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CONTENTS

CHROMIUM MINING INDUSTRY, THE RESERVES POTENTIAL AND THE FUTURE

GOSKOLLI EDMOND*
RRESHKA MURAT**

Abstarct: *Extraction of chromium ore in Albania began in the early in the years of after Second World War. Greater development of the extractive industry chrome was reached in the years of after 1960, manly in 80'-90' years. In Albania there are two Chrome-bearing Ultrabasic Massifs, Eastern Belt, which includes 6 massifs and Western belt which include 11. In 50 years, began extraction the largest deposit of chromium in Albania, Bulqiza. Later opened and many other mines in the Batra, Ternove, Shkalla, Kalimash, Kam Tropoje etc. In total, since 1948 until 2010, have been extracted more 24 million tons of high grade chrome ore. Mainly the principal mining mode of chrome mining was and is underground. Greater production of 1.3 million tons/year of chromium was achieved in 1986. Significant amounts of chrome have been mainly exported to Sweden, China, and Russia. Presently are operating about 200 chrome private companies. Average production for the years after 1990 is 200 000 - 300 000 tons with an average content of 38-42 % Cr2O3. A lot of foreigner and domestic companies are making a lot of geological workings with the aim to find some other chromium deposits or to increase the quantity of ore reserve.*

Keywords: *Mining activity, reserve, current situation, future, concentrate, ferrochromium*

1 INTRODUCTION

For foreign investors, who want to know information of mineral resources of Albania, the information they can access are mineral resources database of AGS (Albanian Geological

∗ *Prof. Assoc. Dr. National Agency of Natural Resources - AKBN Mining Directory. Director-Tirana, Albania E-mail e.goskolli@akbn.gov.al; ++355 67 20 51 825*
∗∗ *Dip. Eng. - Geology. Expert - National Agency of Natural Resources - AKBN Mining Directory. Mines Supervising Sector Tirana-Albania, E-mail; muratrreshka@yahoo.com; Mobile: 00 355 67 20 51 831*

Survey) and exploration report of ore occurrences prepared as well by AGS during 1970's and 1980's years. In these document ore deposits were described using Albanian reserves classification system which is different from the worldwide classifications, particularly those of western. (1, 2)

2. SITUATION OF CHROMIUM MINING IN THE WORLD

Chromium mining of the world is dominated by the countries with large chromium mines of annual ore production of more than 1 million tons, such as South Africa and Kazakhstan. In Albania, the largest producer of chromium ore is Albanian Chrome, operating in Bulqiza mine, with annual production of 80,000 tons.

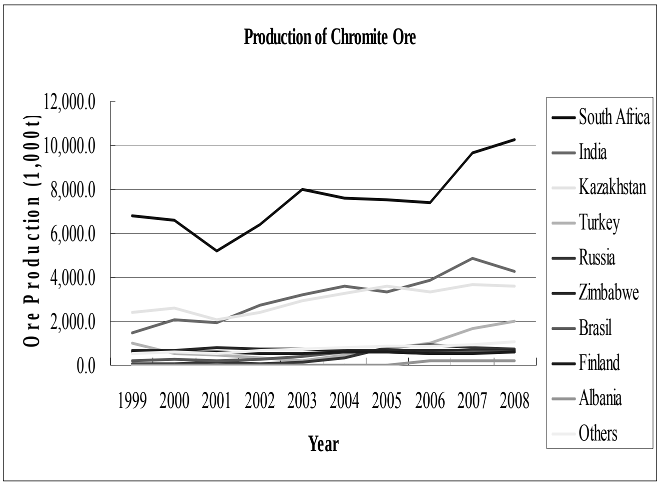


Figure 1. Annual production of chromium by major chromium producing country

The world production of chromium is dominated by four countries (South Africa, India, Kazakhstan,

Turkey) accounting for 84% of the world production. (3)

Chromium ore must meet certain specification to be sold in the world market. There are three classes of ore grade which do not solely depend on chromium contents, but rather its relationship with other associated elements such as Al_2O_3, SiO_2, FeO etc. Table 1 shows general classes of chromium ore.

Table 1 Classification of chromium ore

Metallurgical grade	Cr_2O_3: greater than 48%
	Cr/Fe ratio: greater than 3:1
	Sulfur , Phosphor: less than 0.01%
Chemical grade	Cr_2O_3: greater than 45%
	Cr,Fe ratio: less than 2:1
	SiO_2, Al_2O_3, FeO, MgO, CaO Lesser amounts preferable
Refractory grade	Cr_2O_3: greater than 30%
	Cr_2O_3 + Al_2O_3: greater than 60%
	SiO_2: less than 5%

3. PRESENT SITUATION OF MINING IN ALBANIA

Recent (1995-2011) mining activities, among 201 mining permissions are given for exploitation of the chromium ore in total from 1995 to 2008, 90 (56%) of them belong to Bulqize region and 32 (20%) of them belong to Tropoja region.

From this total number of mining permissions in a periods of 1995-2008, 101 were issued in 1995-2004 period and 61 were issued in 2005-2008 period. After the minimum production of chromium ore during difficult period with low price and low demands of 1999 to 2004, the production of chromium ore began to increase. The production of chromium reached to 200,000 tons/year in 2006 and the figure has been slightly increasing until now (367 327 tons).

The exploitation licenses have been issued for chromium deposits of all ultramafic rock massifs, such as Shebenik-Pogradec massif, Bulqiza massif, except Kukes ultramafic massif and a part of Tropoja massif, where Kalimash and Vlahna mines are located. In April 2010, the concession of Kalimash and Vlahna mines were awarded to a joint venture of Turkish and Chinese companies (Fig. 2)

A distribution of mining concession of chromium classified by amount of production in 2010 is shown in Figure 3 and a histogram of Figure 4shows classification of mine concession by production. The areas of production of chromium in Albania are concentrated in three regions, Tropoja-Kukes, Bulqiza, Shebenik-Pogradec. Among 201 mining concessions registered as of March 2011, 95% of them are conducting small scale operation with annual production of chromium ore less than 5,000 tons and only 4 mining concession have annual production of more than 10,000t. For number of employees, more than 90% of the mining concessions are small scale with less than 20 employees (Figure 4). Further, among these 52 mining concession claim no production in 2009-2010 .

Annual chromium ore productions in 2009 of each areas are shown in Table 1. Chromium ore of more than 10,000tons was produced from 1 concession in Tropoja-Kukes area and 3 concessions in Bulqiza area and most of the rest of concessions are small scale with annual production of less than 5,000 tons.

4 MAIN CHROMIUM MINE

The main chromium mines of Albania are Bulqiza mine and Batra mine in Bulqiza massif and Kalimash and Vlahna mine in Tropoja-Kukes massif.

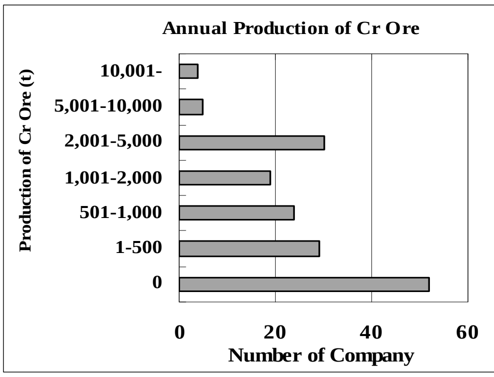
(1) Bulqiza mine

The Bulqize and Batra ore deposits are a continuous ore deposit and they are the biggest chrome deposit in Albania. The ore deposits extend north-west to south-east direction of 5,000 m long and 500-1,200 m in dipping direction with body thickness of 0.50 to 5-10 m (Figure 5). The mineralization occurs in hartzburgite-dunite sequence and the ore body is forming a part of the Bulqiza-Batra anticlinal structure. The ore body has a complicated structure cut by numerous faults and deformed by tectonic movement.

Surface chromium exploitation in the Bulqiza mine was started in 1948 In a period of 1948 to 2005, around 13 million tons of chromium ore was produced from Bulqiza mine. Since the ore resource of upper than Level 16 (-20m) was mostly mined out, the geological resource situation for all categories over level 16 is 60, 000 tons with 46.75 Cr_2O_3 %. While the ore resource of under the level 16 is 2,126,800 tons with 44.91Cr_2O_3 %. The lowest level reached by drillings is -440m and thickness of discovered ore is 3.7 m with 39.99%Cr_2O_3.

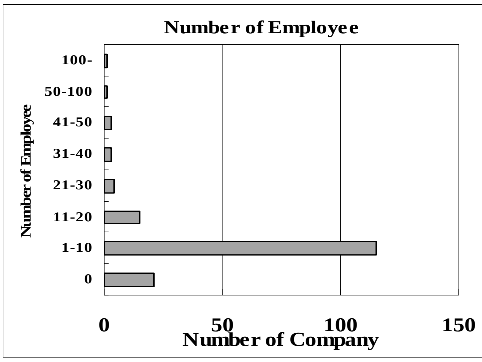
During 2001 to 2007, DARFO was able to continue mining but no progress and no improvement were attained for mining activities. Then in 2007, it was sold to Albanian Chrome (ACR).

In the Bulqiza mine ARC are currently working at the level 16, approximately 800 m below the main gallery (Figure 5).



Production	Number of company	(%)
10,001-	4	2.5
5,001-10,000	5	3.1
2,001-5,000	30	18.4
1,001-2,000	19	11.7
501-1,000	24	14.7
1-500	29	17.8
0	52	31.9
Total	163	100

Figure 2 Classification of chromite concession by annual production of 2010 (4)



Number of employee	Number of company	(%)
100-	1	1
50-100	1	1
41-50	3	2
31-40	3	2
21-30	4	2
11-20	15	9
1-10	115	71
0	21	13
Total	163	100

Figure 3 Classification of chromite mine by number of employee (4)

(2) Kalimash mine

Kalimash mine, located in Kukes ultramafic massif, consists of Kalimash1, Kalimash2, Kalimash3 and Perroi i Batres mines and mining operation of it was started in 1978 (Figure 6). During the period of 1978 to 1997, chromium ore of 1.65 million tons was produced from these four mines before they were closed in 2000.

An international tender for mining concession was

held including both of Kalimash mine of Kukes massif and Vlahna mine of Tropoja massif and the mining concession was give to a joint venture of Kurum Energy, Resources and Metallurgy (Turkey) and Sichuan Jiannanchun Group (China) in April 2010. According to the contract between the government and the Join Venture, 210,000 tons of chromium ore must be produced and concentration plant in Kalimash and Golaj mush be completed within 2 years. Further, smelting plan must be completed to produce 60,000 tons of ferrochrome within three years.

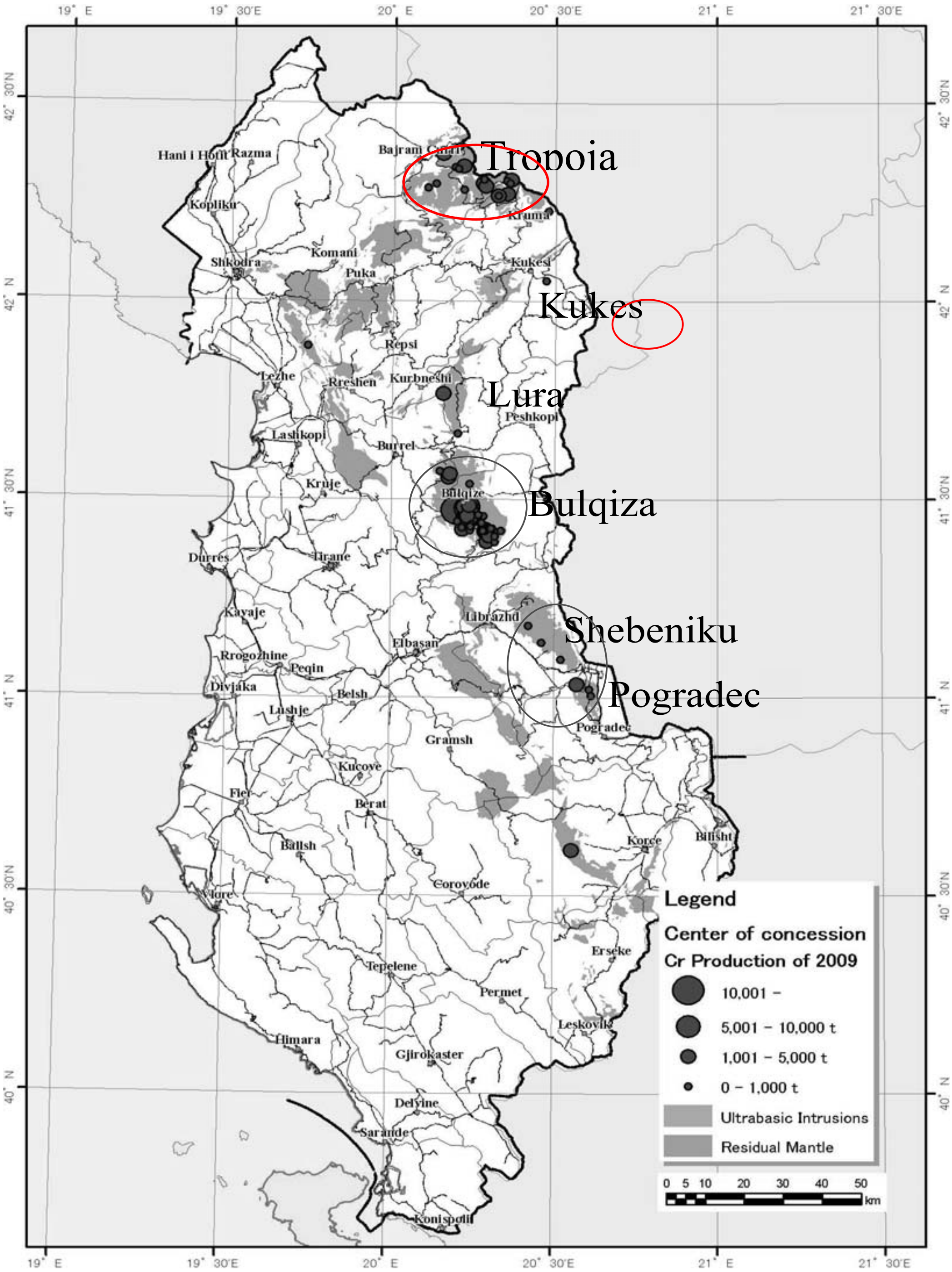


Figure 4 Distribution of mining permit and annual production of chromium ore

Table 1 Classification of chromium mine by production

Tropoja-Kukes

Production of ore /company (t)	Number of company	Total production in 2009 (t)	Total production in 2008(t)	Geological reserves (t)
0	22	0	992	59,907
1-5,000	24	33,394	13,672	330,978
5001-10,000	1	5,335	4,311	262,621
10,001-	1	11,170	4,205	180,000
Total	48	49,899	23,180	833,506

Bulqiza

Production of ore /company (t)	Number of company	Total production in 2009 (t)	Total production in 2008(t)	Geological reserves (t)
0	23	0	8,400	167,243
1-5,000	68	110,643	76,783	1,772,025
5001-10,000	3	17,081	19,670	4,979,063
10,001-	3	105,637	68,191	0
Total	97	233,361	173,044	6,918,331

Shebeniku-Pogradec

Production of ore /company (t)	Number of company	Total production in 2009 (t)	Total production in 2008(t)	Geological reserves (t)
0	5	0	4,042	7,800
1-5,000	9	7,133	10,873	31,263
5001-10,000	1	8,030	0	0
10,001-	0	0	0	0
Total	15	15,163	14,915	39,063

Grand Total	160	298,423	211,139	7,790,900

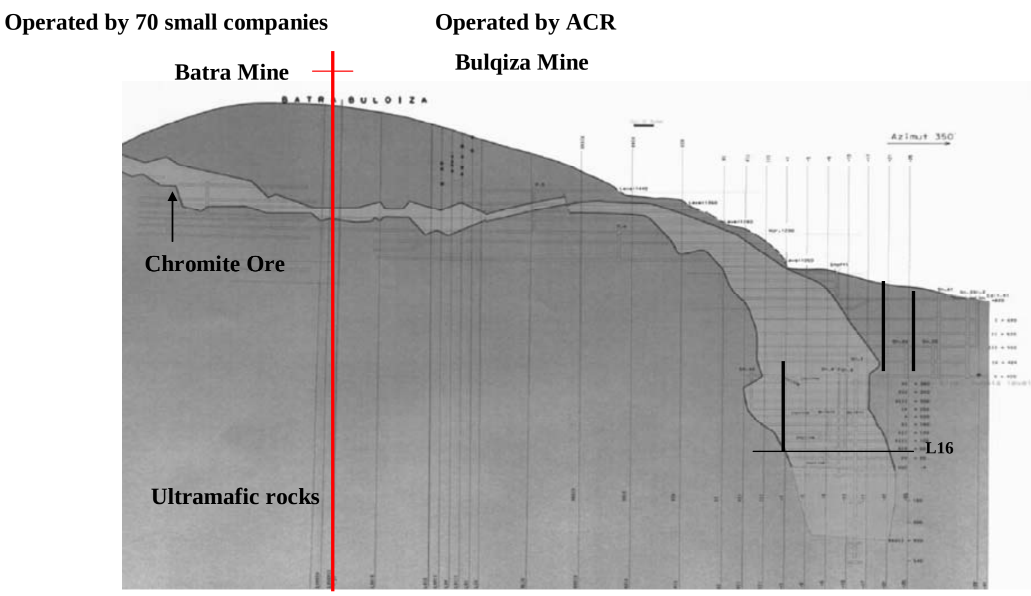


Figure 5 Bulqiza and Batra mines

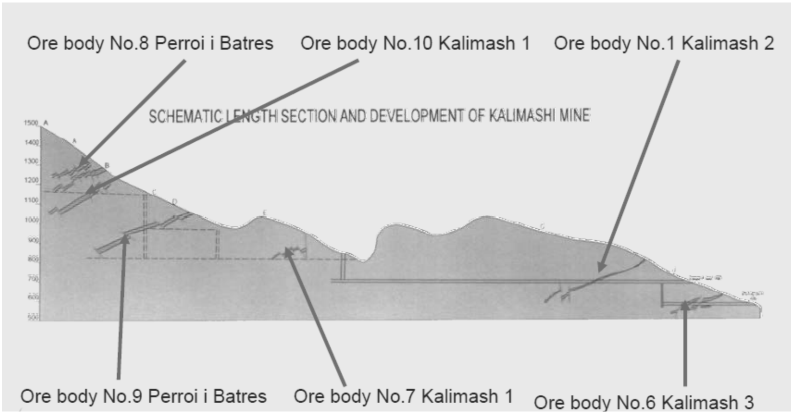


Figure 6 Kalimash mine (After AKBN)

5 EXPLORATION ACTIVITIES

The main exploration activities are conducted by Canadian and Australian junior companies, respectively in Bulqiza massif and Tropoja massif. Other than these, a small scale exploration is conducted in operating mines and at the vicinity of known ore deposits by Albanian companies.

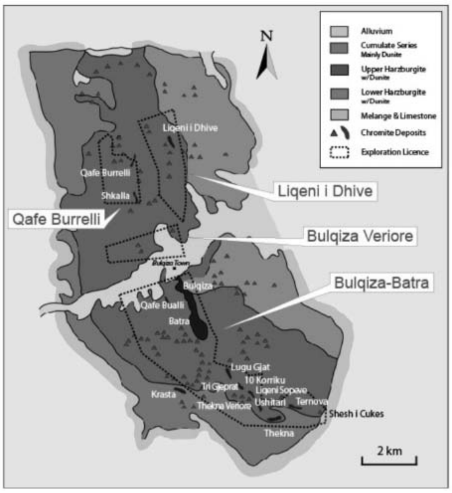
a. Bulqiza massif

Among the chromium exploration projects in Albania a junior company, Empire Mining (Canada), is currently most vigorously working in the Bulqiza area. The Empire Mining obtained prospecting license in the area of 134km2 covering Bulqiza ultramafic massif including Bulqiza mine and Batra mine areas in May 2008 (Empire Mining and EC Terra). Further, within the prospecting licensed area, the Empire Mining obtained four areas of exploration license in January 2009, one of which is covering 35km2 including Bulqiza and Batra mines areas (Figure 4.2.19). Based on their own geological ideas of considering a importance of thrust tectonics for formation of geological structure of the Bulqiza area, they made drilling plan to start drilling. The drilling was started in April, 2010.

b. Tropoja-Kukes massif

Jab Resources (Australia) has three exploration license areas in Tropoja-Kukes massif. They are Kalimash (chromium) over the area surrounding Kalimash mine, Bregu i Bjbest (chromium and Platinum) and Zogai (chromium) (Figure 8).

Kalimash Exploration license area is located surrounding Kalimash mine. In the area Jab Resources has conducted surface geological survey, compilation of existing information, trench survey and reverse circulation drilling of 57 holes (Total 3 834 m) Mineral processing test showed that, using samples of head grade at $10Cr_2O_3\%$, it was possible to produce concentrates of 40 $Cr_2O_3\%$ at recovery of 77%.



F**igure 7** Exploration area of Empire Mining
(After Empire Mining and EC Terra)

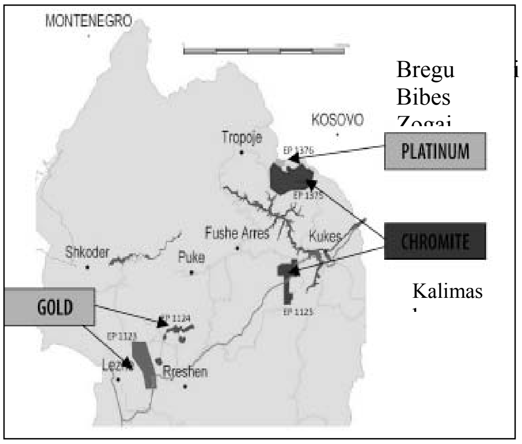


Figure 8 Exploration areas of Jab Resources
(After Jab Resources)

Bregu i Bibes area is located close to Kosovo border in Tropoja massif. In the area, PGE (Platinum Group mineralization has been known by the survey of the state owned exploration company in the past. Boshnjaku and Kulici (2002), from the Geological Research Institute (GRI) of Albania, describe PGE mineralized bodies of chromium orthopyroxenite with strike lengths up to 200m, down dip extents of 20 to 40m and thicknesses from 1 to 10m with PGE grade between 1 and 7ppm with sporadic assays up to 27ppm Pt. Jab Resources conducted surface geological survey, sampling of mineralized zone, compiling of existing data, preliminary mineralogical analysis of PGE bearing chromium mineralization. (5, 6)

Zogaj area is located in the area immediate south of Bregu i Bjbest. In the area there are Zogai mine (mineral resources: 1.238 million tons, operated until 2000) and other relatively large mineral deposits with mineral resources of around 100,000 tons.

6. PROBLEMS EXISTING IN CHROMIUM MINING

Situation of mining operation

Chromium mines of Albania are operated by a few of relatively large scale companies and many small companies. Albanian Chrome, operating in Bulqiza mine, is the only company with number of employees more than 100 and most of the rest of companies engaged in chromium mining are small with number of employee around 10. For the annual production of chromium ore, only 4 companies exceed 10,000 tons and most of the companies have annual production of few thousands tons. Particular in Bulqiza mine, mining operation of many small companies are conducted close to the workings of Albanian Chrome.

The chromium mining operation of small companies has many problems related to management of concession, mining operations, safety and others.

Establishing integrated production system

Since chromium ore is heavy with high specific gravity, it is disadvantageous to exporting chromium ore because of high transportation cost. Further, developed countries, such as Japan and Germany, only import ferrochrome after smelting chromium ore in chromium ore producing countries (South Africa, Kazakhstan and other countries) with low cost of electricity and labor.

Assurance of chromium reserves for future

For sustainable development of chromium mining of Albania, it is necessary to assure minable reserves of chromium ore for future. According to AGS information, chromium resources of 32.8 million tons（B+C1+C2）is repotted to exist in Albania. A total of chromium resource declared by each of concession holder in 2009 is 7.79 million tons and this amount of resource will be depleted in 25 years if production of chromium ore continues at 300,000 tons/year, the same production rate as present.

7. MATERIAL FLOW

As the production of chromium ore drastically increased in 1970's and 1980's, a integrated system of chromium mining industry, continuous process of mining-dressing-smelting, in Albania was considered and concentration plants were established in Bulqiza, Batra, Kalimash and smelting plants were established in Burrel and Elbasan.

In the past, concentration plant was operational in Kalimash, Bulqiza and Batra, and low grade chromium ore of Bulqiza and Batra mines were processed to produce concentrates and then ferrochrome was produced from in smelting plants of Elbasan and Burrel. But mow currently operating concentration plants are Bulqiza and Batra, and Elbasan is the only operational smelting plant (Figure 9). The low grade chromium ore of Tropoja-Kukes area is sent to concentration plant of Kosovo.

8. STRATEGY FOR DEVELOPMENT OF CHROMIUM MINING

The purpose of the strategy is to make best use of chromium resources existed in Albania for the benefit of the country and people living there. For this, mining operation by coordination of few big companies and many small companies by playing each role is necessary for increasing production and continuous operation..

1) Coordination of large company and small company

Chromium mining in Albania is conducted by two clearly different types of company. The one is large company by foreign investment, such as Albanian Chrome operating in the Bulqiza mine and JV of Turkey and China companies operating in the Kalimash and Vlahna mines, and the others are many medium to small scale Albanian companies operating in many places.

For make best use of these chromium resources of small mines and remaining ores in big mines, exploitation work can only be conducted by small companies. Since the mining activities of small companies are economically, socially, strongly connected to local community, these mining activities can not be disregarder for maintaining stability of local community.

2) Establishing association of small companies

Considering future exploitation activities of small companies, it is necessary to establish associations with cooperative function. Chromium ore is produced mainly from three separate areas of Tropoja-Kukes, Bulqiza and Shebeniku-Pogradec. Considering these areas, national level association should be established for sharing equipments and facilities, countering fluctuation of chromium ore price, assisting finances for exploration program and upgrading mining facilities

a. Support mine development by financing

Development of mines is usually accompanied with financial risks because market price of chromium often fluctuates. The association provides the miners with long or short term loans, guarantee against risks and other benefits to promote the development.

b. Purchasing ore and processing in concentration plant

The association purchase chromium ore from small companies and low grade chromium ore is sent to concentration plant before sending it to smelting plant. The price of purchasing chromium ore is set between high price and low price of chromium ore market. If the price becomes higher than the association price, the balance will be saved in the association for preparing the time of low price.

c. Work actively for operation improvement, new technology and countermeasure for environmental problems.

In addition to above, in the case of chromium mining in Albania, it is necessary for the association to have functions of sharing equipments for mining operation and concentration plant. Mine rescue team of regional level should be organized as well. If the association of small companies will be established, they can continue mining operation avoiding closing mine (Table 2). By this way chromium resources of Albania is efficiently utilized and stability of local society is maintained.

3) Establishing integrated system of chromium production

For bringing benefit to Albania by optimal use of chromium resources, it is necessary to establish an integrated system of chromium mining, including processes of mining-dressing-smelting. Chromium ore production of Albania in 2009 is 298,000 tons, but production of ferrochrome is as low as 6,000 tons.

At preset, operational smelting plant of chromium in Albania is only Elbasan Smelting Plan operated by Albanian Chrome and only chromium ore exploited by Albanian Chrome in Bulqiza mine is used for smelting.

Table 2 Target production of chromium ore

Tropoja-Kukes	2009 (t)	Future (t/year)		
	production of ore	Production of ore	Concentrates	Ferro chrome
Association of small companies	49,899	50,000	-	-
Kalimash Mine	0	210,000	90,000	30,000
Sub Total	49,899	260,000	90,000	30,000
Bulqiza	**2009 (t)**	**Future (t/year)**		
	production of ore	Production of ore	Concentrates	Ferro chrome
Association of Small companies	154,924	150,000	-	-
Albanian Chrome	78,437	250,000	-	24,000
Sub Total	233,361	400,000		24,000
Sheb-Prog	**2009 (t)**	**Future (t/year)**		
	production of ore	Production of ore	Concentrates	Ferro chrome
Association of small company	15,163	15,000	-	-
Sub Total	15,163	15,000		
Grand Total	**298,423**	**675,000**		**54,000**

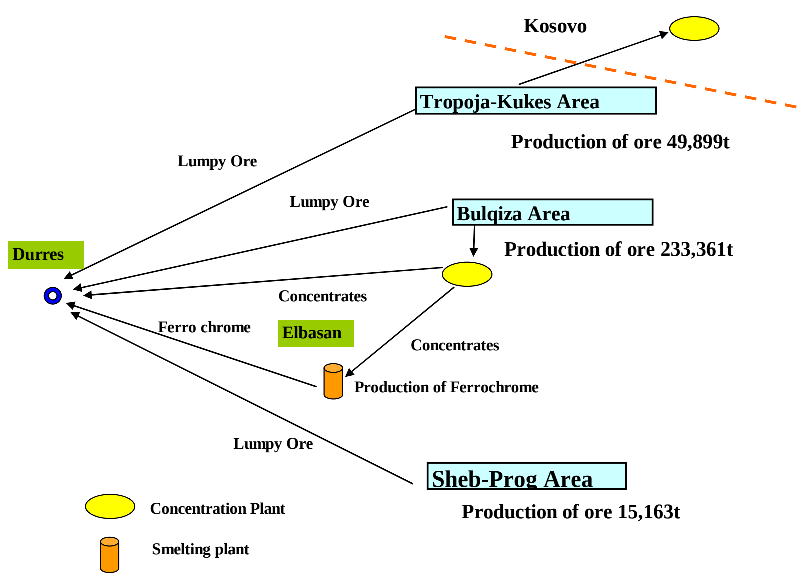


Figure 10. Material flow of present

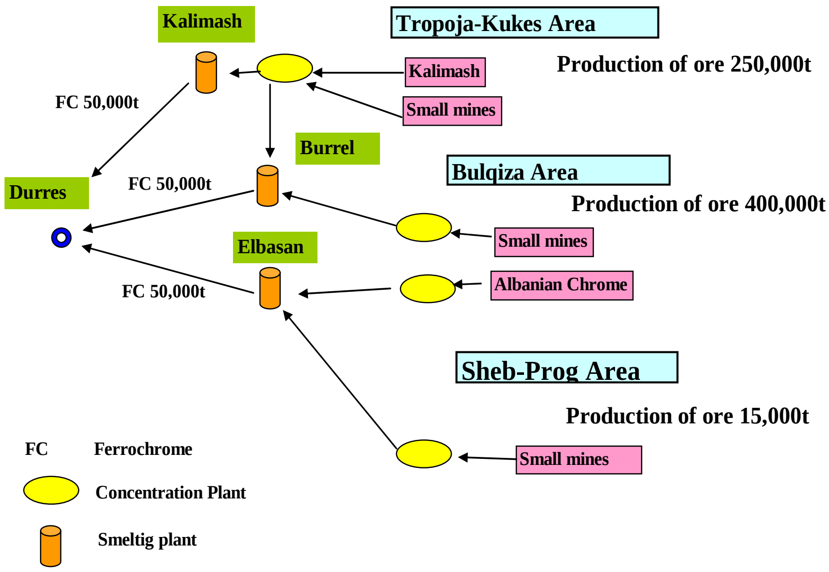


Figure 10 Material flow of future

Concentration plant is only found in Bulqiza-Batra area and, in the northern part of Tropoja –Kukes area, low grade chromium ore, foprvthe moment is sent to two concentration plants of Kosovo (Fig 9). For establishing the integrated system of production (mining-dressing-smelting) in Albania, additional concentration plants and smelting plants are necessary.

The material flow for establishing integrated production system is shown in (Figure 10). Although, in the concession contract between the government and Albanian Chrome, operation of Elbasan smelting plant, operations of Burrel smelting plant and Klos concentration plant after rehabilitation of them are included in addition to operation of Bulqiza mine, so far nothing have done for Burrel and Klos. In the contract of Kalimash and Vlahna mines between the government and JV of Turkish and Chinese companies, building two concentration plants, are included in addition to operation of two mines. The target ferrochrome production of smelting plant of Kalimash-Vlahna mines is 30,000 tons/year and Elbasan smelting plant is setting target of ferrochrome production of 24,000 tons/year. But both of the smelting plants are only considering using chromium ore of their own to attain the targets of ferrochrome production. It is necessary for the government to establish structure in which small company can send ore and concentrate to smelting plant. In future, three smelting plants of will be operational. If the smelting plan of Kalimash, Burrel and Elbasan will accept chromium ore of small companies in addition to their own, the capacity of annual production of ferrochrome at each smelting plant should be around 50,000 tons/year. Among the small companies, there are some plans of establishing small smelting plant by corporation of few companies, but there is no confirmation of this small smelting plant being feasible. The structures charged by mining law with some experts should conduct study to find an optimal system for smelting the chromium ores and concentrates of big companies and small companies at once. More then two concentration plants will be established in Tropoja –Kalimash area because the chromium ore is generally relatively with low grade (around 20Cr2O3 %,) which means that most of the chromium ore must be sent to concentration plant before smelting process.

By completing the material flow described above, it is necessary for Albania to establish chromium mining industry in lime with world market and, then, ferrochrome can be exported to European Countries and China.

4) Assuring mineral resources

According to AGS information, mineral resource of chromium ore is 32.8 million tons (B+C1+C2) and, further, a total of chromium resource declared by each of concession holder in 2009 is 7.79 million tons. For the sustainable exploitation of chromium deposits, it is, consequently, necessary to grade up these values to mineral reserves by adding factors of economy and feasibility by conducting surveys. It is, also, necessary to find new chromium deposits.

Up grading resource to reserves

Most of the amounts of resources claimed by exploitation license holders are B+C1+C2 of Albanian classification, corresponding to indicated or inferred resources of international classification.

Exploration work for finding new deposits

Since chromium deposit do not have geochemical and geophysical halos in their surrounding zone, it is difficult to find concealed deposits. If the fragments of chromium ore are found, tracing of fragments continues toward upper stream to find exposure of the chromium mineralization. In Albania, vigorous exploration work was conducted

Two areas are recommended as prospective area for future exploration program,

Zogaj mine and its surroundings in Tropoja-Kukes massif with mineral resources of 1.238 million tons

Area of transitional zone to tectonite harzburgite between Krasta mine and Ternova mine.

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Scientific Reviewers:
Prof.Ph.D.Eng. KRAUSZ Sanda

OPPORTUNITIES AND PROSPECTS OF COAL EXPLOITATION IN THE AREA OF SIBOVC

**AHMET BYTYÇI*
EDMOND GOSKOLLI****

Abstract: *The great reserves and also convenient conditions of the use of coal, present very strong arguments for the justification of private capital investment in the energy sector of the Republic of Kosovo. The trend of the increase of living standards of population, development of industry and also the demands in South-Eastern Europe market submit the need to build new generative capacities of power. Hereupon, the purpose of this scientific paper is to present the opportunities of supply with coal of generative capacities to 2000 MW, in the perspective of potential areas of Sibovc.*

1. INTRODUCTION

Lignite is of outstanding importance in Kosovo. It contributes 97% of the total electricity generation, with just 3% being based on hydropower. At 14,700 Mt, Kosovo possesses the world's fifth-largest proven reserves of lignite.

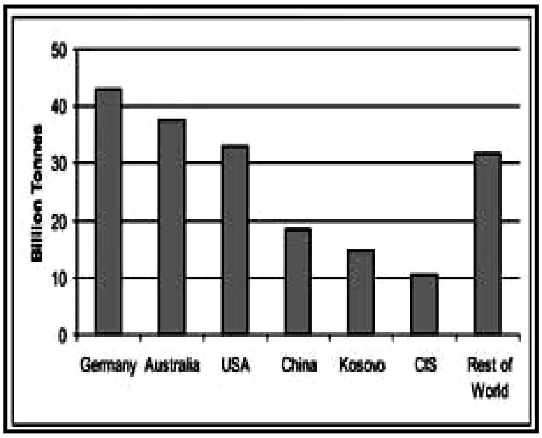

Figure.1 States with large lignite reserves

The lignite is distributed across the Kosovo (Figure.2), Dukagjin and Drenica Basins, although mining has so far been restricted to the Kosovo Basin. The first systematic records of lignite exploitation date from 1922, when small-scale, shallow underground room-and-pillar mining commenced in the Kosovo Basin.

∗ *prof. Ph.D.Eng. at the KEK Prishtina Kosovo, ahmet_ing@hotmail.com*
∗∗ *prof. Ph.D.Eng. at the AKBN Tirana Albania, e.goskolli@akbn.gov.al*

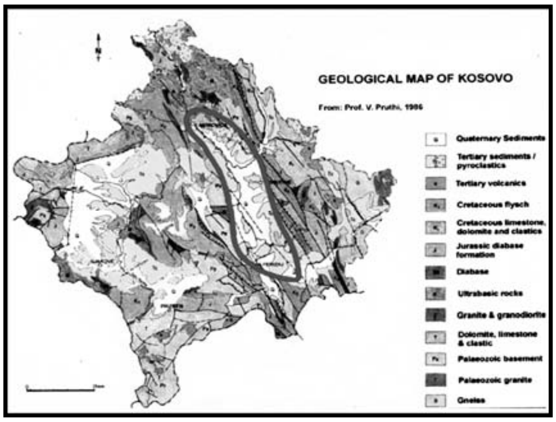

Figure.2 Geological map of Kosovo

Large-scale minning of lignite began with the first production from the Mirash (1958) and Bardh (1969) open-pit mines, using bucket wheel excavators. Cumulative exploitation from the commencement of mining in 1922 up to the end of 2010 has amounted to 310 Mt. Geologically, Kosovo's lignite mines exploit one of the most favorable lignite deposits in Europe. The average stripping ratio is 1.7m^3 of waste to one tonne of coal and the total estimated economically exploitable resource represents one of the richest in Europe, which would allow ambitious power generation and expansion schemes in forthcoming decades. The lignite is of high quality for the generation of electricity and compares well with the lignite resources of neighbouring countries on a range of parameters. Kosovo's lignite varies in net calorific value (NCV) from 6.28-9.21 MJ/kg, averaging 7.8 MJ/kg. The deposits (Pliocene in age) can be up to 100 m thick (coal layer), but average 40 m, and possess an average strip ratio of 1.7:1. This combination has meant that the cost of lignite-fuelled electricity in Kosovo is the lowest in the region. Kosovo's cost of 0.62 €/GJ compares favourably with 0.88 €/GJ in Bulgaria and 1.34 €/GJ in Serbia. Further development of lignite mining in the medium term will continue with the exploitation of the Sibovc mining field in the northern part of the Kosovo Basin, and provides a great opportunity for private investors (Figure.3).

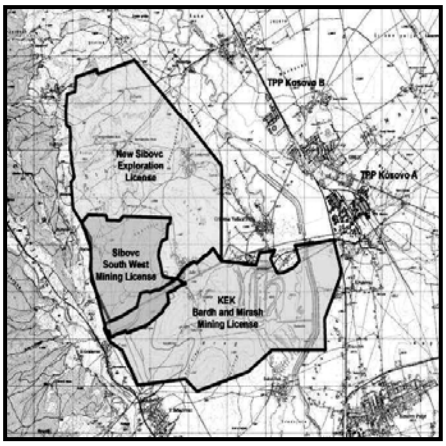

Figure 3.Schematic presentation Location of Sibovc area

2. COAL TYPE, RESERVES AND QUALITY

Sibovc field lies in the central part of Kosovo basin is characterized by an average thickness of 70 m, which is stored under cover tertiary clay, whose thickness varies from several meters up to 125m and lies on clay footwall. In the structure of coverage clay formations distinguished two types: yellow and gray, and green clay layers reported in the coal layer footwall.The basement of the Kosovo basin in the Sibovc area and the exposed surrounding areas to the West are built up by Palaeozoic to Mezozoic crystillane rocks The basin fill consists of Upper Cretaceous strata which are unconformably overlain by Tertiary clays in which lignite is interbedded (Figure 4).

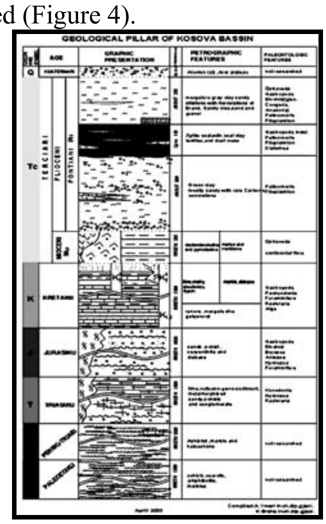

Figure 4. Stratigraphic Standard Profile of the Kosovo Basin

The Sibovc filed was site of luxuriant vegetation growth that finally became overwhelmed by sedimentation and led to the formation of substantial stratiform lignite deposits of up to 90 [m] seam thickness (Figure5).

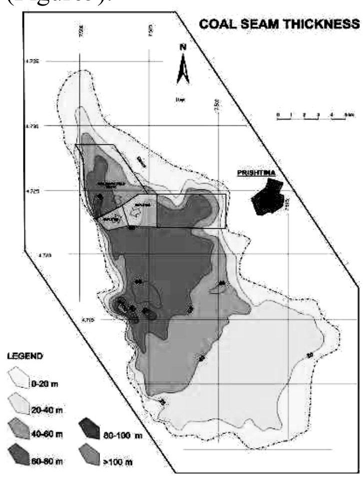

Figure 5 Coal Seam Thickness Kosovo Basin

Towards the West the lignite deposition is tectonically bounded by the development of a series of predominantly NNW-SSE striking faults.The geological work for the Main Mining. Plan for New Sibovc Mine reveals that the tectonic movements were already active during the lignite formation and controlled the deposition of organic material and anorganic clay. These findings differ to the previous geological model which assumes purely post depositional tectonic movements. Thus, in that model depth differences at top or base of the Lignite formation and thickness variations were attempted to explain by extensive faulting. Geological model the Sibovc field is built from information research drilling average 1100 250 X 250 array of working with SURPAC (Figure 6).

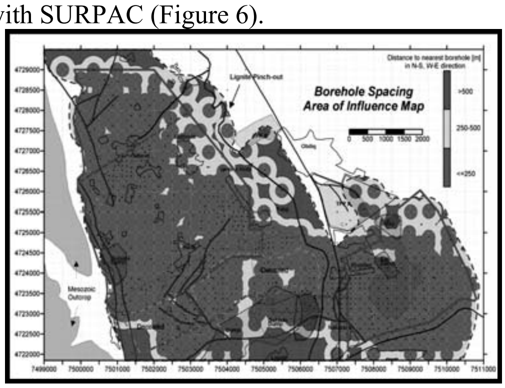

Figure 6 Sibovc Mining Field – Borehole Spacing

Where stocks of coal reaches 990 million tons, whereas 830 million tons are exploitable (Figure 7). Therefore, such reserves and favorable geological conditions in which deposition is Sibovc field are good basis for a long-term use needs of electricity generation energy thinking we increase capacity. But even continuing geological research which should ensure a mirror as the real me during mining activities. For this reason, drilling should be carried out continuously in front of the line excavation, while all nuclei should be described (degree cm) to mapping and sampling.

Parameters which are usually subject to review in KEK. are the quality of coal and geotechnical characteristics, monitoring of groundwater level and in case of any sign of fire coal.

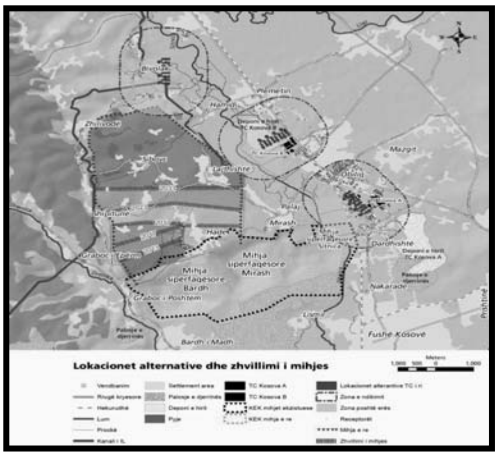

Figure 7 Sibovc Mining Field

- For the value energy lignite reserves are categorized Sibovc field:
  - 29% (> 8.4 MJ / kg)
  - 43% (from 7.7 to 8.4 MJ / kg)
  - 25% (5.8 - 7.7 MJ / kg)
- Lignite moisture ranges (35-50)% and the ash concentration ranges (12-21)%, while sulfur ranges (0.35-1)% (Figure 8).

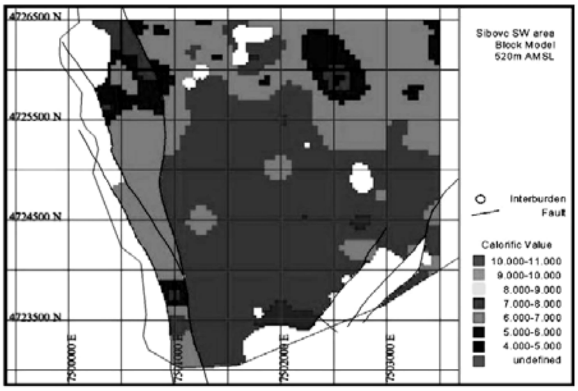

Figure 8 The quality of coal in horizontal sections

Geomechanical conditions that prevail in this area are presented in tab.

Tab.1 Data geomechanical

			2013		2014	
			OB 10^6 [m^3]	Coal 10^6 [mt]	OB 10^6 [m^3]	Coal 10^6 [mt]
Overburden	E_{sh} 10/70		0.19		0.14	
	1	**SR_s1300**	3.47		3.08	
	2	**$S_{ch}R_s$ 650**	4.90		4.90	
	3	**$S_{ch}R_s$ 650**	4.01		4.17	
		SR_s 470	0.85		0.89	
Coal	1	**SR_s470**		4.6		4.7
		SR_s1300				
	2	**SR_s400**		4.4		4.3
		SR_s470				
		SR_s470	0.13		0.14	
Total			13.36	9.0	13.18	9.0

Clay formations composed of overburden cover tends to shift from plastic that are strong under the influence of water and caused as a result of reduced stability. Therefore, further steps are intensified in research needed for management of safe operations Engineer of geo terms. For this reason water removal KEK paid a special importance.

3. TECHNOLOGICAL DEVELOPMENT IN THE FIELD OF SIBOVC

Sibovc areas except large coal reserves that it also possesses favorable conditions characterized by the deposition. Lignite is high quality and impact on excavation represents only the removal of the old furl in its southwestern part. Another advantage of this field is the smallest distance transport to the plant. Lignite excavation in Sibovc offers the best opportunity to supply coal to new power plant with large capacity. Total can provide coal for a plant capacity of 2000 - 2500 MW. Advancing the field of southern Sibovc has the best potential of all other scenarios to fill the spaces with measures of mining overburden Mirash. White and supply of existing plants by 2025 is needed only part of the Sibovc field. Jobs can be optimized starting from the slopes of mining exploitation and utilizing existing infrastructure of the entire deposit existing . Exploitation requires the performance of many journeys. Concentration of exploitation activities in the southwestern part of the field will reduce the number of journeys required. Mining Sibovc not advancing towards the north overlooking the village of Hade initially. According to geotechnical assessments, the eastern slope angle in the jet should 6^{o} and general slope angle should be kept working until 8^{0} for the assessment of new geotechnical. In the wasteland are installed three systems ECS equipment (excavator convey spraider), and lignite-2 sisteme. Operate the removal of overburden is focused on level 3 that was started 2007, with this case are removed about 13 milion m^{3}. Full dynamics of exploitation dote achieved after repair of all equipment where until now only two systems are repaired and others dote completed by mid 2012.

For purposes of calculating the measures used digital model based on topographic maps and data izolines existing drilling database was created. Calculating the volume of measures is implemented with software: MicroStation Program, Datamin and Auto CAD Civil 3D 2010 as well as by other specialized programs that are working on the basis of triangulation (Figure 9)

Tab.2 The presentation of mining equipment in SJP

	φ‘ (°)	c‘ kN/m²	γ kN/m³
Gray and yellow clay	16	30	17.5
Coal seam	40	50	12.2
Green Clay	16	30	18.5

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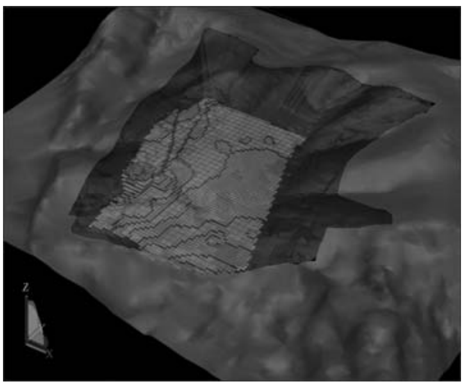

Figure 9 DTM the field in 2024 in the SW Sibovc

In this case we defined the area of the south-western Sibovc should leave 195,075 milion m^3 to issue 123.45 million tons of lignite (Figure 10).

Figure 10 Sibovc field in 2024

The ratio of overburden and lignite revolves around 1:58 m^3 / t, and report [to the field of Sibovc is presented in the map as follows (Figure.11)

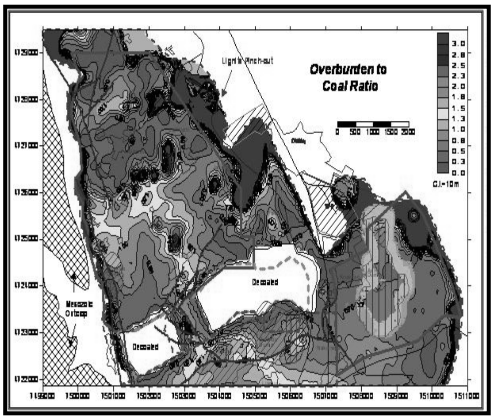

Figure.11 Overburden/Coal ration

Taking into account the specific situation of geotechnical and experience gained so far on potential areas ekskavimin sliding surface mine Mirash-White, are appropriated these technological parameters: height of the stairs in the wasteland h = 25m, whereas in coal h = 15m, slope angle G working wasteland, overburden β < G, Coal β10 ° while the coals β <22 °.

4. EVALUATION OF INFRASTRUCTURE, FACILITIES, HOUSEHOLD FACILITIES AND OTHER FACILITIES

Opening a new field in the basin of Kosovo in all cases will involve the relocation of residents. Right now it is estimated that in the case of field Sibovc would happen displacement greater number of residents at least three villages and nine settlements. Surface Mining Sibovc occupy about 19.76 km^2 land during active stretch of 4.95 km north-south and east west of 4 km. Relocation of villages will change the rural and cost structure. Currently the new law, the Law on Expropriation "is available which requires compensation for land to the next field Sibovc. This area was declared as reserved area for development of mining activity by "Executive Decision No. 2004/28, dated 18 November 2004. " In this case villages Hade, Sibovc, hazel grove, Jani Palaj VOD and the Municipality of Kastriot recognized as locations that constitute the area of special interest for economy. Having entered into force on the date of signing this executive decision, in the villages which represent the specific area of interest should not be developed further construction activities (Figure 12).

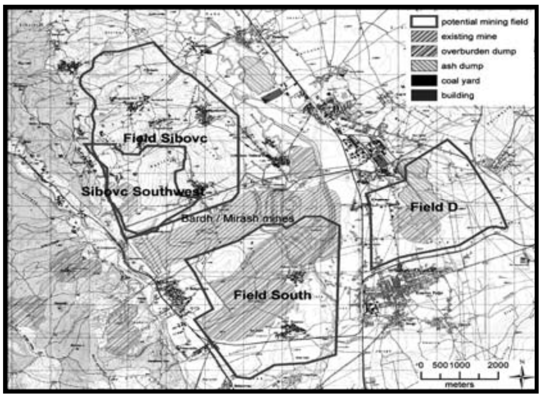

Figure.12 Settlements in the area of Sibovc

Based on criteria set by the Government of Kosovo about property assessment in the villages and the decision to declare the areas in question as an area of particular interest, legal basis on getting Sibovc areas in mining activities and the relocation of villages mentioned.

In this case the property assessment is made by a Government Commission for an inventory of homes, land and agricultural experts at the level of basic state According to a 2004 project submitted by the Commission and to the field of Sibovc appears that the use of coal from this field have moved four locations: Hade, Sibovc, hazel grove and Jani VOD. In such localities residents living in 3960 or an average of 743 houses inhabitant/1house i 5:32.

The structure of the surface land foreseen for expropriation is as follows: -Land of 1158 ha with agriculturefund,610-ha land for pasture ,208-ha land. Thus the total cost of the all expropriation and infrastructure as the main project of exploitation of the field that amounts to 132.637.000 € Sibovc.

5. ANALYSIS OF SOCIO-ECONOMIC FEASIBILITY OF EXPLOITING COAL

For a long time been warning that Sibovc areas will become the area for mining activities. Previous plans have included except such development from south to north. Consequently, the use of land for agriculture continues to dominate in this area, mocking the truth at this zon industrial facilities do not exist apart from the existing mining operation. Based on economic and financial analysis determined the cost of coal production as profit over the period of implementation of this project. Costs for the opening of the mine and annual expenses are for the area through the years are Sibovc.

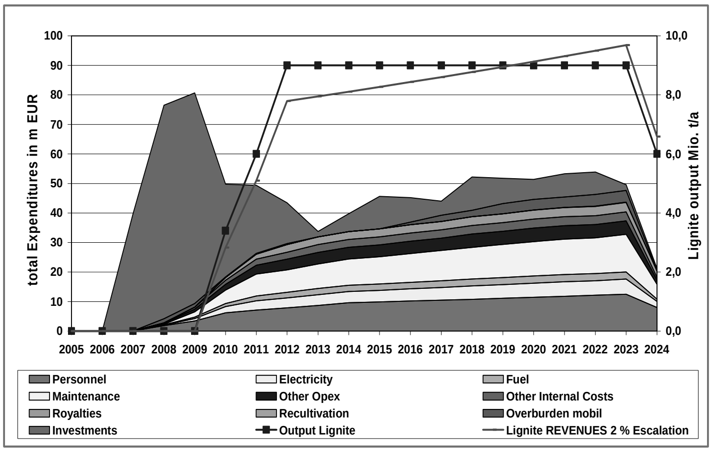

Figure 13. Total expenditures

An important goal of economic research is to determine the correct price of coal production in terms of expenditure. Comparison of costs for electricity production costs indicates that the existing power plants in Kosovo are more favorable than in a new power plant based on coal or a new gas plant. If one considers the power price 0.4 €cents / kWh then the cost for the production of coal is 8.05 € / t.

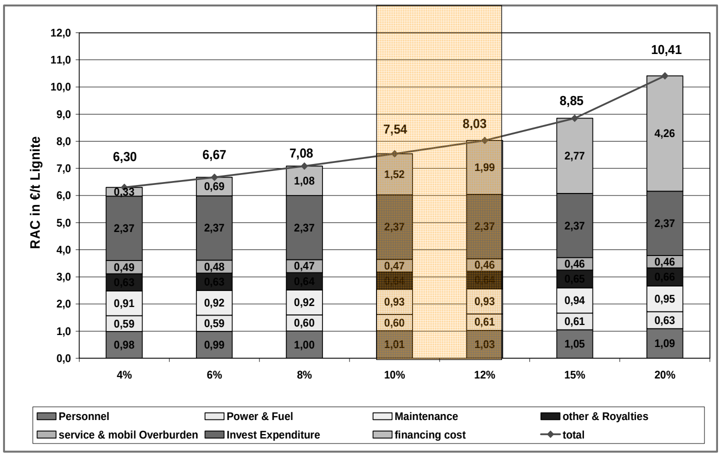

Figure.14 Sensitivity of discount rate

The coal deposit is characterized by favorable report overbord: coal and that it allows in principle the production of coal with extremely favorable price ≤ 7.00 € / t. Based on current studies open up new field of aspect Sibovc: socio-economic and of enviromental is of great importance.

6. CONCLUSION

All these data presented indicate that the scope of Sibovcit as part of the basin of Kosovo possesses a high potential of coal and that can be counted in the group of large areas explatation coal an energy sector development perspective. According to the Kosovo Energy Strategy, mine Sibovc "will provide the entire production necessary to existing units of KEK and the units of TPP" Kosova e Re "until 2030. This mine after only 2030 units will supply the New Kosovo envisioning removal from office of the KEK units by that time. Feasibility study for mine development has determined that the cost of coal available in the new mine is of the lowest in Europe. Mines can supply you with full capacity for the entire duration of her 40 years of age.

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Scientific Reviewers:
Prof.Ph.D.Eng. Dumitru Fodor

GEOLOGICAL AND INDUSTRIAL COAL RESOURCES IN LEADING COAL PRODUCER COUNTRIES

KOVÁCS FERENC*

Abstract: *Nowadays (2008, 2010), the rate of coal in the world's electricity production of $20{\cdot}10^{12}$ kWh/year is a round 40%. It is similarly high in leading coal producer countries: 47% in both the US and Germany. For the future (2020, 2030, 2050), long-term forecasts/plans predict a similarly high rate: 38% in the U an, a round 50% in Germany. with the world average being predicted to be 43% in 2035 with a production of $35{\cdot}10^{12}$ kWh/year. The present (2010) coal production of 6.3 billion t/year may increase to 11 billion t/year by the end of the 21st century with the century average amounting to 8 billion t/year. On the basis of official reports and expert estimations, the forecast data for both geological and, in greater detail, explored industrial coal resources (that can be economically exploited) are analysed. The wide ranges of professional estimations give approximately identical figures: the world's industrial coal resources are 700 -1,000 (1,200) billion tons while estimated geological resources amount to 5,000 – 8,000 (15,000) billion tons. On the basis of the production (demand) data forecast for the 21st century, the average period of supply in industrial resources is 200-300 years in the large coal producer countries (over 1,000 years in Russia) while the world average is 160 years due to China's figure of 40 years attributable to exceptionally intensive production there. The average of the 8 leading coal producer countries (China's 40 year figure included) is approximately identical with this while the average of 7 countries (China excluded) is a round* **400 years.** *On the basis of the estimated geological resources (5,000 – 8,000 billion tons) and subject to further successful explorations, period of supply may even be 500-800 years. With a 10 Mt/year production volume, Hungary has supplies for 330 years, and with an unjustifiably low 4 Mt/year production volume, for 800 years.*

According to publication [1], the rate of coal in electricity production is quite considerable in the current period (2008-2010). In the world's electricity production of $20{\cdot}10^{12}$ kWh/year, the rate of coal is 41%, in the US, it is 47% for a $3.7{\cdot}10^{12}$ kWh/year production while in Germany, it is 43% for a $0.62{\cdot}10^{12}$ kWh/year production volume.

When planning for the future, countries prepare long-term forecasts. According to the global forecast for 2035, coal will be responsible for 43% of the $39{\cdot}10^{12}$ kWh/year production volume. The US forecast for 2050 takes into account a 38% coal rate for the $5.0{\cdot}10^{12}$ kWh/year production. For the period following 2020, the basic German forecast takes into account a 50% coal rate, which may even be higher due to the reduction of the production of nuclear power plants (close-downs) also depending on the amount of imported gas. [2, 3, 4]

* *Prof. Em. Dr. h. c. mult. Dr. Eng. Member of Hungarian Academy of Sciences, Research Group of Geoengineering, Hungarian Academy of Sciences, e-mail: bgtkf@miskolc.hu*

There are different estimations concerning fossil fuel (coal, lignite, mineral oil, natural gas) resources. As regards mineral oil, conventional world supply is estimated to last for 30-40-50 years while the same figure for natural gas is 50-60-70 years. The exploration and exploitation of non-conventional resources (oil shale, oil sand, gas shale, gas hydrate) may significantly expand supply opportunities. In case of coal (hard coal, brown coal, lignite), every estimation indicates significantly larger resources and longer supply periods.

With regard to the above forecasts taking into account a 30-40-50% coal rate in electricity production, this paper gives an overview of what data the different experts and institutions have published about the world's coal resources. In most of the cases, the figures for industrial coal resources (that can be economically produced with the currently available technologies) are given but for some authors, the figures for geological resources and for other expert estimations, the period of supply (how long it will be enough) are presented for the production volume at the time of estimation. As regards the issue of saving/depletion of coal resources, it is investigated what supply volumes can be taken into account for coal, and for how many generations, they will be sufficient.

As early as at the beginning of the 20th century, this was written [5]: 'There is hardly any other issue in natural sciences that scholars would deal with so much as the question of fuels: what will happen if there is no longer any hard coal in the layers of the Earth….. and hard coal **is** running out.' The authors then made the following forecasts:

- The hard coal resources of Great Britain (one hundred million tons) will run out in 435 years…
- Belgium, Prussian Silesia and Russia possess the largest hard coal resources but even these will not be able to satisfy rising demand for longer than 500 years,
- According to Hall, North America may meet current world demand for ten thousand years.

Fifty years later (in 1944), Kálmán Sztrókay [6] wrote the following on the basis of 1929 data: the brown coal resources of the Earth amount to 3,000 billion tons, of which industrial resources are 400 billion tons, and black coal resources give 4,400 billion tons, of which industrial resources amount to 300 billion tons. The figure for the resources of the five continents is 5,662 billion tons in black coal equivalent of 7,000 calories. With the 1929 production volume of 1.25 billion tons, the industrial resources of 700 billion tons (present-day estimates indicate an identical figure for the minimum amount of current industrial resources) meant supply for 570 years (early 2000s).

Now, let us investigate current 'official' and scientific figures and estimations.

According to data from the Hungarian Geological Service [7], the world's industrial black coal resources are 519 billion tons while brown coal resources are 465 billion tons so with a 4.3 billion ton production volume, the 984 billion tons of resources provide supply for 228 years.

According to data from György Vajda [8,9], the world's industrial black coal resources amount to 510 billion tons and brown coal resources are 475 billion tons, altogether 985 billion tons, which ensures supply for 219 years, taking a production volume of 3.6 + 0.9 = 4.5 billion tons/year into account. The coal resources of eight prominent countries (Russia, USA, China, Germany, India, Poland, and South Africa) amount to 817 billion tons. He indicates the world's geological resources to be 5,000 billion tons.

Estimating the world's geological resources to be 4,773 billion tons, the author of publication [10] gives a 136-year period of supply for industrial hard coal resources and a 293-year period of supply for lignite resources.

Investigating the expected prospects of coal production, Klaus Brendow [11] gives the figure 510 billion tce (7,000 calories) for the world's black coal resources and 200 billion tce for brown coal resources, which are 710 billion tce altogether, equivalent to 160- and 460-year periods of supply and an average 196-year period of supply. His figures for geological resources are the following: 6,000 billion tce black coal and 2,700 billion tce brown coal, altogether 8,700 billion tce. Adding up the production forecasts of the different countries, Klaus Brendow expects a 7 billion ton coal production volume for the year 2030 while the World Energy Council (London) gives the coal production forecast of 11 billion tce for the year 2100.

In his study, István Lakatos [12] gives the figure of 1,083 billion tons for industrial coal resources with a 40% rate of black coal.

According to Shashi Kumar's data [13], the world's industrial coal resources (2002) amount to 951 billion tons black coal and 465 billion tons brown coal. He gives the supply data of 204 and 209 years.

The author of publication [2] estimates the world's industrial coal resources to be at least 900 billion tons (Mehr als 900 Mrd Kohlevorräte, 2004), of which the US accounts for 250, Russia for 157, China for 120, India for 80, Australia for 75, Germany for 65, South Africa for 50 and the Ukraine for 30 billion tons, 827 billion tons altogether, other prominent contributors being Brasil, Poland, Indonesia and Colombia.

According to V. S. Kovalenko's data based on former explorations [14], in the world, Russia possesses the second largest coal resources after the US. Russian geological resources amount to 5,335 billion tons, which is 36% of the world's geological resources accoding to him. This indicates the world's geological resources to be a round 15,000 billion tons.

With the above 800-1,000 (1,200) billion ton data of the world's industrial coal resources/reserves (economically exploitable with current technologies) taken from official publications and expert estimates, a supply period of 150-230 years are estimated by experts. Even with the current production volume of 6.3 billion tons and the production volumes of 7 billion tons estimated for 2030 and 11 billion tons for 2100, taking into account an average production volume of 8 billion tons/year for the 21st century, the currently registered industrial resources will safely meet the demands forecast for this century.

With regard to the currently known (estimated) geological resources of 5,000 – 15,000 billion tons and taking into account the expected development of production technologies, further industrial resources of 2,000 – 5,000 billion tons may be forecast for the period after the 21st century, providing supply opportunities for future generations. In view of all this, it is hardly justified to speak broadly about 'ever decreasing fuel resources' – at least with respect to coal types.

In addition to world data and forecasts, here are some data concerning coal in Hungary. [7]

Coal type	**Geological resources M (10^6) tons**	**Industrial resources M (10^6) tons**	**Reserves M (10^6) tons**
Black coal	1,950	200	450
Brown coal	2,170	195	180
Lignite	4,400	2,930	730
Total	8,520	3,325	1,360

Present annual coal production (2010, 2011) is 8.0 – 8.5 M tons of lignite and 1.5 – 2.0 M tons of brown coal. Although emphasizing 'a commitment to coal in principle', the National Energy Strategy [14] takes into account a decreasing future coal rate of 4-5% in electricity production in spite of the present rate of 14%. This decrease cannot be supported with rational arguments and is totally unjustified.

The level of supply of a country or the world with usable raw minerals, namely with coal, with a given production volume, also depends on the number of the population. Next to the data of industrial coal resources, the following table provides the figures of annual production and the number of population, and calculates the per head amount of industrial resources and the expected supply period subject to current production volume.

Country	**Total industrial coal resources M (10^6) tons**	**Production volume M (10^6) tons/year**	**Population M (10^6)**	**Industrial coal resources per head t/person**	**Period of supply years**
Hungary	3,325	10	10	330	330
		4			830
US	250,000	932	310	800	270
Russia	157,000	140	142	1,100	1,120
China	120,000	3,162	1,321	90	40
India	80,000	400	1,210	70	200
Australia	75,000	353	21	3,570	210
Germany	65,000	190	82	790	340
South Africa	50,000	225	44	1,140	220
Ukraine	30,000	80	46	650	375
Total and average for 8 countries	827,000	5,482	3,176	260	150
World	1,000,000	6,300	7,000	140	160

On the basis of the data per head (t/person) (specific values) and the period of supply figures (years), the following conclusions can be made:

- In the world's 8 leading coal producer countries, the amount of coal resources per head is practically twice as much as the 'world average' (260/140,) calculated from currently known (estimated) data. (Obviously, in Asia, Indonesia, Africa or South America, considerable resources may still be discovered.)
- The period of supply calculated for the world's 8 coal producer countries is practically identical with the world average (150/160) although figures reveal quite significant differences between the individual countries (see for example, Russia or China), similarly to the data of industrial coal resources per head (t/person). (Due to export/import data and rates, use parameters may differ for the individual countries.)
- In Hungary, the amount of industrial coal resources per head as well as the supply parameter calculated on the basis of production volumes 10 M t/year, and especially 4 M t/year, well exceeds the world average. In view of this, it is unjustified that the National Energy Strategy only takes into account a 5% coal rate in electricity production forecasts.

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Scientific Reviewers:
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MINING PARAMETERS DETERMINATION IN THE CASE OF THE OPEN PIT AND UNDERGROUND MINING OF THE ROŞIA POIENI COPPER DEPOSIT

ONEȚ VALERIU DAN*
COZMA EUGEN**
ONICA ILIE**

Abstract: *In the future, in the world, the non-metallic minerals requirements are growing and keeping in mining activity of the Roşia Poieni deposit is an opportunity which should not be neglected. Roşia Poieni copper deposit belongs geo-structurally and metalo-genetically to the Neogene eruptive of the Metaliferi Mountains - "Roşia Montană – Bucium – Baia de Arieş" areas, "Roşia Montană – Bucium" District. It belongs to the category of the very big deposits, with small contents of copper – the ore reserves of copper are over* $1.3 \cdot 10^9$ *tonnes and the average content is of 0.34% Cu. For the mining the ore deposit in the deepness there could be applied two mining variants: a) the concomitantly mining of the ore deposit (simultaneously, the open pit and underground mining, delimited by a safety crown pillar); b) the successive mining (open pit, until the depth limit, and then in the underground). In the simultaneous mining conditions, in open pit and underground, of the Roşia Poieni copper deposit, in this paper there will be determined the main parameters of the applied underground mining methods.*

Keywords: *copper deposit, open pit mining, underground mining, room, pillar, ceiling, stability*

1. GEOMECHANICAL ORE DEPOSIT DESCRIPTION

1.1. Geological description

The methodology applied for exploration consisted in 51 ground surface drillings, underground workings (3 levels: 1047m, 945m and 770m) and 209 horizontal and vertical underground drillings, so it runs as to performs a knowledge of the ore deposit into a network of 100 per 100m, on the deepness interval of 1046-851m and 200 per 200m, on the interval of 851-150m.

The copper deposit is stuck in the Fundoaia sub-volcanic body, into the micro-diorite masse, a body with the shape of vertical dome, with the height of about 1200m (it is developed from the level +1030m to the level -150m), with the horizontal shape at the ground surface, irregular and variable as sizes (660m/770m, at the level +956m) and with an elliptic shape (740m/820m, at the level +551.64), to depth.

* *Ph.D. Student - University of Petroşani*
** *Prof., Min. Eng. Ph.D. - University of Petroşani*

2. GEO-MECHANICAL CHARACTERISTICS

The geo-mechanical characteristics [1], [5], [7] of the ore and surrounding rocks are shown in the Table 1 and Table 2.

Table 1. *Mechanical characteristics of the Roşia Poieni copper deposit*

Crt no.	Rock	Hydrothermal transform.	Value	Specific weight	Appar. specif. weight	Compr. strength	Tensile strength	Double shear strength	Internal frictional angle	Cohesion
				kN/m³	kN/m³	kN/m²	kN/m²	kN/m²		kN/m²
1	Andesite of Poieni	Silicified and cloritised	min.	26.4	25.5	524	7 200	14 300	29	920
			av.	27.5	25.7	173	9 500	20 300	-	-
			max.	28.5	26.2	1 189	11 300	26 500	39	1 750
2	Andesite of Poieni	Weakly altered hydrotherm.	min.	25.9	22.6	20 200	2 100	7 000	26	420
			av.	27,5	24.0	35 500	5 500	10 700	-	-
			max.	28.8	25.0	51 400	7 000	13 800	33	2 000
3	Andesite of Poieni	Argillitised	min.	26.0	19.8	6 500	1 500	1 300	27	280
			av.	27.6	22.4	14 800	2 200	4 900	-	-
			max.	29.1	23.8	19 900	2 500	7 000	28	400
4	Andesite of Poieni	Intensively altered and disintegrated	min.	26.7	17.0	-	-	-	11	6
			av.	27.5	20.0	-	-	-	-	-
			max.	28.5	22.0	-	-	-	18	9
5	Andesite of Fundoaia	Silicified and cloritised	min.	25.9	25.8	51 400	51	22 100	33	1 400
			av.	27.2	25.9	77 300	73	23 100	35	2 000
			max.	28.2	26.6	125 800	100	24 200	38	2 000
6	Andesite of Fundoaia	Weakly altered hydrotherm.	min.	22.7	23.0	14 000	2 400	5 800	27	450
			av.	27.3	23.8	31 200	3 100	10 100	-	-
			max.	29.1	25.4	49 900	4 500	18 300	33	650
7	Andesite of Fundoaia	Argillitised	min.	22.8	22.6	6 400	1 100	2 400	26	260
			av.	26.0	22.8	9 500	1 300	4 200	-	-
			max.	27.7	23.0	10 700	1 800	5 800	28	400
8	Andesite of Fundoaia	Non-altered	min.	25.5	21.5	34 600	5 000	-	33	650
			av.	26.0	25.2	93 200	7 900	-	-	-
			max.	26.2	27.0	184 300	16 000	-	37	2 400
9	Andesite of Vârşi		min.	26.0	24.7	35 700	10 300	16 700	33	90
			av.	26.6	25.5	95 300	9 500	19 200	-	-
			max.	27.2	26.4	191 500	13 300	23 000	39	2 700
10	Andesite of Fundoaia with copper mineralization		min.	25.9	25.8	63 000	13 000	24 000	33	1 400
			av.	27.2	25.9	95 000	18 000	42 000	-	-
			max.	28.2	26.6	107 000	24 000	58 000	38	2 000

Table 2. *Elastic characteristics of the Roşia Poieni copper deposit*

Sample no.	Rock characterisation	Elasticity modulus, *E* (10MPa)			Elasticity modulus at the initial loading, *E* (10MPa)	Elasticity modulus at the elastic limit, *E* (10MPa)	Poisson constant (*m*) and Poisson ratio (μ) at the elastic limit	
		Min.	Max.	Average			*m*	μ
I-1 I-2	Andesite silicified and sericitised Andesite silicified and sericitised	144.41 -	350.33 -	273.82 193.93	144.41 193.93	278.53 193.93	7.57 62.85	0.132 0.00159
II-1	Andesite silicified and kaolinised	304.00	758.37	550.87	304.00	758.37	108.56	0,0092
III-1	Andesite silicified and sericitised	110.64	289.97	200.30	112.69	289.97	11.90	0.09
IV-2 IV-3 IV-5	Andesite sericitised, silicified and pyritised Andesite sericitised, silicified and pyritised Andesite sericitised, silicified and limonitic alteration	90.21 57.58 -	223.46 400.54 -	157.11 237.34 91.80	90.21 243.47 91.80	220.15 392.99 91.80	73.23 47.28 6.66	0.013 0.021 0.15
V-1	Andesite silicified, sericitised and pyritised	185.95	431.36	307.91	185.95	409.26	33.30	0.03
VI-2	Andesite sericitised, silicified and pyritised	-	-	471.13	471.13	471.13	18.51	0,054

3. UNDERGROUND MINING METHODS OF THE ROŞIA POIENI COPPER DEPOSIT IN THE CASE OF THE CONCOMITANT MINING

3.1. Mining method selection

Considering the ore and surrounding rocks geo-mechanical characteristics of the Roşia Poieni copper deposit, in the followings it will be taken into consideration the main mining methods that can be applied, namely: open stoping, rooms and pillars, sublevel caving, etc. The selected methods [4] have kept out of low content of the ore deposit, the small value of the ore and the high productivity of the mining technologies.

3.2. Opening, preparation and mining copper deposit in the using case of stopping mining methods

In the Figure 1 it is represented the opening and preparatory workings for concomitant mining (in open pit [2] and underground [3] – open stoping, on the level and sublevel large holes drilling-blasting variants).

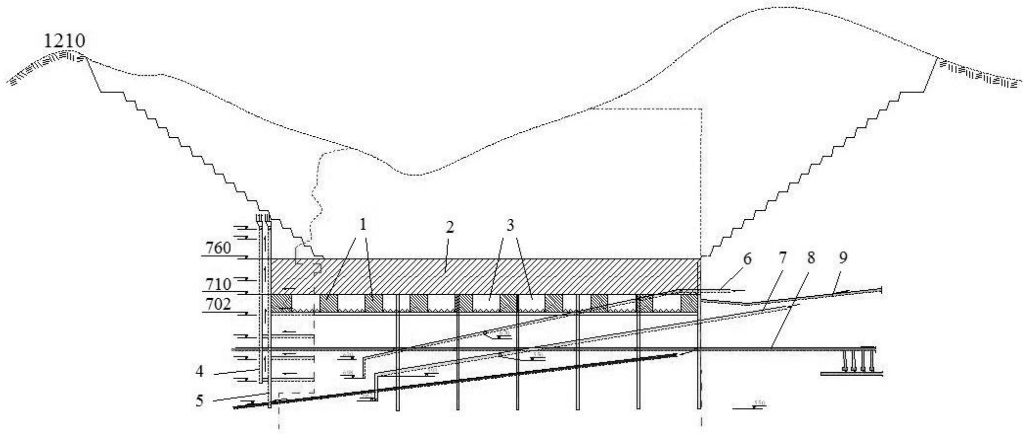

Fig.1. Concomitant mining, in the open pit and underground (stoping mining method), of the Roşia Poieni copper deposit

1- pillars between mining rooms; 2-crown pillar; 3-rooms; 4/5-ventilation/transport shaft; 6/7 – ventilation/transport incline; 8 –main gallery; 9 – adit; 10 – ore crusher

3.2.1. Sublevel stopping, with vertical fan drillings mining method

The application of these methods is possible only by using the high productivity equipments for drilling and holes explosive charging. The favourable conditions for this method application are the followings: large thick and high dip of the ore deposit; hard and stable ore and surrounding rocks; ore without waste rocks intercalations; a low value of the ore. The ore mining begin from a vertical rise situated at a room limit.

