uricani Mine	1.50

4. CONCLUSIONS

- The reform of mining in the Jiu Valley demands an increase of efficiency in the mining activity that implies major efforts to increase competitiveness and embracing the international technical and economical criteria;
- Comparing the designed production capacities of mines in the Jiu Valley coal basin with the achieved production, we can see a 40% - 60% difference. So, in order for the mines to reach designed production capacity, a series of measures regarding technical re-endowment must be taken;
- In order to scientifically base the increase of mine production, the production capacity of all processes and technological links will be determined and the “narrow spots” will be identified in order to increase the production capacity;

- Making the mines more efficient isn't to be achieved only by reducing the personnel costs but by introducing new technologies, and, by doing so, reaching the designed production capacity;
- An economical investment efficiency estimate is needed to determine the annual capacity of mines and the introduction of the concept of investment efficiency.

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Scientific Reviewers:
Prof. PhD. Eng. Afrim Nicolae Băncilă

EXCAVATION STABILITY EXAMINATION THROUGH ADVANCED MODELLING

TUDOR GOLDAN[*]
VASILE GRAMA[**]
CIPRIAN DANCIU[***]

Abstract: *Is presented numerical methods of analysis used in the field of rock engineering. The purpose of carrying out numerical analysis varies. It can be used to carry out qualitative analysis to understand the behaviour of rockmass or the failure mechanisms. Parametric analysis and sensitivity analysis can be carried out for comparison and better qualitative assessment. Each method involves a discretization of the problem domain, which is facilitated by a computer-assisted analysis. Three different numerical models are identified and is presented many methods for analysis.*

Keywords: *excavation, stability, modeling.*

1. INTRODUCTION

Analysis of stresses and strain, of complicated geometrical shapes of openings, intersections of tunnels with tunnels or shafts or galleries, and complex geological environment require discretization of elements and materials. These analyses are very complex and more conductive to numerical methods than performing longhand calculation. Another alternative is to perform analysis by physical or photo elastic methods. Physical modelling is very expensive and time consuming. Photo elastic modelling is becoming a dying art in face of the availability of powerful computers for numerical analysis.

The complex combination of rock mass constituents and its long history of formation make it a difficult material for mathematical representation in numerical modelling. The difficulties are basically reflected in two aspects. One is in developing constitutive models representing the true behaviour of rock mass and its engineering structures, the other is quantitative characterisation of a rock mass for computational analysis using the constitutive models. In coping with these difficulties to achieve the

* *Assoc. Prof. PhD. Eng, - University of Petroşani*
** *Assoc. Prof. PhD. Eng. - University of Chisinau, Republic Moldova*
*** *Assist. PhD. Stud.. Eng, - University of Petrosani*

best numerical representation of a physical rock engineering problem, three categories of numerical methods for rock mechanics problems have been developed and commonly used as follows [6]:

- Continuum models: Finite Element Method (FEM), Boundary Element Method (BEM), and Finite Difference Method (FDM).
- Discontinuum models: Discrete Element Method (DEM) and Discrete Fracture Network (DFN) methods.
- Hybrid continuum/discontinuum models: Hybrid FEM/BEM, Hybrid DEM/DEM, Hybrid FEM/DEM, and other hybrid models.

2. EXCAVATION STABILITY ANALYSIS

In common mining geomechanics modelling, either Mohr-Coulomb or Hoek-Brown criterion is used for excavation stability examination [4]. A limitation resulted from applying these criteria is the inability in capturing the complicated post failure behaviour of rock mass. As a matter of fact, the majority of rock mass immediately surrounding an excavation in mining environment works at its post failure portion of full stress-strain curve. Numerical simulation of this part of rock mass behaviour for mining application purpose is still impractical.

A new advance in rock mechanics modelling has been to create a numerical material which contains bound particles with stress-strain behaviour that is in good agreement with that of the prototype intact rock exhibited in lab testing and structural features that match field geotechnical structural mapping and mechanical behaviour (fig.1).

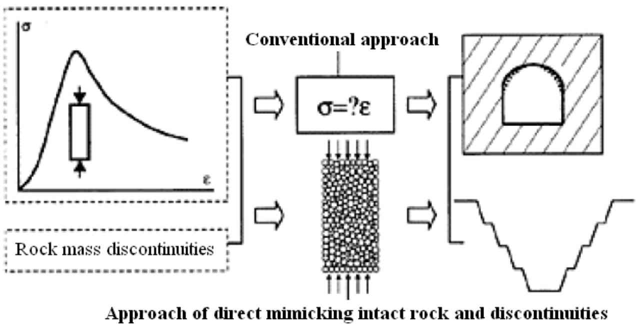

Fig.1. Different modelling approaches

Each numerical method may be used most efficiently if combined with other numerical methods [2]. The purpose of coupling individual numerical methods is typically twofold. First, the strengths of each method can be preserved while its weaknesses may be eliminated. Secondly, the combination of individual methods and their associated models can create a model that best describes the specific problem.

1.1. Finite Element Analysis and Discrete Element Analysis

Fig.2.a shows two-layer rock beam spanning a symmetrical excavation modeled with finite element analysis. The input information required begins with a computing "mesh" establishing the size and shape of the domain to be studied, which in this symmetrical example defines half of the roof. Also to be input is the set of

mechanical properties for each element of the mesh. Fig.2.b shows one part of the output – the deformed mesh – obtained from the displacements of each "nodal point", each corner of an element in this case. Note the shear deformation of the joint elements over the abutments of the beam and the opening of a gap between the layers in the center of the beam. Fig.2.c shows the state of stress in the center of each element, as revealed by vector crosses aligned and proportioned to the directions and magnitudes of the principal stresses.

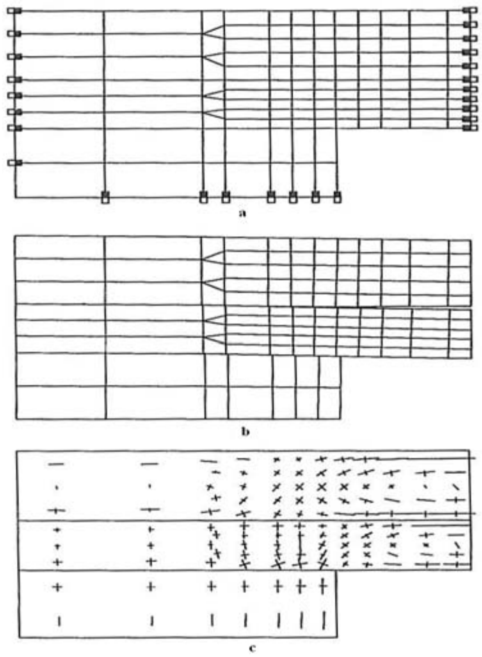

Fig.2. Results of a finite element analysis of an excavation in jointed rock: a - initial mesh; b-deformed mesh; c-stress field (from Hittinger, 1978)

The discrete (or distinct) element method is a numerical model approach with reduced degrees of freedom compared with finite element analysis. By removing deformational modes from blocks outlined by joint elements, only rigid-body modes remain. A finite difference or finite element analysis of the system can then trace the rotations and displacements of the block system as conditioned by the load/deformation relations adopted for the joints. In the distinct element programs pioneered by Cundall [1] and Voegele [8], relatively large two-dimensional block systems are calculated by integrating finite difference approximation of the equations of motion for each block, with changing boundary forces calculated at each time step from the changing block interactions.

Fig.3.a shows the initial position of the blocks. This information had to be input as well as the friction properties of each joint and the unit weight of all blocks. As the computation begins, the blocks displace and rotate under gravity and the deformed mesh can be followed through large deformations. An early stage of output is shoun in fig.3.b. The model achieved stability through arching as the program

continued to run. In contrast, a second model with joint AB rotated clockwise to a more nearly vertical position produced instability and collapse.

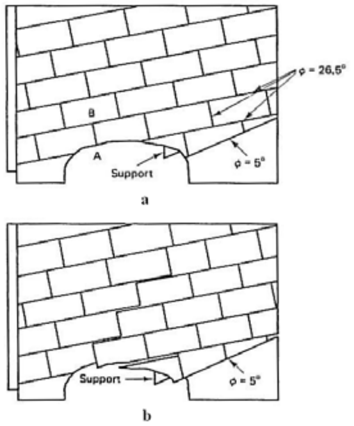

Fig.3. Distinct element analysis (Voegele, 1978): a-input information; b-output after an early stage of deformation.

Distinct element analysis is an instructive tool for excavation engineering in that it permits analysis of large block movement in geologically complex sections having many joint blocks. As with finite element analysis, it is still necessary to compute from a predetermined mesh, incorporating precise locations of all joints.

2.2. Hybrid methods

Continuum model combined with Discrete Element Method: fig.4.a shows the division of the problem domain into two areas. The far field area, away from the opening, is modeled as a continuum. The near field, i.e., close to the tunnel or galerie opening, is modeled with Discrete Elements. This reflects the anticipated ground displacement if jointed rock is encountered and movements are not restrained by support and construction measures [5].

Boundary Element Method combined with Finite Element Method: fig.4.b depicts the two areas that are analyzed differently. The purpose of surrounding the FEM with boundary elements is to eliminate the need for arbitrary and rigid boundary conditions.

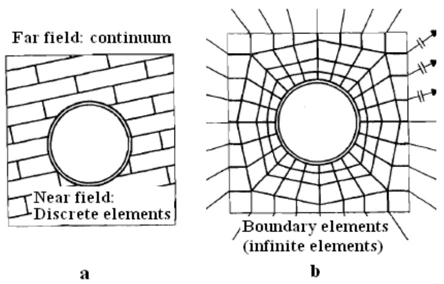

Fig.4. Hybrid methods: a – continuum model combined with discrete elements; b – Boundary Element Method combined with Finite Element Method.

Finite Element Method combined with Discrete Element Method: fig.5 shows the two computation steps, each analyzed with a different numerical approach.

In the first step, the FEM assumes a continuous ground mass around the tunnel or galerie opening. In the second step, joints are introduced forming discrete or rigid block elements along the boundary. The stresses initially calculated from the FEM analysis are used as input to the rigid block analysis [9].

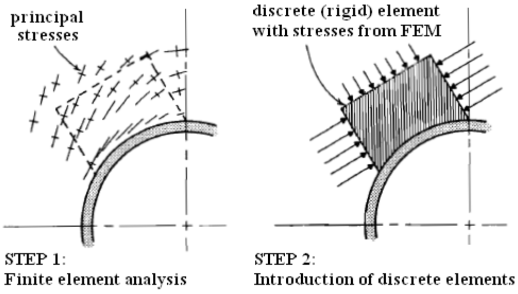

Fig.5. Hybrid method: Finite Element Method combined with Discrete Element Method.

Finite Element Method combined with Discrete Element Method are illustrated in fig.6. The FEM amployed for the first computation step analyzes the stress, strain and deformation of the ground mass including the initial or primary lining. It is assumed that the final, or secondary lining is installed at a later point of time. This lining is analyzed be means of the BEM.

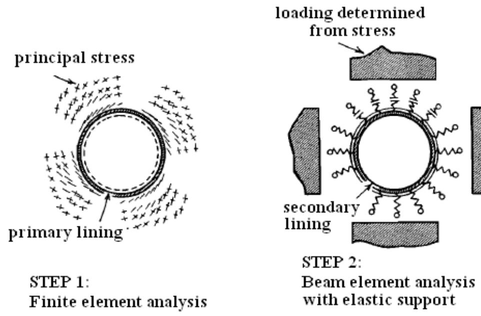

Fig.6. Hybrid method: Finite Element Method combined with Beam Element Method with elastic support.

Finite Difference Method combined with Finite Element Method: the FDM may allow a better validation of parameters that are used as input to the subsequently performed finite element computation.

Discrete Element Method combined with Boundary Element Method: the hybrid scheme DEM and time domain BEM study the effects of radiation damping of far and non-uniform mechanism on discontinuous medium with continuous far field [7]. The domain can be separated in two subdomain: a discrete elasticblocky domain modelled by DEM and an infinite homogeneous continuous domain modelled by BEM. The interface between the two domains is assured by the interface block.

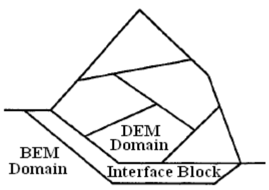

Fig.7. Hybrid method: Discrete Element Method combined with Boundary Element Method.

The boundary grid points along the interface of the interface block is called interface grid points. After the length of boundary element chosen, some interface grid points, whose distance between each other is approximate the chosen length of boundary element, are set to be the nodes of boundary element. So, the displacement of these interface grid points is same as that of boundary element nodes.

Other interface grid points will be approximate consistent with BEM domain. But the interface force is distributed to all interface grid points linearly.

3. CONCLUSIONS

Each numerical method has its advantages and disadvantages. The suitability and applicability of a numerical method must be ascertained for each individual case and on the objective of the study. The combination of individual methods and their associated models can create a model that best describes the specific problem.

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Scientific Reviewers:
Prof. PhD. Eng. Victor Arad

TECHNICAL, TECHNOLOGICAL AND GEOTECHNICAL FACTORS WHICH DETERMINED THE OCCURRENCE OF EXCESS WIDENINGS IN THE CONSTRUCTION OF THE MAIN WATER ADDUCTION AT HYDROPOWER PLANNING OF THE JIU RIVER AND THEIR ECONOMIC CONSEQUENCES

SEMEN CONSTANTIN [*]
SEMEN MANUEL-VIOREL [**]

Abstract: *The objective of this study is to identify end evaluate various causes that led to the occurrence of excess widening when finishing the mining works at a hydropower planning. In order to achieve this goal we adopted specific research directions like the analysis of the geology structure in the area, and of the blasting pattern. The surfaces that have been dislocated where evaluated under processing the topographic measurements in correlation with the rocks characteristics. On this base we identified two groups of causes (technological and geotechnical), depending or not on the rock characteristics. The processing of the database obtained in measurements allowed us to find the relationship between the value of the excess widening and the rock characteristics.*

1. INTRODUCTION

The construction of 14 km from the main water adduction Livezeni-Bumbesti determined the occurrence of excess widening which lead to additional consumption of concrete. The goal of the study was to identify the causes of widening, to evaluate the share of these causes – technical, technological, human, and natural – which involved additional costs for the final lining because of the increased necessary volume of concrete (Figure 1).

In order to achieve its goal, the study approaches the regional geological framework of the hydropower planning, the categorization of the traversed rocks in the geo-mechanical classification related to the geotechnical indexes, the analysis of the blasting patterns used on various locations according to the characteristics of the rocks, the evaluation of the volume of profile deviances on locations and types of rocks and based on these the identification of causes which led to the occurrence of profile deviances and respectively their share in the overall widening.

* *Prof. PhD. Eng. - University of Petroşani*
** *Lecturer PhD .Eng. – University of Petroşani*

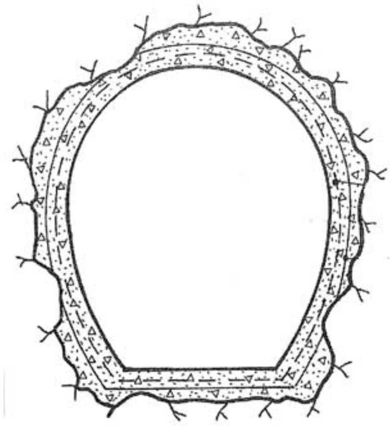

Fig 1. Influence of widening on the increased consumption of concrete

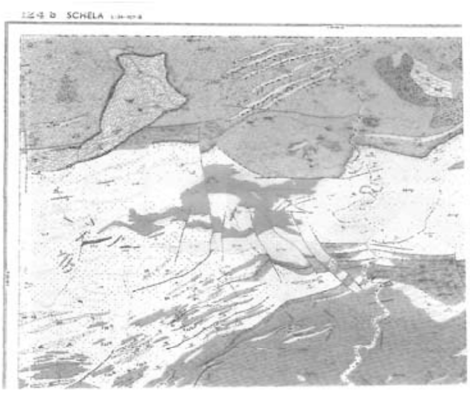

Fig. 2: The geological structure Bumbesti-Livezeni

2. THE REGIONAL GEOLOGICAL FRAMEWORK OF THE HYDROPOWER PLANNING

The main water adduction of the hydropower planning of the Jiu River in the Livezeni-Bumbesti sector is geographically located on the southern slope of the Vâlcan Mountains whose petrography consists of mica schist generated by geo-tectonic pre-hercynical and hercynical cycles and of pre-hercynical granitoids (Figure 2).

In the Danube Autochthon the mica schists are widely spread, being generated under a regional metamorphism of a medium and low geosyncline; this determined their classification into two groups depending on their metamorphisation degree:

- mesometamorphic mica schist – medium degree metamorphism represented by the Dragsani and Lainici-Paius series;
- epimetamorphic mica schist – low degree metamorphism represented by the Rafaila series.

The magmatic rocks intercepted by the hydropower planning are represented by pre-hercynical granitoids of Susita-type.

The Dragsani series – considered a volcanic-sedimentary formation – displays a great variety of rocks: amphibolites, hornblendites, rubanated amphibolites, amphibolitic gneiss, paragneiss rocks, amphibolitic schist and crystallized limestone.

This rocks complex is forming the low level within the series, and in the upper side shows up a series of clastic rocks represented by sericitic and cloritous schist, claritic and quartzous schist, sericitic and graphitic schist.

The Lainici-Paius series includes a broad series of metamorphites-representing volcanogenic metamorphosed formations under the conditions of the amphibolitic facies and under the presence in the close range of granitoids intrusions, being characterized by paragneiss, mica schist, quartzitic schist, quartzite with biotite, feldspatic quartzite, limestone and crystallized dolomites with inclusions of graphitic schist.

The Rafaila series is represented by weakly metamorphosed sedimentary rocks petrographically identified as metaconglomerates: graphite, metapelite, graphitic schist with chloritoid, graphitic schist and limestone intercalations.

The magma rocks concerned by the hydropower planning occur on large surfaces and are represented by the Şuşiţa series with normal granites, porphyric granites, and granodiorites.

The typical geological structure of the Vâlcan Mountains shows complex, plicative and ruptural tectonics, normal and inverse fissures. More important and numerous are the inverse fissures dividing the geological formations into blocks. On the fissure plane the rocks are strongly molded, affecting especially the Lainici-Păiuş series, such as also the tectonic contact between the Drăgşani and Lainici-Păiuş series.

Depending on the physical and mechanical characteristics of the rocks and on their cracking and deterioration degree, the ISPH project for the hydropower planning of the Jiu River assessed the existence of 5 classes of rocks (Table 1).

Table 1: The geo- mechanical classification of the rock massif

Rock category	RMR	$E \cdot 10^3$ [daN/cm^2]	Q	F	Representative rocks
A (FF)	> 80	> 60	> 20	> 8	granitoid, mica quartzite
B (T)	40 - 80	40 – 50	10 – 20	6 – 8	gneiss, mica schist
C (ST)	20 – 40	20 – 30	1 – 10	4 – 6	amphibolite
D (M)	10 – 20	10 – 20	0,05 – 1	2 – 4	cloritous and feldspatic schist
Ds (FM)	<10	< 10	< 0,05	1	Strongly tectonized graphitous, sericitic and cloritous schist

3. THE ANALYSIS OF THE APPLIED BLASTING PATTERNS

In order to identify the causes which led to excess widening occurring, the blasting patterns were analyzed under consideration of the following aspects:

- type of used explosives;
- value of drilling and blasting parameters (specific explosive consumption, number of blasting holes, total explosive quantity and quantity per blasting hole) in correlation to the characteristics of rocks;
- positioning of the blasting holes;
- explosion initiation mechanisms and the ignition order of the explosive loads;
- the manner of applying the designed blasting patterns.

The analysis of the blasting patterns was performed for each location: Murga Mică, Dumitra, Valea Rea, Bratcu, because the rocks types and the magnitudes of the transversal section of the mining work is different.

A first result was the fact that specific blasting patterns were designed for each location under the participation of experts from companies like Austin Powder Austria and UEE Spain and from the University of Petrosani, by using various types of explosives, initiation mechanisms, and explosives quantities correlated to the rocks characteristics.

The designed and applied blasting patterns allowed the achieving of various specific explosives consumptions differentiated on types of rocks, a judicious number of blasting holes, a right positioning and ignition order. The parameters of the blasting patterns depending on the types of rocks and on the locations are shown in Table 2, and the positioning of blasting holes is shown in Figure 3.

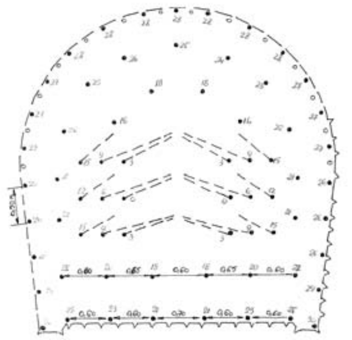

Fig. 3a: Positioning of the blasting holes for very hard types of rocks

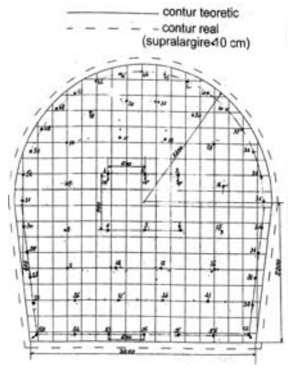

Fig. 3b: Positioning of the blasting holes for hard types of rocks

The analysis of data from Table 2 leads to the following conclusions:

- the explosive type and the explosion initiation mechanisms were correctly chosen:
- the parameters of the blasting patterns match to the explosives characteristics, to the section of the mining work and to the rocks characteristics;
- the evaluation of the compliance degree in the length of blasting holes, in the number of holes and in their role in the dislocation process, in the distance between the holes, in the total and specific per hole quantity of explosives shows their correct application.

All these allow us to assess that the design and the application of the blasting pattern was not responsible for the occurrence of the excess widening.

4. THE EVALUATION OF THE EXCESS PROFILE VOLUMES DURING THE EXECUTION OF THE JIU RIVER HYDROPOWER PLANNING

The evaluation of the excess profile volumes caused by blasting was based on the topographic measurements made in the 3 construction sites involved in the completing of the main water adduction. The purpose of the topographic measurements made after each blast was to determine the various distances R_1, R_2 ... from a fixed point, the equation for the surface resulted from the blasting work being:

$$S_r = \frac{\pi \cdot R\omega^2}{2} + \frac{R_1 + R_2 + R_6 + R_7}{2} R_8 \quad (m^2) \qquad (1)$$

Where:

$$R_{rm} = \frac{R_1 + R_2 + R_6 + R_7}{2} \quad (m) \qquad (2)$$

Table 2: Specific parameters of the applied blasting patterns

Parameters	MU	Value for various locations and types of rocks														
		Murga Mică			Dumitra				Valea Rea				Bratcu			
		A	B	C	A	B	C	D-Ds	A	B	C	D	A	B	C	D
Section	m^2	18,4	18,4	18,4	20,09	20,09	20,09	20,09	19,92	19,92	19,92	19,92	19,9	19,9	19,9	19,9
Hole length	m	2,2	2,2	2,2	2,3	2,3	2,3	1,5-1,8	3	3	2	1,6	3	3	2	1,5
Explosives	-	Goma 2 ECO			Goma 2 ECO				Goma 2 ECO Riocord 100-contur				Goma 2 ECO Riocord 100-contur			
Initiation mechanisms	-	Riodet I CEP – 0,5 cu			Riodet I CEP – 0,5 cu				Riodet I CEP – 0,5 cu				Riodet I CEP – 0,5 cu			
Total number of holes:	buc.	65	53	47	65	52	43	28-35	67	52	43	32-40	68	55	43	37
Explosive quantity	kg	72	60	43	65	55	40	Q30	87	62	52	42	120	93	72	35
Specific consumption for explosives	kg/m^3	1,8	1,5	1,1	1,5	1,2	0,9	0,7	1,5	1,1	0,9	0,7	2	1,6	1,4	1,1
Advance	m	2	2	2	2	2	2	1,2-1,5	2,5	2,5	1,7	1,2	2,5	2,5	1,7	1,2

The processing of the topographic measurements allowed the determination of the excavated surface, and the value of the excess profiles was determined by comparison with the theoretical profile (or with the theoretical profile plus 10 cm):

$$\Delta S = S_R - S_{T+10} \tag{3}$$

- If $\Delta S<0$, underprofiles resulted, which can be acceptable within certain limits; $|\Delta S| < 2\ m^2$;

- If $\Delta S>0$, excess profiles resulted.

Transversal profiles were traced out for each topographic site, thus obtaining all the necessary elements for computations (Figure 4).

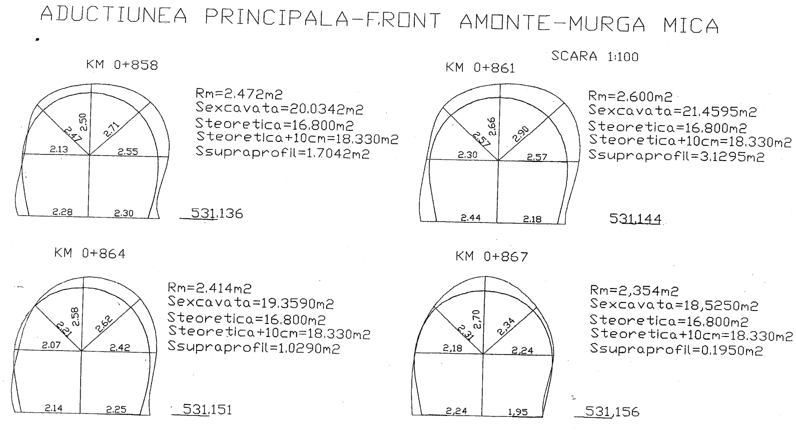

Fig. 4: Transversal topographic profiles

The evaluation of excess profiles was made for the 4 locations (Murga Mică, Dumitra, Bratcu, Valea Rea) upstream and downstream under considering the rock type.

The average value of the found excess profiles for each location and rock type (S_m) was:

$$S_m = \frac{\sum s_i \cdot L_i}{\sum L_i} \quad (m^2) \qquad (4)$$

s_i – excess profile surface in the topographic site i;
L_i – distance between the topographic stations.

For each location the surface of the front excess profiles downstream and upstream was determined, and this allowed the obtaining of a centralizing situation of excess profiles for various rocks types and locations (Table 3).

Table 3: Centralization of the excess profiles for various rocks types and locations

Location / Rock type	Murga Mică				Dumitra				Valea Rea				Bratcu			
	Downstream		Upstream		Downstream		Upstream		Downstream		Upstream		Downstream		Upstream	
	m^2	cm	m^2	cm	m^2	cm	m^2	cm	m^2	cm	m^2	cm	m^2	cm	m^2	cm
A	0,5	13,6	0,54	13,5	0,43	11,1	0,60	14,6	0,54	12,4	-	-	-	-	-	-
B	1,16	17,9	1,28	8,3	1,04	17,3	1,15	16,9	1,14	16,7	1,15	16,7	1,08	16,3	1,58	19
C	-	-	-	-	1,52	18,8	1,51	18,8	1,37	18	1,89	21	1,58	19,2	1,80	20,5
D	-	-	-	-	2,24	21,5	-	-	-	-	2,48	24,3	-	-	2,46	24,2
Ds	-	-	-	-	3,86	28,9	3,08	24,5	-	-	-	-	-	-	3,31	28,9

The complex process of rocks dislocation is influenced by a large number of factors grouped into three categories:

- Natural factors represented by the rocks type, their mechanical characteristics, degree of compactness and fissuring, the existence of schisting planes and their orientation compared to the mining work – transversal, diagonal or directional;
- Technical factors represented by explosives type, its quality expressed by the thermodynamic parameters, total explosives quantity and for each blasting hole, the location pattern of the blasting holes, the ignition order of explosive loads;
- Technological factors represented by the type of the drilling machine, maneuverability of the drilling arms, the compliance with the designed blasting pattern, explosive quantity and ignition order.

5. THE DETERMINATION OF CAUSES WHICH LED TO THE OCCURRENCE OF EXCESS PROFILES AND THEIR SHARE

In order to identify the causes generating excess profiles a profound analysis was performed, involving the geological structure of the traversed rocks, the applied blasting patterns and the manner of their application, the whole technological process; all these led to the identification of two groups of causes:

1. Technological causes determined by:
   The technological limitations of the drilling machine;
   The dexterity of the operator – a human factor;
   The blasting technology.
2. Geotechnical causes

5.1.1. The technological causes

They were determined mainly by the technological limitations of the drilling machine and respectively by the maneuverability of the drilling arms. In all the locations were used highly capable electro-hydraulic equipments TAMROCK AXERA or ATLAS COPCO with high drilling speed. The obtaining of a good profile shape was conditioned by placing the blasting holes at 10 cm distance inside the work front, and the bottom of the blasting hole hat to be placed 10 cm outside of the profile shape, which means an look-out angle of 4^0, called by the constructor the „theoretical" direction (Figure 5).

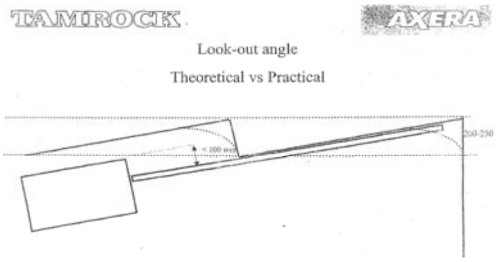

Fig. 5: The variation of the look-out angle of the profile shape blasting holes depending on the maneuverability of the drilling machine

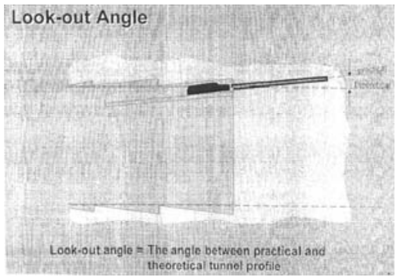

Fig. 6: Excess widening generated by the maneuverability of the drilling machine

This angle cannot be actually achieved, because the maneuverability of the drilling arms causes an exterior distance of 200-250 mm (Figure 6) and a minimum angle on the „real" direction of 7^0. Under these circumstances the excess profile caused by the minimum angle is estimated at 0,42 m^2 in excess to the acceptable profile shape.

5.1.2. Causes generated by the human factor

The observations of the mining hole bottom along the profile shape showed local deviances compared to the acceptable profile, the look-out angle being larger than 7^0 – this phenomenon was found at about 10% of the total work perimeter. These excess distances estimated at 250-300 mm are explained by the position of the drilling operator 8 m away from the front, and under the underground visibility conditions the right positioning of the drilling arms is very difficult. The value of the excess widening generated by this factor was estimated at 0,1-0,18 m^2.

5.1.3. Causes generated by the blasting technology

These causes are determined by the explosives type and quantity and by the correct design and application of the blasting patterns. The detonation of the explosive loads causes a spherical dynamic shockwave and a grid of fissures in the rock massif. A high degree of fissuring may be useful in the central area of the excavation for an easy dislocation of the necessary rock volume, but it may be damaging for the holes on the profile shape, causing excess widening. The classic blasting leads to rocks fissuring on a distance around the underground excavation (estimated at 30-35 cm for very hard rocks, 47-55 cm for hard rocks and 80-110 cm for schist and stratified rocks). Using a smooth blasting, the fissuring degree can be reduced 4-6 times depending on the rocks hardness.

Because of this reason we estimate that the acceptable excess widening of 10 cm imposed by the frame project made by ISPH was ungrounded; in most of the countries this value ranges around 6-30 cm, depending on the rock hardness.

The analysis of the blasting patterns showed the following aspects:

- the distance between the shaping holes of 0,4-0,6 m was rational;
- the used explosives were low-brisant (Lambrex contour or detonating cord with 100 g/m, explosives with low impedance factor of 0,785 compared to 0,983 for dynamite);
- the used explosives were low-diameter ones (much lower that the diameter of the blasting hole), attaining low loading coefficients (0,126-0,23) with direct consequences on the fissuring degree of rocks.

From the facts mentioned above one can conclude that there is a right correlation between the used explosives, their diameter, the diameter of the blasting hole and the rocks type, so that the blasting activity cannot cause excess widening.

5.2. Geotechnical causes

The massif where the mining works are located is very molded by orogenic processes. As a result the rocks were submitted to strong regional metamorphism phenomena, large scale schisting, discontinuities systems with a very high density (7-12 fissures/m), sometimes mixed with clay material.

By analyzing the resulted blasting profile shape one can clearly distinguish the dislocation surfaces caused by explosion from the ones generated by the massif fissuring. Those generated by explosion are irregular and coarse, with freshly broken crystals. Those generated by existing fissuring are slick and glossy. In most of the cases these surfaces are not orientated in the profile shape direction, but in the direction of the fissure planes, generating an irregular contour.

The fissuring and tectonization degree of the massif demanded the elimination of the unstable blocks. The existence of the schisting and fissuration planes intersecting in the vicinity of the contour makes the removal of these blocks necessary, leading to an increased excess widening.

The presented aspects can be synthesized in an evaluation of the share of the various causes depending on the rocks categories (Table 4) by using the values in Table 3, and the relationship between the value of excess widening and the rocks hardness coefficient (RMR) is shown in Figure 7.

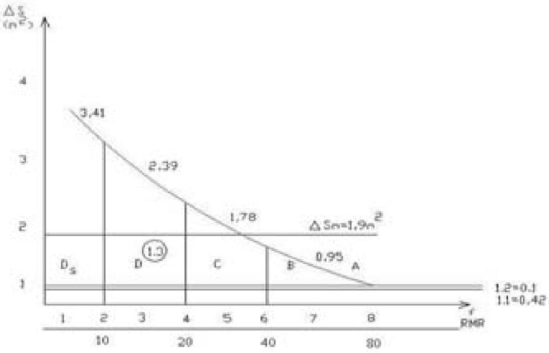

Fig. 7: The dependence of excess widening on the rock hardness and on its quality index

Table 4: The evaluation of the share of the causes for various rocks categories and locations

Location and rock type		**S_T (m^2)**	**S_T+10 cm (m^2)**	**Excess profile (m^2)**	**Value of excess profiles (m^2, %) depending on causes**		
					Drilling machine	**Human factor**	**Geotechnical nature**
Murga Mică Downstream	A	16,8	18,33	0,50	0,42 (84,0 %)	0,08 (16,0 %)	- -
	B	16,8	18,33	1,16	0,42 (36,3 %)	0,12 (10,5 %)	0,62 (53 %)
Murga Mică Upstream	A	16,8	18,33	0,54	0,42 (78,0 %)	0,12 (22,0 %)	- -
	B	16,8	18,33	1,28	0,42 (38,8 %)	0,12 (11,2 %)	0,74 (50 %)
Dumitra Downstream	A	18,89	20.09	0,43	0,42 (100 %)	- -	- -
	B	18,89	20.09	1,04	0,42 (40,4 %)	0,2 (19,2 %)	0,42 (40,4 %)
	C	18,89	20.09	1,52	0,42 (27,6 %)	0,2 (13,1 %)	0,9 (59,2 %)
	D	18,89	20.09	2,24	0,42 (18,7 %)	0,2 (9,2 %)	1,62 (72,3 %)
	Ds	18,89	20.09	3,86	0,42 (10,9 %)	0,2 (5,2 %)	3,24 (83,9 %)
Dumitra Upstream	A	18,89	20.09	0,60	0,42 (70,0 %)	0,2 (30 %)	- -
	B	18,89	20.09	1,15	0,42 (36,5 %)	0,2 (17,4 %)	0,53 (46 %)
	Ds	18,89	20.09	3,08	0,42 (13,6 %)	0,2 (10,0 %)	2,36 (76,4 %)
Valea Rea Downstream	A	18,27	19,92	0,54	0,42 (77,7 %)	0,1 (22,3 %)	- -
	B	18,27	19,92	1,14	0,42 (36,6 %)	0,2 (17,5 %)	0,52 (45,9 %)
	C	18,27	19,92	1,37	0,42 (30,6 %)	0,2 (14,6 %)	0,75 (54,8 %)
Valea Rea Upstream	B	18,27	19,92	1,15	0,42 (36,5 %)	0,2 (17,4 %)	0,53 46,1 %)
	C	18,27	19,92	1,89	0,42 (22,2 %)	0,2 (10,6 %)	1,27 67,2 %)
	D	18,27	19,92	2,48	0,42 (16,9 %)	0,2 (8,1 %)	1,86 (75 %)
Bratcu Downstream	B	18,27	19,92	1,08	0,42 (38,9 %)	0,2 (18,5 %)	0,46 (42,6 %)
	C	18,27	19,92	1,58	0,42 (27,6 %)	0,2 (13,7 %)	0,90 (59,3 %)
Bratcu Upstream	B	18,27	19,92	1,58	0,42 (26,6 %)	0,2 (12,6 %)	1,0 (63,3 %)
	C	18,27	19,92	1,80	0,42 (23,3 %)	0,2 (11 %)	1,18 (65,7 %)
	D	18,27	19,92	2,46	0,42 (17 %)	0,2 (8,1 %)	1,84 (74,8 %)
	Ds	18,27	19,92	3,31	0,42 (12,7 %)	0,2 (6,0 %)	2,69 (81,3 %)
AVERAGE VALUES	A	-	-	0,52	0,42 (80,4 %)	0,1 (19,6 %)	- -
	B	-	-	0,95	0,42 (44 %)	0,18 (18,9 %)	0,80 (36,8 %)
	C	-	-	1,78	0,42 (23,5 %)	0,18 (10 %)	1,18 (66,3 %)
	D	-	-	2,39	0,42 (17,5 %)	0,18 (7,5 %)	1,79 (75 %)
	Ds	-	-	3,41	0,42 (12,3 %)	0,18 (5,2 %)	2,81 (82,4 %)
Average excess widening				1,9	0,42 (22,1 %)	0,16 (8,4 %)	1,32(69,5 %)

The statistical processing using the linear regression allowed us to establish the dependence ΔS=f(f), ΔS=f(RMR), the regression equations being:

$$\Delta S = 3{,}68 - 0{,}37 \cdot f \quad \text{for } f < 8$$
$$\Delta S = 0{,}52 \quad \text{for } f \geq 8 \qquad (5)$$

$$\Delta S = 3{,}32 - 0{,}035 \cdot RMR, \quad \text{for } RMR < 80$$
$$\Delta S = 0{,}52 \quad \text{for } RMR \geq 80 \qquad (6)$$

The obtained correlation coefficients r = 0,96 for ΔS=f(f) and 0,93 for ΔS=f(RMR) show a good correlation between the analyzed parameters. The processing of data was performed both under considering the three causes generating excess widening and with the geotechnical causes alone, because the technological and human causes are not influenced by the rocks type and characteristics, the generated excess widening being constant ΔS=0,52 m^2; the equations for the geotechnical causes alone are:

$$\Delta S = 3{,}15 - 0{,}37 \cdot f \quad \text{for } f < 8$$
$$\Delta S = 0 \quad \text{for } f \geq 8 \qquad (7)$$

$$\Delta S = 2{,}80 - 0{,}035 \cdot RMR, \quad \text{for } RMR < 80$$
$$\Delta S = 0 \quad \text{for } RMR \geq 80, \tag{8}$$

With the correlation coefficients being r = 0,92 and respectively 0,94.

6. THE ECONOMIC CONSEQUENCES OF EXCESS WIDENING

The generated excess widening during the dislocation process out of various causes (technical, technological, human and geotechnical) led to an increased construction cost during the concreting phase of the main water adduction.

The average values of excess widening determined for five rocks categories ranged between 0,5 m and 3,31 m^2.

Taking the fact into account that the main water adduction has a length of 22 km and the weighted average of the excess widening for various rocks categories and traversed distances was 1,9 m^2, the result was an additional concrete consumption of 41.800 m^3, which led to an increase of the concreting costs by 42%. It must be mentioned that the possible excess widening for various rocks categories must be estimated during the elaboration phase of the frame project, and the additional costs must be considered in the general construction costs calculation. The increase of the concrete consumption compared to the initially estimated quantity leads to an increased construction cost for the main water adduction by 9 millions euros; this additional cost will be supported from the position of various and unanticipated costs (limited to 5%) which must be already taken into account in the project.

7. CONCLUSIONS

The purpose of this study was to evaluate the causes leading to the occurrence of excess profiles during the construction of the main water adduction at the Jiu River hydropower planning.

The analysis of the geological structures traversed by the adduction shows a complex structuring characterized by the presence of various categories of metamorphic rocks specific for the Drăgşani, Lainici-Păius and Rafaila series and magmatic rocks – the Susita granodiorites. The area is very tectonically affected with an advanced fissuring degree, distinguishing between a plicative and a ruptural tectonics, large scale or micro folding, normal and inverse fissures.

In order to evaluate the causes we analyzed the applied blasting patterns, their manner of application, and the result was that they were correctly designed and applied and the occurrence of excess profiles was not caused by the blasting technology.

The evaluation of the magnitude of excessively dislocated surfaces was made based on topographic measurements in the 4 locations, correlating these excess profiles with the type of rocks. By studying the technological process and by correlating the excess profiles with the rock category, 2 types of causes were found:

- Technological causes:
    - Technological limitations of the drilling machine;
    - Causes generated by the human factor

- Geotechnical causes like the rocks type and characteristics, their fissuring and deterioration degree, the existence and orientation of the schisting planes.

The share of the 2 categories of causes depends mainly on the rocks characteristics. Thus, in case of the category A rocks (f>8) the technical causes show up with a share of 100% (80,4% because of the drilling machine and 19,6% the human factor); these share diminish with the reduction of the rocks resistance up to 18,5% in case of the DS rocks, the main part (82,4%) being represented by geotechnical causes.

Statistical processing and the use of the double correlation method allowed us to establish the relationship ΔS=f(f) or ΔS=f(RMR) and to obtain very good values of the correlation indexes, 0,93 and 0,96.

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Scientific Reviewers:
Prof. PhD. Eng. Ilie Onica

REINFORCING SOLUTIONS ON THE AREA AFFECTED BY LANDSLIDES FOR THE ACCESS ROAD OVERFLOWING SURGE– SĂSCIORI

MIHAELA TODERAS*

Abstract: *Consequently to the heavy rainfalls during the year of 2005 and at the beginning of 2006, a significant section of the access road towards the Săsciori overflowing surge was affected by landslide phenomena, upwards from the road's platform. Anyway, in spite of these landslides, the observations carried out have revealed that the road did not altered his initial characteristics, the platform width remaining, even after the phenomenon occurrence, at the same value as before, namely 3.5 m. The main issue was imposed by the need for an immediate rebuilding the access road to the valves house Săsciori, on 40 m length of the affected area.*

Key words: *landslide, reinforcement, abutment wall consolidation, catchment's drain, weeper.*

1. GENERAL DATA

The landslide has occurred in the spring season of year 2006, as a consequence of heavy and long-lasting rainfalls, so, while the culvert was clogged, the rainfall water have overflowed the embankment, in the filling area, carrying-over a part of the concrete mantle's ground and the lunette located upwards the existing tubular footbridge. In order to achieve this goal, surveying measurements were done, carried out at the 1:200 scales and, also, a geological and geotechnical characterization of the area affected by the landslide was required. According to the geotechnical study which was done for this working, it was drawn the conclusion that, at the refilling level for the road development, the landslide was shallow. This construction is framed, according to HGR 766/1995 in the category of global importance of designed workings D, namely having low importance.

* *Assoc. Prof. Ph.D. Eng. at the University of Petrosani, toderasmihaela@yahoo.com*

2. THE DESRIPTION OF THE TECHNICAL SOLUTION EMPLOYED FOR THE CONSTRUCTION REINFORCEMENT

In order to provide the getting back to a normal access in safe conditions, there was envisaged the refilling of the rockfill in the road's body and the execution of some supporting workings based on concrete abutment walls.

With a view to faster drainage water evacuation from this perimeter, it was proposed the development of two culverts in the refilling mass, discharged through the abutment wall's weepers. To ensure the proper surface water gathering and evacuation through already existing workings, the alternative was selected to clean out the catchment's drain, to declogge the existing culvert and the footbridge Φ 300 and to restore the lunette destroyed by the landslide. The only feasible solution aimed at diverting the water flow downward the footbridge consisted in resorting to a trapezoid form step-up ditch, supported with cement masonry, having the thickness of 20 cm. For water drainage and evacuation purposes, two elastic raft catchments drains will be constructed, having 1.20 m width and 3 – 5 m height, which will collect the rainfall water through a punched drainage pipe, having 80 mm in diameter and water will be discharged through the abutment wall's weepers. On the affected area, it also was needed to restore the concrete superstructure of the roadway and to mount a metallic railing, at the roadway platform boundary. The railing will be deformable, semi – heavy type with metallic ledge, on the entire section of about 60 m of length, but on the last 9 m section the railing will be mounted in continuous C 12/15 concrete grounding.

To ensure the access in the roadway territory, it was imperatively required the development of a secondary, temporary road, having 250 m of length and 3 m in width, which is practically a downward branch of the existing road. The abutment walls were designed to be built from C 12/16 concrete at the remblai bottom size, with 29 m length, elevation of 3.5 m height and, at the limits of road's platform a 2 – 2.5 m of elevation height on 10 m length. The wall is comprising weepers aimed at refilling and drains-collected water discharge and evacuation, made of pipes Φ 110 mm. The access to the Săsciori overflowing surge is included in 3rd class of exploitation roads, with roadway bench of 2.75 m width and two verges of 0.75 m. The verges were designed to be non – permeable, employing a layer consisting in 20 cm of ballast, stabilized with 6% cement. Before starting the refilling operations in the roadway remblai, the backfilling unproperly slipping material will be removed and linking benches will be digged in the natural soil. The refilling will be completed from local drenant material. Simultaneously with the refilling operations, the drains will be built on the selected locations. For the working development stage there were established also the prevention measures required in order to provide the diminishment of the negative environmental impact, in such a manner that when realizing the infrastructure workings, the potential contamination of surface water flows, underground water and lakes to be avoided. Regarding the protection of terrestrial and aquatic eco – systems, it was not imposed, nor required a special dedicated protection program for eco – systems, bio – diversity and nature preservation. By the adopted technical solution itself, the working and then the use of it, did not induced a negative environmental effect and, consequently, it was no need for achieving specially dedicated ecological restoration and land reclamation workings, the temporarily occupied areas being given back into the natural circuit through special dismantling works at operations cessation. The consolidation of this construction assumes to cover the following necessary steps: the grounding area checking; the embankment digging operations; realization of the route structure with cement concrete mantle; abutment walls, drains and tubular footbridges construction.

3. GROUNDING AREA VERIFICATION. EMBANKMENTS

Before starting any kind of operation, it is required to carry out a geo – technical study. For the case presented in this paper, after taking off the vegetal soil layer and ground soil compacting, it was assessed the compactation degree and the deformability of the grounding land. For this purpose, 3 test trials were carried out for the determination of compactity degree (STAS 2914-84), for each 2,000 m^2 of compacted surface. The grounding land deformability was assessed by measurements, done employing the deflectometer with lever. The measurements with the cross – sectional profile deflectometer were located at maximum 25 m one from another, in 3 points on the road, namely: right, axis and left. At the grounding land's level it is considered that the bearing capacity is reached if the elastic deformation at the test sample has higher values than the allowable value, in at the most 10 % of the measuring locations.

The checking regarding the road bank compactity degree in the bed was done in correlation with the deflectometer measurements, in the points where the results have indicated low bearing capacity values. For quality and state of the material used for refilling purposes, the following characteristics were experimentally determined: granulometry, plasticity limits and compactity characteristics determined with the normal Proctor test. It was considered that the soil moisture content can be measured on daily basis, or at each 500 m^3 of operated material. The achieved compactity degree verification was done by extracting 30 cm deep samples for each elementary material layer, in 3 points evenly distributed at each 2000 m^2 of layer. The check of the compactity degree was carried out only in the cases when there were noticed significant level variations in layer thickness, as a consequence of heavy truck passage during operations. The workings at the road bank are started immediately after the preparatory preliminary operations are brought to completion. The road axis is re – established together with the landmarks for the tracing elements; these landmarks are located outside the working area, being kept and used to materialize the road axis and levels. The preparatory workings are consisting in realizing the road territory, resulted material deposition in waste dumps or other kind of deposits, decapation of vegetal soil layer on a depth of 30 cm. The vegetal soil will be used in order to achieve the valorization of certain lands which will remain non-productive after the work operation cessation and, also, for slopes revegetation. In the road sections where surface water can flow to the roads remblai, these are diverted through catchments drains, which are collecting and discharging the overflow water outside the road territory. Catchments are to be digged before starting the road digging operation.

When higher than 30 cm refilling layers are requested, at their bottom, stone or concrete blocks with sizes of 0.50 m are used, respecting the conditions regarding the filling of cavities and achieving an adequate settlement and homogenous filling, having at least 2 m thickness at the remblai’s upper side. In the refilling, the material will be disposed in uniform and parallel layers to the road axis, on the entire width, the compactation operation being done in successive 15 – 20 cm thick layers. When compact the last layer, the road’s platform slope should be 4 %; the remblai slope’s dip will be 1:1.5.

4. ROADWAY STRUCTURE WITH CEMENT CONCRETE MANTLE

The rigid roadway system corresponds to a low traffic class and has the following structure: 20 cm of concrete BcR 4.0 and 30 cm ballast. The road’s foundation is represented by the layer within the road system which is transmitting to the road bed the vertical strains induced by traffic, in such a way that it does not exceeds the carrying capacity, in the most unfavorable conditions which can appear in the perspective period considered in the design stage. The foundation layer consists in sorted ballast of 0 – 70 mm, which takes – over the function of insulation, draining and grounding functions. The roadway mantle is consisting in cement concrete; the roadway concrete will be selected based on the bending resistance criterion, while this is the main mechanic characteristic of these kind of concrete. The basic materials for preparation of the roadway mantle should be: natural sand sort 0 – 4; crushed gravel sort 4 – 8, 8 – 16, 16 – 31; chippings sort 8 – 16; crushed stone sort 25 – 40; cement CEM I, concrete steel OB 37; bitumen D 80/120; Kraft resistant paper (128 g/m^2) or polyethylene (0,06 mm thick), filer, additives, water. At the surface, the foundation will have the same slopes in cross – sectional profile and declivities in longitudinal sections as the mantle’s surfaces under which they are located. Before the mantle is disposed, the foundation’s geometrical elements will be verified, together with the limit deviations of allowable dislevelments and the compactation degree. On the previously humidified foundation, a layer of 2 cm thick sand is disposed after compactation, which will respect the level and slope angle of the mantle’s inner surface. On this leveled sand layer, the Kraft paper or the polyethylene foil is put, with a superposition of the foils on 5 cm in longitudinal way and 25 cm in cross – sectional direction. The thickness of the un – compacted roadway concrete should be up to 1.15 – 1.35 times higher than the final thickness. Pooring the concrete will be done with longitudinal and cross – sectional dilatation and contraction joints; the longitudinal joints are executed on the entire thickness of the concrete blocks and are equipped with anchors from OB 37 (Φ 10 mm and 1 m length), disposed at the concrete block’s half thickness size, at 1 m distance each from another. The cross – sectional dilatation joints are done perpendicularly on the concrete beam axis, on the entire thickness of the block (they are built at the limit of curves and in points of declivity changes were there are not envisaged convex joints).The cross–sectional contraction joints will be done on depths of 0.2–0.25 of the concrete block’s thickness, in sequences of 4 m, 5 m or 6 m.

5. ABUTMENT WALLS, DRAINS AND TUBULAR FOOTBRIDGES

For the considered case, of the access road consolidation, the walls will be built on alternate sections, each one having 5 m length (see figure 1).

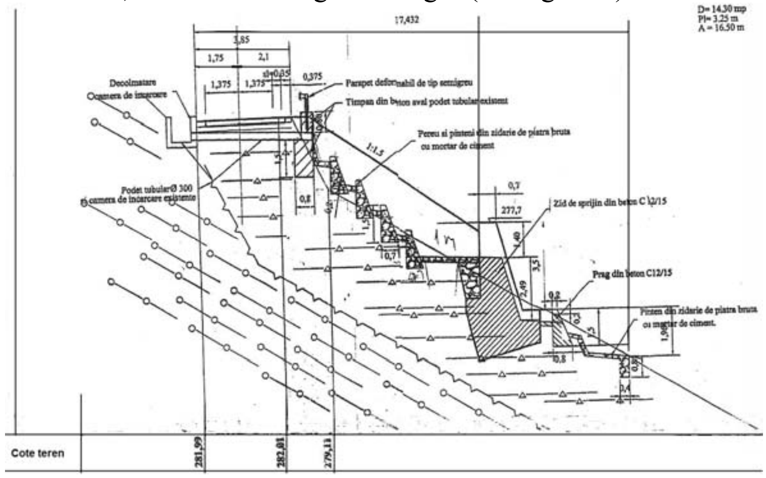

Fig.1 – *Cross-sectional profile in the abutment wall (P3 Hm 0 + 27).*

After tracing and marking – out the foundation hole for the abutment walls, the manual digging is done, to embank the foundation; the surface is leveled and compacted. The foundation of the abutment wall is done with C 12/15 concrete. The concrete surface of the foundation will be protected with matting, permanently moisted during the first 15 days from the concrete disposal. The upper side of the abutment wall will be inclined with a 3:1 dip. Behind the abutment walls, a dry masonry drain will be built, within the scope of collecting the water and divert it to the weepers existing in the elevation. The elevation joints, located between the cemented joints of the masonry sections, will be insulated with tar board in 2 layers, sticked one to another with bituminous mastic. The drain behind the abutment walls should be built in such a manner that he can fulfill his function of collecting the leakage water and divert these water flows to the evacuation weepers. The ends of the abutment walls will be embedded in the versant. The tubular footbridge for catchments drains discharge (see figure 2) will have 300 mm in diameter, providing the water to flow downwards.

A) Dimensioning the Săsciori remblai foot abutment wall (see figure 3)

Computation hypothesis:

$$h_{elevation} = 3.5 \text{ m}$$
$$H_{total} = 4.8 \text{ m}$$

The foundation ground characteristics, according the geo – technical study, are given bellow:

$\varphi = 20^o$ $C = 0.1$ daN/cm^2 $\gamma_p = 17$ kN/m^3 $\delta = \frac{2\varphi}{3} = 13.3^o$

$P_{conv} = 150 + 200$ kPa $= 350$ kPa

$h_{ech} = 1.3$ m $\gamma_{sat} = 21$ kN/m^3

Weights:
- for stress computation on the foundation's bottom: G = 281.40 kN;
- for stress computation in section I – I: G' = 93.36 kN.

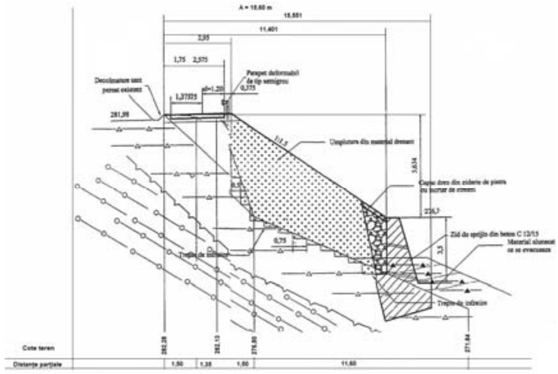

Figure 2- *Footbridge cross – sectional profile (P8 Hm 0 + 44,58).*

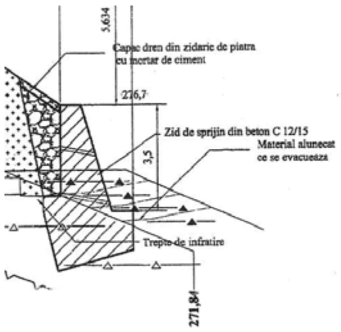

Figure 3- Abutment wall computation hypothesis.

The active land pushing

$$K_a = \frac{\sin^2(90+\varphi)}{\sin(90-\varphi)} \cdot \left(1 + \sqrt{\sin(90+\delta) \cdot \frac{\sin\varphi}{\sin(90-\delta)}}\right)^2 \quad ; \quad K_a = 0,449$$

$$E_a = \frac{1}{2}\gamma_{sat} H^2 \cdot K_a \left(1 + 2\frac{h_{ech}}{H}\right)$$

$$E_a = 167,83 \quad kN/m$$

$$E_{aH} = E_a \cos\delta = 163,33 \quad kN/m$$

$$E_{aV} = E_a \sin\delta = 38,61 \quad kN/m$$

The land pushing diagram:

$$\sigma_0 = K_a\,\gamma_{sat} \cdot \frac{1}{\cos\delta} = 12.62 \quad kN/m^2$$

$$\sigma_H = \left(H + h_{ech}\right)\gamma_{sat} \cdot \frac{1}{\cos\delta} = 5923 \quad kN/m^2$$

$$z_H = \frac{2}{3} H\,\sigma_0 + \frac{\sigma_H}{\sigma_H + \sigma_0} = 1.88 \quad m$$

B) Stability testing

1- Turnover testing

$$M_S = -40.68 \cdot 3.03 - 240.72 \cdot 1.84 = 566.18 \quad kNm \quad ; \quad M_R = 167.83 \cdot 1.05 = 176.22 \quad kNm$$

$$K_S = \frac{M_S}{M_R} = 3.21 > 1.50$$

2- Sliding testing on the foundation bottom

$$K_1 = \frac{N\,f}{T} \geq 1.30 \qquad f = tg\,20^o = 0.364$$

$$\sum G + E_{av} = 281.40 + 38.61 = 320.0 \quad kN$$

$$N = 346.27 \quad kN \quad ; \quad T = 96.06 \quad kN$$

$$\sum E_{aH} = 163.33 \quad kN$$

$$\Rightarrow \qquad K_1 = \frac{346.27 \cdot 0.364}{96.06} = 1.31 \geq 1.30$$

3- Strain testing on the foundation bottom

$$\sigma_{1,2} = \left(\frac{N}{B} \cdot 100\right) \cdot \left(1 \pm \frac{6\,e}{B}\right)$$

where: B = 2,85 m; e – eccentricity, m

$$M_0 = -40.68 \cdot 1.62 - 240.72 \cdot 0.44 + 167.83 \cdot 1.65 = 105.10 \quad kNm$$

$$e = \frac{105.10}{346.27} = 0.30 \quad m \quad ; \quad \sigma_{1,2} = \frac{346.27}{285} \cdot \left(1 \pm \frac{6 \cdot 0.30}{2.85}\right)$$

$$\Rightarrow \sigma_1 = 1.99 \quad daN/cm^2 \quad ; \quad \sigma_2 = 0.44 \quad daN/cm^2$$

<u>Section I – I</u> (see figure 4)

$$G_{b\ I\text{–}I} = 3.89 \cdot 24 = 93.36 \quad kN$$

The active land pushing:

$$E_{a\ I\text{–}I} = \frac{1}{2} \cdot 21 \cdot 3^2 \cdot 0.45 \cdot \left(1 + 2 \cdot \frac{1.3}{3}\right) = 79.38 \quad kN$$

$$E_{a\ I\text{–}I}\,H = E_a \cos\delta = 77.25 \quad kN$$

$$E_{a\ I\text{–}I}\,V = E_a \sin\delta = 18.26 \quad kN$$

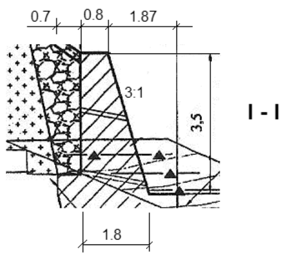

Figure 4 – Strain testing computation.

The land pushing diagram

$$\sigma_0 = K_a\,\gamma_{sat}\,h_{ech} \cdot \frac{1}{\cos\delta} = 12.62 \quad kN/m^2$$

$$\sigma_{I\text{–}I} = K_a\,\gamma_{sat}\,(h + h_{ech}) \cdot \frac{1}{\cos\delta} = 41.75 \quad kN/m^2$$

$$z_H = \frac{2\,H}{3\,(\sigma_0 + \sigma_H)} = 1.23 \quad m$$

Turnover testing

$$M_S = -93.60 \cdot 1.12 = 104.56 \quad kNm \quad ; \quad M_R = 79.38 \cdot 0.78 = 61.92 \quad kNm$$

$$K_S = \frac{M_S}{M_R} = 1.69 > 1.50$$

Strain testing in section I – I

$$M_0 = -93.36 \cdot 0.12 + 79.38 \cdot 0.78 = 50.71 \quad kNm$$

$$e = \frac{50.71}{111.62} = 0.45 \quad m \quad ; \quad \sigma_{1,2} = \frac{108.14}{180} \cdot \left(1 \pm \frac{6 \cdot 0.45}{1.80}\right)$$

$$\Rightarrow \sigma_1 = 1.51 \quad daN/cm^2 \qquad ; \quad \sigma_2 = -0.31 \quad daN/cm^2$$

6. CONCLUSIONS

The access road to the overflowing surge Săsciori was damaged by landslides occurred upstream from the road’s platform, consequently to the heavy rainfall. The road maintained his prior characteristics, with 3.5 m width and infrastructure restoration. The importance category of workings is D, namely low importance. This objective is classified into class IV, being a permanent working, of secondary importance. For the working development stage there were established also the prevention measures required in order to provide the diminishment of the negative environmental impact. According the geo – technical study carried out for this work, the sliding is shallow at the refilling level done at road’s development. In order to provide the getting back to a normal access in safe conditions, there was envisaged the refilling of the rock fill in the road’s body and the execution of some supporting workings based on concrete abutment walls. With a view to faster drainage water evacuation from this perimeter, it was proposed the development of two culverts in the refilling mass, discharged through the abutment wall’s weepers.

To ensure the access in the roadway territory, it was imperatively required the development of a secondary, temporary road, having 250 m of length and 3 m in width, which is practically a downward branch of the existing road. Abutment walls were designed to be built from C 12/16 concrete at the remblai bottom size, with 29 m length, elevation of 3.5 m height and, at the limits of road’s platform a 2 – 2.5 m of elevation height on 10 m length. The wall comprises weepers aimed at refilling and drains-collected water discharge and evacuation made of pipes Φ 110 mm and mounted according the detailed diagrams. Before starting the refilling operation in the road remblai, the un-proper material will be evacuated and jointing benches will be digged in the natural land. The refilling will be done from local draining material. The access to overflowing surge is included in 3rd class of exploitation roads, with roadway bench of 2.75 m width and two verges of 0.75 m. The verges were designed to be non

– permeable, employing a layer consisting in 20 cm of ballast, stabilized with 6% cement. Before starting the refilling operations in the roadway remblai, the backfilling unproperly slipping material will be removed and linking benches will be digged in the natural soil. To ensure the proper surface water gathering and evacuation through already existing workings, the alternative was selected to clean out the catchment's drain, to declogge the existing culvert and the footbridge Φ 300 and to restore the lunette destroyed by the landslide. The only feasible solution aimed at diverting the water flow downward the footbridge was that resorting to a trapezoidiform step-up ditch, supported with cement masonry. For water drainage and evacuation purposes, two elastic raft catchments drains will be constructed, and water will be discharged through the abutment wall's weepers. For water drainage and evacuation purposes, two elastic raft catchments drains will be constructed, having 1.20 m width and 3 – 5 m height, which will collect the rainfall water through a punched drainage pipe, having 80 mm in diameter and water will be discharged through the abutment wall's weepers. For safety purposes, on the affected area it also was needed to restore the concrete superstructure of the roadway and to mount a metallic railing, at the roadway platform boundary. The railing will be deformable, semi – heavy type with metallic ledge, on the entire section of about 60 m of length, but on the last 9 m section the railing will be mounted in continuous C 12/15 concrete grounding.

The computations are based on the saturated soil state of the refilling material disposed behind the wall. In order to confirm this computation hypothesis, it is needed an adequate drainage resorting to catchments and evacuation drains through the wall's weeper.

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Scientific Reviewers:
Prof. PhD. Eng. Constantin Semen

CONSIDERATIONS ON THE GEOTECHNICAL CHARACTERISTICS OF THE OVERLYING ROCKS IN THE QUARRIES OF OLTENIA ACCORDING TO THEIR DEPTH AND THEIR INFLUENCE ON THE GEOMETRICAL ELEMENTS OF SLOPES

ILIE ROTUNJANU*[*]
DANIEL ANTONIE[**]
VIOREL VOIN*

Abstract: *The dependence of slopes geometry of in open – cast mining on the physical and mechanical characteristics of the overlying rocks in quarries calls for the careful study of the latter and the assessment of their variations. The aim of this paper is to analyze the influence of geological formations depth on the rocks strength and to establish the geometrical elements of slopes according to the geomechanical characteristics of the main rock types for the quarries in Oltenia.*

Key words: *cohesive and non – cohesive rocks, geotechnical characteristics, slopes, slopes geometry.*

From a geological point of view, the coal deposits in Oltenia belong to the Dacian – Roman formations and generally consist of clay and sandy clay formations, as well as sands that host 17 coal seams of which quarries mainly ensure the mining of seams V-X and, rarely, in hill areas, the mining of seams XI-XIII.

Quarry depth depends on the geomorphologic structure of the area and ranges from 40-50 m in plain areas to 140-160 m in hill areas.

As quarry geometry and the configuration of margin slopes are correlated with their depth, as well as with the type and the physical – mechanical characteristics of rocks, it is interesting to study the dependence of rocks geological characteristics on the formations depth.

The study of this dependence involved geotechnical drillings into the margin slopes of quarries Tismana I and Tismana II, placed on different mining levels, in order

* *Prof. PhD. Eng. - University of Petroşani*
** *PhD. Eng. - Chief eng. - E.M.C. Rovinari*

to cover the whole stratigraphic column in conditions of limited drilling depth due to the technical characteristics of the drilling equipment.

The drilling levels are shown in table 1.

Table 1: *Geomechanical characteristics of rocks in Tismana mining field*

Type of formation	Open-cast drilling level [m]	Floor level [m]	γ_a [kN/m^3]	w [%]	C [kN/m^2]	φ [^{o}]
CLAY	300	298,4	21,5	14,1	30	15
		287,4	20,3	18,7	28	165
		285	19,4	23	25	15
		282	19	25,8	27	15
	247	243,3	19,6	27,3	26	15
	237	226	17,5	20,4	24	16
	225	216,4	20	24,5	25	14
	182	167,7	18,8	26,8	24	16
SANDS	300	294	20,6	4,1	2	28
		284,2	20,6	9	2	28
	247	244,9	20	20,8	2	28
		240,3	16,1	12,5	2	28
		236,3	15,2	16,9	2	28
		233,8	17,5	12,6	2	28
	237	233,4	16,2	5,5	2	28
	225	214,3	15	13,9	2	28
POWDER CLAY	278	270,8	19,1	19,5	19	19
		262,7	19,6	21,1	18	19
SANDY CLAY	300	289,8	19,9	19,9	16	19
	247	245,2	20,3	20	15	20
		240,4	20,2	20,2	16	20
		235	19,8	18,2	17	21
		232,4	20,7	16,9	15	19
	237	234,5	20,1	15	16	20
		230,4	20,3	20,4	15	19
		227,3	20,4	17	16	19
	225	211	19,7	25,3	16	11
	222	211,2	20,5	19,5	16	18
		208	20,4	21	14	19

The geotechnical mapping was performed for each drilling and samples were collected from the drilled formations.

The analysis of the geotechnical drillings led to the conclusion that the geological formations in the two mining fields, Tismana I and Tismana II, are generally correlated and display just variations of granulometric facies, being predominantly clayish, with a consistent to thick plastic state.

The geomechanical characteristics of the rocks intercepted in the geotechnical drillings, for the two quarries, were determined in the Geotechnical Laboratory of ICSITPML Craiova and are presented on formation types (clay, sands, powder clay, and sandy clay), according to their depth and humidity, in table 1.

The variation ranges of the geotechnical indices for clayish and sandy rocks in quarries Tismana I and Tismana II are shown in table 2.

Table 2: *Te geomechanical characteristics of overlying rocks in quarries Tismana I and Tismana II*

No. crt.	GEOTEHNICAL INDEX	U.M.	QUARRY			
			Tismana I		Tismana II	
			Clayish Rocks	Sandy Rocks	Clayish Rocks	Sandy Rocks
1	Absolute specific weight	kN/m^3	24,6÷26,8	25,5÷27,0	23,5÷26,8	25,6÷27,0
2	Volume weight (Bulk density)	kN/m^3	17,5÷20,4	16,2÷21,2	17,6÷20,3	17,8÷20,4
3	Humidity	%	17,0÷33,9	10,7÷23,8	16,9÷31,8	5,0÷24,5
4	Consistency index	-	0,68÷0,84	0,42÷0,82	0,64÷0,86	0,44÷0,81
5	Plasticity index	%	29,2÷36,4	9,2÷17,3	22,8÷39,2	15,0÷18,2
6	Humidity degree	-	0,82÷1,02	0,36÷1,03	0,80÷1,02	0,40÷1,05
7	Compressibility	daN/m^3	62÷182	58÷182	133÷166	68÷90
8	Specific settlement	cm/m	2,0÷3,9	3,7÷5,9	1,4÷3,0	4,3÷6,2
9	Cohesion	kN/m^2	14÷21	2,0÷13	19÷30	5,0÷7,0
10	Internal friction angle	grade	11÷19	8÷23	11÷19	15÷22
11	Granulometric composition					
	- clay	%	33÷64	7÷28	29÷61	6÷45
	- powder	%	24÷39	4÷28	31÷41	2÷24
	- sand	%	2÷36	44÷95	5÷40	48÷96
	- crushed rock	%	-	3÷17	-	2÷50

The analysis of the geomechanical characteristics leads to the following conclusions:

- for clayish rocks situated from level +300 m to level +168 m, cohesion ranges from C = 30÷24 kPa, the internal friction angle φ = 14-16^o, and humidity ranges from w = 14÷27 %;

- for sandy rocks situated from level +294 m to level +214 m, cohesion is approximately the same, cca. C = 2 kPa, the internal friction angle φ = 28^o, and variable humidity w = 4÷20 %;

- for powder – clay rocks situated from +290 m to + 208 m cohesion C = 18-19 kPa, the internal friction angle φ = 19^o and humidity w = 20÷21 %;

The analysis of the four rock types shows that the geomechanical characteristics vary within very small ranges or remain constant with the growing of their depth, whereas cohesion decreases with the increase of humidity.

Consequently, we appreciate that the geomechanical characteristics of the rocks in Oltenia quarries do not depend on the depth at which they occur, but on their type, structure and humidity. In the case of cohesive rocks there is a higher variation of the strength characteristics (C and φ) according to humidity, unlike the low cohesive or non – cohesive rocks, with which these characteristics are less influenced by humidity. The change of rock humidity is influenced both by surface and underground waters, and have negative consequences on slope stability by reducing it, in some cases, to the limit of the loss of natural balance, thus favoring land – slides.

Under the influence of water, the increase of rock humidity determines an increase of their volume weight, which leads to an increasing weight of the sliding massif and the decrease of its stability margin.

In the design of optimal slope geometry, with imposed safety factor, the influence of the geomechanical characteristics of rocks and of the level of underground water is of utmost importance.

This was pointed out by analyzing the variation of geometrical elements (height and slope angle) through the method of admissible height, paying special attention to non – cohesive materials whose sliding surface is plane.

The admissible slope heights were calculated for different geomechanical conditions and for certain imposed slope angles and safety factors.

The Resal relation was used to calculate the admissible height:

$$H_{ad} = \frac{2 \cdot C'}{\gamma_a} \cdot \frac{\sin\alpha \cdot \cos\varphi'}{\sin^2\left(\frac{\alpha - \varphi'}{2}\right)}$$

where:

$C' = \frac{C}{F_s}$, $\varphi' = \mathrm{arctg}\frac{\mathrm{tg}\varphi}{F_s}$, where F_s – imposed safety factor;

C – rock cohesion, kN/m^2;

φ – interior friction angle of rocks, degrees;

γ_a – apparent specific weight, kN/m^3.

Table 3 shows the admissible heights for slope angles of 55^o, 50^o, 45^o and even 35÷30^o for low cohesive and non – cohesive rocks (with medium to good physical – mechanical characteristics) and safety factors of 1,1; 1,3 and 1,5 for temporary, operating, and long – term slopes.

According to the results presented in the table, there are the following recommendations:

1. For highly cohesive, plastic thick rocks, I_c > 0,75 (clays, heavy clays, carbonaceous shale, or argillaceous marl) with good geomechanical characteristics:

- working levels can be of height h = 20 m for α = 55^o or h = 25 m for α = 45^o;
- levels of temporary slopes can be of height h = 20 m for α = 45^o or h = 25 m for α = 40^o;
- levels of long – term slopes can be of h = 15 m for α = 45^o or h = 20 m for α = 40^o.

2. For highly cohesive, plastic consistent rocks I_c = 0,5÷0,75 (form clays to heavy clays argillaceous marl) with good geomechanical characteristics:

- working levels can be of h = 20 m for α = 45^o and h = 25 m for α = 40^o or h = 15 m for α = 50÷55^o;
- levels of temporary slopes: h = 20 m for α = 40^o or h = 15 m for α = 45^o ;
- levels of long – term slopes: h = 15 m for α = 40^o or h = 10-13 m for α = 45÷50^o.

3. For cohesive, plastic thick rocks (powder or sandy clays), I_c > 0,75, with medium geomechanical characteristics:

- working levels can be of h = 20 m for α = 45^o and h = 25 m for α = 40^o or h = 15 m for α = 50^o;
- levels of temporary slopes: h = 20 m for α = 40^o and h = 15 m for α = 45^o;
- levels of long – term slopes: h = 15 m for α = 40^o and h = 10 m for α = 45^o.

For the same rocks, but with plastic consistent state I_c = 0,5÷0,75, slope heights decrease as it follows:

- for working levels: h = 15 m for α = 45^o and h = 18 m for α = 40^o; or h = 24 m for α = 35^o an h = 36 m for α = 30^o;

- for temporary slopes: h = 11 m for $\alpha = 45^o$; and h = 13 m for $\alpha = 40^o$; h = 17 m for $\alpha = 35^o$; and h = 24 m for $\alpha = 30^o$;

- for long – term slopes: h < 10 m for $\alpha = 45^o$; h = 10 m for $\alpha = 40^o$; h = 13÷18 m for α = 30-35^o;

4. For low cohesive (powder sand, argillaceous sand, and sandy – argillaceous powder) and non – cohesive rocks (sand):

- working levels: h = 10 m for $\alpha = 40^o$; h = 14÷15 m for $\alpha = 35^o$ and h = 25 m for $\alpha = 30^o$;

- temporary slopes: h < 10 m for $\alpha = 35^o$ or h = 10÷15 m for $\alpha = 30^o$;

- long – term slopes: h ≤ 10 m for $\alpha = 30^o$.

Water removal from low cohesive rocks leads to the improvement of their geomechanical characteristics, as well as of the slope geometry.

CONCLUSIONS

As a result of establishing the dependence between the geomechanical characteristics of rocks and the geometrical elements of working, temporary and long – term slopes, the following conclusions can be drawn:

- the height of working levels can be of 25 m, provided the slope angle ≤ 45^o according to the rock type. It is recommended to have h = 20 m and $\alpha = 45^o$ even for the levels consisting of low cohesive rocks, having in view that their structure also contains cohesive rocks with high strength characteristics;

- for temporary slopes with a height of 20-25 m it is recommended to have a slope angle α = 30-40^o according to the cohesive or non – cohesive rocks type.

- for long – term slopes situated on quarry limits, with level heights of 20-25 m it is recommended to have a slope angle $\alpha_{def} = 30^o$.

The proposed geometry is based on the geotechnical safety criterion correlated with the technical and technological efficiency of bucket – wheel excavators used in quarries. This can change with the alteration of the geomechanical characteristics of rocks under the influence of geomechanical, hydro-meteorological, and climate factors that have an impact on slope stability.