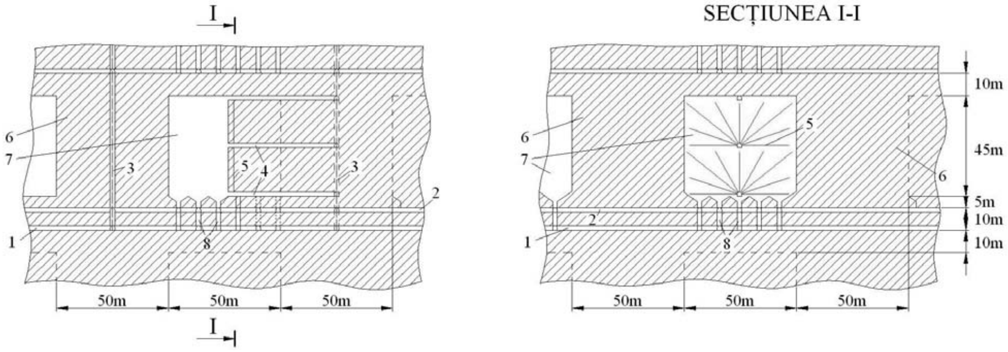

Fig.2. Sublevel stopping, with vertical fan drilling holes mining method

1-transport level; 2-secundary crashing level; 3-rise; 4-sublevel galleries; 5- drillings; 6-pillars, 7-rooms; 8-chutes

3.2.2. Level stopping mining methods

The applied conditions of this mining method are almost similar with the sublevel method. Moreover, for using that method it is necessary a bigger and constant thickness of the ore body, as well as the accurate knowledge of the deposit contour.

The mining method principle consists in the partition, following the deposit strike, the level field into the rooms and pillars. In the first stage there are mined the rooms and in the second stage, the pillars. The slices thickness is between 2 and 5m, depending on the ore hardness.

a. Vertical slice cutting variant of the ore deposit

The particularity of this variant consist in the fact that the ore cutting is made with the parallel or ring drillings, situated into a vertical slices.

In the Figure 3 it is shown the variant with the parallel drilling holes and in the Figure 4 the ring emplacement of the drillings.

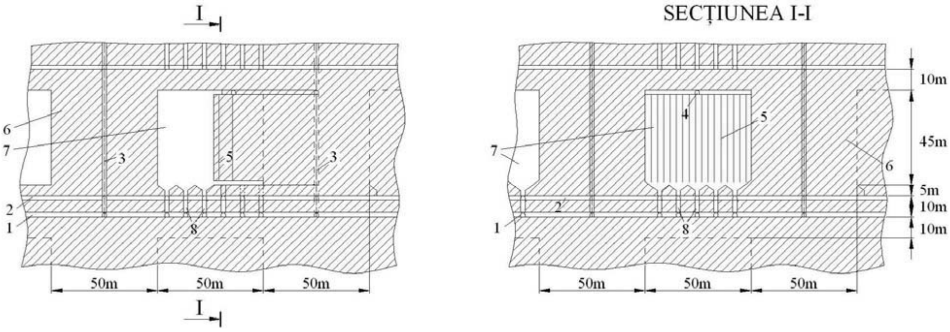

Fig.3. Level stopping, with vertical parallel drilling holes mining method
1-transport level; 2-secundary crashing level; 3-rise; 4-drilling galleries;
5- parallel drilling holes; 6-pillars, 7-rooms; 8-chutes

The method disadvantage consist in the bigger total length of the ring drilling holes (with 50-80% higher then the parallel emplacement) and the bigger grading of the crushed ore (when the distances between the holes tips is more then 1.5-2 times then the distance between the parallel drillings).

b. Horizontal slice cutting variant of the ore deposit

The particularity of this variant (Figure 5) consist in the fact that the ore cutting is made with the parallel or fun drillings, situated into a horizontal slices, achieved from the particular drilling rooms, driven from the panel rises. The preparatory workings are similar with those represented in the vertical variant.

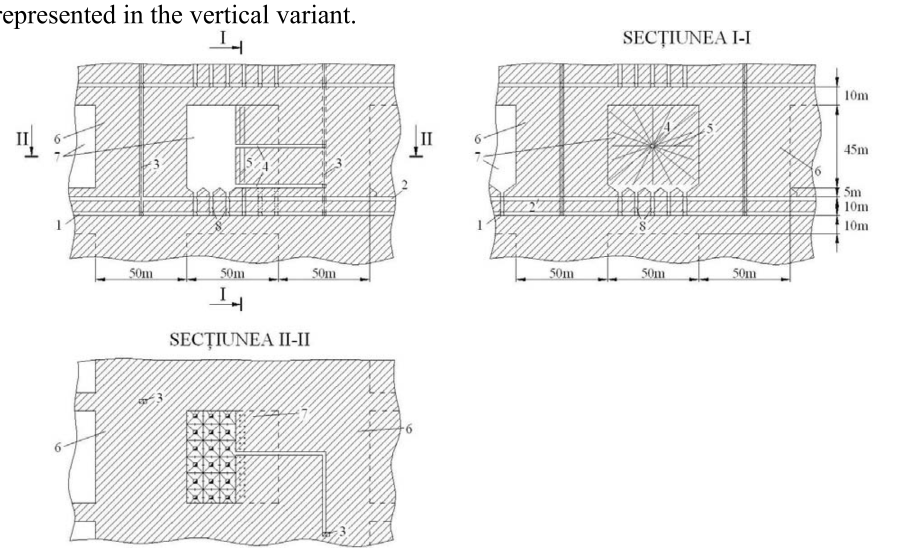

Fig.4. Level stopping, with vertical ring drilling holes
1-transport level; 2-secundary crashing level; 3-rise; 4-drilling gallery;
5- ring drilling holes; 6-pillars, 7-rooms; 8-chutes

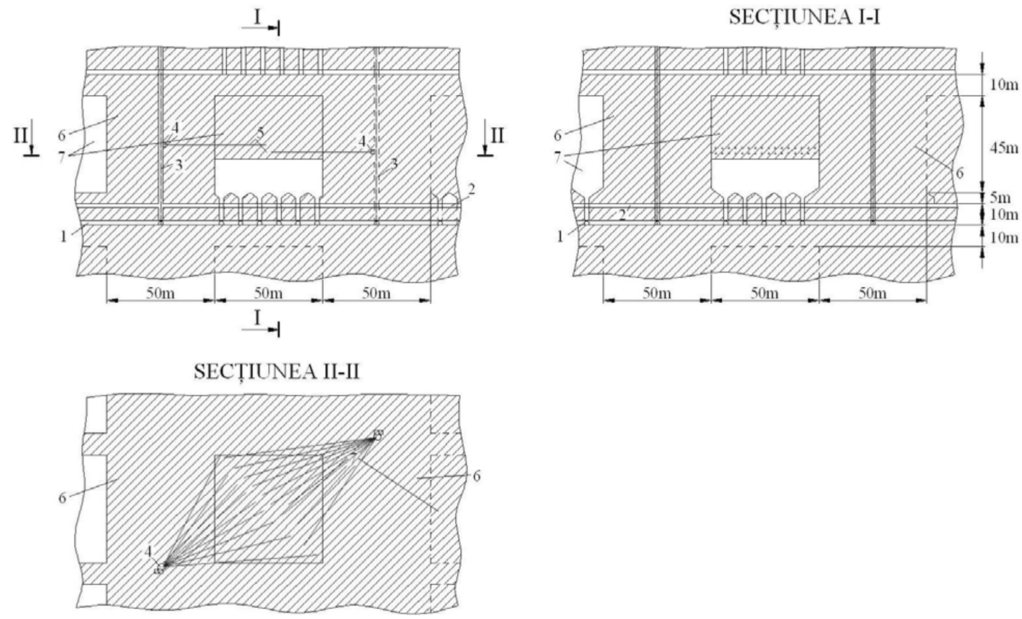

Fig.5. Sublevel stoping, with horizontal fan drillings mining method
1-transport level; 2-secundary crashing level; 3-rise; 4-drilling gallery;
5- fan drilling holes; 6-pillars, 7-rooms; 8-chutes

4. ROOMS STABILITY CHECKING

4.1. Rooms span

From several determination methods of the rooms span, it was taken into consideration two methodologies of the Ritter and Protodiakonov [1], [5], [6]. In conformity with these theories, if it is accepted a safety factor of 4, it results a calculated span of 230m and it is adopted a room span of about 50m.

4.2. Rooms length

Considering the condition that the room walls are stable, under the overburden rocks weight, Borger [5], [6] obtained: for room span of 50m, the rooms' length can reach of 350m. In reality, the rooms' length is 50m and the room stability is ensured.

4.3. Rooms height

Following the Coulomb-Rebhann hypothesis [5], [6], it results that the height of 50m ensures a very good stability of the underground excavations.

4.3. Pillars and ceilings stability assessment

All researchers are in according to the idea that on the pillars acts the compressive static stresses due to the self-weighting and to a certain part of the overburden rocks [5], [6]. For a safety factor value of 3, the pillar width is about 43m, and it is adopted of 50m.

4.3.1. Stamatiu's methodology

This methodology [6] belongs to the procedures that consider the stresses acting on the pillars are distributed non-uniformly. In conformity with this methodology and the obtained results, for the pillars with the width of 50m, the rooms' length must be under the value of 45m.

4.3.2. Sizing of the ceilings pillars between mining levels

Taking into account the fact that the principle generated by the calculation of the ceiling sizes is presented into the works [3], [5], [6], in this paper it is presented the ceiling checking for various loadings. The calculation data are the followings: rooms' width of 50m; pillars sizes of 50x50m; ore density of 2.53t/m^3; ore tensile strength of 58daN/cm^2. It is adopted a safety factor about 3 for a stability period bigger then 70years. After calculation, it is obtained a level ceiling thickness of 6m and it is adopted a practical value of 10m. In these conditions, it is not admitted a supplementary loading on the ceilings.

4.3.3. Checking of the level ceiling thickness

As it is showed previously, the level ceiling thickness is adopted of 10m. For checking there are used several methodologies, namely: the plates bending theories, the ceiling strength to the shear loading, etc. Finally, it results the fact that the ceiling stability is ensured.

5. CONCLUSIONS

After the stability analysis of the support structure in the frame of the open stopping with rooms and pillars mining method, the entire structure remains stable.

The rooms are stable for the proposed sizes, respectively for 50m of width and 350m of length. Regarding the rooms height, these can be of 140m for a safety factor of 5 and 70m for 10. Finally it is adopted a value of 50m.

Concerning the pillars, it is ensured a good stability (safety factor is 3) for pillars width equal to 43-50m. And the ceiling thickness, resulted from calculation, is of 6m and it is adopted the value of 10m.

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Scientific Reviewers:
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ROADHEADERS SELECTION FOR THE TECHNICAL AND GEO-MINING CONDITIONS OF THE ROMANIAN SALT MINES

ONICA ILIE[*]
COZMA EUGEN[*]
MARIAN DACIAN[**]

Abstract: *With the advancement in depth of the mining of the rock salt deposit in Romania, arises the necessity of increasing the support structure stability (pillars and ceilings) and improving the technical and economical performances obtained at the level of every saline. In this way, the introduction of the roadheaders at the cutting operation in the faces could represent an appropriate solution. In this paper, an original algorithm of roadheaders selection is proposed, based on very well grounded technical and geo-mining criteria, both theoretically and practically.*

Keywords: *roadheader selection, rock salt deposit, mining method, rooms and pillars*

1. INTRODUCTION

The most used mining methods, by solid means, in the Romanian salines, in activity in the frame of the Rock Salt Company Bucharest (Cacica, Ocna Dej, Ocna Mureş, Praid, Râmnicu Vâlcea, Slănic Prahova şi Târgu Ocna) are those with rooms and pillars (squared and rectangulared). Within mining technologies at the rooms' level and the opening and preparatory workings, the rock salt cutting is made with explosives, by the drilling - blasting operation. This procedure has some disadvantages such as: low productivity; limited production capacity; ceilings' and pillars' degradation due to the seismic effect of the blast. This last negative effect is the most important one because it is associated with mining depth increase, respectively with the increase of the natural stress- strain state. From here arises the necessity of taking some technical measures to preserve the support structure stability at the acceptable safety level, from the point of view of personnel and equipment. One of these measures that also contribute to the increase of productivity and production capacity would be the introduction of the rock salt mechanised cutting with the aid of roadheaders.

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[*] *Prof., Min. Eng. Ph.D. - University of Petroşani*
[**] *Lecturer, Min. Eng. Ph.D.-University of Petroşani*

The roadheader is a self-propelled unit mining machine that ensures the cutting and loading of the rocks inside horizontal or inclined underground workings (galleries or inclined opening and preparatory workings, tunnels, short mining faces, etc.).

The main classification criterion of the cutting machines is how the cutting head attacks the front face. Following this criterion, the classification of the cutting machines is:

a)cutting machines with integral attack of the face (tunnel boring machines);

b)cutting machines with partial attack of the face (sequential cutting):

b.1)cutting machines with linear attack of the face (continuous miner);

b.2)cutting machines with point attack of the face or roadheaders (the cutting heads or booms having an approximate paraboloid shape):

b.2.1)milling or axial roadheaders (with the rotational axe of the cutting head perpendicular on the working face);

b.2.2)ripping or transversal roadheaders (with the rotational axe of the cutting head parallel with the working face).

Other classification criteria of the roadheaders exist, based on: roadheader mass, power, cutting head power, loading system, boom shape, picks' type, etc.

The goal of this work is to select, from a technical – miner point of view, a roadheader for the technical and geo-mining conditions of the Romanian salines (see the Table no. 1), from the several constructive types made in the world.

From the various types of roadheaders classified after the criterion of front attack, for the Romanian saline situation it is recommended to use roadheaders, being more flexible than the tunnel boring machine and the continuous miner and able to cover bigger excavation profiles. Furthermore, these types of roadheaders were successfully experimented in the world, in similar conditions to the ones of Romanian salines.

In this way, 28 milling and ripping roadheaders were selected, in the 100 tonnes heavy weight class, made by different companies.

The assessment and the selection of these cutting machines were successively provided, in a well established order, depending on the degree of importance of the selection criterion by report to the imposed conditions, as following:

1) The covering of the transversal sizes of the mining excavation;

2) The correlation of directional and transversal dips of the mining working with the technical possibilities of the roadheaders;

3) The tracks penetration in the rock floor of the underground working;

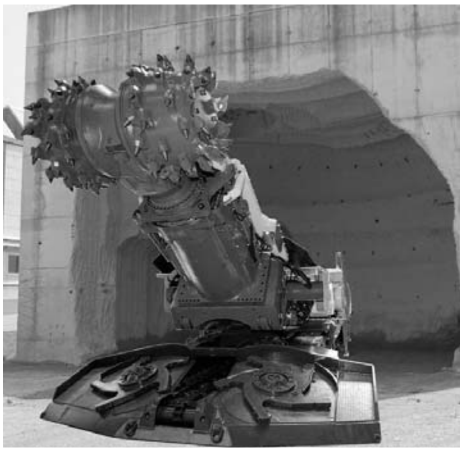
4) The main roadheader performances (the instantaneous cutting rate and cutting picks consumption);

5) The successful experimentation of the roadheader in similar geo-mining conditions.

2. MAIN TECHNICAL AND GEO-MINING CONDITIONS IMPOSED TO THE ROADHEADERS

The selection criteria of roadheaders taken in analysis should be based on the following requirements:

1) It should be able to cut a transversal profile of the mining rooms with a certain maximum cross-section, maximum height and maximum width, mined in slices 4-5m thick, divided into 2-3 bands with 7-8m of width (respectively, to be able to cut a profile with 4-5m of height and 7-8m of width, from a single position of the cutting machine);



a)

b)

Fig.1. Roadheaders: a)ripping roadheader - Sandvik Hardrock Miner MH 620; b) milling roadheader - WIRTH Paurat

2) It should be able to drive (following the dip) an inclined working, with a rectangular profile, and a maximum dip of 10-12^o, cross-section of 30-40m^2, height of 4-5m and width of 7-8m, from a single position of the cutting machine;

3) The roadheader tracks should not penetrate the rock floor of the underground working;

4) A single roadheader should be able to provide a production capacity of rock salt, at the saline level, of about 200-400 thousand tonne/year minimum (depending on the real technical conditions of the saline);

5) The consumption of cutting picks needs to be as low as possible;

6) It should be possible to use an electrical power supply, in the conditions of the existent energetic system of the saline, without substantial investments;

7) It should be a successfully used roadheader in the world, in similar mining conditions of the rock salt or other evaporite rocks;

8) It should have very good reliability, be very easy to maintain and operate and should ensure the best conditions of underground health and safety;

9) It should be able to cover the previous imposed restrictions in the Romanian rock salt deposits conditions, having certain specific geo-mechanical characteristics.

3. SELECTION CRITERIA OF THE ROADHEADER

3.1. Covering of the transversal sizes of the mining excavation

We will compare the maximum sizes of the cutting profile of the roadheaders from a single position, with the mining belt sizes, H_m and L_m, necessary to be mined into a mining room (imposed by the mining rooms or by the transversal profile of the opening or preparatory working). A covering rate of the working height $K_H = \frac{H}{H_m} \cdot 100$, in % and a covering rate of the working width $K_L = \frac{L}{L_m} \cdot 100$, in % (where: H and L are the maximum cutting height and width from a single position, in m) will be calculated for every roadheader. Finally, the roadheaders having a covering rate less than 100% will be eliminated from analysis. Also, the roadheaders for which the covering rate is of about 100% or slightly higher should be capable to achieve opening and preparatory workings inside the rock salt massive, in very good technical and economical conditions.

Table nr.1 *Roadheaders' selection after the criterion of the covering rate of the mining profile sizes*

***H*, [m]**	***L*, [m]**	**K_H, [%]**	**K_L, [%]**	**Selection**	***H*, [m]**	***L*, [m]**	**K_H, [%]**	**K_L, [%]**	**Selection**
MR 520					**AM 105 ICUTROC**				
5,2	8,32				5,9	8,9			
MH 620					**MT 720**				
5,8	8,8				6,6	9,1			
ATM 105 IC					**AM 105-Ex**				
6,6	9,1				5,9	8,9			
ET 120					**ET 170**				
4,05	5,5				4,6	6,1			
ET 180					**ET 210**				
5,1	6,6				4,7	7,1			
ET 250					**ET 380**				
5,5	7,5				7,45	9,8			
ET 410					**ET 410Q**				
5,5	10,0				6,25	10,3			
ET 450					**ET 450L**				
7,3	10,3				7,7	10,4			
ET 480					**Mk-3**				
7,9	11,0				6,0	7,2			
Mk-4					**AQM-200**				
5,223	7,44				5,3	7,6			
T 1.14					**T 1.24**				
4,8	6,6				5,0	6,6	100	88	**0**
T 2.24					**T 3.20**				
6,0	7,8				7,7	9,5			
T 3.20S					**EBZ 318H**				
7,69	8,96				5,59	7			
EBZ 260H					**EBZ 200H**				
5,0	6,2				4,8	6,0			
Observation: On the column regarding "Selection", in conformity with the values of the coefficients K_L *şi* K_H, the following will be written: **0 –** uncovered profile**; X-**covered profile**; XX-** overweight covered profile									

From the analysis of the covering rate values of the 28 milling and ripping roadheaders, presented in the previous table, the roadheaders which do not cover the entire sizes of the mining working profile and the ones which overweightly cover the profile will be eliminated. The overweight cover, besides the large sizes of the roadheader body, means larger weight of the roadheader, increased powers and implicitly a higher cost and reduction of the economical efficiency of these roadheaders, taking in consideration the mining conditions of every saline taken into study.

3.2. Correlation of the straight and transversal dips of the mining workings with the technical possibilities of the roadheaders

We mention that all the roadheaders taken into study ensure the roadheader tramming in the inclined way (upgrade and downward) on the inclination of about 16°-18°, sufficient for a designed incline with a maximum dip of 10°. Also, the maximum transversal dip ensured by the roadheader, ranging between 6° and 8°, is sufficient for the technical conditions required from the mining workings that will be achieved.

3.3. Tracks penetration in the rocks floor of the underground working

In order to avoid the track roadheaders penetration into the rock floor of the underground working, the condition that ground pressure must be less than the compressive strength of the rock salt should be respected. In this way, for the selected roadheaders, a safety coefficient at the penetration will be calculated:

$$K_{SP} = \frac{R_C \cdot C_S}{P_v} > 1 \tag{1}$$

Where: R_c represents the rock salt compressive strength; P_v –ground pressure; C_S=0.5-0.7 is a reduction coefficient of the compressive strength, depending on the fissure and alteration degree of the rocks.

Taking into account the rock salt strength, from the Romania deposits, this criterion is respected by all of the 28 roadheaders taken into study, because their ground pressure is under 0.21MPa.

3.4.Main roadheader performances (the instantaneous cutting rate and cutting picks consumption)

For the assessment of the instantaneous cutting rate and cutting picks consumption, in function of the geo-mechanical characteristics of the rock salt massive mined with aid of the roadheader, there were taken into consideration a series of prediction formulas established through statistical analysis of the data obtained from the measurements achieved in laboratory or in situ by certain authors [1], [3], [4], [5] (see Table no. 2 and 3).

NOTATIONS:

Q - instantaneous cutting rate of the roadheader (at linear cutting), [m³/h];
C_c - specific cutting picks consumption, [picks/m³];
IPR – roadheader penetration index;
IC – roadheader cutter consumption index;
P- cutterhead power, [kW];
m- roadheader weight, [tones];
D – cutterhead diameter, [m];
R_c- uniaxial compressive strength, [MPa];
e – base of natural logarithm;
R_T- tensile strength by the Brazilian method, [MPa];
E – static elasticity modulus, [GPa];
d- cutting depth of the top picks (of 5mm and 9mm) obtained from the linear cutting tests, [mm].

Table no.2. *Prediction models of the roadheaders' performances developed throughout time, in function of the rock uniaxial compressive strength*

References	***Prediction equations of the cutting rate Q and the picks consumption C_c***	***Explanations***
Gehring [4]	$Q=\frac{719}{R_c^{0,78}}$ (2)* -for the ripping (transverse) roadheader	It was based on the performance of a roadheader with a 230kW axial type cutterhead and an Alpine Miner AM 100 with a 250kW transverse type cutterhead
	$Q=\frac{1739}{R_c^{1,13}}$ (3)* -for the milling (axial) roadheader	
Thuro [5]	$Q=75,7-14,3\cdot\ln(R_c)$ (4) -for the ripping roadheader	It was based on the performance of a Atlas Copco Eickhoff ET 120 (132 kW) transverse (ripping) roadheader
Balci [1]	$Q=0,8\cdot\frac{P}{0,37\cdot R_c^{0,86}}$, R^2=0,89 (5) -for *d*=5mm -for the ripping and milling roadheader	All tests are carried out with a conical cutter of S-35/80H manufactured by Sandvik. It has a gage of 80mm, flange diameter of 64 mm, shank diameter of 35mm, tip diameter of 22mm and primary tip angle of 80°.
	$Q=0,8\cdot\frac{P}{0,41\cdot R_c^{0,67}}$, R^2=0,76 (6) -for *d*=9mm -for the ripping and milling roadheader	
Copur [2]	$Q=27,511\cdot e^{0,0023\cdot(IPR)}$, $IPR=P\cdot\frac{m}{R_c}$, R^2=0,93 (7)** -for the ripping roadheader	The calculus relations were elaborated for the ripping roadheaders. *Q* is calculated for the evaporitic rocks (where is involved the rock salt) and C_c for the sedimentary rocks in general.
	$C_c=897,06\cdot(IC)^2+6,1769\cdot(IC)$ $IC=R_c/(P\cdot m\cdot D)$, R^2=0,96 (8)** -for the ripping roadheader	
Observation**:** relations ( ) - with medium degree of confidence; relations ( )* - with great degree of confidence; relations ( )** - with high degree of confidence (from the point of view of this study)		

Specific energy is defined as the amount of work required to break a unit volume of rock and is used to predict the performance of mechanical miners [1].

The prediction models of the instantaneous cutting rate of the roadheader elaborated by Balci [1], in function of the rocks' geo-mechanical characteristics (uniaxial compressive strength, indirect (Brazilian) tensile strength, static and dynamic elasticity moduli, Schmidt hammer rebound values, density, and Cerchar abrasivity index) are based on the optimum specific energy, in kWh/m^3, at which the formation is excavated in optimum cutting geometry [1].

All tests are carried out with a conical cutter of S-35/80H manufactured by Sandvik. It has a gage of 80 mm, flange diameter of 64 mm, shank diameter of 35 mm, tip diameter of 22 mm and primary tip angle of 80° [1].

Table no.3. *Prediction models of the roadheaders' performances (transverse and axial type) developed by Balci et.al., in function of the tensile strength obtained from the Brazilian indirect test and function of the static elasticity moduli of the rocks* [1]

Geo-mechanical parameter	Q, m^3/h	R^2	d, mm
R_T, MPa	$Q = 0{,}8 \cdot \dfrac{P}{3{,}36 \cdot R_T^{\,0{,}71}}$ (9)	0,85	5
	$Q = 0{,}8 \cdot \dfrac{P}{2{,}19 \cdot R_T^{\,0{,}62}}$ (10)	0,71	9
E, GPa	$Q = 0{,}8 \cdot \dfrac{P}{3{,}55 \cdot E^{0{,}71}}$ (11)	0,66	5
	$Q = 0{,}8 \cdot \dfrac{P}{2{,}68 \cdot E^{0{,}4}}$ (12)	0,65	9
***Observation*:** relations ( ) - with medium degree of confidence; relations ( )* - with great degree of confidence; relations ( )** - with high degree of confidence (from the point of view of this study)			

As a consequence of the use of calculus relations, presented in Tables no.2 and 3, the obtained results will be critically analysed and the extreme values will be eliminated, obtaining certain average, technically plausible, values.

Considering the previously obtained instantaneous cutting rate, the selected roadheader must meet the condition that the operational productivity Q_e must be equal or slightly higher than the designed production capacity: $Q_e \geq A_h$ (A_h – the hourly average mine production).

The roadheaders, as well as the other types of mining machines or plants are characterised by 3 kinds of productivity, expressed in tonnes/hour, namely: theoretical productivity Q_t, technical productivity Q_{teh} and the operational productivity Q_e [3].

The roadheader *theoretical productivity* Q_t is determined as the maximum possible productivity for continuous work of the cuttinghead and it is expressed by the quantity of the mining mass detached from the massive in time unit [3].

The roadheader *technical productivity* Q_{teh} is less than the theoretical one and it is the maximum possible productivity for continuous working, taking into consideration the necessary times for manoeuvring operations that do not overlap the working operations and those for worn down picks renewal[3].

The roadheader operational productivity Q_e depends, in addition to the stops listed at the technical stops, on the additional stops due to technical – organizational causes.

Therefore, between these productivities, the following correlation exists:

$$Q_t > Q_{teh} > Q_e \qquad (13)$$

At this rate, it could be considered that the theoretical productivity Q_t of the roadheader is:

$$Q_t \cong Q \cdot \rho \text{, in tonnes/hour} \qquad (14)$$

Where: Q is the instantaneous cutting rate, in m^3/hour (assessed with the empirical relations from Tables no.2 and 3); ρ - rock salt apparent specific density, in tonne/m^3.

Therefore, the theoretical productivity of the last *n* selected roadheaders is presented in Table no.4.

Table no.4. *Theoretical productivity of the selected roadheaders*

Theoretical productivity Q_t, in t/h		
Roadheader type		
Roadheader 1	Roadheader 2	….Roadheader *n*
….	….	…

Between the theoretical and operational productivity of the roadheaders, the following correlation exists:

$$Q_e = Q_t \cdot K_{teh} \cdot K_e \text{, [tonnes/hour]} \qquad (15)$$

In the previous relation, K_{teh} is a coefficient of the continuity of the roadheader work that takes into consideration the technical stops. Taking into account that productive activity is provided on a single shift, the technical maintenance, worn down picks renewal, etc. could be executed without production time. Moreover, the recovery of lost production in conditions of serious damages could be reprogrammed outside the production time (on Saturday and Sunday). Therefore, the value of K_{teh}=0.7-0.85 (namely, 15-30% of the effective work time lost due to technical stoppages) can be considered.

Also, K_e is a coefficient taking into account the technical and organizational stops, other then those directly related to the roadheader. In this case as well, the lost production due to the occasional stops generated by injuries produced at the mine level (such as electrical power interruption, rocks falling down, gas emissions, work injuries, etc.), could be recovered outside the productive time. Therefore, we can consider that the main discontinuities in the effective work time of the roadheader are generated by the auto transport organization of the rock salt production, and a coefficient with a minimum value of K_e=0.5-0.75 (25-50% of the roadheader work time is lost due to the rock salt transport organization, that is specific for every saline). An auto transport along short distances, only on the mining level, mixed with a continuous transport with rubber belts until mine surface should widely increase the roadheader efficiency of use.

Taking into account the values of these two coefficients, the operational productivity is presented in the next table.

Table no.5. *Operational productivity of the selected roadheaders*

Operational productivity Q_e, in t/h		
Roadheader type		
Roadheader 1	Roadheader 2	….Roadheader *n*
….	….	….

The roadheader operational productivity necessary to achieve the annual mine production is:

$$Q_e^{nec} = \frac{A}{N_z \cdot N_{sch} \cdot N_h} \text{, [t/h]} \qquad (16)$$

Where: N_z is the number of the annual work days (N_z=200days/year); N_{sch} -number of shifts during the day (N_{sch}=4); N_h – number of hours on the shift (N_h=6 hours).

Thus, the calculated operational productivity Q_e, for every roadheader restrained into competition, will be compared with the necessary productivity Q_e^{nec} and, finally, only the roadheaders that respect the condition: $Q_e \geq Q_e^{nec}$, will be selected.

3.5.The successful experimentation of the roadheader in similar geo-mining conditions

In the case where we arrive at the end of the procedure of roadheaders assessment and selection, from the last roadheaders remaining, with close performances, selection will be made following the criterion of use experience in similar geo-mining conditions. Finally, we end up with the roadheader with the best performances and that is adapted the best at the technical and geo-mining conditions required by the saline for which the roadheader is purchased.

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PARAMETERS' OPTIMIZATION OF THE LONGWALL FACES WITH TOP COAL CAVING MINING, IN HORIZONTAL SLICES AND INDIVIDUAL SUPPORT TECHNOLOGY

MIHĂILESCU VIOREL*
ONICA ILIE**
COZMA EUGEN**

Abstract: *The coal seam no.3 mining, in the Jiu Valley hard coal basin, made by longwall faces with top coal caving mining technology, in the horizontal slices mining method, using individual supports, could be efficient only if the sizes of the top coal height and the panel length determine o minimum cost of production. The goal of this paper is the optimization of the top coal height and the panel length from a technical and economical point of view.*

Keywords: *longwall face, top coal caving mining, horizontal slice, mining parameters, top coal height, panel length, optimization, cost*

1. INTRODUCTION

Specifically for the mining industry, the operating process obeys the same economical laws as any industrial process. Their modelling respects the following scheme: $X\ \{\rightarrow [S] \rightarrow\}\ Y$; where: $Y = (y_i)$ is the system's outputs volume, $i = \overline{1,n}$; $X = (x_i)$ - the qualitative and quantitative factors that influence the system's outputs y_i , $i = \overline{1,m}$; *S*- system structure through the factors x_i that influence the outputs y_i.

A mining operating process could be represented, schematically [7], by a series of sets as of: the consumption factors that enter in the process $x = \{x_1, x_2, ..., x_n\}$; the rules that the process has a time development $L = \{l_1, l_2, ..., l_k\}$; the mining products (the production factors that are coming out of the process) $Y = \{y_1, y_2, ..., y_m\}$ and the perturbation factors $U = \{u_1, u_2, ..., u_p\}$.

* *Ph.D. Student - University of Petroşani*
** *Prof., Min. Eng. Ph.D. - University of Petroşani*

2. ESTABLISHMENT OF THE TECHNOLOGICAL UNIT FOR OPTIMIZATION

Analysing the mining method (Fig.1) and mining technology (Fig.2) for the thick coal seams, it is observed that they are characterized by a series of parameters classified into 3 importance groups, as in [6], [7]:

a) Main parameters: represent the panel elements and those that are determined based on their interdependences, some auxiliary parameters and the technical and economical indicators;

b) *Auxiliary parameters*: determine or are determined from main parameters in function of the applied technological variant specificity and have a determined role in the mining method economy;

c) *Specific parameters*: are specifically only for a certain mining variant or method and have an influence, especially, on the labour productivity at the coal cutting, loading and exhaustion.

The main variable parameters that characterise this compound of the mining unit and which must be optimized are the panel length (l_{ca}) and the top coal caving height (h_b).

The criteria used in the optimization problem solving are the following: the cost of the unit product c $(c \to \min)$; the output capacity P $(P \to \max)$ and the labour consumption M $(M \to \min)$ [1], [3], [7].

For solving the mathematical and economical models, resulted from the applied certain programming method, only one optimization criterion is taken into consideration, for the other ones being imposed restrictions, thus [6], [7]:

-if the used basis criterion is the cost of the product unit (lei/tonne) then the purpose function has the form: $C \to \min$, with the restrictions: $P \geq P_{plan}$ and $M \leq M_{admis}$ [11];

-if the output and the labour consumption are considered as a basis criterion, then: $P \to \max, M \to \min$, with the restriction: $c \leq c_{plan}$ (where: *plan* – targeted; *admis* – admissible).

The sequence of operations in the case of the top coal caving in the longwall mining technology and horizontal slices, using individual supports (articulated caps GSA-1250mm, transversal beams GSA-570mm and hydraulic props SVJ-2.5m) is the following (Fig.2): *Stage 1*: coal face cutting by drilling-blasting (Fig.2.a); *Stage 2*: articulated caps and wire mesh mounting (Fig.2.b); *Stage 3*: crashed face coal loading and transportation and hydraulic props mounting (Fig.2.c); *Stage 4*: caved top coal drawing (Fig.2.d); *Stage 5*: removal of conveyor on the new position (Fig.2.e); *Stage 6*: roof control by dismantling the props and the articulated caps from the rear position (Fig.2.f) [1], [2], [5], [9].

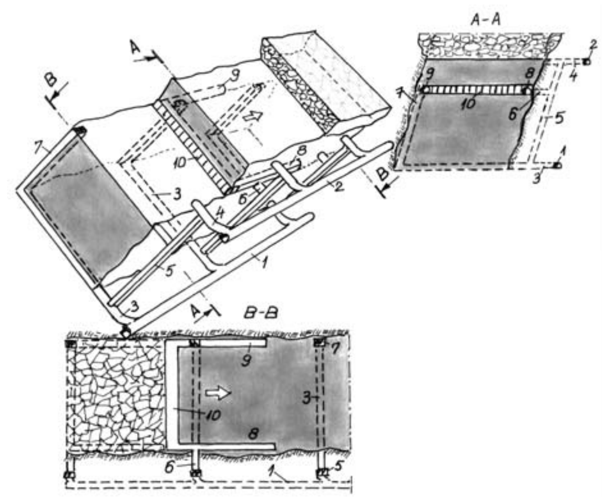

Fig. 1. Top coal caving mining method in horizontal slices, with strike advancement of the longwall mining [9]

1/2 –transport/ventilation directionally gallery; 3/4 – transport/ventilation panel cross-cut gallery; 5-transport rise; 6 – short cross-cut gallery; 7 – ventilation rise; 8 – transport preparatory strike gallery; 9-ventilation preparatory strike gallery; 10 – longwall face with top coal caving

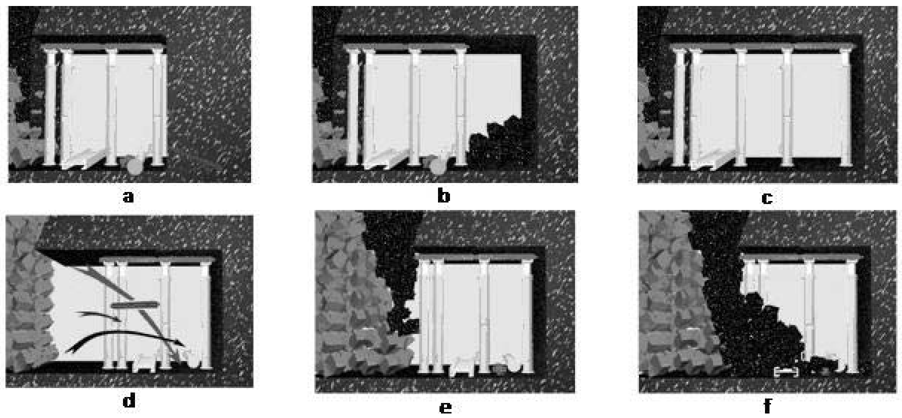

Fig.2. Sequence of operations in the case of the top coal caving in the longwall mining technology, using individual supports

3. ESTABLISHMENT OF THE PURPOSE FUNCTION OF THE OPTIMIZING MODEL

At the level of the *mining economical units,* the cost product on the primary cost elements has the following structure: 1) raw, basic and auxiliary materials; 2) fuel; 3) energy and water; 4) depreciation and fixed assets; 5) other material costs $(1 \rightarrow 15)$; 7)salaries; 8) insurances and social protection; 9) other labour costs (including the recovery of the geological researches); 10) total labour costs; 11) total costs (6+10).

Thus, the general form of the optimization criterion is:

$$c = \frac{\sum C_i}{R_e}\text{, in lei/tonne} \quad (1)$$

Where: c- represents the unit cost of the coal mining at the longwall face level, in lei/tonne; $\sum C_i$ - the total costs, in lei; R_e- the industrial coal reserves of the mining panel, in tonnes.

The panel industrial coal reserves are calculated with the relation:

$$R_e = l_{ca} \cdot \frac{m}{\sin\alpha} \cdot (h_{ab} + h_b) \cdot \rho \cdot k_1\text{, in tonnes} \quad (2)$$

Where: l_{ca} represents the panel length, in m; $l_{ab} = \frac{m}{\sin\alpha}$ - longwall face length, in m; m – normal thickness of the coal seam, in m; α - coal seam dip, in^{o}; h_{ab} – face height, in m; h_b – top coal height, in m; ρ - coal apparent specific density, in tonne/m^3; $k_1 = \frac{\eta}{1-D}$ - coefficient that takes into consideration the recovery factor η and the dilution factor D.

4. MATHEMATICAL MODEL OF THE UNIT COST, AFFERENT TO THE LONGWALL FACE WITH TOP COAL CAVING

The mathematical model of the general costs, elaborated for the longwall face with top coal caving, in horizontal slices, as a basic technological unit, contain all of the cost categories effectuated for the coal mining at the panel level:

$$\sum C = \sum P + \sum I + \sum U + \sum M + \sum E + \sum A\text{, in lei} \quad (3)$$

Where: $\sum C$ represents the total costs effectuated at the panel level, in lei; $\sum P$ - preparatory workings costs, in lei; $\sum I$ - preparatory working maintenance costs, in lei; $\sum U$ - longwall equipments (drilling, support and transportation) costs in lei; $\sum M$ -labour costs, in lei; $\sum E$ - energy (electrical, pneumatic) costs, in lei; $\sum A$ - auxiliary costs (equipment maintenance, materials – wire mesh, wood, explosive material), in lei.

Finally, taking into account the previous equations and every element of the costs, the following mathematical model of the unit cost $c(l_{ca}, h_b)$ is obtained, in lei/tonne, as a dependent function – where: the panel length l_{ca} and the top coal height h_b are independent and variable parameters; coal seam thickness m and the dip α are the known data, but different in order to the geological conditions of the coal deposit. Also, the unit cost function contains a chain of constants $X_1, X_2, \ldots, X_{20}$ that results from the different fixed elements of calculation.

$$
\begin{aligned}
c(l_{ca},h_b)= {} & l_{ca}\cdot\left(\frac{X_8}{4\cdot\frac{m}{\sin\alpha}}+\frac{X_9}{4}+\frac{X_{10}}{1.6}+\frac{X_{11}}{1.6}\right)+h_b\cdot\left(\frac{X_3}{4\cdot\frac{m}{\sin\alpha}}+\frac{X_4}{4}+\frac{X_5}{4\cdot m}+\frac{X_{19}\cdot\frac{m}{\sin\alpha}}{4}+\right)\\
&+\frac{1}{l_{ca}}\cdot\left(\frac{X_7}{4}+\frac{X_{12}}{4\cdot m}+\frac{X_{13}}{1.6}+\frac{X_{14}}{1.6\cdot m}+\frac{X_{15}}{1.6\cdot m}+\frac{X_{20}}{4\cdot m}\right)+\frac{1}{h_b}\cdot\left(\frac{X_1}{1.6\cdot\frac{m}{\sin\alpha}}+\frac{X_2}{1.6}+\frac{X_6}{1.6\cdot m}+\frac{X_{18}\cdot\frac{m}{\sin\alpha}}{1.6}\right)+\\
&\frac{l_{ca}}{h_b}\cdot\left(\frac{X_8}{1.6\cdot\frac{m}{\sin\alpha}}+\frac{X_9}{1.6}\right)+\frac{h_b}{l_{ca}}\cdot\left(\frac{X_{13}}{4}+\frac{X_{14}}{4\cdot m}+\frac{X_{15}}{4\cdot m}+\frac{X_{17}}{1.6\cdot m}+\frac{X_{16}}{1.6\cdot m}\right)+l_{ca}\cdot h_b\cdot\left(\frac{X_{10}}{4\cdot\frac{m}{\sin\alpha}}+\frac{X_{11}}{4}\right)\\
&+\frac{1}{l_{ca}\cdot h_b}\cdot\left(\frac{X_7}{1.6}+\frac{X_{12}}{1.6\cdot m}+\frac{X_{20}}{1.6\cdot m}\right)+\\
&+\left(\frac{X_1}{4\cdot\frac{m}{\sin\alpha}}+\frac{X_2}{4}+\frac{X_3}{1.6\cdot\frac{m}{\sin\alpha}}+\frac{X_4}{1.6}+\frac{X_5}{1.6\cdot m}+\frac{X_6}{4\cdot m}+\frac{X_{18}\cdot\frac{m}{\sin\alpha}}{4}+\frac{X_{19}\cdot\frac{m}{\sin\alpha}}{1.6}\right) \qquad (4)
\end{aligned}
$$

5. DATA ANALYSIS TO ESTABLISH THE OPTIMUM VALUES OF THE PANEL LENGTH AND THE TOP COAL HEIGHT IN FUNCTION OF THE UNIT COST

The optimum values of the panel length and of the top coal height could be obtained by two methods: a) analytical method (by finding the minimum of the unit cost function); b) variants' method (analyzing the sensitivity of the unit cost function to the optimizing parameters variation) [1], [9].

In our case, the input values, in the unit cost function (4) are the following: m={17, 20, 25, 30, 35, 40, 45, 50, 55}m; α ={46, 50, 55, 60, 65, 70, 75}°; l_{ca}={100, 200, 300, 400, 500, 600}m; h_b={1,2,…,15}m. Also, a set of number of workers per shift was taken into consideration, so that: n= {6, 8, 10, 12, 15}.

In the basis of obtained data, there was provided a series of graphical representations of unit cost variation (Fig.3-10) for the values range of the top coal height between 3m and 15m and different face's lengths, in the case of the presence at the longwall face a number of 6, respectively 15 workers.

From the graphics represented in Fig.3 results the fact that the minimum value of the unit cost c=615lei/t, in the case of the placement in the longwall face of 6 workers, is given by the values of l_{ca}=300m and h_b=7m. The rest of the costs, for the other values of panel length and top coal height, are superior to the previous ones. Thus, for h_b=15m and l_{ca}=300m, c=709lei/t, representing an increase of 15.3% of the unit cost of the mined coal. For the maximum value of the panel length of l_{ca}=600m, c=634lei/t, results an increase of 3.08%.

Considering that the unit cost values are contained in a variation range of $\pm 5\%$, it is possible to define an optimum range in this interval, both for the top coal height and the panel length. In the conditions of the 6 workers in the face and the face length of l_{ab}=20m, the optimum range is l_{ca}=150-600m (with optimum point of l_{ca}=300m) and h_b=5-10m.

The data analysis used for elaboration of the graphics from Fig.4 leads to the following conclusions:

The minimum value of the unit cost is c=594lei/t, for l_{ca}=300m and h_b=7m. Also, the range of the unit cost variation of $\pm 5\%$ is for l_{ca}=200-600m and h_b=5-10m. For the maximum value of the panel length, c=610lei/t, representing an increase of 2.7% by report to the absolute minimum value, and for h_b=15m, c=684lei/t, with an increase of 15.5% by report to the minimum value. The 15 workers placed in the coal face leads to a reduction of the unit cost with 19.36%, namely with 115lei/t.

When l_{ab}=24m and n=6 workers in coal face, respectively n=15workers we have the following (Fig.5 and 6):

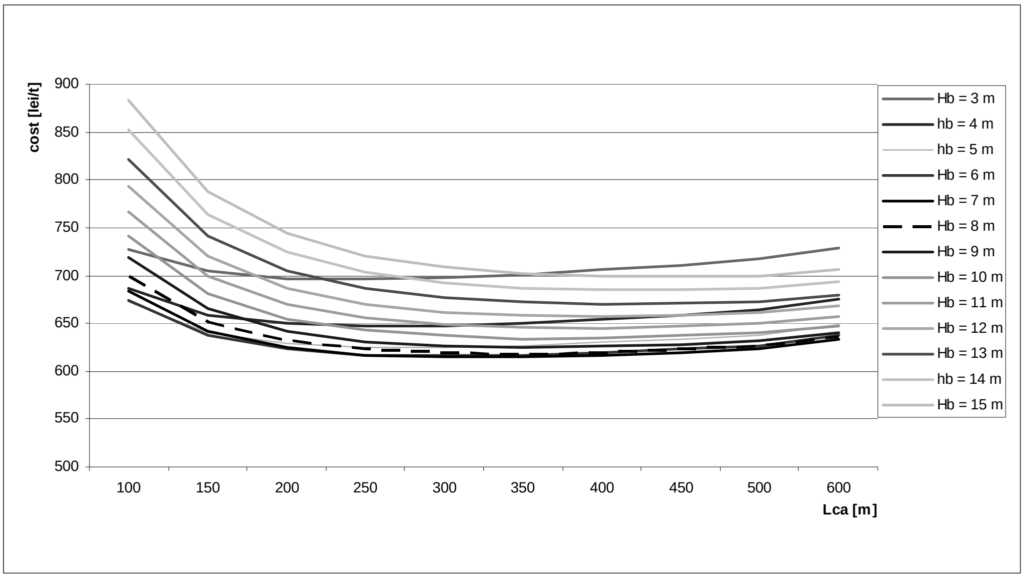

Fig.3. Variation of the unit cost, top coal height and panel length, for l_{ab}=20m and n=6workers/shift

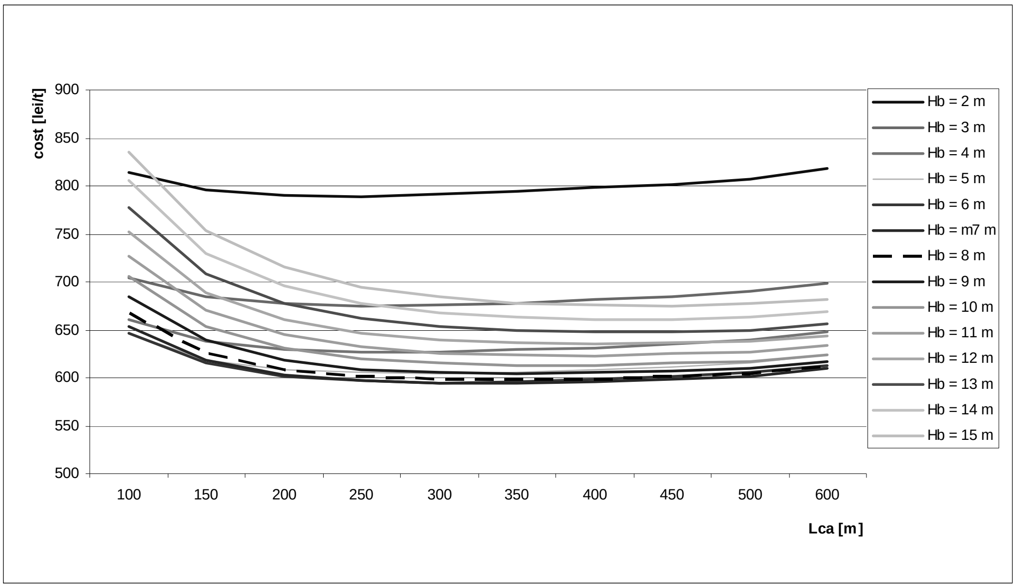

Fig.4. Variation of the unit cost, top coal height and panel length, for l_{ab}=20m and n=15workers/shift

a) For n=6, l_{ca}=300m and h_b=6m, the minimum unit cost is c=566lei/t. The limits for the $\pm 5\%$ of the cost variation are: l_{ca}=150-600m and h_b=4-10m. For l_{ca}=600m, c=587lei/t, representing an increase with 3.71%, and for h_b=15m, c=670lei/t, resulting an increase with 14.13% of the unit cost.

b) For n=15, l_{ca}=300m and h_b=6m, the minimum unit cost is c=545lei/t. For l_{ca}=600m, c=562lei/t (cost increasing with 3.11%), and for h_b=15m, c=645lei/t (cost increasing with 8.34%). The growth of the number of workers from 16 to 15 leads to a unit cost reduction with 21 lei/t or 3.85%.

Comparing the obtained data for l_{ab}=24m and l_{ab}=20m, we have the following conclusions: for a number of 6 workers, increasing the coal length from 20m to 24m leads to a cost reduction with 8.65% (from 615lei/t to 566lei/t), and for a placement of 15workers/shift the cost reduction is of 8.99% (from 594lei/t to 545lei/t).

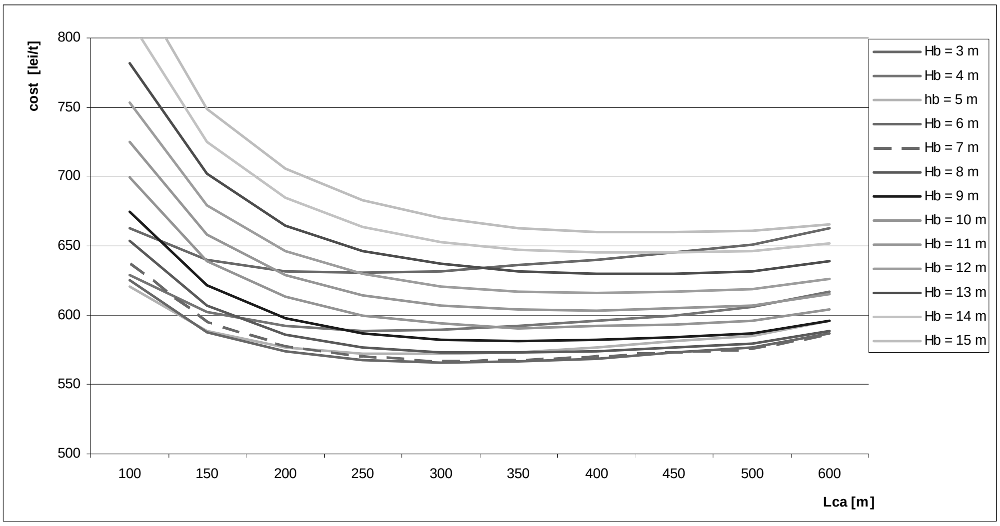

Fig.5.. Variation of the unit cost, top coal height and panel length, for l_{ab}=24m and n=6workers/shift

From the data analysis, corresponding to Fig.7 and Fig.8, we have the conclusions:

a)For n=6, l_{ca}=250m and h_b=6m, the minimum unit cost is c=435lei/t. The limits for the $\pm 5\%$ of the cost variation are: l_{ca}=150-500m and h_b=4-9m. For l_{ca}=600m, c=462lei/t, representing an increase with 6.2%, and for h_b=15m, c=543lei/t, resulting in an increase with 24.8% of the unit cost.

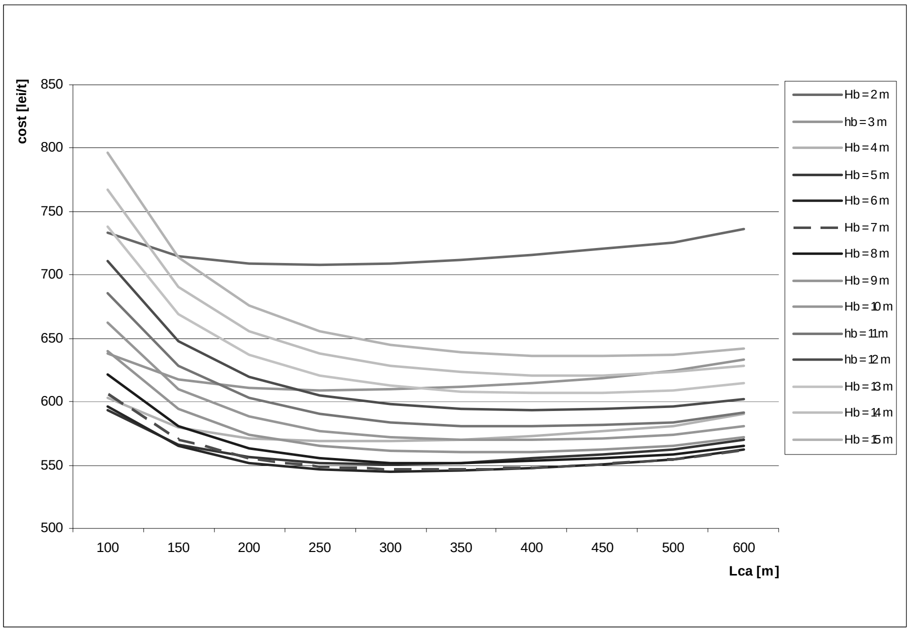

Fig.6. Variation of the unit cost, top coal height and panel length, for l_{ab}=24m and n=15workers/shift

b)For n=15, l_{ca}=250m and h_b=6m, the minimum unit cost is c=412lei/t. For l_{ca}=600m, c=438lei/t (cost increasing with 6.31%), and for h_b=15m, c=514lei/t (cost increasing with 24.75%). The growing of the workers number from 16 to 15 leads to a unit cost reduction with 23 lei/t or 5.58%.

Comparing these data with those ones provided for l_{ab}=20m, result the followings: increasing the coal face length with 20m, for l_{ca}=300m, h_b=6m and n=6 workers/shift leads to the unit cost decreasing with 175lei/t or 38.2%. For l_{ca}=300m, h_b=6m and n=15 workers/shift, increasing the face length from 20m at 40m, cost reduction is with 178lei/t, representing 42.78%. The minimum values of the unit cost obtained for l_{ab}=20m and l_{ab}=40m, c=615lei/t (for l_{ab}=20m, h_b=7m and n=6 workers/shift), by report to l_{ca}=250m and h_b=6m, a cost reduction with 180lei/t or 41,37%; and for a placement of 15 workers, it is c=594lei/t (for l_{ca}=300m and h_b=7m) and c=412lei/t (for l_{ca}=250m and h_b=6m), resulting in a reduction with 182lei/t (44.17%).

Similar technical and economical analyses are achieved for the cases presented in the following graphics (Fig.9-10).

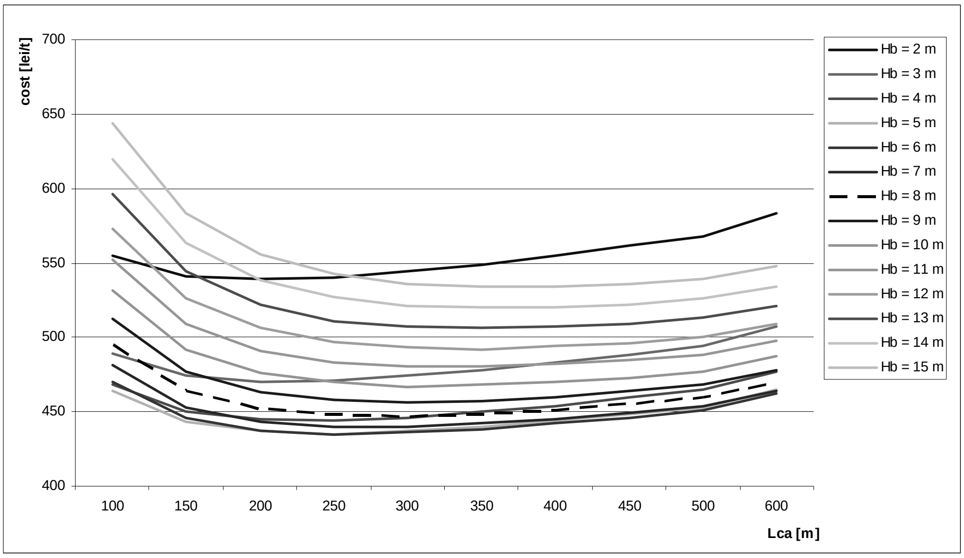

Fig.7. Variation of the unit cost, top coal height and panel length, for l_{ab}=40m and n=6workers/shift

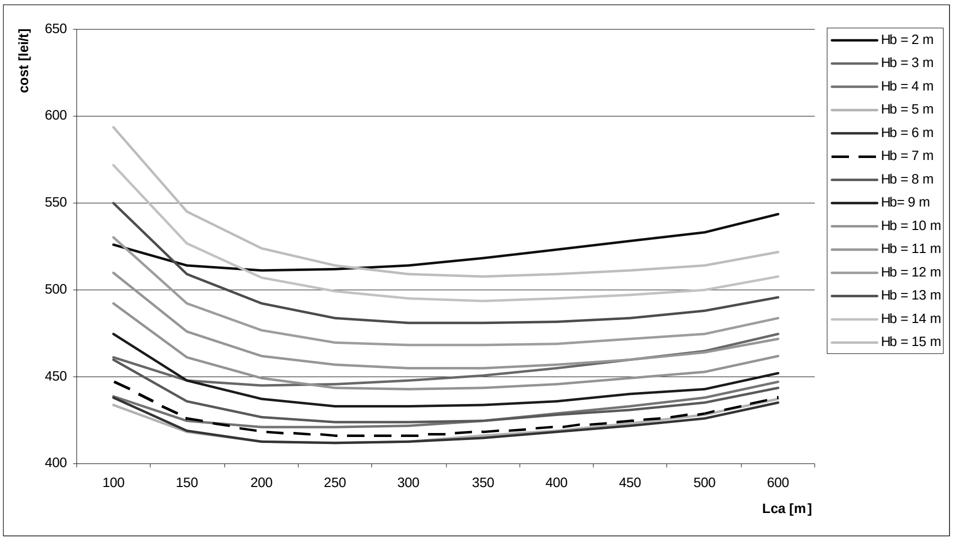

Fig.8. Variation of the unit cost, top coal height and panel length, for l_{ab}=40m and n=15workers/shift

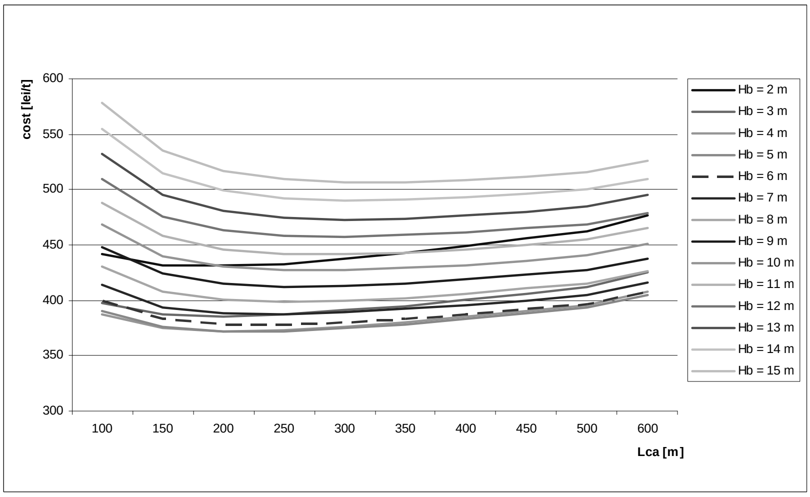

Fig.9. Variation of the unit cost, top coal height and panel length, for l_{ab}=78m and n=6workers/shift

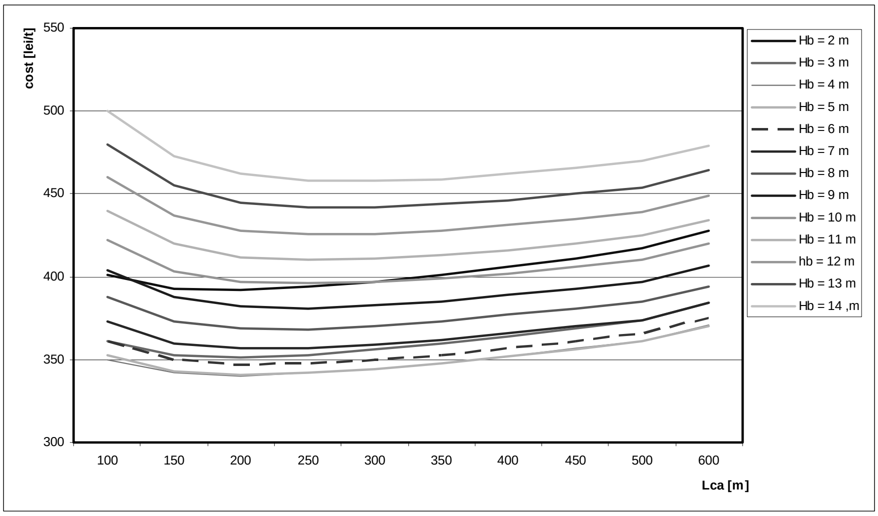

Fig.10. Variation of the unit cost, top coal height and panel length, for l_{ab}=78m and n=15workers/shift

From all these analyses, a series of conclusions is resulted that leads to the finding of the optimum panel length and optimum top coal height, corresponding to a minimum unit cost, for certain technical and geological mining conditions.

CONCLUSIONS

The unit costs obtained for the studied parameters are minimum for l_{ca}=250-300m and h_b=6-7m, and the calculated values for h_b>7m determine the unit costs being up to 40% higher by report to the optimum values, established for every studied case.

The influence of the panel length variation following the coal seam strike, on the total cost is less, being at a maximum of 10%, and the optimum being for l_{ca}=300m.

The increase of the coal face length, from a minimum value of 20m to a maximum value of 78m, has as effect a costs' reduction with about 65%, for 15workers/shift and about 58%, for 6workers/shift.

The minimum unit cost with the lowest value (c=341 lei/t) is provided in the case of l_{ca}=200m, h_b=5m and n=15workers/shift.

The placement in the coal face of a number of 6, respectively 15workers/shift, leads to a reduction of the production cost with about 20% (for 15workers/shift) – because of the high volume of manual operations, in the case of this classical mining technology- an efficient placement of the workers number, in the working face, is very important.

The technical parameters that ensure an efficient economical mining (a minimum unit cost) of the coal seam, for this mining method and technology [4], [8], [10], are l_{ca}=200-600m and h_b=4-9m.

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ASSESSMENT OF THE SAFETY FACTOR OF THE CROWN PILLAR BETWEEN OPEN PIT AND UNDERGROUND MINING IN THE CASE OF THE ROŞIA POIENI COPPER DEPOSIT

ONICA ILIE[*]
COZMA EUGEN[*]
ONEȚ DAN[**]

Abstract: *Roşia Poieni copper deposit is the most important mined copper deposit from Romania and it is mined in open pit. In order to continue the mining into underground, it is necessary to size and check the crown pillar stability between the open pit bottom and the superior limit of the underground mining. In this paper, the synthesis of the crown pillar stability with the aid of the finite element method is presented.*

Keywords: *crown pillar, underground mining, open pit mining, stability analysis, finite element, stress, strain, displacement, safety coefficient*

1. GENERALITIES

As the primary purpose of the crown pillar is to protect surface land users, the mine, and those working in it, from inflows of water, soil and rock, it is vital that such surface pillars remain stable throughout their life. Maintaining the stability of the crown is critical, not only to the success of the mine, but also to ensure the safety for any existing community or infrastructure above [5].

A number of approaches have been proposed over the years to assist in dimensioning crown pillars and addressing their stability. However, because of the significant differences that exist in the behaviour between identified failure mechanisms [4], [1] most of these approaches have been specifically addressed to one or other of the various characteristics failure behaviours [7], [9], [2]. Others have attempted to examine the resulting influence zone of the collapse process [13].

Deep studies concerning the influence factors of the crown pillars stability have been initiated and different approaches of structural analysis [1], [2], [4], [8] have been conceived. These studies had demonstrated that for any rock with given quality, the crown pillar stability depends, mainly, on their geometry. The span, thickness and specific weight of the crown pillar rock mass were found as the most critical parameters of influence [5].

[*] *Prof., Min. Eng. Ph.D. - University of Petroşani*
[**] *Ph.D. Student - University of Petroşani*

At the simplest, the “Scaled Span method” can be deterministically applied by comparing the scaled crown pillar span for any pillar of concern to the critical span value deemed appropriate for the controlling rock mass. When the scaled crown pillar span is determined to be less than the critical span, the crown pillar would be considered stable. If, on the other hand, the scaled span was calculated as greater then the critical span, unless the crown had been used in mining, likelihood of failure would be high [5].

Applying this approach in the case of the Roşia Poieni copper deposit (Fig.1), it is simplest to observe that the value of the safety factor is 16 times less than the value at the balance limit. What it means is that it is impossible to apply an underground mining system where entire deposit surface is excavated on 550x550=302 500m^2 and of 50m crown pillar thickness.

Fig.1. Open pit mining of the Roşia Poieni copper deposit

If the crown pillar thickness is recalculated to have a limit stability, namely for *CS*=1, in the conditions of the scaled span approach, we realise that the necessary crown pillar thickness arises to the order of thousands of meters , which is completely absurd from a technical point of view.

In *conclusion*, the adopted mining system where the crown pillar has a 50m thickness is supported on pillars of 50m width, separated from the large rooms with a 50m span, is a system that could be proved to be efficient from a stability point of view.

The establishment of an exact factor of safety or a safety limit is important because the failure of the crown pillars can appear in a very large variety, from the rock desegregation and various types of collapses, to the sliding of the intact rock blocks and failure of the caving chimneys [3].

Thus, several control mechanisms of rocks failure exist. The failure in the pure shear is rare, from the registered data, but it is easier to be analysed, which led to the analytical methods development [9].

Applied probabilistically, this approach can also better help in defining acceptable or allowable risk for a given situation as per Table no.1 [5].

Table no.1. *Comparative significance of crown pillar failure [5], [6]*

Class	**Prob. of failure [%]**	**Minimum factor of safety**	**Excavation support ratio**	**Design criteria for acceptable probability of failure**				
				Serviceable life	**Years**	**Public access**	**Regulatory position on closure**	**Operating Surveillance Required**
A	50-100	<1	>5	Effectively zero	<0,5	Forbidden	Totally unacceptable	Ineffective
B	20-50	1,0	3	Very, very short term	1,0	Forcibly prevented	Not acceptable	Continuous sophisticated monitoring
C	10-20	1,2	1,6	Very short term	2-5	Actively prevented	High level of concern	Continuous monitoring with instruments
D	5-10	1,5	1,4	Short term	5-10	Prevented	Moderate level of concern	Continuous simple monitoring
E	1,5-5	1,8	1,3	Medium term	15-20	Discouraged	Low to moderate level of concern	Conscious superficial monitoring
F	0,5-1,5	2,0	1,0	Long term	50-100	Allowed	Of limited of concern	Incidental superficial monitoring
G	<0,5	>>2	0,8	Very long term	>100	Free	Of no concern	No monitoring required

2. FINITE ELEMENT ANALYSIS OF THE STRESS AND STRAIN STATE OF THE CROWN PILLAR

2.1. Models description

In view to assess the stability of the crown pillar, the computational models with finite elements in the plane strain were used, in the elastic, isotropic and linear behaviour of the rocks, into a vertical cross-section NW-SE, by ore deposit, perpendicular on the long and quadrangular rooms (in conformity with the mining method). Also, for the elaboration of the 2D models, with the aid of the CESAR-LCPC 2D software [11], [12], simplifications were made by the following suppositions:

-The finite element models were achieved in conformity with the vertical cross-sections from the mining project, at the maximum extension of the open pit mining (the final open pit bottom at the level of +790m) and underground, correspondently with the successive mining of the three levels, namely I level, at +690m, II level, at +620m and III level, at +550m (considered to be critical from the point of view of underground mining), and which could be generated in the plane strain hypothesis;

-The sizes of the rooms (50x50m), pillars (50x50m) and the ceilings between levels (20m) are supposed to be equal on the entire level;

-The crown pillar, at the open pit bottom, has a thickness of 50m (along the width of 183m), increasing progressively until a maximum of 410m (on the width of 367m), toward SE, due to the general slope of the open pit; in the inferior cross-section (at the rooms ceiling at I level, +740m), the crown pillar span is 550m;

-At all mining levels, it is considered that pillars are perfectly coaxial, along the vertical line;

-Both the surrounding rocks and the copper ore are supposed to be continuous, homogenous and isotropic, and the geomechanical characteristics used in the calculus are average ones, representative for the entire rock massive, respectively for the copper ore;

-In situ behaviour of the rocks and ore is considered to be elastic, linear and isotropic;

-Because the study object was not the slope stability analysis and because the scale report between model and the slope sizes, in the model, the open pit was represented only by the general slope planes;

-The ore drawing funnels, because of their high volume, and to simplify the model, it was considered that they belong to the mining rooms;

-The ceilings between levels were divided into a superior zone, affected by the preparatory workings, of about 10m of thickness (related to the secondary crashing horizon, transport horizon and the chutes) and characterized by certain equivalent geomechanical characteristics and an inferior zone, intact, at the roof of the rooms, with the same thickness of 10m;

-The natural stresses state is considered to be geostatical.

The general model (see Fig.2) was generated in three different mining stages, in which the ore deposit was entirely mined in open pit (in conformity with the project), and underground, where the rooms at I level are successively mined, +690m, then II level, +620m and finally the rooms at III level, +550m. Also, because the real measured values of the in situ stresses do not exist, there were generated, for every model, four different situations of geostatic model loading, with the thrust coefficients k=0.2, k=0.5, k=0.75 and k=1.

The achievement of the 2D modelling, in the plane strain hypothesis, for every previous defined model, require the following stages: I) establishment of boundaries, interest zones and meshing of the model; II) determination of zones (regions) and computational hypothesis and the geo-mechanical characteristics input; III) boundaries conditions establishment; IV) initial conditions and loading conditions establishment; V) achievement of calculus and storing of results [10], [11], [12].

Taking into account the mined space sizes, for more precision of the calculus, the models were made with the sizes of X=2 898m/Y=930m (see Fig.2.). The meshing of the model, respectively of every surface region, was made by triangular or quadrangular surface finite elements, with quadratic interpolation (the total number of nodes is 43 851; the total number of surface elements is 19 654).

The rocks and ore characteristics were considered homogenous and isotropic. Thus, the average characteristics were adopted, considered being representative for the in situ behaviour of the rocks, ore massive and the ceiling affected by the preparatory workings (Table no.2).

Table no.2. *Average geomechanical characteristics taken into consideration in finite element models*

Characteristic	**UM**	**Value**		
		Copper ore	***Rocks***	***Equivalent ceiling****
Apparent density, ρ_a	kg/m^3	2 530	2460	2 000
Apparent specific weight, γ_a	kN/m^3	25,3	24,6	20
Elasticity modulus, E	kN/m^2	9 000 000	8 000 000	6 000 000
Poisson ratio, ν		0,12	0,2	0,18
Compressive strength, σ_c	kN/m^2	54 000	48 600	-
Tensile strength, σ_t	kN/m^2	5 800	5 200	-
Shear strength, τ_f	kN/m^2	13 200	12 000	-
Cohesion, C	kN/m^2	9 000	8 000	-
Internal friction angle, φ	,^{o}	30	27	-

***Observation**: the geomechanical characteristics values of the ceiling level affected by the preparatory mining workings (equivalent ceiling) were assessed taking into account the excavations

The initial conditions of the model loading were considered geostatic $[\sigma_o]$, concordant with the variable value of the overburden strata (until the inferior surface of the crown pillar), ranging between about 440m and 50m – after the underground mining and between 440m and 190m – without the open pit mining (also, for to calculate the residual stresses).

The calculus was made taking 60 iterations per increment and a tolerance of 1% of the results, using the "initial stress method" for the resolution. The calculus results were stored in graphical form on the model surface (isovalue, vector and tensor representation) and in the predefined sections – following the superior plane (+790m level) and the inferior (+740m level) of the crown pillar. The results, presented in the following, are processed by „Microsoft Office Excel".

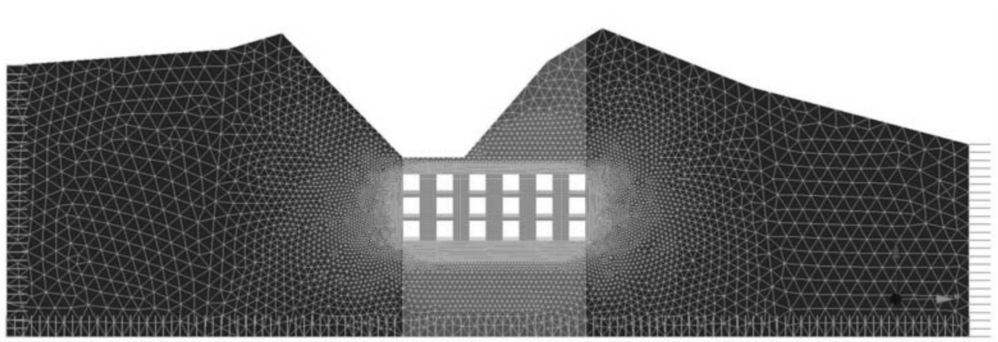


Fig.2. The finite element meshing of the mining models of the Roşia Poieni copper deposit

2.2. Analysis of the results obtained by numerical modelling

Mainly, in order to assess the crown pillar stability, the stress – strain state development was taken into consideration following the superior and inferior cross-section of the crown pillar, as Figure 3.

2.2.1. Stress – strain state analysis

2.2.1.1. Displacements analysis

Because of the variable configuration of the ground surface (generated by the open pit presence), the orientation of the total displacements vector is from NW and SE, towards the open pit centre. In this way, for k=0.2 and 3 mined levels, the maximum value of the total displacement is under 0.3m. These are the results of certain horizontal displacements of 0.09m and vertical displacements of under 0.3m.

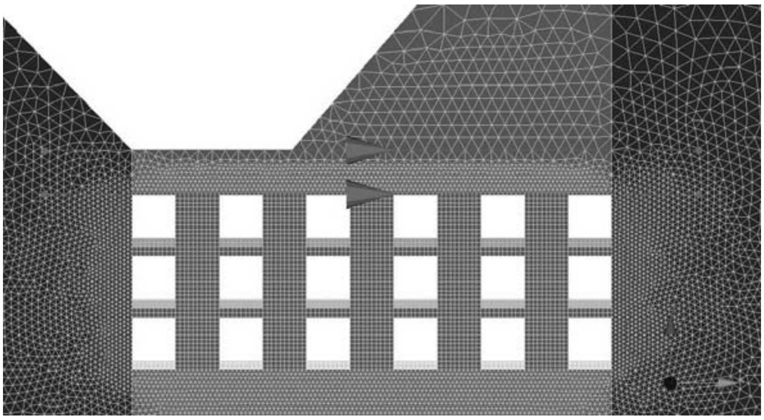


Fig.3. Horizontal sections at the “superior” and “inferior” crown pillar limits

As it can be seen, from the obtained results analysis, the horizontal displacements developed in the crown pillar are in SE towards NW direction, determined by the ground surface configuration related to the underground voids. Also, its vertical displacements under the ground loadings are in totality from up to down.

Regarding the crown pillar, the vertical displacement v is attenuated as the thrust loading increases. In return, the horizontal displacement u increases significantly with thrust loadings.

Obviously, the vertical and horizontal displacements are growing along with the depth advancement of the mining levels.

It is mentioned that the vertical displacements are more accentuated in the crown pillar, in the open pit bottom zone. In this area, the horizontal displacements are the lowest. If in the superior cross-section of the crown pillar the vertical displacements are continuous, in the inferior section, these vary in function of pillar presence, where they are the most significantly reduced.

2.2.1.2. Stresses analysis

The compressive stresses developed in the crown pillar, in the superior section, are over 8-9 times more reduced than the compressive strength of the massive (54 000kN/m^2). Generally, in the inferior section of the crown pillar, the compressive stresses are more important, locally could arise a safety factor at the compression failure of a minimum of 1.8.

Generally, the compressive stresses have the values far below the safety limit, and locally, in certain corners between the pillars and the crown pillar of the level ceilings, it is possible to reach a limit factor of safety, which causes us to affirm that in these zones, a compressive failure phenomenon could appear (probably associated with other type of failure phenomena).

Tensile stresses. If in the upper part of the crown pillar an important reserve of the safety factor exists, of over 500%, due to the fact that, without the stresses from the open pit bottom, which are in the biaxial stresses state, the rest of the crown pillar is in a triaxial state of stresses; in return, in the lower section of the crown pillar certain zones exist where the tensile safety factor is insignificant, such as: the ceiling of the rooms 4, 5 and 6, from I level and the floor of the rooms from III level. The areas with the highest probability of tensile failure are at the level of rooms 2 and 3, I level, NW corner between pillar and the ceiling – floor level.

The shear stresses are the most important stresses from the point of view of the study of the rock massive stability, besides the tensile stresses. In the superior section of the crown pillar, the shear stresses are 3-4 times more reduced by report to the massive strength (13 200 kN/m^2). In the inferior section, they are about 2-3 times lower. This leads us to the conclusion that an obvious risk of arising failure phenomena by shear into the crown pillar does not exist.

If the shear stresses distribution in the rest of the model is studied, it is observed that these can exceed the shear strength limit, especially in the SE corners, of intersections between pillars and the floor, at rooms 4, 5 and 6, located at the III level.

2.2.2. Crown pillar stability analysis after the safety coefficients calculated with the Mohr-Coulomb criterion

A failure criterion will be introduced considering the intrinsic curve of the ore. For any point characterised by a certain state of the stresses, the correspondent Mohr circle is determined and it is reported to the intrinsic curve of the ore.

In our case, to appreciate the crown pillar stability and implicitly for the safety coefficient (factor) *CS* calculus (when: *C*=9 000kN/m^2 and $\varphi = 30^o$) we have the relation:

$$CS = \frac{1}{(\sigma_1 - \sigma_2)} \cdot [15\,588.46 - 0.5 \cdot (\sigma_1 + \sigma_2)] \tag{1}$$

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.

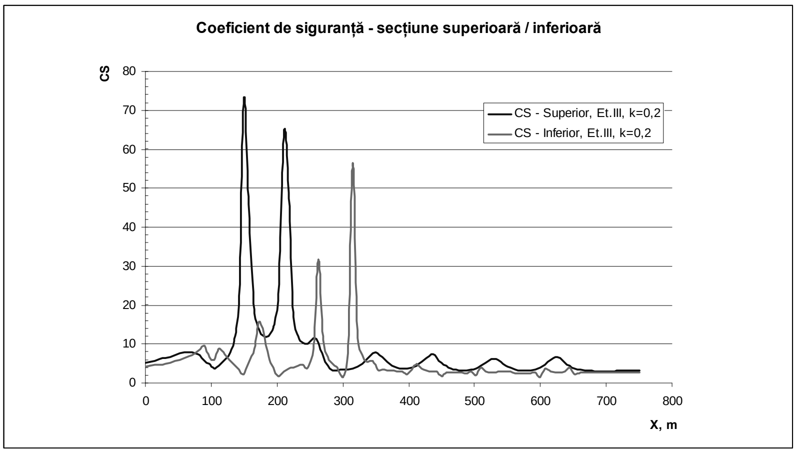
If the safety coefficients (Fig.4.a) are studied, for the three mined levels and *k*=0.2, it is observed that, for both the upper and lower surface of the crown pillar, the failure limit is not reached. But, the lower values of the safety coefficient (*CS*<2) are at the NW corner, room 2 (*CS*=1.8), room 3 (*CS*=1.4) and at the SE level, room 4 (*CS*=1.8) and the room 6 (*CS*=1.4). In these areas, the risk of the local phenomenon of the ore detachment from the rooms ceiling is possible to appear.

With the increase of the thrust, for *k*=0.5 (Fig.4.b), the phenomenon of the crown pillar stability decreasing advances, and the safety coefficients decrease appropriately under the value of 2 in the other upper corners of the rooms situated at the I level, at the level of the crown pillar.

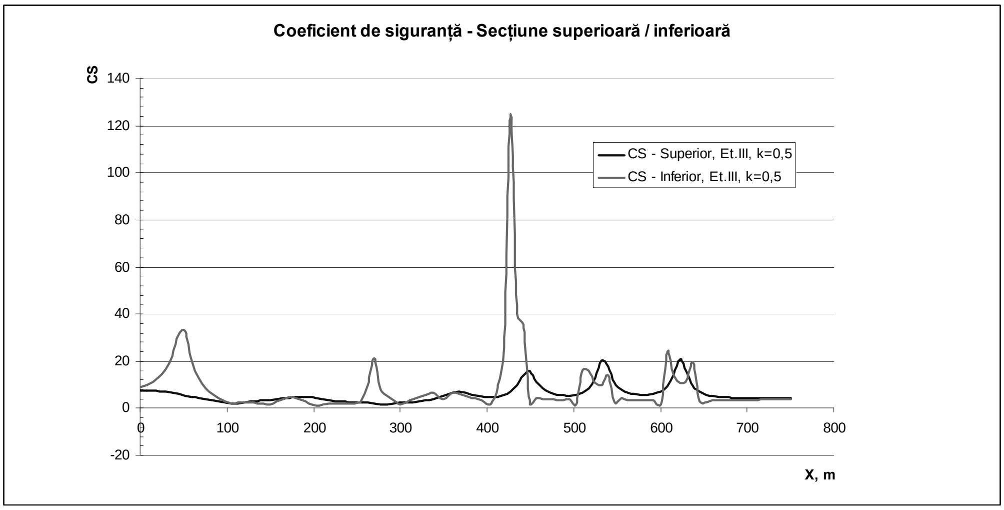
For k=0.75 (Fig.4.c), the reduction phenomenon of the stability is extended to the crown pillar at the entire ceiling of the rooms, and the stability decrease is installed in the upper section of the crown pillar (CS<2), especially in the open pit bottom area, respectively over rooms 1 and 2.

For k=1.0 (Fig.4.d), at the superior level of the crown pillar all the area under the rooms 1, 2 and 3 is affected, at level I. Also, there is a substantial decrease of the safety coefficients at the level of room 2, NW corner (CS=0.924<1), which highlights the arising collapse in that area.

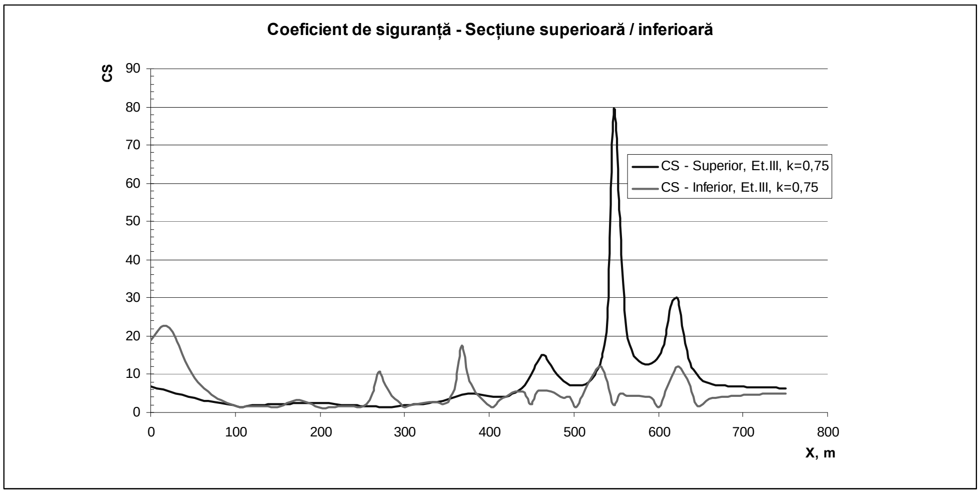
From the analysis of data presented in Fig.4, it is observed that there are no spectacular changes of the crown pillar stability with the levels mining in depth.



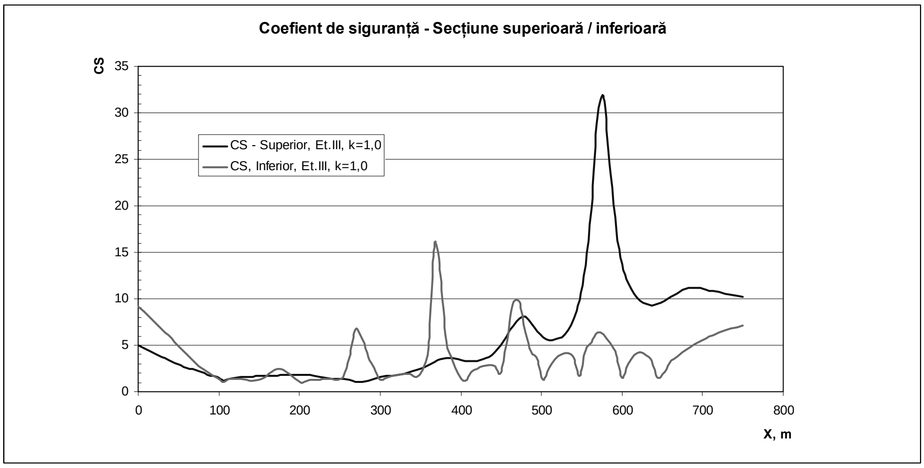
a)



b)

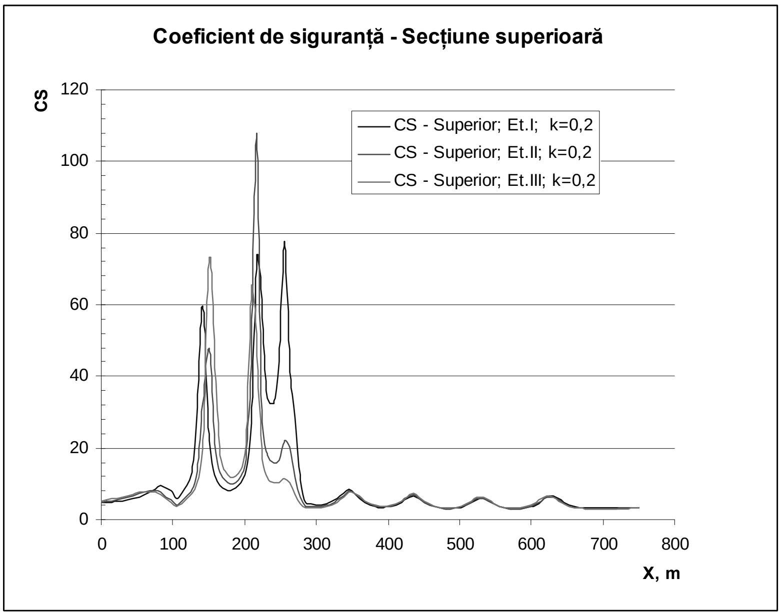


c)

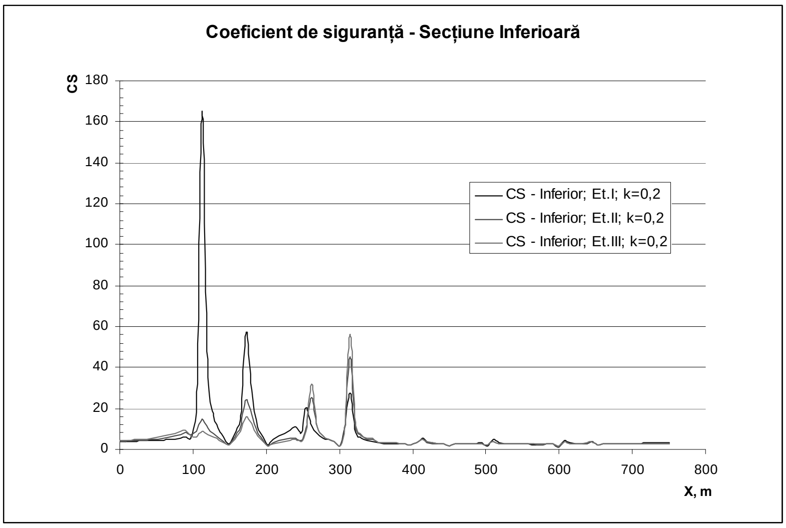


d)

Fig.4. Safety coefficients distribution along the superior surface of the crown pillar, for the followings thrust coefficients: a)*k*=0.2; b) *k*=0.5; c) *k*=0.75; d) *k*=1.0



a)



b)

Fig.5. Safety coefficients variation in according to the mining depth advancement

2.2.3. Stress-strain state variation of the model with the depth advancement of the ore deposit mining and the sensibility analysis of the model results at the thrust coefficients values variation

The analysis was limited at the parameters distribution at the lower section of the crown pillar, considered to be the most representative for the stability analysis.

Such as, from Figure 6 it is observed that the *horizontal displacements* increase by about 2.5 times with the mining passage from the first level to the integral mining of the three levels (Fig.6). Regarding the thrust influence, the horizontal displacements increase with about 50-60% for every 0.2-0.3 increase of the safety coefficient.

In return, the *vertical displacements* are less sensitive to the thrust variation (Fig.7), being more pregnant in the vicinity of the open pit bottom. But, the mining depth increase has a significant impact on the crown pillar's vertical displacements; less in the open pit bottom area and more pronounced towards its SE flank (Fig.7).

The *shear stresses* are influenced both by thrust and by level mining in depth (Fig.8). The shear stresses developed in the crown pillar, lower section, are essentially influenced by *k*, added at the open pit bottom level 1 000kN/m^2 for every 0.2-0.3 increase of k. In the crown pillar rest, towards the SE area, the shear stresses increment is only 200-300KN/m^2.

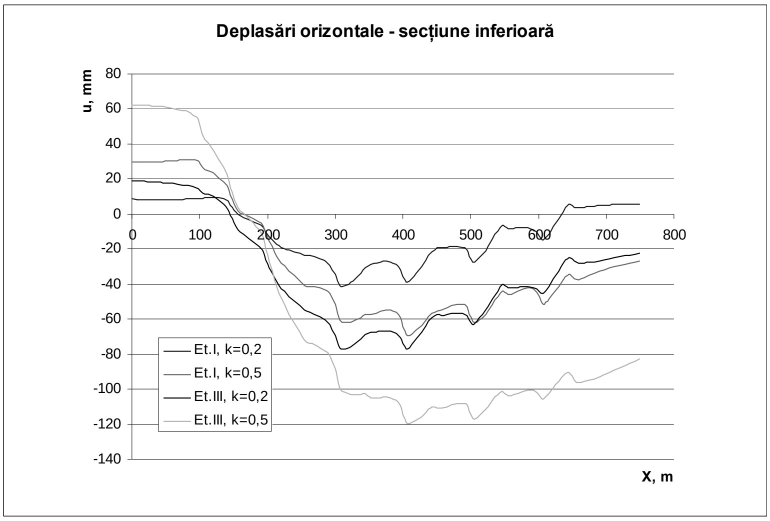


Fig.6. Horizontal displacements variation in the crown pillar, upper section, in function of the levels mining in depth and the thrust coefficient

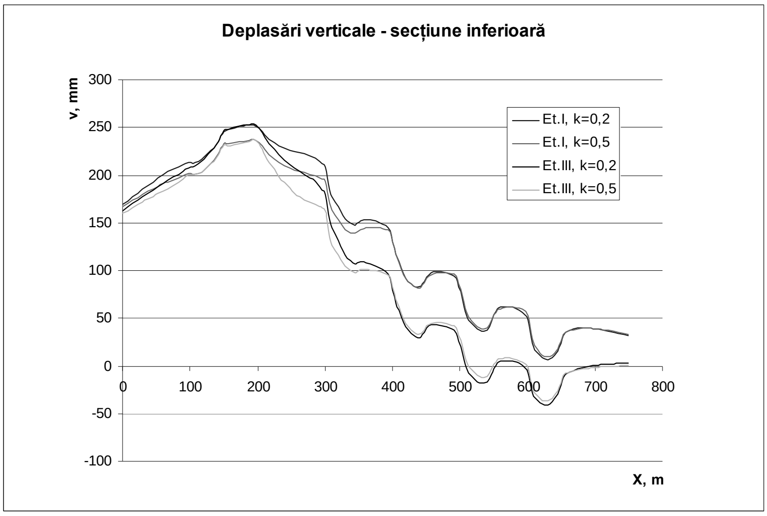


Fig.7. Vertical displacements in the crown pillar, lower section, in function of the levels mining, in depth, and the thrust coefficient

With the mining advancement in depth, with every level, the increase of the shear stresses in the crown pillar – upper section, at the open pit bottom- is with about 30-40%.

Tensile stresses are the most influenced by thrust, observing the important increase with every increase of *k* (Fig.9). If the tensile stresses growth is under 1000kN/m^2 for every new mined level, in the case of *k* it is under 3 000kN/m^2 for every increase of *k* with a value of 0.2-0.3.

In Fig.10 and 11 it could be observed that the *maximum and minimum principal stresses* are influenced, essentially, both by the thrust value and by the mining level descent.

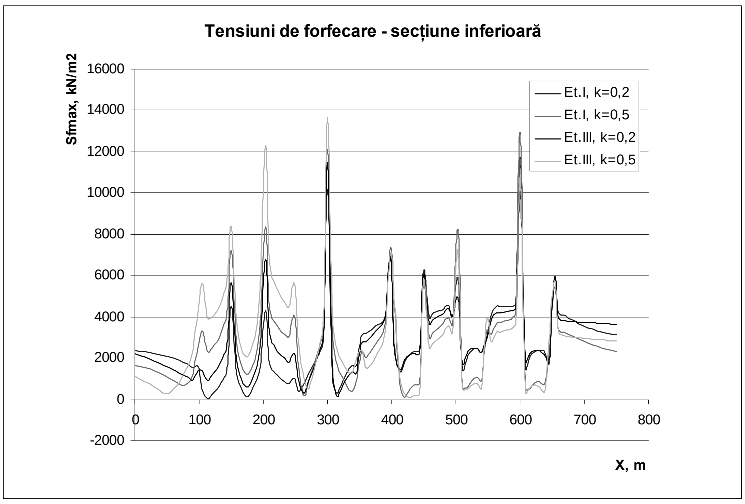


Fig.8. Maximum shear stresses variation in the crown pillar, lower section, in function of the level mining in depth and the thrust coefficient

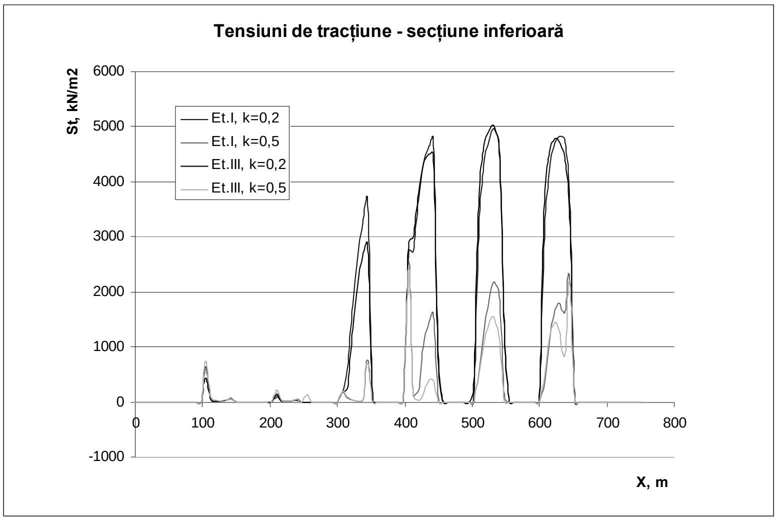


Fig.9. Tensile stresses variation in the crown pillar, lower section, in function of the levels mining in depth and the thrust coefficient

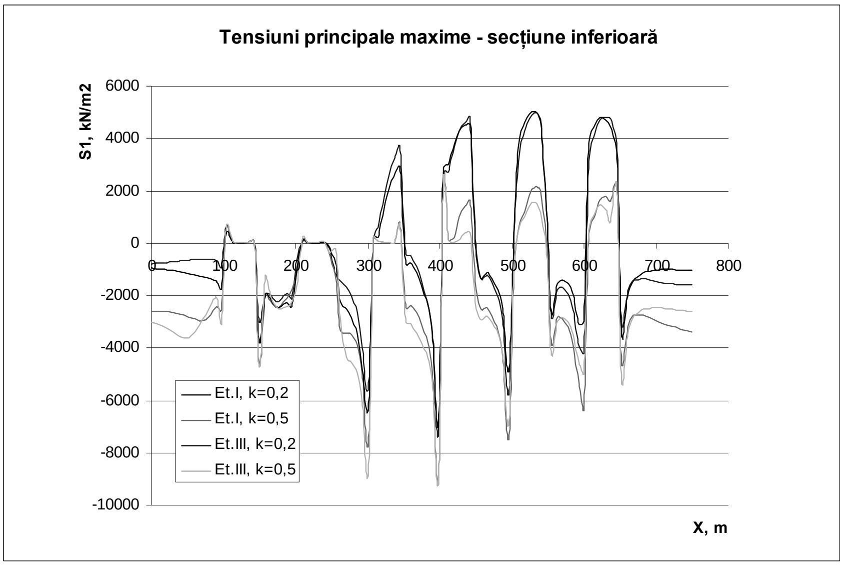


Fig.10. Maximum principal stresses variation in the crown pillar, lower section, in function of the levels mining in depth and the thrust coefficient

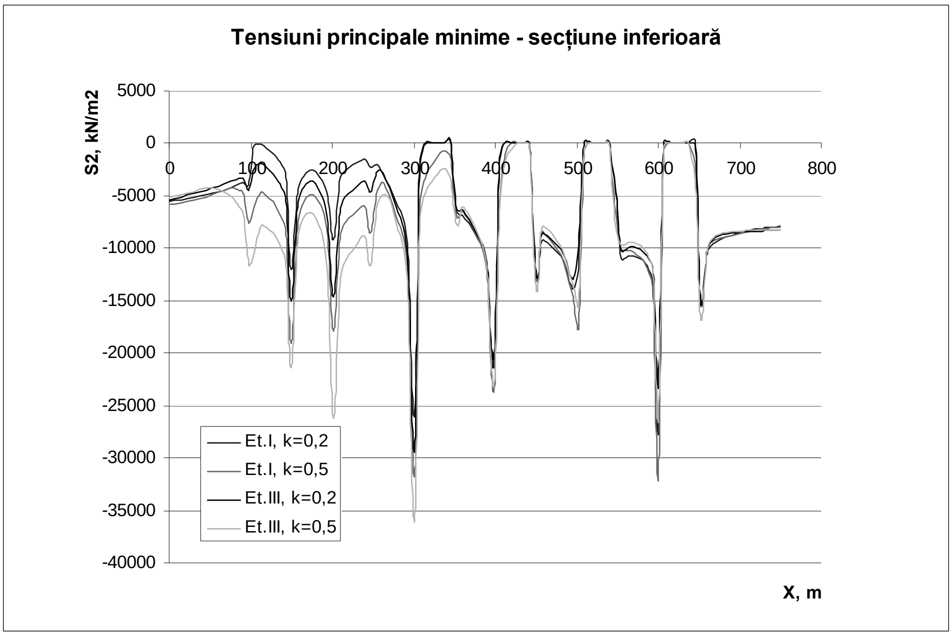


Fig.11. Minimum principal stresses variation in the crown pillar, lower section, in function of the levels mining in depth and the thrust coefficient

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STABILITY ANALYSIS OF THE SETTLING UNDERGROUND RESERVOIR - HYDROELECTRIC ARRANGEMENT OF THE JIU RIVER

ONICA ILIE[*]
COZMA EUGEN[*]
RUSU ADRIAN[**]
MIHĂILESCU VIOREL[***]

Abstract: *The settling underground reservoir Livezeni belonging to the hydroelectric arrangement of the Jiu River, is situated into crystalline rocks and is supported with reinforced concrete and bolts. This paper involves the stability analysis of the settling reservoir Livezeni with aid of the finite element method. The modelling is in 2D, respectively in the plane deformation hypothesis, taking into account the Mohr-Coulomb elasto-plastic behaviour without hardening, for the rocks and elastic behaviour, for the support elements.*

Keywords: *settling underground reservoir, hydroelectric arrangement, stability analysis, finite element, elasto-plastic behaviour, stresses, strains, safety coefficient*

1. GENERALITIES

The hydroelectric arrangement of the Jiu River on the Livezeni sector (Fig.1) and the Sadului Valley involves two powerhouses, on the by-pass, emplaced in the gorge area. The first powerhouse of the arrangement scheme involves as main objective the Livezeni – Dumitra principal head race, which ensures the transit of installed flow of $36m^3/s$, between the bell-mouth intake and Dumitra pressure junction, having a length of 7km.

The by-pass is composed by the following:

-the bell-mouth intake, gate house and the connecting section with the settling underground reservoir;

-the settling underground reservoir (with a length of 130m) that is connected with the principal head race under pressure Livezeni-Dumitra;

-the gallery with circular section (with internal diameter of 3,8m and the length of about 7km); the head race has a permanent reinforced concrete lining with the thickness of 0,3m; the execution of the head race Livezeni-Dumitra is starting after the adits, which are: Livezeni, Murga Mică and the one from the Dumitra pressure junction.

[*] *Prof., Min. Eng. Ph.D. - University of Petroşani*
[**] *Eng. at Ardeal Hydroconstruction Company Cluj-Napoca*
[***] *Ph.D. Student - University of Petroşani*

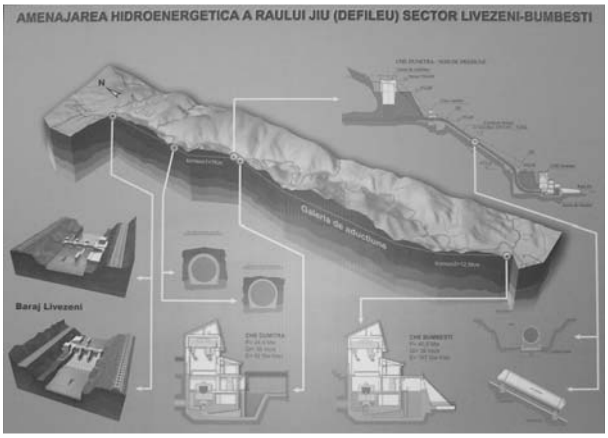


Fig.1. Hydroelectric arrangement of the Jiu River (gorge) – Bumbeşti-Livezeni sector [6]

The construction category is C, the importance class of the construction is III (in conformity with STAS 4273/1987). The area of the working, in conformity with the normative P100 -96, is in according with the E zone of seismicity, having a seismic intensity coefficient K_s=0.12 and a corner period T_c=1.0 [6], [1].

The length of the settling underground reservoir (Fig.2), together with the connection to the bell-mouth intake, measures 175.75m. The gallery sizes (with the width *b* and the height *h*) are the following: for the settling underground reservoir: b=9.20 m; h = 12.30 – 13.40 m; for the connection: b = 4.40 m; h = 4.80 m.

Regarding the morphological point of view, the settling underground reservoir is located on the right bank of the Jiu Valley. The versant is steep, with the slopes ranging between 35^o and 47^o.

The rocks overburdening in the settling underground reservoir area ranges between 30m and 75m and in the connection area, between 5m and 30m.

The settling reservoir is an underground construction, the sizes of which are the following: the settling length itself of 130.0m, the digging width of 9.2m and the height ranging between 12.49m and 13.4m. The slope of the settling reservoir invert is about 0.7%. The settling reservoir design was made for to particle retention with the size of 0.5mm, in the proportion of 95%.

Fig.2. Settling underground reservoir Livezeni

The diggings at the settling underground reservoir reveal a type III rock, for the ante-measurements of the settling reservoir itself with the length of 130m and for the connecting area with the Livezeni adit (with the length of 9.39m).

For the Livezeni second stage adit – diggings (on the depth of 2.4m, in the invert area) and the diggings for the connecting gallery "bell-mouth intake – settling reservoir" are achieved into several rock types (from the type I at the type IV).

The adopted technology, for to excavate the settling underground reservoir, had at base the beginning of the workings execution starting with the existing ones in the working territory, namely the Livezeni adit. An operational technology which starts from the supplementary access (at the bell-mouth intake), into an area like the working territory of the Livezeni sluice, with a great density of the workings on the reduced surface, complicates troubleshooting even more.

2. GEOLOGICAL AND GEO-MECHANICAL CHARACTERIZATION OF THE ROCKS MASSIVE

From a geological point of view, in the area of the settling underground reservoir there are the crystalline rocks that belong to the Danubian Domain, respectively at the Drăgşan series, of the Upper Proterozoic – Paleozoic age. These crystalline formations are identified on about 150m, along the gallery, and involve quartz and chlorite- sericite schist, with granoblastic structure and schistose texture.

Structurally, the entire crystalline complex is intensively tectonized, showing multiple fissure planes, accompanied by intensively altered zones. The rock schistosity is oriented NE-SW, with falls of 20^{o}-40^{o} towards NW (upstream- right slope).

Taking into consideration the petrographic nature of the rocks, their degree of tectonization and alteration, the following rocks categories and their percentage distribution are appreciated for the entire section of the gallery (175.5m), in conformity with the Barton classification (Q system) and Bieniawski (RMR) – Table no.1 - [2]:

Table no. 1. *Percentage distribution of the rocks categories after Barton and Bieniawski classification [1], [2]*

Categoria rocii Rock category	*Barton (Q)*	*Bieniawski (RMR)*	*Ponderea,[%] Share, [%]*
I	>20	>60	0
II	20-10	60-40	16
III	10-1	40-20	60
IV	1-0,05	20-10	14
V	<0,05	<10	10

In conformity with the geological and geotechnical data, related to the settling underground reservoir route, the rocks types founded at the settling reservoir digging are the following [1], [6]:

-*Rocks of category II (rocks of type B)*: are constituted from fissured granites, weakly altered (fissures frequency is of 3-5.5 fissures/m), with frequently altered zones. The values of the geotechnical parameters assessed for this rocks category are: E=4000-5000MPa; K_0=400-500; tg$\varphi(r/r)$ = 0.55; tg$\varphi(b/r)$ = 0.50; ρ =2500kg/m^3, C=0.2-0.25MPa; hardness TS = very hard.

Rocks of category III (rocks of type C): are constituted from much fissured granites, with frequented altered zones, which are founded on the ground surface, in the contact zones with other types of rocks, in the main fault zones, accompanied by fissures connected on the maximal planes S_1. The values of the geotechnical parameters assessed for this rocks category are: E=2000-4000MPa; K_0=200-400; tg$\varphi(r/r)$ = 0.45; tg$\varphi(b/r)$=0,40; ρ =2300kg/m^3; C=0.1-0.15MPa; hardness TS = T – ST (hard – semihard).

Rocks of category IV (rocks of type D): are constituted from much fissured granites, altered, with kaolinization and with breccias zones. The values of the geotechnical parameters assessed for this rocks category are: E=2000-1000MPa; K_0=200-100; tg$\varphi(r/r)$=0,45-0,40; tg$\varphi(b/r)$=0,40-0,30; ρ =2200kg/m^3; C=0,06-0,1MPa; hardness TS = M (soft).

Rocks of category V (rocks of type D_S): The values of the geotechnical parameters assessed for this rocks category are: E<1000MPa; K_0< 100; tg$\varphi(r/r)$=0,30-0,35; tg$\varphi(b/r)$=0,30; ρ =2200kg/m^3; C=0,03-0,05MPa; hardness TS = FM – M (very soft - soft).

The signification of the previously used symbols are the following: E- linear elasticity modulus; K_o- elastic strength; tg$\varphi(r/r)$- frictional coefficient "rock on rock"; tg$\varphi(b/r)$- frictional coefficient "rock on concrete"; ρ - apparent specific density of the rock; C- rock cohesion.

3. FINITE ELEMENT ANALYSIS OF THE STRESS-STRAIN STATE AROUND THE SETTLING UNDERGROUND RESERVOIR LIVEZENI

3.1. Description of the finite element models

Considering the geometry and the location conditions of the settling underground reservoir Livezeni, in view of modelling with finite element of this underground mining working, the plane strain hypothesis [3], [4], [5], [7] was taken into consideration.

The CESAR-LCPC program [10] was used in this paper for to achieve the model of the settling underground reservoir Livezeni.

For this end, the following simplifying assumptions were made:

-For that model to be analysed in the plane strain hypothesis, in spite of the fact that the real length of the settling is of about 130m, it was considered that the settling length is infinite;

-The rock massive, where the Livezeni settling was achieved, is supposed to be continuous, homogeneous and isotropic, with an elasto – plastic behaviour, without hardening, by Mohr – Coulomb type, which allows a symmetrical representation of the model following the "0y" axis.

-The Livezeni settling is modelled in the downstream zone, where the settling excavation height is the largest, of about 13.4m (between the levels +554.6m and +541.19m;

-Also, although the biggest section of the underground settling is situated nearby the rock massive front, which takes a part of the excavation support pressure, it was considered that this section is without this influence;

-In the walls and floor zone, the loading support thickness (of 0.51m) was added at the shotcrete thickness (of 0.09m), resulting a total thickness of 0.6m;

-The loading support of the underground settling, although made of reinforced concreted and the shotcrete, is considered to be elastic, homogenous and isotropic, with weighted average characteristics of the materials, in function of the compound materials volume;

-Although the overburden rocks height of the Livezeni settling is variable, ranging between about 30m and 80m, the most unfavourable value for the settling support stability was taken into consideration, namely of *H*=80m;

-Even though the construction of the underground settling support was made in successive phases, the support stability after integral support achievement was taken into modelling.

Two basic models were taken into study, namely [6]:

A) The model of the Livezeni settling finalized;

B) The model of the Livezeni settling excavation, only with the bolting support installed.

The second model (B) was taken into consideration for visualising, in comparison to the results obtained from the A model, the contribution which has the bolting system on the surrounding rock balance, after excavation achievement.

The achievement of the 2D modelling, in the plane strain hypothesis, for every previous defined model, requires the following stages: I) establishment of boundaries, interest zones and meshing of the model; II) determination of zones (regions), computational hypothesis and the geo-mechanical characteristics input; III) boundaries conditions establishment; IV) initial conditions and loading conditions establishment; V) achievement of calculus and storing of the results [3].

3.1.1 Establishment of boundaries, interest zones and meshing of the model

Taking into account the excavated space sizes, for more precision of the calculus, the models were made with the sizes of X=40m / Y=148,33m (Fig. 3.). Also, the sizes of the interest zone, around the excavations, were established so as to involve the model surface where the stresses and strains variation is maximum. 2D model meshing, respectively of every surface region, was made by triangular surface finite elements with quadratic interpolation. The total number of nodes is of 6113 and elements of 3035, surface elements of 2996 and linear elements of 39.

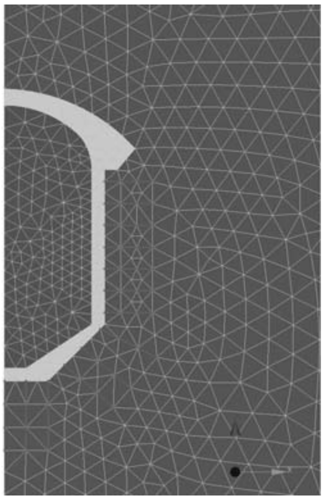


Fig.3. Finite element modelling of the underground settling Livezeni

3.1.2 Determination of zones (regions), computational hypothesis and the geo-mechanical characteristics input

For to symplify the 2D models, two regions were taken with different characteristics (rocks and reinforced concrete) and the bolts as linear elements from the steel bar type. The behaviour of the rocks was considered as having Mohr–Coulomb elasto-plasticity without hardening behaviour and both the loading concrete support and the steel bolts, having elastic behaviour.

3.1.2.1 Geo-mechanical characteristics of the rocks

The rocks' characteristics (mainly, the fissured granite with different degrees of alteration) were presented in the first part of this paper, and taken into calculus in the Mohr – Coulomb elasto-plasticity without hardening behaviour. Thus, some average values of the characteristics were taken, considered representative for the in situ behaviour of the rocks, such as: apparent specific density $\rho_{ar} = 2200$ kg/m^3; linear elasticity modulus E_r=1500000kN/m^2; Poisson ratio $\nu_r = 0.2$; failure compressive strength σ_{rc} = 10000kN/m^2; failure tensile strength σ_{rt} = 1250kN/m^2; cohesion C_r=70kN/m^2; internal friction angle $\varphi_r = 22^o$.

3.1.2.2 Characteristics of the concrete

a) Elastic characteristics

In the calculus we will adopt the concrete elasticity modulus of about E_b=2650000kN/m^2, with a safe reserve of 12%. This leads to the weighted value of the elasticity modulus of reinforced concrete of about 30000000kN/m^2.

The Poisson ratio, for the concrete mark B200 is adopted as ν_b = 0.2 (being very close to the reinforced concrete value).

b) Mechanical characteristics

For the B200 concrete mark with the class III of homogeneity degree, the compressive strength of the concrete is considered R_{ac}=16000 kN/m^2 and tensile strength is taken as R_t=1200kN/m^2 (from the splitting test) and R_t=1800kN/m^2 (from the bending test) [8], [9]. From there, it is possible to deduce (for the minimum values of the concrete strengths) the cohesion C=2200kN/m^2 and the internal friction angle $\varphi = 55^o$.

The reinforced concrete lining thickness is variable, with constant values only at the walls and the floor, of about 0.6m (from which 0.09m is the shotcrete), and at the arching apex of 0.75m, progressively increasing at the springing of arch.

3.1.2.3 Bolts' characteristics

In point of view of elasticity characteristics of the concrete steel PC52 with the diameter of $\varphi = 20$ mm used for operating the cemented bolts, the following were adopted in the calculus: E_a= 210000000kN/m^2 and ν_a = 0.25. In the model, in the axial-symmetry hypothesis, the bolts were taken with a cross section of about 0.00032m^2/m and the length of 2.2m for the walls and 3.0m for the floor. In the stability analysis of the bolts, the ultimate strength of the steel is $\sigma_{c\,\min} = 340$ N/mm^2.

3.1.3. Boundary 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 a maximum underground excavation depth (overburden strata) of about H=80.0m (the ground surface level being variable, and the settling floor, from +541.79m – at the upstream wall, to +542.65m - at the downstream wall): vertical geostatic stresses $\sigma_{oy} = \rho \cdot g \cdot H = 17\ 248$ kN/m^2; horizontal geostatic stresses $\sigma_{ox} = k_o \cdot \sigma_{oy} = 4\ 312$ kN/m^2 (where: k_o=0.25). The induced stress by the presence of the excavation was $[\sigma_e]$, respectively the stress variation represented by the horizontal stress $\sigma_{ex} = -4.31$ MPa and the vertical stress $\sigma_{ey} = -17.25$ 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 ρ_b, g and G_b (where: $\rho_b = 25$ kN/m^2 - concrete density; g=9.8m/s^2 - gravitational acceleration; G_b - concrete support thickness, in m).

3.1.5. Achievement of calculus and storing of results

The calculus was made taking 60 iterations per increment and a tolerance of 1% of the results, using the "initial stress method" for the resolution.

The calculus results were stored in 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 lining). The results obtained correspond to the following parameters: the vertical and horizontal displacement, in mm); the stresses – in kN/m^2 – (horizontal, vertical and tangential; the principal - maximum and minimum-, the maximum shear, compressive and tensile).

4. ANALYSIS OF THE RESULTS OBTAINED FROM THE NUMERICAL MODELLING

In order to ease the stability analysis, a failure criterion a will be introduced considering the intrinsic curve of the rocks (concrete). For any point characterised by a certain state of the stresses, the correspondently Mohr circle is determined and it is reported to the intrinsic curve of the rocks (concrete). In this way, the Mohr-Coulomb line [4], [5], [7] will be taken into consideration, so that, for the cohesion value of C=2200kN/m^2 and frictional angle $\varphi = 55^o$, the safety coefficient is calculated with the relation:

$$CS = \frac{1}{(\sigma_1 - \sigma_2)} \cdot [2\ 523.74 - 0.82 \cdot (\sigma_1 + \sigma_2)] \qquad (1)$$

Where: σ_1 and σ_2 are the maximum and minimum principal stresses, in kN/m^2.

Regarding the calculated value of the safety coefficient CS (Fig.4), 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.

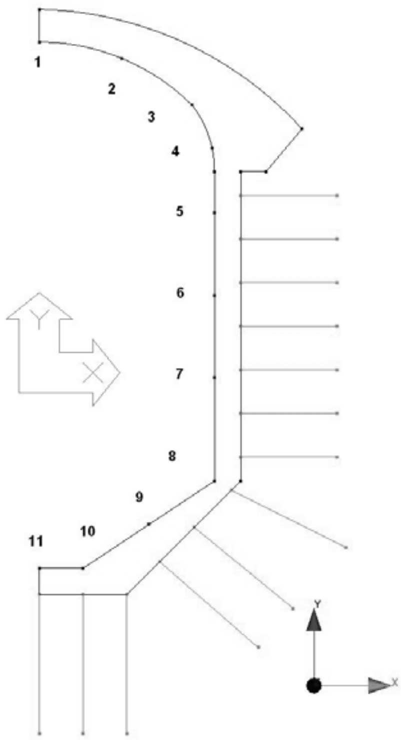


Fig.4. Safety coefficients in the various points on the internal contour of the lining of the underground settling
(1) – *CS*=9,6; (2) - *CS*=1,78; (3) - *CS*=1,06; (4) - *CS*=0,995; (5) - *CS*=1,124; (6) - *CS*=3,7; (7) - *CS*=2,37; (8) - *CS*=5,83; (9) - *CS*=1,67; (10) - *CS*=1,81; (11) - *CS*=6,25

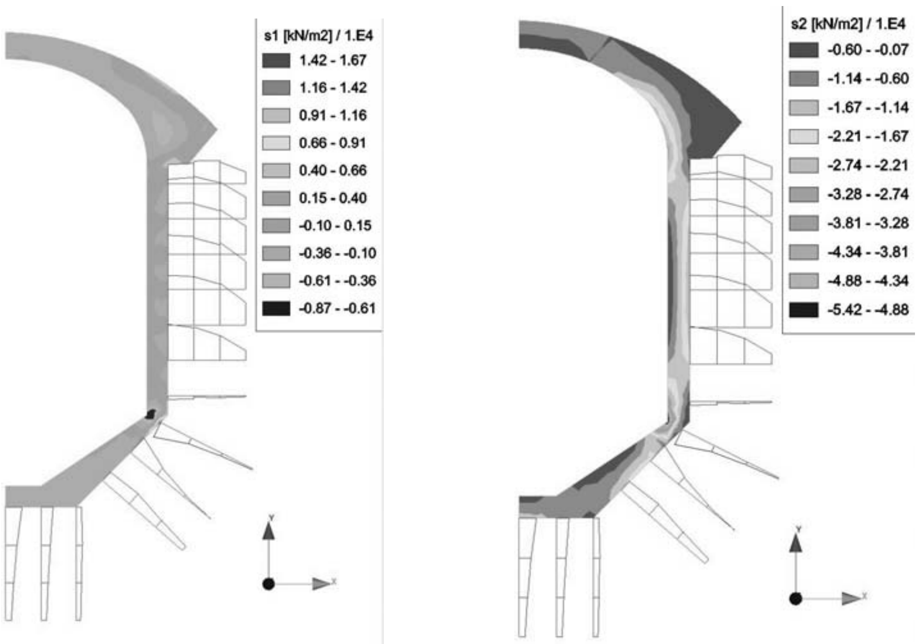
After data analysis concerning the stress-strain state developed around the excavation of the Livezeni underground settling and into lining structure, the following were resulted:

The vertical displacements into the lining structure, on the internal contour, are situated between 13mm at the ceiling, and under 8-9mm at the walls, and 2-3mm at the floor. Also, horizontal displacements in the lining structure at the walls level are of a maximum of $\pm$ 7mm, decreasing steeply toward zero, in the rest of the lining. It is noticed that both the vertical displacements and the horizontal ones are in the limits of the concrete elastic deformations.

If the stresses development in the surrounding rock masse is analysed, it is observed that due to the rigidity of the lining structure, with a preponderantly elastic behaviour, it is freed from the stresses state, which are transferred into the surrounding rocks massive having a preponderantly elasto – plastic toward plastic behaviour, especially nearby the excavation zone.

Regarding the tensile stresses developed in the lining structure (that are the most dangerous stresses, from the point of view of support stability), it is observed that these stresses are missing along the internal contour of the lining (fig.5), which leads to the exclusion of the possibility of failure by lining tensile, starting with the internal surface of the settling lining. The stresses concentrators exist, but, on limited surfaces, in the springing of the arch (under and over the supporting shoe) in the limits of 8300kN/m^2 (of about 4-5 times higher than the tensile strength of the concrete) and in the jointing zone between the wall and the floor, of a maximum of 16 700kN/m^2 (over 9 times the concrete strength).

It is noticed that, although the tensile stresses in these zones are excessively high, the lining structure failure is run down by the steel reinforced structure, that case these loadings very well. As a consequence, from here, the idea of changing the geometry of the joint between the supporting shoes of the arch lining, from the "right angle" configuration to the "obtuse angle" one, which would reduce the tensile stresses in this area, results immediately. Also, concerning the excessive development of the tensile stresses in the connection zone, between walls and the floor, in order to limit these stresses, the construction of a cylindrical connection, progressively, between walls and floor linings, is recommended.



a) b)

Fig.5. Principal maximum and minimum stresses, in KN/m^2, in the lining and the loadings n, in kN, in the bolts at the walls and the floor level

Generally, the compressive stresses developed on the internal contour of the lining range between 800 and 13 000kN/m^2 (regarding to 16 000kN/m^2, the value of the compressive strength of the concrete). The dangerous zones from the point of view of stability at compression of the support structure, on the internal surface, are located in the connecting zone between the ceiling and the wall, where the curve rayon is smaller. The stresses value, in the nodes 200 and 237, being of 20 000-26 000kN/m^2 (about 1.7 times higher than concrete strength). Also, the maximum shear stresses are highest in the same nodes (of 10 000-13

000kN/m^2). It is mentioned that these values were only reported to the concrete, without taking into account the concrete steel, which causes a very important part of the stresses.

To additionally argue the findings above, it is necessary to provide an analysis on the basis of a failure criterion. Analysing the support stability by the concrete characteristic curve and the maximum and minimum principal stresses (the Mohr-Coulomb failure criterion), it is observed, generally, the safety coefficient values of the support are *CS*=2-3, with the maximum over *CS*=9.6 – at the ceiling- and *CS*=6.25- at the floor. But, in the same nodes, 200 and 237, previous mentioned, the safety coefficients are at the stability limit of the support (of *CS*=1.06 and *CS*=0.995) – see Fig.4.

Considering all the previous findings, we can conclude that the "critical" zone of the lining is the connection at the springing of the arch level, where the increase of the curvature rayon and the modification of the supporting shoe configuration are recommended, so that excessive stresses do not appear in this zone. Furthermore, the mounting of the reinforcement steel arch truss at the ceiling and of the frictional and even grouted bolts, in the rock from the ceiling level, substantially improve the support stability of the Livezeni underground settling.

Regarding the bolting system function in the achievement of rock stability, this is dignified by the computational model of the excavation stability supported only with bolts, by report to the underground settling in the final state.

From the model of the underground settling supported only with bolts (model B), the stresses redistribution around the bolts is evident and, implicitly, the reduction of the excavation's internal surface deformation.

Without the loading support, the bolts, both at the walls and at the floor, are charged only at the tensile loading, with up to 1 270kN (the walls bolts being loaded more at the terminal toward the rock massive, and the floor bolts at the terminal toward the excavation). The loading direction is oriented approximately following the axis of the bolts, what means that the bolt are charged less at the shear and more at the pull-out loads.

After loading the reinforcement concrete support mounting a relaxation of the tensile loads of the bolts is observed, at the maximum 438kN. Also, it is observed that the floor bolts (see the Fig.5) reduce their role significantly, and the first floor bolt passes from a tensile loading (of about 1 000kN), to the compressive one (of a maximum -128kN). For the walls bolts, the loading direction is changing significantly, toward a perpendicular direction on the bolts axis; which highlights, in this case, the role of the bolts in the shear loadings developed into rocks massive.

5. CONCLUSIONS

The Livezeni underground settling belongs to the hydroelectric arrangement of the Jiu River, on the Livezeni sector. It has a length of 130m (the width of 9.2m and the height of 12.3-13.4m). It is supported with reinforced concrete (in majority, with a thickness of 0.6-0.7m), shotcrete (of 0.09m thickness and grouted bolts, from PC 52, with $\varphi = 20$ mm).

The excavation and the settler support execution were made in the 6 main phases, starting from the ceiling toward the floor.

The intersected rocks by the underground settling were constituted of granites with different degrees of fissuring and alteration, classified in categories from II to V (from hard rocks to very weak rocks).

In view to verify the support and analyse the stability of the underground settling, both the analytical and numerical models were used. In the performed calculus, the rocks' behaviour was considered preponderantly elasto-plastic and of preponderantly elastic support.

The check by analytical models led to the conclusion that the settling support, achieved in conformity with the project, rests stable.

The approximation degree of the analytical models taken into consideration has not allowed a detailed study of the support stability. Therefore, it came to the stability analysis with the aid of the finite element method.

The modelling with the finite elements was achieved in the plane strain hypothesis, the geostatic loading of the model, and the rocks behaviour being elasto-plastic without hardening of Mohr-Coulomb type.

After the data analysis, resulted from the calculus, on the finite element models, the following were resulted:

Both the vertical deformations and the horizontal ones are between the elastic deformations limits of the concrete.

Due to the rigidity of the support structure, the stresses are transferred from the excavation interior toward the massive of the surroundings rocks (that suffer preponderantly elasto-plastic behaviour, toward plastic one).

The lack of the tensile stresses on the internal contour of the support is observed.

The tensile stresses concentrators (with values of up to 9 times the concrete strength) are developed on the back of the support, in the springing of the arch zone (under and over the supporting shoe). Although the tensile stresses in these zones are excessively high, the failure of the support structure is prevented by the reinforcement steel arch truss, that case these stresses very well.

The dangerous zones from the point of view of support structure stability at the compressive and tensile loadings, on the internal surface, are situated in the connecting zone between ceiling and wall, where the curvature rayon is smaller.

Analysing the support stability on the basis of the Mohr-Coulomb failure criterion, a good general stability of the support is observed, less in the connecting zone at the springing of the arch, where the support stability is at the safety coefficients limit.

Taking into consideration the previous findings, in order to avoid the support failure in the springing of the arch zone, the change of the geometry of the connection between the supporting shoe of the ceiling, from the “right angle” configuration to the “obtuz angle” is recommended, that could reduce the tensile stresses in this zone.

Regarding the excessive tensile stresses development in the junction zone between the walls and floor is recommended the construction a lining with a cylindrical shape.

Beside the above proposals, it is mentioned that the mounting of the reinforcement steel arch truss at the ceiling and the frictional and grouted bolting system, in the rocks from the ceiling level, substantially improve the support stability of the Livezeni underground settling.

Analysing the bolting system role, comparatively, between the finalized support model and the model with the bolted excavation, the very important role of the bolts in the driving faze of the excavation is observed, when the bolts case the pull-out loadings (the role of the bolting system is substantially reduced after the concrete lining mounting).

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Scientific Reviewers:
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INTÉGRATION DES SITUATIONS LIMITE TOLÉRÉES EN FONCTIONNEMENT DANS L'ANALYSE DU RISQUE INDUSTRIEL

MORARU ROLAND IOSIF*
BĂBUȚ GABRIEL BUJOR*

Abstract: *This paper proposes a definition of the Limit Conditions Allowed in Use and emphasize the factors which are making possible to compare a given working task to a effectively performed task, in a human - machine system safety context. Based on brief mathematical description of the concept, the results of a case study performed in two Romanian printing shops regarding the human operator behavior are synthesized. Conclusions are drawn on the safety barriers removal and a systematic approach is proposed for proper incorporation of these cases into industrial risk assessment process.*

Keywords: *Limit Conditions Allowed in Use, risk assessment, human-machine system, prescribed task*

1. L'ANALYSE DES RISQUES ET LA SÉCURITÉ DANS LES SYSTÈMES HOMME - MACHINE

Le risque est défini *„comme un potentiel danger, plus ou moins prévisible"* [1] et peut être interprété comme un niveau d'insécurité potential. Favaro & Monteau [2] précise que le risque est *„un sentiment ressenti par l'individu"*. L'état indésirable du système homme-machine conduit à des conséquences négatives telles que la nature interne ou externe quand elles se rapportent à l'environnement externe du système [18, 19] (Figure 1). Villemeur [24] définit la mesure du risque comme *„la taille d'un danger lié à une mesure de la probabilité d'un événement indésirable et la mesure des effets ou conséquences"*. L'analyse des risques industriels permet de définir des situations industrielles complexes, qui font intervenir l'équipement, produits, personnes et d'autres facteurs [6, 8, 9]. La nature des effets varie considérablement et crée des difficultés dans l'analyse des conséquences [4, 10, 23]. L'analyse de la probabilité de survenue et la gravité des conséquences est la fondation de l'évaluation des risques [11, 13, 14]. L'évaluation des conséquences peut être exprimé de diverses manières (tableau 1).

* *Assoc.Prof., Ph.D at the University of Petroşani, roland_moraru@yahoo.com, gabriel_babut@yahoo.com*

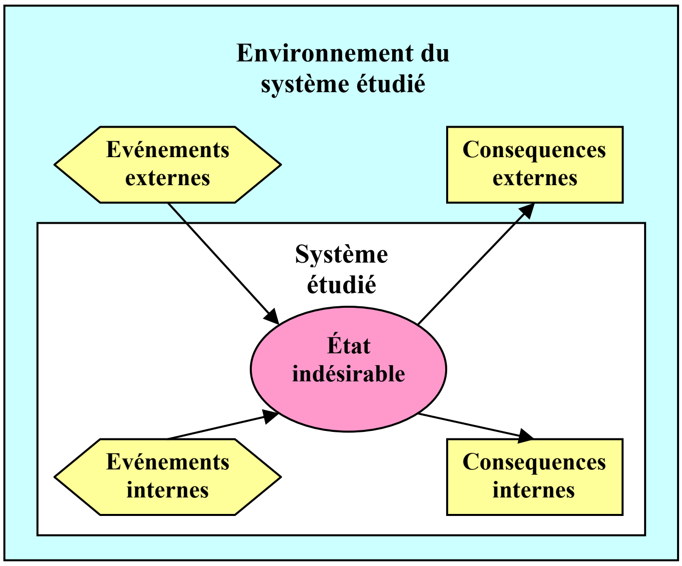

Fig. 1. Le champs général des causes pour l'ensemble générique „événements initiales - conséquences"

Tableau 1. Façons d'exprimer la probabilité d'occurrence et les conséquences

La mesure de la probabilité d'occurrence	Evaluation de la probabilité d'occurrence	Nature des conséquences	Evaluation des conséquences
Probabilité Fréquence Pourcentage, rapport Expression littérale	Par opération, action Sur demande Par unité de temps Par durée de vie Sur un certain temps Par distance parcourue	Physique Physiologique Psycholoqique Financière Politiques Temporelle Expression littérale	Période d'indisponibilité Nombre d'équipements ou de personnes touchées Amplitude Le coût de degâts

La sécurité en fonctionnement permet d'évaluer la confiance dans les services qu'un système fournit. La degré de confiance peut être abordée avec les différents aspects (figure 2) interdépendantes et complémentaires [17, 20] tels que la fiabilité, disponibilité, maintenabilité et la sécurité. L'analyse de la sécurité opérationnelle a comme but d'identifier et de quantifier les obstacles qui empêchent le fonctionnement normal d'un système [12]. Laprie [7] a classé ces obstacles en pannes, erreurs et fautes. La définition proposée par Villemeur pour l'erreur humaine [24] est très proche de celle de l'échec d'une entité: *„différence entre le comportement réel de l'opérateur humain et le comportement imposée, alors quand la différence est supérieure à la limite de l'acceptabilité dans des conditions données"*. D'autres auteurs, dont nous citons Fadier [3], en tenant compte de toutes les activités humaines, définissent l'erreur humaine comme il suit: *„résultat inacceptable (en dehors de la tolérance) de l'action humaine et/ou l'absence d'action d'un opérateur et/ou une équipe, actions qui doivent être prises pour atteindre un objectif spécifique dans des conditions données et dans un laps de temps"* [5].

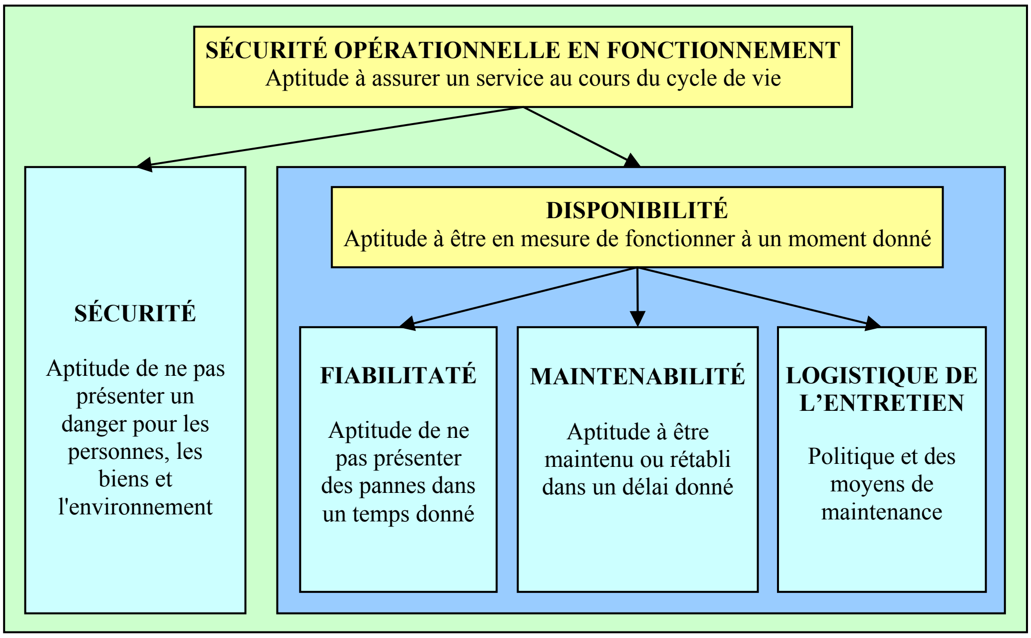

Fig. 2. Composants de la structure de la sécurité opérationnelle des machines

2. SITUATIONS LIMITE TOLÉRÉE EN FONCTIONNEMENT: LE CONCEPT ET SA FORMALISATION

La décision de l'opérateur humain peut être divisé par rapport aux exigences dues à des perceptions différentes d'une situation ou à cause de différents objectifs [15]. Compte tenu des observations ci-dessus peut être faite définitions suivantes exprimées en termes de typologie les Situations Limite Tolérée en Fonctionnement (SLTF) [16, 25]:

Définition 1: **Situations Limite en Fonctionnement.** Soit g_i (t_j) l'évaluation de la gravité par rapport aux critères i. Gravité peut être associé à un seuil d'acceptabilité pour le concepteur $S_{i,C}$, et d'utilisateur de l'équipement d'exploitation $S_{i,E}$. Un cas peut être considéré comme un seuil (SL) si on peut au moins un associé un critère pour lequel le seuil de la gravité est plus grand que le seuil d'acceptabilité du concepteur:

$$SL = \{s_j / \forall i,\ g_i(t_j) < S_{i,C}\} \tag{1}$$

Définition 2: **Situations Tolérée en Fonctionnement.** Un cas peut être considéré comme Situation Tolérée en Fonctionnement (STF) si, quelles que soient les critères considérés, la gravité ne dépasse pas le seuil de l'acceptation associé aux utilisateurs:

$$STF = \{s_j / \forall i,\ g_i(t_j) < S_{i,E}\} \tag{2}$$

Définition 3: **Situations Limite Tolerée en Fonctionnement.** L'ensemble des Situations Limite Tolerée en Fonctionnement (SLTF) est défini comme l'intersection des ensembles de Situations Limite en Fonctionnement (SL) et les Situations Tolerée en Fonctionnement (STF).

$$SLTF = \{s_j / s_j \in (SL \cap STF)\} \tag{3}$$

Définition 4: **Le fonctionnement normal (M_n).** C'est le cas du fonctionnement garanti par le concepteur. Cela ne coïncide pas nécessairement avec la qualité obtenue de la charge de travail, mais les critères de sécurité sont remplis.

Définition 5: **Mode nominal de fonctionnement (M_o).** Le mode nominal est fondamentalement en parfaite adéquation avec la qualité de la mission *„correspondant à répondre aux spécifications énoncées dans les conditions de production"*.

Définition 6: **Mode incorrecte de fonctionnement (M_i).** Est l'opération pour laquelle les exigences de concepteur ne sont pas remplies, mais non intentionnellement.

Définition 7: **Mode volontairement dévié (M_d).** L'opération pour laquelle les exigences du concepteur ne sont pas respectées délibérément.

Définition 8: **Mode d'utilisation ajouté (M_a).** Est un mode qui ne correspond pas soit ensemble des modes prévu par le concepteur, ou la multitude des modes de fonctionnement incorrectes ou déviées.

L'algorithme présenté dans la Figure 3 permet la détermination des cas différents selon le point de vue du concepteur et de l'exploitant.

3. ÉTUDE DE CAS: ANALYSE SLTF DANS DEUX TYPOGRAPHIE

L'analyse des résultats obtenus en deux ateliers de l'impression (typographies) [21], au cours d'une semaine d'obsérvations guidées par la méthode systématique APRECIH [22], a permis l'identification de 32 SLTF, qui ont été regroupés en six catégories, à savoir: procédures mal respectés, annihilation physique des moyens de sécurité, interventions sur l'outil en cours d'utilisation, des problèmes avec les activités déviées d'emploi et de formation dans le maintien et cathacrèse (= utilisation d'une machine ou un dispositif à d'autres fins que celles pour lesquelles il a été conçu).

Les résultats obtenus à partir d'observations sur les deux typographies mentionnées ci-dessus est résumée dans le tableau 2. Près de 61 % de SLTF sont liée à l'anéantissement ou le contournement des barrières et des mesures de sécurité.

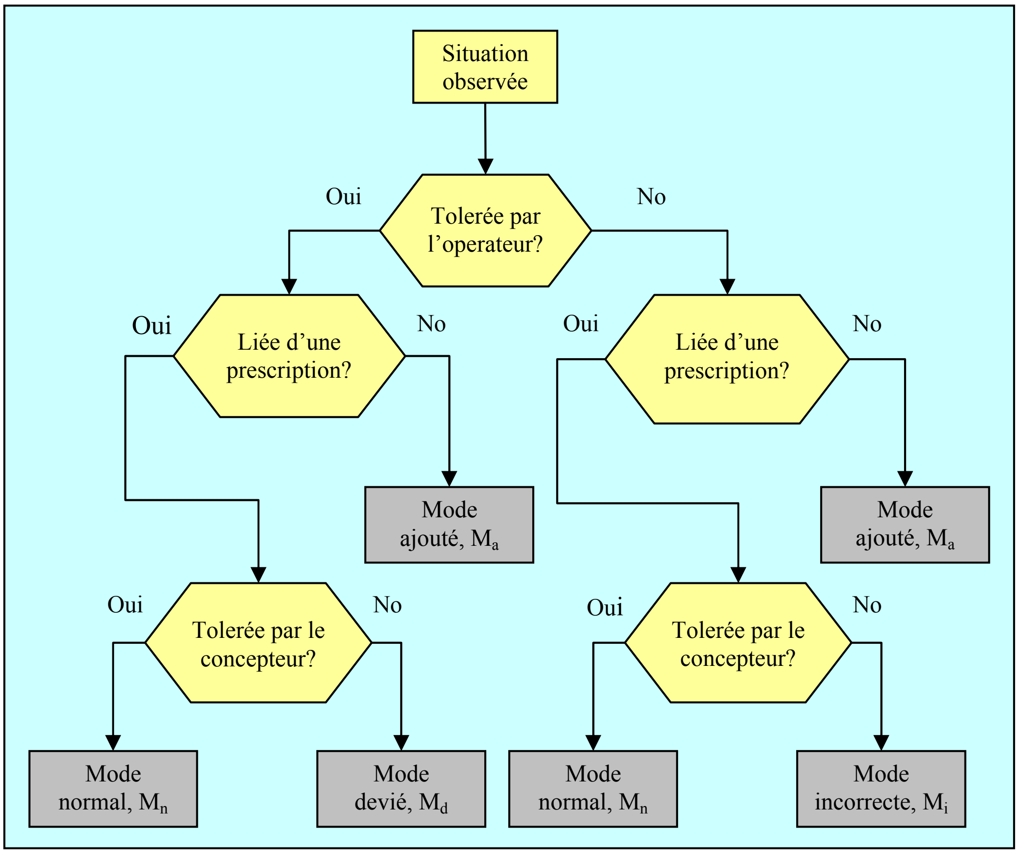

Fig. 3. Détermination des modes de fonctionnement

Tableau 2. Catégories de SLTF observée dans les typographies

Type de SLTF	**Nombre de SLTF identifié**
Le non-respect des procédures	8
Annihilation physique des moyens de sécurité	6
Problèmes avec les activités déviées d'emploi et de formation	7
Les interventions sur la machine en fonctionnement	4
Maintenance	3
Catachrese	4

Pour étudier le bénéfice, coût et du déficit potentiel, pour chacun des quatre critères de définir la gravité une l'analyse multicritères a été mener pour les SLTF identifié. Les résultats obtenus sont présentés dans le Tableau 3. Pour chaque critère a été comptabilisée le nombre de SLTF qui a été constaté que il y a un bénéfice, coût ou déficit, étant donné que certains SLTF ont des effets multiples sur de multiples critères.

Tableau 3. L'analyse multicritères de SLTF en termes de bénéfice, des coûts et de déficit potentiel

Bénéfice immédiat		**Coût immédiat**		**Déficit potentiel**	
Critères	**Nombre de SLTF**	**Critères**	**Nombre de SLTF**	**Critères**	**Nombre de SLTF**
Productivité	20	Productivité	6	Productivité	6
Qualité	5	Qualité	12	Qualité	8
Charge de travail	7	Charge de travail	8	Charge de travail	2
Securité	0	Securité	6	Securité	16

L'analyse des données contenues dans le tableau 3 montre que la plupart des SLTF apporte un bénéfice immédiat en termes de production (≈ 62 %) permettant un gain de temps et la limitation dans l'exécution d'opérations et des interruptions de production. Le déficit potentiel se réfère en 50 % des cas de SLTF au niveau de sécurité. Bien que se concrétise rarement, le déficit de la sécurité est une conséquence de l'exposition volontaire des opérateurs dans les zones dangereuses et de la fragilisation des systèmes de protection en série. Fondamentalement, le coût est exprimé en termes de charge de travail (25 %), résultant en un travail supplémentaire pour les opérateurs qui contourne les mesures de sécurité.

L'analyse en terrain indique l'importance de SLTF, qui se traduit par l'anéantissement ou le contournement des mesures de sécurité en conformité avec réponse des opérateurs en termes de maîtrise du compromis.

4. INTERPRÉTATION DES RÉSULTATS

L'analyse multicritères du bénéfice, coût et du déficit potentiel est un outil qui illustre l'importance du fait que les SLTF doivent être prisent en compte dans l'analyse des risques. Par conséquent, l'approche d'analyse des risques nécessite une nouvelle perspective. Ainsi, nous avons développé l'arbre des causes pour événement indésirable *„présence de l'opérateur à l'intérieur de la machine de pliage"*. L'analyse qualitative de ces arbres montre que l'arbre résultant n'est pas la même pour les situations d'exploitation prescrites (représenté dans la figure 4) et pour le cas de SLTF (représenté dans la figure 5).

Dans le premier cas aucune exception n'est considéré comme volontaire et donc pas d'effet sur la situation de l'emploi. Dans le deuxième cas (SLTF) l'indicateur est éteint, l'opérateur s'engage volontairement à intervenir dans le fonctionnement de la machine.

Cet exemple confirme que les causes d'un événement ne sont pas les mêmes dans les cas prévus pour le fonctionnement et les SLTF. En outre, il est important de noter qu'il n'est pas possible d'associer les aux causes primaires du premier arbre, les causes du seconde arbre.

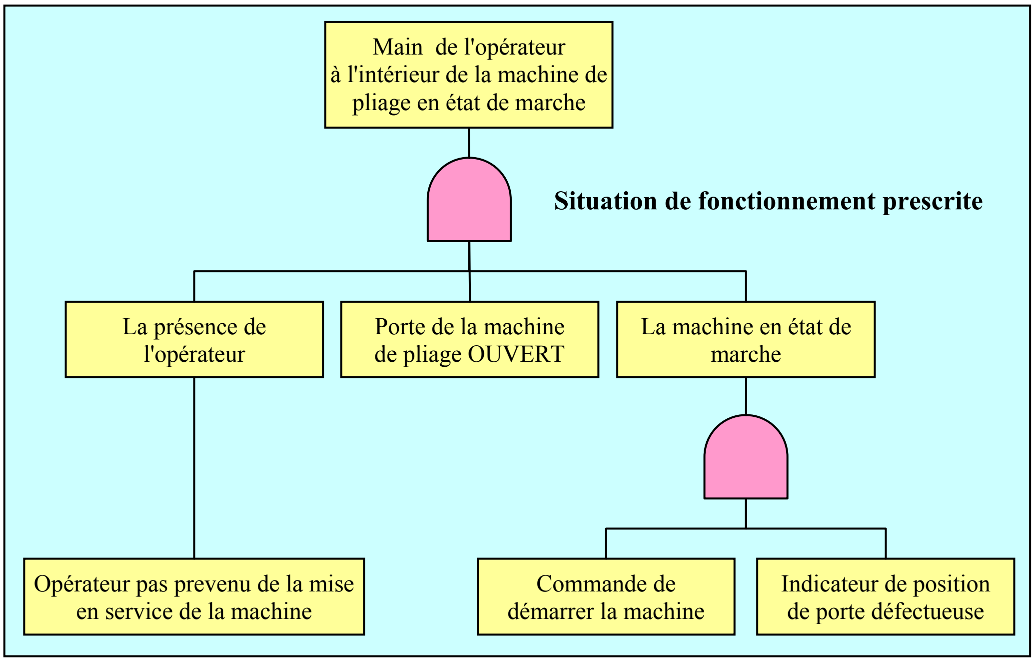

Fig. 4. Arbre des Causes de l'événement *„présence de l'opérateur à l'intérieur de la machine de pliage”* en situation opérationnelle prescrites

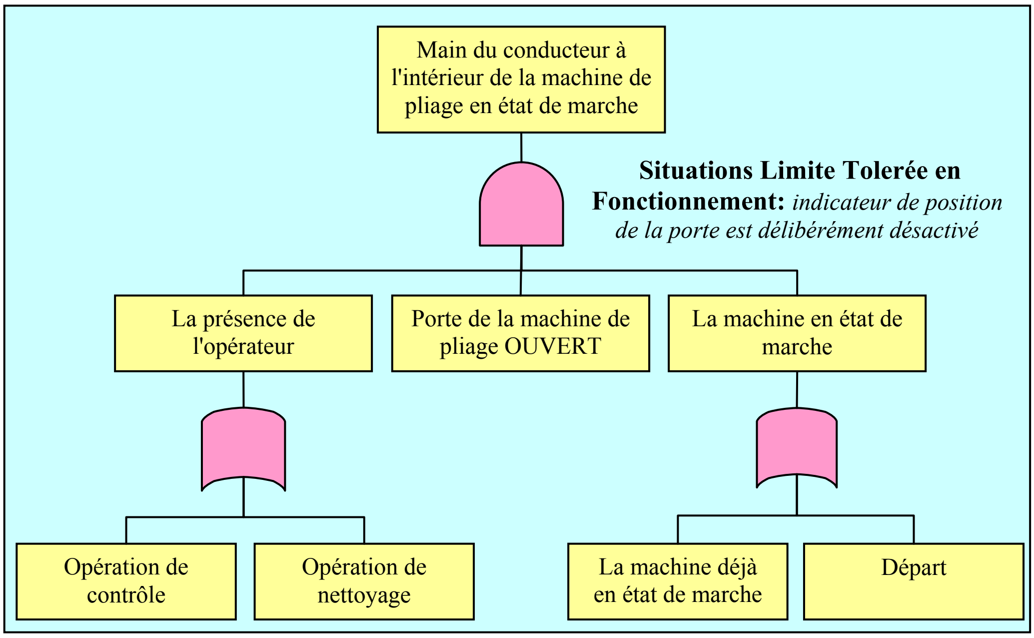

Fig. 5. Arbre des Causes de l'événement *„présence de l'opérateur à l'intérieur de la machine de pliage”* en SLTF

L'échec de la procédure de nettoyage de rouleaux en caoutchouc SLTF conduire à l'exposition des opérateurs à des risques d'écrasement et de l'agression chimique. Etude des modes de défaillance de composants techniques et des erreurs humaines doivent être complétées par l'étude des moyens de supprimer ou de contourner les mesures de sécurité qui conduisent à SLTF. Pour cet effet, il est nécessaire d'identifier les conditions d'exploitation antérieures à la machine. Pour l'examen de SLTF dans quelques approches d'analyse des risques a priori on devrait ajoutée la phase d'identification SLTF. Après une analyse préliminaire des risques, après quoi on identifie les sources de danger, doit suivre l'analyse des mesures de sécurité appliquées. À cet égard, on étudiera la fonction précise de chaque mesure de sécurité particulières et les restrictions introduites dans les voies de l'opérateur humain. Cet ensemble de mesures pour l'identification des situations opérationelles, représentées dans la figure 6, peut être facilitée par l'analyse opérationnelle et les leçons apprises.

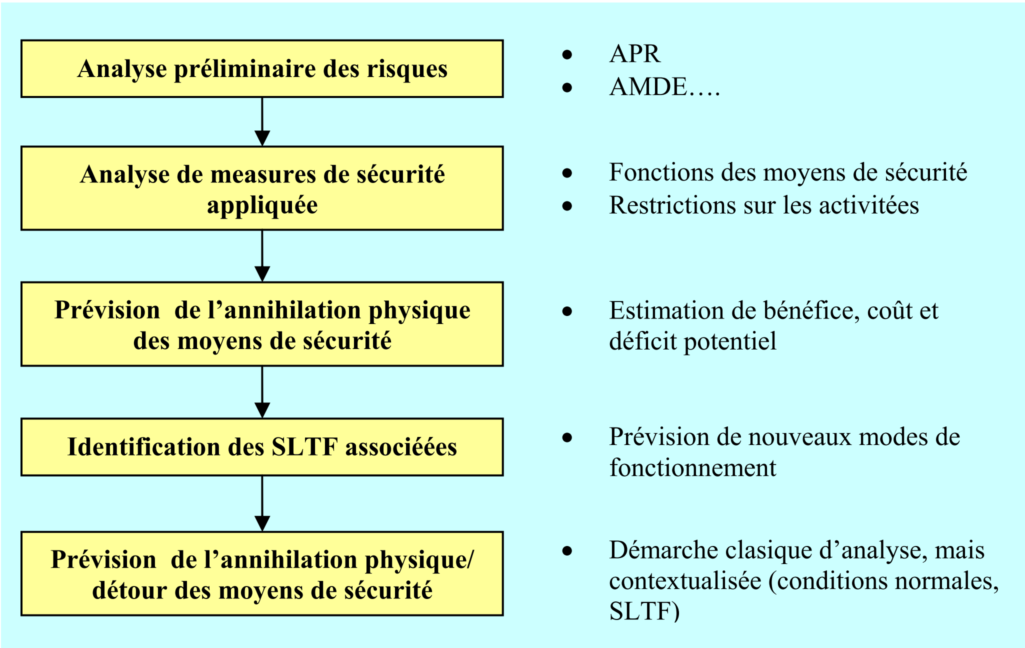

Fig. 6. Approche d'intégration de SLTF dans la structure de l'analyse des risques

Cette analyse devrait être basée sur la prévision de l'anéantissement ou de contournement des mesures de sécurité, en estimant les coûts, les avantages et le déficit potentiel. Il est également possible d'analyser et de prévoir les modes de fonctionnement associés découlant de SLTF. Enfin, l'analyse des risques peut être réalisée pour chaque situation d'exploitation.

5. CONCLUSION

Dans cet article, la notion de **Situations Limite Tolérée en Fonctionnement (SLTF)** comme situation accepté par l'utilisateur, mais non pris en charge par le fabricant ou le concepteur. Les SLTF se manifestent à travers des déviations volontaires et les modes de fonctionnement ajoutées. Jusqu'à présent, ces types de violations n'ont pas été pris en compte dans l'analyse des risques et, en conséquence, le spectre du risque opérationnel n'est pas entièrement couverts par une analyse a priori.