In such situations, it is recommended to divide levels into sub – levels, having in view an upper level of 15 m, and a lower level of 5-7 m, according to the excavating possibilities of the technological equipment.

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Scientific Reviewers:
Prof. PhD. Eng. Victor Arad

Table 3: *Calcuation of admissible height had according to the material type (geomechanical characteristics)*

α	m ctgα	Fs1 Stab.rel	Fs2 Stab.prov	Fs3 Stab.	γ	φ	φ´ Fs1	φ´ Fs2	φ´ Fs3	c	c´ Fs1	c´ Fs2	c´ Fs3	H Fs1	H Fs2	H Fs3
[º]		1,1	1,3	1,5	[kN/mc]	[º]	[º]	[º]	[º]	[kPa]	[kPa]	[kPa]	[kPa]	[m]	[m]	[m]
A. HIGHLY COHESIVE ROCKS (clays to heavy clays, carbonaceous shale, or argillaceous marl)																
I_c > 0,75 (good, plastic thick materials)																
55	0,70	1,1	1,3	1,5	18,5	17	15,5	13,1	11,3	30	27,3	23,1	20	20,37	15,64	12,62
50	0,80	1,1	1,3	1,5	18,5	17	15,5	13,1	11,3	30	27,3	23,1	20	24,95	18,83	14,60
45	1,00	1,1	1,3	1,5	18,5	17	15,5	13,1	11,3	30	27,3	23,1	20	31,61	22,52	17,74
40	1,20	1,1	1,3	1,5	18,5	17	15,5	13,1	11,3	30	27,3	23,1	20	40,16	28,84	22,27
0,5 < I_c < 0,75 (medium, plastic consistent materials)																
55	0,70	1,1	1,3	1,5	18,5	14	12,7	10,8	9,3	24	21,8	18,5	16	14,40	11,41	9,22
50	0,80	1,1	1,3	1,5	18,5	14	12,7	10,8	9,3	24	21,8	18,5	16	17,27	13,53	10,82
45	1,00	1,1	1,3	1,5	18,5	14	12,7	10,8	9,3	24	21,8	18,5	16	21,29	15,91	12,97
40	1,20	1,1	1,3	1,5	18,5	14	12,7	10,8	9,3	24	21,8	18,5	16	26,16	19,85	15,42
B. COHESIVE ROCKS (powder, sandy clays)																
I_c > 0,75 (good, plastic thick materials)																
55	0,70	1,1	1,3	1,5	19,1	18	16,4	13,8	12	20	18,2	15,4	13,3	13,86	10,34	8,32
50	0,80	1,1	1,3	1,5	19,1	18	16,4	13,8	12	20	18,2	15,4	13,3	16,57	12,50	9,94
45	1,00	1,1	1,3	1,5	19,1	18	16,4	13,8	12	20	18,2	15,4	13,3	21,12	15,56	11,78
40	1,20	1,1	1,3	1,5	19,1	18	16,4	13,8	12	20	18,2	15,4	13,3	28,36	19,37	14,88
0,5 < I_c < 0,75 (medium and good, plastic consistent materials)																
55	0,70	1,1	1,3	1,5	19,1	15	13,6	11,5	10	16	14,5	12,3	10,7	9,74	7,51	6,10
50	0,80	1,1	1,3	1,5	19,1	15	13,6	11,5	10	16	14,5	12,3	10,7	11,42	8,94	7,19
45	1,00	1,1	1,3	1,5	19,1	15	13,6	11,5	10	16	14,5	12,3	10,7	14,15	10,91	8,65
40	1,20	1,1	1,3	1,5	19,1	15	13,6	11,5	10	16	14,5	12,3	10,7	18,25	13,25	10,73
35	1,43	1,1	1,3	1,5	19,1	15	13,6	11,5	10	16	14,5	12,3	10,7	24,55	17,44	13,50
30	1,73	1,1	1,3	1,5	19,1	15	13,6	11,5	10	16	14,5	12,3	10,7	36,26	24,42	18,29
C. LOW COHESIVE (powder sand, argillaceous sand, sandy – argillaceous powder) AND NON – COHESIVE ROCKS (sand)																
(good to medium materials)																
55	0,70	1,1	1,3	1,5	19,8	20	18,2	15,4	13,3	6	5,5	4,6	4	4,37	3,21	2,53
50	0,80	1,1	1,3	1,5	19,8	20	18,2	15,4	13,3	6	5,5	4,6	4	5,29	3,93	3,04
45	1,00	1,1	1,3	1,5	19,8	20	18,2	15,4	13,3	6	5,5	4,6	4	6,88	4,79	3,77
40	1,20	1,1	1,3	1,5	19,8	20	18,2	15,4	13,3	6	5,5	4,6	4	9,51	6,33	4,66
35	1,43	1,1	1,3	1,5	19,8	20	18,2	15,4	13,3	6	5,5	4,6	4	14,18	8,87	6,36
30	1,73	1,1	1,3	1,5	19,8	20	18,2	15,4	13,3	6	5,5	4,6	4	24,95	13,87	9,32

POSSIBILITIES OF THE OVER PROFILES EVALUATION RESULTED AT THE EXECUTION OF THE DUMITRA-AHE JIU-LIVEZENI BUMBESTI SECTOR ADDUCTION

GEORGE POPESCU*
MIHAELA TODERAS**

Abstract: *The process of displacement of rocks with explosion energy is a complex process, influenced by several factors, which, in relation to the mechanism of displacement of rocks, were classified in natural, technological and technical factors, whose influence is manifested by the simultaneous deployment efficiency. To identify the causes generating over profiles, it was made a structural analysis of the rocks intercepted by the culvert gallery, the monographs by blasting used and there were assessed the surfaces of the over profiles on locations and types of rocks. The analysis allowed the identification of the groups of causes that led to get the overprofiles.*

Key words: *displacement, structural analysis, overprofiles, blasting, adduction*

1. GENERAL DATA

The LIVEZENI-BUMBESTI hydro technical facility is located downstream of the confluence of the Western Jiu and the Eastern Jiu, at the right bank of the Jiu river, in its clough, between the localities Livezeni and Bumbeşti.

2. THE GEOLOGY AND TECTONICS OF THE HYDRO TECHNICAL FACILITY

Geographically speaking, the LIVEZENI-BUMBEŞTI hydro technical facility's main adduction crosses the southern slope of the Vulcan Mountains, mountains whose constituent geological formations belong to the genetic structure unit known in literature as the. In terms of petrography, the composition of the Danubian Autochthonous is formed of crystalline schist generated by prehercynian and hercynian geotectonic cycles and of prehercynian granitoide massifs. Thus, the Danubian

* *Student PhD. –University of Petrosani*
** *Assoc. Prof. PhD. Eng. - University of Petrosani*

Autochthonous from the southern slope of the Vulcan Mountains contains the following types of petrofacials: mezometamorphic crystalline schist, epimetamorphic crystalline schist and granitoids (magmatic rocks).

In the mezometamorphic crystalline schist two petrofacial entities were separated, known as the Drăgăşani Crystalline (series) and the Lainici-Păiuş Crystalline (series). The epimetamorphic crystalline schist found in the layout of the hydro technical facility is known as the Rafaila Series. The magmatic rocks found in the hydro technical facility are known, in the geologic literature, as prehercynian granitoids, respectively Şuşiţa type granitoids.

The analysis of detailed geological maps and some prospect materials shows complex tectonic issues, that we will succinctly point out: in the area of the hydro technical facility we notice a plicative tectonic and a ruptural one; the plicative tectonic highlights it's self by the presence, in the geological formations crossed by the adduction, of both ample creases and some micro creases. Along some ample anticlinal creases are placed magmatic rocks (granitoids). These anticlinal creases are west-east oriented. The Lainici-Păiuş Series formations are the most affected ones by this ruptural tectonic. South of Rea Valley attack window, for a distance of approximately 2 km, all the way to the contact with the Şuşiţa granitoid, we can find a system of NV-SE oriented transversal fissures and of V-E oriented longitudinal fissures, along who the Rafaila Series formations interbeds in the form of fingering formations.

Along the LIVEZENI-BUMBESTI hydro technical facility this is the most tectonized area: the tectonic contact between Drăgăşani Series and Lainici-Păiuş Series, situated north from the brook Dumitra, proves to also be a heavily tectonized and cataclazed area; in the granitoids body there are no tectonic occurrences that might create difficulties in implementing the hydro technical facility's activity. The rocks found in this body, which we mentioned before, are compact and stabile in comparison with the crystalline schist.

3. CLASSIFICATION OF ROCK MASSIF

The concerns for geomechanic classification of the rock massif have known a continuous development, mostly after 1970. The purpose was to offer the designers and engineers who worked in production a bigger volume of information, in order to more realistically evaluate all the factors involved in the stability of underground constructions and to better choose technically and economically efficient props. Geomechanic classification are based on Terzaghi's (1946) Lauffer (1958) and Deere (1964), pioneering studies. During the last years new classification criteria was proposed from which worth mentioning are the RSR criteria (Wickhman, 1932), RQD criteria (Deere, 1964), RMR criteria (professor Bieniawski 1973) and Q criteria elaborated by Nick Barton (1974). These criteria are most frequently used in the practical work done by the field of subterranean constructions. Most of the existent classifications are based on the study of a variable number of parameters: resistance to compression fracture of rocks, the space between the fissures, orientation and conditions of the discontinuities, presence of groundwater, structure, homogeneity and degree of fracturing of the massive rock, cracks, roughness of fissure and so on. Depending on the physical and mechanical characteristics, the degree of fracturing and alteration, the ISPH project regarding the hydro energetic harvesting of the Jiu River, assesses the existence of five classes of rocks (Table 1).

Table 1- Classification of rocks in five classes:

Rock types	RMR	Q	f	Representative rocks
I (FT)	> 60	>20	> 8	granitoids, silica mica
II (T)	40 - 60	10 - 20	6 - 8	gneiss, crystalline schist
III (ST)	20 - 40	1 - 10	4 - 6	amphibolites
IV(M)	10 - 20	0,05 - 1	2 - 4	clorito feldspathic schist
V (F)	< 10	< 0,005	1	heavily tectonized schist, clorito schist

Table 2 shows a classification for Dumitra - Bumbeşti adduction, also containing the description of the classes of rocks.

Table 2- Types of rocks and geotechnical coefficients for the whole adduction:

Geotechnical coefficients			Geo-mechanic coefficients of the rock aggregation		Prop types
Ee [daN/cm^2]	Ko [daN/cm^2]	F (P)	RMR	Q	
> 600	> 600	> 6	> 80	> 50	I (A)
			60 - 80	10 - 50	
400 - 500	400 - 500	4 - 5	40 - 60	5 - 20	II (B)
200-300	200 - 300	2 - 3	30 - 50	1 - 10	III (C)
			20 - 40	0,5 - 5	
200 - 100	200 - 100	1 - 2	10 - 20	0,05 - 1	IV(D)
< 100	< 100	< 1	< 10	< 0,05	V(D_S)

A first finding is that there are differences between the two classifications, regarding geotechnical coefficients and RMR and Q quality indicators. Moreover, even if the classification of TJ refers to Dumitra-Bumbeşti adduction, the class description refers only to Şuşița Series excluding Lainici, Păiuş and Rafaila Series, intersected by the adduction.

We believe that the frequency of the rock cracks (min. 1-2/ml) shows that the fissuring degree of the rocks is at the most at a medium level (distance between the cracks is 0.3 m-lm). According to the RMR and Q classification, the first and the second classes cover compact rocks with the distance between cracks> 1m, compact rocks that prop themselves or have an occasional local prop, respectively arch anchors placed at a the distance of 2.5 m one from another. The use of systematic solutions for propping in the first and second class reveals that the rocks do not have the stability corresponding to classes I and II of the RMR and Q classification, the degree of failure being advanced.

4. ANALYSIS OF BLASTING MONOGRAPHS USED IN THE DUMITRA AREA

Dumitra's adduction gallery has a geometric shape with straight walls and semicircular arch, the hearth width of 4.6 m, the height to contour curvature and the radius of 2.3 m, a digging section of 18,9m^2 and the upper profile of 10 cm and 20,09m^2. The gallery goes through A, B, C, D, and D_s category rocks. Differentiated blasting monographs were elaborated based on the category of the rock. The monograph's particularities were:

The parameter	Rock type /category			
	A	B	C	D - D_s
Digging section	20,09	20,09	20,09	20,09
Number of holes from which:	57 - 65	43 - 52	40 - 43	40
core holes	9	9	6	6
helping 1	9	7	7	7
helping 2	5	4	4	4
upper hearth	5	5	4	4
contour	20	16	16	14
hearth	9	7	6	6
Quantity of explosive	53 - 65	43 - 50	30 - 40	30
Hole length (m)	2,3	2,3	2,3	1,8
Specific consumption of explosives	1,6 - 1,7	1,4	1,2	0,9
Type of explosive used	**Gama 2Eco dynamite or G dynamite**			
Type of deployment	**CE 05 Cu**			

We mention the fact that the different value of the quantity of explosive used for the same type of rock is dependent on the fissuring degree of the rock and also on the presence of some harder or softer rock inclusions on the working surface.

5. UPPERPROFILE ASSESSMENT FOR DUMITRA ADDUCTION

The Dumitra adduction has a digging section profile of 18,89 m^2, section admissible generated by the upperprofile of 10 cm and 20,09 m^2 and according to ISPH project, goes through A, B, C, D, and D_s category rocks. To assess the upperprofiles, topographic measurements were taken in stations 3 m apart from each other; the summarized situation, with indications related to the type of rock permeated, is presented in table 3.4 for the upstream front and in table 3.5 for the downstream front. We determined the average values of the upperprofiles, differentiated on the rock type. The values for the upstream front were:

Type A rocks $$S_m^A = \frac{\sum L_i s_i}{L} = 1{,}77\ m^2$$

Type B rocks $$S_m^B = \frac{\sum L_i s_i}{L} = 2{,}52\ m^2$$

Type C rocks $$S_m^C = \frac{\sum L_i s_i}{L} = 3{,}02\ m^2$$

Type DS rocks $$S_m^{Ds} = \frac{\sum L_i s_i}{L} = 1{,}43\ m^2$$

Corresponding to the upper profile area we determine the total value (measured in m) for the enlargement of the section + 10 cm using the following equation:

$$\frac{(B+2\delta+2R+2\delta)}{2}(hc+\delta)+\frac{\pi(R+\delta)^2}{2}=20{,}09+S_m$$

$$\frac{(4{,}6+2\delta+4{,}6+2\delta)}{2}(2{,}3+\delta)+\frac{\pi(2{,}3+\delta)^2}{2}=20{,}09+S_m$$

$$3{,}57\delta^2+16{,}42\delta-1{,}19-S_m=0$$

- δ is determined for each rock type:

$3{,}57\delta^2+16{,}42\delta-2{,}96=0$

The result shows us the total δ enlargement opposite to the theoretical contour of 17,3 cm.

- for type **B** δ = 21,5 cm
- for type **C** δ = 24,6 cm
- for type **D_S** δ = 15,4cm

The lower value of the upperprofile in the case if D_S type rocks is justified by the fact that due to the friability of the rock a convulsive blast was performed in the centre of the gallery and the contouring was handmade.

For assessing the enlargement in the case of Dumitra's adduction upper stream front we are:

- for type A rocks: $S_m^A=\frac{\sum L_i s_i}{L}=2{,}29\ m^2$
- for type B rocks: $S_m^B=\frac{\sum L_i s_i}{L}=2{,}68\ m^2$
- for type C rocks: $S_m^C=\frac{\sum L_i s_i}{L}=3{,}03\ m^2$
- for type D rocks: $S_m^D=\frac{\sum L_i s_i}{L}=3{,}24\ m^2$
- for type D_S rocks: $S_m^{Ds}=\frac{\sum L_i s_i}{L}=3{,}86\ m^2$

The value of the enlargement for each rock type is determined with the condition:

$3{,}57\delta^2+16{,}42\delta-1{,}19-S_m=0$

Resulting:

- For type A rocks $\delta^A=20\,cm$
- For type B rocks $\delta^B=22{,}4\,cm$
- For type C rocks $\delta^C=24{,}4\,cm$
- For type D rocks $\delta^D=25{,}5\,cm$
- For type D_S rocks $\delta^{D_s}=28{,}9\,cm$

6. IDENTIFYING THE CAUSES THAT LEAD TO EMERGENCE OF UPPERPROFILES

The analysis permitted us to identify the following sets of causes:

1) Technological causes

Causes determined by technological limits of drilling machines are dependent by the maneuverability level of drilling arms. In all locations TAMROCK AXERA or ATLAS COPCO drilling mobile installations are beeing used, reliable and performant machines, electro-hydraulic installation that are able to achieve apreciable drilling speeds. Achieveing a good contour urges drilling the contour holes at 10 cm inside the work front. Achieveing a good contour urges also drilling of the output at the mine wholes at 10 cm inside the contour that means a bias angle of the exterior whole round about 4°, direction called by the constructor „theoretical". But practically, this angle cannot be realized because the manouvreability level of the arms determines an exterior output of the mine-wholes' sole of 200-250 mm, the realised angle of a whole's lenght of 3 meters is of minimum 7°. Under these conditions, the overprofile determined by the minimum angle of arms' position, represented here by disruption under jagged form(saw's teeth) is 12,5 cm, and so almost 25% higher than admitted over ream of max 10 cm. This overprofiling determined by the drilling machine has an estimate of 2 m^2 2 than the estimate contour, and so an 0,42 m^2 than the admisible contour. The usage of edge blasting reduces the cracking level by 4 to 6 times dependent on rock hardness.

- There have been used low explosive blasters (Lambrex contour or detonating fuse with 100 g/m), blasters with a reduced impedance factor or low charge. The Lambrex contour blaster has an impedance factor of 0,785 in comparison with 0,983of G dynamite.
- There have been used lower diameter blasters than the mine's wholes' diameter which determines low charging coefficient. From this point of view, the following options have been used:

a. Lambrex contour, G dynamite or Gama 2 Eco dynamite with a 25 mm diameter in mine holes with 45 mm in diameter, resulting a charging coefficient of 0,23.

b. detonating fuse Riocord of 100 g/m in mine holes with 25 mm in diameter, resulting a charge factor of 0,126.

Under these conditions we can say that only a small part of the energy is transmitted to the rock, the explosion gases operate on mine holes' walls throughout expansion, reducing the energy. We can say that any of these options can be applicable at the contour blasting process because the transmitted energy at the rock is smaller than 1000 KJ (max. 25% of the explosion energy), the biggest quantity of energy comes under an acoustic wave form. On what we have presented here we can definitely say that there is a strong correlation between the explosives used, diameter of the blasting cartridge and mine's hole, the blasting works don't cause overexpansion of the contour.

2) Geotechnical causes

The main geotechnical cause is the massif in which the mining takes place. The main Livezeni-Bumbeşti adduction is a massif heavily racked by orogenic movements, that passed through all the shriveling fazes (the precursor shriveling, the actual shriveling-which led to the formation of the orogenic system- and the posthumous shriveling which had as an effect the re-shriveling of the formal creases). During these orogenyc movements the getic crystalline was pushed toward west, over the autochthonous crystalline in the form of a big thrust web. The crystalline schist from group I (the getic one) advanced in the form of a very ample thrust web over the crystalline schist from group II (the danubian one). In the case of type C, D or D_S rocks the participation of the whole perforation procedure outside the technical contour is reduced to 7-10 %.

In conclusion, the average upper profiles recorded are generated by:

- The technical possibilities of the equipment 21 %
- The hole perforation procedure 10,5 %
- The geological characteristics represented by the hardness of the rock, the fissure and intensive stratification 68 %.

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Scientific Reviewers:
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SPECIALISED CONSOLIDATION WORKS THROUGH REINFORCEMENT GRADIENTS AND SLOPES TO REDUCE THE RISK OF PRODUCING LANDSLIDES

VALERIU PLEŞEA*

Abstract: *The solutions to fight against the land instability, based on the internal reinforcement of the gradients and slopes starts to gain a large propagation in the Romania's geotechnical practice. Among those, the one who stipulates the land reinforcement with construction geo-synthetic materials, geo-grids and high resistance geo-textiles type detaches abundantly, due to the technical-financial advantages which distances the traditional solutions. Such a solution for ensuring the land resistance, with the presentation of the dimensioning estimation and stability verification is presented in this paper.*

Keywords: *land reinforcement, geo-synthetic materials, landslides, stability estimation, embankment, excavation.*

1. INTRODUCTION

From the work categories and measures for landslide stabilization, the special works of land reinforcement constitutes as reinforcement special situation, respectively a resistance straightening, being activated an older solution group which assumes the using of straw mixed with clayey soils for obtaining the resistant materials.

The Vida, H (1966)'s solution of "reinforcement" soil was extended by land reinforcement with construction geo-synthetic materials, of geo-grids or high resistance geo-textiles especially for consolidating of the main steep of the regressive landslides as for the reconstruction of slipped excavation slopes. In many of the situations, the construction of some embankment works or accomplishing of some fillings behind the support works of the slipped masses, as for erosions and low-depth slides of the sloping land are necessary from the design stage the apply of solutions based on land reinforcement.

The results recorded so far in the field of land reinforcement for consolidating the lands with slip tendency highlights the technical - economical

* *Assoc. Prof. PhD. Eng. – University of Petroşani*

advantages in comparison with the equivalent solutions, the real problems which limits the extension of the applying area of different solutions being linked with the progresses which are to be obtained by producing some more adequate reinforcement materials.

2. LAND REINFORCEMENT PROCEDURES BY INTRODUCING GEO-SYNTHETIC REINFORCEMENT MATERIALS. REINFORCEMENT DIMENSION EVALUATION FOR FIELD REAL CONDITIONS

For slopes of embankment and excavations which lost their stability and for land's low depth slides, the using of reinforcement suppose a prior removal of the slipped soil and bringing the slope or land to its initial state by accomplishing some layer fillings, between are introduced geosynthetic materials (fig. 1 a, b).

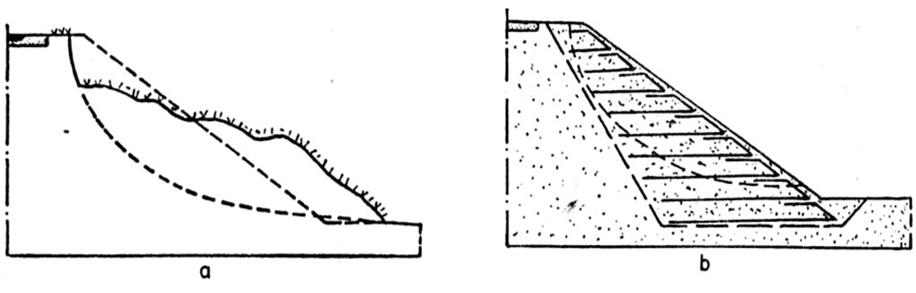

Fig. 1. Embankment slope stabilization solution (accomplishing stages): a – embankment slid; b – embankment slope reinforced with geotextile

The geosynthetic materials thus introduced considerably influences the horizontal strains reduction of the base terrain increasing the land works.

In practice, for stabilizing the embankment slopes, exists a series of solutions based on the principle of soil reinforcement with geotextil materials, of which the one in combination with using the soil capsuled banquets or gabions with geotextil materials filled with soil, respectively anchorage blocks (fig. 2 a,b), records a wide spread [1], [3], [4].

The high variety of reinforce solutions for embankments which uses geosynthetic materials requires an systemic analyze of the main types of works and highlighting the technical and economical characteristics considering the material particularities and by the corresponding work technologies.

For increasing the efficiency in application, as elements for reinforcement of the soil construction, the geosynthetic fitting must have a sufficient concurrence length (L) with the terrain on the both sides of the potential slide surface of the embankment slope.

In this case, for forecast the size of these dimensions, considering the breaking scheme of an embankment on which was interposed a reinforcement material between the base and the foundation soil [1], [3], it can consider that the presence of the casing generates a stabilizing force P in the slide process (fig. 3).

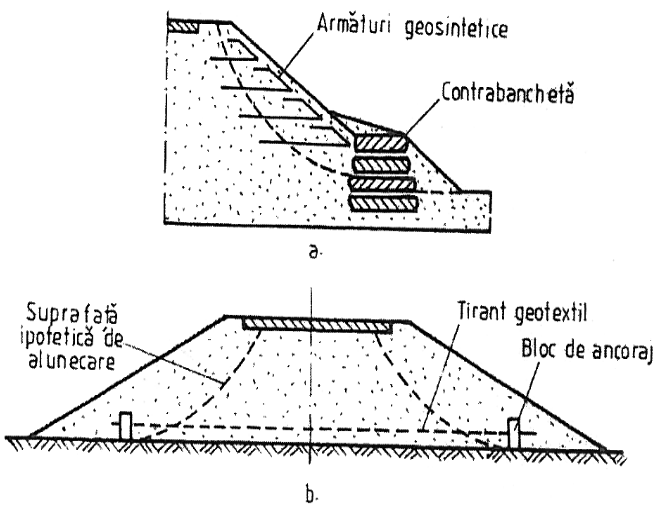

Fig. 2. Embankment slope stabilization solution by reinforcement with geosynthetic casing and geotextil tyrants combined with counterbenches of capsuled soil (a) or with anchorage blocks (b)

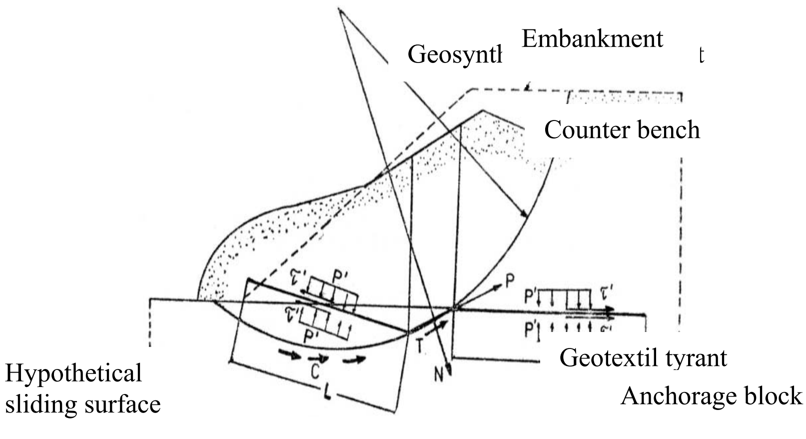

Fig. 3. The geotextil – terrain interaction scheme

For the given situation, the slide coefficient K_s has the expression:

$$K_s = \frac{M_\varphi + M_C + M_P}{M_{al}} \qquad (1)$$

in which: Mφ, Mc – the moments of the stabilizing force which corresponds to the friction and, respectively with the cohesion; M_P – the moment of the stabilizing force P given by the geotextil resistance to traction; Mal – the moment of the forces who produce the slide.

The length of working with land (L), depends of the friction between the casing and the soil and is defined by the relation:

$$L = \frac{R'_c}{2 \cdot \tau'}, \ cm \qquad (2)$$

where: R'c – geotextil (geo-grid)'s resistance to traction, daN/cm; τ' – sheare effort.

$$\tau' = p' \cdot tg\varphi_a, \ daN / cm^2 \qquad (3)$$

3. VERIFICATION ESTIMATION ON STABILITY OF THE REINFORCED LAND

Verification estimation on stability of the reinforced land is made considering two stages, namely: presizing of the work and verification of its safety coefficient.

a) Presizing estimation of the consolidation works

Presizing aims the setting the casing parameters, represented by the number of geo-textile layers (geo-grids) and its length, starting from a size given by the reinforcement height and followed by developing the constructing solution of the work, then to be made the verification of the stability at safety coefficient.

To determine the casing parameters can be used the following two processes [3], [4]:

- **1st Process** which presume the using of the friction circle method (fig. 4).

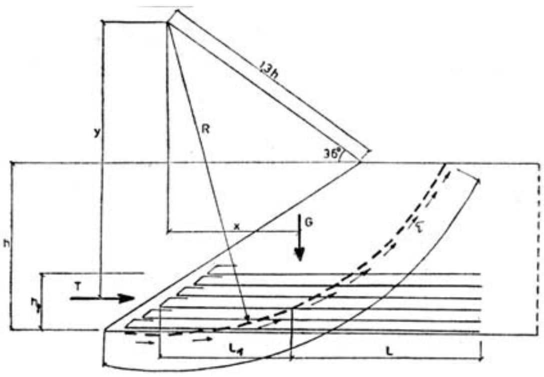

Fig. 4. Calculation scheme for pre-sizing the consolidation work

In this case, from the moments equation compared with the slide circle's center, results:

$$\tau LR + T_y = G_x K_s, \qquad (4)$$

From 4th relation is obtained the total traction force which must be taken over by the casing (T), respectively:

$$T = \frac{G_x K_s - \tau LR}{y}, \ daN \qquad (5)$$

where: G – slide's prism weight (the weight of given soil), DaN; τ – soil's cutting resistance, daN/cm^2; Ks – safety coefficeint; R, x, L – geometric elements (due to 4th fig.).

$$y = R - \frac{h_1}{2}, \; cm \quad (6)$$

h_l – embankment height on which the casing is made, cm.

The casing height is determined due to the casing's resistance on traction, namely:

$$h_l = 0{,}1\ h, \text{ when } R_c \text{ is large} \quad (7)$$

and

$$h_l = 0{,}3\ h, \text{ when } R_c \text{ is low} \quad (8)$$

Usually, the height h_l must be adopted in such a manor that the embankment with height h -h_l to be steady.

Due to the T force is determined the necessary geo-textile layer number, in accordance with the following relation:

$$n = \frac{T}{R_c}, \quad (9)$$

The distance between the casing rows is adopted due to the layer thickness on which is made the consolidation on execution of the soil construction. The minimum thickness of a layer is considered to be 30 cm, same with a minimum distance between the rows. In the case in which for the minimum adopted distance results a casing height different with more than 15 – 20% of the h_l value adopted in the estimation, then the estimation of pre-sizing is repeated, adopting other initial casing parameters.

The soil anchorage length (L) required for a geo-textile row is determined due to the relation:

$$L = \frac{TK_s}{2f \cdot \sigma \cdot n}, \; cm \quad (10)$$

where: f represents the friction coefficient between the geo-textlie and soil

$$f = tg\varphi_a \quad (11)$$

φ_a = 0,9 φ – for nonwoven geo-textiles with a harsh surface; φ_a = 0,6 φ – for woven or polypropylene, polyethylene or polyamide tapes; φ – soil interior friction angle; σ – medium pressure exercised on the geo-textile layer at the T force level;

$$\sigma = \gamma_a (h - \frac{h_1}{2}), \; daN \;/ cm \quad (12)$$

γ_a – soil's specific weight.

From practice, the functionality condition $L \geq l_{min} = 3{,}0$ m (13)

Due to the achievement measure of geo-textile loop, which forms anchoring elements on layer end, under the embankment, is not considered the establishment of L_l length for anchoring the casing to exterior.

- **2nd Process,** which consider the estimation relation (1), namely:

$$K_s = \frac{M_\varphi + M_c + M_P}{M_{al}}$$

Still, the stability estimation goes through the following stages:

- is determined the safety coefficient to stability of the uncased soil, namely

$$K_{nr} = \frac{M_\phi + M_c}{M_{al}}, \tag{14}$$

- is calculated the value of the casing stabilizer moment, according to the equation:

$$M_p = T_y = T(R - \frac{h_1}{2}), \tag{15}$$

In this case, the safety on stability deficient of the consolidation work will be:

$$K_s - K_{nr} = \frac{M_p}{M_{al}}, \tag{16}$$

From the relations (15) and (16) is obtained the expression of the traction force which must be taken by the casing, namely:

$$T = \frac{(K_s - K_{nr}) \cdot M_{al}}{R - \frac{h_1}{2}}, \tag{17}$$

Further, the determination of the casing parameters (n, h_1, T) is made in accordance with the algorithm from the 1st procedure.

b) The verification estimation of the safety at stability coefficient

The K_s safety coefficient estimation is made by the strips method, on which, between the stabilizing forces is included the traction force of the casing material [2], [3], [4].

Due to the fact that the geo-textile material is arranged on multiple overlapped rows, it must be take into consider the number of casing elements existent in every strips, considering the resultant applied in the strip's estimation section center (fig. 5).

In this case, corresponding to the estimation scheme from the 6th figure, the estimation of the stability coefficient is made in accordance with the following formula:

$$K_s = \frac{\sum_{i=1}^{n} N_i tg\, \phi_i + \sum_{i=1}^{n} C_i l_i + \sum_{i=1}^{m} P_i \cos \alpha_i}{\sum_{i=1}^{n} T_i} \tag{18}$$

In the 18 equation, the significance of the parameters is the following:

P_i – the casing traction estimation resistance, as a resultant which corresponds to the casing layer number in a stripe; N_i, T_i – the normal, respectively the tangentially component on the i stripe's weight sliding plan; φ_i –soil's friction angle; C_i – soil's cohesion; i – stripe's disposal number; n – stripe's total number; m – the number of stripes which contain casing; l_i – the length of a stripe.

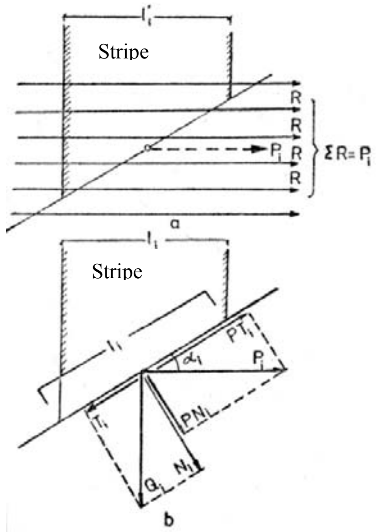

Fig. 5. The force system which manifests on the stripe's estimation section:
a – the result of the casing forces; b – weight and casing resultant components

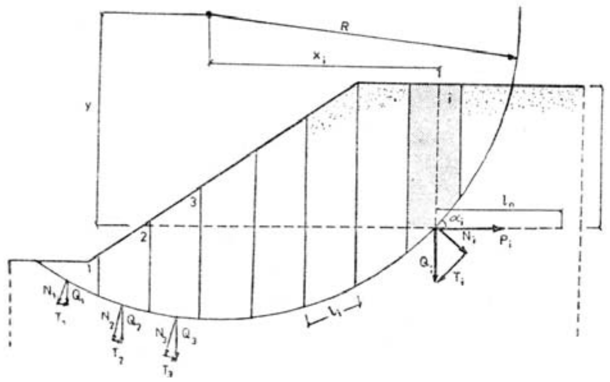

Fig. 6. The K_s stability coefficient estimation scheme

$$\cos\alpha_i = \frac{y_i}{R}; \qquad l_i = \frac{l_i^{'}}{\cos\alpha_i}, \tag{19}$$

Note that if the result is not consistent, is reckon on other casing parameters and the estimation is resumed. The final sizing of the work is made on the casing parameter base which meets the verification conditions of the stability coefficient.

4. CONCLUSIONS

The soil reinforcement with geo-textiles constitutes as a geotechnical measure for support and consolidation of the slopes by considerable reduction of the deformations and so increasing the rock's balance status.

The application domain of the soil's casing solutions with geo-textiles is represented, especially, by the consolidation of the excavation and embankment slopes which lost their stability and in the case of low depth slides of the sloping lands.

In comparison with other classic support solutions, the geo-synthetic material casing procedure is considered to be much more efficient from a technical-economical point of view, claiming for commissioning the works a much lower costs, materials and time, with low price.

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Scientific Reviewers:
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STUDIES ABOUT THE BEHAVIOR OF METALIC MAINTENANCE USED IN UNDERGROUND CONSTRUCTIONS WITH BOLTED ROOF AND VERTICAL WALLS AND THREE ARTICULATIONS

DĂNUȚ CHIRILĂ*
CRISTINA DURA**

Abstract: *This article shows an example about the behavior of metallic structures used in underground constructions. The article shows some results that were obtain in order to see what efforts appear in a metallic maintenance that it is used in a gallery from Livezeni Mine, Romania.*

1. INTRODUCTION

Metalic structures are used in underground construction very often that is why we should give them a little bit more attention. Metallic structures are use as temporally or definitive maintenance and their behavior depend on many things but in this article we will try to study the effort in the elements of a metallic maintenance used in a coal mine in Romania at Livezeni mine.

2. GENERAL DATAS NEDEED

For scaling the metallic maintenance first of all the data's about the soil proprieties, stress and pressure from above, lateral and down where gather.

The rocks meet there had:

σ_c =450daN/cm^2 - the first type of rock

σc =336daN/cm^2 - the second type of rock

f = 3 - 5,5 and it was adapted an f =4

The appreciation of the stability distinction was made by two hypotheses:

The M.M. Protodiakonov hypothesis in which the pressure of the rock from above is calculated:

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* *Prof. Ph.D. Eng. - University of Petrosani*
** *Assist. Prof. Ph.D. Eng. - University of Petrosani, dura.cristina@yahoo.com*

$$P = \frac{8a^2\gamma_a}{3f} = 7{,}85[t/m]$$

The Labasse hypothesis gave the result:

$$P= 2{,}6\ [t/m]$$

The P.M. Țimbarevici hypothesis in which the lateral pressure of the rock is calculated:

Xo= 0,075 m

Da=101,6 daN/cm^2 - active pressure

Dp= 51,3 daN/cm^2 - passive pressure

3. SCALING THE METALIC MAINTENANCE

As we already appreciate it, the metallic maintenance is very often use in coal mines, where the galleries are made very deep in the ground, at high pressure. In this cases not only that the metallic framework are used but those framework must be elastic in order to leave the pressure to stabilize itself after that the maintenance should become to its initial form. In this case the ladder is shown in the figure 1 and the framework is shown in figure 2.

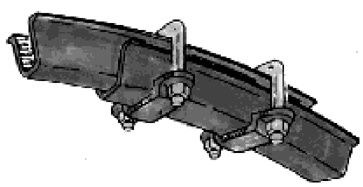

Fig. 1. Ladder

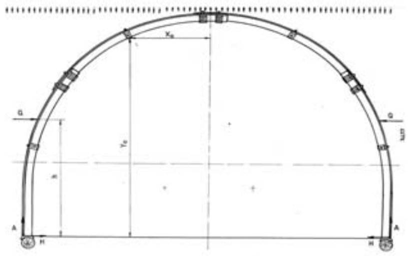

Fig.2. Framework with ladder

After analyzing the proprieties of the rock, the utility of the gallery, the massive pressure and all the factors that influence the gallery in any way the projecting of the maintenance of this gallery was made as it is shown in the figure 3.

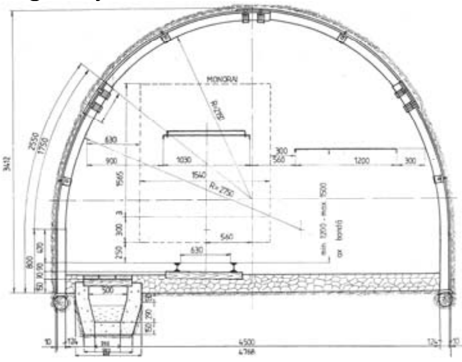

Fig. 3. The profile of the gallery

The main characteristics of this profile are shown in the table below:

Table 1. *Main characteristics of the GDM12 profile:*

Surface	**Unit**	**Value**
Practical surface	m^2	12,00
Digged surface	m^2	13,33
Total surface	m^2	13,76
Air debit	m^3/min	5760-10.800
Wather debit	m^3/hour	240
Height	m	3,412
Width	m	3,768
r (arch radius)	m	1,884

4. CALCULS OF THE EFFORTS THAT APPEAR IN MAINTENANCE WITH THREE ARTICULATIONS

Before we begin we must add the fact that the profile GDM12 have three articulations and so it represents a statically define system. This system is shown in figure 4.

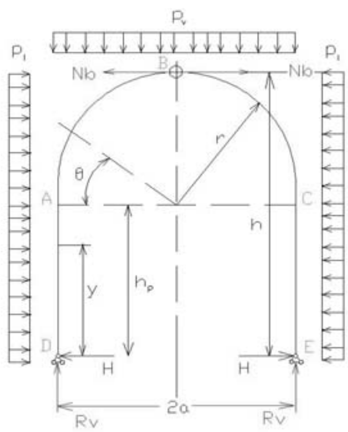

Fig.4. The system with three articulations

The reactions that appear are:

$$R_v = p_v r = p_v a = 7{,}85 \cdot 1{,}88 \qquad R_v = 14{,}75\ t$$

$$H = \frac{-p_v r^2 + p_l h^2}{2h} = \frac{-7{,}85 \cdot 1{,}88^2 + 2{,}6 \cdot 3{,}41^2}{2 \cdot 3{,}41} \qquad H = 0{,}36\ t$$

$$N_B = \frac{p_v r^2 + p_l h^2}{2h} = \frac{7{,}85 \cdot 1{,}88^2 + 2{,}6 \cdot 3{,}41^2}{2 \cdot 3{,}41} \qquad N_B = 122{,}98[t]$$

The hog moments are:

$$M_Y = \frac{p_l h^2 - p_v r^2}{2h} y - \frac{1}{2} p_l y^2 =$$

$$\frac{2{,}6 \cdot 3{,}41^2 - 7{,}85 \cdot 1{,}88^2}{2 \cdot 3{,}41} \cdot 1{,}5 - \frac{1}{2} 2{,}6 \cdot 1{,}5^2 \qquad M_Y = -3{,}053[t/m]$$

- in the section ABC:

$$M_\theta = \frac{p_l h^2 - p_v r^2}{2h}\left(r\sin\theta + h_p\right) + \frac{1}{2}p_l r^2 \sin^2\theta - \frac{1}{2}p_l\left(r\sin\theta + h_p\right)^2$$

$$= \frac{2{,}6\cdot 3{,}41^2 - 7{,}85\cdot 1{,}88^2}{2\cdot 3{,}41}\cdot\left(1{,}88\cdot\sin 20 + 1{,}67\right) + \frac{1}{2}2{,}6\cdot 1.88^2\sin^2 20 -$$

$$\frac{1}{2}2{,}6\left(1{,}88\sin 20 + 1{,}67\right)^2 \qquad M_\theta = -7{,}66[t/m]$$

In the articulations: $M_D = M_B = M_E = 0$

The longitudinal force in any of the sections is calculated:

$$N_\theta = p_v r\cos^2\theta - \frac{p_l h^2 - p_v r^2}{2h}\sin\theta + p_l\left(h_p + r\sin\theta\right)\sin\theta =$$

$$7{,}85\cdot 1{,}88\cos^2 20 - \frac{2{,}6\cdot 3{,}41^2 - 7{,}85\cdot 1{,}88^2}{2\cdot 3{,}41}\sin 20 + 2{,}6\left(1{,}67 + 1{,}88\sin 20\right)\sin 20 \qquad N_\theta = 16{,}68[t]$$

The transverse forces have the expressions:

$$Q_y = \frac{p_l h^2 - p_v r^2}{2h} - p_l y = \frac{2{,}6\cdot 3{,}41^2 - 7{,}85\cdot 1{,}88^2}{2\cdot 3{,}41} - 2{,}6\cdot 1{,}5 \qquad Q_y = -3{,}54[t]$$

$$Q_\theta = p_v r\cos\theta\sin\theta + \frac{p_l h^2 - p_v r^2}{2h}\cos\theta - p_l\left(h_p + r\sin\theta\right)\cos\theta =$$

$$7.85\cdot 1{,}88\cos 20\sin 20 + \frac{2{,}6\cdot 3{,}41^2 - 7{,}85\cdot 1{,}88^2}{2\cdot 3{,}41}\cos 20 - 2{,}6\left(1{,}67 + 1{,}88\sin 20\right)\cos 20 \qquad Q_\theta = -0{,}59[t]$$

in the point B: $Q_B = 0$

5. CONCLUSIONS

The results above shows how does the maintenance with three articulations behave under solicitations. We must say once again that our profile GDM 12 has three articulations and for this profile the results are given above. This results that we obtain means that for the pressure that was given, for the rock characteristics and for all the factors that influence the cavity of the gallery, the three articulations resolve the problem by gliding as much as it is needed for the pressure to stabilize.

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Scientific Reviewers:
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THE PHYSICAL CHARATERISTICS DETERMINATION OF ANDESITES FROM THE SOUTHERN APUSENI MOUNTAINS USING REGRESSION AND CORRELATION ANALYSIS

CIPRIAN DANCIU*
VICTOR ARAD**
TUDOR GOLDAN***
CATALIN NISTOR****

Abstract: *The correlation relations between physical properties were based on regression equations and standard deviation resulting from the experimental data. The type of regression (linear, logarithmic, polynomial, power, exponential) was chosen for each correlation depending on the standard deviation (R^2) derived for that set of data.*

Keywords: *geomechanical, regression, empirical relationships.*

1. INTRODUCTION

The making and preparation of samples for geomechanical characteristics determination is often an expensive and time and energy consuming process, and estimating various properties can be conducted using empirical correlations.

Since many of the geomechanical characteristics of rocks are closely interlinked, with the help of functional correlation relations a quantitative and qualitative analysis can be achieved.

The correlation relations between physical properties were based on regression equations and standard deviation resulting from the experimental data. The type of regression (linear, logarithmic, polynomial, power, exponential) was chosen for each correlation depending on the standard deviation (R^2) derived for that set of data.

This paper will present a series of correlations between physical characteristics of andesites resulting from experimental data obtained in the laboratory.

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* *Assist. Prof. Ph.D. Eng. - University of Petrosani*
** *Prof. Ph.D. Eng. - University of Petrosani*
*** *Assoc. Prof. Ph.D. Eng. - University of Petroşani*
**** *Student Ph.D. Eng. - University of Petrosani*

2. PHYSICAL CHARACTERISTICS OF ANDESITES

Knowledge of andesite physical state can help a quantitative description, but also to estimate the influence on strength and deformation characteristics (Arad, 2009).This can be achieved only by physical characteristics determination: specific density, apparent density, porosity, pore number, compactness and natural moisture.

The determination of physical properties was conducted in the Geomechanics Laboratory from University of Petrosani, in accordance with the standards, rock mechanics recommendations of the International Bureau and the International Society of Rock Mechanics.

Tab. 1. *Average values of physical characteristics obtained for andesites [Danciu 2007]*

Test no.	Rock type	Physical characteristics					
		Specific density (real) $\gamma \times 10^4$ [N/m^3]	Apparent density (Volume) $\gamma_a \times 10^4$ [N / m^3]	Total porosity n [%]	Pore number e	Compactness C [%]	Natural moisture W [%]
1.	Andesite	2.7712	2.6682	3.7167	0.0385	96.2831	0.9595
2.	Andesite	2.7325	2.6129	4.3744	0.0457	95.6254	1.0867
3.	Andesite	2.6558	2.6397	0.6061	0.0060	99.3937	0.7963
4.	Andesite	2.6515	2.4574	7.3201	0.0789	92.6797	2.7668
5.	Andesite	2.6931	2.6734	0.7339	0.0073	99.2660	0.6404
6.	Andesite	2.7124	2.6440	2.5229	0.0258	97.4769	1.1630

Note: Collecting place 1 – Certej-Valea Căpitanului; 2 – Deva-Dealul Motor; 3 – Brad-Criscior; 4 – Săcărâmb; 5 – Albini-Haneş; 6 – Roşia Poieni-Dealul Jgheabului. Note: a place of collection - Certej-Valley Captain, 2 - Deva-Hill Engine, three - Brad-Criscior, 4 - Săcărâmb, 5 - Albini, Hanes, 6 - Rosia Poieni-Hill trough.

3. CORRELATIONS BETWEEN ANDESITE PHYSICAL PROPERTIES AND EMPIRICAL RELATIONSHIPS DETERMINATION

Following laboratory analysis of experimental data obtained on several data sets for the Apuseni South andesites - were determined based on regression equations and standard deviation (R^2) - following empirical relationships for calculating the correlations and physical characteristics were determined. Empirical relations aim at simplifying the way of determination for certain physical properties.

The correlation between apparent density, porosity and pore number.

Analyzing the relationship between apparent density, porosity and pore number for andesites, it was established that there are analytical relationships of polynomial form (ec.1 and 2) between these parameters with standard deviation (R^2) of 0.948.

The graphical representation of equations 1 and 2 is depicted in Figure 1.

Tab. 2. *Empirical relations for determining the apparent density for andesites*

Correlations	Empirical Relation	Correlation Report R^2	Rock type
Apparent density (ρ_a) - porosity (n)	$\rho_a = -0{,}072 \cdot n^2 + 0{,}0282 \cdot n + 2{,}6382$ **(1)** **0.5 <n <8**	**0.948**	**Andesite**
Apparent density (ρ_a) - pore number (e)	$\rho_a = -58{,}93 \cdot e^2 + 2{,}3281 \cdot e + 2{,}6412$ **(2)** **0.005 <e <0.08**	**0.9489**	**Andesite**

Notations: **(1, 2) - polynomial regression;**

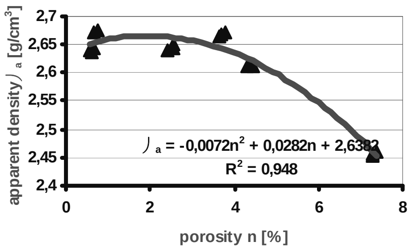

Fig. 1. Correlation between apparent density, porosity and pore number for andesites

Correlation between porosity and moisture.

Porosity is closely related to moisture and as demonstrated by the graphs of both parameters (Fig. 2), which shows a polynomial increase of moisture, corresponding to a relatively large variation in porosity of about 7.5%. The relationship between porosity and moisture is determined using the analytical relations in Table 3.

Tab.3. *Empirical relations for determining the porosity and moisture to andesites*

Correlations	**Empirical Relation**	**Correlation Report R^2**	**Rock type**
Porosity (n) - moisture (W)	$n = -1{,}8656 \cdot W^2 + 9{,}5338 \cdot W + 4{,}7868$ (3) 0.5 <W <3	0.8486	Andesite
Humidity (W) - porosity (n)	$W = 0{,}0552 \cdot n^2 - 0{,}1491 \cdot n + 0{,}8509$ (4) 0.5 <n <8	0.9416	Andesite
Notations: (3, 4) - polynomial regression;			

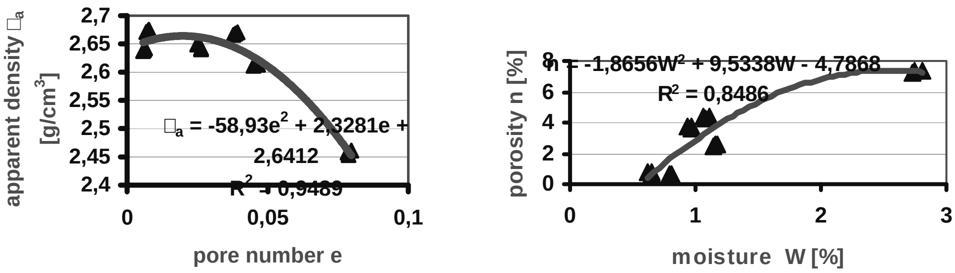

Fig. 2. Correlation between moisture content and porosity for andesites

Correlation between porosity and compactness.

Compactness is a physical parameter that directly depends on porosity, referring to the effect that an increase in porosity causes a decrease of compactness. Based on experimental data it was established that there are linear analytical relations between these two physical characteristics, as those in Table 4, with a correlation coefficient (R^2) of 0.9988.

Tab. 4. *Empirical relations for determining the porosity and for andesite compactness*

Correlations	Empirical Relation	Correlation Report R^2	Rock type
Porosity (n) - compactness (C)	$n = -0,9994 \cdot C + 99,94$ (5) 92 <R <99.5	0.9988	Andesite
Compactness (C) - porosity (n)	$C = -0,9994 \cdot n + 99,998$ (6) 0.5 <n <8	0.9988	Andesite
Notations: (5, 6) - linear regression;			

In fig.3, the graphs show the linear variation of porosity and compactness.

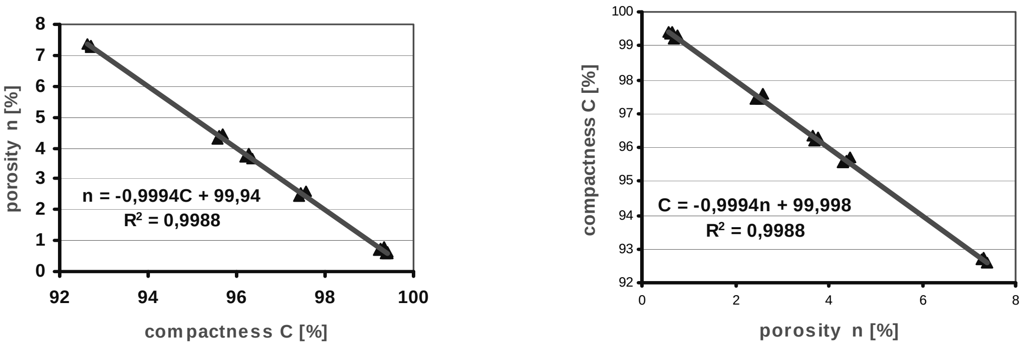

Fig. 3. Correlation between porosity and compactness for andesites

Correlation between moisture content and compactness.

Empirical relations illustrating the link between humidity and compactness for the South Apuseni andesites are presented in Table 5 and graphical representations of equations 7 and 8 are shown in Fig.4. From the graphical representation one can observe that humidity decreases with an increasing compactness.

Tab. 5. *Empirical relations for the determination of moisture and compactness for andesites*

Correlations	Empirical Relation	Correlation Report R^2	Rock type
Humidity (W) - compactness (R)	$W = 0,0552 \cdot C^2 - 10,898 \cdot C + 538,27$ (7) 92 <R <99.5	0.9407	Andesite
Compactness (C) - humidity (W)	$C = 97,176 \cdot W^{-0,048}$ (8) 0.5 <W <3	0.848	Andesite
Notations: (7) – polynomial regression (8) - power regression;			

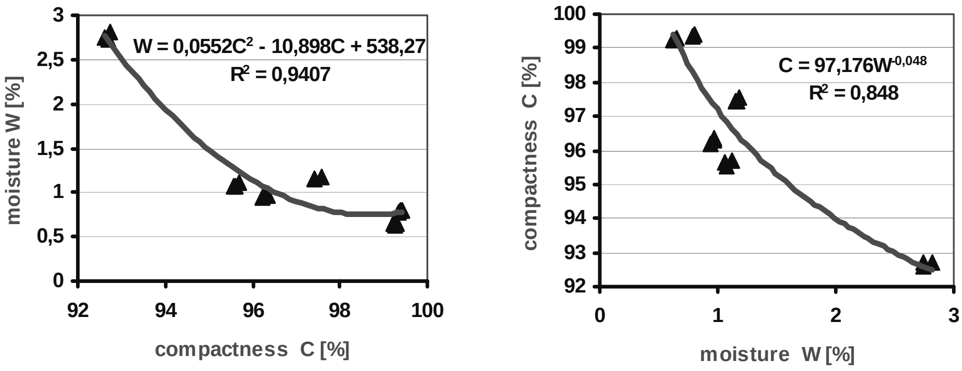

Fig. 4. Correlation between moisture and Compactness for andesites

4. CONCLUSIONS

Laboratory tests were conducted on six types of andesite to investigate the physical characteristics and correlations between them. Analyzing the results yielded the following conclusions:

The correlation between physical characteristics of the South Apuseni andesites was analyzed by the method of least squares regression.

Obtained values for correlation coefficient (R^2) for analyzed correlations are greater than 0.8 and are considered as statistically significant.

Of the correlations examined the link between porosity and compactness proved to be the best correlation of experimental data obtained for andesites, with a correlation coefficient $R^2 = 0.9988$.

Porosity influences compactness directly by the fact that a rise in porosity produces a decrease in compactness.

Humidity is a natural characteristic which depends directly on porosity and compactness as humidity decreases with decreasing porosity and increasing compactness.

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Scientific Reviewers:
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GEOMECHANIC CONSIDERATIONS ON MAGMATIC BASIC ROCKS IN SOUTHERN APUSENI

CIPRIAN DANCIU*
VICTOR ARAD**
TUDOR GOLDAN***
CATALIN NISTOR****

Abstract: *The paper summarizes the results of assessment of geo mechanical characteristics of magmatic rock samples collected from the Southern Apuseni: Căzăneşti, Branişca and Dobra. Because rocks can be used in construction it is necessary that they satisfy certain quality conditions. The papers deal with the physical, mechanical, technology of the dry rocks, saturated, and subjected to 25-50 cycles of freeze-thaw. Coefficients were determined by soaking, gelivity and values obtained were compared with those required by standards. The values obtained for the geo mechanical characteristics of rocks were compared with the conditions of admissibility imposed by standards.*

Keywords: *properties, geomechanical, rocks, standard, construction*.

1. INTRODUCTION

Widespread use of many types of rocks in construction and construction materials industry requires a detailed knowledge of their geomechanical characteristics. Using rocks to the execution of various works is only allowed if they fall within certain quality criteria. In this context, in the Geomechanics Laboratory from University of Petrosani, geomechanical properties of rocks from Southern Apuseni were determined in order to be used in construction and their characteristics were compared with the limits imposed by standards.

2. PHYSICAL CHARACTERISTICS

Physical knowledge of rock can help a quantitative description and to estimate the influence on strength and deformation characteristics. This can be achieved only by

* *Assist. Prof. Ph.D. Eng. - University of Petrosani*
** *Prof. Ph.D. Eng. - University of Petrosani*
*** *Assoc. Prof. Ph.D. Eng. - University of Petroşani*
**** *Student Ph.D. Eng. - University of Petrosani*

physical characteristics: specific density, apparent density, porosity, natural moisture content, saturation moisture, water absorption.

Determination of physical properties was carried out in accordance with the standards, with recommendations from International Bureau of Rock Mechanics and the International Society of Rock Mechanics. Methods of calculation and relations are found in various paper works [1, 2, 3, 4 and 5].

By determining the physical parameters, in accordance with specific standards and procedures, were recorded following average values for the analyzed samples (Tab.1 and 2).

Tab.1. *Mean values for the physical characteristics of rocks*

Sample no.	Rock type	Physical characteristics					
		Specific density (real) γ x 10^4 [N/m^3]	Apparent density (volumetric) γ_a x 10^4 [N/m^3]	Total porosity n [%]	Pore number e	Compactness c [%]	Density degree K_d
1.	Gabbrou	2,8887	2,8449	1,5162	0,0153	98,4837	0,9848
2.	Diabaz	2,7852	2,7499	1,2673	0,0127	98,7326	0,9872
3.	Basalt	2,7289	2,7046	0,8916	0,0089	99,1082	0,9910
4.	Basalt	2,7248	2,6994	0,9309	0,0093	99,0689	0,9906

Note: SAMPLING POINT 1 – Căzăneşti; 2 – Căzăneşti; 3 – Brănişca; 4 – Dobra.

Tab.2. *Mean values for the physical characteristics of rocks*

Sample no.	Rock type	Physical characteristics					
		Water absorption		Saturation coefficient s	Natural moisture W [%]	Saturation moisture W_{sat} [%]	Saturation degree S
		at normal pressure [%]	by boiling [%]				
1.	Gabbrou	0,4274	0,5406	0,7902	0,2271	0,4274	0,5313
2.	Diabaz	0,6951	0,7324	0,9486	0,3922	0,6951	0,5642
3.	Basalt	1,2342	1,2807	0,9636	0,7758	1,2342	0,6285
4.	Basalt	1,3361	1,5229	0,8769	0,9018	1,3361	0,6749

Note: SAMPLING POINT 1 – Căzăneşti; 2 – Căzăneşti; 3 – Brănişca; 4 – Dobra.

3. STRENGTH CHARACTERISTICS

In accordance with the objective, mechanical strength were determined as follows: compressive breaking strength, tensile breaking strength by splitting, cohesion and angle of internal friction, freeze-thaw resistance, hardness, abrasion and strength coefficient Protodiakonov.

Methods of calculation and relations used are according to existing, recommendations of the International Bureau of Rock Mechanics and of the

International Society of Rock Mechanics and some of them are found in specialized paper works [1, 2, 3, 4 and 5].

Tab.3. *Mean values for the strength characteristics*

Sample no.	Rock type	Strength characteristics						
		Monoaxial compressive breaking strength σ_{rc} [MPa]; [N/mm^2]			**Softening coefficient after saturation η_s [%]**	**Softening coefficient after freeze-thaw cycle η_g [%]**	**Relative strength coefficient k_r**	**Saturation coefficient k_s**
		Sample condition						
		dry	saturated	freeze - thaw				
1.	Gabbrou	150,173	131,375	124,644	12,483	16,942	0,875/87,5	1,143
2.	Diabaz	161,117	138,883	129,017	13,757	19,882	0,862/86,2	1,160
3.	Basalt	170,839	162,106	157,856	5,088	7,619	0,949/94,9	1,054
4.	Basalt	179,586	170,944	166,613	4,813	7,187	0,952/95,2	1,051

Note: SAMPLING POINT 1 – Căzăneşti; 2 – Căzăneşti; 3 – Brănişca; 4 – Dobra.

Tab.4. *Mean values for the strength characteristics*

Sample no.	Rock type	Strength characteristics										
		Tensile breaking strength by splitting (Brazilian method) σ_{rc} [MPa]; [N/mm^2]			**Softening coefficient after saturation η_s [%]**	**Softening coefficient after freeze-thaw cycle η_g [%]**	**Cohesion C [MPa]**			**Angle of internal friction φ [°]**		
		Sample condition					Sample condition					
		dry	saturated	freeze - thaw			dry	sat.	f-t	dry	sat.	f-t
1.	Gabbrou	18,829	16,740	16,239	10,966	13,712	26,5	23,4	22,4	50,9	50,7	50,3
2.	Diabaz	15,200	13,460	12,796	11,475	15,771	24,7	21,6	20,3	55,8	55,4	55,0
3.	Basalt	20,404	19,437	19,114	4,708	6,308	29,4	28,0	27,4	51,8	51,8	51,6
4.	Basalt	17,677	16,819	16,651	4,840	5,778	28,1	26,8	26,3	55,1	55,1	54,9

Note: SAMPLING POINT 1 – Căzăneşti; 2 – Căzăneşti; 3 – Brănişca; 4 – Dobra.

Tab.5. *Mean values for the strength characteristics*

Sample no.	**Rock type**	Strength characteristics							
		Freeze – thaw strength			**Hardness after Mohs Δ**	**Abrasion (Baron method) K_{abr}, [mg]**	**Strength coefficient (hardness) after Protodiakonov f**		
		Number or samples with obvious damage	Gelivity coefficient μ_g	Elasticity module decrease after 25 freeze-thaw cycles Δ [%]			Sample condition		
							dry	sat.	f-t
1.	Gabbrou	0	0,022	6,240	5,355	43,25	15,01	13,13	12,46
2.	Diabaz	0	0,033	8,361	5,683	21,83	16,11	13,88	12,90
3.	Basalt	0	0,057	4,246	5,025	34,71	17,08	16,21	15,78
4.	Basalt	0	0,075	3,753	4,999	30,18	17,95	17,09	16,66

Note: SAMPLING POINT 1 – Căzăneşti; 2 – Căzăneşti; 3 – Brănişca; 4 – Dobra.

4. PHYSICAL AND STRENGTH CHARACTERISTICS OF THE AGGREGATES

Using natural aggregates in various works requires knowledge and determination of physical and mechanical characteristics, which depend on the nature and purpose of aggregates used (road works, railways, mortar, concrete).

Physico-mechanical properties values which must be satisfy by natural aggregates are provided in terms of eligibility.

Physical characteristics determined in the laboratory are: specific density, apparent density, density of the pile in loose or pressed dry, porosity, water absorption and pore volume. Of the mechanical characteristics were determined: resistance to crushing dry and saturated, coefficient of softening, breaking strength and compression index in a saturated state, strength and shock index to smash through the dry, wear resistance, freeze-thaw resistance. Mean physical characteristics and strength of aggregates made from basic rocks-Gabbrou, Diabaz, Basalt are shown in Tables 6 and 7.

Tab.6. *Mean values for the physical characteristics of the aggregates*

Sample no.	Rock type	Physical characteristics						
		Specific density $\rho x\ 10^3$ [kg/m^3]	Apparent density $\rho_{ap} x\ 10^3$ [kg/m^3]	Density in dry lot		Water absorption a [%]	Total porosity n_{aw} [%]	Pore volume V_g [%]
				loose $\rho_a x\ 10^3$ [kg/m^3]	compact $\rho_{ai} x\ 10^3$ [kg/m^3]			
1.	Gabbrou	2,8887	2,8233	1,4960	1,7392	0,5693	2,2662	47,0140
2.	Diabaz	2,7852	2,7199	1,4407	1,5792	1,5662	2,3443	47,0310
3.	Basalt	2,7289	2,6716	1,3708	1,6138	1,5104	2,1008	48,6880
4.	Basalt	2,7248	2,6773	1,3936	1,6383	1,3687	1,7441	47,9459
Note: SAMPLING POINT 1 – Căzăneşti; 2 – Căzăneşti; 3 – Brănişca; 4 – Dobra.								

Tab.7. *Mean values for the physical characteristics of aggregates*

Sample no.	Rock type	Physical characteristics								
		Crushing resistance		Softening coefficient	Resistance to crushing by compression in a saturated state R_c [%]	Resistance to crushing by shock in a dry state $R_ş$ [%]	Resistance to freeze-thaw cycles μ_g [%]	Wear resistance		
		sat. R_{sa} [%]	dry R_{su} [%]					LOS ANGELES LA [%]	DEVAL R_{uz}	Quality coefficient C
1.	Gabbrou	76,24	11,13	6,87	77,32	95,08	0,143	12,2	2,52	15,8
2.	Diabaz	71,02	12,02	5,90	75,33	93,76	0,177	15,8	3,04	13,1
3.	Basalt	74,26	11,29	6,58	74,06	95,95	0,085	13,7	2,55	15,6
4.	Basalt	72,64	9,21	7,89	72,88	93,11	0,098	14,8	2,60	15,3
Note: SAMPLING POINT 1 – Căzăneşti; 2 – Căzăneşti; 3 – Brănişca; 4 – Dobra.										

5. ADMISSIBILITY CONDITIONS FOR ROCKS USED FOR RAILWAYS AND ROADS CONSTRUCTION

Rocks (magmatic, metamorphic, sedimentary) used for road works need to be homogeneous in terms of structure and composition mineralogo-petrographic, not showing any physical or chemical deterioration. They must be free of pyrite, limonite or soluble salts and not to contain microcrystalline or amorphous silica, which react with alkali in cements.

Natural aggregates composed of altered rock, soft, friable, porous that contain more than 10% particles for crushed stone, and 5% particles for chippings are not recommended.

Based on the main physical and mechanical characteristics, the rocks used in the manufacture of natural stone are classified into five classes of eligibility, according to Table 8.

Resistance to crushing by compression in a dry state and wear resistance is determined with Los Angeles device. Crushed stone wear is studied on grades from 40 to 63.

Rocks that do not comply with all conditions of eligibility in Table 8, may be classified according to the apparent porosity or wear resistance by using a Los Angeles device, decisive being the lower class indicated.

Rock which does not meet the eligibility for freeze- thaw resistance should not be used to road works.

Based on average values of key physical and mechanical characteristics obtained for the rocks analyzed and compared with the conditions of eligibility, each type of rock is classified in one of the five classes of eligibility (Table 8).

Tab.8. *Conditions of admissibility of natural rocks used in road and railway works*

CHARACTERISTIC	ROCK CLASS								
	A	B	C	D	E	B	B	A	A
	Eligibility conditions					Rock type			
						Gabbrou (Căzăneşti)	Diabaz (Căzăneşti)	Basalt (Brănişca)	Basalt (Dobra)
Apparent porosity at normal pressure, %, max.	1	3	5	8	10	1,516	1,267	0,891	0,930
Resistance to compression in dry state, N/mm^2, min.	160	140	120	100	80	150,173	161,117	170,839	179,586
Wear resistance with Los Angeles device, %, max.	16	18	22	25	30	12,204	15,815	13,724	14,825
Crushing resistance by compression in dry state, %, min.	70	67	65	60	50	77,326	75,338	74,061	72,885
Freeze – thaw resistance: - gelivity coefficient (μ_{25}) ,%, max.	3					0,022	0,033	0,057	0,075
- frost sensibility (η_{d25}), %, max.	25					16,942	19,882	7,619	7,187

CONCLUSIONS

After determining the geomechanical characteristics and comparison with the conditions of eligibility, the following conclusions can be drawn for the analyzed rocks:

• analyzed rocks are compact, having a high density, which results from the comparison of specific densities and apparent densities. Porosity and pore number has relatively low values characteristic of volcanic rocks, once again proving their qualities;

• compressive fracture resistance is quite high, ranging between 150-180 MPa, with a class of eligibility between A and B;

• wear resistance determined using a Los Angeles device has low values and resistance to crushing by compression in a dry state ranks above the class A required conditions for admissibility, recommending these rocks for railway and road works;

• according to the values obtained for gelivity coefficient and softening coefficient after freeze-thaw cycles, it is considered that these rocks are resistant to freeze-thaw conditions;

• after comparing the results obtained with the conditions of admissibility, analyzed rocks can be ranked into two classes. Branisca basalt and Dobra basalt in admissibility class A and the Căzăneşti Diabaz and Gabbrou in admissibility class B.

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Scientific Reviewers:
Prof. PhD. Eng. Ilie Onica

NEW TOOLS FOR ASSESSMENT OF NON-ELECTRICAL EQUIPMENT INTENDED USE IN FIREDAMP UNDERGROUND MINES, RELATED TO EUROPEAN DIRECTIVE ATEX 94/9/EC, ADOPTED IN ROMANIA BY GOVERNMENT DECISION NO. 752/2004

**EMILIAN GHICIOI*
MIHAELA PĂRĂIAN**
LEONARD LUPU***
ADRIAN MARIUS JURCA******

Abstract: *The new tools for assessment of non-electrical Ex equipments consist in the series of specific procedures for evaluating and connected testing procedures. The series of mentioned procedures are based on the harmonized European norms, adopted in Romania, which are included in SREN 13463 standard series and deal "Non electrical equipment for use in potentially explosive atmospheres". The assessment procedures are elaborated and implemented into Product Certification Body – INSEMEX OEC, notified at Bruxelles for ATEX Directive. Also, the testing procedures are elaborated and implemented into Laboratory for Testing in Explosive atmospheres – LIEx.*

Key words: *nonelectrical equipment, assessment of explosion proof protection.*

1. INTRODUCTION

Within the scope of the project PN 07-45-02-05 entitled „Elaboration of assessment tools necessary to assess the safety degree in non-electrical equipment intended for use in potentially explosive atmosphere, according to European Directive ATEx requirements" there had been obtained the following results – new assessment tools implemented in quality system management of the certification body INSEMEX-SECEEx, notified to Bruxelles, NB 1809:

a) Specific procedure of conformity assessment PSp06, under SR EN 13463-1:2003 referent - Non-electrical equipment for potentially explosive atmosphere. Part 1: Basic method and requirements;

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* *Ph.D. Eng. - Senior scientific researcher II – INCD INSEMEX Petroşani*
** *Ph.D. Eng. - Senior scientific researcher I – INCD INSEMEX Petroşani*
*** *Ph.D.Eng. – Senior scientific researcher – INCD INSEMEX Petroşani*
**** *Ph.D.Eng. – Senior scientific researcher III – INCD INSEMEX Petroşani*

b) Specific procedure of conformity assessment PSp-07, under SR EN 13463-2 Non-electrical equipment for use in potentially explosive atmospheres - Part 2: Protection by flow restricting enclosure 'fr'.

c) Specific procedure of conformity assessment PSp-08, under SR EN 13463-3:2006 Non-electrical equipment for potentially explosive atmospheres - Part 3: Protection by flameproof enclosure 'd'.

d) Specific procedure of conformity assessment PSEp-09, under SR EN 13463-5:2006 Non-electrical equipment for potentially explosive atmospheres - Part 5: Non-electrical equipment intended for use in potentially explosive atmospheres - Part 5: Protection by constructional safety "c".

e) elaborated and implemented the tests procedures associated for the new assessment conformity tools in the approved laboratory INSEMEX LIEx.

For elaboration of the new tools for conformity assessment of the non-electrical equipment with explosion protection type have covered the following stages:

a) analysis the specific requirements;

b) design the specific procedure and the associated test procedures;

c) implementation of the new created assessment tools and the associated tests in the INSEMEX quality system.

2. THE NEW CONFORMITY ASSESSMENT TOOL PSP-06

Is apply by the INSEMEX-SECEEx on conformity assessment and product certification – non-electrical equipment design to be use in potentially explosive atmosphere, according SR EN 13463-1:2003 - Non-electrical equipment for potentially explosive atmosphere. Part 1: Basic method and requirements;

For the application of this new conformity assessment tool was revised the following test procedures of the approved laboratory INSEMEX LIEx:

- The test procedure PI-Ex-06.4 – Heat test for determination of the maximum surface temperature (for equipment design to be use in potentially explosive atmosphere generated by the gas, vapors and mist);

- The test procedure PI-Ex-06.5 – Heat test for determination of the maximum surface temperature (for equipment design to be use in the presence of the combustible dusts);

- The test procedure PI-Ex-06.49 – The test of determination of the maximum surface temperature to the equipment design to be use in potentially explosive atmosphere created by the combustible dusts.

Also, it was necessary to elaborate the following test procedures, which have been implemented in the quality system of the approved laboratory INSEMEX-LIEx:

- The test procedure PI-Ex-06.56 - the tests of the nonmetallic parts for non-electrical equipment design to be use in potentially explosive atmosphere;

- The test procedure PI-Ex-06.57 - the tests of electrostatic charge with nonconductive materials for nonmetallic parts of the non-electrical equipments design to be use in potentially explosive atmosphere;

- The test procedure PI-Ex-06.58 – The mechanic test for non-electrical equipments design to be use in potentially explosive atmosphere;

The specific procedure PSp-06 was design and implemented in the quality system of the certification body INSEMEX-SECEEx, notified to Bruxelles with allocated number 1809, having in view all articles from the documents of the quality system. The specific procedures PSp-06 is an application for non-electrical equipments design to be use in potentially explosive atmosphere, according to the SR EN 13463-1 and PS-01-07 SECEEx system procedures for Ex equipment certification.[1][2][3]

Was used the following specific definitions:

- equipment : means machines, apparatus, fixed or mobile devices, control components and instrumentation thereof and detection or prevention systems which, separately or jointly, are intended for the generation, transfer, storage, measurement, control and conversion of energy or the processing of material and which are capable of causing an explosion through their own potential sources of ignition.[1] [3]
- equipment Ex : generic name for equipment and protective systems design to be use in potentially explosive atmosphere[1].
- non-electric equipment – equipment that can perform the function without using electricity.[1]
- the type of protection:
    - Protection by flow restricting enclosure ''fr'';
    - Protection by flameproof enclosure ''d'';
    - Protection by constructional safety "c";
    - Protection by control of ignition source ''b'';
    - Protection by liquid immersion ''k'';
    - Protection by pressurization ''p'. [4]
- potentially ignition source – any ignition source that may occur inside the equipment.[4]
- effective ignition source – ignition source able to ignite an explosive atmosphere.[4]

Specific rules of procedure to asses the non-electrical conformity with requirements from SR EN 13463-1:

a) Is define the general requirements and the correct group and category for equipment according with methodology by the SR EN 13463-1, annex 1.

b) Is asses the ignition hazard:
- hazard analyze;
- equipment assessment for from group I;
- equipment assessment for from group II;
- defects assessment;

c) Setting the maximum surface temperature;

d) assessment the ignition hazard;

e) Assessment report of the ignition hazard;

f) Specific assessment for group and category;
- for equipment from category 1;
- for equipment from category M1;

g) temperature assessment;

h) Checking of the nonmetallic parts of the equipment [4].