Les SLTF sont le résultat d'un compromis en raison de déviation par rapport aux exigences du concepteur et/ou le fabricant en vue d'améliorer la performance de système 'homme-machine. Mettez en surbrillance ce compromis est facilitée par l'analyse multicritère. Afin d'optimiser les performances d'un critère (par exemple, la productivité), les SLTF conduisent à la dégradation des performances par rapport à d'autres critères (par exemple, de la sécurité). Les mesures de sécurité mises en œuvre par les différents partenaires impliqués sont destinées à empêcher (de réduire la probabilité d'occurrence) et/ou de protection (en minimisant la gravité des conséquences) risque d'accident de travail et maladies professionnels. Ils fonctionnent grâce à des restrictions sur le comportement des opérateurs.

L'analyse des résultats obtenus en deux typographies, au cours d'une semaine d'obsérvations guidées par la méthode systématique APRECIH, a permis l'identification la plupart de ces situations ont des causes correspondantes d'annihilation et de contournement des barrières de sécurité. Enfin, il a été proposé une approche d'analyse des risques qui permet l'intégration de ces situations.

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Scientific Reviewers:
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ON THE VERISIMILITUDE OF SAFETY OBJECTIVES AND ACCIDENT SCENARIO SELECTION

MORARU ROLAND*

Abstract: *Risk can only be effectively managed if it is fully understood; therefore a multi-disciplinary approach is often needed to assemble the required knowledge in areas such as probability and statistics, engineering, systems analysis, health sciences, social sciences, and physical, chemical, or biological sciences. A risk assessment must be included in all phases of a system's life cycle to be effective. The paper gives a summery of author's considerations regarding these issues, based on concepts such as verisimilitude of safety objectives, selection of accident scenarios and absolute level of negligible probability. There are emphasized the advantages and drawbacks of probabilistic language and tools in safety studies*

Keywords: *risk assessment, probability, verisimilitude, scenario, safety objective.*

1. INTRODUCTION

In industrial systems operation can appear certain events judged sometimes as quasi - impossible, having extremely low probabilities of occurrence [7]. Risk is defined as the measure of a hazard that combines a measure of the occurrence of an undesirable event and a measure of its consequences [8]. A situation is a hazard if it can be harmful to man, the society, or the environment [2]. The occurrence of an undesirable event is usually measured by its occurrence probability over a given period or by its frequency (number of events occurring per unit of time), or even by its rate of appearance [6]. During the development of safety studies, this kind of events can be taken into consideration in different stages, such as:

a. When defining the study's objectives, where the probabilities of subjective nature are explicitly related to specific unwanted events.

b. In the process of selection of the scenarios considered as potentially generating the unwanted event. Most of the time, based on qualitative analysis, the manager considers the most plausible scenarios from those identified or built by risk assessors.

c. In the process of "a posteriori" evaluation of each scenario's probability, when certain event combinations can lead to negligible values.

Consequently, a major problem in all the stages of a safety study can appear, namely: starting from what probability level or non-verisimilitude can be neglected the events or the combination of identified events, events which will be ignored in the stage of decision making [3, 9]. This should be done considering meanwhile the perceived risk, which is a subjective measure of real industrial and occupational cases (see Fig. 1)

* *Assoc.Prof., Ph.D at the University of Petroşani,* roland_moraru@yahoo.com

Structured on the above - mentioned issues, there will be presented some considerations regarding this field of concern.

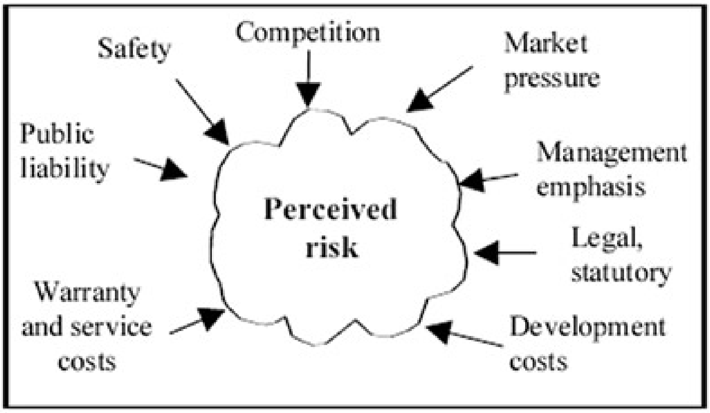

Fig 1 The subjective character of risk perception

2. BASIC CONCEPTS EMPLOYED IN THE SYSTEMS SAFETY FIELD

The system's operation safety represents, through his four specific components (safety, availability, reliability and maintenance), a basic feature of the different life cycle stages of a system. The concept of safety for a system's operation can be illustrated by the diagram presented in the Figure 2. The common point of the safety of the system operation is the use of probabilistic tool, as assessment technique of risks specific for the analyzed system.

It must be noted that, employed in this field, the language suffers a partial lack from his initial rigorousness existing at probabilities theory level. The safety state of a system can be defined as the absence of the circumstances which can disturb the system's operation. The probabilistic and statistical methods allow to assess the occurrence probability of failures and unsafe conditions induced in the system by their propagation as scenarios [5].

Theoretically the absolute (or total) safety of system corresponds with the impossibility of any accident's occurrence, regardless of the considered timing, the system's and his environments status, with regard to all the possible technical failures, human errors and external aggressions. Consequently, the system's design would require a "perfect and complete" knowledge of the system's elements and status, in all the life cycle stages and for all the external environmental conditions.

Such an assumption is not reasonable from reasons related to the scientific and technical level and, mainly, from reasons emerging from the inherent limits of human imagination. These general considerations are leading us to one of the basic principles employed in the study of systems safety: "*The absolute safety is a myth*".

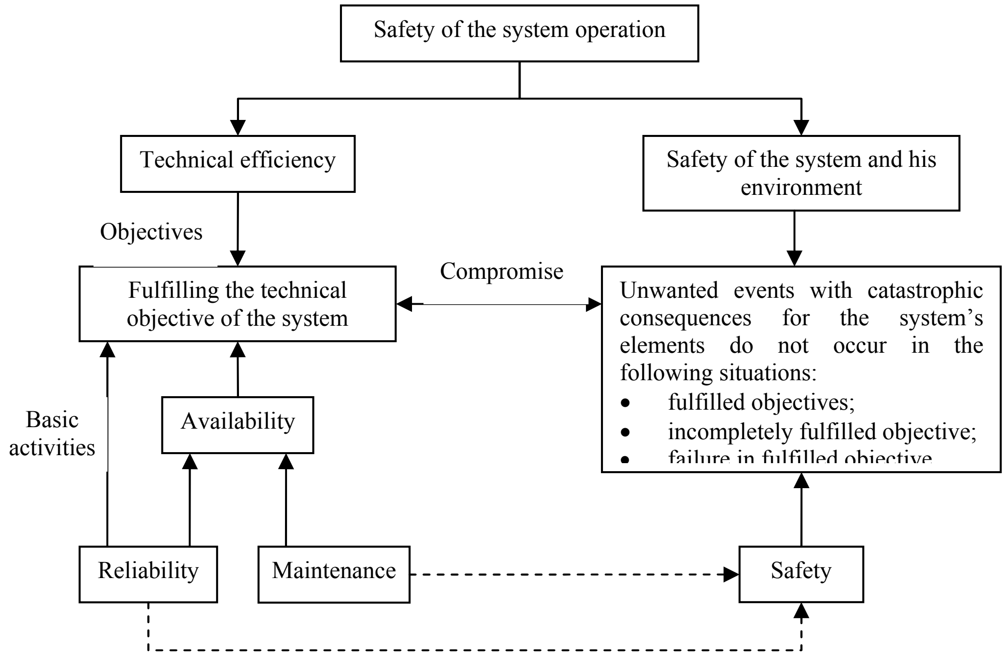

Fig. 2 Basic elements of the safety of the system operation

It follows that the primary notion used is that of "*safety objective*" related to an acceptable risk level, based on rational technical and financial resources, such as illustrated by the ALARP concept (see Figure 3).

For the operation of a system, there can be defined:

- a field of knowledge, in which the accurate description of all the operational status and of malfunctions is possible, together with the predictable consequences on the external environment;
- a field of ignorance, in which the operation status is unknown.

If in the field of knowledge it is possible to assess, with a certain level of accuracy, the occurrence probability for an undesired event and the magnitude of his consequences the estimation of an event's occurrence probability and consequences, if the event is incompletely defined qualitatively, is impossible. The probability of the damage occurrence during the exposure to a risk factor describes the accidental, stochastic and uncertain character. The exposure frequency expresses the time lapse in which the worker is exposed to the risk factor action.

A value must be allotted to the probability of occurrence. This designation is not the result of an inspirational moment. Normally, the risk assessment process should start upwards, by defining for each working task, of the hazards and for each hazards of the risks related. Only after this hazard and risk identification phase (e.g. based on a check-list) the quantification can be initiated.

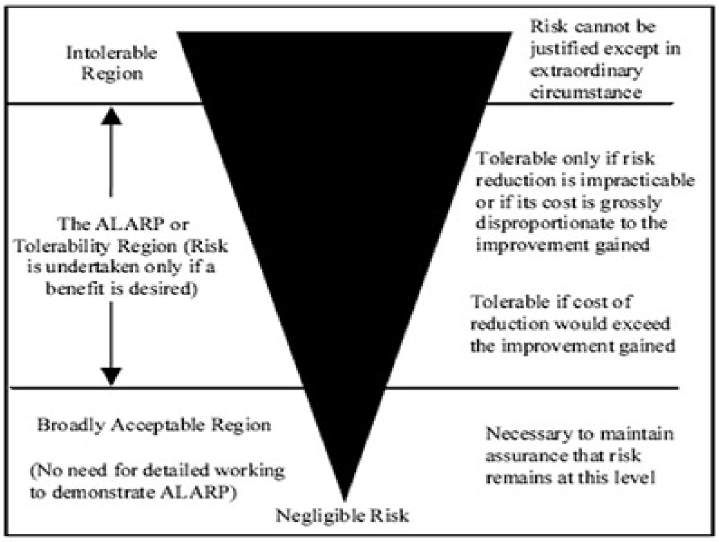

Fig. 3. Levels of Risk and As Low As Is Reasonably Practicable (ALARP)

Quantitative Risk Analysis involves the calculation of probability, and sometimes consequences, using numerical data where the numbers are not ranks (1st, 2nd, 3rd) but rather "*real numbers*" (i.e. 1, 2, 3, 4 where 2 is twice 1 and half of 4). As such, accurate quantification of risk offers the opportunity to be more objective and analytical than the qualitative or semi-qualitative approaches. Most commonly, quantification of risk involves generating a number that represents the probability of a selected outcome, such as a fatality. Following is an example of probabilistic information concerning the risk of a fatality per year (table 1). British Nuclear Industry research suggests the following probability of death from various causes in the UK [4]. The figures are based on past history. The history of fatalities in the Australian mining industry from 1991 to 2001 suggests the following risk of death in Australian mining - .0005 or 1 in 5,000. In western industrialized countries, disease results in a death rate of approximately 10^{-2} per year (1 in every 100 are at risk of death from disease), a high-level risk involuntarily accepted by society. On the other end of the spectrum, natural events such as lightning, flood, and insect bites produce a death rate around 10^{-6} per year, the lowest level of involuntarily accepted risk.

Table 1. Probabilities of risk occurrence of a fatality per year from various causes [4]

Crt. nr.	Cause/activity	Probability of a fatality per year
1.	Lightning	0.0000001 or 1 in 10 million
2.	Fire / explosion at home	0.000001 or 1 in 1 million
3.	Death in a 'safe' industry	0.00001 or 1 in 100,000
4.	Death in a road traffic accident	0.0001 or 1 in 10,000
5.	Death in mining	0.001 or 1 in 1,000
6.	Flying in commercial aircraft	1 -0.00001 or 1 in 100,000
7.	Smoking	0 .05 or 1 in 200

Most Quantitative Risk Analysis for industrial applications attempts to establish probabilities of unwanted events and subsequently the probability of the consequences from the unwanted event. For example, the risk of a total large petroleum storage tank structural failure might be .003 per year. If there are multiple events that must happen before a major loss can occur then assigning numerical probabilities allows for risk calculations that are normally not possible with qualitative or semi-qualitative data.

Structurally, the field of knowledge comprises two areas:

- the area of uncertainty;
- the area of certainty.

The uncertainty area corresponds to a qualitative knowledge of system's status and a transient knowledge of each status in a given situation. The uncertainty can be associated to the existing inaccuracy concerning one of the system's status using a probability density, which allows to characterize the deviations from a mean value [1]. This approach represents one of the bases of the fuzzy logic. The certainty area corresponds to a deterministic knowledge of all the system's status and their consequences.

3. THE VERISIMILITUDE OF SAFETY OBJECTIVES

A safety objective can be defined through the following two basic parameters:

- the denomination of an undesired event (e.g. outrunning of a maximal allowable limit for a noxious gas concentration);
- the frequency or likelihood related to the undesired event, expressed in an adapted measuring unit.

The credibility of a system's safety state is directly linked to the aimed safety level, which can be defined by:

- the "*ambition*" of the safety objective;
- the confidence in achieving the proposed goal, starting from a well identified and documented ensemble of skills and tasks, clearly described in a safety plan.

Available data for probabilistic techniques based assessments will be gathered after studies and actions carried out during the stages of design, development and operation of any systems. As a direct consequence, the real problem of safety objectives credibility arises. Most of the studies concerning the safe operation of technical and/or working systems are employing, as a tool, the modeling of accidents occurrence scenarios, whose credibility is characterized by:

- the representativeness of models employed, specifically determined by the degree of comprehensibility, at his turn defined through the number of variables and parameters considered and the laws describing the significant relationships existing between the internal and external variables of the system;
- the credibility of data employed.

Therefore, a so - called "natural" uncertainty concerning the procedures, results and their interpretation will appears. This should not be mixed up with the uncertainty related to the impossibility of undesired event's occurrence, event deployed in the safety objective. If, in the first case, can be studied and envisaged adequate prevention measures, in the second one the statement "such an event or such a scenario can not occur", will allow and - even - facilitate the perpetuation of a real unsafe state, with potentially catastrophic consequences.

4. SELECTION OF SCENARIOS RETAINED IN SAFETY STUDIES

The identification of scenarios able to lead to any undesired event highly depends on the expertise and imagination of the experts having as task the Preliminary Hazard Analysis for the studied system [10].

The list of the scenarios delivered in this stage will be not ranked, the only classification being done by framing the scenarios in one of the following categories:

- **S_1**: Scenarios already observed and judged as realistic;
- **S_2**: Scenarios already observed, but judged as non - realistic, considering the existing prevention and control measures;
- **S_3**:Scenarios not met before in practice, but considered as realistic;
- **S_4**: Scenarios which were not encountered before and are considered highly improbable.

The quality of judging the scenarios as realistic or not, depends on the amount of knowledge and skills of the assessment work team members and of the decision maker. The role of the decision maker is usually prevalent, due to his responsibility within the organization.

The dilemma of the decision maker, in the most of cases, consists in:

a. Either to accept to consider a possible scenario considered “a priori” as having a low probability during the life cycle of the system. This kind of decision can generate supplementary technical, economical or operational compulsions.

b. Either to reject a scenario judged as improbable, accepting the possible consequences.

It must be noted and stated that, depending on the considered component of risk (probability or gravity), the decision maker must pass from an extreme to another:

- Considering only the low probability, the scenario will be rejected. This is a typical short - term decision.
- If the decision maker considers mainly the gravity of generated consequences, the scenario will be retained and will support further detailed analysis, indifferently of his occurrence likelihood. In this case, the decision is aimed to be as a long - term one.

In the uncertainty area, an applicable decision rule regarding the taking into consideration of scenarios consists to allocate them “a priori” a certain likelihood level, starting from the objective associated with the analyzed undesired event. The level of likelihood can be assessed accepting, for example, the hypothesis that there can not exist more than 100 scenarios, identified or not, leading to the undesired event. There will be considered only the scenarios whose probability is with two magnitude orders lower that the probability of the unwanted event. So for an objective of 10^{-3}/h, there will be retained only the scenarios having an occurrence probability higher than 10^{-5}/h.

5. CONCLUSIONS ON THE USE OF PROBABILISTIC TECHNIQUES IN SAFETY STUDIES

Resorting to quantitative risk assessment techniques has as basic goal a detailed evaluation of the safety level in a working system, in order to achieve a significant improvement of the existing or designed systems safety level.

The basic tool employed in quantitative assessment is the probabilistic computation, which offers various specific advantages, such as:

- yields itself to mathematical processing of data;

- facilitates a better rational distribution of involved accountabilities, through the limitation of erroneous interpretations;
- emphasize the weight which should be allocated to each prevention and/or protection measure;
- facilitates the ranking of occupational accidents scenarios and the elimination of those who are unlikely to occur;
- leads, for any part of the system, to the optimization of design and to a better evaluation of the reached and proved safety level;
- according to the obtained results, it allows a more efficient estimation of the importance of "*weak points*" existing in the system, from the safety state point of view.

The use of probabilistic language without discrimination can lead to two major drawbacks:

- an unwarranted increase of expenses for experimental work if the objective consists in a statistical demonstration of the reached safety level, an objective which is practically difficult to achieve;
- a limitation, or even a diminishment, of the "*confirmed*" safety level, in the case when there are considered only the absolute values of probabilities accepted based on experience acquired on comparable systems, previously analyzed.

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OCCUPATIONAL RISK ASSESSMENT IN LONGWALL UNDERCAVED BENCH FACE NO. 4, SEAM 3, BLOCK 6, VULCAN COLLIERY

GABRIEL BUJOR BĂBUȚ[*]
ROLAND IOSIF MORARU[*]
MONICA CRINELA BĂBUȚ[**]

Abstract: *The paper gives a brief description of the principles applied, method description (structure, tools, working procedure, end users envisaged.), application phases for the most applied Romanian occupational risk assessment method. A case study for a particular working system from a colliery from Valea Jiului basin is included. The presented technique is a practical method for carrying out risk assessment in a structured and systematic fashion, which is, nevertheless, easy to apply by „non-experts”. It was specifically developed for application in occupational injuries across all sectors of industrial activity. It allows not only the identification of active failures, influencing factors and latent conditions, but also their ranking. Conclusions are drawn regarding the strengths and potential limitations of the method.*

Key words: occupational risk assessment, mining, ranking, prevention, safety level.

1. INTRODUCTION

Assessment of workplace risks is the foundation of any company's occupational safety and health risk management. In Romania, since 2006, when the new Occupational Health and Safety Act [12] has stated that the risk assessment is compulsory, several approaches were in use but only one method is extended in application. The main question of risk evaluation is *„how safe is safe enough*?" [4, 5]. A necessary condition is that common criteria are used to assess the acceptability of the risk; this is a challenge for further development [1]. Methods are used to rank risks and to define priorities for actions - what is certainly very desirable - but often this is done by neglecting the analysis of the elements defining these risks and the means of improving the situation. As such, we are focusing on the practical application of the risk assessment process, representing a resource for getting up to speed quickly on the different options available and the means to introduce and implement risk management [7, 8, 10]. Meanwhile, in several industrial areas, the lack of trustworthy and reliable statistical data requires a proper estimation of the frequency and severity classes, which in turn can lead - if tools are not adequately used - to false perception of risks, inadequate ranking of priorities and misuse of available resources [9].

[*] *Assoc.Prof., Ph.D at the University of Petroşani, gabriel_babut@yahoo.com, roland_moraru@yahoo.com*
[**] *Lecturer, Ph.D at the University of Petroşani, babut_crina@yahoo.com*

The aim of the INCDPM Bucharest method [2, 11] is to provide a conservative estimate of global risk to hypothetical members of selected population groups using a semi-quantitative approach. One potential use of the presented approach may be as a „screening” tool, to identify areas for more detailed analysis. When applying the risk assessment tool presented in the next section, it is imperious to stay focus on the goal of risk ranking. Keeping this in mind, the effect of the subjectivity will decrease and the overall aim will be achieved [9].

2. THE RISK ASSESSMENT TOOL: DESCRIPTION AND APPLICATION STAGES

Risk assessment implies the identification of all risk factors within the system under examination and the quantification of their dimension, based upon the combination between two parameters: severity and frequency of the maximal possible consequences for the human body [3, 6]. Thus, partial risk levels are obtained for each risk factor, respectively the overall risk level for the entire system (workplace) under examination. The method provides a semi-quantitative means of estimating individual risk, based on a development of the process used to generate a risk matrix. The method also provides a simplified means of obtaining a conservative estimate of the individual risk to members of defined population/groups. It can also be used to identify those event outcomes contributing most to the risk for each of the population groups specified, and may be used as a screening tool to identify areas for further analysis.

The purpose of the method is, as stated previously, the semi-quantitative determination of risk level for a workplace, based upon the systemic analysis and risk assessment concerning work-related accidents and diseases. The application of the method is finalized with two basic documents:

- **workplace assessment card**, including the partial risks levels for each risk factor and the global risk level for the workplace;
- **prevention actions card**, including the entirety of technical and organizational measures provided by regulations in force, for every risk factor.

The method comprises the following compulsory stages:

a. appointment of the team of assessors;
b. definition of system (workplace) to be examined;
c. identification of risk factors within the system;
d. risks assessment for work-related accidents and diseases;
e. forming the hierarchy of risks and settling the priorities for prevention;
f. prevention measures proposal.

These steps are carried out using specific developed working instruments. The contents and structure of these instruments are presented further.

i). Risk factors identification list is a form that includes, in an easily identifiable and compressed format, the main risk factors categories for work-related accidents and diseases, classified according to the criterion of the generating element within the work system (worker, work task, mean of production and work environment).

ii). List of the possible consequences of the action of risk factors on the human body is a useful instrument for the application of the quotation scale for the severity of consequences. It includes the categories of lesions and injuries of the integrity and health of the human body, the possible location of consequences in relation to the anatomical-functional structure of the body and the generic minimal-maximal severity of the consequence.

iii). ***Quotation scale for the severity and probability of consequences*** on the human body, is a grid for the classification into classes of severity and classes of probability for their occurrence. The part of grid concerning the severity of the consequences is based upon medical criteria for clinical and functional diagnosis and assessment of the working capacity drawn up by the Romanian Ministry of Health and the Romanian Ministry of Labor and Social Protection.

iv). Risk assessment grid (see Figure 1) has the form of a table, with lines representing classes of severity and columns representing classes of probability. The grid is instrumental for the effective expression of the risks that exist in the system under examination, in the form of the coupling severity/frequency of occurrence.

v). Risk/safety levels quotation scale (see Figure 2), drawn up starting from the risk assessment grid, is an instrument that is used for the assessment of the anticipated risk level, respectively safety level. All the couplings severity-probability related to risks levels are presented in the central zone of the form.

vi). Workplace assessment card (whose form is given directly in the case study included in the next section) is the summarizing document of all operations of identification and assessment of work-related accident/disease risks.

		Probability classes					
		1	**2**	**3**	**4**	**5**	**6**
Severity classes	**Consequences**	**P > 10 years**	**5 years < P < 10 years**	**2 years < P < 5 years**	**1 year < P < 2 years**	**1 month < P < 1 year**	**P < 1 month**
7	**Maximum**	**(7,1)**	**(7,2)**	**(7,3)**	**(7,4)**	**(7,5)**	**(7,6)**
6	**Very severe**	**(6,1)**	**(6,2)**	**(6,3)**	**(6,4)**	**(6,5)**	**(6,6)**
5	**Severe**	**(5,1)**	**(5,2)**	**(5,3)**	**(5,4)**	**(5,5)**	**(5,6)**
4	**Important**	**(4,1)**	**(4,2)**	**(4,3)**	**(4,4)**	**(4,5)**	**(4,6)**
3	**Medium**	**(3,1)**	**(3,2)**	**(3,3)**	**(3,4)**	**(3,5)**	**(3,6)**
2	**Limited**	**(2,1)**	**(2,2)**	**(2,3)**	**(2,4)**	**(2,5)**	**(2,6)**
1	**Negligible**	**(1,1)**	**(1,2)**	**(1,3)**	**(1,4)**	**(1,5)**	**(1,6)**

Fig. 1. Risk assessment grid

RISK LEVEL		**SEVERITY - PROBABILITY COUPLING**	**SAFETY LEVEL**	
1	**Minimum**	(1,1) (1,2) (1,3) (1,4) (1,5) (1,6) (2,1)	**7**	**Maximum**
2	**Very low**	(2,2) (2,3) (2,4) (3,1) (3,2) (4,1)	**6**	**Very high**
3	**Low**	(2,5) (2,6) (3,3) (3,4) (4,2) (5,1) (6,1) (7,1)	**5**	**High**
4	**Medium**	(3,5) (3,6) (4,3) (4,4) (5,2) (5,3) (6,2) (7,2)	**4**	**Medium**
5	**High**	(4,5) (4,6) (5,4) (5,5) (6,3) (7,3)	**3**	**Low**
6	**Very high**	(5,6) (6,4) (6,5) (7,4)	**2**	**Very low**
7	**Maximum**	(6,6) (7,5) (7,6)	**1**	**Minimum**

Fig. 2. Risk/safety levels quotation scale

As such, this form includes:

- data for the identification of the workplace: enterprise, division (workshop), workplace;
- data for the identification of the assessor: first name, last name; position;
- generic components of the work system;
- indication of identified risk factors;
- explicit description of the actual forms of occurrence of the identified risk factors (description, parameters and functional characteristics);

- maximal foreseeable consequences of the action of risk factors;
- severity class and foreseen probability;
- partial risk level.

vii). Proposed measures card is a form for centralizing the necessary prevention measures that are to be taken, resulting from the assessment of the workplace from the point of view of the existing risks of work-related accidents and diseases.

3. CASE STUDY: THE RISK ASSESSMENT AT VULCAN COLLIERY

In order to achieve the most relevant results by the application of this method, the first prerequisite is for the system to be analyzed to be a workplace, well defined as with regard to its purpose and its elements (working task, equipments, and environmental conditions). In order to provide a proper understanding, we considered a simple case study, whose aim is to evaluate the occupational risk for "Longwall undercaved bench face no. 4, seam 3, block 6, Vulcan Colliery". All the elements of such a simple working system are well-known, so they should not be listed here anymore. Before initiating the procedure, all members of the assessment team must have knowledge of the assessment method as well as, up to a certain degree, of the workplace to be examined. Another important condition is the existence of a complex and multi-disciplinary assessment team, to include work safety experts, designers, technologists, physicians, occupational medicine specialists, etc., corresponding to the varied nature of the elements of the work system, as well as of the risk factors.

The team leader should be the occupational safety expert, whose leading role will be to attune points of view of the other assessors, in the sense of the subordination and integration of the criteria used by each of them, for the purpose of the analysis: occupational safety assessment. Pursuing the steps and employing the working tools, the risk assessment card is completed, for the considered face from Vulcan Colliery in Valea Jiului coal basin, as given in table 1.

Table 1. Working place „*Longwall undercaved bench face no. 4, seam 3, block 6, Vulcan Colliery*" risk assessment card

<table>
<tr><td colspan="2">Company name:
National Coal Company</td><td rowspan="3">WORKING PLACE RISK ASSESSMENT CARD</td><td colspan="4">Number of exposed individuals: 8</td></tr>
<tr><td colspan="2">Unit: Vulcan Colliery</td><td colspan="4">Exposure length:
6 hours/shift</td></tr>
<tr><td colspan="2">Working Place:
Longwall undercaved bench face no. 4, seam 3, block 6</td><td colspan="4">Assessment team members:</td></tr>
<tr><td>Working system's element</td><td>Risk factor</td><td>Concrete occurrence form of risk factors (description, parameters)</td><td>Maxi-mum seve-rity</td><td>S</td><td>P</td><td>R</td></tr>
<tr><td>0</td><td>1</td><td>2</td><td>3</td><td>4</td><td>5</td><td>6</td></tr>
<tr><td rowspan="3">Equip-ments</td><td rowspan="3">Mechanical risk factors</td><td>1 Slip, falling, grip, muscle tension on work and travel route to/ from work;</td><td>ITM 3-45</td><td>2</td><td>3</td><td>2</td></tr>
<tr><td>2. Collapses, crashes, at workplace and travel route to/from work;</td><td>Death</td><td>7</td><td>1</td><td>3</td></tr>
<tr><td>3. Projection of rock or rock particles during breaking oversized blocks resulting from the blasting operation;</td><td>ITM 45-180</td><td>3</td><td>3</td><td>3</td></tr>
</table>

		4. Compressed air or hydraulic fluid jet under pressure to break hoses or high pressure air or destroying connections between hoses and pipes of pneumatic or hydraulic equipment (hammer, mechanical saws, drills, telescopic columns, pillars SVJ 2500)	ITM 45-180	3	3	3
		5. Hazardous areas or contours (stinging, sharp, slippery, abrasive) represented by wire binding elements used to support, unrocked material, wire mesh, mechanic chain saw	ITM 45-180	3	4	3
		6. Excessive vibration of technical equipment used in drilling mine holes	ITM 45-180	3	2	2
		7. Uncontrolled start-up or shut down of TR3 conveyors	Death	7	1	3
	Thermal risk factors	8. Flames from ignition of methane-air and/or coal-air mixtures	Death	7	2	4
	Electrical risk factors	9. Electrocution through direct, indirect touch or step voltage occurrence due to work in high moisture content environment.	Death	7	1	3
Working environ-ment	**Thermal risk factors**	10. Air currents and bad weather in underground workings.	ITM 3-45	2	5	3
		11. Air temperature differences on the route of travel to / from work	ITM 3-45	2	5	3
	Physical risk factors	12. High levels of humidity	ITM 3-45	2	5	3
		13. Noise at work (equipment operation).	ITM 3-45	2	5	3
	Chemical risk factors	14. Noxious gasses, smoke, gas fumes that may occur during operation of exhaust above the permissible limits (CO_2, CO, SO_2, NO_x, CH_4, etc.)	Death	7	2	4
		15. Gas fumes that may arise from explosions of methane and coal dust	Death	7	3	5
		16.Methane-air mixtures ignition and/or explosion	Death	7	3	5
Working task	**Physical overload**	17. Static load during orthostatic position due to work (muscular and skeletal)	ITM 45-180	3	4	3
		18. Forced or vicious working postures in confined spaces	ITM 45-180	3	4	3
	Psychic overload	19. Psychical load due to permanent mental effort to perform working operation operations	ITM 3-45	2	2	2
Worker	**Human errors**	20. Wrong execution of beams lifting operations, mining pressure control, disposal of coal	Death	7	2	4
		21. Wrong positioning at the assembly/disassembly of beams, coal evacuation, coupling/uncoupling chain of TR3, positioning posts under beams	Death	7	1	3
		22. Wrong handling of haulage equipment (winch 15 kW TAP, transporter TR3) and not using proper signals in accordance with the preset signal code	ITM 45-180	3	2	2
		23. The wrong positioning to equipment, compressed air hoses, hydraulic hoses	ITM 45-180	3	2	2
		24. Improvised connection of compressed air hoses or hydraulic fluid	ITM 45-180	3	3	3
		Etc: Due to limited space availability, we do not list all the identified risks.				
	Omission	42. Omission of an operation mentioned in the working procedure	Death	7	2	4
		43. Failure to use protective equipment	Death	7	2	4

The overall risk level of the working place is given by:

$$N_{rg} = \frac{\sum_{i=1}^{32} r_i \cdot R_i}{\sum_{i=1}^{32} r_i} = \frac{4 \cdot (5 \times 5) + 8 \cdot (4 \times 4) + 22 \cdot (3 \times 3) + 8.(2x2) + 1.(1x1)}{4 \times 5 + 8 \times 4 + 22 \times 3 + 8x2 + 1x1} = \frac{459}{137} = 3{,}35$$

The overall risk level calculated for the „*Longwall undercaved bench face no. 4, seam 3, block 6, Vulcan Colliery*" workplace is equal to 3.35, value that falls within the category of jobs with low to medium risk level, without exceeding the maximum acceptable limit set by definition to 3.5. Analyzing the data included in the risk assessment card, it comes that from the 43 identified risk factors, only 12 are exceeding the acceptability domain (see Figure 3).

It is essential that action is taken as a result of the findings of the risk assessment. Risk assessment should never be just a paper exercise; the entire process will have been a waste of time if the findings are merely noted, but no action taken as a result. Risk assessment should be an integral part of the company's safety management system.

The assessment will almost inevitably result in recommendations for improvements and further actions to control and reduce risk; any identified new or additional risk reduction measures or risk control systems must be implemented. Table 2 summarizes some of the proposed prevention measures, approached in a hierarchical manner, for the car-washer case.

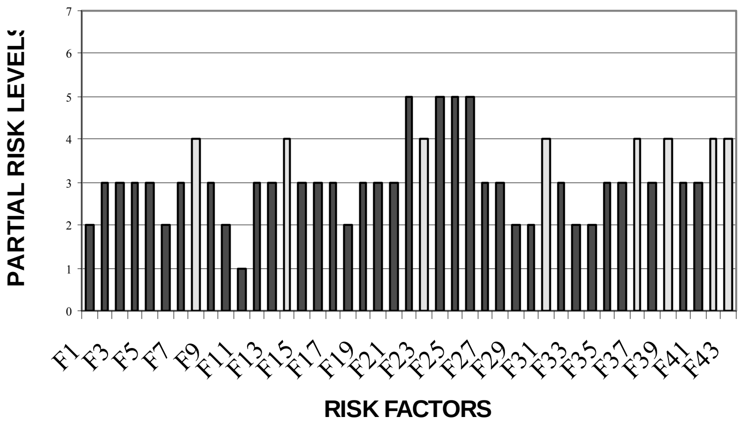

Fig. 3. Partial risk levels of risk factors for the working place „*Longwall undercaved bench face no. 4, seam 3, block 6, Vulcan Colliery*"

Ranking the measures according to risk levels will strongly suggest to the stakeholders that action priorities should follow a precise pattern. Furthermore, the envisaged measures will be part of an extended prevention/protection program, including deadlines and accountabilities.

The distribution of risk factors on generating sources is as follows (see also Figure 4):

- 34.88 %, risk factors characteristic of equipments;
- 25.58 %, risk factors characteristic of working environment;
- 9.30 %, risk factors characteristic of working task;
- 30.23 %, risk factors characteristic of worker.

Table 2. Proposed prevention measures in the case study (excerpt)

PREVENTION MEASURES CARD			
Working place „*Longwall undercaved bench face no. 4, seam 3, block 6, Vulcan Colliery*"			
No	**Risk factor**	**R**	**Envisaged prevention measures**
1.	F22: Airborne sillica dust particles	5	**Technical measures:** • providing water supply and means for combating dust in dust generating points; • providing efficient dust mask dust when generating activities are performed. **Organizational measures:** • regular basis measurements of dust concentration in workplace air; • training staff to the risks induced by silica content airborne dust.
			Technical measures: • use of gas detectors for automatic control of the workplace atmosphere.
No	**Risk factor**	**R**	**Envisaged prevention measures**
2.	F24: Toxic gasses and vapours issued as by-products of methane and/or coal dust explosions	4	**Organizational measures:** • tracking chart to check the means of protection of equipment (both technical equipment and personal protective equipment); • any intervention to electrical equipment will be performed by trained and authorized members of staff; • tracking chart to check the means of protection of equipment (both technical equipment and PPE).
3.	F25: Methane-air mixtures ignition and/or explosion	4	**Technical measures** • use of gas detectors for automatic control of the workplace atmosphere • proper methane drainage system operation; • adequate ventilation provision **Organizational measures:** • staff training on transport and handling of toxic substances in a safe and regular way; • more rigorous checking on how worker comply with safety restrictions and technological discipline.;

The application of this tool into the enterprise will allow:

- to identify all risk factors at workplaces, operation that is necessary to draw up enterprises own instructions concerning safety at work;
- to scan the existing situation of each workplace, in such manner as to ascertain acceptable risks;
- to ascertain the risk levels at each workplace, as well as their hierarchy;
- to set priorities regarding prevention measures for each workplace, respectively the optimal utilization of resources assigned for such purpose;
- to set a hierarchy of workplaces from the point of view of hazards and noxiousness;

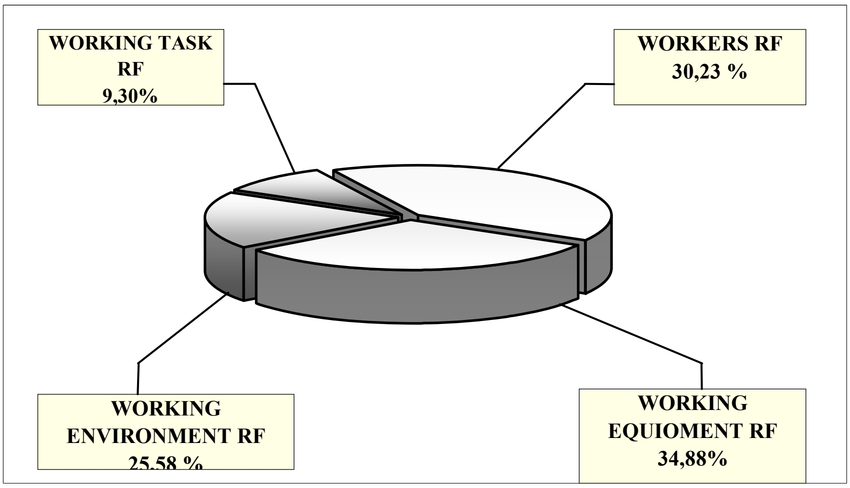

Fig. 4. Weighting of identified risk factors by generating source within the working system

- to compare different workplaces as with regard to occupational accident and disease hazards, with applications for the optimal use of economic lever factors;
- to manage workplace risks with computer-aided techniques, if reliable databases are available.

4. CONCLUSION

The compulsoriness of risk assessment at workplaces in our country resulted from the current law of Romania for this domain, which was harmonized with the legislation of the European Union concerning health and safety at work. The method presented in the paper was approved by the Ministry of Labor and Social Protection and experimented until now in the industrial field, for more than 2,000 workplaces. Furthermore, based on this method, a significant number of safety practitioners have been trained and authorized as assessors. The process of assessors' training is currently developing.

The methodology presented is a development of the approach for calculating individual risk as with elements of the procedure usually undertaken in order to construct a risk matrix. In principle, the tool can be used in a comparative sense in order to judge the effectiveness of proposed risk reduction measures. In practical risk assessment, a simplification of the actual contributing factors is often necessary. Essential factors that should always be considered include the frequency of use, the probability of a hazardous interaction in each occurrence of use and the probability of an injury in each hazardous interaction. Knowledge about human characteristics, perception, cognition and behavior is needed to substantiate assumptions about risk awareness, the likelihood of a particular sequence of activities and users' acceptance of risk. The estimation of likelihood is the main source of uncertainty in this risk assessment method. A necessary condition is that common criteria are used to assess the acceptability of the risk; this is a challenge for further development.

Risk assessment should not be a one off activity, but should be ongoing and be part of the process of continuous improvement. However, used appropriately, a risk assessment is the cornerstone of a successful health and safety management system and provides an invaluable tool in helping to ensure a safe working environment. The method is a qualitative tool, very useful to provide the prevention measures implementation monitoring and, on the other hand, it is a participatory and, consequently, a didactical method. Employed on teamwork basis the method can be a valuable tool, while it is not complex and can facilitate a thorough analysis of elementary risk constituents.

In order to design sound measurement scales it is important that the dimensions proposed reflect the mapping of risk assessors' qualitative judgments onto a scale of quantified options for scoring their judgments. Therefore, the values represented by rating scales should fit with the intuitive perception of risk assessors. There are several pitfalls, with whom the users of these methods may be confronted. The resort to this tool can create a false feeling of safety, induced by the fact that the risk was identified and assessed. But, it is not enough to know and quantify the risk; this means not that risk vanished. While risk analysis is a continuous, dynamic process, assigning some numerical values can not be a goal in itself. On the contrary, this stage should be the starting point for new actions.

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STUDY ON THE PHENOMENON OF CRACKS IN SALT ROCKS FROM SLĂNIC PRAHOVA MINE

ROTARU RALUCA[*]
ARAD VICTOR[**]
APOSTU SUSANA[*]
BARAIAC OANA[*]

Abstract: *One of the main mechanisms that determine the deterioration of salt stability is the occurring of fractures as a result of the influence of mining pressure. Aspects of rock salt cracking can be observed both in situ and laboratory tests on rock samples collected from the mining works. Cracking parameters are determined for different conditions: age pillars, excavation technology, how has the bearing size, depth of mining and other random aspects. The geomechanical assessment of underground workings made in rock salt massif from Slănic Prahova, concluded that salt is homogeneous. Phenomenological approach assumes that all information can be obtained from experiments performed on test samples whose dimensions are large compared to the asperities. Effectively formation of cracks in salt during deformation is very difficult to observe it. This requires a statistical analysis to predict the occurrence of cracking phenomenon based on laboratory tests and to correlate these results with existing theoretical models.*

Keywords: cracking, salt rocks, statistical analysis, geomechanical characteristic

INTRODUCTION

The most geomechanical assessment of underground structures in rock salt assume that salt is homogeneous. This approximation is accurate enough when the geometric extent of heterogeneity is small compared to the size of the modeled structure. The purpose of this paper is to discuss the main aspects of fracture for safety in underground salt in the presence of cracks [2].

Possible use of mines and salt cavities for various purposes, e.g. toxic or radioactive deposits etc., provided the motivation for enhancing the analysis of rock salt properties, mechanical behavior and insulation properties. One of the reasons for using cavities made in the formation of salt deposits - in contrast to other rocks - is its potential to "heal" fractures due to its rheological properties.

Thus, the main focus of analysis of the underground excavations stability carried out in salt has been traditionally investigation and description of creep deformation [3].

∗ *PhD. Stud. Eng.- at the University of Petrosani*
∗∗ *Prof.Ph.D. Eng..- at the University of Petrosani*

Ensure the stability of underground excavations is a priority in mining both economically and in terms of security personnel and existing equipment underground. This objective can be achieved through proper design and execution of underground excavations geometry and support their continuous surveillance [1], [7].

1. PHENOMENOLOGICAL APPROACH OF THE FRACTURING PROCESS

The phenomenological approach assumes that all information can be obtained from experiments on samples whose dimensions are large compared to the roughness. For salt, rough could include salt impurities contained in the matrix micro cracks pre-existing or newly formed [2].

The knowledge of natural and artificial fragmentation of a solid to characterize in a certain structure and texture becomes necessary practical and applied in mining, which is derived from the major influence it exerts massive on geomechanical characteristics, by affecting the stability of underground constructions felt the stress distribution inside the massive and mining pressure regime at a time, to improve the technical and economic indices of extraction of useful mineral substance.

The structural characterization of the massif in the context of stability, reliability mining involves knowing the following structural defects that are to be taken into account geological studies: plans or shear sliding, cracking, fracture and falier of the massif, layering etc.

The texture of the massif will characterize the arrangement and combination in space of the massif of rock components or mineral deposit of useful, i.e. orientation and depth extent of fragmentation or separation [1], [5].

The degree of separation is an important parameter in the characterization of solid rock texture because value can vary from one to another and as such massive blocks need not be entirely separated from the other by which it borders.

Degree of cracking		Cracking Sizing	
Category	Category name	specific (cracks/m)	average block [m]
I	Highly fractured massif (blocks)	>10	to 0,1
II	Massif strongly fractured (medium blocks)	2-10	0,1-0,3
III	Cracking medium massif (big blocks)	1-2	0,5-1,0
IV	Slightly cracked massif	1-0,65	1,0-1,5
V	Virtually monolithic massif (excessive blocks)	<0,65	<1,50

A system consists of cracks which cannot intersect each other. If the cracks intersect sharp right angles to each other, then they must be assigned to different systems. Depending on the angle that intersects and connecting systems failure guidelines, you may encounter: conjugated systems -the cracks intersecting at angles of about 90 ° and changing the orientation of a system change is accompanied by guidance and other and individual systems.

The crack density is defined as the average number of cracks of a system (with allowable deviations from parallelism of ± 10% and for the azimuth) per unit length measured perpendicular to the direction of cracking. Denoting by *n* the crack density, then the relationship can be expressed analytically: $n = 1 / d$, appointment which enables this parameter linear mode of failure of a system and it is a comparative assessment criterion for cracking a system to another [6].

The mechanical properties with implications in underground excavation stability: uniaxial compressive strength, tensile strength, shear strength, cohesion and internal friction angle. There were determined rheological properties of salt, of which an essential role in ensuring the stability we have: long-term resistance limit to compression, tensile respectively.

Geomechanical characteristics of salt analyzed are shown in Table 1.1.

Table 1.1. Geomechanical characteristics of salt at Slănic Prahova

PROPERTY	U.M.	LEVEL		
		IX	X	XI
Specific density,	X 10^4 N/m^3	2,164	2,192	2,258
Volume density, $_a$	X 10^4 N/m^3	2,086	2,125	2,125
Porosity	%	3,462	3,343	3,319
Compressive strength, σ_c	MPa	25,053	23,907	28, 876
Tensile, σ_t	MPa	1,563	1,874	2,689
Shear strength, σ_f	MPa	9,059	6,63	10,5
Cohesion, C	MPa	5,1	5,1	5,2
Angle of internal friction,	[]	30	33	30
Elasticity modulus, E	MPa	2838	2889	3000
Breaking dangerous strain,	%	6,33	7,08	7,25
Strain rate,	% /day	-	-	0,00041
Long-term resistance limit to compression, σ_{lldc}	MPa	15,44	18,56	19,8
Long-term resistance limit to tensile, $_{lldt}$	MPa	11,048	11,07	11,46

Conducting any excavation in rock massif has the effect of redistributing natural state of tension. The components of this state are regrouping focusing on certain critical issues excavations center. If rock around the excavation is strong enough to withstand these newly created tensions induced - secondary state power, then there will be no major cracks and fractures, surrounding rock does not change capacity and excavation will not need any support or building, it will remain stable. Such situations are common. Rock around excavations behavior depends on the following factors: natural state and secondary voltage, size and shape of excavation, the resistance of rocks, excavation method, etc. [1].

Macro fractures, especially faults, always affect local conditions demand. For example, Parker and Scott have established that the pillars located outside macro fractures area are subjected to stresses in excess compared to those located right in those areas.

This situation is attributed to the fact that along fault planes, movements occur which result in a relaxation of efforts from a pillars in the area and transmission of efforts on the pillars of macro fracture neighborhood [3].

2. BEHAVIOR OF PHYSICAL MODELS OF THE OPERATING ROOM SYSTEM - SQUARE PILLARS LONG-TERM STRESS – RHEOLOGICAL

In most situations concerning methods for determining the stress-strain state, methods and means of ensuring the stability of mining works, the salt massif is regarded as elastic, plastic, elastic-plastic or clusters, which naturally excludes from consideration <u>the time factor</u>, and hence <u>the rheological behavior</u>. However, all research and observations in this regard draw attention imperative to consider the influence of time as a key factor in ensuring stability in economic and technical conditions required for mining. All rheological research shows that massive salt deformation occurs over time in a continuous extremely slow [3].

2.1. Laboratory research

To elucidate time behavior, rheological, about dry application system under load through the rooms and pillars, has been made of rock salt model: time scale αt = 500, linear scale αL = 1:320, scale volumetric weight αγa = 1. Dimensions of the model are found in Table 2.1.

Table 2.1 Dimensions of the model

Label	Value
Pillar width (mm) Upper floor Lower floor	50 50,324
Room width (mm) Upper floor Lower floor	44 43,93
Slab thickness (mm)	25
System size (mm)	94

The physical model so constructed was within the limits of tolerance of 2% total height of the model was 127.5 mm, model consisting of two stages. The model was subjected to various degrees of stress which increased in steps.

During testing were measured shortening pillars. Trials lasted 22 days, at the end of which the degree of stress i = 0.827rc large cracks appeared, which led to the rupture model finally. The pillars were peeling their outer side and intermediate floor cracked and then broke. The results obtained were mapped creep curves for levels of stress i = 0.4, 0.5 and 0.82.

Rheological research on models of salt aimed at explaining the influence of time on the stability of the room - pillar. It was found that for specific tasks lower breaking load, or limit the long-term strength σt ≤ σηd pillars stability is not compromised. But after overcoming the values of σt> σηd (degree application j = 0.50) system starts to sag, the pillars suffer slow progressive deformation, the relative instability, which can occur after partial or total failure when exceeding the limit deformation capacity.

2.2. Rheological research based on in situ measurements

Fully elucidate the deformation behavior over time - rheological - the rock salt can be achieved by making observations and measurements of deformation of the resistance elements and in particular the pillars. The set is motivated by the fact that when applying the method of operation rooms and pillars, the main criterion of respect for stability - reliability and safety in operation is related to pillar.

On this basis therefore the main focus of research undertaken at Slănic Prahova salt mine was and is focused on elucidating their behavior in terms of deformability in time and their sizing.

Table 2.2. Values obtained from measurements of deformation mode of the pillars

Pillar	Coef.	Point of deformation	Total deformation	Deformation speed				Average speed of deformation mm/h
			Value of total deformation, ε_t	197 ore	287 ore	243 ore	242 ore	
A	0,71	0,349	0,189	0,162	0,578	0,211	0,0262	0,0339
B	0,66	0,209	0,00465	0,092	0,073	0,0298	0,0296	0,0008
C	0,7	0,228	0,377	0,34	0,282	0,326	0,305	0,0675
D	0,65	0,228	0,499	0,5	0,38	0,064	0,54	0,0865
F	0,66	0,278	0,13	0,13	0,18	0,049	0,074	0,021

CONCLUSIONS

- Knowledge of the structure, texture and composition of the salt is a necessary step in the reconstruction of the processes that generated and affected accumulation of salt over time.
- Deformability of rock in general and rock salt, in particular, depends on the depth to which are, by nature petro graphic, physical factors such as temperature, pressure, size and duration of action of tectonic forces. After how these factors interact, the same rock acts either as a plastic medium or the environment rupture.
- Salt massifs have a particular tension due to weight small volume compared to the surrounding rocks. The consequence of these differences is isostatic redistribution in relation to surrounding rock salt; salt plasticity is an essential part of it.
- The main factors influencing the stability of the largest underground excavations are: transverse profile shape and dimensions of the excavation, the main initial stress value and their reorientation towards mining work; thick layers of rock in the roof and the party was made excavation and geomechanical characteristics thereof; tectonics, cracking and orientation crack groups, presence of support and technical characteristics etc.
- Following geomechanical determinations made on samples of rocks from the massif Slănic Prahova, there were no major phenomena of instability that could be caused by cracks in the rocks of solid products.
- From the point of view of application, all existing pillars to Slănic Prahova salt lie in the stability of creep deformation behavior that time.

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SUSTAINABLE DEVELOPMENT AND MINING

BĂRĂIAC OANA[*]
ARAD VICTOR[**]
IANCU (APOSTU) SUSANA[*]

Abstract: *Sustainable development is a holistic concept that addresses two major problems of humanity: the ability to create and to destroy. Sustainability requires efficient management of resources; reduce waste loads of eco-system and not least to meet the needs of society and future generations. According to the "Mining, Minerals and Sustainable Development, 2002"report: "One of the greatest challenges facing humanity today is integrating economic activity with environmental integrity and social concerns". Mining has changed significantly in recent years. The incidents occurred outside the country and also inside led to discussions at the highest level, which resulted in several briefings and reports. In this way, it was a better understanding of the relationship between mining and environment. Nowadays, mining faces challenges, like: reducing deposits accessible, administrative burdens and high costs that make it difficult to obtain authorizations of extraction etc. Mining must not endanger the natural life support systems - air, water, soil, flora and fauna. Exploitation of useful mineral deposits can generate significant progress and welfare, but they are exhaustible and non-renewable.*

KEYWORDS: *sustainable development strategy, efficient management, environmental and economic reconstruction, objectives, institutional policies.*

1. INTRODUCTION

The 21st century is a time when economical, social and environmental challenges are met everywhere. Consequently, under these circumstances concepts like: energetic efficiency and security, climate changes, rural development, nonpolluting technologies, efficient management etc, are more and more often met in political speeches of the European and international institutions – G 20.

Sustainable development concept sum up all the above mentioned aspects, concepts that concretely suppose having in view for all the measures regarding sustainable development, the impact these measure trigger on long term or to be able to refer to the well known definition given by "Brundtland Report" or "Our Joint Future" elaborated by the World Commission on Environment and Development in 1872 and officially adopted globally during the Rio de

∗ *PhD. student at University of Petroşani*
∗∗ *Prof. Dr. Eng. at University of Petroşani*

Janeiro Summit of 1992: "the ability to satisfy the demands of the present generation without compromising the ability of the future generations to satisfy their needs."

At the Rio Summit of 1992, the key issue of sustainable duration was defined and namely – the opposition between the population growth needs and the limits imposed by the planet resources. The sustainable development is seen as "a new development way that should sustain human progress for the entire planet and for a longstanding future." The States present agreed to a sustainable development plan named Agenda 21.

The sustainable development setting in the vision of the Commission for Sustainable Development has four main dimensions: social, economical, environmental and institutional which include a series of topics and subtopics. This paper emphasizes the environmental dimension and the institutional one in the context of exhaustion of the non-renewable resources and of the rapid climate changes. The economical development must be settled with absolute necessity of valuing the environment, as well as offering social development alternatives.

2. MINING INDUSTRY STRATEGY

In order to answer to the needs of the sustainable development at European and national level a series of strategies have been elaborated, one of which is the mining industry strategy that extends to the year 2020.

The development strategies make possible the dynamic coordination of the development processes that happens on the territory, taking into account the local aspects, the changes in the external environment and make possible the proactive approach carried out in an innovative way, of the territorial development processes.

The strategic plan of socio-economical development, which hints at the mining industry as well, and which is actively elaborated in a geographical place / region is an important tool of territory management that contributes to a more coherent and practical local "government" so that the life of its citizens become more prosperous, the place become attractive for entrepreneurs, the population grow and become responsible of assuring the future, the changes be beneficial so that mass transfer, energy and information between generations should represent evolution.

The overall objective of the energetic department strategy is the attainment of the energy need not only in the present but also in the medium and long run, at a price as small as possible, appropriate for a modern market economy and a proper life standard, under quality and safety diet conditions, obeying the sustainable development principles. [5]

The energetic strategy has in view to achieve the main objectives and measures of the sustainable development policies in the energetic field, these objectives being accepted by Romania as well once our country integrated to the European structures [5]:

- the growth of energetic efficiency;
- supporting the production of the energy based on renewable resources;
- supporting the production of electrical and thermal energy in plants with co-generation, mainly in highly efficient co-generation installations;
- supporting of the research – development activities and distribution of the applicable research result;
- decreasing the negative impact of the energetic department on the environment;
- rational and efficient use of the primary energetic resources.

In the context of economy globalization, any energetic strategy must be carried out according the changes that occur in the world.

For the mining industry, the development strategy promoted previous to year 1989 had as a main purpose the reduction of imports according to the concept of self-support of the economy with mineral resources. The result was not beneficial at all as there was development of a mining field more than the potential allowed by the solid mining resources economically exploitable that the country had. After 1989 the mining field was supported by the state, and this required a considerable budget effort.

The insecure condition of mining between 1990 and 2005 eventually led to the reorganization process and this led to a series of new issues, such as [4]:

- the sudden fallback of the economy in the mining areas affected by the reorganization of the sector;
- growth of the social problems in these areas;
- growth of the environmental problems;
- increasing of poverty.

The mineral resources that can be exploited in the various stages with the present mining technologies are shown in Chart 1. The geo-mining conditions of the deposits in Romania are complex, and the parameters regarding quality are at the lower end in comparison with the similar deposits exploited nowadays worldwide, with upgraded technologies and productivity 5 to 12 times higher [4].

Chart 1. The situation of the base of solid mineral resources exploited and the degree of state intervention

Sustance	**Mineral Resources**		**Degree of State Intervention**
	M.U.	**Quant.**	
Lignite	mil. tons	2800	Low, at granting funding for underground exploitation of social transfers and partly capital allowances
Pit coal	mil. tons	900	Major, by granting funding for exploitation, social transfers and capital allowances
Gold and silver – bearing deposits	mil. tons	40	Very important, by granting funding for exploitation, social transfers, capital allowances and debts grading to electrical energy suppliers.
Polymetallic deposits	mil. tons	90	
Copper bearing deposits	mil. tons	900	
Uranium deposits	mil. tons	*	Major, by granting funding for exploitation, social transfers and capital allowances
Salt	mil. tons	4000	Without intervention
Mineral water	mil. m^3/day		Without intervention

*the quantity of resources as a special plan; [4]

3. MAIN OBJECTIVES OF THE MINING FIELD STRATEGY

In the context of the world and national economic situation Romanian Government established the following main axis of action as objectives in order to attain the claims of sustainable development.

1. The approach of the mining industry activity according to the principles of free market [4**]**, which involves:

- mining products capitalization according to the free market, competing with any other internal and external suppliers;
- reconsidering the exploitation assertions in order to concentrate extraction in the most productive areas;
- carrying out mining production with competitive costs;
- optimizing the personnel number and its wage so that the exploitation should work efficiently;
- renewal, rehabilitation and using new technologies for the sustainable mines so that to facilitate the transfer of the exploitation permits to private operators.

2. Decreasing the Government direct involvement by gradually bringing investments from the private sector [4], taking into account:

- restructuring the production abilities and technological performances improvement, stopping the activity and closing nonviable mines;
- eliminate production grants and those for social security in the mining and lignite field starting with 01.01.2007;
- granting state support for the pit coal department, according to the provisions in Directive 1407 / 23.07.2002 / C.E.;
- granting state support as capital allowances for viable mines and with practicable conditions in the mining and lignite field;
- promoting the state – private partnership to ensure the financing resources necessary for the development and modernizing in order to privatize viable mines and those with practicable conditions;
- promoting a market-orientated management and economical efficiency;
- granting state support as capital grants and allowances for exploitation of the uranium deposits and refining technical concentrates.

3. Carrying out mining activities under environment protection conditions [4], and this has in view:

- inventory of the environmental damages caused by the mining activity in the period prior to receiving the exploitation permit in order for the Government to bear the responsibilities resulting from it until the concession granting moment;
- assessment of the potential impact on the environment by the mining activity, in order to settle the obligations of the mining state-hold operators and private permit holders;
- drawing up the environment protection manual in mining industry to ensure the framework which should offer the certainty of promoting an environment management at European standards;
- revision of the mines closing manual to ensure the framework that should guarantee the permit holders responsibility to the obligation to restore the environment and to their social obligations in the process of mines closing;
- improvement and completing institutional and policing framework that should ensure the monitoring of carrying out the responsibilities of the permit holders in the environment field and regarding society;

- fulfillment of the conformation schedules provisions negotiated by the mining operators with the Environment Protection Agency, in order to get the environment permits.

4. Carrying out mining activities under work security and health conditions [4]:

- drawing up programs for increasing the degree of work security and health;
- increasing the level of work security in the pit coal underground exploitation;
- improving the health of the employees working in pit coal exploitation;
- creating an organizational scheme able to ensure a nationwide rescue service.

5. Lessening social problems triggered by uneconomic mines closing and revitalization of the economy in the affected mining areas [4], having in view:

- promoting the individual and mass dialogue to inform employees regarding the situation and perspective of the unit they work in;
- discussing with the affected staff about the most suitable methods of social protection that are going to be adopted;
- promoting training opportunities for the employees in order to ensure their chances on the work market;
- developing programs for community works for temporary hiring the unemployed;
- social protection for the fired personnel that have small chances of finding a new job;
- using again spaces and assets that became available when the mines were shut down.

4. POLICIES NECESSARY TO FULFILL THE MAIN OBJECTIVES OF THE MINING FIELD STRATEGY

The objectives of the mining field strategy can be fulfilled if enough funding is granted in order to finance their implementation, and this can only be attained by adopting some legislatives policies that could allow this. Below are presented the main policies that must be adopted by our leaders [4].

1. Policies regarding the reconsideration of institutional structures and of the abilities present in the mining field.

The difficulties mining industry in Romania is faced with are the direct result of the inefficient of those involved in mining, which leads to taking new decisions such as the stopping and closing of the mines that have no productivity and at the same time redirecting the funds towards new technologies, modernizing the mines that can make a profit or can become sustainable in the mere future.

These policies include actions such as: revision of the legislative framework, the gradual attenuation of the state's role in the exploring and exploitation activity, drawing up some public policies for the mining sector, and so on.

2. Policies regarding the removal of the financial loses and the growth of the economic efficiency in the mining sector.

Nowadays the Romanian state is forced to assign considerable financial resources for the mining sector, the activities it implies, and all due to a weak management of the mining operators where the state holds a majority stake in the pit coal and ore department.

An efficient management implies the cutback in the funding and losses elimination, by applying actions such as: stopping the activity of inefficient mines, ecological and economical reconstruction of the affected areas, the rise technological performances, and so on.

3. Policies regarding environment reconstruction and protection

Mining operators where the state holds a majority stake do not have yet enough training and financial resources to supervise and execute environment reconstruction works, which determines, in the shortest possible time, the line up to the international standards of solid mineral resources exploitation under environment protection conditions through programs and projects especially destined to this purpose.

4. Policies regarding the improvement of work health and security

Generally speaking these policies require actions such as: personnel training and progress, implementation of special programs especially destined to work health and security, creating a national level mining rescue structure.

5. Policies regarding social protection of the personnel affected by reorganization

Due to the reorganization process that mining activity faces, the most affected was the personnel of the mining operators whose mines are shutting down, as well as the personnel that continue working. The sustainable development implies social policies that could help the personnel made redundant and those who are about to be made redundant.

Institutions such as the Romanian Agency foe Sustainable Development of the Industrial Areas, The County Agencies of Work and others have the role to assess on permanent basis the socio – economic situation in each mining area, in order to avoid the possible difficulties imposed by the reorganization process.

This assessment activity deals with the implementation of the following social protection measures: social protection for the personnel made redundant that have small chances of finding employment, building up sustainable alternatives for the youth, supporting a program for redundant persons to help them get access to the work market, professional training for the personnel made redundant as well as for the hired personnel, information access for the employees regarding the situation and future of the company, constant supervision of the implementation effects of social protection policies for the personnel affected by reorganization.

6. Policies regarding socio-economic recovery of the mining areas affected by reorganization

In the important mining areas of our country, the mines closing generated the poverty of the population as a direct result of incomes drop, which led to a drop in the local economic resources, as well as to the income decrease in local budgets.

National Agency for Mining Areas Development supports the local authorities' initiatives to promote economic recovery strategies which have as purpose a growth in their access to financial resources from the structural funds, as a result of the adherence to the European Community.

In order to introduce the policies regarding economical reconstruction of the affected mining areas, there are stipulated a series of actions and measures, as it follows: the environment quality improvement and that of the infrastructure, integration in the economic circuit of the areas that include technologic annexes and rehabilitated grounds, active strategic planning in order to attract investments, community involvement, exploitation of local resources in the agricultural, forest, piscatorial and touristic field, improving the living standards by restoring infrastructure and utilities, promoting a positive image of the area, and so on.

5. FUTURE PERSPECTIVES

The areas affected by the sector reorganization will stay, for a certain period of time, highly dependent on the mining activity, both through the direct and indirect work occupation level and through the resources that are going to be brought by this activity.

Therefore, the elaborated strategies mainly aim at the continuing of the socio-economical renewal process of the mining areas, of economical reconstruction, supporting the investors regardless of the property form the rehabilitation and modernizing the technology used and the mining activities efficiency growth [4].

The main activities appointed for this purpose are the following:

- Transformation of mining in those areas into a modern and sustainable sector;
- Greening of the areas affected by the mining activity, in accordance with the territory systematization program in order to be identified and capitalized in the interest of the economical development of the area for new opportunities;
- Building the European road lanes that cross those areas;
- Developing tourism as an alternative to the traditional mining activity, in the areas where conditions are favourable for such a development type;
- Youth professional training according to the requirements of the work market triggered by qualification needs of the businesses that will develop in the area;
- Promoting technological development, technologies transfer as an alternative to mining;
- Fighting chronic unemployment through actions meant to occupy the community activities in those areas for those people that are excluded from hiring by the new investors on account of age, sex, professional abilities, training level, and so on.

6. CONCLUSIONS

Industrialization period has been overcome and the future supposes that the main assessment criteria for the quantitative-type developing processes, concerning volume, number and achievement should be replaced by responsibilities, precautions, change and finally growth.

Sustainable development implies promoting energy from renewable sources, impulsion of investments to rise energetic performance, using clean technologies, stimulating research, using geothermal energy, biogas, liquid green fuels and least but not last ecological reconstruction of the areas affected by mining industry.

The requirements imposed by the European Union sustainable development policy within the mining subsector of the energetic sector, in order to link the production abilities to the coal demand in order to produce electrical and thermal energy can be fulfilled only by implementing the following provisions:

- Concentration of production in the sustainable areas and shutting down of the unprofitable areas securely;
- Modernizing and rehabilitation of the equipment in pit coal and lignite mines, which are sustainable;
- Promoting new technologies and equipment modernizing for exploitation of mineral resources;
- Capitalization of the methane in the pit coal deposits;
- Promoting work security and health programs for the personnel in the energetic sector;
- Giving a boost to geological research in order to increase the knowledge regarding coal deposits and their valorization;
- Shutting down mines that have no activity, surfaces rehabilitation and restoring the affected natural environment;
- Conversion of the manpower and recruitment and training of qualified personnel in order to cover the manpower need in the exploitation sector;
- Economic revitalizing of mining regions;

• Providing for coal production sale through medium and long term contracts with negotiated prices according to formulae established according to the market quotations of similar carriers of primary energy;

• Access on the financial markets to assure the resources necessary for the equipment modernizing and rehabilitation and to bring new equipment to mines;

• Granting the necessary funds through the national plan of geologic research to increase the extent to which the identified geological resources are known;

• Assessment and capitalization of mineral resources in the coal heaps with poor ore and in the drainer pools;

All these provisions are added up with a series of terms that belong to the legislative and organizational side, which are mentioned in The Energetic Strategy of Romania, having the following main requirements:

• Changing and completing Mines Law 85 / 2003;

• Making the regulation framework for mines closing a priority;

• Promoting the normative document regarding overtaking of the rehabilitated grounds resulting from the closing of mines by the local public authorities;

• Legislating the norm regarding the reorganizing of the pit coal sector in order to build up a sustainable organizational structure;

• Elaborating some laws that should include actions prior to the removing and dumping works in the lignite pits in order to make available the grounds necessary for mining activities;

• Regulations regarding access to the necessary grounds for mining in case of opening a new uranium deposit;

• Passing some laws regarding revision the technical norms of work security linked with international norms and the performances in the specific equipment construction;

• Other regulations that are as important are those regarding financial restructuring, reorganizing in order to make them sustainable and to privatize them, closing of mines, etc;

• Drawing up and promoting the regulatory document regarding providing for the compulsory personnel structure for societies/companies that carry out mining activities, according to the provisions of the valid legislation;

Development programs for the communities must comprise energy production promotion based on producing energy with renewal resources, so that in year 2010, 24% of the energy consumption should be assured from renewal resources.

Investments play an important role in the improvement of energetic efficiency and they must be stimulated on the entire range: resources – production – transport – distribution – consumption.

An important role in this process is held by the research – development activity and discussions regarding enforceable research results.

A major importance is given to the campaign for "Green Field" projects carried out through the predicted investments in environment protection in Big Combustion Installations in the energetic sector by implementing the transaction mechanism for the greenhouse effect gas emission permits, by carrying out the management of the radioactive wasted securely, including the organization of the financial resources necessary for the final storage and disusing the nuclear-electrical units by providing new production capacities by using clean combustion technologies for the fossil fuels.

We must not neglect the decrease of the social impact on the underprivileged energy users.

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ASSESMENT MODEL OF A MINING PROJECT REGARDING LABOUR FORCE EMPLOYMENT

BUD NICOLAE *
SCHVAB MIHAI **

Abstract: *The paper aims to present an assessment model of a mining project value based on the aggregate income integrating criteria as defined by Keynes. For the purpose we make the transition from traditional cost structures and concepts to a new cost structure to allow the aggregate income calculation. The innovation of the approach involves expressing the quantity elements in two measure units, monetary and labour, and in end the final outcome is expressed as an employed work quantity function. This approach requires the understanding of Keynes's concepts and reasons in order to adjust them to the mining project features. For this purpose, we explained and exemplified, in order to reach a new on what mining project management means, which should include the dynamic of the reserve/resource factor value and relate to the economic system within which it integrates through aggregation. In order to apply Keynes's concepts and reasons for assessing the mining projects, we presented their particularities regarding the start and end of the project, respectively, "life" of a project. These specifications make it possible to correctly apply the reasons behind the short and long term decisions and previously quantifying the costs and incomes during the life and period of the project. During this paper we took in consideration the necessity to difference the mining projects from other business projects, given that the most important characteristic element of these projects consist in the project's life as a function of the resource assimilation degree.*

Keywords: *Project; The life of the project; Technical Capital; Mining project; Entrepreneur's expectations; Aggregate income; The aggregate contribution of the project; Net addition to the resources available and the physical capital;*

1. THE NECESSITY OF DIFFERENTIATION BETWEEN MINING PROJECTS AND OTHER PROJECTS BUSINESSES

In general the meaning of the concept of the project can be defined as "a complex, unusual, limited time, budget and resources with specified performances, designed to meet the needs of the beneficiary". [7]

* *PhD. Eng, PhD. Ec, Deputy in the Parliament of Romania*
** *PhD. Eng, Lecturer at North University – Baia Mare*

The project as defined above is different from the current activities by the following characteristics: a goal, a definite period of life, with a beginning and an end, usual involvement of several participants (institutions, organizations, compartments) and professions, typically is something new that has not been made in this form, specified durations and costs and related requirements for performance. Besides the characteristics regarding the involvement of several participants and a few professions, which may be considered as common to all projects, the other features have real particularities for mining projects.

One of the most important characteristics of a mining project refers to **the life of the project according to the degree of assimilation of the resource.** To initiate a project from a different domain, it is needed to be fulfilled **two requirements**, namely: market and money (market, money). In the case of mining projects it is necessary to **meet the requirement of the three M (mineral, market, money).**

In order to better align the 3 M, one has two differentiate between mineral sector projects as follows: prospecting's -first M has to be found; explorations, the discovered resource must justify attracting more money; development, the resource that has become reserve and the market with its trend must justify a first exploitation project.

In most cases, exploitation projects go beyond the classic simple cycle, opening, and exploitation of initially outlined and quantified reserves, followed by closing. The general case, especially for the minerals in the complex geological, consists of a succession of mining projects with increases or reductions in capacity, with the qualitative changes of reserves and of the conditions of use. There also may arise some temporary closures, creating situations in which deposit is the subject of some projects that go for centuries.

The life of mining projects is less defined than it looks, and to overcome this uncertainty in the assessment of mining projects one should make appropriate clarifications regarding the start of a mining project and the input values of the ***"technical capital."*** Reference is made to this concept introduced by Keynes in order to emphasize the fact that ***one of the unknowns of a mining project*** (without which the project does not exist) ***is the value of the reserves as a component of technical capital.***

Exploitation of a deposit is a succession of projects, with a choice of essential moments which reduce the time for carrying out complex projects. The pertinent question is thus: when do we find ourselves in front of a new project to define it as such?

Defining the mining project beginning is essential because its particularity, represented by the mineral reserve, turns out to be something extremely changing in time, being at the same time the decisive factor of success or failure.

Similarly, one should also take into account the issue of the **mining project ending**, item that is less defined within the large mining projects. This item is related to what may be understood by the generic term **"Entrepreneur's expectations."**

2. ANOTHER APPROACH TO THE CONCEPTS OF COSTS AND REVENUES WITHIN THE MINING PROJECTS

The approach takes into account the mining projects in the "development" phase and continuation of the exploitation activity requiring a new project. The two types of projects start from a technical capital and changes in conditions will lead to a new technical capital. For mining projects, the most important part of the technical capital is valuable only if the value is calculated on the amount of reserves.

Starting from the input data of the project (existing technical capital, technology and resources) hiring a certain volume of work over a defined period of project's life involves two types of expenditures:

1. the amount attributable to the project for the services of the production factors (land, labour and capital) without including the payments to other entrepreneurs for services rendered;

2. the amount attributable to the project, consisting of:

- The amount to be paid to other entrepreneurs for what purchased from them (the cost of the labour factor in question);

- The projects generated cost by simple use of technical capital (the cost of the employed labour).

The two amounts shall be deducted from the value of the sold production and the amount resulting in excess is the developer's revenue (profits expected following the project).

In the light of those described, one can say that, if the economic system does not provide satisfying information regarding wages, rents, interest rates, and especially the prices as basis for assessment of the two categories of costs, a project may appear as unprofitable, or by including it in a concern or branch, but it may be acceptable to the national economic system level.

To remove this drawback, it is necessary to use the aggregate income indicator for assessing a mining project. It consists of a quantity of employed labour, and it is also known as the benefit of that recruitment. Its value is obtained by adding the profit to the cost of the factor of production. The benefit of employing a quantity of labour in a mining project shows the aggregate contribution of the project, manifested through the following aspects:

- employees receive salaries and other benefits;

- suppliers and providers have receipts from the supplies and provisions of which they can earn important profits after they pay their salaries and other costs of the factor of production.

Through aggregation, some revenues become costs and all costs and revenue, although expressed in monetary units are based on a single price, the labour price or wage unit.

3. CHOICE OF THE UNITS OF MEASURE IN THE ASSESMENT PROCESS OF THE MINING PROJECTS

According to Keynes, one of the major problems related to the assessment of the projects lies in the choice of appropriate quantity units in accordance with the global economic system issues.

There will be proposed a series of units of measure, by which it is meant to resolve the following issues:

1. Correct measurement of the value of production by relinquishing the price distorted prices of the mining products

2. Measurement of net margin to the resources available in the economic communities and the current period sacrifices due to economic activities, after taking account of the real "damage" of the initial capital;

3. Measurement of net margin to physical capital;

4. Evaluation of new technical capital compared to the old one when due to changes in technology, the two are not identical.

The correct measurement of the value of mining production considers its value subject to a concern within which the producers is operating, which includes links to community located downstream and upstream of it in integration systems. This is the item referred to by the mining producers when they say that their place within the system is not properly evaluated.

The measurement of the net addition to the resources available and the physical capital mainly concerns reserves as a form of capital. There are issues that the current accounting system does not even approximately resolves. It is about the correct approach of problems involving: exploitation exploration, preparation for exploration, the depletion of reserves and investments for expansion and modernization. These problems concern the clear definition of capital, expressing the correct value and its periodic evaluation based on "expectations in the short and long term".

By their nature the results of mining activity allow for a comparison of action alternatives characterized by different aggregate yields which are characterized by different technical capital with different quantities of labor. To evaluate a mining project two units can be used for quantitative expression, namely:

- the unit of the amount of money, expressed in national currency, which has a strictly uniform character;
- the unit of the quantity of labor force, expressed in number of employed staff and which can be made uniform by more or less fixed relative remuneration.

The unit of measure for the quantity of labor force that is employed, is the work unit, and the nominal salary of the drive unit is called salary unit.

The Salary Fund is a homogeneous quantity which is obtained by multiplying the number of employees expressed in units by wage unit size. Expressing the number of employees in homogeneous working units is resolved by considering that individuals contribute to the supply of labor proportionally to their remuneration. The qualified work hours may be weighed proportionally to their remuneration. Starting from this mode of quantity expression one can determine the current output changes as a result of a new project, with reference to the number of hours of work paid for this output.

With the two units of measure there can be defined a function of the aggregate supply for a certain company, and similarly for a specific branch, or to a combination of economic activities.

- Zr-net cost benefit of use;
- No-volume of employed labour force.

You can write the following default relationship:

$$Z_r = \Phi_r(N_r) \qquad (1)$$

Expectations with regard to the benefit, i.e., the most important economic result expected from promoting a project that will change the employment level will depend on a Nr level of the employed labour force.

Between the volume of employed labour Nr and production Qr expressed in physical units there is the following relation:

$$Q_r = \Psi(N_r) \qquad (2)$$

Consider Ur (Nr), the function of the use cost expected for a given level of employment Nr. with these specifications work supply function retains the following format:

$$P = \left[Z_r + U_r(N_r)\cdot Q_r\right] = \left[\Phi_r(N_r) + U_r(N_r)\right]\Psi(N_r) \qquad (3)$$

In the case of each homogeneous item for which,

$$Q_r = \Psi_r(N_r) \qquad (4)$$

has a clear meaning, you can evaluate the function

$$Z_r = Q_r(N_r) \qquad (5)$$

Subsequently, in general, you can aggregate values No, but not the values Qr because the expression $\sum Q_r$ is not a numerical quantity. The aggregation will be achieved through the function P of work supply.

Through this mode of numerical expression one may resolve the problem of assessing a mining project compiling changes from upstream and downstream entrepreneurs affected by the new project.

This involves assessing the functions Ur, Zr and Qr for all consumer and capital goods influenced by hiring a labour force volume Nr in the mining project for a production Qr resulted from the employment of this work volume.

4. PERSONAL CONSIDERATIONS ON THE SOLUTIONS PROPOSED BY KEYNES REGARDING UNITS OF MEASURE

The approach to deepen the problem of measuring units prior to developing a wide-ranging economic theory can be perceived only if you ask the questions: why was Keynes preoccupied by such a problem? What "units of quantity" (the author's expression) did he want to replace?

The author makes the following remarks in the "*The General Theory of Employment, Interest and Money*":

-"The topic is discussed here for the reason that it simply hasn't been discussed elsewhere in the way that I consider to be appropriate for the needs of my investigation."

-"... we will try to clear up certain issues involved ..."

-"... the three dilemmas which have most prevented me from writing this book and because of which I couldn’t express conventionally until I found a solution for it are: first, the choice of appropriate quantity units for overall economic system problems; Second, the role played by the expectations in economic analysis; Third, the definition of income".

-"... units that economists usually work with are inadequate, which may be illustrated with the help of national dividend concepts, real capital stock and the general level of prices".

These quotes show that the units of measure posed a dilemma. Moreover, the question arises: are the problems of the economic system today resolved with the same units of quantity which Keynes never accepted?

The first indication refers to the concept of economic system. For us an economic system is a set of many entrepreneurs, concept used by Keynes referring to decisions, elections, expectations, and trend. How many and which entrepreneurs does a system include is a question relating to the investigator's ability to master the system’s complexity.

Keynes's merit is that he starts from the basic economic system, the developer as an open system. He aims to clarify aspects of the partially integrated macroeconomic system in order to get to the integrated system. Surprisingly, however, Keynes's theory was of interest almost exclusively at macro-economic level, probably due to political impacts, while contributions to solving the micro-economic problems have passed almost unnoticed.

The two units of quantity proposed by Keynes provide a solution of mixing when it comes to individual "entrepreneurs" for the whole diversity of natural units in which is expressed the production of homogeneous parts. He keeps the amount of money unit, but he doesn't use it to the express production quantities in money but to express the labour quantity in money.

Thus, aggregation is not made using a single measure unit, i.e. the quantity of money, but the aggregation based on consumption of labour force per unit of product.

After the author's assertion "When we wish to speak of an increase in production, we must rely on the assumption that the overall amount of manpower associated with a particular capital is an acceptable index of the resulting production quantity – the two increasing and decreasing together, though not in a predetermined numerical proportion"

To accept the mixing of the final products by price means to accept the imperfections of markets in price formation for all products. Instead, the "homogenization" by indexes of the labour quantity associated with a given technical capital through weighing the quantity of manpower in relation to remuneration, we accept only the remuneration distortion which is formed on a single market - the labor market.

In each situation, an entrepreneur decides to use a given volume of capital, which is the "constant" of the problem of evaluating the results of production activities. To this constant, without employing a volume of manpower, production will be zero and the technical capital at the start of a period without production can reach the end resulting in a certain cost.

This specification is an essential aspect of the evaluation of a project in general and a mining one in particular. Such is clarified how must be dealt the reserves at any time periods, in most cases, quite lengthy, which do not result in "final output".

Reserves form part, or, in some cases, even all the technical capital at an early stage that reaches the end of the period with another value through maintenance and enhancement activities through investment.

When starting up a project by reusing an existing capital, the provided capital should be reassessed at this moment in the life of the project and continuously after putting into operation at a prime capacity. This first capacity is not something arbitrary, but a value that can be accomplished with a minimum investment in technical capital and possibly a bigger capacity if the initial investment is justified by the "herself".

Studying the function of production (Q (N)), resulting in employing an ever-increasing amount of manpower, to a given technical capital we reach the problem of reduced efficiency of labour per unit of salary, i.e. a reduction of income from the production of technical capital.

Instead of believing that the units engaged in addition to a certain volume of labour are increasingly less adapted to use a given technical capital, there is included this non-uniformity of equally remunerated work units in the technical capital which is considered to be increasingly less adapted to employ additional units of work as production increases (for the given capital).

The extreme case is considered where different work units would be so specialized that they would be unable to replace each other. This means that the elasticity of supply of a certain type of capital drops sharply to zero when the entire workforce specialized to work with the technical capital would already be employed.

These items come to support the hypothesis of homogeneity of the work, as there is one case in which homogeneity is questioned, namely, that where there is a great imbalance in the relative remuneration of various units.

Such instability improves much faster on the labour market compared to price fluctuations and distortions.

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ASPECTS REGARDING THE STABILITY OF THE UNDERGROUND MINE WORKINGS FROM BERBESTI – HOREZU BASIN

TODERAŞ MIHAELA**
CHIRILĂ DĂNUȚ**

Abstract: *Horizontal mining stability analysis must primarily be based on the knowledge of the working conditions which, under the observations from the in situ measurements and the analytical calculs, must allow an assessment of the interdependence between manifestation forms of the mining pressure regime, rock massif stability and support parameters. Currently, such an issue should be approached simultaneously in two directions, namely: an analysis of the stability of the mining workings based on criteria of stability, analytical evaluation of the natural state of stress.*

Key words: *stability, criteria of stability, stress state, deformation, mining pressure*

1. THE MINING WORKINGS STABILITY ANALYSIS BASED ON CRITERIA OF STABILITY

In this stage, the literature offers a wide range of stability criteria, for the stability of the horizontal mining workings based either on the theory of static stability or elastic equilibrium theory or elastic – inelastic equilibrium such as plastic type, viscous – plastic type, etc. For the geo-mining conditions of Horezu basin, we selected the stability criteria given in Table 1 (Todorescu, A., Gaiducov, V., 1995, Chirilă, D., 2000).

Table 1- Criteria for analysis of the mining works stability - criteria selected and analyzed:

Category of criteria	Criterion designation / Author	Calculation relationship / Signification
Criteria for assessing the monostadiale loss stability	Criterion based on the evaluation of the secondary stress state of a massive with elastic behaviour Kerisel, Fenner	$\sigma_{\theta\max} = (3-\xi_0)\,\sigma_z \le \sigma_{rc}$; $\sigma_{\theta\max} = (1-3\xi_0)\,\sigma_z \le \sigma_{rt}$; $\tau_{r\theta\max} = (1+\xi_0)\,\sigma_z \le \tau_f$ $\sigma_z = \gamma_a$ H - vertical component of the natural stress state (γ_a - volumetric weight of the rock, H - depth of location of mining work); $\sigma_{\theta.\max}$, $\sigma_{\theta.\min}$ - maximum and minimum stress radial components; $\tau_{r\theta}$ - tangential stress component; σ_{rc}, σ_{rt}, τ_f - breaking strengths in compression, traction and shear rock; $\xi_0 = \mu / (1 - \mu)$ coefficient of active lateral pushing due to the Poisson's coefficient of the rocks μ

* *Assoc. Prof. Ph.D. Eng. at the University of Petrosani*

	The „i" stability criterion based on the condition of producing crack rocks around the mining works G. Barla, Mohr, L.N. Nasonov, Iu Z.Zaslavski, I.L. Davîdovici	$i_s = k_{\sigma_{IId}} \dfrac{C_s}{k_\sigma}$ or $i_s = \dfrac{\gamma_a H}{\sigma_{rc_M}}$ $k_{\sigma_{IId}}$ - coefficient of the long-term limit strength; k_σ - coefficient of concentration of stresses around the mining works; C_S - structural weakening coefficient; σ_{rc_M} - uniaxial compressive breaking strength of rocks from massive
	RQD stability criterion based on the rock quality D.U.Deere	-
	Stability criterion based on uniaxial compressive breaking strength of the rock, based on the BIMR classification	σ_{rc} σ_{rc} – uniaxial compressive breaking strength of rock
Criteria for assessing the multistadiale loss stability	Stability criterion based on the evaluation of the size of displacements "u" of rock around the mining workings	$u = k_{\alpha\delta}\, k_\theta\, k_{il}\, k_s\, k_t\, u_T$ $k_{\alpha\delta}$ - influence coefficient of the slope α and direction δ of the work compared to the main orientation planes of stratification; k_θ - coefficient for evaluation the influence of the event of rock displacement of the mining work perimeter; k_{il} - coefficient depending on the influence of other mining works; $k_s = 0{,}2\cdot(2a - 1)$; k_t - influence coefficient for time to assembly the definitive support; u_T - displacement characteristic of rocks in mm, determined experimentally or analytically according to the depth H and rock strength σ_{rc}
	„n" stability criterion based on the evaluation of the main geomechanical characteristics of rocks	$n = \dfrac{\sigma_{rc} C_s k_{\sigma_{IId}} k_w}{\gamma_a H\, k_\sigma k_{ab}}$ k_w - humidity influence coefficient on the breaking compressive strength of rocks; k_{ab} - influence coefficient of the presence workings

For geomechanical conditions of the location of the mining workings in the Horezu mining area, it is shown that the work can be classified into the 5th class of stability, according to Table 2.

Table 2- Classes of the rocks stability and the value criteria:

Criteria for assessing the stability	Classes of the rocks stability				
	I	II	III	IV	V
	very stable	stable	medium stability	unstable	very unstable
σ_{rc} [MPa]	> 200	100 - 200	50 - 100	25 - 50	< 25
RQD [%]	90 - 100	75 - 90	50 - 75	25 - 50	< 25
i_s	< 0.2	0.2 – 0.25	0.25 – 0.3	0.3 – 0.6	> 0.61
u [mm]	0	0 - 50	50 - 200	200 - 500	> 500
n	0.7 - 1	0.4 – 0.7	0.3 – 0.4	0.2 – 0.3	< 0.2

2. NATURAL STRESS STATE EVALUATION BY ANALYTICAL MEANS

The knowledge of the natural stress state is essential in solving the problems of stability of mining workings. Such naturally complete stress state of the massif is:

(1)
$$|T_\sigma| = |T_{\sigma_G}| + |T_{\sigma_T}| = \begin{vmatrix} \xi_0\gamma_a H & 0 & 0 \\ 0 & \xi_0\gamma_a H & 0 \\ 0 & 0 & \xi_0\gamma_a H \end{vmatrix} + \begin{vmatrix} F_0 + \Delta\bar{\sigma}_t H & 0 & 0 \\ 0 & \xi_{oy}(F_0 + \Delta\bar{\sigma}_t H) & 0 \\ 0 & 0 & \xi_{oz}(F_0 + \Delta\bar{\sigma}_t H) \end{vmatrix}$$

Namely, that consists of two components: a component of gravitational origin $\left(\left|T_{\sigma_G}\right|\right)$ and a component of hereditary origin or tectonic $\left(\left|T_{\sigma_T}\right|\right)$. In this case, the natural gravitational origin stress state was assessed for the massive belonging to E.M. Horezu, in the context of elastic behaviour, plastic and rheological or viscous. Such natural stress state is generally defined by its normal components, $\sigma_v^y = \sigma_z^y; \sigma_x^y; \sigma_y^y$ and tangential components, τ. If we consider that the relief is plan, horizontally, then the vertical normal component is:

$$\sigma_z^y = \gamma_a H \quad (2)$$

Because the rock massif is composed of an uneven distribution, the change in value of σ_z according to depth z is:

$$\sigma_z = \frac{a + bH}{c + H} H \quad (3)$$

Where: a, b, c are the parameters whose values have been established and consequently:

$$\sigma_z = \sigma_v = \frac{\left(1{,}6 \cdot 10^2 + 1{,}7\,H\right) \cdot 10^4}{64 + H} H \quad (4)$$

Both relationships (2) and (4) have been used to determine the value of σ_z in the situation of E.M. Horezu, which are showed quantitatively in Table 3 (Chirilă, D. 2000).

Table 3- Vertical component values:

Depth, H [m]	Vertical component $\sigma_z \cdot 10^4$ [Pa]	
	relation (2)	relation (4)
50	88.65	107.45
100	177.3	201.2
150	265.95	291

Based on the vertical component values σ_z we calculated the natural gravitational origin stress state, for two types of behaviour, namely:

a) elastic behaviour, for which we chose Kuhn's – Terzaghi hypothesis, situation when the horizontal components $\sigma_x = \sigma_y$ and the tangential components are given by relations:

$$\begin{aligned} \sigma_x &= \sigma_y = \xi_0 \gamma_a H \\ \tau &= \xi_1 \gamma_a H \end{aligned} \quad (5)$$

where: ξ_0 is active pushing coefficient due to Poisson coefficient μ:

$$\xi_0 = \frac{\mu}{1 - \mu} \quad (6)$$

ξ_1 is the passive pushing coefficient due to Poisson coefficient μ:

$$\xi_1 = \frac{1 - 2\mu}{2(1 - \mu)} \quad (7)$$

The calculated values of these stresses (for clay $\mu_{med} = 0.27$; $\xi_0 = 0.32$; $\xi_1 = 0.315$ and respectively lignite with $\mu_{med} = 0.24$; $\xi_0 = 0.36$ and $\xi_1 = 0.342$) are shown in Table 4 and Figure

1 and the axial deformation values ε_{lz}, respectively ε_{trx}, ε_{try} in the context of such behaviour are given in Table 5 and Figure 2 (Chirilă, D. 2000).

Table 4 - Calculated values of the horizontal and the tangential components:

Depth [m]	Horizontal component $\sigma_o = \sigma_x = \sigma_y \cdot 10^4$ [Pa]		Tangential component $\tau \cdot 10^4$ [Pa]	
	for σ_z relation (2)	for σ_z relation (4)	for σ_z relation (2)	for σ_z relation (4)
50	49,86	60,44	19,4	23,5
100	99,73	113,2	38,8	44
150	149,61	163,7	58,18	63,6

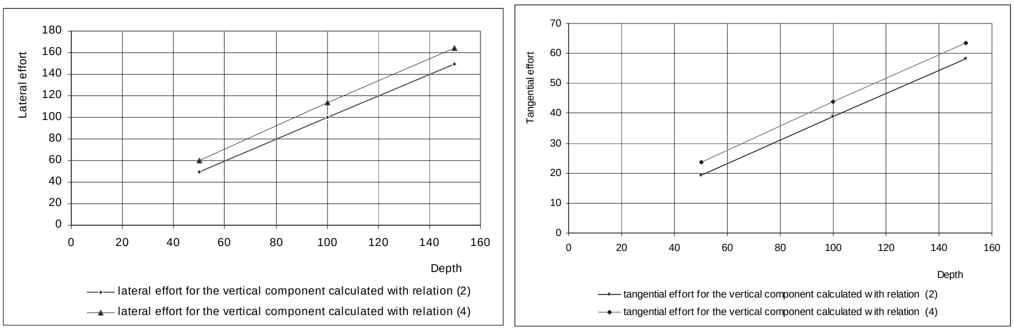

Figure 1 - Variation of horizontal and the tangential components relative to depth, considering the elastic behaviour.

Table 5- Values of the axial and transversal deformations:

Depth, H [m]	Axial deformation $\varepsilon_{l\,z}$, $\cdot 10^{-5}$		Transversal deformation ($\varepsilon_{tr\,x} = \varepsilon_{tr\,y}$), $\cdot 10^{-5}$	
	Clay	Lignite	Clay	Lignite
50	0.03383	0.01907	0.0149	0.00146
100	0.06767	0.02383	0.0298	0.00292
150	0.08947	0.66933	0.0869	0.00532
200	0.11651	0.09029	0.01132	0.00693

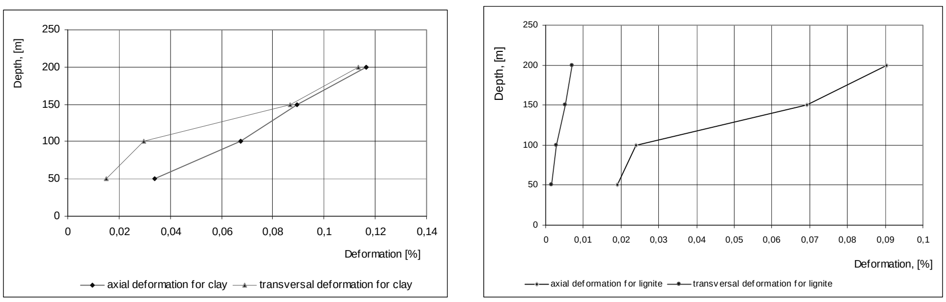

Figure 2- Variation of axial and transversal strains relative to depth, considering the elastic behaviour.

Studies performed along the years, have shown that the rocks under stress action can be characterized by a plastic state with increasing depth, which creates the possibility to use the hydrostatic law distribution hypothesis of normal components of the natural stress state to study the stability of mining workings, (Baklaşov, I.V. 1985, Todorescu, A., Gaiducov, V., 1995, Toderas, M., 1990). Some researchers have established the relationships for the critical depth, H_{critic}, from the downward the elastic-plastic deformation behaviour manifests itself in massive, Table 6.

Table 6 - Relationships for determining the critical depth:

Relationship	Critical depth H_{critic} [m]	Significance
$H_{critic} = \frac{k}{\xi_1 \gamma_a}$	21.14	k – stress value at the plastic limit $\xi_1 = \frac{1-2\mu}{2(1-\mu)}$ $j = \frac{1+\sin\varphi}{1-\sin\varphi}$ γ_a - apparent specific weight σ_{rc} - compressive breaking strength σ_e – stress at the elastic limit
$H_{critic} = \frac{\sigma_{rc}}{(1-j\xi_0)\gamma_a}$	0	
$H_{critic} = \frac{\sigma_{rc}}{2\xi_0\gamma_a}$ (Ţimbarevici)	10	
$H_{critic} = \frac{\sigma_{rc}}{\gamma_a}$ (Slesarev)	11.3	
$H_{critic} = \frac{\sigma_e}{(1-\xi_0)\gamma_a}$ (Belaenko)	1.057	
$H_{critic} = \frac{\sigma_e}{2\xi_0\gamma_a}$ (Ruppeneit)	0.411	

b) plastic behaviour. The natural stress state was determined taking into account the breaking Coulomb's – Mohr hypothesis, expressed in coordinates of normal stress (σ_1, σ_3), using the relations:

$$\left.\begin{matrix}\sigma_x \\ \sigma_y\end{matrix}\right| = i\,(\sigma_z - Q) \quad ; \quad \tau = \frac{1-i}{2}\sigma_z + \frac{i}{2}Q \tag{8}$$

where:

$$i = \frac{1-\sin\varphi}{1+\sin\varphi} \quad ; \quad Q = \frac{2\,C\cos\varphi}{1-\sin\varphi} \tag{9}$$

φ - friction angle of the rock; C – cohesion.

The values of the components of gravitational origin natural stress state for a plastic behaviour (for $\varphi_{med} = 20^0$; C_{med} = 2 MPa; i = 0.4926 and Q = 0.5712 MPa) are shown in Table 7 and Figure 3 (Chirilă, D. 2000).

Table 7 - Values of the natural stress state components for plastic behaviour:

Depth [m]	Lateral components $\sigma_x = \sigma_y \cdot 10^4$ [Pa]		Tangential components $\tau \cdot 10^4$ [Pa]	
	for σ_z relation (1)	for σ_z relation (2)	for σ_z relation (1)	for σ_z relation (2)
50	15.53	24.79	36.56	41.33
100	59.20	70.97	59.05	65.11
150	102.87	115.21	81.54	87.90

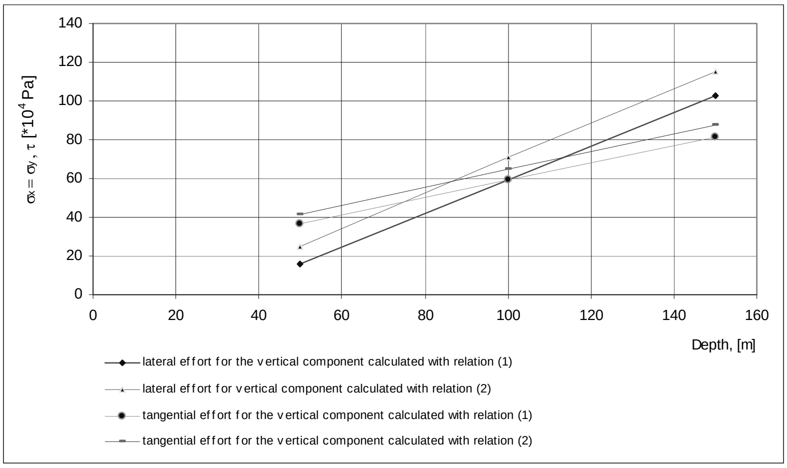

Figure 3 - Variation of the natural stress state components in relation to depth for plastic behavior

3. CONCLUSIONS

The stability criteria taken into study by the stability index values obtained, show certain instability of executed mining workings in the geomechanical conditions from Berbeşti - Horezu. Selection criteria for assessing analytically the stability of mining works and safety of supports was based on the results of in situ observations and measurements made at the mines in this mining area. Criteria considered employ the most representative parameters involved in assessing stability, namely: natural stress state, secondary stress – deformation state, structural weakening of rocks, uniaxial compressive breaking strength, long-term limit strength etc. The rocks from this basin, according to these criteria fall into class IV and V of the stability; that's characteristic for the difficult and very difficult geomechanical conditions according the mining workings which were executed in such rocks, case where the used support will be dispose of (0.25 - 0.4) MPa bearing and even higher.

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SECONDARY STRESS STATE AROUND HORIZONTAL MINING WORKS FROM BERBESTI – HOREZU BASIN

TODERAȘ MIHAELA*

Abstract: *The studies which were developed in the underground mining field showed that the mining pressure is not only a qualitative characteristic of the rock, but it also depends on the size of stress and deformations around the mining works and therefore it is necessary to study the secondary stress - deformation state. This stress state is a consequence of disruption and amplification of the natural stress state of rock massive through the execution the mining works correlated with the effect of weakening of the rock mass, the action of the tectonic forces, the expansion of rocks and their swelling phenomenon, etc. In the context of these specifications, a quantitative assessment of the secondary stress state for the conditions at E.M. Horezu was carried out, in order to characterize the stability of horizontal opening mining works.*

Keywords: *stress state, inelastic deformation, structural weakness, bulking, plastic field, mining pressure*

1. SECONDARY STRESS STATE AROUND THE HORIZONTAL MINING WORKS EXECUTED IN ROCK WITH ELASTIC BEHAVIOUR

Starting from the problem of circular hole achieved in an infinite plate, Fenner establishes in polar coordinates, the relationships that characterize the secondary stress state under the conditions of pre-existence of a natural stress state of gravitational origin (Hirian, C., 1982, Popescu, Al., Todorescu, A., 1982):

$$\sigma_r = \frac{\sigma_z}{2}\left[(1+\xi_0)\left(1-\frac{a^2}{r^2}\right)-(1-\xi_0)\left(1-4\frac{a^2}{r^2}+3\frac{a^4}{r^4}\right)\cos 2\theta\right]$$

(1) $$\sigma_\theta = \frac{\sigma_z}{2}\left[(1+\xi_0)\left(1+\frac{a^2}{r^2}\right)+(1-\xi_0)\left(1+3\frac{a^2}{r^2}\right)\cos 2\theta\right]$$

$$\tau_{r\theta} = \frac{\sigma_z}{2}\left[(1-\xi_0)\left(1+2\frac{a^2}{r^2}-3\frac{a^4}{r^4}\right)\sin 2\theta\right]$$

The maximum value of the tangential stress (for the case when r = a, $\sigma_\theta = 0$) and for the values of the angle $\theta = 0^0, 30^0, 45^0, 60^0, 90^0, 120^0, 135^0$ and 180^0, for the two possible cases, shall be analyzed thus:

- Mining works profile is circular respectively, horseshoe type;
- Mining works is located in lignite or intercepts clay in roof.

These situations are shown in Tables 1 to 6 and Figure 1 - Figure 3 (Chirilă, D., 2000).

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* *Assoc. Prof. Ph.D. Eng. at the University of Petrosani*

Table 1- Maximum value of the tangential stress for a circular profile of mining work (lignite):

Depth H [m]	$\sigma_\theta \cdot 10^4$ [Pa]							
	0^0	30^0	45^0	60^0	90^0	120^0	135^0	180^0
50	3,2	51,2	105,6	144,96	211,4	144,96	105,6	3,2
100	6,4	102,4	211,2	279,82	418,8	279,82	211,2	6,4
150	9,6	153,6	316,8	433,78	633,4	433,78	316,8	9,6
200	12,8	204,8	422,4	570,73	861,2	570,73	422,4	12,8

Table 2- Value of the coefficient of stress concentration k_θ (lignite):

Depth H [m]	$k_\theta = \sigma_\theta / \sigma_z$							
	0^0	30^0	45^0	60^0	90^0	120^0	135^0	180^0
50	0,04	0,64	1,32	1,812	2,68	1,812	1,32	0,04
100	0,04	0,64	1,32	1,812	2,68	1,812	1,32	0,04
150	0,033	0,528	1,089	1,495	2,211	1,495	1,089	0,033
200	0,033	0,540	1,115	1,530	2,264	1,530	1,115	0,033

Table 3- Values of the safety coefficient η for a circular profile (lignite):

Depth H [m]	$\eta = \sigma_\theta / \sigma_z \cdot k_\theta$							
	0^0	30^0	45^0	60^0	90^0	120^0	135^0	180^0
50	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0
100	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0
150	1,0001	1,0001	1,0001	1,0	1,0001	1,0	1,0001	1,0001
200	1,0240	1,0012	1,0001	1,0005	1,0	1,0005	1,0001	1,0240

Table 4- Values of the maximum tangential stress for a circular profile (clay in roof):

Depth H [m]	Clay				Lignite			
	$\sigma_\theta \cdot 10^4$ [Pa]				$\sigma_\theta \cdot 10^4$ [Pa]			
	0^0	30^0	45^0	60^0	90^0	120^0	135^0	180^0
50	-8,8	79,2	149,6	144,96	214,4	144,96	105,6	3,2
100	-17,6	158,4	299,6	289,92	428,8	289,92	211,2	6,4
150	-26,4	237,6	448,8	434,88	643,2	434,88	316,8	9,6
200	-35,2	316,8	598,4	579,84	857,6	579,84	422,4	12,8

Table 5- Coefficient of stress concentration value for a circular profile (clay in roof):

Depth H [m]	Clay				Lignite			
	$k_\theta = \sigma_\theta / \sigma_z$				$k_\theta = \sigma_\theta / \sigma_z$			
	0^0	30^0	45^0	60^0	90^0	120^0	135^0	180^0
50	-0,08	0,72	1,36	1,812	2,68	1,812	1,32	0,04
100	-0,08	0,72	1,36	1,812	2,68	1,812	1,32	0,04
150	-0,0907	0,816	1,542	1,495	2,211	1,495	1,089	0,033
200	-0,0944	0,836	1,579	1,530	2,264	1,530	1,115	0,033

Table 6- Values of the safety coefficient η for a circular profile (clay in roof):

Depth H [m]	Clay				Lignite			
	$\eta = \sigma_\theta / \sigma_z \cdot k_\theta$				$\eta = \sigma_\theta / \sigma_z \cdot k_\theta$			
	0^0	30^0	45^0	60^0	90^0	120^0	135^0	180^0
50	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0
100	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0
150	1,0006	1,0010	1,0005	1,0	1,0001	1,0	1,0001	1,001
200	1,0003	1,0004	1,0005	1,0005	1,0	1,0005	1,0001	1,0240

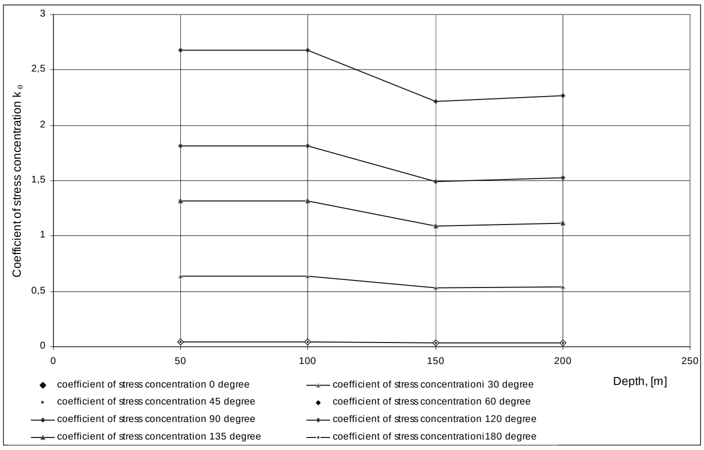

Figure 1- Variation of the coefficient of stress concentration

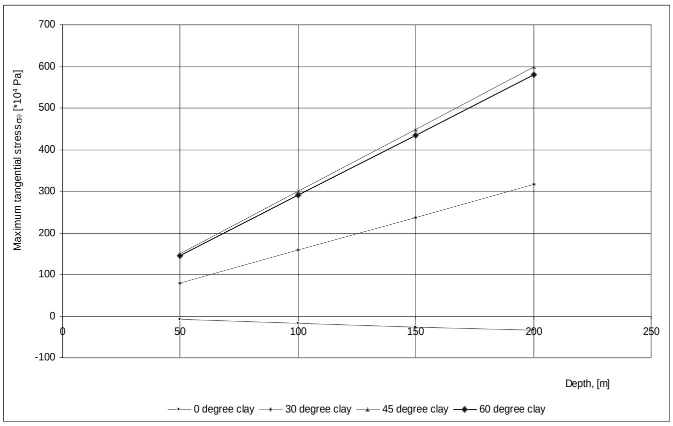

Figure 2- Variation of the maximum tangential stress on the contour of work in areas where clay is present

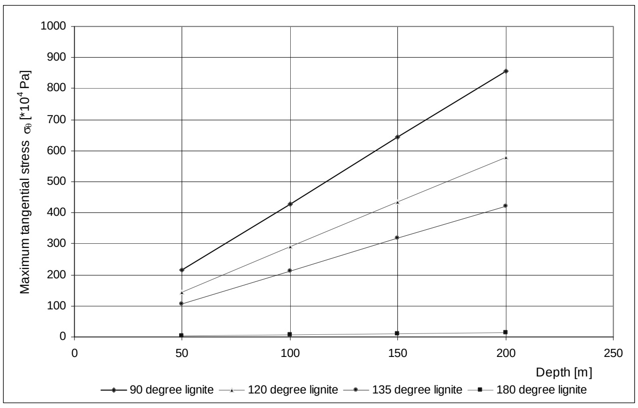

Figure 3- Variation of the maximum tangential stress on the contour of work in areas where lignite is present

The coefficient of stress concentration is determined by the relation:

$$k_\theta = \frac{\sigma_\theta}{\sigma_z} \tag{2}$$

and the value of the safety coefficient η is determined by the relation:

$$\eta = \frac{\sigma_\theta}{k_\theta \sigma_z} \tag{3}$$

2. SECONDARY STRESS STATE AROUND THE HORIZONTAL MINING WORKS EXECUTED IN ROCK WITH INELASTIC - PLASTIC BEHAVIOUR

On contour and around the mining works, the rocks can deform or dislocate, due to the appearance or the existence and development of some fissuration systems or systems of fragmentation. These deformation and dislocation processes can be characterized on the basis of properties and assigned to different forms of occurrence, distribution and manifestation of the secondary stress - strain – displacement state: plastic - clastic - viscous. Immediately after the work is performed, either by blasting or mechanized the rocks are practically instantaneous deformed on the contour (elastic deformation) (Toderas, M. 1990).

Such deformation affects an area around the mining works and there is an extremely high strain (close to the sound velocity), developing almost completely until the installation of support, for which remains unobserved. The inelastic deformation and dislocation process takes place over a long period of time. The development of elastic - inelastic - rupture deformation process leads to the occurrence of relaxation zone of plastic or clastic type. The stress in the inelastic deformation (plastic) zone is determined by the relations (Hirian, C., 1982, Popescu, Al., Todorescu, A., 1982):

$$\begin{aligned} \sigma_r^{p_1} &= \sigma_{ra}^{p_1}\left(\frac{r}{a}\right)^{k-2} - \frac{\gamma_a}{k-3}\left[a\left(\frac{r}{a}\right)^{k-2} - r\right] \\ \sigma_\theta^{p_1} &= (k-1)\sigma_{ra}^{p_1}\left(\frac{r}{a}\right)^{k-2} - \frac{k-1}{k-3}\gamma_a\left[a\left(\frac{r}{a}\right)^{k-2} - r\right] \end{aligned} \tag{4}$$

where:

$$k = 1 + \left(\mu_1 + \sqrt{1+\mu_1^2}\right)^2 \quad ; \quad \mu_1 = \mathrm{tg}\,\varphi = 0{,}53 \tag{5}$$

μ_1 - internal friction coefficient; k = 2.761; r - the height that extends the plastic zone and is determined by the relation:

$$r = \frac{k-2}{2}H \tag{6}$$

The value of stresses in the plastic field are computed depending on the maximum carrying capacity $\sigma_{ra}^{p_1}$ of support:

$$\sigma_{ra}^{p_1} = \gamma_a\, a\left(\frac{1}{2} + \ln\frac{H}{2a}\right) \tag{7}$$

Table 7 shows the maximum value of the carrying capacity and the height of the plastic zone r, respectively the values of the components of the stress state (Chirilă, D., 2000).

Table 7- Maximum value of the carrying capacity, the height of a plastic zone and the components of stress:

Depth H, [m]	$\sigma_{ra}^{p_1}$, [MPa]	R, [m]	$\sigma_r^{p_1}$, [MPa]	$\sigma_\theta^{p_1}$, [MPa]
50	0.0079	19.76	1.109	1.093
100	0.0096	38.55	2.264	2.554
150	0.105	55.32	3.430	6.723
200	0.1127	75.10	4.604	9.237

3) SECONDARY STRESS STATE AROUND THE HORINZONTAL MINING WORKS EXECUTED IN ROCK WITH VISCOUS - ELASTIC BEHAVIOUR (RHEOLOGICAL)

The rock massifs with inelastic behavior, viscous - elastic, are characterized by the fact that generally the stresses are functions of time that can be expressed with relations (Todorescu, A., Gaiducov, V., 1995):

(8)
$$\sigma_r = \frac{\sigma_z}{2}\left[(1+\xi_t)\left(1-\frac{a^2}{r^2}\right)-(1-\xi_t)\left(1-4\frac{a^2}{r^2}+3\frac{a^4}{r^4}\right)\cos 2\theta\right]$$
$$\sigma_\theta = \frac{\sigma_z}{2}\left[(1+\xi_t)\left(1+\frac{a^2}{r^2}\right)+(1-\xi_t)\left(1+3\frac{a^4}{r^4}\right)\cos 2\theta\right]$$
$$\tau_{r\theta} = \frac{\sigma_z}{2}\left[(1-\xi_t)\left(1+2\frac{a^2}{r^2}-3\frac{a^4}{r^4}\right)\sin 2\theta\right]$$

where: $\xi_t = \frac{\mu_t}{1-\mu_t}$ is the rheological coefficient of lateral pushing.

The value of the maximum tangential stresses is summary given in Table 8.

Table 8- Value of the maximum tangential stresses:

Depth H [m]	Lignite				Clay			
	$\sigma_{\theta\,max}$ [MPa]				$\sigma_{\theta\,max}$ [MPa]			
	$\theta = 0^0$	$\theta = 30^0$	$\theta = 45^0$	$\theta = 90^0$	$\theta = 0^0$	$\theta = 30^0$	$\theta = 45^0$	$\theta = 90^0$
50	1,367	0,911	0,456	0,456	1,286	0,858	0,429	0,429
100	2,734	1,822	0,911	0,911	2,573	1,715	0,858	0,858
150	4,1	2,734	1,367	1,367	3,859	2,573	1,286	1,286
200	5,467	3,645	1,822	1,822	5,146	3,43	1,715	1,715

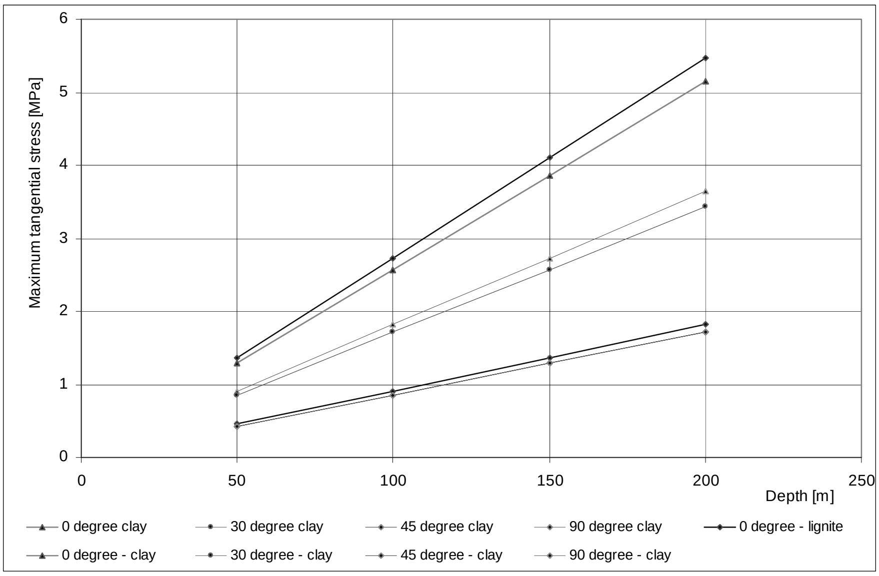

Figure 4- Variation of the maximum tangential stresses on the contour of the mining work in presence of lignite and clay

4) STRESS – DEFORMATION STATE AROUND THE MINING WORK WHICH IS UNDER THE INFLUENCE OF AQUIFER LAYER

In this case, we took into account the situation of the horizontal mining works with a circular shape, located at a distance h from an undrained aquifer layer, which the liquid phase is characterized by hydrodynamic pressure P_{ha} encountered in Berbeşti-Horezu basin (Chirilă, D., 2000). The aquifer layer deforms due to redistribution of stress state, once the excavation of the gallery, deformation that increases the bulking pressure $P_{umf.}$ from the floor of gallery. If the aquifer layer is drained and therefore hydrodynamic pressure $P_{ha} = 1$, then in the floor of gallery the pressure will act (Todorescu, A., Gaiducov, V., 1988):

$$p = (P_{ha} + P_{umf.} - 1) \tag{9}$$

In this context, the maximum tangential stress $\sigma_{\theta\,max}$ before the drainage will be:

$$\sigma_\theta\big|_{r=a} = 2\sigma_z + P_{umf.}(1 + \cos 2\theta) \quad , \quad \theta \in (\pi, 2\pi) \tag{10}$$

And after drainage:

$$\sigma_\theta\big|_{i=a} = 2\sigma_z - p\,(1 + \cos 2\theta) \quad , \quad \theta \in (\pi, 2\pi) \tag{11}$$

The appropriate relative displacement before drainage has the value:

$$U\big|_{r=a} = -\frac{1}{4G} a \cdot P_{umf.} (3 - 4\mu) \sin 2\theta \tag{12}$$

And after drainage:

$$U\big|_{r=a} = -\frac{1}{4G} a \cdot P \, (3 - 4\mu) \sin 2\theta \tag{13}$$

The appropriate relative deformations, before drainage, have the values:

$$q\Big|_{r=a} = -\frac{1}{2G}\left[\left(\sigma_z - \frac{P_{umf}}{2}\right) + \frac{P}{2}\cdot 4\,(1-\mu) - 3\right]\cos 2\theta \tag{14}$$

and after drainage:

$$q\Big|_{r=a} = \frac{1}{2G}\left[\left(\sigma_z - \frac{P}{2}\right) + \frac{P}{2}\cdot 4\,(1-\mu) - 3\right]\cos 2\theta \tag{15}$$

where: G – shear modulus of the rocks, $G = E / 2(1 + \mu) = 76.92$ MPa; $\sigma_z = 7.241$ MPa; K = G E / 3(3 G - E) = 166.6 MPa – cubic modulus of elasticity; H = 200 m; $P_{umf.} = 1.675$ MPa (clay); $E_{clay} = 200$ MPa; $\theta = 120^0$; $P_{ha} = 0.15$ MPa; a = 1.5 m; P = 0.825 MPa.

The results achieved are summary presented in Table 9.

Table 9- Values of the tangential stresses and deformations in the presence of undrained acquifer layer:

Rock type	Maximum tangential stress $\sigma_{\theta\,max}$ [MPa]		Displacement U_{max}		Relative deformation q [%]	
	before drainage	after drainage	before drainage	after drainage	before drainage	after drainage
Gray clay	5,5	6,8	0,625	0,307	5,61	6,43

It is noted that the drainage makes the stress state around the mining work worse, the maximum tangential stress increases from 5.5 to 6.8 MPa and the rock deformation increases from 5.61 to 6.43 % (Chirilă, D., 2000).

5. CONCLUSIONS

The evolution of rock deformation around the mining works shows that, once with increasing the depth location of the mining works, the level of instability and uncertainty of their supports is amplified.

As shown in the literature, the working conditions of the supports can be diversified in 6 classes. Analyzing in terms of geomechanics and of the presence of groundwater the rock massif of Horezu basin where the mining works are executed, we can certainly say that in the real situation encountered in this basin the classes III, IV, V and VI are typical stability and work conditions of the supports. Classification in these classes, confirms that the stability of mining works cannot be assured and is not possible without the support, which must be chosen to account the interaction rock - support – time phenomenon. Also be taken into account that around the mining works a zone of rocks is formed and it's characterized by three areas: a failure – cracking area, inelastic such as viscous - plastic type and an elastic area. In this present case which was exposed, we consider that the general model of interaction is elastic - viscous – plastic type.

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Prof.Ph.D.Eng. Mircea GEORGESCU

NEW AERODYNAMIC PARAMETERS SPECIFIC TO GOAFS

CIOCLEA DORU[*]
LUPU CONSTANTIN[*]
TOTH ION[*]
GHERGHE ION[**]
BOANTĂ CORNELIU[**]
RĂDOI FLORIN[**]

Abstract: *The identification and application of parameters for controlling the goaf leads to the knowledge of the distribution of air leakages, respectively to predicting the development degree of some phenomena which take place at its level.*

Keywords: *aerodynamic parameters, goaf, indexes*

1. INTRODUCTION

In order to ensure a high occupational health and safety level in underground coal mining, at the level of the ventilation network and implicitly of the coal face there are circulated air flows which are established through the ventilation dimensioning process.

For circulating these air flows there are used advanced fans located at the surface and which develop high depressions in order to overcome the aerodynamic resistances related to mine workings.

The depressions performed at the level of the head gallery, are also manifested over the goaf related to the coal face, which inevitably leads to the air circulation through it.

Air which passes the goaf is called air leakage, because it decreases the circulated air flow at the level of active mine workings.

Air leakages through goafs may directly lead to the occurrence of spontaneous combustion phenomena and indirectly to the occurrence of gas accumulations, respectively to explosion type phenomena at the level of active workings.

2. PARAMETERS SPECIFIC TO GOAFS

In order to quantitatively establish the air leakages through goafs, respectively the aerodynamic parameters there have been carried out several measurements and

[*] *PhD., Eng., INCD INSEMEX Petroşani, Romania*
[**] *Eng., INCD INSEMEX Petroşani, Romania*

experimentations at the level of some coal faces belonging to Lonea mining unit, Petrila mining unit, Livezeni mining unit and Vulcan mining unit.

Also, there have been identified specific indexes which provide the possibility of knowing the characteristics of goafs as well as the preparation level from the point of view of spontaneous combustion phenomena occurrence.

2.1. Critical flow index - I_{CD}

The critical flow index – I_{CD}, establishes the preparation level of the goaf in relation to the risk of occurrence of spontaneous combustions. The critical flow index – I_{CD}, represents the circulated air flow per area unit, at the level of the goaf. This index is calculated according to the directly or indirectly determined air flow leakages.

$$I_{CD} = Q_{se} / S \ [m^3/min \cdot m^2] \qquad (1)$$

Where:

Q_{se} - circulated air flow at the goaf level [m^3/min]
S - the section of the goaf at the coal face level [m^2]

According to the critical flow index, the goaf can be characterized by 5 domains, namely:

A. the non-hazardous domain when: $0 \leq I_{CD} < 0{,}1$. This domain is characterized by the absence of the risk of occurrence of spontaneous combustion phenomena at the level of the goaf.

B. the pre-critical domain, when: $0{,}1 \leq I_{CD} < 0{,}4$. This domain is characterized by the increased risk of occurrence of spontaneous combustion phenomena at the level of the goaf. Any increase in flow leads the goaf towards the critical domain.

C. the critical domain, when: $0{,}4 \leq I_{CD} < 0{,}6$. This domain is characterized by the imminent risk of occurrence of spontaneous combustion phenomena at the level of the goaf.

D. the post-critical domain, when: $0{,}6 \leq I_{CD} < 0{,}9$. This domain is characterized by the high risk of occurrence of spontaneous combustion phenomena at the level of the goaf and it is specific to the conditions in which insulation constructions are incorrectly built and present a high permeability level.

E. the unauthorized domain, when: $I_{CD} \geq 0{,}9$. This domain involves aerodynamic paths on which high flows are circulated, specific to un-sealed goafs, which is contrary to the legislation in force.

2.2. Permeability index – I_P

The permeability index – I_P establishes the permeability level of the goaf in relation to the risk of occurrence of spontaneous combustion phenomena. The permeability index – $I_{p,}$ represents the resistance of the goaf to air flowing. This index is calculated according to the pressure loss required for the circulation of the leaked air, determined by "in situ" measurements.

$$I_P = 3600 \cdot H / S^2 \cdot I^2_{CD} \qquad [N.s^2/m^6] \qquad (2)$$

Where:

H - total depression at the level of the goaf [N/m^2];
S - section of the goaf at the coal face level [m^2];
I_{CD} - critical flow index [m^3/min m^2]

According to the permeability index, the goaf can be characterized by 5 domains, namely:

A. the very hard permeable domain, when:

$I_P \geq 360 \cdot 10^3 \cdot H/S^2$

This domain is characterized by a very well-sealed goaf, respectively by the absence of the risk of occurrence of spontaneous combustion phenomena at the level of the goaf.

B. the permeable domain, when:

$360 \cdot 10^3 \cdot H/S^2 > I_P \geq 22{,}5 \cdot 10^3 \cdot H/S^2$

This domain is characterized by a sealed goaf, respectively by the increased risk of occurrence of spontaneous combustion phenomena at the level of the goaf. Any increase in permeability leads the goaf towards the critical domain.

C. the critical permeability domain, when:

$22{,}5 \cdot 10^3 \cdot H/S^2 > I_P \geq 10 \cdot 10^3 \cdot H/S^2$

This domain is characterized by a goaf with critical sealing, respectively by the imminent risk of occurrence of spontaneous combustion phenomena at the level of the goaf.

D. the very permeable domain, when:

$10 \cdot 10^3 \cdot H/S^2 > I_P \geq 4{,}5 \cdot 10^3 \cdot H/S^2$

This domain is characterized by a goaf with a very weak sealing, respectively by the high risk of occurrence of spontaneous combustion phenomena at the level of the goaf and it is specific to the conditions in which insulation constructions are incorrectly built and present a high permeability level.

E. the unauthorized domain, when: $I_P \leq 4{,}5 \cdot 10^3 \cdot H/S^2$

This domain is characterized by a goaf with extremely weak sealing. This domain involves aerodynamic paths on which high flows are circulated, specific to un-sealed goafs, which is contrary to the legislation in force.

2.3. The determination of aerodynamic parameters specific to goafs, at the undermined coal face no. 1-4/3/VIII from Vulcan mining unit.

According to Fig. 1, the actual balance of the flows at the level of the coal face B_{NAR} is:

$$B_{NAR} = 5{,}9 \ [\ m^3/min]$$

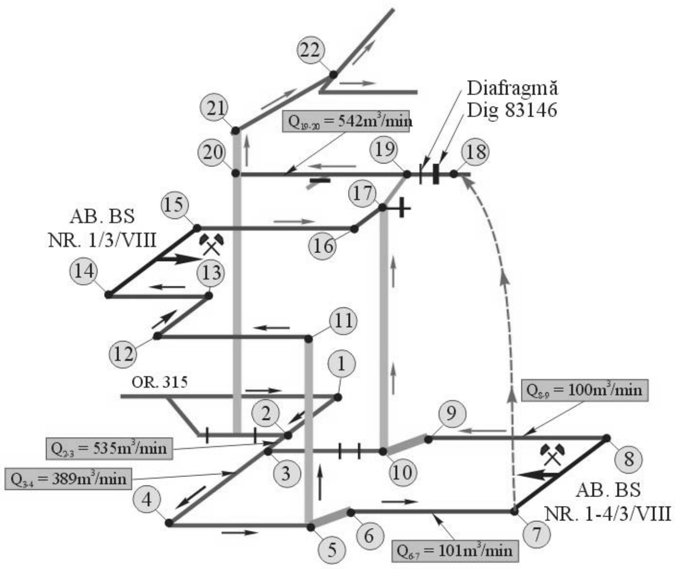

Fig.1 The circulated flow through the goaf up to the insulation dam no. 83146

It has been “in situ” determined and it has been established as being:

$$Q_{se} = 0{,}756\ [m^3/min] = 0{,}0126\ [m^3/s]$$

At the coal face number level, we have:

$$Q_{se} = 5{,}9\ [m^3/min] = 0{,}0983\ [m^3/s]$$

The depression required for the circulation of air flow through the goaf H_{se}, is equal to the pressure loss required for the circulation of air flow over the active mine workings, from the point of entrance into the coal face up to the top of the rising from the floor.

$$H_{se} = 34\ [Pa],$$

at coal face number level: $H_{se} = 47$ [Pa].

The resistance specific to the goaf R_{se} is:

$$R_{se} = H_{se} / Q_{se}\ 2 \quad (3)$$

so:

$$R_{se} = 214.159{,}73\ [Ns^2/m^8]$$

at coal face number level:

$$R_{se} = 4.863{,}969\ [Ns^2/m^8]$$

The critical flow index - I_{CD} has been:

$$I_{CD} = 0{,}00756 \quad [m^3/min \cdot m^2]$$

According to the critical flow index - $I_{CD,}$ the goaf frames into the A. non-hazardous domain

The permeability index – I_P has been:

$$I_P = 62.988{,}157 \cdot 10^3 \cdot H/S^2$$
$$I_P = 214.159{,}73 \quad [N.s^2/m^6]$$

According to the permeability index - I_P, the goaf frames into the A. very hard permeable domain.

2.4. Critical slope index – I_{CP}

The method consists in the virtual application of a consecutive and progressive voluntary stress over the goaf, by the artificial induction of a consecutively increasing resistance at the level of the directional gallery on the floor related to an undermined coal face, applied in the exploitation of a thick bed of high inclination in successive slices or sub-levels.

If at the level of such a coal face which is located at the exploitation of the first sub-level, the obtained geodesic coordinates are inserted in the database of the 3D CANVENT software, the image from Fig. 2 is achieved, image on which the aerodynamic connection through the goaf is also illustrated.

Virtually, at the level of the directional gallery over the floor, a shell door is placed, whose resistance is progressively varied from 0 to 5 [Ns^2/m^8]. At each modification of the aerodynamic resistance, there is noted the lost flow through the goaf and its depression exerted on it and which corresponds to the lost flow.

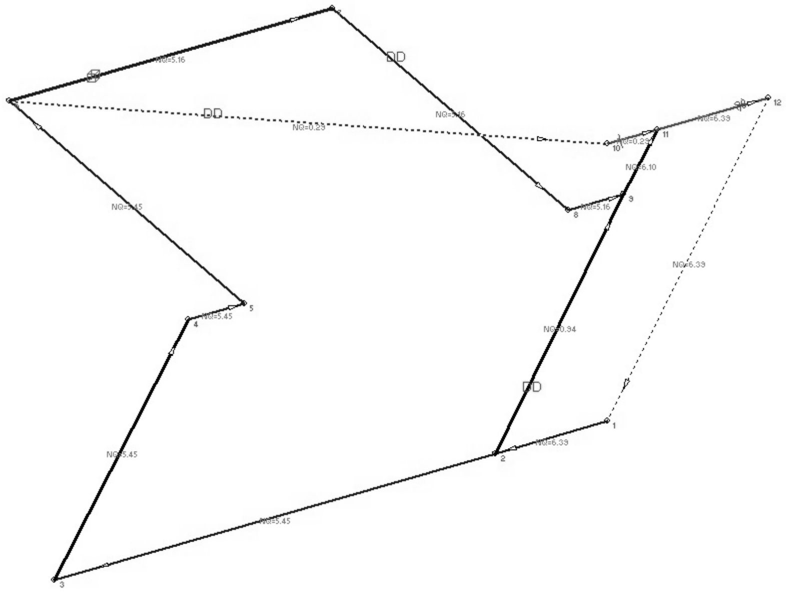

Fig.2

If the parameters concerned, Q and H, are transposed into and H = f (Q) type graphic, there results a curve which through linearization results in a line Fig. 3.

Depending on the structure of the coal face, on the repartition of the aerodynamic resistances at branch level, on the depression exerted at coal face level and on the depression exerted at goaf level, there result linearized curves with different inclinations.

By using this graph, there may be established a "critical slope index I_{CP}" which may be used as a fire index.

$$I_{CP} = \text{ctg}\ \alpha \qquad (4)$$

$$\text{ctg}\ \alpha = \Delta Q/\Delta H \quad [m^5/N \cdot s] \qquad (5)$$

where:

α – the angle of the graph with the abscissa [º];
ΔQ – the increase of the flow lost through the goaf [m^3/min];
ΔH – the increase of depression exerted over the goaf [Pa].

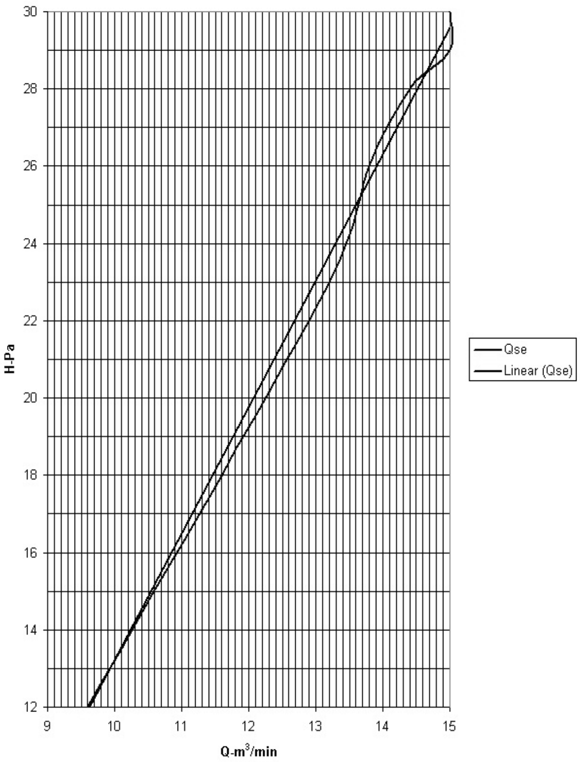

Fig.3

The critical slope index represents the circulated air flow at the goaf level to a depression by one Pa. In order to be used as a fire index, the critical slope index I_{CP} has to establish a correlation between its different values and the risk of occurrence of spontaneous combustions. In order to calibrate de slope index, a detailed analysis of the coal faces from the mining units has been carried out, analysis which took into account the aerodynamic parameters at branch level, the depressions and flows achieved at coal face level. There have been achieved the charts referring to the flow of air through the goaf in the case of minimum, medium and maximum values related to the achieved parameters, Following the analysis, it resulted that the I_{CP} index is used only in undermined coal faces from thick beds of high inclination, having the following values and significations:

$I_{CP} < 0{,}2$ [$m^5/N{\cdot}s$] - normal situation;

$0{,}2 \leq I_{CP} \leq 0{,}8$ [$m^5/N{\cdot}s$] - the occurrence of the self-heating phenomena;

$I_{CP} > 0{,}8$ [$m^5/N{\cdot}s$] - the occurrence of the self-ignition phenomena.

3. CONCLUSIONS

1. Air leakages through goafs decrease the circulated air flow at the level of active mine workings. Air leakages through goafs may directly lead to the occurrence of spontaneous combustion phenomena and indirectly to the occurrence of gas accumulations, respectively to explosion type phenomena at the level of active workings.

2. In order to quantitatively establish the air leakages through goafs, respectively the aerodynamic parameters there have been carried out several measurements and experimentations at the level of some coal faces belonging to the mining units. Also, there have been identified specific indexes which provide the possibility of knowing the characteristics of goafs as well as the preparation level from the point of view of spontaneous combustion phenomena occurrence.

3. The critical flow index - $I_{CD,}$ establishes the preparation level of the goaf in relation to the risk of occurrence of spontaneous combustions. The critical flow index - I_{CD}, represents the circulated air flow per area unit, at the level of the goaf.

4. The permeability index – I_P establishes the permeability level of the goaf in relation to the risk of occurrence of spontaneous combustion phenomena. The permeability index – I_P represents the resistance of the goaf to air flowing.

5. The critical slope index represents the circulated air flow at the goaf level to a depression by one Pa. This index may be used as a fire index.

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WORK AND INTERVENTION PROCEDURES IN CLOSED SPACES WITH POTENTIALLY TOXIC/EXPLOSIVE/FLAMMABLE ATMOSPHERES

GAMAN ARTUR GEORGE[*]
PUPAZAN DANIEL[**]
ILIE COSMIN[***]

ABSTRACT: *Working in confined spaces is a risk factor which generates accidents or unwanted events with impact on human health. The implementation of rules of good practice in this kind of situations represents a mandatory prevention method. A limited space is any place which is not designed for current work, has input or output restrictions, is at atmospheric pressure, has inappropriate ventilation, drowning or suffocation hazard, contaminated atmosphere or oxygen deficit. Therefore, in case of confined spaces, there are required measurements for protecting the worker against the hostile atmosphere, against materials which could cover him or against other hazards which do not confer safety.*

Keywords: *confined space, explosive atmosphere, rescue and intervention staff*

Working in confined spaces [1] is a risk factor which generates accidents or unwanted events with impact on human health. The implementation of rules of good practice in this kind of situations represents a mandatory prevention method.

A limited space is any place which is not designed for current work, has input or output restrictions, is at atmospheric pressure, has inappropriate ventilation, drowning or suffocation hazard, contaminated atmosphere or oxygen deficit.

Therefore, in case of confined spaces, there are required measurements for protecting the worker against the hostile atmosphere, against materials which could cover him or against other hazards which do not confer safety.

A job in a limited space may frame into the definition of working in isolation conditions [3] when working operations are carried out by a single person with which no direct contact can be made. Not all work tasks in limited spaces are carried out as working in isolation conditions. A human entering with the upper body or with the head in limited spaces is considered to be working in such conditions. There are a lot of hazards in limited spaces, and some of them cannot be detected by devices and at least one may have an impact on human health.

[*] *PhD. Eng – II Scientific researcher – INCD INSEMEX Petrosani*
[**] *PhD. Eng – III Scientific researcher – INCD INSEMEX Petrosani*
[***] *Student PhD. – Scientific researcher – INCD INSEMEX Petrosani*

A “confined space” is a work place in which the entrance/exit of workers or only some parts of their bodies is difficult and which presents, in addition, at least one of the following characteristics:

- there is or may occur and persist, a hazardous atmosphere through the improper concentration of oxygen, gases, vapours, smoke, powders, microorganisms and/or through temperature due to improper ventilation;
- there may occur the danger of flooding or discharge of liquids or granules;
- there is known or predictable, the possibility of injuries due to the crumbling or crashing of walls, of vaults, of ceilings, of some parts of them or of materials attached to them;
- the work space from inside the work equipment may become a hazardous area in case of its unexpected start;
- the development of any activity is impossible before ensuring a proper permanent lighting;

Any confined space is characterized by:

- limited opening regarding the access inside;
- it is not designed for developing continuous activities inside or
- it presents poor natural ventilation;
- it presents oxygen deficit [4] and/or toxic or flammable gas accumulations;

Limited spaces may be:

- known and permanent – where planned or periodical workings are carried out (maintenance, cleaning, revision, repairs, interior examination, etc.);
- accidentally occurred – during the construction workings and which become known after some required interventions.

Access / work in confined space

The access / work in confined space is allowed only after the implementation of the safety requirements [2] resulted from the application of a procedure for the safe access/work in confined space developed by the head of the working place.

The procedure for the access / work in confined spaces has to include the following operations:

- the chronological nomination of operations to be carried out and the selection of appropriate tools;
- hazards identification;
- the nomination of specific protection measures;
- the establishment and procurement of specific protective equipment to be used;
- the drawing up of the work permit [2];
- the selection of involved staff, from the occupational health and safety point of view;
- the preparation of specific instructions for safe working in that confined space;
- the nomination of employees from rescue and fire-fighting and fire prevention teams (FFP);
- theoretical and practical staff training on the way of access and evacuation in case of accidents, on the work practices and on the usage of protective, FFP, rescue and first-aid equipment;

For any kind of intervention, revision and repair of installations, machinery, containers, there shall be issued a PERMIT TO WORK WITHOUT FIRE.

Welding, cutting, soldering or other similar operations which present fire hazard shall be carried out in such spaces based only on the PERMIT TO WORK WITH FIRE, after carrying out all measures comprised in this permit. A Permit to work with fire shall not be issued without a Work permit for interventions/repairs to be issued in advance

The work permit may be issued if the following conditions are fulfilled:

- measures for unauthorized entering have been taken;
- the risks in limited spaces are assessed;
- written procedure for entering are available;
- the assessment of conditions from the limited space can be carried out during the activities from inside;
- trained supervisors and performers are available;
- required protection and safety equipment is acquired;
- a specific program for working in limited spaces from the employer is available;
- any work has to be coordinated by the main work performer for the contractors;

The permit is a document issued by the site manager, and has to contain at least the following information [4]:

- the precise identification of the limited space to be entered into;
- purpose of entry;
- authorized period for entry;
- authorized persons that enter;
- supervisor;
- site manager;
- known hazards;
- methods used for controlling the space;
- what are the acceptable conditions for entry;
- initial and periodical test values;
- emergency procedure to be used;
- used communication methods;

Assessments before entering the limited space:

- tests and measurements in order to allow the entry;
- which is the minimum required number of workers and the responsibilities assignment;
- if the limited space has been proper ventilated before the entry of persons;
- personal protective equipment [1] of each individual;
- what devices, tools, safety systems are required for activities in limited space;
- there are available procedures and equipment for emergency situations.

Obligations of workers:

- the worker has to know all safety provisions related to each confined space in which they have to work;
- the worker has the right and must ask for all necessary clarifications regarding the work in confined space;
- the access in confined spaces is not permitted without a work permit;
- the worker has to safely apply all work provisions for the involved confined space;
- the worker has to use PPE specific for the involved confined space;
- the worker has to use, for entering or leaving the confined space, only specially designed entry/exit means;
- the worker has to communicate with the supervisor anytime he is called by him;

- the worker has to notify the supervisor anytime he recognizes an indicator of exposure to a hazardous situation and must immediately leave the confined space;
- the worker must immediately report to the supervisor any accident;
- the worker must immediately evacuate the sealed space when the supervisor gives the evacuation order;
- the worker has to know how to use the rescue equipment if necessary;
- the employee is not allowed to carry out other operations which are not related to the activity for which he has been trained to work in that confined space;
- it is forbidden for the worker to unduly alarm the supervisor;
- it is forbidden for the worker to unduly use FFP and rescue equipment;
- it is forbidden for the worker to remove or make inoperative the systems, mechanisms and elements for isolating the confined space, without prior consent of the person in charge;
- it is forbidden to work with open-fire or to carry out activities that generate enough heat to initiate an ignition, in confined spaces without the permit to work with fire;
- the worker has to maintain the hygienic conditions in the confined space and he must ensure the cleanliness when leaving the work place.

Obligations of the supervisor:

The activity in a confined space is carried out only with supervision from outside, by at least one worker, named supervisor, and who is assigned by the head of the work place.

The supervisor must track the development of the work from the confined space from a non-hazardous place, and has the following obligations:

The supervisor must permanently maintain contact with all workers from the confined space, by at least one mean of communication, as following:

- visually;
- by voice;
- by acoustic signals;
- by knocking on the walls of the space;
- by pulling the insurance rope;
- by phone

In case the supervisor cannot permanently communicate with the workers from the confined space, one or more workers have to be placed in convenient areas, respectively within its sight, in order to ensure the permanent communication of the supervisor with the workers from the confined space.

Workers who ensure the communication between the supervisor and the workers from the confined space have to:

- know and respect the same provisions as the supervisor;
- to wear throughout the surveillance the same protective equipment as the supervisor;

The supervisor has to permanently know the number of workers from inside the confined space.

The supervisor has to know the hazards which may occur during the working in confined space, their possible occurrence causes, their signals and indicators of imminent occurrence and the manner of manifestation.

The supervisor also has to follow the activities carried out near the confined space, activities that may jeopardize the safety of workers from inside the surveyed confined space.

Visually, or by question-answer communication code, the supervisor has to recognize the changes in behaviour of the workers and must ensure them the immediate evacuation in case he realizes that they don't respond correctly to the code set.

In case of an imminent hazard, the supervisor has to give the disposition for immediate evacuation of the confined space.

If the supervisor realizes that he cannot efficiently and in safe conditions fulfil his duties, he must give a evacuation disposal for the personnel from the confined space and he must notify the person in charge for the work place.

Before work resumption, the supervisor has to grant the access to workers only after he has checked from outside the availability of safe conditions for carrying out the activities in the confined space which he supervises.

The supervisor must not leave its place until he is substituted.

The supervisor must forbid the access in confined space to any unauthorized person. In the case in which into the surveyed confined space an unauthorized person has entered, the supervisor must stop the works, evacuate the workers and notify the direct head of the work place.

The supervisor has to use the same personal protective equipment as the workers.

The supervisor must no carry out any other activity which could prevent him to fulfil the obligations regarding the surveillance of workers from the confined space.

The supervisor has to be familiar with first-aid measures and, when needed, to apply them.

In the case of occurrence of any dangerous event [3] or accident, the supervisor must take the following measures:

a) he must immediately notify the intervention [1], rescue and first-aid team, the person in charge with the work place, the coordinator of the work, and, when needed, the staff responsible for the protection to fires.

b) he must not enter the confined space in order to save the workers if his substitute isn't at the surveillance place and if he doesn't wear personal protective equipment.

c) he must provide first-aid to the workers, within his competence.

The supervisor may track the activities from several confined spaces only if the coordinator of the work or if the persons in charge with the workplace note that he can fully and simultaneously fulfil his obligations for each one of them.

The supervisor has to enter into the confined space for rescue purposes [4], only after he has been replaced by another employee trained as supervisor and only with PPE proper for the conditions from the confined space.

Means of access and rescue:

- there have to be provided safe access means (ladders, scaffolding, catwalks, ropes) corresponding to the placement and shape of the openings of that confined space;
- these must be strong and resist throughout the activity carried out in the confined space;
- portable ladders must be secured at least at the top;
- they have to be maintained in operation throughout the work;

Verifying the atmosphere while working in a confined space

The verification of the atmosphere while working in a confined space has to be done permanently or at pre-established periods, depending on the particularities of the confined space.

The access/work in confined space with potentially hazardous atmosphere [3] is forbidden before the verification of the atmosphere.

The inside temperature has to be below 40^{o} C;

When verifying the atmosphere from a confined space, the following requirements have to be respected:

The atmosphere verification operations will be carried out only in the following order:

a) verification of oxygen concentration [2];

b) verification of the nature and concentration of flammable gases and/or powders;

c) verification of the nature and concentration of gases, very toxic, poisonous, corrosive and / or irritating vapours [4];

d) verification of harmful biological agents and/or powders content, other than the flammable ones.

Atmosphere verification in a confined space has to be carried out at different levels, keeping into account the following factors:

- the possible existence of gas bags [1];
- the architecture of the confined space;
- the concentration of the gases;
- the toxicity, corrosiveness, infection level of existing gases;
- the gases density and its' potential variation depending on the temperature;
- the quantity of gases released by the wastes.

After verifying the atmosphere of the confined space, there shall be verified the absence of dangerous animals from inside.

The atmosphere verification operations in confined spaces must be carried out only by authorized persons.

In order to access and work without breathing apparatus in a confined space, the atmosphere has to correspond to the following parameters:

a) the concentration of oxygen in volume percentage, must be between 20% and 22%;

b) toxic, poisonous, corrosive or irritating gases and/or vapours or powders or biological agents have to be below the permissible exposure limit specified in regulations

Warning signals from the automatic devices for detection, measurement, signalling and surveillance of the atmosphere from the confined space should be able to be perceived simultaneously by the supervisor and the workers from the confined space.

The detection, measurement, signalling and surveillance devices have to be in good condition and have to be periodically inspected by metrological bodies, the usage of devices with outdated term of use and/or with inadequate precision class being forbidden.

Devices which will be used in potentially explosive atmospheres have to be certified as explosion-proof.

The devices have to issue warning signals which are easily perceptible by the supervisor and the workers from the confined space, in case of reaching the admitted limit values.

Any worker involved in the activities from a confined space, as well as the legal representatives of the workers, have the right to consult the registrations of the results of the confined space atmosphere verification.

Before starting to wash the confined spaces, these have to be aerated until the decrease of the flammable gas concentration in air below 20% of the lower explosive limit of the atmosphere.

It is forbidden o continue welding, polishing or any other activity which generates sparks in the confined space in which explosive atmospheres may occur, when the gases, vapours or mists concentrations are higher than 20% of their lower explosive limit.

It is forbidden to stop the general and partial ventilation fan, without the consent of th person in charge.

The usage of breathing apparatus with filter is forbidden in atmospheres with low on oxygen. In this situation there shall be used autonomous or non-autonomous protection devices.

The permanent ventilation of the space shall be provided, in order to maintain the atmosphere at proper parameters, without protective breathing apparatus.

Potential hazards in confined spaces

1. Risk of asphyxiation

It can be caused by a chemical work environment, the replacement of oxygen with other gases, the oxygen consumption through bacterial fermentation or through the absorption by plants, soils and seeds, the consumption through fire or due to massive rust.

The effect is asphyxia, respiratory distress, then of the brain and even death.

2. The toxic hazard for the human is given by the work atmosphere in the confined space:

It may consist of gases, vapours, mist, powder, dusts and smoke with toxic effects for humans.

It contains toxic or asphyxiating gases which cannot be detected by the smell: carbon dioxide, carbon oxide, nitrogen, helium, argon, and the people don't directly react to them.

There may be released vapours from different operations (ex. spray painting) which generate a toxic atmosphere.

3. The fire or explosion hazard is due to:

The content of flammable gases, vapours, mists or powders which can ignite in mixture with air by the initiation of an ignition source.

4. The suffocation hazard by inhaling or by chest compression occurs when the person is exposed to some traps or to the spraying of liquids or bulk materials.

5. The choking/covering hazard occurs when there may freely flow solid or granular materials (soil, sand, cereals, other pulverous materials) which can completely cover the human body.

Mind the hazards of collapse, bank, vaults or materials crushing.

6. The mechanical hazards manifest themselves by the direct impact on the human body**.**

Crushing, hitting, fastening, sectioning, cutting, and stinging by moving elements and mechanical bodies.

Where there are mechanisms for mixing, stirring, transport, boring, impact, which are not stopped and blocked correctly.

Hitting, stinging or cutting due to surfaces, edges, peaks, fixed objects or the conformation of the elements of the confined space.

Mind the danger of free falling.

7. The thermal stress in the limited space may affect the persons from inside due to uncomfortable temperatures or variations of temperatures.

Too low or too high outside temperatures (above 40˚C or below -15˚C)

High inside temperatures due to insufficient technological cooling.

Using working procedures which may generate the increase of inside temperatures.

The training of personnel which enter the limited space has to comprise:

- the risks of injury and the means of action in case of their occurrence.
- the manipulation of the technical equipment, tools or devices.
- the proper behaviour in case of faults or critical situations.
- the use of personal protective or rescue equipment.
- the use of the surveillance system.
- the procedures for specific emergency situations.
- First aid measures [3] proper for the carried out activities.

Personal protective equipment:

- They are provided freely and mandatory by the employer.
- PPE has to be in accordance with the work operations and with the encountered risk factors.
- Each part of the PPE is verified before each use.
- Safety belts and anti-falling hanging equipment, support and connection cords are used when there may occur slipping and free falling risks or when the person is not directly visible.
- Where parameters of the working atmosphere are not appropriate or where there is the possibility for the atmospheres subsequent contamination, accessing in the limited space will be granted only with fresh air adduction mask or by using isolating breathing apparatus [2].

Special equipment required:

- Special for controlling the atmosphere: oxygen meter, explosion-meter, toxic-meter.
- Forced ventilation equipment and flexible pipes;
- Inside/outside communication equipment;
- Rescue and extraction from inside equipment;
- Belts, harnesses and cords, fastening systems against falling.
- Personal protective breathing equipment proper for the atmosphere;
- Personal protective equipment while working.

When entering into confined spaces is required for revisions, repairs, inspection before their delivery, there shall be required to be carried out the following preparation measures, in order to avoid intoxication, fire or explosion hazards, measures which are recorded in the work permit:

- the complete emptying of the content from the machinery;
- the disconnection, insulation and armouring of all technological connections (except the ones used for water or steam supply) for preventing the accidental intrusion of substances in the machinery (liquids, gases or vapours) through supply or exhaust pipes from or towards the installations to which the machinery is connected, and the recording of all armouring/dis-armouring operations in the armouring data sheet;
- the steaming, washing with water or the cleaning with inert gas;

The steaming time for the machineries will be separately established, depending on shape and dimensions, as well as on the characteristics of the product which they have contained. The steaming of machinery which contained flammable products is very dangerous because the steam, having a high speed, may produce static electricity and finally an explosion.

In order to avoid the occurrence of static electricity it is advisable to take the following measures:

- the question machinery will be connected to the ground belt;
- the speed of the steam when entering the machinery, will be as low as possible and shall be maintained so until the disclosure (with steam) of an important part from the volume of product or inert gas.

Following the steaming operation, the exhausted gas mixture has to have a low concentration so that it shall not represent an intoxication or explosion hazard, and it shall be conducted, as much as possible, towards free zones, without circulation and ignition sources.

- the aeration and cooling of the machinery to a 40 degrees Celsius temperature;

- collecting samples from the inside atmosphere in order to determine the oxygen and poisonous gases concentrations. When the analysis indicates the presence of flammable or toxic gases, there shall be repeated the steaming and washing operations until the laboratory analysis show the oxygen concentration within the permitted limits and the absence of explosive mixture and/or emissions.

Gas samples shall be collected from at least three points of the machinery (top, middle and bottom). By an assigned person, in the presence of the person in charge, and anytime it is required throughout the development of the workings. Any change of parameters leads to the cancellation of the inspection report issued by the assigned person.

After carrying out all preparatory measures and after the training of the staff involved in the execution of inside revision and repairs workings there shall be issued the permit to work without fire. Measures comprised by the permit aim to remove the possibilities of accidents occurrence, but they don't deplete the measures which may and shall be taken, depending on the specific of the installation and on the hazard level, these measures being nominated in the work permit under the "other measures" section.

When carrying out the workings, there shall be taken into account the following:

- the opening of manholes shall be done only from top to bottom. In case there is no danger of forming and explosive mixture, the manholes may be opened simultaneously;

- in case the presence of pyrophoric sulphides is possible, opening the machineries shall be done only after steaming, followed by water washing. Pyrophoric sulphides resulted from the cleaning of machineries shall be transported outside the work area and shall be buried in a well-established place;

- when workers enter columns of trays, gratings, cyclones or other inside constructions, there shall be verified if these are not displaced from their normal position;

- when entering reservoirs which are provided inside them with electrical equipment (homogenizers, agitators, etc.) there shall be requested to the electrician to interrupt the energy source, according to own OHS provisions for the electrical energy transport and distribution;

- during all operations from inside the machineries, there shall be repeated at certain periods, the determination of flammable gases, oxygen, etc. concentrations in order to establish it there have been developed new hazardous conditions; these time periods will be established for each installation, device or machinery, by the person in charge, depending on the nature of the working carried out; if it is noticed the conditions for which the work permit had been issued have been changed, this permit is cancelled, the work is stopped, and all required measures for removing this new situation are taken.

- after solving this, the work can start again, but only after issuing a new work permit;

- staff to work in confined spaces, manholes, sewers shall be equipped with safety belt, fresh air adduction mask or isolating apparatus, cord, boots without metallic accessories, cotton overalls or coveralls; in atmospheres with oxygen deficiency it is forbidden to use breathing apparatus with filter.