3.THE NEW CONFORMITY ASSESSMENT PSP-07

The specific assessment procedure applied by INSEMEX - SECEEX to assess conformity and certify the products - non-electric equipment intended for use in potentially explosive atmospheres, according to SR EN 13463-2:2006 Non-electrical equipment for use in potentially explosive atmospheres. Part 2: Protection by flow restricting enclosure “fr”.

There had been analyzed the requirements in the ATEX Directive 94/9, transposed as the Government Decision no. 752/2004 with all further modifications, regarding stipulation of the conditions for placing on the market of equipment and protective systems designed to be used in potentially explosive atmospheres, applicable to non-electric equipment, as well as the ones in SR EN 13463-2.

In order to apply this new tool for assessment of conformity, there had been developed and implemented in the accredited laboratory INSEMEX-LIEx, the following procedures:

- The Test Procedure Pi-Ex-06.59 Testing of non-electric equipment type "fr" with verification of restrictive circulation properties after commissionning/maintenance;
- The Test Procedure Pi - Ex - 06.60 Testing of non-electric equipment type "fr" without verification of restrictive circulation properties after commissionning/maintenance;
- The Test Procedure Pi - Ex - 06.61 Testing of non-electric equipment type "fr" where the enclosure's nominal volume varies according to pressure;

Assessment of non-electric equipment protected by flow restricting enclosure “fr” is carried out based on:

a) the explosive atmosphere around an enclosure can penetrate inside it, due to the influence of three mechanisms: ventilation, balancing of the pressure differences between outer and inner volumes (breathing) and diffusion.

b) equipment conforming to SR EN 13463-2 fulfill the requirements stated for the Group II Category 3 Equipment. The protection type "fr" is not adequate to be used in combination with another type of protection to obtain an equipment in a different category but 3.[5]

c) protection by enclosures with restrictive circulation is a type of protection that, through an enclosure, decrease the probability that an explosive atmosphere would penetrate the enclosure at a level sufficiently low so as the value of the concentration inside the enclosure to be lower than the Lower Explosion Limit.[5]

d) The enclosures with restrictive circulation without requirements of verification of the restrictive circulation properties after commissionning or maintenance must undergo the type tests, including the conduits or axis entries etc. These enclosures must, under constant temperature conditions, fulfil the requirement that the time required for an inner pressure 3 kPa (30 mbar) lower than the atmospheric pressure to reach 3 kPa (30 mbar) lower than the atmospheric pressure to be lesser than 80 seconds.[5]

e) The enclosures with restrictive circulation with requirements of verification after commissionning or maintenance must have drawn up provisions for the verifications, to ensure all the requirements for restrictive circulation are fulfilled after commissionning and maintenance. The enclosures must, under constant temperature conditions, fulfil the requirement that the time required for an inner pressure 300 Pa (3 mbar) lower than the atmospheric pressure to reach 150 Pa (1.5 mbar) lower than the atmospheric pressure to be lesser than 80 seconds.[5]

f) equipment where the nominal enclosure volume varies with pressure - the enclosure must be pressurized having the air kept at an overpressure of 400 Pa (4 mbar). The air intake flow required to maintain this overpressure must be measured, in liters per hour (l/h). This value divided by the netto volume of the enclosure, expressed in liters, should not overpass the value of 0,125 h^{-1}.[5]

g) Marking. Additionally to the marking requirements in EN 13463-1, the specific marking for the complete equipment that is required to prove conformity to this document, must include:

- the symbol "fr" (that represents the explosion protection type);
- a warning label to inform the user that, the equipment commissionning must be carried out in such manner to ensure there is no static pressure differences on the enclosure's walls, that could give birth to a circulation through the enclosure.[5]

4. THE NEW CONFORMITY ASSESSMENT TOOL PSP-08

The specific assessment procedure applied by INSEMEX - SECEEX to assess conformity and certify the products - non-electric equipment intended for use in potentially explosive atmospheres, according to SR EN 13463-3:2006 Non-electrical equipment for use in potentially explosive atmospheres. Part 3: Protection by flameproof enclosure “d”.

In order to apply this new tool for assessment of conformity, there had been revised in the accredited laboratory INSEMEX-LIEx, the following test procedure:

- The Test Procedure Pi-Ex-01.01 Testing for flameproof enclosures verification. Tests in explosive mixtures (a new edition and inclusion of the non-electric equipment within the procedure's scope).

Certain types of non-electric equipment intended for use in potentially explosive atmospheres with gas, vapors, mists and/or combustible dusts contain active sources of ignition in normal operation and it must be prevented that these become ignition sources for the surrounding atmosphere where the equipment operates. A method to reach this goal is to enclose the ignition sources into an enclosure, so as an ignition of the internal atmosphere cannot be transmitted in the outer atmosphere. This method is known as protection by flameproof enclosure "d".[6]

The basic principle of ignition protection by the use of a flameproof enclosure, is that gases, or vapour, may enter the enclosure through the cover joints/flanges and if an explosive atmosphere inside the enclosure ignites, neither the enclosure will be deformed significantly, nor flame transmitted through the joints/flanges to the explosive atmosphere outside. For this reason the enclosure has to be both robust and have dimensionally controlled cover joints/flanges with maximum allowable safe gaps appropriate for the types of explosive gas/vapour likely to occur inside the equipment. [6]

For electrical equipment, this type of protection is well known for protecting power arcing components and is defined and described in SR EN 60079-1. As the electrical equipment standard contains the generic testing, verification and marking requirements, unnecessary duplication of the requirements in the non-electrical equipment standard SR EN 13463-3 is avoided by cross reference to the electrical standard. In this standard only those differences necessary for the purpose of providing protection for non-electrical equipment are described in full.

In contrast to SR EN 13463-3, the standard SR EN 60079-1 does not consider explosive atmospheres formed by dusts, except for Group I, category M2 electrical equipment, where its associated general requirements document, SR EN 60079-0, states that flameproof equipment designed, constructed and tested for use in explosive atmospheres of firedamp (explosive mine gas consisting mainly of methane) needs no alteration, or further testing to allow it to be used where a coal dust cloud is present.

The concept of protecting equipment against dust cloud ignition by testing it in a gas / air mixture is also accepted in SR EN 13463-3 for both Group I, Category M2 mining equipment, and Group II, Category 2G and 2D non-mining equipment. This is because the standard introduces an acceptable safety factor against ignition and it allows a much more simple method of testing and verifying its explosion protection properties.

The non-electrical equipment that can be protected by flameproof enclosures fulfils the requirements for the following categories:

- Equipment Group I Category M2 – that does not contain an ignition source arising from severe operating conditions, in particular arising from rough handling and changing environmental conditions;
- Equipment Group II Category 2G or 2D – that does not contain an ignition source arising as a result of foreseeable malfunctions.

The type of ignition protection "d" can be used either on its own or in combination with other types of ignition protection to meet the requirements for equipment of Group I categories M2, or Group II categories 1 and 2 depending on the ignition hazard assessment in EN 13463-1.

Additionally to the standard SR EN 13463-1 requirements, the specific marking required to show conformity with the SR EN 13463-3 standard must include the symbol "d" (showing the type of explosion protection).[6]

5. THE NEW CONFORMITY ASSESSMENT TOOL PSPE-09

The specific assessment procedure applied by INSEMEX - SECEEX to assess conformity and certify the products - non-electric equipment intended for use in potentially explosive atmospheres, according to SR EN 13463-5:2006 " Non-electrical equipment for use in potentially explosive atmospheres. Part 5: Protection by constructional safety “c”.

In order to apply this new tool for assessment of conformity, there had been developed and implemented in the accredited laboratory INSEMEX-LIEx, the following procedures:

- The Test Procedure Pi-Ex-06.66 "Dry run" test for lubricated sealing arrangements;
- The Test Procedure Pi-Ex-06.67 - Type test for determining the maximum engaging time of clutch assembly.

One of the methods of applying ignition protection consists in choosing those types of equipment that do not containing an ignition source in normal service and then apply good engineering principles, so that risk of mechanical failures likely to create incendive temperatures or sparks, was reduced to a very low level. Such protective measures are referred to in the SR EN 13463-5 standard as ignition protection by 'Constructional Safety', or type of protection "c".[7]

Equipment complying with the relevant clauses of SR EN 13463-5 meets the requirements for the following categories:

- Equipment Group I Category M2 (for M1 equipment apply also the requirements given in EN 50303 which specifies the requirements for both electrical and non-electrical equipment);
- Equipment Group II Category 2G or 2D;
- Equipment Group II Category 1G or 1D.

The type of ignition protection constructional safety can be used either on it's own or in combination with other types of ignition protection to meet the requirements for equipment of Group I, categories M1 and M2 or Group II, categories 1 and 2 depending on the ignition hazard assessment.[7]

Before a decision is made to protect equipment or pieces of equipment for use as an assembly including interconnecting parts by the measures described in SR EN 13463-5 standard, it shall have been subjected to the ignition hazard assessment in accordance with EN 13463-1. Furthermore, It shall also have been determined that, by enhancing or increasing the safety of certain vulnerable parts, the required level of protection is ensured against the possibility of ignition sources occurring.

All parts and interconnecting parts of equipment shall be capable of functioning in conformity with the operational parameters established by the manufacturer throughout their foreseeable lifetime and be sufficiently firm and durable to withstand the mechanical and thermal stresses to which they will be subjected.

Additionally to the standard SR EN 13463-1 marking requirements, the specific marking required to show conformity with the SR EN 13463-5 standard must include:

- the symbol "c" (showing the type of explosion protection).[7]

6. THE PROJECT CODE PN 07-45-02-0 UNFOLDING ALONG 2009 HAS AS GOALS:

- a tool to assess conformity of non-electric equipment having the explosion protection type "control of ignition sources "b" according to SR EN 13463-6 Non-electrical equipment for use in potentially explosive atmospheres. Part 6: Protection by control of ignition source "b";[8]
- a tool to assess conformity of non-electric equipment having the explosion protection type "liquid immersion "k" according to SR EN 13463-6 Non-electrical equipment for use in potentially explosive atmospheres. Part 6: Protection by liquid immersion "k".

7. CONCLUSION

Non-electrical equipment had been used for over 150 years in industries having potentially explosive atmospheres. Flameproof non-electrical equipment testing and certification that is necessary in the conformity assessment process are very important, having in view the explosion hazard that occurs due to the presence of potentially explosive atmospheres.

Together with using them in the field of explosive atmospheres, either in underground mines or other surface industries it became necessary to elaborate specific requirements on the concept of protection against explosive atmospheres ignition, as well as to perform equipment assessment based on these requirements.

The paper presents the structure of an assessment tool that is necessary in the evaluation process of the security level of the non-electric equipment designed to be use in potentially explosive atmosphere, according to the ATEx European Directive requirements. This assessment tool is implemented in the quality system of the INSEMEX-SECEEx Notified Body, notified to Bruxelles with the code NB 1809.

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Scientific Reviewers:
Assoc.Prof. PhD. Eng. Roland Moraru

PERFORMING THE DATA USED IN A G.I.S.

HERBEI MIHAI VALENTIN[*]
HERBEI OCTAVIAN[**]
ULAR ROXANA CLAUDIA[***]

Abstract: *The digital map must be made by valorizing all the existent resources based on a rigorous analyze of their content and their costs following to be assured the necessary quality in conditions of maximum efficiency. Each source of data presumes the existence of some special programs which we should bring the respective data in shape correspondent to the digital map by staring from assuring the necessary equipments crossing through the technological problems and conversion ones of the data, reaching to correspondent training of the staff. In this paper it is presented the way of transforming the data from analogical form into digital form in order to be able to be presented and used by a GIS.*

Key words: *geographic data, non-geographical data attributes, GIS, coordinate transformation*

1. INTRODUCTION

The geographic data of a certain area are organized on many thematic layers. (Fig.1). The digital map of a certain area is represented by the sum of all defined layers. A derived map will be constituted in a layer or a certain combination of layers of the existent ones.

A possibility to make an acquisition of the data in order to make the numerical plans is constituted in the operation of computerizing the existent map material in analogical form. There are two methods of digitizing as follows:

- Classical digitize;
- Scanning – vectoring.

[*] *Ph.D. Eng.*
[**] *Prof. PhD. Eng. - University of Petrosani*
[***] *Assist. Prof. PhD. Eng. – University of Petrosani*

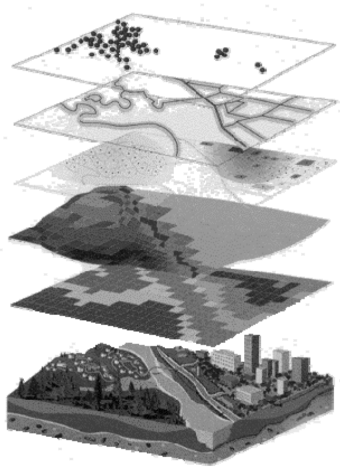

Fig.1. Layers in digital map

Table 1. *Comparison/ digitizing/ scanning – vectoring*

DIGITIZING	SCANNING-VECTORING
Easy way to learn	More complex way of work
Tired for eye and hands	Tired for eye
It is not good for level curves	good for level curves
Indicated for color handwritings	Indicated for editing originals
Possible for a models PC	It needs robust configurations

Data stocked in topographical data bases no matter what is the way of stocking them analogically or digitally may be separated in 2 different categories:

- Graphic data (stocked into maps)
- Non-graphical data or attributes (stocked into registers).

Between the two categories of data there are established mutual relationships with the help of some common codes. Recording on the computer the non-graphic data is not very difficult but crossing from the map or traditional topographical plan to digital form is very difficult.

2. BIDIMENSIONAL TRANSFORMING OF THE COORDINATES BETWEEN 2 REFFERENCE POINTS

The coordinate’s transformations serve to point transposition from a reference system to another. It presumes to be known the position of the two systems or a number of points known into both systems.

Into a rectangular system there are known the following elements:

- Coordinates of a point $P(x_2,y_2)$ into a rectangular system O_2 over the coordinates of the point $P(x_1,y_1)$ into the system O_1(Fig. 2):

$$x_2 = x_0 + a + c = x_0 + x_1 \cos\alpha + y_1 \sin\alpha$$
$$y_2 = y_0 - b + d = y_0 - x_1 \sin\alpha + y_1 \cos\alpha \quad (1)$$

where: x_0 and y_0 are the coordinates of the system 1 or the system 2, and ∝ represents the angle between the two correspondent axes of the two systems.

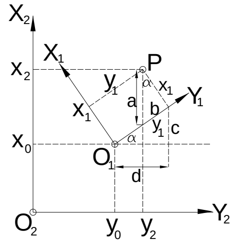

Fig.2. Rotation - translation of the axes

For example if it is considered 2 points $P_a(x_a,y_a)$ and $P_b(x_b,y_b)$ (Fig. 3) known into the systems I and II, if the rotation of the system I is inversed than into the above case, the relation of the 2 points from the two systems is as follows:

$$x_{bII} = x_{aII} + \Delta x_1 \cos\alpha - \Delta y_1 \sin\alpha$$
$$y_{bII} = y_{aII} + \Delta x_1 \sin\alpha + \Delta y_1 \cos\alpha \quad (2)$$

where:

$$\Delta x_1 = x_{bI} - x_{aI}$$
$$\Delta y_1 = y_{bI} - y_{aI}$$

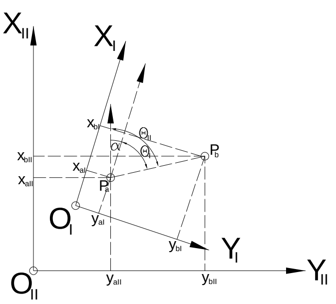

Fig. 3. Transformation scheme

For the transformation of the points there will be introduced a factor of proportion K represented in the report of the distances: $K = \frac{D_{II}}{D_I}$

So, the general formula of transformation becomes:

$$x_{bII} = x_{aII} + \Delta x_1 K \cos\alpha \pm \Delta y_1 K \sin\alpha$$
$$y_{bII} = y_{aII} \mp \Delta x_1 K \sin\alpha + \Delta y_1 K \cos\alpha \quad (3)$$

3. TRIDIMENTIONAL TRANSFORMATION OF THE COORDINATES BETWEEN 2 REFFERENCE SYSTEMS

The problem of the transformation of the coordinates may include transformations between different systems on the same ellipsoid and between different coordinates on different ellipsoids.

It is presumed that for common points we have ellipsoid coordinates **X^{GPS} = $(X, Y, Z)^{GPS}$** and planning coordinates **$(x, y)^{LOC}$** , respectively ellipsoid altitudes **h^{LOC}**. The problem is that the new points determined in GPS measures can be transformed into local system.

3.1. Transforming algorithm is as follows:

a) There will be calculated from the plan coordinates $(x, y)^{LOC}$ the ellipsoid coordinates $(B,L)^{LOC}$ by using the relations known from mathematical cartography.

b) From the ellipsoid coordinates $(B,L,h)^{LOC}$ we can calculated the Cartesian three-dimensional coordinates $(X,Y,Z)^{LOC}$ by using the following relations:

$$X_P = \begin{pmatrix} X_P \\ Y_P \\ Z_P \end{pmatrix} = \begin{pmatrix} (N+h)\cos B \cos L \\ (N+h)\cos B \sin L \\ \left[(1-e^2)N + h\right]\sin B \end{pmatrix} \quad (4)$$

$$N = \frac{a}{\sqrt{1-e^2 \sin^2 B}} \quad (5)$$

$$e^2 = \frac{a^2 - b^2}{a^2} \quad (6)$$

$$h = \frac{\sqrt{X^2 + Y^2}}{\cos B} - N \quad (7)$$

c) It is disposed now for the common points on one hand the coordinates $X^{GPS} = (X, Y, Z)^{GPS}$ and on the other hand the coordinates $X^{LOC} = (X, Y, Z)^{LOC}$:

$$X^{LOC} = X_0 + (1+m) \cdot R \cdot X^{GPS} \quad (8)$$

where:

X_0 - position vector between the origins of the two points
m – scale factor
R - orthogonal and ortho-normal matrix

$$R = \begin{pmatrix} 1 & \varepsilon_Z & -\varepsilon_Y \\ -\varepsilon_Z & 1 & \varepsilon_X \\ \varepsilon_Y & -\varepsilon_X & 1 \end{pmatrix} \quad R^{-1} = R^{T} \tag{9}$$

$\varepsilon_X, \varepsilon_Y, \varepsilon_Z$ - Rotation angles.

For making the transformation we need 7 parameters - $X_0, Y_0, Z_0, \varepsilon_X, \varepsilon_Y, \varepsilon_Z, m$ - which are calculated based on some common points of both systems as follows:

$$\begin{pmatrix} X_1 \\ Y_1 \\ Z_1 \\ . \\ . \\ . \\ X_m \\ Y_m \\ Z_m \end{pmatrix} = \begin{pmatrix} X_1^{LOC} - X_1^{GPS} \\ Y_1^{LOC} - Y_1^{GPS} \\ Z_1^{LOC} - Z_1^{GPS} \\ . \\ . \\ . \\ X_m^{LOC} - X_m^{GPS} \\ Y_m^{LOC} - Y_m^{GPS} \\ Z_m^{LOC} - Z_m^{GPS} \end{pmatrix} = \begin{pmatrix} 1 & 0 & 0 & . & X_1 & . & 0 & -Z_1 & Y_1 \\ 0 & 1 & 0 & . & Y_1 & . & Z_1 & 0 & -X_1 \\ 0 & 0 & 1 & . & Z_1 & . & -Y_1 & X_1 & 0 \\ . & . & . & . & . & . & . & . & . \\ . & . & . & . & . & . & . & . & . \\ . & . & . & . & . & . & . & . & . \\ 1 & 0 & 0 & . & X_m & . & 0 & -Z_m & Y_m \\ 0 & 1 & 0 & . & Y_m & . & Z_m & 0 & -X_m \\ 0 & 0 & 0 & . & Z_m & . & -Y_m & X_m & 0 \end{pmatrix} \begin{pmatrix} X \\ Y \\ Z \\ m \\ \varepsilon \\ \varepsilon \\ \varepsilon \end{pmatrix} \tag{10}$$

with m ≥ 3 common points.

$$L + v = A\,x \tag{11}$$

where:

l – free term as difference of the two systems coordinates;

v – corrections vector;

A – configuration matrix;

x – vector of parameters with the 7 parameters of transformation.

d) The transformation of the coordinates of new points determined by GPS measurements will be made based on the 7 parameters - $X_0, Y_0, Z_0 m, \varepsilon_x, \varepsilon_y, \varepsilon_z$ by using the trans-calculation relation.

e) For the new points from the coordinates $(X, Y, Y)_{LOC}$, there are calculated the ellipsoid coordinates $(B, L, h)_{LOC}$, as follows:

$$\tan B = \frac{Z}{\sqrt{X^2 + Y^2}} \left(1 - e^2 \frac{N}{N+h}\right)^{-1} \tag{12}$$

$$\tan L = \frac{Y}{X} \tag{13}$$

$$h = \frac{\sqrt{X^2 + Y^2}}{\cos B} - N \tag{14}$$

f) Through transformation relations from ellipsoid coordinates it will be done the crossing from the ellipsoid into the projection plan being obtained the plan coordinates $(x, y)_{LOC}$, and from the ellipsoid by using a model for geoids there will be determined the orthometric altitudes.

4. MATHEMATICAL MODEL OF COORDINATES TRANSFORMATION

The operation of digitizing / scanning – vectoring of the existent maps has the following two different stages:

- Taking (reading) the points coordinates from the map into the coordinates system of digitizing equipment (coordinates digitizer).
- Transforming the coordinates digitizer into the coordinates system adopted for the data base (field coordinates – into Stereo 1970 system).

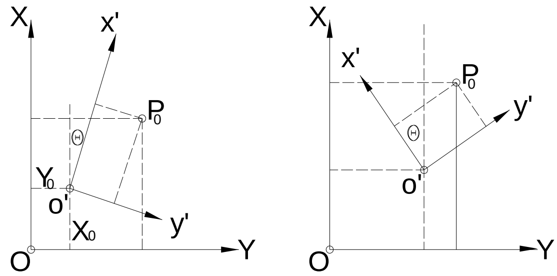

Fig. 4. Orthogonal transformation

Crossing from coordinate's digitizer to field coordinates it is expressed as follows:

$$X_i = t_{11} \cdot x_i + t_{21} \cdot y_i + t_{11} \cdot t_{31}$$
$$Y_i = t_{12} \cdot x_i + t_{22} \cdot y_i + t_{11} \cdot t_{32} \qquad (15)$$

where X_i, Y_i represents the coordinates from the cartographic system (field coordinates) and x_i y_i represents the coordinates digitizer of point i.

If into formula (15) we have:

$$t_{11} = m_x \cdot \cos\theta \quad t_{21} = m_y \cdot \sin\theta$$
$$t_{21} = -m_x \cdot \sin\theta \quad t_{22} = m_y \cdot \cos\theta \qquad (16)$$

it is obtained the semi fine transformation, m_x and m_y being the scale factors on directions X and Y. If $m_x = m_y = m$ results the orthogonal transformation.

The relations (3.18) can be written as a matrix:

$$L = A \cdot T \tag{17}$$

$$L = (X, Y)$$

$$A = (x, y, e)$$

$$T = \begin{pmatrix} t_{11} & t_{12} \\ t_{21} & t_{22} \\ t_{31} & t_{32} \end{pmatrix}$$

where: (18)

The 6 elements of T matrix can be determined if we have 3 points with known coordinates in both systems (digitizer and field). The GIS and CAD systems include interactive procedures of digitizing where it is found the function for determining the matrix of transformation and trans-calculation function based both on relations (15).

The parameters of T matrix can be obtained with the relation:

$$T = (A^T A)^{-1} A^T L \tag{19}$$

and for the precision evaluation we use:

$$s = \frac{(v_x^T v_x + v_y^T v_y)}{(2n-6)^{-1/2}}$$

$$Q_t = (A^T A)^{-1} \tag{20}$$

$$T = (T_1 T_2)$$

$$L = (XY) \tag{21}$$

$$T_1 = (A^T A)^{-1} A^T X$$

where:

$$T_2 = (A^T A)^{-1} A^T Y \tag{22}$$

We note:

$$\mathrm{R} = (\mathrm{A}^{\mathrm{T}}\mathrm{A})^{-1}\mathrm{A}^{\mathrm{T}} \tag{23}$$

$$\mathrm{T}_1 = RX$$

results:

$$\mathrm{T}_2 = RY \tag{24}$$

If the field coordinates have different precisions the relations (24) become:

$$\mathrm{T}_1 = (A^T P_X A)^{-1} A^T P_X X$$

$$\mathrm{T}_2 = (A^T P_Y A)^{-1} A^T P_Y Y \tag{25}$$

$$P_X = Q_X^{-1}$$

where:

$$P_Y = Q_Y^{-1} \tag{26}$$

$$P = \begin{pmatrix} P_X & 0 \\ 0 & P_X \end{pmatrix} \tag{27}$$

In general case the P matrix is full and parameters T_1 *and* T_2 cannot be independently determined.

5. IMPROVING THE FIELD COORDINATES OBTAINED BY DIGITIZING

The transformation parameters can be used for transforming the digitizer coordinates into field coordinates and also for improving the latter ones.

So, the digitizing operation of a map starts by digitizing 4-9 common points (they are usually the corners of the map sheet or intersection points of the map squares).

It is an advantage that for the details (building corners, etc) should be stocked them not the ones obtained by digitizing. The respective points can be considered to have coordinates in two apparently different systems:

- Digitized field system (low precision).
- Calculated field system (high precision).

The transformation parameters can be obtained by using the differential method as follows:

$$t = -(G^T \cdot G)^{-1} \cdot G^T \cdot 1$$
$$v_x = (I \cdot G(G^T \cdot G)^{-1} G^T) \cdot 1 \tag{28}$$

where
$$1 = X_n - X \tag{29}$$

- G – Helmert matrix of differential transformation;
- X_n - field coordinates calculated from the measurements;
- X - field coordinates calculated by digitizing.

In order to evaluate the precision the matrix are as follows:

$$Q_t = (G^T G)^{-1} G^T S_{dx} G(G^T G)^{-1} \tag{30}$$
$$Q_{vv} = (I - G(G^T G)^{-1} G^T) S_{dx} (I - G(G(G^T G)^{-1} G^T) \tag{31}$$

where S_{dx} is the *digitizing matrix* .

After determining the parameters it is useful to make a statistic test for eliminating if it is necessary the extreme values (wrong points). For restarting the calculation it is recommended to be applied the S transformation.

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Scientific Reviewers:
Prof. PhD. Eng. Ilie Onica

THE USE OF INDEPENDENT POLYGON ROUTES, IN ORDER TO ACHIEVE THE MINER BREAKTHROUGHS

FILIP OFELIA-LARISA*
DIMA NICOLAE**

Abstract: *The underground miner workings on various points, define the so-called breakdown miner workings. Their implementation requires appropriate technical terms of their topographical works properly analyzed and the effect of maximum efficiency and accuracy. It is from this point of view, important to study the independent polygonal paths.*

Key words: *the miner breakthroughs, the distances errors*

1. INTRODOCTION

In the execution of the underground mining workings (horizontal, vertical, inclined), there can be encountered situations when it is possible to make an independent polygonal path of support necessary for coordination work with character breakdown (Fig. 1).

This path forms an independent system (called breakdown polygon), the measurement errors of the angles and the lines also form an independent system which has an influence upon the breakdown. This particularity allows a study upon the way of showing the errors, their influence upon the breakdowns, and the distribution of the errors in order to obtain an optimal breakdown point. In what comes next, the measured measures of the errors will be studied (angles, lines) in the breakdown polygons.

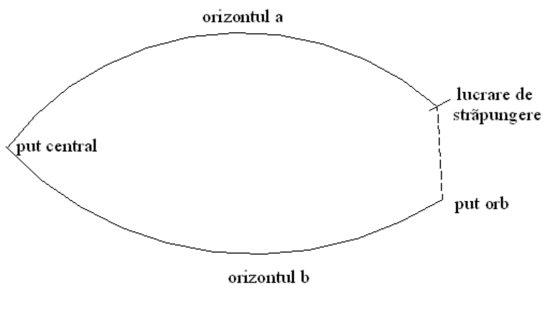

Fig. 1. Called breakdown polygon

* *Assist. Prof. Ph.D. Eng. - University of Petrosani, larisafilip@yahoo.com*
** *Prof. Ph.D. Eng. - University of Petrosani*

2. THE INFLUENCE OF THE MEASURED ANGLES

Consider a simple polygon breakdown of four sides (Fig. 2.); A shot-β1 was measured by real error ε1. Compared with A1 B the side will move to B'and the angles ψ'1 φ'1 have errors and Δψ1 Δφ1 and so there are these relations:

$$\varphi_1 = \varphi_1^{'} + \Delta\varphi_1^{'}$$

$$\psi_1 = \psi_1^{'} + \Delta\psi_1^{'} \qquad (1)$$

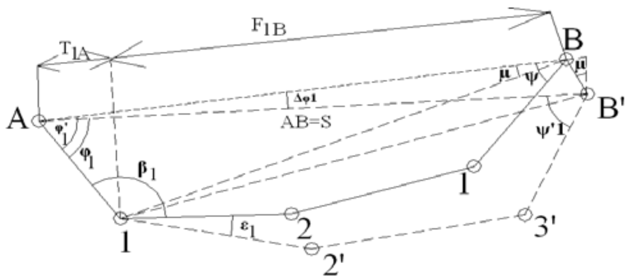

Fig. 2. Simple polygon

Consider a simple polygon breakdown of four sides.

From the figure it results:

$$\Delta\varphi_1 = \varepsilon_1 * \frac{1B * \cos\mu}{AB} = \varepsilon_1 * \frac{F_1^B}{S} \qquad (2)$$

It was noted:

F_1^B - the distance from point B to the projection of the point 1 on S;

S - the connection line between A and B.

The real error ε_2 of the β_2 angle produces upon the φ angle the real error:

$$\Delta\varphi_2 = \varepsilon_2 * \frac{F_2^B}{S} \qquad (3)$$

By continuing and making the sum (2) and (3) there is obtained the total error:

$$\Delta\varphi_t = \frac{[\varepsilon F^B]}{S} \qquad (4)$$

If the fix line is 3B the total error of the ψ angle, because of angle errors is:

$$\Delta\psi_t = \frac{[\varepsilon F^A]}{S} \qquad (5)$$

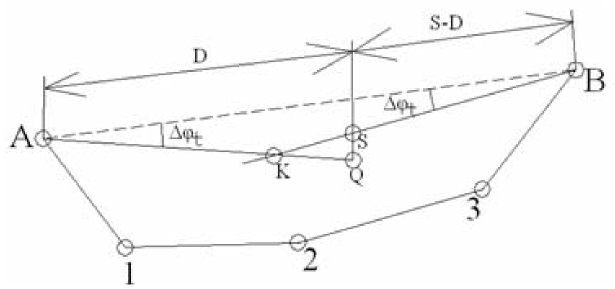

Fig. 3. Polygon A, 1, 2, 3, B

The A,1,2,3,B polygon used in order to realize the AB breakdown, puts the direction angles ψ'_t şi φ'_t mistaken from the real ones with the $\Delta\varphi_t$ and $\Delta\psi_t$ measures given by the relations (4) and (5).

This means that, in case of two directions of going in front, the breakdown working is made upon the directions AK and BK (fig. 3).
It results that if the breakdown point is K point, then the breakdown of the miner working is made with no mistake.

We consider the case in which the going forward speeds are in such a way that the point where the breakdown is made is I, different from K point. In this situation with the notes from the figure, the transversal error Q_t is given by the relation:

$$Q_t = D\Delta\varphi_t - (S-D)\Delta\psi_t \tag{6}$$

Or the relations (4) and (5):

$$Q_t = D\frac{\left[\varepsilon F^B\right]}{S} - (S-D)\frac{\left[\varepsilon F^A\right]}{S} = D\frac{\left[\varepsilon\left(S-F^A\right)\right]}{S} - (S-D)\frac{\left[\varepsilon F^A\right]}{S}$$

It results:

$$Q_t = \left[\varepsilon\left(D-F^A\right)\right] \tag{7}$$

We note the medium square error of angles measurements with $\pm m_\beta$, and (6) relation becomes:

$$Q_t = \pm m_\beta\sqrt{\left[(D-F)^2\right]} \tag{8}$$

It is noticed that the medium transversal error is according to the measure of the variable "D" in a way that am optimal point could be established for which it will exist "D_0" in which the transversal error is minimal. We write:

$$\frac{dQ_t}{dt} = \pm \frac{m_\beta^2}{2} \frac{[(D-F)]}{\sqrt{[D-F]^2}} = 0$$

From where:

$$D_0 = \frac{[F]}{n} \tag{9}$$

(8) relation could be generalized in case that the angles have different precision:

$$D_0 = \frac{\left[\frac{F}{q}\right]}{\left[\frac{1}{q}\right]} \tag{10}$$

It is important to notice that the highest part is considered to be from the point in which the total error is smaller.

The total errors cannot be calculated, so, a medium error is established by using the relations (4) and (5):

$$m_{\varphi_t} = \pm \frac{m_\beta}{S} \sqrt{[\ F^B F^B\ ]}$$

$$m_{\psi_t} = \pm \frac{m_\beta}{S} \sqrt{[F^A F^A]}$$

Or:

$$m_{\varphi_t} = \pm \frac{m_\beta}{S} \sqrt{[(S - F^A)^2]}$$

$$m_{\psi_t} = \pm \frac{m_\beta}{S} \sqrt{[F^A F^A]}$$

So, if:

$$[(S - F^A)^2] < [F^A F^A] \tag{11}$$

Then:

$$m_{\varphi_t} < m_{\psi_t}$$

The most important part is made from A point.

The (11) condition is written simpler as:

$$\frac{[F^A]}{n} > \frac{S}{2}$$

From this it results that the optimal point of the transversal error is found at the middle of AB distance only in case of symmetrical polygons in which the angles were measured with the same precision:

The value of the transversal error in the optimal point is made with the help of (8) relation in which D is calculated with (9):

$$Q_{t,0} = \pm m_\beta \sqrt{nD_0^2 - 2D_0[F] + [FF]}$$

or:

$$Q_{t,0} = \pm m_\beta \sqrt{[FF] - \frac{[F]^2}{n}} \qquad (12)$$

It is obvious the fact that if we go further from the optimal point, the transversal error grows.

If it is noted with "d" the distance from the optimal point, then the transversal error can be calculated with (8) relation**.**

$$Q_{t,0} = \pm m_\beta \sqrt{\left[(D_0 + d - F)^2\right]}$$

Because: $[D_0 - F] = 0$ it results:

$$Q_{t,d} = \pm m_\beta \sqrt{\left[(D_0 - F)^2\right] + nd^2} \qquad (13)$$

If "d" is considered as an independent variable and $Q_{t,d}$ the function, the (13) relation can be written like this:

$$\frac{Q_{t,d}^2}{m_\beta^2\left[(D_0 - F)^2\right]} - \frac{nd^2}{\left[(D_0 - F)^2\right]} = 1 \qquad (14)$$

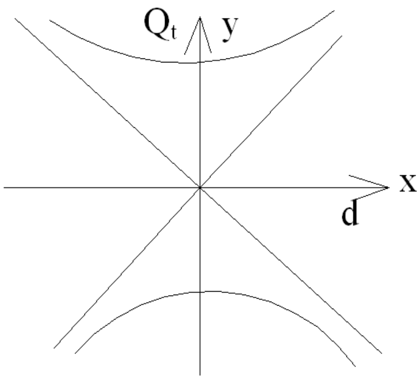

Fig. 4. The variation graphic of the function

The equation of a centered hyperbola was obtained, and its peaks are found on the axe of the dependent variable $Q_{t,d}$, (fig. 4).

From the variation graphic of the function there are the next conclusions:

-the transversal medium error grows from the optimal point after a hyperbolic function;

-the transversal errors grow:

$$\alpha = arctg\left(\frac{m_\beta \sqrt{n}\sqrt{\left[(D_0 - F)^2\right]}}{\sqrt{\left[(D_0 - F)^2\right]}}\right) = \pm m_\beta \sqrt{n} \qquad (15)$$

- the hyperbola becomes more sharpen if the polygon has more points.

3. THE INFLUENCE OF THE DISTANCES ERRORS

We consider for analyze the path from next image (fig.5) where we admit that the line S_1 has the real error ε_{S1}. This produces a transversal error $\varepsilon_{S1}\sin\varphi_1$.
The total error upon the error φ will be:

$$\Delta\varphi_S = \frac{\varepsilon_S \sin\varphi}{s} \qquad (16)$$

The lines error produce upon the angles:

$$\Delta\varphi_S = -\frac{\varepsilon_S \sin\varphi}{S} = -\Delta\varphi_S \qquad (17)$$

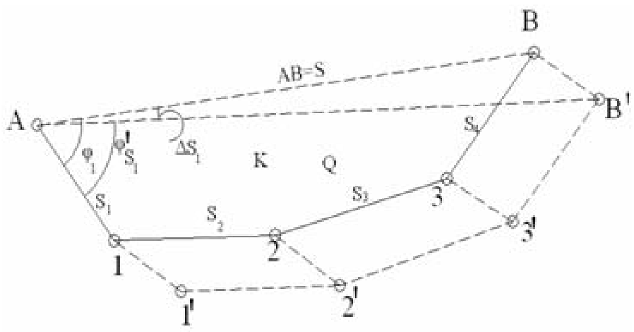

Fig. 5. The influence of the distances errors

The two total errors, being equal, means that the two directions of going forward will be parallel and, the transversal error is the same, independent from the place where is being determined**.**

Going at the medium errors:

$$Q_S = \pm\sqrt{m_S^2 \sin^2\varphi} \quad (18)$$

From (18) relation there are the conclusions:

-the influence of the lines errors is zero;
-at normal lines the whole transversal error participates;
-the optimal point remains valid.

4. THE CUMULATED ERRORS IN BREAKDOWN POLYGONS

Because of the independence of the Q_t and Q_S errors, according to the error law, their cumulating is allowed:

$$Q = \pm\sqrt{m_\beta^2\left[(D-F)^2\right] + \left[m_S^2 \sin^2\varphi\right]} \qquad (19)$$

$$Q = \pm\sqrt{m_\beta^2\left[\frac{1}{q}(D-F)^2\right] + \left[m_S^2 \sin^2\varphi\right]}$$

Q-cumulated transversal medium error.

The growth of the transversal medium error with the distance “d” from the optimal point, is determined with the relations:

$$Q_d = \pm\sqrt{m_\beta^2\left\{\left[(D_0 - F)^2\right] + nd^2\right\} + \left[m_S^2 \sin^2\varphi\right]} \tag{20}$$

$$Q_d = \pm\sqrt{m_\beta^2\left\{\left[\frac{1}{q}(D_0 - F)^2\right] + \left[\frac{1}{q}\right]d^2\right\} + \left[m_S^2 \sin^2\varphi\right]}$$

The analyze of the measured errors in independent polygons leads to the conclusion that from the topographical methods used for the miner breakdowns, the topographical bases are better to use, as they are more secure.

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Scientific Reviewers:
Prof. PhD. Eng. Octavian Herbei

THE SUSTAINABLE DEVELOPMENT OF THE AREAS AFFECTED BY THE UNDERGROUND MINING EXPLOIATIONS

HERBEI MIHAI VALENTIN*
HERBEI OCTAVIAN**
ULAR ROXANA CLAUDIA***

Abstract: *In many life fields it appears more stringent the necessity that in a certain space, a certain region the analysis of some information should be linked to geographic environment. This group of duties needs to be performing spatially the information, which, due to the huge quantity of data to be managed is very great today without using the possibilities, offered by the informatics. The result of this created the basis and contributed to the development of Geographical Information Systems – G.I.S. The area affected by the underground mining are especially the mono-industrial areas that need a permanent monitoring and on long term in order to protect the environment for a sustainable development of them from all points of view.*

Key words: *GIS, Petroşani area, sustainable development*

1. INTRODUCTION

The purpose of this research is to make a GIS in Petroşani area, a system that can visualize, interrogate, manipulate, research different areas from the studied area in order to find some solutions for rehabilitating efficiently or for developing this area by taking account to the field reality.

In order to accomplish a decisional information system for a sustainable development of the areas affected by the underground mining in Jiu Valley mining basin in general and Petroşani area in particular there were followed the next stages:

2. MAKING THE GPS SUPPORTING NETWORK OF JIU VALLEY MINING BASIN

The network is made from 23 points (fig. 1) from which 9 old points and 14 new points and this network was verified and compensated. This network had been made for whole Jiu Valley mining basin, which includes also the studied area.

* *Ph.D. Eng.*
** *Prof. PhD. Eng. - University of Petrosani*
*** *Assist. Prof. PhD. Eng. – University of Petrosani*

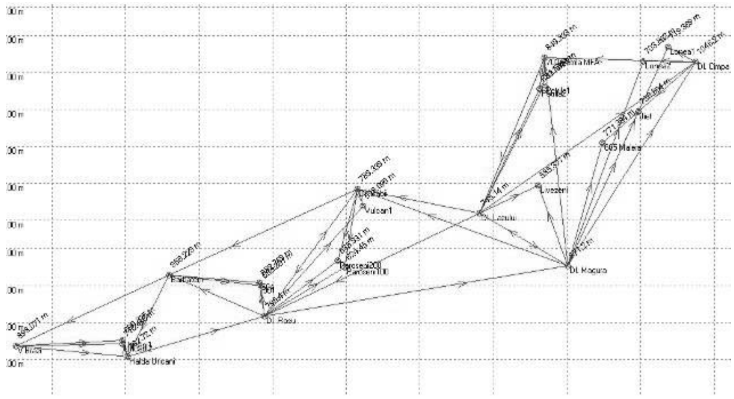

Fig.1. GPS network for Jiu Valley mining basin

3. MONITORING THE SUBSIDENCE PHENOMENON IN PETROŞANI AREA

The decisional process regarding the affected areas will permanently take into account to the parameters of the subsidence phenomenon, parameters which must be determined by using surveying measurements and then be monitories on long and medium term. The parameters which influence the phenomenon of displacement and deformation of the terrestrial surface have been calculated depending on many measurements made at different periods of time, and these measurements will be continued in the future. To study the phenomenon of displacement is made by using the data obtained over the displacement of a group of points from the field.

Their schedule is as follows:

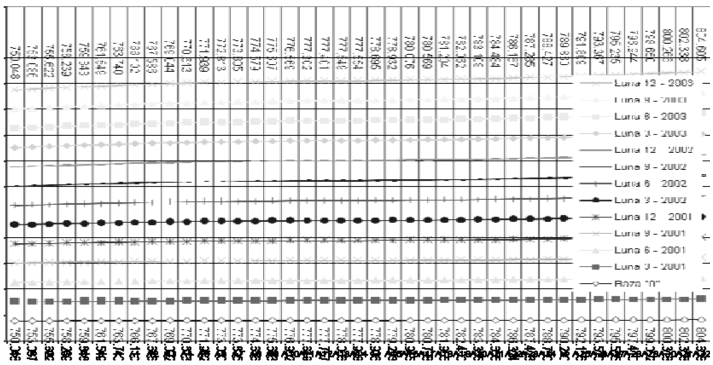

Fig.2. Schedule of heights of following points

After determining the final results it results that the subsidence process in the studied area is active and it must be motorizes in future.

3.1. Adjusting the experimental data by using the regression and correlation analyze

For a certain point P(x, y, z) from the surface of a sinking area there are known the stages of this points as follows:

1.*INCIPIENT STAGE* – where from the repose the point will entering movement till it has the speed v_1;

2.*ACTIVE STAGE* – where the sinking area may cross through all stages (sub critic -supercritical) and where we can anticipate a variation of the subsidence speed around an average value;

3.*FINAL STAGE* – where the sinking speed decreases continuously till stabilizing the terrain.

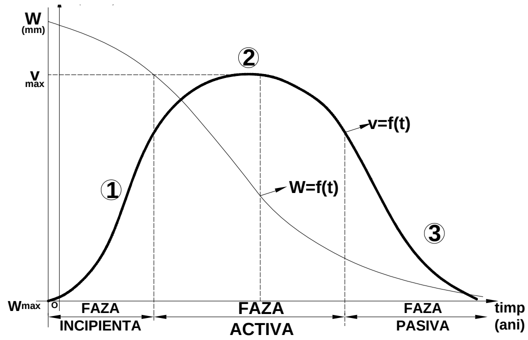

Fig. 3. Evolution of W parameter and of sinking speed V

In correspondence with these stages of the point from the ground it will exist a topic behavior of the subsidence parameter W. As a value, from 0 – moment when the effect is not come yet, toll W_{max} we can observe a continuously behavior even if the fluctuations of speed are discontinuously from a year to another.

In order to obtain a good prognosis we must chose regression functions of the three areas of development. These functions must have different behaviors during the extrapolation period.

So, we looked for functions **W(t)=f(t)** that must have different behavior at extrapolating the values.

- For the (1) are the tendency of decreasing the speeds to zero:

$$W(t) = a_1 \exp(-b_1 t) \quad \text{exponent}$$

where: a_1, b_1- coefficients determined by regression; t – time in months from the first measurements; W(t) – depended variable (sinking), [m].

- For the (2) are the constant tendency of the speeds around an average value:

$$W(t) = a_2 \exp(-b_2 t^2) \quad \text{squared exponent}$$

where: a_2, b_2 – numeric coefficients; t – time (months); W(t) – subsidence, [m].

- For the (3) area the increasing tendency of the speeds to a maximum value:

$$W(t) = a_3 t^3 + b_3 t^2 + c_3 t + d_3) \quad \text{polynomial type}$$

where: a_3, b_3, c_3, d_3 – numeric coefficients.

4. ACCOMPLISH A GIS IN PETROŞANI AREA IN ORDER TO HAVE A SUSTANABLE DEVELOPMENT

The sustainable development means to have healthy and sure models of sustainability. This includes the management, the infrastructure and its services including the facilities of drinking water comparable to the ones from the European Union. In a protected environment they will have an efficient transport and links of communication with the rest of the country and will have available resources of clean energy. The members of this community will have acceptable opportunities for education, will take part to the economic process and they will be able to implicate themselves into the local government.

For this application I used the software ArcGIS 9.2 made by ESRI company and these applications allowed me as follows:

- To create the geo-spatial data bases
- To modify, complete and update the GIS recourses
- To make thematic maps
- To make analyses and interrogations specific to this project
- To build some reports based on geo-spatial data.

The most important feature of a GIS consists in its capacity of making **spatial analysis** so to use the spatial data in order to obtain reports regarding the studding area. These spatial data are worked based on some specific algorithms by using own operations.

The spatial analyses are useful for evaluating, making prognosis, interpreting and understanding. The spatial analyses are **modularly ones**. In GIS there are 4 main types of spatial analysis as follows: spatial superposition and continuity analysis, surface analysis, linear analysis and raster analysis.

The spatial analyze helps us to extract the significant data distributed spatially and that will be worked. So, we can obtain data associations, which can be characterized or which lead to previsions or to understanding the phenomena. The operations can be made directly on the layers together with their attributes.

The result of a spatial analysis is to allow to be interrogated the attributes and to generate some new date by using the existent ones.

The main spatial operations are as follows:

- ➢Operations from a single layer
- ➢Operations from multiple layers
- ➢Statistic analyze
- ➢Network analyze
- ➢Analyzing the areas – making the digital model of the field.

Into the presented application it was defined a series of layers as follows: water network, phone network, underground works, exploited area, GPS network, waters, road, constructions, proprieties.

The data brought to the GIS working environment can be manipulated in different purposes. The data manipulation is made as follows:

a) Finding again
b) Modifying the data structure
c) Interrogating and analyzing.

a) **Finding again the data** (Fig. 4) consists in:

- ✓Selecting a data category;
- ✓Selecting the graphic data or attribute data by using graphic windows, circle, polygon;
- ✓Selecting the graphic data

Fig. 4. Marking and foreseeing the present influence and future area for the next 20 years regarding the Livezeni mining area

b) ***Interrogating and analyzing the data*** (Fig. 5)

When it is put into service the system that contains the geographic data, it was be able to be interrogated by using some simple questions: Who is the owner? What is the distance between 2 points? Etc. or it can be using some analytic questions: *Where are some proper areas for building a house? Etc.*

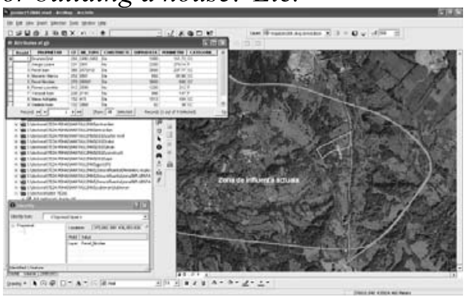

Fig.5. Identifying the proprieties from the influence area

c) **Showing the data** (Fig. 6) and generating the reports has the following processes:

- Creating, memoring, finding again, generating the structure;
- Libraries of conventional signs as points, lines, surfaces having the possibility to generate, edit, insert the new signs;
- Libraries of writing characters having the possibility to generate, edit, insert the new characters, styles;
- Possibility to represent the same frame of some data with different locations, different scales;
- We associate to the attributes some conventional signs, colors, texts;
- Insert the legends.

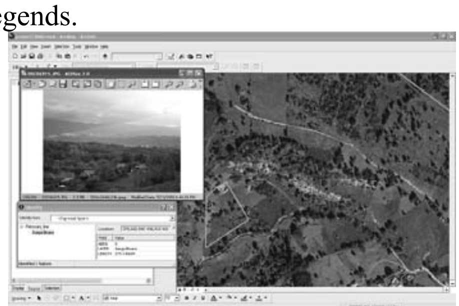

Fig. 6. Visualize some proprieties affected by Livezeni mining exploitation

5. GENERATING AND INTERPRETATION THE DTM FOR THE JIU VALLEY MINING BASIN – HUNEDOARA COUNTY - ROMANIA

The interpolation methods type triangulation one after whom it is obtained a TIN structure **(Triangular Irregular Network)** (Fig. 6**),** are also multiple. Interpolation methods such as triangulations, when it is obtain a TIN (**Triangular Irregular Network**) (Fig. 7) are also multiple. The best one is ***Delaunay Interpolation*** that allows to be obtained some perfect triangles inside a circle so the distance between the points from the picks of the triangle is always minim. For each triangle there are memorized the coordinates and attributes of the three picks, topology and slope and declination direction of the triangle surface.

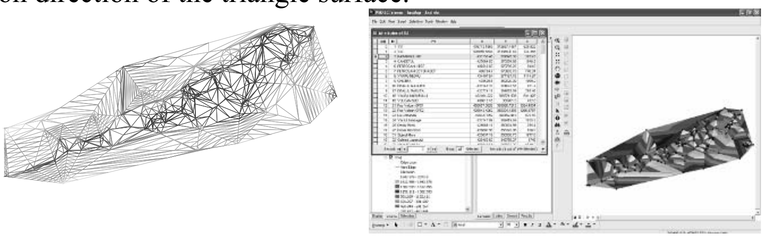

Fig. 7. Elaboration of TIN structure

Three dimensional visualization (Fig. 8) is the preferred way of visualization to comprehend the actual terrain of any place. This can be achieved by techniques using wire frame models or rendering from a 3-D plane to a 2-D plane. For added realism, image based information is added to the rendered primitives. This kind of texture mapping serves to increase the visual appeal and increase the vivid detail. This is popularly known as *draping*, and leads to a greater understanding of patterns in the image and how they relate to the shape of the earth's surface.

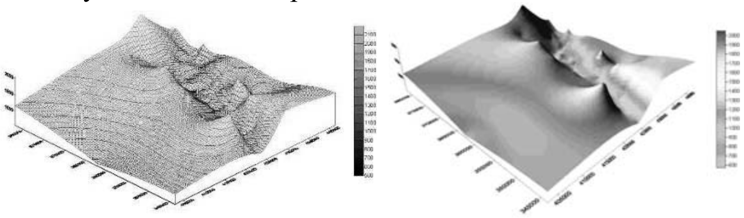

Fig. 8. The 3D model (Wireframe model and Surface model)

5.1. DTM Interpretation

The analysis of DTM's to extract terrain parameters is termed **DTM interpretation**. The extraction can be performed by either visual analysis or quantitative analysis (interpretation). The analysis can be grouped into *general geo-morphometry* or *specific geo-morphometry*. General geo-morphometry deals with quantification of general surface characteristics such as **slope, gradient or aspect**. The concept of measuring **slope** from a topographic map is a familiar one for most professionals in the landscape planning/surveying professions. Slope is a measurement of how steep the ground surface is. The steeper the surface the greater the slope. Slope (Fig. 9) is measured by calculating the tangent of the surface. The tangent is calculated by dividing the vertical change in elevation by the horizontal distance. If we view the surface in cross section we can visualize a right angle triangle. Slope is normally expressed in planning as a percent slope which is the tangent (slope) multiplied by 100:

Percent Slope = Height / Base * 100

Fig. 9. The measuring of slope using ArcGIS software

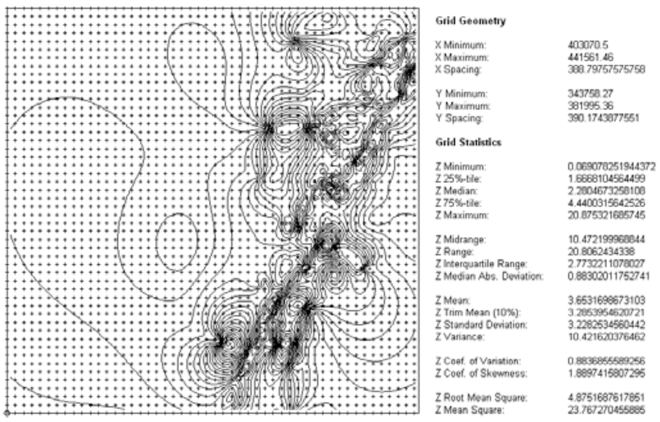

Fig.10. The map of Slope and Grid Information using Surfer Software

S*lope* is an attribute to define surface and comprises **gradient** and **aspect**. When written in the form of a mathematical equation g*radient* (usually calculated in degrees) refers to the first vertical derivative of altitude and represents the rate of change in its magnitude over distance. Similarly a*spect* is the first horizontal derivative of the altitude and represents the direction of the slope. The **curvature** (convexity / concavity) of the terrain can be determined by the second order derivatives. Curvature of the surface helps define the movement of masses. The formulae for calculation are mentioned below:

$$Gradient = \sqrt{\left(\frac{\Delta z_x}{\Delta x}\right)^2 + \left(\frac{\Delta z_y}{\Delta y}\right)^2} \quad / \quad Aspect = \tan^{-1}\left(\frac{\frac{\partial f}{\partial x}}{\frac{\partial f}{\partial y}}\right) \quad / \quad Curvature = \sqrt{\left(\frac{\partial^2 f}{\partial x^2}\right)^2 + \left(\frac{\partial^2 f}{\partial y^2}\right)^2}$$

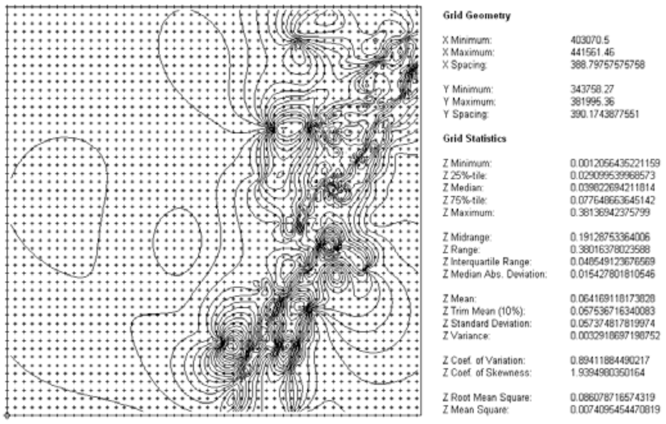

Fig.11. The map of Gradient and Grid Information using Surfer Software

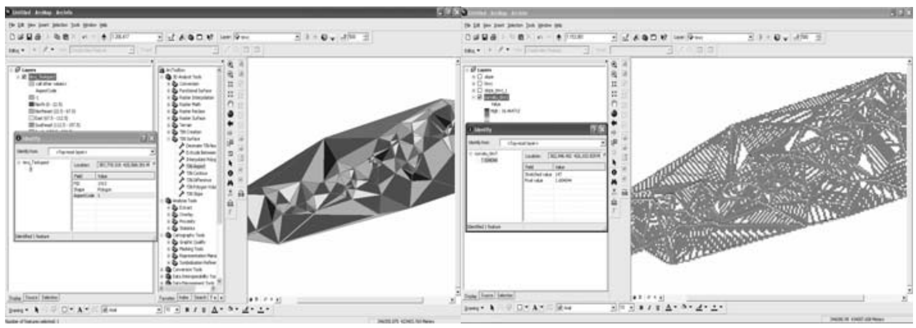

Fig. 12. The calculation of the aspect and curvature using ArcGIS software

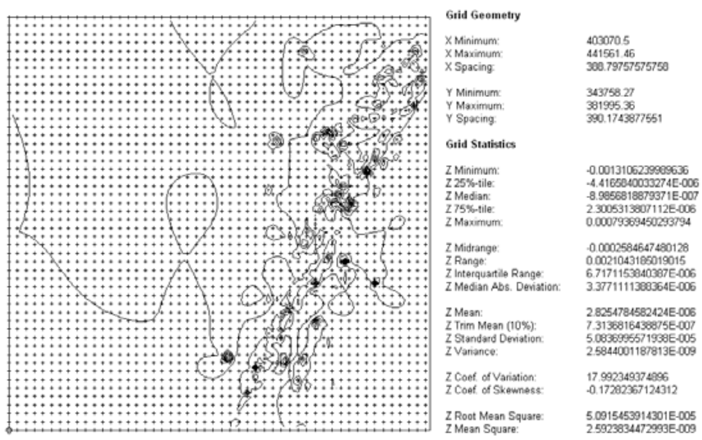

Fig.13. The map of Curvature and Grid Information using Surfer Software

6. CONCLUSIONS

The digital map correlated to different databases offers the possibility to analyze the studied area (Petroşani area). The analysis is the base for the decisional process for making and implementing some feasible projects for developing the area from all points of view. This GIS of Petroşani can be applied in different areas affected by underground mining in Jiu Valley or any other mining areas in order to implement some project regarding their sustainable development. To make this application included automatically also the generation of the digital model of the terrain for the studied area. So, in this way, it can be understood and interpretative the shape and features of the studied area depending on the users needs.

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Scientific Reviewers:
Prof. PhD. Eng. Ilie Onica

SUPPORT STRUCTURE STABILITY ANALYSIS OF THE VALVE HOUSE OF THE BOTTOM DISCHARGE, RĂSTOLIȚA DAM

**ILIE ONICA*
CONSTANTIN SEMEN*
EUGEN COZMA*
ADRIAN RUSU****

Abstract: *This paper relates to the stability of the supports of the valve house of the bottom discharge -Răstolița dam, that is a hydro technical working with a diameter of 10m and the total height of 22m. The temporary supports are composed of steel bolts and shotcrete and permanent support by reinforced concrete. The study is achieved by numerical modelling with the aid of the finite element method, in the axial-symmetry hypothesis and the linear elastic behaviour of the supports and the rocks. After the analysis of stresses, displacements and the safety coefficients, it was concluded that the supports' and surrounding rocks' stability is very good and in conformity with the working purpose.*

Key words: *valve hose bottom discharge, reinforced concrete, bolts, shotcrete, finite element method, stress, strain, safety coefficient, stability analysis*

1. GENERALITIES

Hydroelectric arrangement of Răstolița has as an objective the hydraulic potential exploitation of some effluents on the right side of Mureş river for water supplying Tg. Mureş city (to a flow rate of 6.6 m^3/s) and energetic use by producing in the Răstolița hydro (installed flow rate of Q_i=17m^3/s and generating station capacity of Pi=35,3MW) of about 117.5GWh/an.

The valve house of the bottom discharge is part of the Rastolița hydroelectric arrangement and is an underground cave (a cylindrical shape with a diameter of 10m, covered with a spherical vault with a total high of 22m). The excavation support is made during driving with shotcrete and a bolting system to which is added a permanent

* *Prof. PhD. Eng. - University of Petroşani*
** *Eng. - Ardeal Hydroconstruction Company Cluj-Napoca*

support of reinforced concrete with an average thickness of 0.7m. The complex phases of the underground hydro technical excavation consist, mainly, in the following: access gallery driving; central rise driving; excavation and support of the spherical vault, excavation of the side part of the valve house (in the horizontal slices of 2m) and walls support achievement [2,3,4,11,13].

2. GEOMECHANICAL CHARACTERIZATION

Geological and hydrogeological data of the researched zone show the fact that the rocks in the Răstolița dam emplacement zone are made of pyroclastic rocks and volcanic ashes in which, the decayed and fissured hard andesites are intercalated.

The samples from boreholes F2 and F4 (the Răstolița dam reservation - left and right side) are composed, mainly of pyroclastic rocks with a microconglomeratic aspect (about 87% of samples) cemented volcanic ashes (7%) and andesites (cca.6%). In the boreholes F3 and F7 (the dam reservation - stream bed) the percentages of the previous types of rocks are of 70% and 12% and respectively 18%. In the borehole F2, there are identified the pyroclastic rocks zones being more degraded with a red ferruginous color, fissured and decayed [5].

Laboratory determinations consist of the following: apparent specific density, porosity, water absorption at the normal pressure and temperature, failure compressive strength, shear strength, sonic wave propagation velocity, static and dynamic elasticity modulus, dynamic Poisson ratio [1,6,7] (Table no. 1).

Table no.1. *Range of the geomechanical characteristics of the rocks from the Răstolița dam*

No. crt.	Characteristics		U/M	Rock (range)		
				Pyroclastic rocks	***Volcanic ashes***	***Andesite***
1.	Specific density		kg/m^3	2720-2760	2700-2730	2800-2810
2.	Apparent specific density		kg/m^3	1580-2660	1530-2010	2200-2730
3.	Porosity		%	11.0-34.0	26.0-40.0	3.0-16.2
4.	Water absorption at the normal pressure and temperature		%	2.2-21.5	3.65	0.8-5.2
5.	Failure compressive strength (in the natural state)		MPa	2.7-30.0	8.9-15.5	28-117
6.	Shear characteristics	φ	grade	13-43	10-40	17-34
		C	MPa	0.42-5.00	1.22-3.00	17.00-26.50
7.	Sonic wave velocity (longitudinal)		m/s	1300-4100	1600-2600	2700-4700
8.	Static elasticity modulus (Young modulus)		MPa	3300-19000	550-3960	44700-64000

It is shown a large range of variation of the physical and mechanical characteristics, especially for the pyroclastic rocks and the volcanic ashes which cannot be correctly separated as petrographycal types. In the same sample of the pyroclastic rock arise the andesites fragments in the various percentages (as number and size), compared to the adhesion cement involved by the volcanic ash.

The analyzed rock is represented predominantly by the pyroclastic rocks with gray color, sometime red ferruginous-swarthy colour, in the decayed zone (F2, between 2m and 20m); the pyroclastic rocks aspect is as micro conglomerates well cemented in which, the angular andesite fragments are included into a ash mass, which compose the rock cement; in the reduced percentages, appear the ash samples (maximum 12% from the total tested samples) and totally sporadically; samples composed of the hard, decayed or fissured andesites.

Analyzing, as a whole, the obtained values for the pyroclastic rocks and volcanic ones it's observed that about 80% of the obtained values are grouped into a restrained domain (values considered representatives for these types of rocks and recommended for the design problems) such as [5]: apparent specific density ρ_{nat}=1900-2200kg/m^3; porosity n=10-30%; water absorption at the normal pressure and temperature a_1=3-10%; failure compressive strength (in the natural state) σ_{rc}=5,0-15,0MPa; internal friction angle φ=30-43^o; cohesion C=1,5-4,0MPa; sonic longitudinal wave velocity V_L=1500-2500m/s; Young modulus E=3000-15000MPa (it is recommended that the value of in situ Young modulus to be reduced to about ten times less).

3. FINITE ELEMENT ANALYSIS OF THE STRESS-STRAIN STATE AROUND THE VALVE HOUSE OF THE BOTTOM DISCHARGE OF THE RĂSTOLIȚA DAM

3.1. Finite element model presentation

Depending on the model characteristics, transposition from the 3D space into 2D could be provided following the 3 hypothesis: 1) plane strain; 2) plain stress; 3) revolution symmetry or "axial-symmetry" [9].

The CESAR-LCPC finite element code [17] was used in this work to achieve the model into axial-symmetry of the hydrotechnical underground working "the valve house of the bottom discharge of the Răstolița dam".

In order to realize the model in the axial-symmetry hypothesis, the following simplified hypothesis was made: the access gallery is presented in the model with a cylindrical shape, with internal diameter of 10m and height of 4m, situated on the base of the valve house; the temporary support is composed by shotcrete and bolting system and the permanent support is supposed to be achieved compactly, with a total thickness of 0.8m (0.1m shotcrete + 0.7m reinforced concrete).

Because of the cylindrical shape of the valve house, with 10m diameter, covered with a spherical vault, with total height of about 22m, we consider that the modelling of this underground excavation using the axial-symmetrical calculus hypothesis is the most simple and efficient solution to this problem.

The numerical modelling by axial –symmetrical method could be applied for the bodies having symmetry, of point of view of mechanical loading and of geometry, by reporting to a longitudinal axis of symmetry [10].

The axial-symmetrical structures are the three-dimensional revolution bodies. In the hydrotechnical constructions field these are frequently found: intake towers, unloaded cones, water reservoirs, etc. Also, in the case where the charges are symmetrical, the displacement components, inside of the section passing by the symmetry axis define completely the strain and stress state of the structure [12]. Thus, the valve house bottom discharge of the Răstolița dam could be characterized as an axial-symmetric underground construction.

The profile of a revolution axial-symmetric body can be discretized by the toroidal axial-symmetric finite elements (circular ring of constant section). The transversal section of this element could be in a quadrangular or triangular shape (like our studied models). In every node exist two displacements *u* and *w*, so that the simple quadrangular finite element has eight freedom degree; the material properties (in our case, the rocks, concrete and steel) and the nodal displacements it can be expressed in function of two independent coordinates, radius *r* and height *w*. Usually, the body and the finite element is reported at a cylindrical coordinate system "r, z, θ", but nor the material properties and neither other parameters depend on the angle θ. Also, a particular aspect is that the radial displacements *u* produces the circumferential strains $\varepsilon_\theta = u / r$, respectively the circumferential stresses σ_θ. By consequence, the axial-symmetric element is bidimensional, but the strain-stress state must include a fourth component of strain and stress, so that [10, 12]:

$$\{\varepsilon\} = \begin{Bmatrix} \varepsilon_r \\ \varepsilon_z \\ \varepsilon_\theta \\ \gamma_{rz} \end{Bmatrix} = \begin{Bmatrix} \dfrac{\partial u}{\partial r} \\ \dfrac{\partial v}{\partial z} \\ \dfrac{u}{r} \\ \dfrac{\partial u}{\partial z} + \dfrac{\partial v}{\partial r} \end{Bmatrix} \tag{1}$$

and respectively:

$$\begin{Bmatrix} \sigma_r \\ \sigma_z \\ \sigma_\theta \\ \tau_{rz} \end{Bmatrix} = \frac{E}{(1+\nu)\cdot(1-2\cdot\nu)} \cdot \begin{bmatrix} 1-\nu & \nu & \nu & 0 \\ \nu & 1-\nu & \nu & 0 \\ \nu & \nu & 1-\nu & 0 \\ 0 & 0 & 0 & \dfrac{1-2\cdot\nu}{2} \end{bmatrix} \cdot \begin{Bmatrix} \varepsilon_r \\ \varepsilon_z \\ \varepsilon_\theta \\ \gamma_{rz} \end{Bmatrix} \tag{2}$$

In the case of this study the previous parameters are transformed from the polar coordinates"r, z, θ" into cartesian coordinates „*x, y, z",* and adequately into the displacements *u* and *v* and the stresses $\sigma_{xx}, \sigma_{yy}, \sigma_{zz}, \sigma_{xy}$.

The 2D modeling achievement, in the axial-symmetric hypothesis and the linear elastic behaviour of the rocks, concrete and steel bolts must go over the followings stages: I) establishment of boundaries, interest zones and meshing of the model; II) determination of zones (regions) and computational hypothesis and the geomechanical characteristics input; III) boundaries conditions establishment; IV) initial conditions and loading conditions establishment; V) achievement of calculus and stoking of results [9].

3.1.1. Establishment of boundaries, interest zones and meshing of the model

Having in view the sizes of the underground excavation, for a better precision of the calculus, the models were performed with sizes X=50m/Y = 138.6m. Also, the sizes of the interest zone around the underground excavation were established so as to involve the model surface where the stress and strain variation is maximum. Model meshing, respectively of every region, was made by triangle finite elements with quadratic interpolation. The total number of nodes is of 7521, the elements of 3730, the surface elements of 3700 and linear elements (for the bolts) of 30.

3.1.2. Determination of zones (regions) and computational hypothesis and the geomechanical characteristics input

In order to simplify the 2D models, there were taken into consideration 2 regions with various characteristics (rocks and reinforced concrete) and the bolts as the steel linear elements as bar-type.

a) Geomechanical characteristics of the rocks

The rocks characteristics (especially pyroclastic rocks, volcanic ashes and andesites) previous presented are taken into calculus in the linear elastic and isotropic behaviour. Thus, there were taken some medium values of the characteristics, considered representatives fore the in situ behaviour: apparent specific density $\rho_{ar} = 2200$ kg/m^3; linear elasticity modulus E_r=15000MPa; Poisson ratio $\nu_r = 0.18$; failure compressive strength $\sigma_{rc} = 15$MPa; failure tensile strength $\sigma_{rt} = 2.0$MPa; shear strength $\sigma_{rf} = 4.5$MPa; cohesion C_r=1.5MPa; internal friction angle $\varphi_r = 30^o$.

b) Characteristics of the reinforced concrete support

In the different speciality literature is found the concrete characteristics by reporting of the concrete mark [14]. In the calculus we adopt the concrete elasticity modulus about E_b=26500MPa, with a safe reserve of 12%. The Poisson ratio, for the concrete mark B20 is adopted of $\nu_b = 0.2$ (being very closely of the reinforced concrete value).

In point of view of the mechanical characteristics, the for the B20 concrete mark, the class III of homogeneity degree, the compressive strength of the concrete is R_{ac}=16MPa and tensile strength is R_t=1.2MPa (from splitting test) and R_t=1.8MPa (from bending test) [14, 15, 16]. From where it is possible to deducing (for the minimum values of the concrete strengths) the cohesion C=2.2MPa and the internal friction angle $\varphi = 55^o$.

c) Bolts characteristics

In point of view of elasticity characteristics of the concrete steel PC52 with the diameter of $\varphi = 25$ mm used for operate the cemented bolts was adopted in calculus: E_a= 210000MPa şi $\nu_a = 0.25$. In the model, in the axial-symmetry hypothesis, the bolts were taken with a cross section about 0.0005m^2/m and the length of 2.5m for the vault and 1.5m for the walls. The ultimate strength of the concrete steel is $\sigma_{c\,\min} = 340$ N/mm^2 [8].

3.1.3. Boundaries conditions establishment

The superior side of the model is considered free and the lateral sides, blocked (for the inferior side the vertical displacements $v = 0$ and the horizontals $u \neq 0$ and for the lateral sides $v \neq 0$ and u=0).

3.1.4. Initial conditions and loading conditions establishment

Initial loading conditions of the model were considered as geostatic$[\sigma_o]$, corresponding to underground excavation depth of H=58.6m: vertical geostatic stresses $\sigma_{oy} = \rho \cdot g \cdot H = 12.63$ m, horizontal geostatic stresses $\sigma_{ox} = \frac{\nu}{1-\nu} \cdot \sigma_{oy} = k_o \cdot \sigma_{oy} = 3.16$ MPa. The induced stress by the excavation presence was$[\sigma_e]$, respectively the stress variation represented by the horizontal stress $\sigma_{ex} = -3.16$ MPa and the vertical stress $\sigma_{ey} = -12.63$ MPa. Thus, the loading of the model was performed in the total stress: $[\sigma_T] = [\sigma_o] - [\sigma_e]$. The loading of the regions corresponding to the reinforced concrete support was made gravitationally, separately, in the form$[\sigma_s]$, as a function depending on theρ_b, g and G_b (where: $\rho_b = 0.025$ MN/m^3 - concrete density; g=9,8m/s^2 - gravitational acceleration; G_b - concrete support thickness, in m).

3.1.5. Achievement of calculus and stocking of results

The calculus was made taking 60 iterations per increment and a tolerance of 1% of the results, using for the resolution the "initial stress method". The calculus results were stocked in the graphical form on the model surface (isovalue, vector and tensor representation) and in the predefined sections (following the external and internal contour of the concrete support). The results obtained correspond to the following parameters: the vertical displacement v and horizontal u (in mm); the horizontal stress σ_{xx} andσ_{zz} and vertical σ_{yy}; the shear stress $\sigma_{xy} = \sigma_{yx}$; the normal stress$\sigma_{nn}$ and the tangential σ_{tt}; the maximum principal stress σ_1 and minimumσ_2; the maximum shear stress $\tau_{f\max}$; the compressive stress σ_c; the tensile stress σ_t (in kN/m^2).

Between the stresses following the x and y axis, σ_{xx}, σ_{yy} and σ_{yx}, and the maximum principal stress σ_1 and minimumσ_2 exist the following dependency relations [6]:

$$\sigma_1 = \frac{1}{2} \cdot (\sigma_{xx} + \sigma_{yy}) + \sqrt{\frac{1}{4} \cdot (\sigma_{xx} - \sigma_{yy})^2 + \sigma_{yx}^2} \tag{3}$$

$$\sigma_2 = \frac{1}{2} \cdot (\sigma_{xx} + \sigma_{yy}) - \sqrt{\frac{1}{4} \cdot (\sigma_{xx} - \sigma_{yy})^2 + \sigma_{yx}^2} \tag{4}$$

Also, the maximum shear stresses:

$$\tau_{f\max} = \frac{\sigma_1 - \sigma_2}{2} \tag{5}$$

4. ANALYSIS OF THE RESULTS OBTAINED FROM THE NUMERICAL MODELING

In the way to solving the presented results, the stress concentrations rates σ_{yy}/σ_{xx} and σ_1/σ_2 could describe the stress disequilibrium and implicitly the arising possibility of failure and deformation phenomena (fig.1 and fig.2). A bigger rate leads to the fact that the principal stress circle intersects the rocks characteristic curve, thus developing the failure phenomenon and the opening of certain fissures and cracks inside of the rocks mass or of the concrete support structure. Also, from the point of view of stability, the tensile and shear stress study are very significant because the rocks and the concrete have very reduced limits of tensile and shear and, frequently, the failure arises when these strength limits are surpassed. The vertical and horizontal displacements analysis, as magnitude and the vector orientations, can suggest the amplitude and the direction of the development of deformation phenomena [9].

For to analyse the rocks stability on the excavation contour and the stability of the reinforced concrete support, taking into account the stress state, will be calculated some safety coefficients that quantified the stability of the rocks and the support structure.