- inside staff shall be supervised from the outside on the entire duration of the workings in confined spaces;

- the supervisor shall be equipped with the same personal protective equipment (PPE) as the employees from the confined space and shall permanently keep contact (visually, by voice, by acoustic signals, by knocking on the walls of the space, by pulling the insurance rope, by phone, etc.) with all workers from the confined space;

- working with open fire above or adjacent to confined spaces in which there are workers is forbidden;
- the access in the confined space of unauthorized persons is forbidden;

When there is required to carry out activities which involve fire inside machinery, before issuing the permit to work with fire, besides the above mentioned measures, there shall be achieved the following preparation measures:

- the covering of sewer pits with a wet sand layer of minimum 50 mm thickness;
- cleaning the place on a distance of minimum 10 meters from products or materials which may ignite when working with fire;
- providing steam or water screens for insulation purposes in case there are carried out works at a machinery, at the same time with the operations production area;
- providing access pathways;
- providing materials and equipment required for extinguishing fires as well as for supporting the firemen team (when needed);

During the entire period of repairs and interventions, the person in charge with the workplace as well as the head of the intervention workings are obliged to cooperate closely, to pursue occupational safety and labour protection measures, each one on his line of activity, intervening whenever required for preventing accidents.

After working inside the machineries, reservoirs and before closing the manholes, there shall be checked in detail if there were left people, tools, boards or other objects inside the machineries.

The work permit for interventions, repairs, etc. loses its' validity after the signing or the reception of the work (it corresponds from the OHS point of view) through which it is certified the fact that the work has been finished and that machine may be put into operation; any future remediation will be carried out only based on a new work permit.

CONCLUSIONS

The application of such procedures, according to the current requirements at European level, for the intervention and for carrying out workings in confined space from different industries, leads to:

- the increase of the occupational health and safety level by enhancing the capacity for intervention in safe conditions in case of faults, accidents, disasters, etc;
- the prevention of individual and collective work accidents, with dramatic consequences to which there are generally added adverse consequences on the heritage (national or private) affected by these events.

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IN SITU INSPECTION OF THE AERODYNAMIC PARAMETERS RELATED TO THE VENTILATION FACILITY USED ON THE MAIN FRONT CASTLE UPSTREAM ADDUCTION GALLERY FROM WITHIN A.H.E. SIRIU –SURDUC

GHERGHE ION*
LUPU CONSTANTIN**
CIOCLEA DORU**
BOANTĂ CORNELIU*
MORAR MARIUS*
RĂDOI FLORIN*

Abstract: *Designing an efficient aspirant system is a complex process which involves the knowledge of a great number of factors. Each suction hole has to be designed so that it doesn't allow dust or toxic and/or explosive gases to be recycled, and at the same time, it is required for the debris that clog up the ventilation pipes to be trapped. In order to find solutions to achieve partial ventilation there must be determined the specific parameters related to ventilation columns R_0 (air resistance per unit) and K_0 (unit coefficient of air losses through leakages). These coefficients are of special practical importance because they determine the efficiency of the partial ventilation installations to the greatest extent. The higher the R_0 resistance, the shorter the maximum length over which the air flow circulates towards the work face.*

Keywords: *ventilation, fan, aerodynamic parameters, water constructional works*

1. INTRODUCTION

Partial ventilation facilities are specific to all mining preparation works, as well as to the assembly of mining works for prospecting and geological exploitation.

In terms of structure conditions of use, partial ventilation installations essentially differ from main ventilation installations, being characterised by the following particularities:

- The aerodynamic resistance range at which a partial ventilation installation has to operate is extremely high (from R=0 in the initial phase of the mining work, to R_{max} which corresponds to the final length of the mining work), in a relatively small period of time;
- Continuous and significant change of operational parameters (air flow, pressure) in relation to the advancing of the mining work;

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* *Eng., INCD INSEMEX Petroşani, Romania*
** *PhD., Eng., INCD INSEMEX Petroşani, Romania*

- The installation is achieved step by step and the correct execution over the whole length directly influences the efficient ventilation;
- There are used ventilation columns made of metallic tubes with 400mm, 500mm, 630mm, 760mm, 800mm and 900mm diameters. As means of sealing there are used rubber gaskets attached with flanges and screws, respectively rubber sleeves.

2. VENTILATION SYSTEM OF THE FRONT CASTLE UPSTREAM ADDUCTION

The ventilation of the Surduc – Nehoiaşu main adduction gallery, front Castle upstream, is carried out by an induced system, through a partial ventilation installation in 3.160m length, comprising a column of metallic tubes with the diameter of ϕ = 900mm, two pneumatic fans placed in cascade and a centrifugal fan located at the surface.

In order to seal the ventilation column there are used flat rubber gaskets, the joining of the tubes being done with flanges and screws, and the fans from the column being VP500 type fans (no.2 and no.3).

The centrifugal fan located at the surface is a V496/5 fan with the following parameters:

- flow: - minimum: 17,100 m^3/h (285 m^3/min)
- maximum: 111,600 m^3/h (1,860 m^3/min)
- depression: - minimum: 150 mm CA (1,472 Pa)
- maximum: 370 mm CA (3,630 Pa)
- revolution: 970 RPM
- power: 110 kW

The driving motor of the fan has a 132kW power and a revolution of 1000 RPM.

When the measurements have been carried out, the two fans were located on the column of tubes, as it follows:

- Fan no. 2 at 2,779 m from the opening of the gallery;
- Fan no. 3 at 3,112 m from the opening of the gallery.

Axial fans used are of VP 500 type pneumatically driven, having the following characteristics:

- flow: - minimum: 3,600 m^3/h (60 m^3/min)
- maximum: 18,000 m^3/h (300 m^3/min)
- depression: - minimum: 75 mm CA (736 Pa)
- maximum: 230 mm CA (2,256 Pa)
- compressed air consumption: 5.7 Nm^3/min.

It is mentioned that on a length of approx. 40 m from fan no.3 to the work face, a column of flexible tubes is used for the aeration, each tube having a 5m length and 500 mm in diameter, these being used in order to protect the metallic column when blasting operations are carried out.

3. THE DETERMINATION OF AERODYNAMIC PARAMETERS FOR THE VENTILATION INSTALLATION MADE OF COLUMNS OF TUBES

In order to determine the K_0 and R_0 aerodynamic parameters, there have been carried out measurements for the entire 3,160 m long induced ventilation column.

These measurements focused mainly on the achievement of the following data:

- Static depression(pressure) in the ventilation column;
- Average dynamic depression (pressure) and air velocity in the column and in the air flow measurement stations;

- Air flow at the fan, in the work face and in static depression neutral points;
- Air temperature in the ventilation column and in the mining work, in the flow measurement stations;
- Barometric pressure in the mining work;
- Ventilation column diameter and length;
- Distance between fans and the column length related to each fan.

The results of th carried out measurements are presented in Table 1, according to the measurement stations presented in Fig.1.

Table 1**Results of the measurements carried out in the ventilation column**

Name of the work (adduction)	Measurement station	Static depression [pressure] h_{ST} [mm H_2O]	Average dynamic depression h_{dm} [mm H_2O]	Atmospheric pressure Pa [mm Hg]	Temperature t [^{0}C]	Air density ρ [kg/m^3]	Air velocity V [m/s]	Measured air flow Q_m		Corrected air flow Q_c	
								[m^3/min]	[m^3/s]	[m^3/min]	[m^3/s]
0	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*	*11*
Front castle Upstream main adduction gallery	1	-	-	685	11.0	1.124	9.10	107	1.78	98	1.63
	2	65	11.10	685	11.0	1.113	13.99	165	2.73	165	2.73
	3	8	1.455	685	11.2	1.119	5.05	193	3.22	193	3.22
	4	15	1.81	685	11.6	1.116	5.64	215	3.58	215	3.58
	5	24	1.96	685	11.6	1.121	5.86	224	3.73	224	3.73
	6	0	1.23	685	12.2	1.116	4.65	177	2.95	177	2.95
	7	346	17.86	685	12.6	1.073	18.08	690	11.50	690	11.50
	8	-	-	685	12.8	1.114	12.65	615	10.25	560	9.33

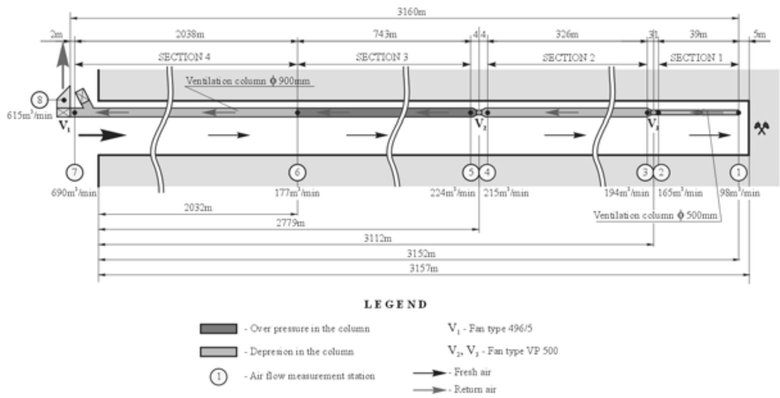

Fig. 1

The partial ventilation installation, for the 3,160 m column length, achieves a 690 m^3/min flow at the centrifugal fan and a 98 m^3/min flow at the work face, resulting in a 592 m^3/min. flow loss through leakages.

According to Fig.1, fan no.3, the closest to the work face (VP500) achieves a 98 m^3/min aspiration flow (station 1) of, resulting in a 67 m^3/min flow loss through leakages, over a 40 m column length.

The air flow from the working face is not enough. By using the ventilation installation in the current situation, there won't be achieved the necessary air flow at the work face.

In order to identify solutions for the ventilation, there have been determined specific parameters related to the ventilation column, R_0 (air resistance per unit) and K_0 (unit coefficient of air losses through leakages).

By using the data from Table 1, there have been determined the aerodynamic parameters related to the ventilation column:

- Aerodynamic resistance per unit, R_0 (kμ/m or 9.81xNs^2/m^8);
- unit coefficient of air losses through leakages over 1 m of the column (m^3/s/m la 1 mm H_2O or 9.81 Pa).

These coefficients are of high practical importance because they mostly determined the efficiency of partial ventilation installations.

In table 2 there are presented the aerodynamic parameters related to the ventilation column and obtained by using the data from table 1.

Table 2 **R_0 and K_0 aerodynamic parameters related to the ventilation column**

No.	Measurement station	Column diameter φ [mm]	Type of sealing	Segment length L [m]	Station no. 1		Station no. 2		Aerodynamic resistance per unit R_0 [kμ/m]	Air leakages through coefficient K_0 [m^3/s/m la 1 mm H_2O]	Ventilation type
					Static depression h_{St} [mm H_2O]	Air flow Q [m^3/s]	Static depression h_{St} [mm H_2O]	Air flow Q [m^3/s]			
0	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
1.	1	500	Flat rubber sleeves	39	65	2.73	0	1.78	0.35722	0.00453	Suction
2.	2	900		326	15	3.58	8	3.22	0.001897	0.0003269	Blowing-Suction
3.	3	900		743	24	3.73	0	2.95	0.003035	0.0003214	Blowing
4.	4	900		2038	346	11.50	0	2.95	0.004184	0.0003383	Suction

4. CRITICAL ANALYSIS OF EXISTING AUXILIARY VENTILATION

The partial ventilation installation in induced system comprises a column of metallic tubes with a diameter of ϕ = 900mm over a 3.120m length, a flexible column with a diameter of ϕ = 500mm over a 40m length towards the work face and three fans placed in cascade, of which two are pneumatic and one is electric (Fig. 1), split in 4 sections, 3 suction sections and 1 blowing section.

According to Fig. 1, under the influence of depression created by fan no.2, a 98m^3/min. flow is suctioned from the work face and circulated over the column towards the fan. From the work face to point 2 (fan), a 67m^3/min flow is suctioned from the gallery into the column through its leakages (flexible column jointed in 8 points over a 40m length), the flow in point 2 being 165m^3/min, and the static depression being 65mmH_2O (637.65 Pa). Under the influence of the pressure exercised by fan no.3 and the depression exercised by fan no.2, the static depression in point 3 is 8 mmH_2O (78.48 Pa), the air flow is 193m^3/min. Under the influence of depression created by fan no.2, air flow is circulated over the column towards point 4, where the flow is 215m^3/min. and the static depression is 15mm H_2O (147.15 Pa).

From point 5, under the influence of the pressure from fan no.2, air is circulated between points 5 and 6, through the leakages from the ventilation column a $47m^3$/min flow is blown into the gallery, the flow in point 6 being $224m^3$/min and the static depression being null.

From point 6, under the influence of the depression from fan no.1 (centrifugal fan), air is circulated between points 6 and 7, from where it is suctioned into the column through its leakages a $513m^3$/min flow, the air flow in point 7 being $690m^3$/min.

The column length related to each fan (Table 2) comprises two sections, of which on is under positive pressure and the other one under negative pressure from the atmosphere from the gallery. From sections with over-pressures there occur air leakages into the gallery, and in sections with depressions there occur air suctions from the gallery into the ventilation column.

Fan no.1 circulates the air on a 2.040m distance, fan no.2 on a 1.077m distance and fan no.3 on a 43m distance, the total length of the ventilation column being 3.160m.

The length of the suction sections is 2.410m, and the length of the blowing sections is 750m, suction sections representing 68.88% of the total length of the ventilation column.

5. ASSESSMENT AND INTERPRETATION OF RESULTS GAINED FROM IN SITU MEASUREMENTS

By processing the results from Table 2, the values of the parameters specific for the ventilation columns (R_0 and K_0) are very close, resulting the following average values:

- For metallic suction sections, the R_0 air resistance per unit is 0.0030405kμ/m;
- For the flexible suction section with the diameter of 500mm, the R_0 air resistance per unit is 0.35722kμ/m;
- For the blowing sections, the R_0 air resistance per unit is 0.003035kμ/m;
- The average air resistance per unit, R_0, of the metallic column with the diameter of 900mm is 0.003038kμ/m;
- The tightness level of the suction ventilation column with the diameter of 900mm expressed through the K_0 unit coefficient of air losses through leakages registered values between 3.383×10^{-4} m^3/s/m at 1mmH_2O and $3.269x10^{-4}$ m^3/s/m at 1mmH_2O;
- The average tightness level of the ventilation column is$1.37915x10^{-3}$ m^3/s/m at 1mmH_2O;
- The average air resistance per unit, R_0, of the ventilation column is 0.91584kμ/m.

By analysing the data from the two tables there may be noticed that the air flow depends on the R_0 air resistance per unit, on the diameter of the column of tubes and on the fans used. In order to increase the air flow at the work face, the replacement of the flexible column with the diameter of 500mm with a column with a larger diameter is required, the replacement of the two VP 500 fans with more advanced one being also necessary.

As the R_0 aerodynamic resistance is higher the maximum length for circulating the flow towards the work face is smaller.

Once with the increase of the length of ventilation columns, the required air flow at the fan increases due to the increased air leakages through the column of the tubes. For these reasons, circulating high air flows towards into the work faces from long mining works involves problems which are very difficult to solve.

CONCLUSIONS

1. For the ventilation of this work the aspirant ventilation system is applied, with ventilation columns made of metal, with a diameter of 900mm over a 3,120m length, a flexible column with a 500mm diameter over a length of 40 towards the work face in extension to the metallic column, their sealing being carried out using flat gaskets made of rubber.

2. The air supply in the ventilation installation is carried out using the centrifugal fan VP 496/5, and on the pathway of the pipes there are located VP-500 axial type fans cascaded on the ventilation column at 2,779m and 3,120m from the opening of the gallery.

3. Due to conditions specific for A.H.E. Surduc –Siriu, it is generally required to be circulated large air flows over long ventilation columns. In this regard, achieving the air flow required at the work face there has to be done using high capacity fans.

4. From the fresh air flow which is circulated on the gallery towards the three fans, a small part of fresh air reaches the work face (98m^3/min). The reduced air flow from the work face is due to the tightness level and to the diameter of the flexible tubes column used, respectively due to the length of the mining work.

5. Due to the recirculation of air, the actual aspirant ventilation is not achieved. The aspirant ventilation system requires that in the column, over its entire length, to be only depression from the atmosphere from the gallery. The column is split into sections, in which, in one there is depression and in the other one there is pressure. In this situation, each fan operates in suction-blowing regime.

6. The air flow required at the work face depends on the R_0 air resistance per unit, on the K_0 unit coefficient of air losses through leakages and on the diameter of the column of tubes and on the fans used.

A proper tightness level can be obtained by decreasing the K_0 unit coefficient of air losses through leakages, using current ventilation tubes, replacing the current flexible ventilation column with a flexible ventilation column having a larger diameter, and the length of the tube to be 5m minimum.

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SYSTEMS USED TO EVALUATE THE PERSONNEL OPERATING IN TOXIC / EXPLOSIVE / FLAMMABLE ATMOSPHERES IN THE MINING INDUSTRY

ILIE COSMIN*
GAMAN ARTUR GEORGE**
PUPAZAN DANIEL***

Abstract: *The evaluation systems represent an essential component part of artificial intelligence that cover the expert knowledge for a specific domain; subsequently, this knowledge shall be dynamically capitalized by a reasoning mechanism. Thus, an artificial reasoning is implemented whose idea is to simulate natural reasoning triggered by the human brain. The assessment system allows knowledge testing by going through a set of questions randomly selected from several chapters, specific to rescue and intervention activities in toxic / explosive / flammable atmospheres.*

Keywords: *evaluation system, human expert, rescue personnel, breathing apparatus, hostile atmosphere*

The assessment systems can be defined as being computer programs that make use of the experience and knowledge of experts in order to solve complex problems in a restricted field and they are based on the research results of the artificial intelligence in the fields of knowledge representation, inference methods and natural languages.

The development of an assessment system requires interaction between an expert, who is a specialized person in the area of knowledge for which the expertise is intended and a knowledge engineer, who is an analyst programmer. The knowledge engineer will take over the expertise of the expert and will convert it into a knowledge database which is then released for the benefit of the users through the system.

The assessment system will be able to simulate the human judgement by obtaining solutions for the analysed problems, solutions which will be explained and justified. The assessment system will allow users to access the expertise in a way similar to consulting a human expert, obtaining a similar result after the consultation.

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* *Student PhD. – Scientific researcher – INCD INSEMEX Petrosani*
** *PhD. Eng – II Scientific researcher – INCD INSEMEX Petrosani*
*** *Phd. Eng – III Scientific researcher – INCD INSEMEX Petrosani*

Development of assessment systems can be seen as 5 interdependent and overlapping phases:

- The identification
- The conceptualization
- The formalization
- The implementation
- The testing

Programming simulation models in order to implement them on computers is done using simulation languages (software packages). The purpose is to obtain high level assessment systems through which there can be tracked the development of a process and its dynamic behaviour.

The use of the assessment systems within the simulation / assessment of rescuers interventions process brings several advantages:

- Performance – assessment systems maintain their knowledge in time, being able to work continuously (they don't forget and don't get tired);
- Reproduction – there can be easily made several copies of an expert system while creating new human experts is often a long and very expensive process;
- Efficiency – they lead to low cost expertise, being able to easily multiply; the development costs can be covered by several users;
- Consistency – similar actions are handled in the same way; it cannot be influenced to draw conclusions, being very objective unlike the human experts that can be subjective;
- Documentation – an expert system can provide continuous documentation regarding the decision process;
- Completeness – an expert system is able to review the entire reasoning, unlike the human expert who retains only the main steps in case of complex expertise;
- Operating speed – a conclusion can be drawn shortly due to the power offered by the computer systems;
- Multilateral approach – the knowledge of several human experts can be comprised into an expert system so that the solutions can be much better than the ones of a single expert;

The environments for developing assessment systems are software packages that simplify the task for building an assessment system. They vary from high level programming languages to low level support systems.

The development environments for achieving the expert system are PHP and MySQL.

The advantages of these development environments are given by the following characteristics:

- Familiarity: the syntax of the language is very easy and it is combining the syntaxes of some of the most popular languages like Perl or C;
- Simplicity: the syntax of the language is free. Library inclusion or compiler directives are not necessary, the PHP code included in a document executing itself between the special markings;
- Efficiency: PHP is used by resource allocation mechanisms, very necessary for a multi-user environment such as the Web is.
- Security: PHP provides the developer with a flexible and efficient set of safety measures;
- Flexibility: being born from the necessity of Web development, PHP has been modular built in order to keep up with the development of different technologies. Not being

related to a particular web server, PHP has been integrated for various existent web servers: Apache, IIS, Zeus, Server, etc.;

- Gratuitousness: it is probably the most important characteristic of PHP. The PHP development under open-source license has led to fast adapting of PHP to the Web needs, code securing and improvement.

The program is installed on a server and is designed so that it can be run on an unlimited number of computers.

The assessment system allows knowledge testing by completing a set of questions randomly chosen from several chapters, specific for the chosen field

The assessment system allows the creation of a database with the recordings of the names of tested persons, of the testing field, of the date and of the test results.

The operating system is built so that it allows modifying the course, respectively the configuration of questions depending on the specific field of activity.

In order to liquidate underground faults [2], the rescuers work – protected by breathing apparatus [1] – in hard areas, in tights spaces, with poor visibility, and most times at high temperatures and humidity, they are subject to a great effort, in the presence of toxic and/or explosive gases, with the possibility to be sometimes repeated and so to affect their personal integrity.

Working in this kind of situations, in which the activity has a high risk level the rescuers have to be recruited, tested, instructed, authorized and trained properly.

The intervention of rescuers involves high effort, and the specific of rescue operations, through the particularities of underground workplaces in case of faults, increases the difficulties in the execution of fault liquidation operations

INCD INSEMEX Petroşani has a unique training facility [1], similar to the ones from the countries with tradition in this field and which comprises training spaces where practical exercises of rescuers are carried out, with several difficulty levels, and which allows the simulation of intervention activities in confined spaces, horizontally and vertically, low visibility environment, high temperature and humidity. The installations, devices and operations carried out during practice are conceived so that to be close to real conditions encountered in different hazardous situations.

After the assessment system is installed on a computer, the use of it by the users involves the following steps:

1. The assessment system displays the following window at start-up:

Fig. 1

2. The program starts with registering the tested person into the programs' database.

The database of the program will be built so as to allow the input of the following information: name and surname of the tested person, personal identification number of the tested person, name of the company where the person carries out its' activity, occupation of the tested person.

The program displays the following window:

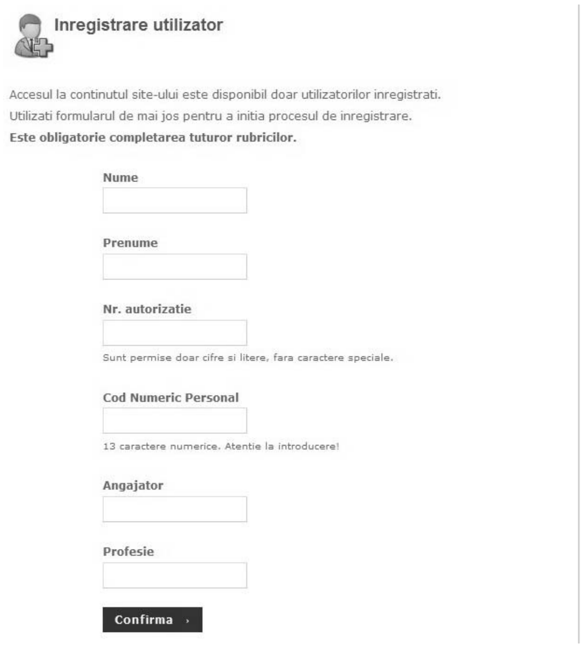

Fig. 2

3. We don't have to register as a new user if we are already a registered user that had been registered before into the database. We have to login using the name, surname and the number of the rescuer authorization [2].

In this case the program displays the following window:

Fig. 3

4. After we have registered as a new user or after we have logged in as an existent user, the program displays the following window:

Fig. 4

5. Depending on the chosen activity field, the program allows viewing the chapters of the course: “Rescuing in toxic environments” [4].

In this case, the program displays the following window:

Fig. 5

For the extractive industry [3] (underground) there can be viewed the following chapters:

- Definition and principles of the intervention and rescue activities in toxic/explosive/flammable environments;
- Normative regarding intervention and rescue activities at industrial facilities with toxic and/or explosive gas emission hazard;
- Insulant respiratory protective equipment based on compressed-oxygen;
- Insulant respiratory protective equipment based on compressed-air;
- Resuscitation equipment;
- First aid providing;
- Mine air;

- Mining breakdowns and methods to combat them;
- The entry of rescuers into action;
- Procedures to be followed by rescuers;
- Regulation of rescuers interventions for the extractive industry;

Viewing these chapters is not mandatory.

6. After viewing the chapters from the course the program allows browsing through the history of previous viewed courses.

In this case, the program displays the following window:

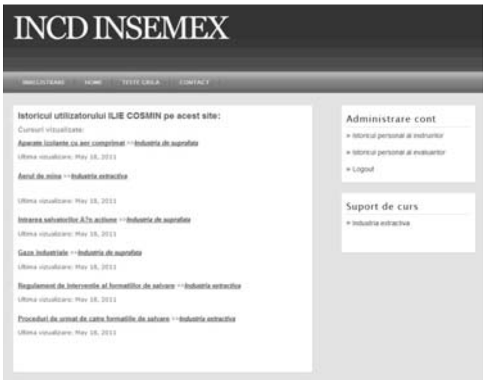

Fig. 6

The test has 30 minutes assigned for work during which the tested person has to answer a 30 questions set, questions that are randomly generated from the complex database of the program.

7. At the end of the 30 minutes allocated time or at the end of the 30 questions, there will be generated an examination report for the tested person.

The tested person passes the test if it responds correctly to at least 24 questions.

In this case, the program displays the following window:

Fig. 7

8. The examination report [4] will comprise:

- name of the tested person;

- domain of activity;
- the unit where it carries out its activity;
- date of the test;
- total number of questions;
- number of correct answers;
- number of wrong answers;
- option for viewing the wrong answers and the correct variants of the questions;
- time used for test completion;
- result of the test (passed/failed);
- name of the person that supervised the test.

CONCLUSIONS:

The assessment system allows knowledge testing by completing a set of questions randomly chosen from several chapters, specific for the chosen field

The assessment system allows the creation of a database with the recordings of the names of tested persons, of the testing field, of the date and of the test results.

The assessment system can be used in the instruction / reinstruction process from within GAS INSEMEX as well as in the process for theoretical reinstruction of personnel for intervention and rescue in toxic / explosive / flammable environments [3] from within the rescue stations.

The operating system is built so that it allows modifying the course, respectively the configuration of questions depending on the specific field of activity.

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RESULTS OF RESEARCH ON MODIFYING THE CONSTRUCTIVE SHAPE OF THE CHARGES FOR THE BLASTING TECHNOLOGY WITH LONG BOREHOLES APPLIED AT THE EXPLOIT METHOD WITH SUB MINED TECHNOLOGY IN JIU VALLEY STAPES

KOVACS ATTILA*[*]
GHEORGHIOSU EDWARD[**]
RUS DANIELA-CARMEN[**]
RĂDOI FLORIN[**]

Abstract: *Currently the most applied exploiting technology in the Jiu Valley for coal extraction is based on sub mined layer for what is necessary long boreholes blasting operations with discontinuous blasting charges. Comparing with the former used exploding charges, tested on the faze of monitoring for approval of the blasting method, in this period it was made a series of changes to the used explosives materials and also the necessity to evaluate the security level for the new discontinuous blasting charges developed with these. These changes appeared on behalf of new disposable explosive materials, with different characteristics should revealed the necessarily of reevaluation of security and technological efficiency aspects.*

Keywords: *Safety discontinuous blasting charges, security level, sub mined layer stapes.*

1. INTRODUCTION

The objective of the research is to assess and compare levels of security for different configurations of special charges for long hole stapes used to extract coal from the Jiu Valley undermined bench, to assess the security of such loads in air-methane and air-methane – coal dust atmosphere, loads consist of explosives for civil uses certified, safe for use in underground flammable and / or explosive environments.

Expected results of research: designing the types of charges for long-hole blasting technology;

- Establishing different configuration for the discontinuous charges for long hole blasting technology;
- Experiments on types of discontinuous charges for blasting technology for evaluation of the safety level of these charges in air-methane and air-methane-coal dust atmospheres.

[*] *PhD. Eng., INCD INSEMEX Petroşani, Romania*
[**] *Eng., INCD INSEMEX Petroşani, Romania*

• Improving occupational health and safety level, economic efficiency corresponding to the long hole blasting operations in mining technology applied undermined workings bench in Jiu Valley;

Work technology of exploitation method consists of extracting of the coal bed in undermined, the basic slice stapes, followed by removal by gravity unloading coal from the undermined bench working behind the front line.

As a basic operation of the stapes start-ups in the formation of free surfaces for separation and gravity flow of coal from the bench undermined to achieve delimiting surface behind stapes, should apply the solution with long-hole mining.

In principle, the technical solution to achieve delimiting surface consists of drilling holes and blasting of three types of holes (Fig. 1):

• Regular mine holes, length 1.5 m, denoted l,1', located at a distance of 0.8 m horizontally between each row of beams (field reinforcement);

• Long holes symbolized 2,2', running on length approximately equal to half the width/ height bench of coal, spaced 1.6 m horizontally;

• Long holes symbolized 3 ', which runs the entire height of the bench 1.5 m from artificial ceiling and 3, the holes is approximately to 1.0÷1.5 m above bench height running 1.0÷1.5 approximately if the ceiling natural work, located at a distance of 1.6 m in the horizontal plane but different from long holes 2. In express cases, when high strength of coal, both long holes will be located at distances of 0.8 m horizontally.

The blasting are steamed to the mouth on length 0.6-0,7 m for the regular ones and 1-1,5 m for the long holes 2 and 3. The steam material is clay.

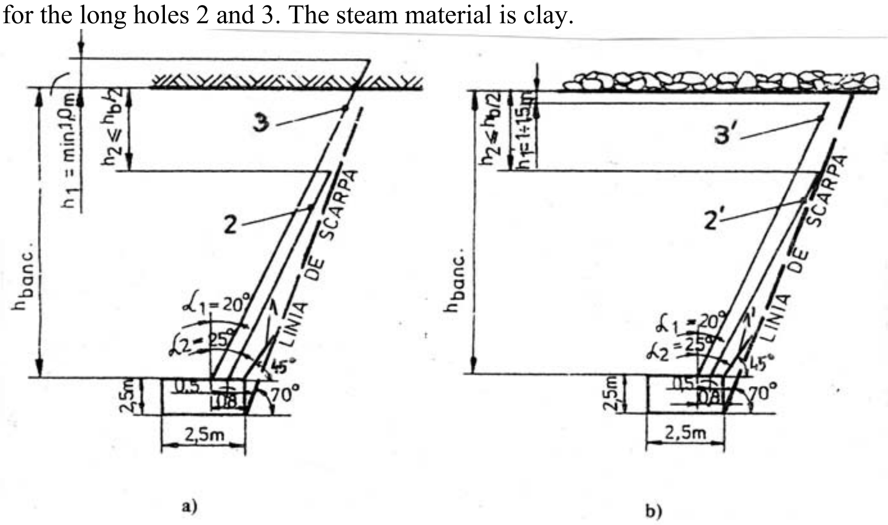

Fig. 1 Long hole placement scheme for training mine delimiting surface start workings of subversion: a - operating under the ceiling nature b – operating under artificial ceiling;

1,1 '- short holes made in the ceiling natural or artificial ceiling; (lg (1), lg (1')); 2,2 '- long hole in the ceiling made naturally or artificially below the ceiling; (lg (2) lg (2')); 3,3 '- long hole in the ceiling made naturally or artificially below the ceiling (lg (3) LG (3)). Explosive charges manufacturing is in accordance with the technical requirements for application in underground long-hole blasting technology of mine, endorsed / approved under Approval no. 6 of 17.05.2001, the Ministry of Lab our, Social Solidarity and Family, specifying:

• As a means of fire and explosives used in blast:
- ✓ Safety explosion-proof explosive (AGP manufactured by NITROEXPLOSIVES Făgăraş);
- ✓ Explosion-proof detonating cord (RIOCORD 6 g / m manufactured by MAXAM Spain);
- ✓ Electrical detonators in explosion-proof construction with probability of ignition of methane in lower than 4% of all tested staples.

- The construction of loads shall be in PVC Omega tubes, assembled discontinuous explosive charge consisting of alternating batch of 1-3 cartridges of explosive safety explosion-proof (100-300 g) and sodium chloride cartridge at diameter and mass as explosive cartridges.
- Sodium chloride used in the construction of long loads discontinuous grain will have the following:
  - ✓ 90 % passing through a sieve with a mesh of 1.0 mm square with sides;
  - ✓ 50 % passing through a sieve with a mesh of 0.5 mm square with sides.

To properly assess the security level on ignite fire damp atmosphere was a case study on the application of loads staple recently used in long-hole blasting technology training bench stapes delimiting start undermining no. 2-3, layer 3, Block VI, skyline 350, E.M. Vulcan.

In practice to minimize the risk of ignition of explosive atmospheres is used to load a configuration similar to that shown in Figure 2: in Omega tube cartridge rock salt 100 grams superfine (minimum 90% passing through the sieve with quadratic side of 1.0 mm with 50% passing through a sieve with square mesh with sides of 0.5 mm) was inserted in a continuous load two cartridges 250 grams of explosive (methane explosion-proof) powdery type Wasagit - Metanit. At the ends of each tube are placed superfine cartridge rock salt 100 grams. Assemble the two millisecond electrical detonators with low intensity type susceptible to firedamp environments RIODET S Microretardo, 30 ms, tube and Cu wires with a length of 3 m, the detonating cord (explosion-proof) type RIOCORD PS 6 g / m at the right of the first cartridge of salt.

For such discontinuous charges the explosive – salt ratio is between 1.34 to 1.90, depending on the length of the assembled charge.

This type of construction of explosive charge, the reports specified contains less than explosive configurations described in the previous step, which leads to higher levels of security.

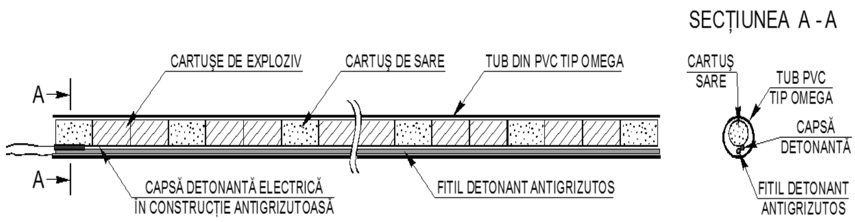

Fig. 2 The model of construction of discontinuous load

The construction of discontinuous charges used in long-hole blasting technology to do the undermining bench stapes delimiting surface starting no. 2-3, layer 3, Block VI, skyline 350, E.M. Vulcan by interleaving a pile of rock salt 100 grams superfine at a continuous load two cartridges explosion-proof explosion of 250 grams, has an appropriate level of security.

All types of explosives used in long-hole blasting technology are tested for every batches at INCD-INSEMEX, with particular emphasis on safety checks to methane and coal dust explosion-proof, explosives safety, the safety from methane electrical detonators safety checks to that of methane and coal dust explosion-proof detonating, observing their proper quality.

Change in the last two three years of explosion-proof explosive type (Slavit V, Wasagit, Metanit, Metanit Specially E7H), means of initiation (Riodet S and Ergodet) and explosion-proof flexible detonating cord (Riocord PS on Riocord PS 6), did not affect security level of long-hole blasting technology while maintaining the same configuration of the charge.

The application long hole blasting technology described in this paper does not generate other additional risks such as unexpected increase in the concentration of methane after blasting, changes in ventilation parameters, disturbance and occurrence of mine support or collapses of rocks.

By conducting the research's occurred proposals were made to improve long-hole blasting technology:

✓ Where coal bench rock intercalations are hard to break down, should be reconsidered blasting monograph by reducing the distance between groups of holes, for each field of mine support applying at least one group;

✓ Increased density of hole groups will lead to a more efficient fore - breaking and avoid oversize to download phase and elimination the emergence non blasted holes;

✓ For fore–breaking and download consequently, an improved percentage recovery of coal bed is recommended to begin the delimiting surface of stapes and leave it unmoved to ensure at least a week under the effect of pressure for mining, coal cracking is more pronounced;

✓ Improving blasting monograph by completing each additional holes, made vertical holes, drilled to 1-1.5 m upper limit of the bench;

✓ To reduce execution time delimiting surface is proposed use of multiple drilling systems to drilling the holes, leading to prevention of dangerous phenomena that occur in the holes;

✓ To evacuation the marginal area proposed drilling holes in groups with the an arrangement parallel to the front line and perpendicular to the lying plane;

✓ In order to optimize the fore breaking of the coal in the marginal area it was recommended to reconsider the drilling and blasting monographic. In this order it is necessary to drill a number of supplemental holes with decreasing the distance between the holes groups and increasing the charges blast effect.

✓ It is recommended that for the first two fields of fans from every stage of shooting , to shoot with the same level of delay, to generate a seismic effect more pronounced with improved results in terms before cruising;

✓ while the natural slope angle is greater than the angle of inclination of the layer, will drill holes in a configuration that allows the execution of the download to bed, ensuring optimal evacuation of coal lying area reducing the risk of developing the phenomenon of spontaneous combustion;

✓ Any modification or change of the building of explosive load used in blasting technology with long holes (explosive explosion-proof / flexible detonating cord / detonators) requires being test by INCD INSEMEX in order to confirm / maintain the level of security against the risk of ignition of fire damp mixture.

For application INSEMEX accredited procedure PI-ETI-6.1 SAFETY CHECK TO METHANE OF THE ELECTRICAL DETONATORS is necessary to use a special stand where to detonate the detonators in the presence of explosive atmospheres air - methane.

Modernization aimed at relocating the stand in a space in which to improve security conditioners and ergonomics for operators. Equipped with an electro mix aeration chamber of the tunnel much faster by purging and also provide complete certainty flue gas exhaust. Measurement of mixed air - methane currently is done with a device type ALTAIR 5 Multigas that besides indicating the concentration of methane measured 0.02% and oxygen concentration.

Make a continuous measurement device operator just follow this indication during the execution of the air - methane and stop gas supply to achieve the specified concentration of 8% - 10% methane.

Triggering the explosion with the blasting - proof is that limited energy pulse in time to 4 milliseconds, that after the detonators function the current to be interrupted.

The test method involves individual testing of 50 detonators each stage of delay, the admission criteria of the lot being susceptible to firedamp atmosphere ignition max.4% of the total number tested detonators.

CONCLUSIONS

Following the evaluation of research results analyzed general conclusion is that technology application with modifications described is safe as far as compliance with all security measures provided both explosives used to make loads and the application of technological operations.

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ESTABLISHING RETENTION EFFICIENCY FOR ELECTRIC FILTER NO. 1 AND NO. 2 RELATING TO BOILER NO. 7 OF THE S.C. CET GOVORA S.A.

KOVACS MARIUS*
GHEȚIE GHEORGHE**
TOTH LORAND*
DRĂGHICI ANGELICA*

Abstract: *The Power Plant Govora SA is a leading producer of electricity and heat in our country. Currently the functioning of the unit is the subject of compliance with a program of gradual reduction of greenhouse gas and dust. This paper aims the determination the parameters of the dust and exhaust gas stack no. 3 of SC GOVORA CET in order to establish these pollutants removal efficiency by electrofilter. INCD INSEMEX - Petroşani determined dust concentration in effluent water on the smoke stuck no. 3 in accordance with Ord 462/1993.*

Keywords: *emission, dust, stationary sources,*

1. GENERAL INFORMATIONS

Profile of activity of the SC Central power plants Govora is to sgenerate power for industrial platform OLTCHIM Rm.Vâlcea, providing thermal energy for heating the district Rm. Vâlcea and producing electricity for the national system. [4]

Design and construction of CET Govora began in 1950, in successive stages, with the development of chemical platform Rm. Vâlcea.

Currently the unit is operating with 5 boilers (C3, C4, C5, C6 and C7) with capacities of each 420t / h each and 4 turboagregate (T3, T4, T5 and T6) of 50 MW.

Fig. no.1 Overall view CET GOVORA

* *Ph.D. Scientific Reasearcher III - INCD INSEMEX Petroşani*
** *Eng. Head of Environmental Protection Laboratory- INCD INSEMEX Petroşani*

The power plant production capacity is 2100 t / h steam at 140 and a power of 200 MW in cogeneration system delivering 485 t / h and 270 industrial steam Gcal / h hot water.

Boilers C3 and C4, which are in an advanced state of decay, gas are connected to the chimney no.1. These boilers with turboagregatele T3 and T4 form large combustion plant LCP1.

The C5 and C6 operated boilers based on lignite are connected to the smok stack No.2 and C7 to the flue boiler No.3.

The C5 and C6 boilers with turboagregatele T5 and T6 form large combustion plant LCP2 and C7 boiler with coal-based operation also with that turboagregatul LCP3 form large combustion plant.

The flow process of Govora is shown in Fig. no. 2

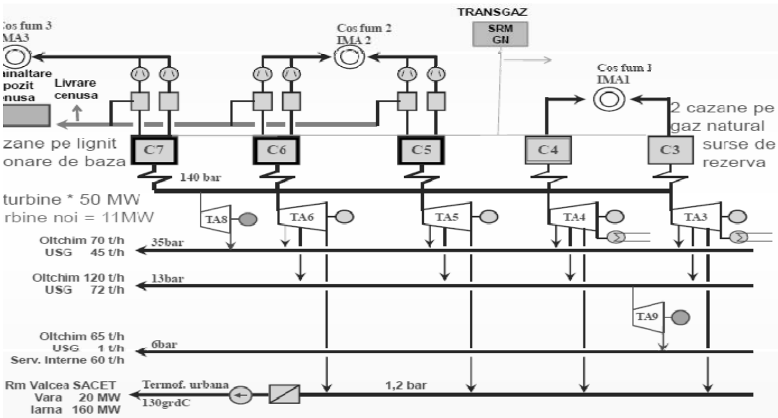

Fig. no. 2 The flow of CET GOVORA

2. DOCUMENTARY RESEARCH ON REDUCTION PROGRAM ELEMENTS OF PROGRESSIVE AND PARTICULATE EMISSIONS FROM SC CET GOVORA SA

2.1. Elements of the program for large combustion plant (LCP) LCP 2

According to the Decision no. 440/2012, owners large combustion plants not complying with emission limit values should have and update programs for the progressive reduction of dust and gas.

The progressive reduction of emissions has been developed with environmental authorities, which were established measures and their deadlines. [5]

LCP2 of CET GOVORA is a large combustion plant type with mixed outbreak.

LCP2 is powered by coal (lignite) and the support of hydrocarbons that lignite has a stake of between 50-88% of the load of the boiler and natural gas, or fuel oil, a stake of between 12-50%.

Smoke stuck No. 3 has a height of 140m and at the base and tip diameters of 15.2 or 7.2 m.

Coal used is the average calorific 1890 Kcal/Kg

The percentage of sulfur in coal is between 0.8-0.92% and the oil between 0.61 - 0.93%.

Ash content measured anhydrous (average reference) is 44.5%.

Burning coal dust is suspended support flame on oil or natural gas.

To this end there are 6 crushing-mill with 6 burners for coal dust, 4 gas burners 4 burners support load and 4 burners 4 burners support heavy fuel load

Flue gas evacuated to stack no. 2 has 2 fans on each boiler gas.

As exhaust gas dedusting systems is used electrofilter.

For these coal boilers have an dust retention efficiency over 99.7%.

Works on the principle of electrostatic retention of ash particles crossing the strong electric field between the electrodes and the emission of deposit.

Slag and ash from silos electro mixed with water and reach the basin Bagger pumps.

With these pumps fluid mixture in a ratio of 1/10 through pipelines is discharged to the ash deposit.

Ash warehouse is equipped with sprinkler systems to prevent emissions of dust.

For analytical evaluation of pollutant emissions, the unit used CORINAIRE method.

Emission factors used by the CORINAIRE method are presented in Table no. 1.

Table no. 1

No. crt.	Fule type	emission factor		
		SO_2 (Kg/GJ)	NO_x Kg/GJ)	Dust (Kg/t)
1.	Methane gas	0.00041	0.15	0,1216*

Emission limit values (ELV) according to HG 541/2003 and those imposed by the environment through their progressive reduction program are given in Table no.2.

Table nr.2

Crt. no	Title of Large Combustion Plant	Thermal power (MW)	ELV SO_2 (mg/Nm^3)	ELV NO_x (mg/Nm^3)	ELV dust (mg/Nm^3)
1	LCP2(C5+C6)	2x349	400	- 490 by the end of 2015 - 200 from year 2016	50
2.	Fuel oil	Depending on the sulfur percentage of the fuel.	0.19	0,42*	
3.	Coal	Depending on the sulfur percentage of the fuel.	0.26	1,04*	

Currently the Guverment Decission (GD) 541/2003 was repealed by GD 440/2010.

For installation LCP2 requested a transitional period until the end of 2013 for SO_2, NOx and particulate late 2011.

Currently Govora is the procedure for obtaining the integrated environment permit.

The proposal for LCP 2 technological measures imposed by the environment, leading to progressive reduction of emissions to limit values are given in Table 3

2.2. The program elements for large combustion plant LCP 3

Large combustion plant no 3 (LCP3) is composed of energy boiler type CR1244 and turboagregatul No.7.

This facility is connected to stack no. 3 with 140m height and diameter at the base and top DE15, 2 or 7.2 m.

The percentage of sulfur in coal and oil is the same as for LCP 2.

The ash content is also measured in anhydrous 44.5%.

Burning coal dust is similar to that described above. Technological process of dedusting is the same as the LCP 2.

For analytical calculation of dust and gas were used the same emission factors as for LCP 2.

Emission limit values (ELV) according to GD 541/2003 and those imposed by the environment through their progressive reduction program are the same as in LCP 2.

The proposal for a technological measures imposed by the environment, leading to progressive reduction of emissions to limit values are given in Table 4.

Table no.3

Crt. No	The proposed measure	Installation	Date upon becoming operational	Effectiveness in reducing of SO_2 (%)	Effectiveness in reducing of NO_x (%)	Effectiveness in reducing of dust (%)
1.	BAT flue gas desulphurisation in power industry	Boilers No. 5, 6, type CR 1244, 420t/h – LCP2	2013	Over 95%	0	0
2.	Reducing nitrogen oxides according to BAT in energy industry.	Boilers no.5,6,type CR 1244, 420t/h – LCP2	2013	0	30%	0
3.	Reduction of dust emissions by following BAT and BREF from energy industry.	Boilers no.5,6,type CR 1244, 420t/h – LCP2	2011	0	0	99,9%
4.	Provide flue gas monitoring systems.	Boilers no.5,6,type CR 1244, 420t/h – LCP2	2006	-	-	-
5.	Purchase of fuel oil containing less than 1% S	Boilers no.5,6,type CR 1244, 420t/h – LCP2	2007	20%	0	0

Table No.4

Crt. no	The proposed measure	Installation	Date upon becoming operational	Effectiveness in reducing of SO_2 (%)	Effectiveness in reducing of NO_x (%)	Effectiveness in reducing of dust (%)
1.	BAT flue gas desulphurisation in power industry	Boilers no.7, type CR1244,420t/h – LCP3	2011	Over 95%	0	0
2.	Reducing nitrogen oxides according to BAT in energy industry.	Boilers no.7, type CR1244,420t/h – LCP3	2011	0	30%	0
3.	Reduction of dust emissions by following BAT and BREF from energy industry.	Bolers no.7, type CR1244,420t/h – LCP3	2011	0	0	99,9%
4.	Provide flue gas monitoring systems.	Boilers no.7, type CR 1244, 420t/h – LCP3	2006	-	-	-
5.	Purchase of fuel oil containing less than 1% S	Boilers no.7, type CR 1244, 420t/h – LCP3	2007	20%	0	0

2.3. Other measures proposed

The future strategic development plan of Govora:

- Compliance with the limits set by the national allocation plan for emission rights of greenhouse gases and will further supply of electricity as green energy, at a rate of 8.3% of national energy,
- Compliance on the acquisition and sale of green certificates for electricity supply delivered
- on-line monitoring of gas emissions from smok stuck,
- gas desulphurisation plant - realization for LCP3 and LCP2,

3. DETERMINATION OF SPECIFIC PARAMETERS AND DUST EMISSION CONCENTRATION IN STUCK NO. 3 OF LCP 3

At the request SC Govora SA, Laboratory of Environmental Protection in from INCD INSEMEX Petrosani has made determination of specific parameters and concentrations of particulate emissions discharged from stack no. 3, No. 7 respectively for the electro boiler No. 1 and 2 of the LCP 3. [1], [2], [3], [4].

Equipment used for measurements were:

- isokinetic sampling equipment -TECORA-BASIC ISOSTACK 748589P Series

-type analytical balance METTLER TOLEDO XS205DU / M

The test method used:

INSEMEX procedure PI-02 " Manual Determination of mass concentration of particulate ".

Referential assessment:

-Order no. 462 of July 1, 1993 for approval of technical conditions for the protection of air and methodological standards for determining emissions of air pollutants produced by stationary sources.

-GD. 440 of 28 April 2010 laying down measures for the limitation of emissions of certain pollutants from large combustion plants.

Results of measurements made are centralized in Table no. 5. [6]

Table no.5

<table>
<tr><th rowspan="2">Place of sampling</th><th rowspan="2">Operation</th><th colspan="10">Auxiliary parameters *</th><th rowspan="2">The average concentration of dust [mg/Nm^3]</th></tr>
<tr><th>H [m]</th><th>D [m]</th><th>T_s [^{0}C]</th><th>T_a [^{0}C]</th><th>Va [m/s]</th><th>Q [m^3/h]</th><th>Pa [KPa]</th><th>O_2 [%]</th><th>CO_2 [%]</th><th>W [%]</th></tr>
<tr><td rowspan="3">Smoke stuck of boiler no. 7</td><td>Maximum operating system</td><td rowspan="3">75</td><td rowspan="3">7,4</td><td>144,55</td><td rowspan="3">20,4</td><td>5,7</td><td>882.360</td><td rowspan="3">97,2</td><td>9,6</td><td>3,12</td><td>8,46</td><td>57,02</td></tr>
<tr><td>Functioning of electrofilter no. 1</td><td>141,31</td><td>5,1</td><td>789.480</td><td>11,8</td><td>2,4</td><td>8,1</td><td>61,1</td></tr>
<tr><td>Functioning of electrofilter no. 2</td><td>137,48</td><td>5,3</td><td>820.440</td><td>11,6</td><td>2,1</td><td>9,24</td><td>65,59</td></tr>
</table>

Tabel legend:

H- Height of source [m];

D- The diameter of the measuring section [m];

T_s- The average air temperature in the source [^{0}C];

T_a- Ambient temperature [^{0}C];

V_a- Average air velocity from effluent [m/s]

Q- Average flow of gas [m^3/h]

P_a-Atmospheric pressure [KPa]

O_2- Oxygen concentration from effluent [%]

CO_2- Carbon dioxide concentration from effluent [%]

W- Humidity [%]

Specific parameters from measurements of stack effluents discharged No.3 resulting dust concentrations exceed 50 mg/Nm3, one that calls for further measures in order to comply with the limit values of GD no. 440/2010.

Dust collected from stuck is made by isokinetic compliance so were determined and specific parameters of air chimney No.3 (effluent velocity, gas flow, mean temperature, pressure, humidity and oxygen concentration) in 3 modes flue gas fan running IDE

With these parameters were recalculated exhaust flow from the stuck.

The results of the measurements are shown in table no. 6.

Table no. 6. -Regime 1 - Flow boiler 420 t / h total gas flow at the entrance IDE 1,025,000 m^3 / h

Measured parameters			
Crt. no.	**Name**	**Unit of measurement (UM)**	**Value**
	The average temperature	°C	144,55
	Pressure	hPa	972
	Humidity	%	8,46
	Oxygen	%	9,6
Calculated parameters			
1	Dry gas flow	m^3/h	882.360
2	Normal condition of gas flow	Nm^3/h	218.971,13
3	Gas flow corrected 6%0_2		1.161.000
4	Dust concentration Nm^3/6 % O,	mg/Nm^3	57,02

Regime 2 - Maximum flow rate ELFI 565 000 m^3/h

Measured parameters			
Crt. no.	**Name**	**U.M.**	**Value**
	The average temperature	°C	141,31
	Pressure	hPa	972
	Humidity	%	8,1
	Oxygen	%	11,8
Calculated parameters			
1	Dry gas flow	m^3/h	789.480
2	Normal condition gas flow	Nm3/h	200.413,69
3	Gas flow correct 6%02		1.287.195,7
4	Dust concentration Nm3/6 % O,	mg/Nm^3	61,1

Regime 3 - Maximum flow rate ELFI 565 000 m^3/h

Measured parameters			
Crt. no.	**Name**	**U.M.**	**Value**
	The average temperature	°C	137,48
	Pressure	hPa	972
	Humidity	%	9,24
	Oxygen	%	11,6

Calculated parameters			
1	Dry gas flow	m^3/h	820.440
2	Normal condition gas flow	Nm^3/h	208.273,05
3	Gas flow correct 6%02		1.309.212,8
4	Dust concentration Nm3/6 % O,	mg/Nm^3	65,59

CONCLUSIONS

At the request SC Govora SA, Laboratory of Environmental Protection from INCD INSEMEX Petrosani has made determinations of specific parameters of the emission of dust from stack no. 3, which ensures the boiler flue 7.

The purpose of these measurements was to see efficiency for retaining particulate matter of electrofilter. These electrostatic precipitators were made based on the previous year and are on probation.

To this end, we made a documentary study on the progressive reduction program and particulate emissions SC Govora SA, measures program and deadlines set imposed by the environment, which allows operation of the unit in compliance with the law.

Specific parameters from measurements of stack effluents discharged No.3 resulting dust concentrations exceed 50 mg/Nm^3, one that calls for further measures in order to comply with the limit values of GD no. 440/2010.

REFERENCE:

[1] *Ordin nr. 462 din 1 iulie 1993 pentru aprobarea Condiţiilor tehnice privind protecţia atmosferică şi Normele metodologice privind determinarea emisiilor de poluanţi atmosferici produşi de surse staţionare*

[2] HG nr. 440 din 28 aprilie 2010 *privind stabilirea unor măsuri pentru limitarea emisiilor în aer ale anumitor poluanţi proveniţi de la instalaţiile mari de ardere.*

[3] *Ghid pentru monitorizarea şi automonitorizarea emisiilor de dioxid de sulf, oxizi de azot şi pulberi, provenite de la instalaţiile mari de ardere*

[4] SC CET GOVORA SA-*Tradiţie şi inovare*

[5] *Program de retehnologizare şi ecologizare*

[6] Raport de evaluare nr. 92/2012 *privind concentraţia de pulberi din suspensie realizat de INCD INSEMEX – Petroşani*.

DEVELOPING A DATABASE ON ACCIDENTS CAUSED BY EXPLOSION IN THE INDUSTRIAL AND CIVIL FIELD

SZOLLOSI-MOTA ANDREI*
PRODAN MARIA*
NALBOC IRINA**

Abstract: *According to Council Directive 96/82/EC on the control of major accidents involving dangerous substances - Seveso II, site operators have several obligations including developing internal and external emergency plans -12, GD 804/2007. To develop an internal emergency plan, among other documents is appropriate to make specific accident scenarios involving hazardous activities to the entity concerned. There have been investigated the occurred events that were based on flammable substances explosions, there was analyzed the explosive environment, the source of ignition, causes and effects, depending on the specific activities of specific technology flows and the specific technological areas where these events took place. Based on this research has been developed a database that provides information for developing specific accident scenarios of their activities, scientifically grounded. Knowing the specifics of the entity for which are developed the accident scenarios, the database will be able to extract the events in similar situations, providing essential information on dynamic and thermal effects, of the blast damage area, of space technology and material destruction, number of victims, etc.*

Keywords: *explosion, accidents, database*

1. INTRODUCTION

The explosion risk is a major industrial risk in the economy, being present in all establishments that produces, uses, handles, stores and carries dangerous substances, namely products with flammable and explosive properties. Explosions and detonations are rapid combustions that occur in confined spaces (tanks, pipelines, mining and so on) that have flame propagation speeds of the order of thousands of m / s, depending on the speed of reaction. The phenomenon produced by burning an explosive mixture (methane-air, coal dust-air) which propagates with subsonic speeds (tens to 100 m / s) is conventionally named deflagration [2].

The flame follows the initial wave, accelerates, the air is compressing, and there are increased pressure until it reaches the maximum explosion pressure that coincides at some point in time and space with the flame speed. Maximum pressure continuous across the route as the strong compression wave propagates away from the point of maximum flame speed The main

* *PhD. student, INCD INSEMEX Petroşani, Romania*
** *Chem. Eng, INCD INSEMEX Petroşani, Romania*

condition for an explosion whether or not followed by fire to occur is the simultaneous presence of an efficient source of ignition / initiation with a preformed explosive atmosphere. Ignite capacity of a source of ignition should be compared with ignition susceptibility of flammable substance which is the combustible atmosphere preformed explosive.

Combustion and explosion occur only in the gas phase or in gaseous combustible substances, or combustible dusts such as coal dust when there is gas or vapor release through decomposition, followed by evaporation, capable of forming explosive atmospheres. If gas explosion is homogeneous (methane) in the other hand coal dust explosion is heterogeneous, that if it is primed in a point of the mixture spreads progressively on its support, by its ability to self-healing the reaction.

Depending on the specific activities are being used different types of substances with flammable properties.

In industrial and civil field can be found various flammable atmospheres, however in the mining industry and firedamp mines, the most common flammable atmospheres are methane gas and coal dust or a mixture thereof. Methane burns with a bright flame than with the release of large amounts of heat. Regular ignition temperature of methane is 535^0C (650^0 C - by other authors), but this can be reduced or increased depending on the conditions of the mixture. For methane-air mixtures, the explosion limits are lower explosion limit (LEL) = 5% and upper explosion limit (UEL) = 15%. The maximum intensity of the explosion will occur when methane content is 9.47%. The main parameters that characterize an explosion are explosion temperature, maximum explosion pressure that develops in the explosion, heat of explosion and propagation speed, these factors are strongly influenced by the composition of the explosive mixture [2].

Possible sources of release of flammable gas (methane): Distribution networks to final consumers, for the most part are much older than the normal operating between 12 and 18 years provided in the regulations (regulations) in force. Even the new ones that are made after old technologies from integrated steel pipes buried underground or extensions may have faults after some time. These occur at welded seams, where isolation can not be made in at fabrication, but is done on site. Depending on the chemical composition of the soil, the way of protecting the existence of power grids in the proximity of gas pipes, they are exposed to corrosion. Depth penetration of the corrosion can lead to the formation of pores or holes in pipe walls, which thus become sources of gas release.

Where gas leaks are not detected in time if air vents are not located in appropriate places, or are not properly maintained, gas can migrate through the soil directly and / or through preferential paths: sand bed pipe, sewage pipes, cable ducts, heating canals, then enter confined spaces and form explosive mixtures.

In most civil events caused by explosions that have resulted in material damage and victims the methane gas was combustible material. In most civil events caused by explosions have led to material damage and casualties due to methane (natural gas) release from distribution pipes dysfunction (perforation due to corrosion or cracks in seams) and the migration through soil and by various underground ducts to the place of storage in basements and / or other spaces (rooms / apartments).

Coal dust has explosive properties and forms a cloud in the air with dangerous levels and comes into contact with an ignition source that has the minimum ignition energy. From the source, the flame propagates in the cloud, flame radiation heats neighboring particles, resulting in oxidation reactions at the surface or pyrolysis.

In developing of coal dust explosions there are three phases in which they occur the following physicochemical processes:

1. Particles of coal pyrolysis, distillation or gasification of coal. Coal particles surrounding the source of ignition, under the influence of heat absorbed decomposes, releasing volatile substances. Following this process remains a solid residue consisting of coke and ash. Substances contain volatile flammable gases and vapors (hydrocarbons, carbon monoxide, hydrogen, tar etc.) carbon dioxide and water as vapors.

2. Combustion of volatile substances - gas phase reactions

Flammable gases and vapors, forms around solid nucleus in the presence of atmospheric oxygen, a flammable mixture coating. When the energy source is greater than the minimum ignition energy of the flammable mixture, after some time, it ignites.

3. Partially solid residue burning in the flammable mixture

After the coal particle passes through the first two phases, still contains some volatile substance that is released more slowly. The last phase is characterized by the reactions gas - solid surrounding atmosphere and has a longer duration compared to previous phases. [2]

Underground coal exploitation raises special character security issues determined by specific underground environmental conditions and the existence of potentially explosive atmospheres. Fires and explosions in coal mines are part of events with particularly with serious consequences in the economic and social plan, as well as negative influences on the environment.

Dynamic effects: The effect of an explosion, especially due to the size of the shock wave propagated in the surroundings is determined by the size and shape of the flammable substance and the reactivity explosive atmospheres.

Thermal effects: Estimation of thermal effects generated by explosions on structures and people in their area of influence is difficult. These effects are the result of emission or energy transfer from the flame generated by an explosion on structures and people in the area of influence.

2. DEVELOPING THE DATABASE OF ACCIDENTS FROM EXPLOSIVE ATMOSPHERES

For the scientific base of the accident scenarios, required to develop internal emergency plans by anticipating the effects of future events and the development for this purpose of a database of accidents caused by explosions in industrial and civilian domain they were extracted, processed and selected from the technical expertise reports, available at INCD INSEMEX in the archives of the institution, in the institution's technical library and those under the custody of the research laboratories, information and data relevant to flammable atmospheres, release source, open source dynamic effects and thermal victims if necessary and conduct events. There were also retained data regarding the time of the events and their location

For achieving a database containing key information about occurred events that were based on explosions of flammable substances was performed the analysis of the causes and effects of events, depending on the specific activities of specific technological flows and specific technological areas where took place these events.

The database - built by multicriteria searches such as: type of plant technology, hazardous substance that caused the event, etc.- helps to expeditious retrieval of information on the causes and effects of one or more events that have occurred and that match the search criteria. [1]

By the database development was achieved rapid establishment of the relevant elements for the development of accident scenarios closer to a possible reality in developing measures most appropriate facility, technology, and hazardous substances handled.

The most basic form of storage is to create and maintain data in data files using EXCEL application.

A database is a collection of information that is intended to achieve a particular purpose.

For the creation, maintenance and consulting databases are used - specialized software called Database Management Systems (DBMS).

To develop database has been used the application 2010 Microsoft Office Professional ACCSESS which is a program designed for developing and managing databases.

Access database program consists of one or more two-dimensional tables. Separating data by type of information allows easy operation of the database.

Creating forms facilitates the introduction, viewing and updating data. Search, quick access to information is ensured data collection queries. Analyze, display and print data required by certain criteria is provided by reports.

The program has been used to develop database provides user assistant procedures (wizards) that can be done easily these forms, queries and reports.

In addition and in parallel with the search for sources of information on events (technical expert reports as they are analyzed), has been made an analysis of how data can be systematized.

Below are given examples of ways to access information from the database. The report drawn up from accessing the database has several fields, such as (Figure 1,2,3):

- Year of production
- Domain
- Location
- Installation type
- Hazardous material
- Source release
- Ignition source
- Thermal effects
- Dynamic effects
- Victims
- Direct effects neighborhood
- Environmental impacts
- Name of economic agent
- Deployment mechanism

In parallel with the search for the information on events sources (technical expertise reports in which they are analyzed), has been done an analysis of how data can be systematized.

Given the relatively large number of works inventoried is concluded that they provide sufficient data to build the core of the database, which can be extended throughout the identification of other works (technical expertise).

To systematize information Access the application was chosen from the set of Microsoft Office programs. There were presented the main features of this application as well as examples of forms of input, query and reporting information from the database.

Preliminary investigations have been made additionally for the choice of database management and how to build the collection of information to be stored.

Due to the facilities provided the conclusion is that the application Access is the option for building and managing database.

Figure 1:
Examples of report generated by the database, by year

Figure 2:
Example of a report generated by economic agent location

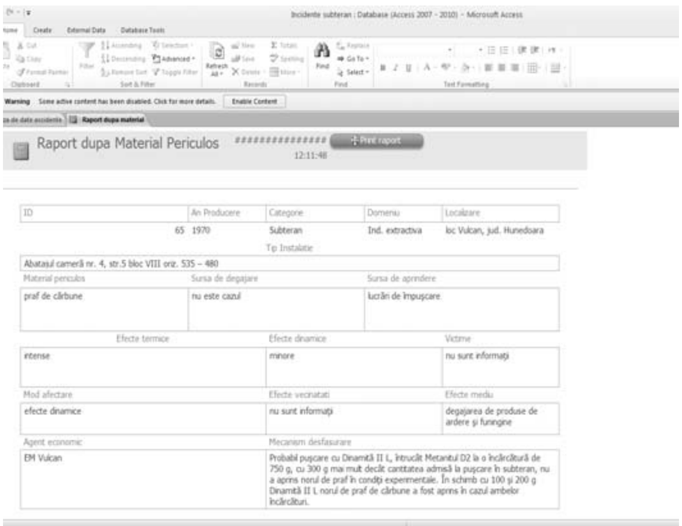

Figure 3: Example of a report generated by hazardous material

3. CONCLUSIONS AND PROPOSALS

1. Of accident statistics at the county level as a hierarchy of frequency on trades, coal mining and preparation comes first. Of mining damage „explosion of gas/coal dust” is ranked 2 after mine fires and fires. Analyzing type events held in underground explosions, it can be concluded that methane gas mixed with air and / or coal dust is explosive environment, which is supported explosions.

2. Factors that contribute to the accumulation of dangerous methane are: poor ventilation, improper ventilation methods of dams and changes in geological features.

3. The sources of initiating explosive mixture were: spontaneous combustion and self-heating, blasting operations, electrical hazards, welding or open flame operations.

4. Explosions are part of the major risks in the economy because they are linked to uncontrolled development of industrial processes, resulting in serious danger, immediate or long-term, both for man and for interior or exterior installations, or the environment.

5. To achieve database was purchased Microsoft Office 2010. Under this package was work with ACCESS program, specially designed for database. There was developed a multicriterial database search. To achieve database were used the results derived from the technical expertise/technical expertise reports existing in archive of INCD INSEMEX Petroşani about underground events.

6. For civil domain most accidents type or explosions, whether or not followed by fire, natural gas is flammable material.

7. Causes of accumulation of natural gas in residential areas and in the civil domain are multiple, so the damage at the level of pipelines to failure of machinery used for cooking or heating.

8. Is proposed to manage the database, completion to new events to update the database daily.

REFERENCES

[1] Microsoft Office Professional - *Access* – 2010

[2] PN-07-45-02-23: "*Research for scientific substantiation of accident scenarios, required to develop internal emergency plans by anticipating the effects of future events and development for this purpose of a database on accidents caused by explosions in industrial and civil field*", INCD-INSEMEX, 2010-2012

THE AUTOMATION OF THE STAND FOR TRACKING THE EVOLUTION OF THE SPONTANEOUS COMBUSTION OF COAL BEDS

TOTH ION*
LUPU CONSTANTIN*
CIOCLEA DORU*
TOMESCU CRISTIAN**

Abstract: *Within the framework of the "Nucleu" research program there has been carried out the project entitled "The adjustment of the technologies for preventing and fighting against spontaneous combustions for the undermined coal bed method", which comprised the phase entitled "Increasing the research capacity by using modern methods and equipment in order to prevent and fight against the spontaneous combustion of coal". Within this phase, the stand for tracking the evolution of spontaneous combustion of coal beds has been upgraded. The stand subject to upgrade dates from the 60s. Following the automation, the stand has been equipped with: a Heating system with 500 W encapsulated resistances, 4 pieces serially; b. System for monitoring the temperatures from the coal bed using thermocouple type sensors with a 50-1600℃ measuring range; c. System for measuring the temperatures from the air cushion using the "thermo-resistance type PT100 with 50-400℃ measuring range"; d. TSI 4043 flow meter with 0-100 l/min flow rate and calibration certificate; e. Pre-settable valve – proportional valve – 3 way with control loop, 24 VDC; f. Specialized software for programming work parameters and for monitoring the characteristics of the evolution-involution of the spontaneous combustion process with Siemens Simatic Net CP 5611 A2 – 6GK 1561 AA 01 data acquisition board and Siemens Simatic Win CC V7, OSP3, 6AV 6381-2BC07 – oAx0 user licence. By creating the automation system of the stand for tracking the evolution-involution of the spontaneous combustion, the use of the stand is extended by achieving the evolution of the spontaneous combustion in adiabatic system, not only by observing the evolution at programmed heating.*

Keywords: *spontaneous combustions, simulation stand, coal*

1. INTRODUCTION

At the National Institute for Research and Development in Mine Safety and Protection to Explosion – INSEMEX Petroşani, within the „Nucleu" national research program, "Developing the national capacity for assessing, preventing and limiting the risks generated by

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* *PhD., Eng., INCD INSEMEX Petroşani, Romania*
** *PhD. Student, Eng., INCD INSEMEX Petroşani, Romania*

industrial applications carried out in explosive and/or toxic environments in the occupational health and safety field, environmental protection, mineral resources and materials", there has been carried out the project entitled "The adjustment of the technologies for preventing and fighting against spontaneous combustions for the undermined coal bed method", project which comprised the 5th phase entitled "Increasing the research capacity by using modern methods and equipment in order to prevent and fight against the spontaneous combustion of coal".

Within this phase, the stand for tracking the evolution-involution of the spontaneous combustion of coal beds has been automatized.

2. GENERAL ASPECTS REGARDING THE DEVELOPMENT PHASES OF THE SPONTANEOUS COMBUSTION

During the development of the spontaneous combustion and in the burning phase, there arise gaseous oxidation products, water vapours and smoke particles [2].

The gaseous oxidation products comprise: oxygen, carbon dioxide, carbon oxide, hydrogen, methane, aromatics, etc.

The concentrations of the gaseous oxidation products follow the evolution of the combustion process, as well as its involution. In the evolution phase, the concentrations are strictly increasing and in the involution phase these concentrations decrease, but this decrease is slower due to the influence of remanent gases absorbed by the deposit and water.

2.1. Spontaneous combustion and coal burning process development

One of the oxidation gases which faithfully follow the development of the spontaneous combustion process is the carbon oxide (CO).

Professor Bystron developed a model for the evolution and involution of the spontaneous combustion process, based on the relation between the variation of carbon oxide in time (τ) and temperature (t) developed during the process. This relation was the result of observations gained over tens of years of experience in fighting against underground fires [3].

The development model of the spontaneous combustion and coal burning process is presented in Fig. 1.

Professor Bystron splits the process into 6 phases.

● ***Incubation phase 1 - 2 - 3.*** This phase is characterized by the initial temperature (t_p), deduced from the expression:

$$t_p = 8 + \frac{H - 30}{\sigma} \tag{1}$$

where:

H – depth from the surface

σ - geo-thermal gradient (m/°C).

The specific temperature in point 1 is t_1 = 30°C, being the temperature of the coal massif. Through oxidation, coal heats up to the point 2, considered to be non-hazardous t_2 = 40°C.

If the heat storage in the massif conditions are not fulfilled, then coal is cooled to the temperature t_1 = 30°C of the massif. If the heat storage conditions are fulfilled, then coal heats up to a temperature t_3 = 60°C, called critical point to whom it corresponds a carbon oxide (CO) concentration of 10 ppm = 0,001% vol. CO.

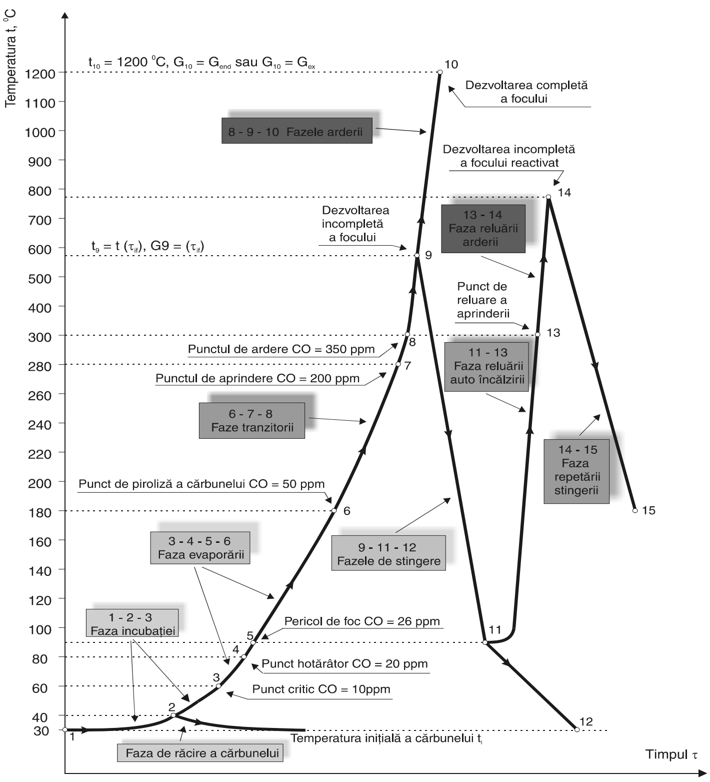

Fig.1

● ***Evaporation phase – its first part - 3 - 4***

The phase comprises the heating of coal from t_3 = 60°C to t_4 = 80°C, corresponding to point 4 from the diagram, called decisive critical point where the CO concentration is CO = 0.002% vol. = 20 ppm.

● ***Second evaporation phase- 4 - 5 - 6***

It comprises the temperature increase from t_4 = 80°C through point 5 (point when fire hazard occurs) with a concentration of CO== 0.0026% vol. = 26 ppm, towards point 6 with the temperature of t_6 = 180°. Point 6 is called pyrolysis point and it is characterized by a concentration of CO= CO = 0.005 % vol. = 50 ppm.

● ***First transitory phase 6 - 7***

This phases corresponds to the transition from the pyrolysis point 6 with the temperature t_6 = 180°C to point 7 – ignition point with the temperature t_7 = 280°C and the carbon oxide concentration of CO = 0.02% vol. = 200 ppm.

● ***Second transitory phase - 7 - 8***

This phase corresponds to the transition from point 7 with the temperature of t_7 = 280°C to point 8 with the temperature of t_8 = 300°C. A concentration of carbon oxide of CO = 0.035 % vol. corresponds to point 8. Point 8 is also a self-ignition and fire starting point. The temperature t_8 from point 8 represents the minimum fire temperature.

● ***Burning phase 8 - 9 - 10***

It makes reference to the temperature increase from t_8 = 300°C, through point 9 – fire development point, up to point 10. The temperature of the point 10 (complete fire development point) is t_{10} = 1200°C and it stands for the temperature from the complete fire development phase. If it is intervened upon the coal which reached point 9 and the burning process is stopped, then coal passes into the extinction phase 9-11.

When the temperature of the affected massif falls below the critical temperature (80 - 90°), then it further cools down until the initial temperature 11-11. But if the extinction phase is stopped with the cooling of massif above the critical temperature, the phenomenon may be reactivated, pathway 11-13-14 on the diagram. By intervening again upon the fireplace from the incomplete fire development point (14), the phenomenon may again pass the extinction phase, pathway 14-15 on the diagram.

3. DESCRIPTION OF THE EXPERIMENTAL STAND BEFORE THE UPGRADE

In order to track the coal oxidation process in laboratory conditions, an experimental stand is used for simulating the self-heating and self-ignition of coal processes, so that the work conditions achieved on the stand to be similar to the ones from the mine [1].

3.1. The description of the initial experimental stand

The experimental stand (Fig. 2) from INCD INSEMEX Petrosani has been developed in the 60s and has the following main components:

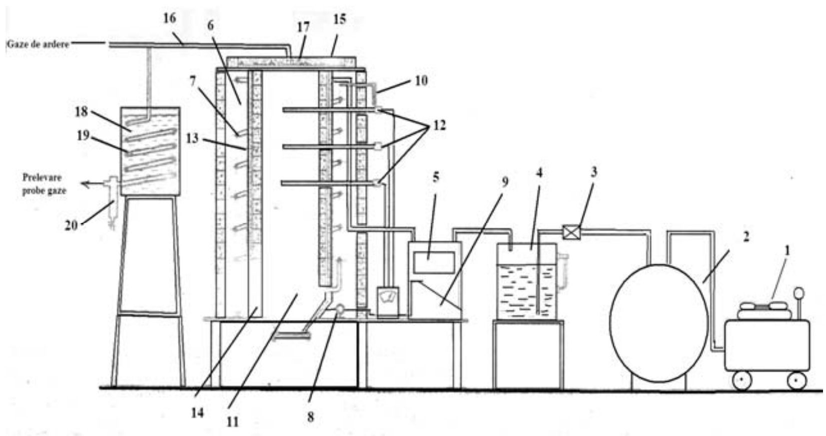

Fig.2 Schematic diagram of the experimental stand

Legend

1.	Compressor	11.	Combustion chamber
2.	Air buffer tank	12.	Thermocouples
3.	Pressure reducer	13.	Firebrick
4.	Humidifier	14.	Asbestos plate
5.	Flowmeter	15.	Combustion chamber cover
6.	Heating pad	16.	Burning chamber exhaust pipe
7.	Compressed air spiral tube	17.	Mineral wool layer
8.	Heating resistances	18.	Roast gas cooling vessel
9.	Electronic thermostat	19.	Spiral pipe
10.	Mercury thermometer	20.	Settling vessel

- The compressor (1) which produces the compressed air required for the coal oxidation process;
- The air buffer tank (2) having the role to store a 200 l volume of compressed air;
- The pressure reducer (3) which increases the pressure of the compressed air from 3-4 atmospheres to 164-245 mm col H_2O;
- The humidifier (4) in which, on the one hand the compressed air is enriched in humidity through its ebullition in water, and on the other hand the solid particles are retained in the air;
- The flowmeter (5) which measures the air flow that passes through the coal sample;
- The heating chamber or the heating pad (6), comprising the tube through which compressed air circulates, mounted in spiral shape (7), as well as the resistances (8) which heat the air from the tube. Coupling and decoupling these resistances is commanded by an electronic thermostat (9) which ensures a minimum threshold (when coupling) and a maximum threshold (when decoupling) of the temperature from the heating chamber. The difference between these thresholds varies between 3 and 5 ° C. The temperature of the heating chamber (pad) is shown by the mercury thermometer (10).
- The combustion chamber (11) is provided with three thermocouples (12) by means of which it is tracked the evolution of the temperature in the mass subject to self-heating and self-ignition processes. This chamber is firebrick (13) lined and at the same time it is isolated from the heating chamber. This insulation is carried out using asbestos plates (14) which plank the combustion chamber.
- The combustion chamber cover (15) aims to close these chambers, and it is provided with a pipe for exhausting the gases resulted from the coal oxidation process (16). In order for the resulted temperature not to be dissipated outside, the cover of the combustion chamber has been insulated with a mineral wool layer having a thickness of approx. 10 cm (17).
- The roast gas cooling vessel (18) in which a spiral tube (19) is located, tube through which gases are collected in order to determine them. In the vessel (18) flows continuously an open system water circuit, so that the collected gases are cooled to the temperature of the water (16-25 ° C).
- The settling vessel (20) aiming to store the condensation and the tar resulted from the coal dry distillation process.

4. THE AUTOMATIZED STAND

The assembly of the automatized stand for tracking the evolution-involution of the spontaneous combustion comprises the following components:

- Heating system with 500 W encapsulated resistances, 4 pieces serially and 4 spare pieces;
- System for monitoring the temperatures from the coal bed using thermocouple type sensors with a 50-1600°C measuring range;
- System for measuring the temperatures from the air cushion using the “thermo-resistance type PT100 with 50-400°C measuring range”;
- TSI 4043 flow meter with 0-100 l/min flow rate and calibration certificate;
- Pre-settable valve – proportional valve – 3 way with control loop, 24 VDC;
- Specialized software for programming work parameters and for monitoring the characteristics of the evolution-involution of the spontaneous combustion process with Siemens

Simatic Net CP 5611 A2 – 6GK 1561 AA 01 data acquisition board and Siemens Simatic Win CC V7, OSP3, 6AV 6381-2BC07 – oAx0 user licence.

The thermal process achieved by the automatized system is presented in Fig. 3.

By creating the automation system of the stand for tracking the evolution-involution of the spontaneous combustion, the use of the stand is extended by achieving the evolution of the spontaneous combustion in adiabatic system, not only by observing the evolution at programmed heating [4].

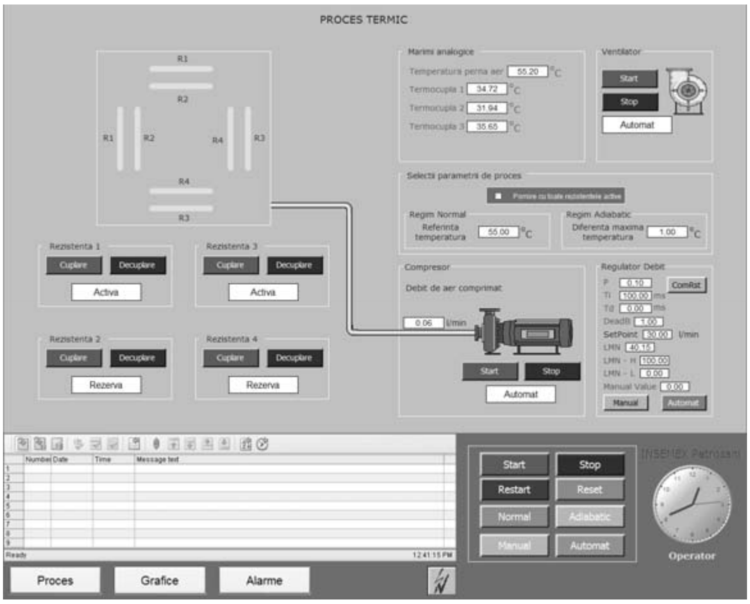

Fig.3

5. CONCLUSIONS

- The spontaneous combustion is a complex physical-chemical process for "in situ" coal oxidation.
- The evolution of the spontaneous combustion may be modelled using the stand from INCD INSEMEX Petroşani.
- The stand was built in the 60s and needs to be upgraded.
- Upgrading the stand consisted in the replacement of the:
    a) Heating system;
    b) Temperature measurement system;
    c) Flow measurement system;
    d) Hall ventilation;
    e) Computer software
- The work has been received by acceptance report 14/28.05.2012.
- Using the upgraded stand there may be carried out the study on the evolution of the spontaneous combustion in adiabatic conditions.

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CAUTION LEVEL ESTIMATION AND ASSESSMENT IN MAKING DECISIONS REGARDING THE DIAGNOSIS AND PROGNOSIS OF THE HEARING IMPAIRMENT RISK

VASILESCU GABRIEL – DRAGOŞ[*]
SIMION SORIN – VICTOR[**]
MIJA NELU[***]

Abstract: *This paperwork presents a methodological approach on the analysis and on the statistical and probabilistic assessment of the hearing impairment risk, based on the hazards quantified as risk predictors, in order to establish the modalities for assessing the caution limits related to the acceptability ranges. The statistical approach is based on the rational quantification of what exists and can be observed, the probabilistic part of this fact representing the extrapolation to what it can be reasonably deduced from these statistics of hearing impairment occurrence probability.*

Keywords: *noise, auditory handicap, diagnosis, prognosis, distribution, algorithm*

1. Generalities regarding the hearing impairment due to noise

Individuals regularly exposed to noise may suffer from hearing loss with variable severity. Due to hearing loss, there may be damaged both the speech understanding as well as the perception of acoustic signals generated during the work process or daily life. Excluding the exposure to explosions, to high impulse noise and extremely high levels of stationary noise, the permanent damage of the organ of hearing may be achieved progressively in time, depending on the exposure time [4].

The term regarding the *"permanent displacement of the threshold caused by noise"* is discussed in the paper as a component independent of other elements of the audibility threshold levels, which is usually equal to zero, and for a given noise exposure has a range of positive values representing the variation of individual susceptibility of hearing impairment to noise. The permanent displacement of the threshold caused by noise is generally preceded by a temporary reversible effect on the hearing named *temporary modification of the threshold cause by noise,* whose severity and recovery mode depend on the exposure level and time.

Since the precise determination related to the clearly differentiation of the changes of the hearing threshold levels caused by noise or other factors is difficult in case of punctual approach for each individual, there has been choses to determine the statistical distribution changes of the hearing threshold levels for a population exposed to a specific kind of noise. In

[*] *PhD., Eng. INCD INSEMEX PETROŞANI*
[**] *PhD. Student, Eng. INCD INSEMEX PETROŞANI*
[***] *Eng. INCD INSEMEX PETROŞANI*

this regard, in order to highlight the differences between the hearing threshold levels of two groups of persons which are similar in all important aspects, excepting the fact that one group is exposed to occupational noise, there may be successfully used the average or median parameters related to the permanent displacement of the hearing threshold caused by noised. [5]

In some countries, the haring handicap caused by noise exposure may have legal consequences regarding the responsibility and compensation. Levels of the hearing threshold at different frequencies at which it is considered that there is a hearing handicap (limit level) depends not only on the damage itself, but also on the legal aspects based on social and economic considerations. In addition, defining the hearing handicap depends on the required speech understanding quality, on the average background noise level and, in relation to the relative importance of different frequencies, perhaps even on the spoken language [3]. Because the hearing impairment caused by noise isn't only the results of exposure to occupational noise, but also the result of the entire exposure to noise of the population, there is necessary to take in to account the non-occupational exposure of individuals (during their travelling to or from their jobs, home and during recreational activities etc.).

The prognosis of hearing impairment due to occupational noise exposure is possible when the non-occupational exposure is negligible compared to the occupational one, case in which it is required to calculate the hearing impairment due to total daily and combined exposures (occupational and non-occupational) to noise, and, if required, there may be also estimated the contribution to the total hearing impairment due to occupational noise exposure.

2. GENERALIZED MATHEMATICAL MODEL FOR ASSESSING THE RISK OF HEARING IMPAIRMENT CAUSED BY NOISE

2.1. Theoretical and practical considerations on the prognosis of the effects of noise on the hearing threshold.

The hearing quality depending on the age of a population which is to exposed to noise depends on the extent to which there are accidentally included other factors besides natural ageing. Also, some diseases, ototoxic drugs problems and unknown exposure to occupational noise may influence the *hearing threshold level related to age.*

In order to substantiate the hearing threshold level related to age, there are used two databases, A and B, where one is fully specified (A), and the other one being at the latitude of the user (B) [5].

So, the A database is provided from otologic normal people, with normal health condition, who do not present signs or symptoms or ear diseases and wax plugs in the auditory channels and which have not been excessively exposed to noise. The statistical distribution of the thresholds of a population of this kind "very protected" have been standardized for male populations, as well as for female populations.

For the B database there is recommended a dataset collected from a control population which has not been exposed to occupational noise.

Also, choosing the proper database depends on the problem supposed to be solved.

The determination of the permanent displacement of the hearing threshold caused by noise

a) The calculation of $N_{0,50}$

Values of the permanent displacement of the threshold caused by noise depend on the audiometric frequency, on the exposure time, on the θ/θ_0 ratio and on the exposure level for a 8h working day, $L_{EX,8h}$ mediated on the θ exposure time. So, for exposure times between 10 and 40 years, values of permanent displacements of the threshold caused by noise (potential medians), are given for both sexes by the following relation: [1; 5]:

$$N_{0,50} = \left\{ \begin{bmatrix} -0,033^{(500\ Hz)} \\ -0,020^{(1000\ Hz)} \\ -0,045^{(2000\ Hz)} \\ 0,012^{(3000\ Hz)} \\ 0,025^{(4000\ Hz)} \\ 0,019^{(6000\ Hz)} \end{bmatrix} + \begin{bmatrix} 0,110^{(500\ Hz)} \\ 0,070^{(1000\ Hz)} \\ 0,066^{(2000\ Hz)} \\ 0,037^{(3000\ Hz)} \\ 0,025^{(4000\ Hz)} \\ 0,024^{(6000\ Hz)} \end{bmatrix} \lg(\frac{\theta}{\theta_0}) \right\} x \left\{ L_{EX,8h} - \begin{bmatrix} 93^{(500\ Hz)} \\ 89^{(1000\ Hz)} \\ 80^{(2000\ Hz)} \\ 77^{(3000\ Hz)} \\ 75^{(4000\ Hz)} \\ 77^{(6000\ Hz)} \end{bmatrix} \right\}^2 \tag{1}$$

where:

$[93^{(500\ Hz)}, 89^{(1000Hz)}, 80^{(2000Hz)}, 77^{(3000Hz)}, 75^{(4000Hz)}, 77^{(6000Hz)}]$ - range of values of the limit acoustical pressure level defined depending on the frequency, L_0(dB) ;

θ(years) – exposure time, θ_0=1 year;

$L_{EX,8h}$ – represents the level of exposure to noise for an 8h working day;

[500, 1000, 2000, 3000, 4000, 6000] - audiometric frequency values; (Hz).

Equation (1) is applied in case in which $L_{EX,8h}$ is higher than L_0, otherwise the level of exposure to noise for an 8h working day ($L_{EX,8h}$) being equal to L_0, so that $N_{0,50}$ is equal to 0.

Also, for periods of time smaller than 10 years, N will be extrapolated from the value of $N_{0,50}$ corresponding to a 10 year period, according to the following relation [1;5]: [1;5]:

$$N_{0,50;\theta<10ani} = \frac{\lg(\theta+1)}{\lg 11} N_{0,50;\theta=10ani}, \tag{2}$$

b) Statistical distribution of N

The distribution of N is approximated by two different halves of normal distributions (Gaussian), respectively: upper half for the quantile with the hearing weaker than the median found over the $N_{0,50}$ median value and the lower half which is found below the $N_{0,50}$ median value [1;5].

So, there are two situations:

1. If 0,05≤Q≤0,50 then the permanent displacement of the threshold caused by noise, N_Q , is given by the following relation (3):

$$\begin{bmatrix} N_{0,05;0,95} \\ N_{0,10;0,90} \\ N_{0,15;0,85} \\ N_{0,20;0,80} \\ N_{0,25;0,75} \\ N_{0,30;0,70} \\ N_{0,35;0,65} \\ N_{0,40;0,60} \\ N_{0,45;0,55} \\ N_{0,50} \end{bmatrix} = N_{0,50} + \begin{bmatrix} 1,645 \\ 1,282 \\ 1,036 \\ 0,842 \\ 0,675 \\ 0,524 \\ 0,385 \\ 0,253 \\ 0,126 \\ 0 \end{bmatrix} x \left\{ \left[\left[\begin{pmatrix} 0,044^{(500\ Hz)} \\ 0,022^{(1000\ Hz)} \\ 0,031^{(2000\ Hz)} \\ 0,007^{(3000\ Hz)} \\ 0,005^{(4000\ Hz)} \\ 0,013^{(6000\ Hz)} \end{pmatrix} + \begin{pmatrix} 0,016^{(500\ Hz)} \\ 0,016^{(1000\ Hz)} \\ -0,002^{(2000\ Hz)} \\ 0,016^{(3000\ Hz)} \\ 0,009^{(4000\ Hz)} \\ 0,008^{(6000\ Hz)} \end{pmatrix} \lg\frac{\theta}{\theta_0} \right] x \left[L_{EX,8h} - \begin{pmatrix} 93^{(500\ Hz)} \\ 89^{(1000\ Hz)} \\ 80^{(2000\ Hz)} \\ 77^{(3000\ Hz)} \\ 75^{(4000\ Hz)} \\ 77^{(6000\ HZ)} \end{pmatrix} \right] \right] \right\}$$

2. If 0,50<Q≤0,95 then the permanent displacement of the threshold caused by noise is given by the following relation 4)

$$\begin{bmatrix} N_{0,05;0,95} \\ N_{0,10;0,90} \\ N_{0,15;0,85} \\ N_{0,20;0,80} \\ N_{0,25;0,75} \\ N_{0,30;0,70} \\ N_{0,35;0,65} \\ N_{0,40;0,60} \\ N_{0,45;0,55} \\ N_{0,50} \end{bmatrix} = N_{0,50} - \begin{bmatrix} 1,645 \\ 1,282 \\ 1,036 \\ 0,842 \\ 0,675 \\ 0,524 \\ 0,385 \\ 0,253 \\ 0,126 \\ 0 \end{bmatrix} x \left\{ \left[\left[\begin{pmatrix} 0,033^{(500\ Hz)} \\ 0,020^{(1000\ Hz)} \\ 0,016^{(2000\ Hz)} \\ 0,029^{(3000\ Hz)} \\ 0,016^{(4000\ Hz)} \\ 0,028^{(6000\ Hz)} \end{pmatrix} + \begin{pmatrix} 0,002^{(500\ Hz)} \\ 0,000^{(1000\ Hz)} \\ 0,000^{(2000\ Hz)} \\ -0,010^{(3000\ Hz)} \\ -0,002^{(4000\ Hz)} \\ -0,007^{(6000\ Hz)} \end{pmatrix} \lg \frac{\theta}{\theta_0} \right] x \left[L_{EX,8h} - \begin{pmatrix} 93^{(500\ Hz)} \\ 89^{(1000\ Hz)} \\ 80^{(2000\ Hz)} \\ 77^{(3000\ Hz)} \\ 75^{(4000\ Hz)} \\ 77^{(6000\ HZ)} \end{pmatrix} \right] \right] \right\}$$

where:

[1,645;1,282;1,036;0,842;0,675;0,524;0,385;0,253;0,126;0] - is represents the values of the k multiplication factor in 0.05 intervals for the Q quantile;

[$93^{(500\ Hz)}$, $89^{(1000Hz)}$, $80^{(2000Hz)}$, $77^{(3000Hz)}$, $75^{(4000Hz)}$, $77^{(6000Hz)}$] - is the range of values of the limit acoustical pressure level defined depending on the frequency, L_0(dB) ;

θ(years) – is the exposure time, θ_0=1 year;

$L_{EX,8h}$ – represents the level of exposure to noise for an 8h working day;

[500, 1000, 2000, 3000, 4000, 6000] - audiometric frequency values (Hz).

The values which correspond to the statistical distribution queues for situations in which 0<Q<0.05 and 0.95<Q<1, aren't sure and therefore they aren't estimated following the difficulties to validate these domains.

2.2.-The determination if the hearing impairment and of the hearing handicap caused by noise.

Potentially hearing impairment due to occupational exposure to noise is directly assessed through the permanent displacement of the threshold caused by noise, which may be [4]:

- Separately considered for each frequency of interest;
- Gathered for a certain number of frequencies having as result a total threshold displacement;
- Averaged over a number of selected frequencies which usually represent the main frequency domain for speech understanding..

In order to calculate the hearing handicap there may be used (for each ear; for the average of both ears; for weighted average of both ears) a combination of hearing threshold levels at mentioned frequencies, respectively: average hearing threshold level at 500 Hz, 1000Hz, 2000Hz; average hearing threshold level at 500 Hz, 1000Hz, 2000Hz and 3000Hz; average hearing threshold level at 1000Hz, 2000Hz and 4000Hz; average hearing threshold level at 1000Hz, 2000Hz and 3000Hz; average hearing threshold level at 1000Hz, 2000Hz, 3000Hz and 4000Hz; average hearing threshold level at 2000Hz and 4000Hz; average hearing threshold level at 2000Hz, 3000Hz and 4000Hz, etc.

The risk of hearing handicap due to noise exposure and age or only due to noise exposure frequently represents measures of the negative effects of noise exposure over the population [2].

2.3.-Calculating the A database

Equations applied for the H hearing threshold level depending on the Y age (years) for different intervals of the Q quantile which has the value of the threshold higher than the H_Q are the following [1;5]:

1.If 0.05≤Q≤0,50 then le hearing threshold level related to the H_Q age is given by the equation (4.1) for a male population and by equation (4.2) for a female population.

2.If Q=0.50 then le hearing threshold level related to the H_Q age is given by the equation (5.1) for a male population and by equation (5.2) for a female population.

$$\begin{pmatrix} H_{0,05;0,95} \\ H_{0,10;0,90} \\ H_{0,15;0,85} \\ H_{0,20;0,80} \\ H_{0,25;0,75} \\ H_{0,30;0,70} \\ H_{0,35;0,65} \\ H_{0,40;0,60} \\ H_{0,45;0,55} \\ H_{0,50} \end{pmatrix} = H_{0,50} + \begin{pmatrix} 1,645 \\ 1,282 \\ 1,036 \\ 0,842 \\ 0,675 \\ 0,524 \\ 0,385 \\ 0,253 \\ 0,126 \\ 0 \end{pmatrix} x \left[\begin{pmatrix} 7,23^{(125\ Hz)} \\ 6,67^{(250\ Hz)} \\ 6,12^{(500\ Hz)} \\ 6,12^{(1000\ Hz)} \\ 6,67^{(1500\ Hz)} \\ 7,23^{(2000\ Hz)} \\ 7,78^{(3000\ Hz)} \\ 8,34^{(4000\ Hz)} \\ 9,45^{(6000\ Hz)} \\ 10,56^{(8000\ Hz)} \end{pmatrix} + 0,445\ H_{0,50} \right]$$

Male population (4.1)

$$\begin{pmatrix} H_{0,05;0,95} \\ H_{0,10;0,90} \\ H_{0,15;0,85} \\ H_{0,20;0,80} \\ H_{0,25;0,75} \\ H_{0,30;0,70} \\ H_{0,35;0,65} \\ H_{0,40;0,60} \\ H_{0,45;0,55} \\ H_{0,50} \end{pmatrix} = H_{0,50} + \begin{pmatrix} 1,645 \\ 1,282 \\ 1,036 \\ 0,842 \\ 0,675 \\ 0,524 \\ 0,385 \\ 0,253 \\ 0,126 \\ 0 \end{pmatrix} x \left[\begin{pmatrix} 7,23^{(125\ Hz)} \\ 6,67^{(250\ Hz)} \\ 6,12^{(500\ Hz)} \\ 6,12^{(1000\ Hz)} \\ 6,67^{(1500\ Hz)} \\ 7,23^{(2000\ Hz)} \\ 7,78^{(3000\ Hz)} \\ 8,34^{(4000\ Hz)} \\ 9,45^{(6000\ Hz)} \\ 10,56^{(8000\ Hz)} \end{pmatrix} + 0,445\ H_{0,50} \right]$$

Female population (4.2)

$$H_{0,50} = \begin{pmatrix} 0,0030^{(125\ Hz)} \\ 0,0030^{(250\ Hz)} \\ 0,0035^{(500\ Hz)} \\ 0,0040^{(1000\ Hz)} \\ 0,0055^{(1500\ Hz)} \\ 0,0070^{(2000\ Hz)} \\ 0,0115^{(3000\ Hz)} \\ 0,0160^{(4000\ Hz)} \\ 0,0180^{(6000\ Hz)} \\ 0,0220^{(8000\ Hz)} \end{pmatrix} x (Y - 18)^2 + H_{0,50;18}$$

Male population (5.1)

$$H_{0,50} = \begin{pmatrix} 0,0030^{(125\ Hz)} \\ 0,0030^{(250\ Hz)} \\ 0,0035^{(500\ Hz)} \\ 0,0040^{(1000\ Hz)} \\ 0,0050^{(1500\ Hz)} \\ 0,0060^{(2000\ Hz)} \\ 0,0075^{(3000\ Hz)} \\ 0,0090^{(4000\ Hz)} \\ 0,0120^{(6000\ Hz)} \\ 0,0150^{(8000\ Hz)} \end{pmatrix} x (Y - 18)^2 + H_{0,50;18}$$

Female population (5.2)

3. If 0.50≤Q≤0.95 then le hearing threshold level related to the H_Q age is given by the equation (6.1) for a male population and by equation (6.2) for a female population.

$$\begin{pmatrix} H_{0,05;0,95} \\ H_{0,10;0,90} \\ H_{0,15;0,85} \\ H_{0,20;0,80} \\ H_{0,25;0,75} \\ H_{0,30;0,70} \\ H_{0,35;0,65} \\ H_{0,40;0,60} \\ H_{0,45;0,55} \\ H_{0,50} \end{pmatrix} = H_{0,50} + \begin{pmatrix} 1,645 \\ 1,282 \\ 1,036 \\ 0,842 \\ 0,675 \\ 0,524 \\ 0,385 \\ 0,253 \\ 0,126 \\ 0 \end{pmatrix} x \left[\begin{pmatrix} 5,78^{(125\ Hz)} \\ 5,34^{(250\ Hz)} \\ 4,89^{(500\ Hz)} \\ 4,89^{(1000\ Hz)} \\ 5,34^{(1500\ Hz)} \\ 5,78^{(2000\ Hz)} \\ 6,23^{(3000\ Hz)} \\ 6,67^{(4000\ Hz)} \\ 7,56^{(6000\ Hz)} \\ 8,45^{(8000\ Hz)} \end{pmatrix} + 0,356\, H_{0,50} \right]$$

Male population (6.1)

$$\begin{pmatrix} H_{0,05;0,95} \\ H_{0,10;0,90} \\ H_{0,15;0,85} \\ H_{0,20;0,80} \\ H_{0,25;0,75} \\ H_{0,30;0,70} \\ H_{0,35;0,65} \\ H_{0,40;0,60} \\ H_{0,45;0,55} \\ H_{0,50} \end{pmatrix} = H_{0,50} + \begin{pmatrix} 1,645 \\ 1,282 \\ 1,036 \\ 0,842 \\ 0,675 \\ 0,524 \\ 0,385 \\ 0,253 \\ 0,126 \\ 0 \end{pmatrix} x \left[\begin{pmatrix} 5,34^{(125\ Hz)} \\ 4,89^{(250\ Hz)} \\ 4,89^{(500\ Hz)} \\ 4,89^{(1000\ Hz)} \\ 5,34^{(1500\ Hz)} \\ 5,34^{(2000\ Hz)} \\ 5,78^{(3000\ Hz)} \\ 6,23^{(4000\ Hz)} \\ 7,12^{(6000\ Hz)} \\ 8,45^{(8000\ Hz)} \end{pmatrix} + 0,356\, H_{0,50} \right]$$

Female population (6.2)

2.4.- Algorithm of the generalised mathematical model for assessing the hearing impairment risk

In order to substantiate a hearing impairment risk assessment system, there has been designed a graphical-analytical mathematical mode which confers the possibility to estimate and appreciate the risk of hearing handicap due to noise exposure, based on the difference between the hearing handicap risk due to age and noise and on the hearing handicap risk of the population [4;5].

The application of the mathematical prognosis model of the hearing impairment risk, involves the following steps [3]:

(S1): Defining the problem to solve (establishing the type of population: male or female; age of the subjects from the analysed population; daily noise exposure for n years (8h/day, 5 days/week, 50 weeks/year);

(S2): Establishing the combination of frequencies for the mediation of the hearing threshold levels;

(S3): Calculating the hearing threshold level related to the H_Q age for a certain type population (male or female) unexposed to noise according to the A database; Also, verification of the equation related to the sum of values between permanent threshold displacement caused by noise and the level of the hearing threshold related to age, which if it is higher than 40dB, then it significantly modifies the result and therefore the value of the permanent displacement of the threshold caused by noise shall be corrected according to the equation: N-(HxN)/120;

(S4): Calculating the permanent displacement of the threshold caused by noise:

$$H' = H + N - \frac{HxN}{120}, \tag{7}$$

(S5): Determining the hearing threshold level related to age and noise for the population exposed to noise;

(S6): Graphical representation in Gaussian coordinates (within a rectangular axis system in which, on the abscissa there are highlighted as percentage, at the lower part the values of people with weaker hearing (from right to left)/at the upper part the values of people with better hearing, and on the ordinate the values of the hearing threshold level in dB);

(S7): Determining the hearing impairment risk corresponding to a daily exposure to noise level in each day, over a period of time (expressed in years). With data specific for the hearing threshold level related to the age from the A database.

2.5.- Case study on the hearing impairment risk assessment.

In order to apply the mathematical model for assessing the hearing impairment risk, there is taken into account a 50 years old male population, which has been exposed to a medium level of daily noise of $L_{EX,8h}$ = 90 dB, each day, during 30 years [4]. Application of the mathematical prognosis model of the hearing impairment risk, involves the following steps:

(P1): Defining the problem to solve: population: male; age 50 years; daily exposure level each day during 30 years (8h/day, 5 days/weeks, 50 weeks/year): $L_{EX,8h}$ = 90 dB;

(P2): Establishing the combination of frequencies for the mediation of the hearing thresholds level: in order to determine the hearing handicap there is used the combination of frequencies 1000Hz, 2000Hz and 4000Hz;

(P3): Calculating the hearing threshold level related to H_Q age for a certain type population (male or female), unexposed to noise according to database A: The level of the hearing threshold level related to H_Q noise for the unexposed to noise population is calculated according to the A database (Table no.1) and it is mediated for 1000Hz, 2000Hz and 4000Hz frequencies, respectively:

Table 1

HEARING THRESHOLD LEVEL, dB									
Frequency Hz	Age: 50 years								
	0,9	0,8	0,7	0,6	0,5	0,4	0,3	0,2	0,1
Men									
500	-4	-2	0	2	4	6	8	10	13
1000	-4	-1	1	2	4	6	8	11	14
2000	-4	0	3	5	7	10	13	16	21
3000	-2	3	6	9	12	15	19	23	28
4000	0	6	10	13	16	20	25	30	36
6000	0	7	11	15	18	23	28	33	41

Selected values of the hearing threshold level, in dB, from the A database

$$\begin{pmatrix} H_{0,9;50} \\ H_{0,8;50} \\ H_{0,7;50} \\ H_{0,6;50} \\ H_{0,5;50} \\ H_{0,4;50} \\ H_{0,3;50} \\ H_{0,2;50} \\ H_{0,1;50} \end{pmatrix} = \begin{bmatrix} (-4-4+0)/3 \\ (-1+0+6)/3 \\ (1+3+10)/3 \\ (2+5+13)/3 \\ (4+7+16)/3 \\ (6+10+20)/3 \\ (8+13+25)/3 \\ (11+16+30)/3 \\ 14+21+36)/3 \end{bmatrix} = \begin{pmatrix} -2,7 \\ 1,7 \\ 4,7 \\ 6,7 \\ 9,0 \\ 12,0 \\ 15,3 \\ 19,0 \\ 23,7 \end{pmatrix}$$

Values of the hearing threshold level related to H_Q age for the unexposed to noise population

(P4): Calculating the permanent displacement of the threshold caused by noise and the verification of the equation related to the sum of values between the permanent displacement of the threshold caused by noise and the threshold hearing level related to age: At a 4000 Hz frequency, the sum of the values related to the permanent displacement of the threshold caused by noise and the hearing threshold level related to age for the quantile 01. Is higher than 40dB (table 2). Therefore, the 19 dB value from table 2 is reduced as following:

$$19 - \frac{36x19}{120} = 13,3dB$$

Table.2

NIPTS, dB Noise exposure level $L_{EX,8h}$ = 90 dB									
Frequency Hz	Exposure time: 30 years								
	0,9	0,8	0,7	0,6	0,5	0,4	0,3	0,2	0,1
500	0	0	0	0	0	0	0	0	0
1000	0	0	0	0	0	0	0	0	0
2000	3	4	4	5	5	6	7	8	9
3000	8	9	9	11	11	13	14	16	18
4000	10	11	12	13	14	15	16	17	19
6000	5	7	8	8	9	10	11	13	15

Permanent displacement of the hearing threshold level caused by noise, in dB

$$\begin{pmatrix} N_{0,9;30} \\ N_{0,8;30} \\ N_{0,7;30} \\ N_{0,6;30} \\ N_{0,5;30} \\ N_{0,4;30} \\ N_{0,3;30} \\ N_{0,2;30} \\ N_{0,1;30} \end{pmatrix} = \begin{bmatrix} (0+3+0)/3 \\ (0+4+11)/3 \\ (0+4+12)/3 \\ (0+5+13)/3 \\ (0+5+14)/3 \\ (0+6+15)/3 \\ (0+7+16)/3 \\ (0+8+17)/3 \\ 0+9+13,3)/3 \end{bmatrix} = \begin{pmatrix} 4,3 \\ 5,0 \\ 5,3 \\ 6,0 \\ 6,3 \\ 7,0 \\ 7,7 \\ 8,3 \\ 7,4 \end{pmatrix}$$

Values related to the permanent displacement of the threshold caused by noise

(P5): Determining the hearing threshold level related to age and noise for the population exposed to noise: Through the application of equation (7) referring to the hearing threshold level related to the H age and to the permanent displacement of the threshold caused by noise, real or potential N, there is obtained an approximation of the biological events, considered to be precise enough according to the specialized literature:

$$\begin{bmatrix} H'_{0,9} \\ H'_{0,8} \\ H'_{0,7} \\ H'_{0,6} \\ H'_{0,5} \\ H'_{0,4} \\ H'_{0,3} \\ H'_{0,2} \\ H'_{0,1} \end{bmatrix} = \begin{bmatrix} -2,7 \\ 1,7 \\ 4,7 \\ 6,7 \\ 9,0 \\ 12,0 \\ 15,3 \\ 19,0 \\ 23,7 \end{bmatrix} + \begin{bmatrix} 4,3 \\ 5,0 \\ 5,3 \\ 6,0 \\ 6,3 \\ 7,0 \\ 7,7 \\ 8,3 \\ 7,4 \end{bmatrix} - \frac{1}{120}\left\{ \begin{bmatrix} -2,7 \\ 1,7 \\ 4,7 \\ 6,7 \\ 9,0 \\ 12,0 \\ 15,3 \\ 19,0 \\ 23,7 \end{bmatrix} x \begin{bmatrix} 4,3 \\ 5,0 \\ 5,3 \\ 6,0 \\ 6,3 \\ 7,0 \\ 7,7 \\ 8,3 \\ 7,4 \end{bmatrix} \right\} = \begin{bmatrix} 1,7 \\ 6,6 \\ 9,8 \\ 12,4 \\ 14,8 \\ 18,3 \\ 22,0 \\ 26,0 \\ 30,0 \end{bmatrix}$$

(P6): Graphical representation in Gaussian coordinates: The results obtained in (S5) are graphically represented in Gaussian coordinates with different handicap risks represented for an arbitrary limit of 27 dB, in order to study the dependency of the risks values with limits values (Fig.1):

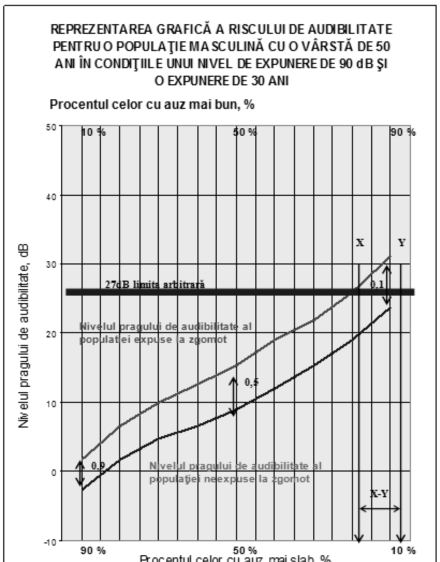

Fig.1-Graphical representation in Gaussian coordinates for the hearing impairment risk

(P7): Determining the hearing impairment risk corresponding to the daily noise exposure, for each day during a time period (in years), with specific data for the hearing threshold level related to age from database A: The graphical-analytical determination of the hearing impairment risk for a 50 year old male population exposed to average daily exposure to noise level of $L_{EX,8h}$=90 dB / day, during 30 years (8h/day, 5days/week, 50 weeks/year), has led to the following results:

- the auditory handicap risk due to age and noise is 18% (X point);

- the handicap risk of the population is 6.5% (Y point);
- the auditory handicap risk due to noise exposure is 11.5% (the horizontal difference between X and Y).

4. CONCLUSIONS

4.1. Starting from theory to practice, the paperwork highlights objectively through modern mathematical tools from the field of risk generated by noise in the occupational process, structural and process components which allow the proper determine a relative percentage of hazards identified as risk predictors, as well as the implications of these results on health and safety of the exposed population. Also, there is presented a method for estimating the hearing impairment due to noise exposure, in terms of conceptual and methodological defining for the auditory handicap assessment, taking into account both the legal consequences related to liability and compensation, as well as the legal definitions and interpretations based on economic and social considerations.

4.2. The graphical-analytical method determining the hearing impairment risk presented in the paperwork, provides the possibility to estimate this type of risk based on the auditory handicap risk of the population (determined in relation with the auditory threshold level of population un-exposed to noise) and on the auditory handicap due to age and noise (determined in accordance with the auditory threshold level of population exposed to noise). At the same time, the generalised mathematical model for assessing the hearing impairment risk, ensures, based on an algorithm, the method for estimating the auditory risk for a certain type population (male or female) which is exposed to daily noise over a certain period of time (years), representing calculation tool which is very useful for making decisions regarding the caution level required to be established for clarifying the acceptable/tolerable risk domain.

4.3. Results gained from the application of this mathematical model may contribute to the development of specialised databases for various fields of activities carried out at the level of the national economy, in order to develop the knowledge from the occupational risk prognosis and diagnosis field, having a preventive role for an occupational safety problem.

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ARTISANAL MINING IN ROMANIA AND WORLDWIDE

BUD IOAN*
DUMA SIMONA**
GUSAT DOREL***

Abstract: *Raw materials prices increasing, started in 2005, leaded to mining activity development worldwide excepting Europe and specially Romania. 2009 – 2010 world crisis doesn't seem to affect this sector, meaningful, at global level. Raw materials huge producers, especially metallic ores, have known important and continuous development. Romania prefers to close this sector in detriment of creation new strategies for efficiency development even where the profitability was obvious or easy to obtain. The price of gold, silver, copper, lead and zinc, with historic evolutions, could make profitable the ores exploitation situated at the limit of profitability (eventually with reasonably subventions) with some organizational measures. We consider that the lack of a strategy – at national level – concerning mining activity leaded to this sad situation of an important sector of national economy, inducing in public opinion – more or less informed – the idea that this activity have no reason. The strategy of a nation should be based on pertinent analyses and not on miscellaneous opinions, the less on public opinion manipulation by conjuncture political decisions.*

Keywords*: raw materials, artisanal mining, preserved areas, limit of profitability, strategy, historic evolution*

1. BACKGROUND OF MINING IN THE WORLD

Since 2005, changes in commodity prices and in particular metallic minerals were constantly increasing, culminating in gold and silver prices during the crisis. In this context, the mining industry in the world has reacted in a natural way, countries possessing the resources and experience in the mining and quarrying industry promoted and developed through existing production capacity, reopening of closed mines and exploitation of new deposits.

1.1. Artisanal mining in the world

Currently artisanal and small scale mining (ASM) refers to exploitation by individuals, families or cooperative with little or no mechanization. In some countries, there is a distinction between "Artisanal Mining" which is purely manual and on a very small scale, and "small-scale mining" which is mechanized and on a larger scale.

∗ *Prof. Dr. Eng. at Technical University of Cluj Napoca, University North Center of Baia Mare, Faculty of Mineral Resources and Environment, alizeuro@yahoo.com*
∗∗ *Assoc. Prof. Dr Eng. at Technical University of Cluj Napoca, University North Center of Baia Mare, Faculty of Mineral Resources and Environment*
∗∗∗ *Assist. Prof. Dr Eng. at Technical University of Cluj Napoca, University North Center of Baia Mare, Faculty of Mineral Resources and Environment*

Small-scale and artisanal mining frequently use one semiskilled or unskilled labor with low levels of mechanization, production, efficiency and recovery.

Artisanal mining techniques use to work with simple tools and equipment, usually in the informal sector and outside legal and regulatory framework as well. Craft operations are characterized by low productivity, lack of safety measures and a high environmental impact.

Artisanal mining occurs worldwide but is especially prevalent in developing countries in Asia, Africa, Oceania and Central and South America. According to a recent study done by the ILO, currently between 13 and 20 million people (men, women and children) are working in artisanal mining. Over 50 developing countries have involved in artisanal mining their work.

1.2. Scale artisanal mining and its effects

Artisanal mining activities can range from the most simple such as sand gold washing from riverbeds to the most complex such as underground mining and flotation processes on a small scale. In these different situations, small complex is very difficult to regulate, to ensure environmental protection and work safety conditions.

As a result of child labor is a very large number of accidents reported in artisanal mining (especially coal mines, gold mines, hard rock quarries). Global artisanal mining contributes over 12% or 330 tons per year of gold production. This rate is valid for world gold mining industry and for special community.

The gold price, which rose from U.S. $ 8,937/kg in early 2002 to more than $ 57,870 U.S./kg in August 2012 seems to be reflected in the growing number of people opting for exploitation.

In many countries, the mining activity is focused on gold production. For example in Ghana and Ecuador, gold is 2/3 of production in the Philippines 90%, Peru almost 100%. Other minerals: bauxite, gemstones, iron ore, marble, limestone, building materials etc. In India over 40 different minerals are exploited, in China more than 20.

These minerals are mainly produced for the local market and sold in villages along the road. In some countries, Ag, Zn and other base metals are produced on a significant scale.

In a largely way the artisanal mining is a survival strategy adopted primarily by rural populations and that seem to be the most promising revenue opportunity.

2. ARTISANAL MINING IN ROMANIA

2.1. Considerations on mining in Romania

In the past, around mining have been created artificially operating activities centered by community resources.

Names of municipalities were for mine: Baia Mare - Big Mine, Baia Sprie - Mine Sprie. At one point, in particular economic and political conditions, mining became unprofitable and considered "an economic black hole". In this context it was decided to stop such activities when mining countries recorded rates of development potential historical backdrop of unprecedented mining prices.

The mining activity took place in a structure socialist - communist leadership was called by unions, political, ignoring any criterion of competence, defying every rule of civilized world. Organizing this activity has all the fundamentals of obsolete structures at all levels. The normalization with unsubstantiated working time, personnel policy of the enterprise, the supply, operation of equipment, etc. turned in one unproductive activity, with high costs and low productivity.

Firms "tick" with the purpose of providing products and services to mining industry - held by people involved in the system, with access to the information within it - have strongly contributed to the undermining and destruction of the mining sector.

In a controversial economic context of post-revolutionary period, the Romanian industry began to be considered - in order to be redeemed at ruinous prices - a "scrap heap" and the mining industry a "black hole" that disappeared without profit subsidies oversized and had a devastating effect on the environment.

In Romania, all these premises were mining activity "damaging", while the world continued development of mineral resource exploitation, representing national economic engine in many countries outside Europe, even in times of crisis.

The negative image created by the political game in the 90s - when the miners were manipulated and involved in riots, underplayed by public opprobrium - was complemented by the European philosophy, which believes that there is a need of mineral resources. Europe says is an ecologist one, not become dirty with mining and metallurgy, preferring to create high technology, ignoring empires with a high rate of development rate, defying the crisis, based on the exploitation and resource recovery.

This was the posture of Romanian mining industry, undermined from within and helped to fall into the abyss, from the outside.

We believe that small-scale mining could be an alternative for Romania by encouraging private initiative in a legal framework. Ignorance authorities on social issues led much of the population in industrial areas in closure and post-closure to practice a primitive form of artisanal mining through recovery and use of metallic materials.

2.2. Artisanal exploitation cracked andesits

The Maramures County is strongly affected by the closure of mining activity in the economic, social and environmental way. In these circumstances it was used the metal recovery activities from old mining structures and a form of artisanal mining focused on andesits cracked plate. Two aspects of this activity is revealed:

- artisanal exploitation even in protected areas in multiple locations, situated at relatively low distance between them (Figure 1 and 2)
- Industrial-scale exploitation by drilling and blasting on high benches which endangers the degree of recovery of valuable reserves.

Our county has 34 protected areas of national interest, of which the largest are Maramures Mountains National Park (with 148,850 hectares) and Reserve Pietrosul Mare (3,300 hectares), plus the protected areas of local interest such as peduncled oak Reserve Bavna - Lucacesti (with 26 acres).

In the absence of technical documentation and insufficient involvement of the authorities, a large number of entrepreneurs began the operation of andesits cracked plate in artisanal manner in protected areas.

So are violated two fundamental principles of a civilized society:

- the exploitation of reserves is in agreement with the authorities, involving the development of specific technical documentation and payment of taxes and royalties to the state;
- protecting the environment and natural resources should be a goal of all human activities, especially economic ones. Protected areas are defined and delimited, but these rules are not respected.

The Scale of the phenomenon is significant (Figure 3), so in the first stage operation was performed rudimentary by family members or close with simple tools (shovel, pickaxe, etc.) and animal drawn carts transport or tractors.

Figure 1: Artisanal excavation in a protected area Natura 2000

Figure 2: Animal traction in proximity operating area

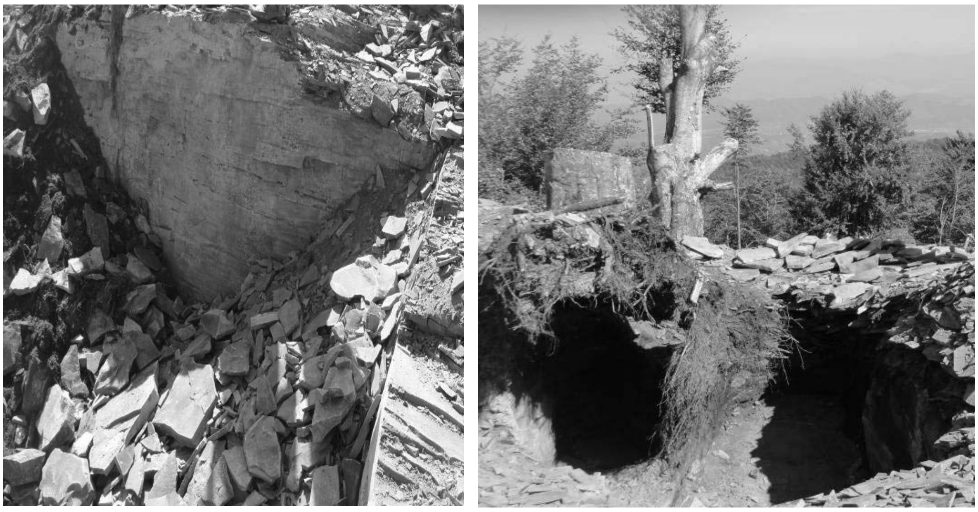

Figure 3: Exploited zones in an artisanal way

Pictures taken on the ground in the autumn of 2011, highlighting the expansion, and specific positioning this illegal activity we called artisanal mining, drawing after the international literature (Figure 4).

In the XXI century the world reached a highly advanced technology that was implemented in the mining too, leading to the production capacity of ten million tons per year (mining and processing).

Simultaneously, another pole of civilization grow subsistence activities, providing minimum livelihood for families or small communities on the edge of survival.

Exploitation of artisanal mining is a self-service market; anarchic, with no informality borderline of the country with no benefit. However, it allows those involved temporary survival. In these circumstances it would be illusory to rely on artisanal exploitation products to provide material for construction / reconstruction.

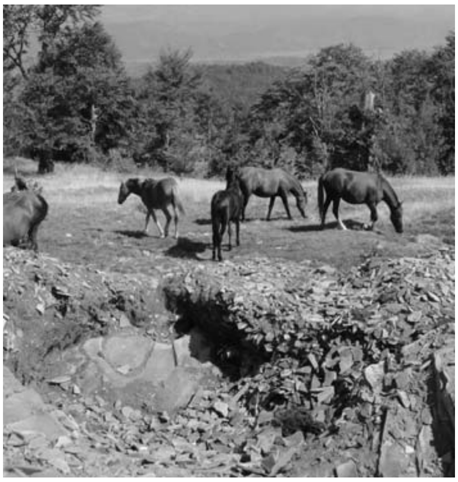


Figure 4: The Andesite plates are carried with animals or tractors

In the subsequent phase, potential entrepreneurs with financial possibilities have started to use the possibility of stripping machines and mechanical load and dump transport (Figure 5).


Figure 5: Equipment operating in proximity perimetersof the exploatation

Legalization of such activities (example Mongolia, South America and several African countries) can bring multiple benefits:

- Taxes to the state or community
- Technical advice, and HSE from the authorities as well
- Establishment of a social protection system in which these people - which are normally outside of the social security system - can be integrated.

3. THE PERSPECTIVE OF MINERAL RESOURCES EXPLOITATION

Europe starts to be aware concerning the role of raw materials exploitation in the community development and interior development of states members. One of the firs consequences is the equilibrium between environmental politic and industrial development, being attentive to the necessity of reducing dependence to the world huge raw materials producers.

There are some administrative impediments, the image of mining sector should be improved, and there is an important need of qualified work force, management techniques, education and formation.

In the Official Journal of EU (2009/C27/19) was published the Notice of Economic and Social European Committee (C.E.S.E) concerning mining non-energetic industry in Europe.

These documents present in details the role of minerals in life quality and development of sustainable communities, classifying non-energetic minerals in: metallic minerals (copper, iron, silver, etc), industrial minerals (salt, feldspar, kaolin, etc) and building minerals.

Europe has a very limited capacity to assure metallic minerals from indigene extraction (in 2004 Europe imported 177 millions tones and the production was 30 millions).

The extractive industry sector of non-energetic minerals offers job places for 295.000 persons and 18.300 enterprises.

Several European peoples don't realize the importance of mining activity but, in the future, the European sustainable development will depend on local mineral resources.

Promoting renewable energy sources and efficient energy utilization lead to technologic equipments, which incorporate important quantities of metals, especially rare and precious metals.

The new members states – the east European countries - have a very advantageous geological structure for mining activities so substantial investments in this sector will improve the security of raw materials for European industries. In these countries the mining sector has very few state investments and the private capital investment in these enterprises is essential.

The development of mining sector generates job places in other industrial sectors; usually an extraction operation creates an average of four times more indirect job places in the region where it is placed. The regional potential development is considerable especially in areas where other forms of economical development are unavailable.

There are several areas in Romania, in similar situation, such as Baia Mare, Baia Borsa area, Apuseni Mountains, etc. The notice of Social and Economic Committee asks to European Community to deliver and recommend rules to promote extractive industry, such as: improvement of legal frame and authorization system, improvement of extraction compatibility with environmental protection, improving the quality of minerals database.

Among these recommendations is placed the exploration activity, stimulants for exploration, reducing the time for land – which is object of an extraction activity - acquisition, to encourage the creation of exploration enterprises and attract enterprises from outside the European Community. Sustaining the proximity principle in minerals supplying inside European Union, will reduce the effect of "not in my back yard" thereby increasing the acceptance by the community of the extractive industry.

CONCLUSIONS

The increase in raw materials prices, started in 2005, lead to mining development world wide, excepting Europe, especially Romania. World crisis 2009 – 2012 doesn't seem to affect globally and significantly this sector. Huge raw materials producers, especially metallic minerals, have experienced a continuous development.

Instead of creating strategies for increased efficiency in the sector, Romania preferred to close it, even where it was apparent profitability or make it simple. The gold, silver, copper, lead and zinc prices, with historic evolution, could make profitable exploitation of deposits that were at the limit of profitability (possibly reasonable subsidies) by taking measures such as organizational.

Even if prices had not risen and mining had closed, we consider that the "army" of miners who had a certain level of skill and education could be used by a proper strategy to develop infrastructure of roads, railways, tunnels, etc.

"Leaving to the hearth" of this force by the granting of compensatory payments has generated a host of social problems. Unfortunately, the interests of a group of individuals, on the closure of the mines, prevailed before the national interest.

These issues can be regarded only with consternation.

The extractive industry of industrial minerals, building minerals and energetic minerals continued the same stream of anarchy, missing engineering component, remarking the irrational exploitation with economic implications (no financial rules, inefficiency, vandalism, etc.) and compromising reserves, in many cases.

We believe that the lack of a strategy-nationwide-concerning mining industry has brought in this sad situation an important sector of the economy, decided to loom the fatal stigma generated by "mineriade - miner`s battle" (phenomena of handling in the political interests of the masses of miners), inducing in public opinion, more or less competent and informed, the idea that this domain does not have any sense.

The strategists of a nation must base their vision on relevant analysis and not on the various views, the less on manipulate public opinion through political decisions of the moment.

If the notice of an activities likely to cause environmental problems can be discussed in public hearings - subject to the approval of public opinion in the area, the medium and long term development strategy of a country must be based and correlated with economic interests and promotion of its human potential, and with environmental protection in the context of sustainable development, involving specialists and strategists with experience and overall vision, beyond petty personal interests.

In the context of the current state of the global economy and the difficult economic and social situation, in Romania there is potential and the need to provide a legal framework for the development of small-scale mining activities and artisanal mining.

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EXPLOITATION OF METHANE GAS OF COAL DEPOSITS ASSOCIATED

FLOAREA DANCI[*]

Abstract: *On exploitation of coal with high level methane, we are talking about the benefits of improving the health and the safety of work, increasing production and productivity, reducing overall ventilation costs, etc..., is obvious that the implementation of modern technologies and capture of methane dranaige shoul be a priority for all mining operators. In the same time, we know that the methane obtained from coal extraction activity is 18%, in year 1990, from total emission of greenhouse gases worldwide, capturing si destroying methane drnaige should be a priority of environmental policies. Is obvious that mining operators who exploit deposits of coal with high level of methane, can be a cheap resource of energy, that can be used to cover a part or even all of the self necesary, reducing by that costs of production, of may be transformed in a product that can greatly benefits by selling to other economic operators.*

1. INTRODUCTION

Methane in content of 80-95% is found naturally in coal layers, being a product of the process of carbonization of organic substances and is released naturally by changing the tension existing in massive mining (excavation of underground mining operation itself proriu coal). In many cases the geological conditions in a particular sedimentary basin while methane can be released significant amounts of carbon dioxide gas whose presence in coal layers is related to the process of carbonization, but whose presence affects most or substantially the degassing process management strategies and layers of coal.

Reducing the risk of potentially explosive methane concentration in coal mines is mainly based on the existence of an overall ventilation system and/or partially effective to ensure rapid dilution of methane desorbed from coal hazardous concentrations ,but when the amount of methane absorbed in coal is high, it requires dilution air circulation volumes that exceed the capacity of general ventilation systems or in part, or require significant increases in the costs of operation of these systems, the only viable technical solution to ensure extraction activity in complete safety remains implement systems that can ensure capture and degassing methane drainage before it is released into the network of underground mining, or in the air stream circulated through them.

An efficient drainage system and capture the methane gas content in coal involves first selecting the most effective ways to capture it and on the other hand the implementation and execution of a drainage system to ensure the capture, transport and use of gas captured complete safety and at the concentrations of 30% CH_4 harmless concentration considered in terms of the risk of explosion.

[*] *PhD Student, University of Petroșani*

Is obvious that in terms of safety in the work of exploitable coal underground, in terms of the presence of explosive gases in the mine atmosphere, the solution is the presence of an integrated system consisting of a general ventilation system effectively and a degassing system to ensure capture and drainage of a volume of gas large enough to allow the operation of general ventilation optimum technical and economic parameters.

More than that, the existence of an adequate legal framework that encourages use of mine gas degasification systems captured by applying energy and reduce emissions of greenhouse gases generated from the extraction of coal will have a beneficial effect in stimulating investment in drainage systems capture methane gas and mining operators.

Except for security reasons and economic work should be taken into Buckets and an efficient system that will allow degassing mine gas capture the parameters of high purity (typically at concentrations ranging from 45-90% for CH_4) thus creating the conditions for its use in various energy applications leading to obtain additional income for the mine operator to direct infleuență reduce operating costs.

2. CLASSIFICATION OF METHANE GAS ASSOCIATED COAL DEPOSITS

Depending on the origin (in terms of the overlap of the degassing process at coal mining) and how to extract methane gas associated with coal deposits that can be classified.

Methane gas from virgin ores (VCBM) - is extracted from coal layers and the surrounding waste rock by drilling carried them from the surface, before the opening of the works of the deposit and its exploitation, or for those layers or portions of layers which are exploitable technical-economic considerations. Usually the content of methane si over 60%, but the results is determined by the concentration and pressure of methane gas and methane permeability layer and waste rock surrounding.

Methane gas from active mines (CMM) - is extracted by drilling underground layer made of coal mining performance while training new fields of slaughter, by the performed drilling adjacent to areas used for mining or drilling performed by surface, where the operating depth is reduced.

Methane gas from abandoned mines (AMM) - is extracted by drilling carried out from the surface or abandoned pipeline opening main proceedings, works from closing changes in methane gas storage tanks. In this case the extraction solution adopted is determined by solution of closure.

Methane gas from current general ventilation (VAM) - is due to emissions that occur continuously in the rock massif and coal mining underground route from active employment. Due to large volumes of air circulated methane concentration is generally below 0.6%, technologies of the air-methane recovery for energy is constantly growing

3. DETERMINATION OF GAS COAL DEPOSITS

Measurements on methane content of coal are commonly used both for reasons of safety at work (establishing flow of fresh air required to dilute methane emitted from the reservoir below the maximum allowed by law) and to establish the level of resources available Gas to implement recovery and recovery applications thereof.

For the measurements of gas they use two types of methods, namely:

- direct methods are generally limited to measuring the volume of gas released from coal samples provided that they are placed in sealed containers desorption;

- indirect methods based on empirical correlations, or data derived from laboratory tests to determine the isothermal absorption curves.

3.1 Direct methods

Determination of in situ gas desorption is based on the containers of coal samples collected in a fresh state as possible. During desorption samples are maintained under temperature conditions similar to those in layer measured desorption rate allows estimating the volume of gas released before the introduction of the sample container desorption.

Periodically released gas is passed into a measuring (foto 1) cylinder and may be subject to chemical analysis to determine the composition. Determining the amount of gas remaining in the sample after grinding coal is m). µrelizează sample size of less than 1 mm (USUALY 20 MICROMETRII, PUI TU SEMNUL) Proceura for determining the in situ gas content requires periods from a few days to a few weeks, although in Europe and Australia have been developed methodology for rapid determination of this parameter.

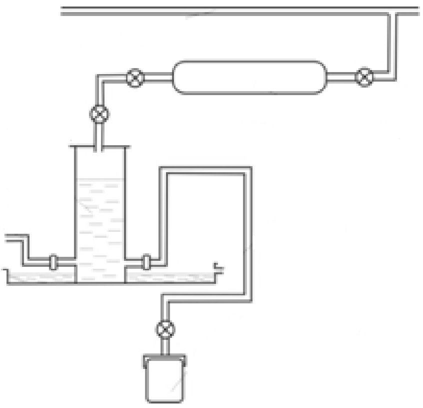

Foto. 1 Equipment for measuring gas content (Australian standard)

Total gas content of coal is composed of a series of partial contents to determine every procedure is different (Foto 2). The gas lost during sample collection (Q1) is determined from the gas container found free sampling (Q2) based on time spent for collecting the sample and the time it was released quantity Q2 (the period since the sealed container and measuring the time it) and Q3 is determined by measuring gas gathering and released after the grinding.

Under these conditions desorbed gas content of coal just is given by:

$$Q = \left(\frac{Q_1 + Q_2}{M} + \frac{Q_3}{M^I} \right) \frac{1}{1 - 1{,}1c} \quad [cm^3/g][m^3/t] \qquad (1)$$

where:

M - mass of sample taken (g)

MI - subject to breakage sample mass (g)

C - ash content (%)

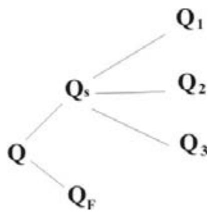

Foto. 2 Three components of the total gas content.
Q-total content;
Qs-adsorbed gas content,
QF-free gas content;
Q1-containing gas lost during sample to obtain,
Q2-found free gas content sampling container after sample collection;
Q3-gas content released by advanced grinding the sample.

3.2 Indirect methods

This method involves determining the gas pressure in coal seam and establish adsorption isotherms plotted using laboratory gas content of the coal layer. The relationship between pressure and volume of gas adsorbed in coal is:

$$V = V_L\left(1 - a_d\right)\frac{P}{P + P_L} \qquad (2)$$

where:

V-volume of gas adsorbed / desorbed from coal (m3 / t),

P - gas pressure of massive (Mpa)

VL - Langmuir volume (m3 / t),

PL - Langmuir pressure (Mpa)

ad - content ash on dry sample.

Langmuir volume and pressure is determined in laboratory conditions at steady state in the adsorption / desorption under fluctuating pressure.

3.3 Estimating gas content of coal strata based on empirical relationships

The methodology used for the determination of methane in coal is based on empirical formula and Kim Meinser of the main variables are moisture, volatile matter content, the volume of gas absorbed by wet coal, fixed carbon, coal thickness and temperature .

$$V_{CH4} = -325.6 \times \log (V.M/37.8) \qquad (3)$$

Evaluation of methane content of coal in situ can be made using equation Kimm:

$$V = (100 - M - A) /100 \times [Vw /Vd] [K(P)^N - (b \times T)] \qquad (4)$$

where:

V = volume of gas absorbed (cc / g)

M = Humidity (%)

A = ashes content (%);

$$Vw/Vd = 1/(0.25 \times M + 1) \qquad (5)$$

Vw = volume of gas absorbed by wet coal (cc / g)

Vd = volume of gas absorbed by dry coal (cc / g);

Values for K and N depend on the quality coal and can be expressed as ratios of fixed carbon (FC) and volatile matter content (VM);

$$K = 0.8 (F.C /V.M) + 5.6 \qquad (6)$$

where:

FC = fixed carbon (%)

VM = volatile matter content (%)

N = Grade Coal (for most coal,

N = (0.39 - 0.013 × K)

b = absorption constant change in temperature (cc / g / ◦ C),

T = temperature gradient × (h / 100) + To

T = temperature at depth positioning of Layer

To = temperature at ground level,

h = depth (m);

4. TEHNOLOGIES TO CAPTURE MINE GAS

Technologies for capture and methane drainage can be divided into two main groups, namely:

- technologies for carbon capture and coal seam methane drainage Inna and during operation of coal

Layer;

- technologies for methane drainage capture and exploit space during operation or post-extraction.

Technologies for capture / drainage layer existing gas or coal Starting as during operation include:

- v*ertical drilling from the surface to the layer or layers of coal* which consists in performing vertical drilling from the surface to intercept the target layers, application of technologies prefisurare usually hydraulic drilling decompression by drainage, which will enable methane desorption from the coal matrix and its migration by drilling through the newly formed cracks.

- d*rilling conducted from the surface to the coal* layer or layers applied while the gas migration favors coal properties without the need for methods of cracking of the layers. A second aspect that limits the application of technology is related to the flow of water in wells, their geometry raising special problems, technical solutions escape from economically prohibitive. The technology was perfected in Australia for degassing packages layers by introducing magnetic device guide drilling conducted to allow interception of a traditional vertical drilling.

Most often resort to running drilling intersecting the same drilling degassing of a single layer, or resot to more drilling in layers located at different depths, which intercepts same vertical drilling production (foto 3).

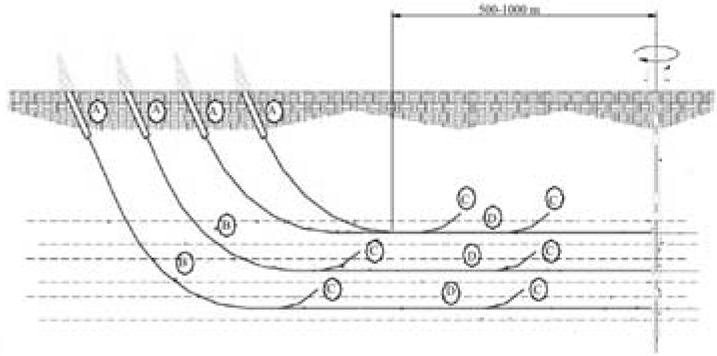

Foto.3 The schematic diagram for packet layer degassing of drilling from surface. A-drilling Φ Φ 150 mm Cased to 100 mm, B-drill interception layer Φ 96 mm, positioning the C-wells to roof layer, D-drilling Φ 96 mm layer

With this technology the recovery, under favorable conditions the permeability of coal amounts to 80% of the total gas, but degassing rate is higher than for vertical drilling technology. In terms of quality gas is extracted in terms of delivery transport networks with minimal processing costs are generally lower than for vertical drilling technology costs by eliminating fragmentation, even while the actual cost of drilling are in the range 30-100 $ / m, depending on the depth from the surface of the target layer.

5. POSSIBILITIES FOR METHANE GAS CAPTURE FROM COAL LAYERS

Depending on the concentration of methane gas mining can be considered a range of options for commercial use of mine gas, such as:

Substitute natural gas - made in three forms

-direct injection gas mine in the distribution of natural gas, where methane concentrations exceed 95% combustible gas content is below 4% and practically does not contain oxygen. In this case mine gas should be subjected to a refining process and or enrichment to be delivered to the natural gas network, a process that requires additional investment costs;

-gas liquefaction mine and use as an alternative to liquefied natural gas. Option became available with the development of technology and refrigeration and liquefaction small;

- *Production of synthetic fuels* - gas mining can be used as raw material for production of dimethyl ether, methanol and other synthetic fuels, the disadvantage is the large amount of methane given daily requirement (less than 14 x 104 m^3/day) .

Use without reprocessing mine gas - in this case usability and mine gas recovery can be different and become very attractive if the concentration of methane has values that fall outside the requirements of other applications. May consider the following possibilities:

- *Use the process of burning coal in power plants.* Viable option given that the distance from the mine to the power plant operator is less than 40 km. Benefits arising from the reduction of coal consumption for electricity and / or heat, while the positive impact on the environment because due to the reduction of NOx, SOx, and particulate matter;
- *Use in the treatment of mine water* - in many cases involves removal of underground coal mining of large volumes of contaminated water with minerals that may alter water quality supreme in the region. According to environmental requirements, in some cases it is necessary to desalination and water purification mine to be discharged into surface waters in which case mine gas can be used as energy resource in the process of desalination.
- *Use as a fuel for heating obtaining necessary mining operator* - option requires low investment and small amounts of methane is usually seasonal
- *Use direct local industrial activities using gaseous fuels in the process* - is restricted by the distance between the mine operator and the user, distance to be less than 10 km but requires providing constant flow rates and concentrations.

Electricity generation and cogeneration or trigeneration system – although the implementation of this solution requires a high initial investment effort, this solution use mine gas is the most favored by mining operations, as:

- have lately been made cogeneration internal combustion engines, gas turbines or microturbines using mine with CH_4 concentrations below 35%, leading to 25%;
- operation of groups is possible under varying concentrations of methane gas in the mine;
- cogeneration plants can be located near the source of damp usually premises mining operator;
- all mining operators are large consumers of electricity and heat, so the solution adopted for the use of mine gas in CHP system using energy produced by the operator has positive effects on production costs;

- current mine gas plants using that works in cogeneration mode touch cumulative yields between 82 and 85% compared to the solution is chosen only to produce electricity.

Recovery options for VAM

Need to ensure a proper environment underground coal mining business conduct safe, involves circulating through a network of mining large volumes of air in the exhaust into the atmosphere through the main fan stations containing methane concentrations of usually do not exceed 0.6%.

Original equipment manufacturers primary objective was to develop technological solutions to ensure methane destruction (powerful greenhouse gas) in order to obtain „carbon credits". Further development of combustion technology mixtures with low concentrations of volatile organic compounds under certain conditions allowed the process the heat is recovered and used to produce energy.

6. CONCLUSIONS

If the exploitation of coal with high content of methane, if we refer only to the benefits from improved health and safety conditions at work, increasing production and productivity, reduce overall costs the ventilation, etc..., it is clear that implementation of the most technologies methane capture and drainage should be a priority for all mining operators.

At the same time if we take into account that methane gas from coal extraction activity is attributed in 1990, 18% of all emissions of greenhouse gases worldwide, capturing and destroying methane drainage should be constitutes a priority environmental policies.

It is obvious that mining operators exploiting coal deposits with a high content of methane, it can be a cheap energy resource that can be used directly to cover part or all of their needs, significantly reducing production costs, or may become a product that can greatly benefit by selling to other traders.

After 1990 technologies using methane derived from coal extraction activities have been continuously developed so that now there is the opportunity to use air-methane mixtures at almost any concentration of methane in the mixture.

From the presentation that however the agreed and gas recovery technology used mine is the production of energy in cogeneration system using either internal combustion engines or turbines or microturbines, because not very large values of the initial investment, short payback period and especially the possibility of direct use of energy by the mine operator or the local / regional level, which requires minimal investment in transmission and distribution.

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Scientific Reviewers:
Prof.Ph.D.Eng. SÂRBU Romulus

GAS RECOVERY TECHNOLOGY OF MINE

FLOAREA DANCI*

Abstract: *Existing technologies for mine gas recovery ensures optimum efficiency energy recovery and ensures removal of a substantial percentage of methane emissions due to coal mining. Best practices and standards in the collection and drainage of mine methane gas will ensure the concentration and stable flow which will allow its use in various applications at very low cost. At this time there is available a wide range of potential applications from use as fuel in furnaces, boilers or furnaces use as fuel for internal combustion engines, turbines or microturbines, to produce energy, liquefied gas production, field the compost raw chemical industry, natural gas substitute.*

1. USING CMM FOR INJECTION INTO NATURAL GAS NETWORKS

Currently over 50% of methane emissions associated with coal mining are released into the atmosphere by power stations of fans every foul air. If coal mines operating strategies with high methane content to maintain the required levels of safety and health legislation at work, assume that in parallel with the ventilation parameters (flow rates, air circulation, etc.) to ensure dilution of methane at levels below those prescribed to implement systems capture surface drainage and disposal of CH_4 before, during, or after the actual extraction of coal (degassing systems). Most times when using sitemelor degassing, CMM parameters recovered from coal layers (CH_4 concentration, oxygen, carbon monoxide and dioxide, nitrogen, water content, etc.), it is possible to inject gas networks after it has been subjected to processing which lead to achieving quality parameters required by specific regulations for gas transmission pipeline networks. In general parameters of injection gas networks are highly restrictive, as shown in Table 1.

Table 1. Quality parameters for natural gas transported by pipeline (American standard)

Parameter	value
Content of oxygen (O_2)	< 0,2 %
Nitrogen (N_2)	Max. 3 %
Carbon dioxide (CO_2)	Max. 2 %
Low calorific power	Min. 967 Btu/scf (36 MJ/Nm^3)
Water vapor	7 lbs/mmscf (0,028 kg/Nm^3)

Generally mine gas derived from prior degassing process values approaching standards imposed by injection into natural gas networks, unlike mine gas coming from degassing facilities operated or closed mines because contamination with nitrogen,

* *Ph.D. student at University Petroşani*

oxygen, dioxide carbon and water does not meet these requirements. Whatever the origin of mine gas under the conditions in which it has a content of 50-90% CH_4, purification and distribution of natural gas as a replacement is the first option to be considered.

May be taken into account four parameters technical solutions to improve mine gas to be injected into the natural gas network:

1. Investing in technology to improve capture and mine gas drainage layers virgin focused on optimizing drilling technologies, capture and drain and continuous quality monitoring systems of gas.
2. Making colliery gas mixtures with low quality gas mine high quality, given that it is available with major effect on cost of purifying mixture to achieve the specified injection gas networks.
3. Improve the calorific value by mixing with high calorific gas (eg propane), while the injected gas quality requirements of the network allow.
4. Using enrichment and purification technologies for mine gas concentrations required to achieve the main components.

1.1. Technologies to improve the quality CMM)

a) Technology to remove nitrogen (N2) From a technical standpoint

Elimination/reduction of nitrogen from mine gas (CMM), raises the greatest difficulties, tehnologiilem available as inb the same time the most expensive. In terms of availability for industrial applications at this time can be nominated five technologies, namely:

- cryogenic technologies;
- technologies based on absorption of nitrogen pressure variations (pressure swing absorbtion PSA);
- solvent absorption technologies;
- technologies of molecular absorption filters (molecular gates technology NRU);
- membrane technology absorption.

Cryogenics - using a series of heat exchangers to liquefy the gas flow at high pressure mine and a splitter distillation nitrogen is excreted as current high nitrogen content gas. The system provides the highest methane recovery and is the most economical in terms of investment while the processing capacity exceeds 5600 Nm^3/oră.

Nitrogen absorption technologies based on variations in pressure (pressure swing absorbtion) - is based on gas absorption property at a given pressure by solid surfaces, absorption capacity is much greater as the gas pressure is higher. The technology is based on different affinity to various gas absorption sound absorption area. If CMM nitrogen removal is accomplished by passing the gas stream at high pressure through a solid bed room absorption affinity for nitrogen, exhaust gas stream is enriched in methane.After saturating the absorbent bed by reducing the absorbed nitrogen is released from the absorbent bed facility is ready for a new cycle. In general, porous materials are used to create a large contact surface as the flow of CMM, the most commonly used bedding material is activated carbon, zeolites, alumina, silica gels.

Solvent absorption - technology is based on selective absorption capacity of specific solvents for various gases. Generally resort to solvents commonly used in petro-chemical solvents released sparing methane-rich gas stream nitrogen in a low temperature environment.

Molecular absorption filter technology - based on molecular filters are the property of accurately adjusting the pore size of 0.1 Angstrom. For CMM pore size is controlled in molecular size larger than the size of your methane and molecular nitrogen, oxygen, carbon dioxide and water.

This will allow the pores to absorb nitrogen, carbon dioxide and other constituents except bed methane absorber passes over the same pressure as the input. To remove contaminants reduce supply pressure, thus allowing desorption of contaminants from absorbing filter.

Membrane absorption - technology is based on using membranes that allow selective passage of methane, ethane or other hydrocarbons, but are impermeable to nitrogen. For the nitrogen content of 6-8% is based on single-stage filter for higher concentrations will consist of two stages process with membrane filtration.

b) Technologies for removing oxygen (O_2), carbon dioxide (CO_2) and water vapor.

Disposal systems oxygen, carbon dioxide and water can function as separate units but are usually integrated in complex systems which eliminate all four contaminants, interconnected processes on the same location. Available technologies for removing other constituents of the gas are still mine. Removing oxygen from mine gas is considered as the second most important component of technologically and economically CMM purification process. This process is extremely important because of limitations imposed quality standards for gas transmission pipeline networks usually oxigen.În concentrations below 0.1% when using PSA technology (pressure swing absorbtion) and molecular gates NRU technology, demonstrated that nitrogen removal is achieved by removing the oxygen so that oxygen deficiency in bottom proportionate final is very easy, but when using cryogenic technology or organic solvents to remove oxygen system must occupy the first position in the purification process due to substantial increase in the risk of explosion high purification process.

To remove carbon dioxide available technologies include absorbent membrane, selective absorption, or units containing organic compounds ammonia derivatives. Amine units are tolerant only at very low oxygen content and therefore disposal unit of carbon dioxide must be located in flux after oxygen removal unit.

Removing water vapor from CMM is the simplest component of the gas purification mine and often is fully a molecular filter dewatering preferred over other solutions because of lower operating costs.

2. USING THE CMM TO REDUCE HUMIDITY OR EXTRACT COAL DELIVERED TO POWER STATION.

Application demonstrates its effectiveness in conditions where extraction-combustion-energy production is regarded as an integrated system, since most benefits from reducing humidity (dry) coal are found in the combustion process, namely:

- improved temperature distribution of combustion effects in reducing maintenance costs grates with bars combustion systems;
- reducing corrosion in the flue gas desulphurisation;
- the emission of fine particles the combustion process;
- drying process does not require high quality parameters for mine gas;
- reduction of greenhouse gas emissions through recovery and use of mine gas can be significant.

Humidity coal delivered termocentraleleor is a parameter characteristic of commercial contracts between mine operator and power producer and is reflected in the selling price weightings, ensuring transportation costs and additional costs generated by the combustion process with a power coal with high humidity. For inclusion in this parameter as coal producers turn to technology to reduce moisture by mechanical (centrifugal screeners asecarea on), or thermal methods using rotary dryers, fluid bed etc. However using the burners to produce hot air or steam. In general for this purpose (dry) in the production of hot air or steam coal is used for daily consumption can reach la150 tons to over 9000 tons production delivered. Meanwhile drying process using coal as a primary energy source involves consumption of about 2 MW of electricity.

3. USING THE CMM TO PRODUCE METHANOL

Methanol is widely used (about 37 million tons / year worldwide), as raw material for the production of formaldehyde as a basic component in the production of plastic explosives and dyes to obtain acetic acid component of the paint industry and adhesives. Currently has the fastest growing market for producing methanol dimethylether (DME), used as an alternative to LPG, LNG, diesel and gasoline.

4. CMM USE IN FUEL CELLS

The fuel cell is a galvanic cell in which the free energy of a chemical reaction is converted into electrical energy. All fuel cells have a similar structure and contain two electrodes separated by an electrolyte, connected to an external circuit. The anode is fed with gaseous fuels, here with their direct oxidation and the cathode is supplied with an oxidant (eg oxygen in the air) (Figure 4.7). The electrodes must be permeable structure is porous, and the electrolyte is as low permeability. If a conventional fuel cells that run on hydrogen and oxygen, the reaction that occurs is:

$$H_2 + \frac{1}{2}O_2 \rightarrow H_2O$$

5. CMM UTILIZATION FOR ENERGY IN COGENERATION SYSTEM USING GROUPS WITH INTERNAL COMBUSTION ENGINE

CMM use in internal combustion engine (CHP modules) is perhaps the most used application in the world. Of the major advantages can be mentioned the following:

- can work with mine gas from both prior degassing and gas layers derived from degassing mine exploited space;
- can be an important source of electricity and usable heat by the mine operator on-site, resulting in a significant reduction in operating costs;
- systems are compact and modular allowing substantial changes in the volume of resources available;
- to generate electricity and heat that can be distributed to existing transmission and distribution networks;
- from commercially based systems with internal combustion engines have demonstrated viability;
- installed power for one group of cogeneration have already exceeded the 3 MW;
- can operate in cogeneration or trigeneration;
- for large power plants (over 1 MW) for initial investment costs amount to about 800 € / kW installed;
- the cumulative yield cogeneration system is usually 82-85% of the 39-42% electrical efficiency and 42 43% yield trermic;
- can operate at methane concentrations between 30-85%, up to 10% CO_2, şi 5% N_2;

- emissions of nitrogen oxides, sulfur oxides and particulate matter are reduced.