We note that the CESAR-LCPC code [17] not provide the safety coefficients following certain failure criteria. Therefore, there will be introduced a failure criterion considering the intrinsic curve of the rocks and the concrete. For any point characterised by a certain state of the stresses, it is determined the correspondently Mohr circle and it is reported to the intrinsic curve of the rocks and the concrete. In this way, there will be taken into consideration the Mohr-Coulomb line (defined by the relation: $\tau = C - \sigma \cdot tg\varphi$) and will be established the following conditions:

a) If $\sigma_1 < R_t$, results $CS = R_1 / R$

$R_1 = 1299.04 - 0.5 \cdot S_c$ - for the rock on the excavation contour (when C= 1500kN/m^2 and $\varphi = 30^o$);

$R_1 = 444.46 - 0.156 \cdot S_c$ - for the rock on the excavation contour (when taking into account the structural weakening coefficient, after V.V. Raiski [1], C_{ss}=0.3, results C=450kN/m^2 and $\varphi = 9^o$);

$R_1 = 1261.87 - 0.82 \cdot S_c$ - for the reinforced concrete support B 200 (when C=2200kN/m^2 and $\varphi = 55^o$).

b) If $\sigma_1 \geq R_t$, then CS=0.

In the previous relations: $S_c = \frac{\sigma_1 + \sigma_2}{2}$ represents the abscissa of the Mohr circle; $R = \frac{\sigma_1 - \sigma_2}{2}$ - Mohr circle rayon; R_1 – Mohr circle rayon of the tangent at the Mohr-Coulomb line; CS - safety coefficient; R_t – rocks or concrete tensile strength; C - cohesion; φ - internal friction angle.

Regarding the calculated value of the safety coefficient CS, three stability cases exist:

1) *CS*=1, when the Mohr circle and the intrinsic curve are tangent – resulting a limit stability;

2) *CS*<1, when the Mohr circle and the intrinsic curve are secant – resulting the conditions for arising the failure phenomena;

3) *CS*>1, when the stresses state is far from the failure phenomena – resulting a certain stability degree, depending on the value of the safety coefficient.

Considering the previous relations, in the fig.3 are represented synthetically the values of the safety coefficients for the support and for the surrounding rocks. It is established that the values of these are well above the stability limits (*CS*>1.6-2). Even in the conditions of the reducing with 70% the rocks characteristics, the safety coefficients of the rocks are close to the stability limits. Taking into account the support presence it is possible to consider that the excavation stability is assured.

Also, if is compared the shear stress values ($250 \div 850$kN/m^2-on the external contour of the support; $200 \div 2000$ kN/m^2- on the internal contour of the support), the tensile stresses ($0 \div 300$kN/m^2-on the external contour; $0 \div 650$ kN/m^2- on the internal contour) or the compressive stresses ($-2200 \div 700$kN/m^2 - on the external contour; $4500 \div 0$ kN/m^2 - on the internal contour) of the support structure, it is seen that these values are well below the strength values of the reinforced concrete support. Mentioning that, the biggest values correspond to the supporting shoe zone because it's angular geometry, although it is in a good state of stability.

As regards to the horizontal and vertical displacements of the internal contour of the support it is established that these values are below 1mm, it is well below the reinforced concrete support capacity (fig.4).

If there are analysed the bolting loadings (fig.5.a), it is observed that the majority of bolts are loading at the tensile, with the most important values for the first three rows at the vault level, of maximum 43.1kN, the following two rows being loaded partially at the compressive loadings of maximum - 6kN, and the wall bolts at the tensile loads of max. 10kN. Which determine some total displacements in the bolts of max. 3.2mm, at the vault apex leading at the conclusion that because the big ratio between the bolts length and the excavation sizes these bolts has only a temporary function, with the shotcrete reinforced with the wire mesh.

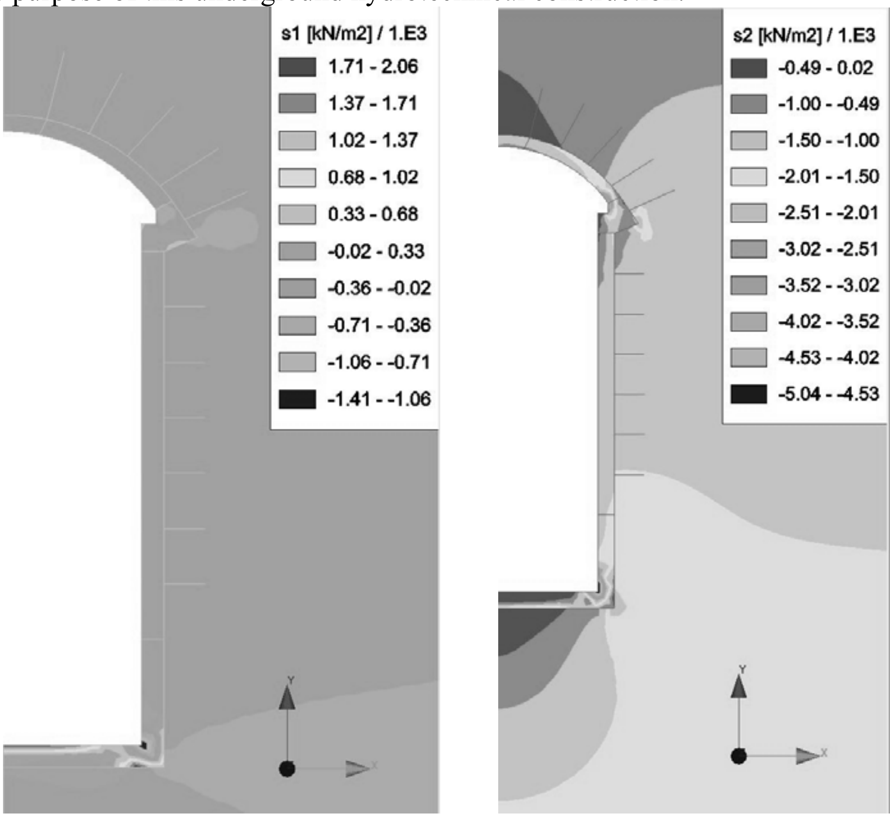
5. CONCLUSIONS

Hydroelectric arrangement of Răstoliţa has as an objective the hydraulic potential exploitation of some effluents on the right side of Mureş river for water supplying Tg. Mureş city. The valve house of the bottom discharge is part of the Rastoliţa hydroelectric arrangement.

The rocks in the Răstoliţa dam emplacement zone, with a good stability, are made of pyroclastic rocks and volcanic ashes in which, the decayed and fissured hard andesites are intercalated.

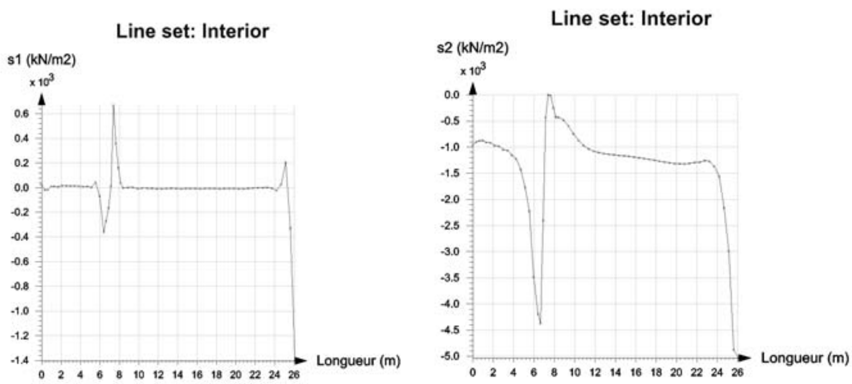
The stability study of the valve house of bottom discharge Răstoliţa dam is achieved by numerical modelling with the aid of the finite element method (CESAR-LCPC code), in the axial-symmetry hypothesis and the linear elastic behaviour of the supports and the rocks.

Finally, analysing the stress - strain state and the safety coefficients, the conclusion is that excavation support of the valve house of bottom discharge Răstolița dam has a good state of the stability, in conformity with the necessary tenacity imposed by the purpose of this underground hydrotechnical construction.



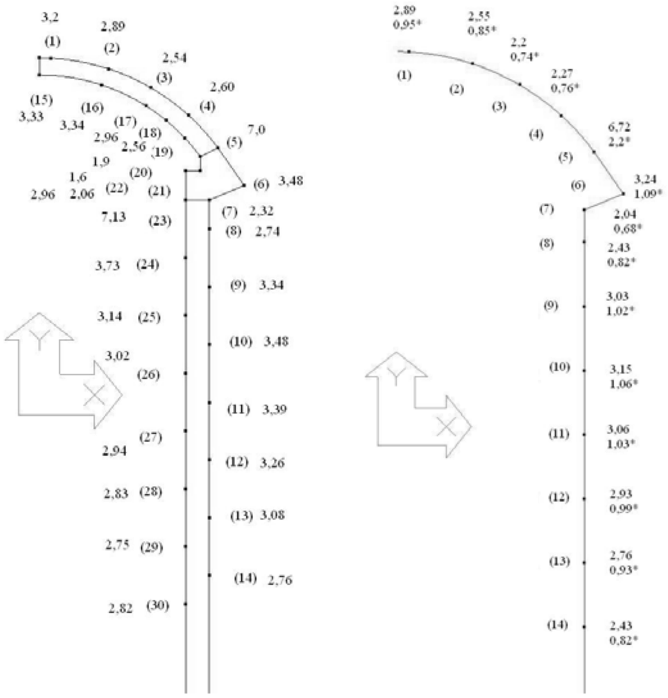
a) b)

Fig.1. Maximum principal stress σ_1 and minimum σ_2, in kN/m^2



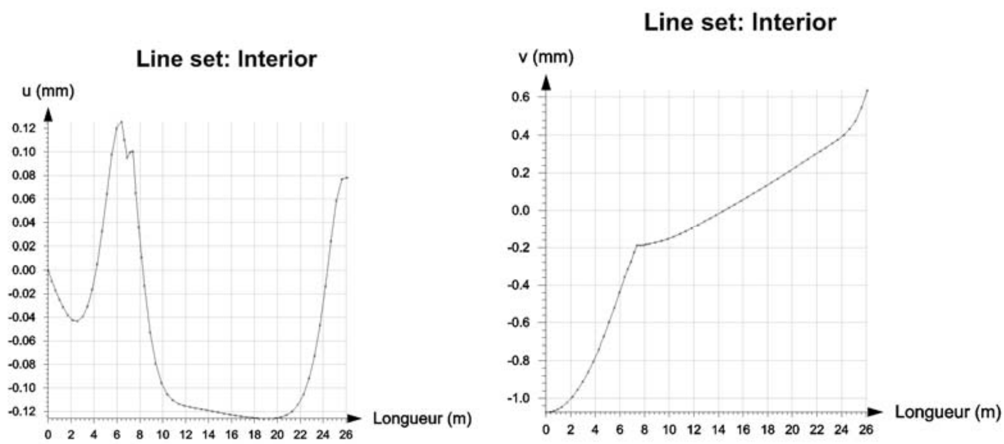
a) b)

Fig.2. Maximum principal stress σ_1 and minimum σ_2, in kN/m^2, on the internal contour of the support structure



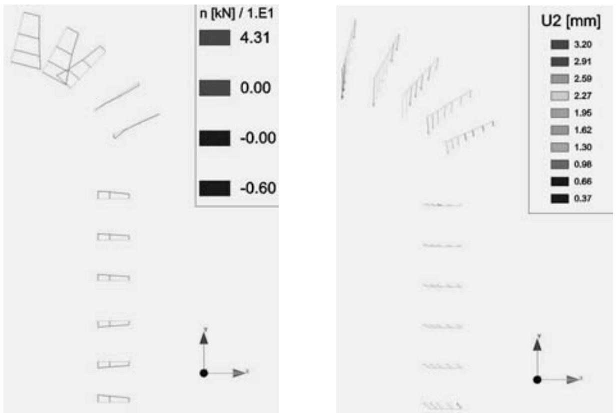
a) b)

Fig.3. Safety coefficients in the various points on the internal contour of the support and on the excavation contour of the surroundings rocks



a) b)

Fig.4. Horizontal u and vertical v displacements, in mm, on the internal contour of the support structure



a) b)

Fig.5. Bolt loading

a) Loadings *n*, in kN ; b) Total displacement vectors, in mm

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Scientific Reviewers:
Prof. PhD. Eng. Mircea Georgescu

IMPLEMENTATION OF THE EUROPEAN PROVISIONS INTO THE NATIONAL LEGISLATION – RESEARCHES ON THE DESIGN OF A FACILITY FOR THE STORAGE OF INDUSTRIAL WASTES

CONSTANTIN LUPU[*]
LADISLAU KOVACS[**]
EMILIAN GHICIOI[***]

Abstract: *INCD INSEMEX – Petroşani drew up the documentation necessary for an undertaking that dealt with the getting of the integrated environment authorization with the view to producing a clean facility for the storage of industrial wastes. Consequently there have been drawn up 4 study reports on the location of the new storage facility, its impact over the surrounding environment, the management of waters located on the new location and the design of the storage facility in compliance with the new regulations in force.*

Key words: *industrial wastes, surrounding environment, wastes dump, furnace slag, levigation test*

1. PRELIMINARY RESEARCHES VIEWING THE SELECTION OF THE LOCATION FOR THE NEW (STORAGE PLACE) STOREHOUSE

1.1. Location of storage place

„Study Report on the Location and Classification of the Storage Place in Relation to the Nature of the Material” was drawn up during the first stage of this process. According to the results of the investigations carried out to settle the behavior of levigated matters in the presence of water (see the Order no. 95/2005 issued by MMGA on the acceptance criteria and the preliminary procedures for the acceptance of wastes for furthers storage and the national list covering the wastes that are being accepted from each class of wastes and SR EN 1425/1, 2, 3, 4/2005), there has been concluded that the new storage place shall be classified as non – hazardous one [1].

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* *PhD. Eng. – Senior scientific researcher II – INCD INSEMEX Petroşani*
** *PhD. Eng. – Senior scientific researcher I – INCD INSEMEX Petroşani*
*** *PhD. Eng. - Senior scientific researcher III – INCD INSEMEX Petroşani*

All these investigations were carried out on slag wastes sampled during the months of July and August 2006.

As a result of the analysis carried out on the site, the proposal was to locate the new storage place on the land that belongs to the undertaking enclosure within the borders of the ancient industrial waste dump; this is because the ground the underground and the underground waters where the new storage place is located are already polluted due to the long- time storage of hazardous wastes come from different engineering processes.

The selection of location depended on the minimum distance that should be kept up to the nearest inhabited dwelling place, i.e. 1000m and the minimum distance up to the nearest river (in accordance with Law no. 107/1996 updated). The location of the new storage place has been arranged in accordance with the topometrical measurements (in STEREO 70 coordinates) on the former slag waste dump of the undertaking.

Three drillings were carried with the view to determining the physical and mechanical parameters of the base and monitoring: one drilling was to determine the lithology of the base and two drillings were for the monitoring the quality of the underground water both during the operational period of the deposit and during post – closure period. The result showed that the supporting surface of the base meets the legal requirements. Laboratory tests were carried out together with an expert company and they revealed the engineering characteristics regarding the settlement of slag on the storage place (slope angle, maximum height of the waste dump and calculation of the supporting power of the surface).

By taking into consideration the requirements in the environment legislation, the physical and mechanical parameters of the base and the requirements said in the Ord. no. 757/2004, there has been designed a new storage place of 60 x 60m, that can be enlarged in the following years [5].

Based on the environment protection legislative, the requirements of Ord. 757/2004, and physical- mechanic parameters, the new waste dump has dimension of 60 x 60 m, with possibility to extend.

1.2. Impact of the waste storage place over the surrounding environment

For a proper evaluation of the impact over the surrounding environment, INCD INSEMEX – Petroşani has drawn up a study that analyses the impact of the waste storage place over the environment: "Evaluation study of the impact over the environment" and "Report on the evaluation study of the impact over the environment", with the observance of the requirements stated in the Ord. 860/2002. Accordingly, there was monitored the present state of the environment parameters on the location, there were identified the potential sources that pollute the environment parameters within the area under analysis and their impact over the component elements of the environment, as well the suitable measures for diminishing this pollution. Consequently, the proposal covers periodical measurements of powders in suspension found on the location of the storage place and of imissions into the surrounding environment, so as to gather information on the impact generated by the activity carried on in the slag storage place. Thus, there shall be measured:

- the suspended powders at the working place (the working face);

- the imissions of suspended particulate matters (PM10) and of settling powders within the area adjacent to the storage place;
- the quality of underground waters through monitoring drill: F001, F002, F03, F004;
- the parameters of the levigated matters arrived in the pool that neighbors the storage place;
- the indicators of rain waters collected by the pool;
- the qualitative parameters of surface waters;
- the quality of soils adjacent to the storage place

The monitoring period shall be stated by the region authoring for the protection of the environment. The environment parameters shall be monitored both during the operation period of the storage place and during the closure and post- closure period. For a thorough analysis of probable occurrence and seriousness of adverse effects over people's health and over the surrounding environment, there have been identified and evaluated the risks induced by the storage of these slag wastes.

As the furnace slag contains heavy metals (lead, zinc and compounds, cadmium and compounds, arsenic and compounds), there have been analyzed the effects of these toxic slags over man and environment.

CEPROMIN Deva carried out levigation tests on two furnace slag samples taken from the base level of the former storage facility in order to characterize the slag waste and settle the class of the deposit in compliance with the Ord. no 95/2005.

Previous to levigation testing, the slag samples were analyzed from a physical and chemical point of view and there were determined the main features: grain size, humidity, content in heavy metals (Cu, Pb, Zn, Fe, Ni, Co, Cr) and different oxide compounds (FeO, Fe_20_3, $Si0_2$, Al_20_3, $Si0_2$, CaO, MgO), as well the content in As and S [3].

The levigation tests were carried out in compliance with the specifications stated by the standard SR EN/12457-4; the eluates resulted from the levigation test were analyzed by the help of atomic absorbtion (heavy metals), photocolorimetry (As, Hg, Mo), complexometry and all the other parameters were analyzed by gravimetry.

The content of chemical indicators from the eluate of the levigated slag was diluted under the border values approved by the Order no. 95/2005 that correspond to non- hazardous, grain wastes.

The experts at chemical laboratory of S.C. CEPROMIN S.A. Deva analyzed soil samples taken from the bottom of the pit (Fig.1) and from the left bank of the river near by the storage facility, at the borderline of the industrial platform.

Lead, zinc and cadmium concentration measured in the witness soil sample taken from the left bank of the river showed higher values- than the border values admitted by the intervention threshold for soils with less sensitive use, according to the Order no. 756/1997.

Also, lead, zinc and cadmium concentrations measured in the soil sample taken from the bottom of the hole exceeded tens of times the intervention threshold for soils with less sensitive use, according to the above said order.

For the environment protection, there have been stated specific measures to diminish pollution of water, air, soil, human dwellings and vegetation.

Fig. 1. Excavation of a hole on the ancient waste dump

1.3. Water management for the furnace slag storage place

With the view to meeting the legal requirements in force on the water protection, INCD INSEMEX – Petroşani has drawn up the necessary documentation to get the water management permit in compliance with the current legislation.

The system used to collect the levigated matter comprises the drainage layer for the levigated matter, the drainage pipe for the levigated matter, the pipes that collect the levigated matter, manholes, pump, and the pipe that removes the levigated matter.

A collecting pipe made of PEHD is laid on the floor of the storage place, at a depth of 2,5m; 2/3 of the pipe cross – section area has round perforations, having D $\geq$ 200 mm. Perpendicular to this pipe, there are laid the same type of pipe, with 15m between them. These pipes are intended to collect the levigated matter come from the extreme points of the storage place.

The levigated matter is driven towards the base of the storage place; subsequently it is conveyed with the help of PEHD type polyester pipes towards the proofed pool located at one side of the storage place. The last stage comprises the discharge of the levigated matter into the deaning station located on the industrial platform of the undertaking.

2. DESIGN OF THE PLACE FOR THE STORAGE OF FURNACE SLAG

The design comprises the arrangement of a new place for the storage of furnace slag and it uses a surface of land adversely affected by pollution; this surface is located within the ancient waste dump that doesn’t allow other uses.

After clearing a surface of land, there was produced the new waste storage place for furnace slag that meets the provision of the Government. Decision no. 349/may 2005 on the “Storage of wastes and the Technical Norms no. 757/ November 2004 on the storage of wastes and the provisions stated by the Decision no. 349/2005 [2].

The waste storage place for the furnace slag consists of cells (the first cell of 60 x 60m), on the North – North Western side of the industrial waste dump. The surface of first cell allows the storage of an amount of slag generated during an one year period, i.e. 60.000 to in 2006.

The cell no.2 was produced during the operational period of the first cell; the land surface available for the rest of cells provides a 20 year period of operation for this new storage place. The project includes execution details of the proofed pool (the specific parameters of the bentonitic geocomposite SEALTEX 5000, of JUNIFOL HDPE2 diaphragm of NETEX APP 1200 UVLS geotextile), the system that drains the levigated matter, the system that discharges the levigated matter, the method used to close the storage place and the guarding trenches of the storage place. The constructive elements are shown in Figs. 2, 3 and 4.

Slag wastes are settled in horizontal ascending layers of 1,5 – 2m. The first layer is accomplished by the settlement of the slag in transverse layers in advance; the rest of slag layers are settled in retirement.

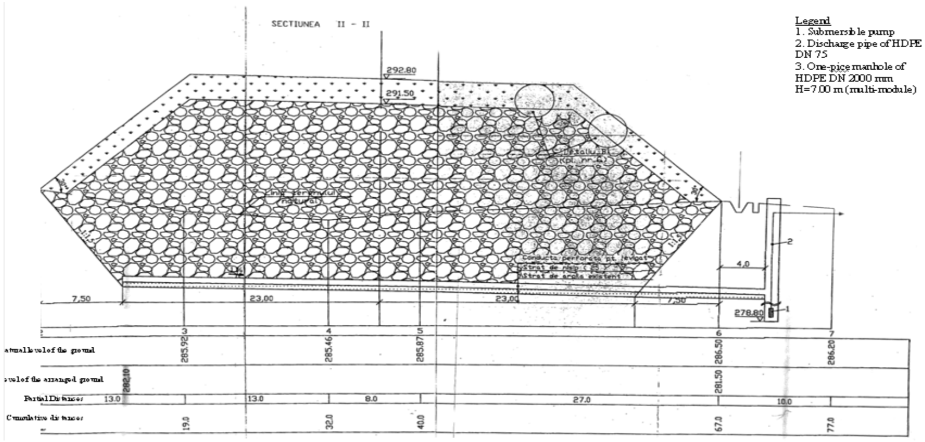
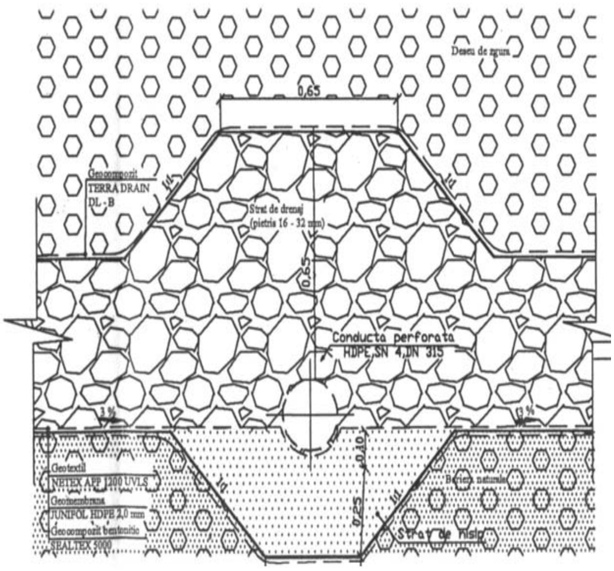


Fig 2. Constructiv elements of the slug

The levigated matter resulted from slag humidity and of rain waters fallen on the storage place shall be collected in a manhole and then, it shall be discharged with the help of a submersible pump into the system that cleans the waste waters of the undertaking [6].

Once the cell of the storage place is filled, the upper layer of the storage place is proofed with the help of a geo membrane to prevent the ingress of rain waters into the body of the waste dump and there shall be produced guarding trenches. A drainage layer shall be put above the geomembrane; it is a geotextile layer and it is the support for the vegetable soil.



Note
The pipe shall be drilled at the upper side at 2/3 of the hight

Fig. 3 – Levigat draining system

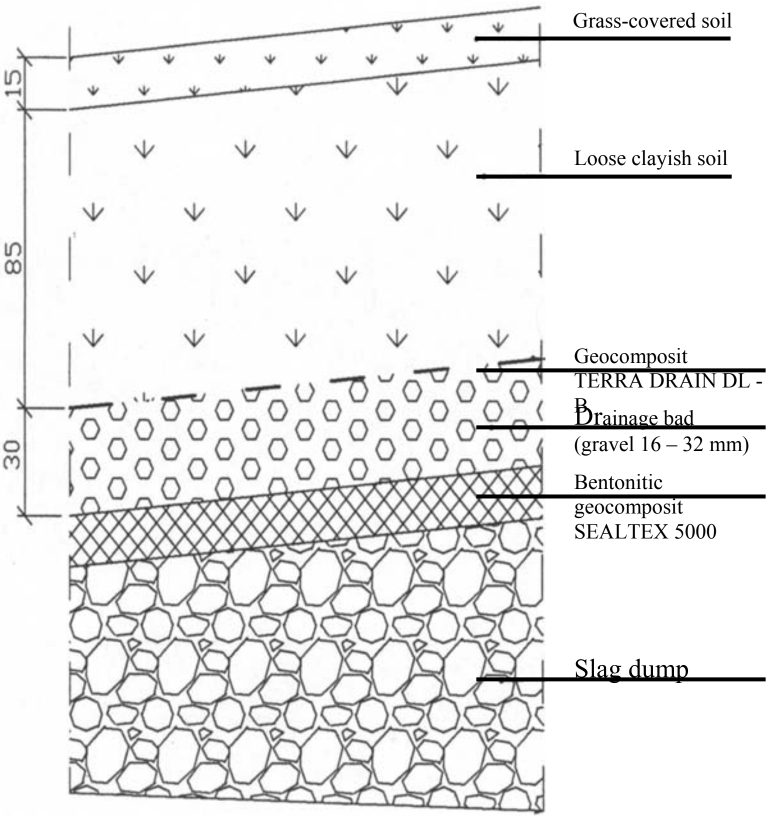


Fig. 4 – Closure procedure of the slag dump

4. CONCLUSIONS

The storage of the furnace slag on the storage place belong to the undertaking is under the incidence of OUG no 152/2005 regarding the prevention and integrated control of pollution [4]. The program of compliance states the cessation of storage for the slag furnace on the former location until 2006 and there shall be arranged a new storage place, in accordance with the requirements stated by HG 349/2005 and by Ord. no. 757/2005.

The selection of the location related to the requirement about the minimum distance of 1000 m up to the nearest dwelling house and the minimum distance of 15 m up to the nearest waterway (according to the law no 107/1996 uploaded).

The location of the new deposit meets the results of topo measurements made in STEREO 70 coordinates.

The location of the new storage places on a surface of land owned by the undertaking and the use of a polluted surface of land which couldn't have been used for other purposes represent to positive aspects for environment protection.

There were made physical and chemical analyses on the been analyzed the wash ability of soluble salts from slag, furnace slag; the quality of the soil related to the location of the storage place has been analyzed with the help of drill holes and the quality of water has been analyzed with the help of monitoring drill holes.

The results gained from all these analyses helped to evaluate the impact over environment parameters and to forecast the impact of certain activities (such us transportation, storage and arrangement of furnace slag on the storage places over the surrounding environment.

The construction and the operation of the places for the storage of furnace slug doesn't influence to a great extend the air, the water, the soil and the vegetation whiting this area. As a result, this storage places is classified as non-hazardous one, on condition that all the measures stated in these paper should be observed.

In accordance with the legal provisions in force, the operator of the storage place shall have to perform post-closure monitoring all trough a period that has been previously determined by the competent environment authority (minimum 30 years).

Monitoring includes analyses and measurements of emissions and imissions as well a control of the environment parameters in accordance with the legal provisions in force.

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Scientific Reviewers:
Prof. PhD. Eng. Mircea Georgescu

NEW TECHNOLOGY IMPLEMENTED IN THE SETTLING OF COMPLEX VENTILATION NETWORKS

**DORU CIOCLEA*
CONSTANTIN LUPU**
LIVIU JURCA***
ION GHERGHE******

Abstract: *For providing the best possible occupational health and safety conditions for the staff working in potential explosive and/or toxic atmospheres, there are used special ventilation installations. For the case of underground operations, this aspect is quite complex because the ventilation network covers tens of kilometers. n order to settle the ventilation network related to Paroşeni mine, there has been used a Canadian expert software called CANVENT. His paper presents both the specific elements regarding the settlement of the ventilation network and the above said simulations.*

Key-words: *mining ventilation, software, ventilation network analysis*

1. GENERAL NOTIONS

The mine network necessary for the mining of useful mineral substances displays a high complexity, different shapes and cross-sectional areas and can reach tens of kilometres in length.

For getting the best possible working conditions in underground, it is necessary to provide the primary protection, i.e. suitable ventilation [3]. The purpose of this ventilation is to:

- provide the concentration in oxygen necessary for the personnel currently working in underground;
- dilute the explosive and/or toxic gases existing in the mine network;

* *PhD. Eng. – Scientific Researcher II, INSEMEX Petroşani*
** *PhD. Eng. – Scientific Researcher II, General Director of INSEMEX Petroşani*
*** *Eng. - Scientific Researcher II, INSEMEX Petroşani*
**** *Eng. - Scientific Researcher III, INSEMEX Petroşani*

- diminish the heat emitted inside mine workings, both due to human activities and to thermal gradient. A good ventilation of each mine working involves the best possible repartition of air flows along each branch of the ventilation network [4]. In this spirit it is necessary to settle the ventilation network of each mine. An example of complex ventilation network is the one belonging to Paroşeni mine.

2. ANALYSIS OF THE VENTILATION NETWORK BELONGING TO PAROŞENI MINE AND MEASUREMENTS CARRIED OUT IN SITU

The general ventilation of Paroşeni mine is of the upward type, under the influence of the depression s created by the main ventilation station no. 18 VOD 30.

At Paroşeni, coal is extracted through four workings:

- face working panel 4, bed 3, block VI level 300;
- face working panel 24, bed 3, block V level 250;
- working with short face (pillar), bed 3, block V level 250;
- working with short face, panel 3 bed 3, block V level 250.

According to the annual project for general ventilation for the second, third and forth trimesters that takes into consideration the structure of the ventilation network, the dispersion of the working faces in operation, as well the operation of the main ventilation station no. 18 VOD 30, there have been devised three main ventilation circuits:

- the ventilation circuit of the level 250;
- the ventilation circuit of the level 300;
- the ventilation circuit of the level 360.

In order to find out the real values of the aerodynamic parameters specific to the mine workings that are part of the mine ventilation network, there have been carried out measurements of the air flows and of the depressions.

The whole ventilation network includes 171 junctions (knots) and 216 branches [2].

3. SETTLING THE VENTILATION NETWORK OF PAROSENI MINE

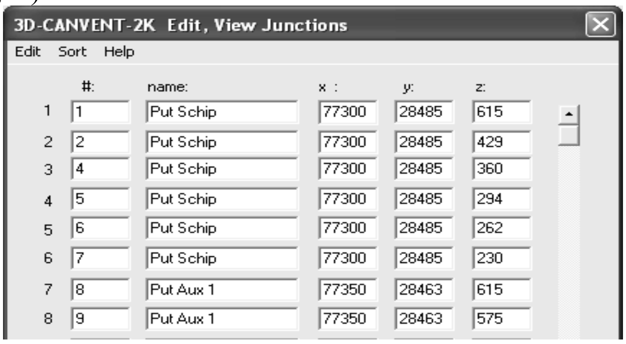
For providing the best solution available for such a complex ventilation network, we have used the Hardy-Cross method for successive approximation. This method represents the grounds of expert software CANVENT designed in Canada [1]. 3D – CANVENT is a Window type application and it has been designed to support the planning, design and analyze the mine ventilation systems. 3D – CANVENT represents a mixture between a friendly use of graphic representations and the capability to design 3D networks that makes possible to visualize the ventilation network in a 2D system (X – Y, X – Z, Y – Z) and/or 3D system (X–Y –Z).

This software helped us to provide the solution for the ventilation network as well an optimization of the air flow distribution within the ventilation branches.

Settling the ventilation network of Paroşeni mine made necessary to run several stages:

– Marking the junctions of the ventilation network on the spatial diagram;

– Determining the geodesic coordinates of the identified junctions and inputting the geodesic coordinates of junctions and the existing branches into the database of the software (see Fig. 1).



3D-CANVENT-2K Edit, View Junctions

Edit Sort Help

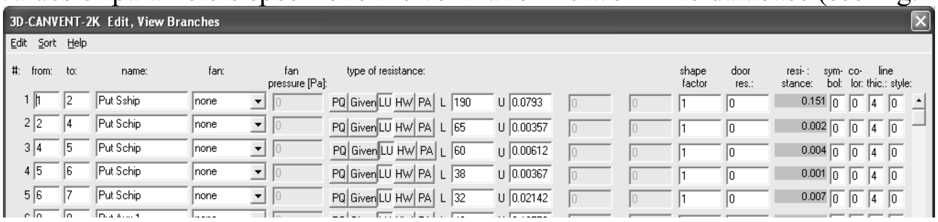
	#:	name:	x :	y:	z:
1	1	Put Schip	77300	28485	615
2	2	Put Schip	77300	28485	429
3	4	Put Schip	77300	28485	360
4	5	Put Schip	77300	28485	294
5	6	Put Schip	77300	28485	262
6	7	Put Schip	77300	28485	230
7	8	Put Aux 1	77350	28463	615
8	9	Put Aux 1	77350	28463	575

Fig. 1. Table with knots

– The carrying out of measurements in situ; these measurements include:

- measurements of the aerodynamic parameters of mine workings;
- measurements of the geometrical parameters of mine workings;
- measurements of the physical parameters of the air;

– Calculation of aerodynamic strength specific to each branch and the inputting the values of parameters specific to the ventilation network into database (see Fig. 2);



3D-CANVENT-2K Edit, View Branches

Edit Sort Help

#:	from:	to:	name:	fan:	fan pressure [Pa]:	type of resistance:					shape factor	door res.:	resi-: stance:	sym-bol:	co-lor:	line thic.:	style:
1	1	2	Put Sship	none	0	PQ Given LU HW PA	L 190	U 0.0793	0	0	1	0	0.151	0	0	4	0
2	2	4	Put Schip	none	0	PQ Given LU HW PA	L 65	U 0.00357	0	0	1	0	0.002	0	0	4	0
3	4	5	Put Schip	none	0	PQ Given LU HW PA	L 60	U 0.00612	0	0	1	0	0.004	0	0	4	0
4	5	6	Put Schip	none	0	PQ Given LU HW PA	L 38	U 0.00367	0	0	1	0	0.001	0	0	4	0
5	6	7	Put Schip	none	0	PQ Given LU HW PA	L 32	U 0.02142	0	0	1	0	0.007	0	0	4	0

Fig. 2. Table with branches

– The 2D or 3D drawing of the ventilation network;

– Settling the ventilation network. Both the direction and the optimum distribution of the air flows along each branch are being identified in this stage (Fig. 3).

4. MODELING / SIMULATIONS IN THE GENERAL VENTILATION NETWORK OF PAROSENI MINE MADE WITH THE HELP OF 3D CANVENT SOFTWARE

With the view to evaluating the future changes that may come up in the mine general ventilation network in relation to the distribution of the air flows [2], both from the point of view of their size and direction, the locations of the structures for the control and drive of air flows, as well the types of mine ventilation structures, there have been settled, in full agreement with the department for occupational health and safety at Paroseni mine, the following simulations:

Simulation no. 1 – Settling the ventilation network with the change of the route used to discharge the poisonous air on the new inclined plane Panel 4/3/VI level 300, implicitly of the related ventilation structures.

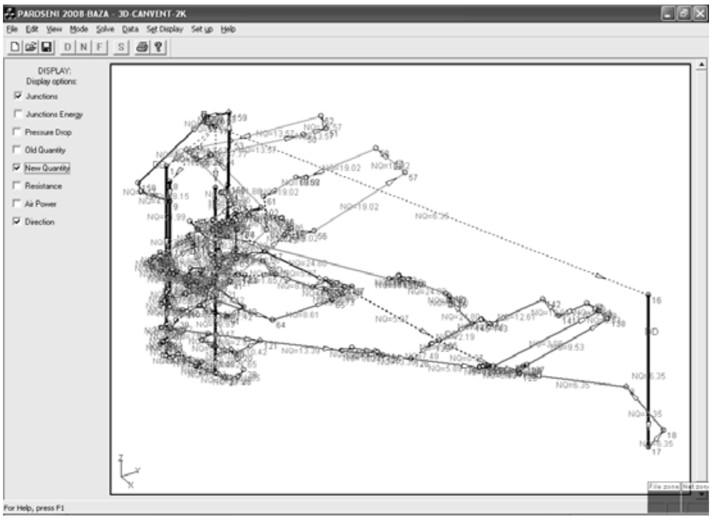


Fig. 3. The direction and the optimum distribution of the air flows along each branch are being identified

Simulation no. 2 – Settling the ventilation with the opening of a group comprising 3 air doors at the level 250, at the base of the ventilation rise 250 – 360.

Simulation no. 3 – Settling the ventilation with the opening of a group comprising 3 air doors at the basis of the behind shaft 360 – 575.

Simulation no. 4 – Settling the ventilation with the opening of a group comprising 3 air doors at the connecting gallery no. 4, between the west longitudinal way level 250 and the conjugated ventilation longitudinal way level 250.

There follows the presentation of results gained after performing two of the above-said simulations (i.e. simulation no. 1 and simulation no. 4).

4.1 Simulation no. 1 – Settling the ventilation network with the change of the route used to discharge the poisonous air on the new inclined plane Panel 4/3/VI level 300, implicitly of the related ventilation structures.

There have resulted the following aspects after the simulation no. 1 (Fig. 4):

– There has been a significant change in the distribution of the air flows inside the ventilation network of Paroşeni mine, especially within the mining area related to the block VI; subsequently, the flow rate diminished from12.61 m^3/s to 1.29 m^3/s at the face no. 4/3/VI;

– As a result of settling the ventilation network (after the location of 5 ventilation structures), there resulted a distribution of the air flows almost similar to the distribution of air flows at the basic model, except the flow rate gained at the face panel 4/3/VI where it diminished from 12.6 m^3/s to 10.97 m^3/s;

– A diminution of the annual costs related to air conveyance along the blind shaft (branch 156-159) with 3.82% from 387,019 lei to 372,219 lei;

– An increase of the annual costs related to air conveyance along the air way W that connects to the blind shaft level 360 (branch 100-155) with 7.73%, from 3,168 lei to 3,413 lei,

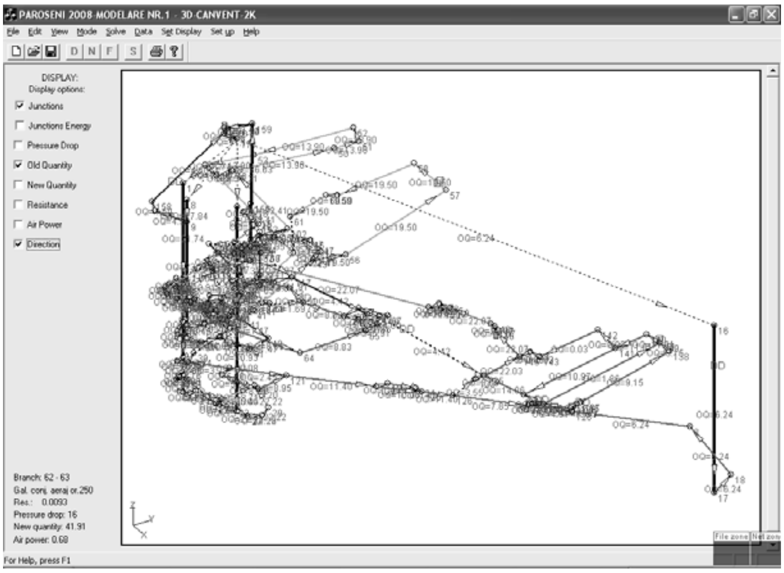


Fig. 4. Aspects after the simulation

– A diminution of the annual costs related to air conveyance along the air way that connects to the blind shaft level 425 (branch 154-156) with 24.91%, from 562 lei to 422 lei;

– There has been registered a diminution with 3.53%, from 95,519 lei to 92,150 lei for the ventilation canal of the main ventilation station no. 18 VOD 3.0 (branch 159-160);

– There has been registered a diminution of the annual costs related to air conveyance with 0.35%, from 1,118,298 lei to 1,114,401 lei for the whole main ventilation station no. 18 VOD 3.0.

4.2 Simulation no. 4 – Settling the ventilation with the opening of a group comprising 3 air doors at the connecting gallery no. 4, between the west longitudinal way level 250 and the conjugated ventilation longitudinal way level 250

Simulations no. 4 relied on the simulation no. 1 and it comprised a removal of the air doors located on the connecting gallery no. 4 between the longitudinal way W level 250 and the conjugated ventilation longitudinal way level 250, branch 48-55 (Fig. 5).

There have resulted the following aspects after the simulation no. 4:

– The distribution of the air flows in the ventilation network of Paroşeni mine didn't change significantly for the ventilation circuit panel 1/3/V level 250; there were moderate changes for the face panel 4/3/VI, where the air flow rate diminished from 12.61 m^3/s to 10.89 m^3/s and for the face panel 2/3/V where the flow rate diminished from 19.02 m^3/s to 17.2 m^3/s. The distribution of air flows changed significantly for the ventilation circuits panel 1/3/III level 250, where the air flow rate diminished from 13.57 m^3/s to 4.47 m^3/s;

– A diminution of the annual costs related to air conveyance along the blind shaft (branch 156-159) with 2.72% from 387,019 lei to 376,492 lei;

– An increase of the annual costs related to air conveyance along the air way W that connects to the blind shaft level 360 (branch 100-155) with 10.48%, from 3,168 lei to 3,500 lei;

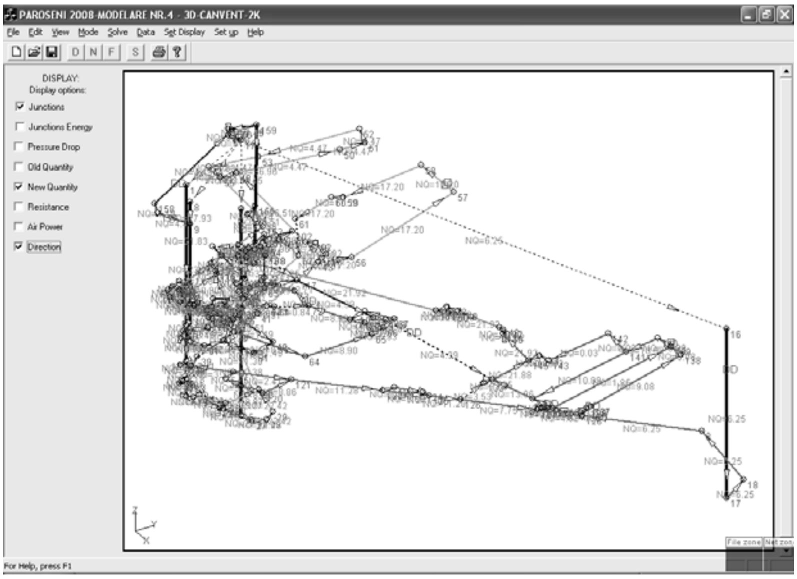


Fig. 5. The conjugated ventilation longitudinal way

– A diminution of the annual costs related to air conveyance along the air way that connects to the blind shaft level 425 (branch 154-156) with 26.3%, from 562 lei to 414 lei;

– There has been registered a diminution with 2.51%, from 95,519 lei to 93,120 lei for the ventilation canal of the main ventilation station no. 18 VOD 3.0 (branch 159-160);

– There has been registered a diminution of the annual costs related to air conveyance with 0,25%, from 1,118,298 lei to 1,115,557 lei for the whole main ventilation station no. 18 VOD 3.0.

5. CONCLUSIONS

– For modeling the ventilation network of Paroşeni mine, there has been used the iterative method of successive approximations; the whole software for settling the ventilation networks (3D - CANVENT) relies on this method.

– The elements necessary to run the software (i.e. the inputs) are gained after monitoring the ventilation process.

– Based on the current ventilation network, there have been determined the simulations of possible situations.

– The simulations have underlined both the important part played by the group of air doors and their influence over the distribution of air flows at each branch.

– Each simulations intended to acquire the necessary flow rates in accordance with the Annual ventilation project, measured at faces of the main ventilation circuits and at the main ventilation station. It has to be stipulated the type of control and/or regulating structures, as well their locations in order to be able to reach the desired distribution.

– For each simulation, there has been determined the operating mode of the two fans at the main ventilation station no. 18 VOD 3.0; these simulations have also allowed to evaluate the ventilation capabilities of the mine for each situation apart.

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Scientific Reviewers:
Assoc. Prof. PhD. Eng. Roland Moraru

PREDICTION OF FUGITIVE DUST DISPERSION AND DEPOSITION WITHIN AND FROM SURFACE MINING OPERATIONS THROUGH COMPUTATIONAL MODELLING TECHNIQUES

GABRIEL-BUJOR BĂBUȚ*
ROLAND-IOSIF MORARU*

Abstract: *The extraction and processing of minerals from surface mines and quarries can produce significant fugitive emissions as a result of site activities such as blasting, unpaved road haulage, loading, primary crushing and stockpiling. Uncontrolled fugitive dust emissions can present serious environmental, health, safety and operational issues impacting both site personnel and the wider community. This paper proposes that optimal modeling of open pit emissions may be more accurately achieved by the use of a multi-scale predictive modeling approach utilizing computational fluid dynamics methods for high resolution near source dispersion and conventional Gaussian based methods for far field dispersion modeling. Typical operating emissions and meteorological conditions are obtained from long term data records collected at a large operating quarry extraction operation. Emissions can be modeled using a specific framework within conventional atmospheric boundary layer profiles expressed as functions of turbulence and velocity parameters under assumed neutral conditions.*

Key words: *dust, dispersion, turbulence, open pit, flow mode*

1. INTRODUCTION

As open pit mines and quarries become deeper and more productive the potential to produce greater pollutant emissions including fugitive dust emissions will increase. To maintain and enhance the health and safety of the extractive and transport operations and to minimize off site dust emissions it is necessary to design effective mitigation measures to minimize fugitive dust emissions, and to maximize the ventilation of the pit opening to dilute, disperse and remove fugitive dust from the workings. The principal tool available to environmental engineer is to use the shape of the excavation and the surrounding topography to harness the penetration of the natural

* *Assoc. Prof, Ph.D. - University of Petroşani, Romania*

wind systems to maintain the air exchange rates within the mine opening. The determination of the internal ventilation regime within the open pit is a complex process as the driving natural wind system will change subject to diurnal and seasonal changes in its strength and direction, which are dictated by the local wind systems and the differential heating of the earth's surface by the sun.

The background local wind system may be modeled by the construction of a three dimensional velocity profile called the atmospheric boundary layer (ABL). The ABL model requires the definition of the surface roughness of the surrounding topology, a measured reference height velocity above the terrain and the determination of the thermal stability of the atmosphere. The strength and the direction of the resulting downwind ABL will be influenced by the topography of the terrain surrounding the open pit and the shape and depth of the excavation. The internal microclimate is created by a combination of: the degree the external ABL penetrates the mine opening; the in pit topography; and the diurnal heat exchanges between the sun, the internal surface of the pit and the atmosphere.

These internal ventilation flows will be responsible for the initial dilution, dispersion of any fugitive dust emissions within the open pit. In turn, the amount of air exchange affected between the internal ventilation regime and the mainstream ABL flowing over the mine opening will determine the degree of off-site fugitive dust emission experienced. Dust dispersion rates may be attenuated by low airflow exchange rates caused by the recirculation of local ventilation flows or containment by thermal inversions. The retention of ambient dust levels will potentially decrease visibility and increase the exposure of workers.

2. PREVIOUS OPEN PIT VENTILATION MODELS

Russian researchers have over the past thirty years published the results of research studies detailing the development and application of mathematical models to simulate the ventilation of open pits. The basic theoretical and practical principles of the design of natural open pit ventilation systems are contained within the books authored by Nikitin and Bitkolov [14], and Bukhman et al [9]. These studies identified that the natural dynamics of the ventilation air exchanges experienced within open pit excavations, is principally influenced by the background ABL, the generation of any subsequent recirculation flows or a combination of the two. The physical factors determining the in pit ventilation flows are: the ratio of the width of the mine excavation in the direction of the prevailing wind to the depth of the mine and the angle of the leeward facing wall. Bukhman established that the ventilation of open pit mines is also affected by the surrounding topography. Thus, in areas of low lying relief the location of such features as waste rock dumps will greatly influence the in-pit ventilation. Belousov [7] reports the results of a theoretical study which confirm that the intensification of the internal ventilation dilution efficiency of an open pit mine due to the channeling of the ABL by external rock dumps by channeling air walls. In a later modelling study, Belousov [8] demonstrated that the ventilation of an open pit could be enhanced by reducing the lee ward wall recirculation zone by the use of variable guide vanes or the use of ducted forcing fans [6]. Baklanov [3] presented the results of a field validated modeling study that investigated the dispersion of pollutants from within naturally ventilated open pits. Baklanov [4] established a thermodynamic model of the dynamics of open pit ventilation systems. The model can simulate the diurnal change that takes place during the differential solar heating of the surface during the day and the release of this energy during the evening. In a later paper Baklanov [5] presents the results of a series of numerical modeling case studies that consider the pollutant dispersion within naturally ventilated open pits, for a range of different external wind speeds and directions, internal thermal stratifications, recirculation flow regimes and the interaction of local open pit thermal circulations with the external atmospheric flow field. The most complex and interesting example in terms of the physical processes involved is the modeling of the atmospheric dynamics, micro climate and pollutant dispersion within deep open pits. Relative to the more natural complex topographies, open pits have a greater isolation from the external atmospheric flows and possess a greater pitch angle of slope. However, the modeling of the ventilation of open pits presents several unique features:

- the presence of more complex internal inhomogeneous flow surfaces, and in particular the great influence of the external atmospheric boundary layer upon the circulation in the inner open pit, which demands a three dimensional fluid modeling approach;
- the great influence of the slope radiation effects upon the open pit micro climate, forming local temperature inversions and local winds;
- the open pit atmosphere is more closed than the external one, leading to a sharp decrease of the inversions.

3. DUST DISPERSION MODELS FOR SURFACE MINING OPERATIONS

Reed [17] has produced a comprehensive review of the dust dispersion models that have been developed or applied to the prediction of dust from surface mining operations including quarries. The dust dispersion models used to predict emissions from surface mining operations are generally adapted from existing regulatory industrial air pollution models. A major challenge to the modeling the dispersion of fugitive dust emissions from deep surface mines or hard rock aggregate quarries is the influence of the in pit meteorology. As most Gaussian plume dispersion models used for regulatory purposes have been developed to model downwind dispersion of dust from sources across a flat or undulating terrain, these models cannot account for the influence that the complex flow regimes that exist within quarry openings. As fugitive dust emissions within a quarry are transported and dispersed by the local airflow field within the quarry, there is a need to develop transport and deposition models that reproduce the local effects produced by these flows.

The airflow regime within a deep quarry opening is produced by the combined action of the mechanical shear of the atmospheric boundary layer across the surface opening and the thermal buoyancy forces created by the differential heating of the quarry surface by the passage of the sun during the day. In addition, the occurrence of thermal temperature inversions at night may also assist trapping the dispersion of the dust emissions from within the quarry. The combination of these forces creates: (1) an external flow field across the surrounding terrain and across the interfacial quarry opening that is governed by the atmospheric boundary layer, and (2) an internal flow field driven by the combination of the mechanical shear of the atmospheric boundary layer across the surrounding terrain and the airflow within the quarry opening, and the thermal effect created by the differential heating of the internal quarry surfaces by the sun (see Figure 1). To improve the understanding and modeling of these processes requires the adoption of a multi-scale modeling approach.

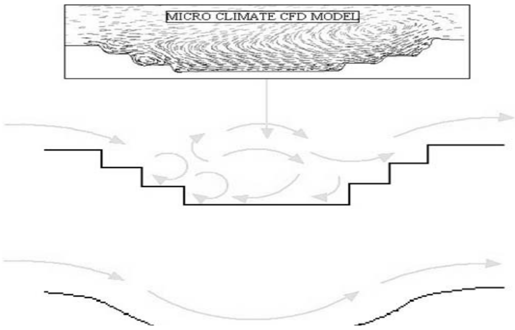

Fig. 1. The wind pattern affecting dust dispersion in quarries

Thus, a fugitive dust emission within a quarry will be transported and dispersed by this locally generated flow field. The creation of this chaotic, and often recirculatory in-pit flow regime within the quarry will increase the residence time of the entrained dust particles within the confine of the quarry. As the deposition of particles is governed by Stokes Law, any increase in the dust residence time within the quarry may allow either: (1) allow the settlement of many of the dispersed dust particles or (2) increase the probability of their removal by impaction on the internal surfaces of the quarry. Consequently, only a fraction of the fugitive dust emitted and dispersed within the confines of the quarry will cross the interfacial layer defined between the quarry opening and the atmospheric boundary layer to be dispersed and potentially deposited downwind across the surrounding terrain. The fraction of the dust emission which crosses the interfacial layer between the quarry opening and the ABL will form an equivalent emission source whose downstream dispersion may be modeled by conventional Gaussian plume models.

The existence and effects that these internal pit flows have on the retention of dust emitted within the quarry opening was recognized by research work conducted by Cole and Fabrick [10]. The authors note that the earlier work of Shearer concluded that for shallow open pit mines that approximately one-third of the fugitive dust emissions from mining activities escape the open pit. This is a very simplistic model that is representative of the box model algorithm. Further discussions are provided for a Gaussian plume model described by Winges [20]. This model calculates the mass fraction of dust that escapes an open pit in terms of an mathematical expression containing the particle gravitational settling velocity (determined from Stokes' Law), the vertical diffusivity, and the depth of the open pit. The resulting mass fraction estimate was employed to scale the areal dust emission factor employed within a Gaussian plume dispersion model to predict the particulate deposition downwind.

4. THE INFLUENCE OF THE IN PIT TOPOGRAPHY

Figure 2 illustrates the predicted influence that the downwind surface topography can have on the dispersion and deposition of fugitive dust emissions. The modeled stationary areal dust source is located at the lowest elevation of the quarry. The prevailing wind direction crosses the quarry opening from the westerly direction. The influence that an increase in the detail of the elevation of the in pit topology has on the dispersion and deposition of the fugitive dust source can be observed from a comparison of the results of successive simulations that increase the density of the in pit domain mesh (from top to bottom). It is concluded that as the detail of the topology increases, the greater is the degree of near source deposition observed, accompanied by a reduction of the downwind dispersion and deposition.

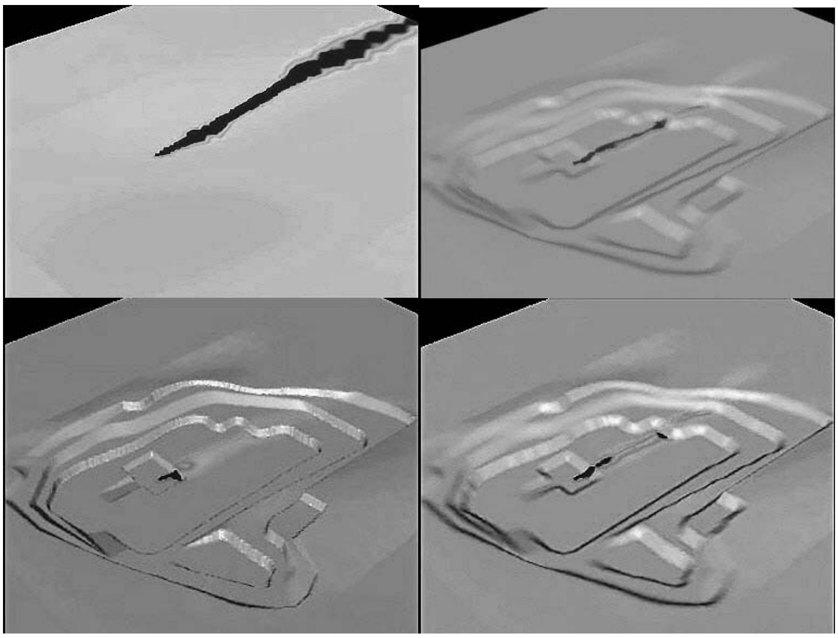

Fig. 2. The influence that the detail of in pit topography has on the dispersion and deposition of a dust plume [16]

Consequently, it is concluded that under neutral stability conditions that the combination of the prevailing wind direction and speed together with the in pit topography can create an in pit ventilation regime and micro climate that will greatly influence both the dispersion and deposition of fugitive dust sources within the confines of the surface opening. The influence of the microclimate may both contain or enhance or attenuate the dispersion and hence dilution of the fugitive dust that may decrease the visibility within the open pit, or may attenuate the fugitive dust emissions within the opening due to the increased impaction and removal afforded by the recirculatory ventilation flows induced within the quarry.

5. FACTORS INFLUENCING IN PIT DISPERSION AND DEPOSITION OF DUST

The 3D computational flow modeling of the ventilation flows within the open pit have been shown to influence the observed dispersion and deposition of fugitive dust emissions within the excavation. It is concluded that the dispersion and deposition of fugitive emissions is governed principally by the location of the emission source and the direction and strength of the prevailing wind characterized by the ABL.

For example, figure 3 illustrates, in his left image the CFD model predictions of the dispersion and deposition of a fugitive dust source close to the leeward high wall of the open pit at the base of the quarry. The moderate strength prevailing wind is from the south westerly direction. There is an observed initial dispersion and deposition of the dust in the shadow of the leeward high wall due to the recirculatory flows generated in this region, followed by a general downwind dispersion and deposition of the remaining emission.

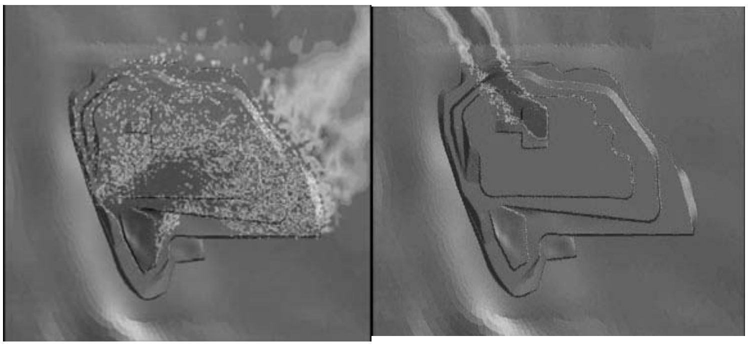

Fig. 3. The dust dispersion and deposition predicted for a fugitive source near the leeward shadow of the prevailing ABL (left) and on the windward side of a strong prevailing ABL (right) [11]

The right image on figure 3 shows the CFD model predictions of a dust source located at the base of the quarry on the windward side of the quarry subject to a strong south easterly prevailing wind. The deposition and dispersion experienced are observed to follow a more conventional downwind pattern. These predicted phenomena were confirmed from both visual observations and quantitative dust sampling studies conducted during blasting and truck haulage events at similar locations within the quarry [1, 11 and 19].

6. CONCLUSIONS

An analysis of the results of the series of previous recent research papers and studies had revealed that a multi-scale modeling approach to assessment of the emission, dispersion and deposition of fugitive dust from in pit fugitive dust sources is useful.

As detailed above, there have been a number of recent research studies concluding that the use of the dust emission models for large open pit and quarry operations, together with the in pit dust retention models together with conventional Gaussian plume dispersion models can produce over predictions of off site emission and deposition. The use of an appropriate three dimensional field validated computational model, may allow the improved simulation of these events, which could allow the mine operator to predict the occurrence of in pit reduced visibility. It is therefore proposed the development of a three scale modeling approach:

a. The development of improved dust emission models to more accurately represent the emission characteristics of the various stationary and mobile in pit fugitive dust sources, including unpaved road truck haulage models.

b. The development of a more complex three dimensional computational fluid dynamic model, that allows the engineer to more accurately predict the influence of the generated in pit microclimate has on the dispersion and deposition of fugitive dust within the open pit workings.

c. The development of an interface between the models described above, with which to determine an areal emission factor dust through a defined area across the mine opening and the transference of the determined area dust emission to the background ABL.

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METHODOLOGY FOR A MAJOR ACCIDENT'S PROPAGATION FLOW INDEX DETERMINATION

MONICA-CRINELA BĂBUȚ*

Abstract: *The quantification of the influence exerted on the targets from an establishments concerned by the Seveso II Directive requirements can be performed through the potential danger resulting by the aggregation of the gravity index and the propagation flow index. Starting from the environment's specific characteristics, the major accident's propagation kinetics and the distance between the target and the danger source, the paper has as goal to depict a propagation flow determination method. In this index determination it was considered the phenomenon isotropy and anisotropy, and the definition of lethal, irreversible, reversible and without effect zones was done based on the national and EU legislation requirements concerning the prevention and control of major accident hazards involving dangerous substances.*

Key words: major accident, propagation flow index, Seveso II Directive

1. THE PROPAGATION FLOW

The flow intensity generated by a major accident develops according various parameters, specific to his propagation environment (air, water, soil). The propagation flow index allows the taking into consideration of these phenomena. So, the gravity index of the physical considered effect is weighted through the propagation flow value [3], [4], [5].

The propagation flow intensity depends on the following parameters:

- the distance to the source of danger (the distance source - target) [3], [4];
- the environment's characteristics, such as the relief and the wind rose [2];
- the propagation kinetics [4], [5], which characterizes the major accident propagation speed; this, at his turn, depends on the environment (air, water, soil) and his characteristics.

The last cited parameter is difficult to be considered. For example, the explosion phenomena are having o very fast kinetics, but the time span required for the

* *PhD-Student.Eng. at the University of Petroşani, babut_crina@yahoo.com*

explosion to occur can be long and, for this reason, it is possible to perform the protection or the targets evacuation from the site. From the environmental characteristics, only the wind rose was considered.

2. FLOW ATTENUATION AND FLOW INTENSITY COMPUTATION

2.1. The distance

In the classic approach, the danger flow is inversely proportional with the square of the distance between the source and the target [1], [4]. This way of approaching the problem is valid when the danger source is quantified by an energy value. In our case, the danger source is quantified through gravity levels, which do not represent an energetic potential. Consequently, the approach proposed in the paper will differ significantly, with respect to this classic approach.

So, in order to define the attenuation as a function of the distance, we have chosen the use of the threshold effects and, based on their values, there were defined four specific effect zones for each major accident. To this purpose, there is proposed the application of the following three effect threshold values:

- the lethal effect threshold (LET);
- the irreversible effect threshold (IET);
- the reversible effect threshold (RET).

The threshold limits for lethal and irreversible effects occurrence are representing concepts and, consequently, classic quantities, specific to the safety reports elaboration. The reversible effect threshold limit allow to define the distance from where a major accident's effects are not significant and then, to define a non - effect zone. Proceeding in the previous way there can be highlighted the following effect zones (see figure 1):

- the lethal effect zone (LEZ);
- the irreversible effect zone (IEZ);
- the reversible effect zone (REZ);
- the no - effect zone (NEZ).

The values corresponding to these threshold limit values, according to the type of physical effect induced are synthesized in table 1 [3], [6], [7], [8].

Table 1. *Effect threshold values for various physical effect categories*

	LET	IET	RET
Overpressure	140 mbar	50 mbar	30 mbar
Thermal flow	5 kW/m^2	3 kW/m^2	1,5 kW/m^2
Toxicity	LC 1 % (30 min.)	IDLH	NOAEL
Liquid pollution	LC 1 % (30 min.)	EC_{50}	NOAEL

In the case of overpressure and thermal flow phenomena, the retained threshold limit values are constant, whatever the involved substance's nature. On the contrary, for accidental atmospheric pollution and accidental aquatic pollution, the proposed threshold limits are specific to the dangerous substance.

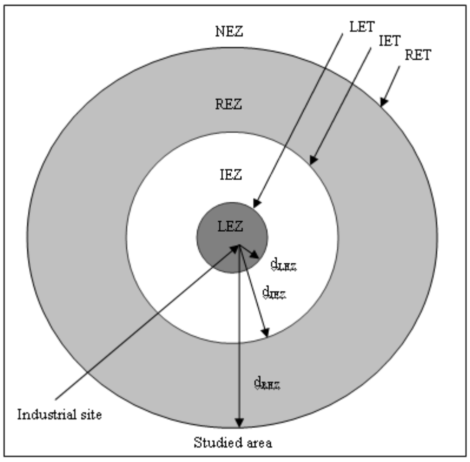

Fig. 1. The representation of the effect zones around the studied area

The physical effect attenuation is not linear but, in a first approach, it will be considered that it is linear on zones. Indeed, the flow vector intensity and the attenuation factor are assessed for each effect zone. The attenuation factor will be determined based on the principle extensively presented in the following considerations, being formed from the following ratios:

• the distances ratio d_i/D, where d_i is the distance source - target, and D, the effect zone length i in which the target is located. The ratio depicts the advancing degree of the accidental phenomenon in a specific effect zone. This ratio depends on the considered effect zone, being defined as it follows:

- if $0 < d_i \leq d_{LEZ}$, then $D = d_{LEZ}$;
- if $d_{LEZ} < d_i \leq d_{IEZ}$, then $D = d_{IEZ} - d_{LEZ}$;
- if $d_{IEZ} < d_i \leq d_{REZ}$, then $D = d_{REZ} - d_{IEZ}$.

The fact that the phenomenon advancement degree in an effect zone is represented by a ratio, involves assuming that the attenuation process is linear, in each defined effect zones.

• the ratio d_{EZi}/d_{REZ}, with $d_{EZi} = d_{LEZ}$, d_{IEZ} or d_{REZ} as a function of the effect zone i. This ratio allows to consider the importance of the studied effect zone (IEZ, LEZ or REZ), with regard to the global effect zone, which through the assumed hypothesis, is the reversible effect zone (REZ).

In order to obtain the attenuation factor α, these two ratios are multiplied, and then the obtained result is normalized by the $\alpha_{normalization}$. The attenuation factor α is the variable representing the accidental phenomenon attenuation, as a function of the distance source - target and the distances of threshold limits specific for the accident type and/or the involved dangerous substance. The attenuation factor is dimensionless and represents a percentage from the distance.

The computation formulas of the attenuation factor α are the following ones:

Case 1: **The target is located in the lethal effect zone** ($0 < d_i \leq d_{LEZ}$) (fig. 2)

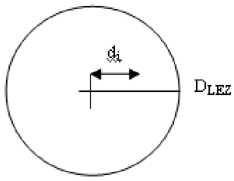

Fig. 2. Advancing degree of the accidental phenomenon in the lethal effect zone

$\alpha = [(d_i - 0) / (d_{LEZ} - 0) \times d_{LEZ} / d_{REZ}) / \alpha_{normalization}$
where: $\alpha_{normalization}$ is defined by $\alpha_{normalization} = (d_{LEZ} + d_{IEZ} + d_{REZ}) / d_{REZ}$.

For this zone, the attenuation factor has the minimum value 0 (for $d_i = 0$), the maximum value being $d_{LEZ} / (d_{LEZ} + d_{IEZ} + d_{REZ})$ pentru $d_i = d_{LEZ}$.

Case 2: The target is located in the irreversible effect zone ($d_{LEZ} < d_i \leq d_{IEZ}$), (fig. 3)

$\alpha = [(d_i - d_{LEZ}) / (d_{IEZ} - d_{LEZ}) \times d_{IEZ} / d_{REZ} + d_{LEZ} / d_{REZ}] / \alpha_{normalization}$

The term d_{LEZ}/d_{REZ} represents the advancement percentage due to the previous zone, namely the lethal effect zone.

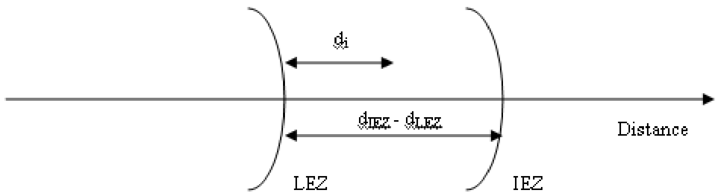

Fig. 3. Advancement degree of the accidental phenomenon in the irreversible effect zone

For this zone, the attenuation factor has the minimum value $d_{LEZ} / (d_{LEZ} + d_{IEZ} + d_{REZ})$, for $d_i = d_{LEZ}$, and the maximum value is given by the expression $(d_{LEZ} + d_{IEZ}) / (d_{LEZ} + d_{IEZI} + d_{REZ})$, for $d_i = d_{IEZ}$.

Case 3: The target is located in the reversible effect zone ($d_{IEZ} < d_i \leq d_{REZ}$), (fig. 4)

$\alpha = [(d_i - d_{IEZ}) \times (d_{REZ} - d_{IEZ}) \times d_{REZ} / d_{REZ} + (d_{LEZ} + d_{IEZ}) / d_{REZ}] / \alpha_{normalizaation}$

The term $(d_{LEZ} + d_{IEZ}) / d_{REZ}$ represents the advancement percentage of the two previous zones, namely the lethal effect zone (LEZ) and the reversible effect zone (REZ).

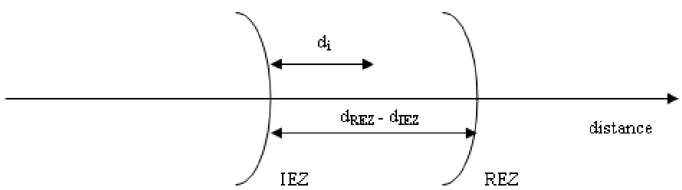

Fig. 4 Advancement degree of the accidental phenomenon in the reversible effect zone

For this zone, the attenuation factor has the minimum value $(d_{LEZ}+d_{IEZ})/(d_{LEZ}+d_{IEZ}+d_{REZ})$, for $d_i = d_{IEZ}$, and the maximum value is 1, for $d_i = d_{REZ}$.

Case 4: $d_i > d_{REZ}$, then α = 1, while the target is located in the no effect zone. Consequently, it can be considered that the accident do not propagates beyond this limit.

The propagation flow index F is the complement of 1 of the attenuation factor and, consequently, it is determined with the relationship:

$F = 1 - \alpha$ [%]

For example, if we consider a major accident having the following effect zones:

- $d_{treshold\ LEZ}$ = 200 m;
- $d_{treshold\ IEZ}$ = 1500 m;
- $d_{treshold\ REZ}$ = 4000 m;

the result, graphically represented in figure 5, is obtained both for the attenuation factor α, and for the propagation flow index F.

The example given in figure 5 indicates that, for each effect zone, the flow index decreases differently, fact that corresponds to the initial hypothesis, according to which it was admitted that the phenomenon attenuation for an effect zone depends on:

- the advancing degree in the in the respective zone;
- the size of the effect zone considered with respect to the global effect zone.

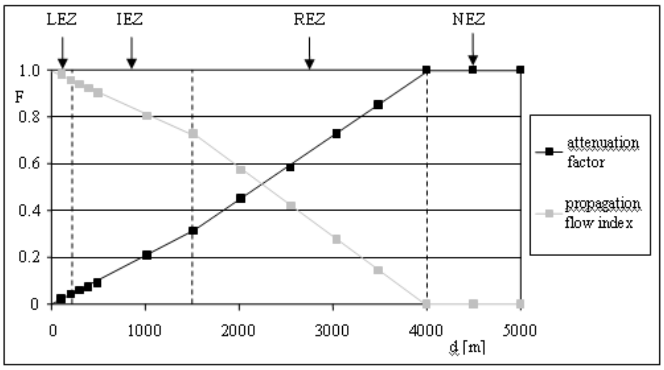

Fig. 5. Example regarding the propagation flow index and attenuation factor evolution

2.2. The phenomena isotropy and anisotropy

The isotropy concept is considered in a spatial manner, and the application rules are outlined bellow:

- for the isotropal phenomena, the effect zones are circular;
- for the anisotropal phenomena, the effect zones are partitioned in sectors.

As an example, for the case of toxic gas dispersion, the anisotropy is considered through the data concerning the wind rose, which allows dividing in sectors the propagation space, on privileged directions. Considering simultaneously the concept of effect threshold limit and the isotropy or anisotropy of the accidental phenomenon, there can be proposed two ways of cutting-out the studied area:

- for the isotropal phenomena, the industrial site's environment is divided in the manner presented in figure 1. The retained effect zone corresponds entirely to the studied area; this zone can be affected by the analyzed major accident. In this case the propagation flow factor will be evaluated for the entire analyzed zone.

- for anisotropal phenomena, the industrial site's environment is cut - out in sectors, as represented in figure 6 [4].

The retained effect zone corresponds to the coupling of the circular area LEZ with the sector or sectors able to be affected from a spatial point of view (triangle marked area for IEZ and REZ).

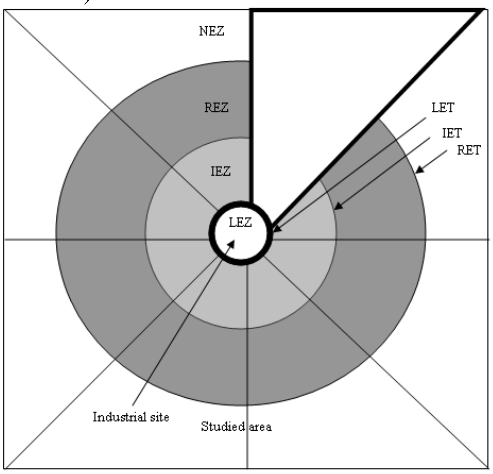

Fig. 6. Example of evaluation area of the propagation flow zone in the case of a anisotropic phenomenon

Inside the lethal effect zone it is difficult to propose a preferential direction, while it is possible, for example in the case of a toxic gas dispersion, to register significant wind direction variations. Consequently, the propagation flow factor will be evaluated for the marked zone.

Anyway, one or more sectors can be considered for the effect zone determination, if this became necessary.

3. CONCLUSIONS

Starting from a literature review and the national and EU legal requirements concerning the prevention and control of major accident hazards involving dangerous substances, the paper has as main goal to present an expeditious methodology for propagation flow index determination.

The following parameters were taken into consideration for the propagation flow determination: the distance from the danger source (distance source - target), the environment's characteristics (the wind rose) and the propagation kinetics.

The propagation flow index F represents the complement of 1 of the attenuation factor α, a variable value which characterizes the accidental phenomenon attenuation, according the distance source - target and the distances of the effect threshold values, specific to the accident type and the involved dangerous substance.

The methodology proposed in the paper, for the propagation flow determination takes into consideration the isotropy and anisotropy of the phenomena involved in the occurrence of a major accident. In order to define the lethal, irreversible, reversible and non - effect zones, there were employed the national legislation requirements and the experience gained in the European Union state members, regarding the urbanization control for the industrial sites with high risk levels.

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DEVELOPING A PARTICIPATIVE MANAGEMENT STRATEGY FOR OCCUPATIONAL HEALTH AND SAFETY RISKS

ROLAND-IOSIF MORARU*
GABRIEL-BUJOR BĂBUȚ*

Abstract: *The principles that underlie a coherent and efficient prevention program for occupational health and safety are outlined in this paper: the need of a global approach of these problems not only at the workplace but for the whole of the living conditions at work; the role of actor of the workers and therefore the absolute necessity of a participative approach; the real usefulness of measurements and of risk quantification in general; the differences between risk assessment and risk management and the specificities of small and medium size enterprises. On the basis of these principles, the various steps and levels of intervention are defined. A dynamic strategy can prove to make it possible to approach the work situations progressively in small as well as in large companies, to coordinate the cooperation between the workers, the technical staff and the occupational health practitioners and to prevent the problems more rapidly, more efficiently and more economically.*

Key words: *prevention, risk management, risk assessment, participation, strategy*

1. INTRODUCTION

Following the Directive 89/391 [1], the various European states, including Romania, had to restructure, sometimes considerably, their legislation concerning the organization of health, safety, well-being at work. In particular, the companies are since required to carry out a risk assessment for all their workplaces. Various methods were developed and proposed to carry out this assessment. Many of them actually draw up only one inventory of the hazardous situations, with some general and usually stereotyped recommendations [8]. It is not of a method, a checklist nor a tool, but a strategy seeking to organize efficiently, economically and durably the efforts of the various protagonists of health and safety at work: the employees, the hierarchy, the occupational physicians, the occupational health and safety (OHS) practitioners and the experts.

* *Assoc. Prof. PhD. Eng. - University of Petroşani*

A dynamic strategy is based on a certain number of fundamental principles presented and discussed in the first part of the paper.

2. BASIC PRINCIPLES FOR THE RISK MANAGEMENT STRATEGY DEVELOPMENT

21. Workplaces and work situations

By "workplace", one generally understands, in a restrictive way, the place and the conditions in which a worker has to perform a stereotyped task [3]. This concept is now out-of-date and, in the new forms of work organization, the work is more changing and the operators work in a group of workplaces, that we will call a "work situation", where they interfere the ones with the others. Moreover, the behavior, satisfaction, quality of work and well-being of any worker do not depend only on the physical or chemical factors of his working environment, but also on the work organization, the responsibilities and the collective relations [2]. The expression "work situation" refers to all the aspects, physical, organizational, psychological, social of the working life, that is likely to have an influence on the health, the behavior and the well-being of the employee.

2.2. Risk factors

Are called risk factors all the aspects of the work situation that have the property or the capacity to cause a damage. These factors can relate to safety, to physiological health or to psychosocial health. The risk in itself is the probability of a damage of a certain severity, taking into account the exposure to the risk factor and the circumstances of this exposure [9].

When a rigorous use of the terms is essential - and thus in discussions between OHS practitioners and in the regulations - the terms of risk factors should be used rather than the terms of danger (referring mainly to the risk factors of safety) or nuisance (used rather for the factors of environment, in the discomfort zone). It appears unrealistic to seek to impose this rigorous terminology in industry but, however, a clarification of what the interlocutors imply by these terms is needed in many occasions. This definition of the terms of risk factors differs from that adopted in medicine, where, for example, cholesterol is called a risk factor for cardiac problems. These individual characteristics (age, gender, weight, personal sensitivity, etc) are thus actually risk co-factors since they increase the risk for a given person [6].

2.3. OHS practitioners and experts

We will designate by OHS practitioners the persons, such as safety officers, occupational nurses, occupational physicians, industrial hygienists, ergonomists…, who received some training in health and safety at work and who developed a particular motivation to recognize, evaluate, prevent and limit the risks. The training and competences of these people can be varying and one will make a distinction between the OHS general practitioners and those more specialized, for example, on musculo-skeletal disorders, occupational hygiene, stress etc.

There are considered experts the people, coming in general from specialized laboratories, who have the competences and the methodological and technical means to look further into a particular problem. In general however, these competences and means are limited to a particular aspect: electricity, toxicology, acoustics, mental effects, stress…

2.4. The small and medium-sized enterprises (SME)

In the western countries, less than 40% of the employees work in companies employing more than 250 people. Usually, in these large companies, a well trained OHS practitioner is present, competences are available, consultation bodies' function rather well, the problems are dealt with and the frequency and severity rates of accidents and occupational diseases are lower by 30 to 50% to those in small and medium-sized enterprises (SME).

The majority of the employees work in SME where the situation is much more variable. In the medium-sized companies, an internal OHS practitioner is sometimes available, but often he remains isolated and appointed part-time to this mission of prevention. In the smallest enterprises, the employer himself is often theoretically in charge of this mission.

These external OHS practitioners are or should be general OHS practitioners, since they are confronted, here with a safety problem in a garage, there with an occupational disease in a dry cleaning shop, or still with a stress problem in an office. They have in general at their disposal basic material for ordinary measurements. The methods to be developed must therefore be addressed in priority at these SME, by taking account of the limited means and competences that are there available.

2.5. Available quantifications

Knowledge from what really occurs in the work situation is decreasing from the employee to the expert. Quite realistically, the situation can be described as follows in the majority of the cases:

- the employee knows what he does and what he lives everyday (real work);
- the foreman and even more so the management of the company know what the employee is supposed to do (prescribed work) and believe to know what he lives;
- the internal OHS practitioner knows what he has time to study;
- the occupational physician knows what he asked and what he heard (complaints) on the occasion of the periodic medical checkups with the employee and what he sees, feels, hears (noise) when he visits the company;
- the external OHS practitioner called for a specific problem knows what one told him and, again, what it sees, feels, hears during the 2 hours - 2 days when he stays in the company;
- the expert knows only what interests him for the specific problem for what he was called in.

On the other hand, qualification in health and safety increases in the opposite direction.

- employees, foremen, direction… are not or are little aware of the risks they incur, depending upon the health education they received or acquired;
- the internal OHS practitioner, depending upon the training he received, knows the main legal requirements and the general principles of prevention;
- the external OHS practitioners have, the ones a rather general qualification, the others more specialized competencies;

- the experts are specialized in a certain field and very often unaware of the others.

It thus appears logical to consider that the two sets of knowledge - about the work situation and about the principles of the well-being - are complementary. Remain to organize this cooperation in an interdisciplinary way.

2.6. The employer's main role

It is understood that many studies undertaken by an OHS practitioner or an external expert, the day that is convenient for him, on a specific problem not put in its context, have very little effects, or even a negative effect due to the missed opportunity for a more coherent action.

The employee must thus be the main actor - and not only the object - of prevention and must be regarded as such by all the OHS practitioners or others. This means that participation - and not only consulting - of the employees is indispensable. This is only possible if the qualification of the employees concerning their work situation, and their integrity is explicitly recognized [4].

2.7. The global character of the problems

20 to 30 years ago in the western countries, the priority was to avoid very handicapping accidents and occupational diseases: specific actions on specific factors (electricity, falls, fire, chemical agents…) had to be taken rapidly to reduce the hazardous exposures. Since then, industrial disastrous situations have been eliminated and a particular problem can no longer be isolated and solved independently of the context.

The employee "sees" his work situation like a whole and not like a set of distinct and independent facts: he is "being well" or not, he likes is job globally or not. In addition, all aspects of the work situation are inter-related: the noise influences the relations between the people; the technical organization between workstations influences the risks for musculoskeletal disorders; the division of the responsibilities influences the work content, the accidents, etc

The employees can hardy understand and furthermore actively cooperate to a prevention program focused on a specific aspect, while other aspects, more significant for them are ignored. It is also the reason why, training programs on manual handling or actions on the stress conditions are doomed to fail when they are not preceded or accompanied with a revision of the machines, of the work organization, with noise abatement, etc.

2.8. The risk analysis stage

The number of methods aiming at 'assessing' the risks is definitely greater than the number of methods aiming at "preventing" them and these methods relate to generally only one factor of particular risk. Most of them were developed by experts (as we defined them) whose responsibility and interests are mainly to establish the dose-response relationships, rather than to solve a particular problem in a particular work situation. That is particularly obvious in the case of the environment factors: evaluation of the concentration of a pollutant in the air, of the personal exposure to noise, of the exposure to heat. Extremely sophisticated methods were published to this end [5, 7, 10]. They are little used and, most of the time, are misused, because difficult, heavy and costly.

From these methods and handbooks, it should be concluded that the representative and correct quantification of the exposure to any risk factor is very difficult and expensive and that the many measurements or quantitative evaluation do have little or not value. It is thus necessary to draw the attention of the OHS practitioners who measure systematically and of the employers who require these evaluations, on the real interest of these measurements, their validity, and their cost and to encourage them to quantify better and more validly but more advisedly and for explicit prevention objectives [11]

It is thus necessary to discourage the systematic and at first quantification, which is likely to distract from the first goal, prevention. In each case, it is up to the OHS practitioner to determine if he must or not conduct a quantification of the risks and the reasons (epidemiologic, technical, political…) for which he must conduct it.

2.9. Risk assessment vs risk management

This tendency for systematic quantification also exists concerning the risk of accident. Methods are used to classify the risks and to define priorities for actions – what is certainly very desirable - but often by neglecting the analysis of the elements defining these risks, the reasons and the means of improving the situation.

The prevention approach consists in seeking the most effective means to reduce the risk, by acting on one or several of its components: elimination of the risk factor, reduction of the exposure, increase of the reliability of the work system… It is thus essential that the analysis of the risk be not simply a recording of its components, but consist in a careful analysis of the reasons of the exposure, the circumstances of this exposure, the severity of the consequences and the most relevant and reasonably practicable means to reduce them.

The final quantitative evaluation of the risk is consequently secondary, the most important thing being to study the components and the details on which it is going to be possible to act. Rather than speaking about risk assessment, it is thus more appropriate to speak about risk management.

3. DYNAMIC MANAGEMENT: THE VARIOUS STEPS OF INTERVENTION

Although all the problems are dependant, it is neither realistic nor possible to solve all of them at once. Considering the Swiss-cheese accident model (see figure 1) the first stage consists, for example, in replacing a defective tool, leveling the ground, improving the ventilation system or modifying a hierarchical relation.

Essential step, it is not sufficient because the reasons for which the tool was defective, the ventilation was degraded, the hierarchical relation was aggressive… did not disappear and the situation will return soon or later towards the initial state.

The second stage can consist in reexamining the general work organization, the institutional links between people or in rearranging the operating area. Perhaps a third stage will relate to the workers training: vocational training to perform the tasks, education to their well-being, leading them to recognize themselves the problems, to manage them directly as they arise, bringing the employees to a degree of self-management of their health, safety and well-being to work. Maybe the next stage will relate to the culture of the company, the integration of the concerns of well-being in the overall management of the company.

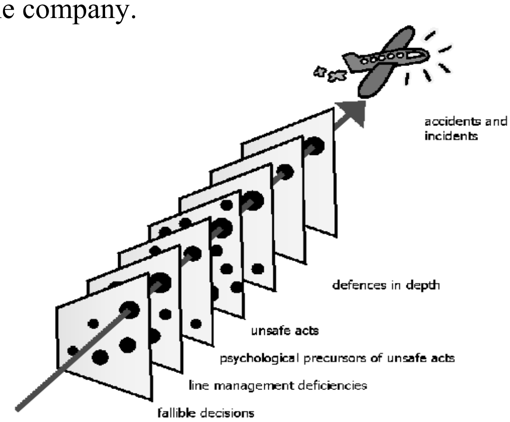

Fig. 1. The Swiss cheese accident model

The amplitude of each stage will vary as a function of numerous parameters and the number of steps will be infinite, as, at any step, the risk is great to regress to a state of carelessness and improvisation.

The knowledge, information and data necessary during the first steps relate primarily to the work situation: the tools used, the machines circulating, the chemical products to exhaust, etc. Knowledge in ergonomics, medicine and safety is certainly desirable to select the good tool or to ventilate more effectively, but is less essential than the knowledge of the work situation day after day. This first step must therefore be carried out as close as possible of the work situation and its output will be especially a function of intimate knowledge of what occurs in the course of time in this work situation.

Conversely, at a more advanced step of the process, the problems require more qualification in work organization, training, management of the relations…. The analysis must be finer, more specific and requires tools and competences that only OHS practitioners generally have. According to the step, the necessary competences will thus rather be those of an OHS practitioner or of the workers themselves, these remaining the main actors of the prevention, for whom and by whom prevention is implemented.

4. CONCLUSIONS

Various competences being complementary and necessary at the various stages of the risk management process, a 'strategy' is essential to coordinate these various partners and to use advisedly their competences and resources. The use of the any systematic strategy to coordinate the actions in health, safety and well-being at work must not be improvised. As discussed already, the optimal state of health, safety and well-being for the workers and of physical and economic health for the company cannot be reached in once. It is necessary finally to insist on the fact that this OHS practitioner is indeed a *facilitator* and not the person *responsible* for the participative process and his success. His role remains external: to smooth things over, to train the people so that they assume the full responsibility for the process, to make so that the partners deal with themselves gradually and manage *their* problems jointly. If this OHS practitioner is the facilitator of the participation within the work situation, he is also the facilitator of the relationship between the company and external consulting OHS services in order to insure the coherence of the external interventions. He becomes therefore the coordinator of the external interventions, leaving to the external OHS practitioners the task to provide the specialized technical assistances. He thus holds a 'hinge' position, supervising the evolution of the company and ensuring the recourse to the external assistances when necessary. The objective of the participative approach is, indeed, as said, not to do without the OHS practitioners but to utilize them advisedly and more effectively.

An in-depth analysis of the dynamic policy of risk management results in proposing a strategy whose purpose is to gradually approach the work situations in the small as well as in the large companies, to coordinate the cooperation between employees, management, internal and external OHS practitioners and to arrive faster and less expensively to effective prevention. Tools are proposed to implement the strategy. The employer remains fully responsible for the implementation of the policy. This implementation however requires the intervention of an OHS practitioner, oiling the wheels of the participative process.

In systems of health and well-being at work such as those in use in Belgium and France, the OHS practitioner-facilitator is in all probability the occupational physician, who remains the only one in contact with the SME and the only one to enjoy there some moral capacity of influence. Concurrently to his surveillance mission of the individual health of the employees, he is thus invited to play the role of coordinator of the collective and individual actions of prevention, being interested in the coherence and effectiveness of the interventions and leaving to more specialized OHS practitioners the technical details of these interventions.

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Scientific Reviewers:
Prof. PhD. Eng. Vasile Oros

INTOXICATION RISK GENERATED BY THE NATURAL GAS'S ROAST GAS

PAVEL GHEORGHE[*]

Abstract: *Due to improper manipulation of devices that consume gas fuels or to improvisations of the natural gas usage installations, many intoxications with natural gas roast gas and even deceases produce every year. Together with the cold season, the risk of these incidents producing increases, due to the inappropriate usage of the devices that consume gas fuels (cooking machines, stoves, heating devices, etc.) and due to unauthorized modifications of the natural gas usage installations. In this paper, the author proposes to present in detail the way in which the human organism is affected by the roast gas that can accumulate in spaces where devices that use gas fuels are installed.*

Key words: *oxygen, intoxication, carbon monoxide, carbon dioxide*

1. GENERALITIES

Breathing is a complex physiological process that involves three different functional systems: the respiratory system, the cardiovascular system and the nervous system.

The quantity of air that passes through the lungs in one minute represents the respiratory debit, which has a value of 5-10 liters/minute during repose, and reaches values between 120 and 150 liters/minute during physical effort. Let us not forget that there are great differences when it comes to the way in which different persons adapt to effort, and training and stress have an important role.

Dysfunctional breathing can provoke severe or even deadly accidents. In this context, it is very dangerous for people to go in environments that contain toxic gas or with low oxygen content, if they do so without proper protection.

The harmfulness degree for the respiratory system is determined by a series of factors that refer to the quantity and active concentration of gas, to the cumulative action in the organism, to the exposure time, to the temperature of the working environment, to physiological particularities.

In certain conditions, a person can live in an atmosphere that contains less oxygen then normally.

[*] *Ph.D. candidate Eng - Emergency Situations Specialist E.ON Gas Distribution*

It is important to underline that the person cannot sense the decrease of the oxygen concentration, because the breathing is not regulated by the oxygen, but by the carbon dioxide, which is a stimulus for the respiratory nervous system.

The symptoms that appear with the decrease of the oxygen concentration in the air inhaled:

- between 21 and 18% O_2 – normal breathing with slight fatigue signs;

- between 18 and 15%O_2 – faster yet heavier breathing, the volume of air inhaled increases, the pulse accelerates, the concentration capacity decreases, breathing dysfunctions can appear with physical effort;

- between 15 and 10%O_2 – heavy breathing, sickness symptoms, fatigue and exhaustion, the person is aware (conscious) but cannot make decisions and cannot perform physical activities;

- between 10 and 8%O_2 – unconsciousness, dizziness, nausea, the persons leans down to the ground, cannot walk or starts to drag, fainting intervenes, and even if resuscitation is possible, permanent lesions can remain on the brain;

- under 8%O_2 – members paralysis, loss of consciousness, convulsions, breathing stops but the heart continues to beat for a few more minutes and after that, the victim dies.

2. CARBON MONOXIDE INTOXICATION (CO_2)

Carbon dioxide is a gas without color, without odor, without taste, it does not support life, it does not burn (burn-proof) and it is heavier then air (it flows down to the ground, especially in rooms without ventilation (sewers, basements, trenches, holes, closed rooms or spaces).

Carbon dioxide is produced through the complete burning of combustible materials and substances, following fires, following the usage of natural gas for household activities, and, in small quantities, through the breathing of all living creatures.

The carbon dioxide excess in the blood is defined in medical terms as „hypercapnia" and is caused by the increase of the carbon dioxide quantity in the breathable mixture.

The increase of the carbon dioxide level leads not only to the increase of the carbon dioxide concentration in the blood, but also to a decrease of the oxygen concentration in the blood and cells and it is felt especially during intense physical activity (the „lack of air" sensation appears, which leads to the shortening of the inhalation, to the acceleration of the breathing rhythm (panting), to fatigue, to muscular cramps, to head aches and finally to the loss of consciousness and death).

A person sitting in an environment with a carbon dioxide excess feels a certain breathing discomfort, fatigue and the need to inhale a larger quantity of air. It is recommended for that person to interrupt all activities, to relax and breathe deeply, to prevent or to slow down the normal gas exchange in the lungs.

A person who inhaled smaller quantities of carbon dioxide suffers from head aches, nausea with or without vomit, blurry vision, heavy breathing. If the gas concentration in the air is very high, that person can loose conscience in a matter of minutes, falls down to the ground and can die quickly.

It is very important to know that the person is not aware of the intoxication and cannot take measures to save himself/herself.

When someone goes alone inside an environment with carbon dioxide and gets intoxicated, they can loose their lives, since there is nobody around to save them.

In the presence of several persons, the person intoxicated with carbon dioxide can be saved, if in the first minutes of unconsciousness that person is taken away from the toxic environment, to fresh air.

The victim will be laid down on the back with the head turned to one side, the collar will be unbuttoned and the belt will be unbuckled and if there is no pulse, cardio and respiratory reanimation is to be performed.

Oxygen breathing is life saving if there is an oxygen tank with pressure reduction around.

Attention! The rescuer must know that by entering the toxic environment (room, basement, sewer, etc.), without protection, he/she will also intoxicate. To avoid that, he/she will wear a gas mask or a mask foreseen with a hose through which to breathe air from outside the gas contaminated environment. Lacking these, the rescuer will be tied with a rope around the body or by the belt and will be pulled out as soon as he/she losses conscience (there were cases when several persons died trying to help each other).

In general, the human body supports a concentration of up to 4% carbon dioxide in the inhaled air.

Symptoms that appear when the carbon dioxide concentration in the air inhaled increases:

- up to 0,5%CO_2 – it is safe to work, 8 hours/day, the ventilation of the lungs increases by up to 5%.

- between 0,5 and 2%CO_2 – a sensation of warmth and moisture appears, lack of attention to details, fatigue, anxiety, energy loss and clumsiness, knee weakness feeling (lack of leg power), increased breathing rhythm and amplitude, head aches appear and an extended exposure can affect the body's functions even after the person leaves the infested area (a person can require a few days in a normal environment for the metabolic functions of the body to restore to normal), the ventilation of the lungs increases by up to 50%.

- between 2 and 4%CO_2 – panting appears after expiration, severe head aches, dizziness and possible blurry vision, (dyspnoea), a state of agitation, the sensation of chest pressure, the frequency of the pulse increases and the breathing accelerates, the ventilation of the lungs increases by 100%.

- between 4 and 10%CO_2 – breathing becomes violent, fatigue leads to exhaustion, the head aches are very strong, dyspnoea accentuates, the prolonged exposure to a 5-6% concentration can lead to irreversible health effects or even death, and a prolonged exposure to a concentration higher then 8% can lead to unconsciousness and death.

- between 10 and 15%CO_2 – breathing becomes very heavy and the expiration very fast, head aches are very strong, exposure for even as little as a few minutes will lead to unconsciousness, suffocation without warning and death.

- over 15%CO_2 – comatose states, convulsions and death in a few seconds, exposure to a concentration higher then 20% leads to instant death.

3. CARBON MONOXIDE INTOXICATION (CO)

Carbon monoxide is a very dangerous poisonous gas, because it is unperceivable, it is without color (invisible), without smell, without taste and slightly diffusible (having a density of approximately the same as the air density: d=0.967, the dispersion – spreading – is done in the entire atmosphere of the room).

Carbon monoxide is also known as the „perfect asphyxiation agent" due to its super fast intoxication, followed by asphyxiation, and due to the impossibility to detect it.

Carbon monoxide forms following the incomplete burning of the natural gas, practically of the methane (natural gas is composed of approximately 99% methane), incomplete burning caused by the insufficient oxygen in the atmosphere.

When natural gas is used, in general by the household consumers, the formation of toxic carbon monoxide concentrations is caused by:

- Lack of total evacuation of burned gas from broken stoves (with inappropriate draught),
- Lack of total evacuation of the burned gas from closed spaces, where devices that use gas or burners with open flame function (for example: cooking machines),
- Over-heating of the metallic stoves or of the terracotta stoves with cast-iron cooking plate, in closed spaces or rooms, without ventilation;

Carbon monoxide intoxications can appear in cases such as:

- Household accidents (most often),
- Professional accidents (rarely, but more severe due to the increased number of victims),
- Suicide (very rare),
- Homicide (extremely rare).

Toxic-kinetic and toxic-dynamic effects of CO:

CO goes into the organism via the respiratory ways without having an irritating action, it attaches to the hemoglobin (Hb), practically replacing the oxygen attached to the Hg, the oxygen does not reach the tissues and internal organs of the human body (heart, brain, etc.), causing them to malfunction (anoxia or hypoxia); a small amount of CO dissolves into the plasma.

The reversible chemical reaction that takes place: $HbO_2 + CO \leftrightarrows HbCO + O_2$

Hemoglobin (Hg) is a substance that represents the colored matter of the red blood corpuscles, with the role to spread the inhaled oxygen through the body ($Hb + O_2 \rightarrow HgO_2$: oxy-hemoglobin).

CO attachment to the Hb leads to the formation of carboxy-hemoglobin (HbCO), which affects the Hb's capacity to attach and transport oxygen (the so-called transport anoxia).

Hb's affinity for CO is of 200-250 times higher then for oxygen, so even reduced CO concentrations are very harmful.

CO acts as an inhibitor and disturbs the oxidation processes at the muscular level, leading to pronounced muscular hypo-tonicity.

Practically, the intoxication with CO leads to lack of oxygen at cellular level.

By breathing in a normal atmosphere, CO eliminates very slow. The accelerated elimination is done though oxygen therapy with normal or hyper-bar oxygen.

The toxic effect of CO depends on its concentration in the air inhaled:

- at a concentration > 0.01%, first harmful signs appear,
- at a concentration > 0.10% it is already dangerous,
- at a concentration > 0.20% it is very dangerous and leads to death.

Normally, the intoxication degree is determined by the HbCO concentration in the total Hb of the blood, or by the quantity of HbO_2 transformed in HbCO.

The lethal concentration of HbCO begins from 65% and is given by:

- 0.15% CO in the air inhaled during 3 hours,
- 0.20% CO in the air inhaled during 1-2 hours,
- 0.40% CO in the air inhaled during less then 1 hour.

<u>The clinical picture</u> depends on the concentration of CO in the volume of air inhaled and on the concentration of HbCO in the blood.

The HbCO concentration in the blood of up to 5% is considered acceptable (5-8% is the HbCO concentration in the blood of a smoker, and if this is associated with alcohol consumption or other sedatives, it can lead to disorders of the cerebral circulation).

A normal environment is considered to have a concentration of CO of up to 0.01% Vol. (or 1 mg/cm).

A classification of the intoxications can be done as follows:

a. minor intoxication – when the CO concentration in the air is of up to 0.1%, and HbCO of up to 15% and manifests through minor symptoms, „pseudo-influenza": head aches, fever, stomach aches, nausea, vomit, drowsiness, fatigue, confusion.

b. advanced/acute intoxication – when the CO concentration in the air is between 0.1 and 0.4%, and:

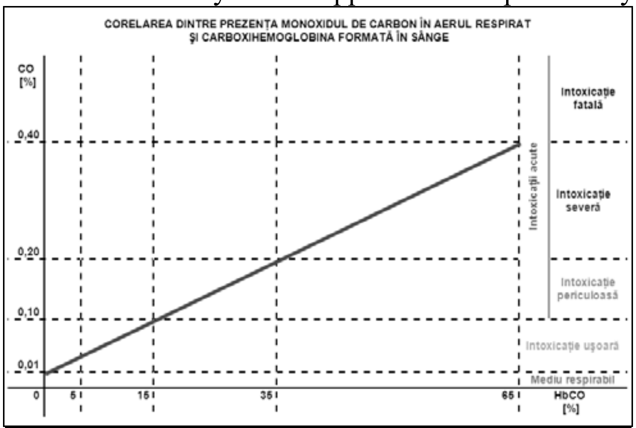
➡ for HbCO between 15 and 35%, a **dangerous acute intoxication** manifests: cephalalgia, dizziness, rapid pulse and cardio rhythm, weakened eye sight, chest pains, maniac excitation, euphoria, it can be confused with alcohol intoxication.

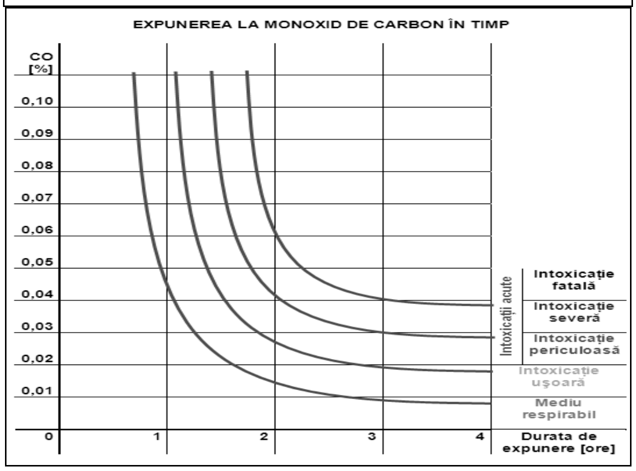
➡ for HbCO between 35-65%, a **severe acute intoxication** manifests: violent cephalalgia, superficial, irregular breathing, pronounced muscular weakness – the patient is aware of the danger, but cannot move because of muscular hypo-tonicity, comatose state, intermittent convulsions, the teguments have a rose-cherry red color.

➡ for HbCO of over 65%, a **fatal acute intoxication** manifests: death, caused by failure of vital organs.

c. over-acute intoxication (instantaneous) – when the CO concentration in the air is higher then 1.0% and HbCO is of over 65% and manifest through: rapid loss of consciousness, convulsions, death in a matter of minutes.

d. chronicle intoxication appears after a few days or weeks, with persons that live in toxic environments, who have a constant quantity of HbCO in their blood, and manifest through: nervous disorders, cardio disorders, acoustic disorders, eye disorders, digestive disorders, difficulties to memorize and even memory loss, personality disorders and modification, disorientation, decrease or loss of hearing. It must be remembered that secondary affects appear after 2 up to 40 days.





<u>Therapy in case of intoxications with CO:</u>

The victims must be urgently removed from the toxic environment and must be consulted by a doctor;

The special treatment consists of administering oxygen through oxygen therapy, as follows:

- Administering 100% oxygen at normal pressure, with the help of a mask, through normal breathing (physiological) or intubations through artificial breathing;
- Administering 100% oxygen hyper-bar; in isolated rooms pure oxygen is introduced at high pressure to rapidly reduce the CO level in the blood and to re-establish the oxygen transport capacity at cellular level.

The CO test is performed by the doctor, to determine the HbCO level in the blood (the quantity of CO in the air inhaled can also be measured with the help of an analyzer).

To remember: Oxygen therapy must not be delayed by medical tests and analyses.

CO elimination from the organism is done:

- very slow in a normal environment, in 12-24 hours;
- in approximately 4 hours in a 100% oxygen environment;
- in approximately 1 hour in a hyper-bar oxygen environment.

The harmful effects of the carbon monoxide are determined by:

- The CO concentration in the inhaled air, in the contaminated space,
- The quantity of CO inhaled,
- The duration of the exposure to CO,
- Altitude (at high altitude, the air contains a smaller quantity of oxygen!),
- Intensity of the physical effort in the contaminated space,
- Physical build, gender (small children, pregnant women, old persons, smokers, anemic persons and the persons who have other problems – cardio problems for example, are more easily affected and present more severe symptoms).

Measures to be taken in case of carbon monoxide intoxication:

- The CO concentration of the space that will be entered is measured, without entering that space;
- The intoxicated persons are urgently removed from the toxic environment and taken to fresh air;
- All persons in the room or in the building are announced to leave the contaminated area;
- The emergency medical unit is immediately announced.

If during the treatment week after the carbon monoxide intoxication, eye sight modifications, movement modifications or behavior modifications appear, the doctor must be urgently announced.

The evaluation of the carbon monoxide intoxication symptoms, delayed or lasting, is done by the neurologist and by the psychologist.

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Scientific Reviewers:
Assoc. Prof. PhD. Eng. Roland Moraru

DANGER OF EXPLOSION AT THE PARKS OF TANKS FOR LIQUID FUEL

SORESCU FLORIN - MARIN*

Abstract: *Areas having potentially danger of explosion are the areas where an explosive atmosphere due to mixtures of steam/air may occur in such quantities, as there are needed special protection to maintain the employee safety and health.*

1. AREAS WITH POTENTIALLY DANGER OF EXPLOSION AND/OR FIRE AT TANKS PARKS

1.1. Areas with potentially danger explosion at liquids fuel tanks

A *potentially danger of explosion area* means that space where, under normal operating conditions can accumulate permanent or accidental gases, fumes or dust, powders in sufficient quantities to generate an explosive atmosphere in a mixture with air or oxygen [2, 3].

In accordance with German Technical Norm for Flammable Liquids T.R.b.F 20" since 2001 [11], Annex 1 of HG1058/2006, on „Minimum Requirements for Improving Safety and Health Protection of Workers who May Be Exposed to potentially Risk Due to Explosive Atmospheres [7]" and art. 2.19 in NP 099 - 04 [16], the areas with explosion hazard are classified according to the frequency and duration of occurrence of explosive atmosphere in three zones:

a)**Zone 0** - areas in which there are permanently explosive atmosphere, consisting in mixture of air with gas fumes or flammable foggs, for longtime or frequently periods (e.g. the tanks, the apparatus of tubes and pipes in the plant);

b) **Zone 1** - areas in which the explosive atmosphere, consisting in of mixture of air with gas fumes or flammable foggs occur occasionally (e.g., in the neighberhood of the loading intakes of zone 0, the neighboring opened joints, vats retention , around the pumps areas);

c)**Zone 2** - areas that are not expected to an explosive atmosphere consisting of air mixed with gas, flammable vapors with foggs, but if it still appears most likely only

* *Student PhD. Eng. - University of Petroşani*

rarely and briefly (e.g. area around zones 0 and 1 and around demountable joints of pipes) according fig.1.1 and fig.1.2.

Areas with danger of explosion can be limited by [11]:

•special construction measures;

•ventilation technique;

•constructive measures, in light of land which restricts the spread of mixtures air / steam, flammable or explosive.

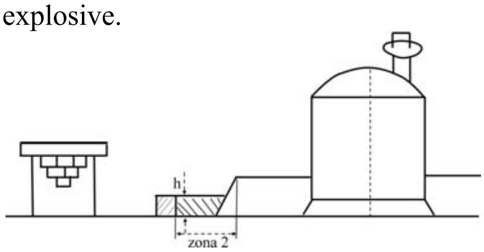

Fig. 1.1 Limitation by the wall areas with danger of explosion

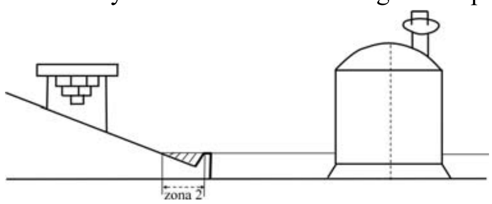

Fig.1.2 Limiting danger of explosion areas by slope of the land

Inside pipes canals, fittings and parts plants in operation that are not always filled with liquid there are dangerous of explosion zones. Around outdoors pumps a radius Ra area, measured from the pump casing is zone 1, zone 2 is the pump, the pump during operation that are not permanent filled is an area with danger of explosion. If inside a tank insulated to the atmosphere, which would lead to a breathing tank, shall ensure a sufficient supply of inert gas (CO_2), so the tank can not create an explosive atmosphere than the inside tank area is not a danger of explosion. Around the air exhausting devices of tanks, the zone 1 is the area limited by a cylinder shape with a radius R from 1 to 1.5 meters.

In the case of tanks with a floating cap, zone extending to a distance of 1.5 meters around the tank's mantle, up to a height of 1m above the upper edge of the tank's mantle area is a zone 1.

Around openings in the vapor space of the tank, which need to be opened to take laboratory samples there are distinguished the following [7, 11, and 16]:

♦ area of up to 3 meter is zone 1, while the inner space tank is classified in zone 0;

♦ area of up to 3 meter is the zone 2, while the inner space tank is classified as zone 1 or zone 2;

♦ area of up to 3 meter is the zone 2 where inert tanks.

Maximum flow [m^3/h] Temperature of ignition [^oC] R [m]

Table 1.1 Areas with potentially danger of explosion

Maximum flow [m^3/h]	Ignition Temperature [^oC]	R [m]
60	" < 0	2
	0 ≤ " < 21	1
	21 ≤ " < 35	0,5
	35 ≤ " ≤ 55	0,5
180	" < 0	3,
	0 ≤ " < 21	1,5
	21 ≤ " < 35	1
	35 ≤ " ≤ 55	0,5
450	" < 0	5
	0 ≤ " < 21	2,5
	21 ≤ " < 35	1,5
	35 ≤ " ≤ 55	1
900	" < 0	7
	0 ≤ " < 21	3,5
	21 ≤ " < 35	2
	35 ≤ " ≤ 55	1
1350	" < 0	8,5
	0 ≤ " < 21	4,5
	21 ≤ " < 35	2,5
	35 ≤ " ≤ 55	1,5
1800	" < 0	10
	0 ≤ " < 21	5
	21 ≤ " < 35	2,5
	35 ≤ " ≤ 55	1,5
2400	" < 0	12
	0 ≤ " < 21	6
	21 ≤ " < 35	3
	35 ≤ " ≤ 55	2
3000	" < 0	14
	0 ≤ " < 21	7
	21 ≤ " < 35	3,5
	35 ≤ " ≤ 55	2

At the tanks with floating cap, the area extending to a distance of 4.5 meters around the mantle of the tank to 1meter height above the upper mantle edge is the zone 2 conformly to fig 1.2 and fig.1.3.

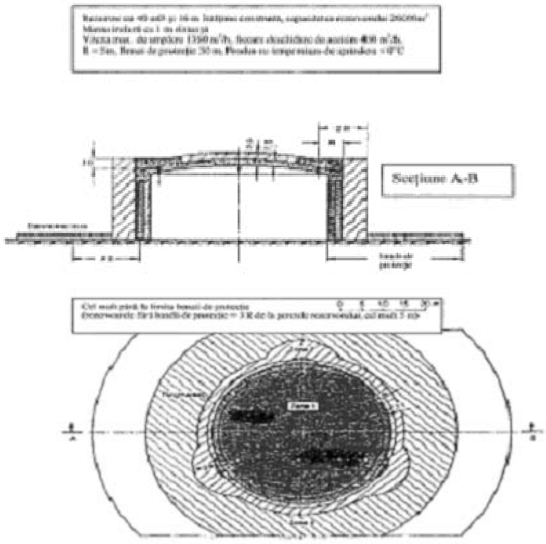

Fig.1.3 Areas with potentially danger of explosion for stationary tanks

The outside retention area bounded by a height of 0.8 m from ground level and a distance of 3R from the retention vat is zone 2.

If the tanks are filled and emptied using the technique of gas oscillation, whose breath is diverted through - an oscillation gas system, are not fitted with airing in the atmosphere, the area covered by the h = 1m around tank is zone 2, and vats retention area are zone 2 to h = 0.8 m according to fig.1.4.

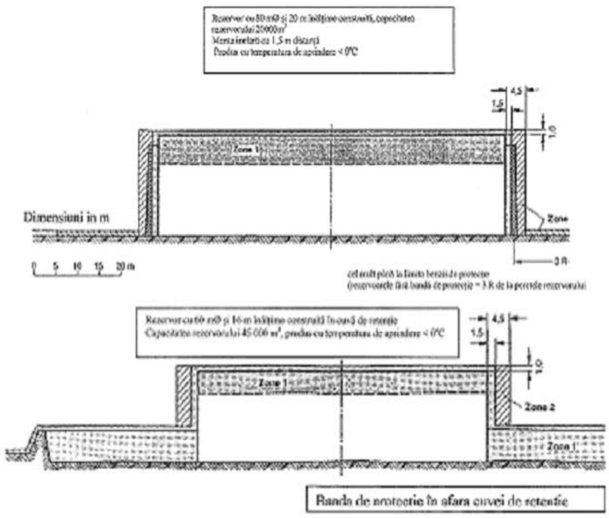

Fig.1.4 Areas with danger of explosion from the tanks with a floating cap

Around tanks where the occurrence of explosive atmosphere is prevented by inertization it is not a hazard explosion zone, according fig.1.5.

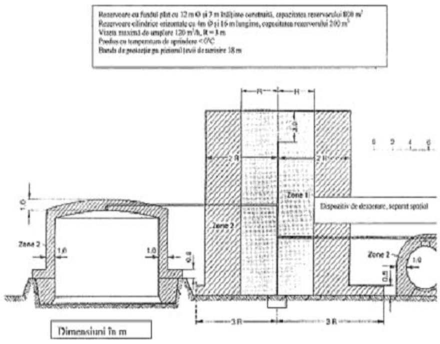

Fig.1.5 Areas with potentially danger of explosion at oscillation tanks

2. AREAS WITH POTENTIALLY DANGER OF EXPLOSION MIXTURE EXHAUST STEAM/AIR

Areas with potentially danger of explosion in admission pipelines to the recovery and purification of polluted air (exhaust) installation may be an area associated with a low risk if protection measures are taken to reduce the risk (probability reduction) to create an atmosphere explosive in the pipeline [11]:

a)reducing the concentration of operational inflammable steam exhausted air through feeding clean air to a value of 50% below the lower limit of explosion;

b) increasing operational concentration of inflammable steam in discharged air through feeding inflammable gas or steam(s) to a value above the lower limit of explosion;

c)inertizing enough so that one area of the existing danger of explosion (Zone 0) reaches an area with high risk of lower (zone1) or without danger of explosion.

A inertizing is considered sufficient if the facility in a tank or a pipeline to pipe usually occur operational 50% of the allowable limit for the concentration of oxygen determined by "BGR 104-Rules of safety and health rules and work to prevent explosions (EX - RL) of the Professional Associations in the Working Sector" [1].

Mixed steam/air removed from the tank must be discharged as to not result in hazards to employees by means of safety in situations:

- at the filling a tank with flammable liquids;
- by respiration due to heating by sunlight;
- the introduction of other media in the reservoir (air, water, water vapor, inert gas).

Steam – air mixtures may be:

o discharged without danger in the free atmosphere through vent pipes;
o emptying (transferring) in another tank (storage) where is bottled;
o drived in a purification exhaust air (bad) or recovery;
o wiped out without danger by burning (by ignition).

Around the pumps, in particular pumps with magnetic coupling are omitted areas with danger of explosion. Thus, around the pumps in the open air, over a distance radius Ra, measured from the wall is the pump casing 1, the area around the pump Ra distance is an zone 2. Pump hole is inside zone 1. Inside the pump hole around the area covered by a distance of around 2m in opening up to h = 0.8 m above the ground is the zone 2 according fig.1.6.

If the pumps which power air cooled engine training is directed to the pump and is placed in depth (holes) no more than 1.5 m deep, the whole area is zone 2. Around pumps located in rooms without special ventilation requirements, radius range Ri for a distance measured from the pump casing wall is zone 1. In this context radius contents to a remote area of 2Ri is zone 2 [11].

In conclusion, the pumps in operation which are not permanently filled with liquid fuel are the hazard of explosion.

3. SAFEGUARD MEASURES AT PARK TANKS

In areas with danger of explosion and/or fire will take protective measures to reduce the risk of ignition of hazardous atmosphere or limit the effects of an explosion in the size of non-hazardous such as [11]:

In Zone 2: sources of ignition, operation that can occur during normal operation: natural ignition sources (solar heat), sources of self-ignition of chemical, physico-chemical, exothermic chemical reactions.

In Zone 1: Ignition sources and those that can often occur through failure: sources of ignition, flame, ignition sources such as electricity (static electricity).

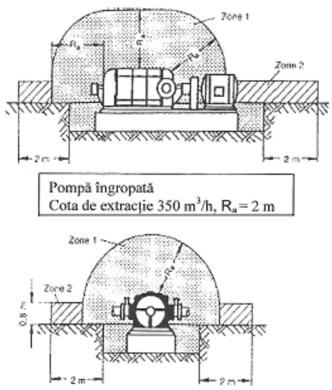

Fig.1.6 Areas with potentially danger of explosion around the grave located pumps

In the zone 0: Ignition sources and those that may occur in very rare failure: indirect sources of ignition (radiation of an outbreak of fire).

Devices, means of production facilities and the facilities that are used in areas with danger of explosion and/or fire can be put into service only if they have been verified and do not produce sparks clash.

According to art.37 of Order 163/27.02.2007, buildings and installations should be done so throughout life in case of fire initiation to ensure [14]:

a)estimating the stability of the bearing elements for a specified period;

b) limiting the emergence and the spread of fire and smog inside the building;

c)limiting the spread of fire from neighborhood;

d) the possibility of users to move safely or be saved by other means;

e)the security forces and means;

Places where explosive atmospheres may arise under Article 5, 6,9,10 of GD 1058/2006 are as follows [7]:

• a place where explosive atmospheres may occur in concentrations so high as to require special precautions to protect the health and safety is considered dangerous within the meaning of the decision;

• a place where no explosive atmospheres may occur in concentrations so high as to require special precautions to protect the health and safety of workers involved is considered devoid of danger within the meaning of the decision;

• flammable substances and/or combustible materials are considered that may form explosive atmospheres, except - if it is an analysis of their properties reveals that in contact with air can not propagate independently explosion;

4. PROTECTIVE MEASURES AGAINST EXPLOSIONS

According to HG1058/2006 are established in Annex 2 the following minimum requirements for improving health and safety protection for workers in the potential danger of explosion such [4, 7, and 16]:

• any leakage and / or intentional loss or flammable gas, vapor or mist that can produce fuel for explosion hazards, or be removed properly diverted to a safe place or, if possible, be stopped in a safe or should be remedied other appropriate methods;

• if an explosive atmosphere contains several types of gas, flammable vapors or mist, protective measures must be appropriate to the greatest possible danger;

• Prevention of ignition hazards, according to article 6 of the decision must take into account electrostatic charges, where workers or the working environment act as carriers or manufacturers of electrical load;

• plant, equipment, protective systems and associated connecting devices must be put into service only if the explosion protection document (emergency plan) allows their safe use in explosive atmospheres. This also applies to work equipment and associated connecting devices which are not regarded as equipment or protective systems in accordance with GD 752/2004 [8], which transposes Directive 94/9/CEE;

• must take all necessary measures for the workplace, working equipment and all associated log devices, available to workers, to be designed, constructed, assembled, installed, maintained and operated so as to minimize the danger of explosion or expansion in the job and/or work equipment;

• in the perimeter of the explosion, workers have warned the optical signals and/or acoustic and withdrawn before they reach the explosion;

• shall be provided in the explosion protection document the rule to ensure maintenance and safety to make it possible in case of danger, leaving the workers quickly and security sites in danger;

• before a job where explosive atmospheres may be used, safety must be checked against the global explosion of persons competent in the protection against explosions having training in the field;

• if necessary risk assessment should be taken the following measures:

a)it should be possible to maintain equipment and protective systems in safe operating independently of the rest, where a flat can cause the current expansion of eminent danger;

b) it should be possible to manually stop the installation so as not to compromise the safety equipment and protective systems involved in automatic processes which deviate from the conditions of work determined under SREN 418 [18] and SRCEI 60364 - 4 - 46 [19];

c) to stop in an emergency, accumulated energy must be dissipated as quickly as possible and secure so that it no longer constitutes a potential source of danger.

In areas with danger of explosion can be used [16]:

- Cables approved for implementation in the area (eg aluminum cables with s=16mm^2 excluding their use in plants with intrinsic safety);
- Insulated conductors installed in pipeline protection;
- Encapsulated bars having an antiex protection according SREN 50014 [20].

5. CONCLUSIONS

Knowledge of the hazards associated with chemicals, their classification, and characterization accordingly to the impact on the human or the environment, measuring or controlling the level of pollutants represent essential steps to achieve the purpose of risk reduction associated to chemicals.

Action taken by the European Union in domain of chemicals is designed to achieve two major objectives:

- Promoting the free transfer of chemicals in the territory of the EU countries;
- Health protection, public safety, animal and environmental protection;

Measures were based on the principle that a balance should exist between environmental protection and environmental health on the one hand, and other hand, the fierce competition in the chemical industry.

Pollution due to chemicals requires concerted international action. European Commission introduced restrictions for the entire Community market in respect of a particular substance which presents unacceptable risks, taking into account socioeconomic factors.

With the Romania entry into the European Union and the Acquis Communautaire, it is necessary to harmonize laws and technical regulations with particular EU legislation.

Thus, by Decision no. 804/2007 were transposed Council Directive 96/82/CEE regulations on the danger of major accidents involving dangerous substances.

It requires that parks specialists will develop specific rules for liquid fuel tanks with respect to EU requirements on health and safety, as well as fire prevention efficient measures.

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Scientific Reviewers:
Prof. PhD. Eng. Vasile Oros

KEY MEASURES FOR FIRE PREVENTION IN PARK TANKS OF LIQUID FUEL

ROLAND MORARU*
SORESCU FLORIN-MARIN**

Abstract: *The congestion of modern technological petrochemical installations has increased the dangers involving large light and very light quantities of hydrocarbons fallout, because evaporating forms huge free explosive. The site of tanks will be arranged to be away from the flood and to ensure proper drain pluvial waters, it will have ensured the supply of required water (drinking, industrial and extinction reserve) and will be provided with adequate drainage, and neutralizing wastewater systems.*

Keywords: *fire, prevention, park, tank, fuel, threat, occupational risk.*

1. BACKGROUND

General fire prevention in facilities construction and operation under Article 80 of Order 163/2007 for approving the general rules on fire defense concern [14]:

a) control/supervision of activities in terms of fire prevention, and fire following consequences;

b) establishing technical measures - to reduce organizational risk of fire or the consequences of fire;

c) keeping the done conditions for the safe evacuation of users and security intervention teams in the event of an fire outbreak;

d) maintenance of operational status of technical means for fire defense;

During the operation of technological installations are prohibited [2, 11, 14]:

♣non-compliance supervision tasks according to the instructions for operation;

♣operation without systems, devices and equipment required by the instructions for operating and maintaining the parameters for safety in operation or their replacement with more capacity;

♣in premises with high risk of fire or explosion prohibit access of employees and other persons without adequate protective equipment working conditions;

* Assoc.prof.PhD. Eng. -

♣works welding, cutting, soldering or other operations that present a hazard of fire, technological installations in the risk of fire or explosion, are running only after taken steps to evacuate people, emptying, washing tightening pipeline routes, ventilation or ventilation spaces, will be endowed with means of fire termination and only on the basis of the fire work permit, with a only one day validity;

♣smoking sites are located at a distance more than 40 m from the places where there is danger of explosion: Gas and liquid fuel, inflammable vapors.

On the interior roads will be planted appropriate route indicators (also at the crossing railway lines) will be mounted resistant footbridge to the passage from ditches, canals, and if this is not possible to provide pathways access and movement by-pass marked.

On the place of loading and unloading petroleum products will be provided gutter or sewage intakes to collect the leakage of petroleum products.

2. UNLOADING RAMPS FOR RAILWAY COACHES PETROLEUM PRODUCTS

The main measures of prevention and firefighting are [10]:

➢ operations of unloading, loading, transport and handling of petroleum products will be executed under the supervision of a work leader, specifically trained for this purpose and with strict instructions to work (using the procedures for hazardous work such operations);

➢ The use of helper means (key, crowbar, hammers, extension tools etc.) made by ferrous materials to avoid spark;

➢ construction of metal ramps will be linked to ground by fixed links (made under the rules in effect) for drain static electricity, they will be regularly maintained and inspected;

➢ trucks entering the discharge line will run with Diesel engines operating;

➢ at rail ramps for loading and unloading petroleum products shall be done the same potential in metal elements;

o Circulation staff at railway discharge line will be made only under the following rules:

o crossing only perpendicular direction over lines, after the person is convinced in advance that there are not moving locomotives and wagons;

o will not put the foot between the needle and needle or rail and contrarail;

o walking near cars will run at at least three meter distance from them;

o will not pass through the carriages and among the appropriate swabs;

o will not run on stairs and on the pads during maneuvers;

o will not jump from one car to another.

➢ winter season discharge of frozen petroleum products will be after their preliminary heating, the tanks will be connected to the steam store, or that of a operating locomotive; unfrozing is made using steam or hot water, the use of open fire is entirely prohibited. Steam pipes are well insulated and the insulation protected by a coat of plates to avoid soaking insulation with oil;

♦ concrete or metal platforms (ramps), coke, barrels will be constructed from materials incombustible, with a slope of about 3% to a corner of them, which will be linked to the deposit of sewage through a grate collector; ramp will be raised by at least 20 cm from the surrounding land;

♦ pipelines that cross oil outlet is connected to ground.

3. PUMP HOUSES

♦ pump houses will be built from materials provided by incombustible flooring and executed in a nonskid material, impervious to petroleum products, which do not produce sparks by striking, the floor will be provided with a slight slope (2%) to allow outflow liquid dissipated accidentally discharges into the collection vessel;

♦ pump houses will usually have natural ventilation (as appropriate and mechanical ventilation) will be provided with at least two exits and doors will be open only from the outside; height pump house will be at least 3 m;

♦ prohibiting installation in the same building with pumps, the electric motors operating centrifugal pumps are not executed if the construction anti-explosive;

♦ if the existing pump houses, equipped with electric motors of normal construction, with increased security they (and starting devices) will be mounted in a room separate from the pump room through an airtight septum;

♦ distance between the most prominent of pumps mounted on the same line and the wall will not be less than 1 m;

♦ Parts of mobile pumps will be provided with protective defenders; also each pump will be fitted with pressure gauge, which will have the maximum indicated by the sign (line) red;

♦ bare conductors or broken, disposed of elbows, etc.) in such cases is put into service a facility to remedy the defect;

♦ Free running operation is prohibited for electrical-pumps

4. STORAGE HANDLING, RAMPS FOR PACKED PRODUCTS, LOADING RAMPS FOR PETROLEUM PRODUCTS IN TANKERS

All ramps and facilities for loading and unloading will be connected to ground through flexible links to ensure leakage of static electricity formed during transport and pipelines during loading and unloading, making it before the start of operations of loading - unloading. Ramps will be made of incombustible materials having a length depending on the volume available, which should not exceed the maximum length of a gasket of rail tankers.

The main measures relate to:

♣in the handling warehouse is placed the distribution line for petroleum products in barrels or tanks, to the beneficiaries, this magazine will be produced as a completely separate rooms, which will perform only operations of handling and distribution;

♣handling warehouse will be constructed of materials having incombustible floor made of a nonskid material, impervious to petroleum products, which do not produce sparks by striking, will be provided with straggling accidental leakage of liquid and discharges it into the collection vessel;

♣ prohibited blocking access routes, evacuation and intervention materials that reduce the width or height set free movement or the peril of fire or explosion, and making changes to them;

♣ windows and doors will open out. If the old building with sliding doors, they will be secured against removal from the guides (which must not form the threshold but are buried in the floor), and will remain in the open position during the program;

♣ singer-bascule will be placed in concrete vats, whose genteel basic will have a slight slope to capture the vessel; weigher platform will be built at floor level storage handling with a space between the tank wall and the platform for more than 10 mm;

Loading road tankers shall comply with the following:

♦ ensuring vehicle movement against the move and hand brake, if slopes putting wooden wedges at the wheel;

♦ introduction of road tankers to the ramp and dropping their loading holes;

♦ making links to make land in tankers;

♦ mounting extension tools from loaders ramp and placing them in the tankers loading intakes;

♦ start of the operation itself to load, by opening the air charger;

♦ platform ramp car park will only road tankers being uploaded, the tankers will stand with the engine stopped, the ramp outside the places designed for storage;

♦ not be permitted verification engine repairs road tankers or platforms on the loading ramps;

♦ is not allowed by road tankers loading flow products directly without the extension tools;

In auto ramps for tanks should be provided to each filling or discharge a flexible link connected to the coupling to the ground and must be free to end a claw type clasp that link to car.

5. TANKS, DAMS AND PIPELINES

The main measures of fire prevention refers to [13.14]:

→ tanks for petroleum products will be placed in parks, on platforms, with the rate lower than technological platforms and facilities will be enclosed with dams in accordance with the rules in force;

→ land around tanks embankment will be level and will have a slight slope towards the gutters, to capture water meteorite, a condensation of heating products delivered loose or otherwise damaged. Gutters will be connected to sewerage by industrial decanter with hydraulic closure and be covered with barbecue;

→ land around tanks acceptance of the embankment will be paved and will slope to drain sewage mouth (according to paragraph above);

→ buried tanks will be placed over the surrounding land so that the top of the tank must be at least 20 cm below the share of land, around the reservoir will be filled with sand, the tank will be provided with a hole visit with a diameter of at least 45 cm and cover it will mount pipes and fittings for loading, firing, vent, measurement and protection against flame, measuring pipe will be provided with fittings and non-ferrous metal cap;

→ tanks will be located such that the bottom tank to be located below ground at least half of its height (for vertical tanks) or half the diameter (for horizontal tanks);

→ tanks will be equipped with health and safety at work, as follows:

- o reception tanks grouped in parks will be connected by a footbridge located at the height of lids, with railings with 2 access roads mounted in the opposite sides;
- o on access roads to parks tanks will place signs with the panel statement "passing off";
- o boarding ladders on the "cat" in tanks, both workers will be free tomorrow, all the instruments and tools (including roulette, vessels took samples, lamp etc..) will be placed in a bag to be worn on belt ;
- o the operations of sampling, measurements in tanks, etc.., operators will be back to the wind to avoid inhaling noxious vapors;
- o dams will be equipped with at least 2 stairs to enter the interior, located on opposite sides and with the current hand;
- o links to a ground check daily (visual) and periodically by qualified personnel who measured the resistance of the dispersion of gripping the ground and ensure the continuity of electrical installations;
- o distance pipelines for oil, water, air, steam will be labeled grouped at the appropriate distance (typically 10 m) as follows:
  - ▪ air piping will be marked with the symbol or name of the content and color in the standard force; Burn will be the entry and exit from the pump house, change direction and out of tanks;
  - ▪ the manholes of valves are mounted directly on the panel marked valves;
  - ▪ buried pipeline routes will be marked with appropriate panel mounted to change direction.
- o pipes, valves, fittings will be replaced if:
  - ▪ wall thickness has reached minimum operation;
  - ▪ wall thickness has not reached the minimum operation but estimated it could reach the limit during the next cycle of operation;
  - ▪ test with hammer blows is found that the material is degraded or that the wall thickness should be considered insufficient;
  - ▪ expired time allowed to run threaded flange is worn or is distorted;
  - ▪ does not provide closure (if valves channels).

6. CLEANING AND REPAIR OF CONTAINERS FOR STORAGE OF COMBUSTIBLE LIQUIDS

The main measures for cleaning and repair of tanks which combustible liquids refer to [14]:

►all tanks storing liquid petroleum products will be subject to periodic cleaning, according to a plan (determined in each enterprise according to the findings of inspections carried out under the tank inspection unit and the deposition of each type of product stored in different tanks), the work of preparation of the various tanks for cleaning and the cleaning will be performed in accordance with instructions at work and protection of each developed and approved by its management;

►instructions tanks will develop depending on the assortment of products contained, as follows:

- reservoirs for oil;
- reservoirs for products deflagrability below 45 ° C, no sulphur content;
- reservoirs for products deflagrability below 40 ° C, sulphurous;
- reservoir for products deflagrability from 40 ° C of not filing separate solid;
- reservoir for products deflagrability from 40 ° C, which is separated from solid deposits, frozen, etc..;
- etilene gasoline storage tanks;
- tanks for chemical reactive.

►tanks preparing for intervention will be subject to the following measures:

- all tanks that were stored with the product deflagrability below 38 ° C with a sulfur content higher than 0.02%, gas and vapor discharges from the reservoir will be steaming; steaming will run through boiling a layer of water at least 1 m thick, which will inject water directly or serpentine;
- during steaming tanks visiting mouth will remain open at all times to avoid crushing tank lid;
- cessation products steaming tank with sulfurous deflagrability point below 38 ° C, will be made only after the steam exhaust through the mouth by visiting the cap is no longer detected vapor hydrocarbons (usually through laboratory tests and by exception smell);

➢ work will only begin after approval and issuance of work permits;

➢ if the work lasts several days, the content analysis of the gas tank and the atmosphere around the reservoir will be repeated daily approve the extension permit for entry into the reservoir;

➢ foreman, head of training or the person designated to lead the work is required to train staff properly equipped to provide proper equipment, and equip with the necessary tools are required, also to supervise the execution of works and provide technical assistance, a staff member will monitor the work done by tank cleaner intakes lower visitation;

➢ entirely prohibiting washing or cleaning tools in the tank; the washing tanks of petroleum products, water velocity should not exceed 1m/s and 3 atmosphere pressure steam;

b) mechanical ventilation forced Desktop.

Access of personnel and operational interventions in case of fire, to rescue and first aid people in danger, fire and their effects must be ensured at all times [14]:

a) buildings and their premises;

b)technological facilities and annexes;

c)liquid fuel depots;

d) the technical means of fire defense: pumping stations, hydrants interior and exterior fire, extinguisher, ponds, tanks, castles, water, ramps of natural water sources;

e) paintings and distribution switch general electrical installations for lighting, power, security and backup;

f) technological valves or auxiliary facilities to be operated in case of fire and the command of the (gas and liquid fuel);

g) other means used to intervene in case of fire: traction for vehicles and transport tanks or tankers for water, etc.

Table 1. The number of connections to the ground

Tank diameter d [m]	d<2	2<=d<10	10<=d<20	20<=d<40
Number of links to ground	1	2 la 180°	3 la 120°	4 la 30°

7. CONCLUSIONS

Scientific-technical progress, boosted by meeting the requirements of a constantly increasing population, has generated an amplification and diversification of industrial production, leading to an exponential increase in risk for persons.

Maintaining control on chemical risks is achieved through continuous monitoring of the level of exposure, implementing measures in the prevention and protection of the deposits, rigorous monitoring of the health status of workers, informing workers about the dangers at using the new information available own experience and other producers, users and researchers.

Risk assessment for machinery producer may be limited to assessing compliance with the essential health and safety standards and approvals.

The best way to protect workers from chemical risks to chemical operators is to eliminate or reduce exposure by reducing concentrations in the work area, and the duration of exposure.

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Scientific Reviewers:
Prof. PhD. Eng. Vasile Oros

ISSUES AND INTERPRETATIONS OF THE IGNITION RISK RISED FROM MECHANICAL SPARKS IN EXPLOSIVE ATMOSPHERES

ADRIAN JURCA*
NICULINA VĂTAVU*
SORIN SICOI**
LEONARD LUPU***

Abstract: *The potential explosive zones are those industrial areas where there is a risk to be produced a mixture of air with gas, vapors, mists or combustible dust which can be ignited by the different ignition sources, resulting explosions which can produce life losses. The metallic materials of the equipment case or their components, from industrial installation, can become ignition sources by mechanical sparks and/or hot surfaces. The authors had identified and analyzed the ignition capacity by mechanical sparks on the main metallic materials and had established original solutions to prevent ignition of the explosive atmospheres.*

Key Words: *mechanical sparks, ignition, explosive atmospheres, explosion protection*.

1. INTRODUCTION

An explosive mixture formed by air with combustible gas, within explosive limits, can be ignited by different ignition sources of electrical, mechanical or thermal nature etc.[10][11]

The mechanical sparks represent ignition sources of a mechanical nature generated by friction phenomena. They are metallic particle resulted from mechanical interaction products between two metallic parts. Depending on the mechanical interaction conditions (light collision, intermittent friction or continuous friction) can produce mechanical sparks in form of particles with high or very high temperature.[4]

In literature are found two theories that explain the mechanism of mechanical spark formation:

* *D.Eng. – Senior scientific researcher III – INCD INSEMEX Petroşani*
** *Eng. – Senior scientific researcher II – INCD INSEMEX Petroşani*
*** *D.Eng. – Senior scientific researcher – INCD INSEMEX Petroşani*

- alumino-thermy process where through metallic particles arrive to incandescence as a result of the exothermic reaction between aluminum and iron oxides (rust).

- oxidation ignition and metallic particle burning theory where through metallic particle have a successive oxidation process, through initial temperature which was separated particle, ignition and burning of them under protection of a metal oxide layer formed on the outer surface which allows penetration of the necessary oxygen.[1][2]

2. ASSESSMENT

Testing the mechanism generating the ignition explosive mixture from mechanical spark, known up to the present, described in the literature, shows the following:

a) alumino-thermy process (thermal) producing metallic sparks is found both theoretically and experimentally in the case of mechanical interaction like impact and friction between aluminum and alloys with the rusty steel;

The thermal theory cannot explain generation of metallic sparks in the case of other industrial metals like Mg, Ti, Fe and their alloys.[6]

b) oxidation, ignition and metallic particle burning theory is argumentative in the case of majority industrial metals.

According to this theory a significant rising of ignition capacity of the mechanical spark has an oxidation process of the hot particle resulting by metals friction, in its trajectory thought explosive atmosphere.

c) In the theoretical and experimental researches based on these two theories presented in the literature has not been found a classification on the ignition capacity of all industrial used metals, except for a generally based grouping which put the metals in two main groups:

- Al, Mg, Ti – considered as the metal group having maximum ignition capacity;

- Fe and Cu with their alloys – considered the alloy group with reduced ignition capacity.

d) With all this knowledge acquired on theoretic and experimental ways on the complex phenomena of explosive mixture ignition by mechanical sparks, there has not been found any pertinent explication for all metal range and especially for hardly oxidizing steels, which led to incompatible technical solutions from the explosion protection point of view. It is mentioned that, when different metals had been tested, by the authors, to generate mechanical sparks in the INSEMEX laboratories, the stainless steels induced ignitions on explosive atmospheres, proving a ignition ability which has to be taken into account when risk assessment are ran, in industry or at designing the equipment and protective systems intended for potentially explosive atmospheres. [3][5]

3. ANALYSIS

The theoretic and experimental results obtained by the authors prove that besides the metals of the first group, stainless steels show, in certain situations, ignition capacity through mechanical sparks which are comparable with Al, Mg or Ti. Thus, in the explosion protection technique it had been arrived at applying non-adequate technical solutions (e.g. use of stainless steels in case of fans).[9]

The ignition principle of the mechanical sparks is well-known; this is given by the ratio of the energy developed in the metallic incandescent particle and ignition energy of the combustible mixture. The ignition takes place when the caloric energy of the metallic particle reaches or exceeds the ignition energy of the combustible mixture.[7]

In the specialty literature, to complete the ignition mechanism of the combustible mixtures with the mechanical sparks from very oxidizing metals it is used the curve d of the minimum ignition energy variation according to the amount of combustible gas, (determined having as ignition source an electric spark), as well as the variation curves of the caloric energy in the metallic particles depending on the oxygen content in the combustible mixture a, b, c, presented together in figure 1. [3]

In the inflammable mixture of air with combustible gas, increasing the combustible gas content results in a decreased oxygen amount. A lowered oxygen amount will result in a decreased temperature and caloric energy of the metallic particle, according to diagram a, b, c of the caloric energy by the mechanical sparks depending on the combustible gas content.

Determination of the optimum conditions of igniting the combustible mixture according to the combustible and oxygen concentration in the mixture can be realized through examination of the curves corresponding to heat emission from mechanical sparks together with minimum ignition energy diagram depending on the fuel content.

Examining the curves shown in figure 1 points out that the caloric energy curve a corresponding to metallic particles having an initial temperature T_a intersects diagram d in two points. The points a' correspond to the poor concentration of fuel C_1, and the point a'' corresponds to the rich concentration of fuel C_3. In the zone limited by the points a', b', a'', the heat emission from sparks attains the value of the minimum ignition energy of the mixture and so the mixture is ignited.

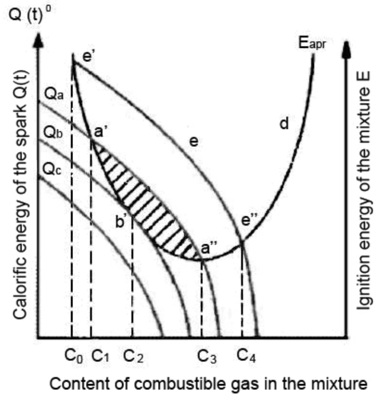
In the case of formation of sparks having the lower initial temperature $T_b < T_a$, heat emission will be lesser (on the same mixture fuel-air). In the situation of an initial temperature T_b of the metallic particles detached, the heat emitted represented by the b curve, intersects the curve d in the point b', corresponding to the poor mixture concentration at a C_2 fuel concentration.

In the case of sparks with initial temperature $T_c < T_b$, the diagram of the heat amount does not intersect the d curve, and as a result, the sparks with the initial temperature $T_c < T_b$ emits an amount of heat insufficient to ignite the fuel-air mixture in any concentration.

The a' and a'' points realize: a reduced range of concentrations (C_1 and C_3), within which the friction sparks can ignite the explosive mixture – respectively the explosive concentration limits possible by ignition by mechanical sparks from oxidisable metals. These limits of ignition by mechanical sparks represent a narrowing of range of concentration limits that are produced by the ignition sources which not depend by the oxygen amount, compared to the case of mechanical sparks resulted from hardly oxidizing metals (see figure 2). This reduction of the ignition range is produced generally because of the mixtures rich in fuel content, that will determine, at their turn, a decreased content of oxygen, respectively a lowered temperature of the metallic particle.

Figure 1. Determination of the optimal content for ignition of fuel mixture by mechanical sparks of oxidizable metals.

$T_a > T_b > T_c$



The b' point corresponds to the optimal fuel concentration in air when ignited through mechanical sparks. Thus it can be concluded that the mechanical sparks by the intensive oxidizable metals can easily ignite a mixture poor in content. On the other hand, determining the point b' is important to find out the optimal concentration to ignite the fuel-air mixture, which represent the point of maximum sensitivity of the ignition, necessary in experimental research activity of laboratory tests in explosive atmosphere.

It is to be noticed that the maximum ignition sensitivity by mechanical sparks is found in a point differing from the one set when ignited by electrical sparks, which is given by the minimum of the curve d.

This mechanism for ignition the combustible mixture is valid for metals which maintain intensive processes of oxidization, ignition and burning the metallic particles.

To the metals which not oxidize intensive or hardly, as those classified as non-oxidizable, the metallic particle temperature depends on the initial friction temperature had at the moment of detaching in the mechanical process. In this case, the metallic particles have a temperature which does not depend on the oxygen quantity, respectively on the fuel from inflammable mixture.

$$T_a > T_b > T_c\ ;$$

$$Q_a > Q_b > Q_c$$

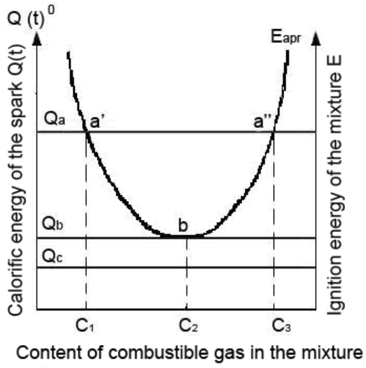


Figure 2. Determination of the optimal content for ignition of combustible mixture by hardly oxidizable mechanical sparks

The caloric energy of the metallic particles depends on: the temperature at the moment of being detached and the metal type. In figure 2 are presented the ignition energy diagram of the flammable gas mixture and the lines representing the caloric energy of the metallic particles which have constant values related to the combustible mixture concentrations. Ignition of the inflammable gas mixture from to mechanical sparks created by the hardly oxidizable metals can be analyzed through intersection of the ignition energy line by the inflammable gas mixture depending to the combustible quantity with caloric energy line of the mechanic particles.

In the case of the metallic particles detached with high temperature, which can be in some cases even the fusion temperature, when the particle is luminescent, the caloric energy of the particle Q_a can ignite the explosive mixture in the combustible concentration range C_1-C_3. This ignition range is two times higher than in the case of the oxidizable metallic particles.

Depending on the initial temperature, the mass or the particle volume and the metal type, the caloric energy by the metallic particle can achieve the diagram $E_{apr.}$ in the limit b point. But the b represents the minimum ignition energy of the inflammable mixture agreed through ignition determination in laboratory having ignition source the electrical spark.

So, in the case of inoxidizable metallic particles, the minimum ignition energy of the inflammable mixture coincides with the minimum ignition energy determinate with electrical sparks.

In the case of metallic particle which is detached with small temperature respective with small caloric energy, the line the caloric energy Q_c does not intersect the curve of ignition energy of the flammable mixture and in this case ignition does not occur. The mechanical sparks in this case are considered “cold mechanical sparks”.

4.SOLUTIONS FOR EXPLOSION PROTECTION

The solutions for protection against explosions are based on two main criteria:

a) Choosing the metals with a low ignition capacity;

b) Adopting spark-proof protective coverings;

When conceiving and designing of the technical equipment intended for use in potential explosive atmospheres in the industrial areas or in firedamp mines, designers must to have in view many criteria for choosing of the metals materials.[8]

A special case is represented by the equipment which requires often handling, mounting and dismantling, when the design requirement is to lower the weight. In this case, use of aluminum is most of the times the basic solution. The solution to prevent ignition hazard from mechanical sparks consist in the non-sparking protective covering. An example of application of this solution is the individual hydraulic light prop made from aluminum, for powered roof supports in firedamp coal mines.

Another special case is represented by the explosion-proof designed fans for potential explosive atmosphere with acetylene. At this equipment the danger of acetylides forming at contact between acetylene and different metals determined usage of aluminum alloys in manufacturing of rotor-stator items, but the final technical solution consisted in a protective covering made from inorganic corrosion protected materials.[9]

5. CONCLUSIONS

The results of the fundamental research obtained are in accordance with the results known up to present in the specialty literature, which are completed through them; this confirms as principle, the generality of employing the caloric criterium for the whole range of metals. The ability to ignite by mechanically generated sparks of the metals of industrial use shows a great importance both under theoretical and practical aspects in the technique of explosion protection of the technical equipment an industrial plants employed in potentially explosive atmospheres in the firedamp mines or industrial sectors other than the mining ones.

Classification of metals in groups according to their ignition ability through mechanical sparks makes a database for the design, conceiving and manufacturing activities within the explosion protection against the danger of mechanically generated sparks in industrial environments with potentially explosive atmospheres.

The technical solutions of explosion protection identified by research require technological development so as, through decreasing the costs in the intermediate products, they can be enrolled in the market economy challenges.

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Scientific Reviewers:
Prof. PhD. Eng. Mircea Georgescu

EXPLOSION RISK GENERATED BY THE NATURAL GAS DISTRIBUTED TO THE CONSUMERS

ROLAND MORARU*
PAVEL GHEORGHE**

Abstract: *Due to improper manipulation of devices that consume gas fuels or to improvisations of the natural gas usage installations, several explosions produce annually, of a smaller or larger intensity, resulting in important material damages, serious human accidents or even losses of life. Together with the cold season, the risk of these incidents producing increases, due to the inappropriate usage of the devices that consume gas fuels (cooking machines, stoves, heating devices, etc.) and due to unauthorized modifications of the natural gas usage installations. In this paper, the author proposes to present in detail the process of very fast and violent burning of the natural gas that can accumulate in spaces where devices that use gas fuels or natural gas usage installations are present.*

Key words: *methane, explosion, explosiveness limits, levels of danger*

1. GENERALITIES

Natural gas is a complex mixture of saturated hydrocarbons (alkanes) and other gases; it is known as **dried natural gas** and is part of the category of highly dangerous substances.

The natural gas distributed in Romania can be considered **methane gas (CH_4)**, as it results from the following data regarding its components:

Name	Chemical formula	Concentration [%Vol.]	Danger class	Symbol
Methane	**CH_4**	**99,31**	- highly dangerous	F+
Ethane	**C_2H_6**	0,11	- Highly dangerous	F+
Butane	**C_4H_{10}**	0,05	- Highly dangerous	F+
Nitrogen - azote	**N_2**	0,35	- Without danger	-
Carbon dioxide	**CO_2**	0,14	- Without danger	-
Other gas	-	0,04	-	-

* *Assoc. Prof. PhD. Eng. at the University of Petroşani*
** *Ph.D. candidate Eng. - Emergency Situations Specialist E.ON Gas Distribution*

Methane's general characteristics:

Inflammability temperature	161^0C
Ignition temperature in air *)	595^0C (550^0C ÷ 750^0C)
Ignition temperature in oxygen	550^0C
Minimum ignition energy	0,29 mJ
Explosion speed	2.300 m/s
Inferior explosion limit *)	4,5 % Vol. (4,5 % ÷ 6 % Vol.)
Superior explosion limit *)	16,5 % Vol. (14 % ÷ 16,5 % Vol.)
Maximum explosion point *)	9,5% Vol. (9,5% ÷ 10 % Vol.)
Vapor density in report with air (ρ = 1)	0,55
Inferior calorific value	35,80 MJ/m^3
Burning temperature *)	2500^0C (2.200^0C÷ 3100^0C)
Flame temperature in the air	2210^0C
Flame temperature in oxygen	3030^0C
Quantity of heat released through burning	8.400 Kcal/m^3 or 13.300 Kcal/Kg

*) by some authors the results of the research can differ within certain limits

Methane ignites very easy from a spark or open fire and the flame travels at very high speed within the gas mass, with this phenomenon producing instantly.

By burning in air, the methane gas transforms in carbon dioxide and water, releasing a large quantity of heat. ($1CH_4 + 2O_2 \Rightarrow 1CO_2 + 2H_2O + Q$)

In laboratory conditions, when heated to approximately 900^0C, methane decomposes in carbon and hydrogen ($CH_4 \Rightarrow C+2H_2$).

2. EXPLOSIVES CLASSIFICATION

The Explosion is a process of **very fast and very violent burning** of the explosive mixtures, which produces in fractions of a second, releasing heat and light, and generating very high pressures.

Depending on the burning (reaction) and explosive mixtures decomposing speed, explosions classify in:

- **Deflagration –** with a burning speed of cm/s and up to one hundred m/s. The deflagration is a quick and uniform burning which produces in steps, usually in closed spaces, where air access is constant.
- **Actual explosion –** with a burning speed of one hundred m/s and up to one thousand m/s. The actual explosion is a very fast burning that produces when the gas mixture fuel-air, existent in a closed space (room, recipient, pipe, etc), is between the explosiveness limits and comes in contact with a ignition source.
- **Detonation –** with a burning speed of over one thousand m/s. The detonation is an extremely fast burning that produces when the gas mixture fuel-air, existent in a closed space (pipes with sufficiently large diameters and lengths), is within the explosiveness limits and comes in contact with an ignition source, characterized by the appearance of a shock wave (compression).

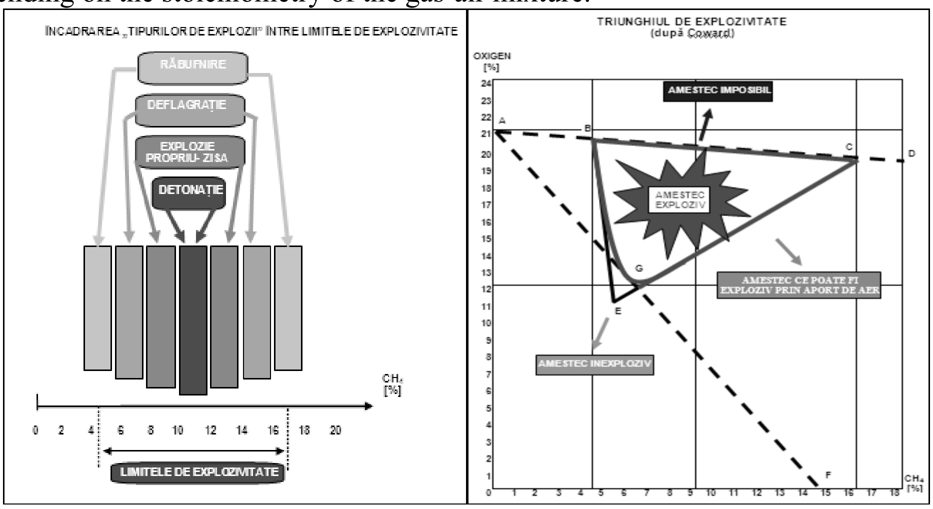
There is another type of explosion, **Outburst,** with a burning speed of m/s, which can be considered as a limit effect of the explosion, produced near the inferior or superior limit of explosiveness. The pressure that accumulates is not too high, but it lasts a little longer and it usually produces in the burners of the boilers, stoves or even in some spaces where explosive mixtures exist.

In another approach, the explosion can be defined as a fast transformation of potential energy, physical or chemical, in mechanical energy, which also involves a significant gas expansion. Thus, we have:

- **Physical explosions** that take place, for example, when a recipient is pressurized over its resistance limit and fails. When a gas contained within a recipient is heated excessively, an explosion can produce, which involves a shock wave, a „fire ball”, fragments projected from the recipient and strong heat radiations.
- **Chemical explosions**, which are more violent, or close to the stoichiometryc levels. A methane-oxygen mixture is stoichiometryc when the molar ratios are: $\mathbf{1CH_4 + 2O_2 \Rightarrow 1CO_2 + 2H_2O}$**.** For each methane molecule, the reaction requires two molecules of oxygen, in order to be in a stoichiometry proportion. A chemical explosion inside a reservoir can induce a physical explosion if the pressure increases sufficiently.

Depending on the flames travel speed during an explosion, reported to the sound speed (345 m/s), some scientist’s classified explosions as follows:

- **Deflagration,** which is a burning reaction in which the speed of the flame through the fuel environment not involved in the reaction is smaller then the speed of sound. Because the speed is smaller then the speed of sound, there is no shock wave.
- **Detonation,** which is a burning reaction in which the speed of the flame through the fuel environment not involved in the reaction is higher then the speed of sound. The shock waves that form help maintain the reaction. Detonations can move in any direction, for example if they produce inside a pipe, they are not stopped or restricted by the gas flow direction. Within a closed recipient, the maximum pressure caused by a detonation can be of 18 up to 30 times higher then the initial pressure, depending on the stoichiometry of the gas-air mixture.



3. EXPLOSIVENESS LIMITS

Inferior explosion limit (ignition) is the minimum concentration of combustible gas, vapors or dust in the air, which can explode. Bellow the inferior limit, the explosion cannot produce because of the air excess.

Superior explosion limit (ignition) is the maximum concentration of combustible gas, vapors or dust in the air, which can explode. Above the superior limit, the explosion cannot produce because of lack of air, with the mixture being too rich in combustible substance.

The explosive area increases if the pressure increases, which determines the slight decrease of the inferior limit and the substantial increase of the superior explosion limit, of the oxygen concentration, high temperatures and higher energy of the ignition source.

Moisture and other inert gases that form the combustible gas-air mixture decrease the explosive area.

Observations: If the methane burns (outside of the explosiveness limits), even though it is considered that a risk of explosion no longer exists, it continues to exist, because:

- If we are under the inferior limit (under **4,5 % Vol.**) and the methane burns with flame, in case of an unexpected flow of methane, the 4.5% Vol. concentration is exceeded and the explosion can produce;
- If we are over the superior limit (over **16,5 % Vol.**) and the methane burns with flame, if there is an air affluence, the concentration can decrease bellow 16.5% Vol. and the explosion can produce.

4. METHANE'S EXPLOSION DANGER

The explosion danger status consists of a complex of techno-organizational conditions in which a working system with fixed boundaries in space is at a certain point and that can determine the complete or partial removal of this system from the normal status through the unwanted event (explosion), having as consequence destruction of material goods and human victims.

The explosion danger state is determined by:

- The existence of combustible gas (methane) and of the oxygen in the air, which together make an **inflammable system,**
- The presence of a **ignition source** (the energy necessary for the methane ignition) and of the means that can produce it, and
- The appearance of a **favorable circumstance** (the concentration of the combustible gas-air mixture is within the explosiveness limits).

The inflammable system and the ignition source are determining for the production of explosions.

The imminent danger of explosion state is a concrete, real and actual situation that misses only the ignition source to produce an explosion at any given moment.

The parameters of the danger of explosion for methane gas are:
- inferior calorific value: 35,80 MJ/m^3
- flammability temperature: 161^0C
- ignition temperature: 595^0C
- minimum ignition energy: 0,29mJ
- explosion interval: 4,5 % ÷ 16,5 %
- minimum oxygen content for the explosion (burning): 12-14%
- explosion speed: 2.300 m/s
- explosion pressure: over 2000 KPa

The self-ignition (spontaneous ignition) temperature influences the parameters of the danger of explosion. Thus, a methane-hot air mixture can self-ignite without the action of an ignition sources or if it is exposed to a substantial air (oxygen) flow. Also, together with the increase of the pressure, the ignition temperature reduces, thus the methane can ignite beginning with the temperature of 540^0C.

The minimum ignition energy of methane is explained as the necessary energy for the most reactive methane-air mixture to ignite, influenced by the temperature of the surrounding environment, the quantity and period of time during which the energy is supplied and by the area in which it is supplied

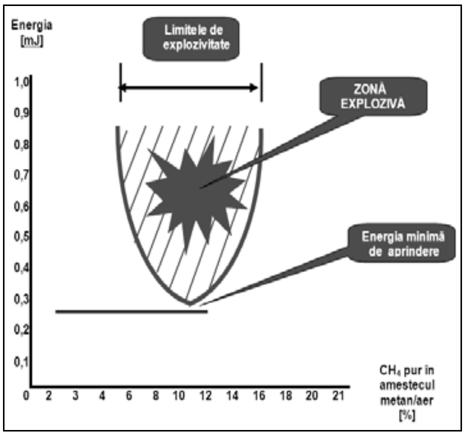
A mixture that is closer to the superior or inferior explosion limit, in order to ignite, can require a higher quantity of energy then the minimum ignition energy.

The effects of the explosive over-pressures increase together with the increase of the burning speed, with the over-pressure created during the explosion depending on the capacity of the flame to accelerate and to obtain higher and higher speeds.

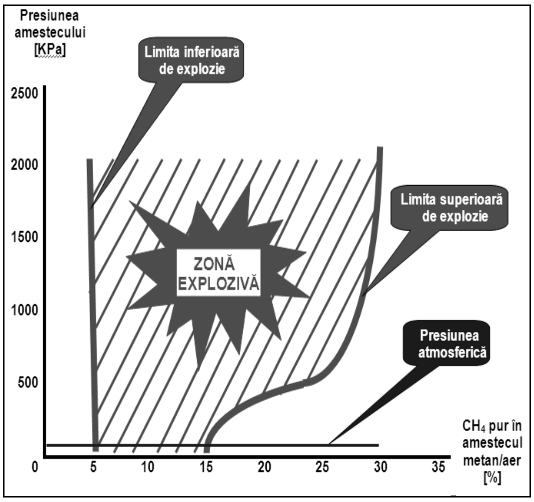
Obstacles, equipments and other blocked objects help increase the speed and acceleration of the flame. The turbulences created by the obstacles increase the speed of the flame, and, as the flame moves faster, more intense turbulences are generated.

The combustible gas – air mixtures have the maximum burning speed when the gas concentration is a little over the stoichiometry level.

The explosion pressure that is created after the ignition of the gas propagates under the form of shock waves and can exceed 2000 KPa (20 times higher then the atmospheric pressure).



Explosive limits for methane-air mixtures at atmospheric pressure and 26ºC temperature



The effect of the pressure on the explosive area of the methane in the air

5. DANGER OF EXPLOSION LEVELS

Depending on the gas concentration (Cg) in a space

1. **without danger - Cg ≤ 200 ppm (0,02% Vol.)**
2. **notable danger - Cg = 200 ÷ 5000 ppm (0,02 ÷ 0,5% Vol.)** – requires the ventilation of the space
3. **potential danger - Cg = 0,5 ÷ 2,2% Vol. -** requires the ventilation of the space, the interruption of the supply by closing the valves, the localization and remedy of the defect by authorized persons
4. **serious danger** - **Cg = 2,2 ÷ 4,4% Vol.** – requires the immediate interruption of the gas supply by closing the valves, the evacuation of the dangerous area, the ventilation of the space, the localization and remedy of the defect by authorized persons.
5. **serious and imminent danger – danger of explosion** - **Cg = 4,4 ÷16,5% Vol. –** requires the interruption of all activities, the evacuation of the dangerous area, the immediate interruption of the gas supply by closing the valves located outside the space, the ventilation of the space, the verification of the gas concentration decrease in the space and then the localization and remedy of the defect by authorized persons.

4. severe danger - Cg = 16,5 ÷ 100% Vol. – requires the immediate interruption of the gas supply by closing the valves located outside the space, the evacuation of the dangerous area, the ventilation of the space, the permanent verification of the gas concentration decrease in the space and then the localization and remedy of the defect by authorized persons.

Attention: In case of methane concentration of over 40% Vol., there is a danger of asphyxiation or suffocation because of the lack of oxygen replaced by methane!

6. CONCLUSIONS

To prevent and eliminate the danger of explosion, it is forbidden to modify the natural gas usage installations without the necessary approvals and/or by unauthorized persons. The consumers who want to modify the in-house installations should only appeal to the services of a company or persons authorized to project and execute natural gas installations.

The verification and technical revision of the installations are compulsory and are part of the duties of the consumers, but companies authorized by ANRE (National Regulation Authority in the Energy Field) must perform them.

The technical verification of the installations is performed every 2 years.

The technical revision of the installations is done every 10 years, in case the installation is not used for a period higher then 6 months, and after every event that could affect the safe functioning of the installation.

Also, the verification, maintenance, and repair of ACCG (devices that consumer gas fuels: cooking machines, heating devices, etc.), are responsibilities of the consumers, but must be performed by persons or companies authorized by ANRE.

To prevent accidents resulting in human lives losses and destruction of material goods, which could produce as a result of explosions, the consumers must strictly follow these instructions:

<u>Before igniting the fire</u>:

- Ventilation of the rooms in which ACCG function;
- Verify the draught of ACCG connected to the chimney;
- Verify the valves of ACCG;
- Ensure air access to ACCG with burners;
- Verification of the functioning of the automation equipments, by case.

<u>When igniting the fire (for non-automated ACCG and burners)</u>:

- Ventilation of the burner for minimum 5 minutes before turning on the fire;
- Usage of the special igniter for the burner (matches, lighters, paper, etc., are forbidden);
- Slow opening of the maneuver valve and ignition of the fire, with the supervision of the flame's stability.

Igniting the fire in the case of automated ACCG is done according to the instructions elaborated by the producer.

<u>During the usage of ACCG</u>:

- it is forbidden to leave the fire unsupervised, in case of non-automated ACCG;

If a specific natural gas smell is felt, these are the following measures to be taken:

- all fires are put out and all doors and windows are opened;
- it is forbidden to turn on the light or to maneuver electrical appliances;
- the licensed distribution operator is announced immediately;

The verification of possible natural gas leaks is done by smell and with water and soap foam. It is forbidden to verify the natural gas installations with open flames.

When putting out the fire:

Putting out the fire at the ACCG connected with a hose is done by closing the safety valve located ahead of the hose, and after the flame is out, the maneuver valve of the ACCG is also closed.

In companies and institution, the fire is ignited and put out by trained personnel, responsible for this operation, and in case of economic operators and apartment unit heating devices, by qualified personnel.

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Scientific Reviewers:
Prof. PhD. Eng. Vasile Oros

CONTRIBUTIONS TO KNOWLEDGE OF CHEMISTRY AND MINERALOGO-PETROGRAPHIC FEATURES OF AMPHIBOLITES FORMING DEBRIS GORGES IN JIU

POSTOLACHE MIHAIELA*

Between amphibolite rocks in amphibolite composition which type hornblende common green mineral opaque as representative and predominant in the study were three types namely: massive amphibolites, amphibolites and gneiss rubanate.

Following a complete microscopic study of the use of micrometric eyepiece and device integration (measurement) "Eltior" were established following modal data on mineralogical composition and metamorphic rocks forming minerals dimensions mentioned above (Table 1).

Table 1

Rock type / Minerals	Massive amphibolites		Rubanate amphibolites		Gneiss amphibolites	
	%	Size (mm)	%	Size (mm)	%	Size (mm)
Hornblende	61,6	0,064 x 0,096 1,305 x 1,522	56,9	0,048 x 0,080 0,840 x 2,827	50,7	0,080 x 0,112 0,483 x 1,288
Plagioclase	27,9	0,064 x 0,096 0,402 x 0,885	20,7	0,064 x 0,096 0,402 x 0,966	31,5	0,064 x 0,080 0,322 x 0,885
Quartz	4,2	0,015 x 0,022 0,187 x 0,225	12,2	0,016 x 0,032 2,175 x 4,350	4,7	0,032 x 0,048 0,241 x 0,563
Biotite	0,1	0,128 x 0,724	–	–	3,4	0,048 x 0,161 0,322 x 1,610
Titanite	1,5	0,032 x 0,048 0,112 x 0,181	1,3	0,015 x 0,030 0,037 x 0,112	2,3	0,032 x 0,048 0,080 x 0,161
Apatite	0,7	0,032 x 0,048 0,096 x 0,112	0,8	0,016 x 0,048 0,161 x 0,644	0,5	0,048 x 0,064 0,048 x 0,161
Actinote	0,2	0,022 x 0,037 0,030 x 0,112	–	–	–	–
Chlorite	0,4	0,015 x 0,030 0,015 x 0,075	1,5	0,015 x 0,037 0,037 x 0,075	1,8	0,048 x 0,044 0,112 x 0,402
Sericite	1,3	0,007 x 0,037 0,030 x 0,075	0,9	0,007 x 0,013 0,010 x 0,026	4,2	0,007 x 0,027 0,030 x 0,075
Epidote –zoizite	0,4	0,015 x 0,022 0,030 x 0,075	–	–	–	–
Opaque minerals	1,7	0,048 x 0,080 0,128 x 0,483	5,7	0,016 x 0,032 0,126 x 1,127	0,9	0,032 x 0,032 0,161 x 0,322
TOTAL	**100**		**100**		**100**	

* *Assist. Prof. PhD. Eng. - University of Petrosani*

Of those presented can be seen in terms of primary quantitative mineralogical composition, in which hornblende and feldspar plagioclaz allows with certainty that analyzed rocks to fall within amphibolites category.

Quartz which does not exceed 4.7% of all components is another characteristic of those rocks, in which a feromagnezian silicate (in this case hornblende) is predominant.

A special case is presented amphibolites rubanate in which the silicon dioxide appears in a higher percentage of 12.2%.

This is explained by the fact that the composition of these types of rock, white bands alternating with ones hornblendics, are made of mineral Fels (such as feldspar and quartz plagioclaz migmatics).

In terms of absolute size of components, there is a clear predominance of those who granulometry subunit.

This feature is specific for nematoblastic texture of rocks and grano-nematoblastic texture which is microcrystalline and very occasionally mezocrystalline.

Fineness of mineral components for a top draw attention to the possibility that the amphibolite rocks Jiu Defile, mostly, to provide superior physical and mechanical parameters, features rock with great cohesion of a kind "hard-rocks".

Origin of igneous amphibolites rocks was presented since 1937 by G. Paliuc, and then more studies L. Pavelescu, Maria Pavelescu and collaborators (1958, 1967, 1969, 1970 and 1971).

This study provides new contributions to understanding the origin of complex lower amphibolites Drăgşan series making up the gorge Danubian Autochthonous Jiu. For starters, have pursued chemical characters of rock and especially situation reports of TiO_2, MgO and FeO.

As is known, titanium and iron tend to separate the early stage of magma crystallization, these elements focus on ilmenite and titanomagnetit. However, the ilmenite of magmatic rocks formed in the first stage of crystallization is replaced partly or wholly of titanium. It follows therefore that the crystalline schists from igneous rock will contain mostly titanite and less ilmenite, and metamorfites of sedimentary origin are devoid of this mineral, the titanite being present as oxide in small percentage.

P. Giraud (1960) finds in several amphibolites analyzed that the value of TiO_2 is more than 0.8% in rocks poor in MgO and richer in FeO. As the molar ratio of MgO/FeO increase, the amount of TiO_2 decreases.

Based on these observations, case reports TiO_2, MgO, FeO and other oxides, in the case of amphibolite rocks of the lower complex series Drăgşan can be followed in Table 2.

Oxide percentage values presented in the table below highlight from the outset, a characteristic of amphibolites composition. Thus, SiO_2 and alkalis are relatively small, while ferromagnesians and calcium oxides appear rather high percentage. This is clear, moreover, from ternary diagram MgO - FeO + Fe_2O_3 - Na_2O + K_2O (Fig. 1).

The determination of material and finding that premetamorphic during regional metamorphism, overall chemical composition of rock original has been preserved or not, amphibolite rocks chemistry was performed by specific methods eruptive rocks (lava). Among them was called, the method of Niggli, Köhler and Raaz. Niggli parameters are presented in Table 3.

Table 2

no. sample	1	2	3
Rock type	Massive amphibolites	Rubanate amphibolites	Gneiss amphibolites
Oxides	%	%	%
SiO_2	46,82	54,75	50,02
Al_2O_3	16,57	16,00	20,32
Fe_2O_3	5,99	2,31	3,25
FeO	6,93	5,06	7,79
MnO	0,04	0,08	0,06
MgO	3,68	6,50	3,50
CaO	11,81	8,68	9,68
Na_2O	2,34	2,56	2,69
K_2O	1,86	1,42	0,50
TiO_2	1,04	1,10	0,80
P_2O_5	0,09	0,10	0,07
S	0,14	–	–
CO_2	–	0,10	0,39
H_2O	2,43	1,23	0,92
P.C.	3,14	1,34	1,31

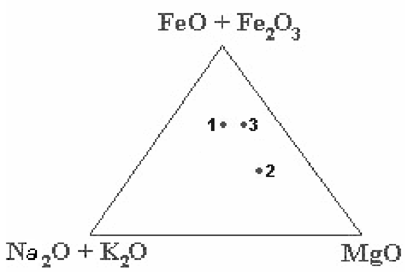

Fig 1 – Ternary diagram of the molecular units of alkali, iron and magnesium.

Table 3

Rock type / Niggli parameters	Massive amphibolites	Rubanate amphibolites	Gneiss amphibolites
si	114	145	130
al	22,50	25	31
fm	38	42	36
c	30,50	25	27
alk	8	9	7
k	0,34	0,27	0,10
mg	0,48	0,62	0,37
o	0,28	0,10	0,17
c/fm	0,65	0,59	0,75
q_z	1,50	9	2
ti	1,80	1,50	1,50

Examining Niggli parameters values from the table, stands low value of the parameter „q_z” which accords with reality. Amphibolite rocks examined except rubanat amphibolite (migmatizat) for the „q_z” has a slightly higher value, being poor free quartz. While parameter „c” in conjunction with the „fm” is over 60% of all the four basic parameters „al”, „fm”, „c” and „alk” calculated for these rocks.

In one of the tetrahedron Niggli sections with the parameters mentioned above (Fig. 2) analyzed three types of amphibolites designing field eruptive rocks (lava).

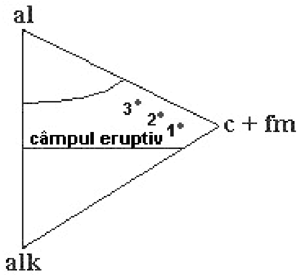

Fig 2 – Amphibolites rocks projection of sections in a tetrahedron **al – alk – c + fm** (**Niggli**)

Since projection method of calculation of Köhler and Raaz, the double ternary diagram „+q_z – F – fm – (-q_z) (fig.3), shows also that the amphibolite rocks Drăgşan series, present in some fields of debris in Jiu Defile, derived from basic rocks, poor in quartz and high content in fenice minerals.

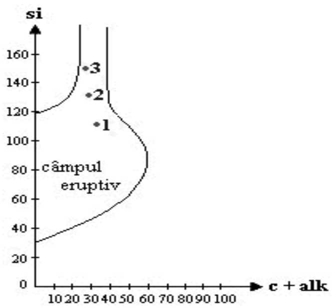

Fig 3 – Diagram si – c + alk, for the amphibolite rocks with eruptive rocks delimit the field after Burii.

Rubanat amphibolite having a higher sensitivity in quartz was designed to limit the field of basic and neutral eruptive rocks.

CONCLUSION

In conclusion, amphibolite rocks analyzed in terms mineralogo-petrographic and petrochemical derived from metamorphism of basic rocks and neutral within amphibolite facies, subfacies staurolit-almandin. Evolution of geosynclinals areas during prebaikalian cycle was accompanied by a first activity of the initial magmatism, represented by complex ofiolitice rocks, which then have derived the main types of amphibolite rocks Drăgşan series.

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Scientific Reviewers:
Prof. PhD. Eng. Grigore Buia

FLOTATION OF SEDIMENTS FROM THE CERNY POTOK STREAM

IVA JANAKOVA*
PETER FECKO*
NIKOLAS MUCHA*
BARBARA TORA*

Abstract: *The paper deals with an evaluation of flotation application suitability in the sludge decontamination from the Cerny Potok situated in the City of Ostrava. Flotation of organic pollutants is very effectual and more than 70 % of NEL, 90 % of PCB and 80 % of PAH from the sediments was removed.*

1. INTRODUCTION

The environment has been polluted by a variety of hydrocarbons, oil substances in particular, since the beginning of the last century. In the 1980s their quantities reached alarming 6,000,000 tons a year. A small amount of such contaminating hydrocarbons changes in the environment by oxidation or photochemical reactions. The rest is dispersed by vaporization, dissolves in water, remains in sediments, is covered by ice and a part is degraded by microorganisms. However, the hydrocarbons are really removed from the environment by combustion or by the action of microorganisms. Otherwise, they stay in the environment for many years.

1.1 Description of the Cerny Potok locality

The Cerny Potok rises in Ostrava, the municipality of Marianské Hory, approximately 230 m above the sea level. It runs through the city district of Moravská Ostrava and Privoz and near the confluence of the Odra and Ostravice Rivers, it issues into the Odra. The course is about 5 km long and Q_{355} is 0.074 $m^3 s^{-1}$. Waste water from the Jan Sverma Coking Plant and Central Waste Water Treatment Plant for Ostrava (UCOV) is discharged into the Cerny Potok. The average value of BSK_5 in 2007 was 7.2 mg/l (4th quality class) and CHSK 37.5 mg/l (5th quality class). Also the parameter of ammoniated nitrogen of 3.28 mg/l places this water course into the 5th quality class.

* *Prof.Ph.D.Eng. - VSB-Technical Univerzity of Ostrava, peter.fecko@vsb.cz*

Out of the monitored parameters only total phosphorus (0.25 mg/l) and electrolytic conductivity of 122 mS/m are placed in the class 3. The water quality in the Cerny Potok is predominantly determined by the quality of the discharged water from the Jan Sverma Coking Plant (0.014 m^3s^{-1}) and water from the Central Waste Water Treatment Plant for Ostrava (0.038 m^3s^{-1}), which participate in the course aquosity with two thirds. According to a water-management decision only the parameters in Table 1 are observed with the both sources.

Table 1. *Observed parameter average values with the discharged waste water for 2007*

Source	BSK_5	CHSK	NL	RAS	$N\text{-}NH_4^+$	N_{inorg}	P_{total}
	(mg/l)						
OKD Jan Šverma Coking Plant	7.1	31.6	17.2	340	2.3	5.3	0.4
ÚČOV Ostrava	11.3	33.5	13.7	503	6.7	8.1	0.6

Within the construction of D 47 motorway a partial relocation of the water course into a new bed (2.47 km) had been planned below the waste water discharge from the Jan Sverma Coking Plant. The relocation of the water course also required a removal of the sediments. The sediment reached the thickness of 1.8 m, the average thickness of the sediment in the bed was 1.2 m and the minimum thickness of the sediment was 0.2 m (shipyard). The sediment comprised of organic pollutants in the concentrations (See Table 2), which prevented its dumping on a disposal site. Also the content of organic carbon in the sediment solids (30.82%) was the second key parameter, which limited its dumping on a disposal site (5% TOC). The content of heavy metals in the water extract meets the requirements of Regulation 294/2005 Coll. The problem sediment pollutants are mainly PAH, PCB and NEL (hydrocarbons C10 – C40).

These pollutants get in the sediments predominantly from the waste water from both the above mentioned sources, larger part being on the part of the Jan Sverma Coking Plant and probably there is seepage from the industrial premises of the Jan Sverma Coking Plant and the near-by oil lagoons Ostramo, which affect the content of NEL and PCB in the sediment. According to the requirements NV 229/2007 Coll. six analytes of PAH represented in ΣPAH (fluorantene, benzo(b)fluorantene, benzo(k)fluorantene, benzo(a)pyrene, benzo(ghi)perlene, indeno(1,2,3-cd)pyrene) are observed on the outlet from the UCOV Ostrava; the free of charge limit is 0.5 g/day. The substance balance for the above mentioned PAH analytes implies that in case of the observed analytes the UČOV Ostrava discharges as much as 10.42 g/day, 1.14 g/day of anthracene and in □PAH (15 analytes) it is as much as 67 g/day. PCB on the outlet from the ÚČOV is under the detection limit. A significantly higher amount of PAH is contained in the waste water from Jan Sverma Coking Plant, where the daily amount for the 15 analytes reaches the value of 133 g/day.

With regard to the fact that the content of the organic pollutants in the solids exceeded the limits for dumping this waste on disposal sites a suitable decontaminating technology had to be selected, which would permit a safe disposal of the waste on the disposal site or a new method of utilization of the decontaminated sediment (e.g. for land reclamation).

Table 2. *Organic pollutant content in the sediment solids from the Cerny Potok (mg/kg solids)*

Benzene	BTEX	EOX	NEL	Tetrachlorethene	Trichlorethene	PAH(15)	PCB(6)
3.58	8.722	14.4	2880	0.76	1.06	541	0.644

Notes: PCB congeners 28, 52,101,138,153,180

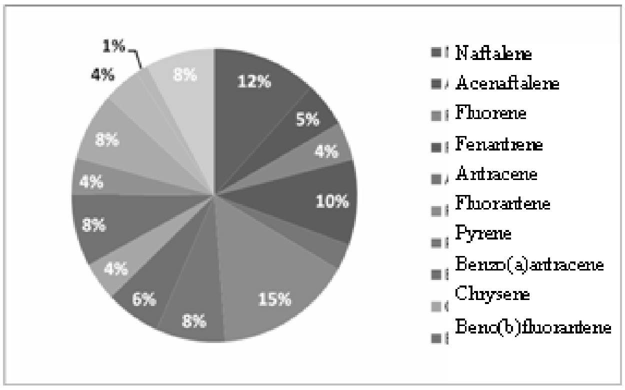

Fig.1. Organic hydrocarbons

Figure 1 shows the representation of the individual analytes of PAH. It is apparent from the figure that the so-called “light PAH” amount to about 21% and the so-called “heavy PAH” represent 79%.

1.2. Mineralogical composition of the Cerny Potok sediment

Four samples of sediments from different sampling points were taken. The mineralogical composition analysis of the samples was carried out on an all-levels sample. An X-ray diffraction analysis was done in the laboratories of the Institute of Geological Engineering at VŠB-TU Ostrava. The measurements were carried out on a modernized, fully automated diffractometer URD-6 (Rich. Seifert-FPM, SRN). The following phases were identified on the samples in question: ankerite, chlorite, kaolinite, muscovite, orthoclase, plagioclase, albite, quartz (See Figure 2).

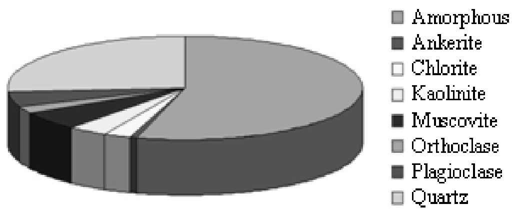

Fig. 2. Mineralogical composition of the Cerny Potok sediments

2. FLOTATION OF CONTAMINATED SEDIMENTS AND SOILS

Flotation has not been applied in the cleaning of organically contaminated soils and sediments in the Czech Republic so far. In the CR there are only references on the application of flotation predominantly in the cleaning of oiled water which can be found in the textbooks on the theory of flotation Kmet (1986), Spaldon (1983). Michael J. Ahrens and Craig V. Depree (2004) studied the representation of polycyclic aromatic hydrocarbons in 6 grain sizes of Auckland Harbour sediments in New Zealand. Flotation was used only to distribute PAH in the individual fractions. A solution of sodium-polytungstate collector was used for the flotation.

In a Dutch scientific study of Mulleneers et al. (2002) a new flotation method on laboratory scale was used as an alternative remediation technique. Dissolved air was used to create small bubbles. Tests were performed with the finest fractions of sediments of Overschie (Rotterdam) and Petrol Harbour (Amsterdam) sludges contaminated with Polycyclic Aromatic Hydrocarbons (PAH). Several agents and conditions were tested with respect to the flotation efficiency. For Overschie sludge the best results were obtained without collector and SDS as frother. The PAH concentration in the froth was up to 8 times higher than that in the non-floating fractions. The collected amount of dry matter in the froth was around 13%. With a two step flotation, the PAH concentration of the non-floating clean fraction was reduced from 240 mg (kg.d.m^{-1}) to 99 mg (kg.d.m^{-1}). For Petrol Harbour sludge the best results were obtained with the alcohol-based frothers Aerofroth and Montanol. The flotation was less selective than with Overschie sediment. Around 50% solids were collected in the froth and the PAH concentration in the froth was around 2 times higher than in the settled fraction and 3-5 times higher than in the fraction remains.

2.1 Flotation experiments

The flotation experiments were carried out in the laboratories of the Institute of Environmental Engineering at the VSB-TU Ostrava on a flotation machine VRF-1, a product of RD Pribram with an active capacity of 1 litre, under the following conditions:

Condensation: 150 g 1-1

Flotakol NX collector dose: 500 g. t^{-1}

Pulp collector agitation: 1 min

Flotation time: 10 min

Having finished the flotation, the flotation products (both the concentrate and the tailings) were filtered, dried and underwent chemical analyses for PAH, PCB and NEL, which were carried out in the laboratories of VÚHU, a.s. in Most. The flotation experiments were done on 4 samples from different sampling points.

Sample 1

Sample 1 was taken from a levee situated 200 meters from the chemical plant of Borsodchem, a.s. Ostrava (Figure 3). The flotation experiment results are given in Table 3. The obtained results imply that applying flotation it is possible to remove 73 % of NEL, 93 % of PCB and 80 % of PAH from the sediments.

Fig. 3. Sampling point of Sample 1 – Levee

Table 3. *Flotation results – Sample "Levee"*

Product	Recovery	Content			Recovery		
		PAH	ΣPCB	NEL	PAH	ΣPCB	NEL
	%	mg/kg			%		
K	65,44	216	0,291	1900	80,19	92,89	72,71
O	34,56	101	0,042	1350	19,81	7,11	27,29
P	100	176,26	0,205	1710	100	100	100

K – concentrate, O – tailings, P – feed

Table 4. *Flotation results – Sample Sverma*

Product	Recovery	Content			Recovery		
		PAH	ΣPCB	NEL	PAH	ΣPCB	NEL
	%	mg/kg			%		
K	68,23	840	0,375	2050	83,91	75,48	78,58
O	31,77	347	0,261	1200	6,09	24,52	21,42
P	100	683	0,339	1780	100	100	100

K – concentrate, O – tailings, P – feed

Table 5. *Flotation results – Sample 3*

Product	Recovery	Content			Recovery		
		PAH	ΣPCB	NEL	PAH	ΣPCB	NEL
	%	mg/kg			%		
K	63,47	548,67	352,44	4636	77,31	72,23	95,99
O	36,53	279,76	235,32	1812	22,69	27,77	4,01
P	100	450,44	309,66	3065,4	100	100	100

K – concentrate, O – tailings, P – feed

Sample 2

The second sample was taken from a point placed near the Jan Sverma Coking Plant (Figure 4). Table 4 shows the results of its flotation experiments. It is apparent from the acquired results that applying flotation it is possible to remove 79 % of NEL, 75 % of PCB and 84 % of PAH from the sediments.

Fig. 4. Sampling point of Sample 2 – Sverma

Sample 3

The next sample was taken from the centre of the Cerny Potok's bed, which is between the sampling points 1 and 2 (Figure 5). The flotation experiment results are given in Table 5. The results imply that flotation is able to remove 96 % of NEL, 72 % of PCB and 77 % of PAH from the sediments.

Fig. 5. Sampling point of Sample 3

It is apparent from the flotation experiments (See Tables 3 to 5) that the application of flotation on the given samples is highly effective and the acquired results are excellent. In case of all the samples applying flotation it was possible to reach a pollutant recovery into the concentrate higher than 70 %, which means that the given inexpensive technology it could be possible to treat such sediments which are still a great environmental strain for the City of Ostrava.

3. CONCLUSION

The objective was to verify the application of separation by flotation in the decontamination of sediment samples from the Cerny Potok situated in the City of Ostrava. The obtained results imply that the application of flotation is very effective and it is possible to apply it to eliminate harmful organic pollutants from the sediments in question.

4. ACKNOWLEDGEMENT

In conclusion we would like to thank GACR. The work benefited from the project financial support grant No. 105/09/P221.

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Scientific Reviewers:
Prof. PhD. Eng. Sanda Krausz

PYROLYSIS OILS FROM WASTE MATERIALS AS A COLLECTORS IN BLACK COAL FLOTATION

PETER FECKO*
ALENA KASPARKOVA*
VLASTIMIL KRIZ*
JOSIP ISEK*
TIEN PHAM DUC*
MONIKA PODESVOVA*

Abstract: *The paper deals with verification of floatability of classical collector Montanol 551 (commonly applied collector in the Czech Republic and Poland) and pyrolysis oils, which were obtained through pyrolysis of waste, namely mixed plastics, tyres and waste rubber in combination with black coal from Lazy Mine, in black coal flotation. Black coal from ČSA OKD, a.s. coal preparation plant was used for flotation tests. The results imply that it is possible to produce collectors from waste materials which may be applied in the flotation of black coal.*

Keywords: *flotation, black coal, pyrolysis, plastic waste, tyre and rubber*

1. INTRODUCTION

Coal is a raw material mined for the purposes of numerous industries. At the beginning of the 21st century, the global significance of coal for the civilization is universal. It is used in the metallurgy, power-engineering, chemistry and many other industrial branches.

Currently, mining of black coal is stagnating despite the fact that the consumption of industrial raw materials, the natural resources of which are not renewable and depletable by human action, is increasing. The level of slump in the coal mining is dependent on the development of both international and national conditions. Another significant factor having an influence on this is the fulfilment of international agreements on gradual reduction of sulphur oxides released into the atmosphere. There is a question how to deal with the issues in connection with the slump in coal mining.

One possibility is as perfect preparation of the mined coal as possible as well as maximum utilization of its combustible component.

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* *Prof. Ph D. Eng. - VSB-Technical University of Ostrava, peter.fecko@vsb.cz*

This trend represents new objectives for mineral dressing – to prepare and comprehensively use the finest coal fractions that originate from the processes of black coal preparation. The created large share of slurry fractions cannot be processed applying classical dressing methods. Those fractions then leave the preparation plant unused, they get into the circuit water of waste water treatment plants, they deteriorate the processes of sedimentation, filtration, water courses get polluted and there are losses in valuable raw materials as coal slurries are dewatered in sludge beds or lagoons and they remain unutilized.

The technological procedures of processing of mineral and secondary raw materials, which can be used to deal with the difficult task of ensuring effective, economical and complex utilization of material resources, are flotation, flocculation or their combination, i.e. flotoflocculation.

Thanks to flotation as a technological method of dressing mineral resources and secondary raw materials it is possible to deal with an uneasy task of ensuring an effective, practical and complex utilization of materials sources. Flotation belongs among methods which permit processing of fine, non-homogeneous black-coal slurry. Coal flotation deals with cleaning the finest fractions of mined coal that originates due to abrasion or disintegration in the course of coal mining and preparation. In practice, it is used only for preparation of cokeable coal. Flotation in coal preparation is a complementary method of other classification processes but very important one and often necessary, especially in connection with ever increasing share of fine fractions due to an extended degree of mechanization of coal mining.

Along with improving the selectivity of flotation it is currently possible to make use of old dumps as the source of material. There was no possible utilization for such dumps in the past. In the course of flotation, very fine fractions get separated, which would otherwise transfer into waste. Next, it is possible to apply flotation to retrieve coal for power-engineering purposes from old settling pits.

This way a significant share of waste constituent is removed from the pulp and thus it gets enriched. Selective flotation can be a final cleaning method or a part of a complex preparation process, which leads to preliminary enrichment of pulp before further preparation process. Application of a suitable collector, which would be ecologically as well as economically interesting, makes an inseparable part of the flotation process. Therefore, it is vital to search for new collectors that would comply with the conditions in question.

2. EXPERIMENTAL MINERALOGICAL-PETROLOGIC CHARACTER OF THE TESTED COAL

Coal polished sections were prepared in compliance with CSN ISO 7404-2 Standard. Maceral analysis was implemented using grain sections in compliance with CSN ISO 7404-3 Standard. Coal-petrographic analyses were carried out using a microscope of NU 2 of Carl Zeiss Jena in oil immersion under the following conditions: $n_D = 1.515$, $\lambda = 546$ nm and temperature of 20^{o}C. In the photos of the coal polished sections the magnification scale is 50 μm. Petrographic analyses were carried in the laboratories of

Research Institute of Arcelor Mittal, a.s. Ostrava. Table 1 gives an evaluation of petrologic analyses, where the individual symbols are to mean:

D - candle coal
G - gas coal
Ž - medium volatile bituminous coal
Kž - low-volatile bituminous coal
Ka - baking coal of the 1st type
Kb - baking coal of the 2nd type
T – anthracite coal
Rn - random reflectivity according to CSN ISO 7404-5

Table 1. *Evaluation of petrologic analysis of coal from ČSA Mine*

maceral analysis (%)	**type analysis** (%)		**vitrinite reflectivity** (%)
Vitrinite: 49,2	D:	Ka:	Rn: 1,028
Inertinite: 44,8	G: 22	Kb:	Rmin: 0,853
Liptinite: 6,0	Ž: 60	T:	Rmax: 1,294
	Kž: 18	A:	s: 0,113

Figure 1 displays the reflectivity of black coal from the ČSA locality. Figure 2 shows a megasporinite grain and Figure 3 shows a grain of cutinite and vitrinite.

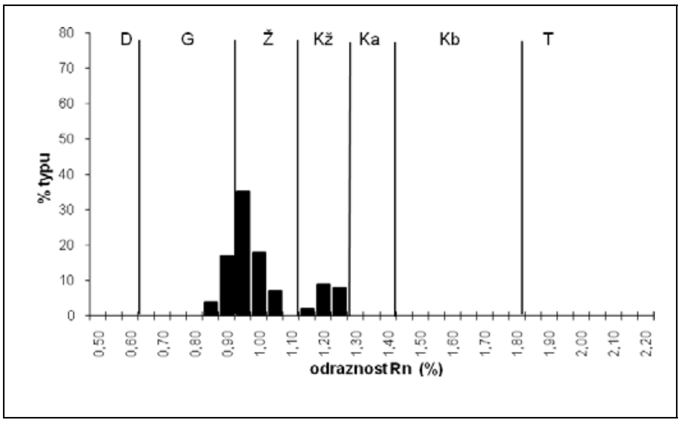

Fig. 1: Reflectivity of ČSA Mine black coal

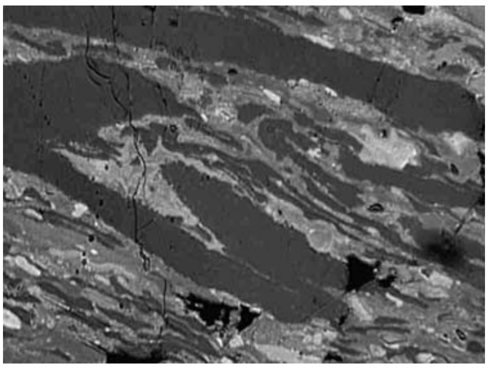

Fig. 2: Megasporinite grain –ČSA Mine

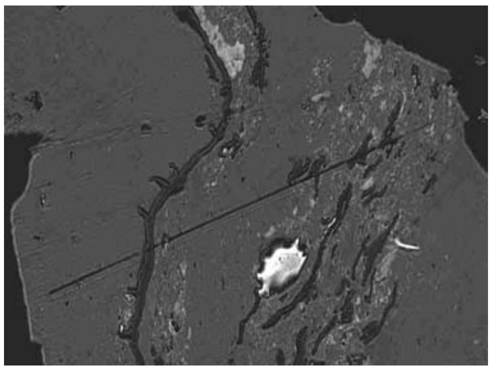

Fig. 3: A grain of cutinite and vitrinite (ČSA Mine)

3. METHOD AND RESULTS OF FLOTATION TESTS

For all the pyrolysis tests a laboratory pyrolytic unit was used, built in IRSM AS CR Prague. Pyrolysis took place in a quartz reactor (ø 60 mm, length 450 mm) placed into a vertical tube oven controlled by a microprocessor.

Conditions of Pyrolysis:

Heating rate: 5 K·min^{-1} up to the final temperature of 900°C, soaking time 30 minutes at 900°C.

The sample weight was 50 g (grain less than 3 mm). For pyrolysis tests we used walnut shells, peach stones, green beans, poppy heads, peanut shells, onion peel, corn leaves and apple tree leaves. Figure 4 shows equipment used for the pyrolysis tests.

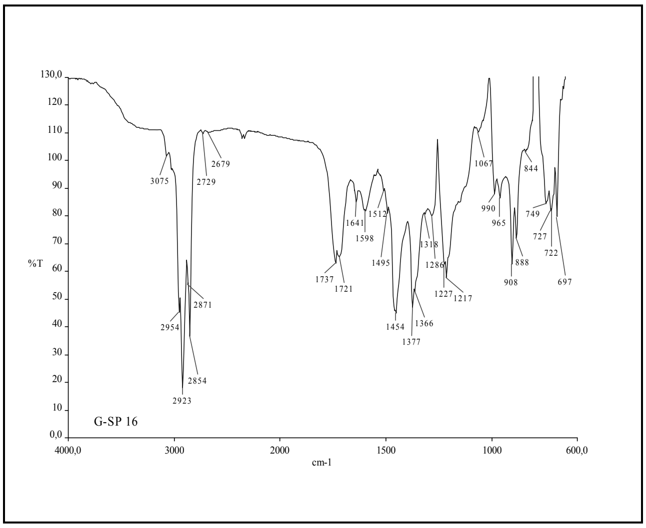

Fig. 5: FTIR – analysis of pyrolysis oil obtained through pyrolysis of mixed plastics and black coal

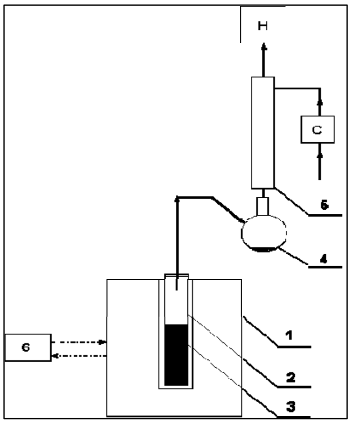

Fig. 4: Laboratory pyrolytic unit

1 – vertical tube oven, 2 – quartz reactor, 3 – pyrolyzed sample, 4 – flask with liquid products, 5 – cooler, 6 – programmable temperature control unit, H – gas holder with gas volume registration, C – cooling unit

The characteristics of the obtained oils were analyzed in the laboratories of VUCHEM VŠB-TU Ostrava using an infra-red (IR) spectrometer. To be illustrative, only two IR analysis results are given, namely of sample G-SP16 (oil obtained through pyrolysis of mixed plastics and black coal from Lazy Mine) and sample G-PR18 (oil obtained through pyrolysis of tyres, rubber and black coal from Lazy Mine) as shown in Figures 5 and 6.

It was 20 % of mixed plastics and 80 % of black coal that were used. The analysis results imply that the oil lacks the characteristic vibrations of hydroxyls in a water molecule, which means that it has a minimum water content. In the sample there is a **significant representation of C-H** bonds present in the unsaturated hydrocarbons (4 peaks in 2954 – 2871 cm^{-1} spectrum area); next, there is carbonyl in 1737 – 1721 cm^{-1} spectrum area**. The presence of aromates is also possible** (vibrations over 3000 cm^{-1}, overtones around 2000 cm^{-1}, peaks around 1600, 1500 and 1400 cm^{-1}).

Pyrolysis was carried out combining 85 % of black coal and 15 % of tyre and waste rubber mixture. The FTIR results imply that in the band 3372 cm^{-1} **the presence of water is possible, in bands** 2953 – 2855 cm^{-1} C-H bonds were identified and around 1700 cm^{-1} band **carbonyl** was identified**. In the sample the presence of aromates is possible** (vibrations over 3000 cm^{-1}, overtones around 2000 cm^{-1}, peaks around 1600, 1500 and 1400 cm^{-1}).

In the majority of the samples the presence of C-H bonds was proved in saturated hydrocarbons (sharp vibration bands between 2800 and 2960 cm-1). The highest intensity of the peaks was in case of mixed plastic sample and the sample with used combination of tyres and rubber. In the majority of the sample the presence of the carbonylic group C=O (around 1700 cm-1) in ketones or carboxylic acids was determined. The presence of aromates was determined in the samples of plastics, tyres and rubber.

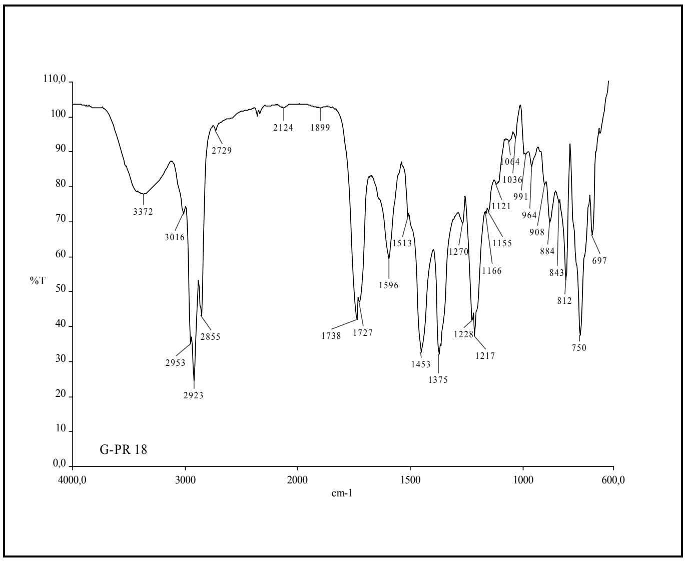

Fig. 6: FTIR analysis of pyrolysis oil obtained through pyrolysis of black coal and mixture of tyres and rubber

4. METHODOLOGY OF THE FLOTATION TESTS

Flotation tests were carried out in the laboratory of the Institute of Environmental Engineering of the Mining College - Technical University of Ostrava using a laboratory flotation machine VRF-l, a product of RD Příbram. It is an agitating flotation machine with an own air intake. Oils produced during pyrolysis were used to test flotation oil selectivity and were compared with Montanol 551 collector. Flotation was carried out under the following conditions:

- pulp density: 150 g/l
- collector dose: 500 g/t.
- agitation time with agents: 1 minute
- flotation time: 5 minutes

Tables 2 to 12 show the results of the individual experiments:

Table 2: *Flotation test results with Montanol 551*

Montanol 551	yield	ash content
	%	%
K	94,02	8,14
O	5,98	78,17
P	100	12,33

K – concentrate, O – tailings, P – flotation input

Table 3: *Flotation results with oil G-SP16 (black coal from Lazy Mine + mixed plastics 20 %)*

G-SP16	yield	ash content
	%	%
K	78,17	4,67
O	21,83	41,97
P	100	12,81

Table 4: *Flotation results with oil G-SP18 (black coal from Lazy Mine + rubber 15 %)*

G-SP18	yield	ash content
	%	%
K	70,34	5,13
O	29,66	24,22
P	100	10,79

Table 5: *Flotation results with oil G-SP17*

G-SP17	yield	ash content
	%	%
K	62,57	8,00
O	37,43	16,73
P	100	11,27

Table 6: *Flotation results with oil G-PR19*

G-PR19	yield	ash content
	%	%
K	59,11	6,22
O	40,89	21,27
P	100	12,37

Table 7: *Flotation results with oil 4G8-PO1*

4G8-PO1	yield	ash content
	%	%
K	81,82	6,59
O	18,18	44,01
P	100	13,39

Table 8: *Flotation results with oil GSP16+Montanol*

GSP16+Montanol	yield	ash content
	%	%
K	93,64	11,02
O	6,36	64,66
P	100	14,43

Table 9: *Flotation results with oil GSP17+Montanol (1:1)*

GSP17+Montanol	yield	ash content
	%	%
K	95,68	10,7
O	4,32	78,35
P	100	13,62

Table 10: *Flotation results with oil GPR18+Montanol (1:1)*

GPR18+Montanol	yield	ash content
	%	%
K	94,63	9,31
O	5,37	89,16
P	100	13,60

Table 11: *Flotation results with oil GPR19+Montanol (1:1)*

GPR19+Montanol	yield	ash content
	%	%
K	94,57	9,83
O	5,43	76,14
P	100	13,43

Table 12: *Flotation results with oil 468-PO1+Montanol (1:1)*

468-PO1+ Montanol	yield	ash content
	%	%
K	95,01	10,43
O	4,99	66,54
P	100	13,23

5. CONCLUSION

The objective of the paper was verification of floatability of the classical collector Montanol 551 and pyrolysis oils obtained through pyrolysis of waste, namely mixed plastics, tyres and waste rubber, combined with Lazy Mine black coal in black coal flotation. Black coal from ČSA OKD, a.s. coal preparation plant was used for flotation tests. The paper results imply that it is possible to produce collectors from waste materials which may be applied in the flotation of black coal. Combining pyrolysis oils and Montanol 551 in 1:1 proportion we succeeded in obtaining excellent quality of flotation concentrates, at the concentrate yield over 90 %. It is thus apparent that each reduction in Montanol 551 consumption may bring a significant economic effect.

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Scientific Reviewers:
Prof. PhD. Eng. Sanda Krausz

EFFECT OF AURIFEROUS PYRITES ROASTING IN MICROWAVES FIELD, ON THE CIANYDATION RESULTS

SANDA KRAUSZ[*]
NICOLAE TOMUS[**]
LILIANA CIOBANU[**]
EUGEN CRACIUN[***]
STEFAN CRACIUN[***]

Abstract: *Using the microwaves energy was extended in the last times its applications being numerous and in various domains like extractive metallurgy. It is also known that it was recorded a wide change in the different materials behavior under the microwaves action. Based on this fact it have been studied the possibility of the coarse ores and concentrates pretreatment with microwaves before applying the hydro or pirometallurgical processes and also before applying some processing methods on the different wastes of different compositions. In that context it's registered our research that wants to establish the roasting effect on the auriferous pyrites in the microwaves field, by comparing the obtained results on the cyanidation of the roasted and unroasted pyrites.*

1. INTRODUCTION

The sulphide concentrates roasting is a pirometallurgical operation frequently applied having as goal the removal of the sulphur excess but also to realize some material transformations by heating the material under the melting temperatures. Except the roasting operation applied for the material drying, that don't produce any structural or chemical changes, all other roasting processes have as results a chemical transformation (decomposition, redox operations, etc.) of the material that is prepared in this manner for the metal extraction phase. It is also assured by this way the thermal breakup of the sulphides, and the elimination of the gaseous compounds like: S_2, O_2, As_2O_3, SO_2, followed by the apparition of some pores, microchanells and cracks that insures a good penetration of the attack solution (Kheil, Golcea, Krausz, 2006).
The precious metals recovery from the pyrites that contains such metals can be improved by a preliminary roasting.

* *Prof. Ph.D. Eng. - University of Petrosani*
** *Research and development National Institute for Metals and Radioactive Resources*
*** *S.C INTEC SA*

As it is known, the pyrite breakdowns at 740°C and the sulphur is liberated as it is shown in the following reaction:

$$FeS_2 = FeS + \frac{1}{2} S_2 \quad \Delta G^o_{743} = 14110 \text{ cal/mol} \qquad (1)$$

The iron sulphides oxidation leads mainly to form oxides as Fe_2O_3 and Fe_3O_4, but it is possible to form a thin layer of ferrous sulphate that interfere with the residual FeS in a very fast reaction, that can be resume in the following reactions establish by Pinkovski (aped Frenay,1994):

$$FeS + 2O_2 = FeSO_4 \qquad (2)$$
$$FeS + 3FeSO_4 + O_2 = 2Fe_2O_3 + 4SO_2 \qquad (3)$$
$$2FeSO_4 + SO_2 + O_2 = Fe_2(SO_4)_3 \qquad (4)$$

The FeS obtaining is realized by the FeS_2 breakdown at 1000°C in argon atmosphere. Then, the FeS roasting begin at 320°C, with the ferrous sulphate formation which is discomposing partially in the reaction with the residual FeS that suffer a thermal decomposition at 850°C and the SO_2 liberated initiates the ferric sulphate formation. Vary researches shown that is excluded the primary formation of the ferric sulphate, this one is formed with no doubt in the reaction (4) (Bolgiu, Dumitrescu and Thornhill, Pidgeon, apud Frenay, 2004).

For some compounds that appear during the pyrites roasting we know the exact values of the thermodynamic functions. The pyrites roasting occur in an oxidizing atmosphere and that fact shows the type of the stable compounds that forms the roasting gases: O_2, N_2, SO_2, H_2O and CO_2. Between all these gases appear only one interaction rended by the following reaction:

$$SO_2 + 1/2\, O_2 = SO_3 \qquad (5)$$

Due to its avidity, SO_2 can be cached, by gases leading, in a buffer bowl containing $Ca(OH)_2$, at pH = 12; in this case it's obtained the sulphurous acid:

$$SO_2 + H_2O = H_2SO_3 \qquad (6)$$

After its interaction with the $Ca(OH)_2$ the solution pH decrease at 3,5 and take place the following reaction:

$$Ca(OH)_2 + H_2SO_3 = CaSO_3\downarrow + 2H_2O \qquad (7)$$

The actual knowledge regarding the cyanidation process show that only a small part of gold is leached after the Elsner reaction, the mainly part is leached in the second phase (Adamson, 1970), in conformity with the reaction bellow:

$$4Au + 8CN^- + O_2 + 2H_2O = 4Au(CN^-)_2 + 4OH^- \quad (8)$$
$$2Au + 4CN^- + O_2 + 2H_2O = 2Au(CN^-)_2 + H_2O_2 + 2OH^- \quad (9)$$

The gold solubilization – as any other – depend on the cyanide solution concentration, on the solution alkalinity, pH and on the temperature. Our investigation has as target the overlap of the thermochemical effect expressed by the reactions from above with the influence of the microwaves action, to the auriferous pyrite preparation for the precious metals extraction.

2. MATERIALS AND INSTALLATIONS USED

2.1 The characteristics of the sample material

The experimental research has been realized on a sample taken from the auriferous pyrite concentrate deposit of the Central Flotation Plant – Baia Mare; the chemical composition of this pyrite (the main components) is presented in table 1.

Table 1. *Chemical composition of the auriferous pyrite sample (main compounds)*

Substance	Content %	Substance	Content %
SiO_2	9,38	Pb	0,57
TiO_2	0,80	CaO	0,80
Al_2O_3	0,06	MgO	0,06
Fe	39,16	K_2O	0,45
P_2O_5	1,70	Au	8,4*
S	42,30	Ag	59*

* în g/t

The difractometric analyze show the presence of the following minerals: pyrite, chalcosine, native gold, aurostibil, stromeyerit, pyrostilpnit and subordinated quartzite, quartz, feldspar, galena, blende, antimonite, magnetite, heavy minerals. It has been establish their balance on the granulometric classes.

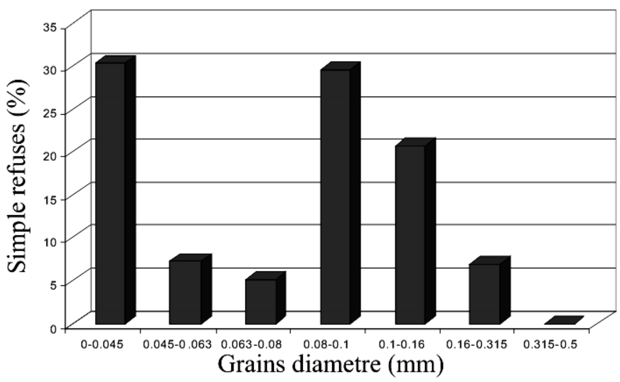


Figure 1. The percentage balance of the granulometric fractions

The results of the granulometric analyze are presented in figure 1 by percent balance of the granulometric classes and this show a relative irregular granulation of the material, the predominant classes being those under 0,045 mm, 0,063 – 0,08 mm şi 0,8 – 0,16 mm.

To remark the pyrite roasting effect in different conditions, from the pyrite concentrate have been separated 3 samples with identical characteristics; one of the sample is called “raw pyrite” and the others two have been submitted to a roasting process in the microwave field at 400^{o}C and 700^{o}C.

2.2 The experimental installation

Because the auriferous pyrite concentrate has a very good electromagnetic coupling with the microwave radiation, it have been design and realized a roasting installation in fluidized bed that contain: the microwave generator, the transmission line of wave guide type, the microwave applier – the process reactor in fluidized bed, the fluidized agent source, the gas-powder separator as cyclone and the neutralization reactor for the products issue from the reaction (figure 2).

The temperature variation inside the microwave applier during the roasting process is presented on figure 3. This variation must be known because it influence in an appreciable manner the dissolution speed of the gold due to the activation energy needed in the process and due to diminishing of the oxygen solubility with the temperature increasing.

To remark the roasting effect on the samples, each of them have been chemically and on the electronic plunger analyzed before and after processing.

1. Panou comanda reglare magnetron
2. Reflectometru
3. Ghid de unda
4. Fereasta din cuart transparenta la microunde
5. Reactor fluidizare
6. Rotametru
7. Ciclon
8. Manometru
9. Compresor
10. Sistem de neutralizare
11. Termocuplu
12. Camera de fluidizare

Figure 2. The microwave heating installation in fluidized bed

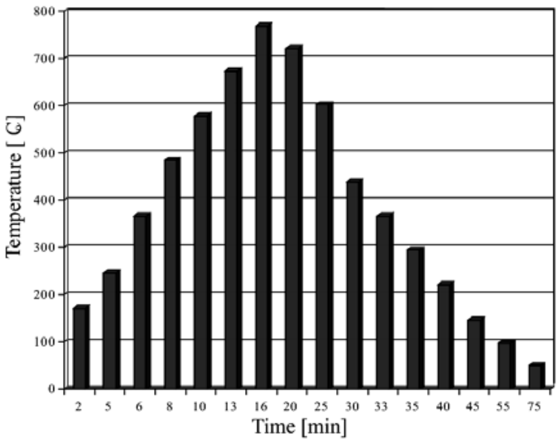


Figure 3. Temperature variations inside the microwave applier during the pyrite roasting

3. EXPERIMENTAL RESEARCHES

The experimental investigation passes through three steps: the first consist in the raw pyrite cyanidation, in the second the pyrite is roasted at 700°C and in the third the pyrite is roasted at 400°C.

3.1 The raw pyrite cyanidation

It has been realized by the "rolling bottle" method in the following work conditions:

To the pyrite sample (0,5 kg) have been added 750 ml NaCN solution containing 400 ppm CN^- ions (1,5 g NaCN for 2000 ml water). The pH value was initially 10,3 and after 5 minutes of agitation drop to 1,44. As a consequence, it was added a lime solution; the addition of 55 g of lime (corresponding to a specific consumption of 100kg/t) ensure a Ph of 12,1. The solution temperature has been maintained at 24,7°C.

All along the solubilization process it has been followed the concentration of the CN^- free ions (by titration with a $AgNO_3$ solution) and the gold concentration passed in the solution after 2, 24 48 and 72 hours.

The CN^- content in the initial leaching is high (400 ppm) but after two hours of stirring its drop at 100 ppm, and for that reason it have been added a new quantity of NaCN to restore the initial concentration of the CN^- to 400 ppm.

After 24 hours of stirring, the CN^- concentration drops again to 6,24 ppm, went it have been added again 1,4 g of NaCN; the CN^- concentration rise to 340 ppm and rested nearly constant after 72 hours of stirring.

The gold content increasing from 0 to the maximal value of 2.72 mg/l (figure 3) is realized in time with two different values; so, in the first two hours the increasing is from 0 to 0,68 mg/l, with an average of 0,34 mg/l, and in the 22 hours that follows to 2,05 mg/l, with an average of 0,06 mg/l; its take place a normal and predictable diminution of the dissolution speed relayed to the diminution content of the free CN^-. Between 24 and 48 hours the gold concentration increasing is much more slow, from 2,05 to 2,18 mg/l, with an average of only 0,005 mg/l,h, even if there is an increasing of the CN content by adding NaCN. In the last period of time, the dissolution speed increase to 0,023 mg/l,h, at the same time with the significant increasing of the CN concentration, that the effect is diminished (and then maintained constant) due to the existing and available gold from the pyrite surface that pass in the solution (figure 4).

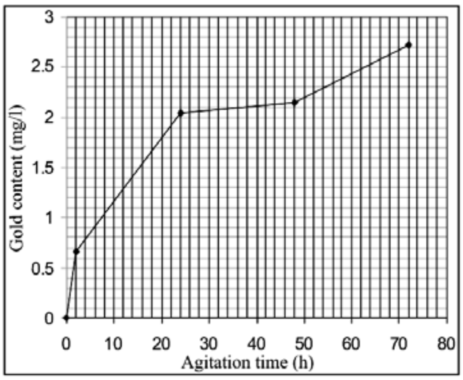


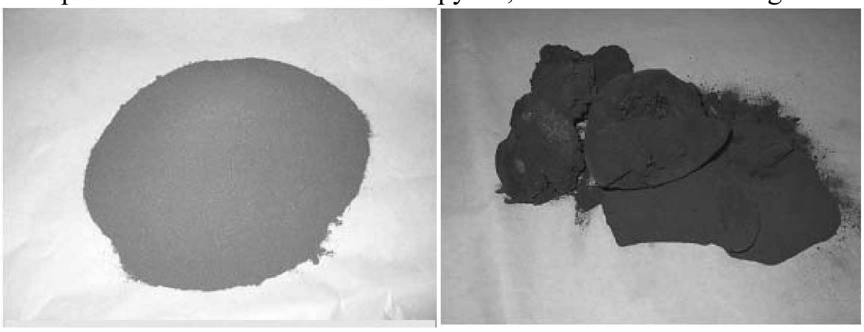
Figure 4. The variation in time of the gold content from the solution, at the raw pyrite cianidation

The total NaCN consumption for the raw pyrite leaching has been of 2,8 kg/t.

The leaching efficiency has been calculated taking into account the initial gold content from the 8,4 t of pyrite; from the quantity of 0,004 g of gold that can be find in the 500 g of pyrite sample submitted to the cyanidation process, it have been dissolved, after 72 hours only 2,72 mg/l x 0,75 l/1000 = 0,002 g; it result an efficiency of 50%.

3.2 The pyrite cyanidation after roasting in the microwave field, at 700°C

A pyrite sample has been roasted at 700°C, in the previous installation. After roasting, the pyrite aspect was different from the raw pyrite, as it can be seen in figure 5.



a) b)

Figure 5. The raw pyrite aspect (a) and the roasted (at 700°C) pyrite aspect (b)

The pyrite leaching was realized in identical conditions, with the same quantity of material.

The solution pH diminution after 5 minutes of contact with the sample occurred from 10,3 to only 6,72; its correction to the value of 12,4 was realized with a smaller quantity of lime, of only 10 kg/t.

It have been followed the same parameters, at the same periods of time and the CN^- concentration maintenance was realized by adding new quantities of NaCN from time to time. The totally adding of NaCN was about 0,853 g that represent a specific consumption of 1,706 kg/t.

With an eye on the dissolution process dynamic (represented in the figure 6), it can be observed that in the first period of time of 2 hours, it occurred the same pronounced increasing content of gold in the solution from 0 to 1,63 mg/l (0,815 mg/l,h) and the diminishing of the CN^- to 108 ppm. It have been added NaCN (0,37g) and realized a concentration increasing to 139 ppm. In 24 hours. During the same period, the gold content that dissolve in the solution increase to 3,28 mg/l realizing an average dissolution speed of 0,075 mg/l,h. adding a new quantity of NaCN (0,33 g) ensure, after 48 hours of stirring, a CN^- concentration of 280 ppm but the gold dissolution run slower until 3,49 mg/l, with an average speed of 0,008 mg/l,h. Adding a new NaCN quantity (0,153 g) it has realized an increasing of the gold content in the solution about 4,32 mg/l with an average og 0,035 mg/l,h.

As it can be determined, the evolution in time of the dissolution process is similar but the content of the dissolved gold is considerable higher, recording an extraction efficiency increasing from 50 % to 81 %, with 31 % higher than in the case of the raw pyrite. It can be added to that the advantage of a considerable reducing of the specific consumption of alkalinization reagents and NaCN.

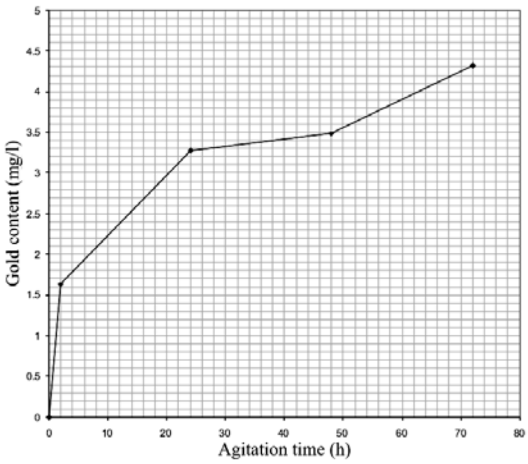


Figure 6. The variation in time of the gold content from the solution, at the cyanidation of the roasted pyrite at 700°C.

3.3 The pyrite cyanidation after roasting in the microwave field, at 400°C

The aspect of the roasted pyrite at only 400°C can be seen in figure 7. It defers essentially from the roasted pyrite at 700°C and that can suggest a different behavior during the dissolution.

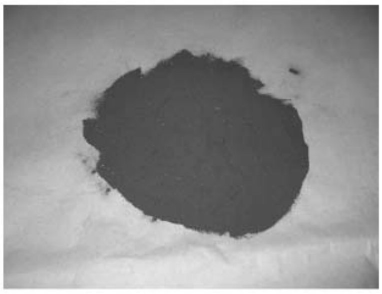


Figure 7. The aspect of the roasted pyrite at 400°C

The cyanidation has been realized in the same conditions. The pH value drop, in the first 5 minutes of stirring from 10,3 to 2,52; to ensure an alkalinity of 12 it was used a quantity of 45 g of lime, meaning 90 kg/t. By the same modality it was followed the free CN^- and the gold content dissolved in the solution. The NaCN added, step by step, was this time of about 2.796 kg/t.

It have been remarked (figure 8) that, after 2 hours of cyanidation, the gold content was about 1,55 mg/l, so, the dissolution have been realized with an average of 0,775 mg/l,h. After 24 hours of stirring the gold content increased to 2,06 mg/l, with an average of 0,023 mg/l,h, after 48 hours to 2,15 mg/l (0,0035 mg/l,h) and finally, after 72 hours to 2,88 mg/l 0,03 mg/l,h). The gold extraction efficiency was about 54 % in the conditions of a small reduction of the specific consumptions of lime and NaCN than these needed for the raw pyrite.

The comparing results obtained can be seen in table 2 and their show the important effect of the pyrite roasting in the microwave field, with the condition that the pyrite is roasted at 700°C.

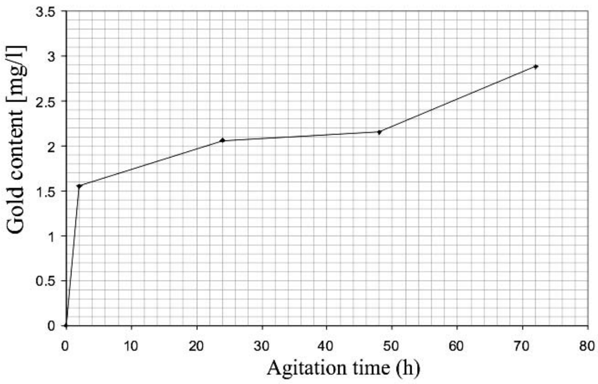


Figure 8. The variation in time of the gold content from the solution, at the cyanidation of the roasted pyrite at 400°C

Table 2 *The parameters of the cyanidation process and the comparative results*

Parameters, indices	Raw pyrite	Roasted pyrite 700°C	Roasted pyrite 400°C
Sulphur content, %	42,3	15	25
Final gold content , mg/l	2,72	4,32	2,88
Dissolution average speed, mg/l,h	0,038	0,06	0,04
Gold extraction efficiency, %	50	81	54
Specific lime consumption, kg/t	110	10	90
Specific NaCN consumption, kg/t	2,8	1,7	2,796

5. CONCLUSIONS

This research presents the first essays to establish the possibility of using the roasting in the microwaves field, to the auriferous pyrites roasting with low content of precious metals, in view to improve the metal extraction at the cyanidation. As we can see, it has been confirmed the assumption of a benefic effect of that processing on the gold extraction efficiency.

The roasting of that type of pyrites concentrates in these conditions is possible due to the fact that these materials are microwave absorbents.

The research results shown in table 2 remark that is important to roast the material at 700°C because in that case the material has a reduce content of sulphur (from 42,3 to 15 %) and this ensure a faster reduction of the pH and automatically a lower consumption of the alkaline reagent. On the other side, the pyrite roasting at that temperature ensure the formation of cracks, microchannels and pores, that offer a contact surface much bigger with the NaCN solution and reduce the specific consumption of NaCN and increase the gold extraction.

The encouraging results obtained recommend the pursuit of these researches in view to optimize the working parameters to increase the gold extraction.

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Scientific Reviewers:
Assoc. Prof. PhD. Eng. Dorina Baciu

GLOBAL POLLUTION INDEX – GPI – EVALUATION ALGORITHM

ADRIAN FLOREA*

Abstract: *The paper deals with the concept of Global Pollution Index – GPI – and the algorithm developed in order to compute the GPI so this become a versatile tool for assessing the environment quality. After a briefly presentation of the global pollution index method and the computing algorithm, an example for GPI calculus is presented.*

Key words: *Global Pollution Index - GPI, environment quality, environmental parameters, global environmental quality assessment*

1. INTRODUCTION

The necessity of a global evaluation method of environment quality developed in order to assess the environmental impact of various human activities and to follow the evolution of pollution phenomenon in time.

Globally there were several attempt to assess the environmental quality through synthetic indicators, which usually refer to a single environmental factor, like: quantity of pollutants depleted in air or water, expressed with chlorine index, soil pollution with heavy metals, expressed with zinc equivalent, etc.

The simple counting of each factor status (air, water, soil, human health) in some cases is not enough, because it does not underline the interdependences between environmental components and the synergic effect in ecosystems behavior.

Such a method has to facilitate the comparison between the environment status at a moment and a previous or future environment status and needs to support the regional environmental quality mapping process.

Therefore, it was developed a method to judge the environmental health and pollution status and to estimate the value of this status based on an indicator obtained from the report between the ideal value and the given value at a moment of some quality indicators, considered essential for the analyzed environmental factors (Rojanschi, 2004).

* *Assoc. Professor Eng., Ph.D. at the Mining Engineering Faculty, University of Petroşani, E-mail: adflorea@upet.ro*

2. THE GLOBAL POLLUTION INDEX METHOD

The method implies several steps of synthetic evaluations based on quality indicators able to reflect the general status of every analyzed environmental factor and then their graphical representation.

This way there is evaluated the quality of every environmental factor at a moment according to a quality scale and it receives a mark that reflects *de facto* situation, compared to ideal situation. The quality scale is graded from 1 to 10, where 10 represents the natural status, unaffected by human activity and 1 represents the irreversible situation with very severe deterioration of the analyzed environmental factor.

Usually, we consider that it is possible an environmental evaluation from a certain area at a moment using the following parameters:

- Air quality,
- Soil quality,
- Water quality,
- Human health,
- Deficit of plants and animals species (biodiversity index).

Every environmental professional who uses this method may use also other parameters (ex. landscape, socioeconomic ambient, etc.) with relevance for the analyzed situation.

Every of these environmental factors may be characterized by several quality indicators with known admissible limits in order to evaluate the pollution degree. According to measured values, they are given quality marks. These quality marks received by every environmental parameter from the analyzed area will serve to draw a graphical diagram, as a method of synergic effect simulation.

The global pollution index of an ecosystem – GPI – is the result of the rapport between the ideal status area and the real status area. According to the computed value of GPI, the environment fits one of the following quality categories:

Table 1. *Environmental quality categories according to GPI value*

GPI value	Effect on environment quality
1	Natural environment unaffected by human activities
1÷2	Environment subject to effect of human activities in admissible limits
2÷3	Environment subject to effect of human activities, generating a discomfort grade to life forms
3÷4	Environment subject to effect of human activities, generating an alteration grade to life forms
4÷6	Environment bad affected by human activities, dangerous to life forms
> 6	Degraded environment, inappropriate to life forms

The graphical representation of ideal status is a regular geometrical shape with equal radius that has a value of 10 quality units. Joining the points resulted from real status assessment obtain an irregular geometrical shape with a smaller area, inside the regular shape associated with ideal status (fig. 1), where P1÷P8 are the analyzed environmental parameters. If there is no modification for the quality of environmental factors, so there is no pollution, the GPI value is equal to 1 and the graphical representation of real and ideal status is practically the same.

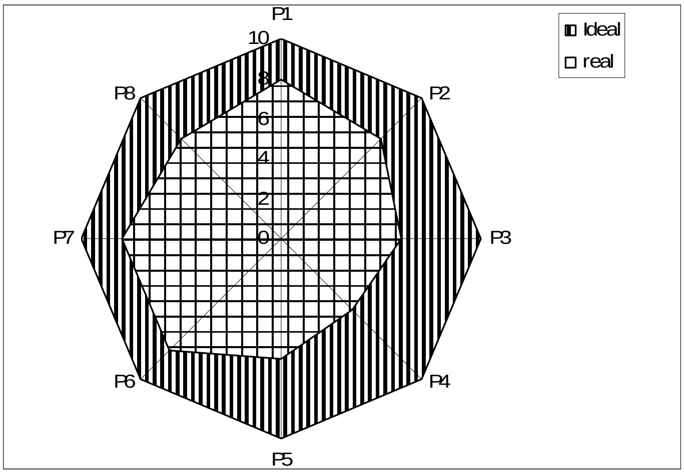

Fig. 1. Graphical representation of GPI

3. COMPUTING ALGORITHM OF GLOBAL POLLUTION INDEX

The analytical determination of GPI is simple. As mentioned above, first we choose the "n" (where $n \geq 3$) environmental parameters and then we give quality marks between 1 and 10 to each parameter. The resulted surface is a polygon with "n" sides formed by "n" triangles that have a common point named pole and the correspondent angle equal with 360°/n. The area of "n" side's polygon is equal to the sum of the triangles area that composes the polygon. Therefore, the problem consists in the determination of the area for each individual triangle component (Florea, 2010). There are two situations:

a) Each of "n" parameters has the quality mark equal to 10. In this case the polygon area "A" reflect de ideal situation and may be calculated as sum of each triangle area "$A_{\Delta i}$" with the following relation:

$$A = n \cdot A_{\Delta i} \tag{1}$$

b) The "n" parameters have different quality marks in the range 1 to 10. In this case the real situation reflected by the polygon area "A" may be calculated as sum of each triangle area "$A_{\Delta i}$" with following relation:

$$A_r = \sum A_{\Delta i} \tag{2}$$

Lets take into consideration the triangle „i" which has two of the sides equal to the quality notes for the parameters "i" respectively "i+1" (fig.2).

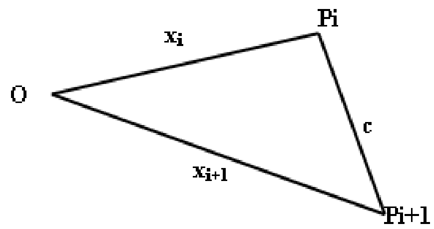

Fig. 2. Triangle „i”

In each of “n” triangles we know two sides, x_i respectively x_{i+1}, which represent the quality notes for the parameters “i” respectively “i+1” and the angle between them. The third side, c, is determined applying cosines theorem in any triangle (Bachmann, 1980), using following relation:

$$c = \sqrt{x_i^2 + x_{i+1}^2 - 2 \cdot x_i \cdot x_{i+1} \cdot \cos(\tfrac{2\cdot\Pi}{n})} \tag{3}$$

The area of triangle „i” is calculated with the Heron formula:

$$A_{\Delta i} = \sqrt{S \cdot (S - x_i) \cdot (S - x_{i+1}) \cdot (S - c)} \tag{4}$$

where „S” is determined with the relation:

$$S = \tfrac{1}{2} \cdot (x_i + x_{i+1} + c) \tag{5}$$

For the ideal case, when $x_i = x_{i+1} = 10$ the relation (3) becomes:

$$c = 10 \cdot \sqrt{2 \cdot (1 - \cos(\tfrac{2\cdot\Pi}{n}))} = 20 \cdot \sqrt{\frac{1 - \cos(\tfrac{2\cdot\Pi}{n})}{2}} \tag{6}$$

and the relation (5) becomes:

$$S = \frac{1}{2} \cdot (10 + 10 + 20 \cdot \sqrt{\frac{1 - \cos(\tfrac{2\cdot\Pi}{n})}{2}}) = 10 \cdot (1 + \sqrt{\frac{1 - \cos(\tfrac{2\cdot\Pi}{n})}{2}}) \tag{7}$$

while the relation (4) becomes:

$$A_{\Delta i} = 100 \cdot \sqrt{\frac{1 - \cos(\tfrac{2\cdot\Pi}{n})}{2} \cdot \frac{1 + \cos(\tfrac{2\cdot\Pi}{n})}{2}} = 100 \cdot \sqrt{\frac{1 - \cos^2(\tfrac{2\cdot\Pi}{n})}{4}} =$$

$$= 100 \cdot \sqrt{\frac{\sin^2(\tfrac{2\cdot\Pi}{n})}{4}} = 100 \cdot \frac{\sin(\tfrac{2\cdot\Pi}{n})}{2} = 50 \cdot \sin(\tfrac{2\cdot\Pi}{n}) = 100 \cdot \sin(\tfrac{\Pi}{n}) \cdot \cos(\tfrac{\Pi}{n}) \tag{8}$$

So the area of the ideal polygon becomes:

$$A = n \cdot A_{\Delta i} = 50 \cdot n \cdot \sin(\tfrac{2\cdot\Pi}{n}) = 100 \cdot n \cdot \sin(\tfrac{\Pi}{n}) \cdot \cos(\tfrac{\Pi}{n}) \tag{9}$$

We have to mention that for the last triangle, "n", which the two sides equal to quality marks for parameter "n" respectively "1", in the above relations the value x_{i+1} becomes x_1, so $x_{n+1} = x_1$.

The logical scheme for the computing algorithm of global pollution index – GPI is shown in the figure 3.

Table 2 presents synthetically an example for the algorithm application. In the first row of the table is the order number of the parameter y_i, mentioned in the second row. The third row shows the quality marks x_i, given to each environmental parameter taken into consideration, and in the following rows there are the calculated values with relations (3), (5) respectively (4) for c, S respectively A_i. In the last cell of the last row is the area of the real situation computed as sum of the five triangle areas.

Table 2. *Example of algorithm application for GPI calculus*

i	**1**	**2**	**3**	**4**	**5**	Real polygon area
Parameter, y_i	Water	Air	Flora	Fauna	Landscape	
Quality marks, x_i	7.00	8.00	7.00	7.50	6.50	
c	8.85	8.85	8.53	8.27	7.95	
S	11.93	11.93	11.52	11.13	10.72	
A_i	26.63	26.63	24.97	23.18	21.64	123.04

The area of the ideal polygon was calculated with the relation (9), so:

$$A = 100 \cdot n \cdot \sin(\tfrac{\Pi}{n}) \cdot \cos(\tfrac{\Pi}{n}) = 100 \cdot 5 \cdot \sin(\tfrac{\Pi}{5}) \cdot \cos(\tfrac{\Pi}{5}) = 237.76 \qquad (10)$$

As mentioned above, the global pollution index of an ecosystem – GPI – is the result of a rapport between the ideal status area and the real status area:

$$\text{GPI} = 237.76 / 123.04 = 1.932 \qquad (11)$$

In this situation $1 < \text{GPI} < 2$ and the environmental quality category for this value is "**Environment subject to effect of human activities in admissible limits",** according to table 1.

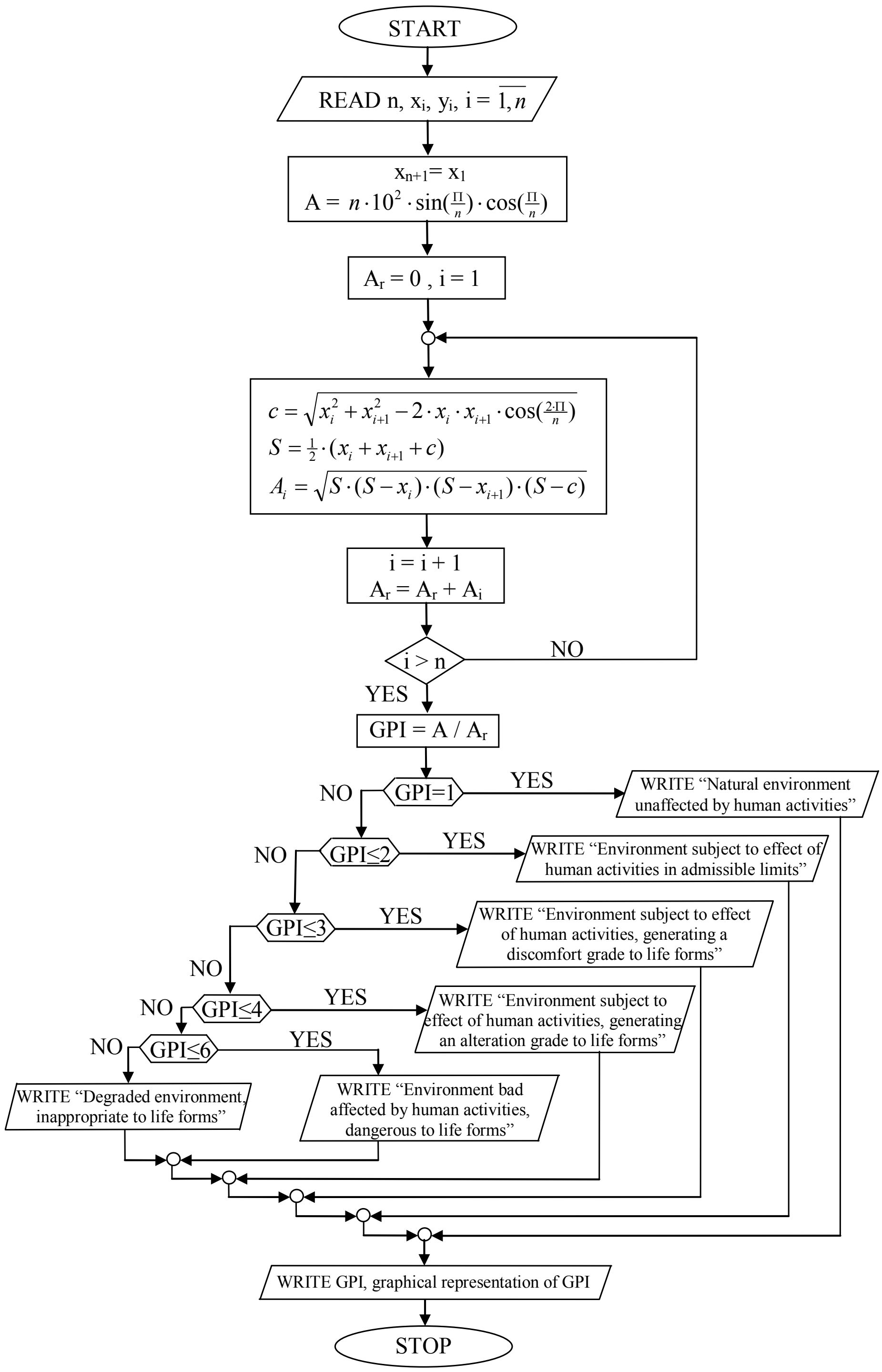

Fig. 3. The logical scheme for computing algorithm of global pollution index – GPI

The graphical representation of GPI for the analyzed example is presented in figure 4. The ideal area is represented by hatched polygon, while the interior polygon represents de real area.

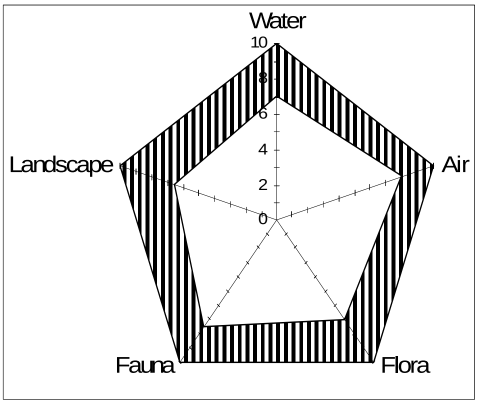

Fig. 4. Graphical representation of GPI for the analyzed situation

4. CONCLUSIONS

The global pollution index method – GPI – is an environmental quality status evaluation method based on various environmental parameters (air, water, soil, flora, fauna, landscape, human health, socioeconomic ambient, etc.) analyzed in a certain region at a moment, which allows an objective tracking in time of the pollution phenomenon in that region. The method also allows a comparison of the environmental quality from several regions subject of analyze, at a moment.

The mathematical model described in the computing algorithm of global pollution index allows the analytical determination of GPI value for the case of analyzing any number of parameters (for $n \geq 3$), with any values for quality marks of these parameters. The granting process of quality marks implies an adequate number of data related to analyzed parameters, data provided by an environmental monitoring program.

The logical scheme from figure 3 which describes the computing algorithm of global pollution index may be included into a computer code or into an Excel spreadsheet (Somnea, 1994) and has the purpose to become a versatile and easy to use tool for the environmental professionals who use this method for global environmental quality evaluation.

Finally, we need to mention that this method of global environmental quality evaluation may be improved. A possible improvement direction is decreasing the subjectivity degree in the granting process of quality marks, process influenced by the experience and exigency of the person who makes the analysis. Another improvement implies the estimation of the limits for all indicators that characterize the environmental parameters and their weight in determining the general quality status of the environment.

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Scientific Reviewers:
Prof. PhD. Eng. Mircea Georgescu

HARNESSING METHANE GAS FROM THE MINES OF JIU VALLEY

FLOAREA DANCI*
OLGA MARKOŞ*

Abstract: *The Jiu Valley mining sector strategy should focus on upgrading mine and/or sectors cost/most efficient. It should be noted that mines remain a chance for the economic future of Jiu Valley. We have a resource of raw materials from coal mines in our country and yet untapped: methane gas.*

1. GENERALITIES

First, it should be noted that mines remain a chance for the economic future of Jiu Valley, even if 2010 coal mining subsidy will be withdrawn and in our country and the mines on Jiu Valley will have to be competitive with other countries, which they are far ahead of us. In pur opinion, the Jiu Valley mining sector strategy should focus on upgrading mine and/or sectors cost/most efficient. In terms of organizational structure, superstructure, CNH, believe it is better to be replaced with integrated complex projects such as Itochu Kopex, which involves mines Paroseni, Vulcan preparation and thermal Coroiesti Paroseni.

We also consider appropriate establishment of a specializing company to clean and prepare the location. Operation activities and preparation of coal have major negative impact on the environment in the Jiu Valley. A company that aims to prepare for future ecological and alternative economic activities to areas affected by mining is crucial for the future of the Jiu Valley. This company can and must rely on specialists on CNH, University of Petrosani, INSEMEX and ICPM, which require experience and knowledge for it. Is essential requirement from any companies wishing to do business is to be green"land with clear legal status and access to utilities.

We discussed about closing mines, the greening area, even monitoring after closing, but not post-closing capitalization. We have a resource of raw materials from coal mines in our country and yet untapped: methane gas.Harnessing these resources mean:

* *Student Ph.D. - University of Petrosani*

- Greening (a powerful greenhouse gas is no longer released into the atmosphere);
- Security (we avoid any fire situation on former mines);

Continue exploitation of a profitable economic source of the mines, where coal is taken at least partially, the methane gas.

Coal mining activity produces fractures of rock layers, producing a relaxation of the rock and therefore in that methane issue. Methane is usually removed to allow work safety underground (to avoid explosions and intoxication with methane), but the methane emanation process continue after mine closure and the gas is released in a controlled way, or simply escape to the surface. These issues of greenhouse gas potential of the planet's warming temperature 21 times faster than carbon dioxide for a time horizon of 100 years.

Using methane from coal mines will not only contribute substantially to reduce of greenhouse gas producing fact, but will have a positive effect on regenerating former mining areas, and because the investments are appropriate.

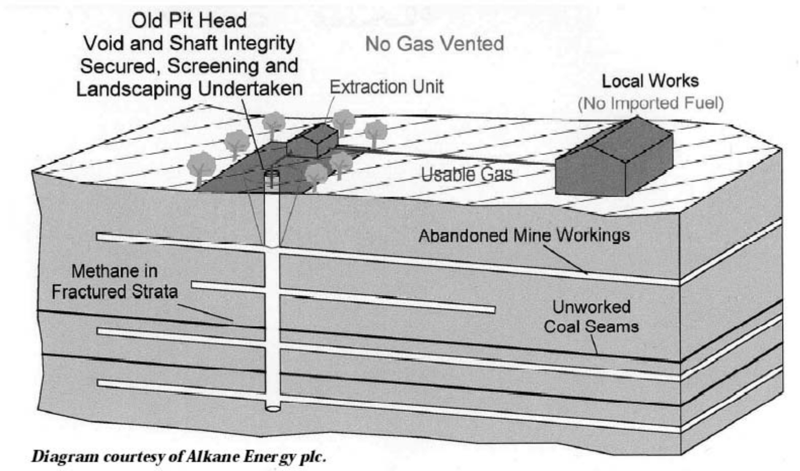

Fig.1. Diagram courtesy of Alkane Energy plc.

2. DIFFERENT TYPES OF GAS FROM COAL MINES

There are three different types of gas from coal mines:

- Gas issued from layers that contain un-used-CH$ on a rate of over 90% and can be collected independently of coal exploitation in some places. Gas composition is generally stable, which means that gas can be fed directly to the gas network.
- Gas from operation of assets, it is a mixture of methane/air released during coal exploitation and to be evacuated for security reasons at work. Usually this type of gas contains about 5-12% oxygen by volume and 25-60% methane. Because the proportion of methane/air can change suddenly from complication arise for use on gas engines.
- Gas from closing mines, even after closing the gas continues to emanate from the surface. Normally closed gas from exploitation does not contain oxygen and composition is relatively stable. Content is 60-80% methane.

Harnessing of methane had some advantages:

- avoid gas issues in atmosphere;
- mine gas is a fuel alternative to conventional;

- very efficient for generating electricity and heat at the central place of mining gas.
- using appropriate technology (eg. Jenbacher engines that use controllers for gas, food and turbo bypass LEANOX system) can provide continuous operation even in conditions of rapid change on gas content mines;
- it depends of using time and location of the plant using heat, gas use in gas engines Mine offers a potential "saving" of CO_2 emission by approximately 30.000 to 40.000 tones/year and MW.

Existing pipelines gas supply are often the most accessible markets for gas if it responds standards mine gas transported by gas suppliers. if gas has a concentration of a least 95% (approximately 31.8 GJ/1000 m3) and contains less than 4% of non-hydrocarbon gas concentration and virtually 0% oxygen, requiring only a minimal processing before compression and injection pipelines gas. Mine gas which does not meet network requirements can be improved removing unwanted constituents, and/or mixing gas with another high calorific value added or propane. A boiler to produce industrial or utilities (heat, hot water) located less than 40km from the mine can be an excellent market. Mine gas is often a fraction of total fuel used by air and because of this flow variations and other constituents besides methane, which is in mine gas composition have a negligible impact. Cogeneration produces methane and other benefits besides saving coal. Benefits of nature environmental protection (reduction of NOx, Sox and particles suspended in air) occurring in methane cogeneration, even if its contribution is small compared to other fuel. Cogeneration provides other operational advantages (easier operation, less fluctuation in operation and a better economic efficiency).

3. POSSIBLE USES OF GAS FOR MINE

The future on mining in the Jiu Valley must aim at major transformation mine gas from a traditional enemy into a friend with an essential contribution to economic revitalization and ecological regenerate the area.

Making a list of possible uses of gas for mine in the technical, economic or simply to reduce environmental impact, they may be:

- Natural gas substituent;
- Direct use of methane;
- Cogeneration (heat production);
- Coal drying in preparation;
- Evaporating salt water;
- Heating building in the premises and mining and air aeration system;
- Other uses by local industry (companies locates within 10 km of gas supply with a continuous demand for gaseous fuel, for example, metallurgical and chemical industry);
- Underground exhaust air oxidation to produce heat;
- Electricity generation and cogeneration;
- Cogeneration (electricity production);
- Internal combustion engines;
- Turbines;
- Cogeneration;

- Fuel cells with similar working principle of batteries (mine gas is used to produce hydrogen necessary operation):
- Oxidation of exhaust air to produce electricity from underground;
- Destruction (complete burning, exhaust air oxidation);

Because in the Jiu valley extends gas supply network considered it mine gas recovery from the mines in this area by injecting gas network.

Jiu Valley eliminates an average 170.000-240.000 m^3/24 hours. Jiu valley mines are guarded by experts INSEMEX in category II, 15 m^3/ hour me flow.

Proof that gas in the Jiu Valley mining gas can be economically exploited is the EM Lupeni Heating, which works with mine gas power recovered by the mine degassing station.

4. CONCLUSIONS

Using methane from coal mines will not only contribute substantially to reduce of greenhouse gas producing fact, but will have a positive effect on regenerating former mining areas, and because the investments are appropriate.

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Scientific Reviewers:
Prof. PhD. Eng. Romulus Sârbu

THE INFLUENCE OF METHANE SOURCES ON CLIMATE CHANGES

OLGA MARKOŞ*
FLOAREA DANCI*

Abstract: *the effects of methane sources on climate changes may be twice higher than we`ve thought in the beginning. The methane sources are varied: swampy zones, gaseous hydrates from the oceans soil, oceans, fresh water micro-organisms, organic coal, hollow waste, oil fields, used or abandoned mines, rice plantation etc.*

1. GENERALITIES

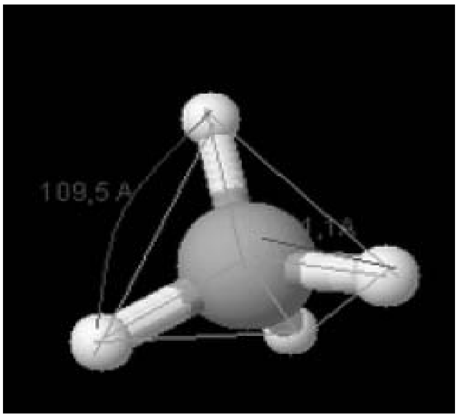

Fig. 1. The geometry of methane molecule

Methane (CH_4) is an inodorous, colorless, lighter than air, it was discovered in 1778 in the mires, that's why it was named "mires gas". The gas can be found on earth atmosphere, in concentration around 1745 parts on billion (1998) growing 700 parts on billion (1750). A group of research workers studied the effects of methane on global warming; they were set about a comparison between the earth and a greenhouse. Even in a cold winter day there is proof that being in a greenhouse there can be nice if the sun is shining. The glass walls let the sun light in, but never let the new warmth wasting on outside. Earth like a greenhouse action in the same way, also like in the greenhouse, the gas combination that creates the earth atmosphere works like a glass wall, the wall let the warm only in, not also back in outer space. They also proved that knowing all greenhouse gas that forwards on global warmth is so difficult to find out.

Once greenhouse gas, like methane or ozone molecules are sent on air, they combine and get together, that's why the composition appears modified. When gas is

* *Student Ph.D. Eng. - University of Petrosani*

spoiled, its contribution on global warmth, like greenhouse effect, is modified too. So, the real effect about the climate of only one issue gets hard to establish.

The most important greenhouse gases are: carbon dioxide, methane, nitrogen oxide, halogen hydrocarbon. All these gases are called "well mixed" because of their long life, they can live even a decade. These gases are created also by natural sources and human activities. The ozone disposed on the ground atmosphere, called troposphere, also works on global warmth. On outward atmosphere the ozone protects life on earth by dangerous UV sunlight. Some of the most important researches about the level on earth warming are based on a kind of Intergovernmental Panel on Climate Change (IPCC) reports.

These reports involve the hard work of thousands of climate scientific explorers. The reports are produced over greenhouse gas measures, exactly in the same way they are in atmosphere after mixing with another gas.

2. THE EFFECT OF METHANE ON GLOBAL WARMTH

Because of studies, the methane effects about global warmth looks like it could be twice worst than it was considered in the beginning. New studies prove the fact that methane issues are so important in global warmth, from "well mixed" gas in 1975 and nowadays.

The report also treats the methane effects about air pollution. An important part in air pollution is near tropospheric ozone section. IPCC report treats the effects of ozone section growing but it can't be connected to a particular source.

Classifying climate effects by issues, Shindell and his fellow workers found that methane effects issues are more important. So, the real guilty gas in global warmth is methane not smog, but methane helps growing smog. Shindell said that controlling the methane is possible and this is how we will control global warmth better than we expected.

Fig. 2. Gas issues on Mart is a mix of watery vapors and methane

NASA announced on the first part of this year that gas issues on Mart is a mix of watery vapors and methane. The revelation can be the proof of life on red planet.

3. METHANE SOURCES

Methane is produced by microorganisms called methanogens. This is a kind of germs that eliminate methane like the rests on life process. Anyway these germs depend by water so we are not sure about their existence on Mars.

Another explanation could be the geological one. Gas could come from volcanic issues or could be the results of chemistry between different types of soil and rocks.

On the earth, methane and methane storages in atmosphere have contributed towards the greenhouse effect and global warmth. Anyway this gas is the first component in natural gas, a useful fuel. Now, science people study ways of using methane from Mars in our benefit. All we have to do is to hope this discover will influence in good our life. Science people realized they omitted another important methane source: plants.

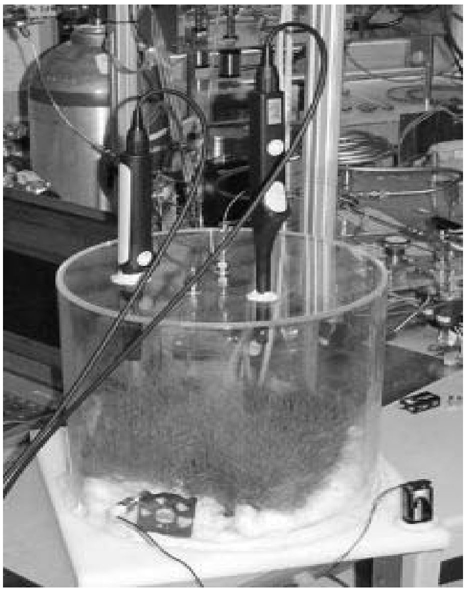

Fig. 3 The study of the difference between living plants and death plants

The methane concentration in air almost doubled in the last 150 years. Even methane is known like a natural gas, a little part of it came from industrial activities.

The most important source of methane is “biogenic sources” like rice cultures or domestic animals, in correlation with human being. Today most of air methane results from bio sources.

Near people thought that methane results only from anaerobic process, with micro-organisms without oxygen. This way acetate or hydrogen and carbon dioxide are converted into methane.

The first source of methane is rice cultures and also the digestion of ruminant animals, spaces for keeping waste and gas resulted in cleaning sewerage system. Like we thought, that kind of sources is guilty for 66% from all air methane production in one year.

Fig 4 A clean medium

But the research people from physics nuclear Max Planck Institute discovered that even plants produce methane and issue it in atmosphere, even in normal conditions, in aired medium. The science people saw this when they had in view gas issued by corn and rye and which is the difference between living plants gas and dead plants gas. They found that living plants issue from 10 to 1000 more methane than death ones. The science people succeeded into proving that methane issue was hardly growing if plants were exposed on sunlight. We are not sure yet about how methane appears in plants system. Research people from Heidelberg think that there is a chemistry way which we don’t know about, another area to study bio-chemistry is plants physiology.

These new discoveries explain some things that we never understood yet. So, a group from Heidelberg University measured recently the methane above tropical forests. They found that methane level is astonishing high. And we know that: forests do this.

They valued between 10 and 30 percent of methane produced every year is because of plants. All the books told us that biogenic methane can be produced only without oxygen. That's why no one studied this possibility.

To know the issued quantity, the Heidelberg research workers began a kind of experiments, most of them in artificial atmosphere, without methane so they can be sure about the methane that they will find is not the normal one in air. Over this they have done a kind of experiments with isotopes so they can watch the process, the methane production. So watching closer, as everybody knew, they discovered something that needs to rewrite all the books of this type.

An important question after that can be: which is the biosphere role in methane production in earth history, and how influenced global warmth? This kind of questions is important to understand the feed-back process between climate changes and global warmth gas production.

4. GLOBAL WARMTH AMPLIFIED BY METHANE FROM OCEANS

Large quantity of fuel came out on surface in north of Siberia, and creates problems that could hasten global warmth.

Anyway they don't know if Arctic methane issues are new or just weren't studied yet. The article from Science said that 8 million tones of methane came out on surface every year, equals the quantity issued by all the oceans on earth. The underwater permafrost is losing his capacity of keeping captive this gas, declared Natalia Shakhova, from University of Fairbanks, Alaska. The experts measured the methane level in 5.000 places in east on Siberia; from 2003 until 2008 they found that in some places methane just came on surface in a kind of bubbles. Today, the methane concentration in Arctic zone is the highest in last 40.000 years.

The supervising of this methane sediment is not easy stuff because issues are not a routine and there is no any signal to announce the start of issue. Sometimes gas sediment were over sea level, but when they are under water the temperature is over 12-17 ^{o}C. 60% from methane issues came from people activities, like farming, and only 40% in natural causes.

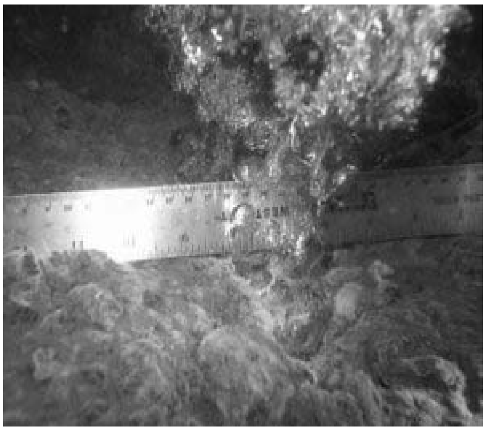

Fig. 5 The methane in ocean's water

The methane from oceans is an important part in greenhouse effect, says that an article from Nature Geosciences.

Research workers from San Diego University, USA, studied six sits from the Mexico golf where methane bubbles came on surface from 500-600 meters. So, as we all have expected, bubbles are so deep and they climb up to the water surface, and methane is squandered in air

Methane has a high influence in greenhouse effect, higher than carbon dioxide, and its warmth ability in 100 to 250 years is 25 times higher than carbon dioxide. Science workers used a robot to obtain water samples on every 20 meters in a column near methane bubbles, and then they studied this gas. They saw that out of sediment the methane level was critical decreased, but in surface water the methane grown back.

Starting to quantity of methane from surface water, research workers graveled the speed for dissipation of methane in atmosphere. Science people found new values from 10 to 10.000 higher, before that, they thought that bubbles issued from more then 200 meters never came on surface. Research workers prove the importance of methane sediment like a source for the methane in air; this source is ignored now in climate studies.

5. MEASURES TO REDUCE THE METHANE QUANTITY

Because of evolution of the greenhouse effect, gas issues have grown with 20% during the last years. That's because of methane produced by animals day by day. Some numbers: the gas quantity which human living issues in atmosphere every year equals 49 billions of tones carbon dioxide, now 13,5% is methane produced by cattle breeding, 25,9% is energetic system, 19,4% industry, 17,4% forests domain, 13,1% transports. The problem of gas issues from farming is one of targets in the war against global warmth.

Fig.6 Growing of methane issues because of farm animals.

This problem mobilized so many groups from New Zeeland to Great Britain, France, Australia, Germany and Canada. A cow can issue every year, ruminating, over 100kg of methane, a sheep issues 6 times less.

Remember that methane is important in the greenhouse effect; its warmth ability is twenty times higher than carbon dioxide. So, we are not amazed, cows` eructation is 70% from methane issues on farming domain in Europe. This percent also can grow because people need more and more animal products. The milk and meat products in 2050 year must be twice higher then 1999-2001. That's why, the research workers needs to limit the gas created by ruminate animals.

The system which produces methane in cows' orgasms is known: cow eats fodder; some micro-organisms in stomach cavity extract the hydrogen. Hydrogen is used by "methanogenic" germs to reduce carbon dioxide on methane. This process costs from 2% to 15% from all energy. This energy is just lost because methane is scattered in air. If digestive process could allow forming less quantity of methane for more useful product for animal or just producing larger quantities of milk, the advantage could be double: a lower greenhouse effect is a better production.

"Since 1970 there has been an idea for keeping the lost energy from digestive process, so they studied how to reduce this losing methane." Said Cecil Martin from farming researching National Institute from Clermont-Ferrand, she obtained god results with using linseed oil in cows diet., on milk cows. The research has been modified in 1995, at the same time with the negotiation for the Kyoto protocol. The governors need to watch closer for gas issues guilty for greenhouse effect in their country.

In New-Zeeland, where animals' gases issue (34, 2 million from sheep and 4, 2 million from human activities in 2008) is 99% from all, the gas is because of farming domain. In 2002 they organized all their science people to do something with that problem. The target was finding a strategy, for farmers, to reduce animal's gas issue until 2013; finally they want 10% lower methane pollution from farming.

6. STRATEGY TO REDUCE METHANE ISSUE

Mark Aspin, leader on Neo-Zealander group, said that all the steps for reducing gas issue is based on the micro-organisms inhibit the micro-organisms responsible for producing methane.

I Select the "right" animals

The idea is to choose cows that produce, natural, less methane than others. The actual research tries to discover how genetics influence this phenomenon. They know that the same cow, obedient on a constant diet, can produce variable gas quantities. Also, methane product is germ "methanogenic" work in digestive tract.

II Food adjustment

The difference between fodder, cereals and lipids is that cereals and lipids keep hydrogen, the basic element in methane production. So, the methane quantity is a halt lower for sheep if 80% from daily allowance of gramineous plants will be replaced with rice. Cows that received linseed oil in 6% had lower methane issues with 27 to 37%. They also proved that fresh fodder produces lesser methane in digestive tract. Research workers need to study the economic validity in this solution and to evaluate the impact in the medium. In Australia, a biotechnology society works on creation of a plant which will produce less methane in digestive system.

III Changes in the ecosystem of before-stomach cavity

Some plant extracts, like lucerne or yucca, looks to be able to destroy the microorganisms which, in digestive process, produce hydrogen and serve like a base for some germs responsible in using hydrogen in methane product.

Using antibiotics can be a solution but this is forbidden in Europe. We must understand and control the action of these additives in the future.

IV Animals' inoculation.

This solution is to force the ruminants' self-protection system to destroy the germ which produces methane. An experimental vaccine in Australia offer hopeless different results, in different parts where the vaccine has been done.

V Kangaroos` digestive system a model to follow

Kangaroo produce less methane. Hydrogen from its digestive process is caught by some germ with produce acetate. This germs also live in ruminants digestive system but they are "sleeping" because of methanogenic ones. So, they could try to stimulate this germ which produces acetate. In the future we follow the reason why is the digestive system unfavorable for acetate germs.

All this wais should be complementary because using of only one method this will not work. For example, destroying methanogenic germs can start to accumulate the hydrogen and that can create a disaster, also for the products and the animals' health. To accumulate the hydrogen modify the digestive pH, that can have complication like acidize, a phenomenon that will regress the animals products a finally can cause its death. "To issue the methanogens is a short time solution; in long time we must find another solution to replace the first one is to use somehow the hydrogen." Said Athol Kliere from Queensland University, Australia. They also find the long time solution for methanogens replacement with acetate germs, like kangaroos digestive process.Until the way to keep methane issue lower will be found they will try to change the animals' diet, by using cereals or lipid instead of fodder, this also is the most studied solution.

We can reduce the issue with 20% in the close future, but we must convince the farmers that we don't want to support the costs rising for this adjustment. Now the problem is less scientific and more economic or political.

7. CONCLUSIONS

Methane is an important effect of the greenhouse, whose warmth potential is twenty times greater than that of carbon dioxide and its warmth potential on a 100 years period length is 25 times greater than that of carbon dioxide.

The real source that is mostly guilty for global warmth is not smog but the methane that leads to smog quantity growth; if methane can be controlled, and this can be done, then it`s probable to reduce global warmth more than we could have thought, obtaining this way a positive result.

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ISSUES CONCERNING THE BEST AVAILABLE TECHNIQUES FOR COMBUSTION OF SOLID FUELS

DANIELA-IONELA CIOLEA*
EMILIA-CORNELIA DUNCA**

Abstract: *Over time there have been harm reduction research results from combustion of solid fuels, so there are different processes and a variety of equipment and techniques. Power generation generally uses a variety of combustion technologies. For solid fuel combustion, spray combustion, fluidized bed combustion and burning on the grill are considered best available techniques (BAT). Technology used in thermal power plants involve combustion or gasification of solid fuels to produce electricity.*

Key words: *combustion, fuel technology, energy, environment*

1. MAIN COMBUSTION TECHNOLOGIES CURRENTLY APPLIED

1.1. Fluidized bed combustion

Basically, fluidized bed combustion process consists of burning solid fuel particles in suspension. In an oxidizing current limit distinguished two situations are determined by the speed of air insufflation: stationary fluidized bed combustion or dense (ASF) circulating fluidized bed combustion (ASFC). [2]

Circulating fluidized bed combustion technology is relatively new, the first steam generator based on this technology was developed in 1979 when the company Alstrom as a 15 MWt steam generator running on fuel oil to flow in fluidized bed combustion working of peat and wood waste.

Offers many advantages that have led to rapid growth, currently being operated steam generators of 250 MWe and 460 MWe contracted to. In Romania, this technology was applied to a first ASFC CAC (hot water boiler) 120 MW, whose design began in 1990.[2]

A fluidized bed is a system in which a gas, distributed through a distribution grid (grid or injection nozzles), is expelled from the bottom up, through a layer of solid particles, so particles floating in the stream of gas and is in a constant turmoil.

* *Assist. Ph. D. Eng. Math. – University of Petrosani, cdipentrucdi@yahoo.com*
** *Lecturer Ph.D. Eng. - University of Petrosani, duncaemilia@yahoo.com*

Biphasic behavior of the medium in which solid particles can move some over others, is comparable to that of a boiling liquid, hence the name of *fluidized bed.*[2]

Basically, the process consists of burning coal particles suspended in an oxidizing current, two situations are distinguished limit determined by the speed of air insufflation: fluidized bed combustion in stationary and circulating fluidized bed combustion.

1.1.1. Stationary fluidized bed combustion (ASF)

The minimum flow velocity field and the layers with large particle size (as happens when crushed coal) is the phenomenon of segregation, characterized by the fact that fine particles gather at the upper layer and large based.

If process flow is going into a furnace in which are inserted properly and air solid fuel and ignition and combustion conditions are met, combustion occurs in heavily, and the process is known as a *stationary fluidized bed combustion (ASF).*

1.1.2. Circulating fluidized bed combustion (ASFC)

Lately, this technique is used increasingly more in technology enhanced combustion of solid fuels, because the facilities they offer in comparison to burning such fuels. This is primarily the burning rate increase, increased exchange convective heat but also increase the time of stationary fuel particles in an outbreak. [1]

ASFC technology has increased worldwide over the past 20 years. Stands tend to build circofluid boiler combustion systems, which is an evolved version of boilers with circulating fluidized bed combustion.

1.1.3. Fluidized air combustion (AFBC)

Technology is to maintain the coal particles with a grain of millimeters or tens of millimeters in an upward airflow.

Because the density change by burning coal bed will be introduced only at the bed surface and as we will burn down to the bottom, the movement of particles creates visual sensation of boiling fluid, where technology has also received the name of bed fluidized boiler. Airflow speed is to strike a balance between weight and carbon particles created by the Archimedes force.

In 1200 the world are central in circulating fluidised bed combustion, with a total of 65 thermal power GWT, distributed as follows: Asia 52% North America 26%, other 1%.

Leading companies in this market are detached Foster Wheeler/Ahlström (about 180 units in operation) and Lurgi Lentjes Babok (about 90 units), other companies are Alstom Power, Babcock & Wilcox and Kvaerner.

Currently there is a noticeable tendency to develop this technology, reaching powers as high as 2020 can be produced 150 GW due to the positive evolution of the market.

AFBC technologies are:

- adaptable to both new and existing installations as well;
- suitable for refurbishment (replacement of existing boiler with an AFBC)
- suitable for conversion of boiler (replacing a portion of an AFBC boiler) in various applications;
- can burn low quality coal (eg lignite of low calorific value waste left from washing coal, petroleum coke and other waste materials).

AFBC technology has proven effective and commercially available power modules larger than 300 MW. In North America are in operation more than 600 boiler (installed capacity of 30 GW), similar functions and capabilities in Europe, and China has over 2,000 small bubbling AFBC boilers in operation.

Several projects are planned or are currently being implemented in the field of 250-350 MW. Development Corporation of Japan has made power plants convert a 350 MW PC boiler Takehara, *bubbling* AFBC technology. EDF in France has built a 250 MW circulating AFBC *(Lurgi technology)*. In general, projects over 300 MW have greater technological risk, why should thoroughly analyze the data for each project. [1]

As demands on developed countries remove pollutants SO_2 are large (usually over 90% removed), most recent projects using *circulating* AFBC option.

1.1.4. Pressurized fluidized combustion (PFBC)

PFBC technology uses a combustion process similar to AFBC technology, but the difference compared to AFBC are:

- boiler works at a pressure higher than atmospheric (0.5-2 MPa);
- gas is cleaned out of the PFBC boiler;
- gas is expanded in a gas turbine.

PFBC technology includes all the advantages of AFBC (removing most of the pollutants SO_2, NO_x emissions low, the capability to burn fuel of low quality and flexibility in choosing fuels) and has in addition:

- gompact and modular design. Upgrading is easier than for AFBC existing power plants because of reduced space requirements;
- potential for achieving higher power output (over 45%) than conventional pulverized coal plant or AFBC (36.5% efficiency) and
- lower capital costs than IGCC technology for pulverized coal or gas from scrubere wet.

The latest and most advanced plants with circulating fluidised bed combustion pressure built by the world leader in this field, Alstom Power, are:

- Quantity Turkey. The power plant has an installed power of 2 x 160 MWe and runs on coal. Each steam generator has four cyclones.
- Red Hills (USA). It is a plant with a capacity of 500 MWe (2 x 250 MWe), put into operation in 2002, and coal burning.
- Guyana (Puerto Rico Power Authority). Put into operation in 2002, with a power of 2 x 250 MWe. Due to stringent emission limits, the center was equipped with depollutant and desulphurisation.

1.2. Burning pulverized

Coal combustion by spraying powder in furnace steam generators, is the most widely used combustion technology in the world today and certainly the most widely used combustion technology in Romania.

For plants that burn coal is by spraying, the most widely used method of gross global output growth is an increase in average temperature above the thermodynamic cycle, specifically by increasing the live steam parameters. Currently us build plants with steam generators Benson type, with forced crossed unique to ensure supercritical parameters of live steam turbines will be used in specially adapted for legally constructive raise these parameters. For the same operating system of the plant and the same atmospheric conditions, a 10% increase in efficiency means a reduction in fuel consumption, let's say, 8%, which means a reduction in CO_2 emissions by approximately 80,000 t CO_2/ year (ie about 8%).

By a simple calculation can be seen that the same technology for combustion, for the same fuel and the same time, efficiencies achieved today by modern power stations burning fossil fuel spray, operating with supercritical parameters of live steam (temperatures around 600 °C and pressure of approx. 250 bar) are between 40 and 52%. Efficient plants in Romania is in the best cases of 37%.

1.3. Gasification

Gasification technology (coal gasification combined cycle-IGCC) is obtaining synthetic gas from solid fossil fuel. [1]

There is no general tendency to believe that this technology is already commercial, mostly due to cost 10 -20% higher than pulverized combustion plants. But most times was not taken into account for conventional power plants and cost reduction of $SO_{x,}$ $NO_{x,}$ particulates and CO_2

Considering a reduction in CO_2 emissions by 85-90% for both technologies, the price difference is reduced or change their meaning, IGCC becomes cheaper.

It can be concluded that gasification is a technology in coming years is likely to become a feasible solution of increasingly used in construction of new ones, which wants to have all included environmental systems, especially those limiting atmospheric emissions.

2. CONCLUSIONS ON THE COMBUSTION OF SOLID FUELS ON A PILOT INSTALLATION - ASFC

Once prepared coal samples, reagents needed by the pilot plant fluidized bed combustion (ASFC) have verified the experiments began, and with gazanalysis TESTO 350 have measured concentrations of sulfur dioxide, nitrogen oxides, carbon monoxide and dioxide carbon, excess air and other parameters. With MENER program have been online focus lower temperatures in fluidized bed/upper fluidized bed ash cooler, input/output body convection, entry/exit cyclone etc.

Process that provides the best results on gas desulphurisation pilot plant fluidized bed combustion flue gas is washing the reactor with an alkaline solution of sodium hydroxide from 1.5 to 5% NaOH (Table no. 1). [1]

Results obtained by placing limestone in the outbreak are not equally effective as for other reagents used. The solution is simple and cheap but which may require facilities to solve desulfurării old.

Capital cost for the wet lime / limestone is especially influenced by the flow of gases.

Capital costs for wet limestone process varies between *35-50 Euro/kW* and operating and maintenance costs are between *0.2 to 0.3 Euro/kWh.*

Cost containment features of SO_2 is between *750-1150 euros/ton* of SO_2 retained, and the effect on electricity price is *3-6 Euro / kWh* (electricity produced).

Table no. 1. *Values measured after the SEGA - Injection 5% NaOH solution*

Exit	**T**	**O_2**	**CO^***	**CO^*_2**	**NO^***	**NO_x^***	**SO_2^***	**λ**
No.	**°C**	**%**	**mg/m^3_N**	**g/m^3_N**	**mg/m^3_N**	**mg/m^3_N**	**mg/m^3_N**	**-**
1	44,9	13,55	669,46	252,17	166,03	268,29	11,48	2,82
2	45	13,82	692,03	252,23	158,38	256,96	11,91	2,92
3	45,6	14,11	737,48	252,18	147,67	241,00	6,20	3,05
4	47,2	14,34	774,21	252,06	146,78	235,47	6,42	3,15
5	47,9	14,47	798,24	252,13	143,59	235,45	13,09	3,22
6	47,9	14,38	784,55	251,81	150,68	246,19	25,83	3,17
7	48,2	14,16	759,32	252,31	160,42	256,25	18,75	3,07
8	49,2	14,21	762,15	252,00	155,72	253,61	18,89	3,09
9	50,2	14,33	781,48	252,13	158,52	253,56	12,82	3,15
10	50,5	14	763,39	252,00	168,15	272,36	18,32	3,00
11	50,6	13,78	773,89	252,06	171,32	276,84	11,84	2,91
12	50,2	14,47	660,41	252,13	174,14	282,54	26,19	3,21
13	49,8	14,79	691,43	252,34	179,90	287,20	13,77	3,38
14	49,6	15	706,25	252,35	176,23	287,00	21,38	3,50
Average	48,32	14,32	724,52	252,11	166,78	267,75	17,61	3,16

$^{*)}{}_{Ref} = 6\%\ O_2$

The cost for spray drying system depends mostly on the ability of installation of main machine - absorber and injection system. Capital costs reported vary depending on the type of energy facility.

Electricity prices could rise in coming years due to operating costs of facilities to be attached denoxare European standard power plants.

Not to increase energy prices in the market, these costs may be covered by increasing the efficiency of energy production. This depends, however, the management of each country's power plants.

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Scientific Reviewers:
Prof. PhD. Eng. Romulus Sârbu

BEHANDLUNGSVERFAHREN VON GEFÄHRLICHEN ABFÄLLEN

VALENTIN IORDĂCHIȚĂ*

Abstract: *In general, treatment of hazardous waste requires that a hazardous waste permit be received before the treatment can be done. "Treatment" covers a broad spectrum of activities, almost anything that can be done to a hazardous waste prior to disposal. Fortunately, under a limited set of circumstances, generators are allowed to treat their own hazardous wastes without first going through the complex regulatory process of getting a hazardous waste permit. The broadness of this definition, though, creates many areas of confusion about when a hazardous waste treatment permit is required and when a particular activity is excepted from requiring a treatment permit. The paper analyses the possibilities of treatment of hazardous waste in world and Romania, for storing dangerous waste and suggests some viable solutions.*

Key words: *Behandlungsverfahren, Gefährlichen, abfällen.*

1. EINFÜHRUNG

Abfälle jeglicher Art, die durch mehrere menschliche Aktivitäten produziert wurden, ist ein aktuelles Thema in allen europäischen Ländern, aufgrund der Erhöhung ihrer Mengen und Arten.

Weil die erzeugten gefährlichen Abfälle, in den industriellen Aktivitäten, nicht immer eine feste Konsistenz haben, ist erforderlich vor einer Deponie oder Verbrennung, Anwendung von physikalisch-chemischen Behandlungsprozessen, die als Ziel haben, die Neutralisation oder Trennung der gefährlichen Substanzen als festen Massen, die leicht behandelt werden können und langfristig in sicherer Umgebung für die Umwelt aufbewahrt werden können.

2. SITUATION DER BEHANDLUNG UND DER ENTSORGUNG VON GEFÄRLICHEN ABFÄLLEN IN RUMÄNIEI

In Rumänien, das Management der organischen Abfälle, aus der Mineralölindustrie und dem organischen Chemiebereich, war in der Vergangenheit gut genug und ist genügend auch heutzutage.

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* *Student PhD. Eng. – University of Petrosani*

Leider, die Behandlung von anorganischen Abfällen ist sehr arm, die meisten Behandlungsanlagen sind heute stillgelegt oder in Verfall.

Allerdings, mehrere Anlage, verwendeten in der Vergangenheit für die Behandlung und/oder Beseitigung von Abfällen, wurden verschlechtet und sind heute unbrauchbar, aufgrund der wirtschaftlichen Schwierigkeiten der meisten Betrieben.

Für die Behandlung und Entsorgung der Abfälle wird in Rumänien die termische Behandlung(Verbrennung oder Verwertung in der Zementindustrie) bevorzugt.

Die Firma Sotem România SRL besitzt eine Anlage in Holcim Zementwerk, Campulung, zum Mischen von organischen Abfällen, die in der Abbildung 1 gezeigt wird.

Die Operationen, von dem Zementofen erfüllen derzeit die EU-Vorschriften nicht, das heißt die Automatisierung und Überwachung des Ofenes sind nicht optimal, aber Sotem/Holcim führt derzeit ein Ausrüstungsprogramm durch, das in der Zukunft die derzeitigen EU-Vorschriften erfüllen wird.

Abbildung 1. Sotem-Anlage zum Mischen von organischen Abfällen [1]

3. BEHANDLUNGVERFAHREN VON GEFÄHRLICHEN ABFÄLLEN, DIE WELTWEIT VERWENDET WERDEN

Zur Einhaltung der europäischen Rechtsvorschriften über Abfälle, betrefend die zulässigen Höchstgrenzen für gefährliche Stoffe in Abfälle, die Unternehmen, die gefährliche Abfälle generieren, müssen durch entsprechende Investitionen, eine interne oder externe Behandlung durchführen (auf eigene Kosten), um die Konzentration von gefährlichen Substanzen vor einer eventuellen Deponie oder Verbrennung (Übertage- oder Untertagedeponie) zu reduzieren.

Die Zuordnung der einzelnen gefährlichen, nicht verwertbaren Abfallarten zu einem bestimmten Verfahren ergibt sich aus den Stoffeingenschaften der Abfälle. Je nach der Beschaffenheit werden Abfälle durch biologische, thermische, oder chemisch-physikalische Verfahren behandelt. Feste Rückstände sind möglichst reaktionsarm und konditioniert auf geordneten Deponien abzulagern.

In der Tabelle 1 sind die Behandlungsprozessen, wie in der europäischen Geseztgebung presentiert.

Unter Konditionierung versteht man die Vorbahandlung des Abfalls, chemische und/oder physikalische oder biologische Verfahren, die dazu dienen einen

Abfall für eine bestimmte Behandlung (C(P, BB, TB, D, V) geeignet zu machen. Beispiele dafür sind:entwässern, staubfrei machen, befeuchten, zerkleinern, sortieren.

Tabelle 1. *Behandlungsarten gemäß EU-Normen [2]*

Kurzzeichen	Benennung
C/P	Chemisch-physikalische Behandlung
BB	Biologische Behandlung
TB	Termische Behandlung
D	Deponierung
K/V	Konditionierung/Verfestigung

3.1 Thermische Verwertung und Behandlung der gefährlichen Abfällen

Thermische Behandlung von gefährlichen Abfällen ist die einfachste Methode der Abfallwirtschaft, der bekannteste und weltweit angewandt.

Das Ziel der termischen Behandlung von gefährlichen Abfällen ist Energiegewinnung, die zurückgewonnen wird, oder ihre Inertizierung.

3.1.1 Verbrennung der gefährlichen Abfällen

Der anlagentechnische Aufbau einer Verbrennungsanlage für gefährliche Abfälle wird durch ihre Aufgaben im Rahmen eines übergeordneten Entsorgungssystems bestimmt.

Öffentlich zugängliche Anlagen müssen ein umfangreiches Angebot an Annahme-, Vorbehandlungs- und Lagerungseinrichtungen vorhalten, da nicht wie bei betriebseigenen Verbrennungsanlagen auf vorhandene Einrichtungen zurückgegriffen werden kann.

In der Tabelle 2 sind verschiedene Design-Alternativen und deren Kompatibilität mit bestimmten Arten von Abfällen presentiert, die durch Verbrennen behandelt werden.

Tabelle 2. *Eignung verschiedener Verbrennungssysteme zur Behandlung bestimmter Abfallarten [2]*

Abfallart	**Drehrohr-ofen mit Nachbrenn-kammer**	**Rost-feuerung**	**Wirbelschicht-feuerung**	**Etagen -ofen**	**Brennkammer /Muffelofen**
Flüssige Abfälle	++	-	+	-	++
Pastöse Abfälle mit hohe Zähigkeit	++	-	+	-	-
Schlämme mit hohem Wasser-gehalt	+	-	++	+	-
Feste Abfälle -stückig -sperrig	 ++ +	 + ++	 + -	 - -	 - -
Gebinde	++	+	-	-	-
Gase	+	-	+	-	++

Aufgrund der Tatsache, dass derzeit noch der überwiegende Teil der gefährlichen Abfälle als feste, pastöse oder flüssige Stoffgemische anfallen, geht der

Drehrohrofen als universell einsetzbares Verbrennungsverfahren hervor. Derzeit sind 80% der eingesetzten Systeme Drehrohrofenanlagen.

3.1.2 Ent- und Vergasung der gefährlichen Abfälle

Organische Verbindungen werden bei Erhitzen instabil und infolge der Entgasung in flüchtigen Produkte und Kocs zersetz. Anschließend wird in der Vergasung der entstandene Kocs durch Zugabe von reaktivem Gas in weitere gasförmige Produkte und Asche umgewandelt.

Durch die Pyrolise entstehen, abhängig von den Prozessparameter, gasförmige, flüssige und feste Produkte in unterschiedlicher Verteilung.

Diese Verfahren befinden sich im Übergangsstadium vom Labor zum industriellen Maßstab, daher existieren nur wenige Anlagen:

- Salzgitter Pyrolyse- Pyrolyse im Drehrohr;
- Plasmox Verfahren- Pyrolysekammer mit Zentrifuge;
- Thermoselect Verfahren- Entgasung im Entgasungskanal mit anschließender Vergasung im Hochtemperaturreaktor.

3.1.3 Nassoxidation der gefährlichen Abfälle

Die Nassoxidation ist ein Prozss, bei dem organische und anorganische Verbindungen bei erhöhten Temperaturen und Drücken mit Luft- oder Reinsauerstoff in wässrigen Lösungennund Suspensionen flammenlos oxidiert werden. Daher müssen die Substanzen als Suspensionen vorliegen. Bekannte Verfahren sind hierbei das Bayer-Loprox-Verfahren sowie das Vertech-Verfahren. Das Vertech-Verfahren wird für die Nassoxidation von Klärschlämmen eingesetzt.

3.2 Zuordnung von gefährlichen Abfällen zur chemisch-physikalischen Behandlung

Behandlung des Abfalls mit chemischen und/oder physikalischen Methoden hat als Zweck, Veränderung seiner chemischen, physikalischen, biologischen Eigenschften.

Für diese Behandlung sind Abfälle mit folgenden Eigenschaften gut geeignet:

Pentru tratare fizico-chimică sunt recomandate deşeuri cu următoarele proprietăți:

• Hohes Schadstoffpotential- Der Eintrag großer Schadstoffmengen auf die Deponie ist zu verhindert; daher werden in der Deponieverordnung auch Grenzwerte für verschiedene Schadstoffe vorgeschrieben.

• Hoher Wassergehalt- Ein hoher Wassergehalt bewirkt einen niedrigen Brennwert und verhindert dadurch eine selbstgängige Verbrennung. Eine Deponierung stark wasserhaltiger Abfälle ist zu vermeiden, da diese das Deponieentwässerungssystem belasten und die Standfestigkeit des Deponiekörpers negativ beeinflussen. Aus diesem Grund ist die Ablagerung flüssiger Abfälle verboten, Trennungsprozesse, Konzentration beinhaltiger gefährlicher Stoffe oder eine Inertisierung/Verfestigung des Abfalls empfohlen sein, vor der Deponie.

In der Tabelle 3 sind Beispiele jener Abfallstoffe angeführt, welche mit chemisch-physikalischen Verfahren zu behandeln sind.

Tabelle 3. *Zuordnung von Abfallstoffen zum anorganischen/organischen Strang eine chemisch-physikalischen Behandlungsanlage [2]*

Anorganische Abfälle	Organische Abfälle
Galvanikschlämme	Bohr- und Schleifölemulsionen
Säuren und Säurengemische	Öl- und Benzinabscheiderinhalte
Laugen und Laugengemische	Sandfanginhalte, öl- oder kaltreinigerhältig
Bleichereiablaugen	Öl-Wassergemische
Metallsalzhaltige Konzentrate	Schlamm aus der Tankreinigung
Entwickler- und Fixierbäder	
Deponiesickerwasser	

In der Genehmigungspraxis kann die Zuordung der Abfälle zu einer bestimmten Behandlungsanlage auf unterschiedliche Weise erfolgen:

- Durch einen Positivkatalog, der alle Schlüsselnummern und Bezeichnungen der Abfälle enthält, die behandelt werden können,
- Durch einen Negativkatalog derjenige Abfälle, die nicht behandelt werden können,
- Durch chemische Ausschlusskriterien (z.B Quecksilbergehalt)
- Durch Angabe anderer Abfalleigenschafften, die einer Behandlung entgegenstehen (z.B. penetranter Geruch)
- Eine Kombination der oben genannten Kriterien.

Mit der Behandlung von stark schadstoffbelasteten und wasserhaltigen Abfällen durch chemisch-physikalische Verfahren werden folgenden Ziele verfolgt:

- Rückgewinnung von Wertstoffen (z.B. Ol),
- Abtrennung (z.B. Schwermetalle) oder Umwandlung (z.B Cyanide) von Schadstoffen,
- Gewinnung eines Teilstromes, der gefahrlos in die Umwelt entlassen werden kann,
- Reduzierung der Masse der zu deponierende Abfälle,
- Konditionierung der Abfälle vor der Deponierung (z.B Verfestigung)
- Geringstmögliche Emissionen beim Betrieb
- Weitgehende Betriebs- und Störfallsicherheit.

3.2.1 Verfahren der chemisch-physikalischen Behandlung

Zum Erreichen der oben genannten Ziele ist der Einsatz von drei Verfahrenstypen notwendig:

- Trennung von Gemischen,
- Umwandlung von toxischen Substanzen,
- Verfestigung, Immobilisierung.

Eine Übersicht der einzelnen Verfahren liefert die Abbildung 2. [2].

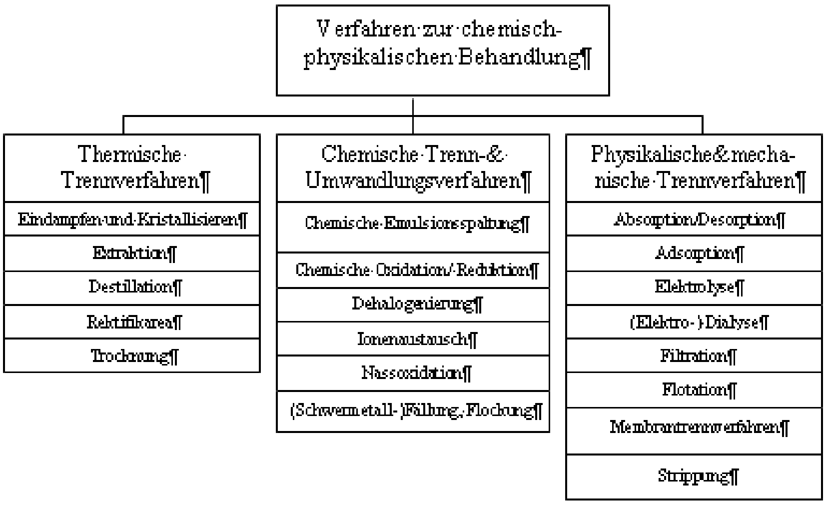

Abbildung 2. Zuordnungen einzelner Verfahren zur chemische-physikalischen Abfallbehandlung

3.3 Verfestigung der gefährlichen Abfällen

Die Anforderungen, die bei der Verfestigung flüssiger, pastöser oder schlammförmiger Abfällen an die Art der Schadstoffbindung zu stellen sind, werden auf der Abfallseite bestimmmt von den ökotoxikologischen Eigenschaften der Schadstoffe und deren Mobilität in den verschiedenen Teilen der Umwelt.

Die Art der Verfestigung gibt vor, mit welchen mechanischen, physikalischen und chemischen Mechanismen der Schadstoffbindung gerechnet werden kann. Das Ziel der Verfestigung definiert schließlich das qualitative und quantitative Ausmass der Emissionen, die kurz- oder langfristig toleriert werden können.

3.3.1 Anforderungen an die Schadstoffbindung- Ziel der Verfestigung

Die Anforderungen an die Vollständigkeit, Festigkeit, Irreversibilität und Dauerhaftigkeit der Schadstoffbindung im Verfestigungsprodukt steigen mit höheren Zielvorgaben.

Das erste Ziel ist lediglich, den Abfall, umschlagbar und transportierbar zu machen, ohne dass von ihm Gefahren für diejenigen ausgehen, die mit ihm umgehenden.

Dazu ist der Abfall aus seiner flüssigen bzw. Pastösen/schlammigen Form in eine besser handhabbare stichfeste, erdige bis betonharte Konsistenz zu überführen.

Das Verfestigungsprodukt soll nicht thixothrop sein (das heißt sich beim Schütteln oder Rühren verflüssigen) und darf nicht leicht entzünbar sein. Die Emissionen wie Staub, Gas, flüchtige Schadstoffe müssen beherrschbar sein, das heißt die Anforderungen an die Schadstoffeinbindung sind zu messen an einen gegebenenfalls hohem technischen Stand zur Erfassung der Emissionen.

Das Material soll mit Wasser nicht in nachteiliger Weise (z.B. unter Hitzeentwicklung und Freisetzung niedrigsiedender Bestandteile) reagieren.

Sonderanforderungen richten sich nach dem Ziel der Weiterbehandlung/-verwertung.

Ist eine thermische Behandlung das Ziel, bieten sich Verfahren mit brennbaren Verfestigungsmaterialien (z.B. Bitumen oder Kohlenstäube) an, so dass das allgemeine Ziel „nicht entflammbar“ in diesem Fall nicht durch das Verfestigungsmaterial, sonders durch entsprechende Anforderungen an das Verfahren und die Lagerungstechnik zu realisieren ist.

Die nächsthöhere Stufe wäre die Verfestigung für die Entlagerung auf einer unter- oder oberirdischen, auf hohem technischen Sicherheitsstand betriebenen Hochsicherheits- oder Sonderabfalldeponie.

Bei diesem Deponietyp wird der Wasserzutritt verhindert bzw. Minimiert, Deponieabwässer werden kontrolliert abgeleitet und behandelt.

Die wichtigsten, über die erste Stufe hinausgehenden Anforderungen wären stärkere Einschränkung der Ausgasung und insbesondere hohe Anforderungen an die Langzeitstabilität des Produktes, wobei allerdiengs die Beanspruchungen durch Wasser, Verwitterung und andere Umwelteinflüsse durch bauliche Massnehmen relativ gering gehalten werden sollten.

Wesentlich höher sind die Anforderungen an ein Verfestigungsprodukt, das Untertage abgelagert werden soll. In solchen Fällen ist davon auszugehen, dass der Abfall langfristig bis zur Wasserkapazität mit Sickerwasser gesättigt wird und über eine-möglichst geringe-Restdurchlässigkeit der Deponiebasis eine Grundwasserbelastung erfolgen kann.

Als Zielvorstellung gilt, dass langfristig das Eluat Trinkwasserqualität besitzt, zumindest aber Immissionsneutral ist.

Im einzelnen gelten folgenden Anforderungen:

- Geringe Wasserdurchlässigkeit des Verfestigungsproduktes,
- Mechanische Langzeitintegrität und –stabilität,
- Keine über das Ziel “Trinlwasserqualität” hinausgehende Schadstoffreisetzung bei Verwitterung und Korrosion oder bei veränderten Milieubedingungen in der Ablagerung (z.B. Redox-Verhältniss, pH, Temperatur, chemisches Milieu),
- Keine Schadstofffreisetzung bei langfristig ablaufenden biochemischen Umsetzung- bzw. Abbauprozessen,
- Keine Abgabe von schädlichen Reaktionsprodukten bzw. Metaboliten.

Die Verwertung eines verfestigten Abfalls als Wirtschaftsgut, z.B. für den Wege- und Straßenbau, stellt die höchstens Anforderungen, insbesondere, wenn der Abfall nicht nur in einer abgedeckten Tragenschicht eingebaut wird. Straßenbauten haben im Durchschnitt Lebenszeiten von etwa 30 Jahren, das Schicksal des ehemals eingebauten Abfalls kann kaum noch verfolgt und kontrolliert werden, bestensfalls bleibt er Baumaterial oder wird auf eine Deponie verbracht.

3.3.2 Versuchen und Ergebnissen betreffend Verfestigung in Rumänien und im Ausland

In die EU, konzentriert sich die Prioritäten und die Richtungen in diesem Bereich, auf die Entwicklung von Lösungen, die als Ziel haben, die Rückgewinnung, Recycling und Verringerung der gefährlichen Abfälle.

Gehend von diesen Zielen aus, in allen Ländern der Europäischen Union und überall in der Welt, werden Programme ausgeführt, die privat oder statlich finanziert sind, die als Ziel haben, Forschung und Entwicklung von Tehnologien, die einen sicheren Umgang mit den gefährlichen Abfällen ermöglichen können.

Die ersten Ergebnissen ließen zu viel nicht erwarten, das Institut für Chemische Tehnologie, aus Prag, Tschechien, schlug ein Verfahren zur Säureauslaugen von gefährlichen Schlämmen, gefolgt von einer selektiven Fällung in mehreren Etappen, und nämlich:

- Ausfällung der dreiwertigen Metallen, Cu, Cd, Ca, Mg and Si;
- oxidative Ausfällung von Mn;
- Fällung von Zn als Carbonat.

Nach Säureauslaugen von gefährlichen Schlämmen ergibt säuren Lösungen reich an Schwermetallen. Aufgrund der Eigenschaften dieser Gewässer (niedriger pH-Wert, Anwessenheit der Schwermetalle, neutrale Salze in der Lösung), es wird notwendig, bevor sie in die Umwelt entlassen dürfen, eine Reinigung.

Üblich, diese Gewässer werden konventionell behandelt, wie folgt:

• Korrektur des pH-Wertes und Metall-Fällung,
• Trennung der Metallhydroxide und dem Gips durch Absetzen,
• Verwertung oder Beseitigung dieser Metalle.

Das vorgeschlagene besteht in:

• der Behandlung mit Kalk, in zwei Schritten, in Anwessenheit der Luft, bis pH=11 erreicht wird, für die Fällung der Metalle (inklusiv Mangan) und Flockung-Dekantieren.

• der Korrektur des pH-Wertes mit CO_2 (pH=7,5-8) und Fällung des Aluminium gefolgt von Flockung-Dekantieren.

Dieser Verfahren ist Teil der empfolenen Lösungen BAT, für Behandlung von Grubenwässers und ist in zahlreichen Minen in Europa und weltweit angewandt.

Korektur des pH-Wertes, in den resten Schritt, kann man mit Kalk, Kalkmilch oder Kalkstein gemacht werden.

Ein anderes Verfahren wird von der Universität Nis, aus Srebien vorgeschlagen und es sieht Aufnahme der Schwermetalle (Cu, Cr, Cd, Ni, Pb, Zn), aus den Schlämme in Glaskeramik voraus, die stark chemisch gegen der Wirkung von säuren und alkalischen Stoffen widerstandsfähig sind.

In den Verreinigten Staaten wird ein Verfahren, nach Patent 6962562 angewendet, das Kalzinierung von giftigen Abfällen, Behandlung mit Phosphorsäure und wenden einem inerten Produkt an, voraussieht.

In der Türkei wurde als Verfahren, das Vermischen gefährlicher Abfälle mit Zement Portland, wenn eine inerte Masse erhält, vorgeschlagen.

In Portugal wurde eine Inertisierungstehnologie entwickelt, die die toxische Metallen aus den Schlämmen auf Keramik, basierend auf Ton fixiert.

Die Firma VOEST Alpine Stahl GmbH aus Linz, Österreich, die Metall bearbeitet, versucht ein Verfestigungsverfahren dem Metallstaub aus der Produktionsaktivität zu entwickeln. In der Abbildunf 3 ist ein Verfestigungsprodukt prezentiert, das nach folgendem Rezept erhalten wurde:

Metalischer Staub	83,3%
Bindemittel-Zement	16,7%
Wasser/Bindemittel	2,86
Konsistenz	Erdfeucht, verdichtbar

Abbildung 3. Verfestigungsprodukt, Firma VOEST, Österreich

Auch in Rumänien gibt es Interesse, von mehreren Jahren, zu entwickeln und zu verbessern von Verfestigungsverfahren der gefährlichen Abfällen. Als Ergebniss kann man ein Verfahren erinnern, das die Emailabfälle, für die Beschichtung von keramische Materialle (dekorativen Ziegel, Keramik, Fliesen, usw.) verwendet (Registriert bei OSIM unter Patentnummer A/006232/23.07.2003).

Die Vorteile diesem Verfahren sind folgenden:

- bietet eine besere Wiederverwendung der Emailabfällese, durch Aussetzen von einfacher Verarbeitung;
- als Abfall wird zu einem Preis weit unter dem ursprünglichen Wirkstoff gekauft;
- die Tehnologie ist sehr einfach und anpassbar;
- der Abfall kann ohne andere Verarbeitung oder als Zusatzstoff, im Rezept, verwendet;
- garantiert nidriege Produktionskosten.

Die Versuchen wurden im Labor der Universität Petrosani durchgeführt und wurde für diesen Zweck einen Ofen mit maximaler Temperatur von 1200° C verwendet.

4. KONKLUSIONEN

Die Abfallwirtschaft ist ein Thema, das fast den gesamten Planeten betrifft, aber vor allem manifestiert sich in stärkerem Maße in die EU aufgrund ihrer zunehmenden Größe und Vielfalt und ihre negativen Wirkungen auf die Umwelt.

Wenn in den Industriestaaten bereits eine erfolgreiche Aufklärung der Bewölkerung und Unternehmen über den Umweltschutz gelungen wurde, in den ehemaligen sozialistischen Ländern ist dieses Prozess nur am Angang, ihre Wirksamkeit hängt von der einwandfreien Durchführung und Durchsetzung.

Die Abfallbehandlungsverfahren, angewendet heute in der Welt, sind nicht immer in der Lage maximalle Effizienz zu erreichen, deshalb ist die Entwicklung neuer oder besserer Tehnologien eine Priorität für alle Länder, die eine saubere Umwelt für die zukunftige Generationen schaffen wollen.

Deshalb ist derzeit die Erforschung und Entwicklung neuer Methoden zur Handhabung gefährliche Abfälle und Verfestigung eine wichtige Thema in fast allen Forschungseinrichtungen auf dem Gebiet geworden.

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Scientific Reviewers:
Prof. PhD. Eng. Mircea Georgescu

HOW TO DEVELOP A FORMER MINING AREA IN A SUSTAINABLE MANNER

RAREŞ MUNTEANU*

Abstract: *The sustainable developmentie although not a new idea, is more and more actual. The development of any area must be based on the principles of the sustainability in order to be successful.*

Key words: *rehabilitation, development concept, landscape*

1. INTRODUCTION

The rehabilitation of the mining regions is meant to ensure the development in a sustainable manner for a valuable living and landscape area with multiple options for use.

Redevelopment efforts should be geared towards eliminating hazard potentials to permanently ensure public safety, to develop a post-mining landscape with multiple options for use and an acceptable design which requires no secondary treatment. The rehabilitation area should be developed into a self-sufficient landscape of trans-regional significance for tourism. Simultaneously, issues of flood protection, leisure and recreation, nature, landscape and additional forest area should be resolved in a compatible manner and based on a clear-cut functional division between areas intensively used and areas sensitive to noise. Rehabilitation measures relating to water should focus on the re-creation of a stable and mostly self-regulating regional water balance.

2. THE SITUATION IN THE JIU VALLEY

As a result of the major job losses that took place in the National Pitcoal Company (Compania Nationala a Huilei – CNH – Petroşani, Romania), the number of the employees was reduced from 45.141 at the beginning of 1997 to about 9.000 in 2010. Those who retired represent only a minority, most of those who left CNH became unemployed. By the time the system of mass-dismissal with reparatory

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* *Lecturer PhD. Eng. – University of Petrosani*

payment was introduced no alternative new jobs were created, considering that many of them came in the Jiu Valley from other regions of the country and that those people would try to go back to the places of origin. This supposition was not realistic and the economic difficulties arose in short time for the unemployed who remained in Jiu Valley. These problems had two reasons:

a) the amount of money received as compensatory payments was not sufficient to ensure the future of the unemployed;

b) the money received as compensatory payments was mismanaged by the owners, as they were used mainly to buy goods and even if some very few persons tried to start a business, the lack of managerial skills caused those business to go bankrupt very fast.

A realistic analysis would show that the number of persons working at CNH has decreased by more than 80% and this is a very serious problem for Jiu Valley. No matter how we put the problem, the Jiu Valley hasn't the potential to create new jobs for all these persons. But this doesn't mean that the Jiu Valley has no future, because, beside the weaknesses there are opportunities as well.

2. REQUIREMENTS FOR THE DEVELOPMENT OF THE JIU VALLEY AREA

The facilities for sports, leisure and recreation must be maintained, extended and developed to upgrade this recreational area of trans-regional significance.

Through the formation of a tourist water association*,* water-bound options for recreation at the Campu lui Neag lake can be combined in a synergic way with the mountain tourism in the western part of the Jiu Valley.

Prerequisites are to be created for a large coherent, richly structured forest area, to systematically increase the percentage of forest and provide protection for the current forest area.

An effective protection of landscape, nature and species shall be guaranteed in the mountain zones (Parâng and Retezat), including their spatial and functional integration into landscape elements in the rehabilitation area and in the remaining unexploited environment.

Traffic access and internal development of the rehabilitation area shall be improved essentially and in a target-oriented manner by providing large-area traffic connections, demand-driven traffic development of recreational areas, re-building of devastated or interrupted historical traffic routes and the creation of a multi-use biking and hiking trail network including its integration into the trans-regional and regional traffic and trails network.

Moreover, the general rehabilitation plan must contain stipulations and goals on the following items:

► the geographic position (including boundaries) of the rehabilitation area,
► soil protection and material disposition,
► slope remediation and landscaping,
► areas of potential contamination, contaminated sites and waste dumps,
► regional water balance,

- ► preventive flood protection,
- ► noise and dust protection,
- ► fishing,
- ► recreation,
- ► agriculture,
- ► increase in forest area and forest protection,
- ► nature and landscape and
- ► traffic infrastructure and local public transport.

The plan must describe the goals and provides reasons for goal setting while detailed rehabilitation measures and methods are to be laid down in the final operating plans of the Jiu Valley mining area. In addition, the general rehabilitation plan must contain so-called priority areas which cannot be changed by the rehabilitation company or the relevant addressee. A good general rehabilitation plan comprises priority areas for:

- ► increased forest area,
- ► forest protection,
- ► nature and landscape (succession areas),
- ► nature and landscape (forest areas),
- ► nature and landscape (water areas),
- ► recreation.

In contrast to priority areas whose use has been ultimately defined, the plan also comprises so-called reserve areas, where a particular type of use should be given special weight compared to other options. The reserved areas in the general rehabilitation plan comprise areas for:

- ► increased forest area,
- ► agriculture,
- ► nature and landscape,
- ► nature and landscape (lakes),
- ► recreation.

After the general rehabilitation plan of the Jiu Valley area becomes legally effective, the in the final operating plan the environmental needs must be harmonized with the economic needs of the society. The harmonization of all these requirements must be done using a regional development concept.

In order to achieve the integrated approach for the development of the Jiu Valley, we consider useful to apply a logical frame (see Table 1).

A development concept for the Jiu Valley can be expressed as follows:" The Jiu Valley is to be taken into consideration as a single administrative unit, the social, economic and environmental rehabilitation process will be unitary implemented and under these circumstances the tradition and the novelty, with new ideas, interpenetrate for a sustainable development".

Table 1. *Logical framework for development of the Jiu Valley region*

	Intervention logic	Objectively verifiable indicators of achievement	Sources and means of verification	Assumptions
Overall objectives	► Development of alternative activities in the Jiu Valley to create new jobs as the number of jobs in the mining sector is decreasing.	► New jobs and new business will be created	► Register of Commerce ► Labour Agency ► Information from the press	► The desire of the inhabitants of the Jiu Valley to continue their lives in the same region after the wane of the mining industry ► Allocation of important funds to implement programmes concerning the ecological issues and rehabilitation of the infrastructure
Specific objective	► Ecologisation of the surfaces affected by the industrial activity ► Elucidation of the legal status of the lands ► Rehabilitation of the access ways and utilities ► Aggressive publicity for the Jiu Valley ► Use of the usefull minerals inthe sterile heaps ► Use of the mine gas fro the active as well as from the closed mines	► Ecologized surfaces handed over to the local communities / to the nature ► Enhancing the purity of the air ► m^3 of gas used for the economy ► direct income from the use of the mine gas ► Length of the access ways rehabilitated ► Various buildings and areas made available for business	► Environmental reports ► Reports from the CNH and local authorities ► Information reports from the Register of Commerce	Risks: ► Posibility of finance shortage ► Lack of unity in actions
Expected results	► The companies working in the Jiu Valley will thrive and thus new jobs will be offered to the population ► New business will be created ► The health of the population will be better due to a clean environment	► Number of new bussiness ► Number of new jobs	► Reports of the Register of Commerce and the Labour Agency ►Reports of the local authorities ►questioning the inhabitants of the Jiu Valley	► Good and effective laws ► Correct and complete information of the public outside the Jiu Valley

Activities	► Evidence of all the surfaces that need ecologization, as well of the negative effects that need to be removed ► Evidence of the re-usable buildings and of the works to be done in order to rehabilitate them ► Evidence of the transportation ways (e.g. roads) that need rehabilitation works ► Evidence of the utilities (water, power, sewerage) to be rehabilitated ► Evidence of the available accommodation and of the improvements that need to be done	Means*:* ► Companies specialised in civil engineering and building ► Support from the local authorities	► Available statistics. ► Evaluations done by the actors interested and involved	► Agreement between the interested actors (companies, business persons, local authorities) in order to carry on a joint actions

3. CONCLUSIONS

Although the mining industry in the Jiu Valley, Romania, is on the wane and, as a result, the economic situation is very difficult, there are enough possibilities for the local community to continue the development “at home”. New activities must be developed. The core issues are the ecologization and making clear the legal status of the land, in order to attract new investments. The position of Jiu Valley at the crossing of important national routes is an advantage that must be carefully put in use in order to ensure the sustainability of the region.

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Scientific Reviewers:
Prof. PhD. Eng. Florian Buşe

FUNCTIONAL AND AESTHETIC REINTEGRATION OF ABANDONED COAL PITS

CIPRIAN NIMARĂ*

Abstract: *Mining pits are the result of the morphological process of rock excavation site with a reserve of useful mineral substances. Mining pit's morphology is particularly spectacular in terms of dimensional and designed landscape being characterized by an inversion of relief. Because of strong visual impact, measures to rehabilitate land affected by mining must seek aesthetic problem with industrial design projects, for a restful ambience.*

Key words: *functional reintegration, anthropic landform, coal pit, aesthetics, anthropic landscape*

1. INTRODUCTION

Mining activity usually results in two kinds of anthropic landforms: one is a positive landform (waste dumps, anthropogenic terraces, etc.) and the other is a negative landform (coal pits, excavation, uncovered land, etc.). The basic principle regarding the rehabilitation of the degraded land is that the negative landforms need to be filled in, while the positive landforms need to be smoothed out. This "leveling the high land and filling the low land" is an important engineering component in the rehabilitation of degraded mining lands (Georgescu M, Dumitrescu I, Biro C.).

Land rehabilitation ($\mathbf{R_p}$) requires extensive research to be based on a recreational landscape development, using landscape architecture techniques ($\mathbf{A_p}$) and industrial design ($\mathbf{D_I}$). Landscape architecture has its own ways of working, often turning to vegetable item as "building material" (Bradshaw A.).

$$R_p = f\,(D_I, A_p) \tag{1}$$

Anthropic landscape can join the functional and aesthetic reorganization simple or complex programs, in terms of: financial resources available, degree of impairment of environmental components and local specificity. Quick solutions are welcome to have low

* *Student Ph.D. Geographer –University of Petrosani*

costs, which in turn degraded territorial complexes with valences recreational facilities.

Completion of mining activities to date and conversion of degraded environmental areas within require administrative and engineering measures. Specific technical operation after closure and preservation of mining fields, landscaping is functional rehabilitation and reintegration of former mining field. Is needed to take account of the specific site, the latest techniques and methods in terms of organic conversion and expectations of local community.

Of aesthetically, the created new space should take into account certain psychological principles of landscape perception (fig.1, fig.2). For the perception of landscape, like a performance, it is important the light, the angle of illumination and its intensity and how the background is made – the arrangement of the morphological features and vegetation.

From the point of view, green areas for recreation or tourism officials to be achieved in a particularly degrading territory, bearing in mind that these activities were not typical of this area before the start of mining activity. As a result, tourism or leisure

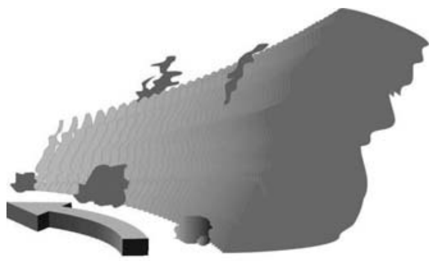

Fig. 1. Area that directs

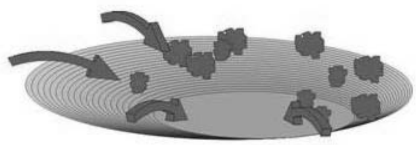

Fig. 2. Collecting area

activities (viewed as desires or needs of human beings to delight the senses), should be concentrated in those areas that have the great potential for scenic beauty and quiet.

Usually the created function of the new made land is perceived locally or regionally. If it is possible that tourism and recreational activity to replace the previous industrial activity, then it can play an important role in the social and economical terms (Law DL.).

While it is possible for tourism to supplement local industry, it will rarely play a vital role at the local level.

If the territory, in which the anthropogenic factor has held industrial activity, is not in terms of potential landscape and harmonious relations with any other tourist resources, when recovered a degraded land, it can be improved by building parks and public gardens, artificial lakes, water parks and botanical gardens (fig. 3). Of course, greening and tourism areas represent a complex combination of all kinds of landscape functions.

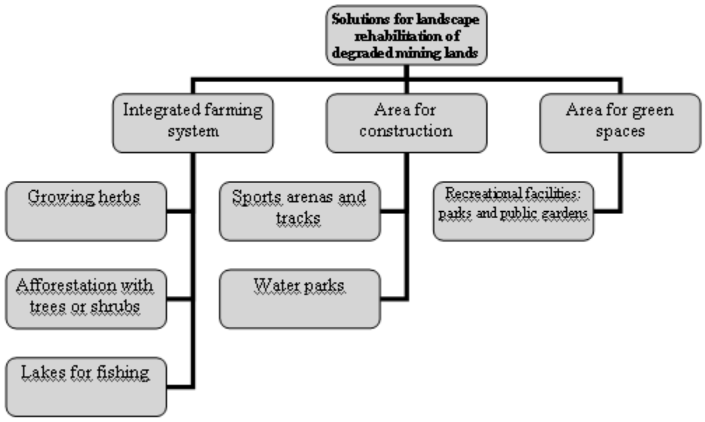

Fig. 3. Solutions for rehabilitation of degraded mining lands

2. LANDSCAPE PATTERNS OF REINTEGRATION OF COAL PITS

2.1 Arranging the coal pit for recultivation

During operation, the tailings can be transferred from a coal pit operation in other sectors which are closed or inactive. After filling their former careers or planning as tailings dumps inside, they enter a rehabilitation program similar to external waste dumps.

Because most of the usable land destroyed as a result of mining activity was previously farmland, it can be recultivated following adequate reclamation procedures. Forest recultivation involves a series of preparatory work as: building works, land preparation work and works to improve soil conditions.

Results of experiments showed that the soil improving species (which fix nitrogen) such as acacia, alder and especially sea buckthorn have been successfully used in all cases. They have contributed to substantial improvements and have stimulated growth of other wood species when grown in mixture with them (Lazăr M.).

For example, in Russia, on silty - clay alkaline deposits with low phosphorus and potassium, good results were obtained from plantation with: sea buckthorn (Hippophae rhamnoides), black alder (Alnus glutinosa), alder (Alnus incana) and lupine (Lupinus albus) - a row of two rows of sea buckthorn species. Also good results were achieved by planting locust tree (Acacia melanoxylon). For soils of wet regions from Germany is successfully used the willow (Salix incana).

In Romania, forest recultivations were made in the mining basin of Oltenia, in the mining perimeter of Tismana, Peşteana and Roşia de Jiu (Red of Jiu) where used pine species were.

The mixture is best at the bouquets, which occupy areas of 50-100 m^2 or pure bands alternating between them, 5-10 m wide. Crop density of seedlings will be 5000-6700 per hectare with a range of 0,8-1 m row from 1,25-3 m between rows (Fodor, D.).

Agricultural re-cultivation involves a sequence of processes and draining work, re-fertilization, collection and selection of seeds and is done in two stages (Georgescu, M.):

- the first stage aims to regenerate soil fertility through crop production as: alfalfa, clover, etc.;
- the second stage, after 4-5 years, aims the basic agricultural crop production as: wheat, corn, rape, etc. (fig. 4).

In Romania, experimental research were made in the mining basin of Oltenia on the outside waste dumps of the Tismana coal pit, being used four types of crops: wheat, corn, potato and alfalfa. Another experimental research was made on the Cicani – Balta Unchiaşului (The Greybeard Pond) waste dump.

Some of the Rovinari mining basin's dumps (Cicani and Gârla dumps) were used for growing fruit trees and vine. Tree species used, were: apple and plum trees, and vine species used were: Italian Riesling and Merlot.

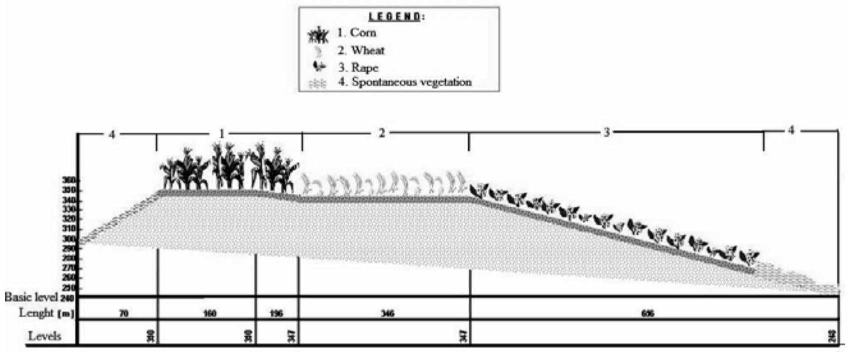

Fig. 4 Model of agricultural re-cultivation of a waste dump

2.2 Arranging the coal pit for waste storage

This function is suitable for mining closed, where geological and climatic conditions are appropriate and if is available a large amount of garbage for landfill.

The current trend worldwide in terms of mining and controlled storage of household deposits is that of a single project combines the exploitation of minerals, waste storage and environmental recovery of the land even and rendered the economic circuit.

Using sterile material derived from coal operation may be used for daily deposited layer of material over the waste and for the construction of impermeable barrier (permeability than 10^{-9} m / s).

Arrange removals deposit using sterile material, reducing investment costs by reducing or eliminating the necessity of use of building materials from the adjacent land.

Simultaneously with filing sealing in areas where storage will take place (active cells) will put in work and leachate collection system. As there is residual waste storage and filling is installed the gas storage wells for the disposal and collection pipelines that carry gas to the combustion plant. By the increasing the thickness of deposited waste (which are regularly covered with a layer of soil), the cleaning wells of the collecting and exhaust leachate system are built up. After submission of household waste can set capture biogas resulting circuits (fig. 5).

After occupying the projected volume for each storage cell is passed to the third stage, to reinsertion in the economic cycle.

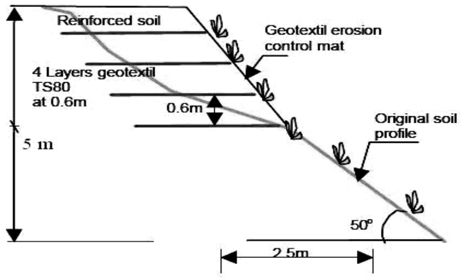

Fig. 5. Collection and storage of biogas for use of generation of electricity

2.3 Arranging the coal pit as artificial lakes like water parks and farmland pond-units

Excavated areas become artificial lakes with complex functions. Use as artificial lakes requires, besides strict compliance technical requirements for construction slopes, a number of other studies and engineering:

- Carrying out works to regulate surface water and groundwater;
- A study of geotechnical and topographic surveys before and after execution of stabilization of human structure;
- Execution of engineering works to stabilize and enhance the coal pit (fig. 6);
- Sealing platform and future lake shores;
- Monitoring the stability and mechanical behavior of the assembly formed by the lake and surrounding morphology;
- Natural or controlled flooding of coal pit's lakes;

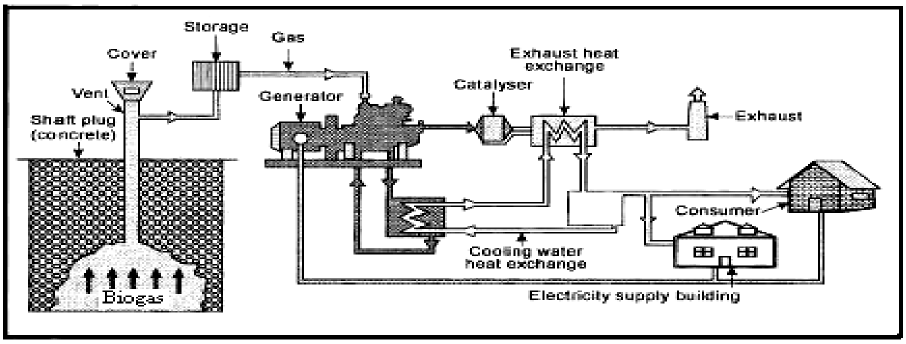

Fig. 6. Anti-erosion works for a coal pit slope using geotextile

The water park can be designed in the place of abandoned coal pits where land surface has undergone impressive. In terms of space, water parks are in the form of artificial lakes and to achieve harmony between natural and created landscapes, buildings for recreation, fun has to be covered by forest or shrubs plantations. To get a very pleasant landscape, plants such as poplars, willows, wicker, grass (turf) or different types of flowers that are suitable for specific climate area, have to be planted along or around the lake (Fig. 7).

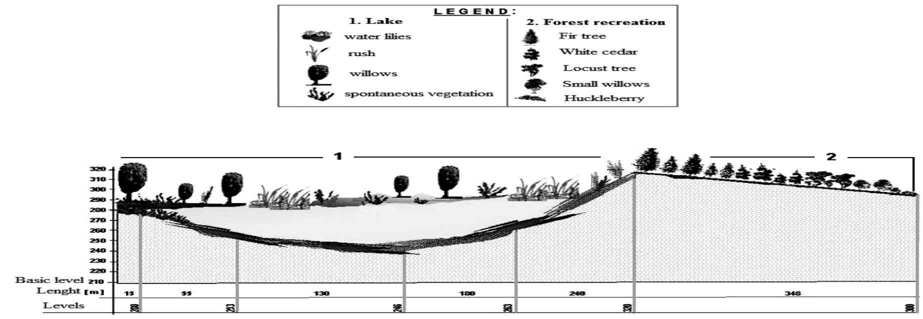

Fig. 7. Lake - type arrangement of a coal pit

In a farmland pond-unit, the grass, vegetables and grains (including an animal farm) forms a concentric cycle, encircling the pond. This process is a production cycle, where vegetables and grain are fed to pigs, vegetables and grass are fed to fish and pond silt and the excrement of both pigs and fishes are used to fertilize the soil. In this way, the land and water ecosystem interact as a compound ecosystem with multiple levels and functions (Wang Y., et all).

Currently, structural designs of this type are being applied to coal-mining collapsed areas in Tongshan County, Jiangsu Province, China.

2.4 Arranging the coal pit for scientific research

Coal pits are central elements in the landscape created by mining and provide an unusual potential for biodiversity, even higher than that offered by land unaffected. Openings generated by mining industry and active geomorphological processes can be traced easily, is unique scientific attractions. Coal pits may become an integral part of the cultural identity of the mining region , a good example is in Gera Ronneburg where in 2007 it was organized a federal exhibition of landscape design where were exposed remains of cultural heritage, and local mining (photo 1). Another coal pit from Saxonia, Brandenburg was used in 2001 for the same purpose.

Photo 1. Gera, Ronneburg (Thuringia), Germany

CONCLUSIONS

The mining activity not only destroys the valuable land resources, but also pollutes the environment surrounding the mining area.

Reclamation procedures viewed as an economic process, by integrating the use of land after reclamation, can lead to profitable post operations for mining companies and local communities, even if the economic factors were not taken into consideration during the feasibility studies or when the reclamation plan was developed. It is important to note that the cycle of ***exploitation – redevelopment – re-use*** is particularly beneficial to the sustainability of mining operations as well as achieving a balance between development, environmental, social and cultural objectives.

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Scientific Reviewers:
Prof. PhD. Eng. Romulus Sârbu

HAZARDS GENERATED BY HUMAN ACTIVITIES IN THE NORTH-EAST OF PETROSANI MOUNTAIN VALLEY

CIPRIAN NIMARĂ*

Abstract: *The social and economic development determines the acceleration of the environmental components by changing the status and their response is apparent by highlighting conflicted relations in the territory. As technological development and anthropic areas within, started to extend, there has been a contradictory enhancement with the natural environment. Following this feed-back, it will be generated a new territorial dimension and architectural space, resulting in the relief by the appearance of the inversions and critical environments.*

Key words: *human activity, hazard, anthropic landscape, vulnerability.*

1. INTRODUCTION

This study aims to identify vulnerable perimeters and hazards generated by human activity. Anthropogenic activity in the Petrosani mountain valley (figure 1.), generates many changes in the natural conditions, among which are the major negative effect on land. Inventory of the economic activities that generate such effects is the first step in properly assessing the types of land degradation by them.

The features and magnitude of impact, caused by human activities are influenced by the fact that a relatively small association on areas of intervention and that they are subject to the law of cumulative effects (Filip, S.).

2. ANTHROPOGENIC ACTIVITY THAT GENERATES VULNERABLE AREAS AND HAZARDS WITHIN

Human actions with significant environmental impact in the north-east of Petrosani basin are represented primarily by surface or underground mining, extraction and storage of tailings produced from this activity, household waste storage activities and forest exploitation activities.

* *Student Ph.D. Geographer –University of Petrosani*

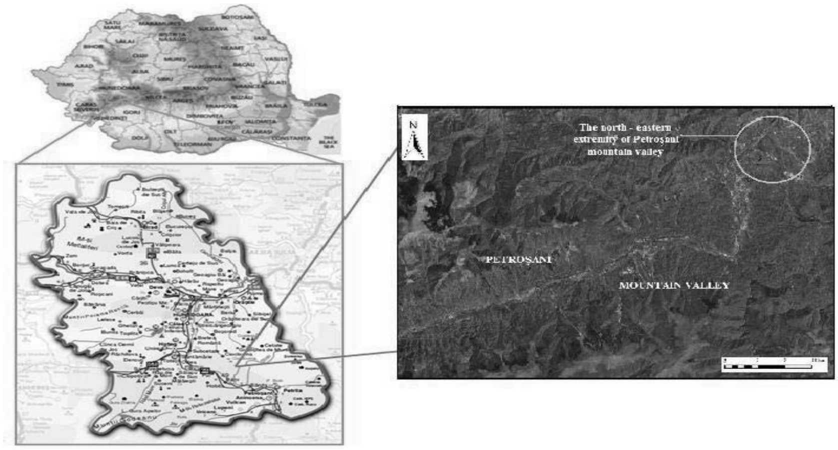

Figure 1. The mountain valley of Petrosani

2.1. Hazards caused by the mining activity

By mining activities and minerals processing, the landscape in a relative balance, changes its dynamics by an acceleration step, generating other landscapes that function in a high degree of entropy. Geomorphological elements change, creating new surface formation and accelerate soil physical and chemical processes.

The area next to the surface or underground mining is destabilized by mining, underground or surface water infiltration, vibration, explosions, mining transport that are likely to produce effective risk.

Surface land degradation occurs depending on the thickness of coal layers. Operation thin layers cause only small surface diving. Fractured rocks on the perimeter of excavation is set in motion, sending it to the massive displacement over a distance which depends on their ability to break up and fill the resulting gap.

Land areas subjected to mining subsidence in the north-eastern extremity of the Petrosani basin, can be identified in the field in the mining perimeter of Lonea (photo 2.1) and Livezeni. Next to those areas, there can also be identified cracks and irregularities on the land surface that can foretell the extended compaction process.

Fractured and unstable land affected nearly 70 individual peasant households and in some cases necessitated the evacuation and demolition of residential buildings in the Petrila town.

In the mining perimeter of Lonea and Defor, the surface deformation is visible due to roof collapse of mining works directly after mass extraction of active mining blocks III, IV and VII, layer 3.

The geomorphic risk occurs in the former coal pit Defor, producing the collapse of the banks and subsidence (photo 2.2).

Factors that make possible the mass movements of land are:

- gullies and ravines made by the rainfall erosion;
- water accumulated in the coal pit;
- underground mining.

Photo 2.1. Mining subsidence in Lonea **Photo 2.2.** Subsidence phenomenon, mining perimeter Defor coal pit

Hydrological potential risk area is determined by the possible lifting of the lake water level, which could cause flooding. Also, the storm waters, waste water from the mine and the surface diving due to underground exploitation are a hydrological potential risk factor.

In the north-east of Petrosani basin, near Petrila town, is the mining waste dumps perimeter having an occupied area of 49,59 ha. The waste dumps plateau takes place as a fan composed by five branches, forming angles between them of 9°, 14°, 16° and 24°. Active geomorphological processes are highlighted by the phenomenon of mass movement of materials, eroded surfaces, areas covered by water and forming lakes and swampy areas.

Table 1. *Type areas of degraded land in the Petrila mining perimeter*

No.	Type areas of degraded land	Area (ha)
1	Degraded land with deep erosion (gullies and ravines)	0,1
2	Land with excessive surface erosion	7,008
3	Degraded land submerge	6,21
4	Eroded surfaces form plateaus	3,025
5	Degraded land by swamps	1,48
TOTAL		**17,823**

The ravines have a 2-3 m width and 1,5-2 m depth. The high pitch slope 45-50° and the high-level differences make possible the landslide process. The land with excessive surface erosion is 7,008 ha and the degraded land submerge is 6,21 ha (table 1).

In conclusion, in the north-eastern part of the Petrosani basin were identified 10 vulnerable areas affected by mining hazards (figure 2) and the total area of degraded land in this perimeter is 162,26 ha.

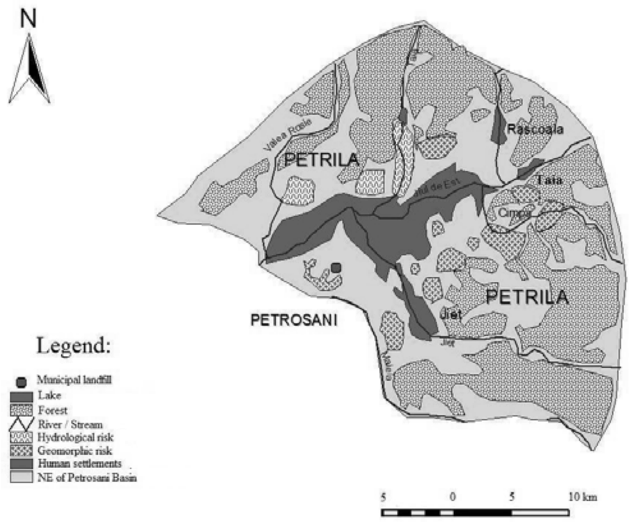

Figure 2. The anthropogenic hazards in the north-east of Petrosani mountain valley (Nimară, C.)

2.2. Hazards caused by the waste storage activity

Storage of household waste is another example of anthropogenic hazard, affecting land areas where storage mode is not made under international standards required. In this case, the household filing was made between the branches of the mining waste dumps. It is considered that the execution does not correspond with the rules because it didn't take into account the intensity and direction of the wind.

This storage of household waste is a hazard because the dust particles, the ashes and the unpleasant odors are moved by the strong wind (of south-west and south-east), to the Petrosani and Petrila town.

Also a significant risk represents the leachate that affects the soil and the fermentation gases that affects the atmosphere (Biro, C.).

2.3. Hazards caused by timber exploitation

The timber exploitation involves economic activities which take place over wide areas, which generally requires construction of accessible routes.

Construction of side roads, very near to the line of greatest slope, on a brittle substrate, was a premise for starting the rain erosion which resulted in the formation of some impressive gullies and ravines. In the north-eastern part of the Petrosani basin, the area affected by the timber exploitation is 17,50 ha.

3. CONCLUSIONS

In the north-east of Petrosani mountain valley, like in the whole basin, the human intervention process has increased exponentially and the most affected areas were the urban perimeters and areas with mineral resources. The area affected by anthropogenic hazards caused by the mining activity, waste storage and timber activity is 185,76 ha.

This study aims to be a representative one, seeking to highlight the territorial reality of the north-eastern part of Petrosani basin with the existing hazards and potential risks in a territory in which the items that has generated the system are altered by an environmental mismanagement and the final result being the emerge of a dynamic metastable equilibrium (Nimară, C.).

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Scientific Reviewers:
Prof. PhD. Eng. Romulus Sârbu

HUMIDITY, IMPORTANT FACTOR IN COAL SELF-IGNITION

CLEMENTINA MOLDOVAN*
CLEMENT IONESCU**

Abstract: *By release of some gases, explosive mixtures can occur in the Jiu Valley mines; on the other hand, there is danger of coal spontaneous self-ignition that can produce underground fires. For this reason it is necessary to avoid all the factors that might lead to the occurrence of underground fires in coal mines, where coals have self-ignition characteristics. This paper focuses especially on one of these factors that influence the occurrence of underground fires, the humidity. Both, the influence of coal inherent humidity and the underground air relative humidity over coal self-ignition are analyzed. The final results of this paper and the graphic representations emphasize the important part that humidity can play over coal self-ignition in the Jiu Valley mines.*

Key words: *coal self-ignition, humidity, coal oxidation*

1. INTRODUCTION

Mining industry is characterized by specific working conditions, imposed by natural particularities as well as by the production process character. These necessitate a series of special measures that must not endanger workers life by accidents at workplace or by illness due to particularly underground working conditions.

It is known, on one side, that in Jiu Valley mines there is a hazard of developing explosive mixtures, caused in first place by occurrence of methane and other gases; and on the other side the hazard of spontaneous coal self-ignition, that leads to underground fires occurrence. The occurrence of underground fires is unwanted because these fires would take out large quantities of coals from the economic circuit and would necessitate enormous extinguishing costs [1].

* *Assist.Prof. PhD. Ch. - University of Petroşani*
** *Assoc.Prof. PhD. Ch. - University of Petroşani*

For this reason, it is necessary to pay a particularly attention to avoid all factors and causes that could lead to underground fires occurrence, especially in mines where there are coals with self-igniting properties, like the mines in the Jiu Valley.

There were issued many hypotheses on the manner that coal self-ignition takes place, which can be incorporated in the following theories:

- pyrite oxidation theory;
- coal oxidation theory, actually considered the most important;
- phenolic theory;
- bacterial theory.

For the burning to take place, three elements are necessary to participate in this process: the fuel, oxidizing environment and ignition source.

The participation manner of the three elements and their actuating conditions lead to endogenous fires occurrence, especially in Jiu Valley coal mines.

In practice, the self-ignition of coal in the massif or from the neighborhood coal formation (apophysis, unexploited beds) can take place, as well as the stope or caved coal.

Spontaneous coal ignition represents an exothermic process that depends on coal endogenous factors (physico-chemicals and petrographics) and exogenous factors (mining and geologic) [4].

Coal in contact with the atmosphere absorbs oxygen, which affects its coking properties.

A very important effect is that coal oxidation can lead to its spontaneous ignition and creates difficulties for open-mining or underground exploitation.

Because of negative effects that the occurrence of underground fires involves, the coal self-ignition has made and makes an interest among many researchers, with a vast bibliography serving for this purpose.

From the researches presented in the technical literature, the main cause of coal self-ignition is due to coal oxidation.

The issue that arise is if this exothermic sporadic process is sufficient or not to initiate an independent burning process. The technical literature counts a series of factors that affects the oxidation process. With all of this, until now it was not possible to exactly reproduce into laboratory the same conditions like in the mine underground and the process mechanism is not completely edified.

One factor sketchily mentioned in technical literature, which has a high influence over coal spontaneous self-ignition is represented by humidity.

It is known the fact that also coal and air are capable to retain a quantity of humidity under different forms: adsorption, absorption, free and capillary water, water in form of vapors, function of the temperature and pressure conditions.

In case that atmospheric humidity that surrounds the coal is maintained constant, the coal is in equilibrium with the environment vapors. In these conditions, any heating of coal will be only due to its oxidation. When atmospheric humidity modifies, a new equilibrium is made between coal, environment and coal temperature, because adsorption-absorption or to desorption. This will accelerate or decelerate the oxidation process.

Taking account of Jiu Valley microclimate which is characterized by relative humidity between 35 ÷ 85 %, while the natural coal humidity is low (its maximum being of approximately 3 %) the question which arise is that if not these parameters can play an important part in developing auspicious conditions for coal self-ignition.

The purpose of this paper is to trace the influence, for inherent coal humidity and for the relative or atmospheric humidity as well, from mines underground over coal self-ignition.

EXPERIMENTAL PART

In order to trace the influence of humidity over coal spontaneous self-ignition, the installation from figure 1 was used.

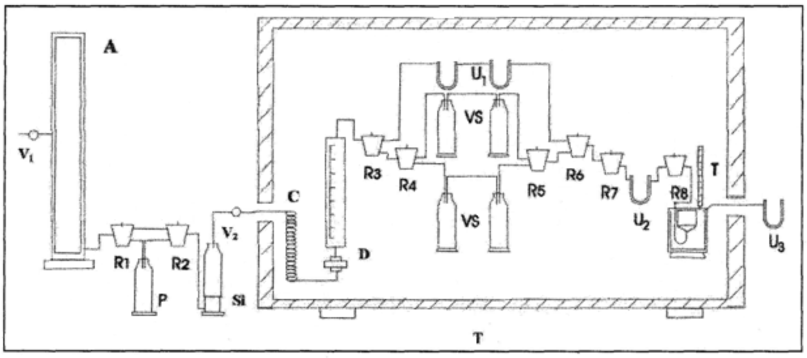

Figure 1. Installation for coal oxidation

Key:

P – pyrogallol container	V_1, V_2 – safety valves	C – copper coil
D – flow meter	$R_{1, 2, 3, 4, 5, 6, 7, 8}$ – valves	$U_{1, 2, 3}$ – tubes with P_2O_5
V_s – recipients with substances [(K_2CO_3 dist., (NH_4SO_4))]	Si – recipient with silica gel	A – air tube

By analyzing coals behavior in hydrogen peroxide, it was concluded that they have different behavior, by taking into account the allure of resulted curves in graphical representation of temperature function of time (Fig.2):

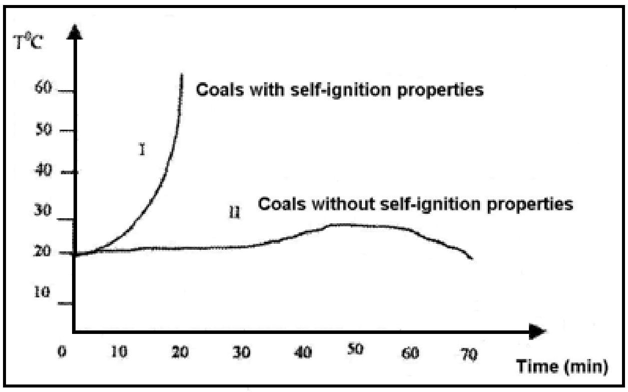

Figure 2. Coal oxidation with hydrogen peroxide – graphical representation of temperature function of time

Curve I is specific to coal from E.M. Petrila and curve II is specific to anthracite from Schela without self-ignition properties [2].

Coals samples were placed so as to be in contact with the air coming from an air cylinder. For pre-dry, the air passes through the tube with silica gel. After drying, the temperature of used air is conditioned by its passing through a copper coil. In the next place, the air enters to a flow-meter in order to maintain a 2,5 ml/min constant flow and after that it passes through the tube with P_2O_5 in order to achieve a 0% humidity.

The relative humidity: 45%, 80%, 100% are achieved by passing the air through their containing washing recipients; K_2CO_3, $(NH_4)_2SO_4$ and distilled water (tab.1).

Table 1. Saturated solutions of salts with given relative humidity

Crt. No.	Substance	Relative humidity above the solution at 20°C (%)
1	Potassium carbonate (K_2CO_3)	45
2	Ammonium sulphate $(NH_4)_2SO_4$	80
3	Distilled water	100
4	Calcium chloride ($CaCl_2 \cdot 6H_2O$)	25
5	Potassium chloride (KCl)	86
6	Sodium chloride (NaCl)	76
7	Sodium azotate	63
8	Calcium azotate $Ca(NO_3)_2 \cdot 6H_2O$	55

During work, the entrance of gas into the solution and carrying of the liquid into the circuit are avoided. In the end, the air enters into the calorimeter, where it meets the coal that is placed in a G3 funnel; and the resulted heat by oxidation is measured with the help of a Beckmann thermometer.

In the first stage of the research coal humidity and the relative humidity of air were adjusted; and coal quantity and air flow were kept constant.

In the second stage the granulometry, coal humidity and relative humidity were adjusted [3].

The obtained data are summarized in tab. 2, 3, 4, and 5 and graphically represented in fig. 3, 4, 5 and 6.

In order to compare the behavior of self-igniting coals and a coal without self-ignition tendency some values of the temperature function of time were tracked, for the anthracite coal from Schela of 0,2 mm granulation and W=4,2% humidity, values given in tab. 6 and graphically represented in figure 7.

Table 2.

	Time (h)													**Relative humidity (%)**
	0	1	2	3	4	5	6	7	8	9	10	11	12	
Temp. °C	1,70	1,60	1,80	1,35	1,30	1,36	1,41	1,43	1,41	1,53	1,69	1,86	1,90	0
	1,70	1,73	2,18	2,69	3,02	3,12	2,93	2,69	2,45	2,17	1,97	1,87	1,88	45
	1,70	1,84	2,21	2,58	2,60	2,54	2,43	1,71	1,57	1,60	1,77	2.03	2,10	80
	170	2,25	2,55	2,66	2,64	2,60	2,61	2,04	1,90	1,75	1,79	2,04		100

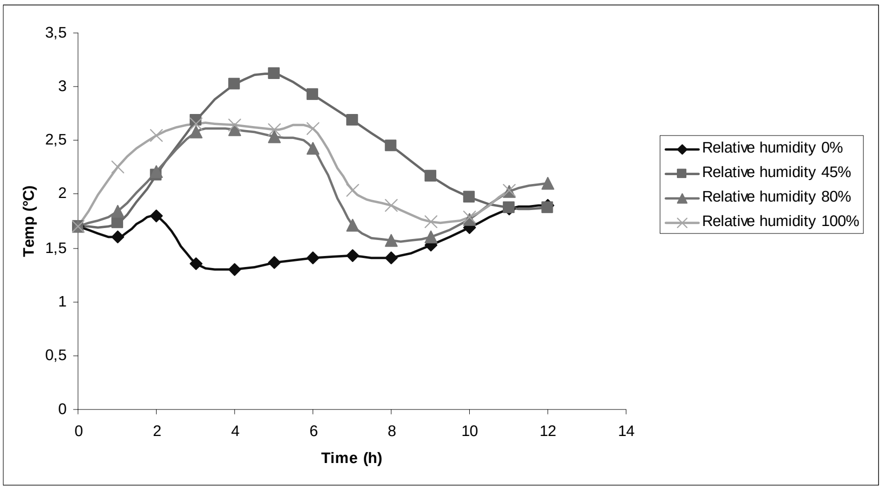

Figure 3. The curves alure for coal with W=2,6% and 0,2 mm granulation

Table 3.

	Time (h)													Relative humidity (%)
	0	1	2	3	4	5	6	7	8	9	10	11	12	
temp. °C	1,70	1,20	0,80	0,88	1,00	1,40	1,84	2,30	2,90	3,20	3,30	3,4	3,50	0
	1,70	1,68	1,60	1,50	1,38	2,20	2,80	3,00	2,76	2,70	2,88	2,98	3,04	45
	1,70	1,59	1,40	1,22	0,95	0,72	0,48	0,68	0,78	0,84	0,94	2,03	2,16	80
	1,70	1,60	1,58	1,40	1,20	1,26	1,40	1,60	1,80	1,88	2,10	2,38	2,50	100

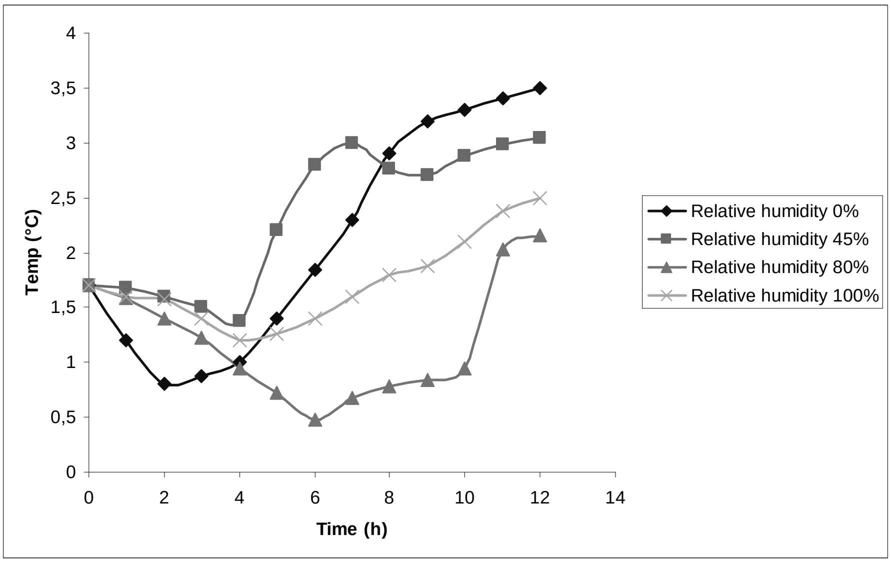

Figure 4. The curves alure for coal with W=18,7 and 0,2 mm granulation

Table 4.

	Time (h)													Relative humidity (%)
	0	1	2	3	4	5	6	7	8	9	10	11	12	
temp. °C	1,70	1,76	1,93	2,13	2,37	2,35	2,28	2,21	2,10	2,12	2,16	2,35	2,40	0
	1,70	2,36	2,76	3,12	3,27	3,31	3,28	2,88	2,80	2,75	2,68	2,71	2,80	45
	1,70	2,48	2,79	3,08	2.94	2,92	2,68	2,48	2,28	2,08	1,80	1,76	1,60	80
	1,70	2,01	2,18	2,40	2,63	2,70	2,65	2,34	2,14	1,96	2,16	2,46	2,60	100

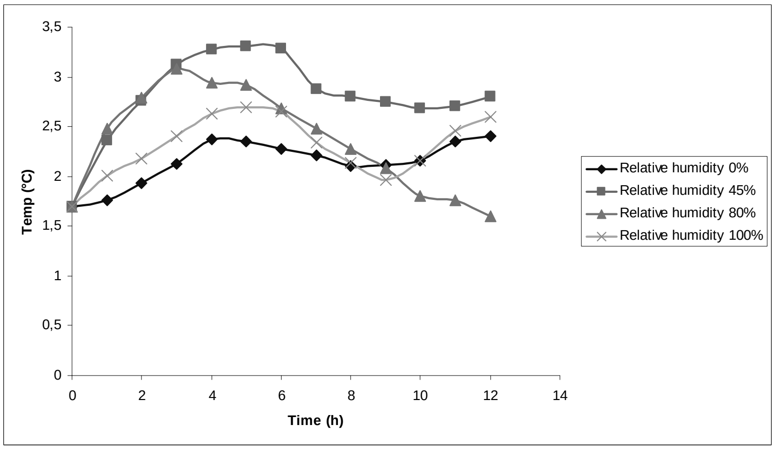

Figure 5. The curves alure for coal with W=2,9% and 0,5 mm granulation

Table 5.

	Time (h)													Relative humidity (%)
	0	1	2	3	4	5	6	7	8	9	10	11	12	
Temp. °C	1,70	0,70	0,31	0,45	0,30	0,69	0,49	0,47	0,89	1,24	1,53	1,70	1,80	0
	1,70	1,50	1,31	1,30	1,40	1,24	1,48	1,27	0,94	0,64	0,60	0,86	0,94	45
	1,70	1,22	1,09	0,94	0,45	0,06	0,02	0,22	0,30	0,38	0,46	0,60	0,74	80
	1,70	1,30	1,12	1,17	1,20	1,18	1,04	0,79	0,53	0,30	0,40	0,78	1,08	100

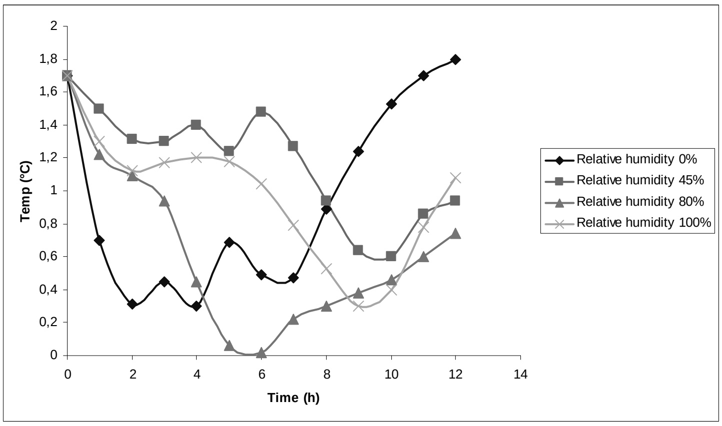

Figure 6. The curves alure for coal with W=15,8 and 0,5 mm granulation

Table 6.

	Time (h)													Relative humidity (%)
	0	1	2	3	4	5	6	7	8	9	10	11	12	
Temp. °C	1,70	1,72	1,70	1,67	1,60	1,54	1,52	1,55	1,56	1,54	1,54	1,55	1,56	0
	1,70	1,80	1,85	1,82	1,75	1,62	1,50	1,42	1,45	1,50	1,52	1,56	1,58	45
	1,70	1,75	1,78	1,65	1,50	1,42	1,30	1,34	1,40	1,43	1,45	1,49	1,50	80

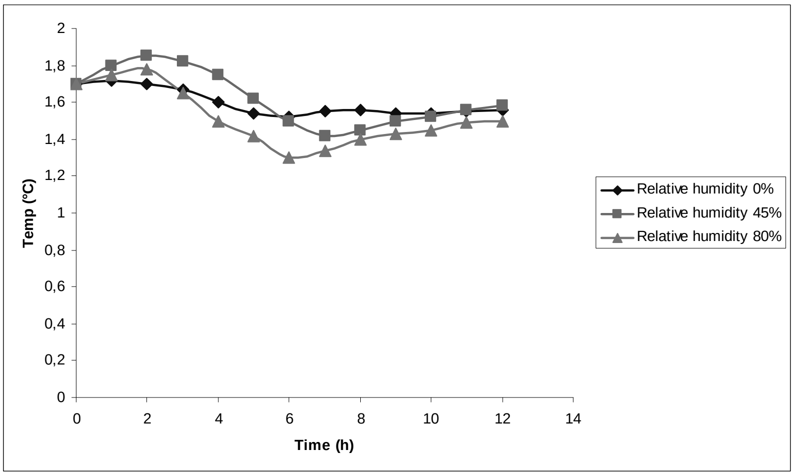

Figure 7. The curves alure for anthracite from Schela with W=4,2% and 0,2 mm granulation

3. CONCLUSIONS

From the graphical representations the following conclusions have come:

- the exothermal effect, which follows the coal oxidation process, is higher as the coal humidity and granulation are lower;
- self-ignition conditions became propitious, especially when the coal has a natural low humidity, case in which it doesn't exist the possibility of a cooling process to occur, as a result of coal evaporation;
- coal self-ignition is influenced in some measure by the air relative humidity;
- a higher exothermal effect is recorded in case of 45% relative humidity, followed, with small exceptions, by 100% and 80% relative humidity, especially in case of low humidity coals;
- in case of low humidity coals, in the same conditions of relative humidity, the increase of granulation has influence in some manner over the exothermal effects, some diagrams with similar allures being recorded, but with not so pronounced maximal values;
- as the coal humidity increases, for the same air relative humidity (45%, 80%, 100%), a decrease of temperature takes place at the beginning of oxidation, followed by an exothermal effect that is higher as the coal oxidation time increases;
- the increase of coal oxidation time in the presence of air relative humidity leads to the occurrence of some exothermal effects with higher maximal values than

the maximal values recorded at the beginning of oxidation, which explains the increase of temperature in time and occurrence of underground fires;

• the presence of relative humidity in the air represents an important factor, that seems to have a high role and effect in encouraging the self-ignition of Jiu Valley coal;

• also, air relative humidity has a very small influence over the coal without self-ignition properties, but the increase of temperature is insignificant and can not lead to underground fires;

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Scientific Reviewers:
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MINING SECTORAL PROFILE IMPACT ON WORKING CONDITIONS: SAFETY ISSUES IN JIU VALLEY REGION

SABINA IRIMIE*
VIRGINIA BĂLEANU**
ANDREEA IONICĂ***

Abstract: *The paper draws a profile of the mining sector from the working conditions perspective as it is seen comparative with other sectors in European Union (EU), emphasizing the occupational safety side. Thus, in order to highlight the sectoral profile impact on the safety of workers in regions with mining activity as reflected by the state of accidents at work, we present and discuss the particular case of the Jiu Valley – a Romanian region still heavily dependent on mining.*

Key words: *mining, working conditions, occupational safety, accidents at work, incidence rate.*

1. INTRODUCTION

A basic component of EU political actions addresses the concern to improve the working conditions, including the safety and health of workers, as a major driver of the economic and social progress. So, the ultimate aim of the policy in this area is to reduce accidents at work and prevent the suffering of workers which affect their families and employers too, with repercussions on society as a whole. After outlining the background of development the framework for supporting the needed actions at the EU level our paper points out the sectoral perspective of working conditions. Such a perspective reveals differences between the economic activities, implying different level of risk for health and safety of people who works in some sectors comparative with others. Against this background, the paper aims to profile the mining sector and explore the sectoral profile impact on the safety of workers in regions with mining activity as reflected by the state of accidents at work. Accordingly, we present these safety related issues in one such representative region of our country – Jiu Valley –

* *Prof. PhD. Eng. - University of Petrosani, bina2932@yahoo.com*
** *Lecturer PhD. Ec. - University of Petrosani, ginabaleanu@yahoo.com*
*** *Assoc. Prof. PhD. Eng. - University of Petrosani, andreeaionica@yahoo.com*

making a comparative analysis of the main indicators on accidents at work (regional vs. national level).

2. MINING SECTORAL PROFILE FROM WORKING CONDITIONS PERSPECTIVE AT THE EU LEVEL

The framework for supporting the needed measures in the Member States and monitoring the progress at the European level was continuously developed in order to harmonize the national regulations, as well as the criteria and methodologies used for recording and processing information in the field of Occupational Safety and Health (OHS). In this context, several interrelated projects carried out since 1990 such as those providing an overview of the state of working conditions throughout Europe – European Working Conditions Survey (EWCS), and respectively the methodology for gathering comparable data on accidents at work – European Statistics on Accidents at Work (ESAW). Also, the European Commission defined a Community strategy for the period 2002-2006 „based on an overall approach to wellbeing at work which took account of changes in the workplace and the emergence of new risks, especially those of a psychosocial nature. ... The new strategy for 2007-2012... aim for a 25% reduction in the total incidence rate of accidents at work by 2012 in EU-27 by improving health and safety protection for workers and as one major contribution to the success of the Growth and Jobs Strategy" (COM/2007/62final, p.2). Actions in the field of health and safety at work are now supported by the PROGRESS programme (2007-2013) – the programme established to support financially the implementation of the objectives of the EU, as set out in the Social Agenda.

The latest edition of the EWCS was carried out in 2005 by Eurofound (European Foundation for the Improvement of Living and Working Conditions) covering data from EU-25 countries as well as Bulgaria, Romania, Turkey, Croatia, Norway, and Switzerland. Based on this survey results (completed with those of the three precedents) was performed a secondary statistical analysis with the aim of providing a sector perspective of working conditions. According to the recently published report, the study used several working conditions and outcome variables for drawing sectoral profiles (Jettinghoff and Houtman, 2009, p.4) as follows:

- Physical environment: ambient conditions; ergonomic conditions;
- Time: duration of work; non-standard working hours; work-life balance;
- Organisational environment: job demands; job control; skilled work;
- Social environment: social support; discrimination;
- Outcomes: mental health problems; musculoskeletal health problems; absence due to health problems; job satisfaction.

As a main result, the report presents the position of the 26 sectors considered (NACE classification, 2-digit level) into a four-cells matrix:

- Sectors with favourable working conditions and outcomes (top three: Insurance, Financial intermediation, Education);
- Sectors with favourable working conditions and unfavourable outcomes (Public administration, Public utilities, Health and social work);

• Sectors with unfavourable working conditions and outcomes (Agriculture, Land transport, Hotels and restaurants, Mining and quarrying);

• Sectors with unfavourable working conditions and favourable outcomes (Wholesale and retail, Water, air sampling activities).

So, mining and quarrying is one of the sectors perceived by the European workers as providing unfavourable working conditions and outcomes. This profile of unfavourable sector is marked by the risks and problems associated with the following variables: ambient and ergonomic conditions, working hours, job control, skilled work, stress and musculoskeletal diseases. The report also discuss some differences between sectors by socio-demographic characteristics (gender, age, years in organisation, employment status, employment contract, educational level, etc.) that may partly explain the previously mentioned ranking of the sectors. From this point of view, mining and quarrying is characterised as typical 'male-dominated' (90.4%), with relatively high values of: the average age of workers (45.2 years), the percentage of employees (91.8%), with a permanent contract, and the incomes level. Also, it is a sector with relatively low values for the rate of staff turnover (workers staying with their company for a considerable period) and for the educational levels.

3. OCCUPATIONAL SAFETY ISSUES IN THE JIU VALLEY REGION

The ESAW methodology considers two main categories of indicators on accidents at work (DG EMPL and EUROSTAT, 2001, p. 21): the number of accidents and the incidence rate (frequency related to the reference population of persons in employment).

The first indicator evolution from 2001-2002 to 2007-2008 years (showing the distribution by the major types of accidents – fatal and non-fatal) is presented for Romania and for the Jiu Valley region in table 1.

Also, for comparative purposes, in the table 1 were added comparable data on number of accidents at work by type (fatal and non-fatal) at the EU level (available data for 2001-2002 years). Thus, the overall trend of the accidents number evolution (as absolute values) seems to be positive both at the national and regional level, aligning with European evolution. Obviously, this situation may be related with the specific measures and actions in our country during this period (between 2001 and 2008), including the adoption of a new OHS legislation (Law 319/2006).

Table 1. *The number of accidents at work by type*

Specifications	Fatal accidents	Non-fatal accidents
ROMANIA*		
Year: 2001	440	6368
Year: 2002	415	5854
Year: 2007	472	4391
Year: 2008	331	4261
Jiu Valley region**		
Year: 2001	18	1338
Year: 2002	17	1164
Year: 2007	7	286
Year: 2008	17	247

EU***		
Year: 2001	4922	4702295
Year: 2002	4790	4408616

Sources: *National Labour Inspection Statistics; **Territorial Labour Inspection of Petrosani Records; ***EUROSTAT

On the other hand, comparing the 2001-2002 data related to the percentage of fatal accidents in the total number of accidents allow us to observe that if this percentage was arround 0.1% at the EU level, its values were and remain much higher in Romania (over 6.5% in every year, including 2007 and 2008). Also considerable higher values resulted for the Jiu Valley region too, where records show an increase of this percentage (from 1.3%-1.4% in 2001-2002 to 2.4% in 2007 and respectively 6.4% in 2008). But, even these regional values are apparent more favourable than those for the country level, they have to be related to the reference population of persons exposed to the risk of accidents at work (employees) at the respective level (national and regional). That is, a realistic interpretation requires a comparative analysis of the incidence rates (table 2).

Table 2. *Incidence rates of accidents at work by type*

Specifications	**Fatal accidents**	**Non-fatal accidents**
ROMANIA		
Year: 2007	10.00	93.02
Year: 2008	6.89	88.66
Jiu Valley region		
Year: 2007	19.51	797.19
Year: 2008	46.58	676.71

Primary data sources: National Institute of Statistics (NIS, 2007; NIS, 2008), and respectively Statistics Directorate of County Hunedoara

Incidence rates in table 2 were calculated as number of accidents per 100000 employees (according ESAW methodology), based on national and territorial statistics on employment.

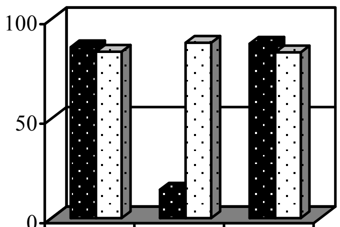
The results of such an analysis for the last two years show a very different situation: the overall incidence rate (considering the total number of accidents) at the national level is 103.01 in 2007 and decreases in 2008 (95.55), but the values resulted in the same years for the Jiu Valley region are 816.70 and respectively 723.29. Furthermore, as it can observe in table 2, the incidence rate of fatal accidents in the two years at the regional level are 19.51 and 46.58, also much higher than at country level (10.00 and 6.89).

It can't be possible to explain this contradictory situation without considering that coal mining was for a long time the core activity in the Jiu Valley region. On the one hand, in this area is located the country's most important geological deposits of hard coal. On the other hand, the geo-mining characteristics are complex (e.g. a quite complicated tectonics, medium and high depths of exploitation, with frequent inclusions of sterile/rocks, strong emissions of methane and underground water, etc.).

Therefore, the working conditions related to the physical environment are very difficult comparative with those from most of other activities (including the ones in the same branche of mining and quarrying). Obviously, it is a specific major risk factor for the safety of people who works in these conditions. Even if after 1990 the number of employees in mining decreased continuously (as a direct consequence of sectoral restructuring actions) it still remains an important component of reference population (total employees) in the region (over 30% in 2007-2008 years).

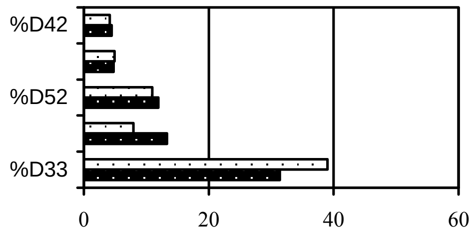
According to the institutional local body responsible with recording and monitoring OSH problems – Territorial Labour Inspection Office of Petrosani (TLIOP), over 80% of total number of accidents at work in the Jiu Valley is constantly registred in the hard coal mines. Indeed, the records for the two last years allow us to observe the dominant relative contribution of the accidents from mining to the accidents in the region both as weight in total (%TA) and in the category of non-fatal accidents (%NFA). Moreover, a maximum weight of 88.2% appears in the category of fatal accidents (%FA) registred in 2008 (see figure 1).

A further causal analysis based on TLIOP available data on distribution of the work accidents registred in the two years shows mainly the same *deviations* from normality that leading to accidents. The term *Deviation* is defined in the ESAW methodology (DG EMPL and EUROSTAT, 2001, p. 20) as: "…the description of the abnormal event, i.e. the Deviation from the normal working process. The Deviation is the event that triggers the accident". There are ten types of deviations considered in the ESAW classification system (codified with two numeric characters, from 00 to 99). According to the TLIOP records for 2007 and 2008 years the prevalent types of deviations leading to the work accidents in the Jiu Valley region (reaching about 80%) are: Breakage, bursting, splitting, slipping, fall, collapse of Material Agent - Not specified (30 type); Loss of control (total or partial) of machine, means of transport or handling equipment, handheld tool, object, animal - Not specified (40 type); Slipping - Stumbling and falling - Fall of persons - Not specified (50 type).



	%TA	%FA	%NFA
■ 2007	86	14,3	87,8
□ 2008	83,7	88,2	83,4

Fig. 1. Weights of accidents at work in mining



	%D33	%D44	%D52	%D51	%D42
□2008	39,02	7,95	10,98	4,92	4,17
■2007	31,4	13,31	11,95	4,78	4,44

Fig. 2. Top five deviations leading to work accidents mining

Figure 2 presents the resulted top five deviations (D) from the above mentioned types leading to the work accidents in the Jiu Valley region (with their frequences of occurrence in the two considered years), codified and labeld according the ESAW system as follows:

- 33 – Slip, fall, collapse of Material Agent - from above (falling on the victim);
- 44 – Loss of control (total or partial) - of object (being carried, moved, handled, etc.);
- 52 – Slipping - Stumbling and falling - Fall of person - on the same level;
- 51 – Fall of person - to a lower level;
- 42 – Loss of control (total or partial) - of means of transport or handling equipment, (motorised or not).

As it can observe, the first rated deviations may be explicitly or implicitly related with causes depending on the working conditions previously profiled for the underground mining activity in the Jiu Valley region.

4. CONCLUSIONS AND INTENTIONS

Becoming a Member State of the EU, Romania committed to the Community policy and goals established in the field of OHS. Accordingly, were adopted some important measures for aligning the national regulations and monitoring systems, several with refer to mining and quarrying sector (e.g. Government normative act no.1049/2006, or Labour Minister order no. 448/2008).

Our paper make a contribution on the factual side by revealing that despite such measures leading to a positive trend in evolution of work accidents number at the national level incidence rates still exceed the ones at the EU level, and a plausible explanation may be the regional differences induced by the sectoral profile of working conditions.

The results of our analyses for the Jiu Valley as a region marked by the impact of unfavourable profile of the mining sector may be the starting point for:

(1) similar studies directed to identify other regions with incidence rates of accidents at works higher than those at the national level;

(2) additional adequate measures and efforts for improving health and safety protection for workers in such regions.

Also, a future research direction for us is to explore the psychosocial emergent risks of mass-layoffs in Romanian mining sector.

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Scientific Reviewers:
Prof. PhD. Eng. Florian Buşe

SUBSTANTIATION THE DISCOUNT RATE OF CASH FLOWS IN THE ECONOMIC EVALUATION OF MINING PROJECTS

MIRELA ILOIU*
SORIN ILOIU**
DIANA CSIMINGA*

Abstract: *The magnitude of uncertainties in a mine development project is much larger than in most other manufacturing industries. The determination of the discount rate to be implemented for a given project is a very difficult task because it has to reflect the project's risk and there are many risk factors that have been considered in order to substantiate this parameter.*

Keywords: *mining projects, evaluation, discount rate, cash flows, risk, uncertainty, IRR, NPV, interest rate, country risk*

1. INTRODUCTION

Investment decision in mining projects is usually made after an economic evaluation as well as in every business application. The evaluation process involves the considering of risks associated with mineral exploration and development. These are commonly classified as technical, economic and political risks, and are accounted for in the investment decision by changing the discount rate. Thus, the determination of the discount rate to be implemented for a given project is the most difficult item of economic evaluation process.

2. DETERMINATION OF THE DISCOUNT RATE

All modern texts on project evaluation conclude that the preferred methods of evaluation, when sufficient data is available, are those that incorporate annual cash flows projections and that recognize the time value of money: the net present value (NPV) and the internal rate of return (RIR).

* *Lecturer, PhD. Eng. - University of Petrosani, mirelailoiu@yahoo.com*
** *Ph.D.Eng. - University of Petrosani, soralisu@yahoo.com*

The IRR evaluation generates a percentage figure which is equal to the interest rate at which the project capital would have to be invested to generate the same series of annual cash flows that the project will generate.

The NPV gives the value of the project in as a monetary unit today. Each year's cash flow is discounted to the present at a predetermined discount rate which reflects the project risk and the investor's minimum investment criteria. The NPV is the sum of these discounted annual cash flows.

Unfortunately, the literature on discounted cash flow evaluation does not deal specifically with the selection of discount rates for mineral project evaluation. Most texts focus on the calculation of the corporate cost of capital because it is possible to determine a discount rate that is appropriate for an individual project, on the basis of industry expectations for project returns (IRR), the risk factors associated with mineral projects in general, and the risks related to the specific project.

Economic and finance theory proposes the use of the corporate cost of capital as a discount rate. This value is the weighted average cost of the funds available to a company, including equity (common stock), debt (after tax rate) and preferred shares. Referred to as the Weighted Average Cost of Capital (WACC), it is expressed as an interest rate and is calculated as follows:

$$r_{WACC} = r_e p_e + r_d p_d + r_p p_p \qquad (1)$$

where:

r_{WACC} – weighted average cost of capital (expressed as %)

$r_{e,d,p}$ – proportional costs of equity capital, debt (after tax) and preferred stock (all expressed as %)

$p_{e,d,p}$ – proportions of equity capital, debt (after tax) and preferred stock that make up the corporate capital where $p_e+p_d+p_p = 100$

For evaluations on an all equity basis only the cost of equity capital needs to be considered. The Capital Asset Pricing Model (CAPM) is perhaps the most widely used method of assessing the cost of equity capital and expressing it as an interest rate. The basis of this method is that the return on an individual corporate stock can be related to the stock market as a whole by the relationship:

$$r_e = f + R\beta \qquad (2)$$

where:

r_e – expected return on the common stock

f – risk-free return (usually based on government bond rates)

R – risk premium of market returns above long term risk free rates

β – beta factor for the common stock

The beta factor expresses the variability of the common stock with respect to the variability of the market as a whole. By definition, the beta of the market is 1.00.

There are some disadvantages with using a market based beta to evaluate an individual mineral project:

• betas measure the variation in a stock price relative to the market – as the market fluctuates so does the beta;

- betas measure the variability of the share price of an entire company, not of an individual project;
- beta values for a company (or industry) vary over time, implying that the value of a specific project would vary over time (via the discount rate) with the fluctuations of a company's share price;
- relative betas for gold and base metals vary over time so there is not necessarily a consistent relationship between gold and base metals discount rates (although the pattern has been to apply lower discount rates to gold projects).

In practice, for cash flow evaluations at the feasibility study level of projects in low risk countries, mining companies use a discount rate in the region of 10% for evaluations in constant (real) currency, at 100% equity, after tax. This is based on:

- a survey conducted by CIM Mineral Economics Society which indicated that they were suing the following rates for feasibility studies (fig.1):

  base metals 11.3%
  gold 8.8%
- discussions with other mining companies;
- published evaluations by mining analysts;
- direct experience in studies undertaken for mining companies;
- various published references.

A discount rate in the 10% range seems to have no theoretical basis, other than the fact that a 10% rate of return (no inflation) after taxes is a reasonable rate of return compared with the return on government bonds. Since this rate is used by major mining investors to make decisions that involve large amounts of money, it must be felt to have validity. The conditions under companies apply this rate are specific:

- constant currency. It is difficult to obtain agreement on inflation forecasts and many evaluations avoid the problem by leaving inflation out;
- 100% Equity. The reason of 100% equity cash flows is that an evaluation should measure the inherent value of a mineral project, not the ability of an owner to finance a project on favourable terms. Financing is as much a function of the owner's credit rating and the money market as the project itself. If financing is involved it would be necessary to modify the discount rate accordingly, by means of a lower discount rate to reflect the lower risk in the debt portion.

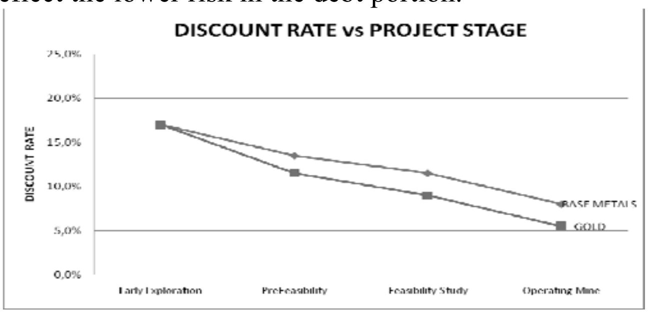

Fig.1. Discount Rate vs. Project Stage

- After tax. Since tax is a cost of operating, it must be included in the calculation of cash flows With the exception of a radical change in taxation policy (which is a function of country risk) it is possible to make an accurate estimate of the amount and timing of the tax liabilities incurred by a project since the method of tax calculation is set out in detail in tax legislation.

- Feasibility studies. This condition implies a high level of data development and a high level of certainty. The term feasibility study has a specific meaning for mineral projects, particularly to the banks and the major stock exchanges.

3. DISCOUNT RATE COMPONENTS IN A MINERAL PROJECT

A discount rate for a mineral project comprises three principal components: the risk-free interest rate, mineral project risk and country risk.

Risk-free interest rate is the value of the long-term, risk-free, real (no inflation) interest rate.

Mineral project risk include risks associated with reserves (tonnage, mine life, grade), mining (mining method, mining recovery, dilution, mine layout), process (labour factors, plant availability, metallurgy, recoveries, material balances, reagent consumption), construction (costs, schedules, delays), environmental compliance, new technology, cost estimation (capital and operating), and price and market.

Country risk refers to risks that are related to country-specific social, economic and political factors.

Using these components, it is possible to calculate a project specific discount rate:

  Real, risk-free, long-term interest rate
+ Mining project risk (varies with level of knowledge)
+ Country risk
= Project specific discount rate

3.1. Mineral Project Risk Component

The knowledge of a mining project at the feasibility study stage describes a certain comfort level and a degree of certainty as to the outcome of the project, and therefore a measure of risk, then reflected in the selection of the mining project risk component of the discount rate.

Studies are often made at much earlier stages of project development than the feasibility study. For example, a broad order-of-magnitude study is usually undertaken to rank and possibly reject potential projects in the early stages. A pre-feasibility study is undertaken when more data are available and is used to justify continuing expenditures towards a final feasibility study. Because these studies are made at earlier stages of development, there is less data; the degree of uncertainty is higher so the risk level higher and the discount rate will higher accordingly.

As the project moves toward the feasibility stage and into detailed design, construction, start-up and full operation, the uncertainty associated with the risk components is reduced. Once construction is complete, the capital cost risk is reduced to zero, since all of the capital has been spent and the costs are known. Similarly, uncertainty related to operating costs diminishes after the first year of operation. Metallurgical recovery levels are well established after several years of operation.

3.2. Country Risk

Traditionally, there are two mining nations that have been considered to have zero risk with regard to political and economic stability, the country risk portion of the discount rate being zero. But not all projects are developed in countries with zero country risk, so it is necessary to assess the effect that the geo-political location of a mineral project can have on the discount rate and valuation.

The components of country risk are listed in the Table 1.

The level of risk varies in time and from country to country. Measures of country risk can be obtained from different sources, but the difficulty is to obtain a complete listing of all countries where mining may take place, and to obtain a country risk figure expressed as an interest rate in order to be added to the discount rate.

There are three main sources of country risk measurements:

• Country Rating Services: some agencies provide country risk ratings that take the form of a score assigned to a country on the basis of several significant variables: debt levels, debt repayment record, current account position, economic policy, political stability. The disadvantage is that the scores cannot be readily converted to discount rate components.

Table 1. *Components of Country Risk*

Political Risk	Government stability Political Party Constitutional Risk Quality of government Foreign ownership policy (risk of nationalization) Government crises Taxation instability Environmental policy, environmental protectionism
Geographic Risk	Transportation Climate
Economic Risk	Currency stability Foreign exchange restrictions
Social Risk	Distribution of wealth Ethnic or religious differences within the indigenous population Literacy rate Corruption Labour relations

• Bank Rating Services: banks express their opinion of a country risk level in two ways: by the terms of the loans they will make to a country, meaning life and interest rate, and by a country credit rating. The former are confidential and not generally available. The later are published regularly and are expressed by a letter scale. But because the scale excludes many countries where mining is carried out, it cannot be used for determining a discount rate.

• Forfaiting Rates: Forfaiting rates are the discount rates that forfaiters apply when purchasing governments' bonds and include a basic interest rate and a risk component. They are expressed as interest rates so they are useful for estimating discount rates.

4. CONCLUSIONS

The risk associated with a project varies with the stage of development of the project. This variation must be reflected in the discount rate that is used to evaluate the project. Each project has a specific set of risk characteristics. Although a consistent set of criteria for feasibility studies is helpful to provide a common basis for comparison, there aren't two identical projects.

Increments of country risk can range from 0% in low risk countries to values as high as 10% and more. These can increase a discount rate substantially and have a corresponding reduction in the NPV of the project. It is important to distinguish between the IRR used for decision making and purposes and the discount rate used to value the NPV of a property. An exploration prospect that indicates an IRR of 15% may be worth spending more money on, but one may use a 20% discount rate to determine what to pay for it. The 15% reflects the project's potential, but the 20% reflects its risk at the exploration stage.

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THE TITLE OF THE PAPER WILL BE WRITTEN WITH CAPITAL LETTERS, CENTERED, AT 7.0 cm FROM THE UPPER EDGE OF A4 FORMAT, ALONG THE ENTIRE WIDTH, TIMES NEW ROMAN, 14 POINTS, BOLD.

· (14 points)

· (14 points)

The first name and the family name will be written with CAPITAL LETTERS, BOLD, CENTERED, 130 mm width, Times New Roman, 12 points, each name being followed by number, *and in the footnote the didactic and scientific degree, the position and place of work of the authors (possibly fax, E-mail) are indicated, with italics, 130 mm width, Times New Roman, 11 points.*

· (12 points)

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<Tab> **Abstract: (Times New Roman, 10 points, bold)** The Abstract will only be in English and will have between 50 and 100 words, arranged all over the entire width, single, left – right alignment, all over the width of the printed space, Times New Roman, 10 points.

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<Tab> **Key words: (Times New Roman, 10 points, bold)** 5 – 10 words, in English, along the entire width, Times New Roman, 10 points.

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<Tab> **1. THE TITLE OF THE CHAPTER IS WRITTEN WITH CAPITAL LETTERS, BOLD, TIMES NEW ROMAN, 11 POINTS AND WILL BE NUMBERED WITH ARABIC NUMBERS.**

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<Tab> The paper text paragraphs will be left – right aligned. The lines of the entire text will be single spaced with Times New Roman fonts, size 11 points.

The paper will be edited in a language of international circulation (English, French, German) on A4 format (210 x 297), page setup: top 5,6 cm, bottom 4,6 cm, left 3,7 cm, right 3,7 cm, in WORD or WORDPERFET or WINDOWS and the number of pages must be even between 4 and 16. The size of the symbol is given next and is

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* *Professor Eng., Ph.D. at the University of Petroşani, adresa email*

written in the instructions used to assist in editing the text. It indicates a blank line and does not show in the text of the paper, being obtained by pressing <ENTER>. The size of a <Tab> is 1,27 cm. No blank line is left between the paragraphs. Between the last line before the title and the title of the chapter and between the title and the first next line a blank line is left for each.

<Tab> **1.1 Subchapter. Subchapters can be used in the text, numbered with the number of the chapter and a number showing the number of the subtitle within the chapter. The subchapters and the numbers are Times New Roman, 11 points, bold.**

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Figures should be enclosed in the text in the order of their presentation, as far as possible on the page where reference is made to them. They shall be numbered with Arabic numbers. Black – and – white, high contrast figures are recommended. Photos can be used as well, but they should be of good quality, clarity and sufficient contrast. The figures will have a legend (name of the figure), which, along with the number, will be written underneath with Times New Roman, 10 points, centered as to the figure. The figures will be surrounded by text.

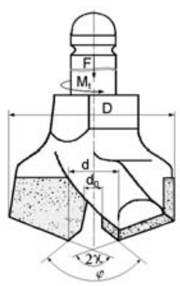

Fig. 1. Detachable bit

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The tables will be enclosed in the text, a 10 points blank line being left above and under the table, and will be numbered with Arabic numbers. Both the number and the explanations to the table are written with Times New Roman, 10 points, italic, for the number and bold for the explanation to the table, centered in the space of the table and above it. The table entries will be Times New Roman, 10 points, bold, and the data in the table will be Times New Roman, 10 points. The thin lines of the table will be ½ points (0,02 cm), and the thick ones will be ¾ … 1 points (0,03 … 0,04 cm).

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4. PRESENTATION OF THE MATHEMATICAL EQUATIONS

· **(**11 points)

The mathematical equations will be with times New Roman, 11 points, center of the page and numbered on the right with Arabic numbers between round brackets.

· (8 points)<Blank line 8 point high>

$$X^2 + Y^2 = Z^2 \tag{1}$$

· (8 points) < Blank line 8 point high>

An 8 point high blank line is left between the last line before the relation and the relation and between the latter and the next first line.

The last page will be at least 2/3 full.

The pages of the paper will only be numbered by a pencil outside the printing space.

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[1]. **Marian I.,** *Mecanizarea în minerit,* Editura Tehnică, Bucureşti, 1969.

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