6. USING CMM FOR POWER GENERATION USING MICROTURBINES

Using micro-turbines for energy production based on CMM has a number of advantages compared with other solutions mine gas recovery, such as:

- can work with mine gas from both prior degassing and gas strata of mine came of degassing exploited space;
- can be an important source of electricity and usable heat by the mine operator on-site, resulting in a significant reduction in operating costs;
- systems are compact and modular allowing substantial changes in the volume of resource available;
- to generate electricity and heat that can be distributed to existing transmission and distribution networks;
- in terms of commercial systems based on internal combustion engines have demonstrated the viability;
- installed capacity is in the range 30-2000 kW microturbines use batteries allowing substantial changes configuration depending on the volume of gas available;
- ideal for CMM with low calorific power (less than 350 Btu / scf);
- the cumulative yield cogeneration system is usually 82-85% of the 39-42% and 42-43% electrical efficiency trermic performance;
- can operate at variable methane concentrations between 30-80% - noise is low, typically below 60 dB at 10 m distance.

7. USING THE CMM TO PRODUCE SUBSTITUTES LIQUIFIED NATURAL GAS (LNG, LIQUEFIED NATURAL GAS)

Use mine gas to produce liquefied natural gas a replacement is an attractive option for mining operators because:

- technological progress achieved for small refrigeration facilities used to convert CMM to LNG facilities has led to charges characterized by specific low, especially in areas without access to pipeline transport networks of the final product;
- all available technologies can operate mine gas derived from prior degassing activity (with high methane gas) and gas mine degassing activity coming from areas operated or closed amines (average content of methane gas);
- in all cases, mine gas is used at a pressure close atmosfrică pressure, reducing gas compression costs;

- can use gas mine the concentration of nitrogen amounts to up to 30%;
- do not assume the existence of pipeline networks and can be transported by car or rail;
- existence of a local or regional markets for LNG leads to a substantial reduction in transport costs.

8. CMM CONVERSION INTO SYNTHETIC FUELS

Technological transformation of liquid gas became available since the 1920s, but large initial investment costs related to the construction of facilities for processing natural gas in synthetic fuels have been the main obstacle to their widespread, especially given that demand for fuel Petroleum products demand was much higher. Conversion technology in liquid methane best results is Fischer-Tropsch synthesis technology, based on a series of chemical reactions leading to a series of hydrocarbons, of which the most important are the saturated hydrocarbons (alkanes):

$$(2n+1)\ H_2 + n\ CO \rightarrow C_nH_{(2n+2)} + nH_2O$$

Gaseous compounds necessary to obtain the Fischer-Tropsch reaction, first be desulfurized gas to protect the catalyst, followed by a set of reactions that improves the H2/CO ratio. Of these two reactions are most important, namely:

- water reaction transforming gaseous compounds

$$H_2O + CO \rightarrow H_2 + CO_2$$

- and the reaction of methane conversion into carbon monoxide and hydrogen

$$H_2O + CH_4 \rightarrow CO + 3\ H_2$$

Discoveries of the last decade about the necessary catalysts in Fischer-Tropsch technology have led to a significant reduction in production costs so that synthetic fuel production plant with a capacity of less than 5,000 barrels / day is perfectly feasible. The technology is also applicable while the methane concentration is less than 40% and nitrogen concentration above 30%.

CONCLUSIONS

The decision to implement a project of mine gas recovery is finally captured economic decision but this decision requires careful consideration of all the variables and constraints of technical, technological, commercial, legal and/or institutional intervention. Finally this decision may have effects that at first sight are not obvious but that could materially affect the main business of mine operator or underground exploitation of coal.

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Scientific Reviewers:
Assoc.Prof.Ph.D.Eng. Mihaela PODARIU

GEOARCHAEOLOGICAL STUDY OF SOME NEOLITHIC TOOLS OF POLISHED STONE RECENTLY DISCOVERED IN THE SOUTH-WEST OF TRANSYLVANIA

LORINȚ CSABA*
IOAN ALEXANDRU BĂRBAT**

Abstract: *There are just less information about the raw material from which they were made polished stone tools such as hammers used in Prehistory. In this context, there are necessary numerous interdisciplinary studies, such as the geoarchaeological studies like this one. The article was made based on the field investigation of the archaeological sites, archaeological artefacts mineralogical and chemical determinations made in the laboratory, but also the bibliographical study was very important. Based on the interdisciplinary investigations, we hope to be able to identify, in the future, the main sources of the local raw material, used by the Starčevo-Criş communities settled in the Middle Mureş Valley.*

Keywords: *chemical and petrographical analysis, Hunedoara County, preventive archaeology researches, early neolithic*

1. INTRODUCTION

The European Union and our country are developing operational programs for the period of 2007-2013, co-financed by the European Regional Development Fund (ERDF) and the Cohesion Fund (CF), (Ex.: Operational Programme - Transport POS-T). Their main objective is to develop a modern and sustainable transport infrastructure that will facilitate a safe and efficient movement of people and goods at national and European level and contribute positively and significantly to the economic development of Romania. These programs have generated significant engineering challenges, as well as the decommissioning of large areas of land. Among the difficulties encountered have proved to be significant, as well, the issues related to the archeological downloading of important sites intercepted by the new road routes in various stages of design or execution. In the general context of the intensification of archaeological research at national level, interdisciplinary sciences as Geoarchaeology have increasingly individualized in Romania as well.

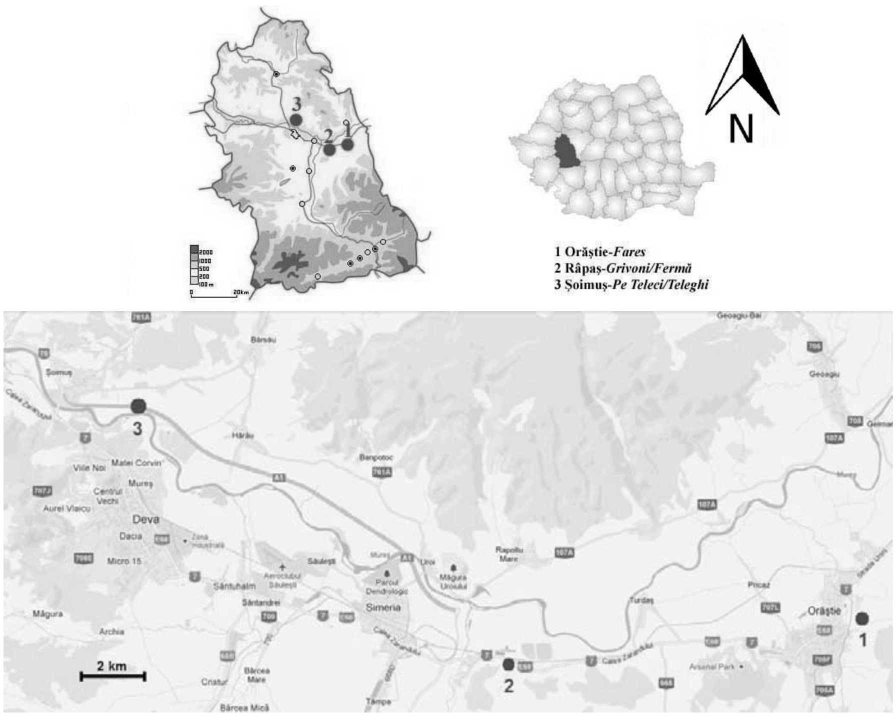
∗ *Ph.D. Eng., Lecturer at University of Petroşani, Mining Faculty, Department of Management, Environmental Engineering & Geology, csabigeo@yahoo.com*
∗∗ *Archaeologist, Museum of Dacian and Roman Civilization Deva; Ph.D. Candidate at the „1 Decembrie 1918” University of Alba Iulia, ioan_alexandru_barbat@yahoo.com*

A recent field of research, the Geoarchaeology is a multi-disciplinary approach which uses the techniques and subject matter of geography, geology and other Earth sciences to examine topics which inform archaeological knowledge and thought.

This article is based on the geological and chemical investigations of a few samples from polish stone tools – the hammers category – and its purpose being the determination of the main raw materials used in prehistory. In this case, they have been analyzed artefacts from the first period of the Neolithic, between 6150/6050-5500/5400 B.C. (Mantu 1999-2000; Luca, Suciu 2006; Luca, Suciu 2007; Luca, Suciu 2008). In that period, the Starčevo-Criş communities made this type of polished stone tools used especially to mill the grains, to break the bones to extract the marrow, to make the tools or weapons etc. (Lazarovici 1969; Lazarovici 1977; Lazarovici 1979; Draşovean 1981; Ursulescu 1983; Ursulescu 1984; Lazarovici 1984; Maxim 1999; Ciută 2005).

2. GEOGRAPHICAL POSITION AND RESEARCHES HISTORY

The archaeological sites which we refer have been located during the archaeological researches since 2009 until 2011, in Hunedoara County, on the Mureş Valley, between Deva and Orăştie (Map 1).



Map 1. The position of archaeological sites in the Mureş Valley: Orăştie-*Fares*, Râpaş-*Grivoni/Fermă*, Şoimuş-*Pe Teleci/Teleghi*[1]

[1] Graphics processing after: http://maps.google.com/ and http://commons.wikimedia.org/wiki/File:HD_County_Blank_Map.png

In this article we will analyze a part of the polished stone tools discovered at Orăştie-*Fares*, Râpaş-*Fermă* and Şoimuş-*Pe Teleci/Teleghi.* All the material is stored in the archaeological deposit of The Museum of Dacian and Roman Civilization Deva[2].

Orăştie-*Fares*[3] (Map.1/1). The archaeological site is located on the right shore of the river in Orăştie, on the second terrace, of medium height and has a dominant position in the area. To the north of the site is placing the Starčevo-Criş phase IIIB-IVA settlement known as *Dealul Pemilor* X_8 (Luca, Cosma 1993, p. 86; Paul et alii 1994, p. 45; Paul et alii 1995, p. 63; Luca 1997, p. 19-20; Luca et alii 1998, p. 17-29; Maxim 1999, p. 173; Luca, Pinter 2001, p. 23; Luca 2005, p. 112; Luca 2008, p. 123; Luca et alii 2011, p. 113). The location of the recently discovered Neolithic settlement is on the north side of the terrace. The biggest concentration of the material is in this area but there are few ceramic pieces in the south part. The archaeological site of Orăştie-*Fares* point, has been identified in 2004, during some field researches. During the year 2009 have been carried out farming activities which have disturbed archaeological deposits. On this occasion have been recovered some archaeological pieces among many polished stone hammers.

Râpaş-*Grivoni/Fermă* (Map.1/2). Archaeological point has been found on a terrace of average height, located on the right-hand side of the road connecting Râpaş village and DN 7. The settlement is located on the second terrace on the left bank of the Râpaş Valley, a southern afluent of Mureş River. At approximately 100 m south of the archaeological point presented begins the hearth of the actual village. The site was discovered by Mihai Căstăian and Antoniu Marc, archaeologists from MCDRD, on a project that included the network of water for supply Râpaş village (Luca 2008, p. 139; Marc 2010, p. 42). Subsequently, another collective from the same institution have conducted research to determine the extension of the site (2009). First Neolithic materials were recognized during a preventive archaeological research in 2010[4].

Şoimuş-*Pe Teleci/Teleghi* (Map.1/3). Average terrace is located on the shore of the river law Mureş, between Bălata and Şoimuş villages, being bounded to the north of poultry holdings and to the south of the Mureş River. Archaeological research carried out at this point met, over time, a series of forms – casual, surface and preventive (Andriţoiu 1979, p. 27, nota 35; Draşovean, Rotea 1986, p. 9; Lazarovici, Kalmar-Maxim 1991, p. 131; Maxim 1999, p. 187; Luca 2005, p. 151; Luca 2008, p. 165). Recent archaeological research carried out in the autumn of 2011 in the Şoimuş area and in *Teleci/Teleghi* point by a archaeological collective from MCDRD[5] to save some of the key objectives of the highway Deva-Orăştie, have led to the identification on the west side of terrace of some human habitation traces from the early and classical phases of the Starčevo-Criş cultural complex (Schuster et alii 2012, p. 291-292).

3. ARCHAEOLOGICAL CONTEXT AND TYPOLOGICAL DESCRIPTION OF POLISHED STONE TOOLS

3.1. Cultural and chronological framing of discoveries

For archaeological site Orăştie-*Fares*, the lithic artifacts, in large part, has been gathered from the surface based agglomerations of pieces that can come from possible

[2] In the text we will use the abreviation MCDRD;
[3] The terrain is holded by a private company;
[4] The members of the research team were: Romică Pavel, Costin Daniel Ţuţuianu, Oana Tutilă and Alexandru Bărbat;
[5] The members of the research team were in the West part of archaeological site (also called – B Area): dr. Cătălin Rişcuţa, Antoniu Marc and Alexandru Bărbat;

archaeological complex dispersed by the agricultural work. Based on ceramics discoveries from this point, among which a hammer stone analyzed in this article, may be placed in Early Neolithic IVA phase of the Starčevo-Criş cultural complex. From the Râpaş-*Grivoni/Fermă* site, in the hut L1/2010, were recovered two stone hammers, one of the pieces being subject to analysis more complex than macroscopic observation. During the archaeological research at Şoimuş-*Teleci/Teleghi*[6], have been identified several parts of the stone hammers category, inside of early neolithic dwellings of the phases IC-IIB and IIIB of the Starčevo-Criş cultural complex. These artefacts come from two archaeological complexes: one of them a semi-subterranean house (C18), chronologically situated in phase IC-IIA of the Starčevo-Criş cultural complex[7]. The second piece comes from the structure of the aboveground dwelling from IIIB phase of the same archaeological frame[8].

3.2. Typological description of polished stone tools

We will describe the pieces from archaeological[9] and typological[10] point of view, following the alphabetical order in which they are submitted in the first part of this study:

- Stone hammer (Figure 1/1), Orăştie-*Fares*, traces of use to a single end, L=92,96 mm, l=82,14 mm, transversal profile in ellipsoidal form, the longitudinal one in ovoidal, elongate and flattened form, type Antonović VI/1/f (Antonović 2003, p. 56, Sl/Fig. 34/VI/1/f).
- Stone hammer (Figure 2/1), Râpaş-*Grivoni/Fermă*, fragment, traces of use to one of the sides, L=145,72 mm, l=76,72 mm, transversal profile in triangular with rounded corners form, the longitudinal one in rectangular rounder corners form, type Antonović VI (Antonović 2003, p. 56, Sl/Fig. 34/VI).
- Stone hammer (Figure 3/1), Şoimuş-*Pe Teleci/Teleghi*, fragment, traces of use to a single end and to one edge, L=88,12 mm, l=91,95 mm, transversal profile in rhomboidal with rounded corners form, the longitudinal one possible ovoid form, type without correspondence.
- Stone hammer (Figure 4/1), Şoimuş-*Pe Teleci/Teleghi*, fragment, traces of use to the wide surface and to one end, L=63,66 mm, l=83,47 mm, transversal profile in circular form, the longitudinal one possible ovoid form, type Antonović VI/3/c (Antonović 2003, p. 56, Sl/Fig. 34/VI/3/c).

4. RESEARCH METHODOLOGY

Samples were submitted to chemical analysis in the Laboratory for Monitoring Environmental Factors – CNH S.A. Petroşani, using the S4 Pioneer X-ray spectrometer and Spectra Plus software. Mineralogical and petrographical microscopy analysis was also performed in Geology Laboratory of Petroşani University, Mining Faculty, Department of Management, Environmental Engineering and Geology. For Mineralogical and petrographical investigations were performed thin sections on four pieces, pointing out (for a small group of

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[6] One of the pieces was found under the ground, between 1.25-1.35 m, in a sub-terrain house named L1/2010 from S1;

[7] The object was discovered in the deepest part (0.45-0.60 m) of C18 archaeological complex, in F-G, J-K square;

[8] The last artefact was discovered in the SW corner of a surface house named by us *C Limită sit*, at 0.25/0.30 m under the ground;

[9] For the artefact description we have been followed the next order: artefact type, place of discovery, preservation, traces of use, dimensions (maximal length and weight), transversal and longitudinal profile form;

[10] For the typology of the polished stone tools we have been used Antonović 2003.

artifacts) some raw materials used in Early Neolithic for making these kinds of artifacts like hammers[11]. To complete geological data were performed also chemical analysis, we provide more data, useful both this study and other works, about the chemical composition of rocks, especially for identifying source area of lithic material that formed the basis of manufacturing artifacts[12].

5. ANALYSIS AND RESULTS

5.1. Sample X8, Orăştie-*Fares*; (Table 1, Figure 1/1-3).

1. The structure: crystalloblastic and granoblastic	**2. The texture:** compact, unoriented
3. Mineralogical composition	**Main mineral:** quartz The quartz – is in the form of crystalloblastic and granoblastic with irregular contour with translucent appearance and irregular cracks. **Secondary minerals:** limonite, occurring in cracking areas.
4. Name of rock: quartzite	

Table 1. Chemical analysis results for sample X8 (Orăştie-*Fares*).

Oxide/chemical element	**SiO_2**	**Vol.**	**Al_2O_3**	**Fe_2O_3**	**Na2O**	**CaO**	**K_2O**
[%]	74.7	22.3	2.17	0.502	0.186	0.107	0.0432
Oxide/chemical element	**P**	**Cl**	**S**	**Zr**	**Mn**	**Zn**	**Y**
[%]	0.0189	0.0057	0.0043	0.0017	0.001	0.0008	0.0005
Oxide/chemical element	**Ni**	**Ga**	**Total: 100,0416 [%]**				
[%]	0.0003	0.0002					

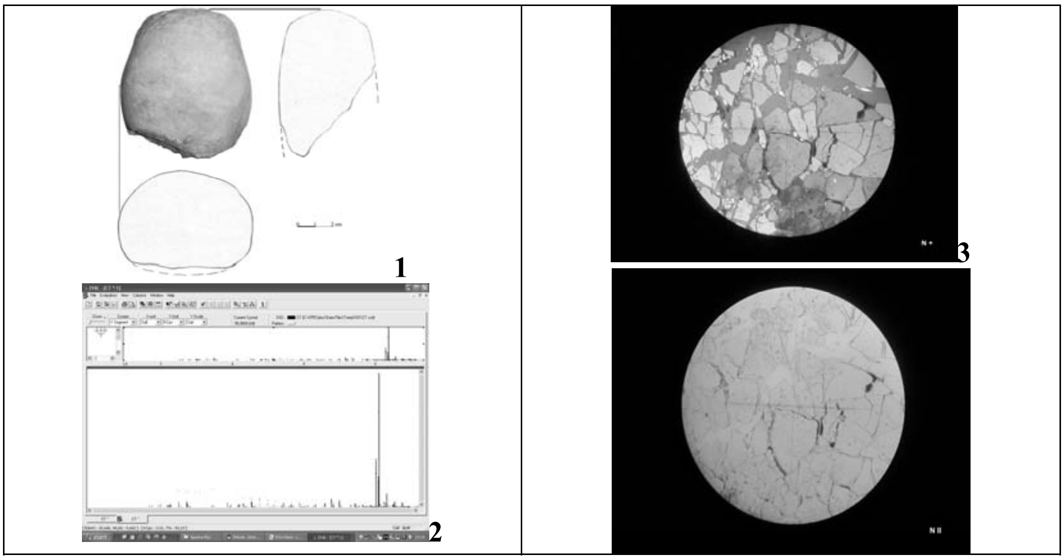


Figure 1. Hammers (Crusher / striker) of quartzite found at Orăştie-*Fares* (1); Chemical spectrum image (2); Microscopic view (40X) (3).

[11] Analysis performed in according with: STAS 6200/4-1981; SR EN 932-3/1998;

[12] Chemical analysis was performed in the Laboratory for Monitoring Environmental Factors – CNH S.A. Petroşani, using the S4 Pioneer X-ray spectrometer and Spectra Plus software and „calibration" method.

5.2. Sample SI-L1, Râpaş-*Grivoni/Fermă*. (Table 2, Figure 2/1-3).

1. The structure: granoblastic	**2. The texture:** oriented - schistose
3. Mineralogical composition	**Main mineral:** orthose, quartz, muscovite, biotite. 1) The orthose – appears granular with irregular contour and turbid surface due to chemical alteration with the formation of secondary minerals like kaolinite and sericite type; some orthose granoblaste present as inclusions quartz granoblaste and lepidoblaste of muscovite; 2) The quartz - appears as granoblaste with irregular contour and smaller than the orthoses, present translucent surfaces furrowed sometimes by irregular fracture lines; 3) The muscovite – appears as (lamellar) lepidoblaste, included most of the times in the mass of orthosies granoblaste; present alteration processes such sericitizations; 4) The biotite – rarely occurs in the form of brown lepidoblaste with surface that shows intense alteration processes, respectively transitions in limonite; **Secondary minerals:** Kaolinite, sericite and limonite; These occur as a consequence of chemical alteration processes of main minerals respectively the orthose, quartz, muscovite and biotite.
4. Name of rock: Gneiss (Orthogenesis – resulting from metamorphosis of granite).	

Table 2. Chemical analysis results for sample SI-L1 (Râpaş-*Fermă*).

Oxide/chemical element	SiO_2	Al_2O_3	CO_2	Na_2O	K_2O	CaO	Fe_2O_3
[%]	63.0	14.5	11.4	3.87	3.18	1.73	1.52
Oxide/chemical element	MgO	Ba	Ti	Sr	P	Mn	Zr
[%]	0.335	0.1675	0.1275	0.1012	0.0421	0.0148	0.0080
Oxide/chemical element	Cl	S	Zn	Rb	Pb	V	Ga
[%]	0.0065	0.0052	0.0036	0.0025	0.0025	0.0022	0.0015
Oxide/chemical element	La	Y	Nb	Ni	**Total: 100.0227 [%]**		
[%]	0.0009	0.0006	0.0006	0.0005			

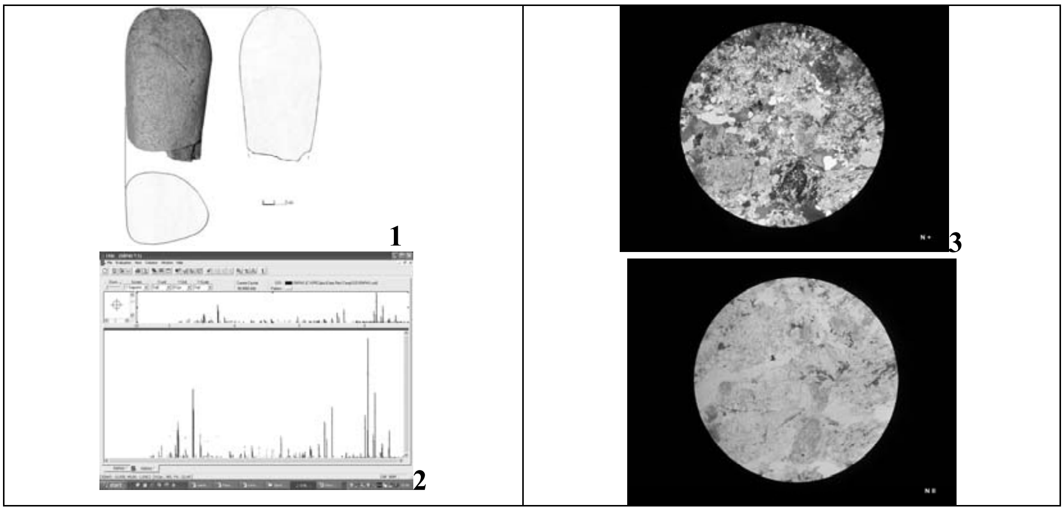


Figure 2. Hammers (Crusher / striker) of gneiss found at Râpaş-*Grivoni/Fermă* (1); Chemical spectrum image (2); Microscopic view (40X) (3).

5.3. Sample B, Şoimuş-*Pe Teleci/Teleghi*. (Table 3, Figure 3/1-3).

1. The structure: granoblastic	**2. The texture:** compact, unoriented
3. Mineralogical composition	**Main mineral:** quartz, biotite 1) The quartz – appears as translucent granoblast with irregular contour and cracks; 2) Biotite occurs sporadically as brown lepidoblaste; **Secondary minerals:** limonite, occurring in cracking areas and on the lepidoblast of biotite.
4. Name of rock: quartzite	

Table 3. Chemical analysis results for sample B (Şoimuş-*Pe Teleci/Teleghi*).

Oxide/chemical element	SiO_2	Vol.	Al_2O_3	K_2O	Fe_2O_3	Na2O	CaO
[%]	78.0	17.5	2.91	0.739	0.602	0.126	0.0569
Oxide/chemical element	**P**	**Ti**	**Ba**	**Cl**	**Zr**	**S**	**Mn**
[%]	0.0207	0.0138	0.0115	0.0076	0.0047	0.0021	0.0014
Oxide/chemical element	**Nd**	**V**	**Sr**	**Rb**	**Zn**	**Y**	**Ni**
[%]	0.0011	0.0011	0.0009	0.0008	0.0007	0.0007	0.0003
Oxide/chemical element	**Ga**	**Nb**	**La**	**Total: 99,9944 [%]**			
[%]	0.0003	0.0002	0.0002				

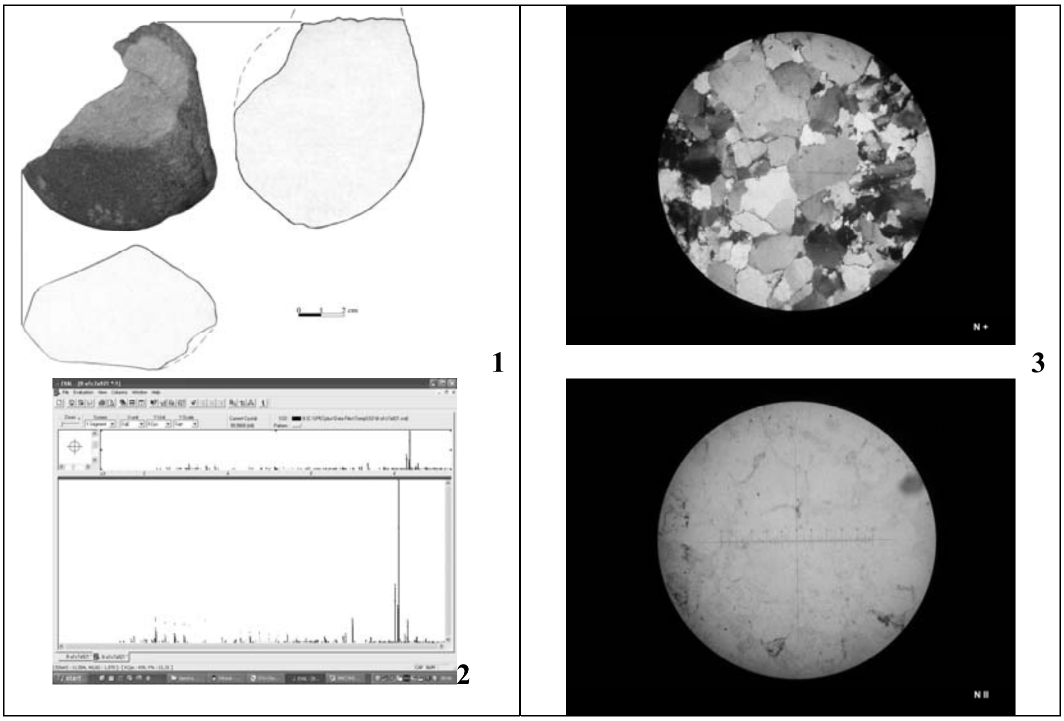


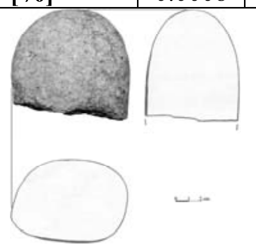
Figure 3. Hammers (Crusher / striker) of quartzite found at Şoimuş-*Pe Teleci/Teleghi* (1); Chemical spectrum image (2); Microscopic view (40X) (3).

5.4. Sample C403, Şoimuş-*Pe Teleci/Teleghi*. (Table 4, Figure 4/1-3).

1. The structure: granoblastic	**2. The texture:** oriented - schistose
3. Mineralogical composition	**Main mineral:** orthose, quartz, muscovite, biotite. 1) The orthose – appears as granoblast with prismatic - irregular contour and turbid surface due to chemical alteration with the formation of secondary minerals like kaolinite and sericite type; some orthose granoblaste present as inclusions quartz granoblaste and lepidoblaste of muscovite; 2) The quartz – Is frequently, appears as granoblaste with irregular contour, present translucent surfaces furrowed sometimes by irregular fracture lines; The granoblastic small quartz, which occurring as inclusions in feldspar mass, are of the second generation; 3) The muscovite – appears as (lamellar) lepidoblaste sometimes with prismatic-elongated shape, included most of the times in the mass of orthosies granoblaste; present alteration processes such sericitizations; 4) The biotite – rarely occurs in the form of brown lepidoblaste with surface that shows intense alteration processes, respectively transitions in limonite; **Minerale secundare:** caolinit, sericit, limonit. **Secondary minerals:** Kaolinite, sericite and limonite; These occur as a consequence of chemical alteration processes of main minerals respectively the orthose, quartz, muscovite and biotite.
4. Name of rock: Gneiss (Orthogenesis – resulting from metamorphosis of granite).	

Table 4. Chemical analysis results for sample C403 (Şoimuş-*Pe Teleci/Teleghi*).

Oxide/chemical element	**SiO_2**	**Al_2O_3**	**CO_2**	**Na2O**	**K_2O**	**Fe_2O_3**	**CaO**
[%]	65.5	14.1	10.1	3.43	3.20	1.58	1.32
Oxide/chemical element	**MgO**	**Ti**	**Ba**	**P**	**Sr**	**Zr**	**Mn**
[%]	0.399	0.1135	0.0984	0.0720	0.0498	0.0100	0.0098
Oxide/chemical element	**S**	**Cl**	**Zn**	**Rb**	**Pb**	**Ga**	**V**
[%]	0.0097	0.0061	0.0060	0.0029	0.0021	0.0019	0.0017
Oxide/chemical element	**Nb**	**La**	**Y**	**Ni**	**Total: 100,0157 [%]**		
[%]	0.0008	0.0008	0.0008	0.0004			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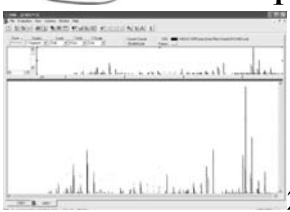
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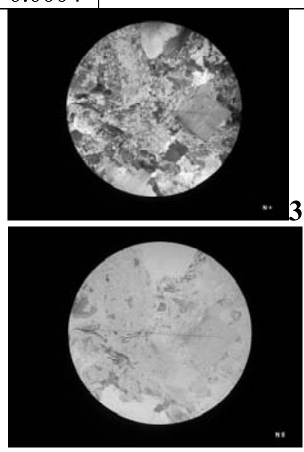
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Figure 4. Hammers (Crusher / striker) of gneiss found at Şoimuş-*Pe Teleci/Teleghi* (1); Chemical spectrum image (2); Microscopic view (40X) (3).

6. CONCLUSIONS

Polished stone tools (hammers-crusher/striker) have been barely analyzed from geological and chemical point of view. The first studies of Romanian literature which have treated these issues, took into account axes or other pieces (Stoicovici, Blăjan 1979, p. 31, 51-61) and less hammers-crusher/striker.

In Romania, petrographic determinations were performed on polished samples from Gura Baciului (Lazarovici, Maxim 1995, p. 159-162) and Trestiana (Ursachi 1999-2000, p. 28-29). In the last case there wasn't specified the type of analyzed item. Geological analysis revealed that early-neolithic populations, from Gura Baciului, use andesite for manufacture tools from group of hammers (crusher/striker), and sometimes the axes, after accidental cracking, were reused as a hammers (crusher/striker). In this case we are dealing with a diabase from the raw material point of view (Lazarovici, Maxim 1995, p. 160).

In Hungary, for Körös type discoveries, which is approximately the same cultural phenomenon as here, petrographic results revealed the use of raw materials such as quartzite for manufacturing pieces like hammers (crusher/striker) in the archaeological sites from Endröd-*39* (Starnini, Szakmány 1998, p. 341), Endröd-*119* (Starnini, Szakmány 1998, p. 337-339) and Méhtelek-Nádas (Starnini 1993, p. 81).

In Serbia, in Archaeological sites Starčevo-Criş from Divostin, Donja Branjevina, Velesnica and Tečić, have been identified polished tools manufactured from category of gneiss (Antonović 2003, p. 29, 31-32, 49, 138, Sl/7).

Mineralogical and chemical results obtained on polished tools analyzed in this study, shows that hammers (crusher/striker) used by Starcevo-Cris cultural complex communities in the considered area are made from local raw materials, until now being outstanding quartz and gneisses. The boulder stones shapes and their occurrence indicates the probable use of river beds of Orăştie, Strei and Mureş as sources for collecting this pieces/ raw materials. Also, references and other specialized sources indicate the emergence of gneisses in this area (Savu et al. 1968, p. 17).

Very small number of artefacts does not provide a complete picture of their percent in the current activities (in particular cereals milled), inside a Starčevo-Criş settlement or dwelling. Thus, in the future, these analysis should be extended for the other Early Neolithic sites, as well, to compare the results and to extract relevant conclusion.

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Scientific Reviewers:
Prof.Ph.D.Eng. Laura COMĂNESCU

SMOOTH SLOPE FOR THE ROAD SECTOR NO 3, THE A3 MOTORWAY, USING EXPLOSIVES

BOLGAR MIHAI*

Abstract: *The road no. 3 of the A3 motorway is planned to run on the itinerary localities Comarnic - Sinaia - Busteni - Azuga - Predeal, on Prahova river bed and right next to the river, going so Prahova Valley. Located in the center - south of Romania, Prahova Valley is one of the most important gateways between Wallachia and Transylvania, with here and there an aspect of the gorge, separating Baiului and Bucegi Mountains. Moreover, we can say that separates the Eastern Carpathians of Southern. The question, however, is the way to achieve this stabilization, if necessary, and smoothing the slope for the road. Smoothing is cleaning his stumps and roots, which can be done either by using excavators or using explosives. Next we consider smoothing slope with industrial explosives.*

Keywords: *smoothing, slope, explosives, cartridge, shooting.*

1. INTRODUCTION

In terms of administrative, this sector across Prahova county, in 80% proportion and 20% Brasov county. Figure 1.

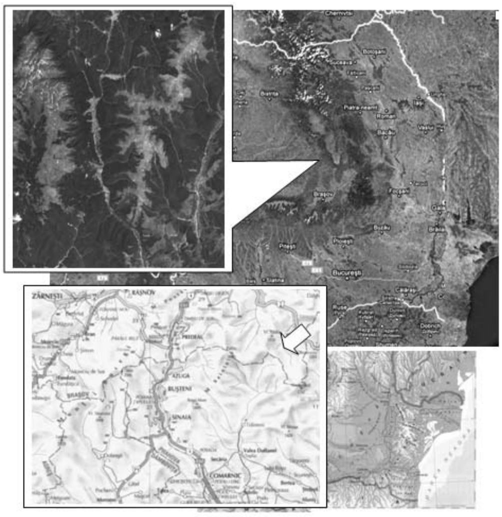

Fig. 1. Geographical location of the road sector

∗ *Ph.D. student at University of Petroșani*

Highway in this area ranges from south - west of the Curvature Carpathians where we find Comarnic city in Prahova County, north - east is bordered by mountains called Baiului and Garbova, mountain that is part of the Eastern Carpathians and depressions intramontane of Prahova Valley adjoins Bucegi of the Southern Carpathians.

The landscape is shaped by petrographic and tectonic nature of the basic, ranged from Sub Carpathian area and Brasov Depression, ± 1000 m level, proper of sector-higher and runs from Bucegi massif and Garbova massif. Deepening of Prahova Valley was made in layers of petrography - marl - limestone heavily wrinkled and diaclase and his own tectonic in the limestone blocks embedded in these formations but also according from the strong erosion imposed by the Prahova River.

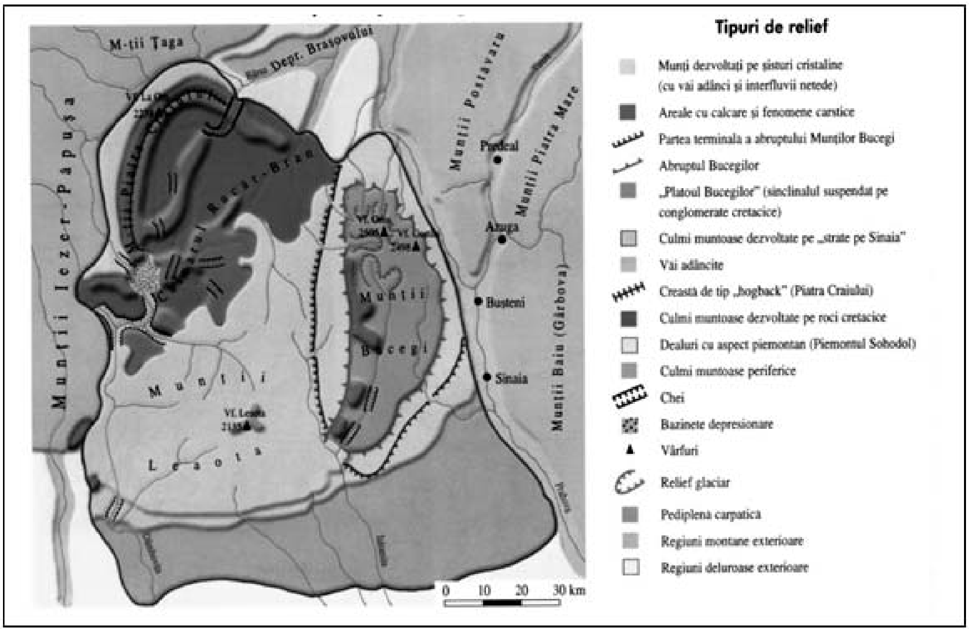

Fig. 2 Types of terrain encountered

Mutations of the watershed in the north and south of the Upper Prahova current flow are recorded from configuration from Predeal level and valleys generations. The down basic level formed by the Prahova River and piercing from Posada canyon are boosted the regressive erosion. Prahova River tributaries from the Bucegi Mountains have uncovered a true alignment of springs at the base of the conglomerate, significantly increasing river flow and hence their erosion. The specific relief of Prahova lane, alongside morphogenetic relations with neighboring areas pleads for individualization of some distinct stages of evolution.

2. BLASTING STUMPS WITH ROOTS WITH EXPLOSIVES

With blast works can be snatch, cut and remove stumps with roots from the soil. Such works may be performed for construction of roads and industrial objectives in forested areas and deforested and to obtain firewood.

In our case this must be achieved before building the support wall because on the slope corresponding to the road tunnel are trees and stumps which must be removed. The diameter stumps encountered on slope is approximately equal to 30 dm in diameter.

At stumps with large roots at small depth and diameter greater than 0.5 m, as well as at stumps with dense intertwining roots, the explosive charge is inserted through a hole practiced until under the center of the stump.

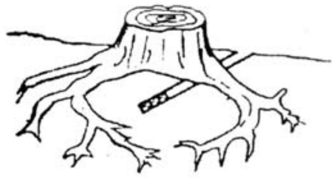

Fig.3 Location of explosive charge to remove stumps with large roots to small depth.

At stumps with pivoting roots, explosive charge is inserted through drillings along pivot with a length of 1.5 to 2 times the diameter and finished with an oven in order to place a concentrated load.

At rotten stumps the load size of explosive can be inserted through a hole in hollowed in the center of rotten and then tamping.

The load size of explosive for pulling out stumps from the ground is determined by the pivoting stump diameter:

$$Q = q \cdot d \qquad \text{(kg)}$$

with:

d – stump diameter, dm;

q – specific consumption of explosive, kg/cm^3.

$$Q= 0{,}22 \cdot 30= 6{,}6 \text{ kg}$$

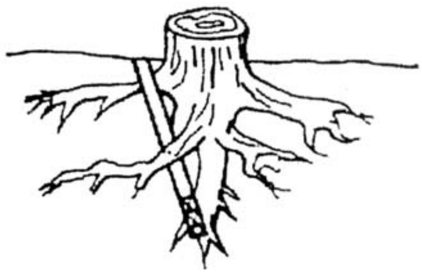

Fig. 4 Location of explosive charge to remove stumps with pivoting roots

When stumps diameter is over 0.6 m and with large roots can be used 2 to 3 loads.

Specific consumption of explosive depending type of wood used, type root and soil type is chosen as in Table 1.

Splitting of stumps with roots, after being removed from the soil can be also by shooting with explosive charge applied or placed in holes. The holes are drill either by cut surface to ¾ of the height stump, or laterally between two ramifications up to ¾ of the stump diameter.

The load size of explosive for splitting stumps with vertical holes on the surface of cutting and then tamping is:

$$Q = 0{,}0001\ d^2 \quad (kg)$$

Table 1

Wood used	The way root	Soil type	
		hard, stony and rocky	**sandy and sandy clay**
		Specific consumption of explosive (kg)	
Hardwood	- flat and stretched - pivoting	0,22 0,250	0,300 – 0,400 0,350 – 0,450
Soft woods	- flat - pivoting	0,100 0,150	0,22 0,250 – 0,350

and with lateral holes:

$$Q = 0{,}00003\ d^2 \quad (kg)$$

where:

d – stump diameter, cm.

Thus, in our case, the load size of explosive for splitting stumps with vertical holes on the surface of cutting is:

$$Q = 0{,}0001\ .\ 30^2 = 0{,}09\ kg$$

3. CONCLUSIONS

Given the amount of explosive required slope smoothing, we can assert that it is much less expensive to use this method because it assumes personnel costs, costs of equipment and lower energy. Also duration of the smoothing is much shorter which leads to an advantage in addition to those outlined above, reason for is proposed to realize smooth slope along the route of the A3 highway with explosives.

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Scientific Reviewers:
Prof.Ph.D.Eng. ROTUNJANU Ilie

AIR QUALITY CONSIDERATIONS IN UNDERGROUND AND SURFACE WORKINGS ATMOSPHERE AND IN THE SURROUNDING ENVIRONMENT PRAID SALT MINE

GHEȚIE GHEORGHE *
DRĂGHICI ANGELICA **
MARIUS KOVACS **
TOTH LORAND **
SIMION SORIN ***

Abstract: *While performing professional activities, workers may be subjected to the action of pollutants that affect their health and physical integrity directly or indirectly. This paper presents air quality parameters (dust, gas) both in the workplaces of the salt mine, in the treatment base and in the surrounding environment. For the knowledge of air quality parameters at underground and surface workings and at the treatment base and the surrounding environment the team within INCD INSEMEX under the contract no. 5907/2012 conducted a series of measurements to assess the state of health and safety at the unit level.*

Keywords: *assessment, air quality, surrounding environment, work places*

1. HISTORY OF PRAID BASIN

Praid Basin is located on the eastern side of the Transylvanian Basin, at the base of Gurghiu Mountains forming a well-defined micro-region called "Salt Mines Area." Praid Depression presents itself as a sub-mountainous grooved longitudinally to the axis of the volcanic mountains and it is from the Gurghiului gorge until Kalonda hill.

The salt mines and Praid basin lies at the boundary of two major geological structures, respectively volcanic chain neoeruptiv Călimani-Gurghiu-Harghita and sedimentary Cretaceous, Paleogene and Neogene of Small Târnavei valley. Lying volcanic plateau is eroded and partitioned since Quaternary, the headwaters of young streams.

The material is rock salt accumulation, a monomineral rock (sodium chloride - NaCl), consisting of "halite" mineral that crystallizes in the cubic system, or less octahedral system. His cleavage is uneven, shaped shell, with breaks along the cleavage plans.

Mine depletion of Doja forced to make works of ore exploitation to delimit the new location of the mining field.

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* *Eng., Head of Environmental Protection Laboratory INCD INSEMEX PETROŞANI*
** *Ph.D. eng. Scientific Researcher III INCD INSEMEX PETROŞANI*
*** *Ph.D. eng, Scientific Researcher III INCD INSEMEX PETROŞANI*

The exploiation continued in the NE of the salt mountain, after execution of the necessary opening workings (ventilation shaft, coastal slope connecting the Horiz. I and gallery Horiz. I), which were then taken underground drilling horizontal transverse galleries in horizontal directional. V and the Horizont. VIII.

For a possible extension of Doja mine underground drilling were performed, which located a higher quality salt deposit.

After 1978 begins the opening of lower horizons by performing adits, the inclined plane and blind pit extraction. At lower horizons were designed operating rooms long rectangular pillars in an orderly arrangement, with dimensions of 20m wide, 12 m high and 100-250 m long.

2. METHOD OF EXPLOITATION

Mining method used today is stopping with explosives. At the salt mine are used two methods, depending on the size and layout of the operating room. At lower horizons (as mine Doja), named after absolute share horizons: 286 m 266 m 246 m and 230 m, is used "salt mining method with small rooms and rectangular pillars". Operating room size is: width 20 m, height 12 m and lengths between 200-275 m and the pillars, as of 20 mx 12 m, with lengths of 95-100 m thickness floor of horizons is 8 m should be noted that the size of the rooms is used only at Praid mine [2].

Technology work comprises the following: salt-cutting machine cutting path from the hearth, punching front, two – stage blasting front, airing the room, , breaking oversized and salt removal from front.

To ensure continuous extraction process, operating activities are carried out simultaneously in at least three operating rooms, each executing one phase of work is distinct from the stopping methodology. Amount of salt resulting from the blasting cycle is 930 tons.

In preparation plant salt is ground to different sizes and types, is treated with an anti-caking substance and comes packaged in bags or bulk, taking into account the demands of internal and external beneficiaries.

3. TREATMENT BASE

From Transylvania and Romania Salt mines, to at the Praid mine is practiced the most developed medical treatment. Starting from the therapeutic results achieved in salt mines (e.g., Wieliczka, Poland), the managers of the salt mine, attempted to organize underground treatment at respiratory issues.

The underground treatment hall was moved in 1980 to a more appropriate dried horizon, (402 m horizon, called "50") into a camera system, which currently operates. Here at 120 m depth in salt massif, the number of visitors in peak season, reaches 2500-3500 people / day. Treatment rooms and annexes are electrically lighted and provided with a ventilation system with continuous operation. Also equipped playgrounds, chairs and loungers, pool tables, table tennis, an ecumenical chapel and a bar, all to make it bearable and pleasant, the minimum required time is 4 huors for underground treatment.

The rooms are large so that the patients can practice various ball games: badminton, volleyball, handball, etc

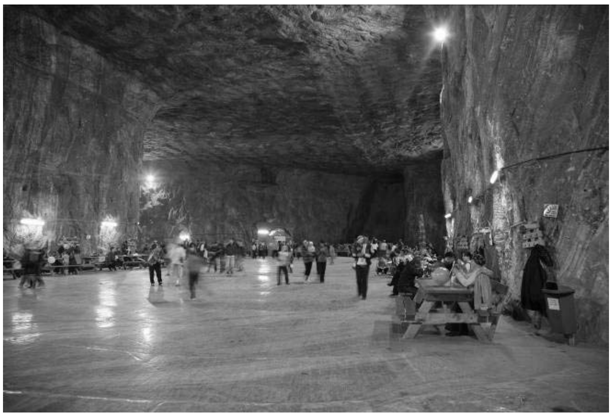

Foto no. 1 – Treatment hall Praid Mine

Foto no. 2- Restaurant in Praid Mine

The set of physical, chemical and biological conditions, treatment base occurs together, causing complex effects on the human body.

Factors that contribute to specific speleotherapy are:

- High degree of air purity (sterility);
- Relatively high air humidity (vapor content);
- Content of favorable vapor condensation;
- Constant air temperature;
- Secondary action of relatively low temperature;
- Low speed winds;
- High content of carbon dioxide in the air;
- high negative ionizers;
- low pH value in air (acid character);
- High partial pressure of oxygen.

In 2009 was made the underground restaurant at a higher horizon (elevation 426 m), connected to the waiting room to the exit . Thus the expanding treatment halls of the two horizons underground, will become the largest asthma treatment resort in the country. In the future expansion of this underground bases provided by another horizon (at the bottom of the current at the rate 375 m), achieving a huge underground complex treatment and visitation site [4].

The obtained results show that salt mine underground speleo-and climatotherapeutic treatment is a simple and efficient, which contributes to the improvement of patients suffering from respiratory diseases, maintain and restore spiritual balance and autonomic nervous system tone.

4. THE RESULTS OF MEASUREMENTS

This year INCD INSEMEX Petrosani made at the salt mine a variety of tests, both underground and surface, respectively [1]:

- Ventilation parameters, concentrations of dust and gas in the underground workplaces and the surface.
- Dust immission at salt mine limit.
- Concentrations of gas and dust in the underground treatment base and at adjacent areas.

Measurements were performed in summer season because during this period are recorded the highest number of tourists and quarrying salt is diminished.

Determination of the gaseous components of the working atmosphere and at the treatment base were performed by gas analyzers with electrochemical sensors, calibrated MSA ORION plus type.

The concentration of respirable suspended particulates were performed gravimetrically by sampling Apex unit.

Determination of dust immission were performed with LVS head unit sampling for PM - 10.

The results of the gas at the workplace are presented in tab. no. 1.

The results of gas measurements made at work places were found with no exceeding the maximum permissible concentration for the analyzed chemical agents (CO_2, CH_4, CO, SO_2, NO_2 and Cl_2).

It also were determinate gas measurements at the treatment base, the evaluation realized in accordance with Law 104/2011 for the protection of human health and the environment. Following the evaluation were exceeding for any determined chemical agents.

In tab. no. 2 are shown the values obtained by measurements of dust immission at the limit of the salt mine. Under the legislation, there was excess dust concentration at a single point determination (at the end of the loading ramp).

Tab. no.1. Gas determinations for underground workplaces

No.	Workplaces	O_2 (%)	CO_2 (% vol.)	CH_4 (%LEL)	CO (ppm)	SO_2 (ppm)	NO_2 (ppm)	Cl_2 (ppm))
1	2	3	4	5	6	7	8	9
1	Coastal gallery concrete at the end zone, to salt (section 6-pl.nr.2)	20.9	0.03	0	6	0	0	0,1
2	Horiz.+448 Telegdy Sector	20,9	0,03	0	1	0	0	0,1
3	Horizontal. 266. Gallery access to Horiz. 246 (at 80 m from the mouth gallery section 12-pl. No.2)	20,9	0,04	0	1	0	0	0,1
4	Horizontal. 208, room 6500 (end room S-V)	20,9	0,03	0	1	0	0	0
5	Main Ramp, Horizontal. 426 (point 2-pl loading the dump. No.2)	20,9	0,04	0	1	0,2	0	0,1

Tab.no.2 Dust immission limit enclosure

Sampling point	**Environ mental factor analyzed**	**Parameter determined**	**Average concen-tration determi-ned**	**Unit**	**Limit values according to Law 104 /2011**	**Exceeding the limit value admitted**	**Obs.**
1	2	3	4	5	6	7	8
At the gate, behind store	ambiental air	Particule matter	26,0	$\mu g/m^3$	50	-	T =13-15°C Wr=45 -52%
At the end of the loading ramp		Particule matter	78,0	$\mu g/m^3$	50	28	
At 4m distance rail car wash		Particule matter	43,0	$\mu g/m^3$	50	-	
Inside strand		Particule matter	29,0	$\mu g/m^3$	50	-	

5. CONCLUSIONS

Therapeutic effect of salt mines was covered more in the last decades of the twentieth century, Thus the number of people who use this therapeutic treatment is increasing.

Considering the salt mines microclimate factors, we can say that the underground bases are adequate treatment to treat sufferers of respiratory diseases.

Evaluation of measurements was performed both under Law 104/2011 on ambient air quality and on HG. 1/2012 laying down the minimum safety and health for protection of workers.

Following measurements and assessments made by INCD INSEMEX, air quality in the workplace, and at the environment and the treatment base, it was found that no exceedings of analyzed parameters except dust immission measured at the load ramp .

Given the results of the measurements made, it is estimated that salt mining activity does not influence air quality at the treatment base.

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Scientific Reviewers:

Ph.D. eng. Scientific Researcher I INCD INSEMEX PETROŞANI GHICIOI Emilian

OPPORTUNITY FOR A GIS IN MINING

LUPU – DIMA LUCIAN*
EDELHAUSER EDUARD**

Abstract: *Geographic Information Systems, they are becoming more daily presence felt in as many areas. Development of IT tools, approaches that have allowed to be a dream until yesterday has now become commonplace. Through this technological leap, to get in the situation as in areas with a special dynamic, and can be very useful to use systems for GIS. In this way you can see and use GIS in mining. Implementing a GIS for mining, where there are already tools, can bring the reach of all departments involved and arguably the highest level management, tools for the management of production activities. It also allows management to make estimates required for the next stages in the life of mines. This paper aims to show a picture of the possibility that a GIS system, to help the smooth running of a mining.*

GENERAL CONSIDERATIONS

Currently the National Coal Company Petroşani does not use a geographic information system, in neither of possible situations where it could be used. However, the production, maintenance, all the processes carried out in the mining units and centralized at the headquarters of the National Coal Company, made that in fact to be obtained the layers needed to obtain the GIS, capable of providing the necessary information, coordinated and in conjunction with each other.

In this paper we will try to do an analysis of the availability of existing information, to compare it with the prerequisites and to show what could be achieved by combining in a useful way, in the sense of technical needs required to create a GIS. I will also try to show the utility of the estimated result in the production process.

THE CURRENT SITUATION OF THE INFORMATION NEEDED FOR THE REALISATION AND OF THE USING OF GIS IN JIU VALLEY UNITS.

The National Coal Company extracts coal by underground mining. In the extraction process, as normally, several types of processes are required and most of them contain at least one informatics or graphics component.

Because the extraction of coal, could not be done without knowing the coal basin, scientists have made maps of it, including in these the existing mining structure. One can easily realize that it actually becomes the base layer, the one on which the GIS system can be built. In the organizational structure of CNH a surveying department can be found not only at headquarters but also at each mining unit. This department is the one that creates this layer and

* *PhD Associate Professor, Manager Info 98 Computing Centre Company*
** *PhD Professor, Head of the Management Department, University of Petrosani*

provides this information to the other departments, the main beneficiary being the production department. This department elaborates operative and development plans, or investment propositions in direct coordination with the topographers.

Other departments also in direct relation are also direct beneficiaries. Electro-mechanical department is also directly interested to have the correct and completely updated information regarding the structure of mining works of any categories, this department managing the supply of energy, water and compressed air. The interplay between the needs and the interests in order to attain a common goal- the extraction of useful mineral substance- is obvious. This department develops single conductor electric diagrams and updates them, aiming to have in each moment the full access over the electromagnetic system of a mine unit and this way to be able to serve the department with it collaborates. There are the electro-mechanics those who deal with the maintenance of the equipment with which people work in a mining unit. It is important for one to know exactly the place where the equipment is, the state in which it is, the revision periods and the implication of all the above mentioned. At the same time, the ventilation department, who provides the management of the ventilation system underground, is one that must know the mining work structure , so that to be able to adapt its ventilation network at given conditions. They also elaborated a graphical diagram of ventilation network. The transport department that includes all the transport components for the useful mineral substance extracted, raw material and personnel in very close connection with the "base layer".

As can be seen, in very few words, I have shown only a part of the information that can be required for the creation of a GIS system. I haven't said anything about the personnel or about the materials needed in the production process and I have done it on purpose because I have intended for this paper to form a basis for discussion for those interested or for the management from local mining industry with regard the existence of a GIS.

THE CURRENT USE OF GIS IN MINING INTERNATIONAL LEVEL

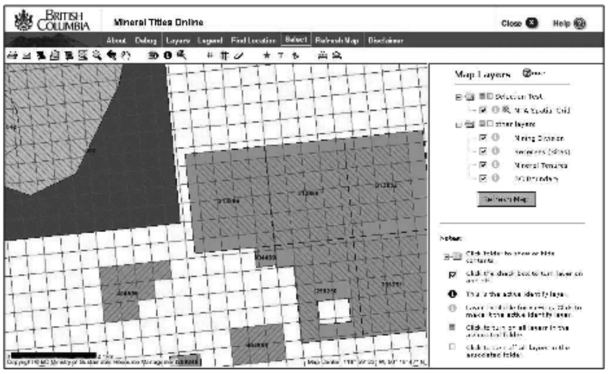
A brief and concise view over GIS at international level, shows us that GIIS are used in mining and that they bring significant benefit in current activity.

Beginning with ESRI we can find references about what the presence of GIS in mining means. It is shown[1] n August 2006 in a practice material in mining domain in a paragraph cited from Andy Lang, Pacific GeoTech Systems Ltd that "with the election of a liberal government in British Columbia in 2001, a new vision for efficient government administration was conceived". Government was reorganized, and most mapping and database management functions were grouped together.

These new ministries faced a reduction in the workforce as well as an increased need for reengineered technical applications and a leaner, enhanced administrative system. This called for a collaborative and integrated approach that forced government to utilize common infrastructure and IT services to achieve service delivery.

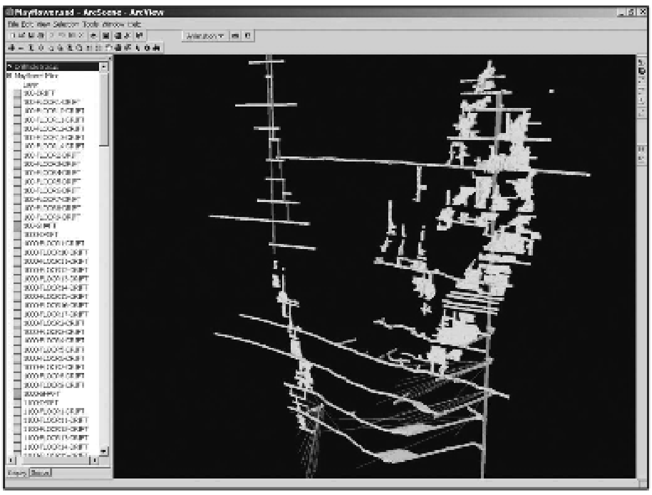
In 2002, Richard Neufeld, the minister of Energy and Mines, announced the move toward a map selection acquisition system. An industry consultation process and research began in earnest. The need to make the selection of a title and the actual title issuance a nearly instantaneous process was imperative.

1 http://www.esri.com/library/bestpractices/mining.pdf



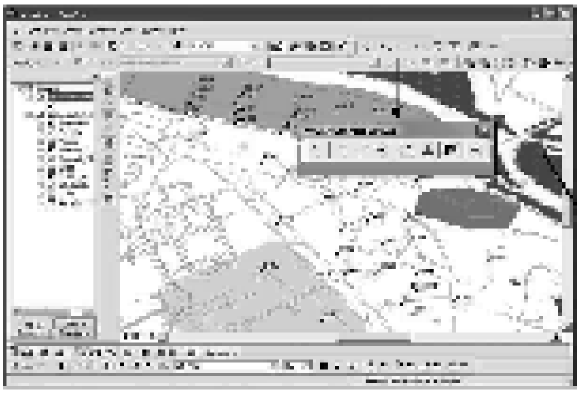
Mineral Titles Online is the first e-commerce, GIS Web-enabled system for mineral title acquisition in British Columbia.

MTO incorporates a digital map selection process and capitalizes on a host of government wide initiatives that offer Internet-based delivery of digital maps, greater access to information, and a secure administration system for electronic payments. Powered by Pacific GeoTech Systems' truePERMIT solution, MTO was developed and implemented using ArcIMS, ArcSDE, XML, Java, Java 2 Platform, Enterprise Edition (J2EE), Apache Ant, Log4J, Oracle, Apache Struts Framework Model-View-Controller-2 (MVC2), design patterns, and the government's MIRA Java Payment and Internet Mapping Framework (IMF). Here you can see a Virtual 3D Model of the Mayflower Mine. In the mid-1990s, Brimstone Mining, Inc., purchased the historic Mayflower Mine in southwest Montana. The Mayflower Mine had been a top gold producer in the 1930s. The value of gold when Brimstone bought the mine, along with the prospect of applying more advanced mining techniques, provided the stimulus to reopen the mine.



Billiton Illawarra Coal Environment Manager Gary Brassington, declared[2], referring to the advantages of a Geographic Information System (GIS) "This research project demonstrated that existing technology could be used to solve unique mining problems, and the way GIS was used and trained to predict outcomes was very new for the mining industry. The use of GIS was prompted by the fact that the process of understanding and managing coal mine subsidence impacts is largely a spatial one and that many of the factors that are critical to the assessment of subsidence impacts have a strong spatial component, such as proximity to longwalls, terrain attributes (slope, relative height, valley shape and height), and the distribution of sensitive features. The project involved industry participants in demonstrating the best methods of adapting and using GIS technology; consequently, there has already been significant take up of the technology following on from this project. Successfully managing subsidence impacts is the key to unlocking access to future coking coal resources within the Southern Coalfield and is an integral part of underground mine planning."

The number of the January / February 2009 Mining Magazine[3] presents tools and capabilities of these tools, applicable in mining domain.. Thus, RockWare is a leading distributor for the ESRI and MapInfo/Encom desktop GIS products, but also supplies its own solution, RockWorks, which focuses on the visualization of spatial data from subsurface, including drillhole logs, stratigraphic profiles and cross-sections. This is in contrast to ArcGIS products, which are focused on surface spatial-data analysis and visualization, so there is a good fit between them.



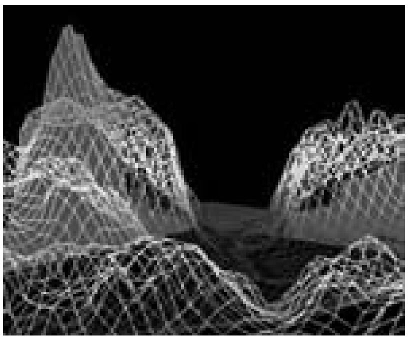
Rockware GIS Link screenshots

There is some overlap in that ArcGIS 3D Analyst provides some subsurface visualization and RockWorks has a number of tools for surface-data spatial analysis and mapping, but overall they work well in combination. RockWorks can export 3D Shapefiles for visualization in 3DAnalyst. Also, RockWare GIS Link installs as a toolbar into ArcMap, importing RockWorks borehole locations into ArcView and offers easy linking of the two programs for the generation of cross-sections, profiles, fence diagrams, logs, and elevation and isopach contours.

2 http://www.miningaustralia.com.au/news/gis-technology-improves-decision-making
3 http://www.geosoft.com/media/uploads/news/articles/GISfeature_MiningMagazine_0209.pdf

Marie Phifer also shows in TechnoMine[4] he clear advantages of using GIS in mining domain. With the advent of Geographical Information System (GIS), many mining activities (from exploration to stope development, and production to mine rehabilitation) evolved from pure luck to science. Gone were the days when operations would rely on linen and paper maps and old surveys and drawings and superimposing transparencies to create layers and composite images. GIS replaced old map-analysis processes, traditional drawing tools, and drafting and database technologies.

GIS is ideal for integrating various exploration datasets such as geophysical images, geochemistry, geologic maps, radiometric surveys, boreholes, and mineral deposits. GIS gives the explorationist tools to manage, display, and analyze data, resulting in successful, cost-effective discovery of new mineral deposits. GIS is ideal for integrating various exploration datasets such as geophysical images, geochemistry, geologic maps, radiometric surveys, boreholes, and mineral deposits. GIS gives the explorationist tools to manage, display, and analyze data, resulting in successful, cost-effective discovery of new mineral deposits.



Mapping of mineral potential using GIS is conducted to delineate areas with different probabilities of hosting certain types of mineralization. The main steps in generating mineral potential maps are:

1. establishing the exploration conceptual model
2. building a spatial database
3. spatial data analysis (extraction of evidence maps and assigning of weights); and
4. combination of evidence maps to predict mineral potential.

4 http://technology.infomine.com/reviews/GIS/welcome.asp?view=full

OPPORTUNITIES OF GIS DEVELOPMENT IN JIU VALLEY MINES

Understanding the informatics level of CNH and taking into account my experience in the mining domain, I can say that is attainable to develop a GIS for the entire mining basin of Jiu Valley. However to be true even if possible it would be feasible to start from a lower level.

The solution we could think about and that could be developed would be that of using a web GIS solution, to convert the existing and uncoordinated information from a coal unit in that required to start the project and then to build the system. At first stage, from the base layer could be added layers of power, ventilation, water, transport After a period of time, during which the system works, one can complete the layers with those suitable to be added, namely, human resources, planning of development work.

The solution to adopt a web system is motivated by the perspective, namely by the fact that an extension of the developed system at the unit level mining, to a higher level, able to cover the whole area of the Jiu Valley mining becomes much easier to be done. At the same time, this choice is very attractive for the development in the context of the energy complex existence, thus extending the informatics system for all implied units.

CONCLUSIONS

The implementation of a GIS in mines from Romania and more exactly in the Jiu Valley is perfectly possible and with great prospects in terms of benefits. The development and the implementation process would not be so difficult at least in the early stages. The infrastructure will enable the transformation of the dispatcher in a GIS operator and this system would undoubtedly bring a set of significant benefits:

- increase of the production efficiency;
- full control over the work performed underground;
- prediction of the situations that might lead to dysfunction in productive activity;
- optimal use of human resources;
- fair and effective involvement of all decision makers in the production process.

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GEOMORPHOLOGICAL HAZARDS GENERATED BY HUMAN ACTIVITIES IN THE EASTERN REGION OF PETROŞANI MOUNTAIN VALLEY

BUIA GRIGORE*
NIMARĂ CIPRIAN**

Abstract: *Mining, the main economical activity from Petroșani mountain valley, represents for the moment, a vital necessity for the comunity evolution of this territory. The products of this activity can be observed in the environment through the man made morphology, characterized by the variety of dimensions, shapes, and morphological processes. As a result of anthropogenic processes of excavation-relocation and storage, the initial territory is modified in terms of shape and function, which results in appearance of a geomorphological landscape, accompanied by associated hazards and risks. The main geomorphological hazards from the eastern region, made by the anthropogenic activities are: landslides, land collapsing, settling processes, mud flow and soil washing.*

Keywords: *hazard, human activity, geomorphological process*

1. INTRODUCTION

The present study aims to identify the areas which are subject to hazards generated by anthropogenic activities in the Eastern region of Petroşani mountain Valley, starting to the geologic and geomorphological features regarding the vulnerability of the territory to land slides and flooding.

The social and economical evolution causes the acceleration of transformation induced to environmental components and its answer is seen by highlighting the conflicting relationships in the field. As technological development and expanding the anthropogenic areas, it has produced a contradictory relationship with the natural environment.

The anthropogenic modeling differs from the natural one by: intensity, duration, complexity and design of the products, by printing an irreversible development to the territory. Thus, areas with positive landforms, subject to anthropogenic modelling, were converted from the original form to an almost flat landform and the flat areas have been uplifted with tens of meters. As a result of the feedback it generates a new spatial dimensioning and territorial architecture, finally resulting the appearance of relief inversions and critical areas.

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* *Prof. PhD. Eng. at University of Petroșani*
** *PhD. associate lecturer at University of Petroșani*

2. LOCATION OF EASTERN REGION

The limits of this region (Figure 1) have been drawn by analyzing the phisical and geographical complex, following the dominant items and the values of the subordinates.

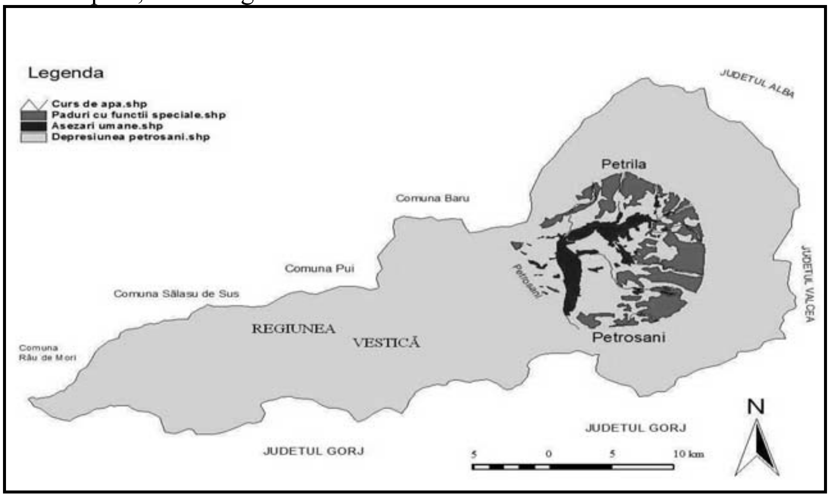

Figure 1 Location of Eastern region of Petroșani mountain valley

In the northern limit of the analyzed region, was taken into account the tectonic line. Along this, it can be observed an over-riding of crystalline over the deposits series of conglomerate of sedimentary .

At the two formations contact, can be observed the line of the split slope, which on the direction South West-North East goes under the following peaks: Boțonilor Hill (950 m), Red Stone (1192 m), North Cimpa (978 m). Between Petroşani and North of Cimpa, the northern boundary separates the basin from the Șureau Mountains, formed by crystalline and Mesozoic Domain (Jurassic limestones), which takes the form of a steep.

North-Eastern limit, on a small side, is made up by the Rascoala erosion hollow, sunk on the lower course of this river. It seems that, this hollow was a bay of the Miocene Lake, which came into contact with this corner of the basin.

South and South-East Limit separates the Eastern Region of the Petrosani Depression from Parang Mountains. To the South-East of Jiu gullet, the limit passes to the North of Măgura Hill (970 m), crosses the upper course of Salatruc Valley and the middle course of Maleia, continues to the West of Plaiul Godeanu and Cimpa Hill, reaching to Eastern Jiu Valley upstream of the Cimpa village. Aceeaşi limită morfologică, bine exprimată între zona montană şi depresiune, se observă între sectorul dintre Jieț - Cimpa; aici piemontul apare clar detaşat față de zona montană, ieşind bine în evidență ruptura de pantă dintre cele două unități morfologice.

The ***Western limit*** of the analyzed area was considered to be the section between Eastern Jiu Valley and Aninoasa River.

3. GEOLOGICAL AND GEOMORPHOLOGICAL FEATURES OF THE ANALYZED REGION

The Eastern region of the Petrosani mountain valley takes the form of a synclinal with the approximate orientation on the NNE-SSV direction.

Geological structure and modeling activity of exogenetic factors allowed the individualization of th analyzed region towards the neighbouring mountain frame.

The sedimentary lithology, differentiated from one place to another has resulted in an even greater complexity of landscape, creating structural surfaces by river erosion, in the climatic conditions which have succeeded, as shown especially in the right versant of Slătinioara Valley.

Sedimentary filling of Petroşani Basin has in its constitution, six stratigraphic units, grouped into five formations. Of the five horizons, the horizon 2 or "lower productive" is highlighted, having 25 coal layers. Layer 3 is the most important from the basin, due to its expanding in almost all the mining premises. The thickness is variable from 1-50 m, and the highest values being known in the eastern region on the premises: Petrila, Lonea, Dâlja.

Tectonical speaking, in this area, can be seen the Northern marginal fault, reverse, bisected, oblique, North-East oriented 60°-70° East, with a tilt towards the North-West of 65°-70° and a lifting of the North-West segment of 270 m at Petrila. This fault can be seen on the surface in Maleia hill from Petrila, in Dâlja hill and near the Aninoasa brooks.

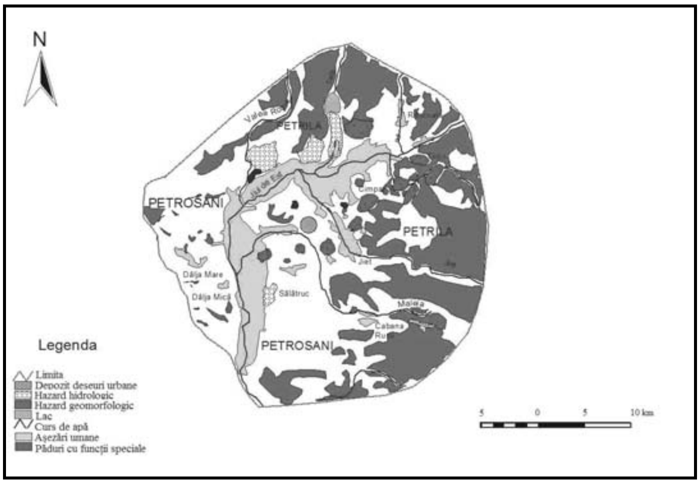

Figure 2 Hazards identification generated by human activities

Although the relief energy is about 100-150 m, though the current forms and processes are reduced by fixing the versants with tree and vegetation. Among these processes we mention: the existence of the heavy erosion, landslide, surface washing, crashes, separation of blocks, etc., especially in the area of roads, steep slopes, which have a thin adobe covering.

4. IDENTIFICATION OF GEOMORPHOLOGICAL HAZARDS FROM THE VULNERABLE AREAS

Anthropogenic actions with significant environmental impact within the Eastern region of Petroşani mountain valley are represented primarily by open pit activities or underground activities, coal extraction and dumping, urban waste dumping and forests exploitation (Figure 2).

The main anthropogenic structures created as a result of mining in the Eastern Region of the basin are represented by dumps and coal pits.

The vast majority of dumps are located on hillsides or along some valleys with or without a hydrological regime. Land levels for dumping the sterile varies between 650 m in the axis of valleys and 750 m on versants which have a slope between 6° and 35°.

In some cases, the waste dump were constructed so that they impound the valleys without a permanent water courses; in periods of rainfall, lakes are forming whose presence is particularly conducive to the stability of the heaps.

Depending on the exploiting area, the dumps material is different, being made up, most of the time by clays, marls and argillaceous sandstone.

The infiltrated water through dumps changes the physical and mechanical properties of materials and of land and can give rise to erosion phenomena. It also can produce hydrostatic pressure and ultimately can cause landslides or afterflows.

On the anthropic landscape, as a result of starting the geomorphological processes, are generated small landforms overlapped on the main anthropic landform.

In terms of mining, the anthropic landforms represent an interest for typology, method of dispersion, the complexity of the original surface modification, the rate of area coverage, potential risk, and its actions.

Thus, the affected landforms by mining activities in the eastern region of the Petroşani mountain valley and subject to anthropogenic hazard are: meadow of Eastern Jiu, Maleia Piedmont, Dâlja Piedmont, Arsului Valley and Defor Valley.

The meadow of Eastern Jiu of the immediate vicinity of Petrila town is affected by the exploitation and dumping activities of Petrila Mine.

The dumping perimeter is close to the mining premises and continues up to the Southern slope of Rusalin brook and the Northern slopes of Maleia brook.

Geomorphological processes and their products are found in dump sections in the form of: mass displacement phenomena, ravines and areas covered by water in the form of lakes or swamp areas (Photo 1-2).

The ravines present the openings of 2-3 m and depths of 1,5-2 m; the existing lakes are present in the North of the sector V, of which the one from the area of the pile P2 expands on about 100 m long and 10-30 m wide, and the one from the pile P3, P4 area, ranges of about 345 m in South-West direction and a width between 80-100 meters. Thereto are added the lakes formed in the right pillar P9 and P10 at the South of the dump V with an area of about 1800 – 3000 m^2.

Photo 1-2 Swamps in Petrila dumps area

Maleia Piedmont takes the form of an accumulation piedmont which has a direction of SE-NW.

The alluvial fan from Maleia perimeter is located at approximately 40 m North of County Road 709 F and approximately 3 km away from the Petrosani town, having a relative altitude of 744 m.

The catchment basin of the torrent is overlapped on a sterile dump, which is a result of the mining area (Photo 3). In the presence of excessive rainfall, these deposits can be mobilized on the torrent drain and in this circumstances is being a high geomorphological risk factor (Photo 4).

Soils with variable thickness make up a continuous mat throughout the region, being made up of yellow, grey, greenish-yellow adobe, occasionally black. For areas with excess of water, the algorithm based on texture and package degree forecast a high vulnerability (very vulnerable) compared to the other algorithm that forecast moderate values of vulnerability (stable/unstable).

For Eastern region of Petroşani mountain valley and for Maleia perimeter as well, the high compaction rate and a P_v/ρ_0 ratio with values between 1.2 and 0.8 represent a very high rate of instability.

Photo 3 Catchment basin

Photo 4 The inferior part of alluvial fan

Arsului Valley with the valley sector adjacent to the Lonea mining perimeter, presents active geomorphological processes which are caused by mining activity, are shown by: subsidence, collapses, crashes with negative influences on a larger area than the extraction place.

The compaction phenomena by subsidence is a physical and mechanical process which is generated by the empty spaces from rocks deposits The tension release of fragile sedimentary deposits and a new layers arrangement has as a result the deformation of the primary surface. The process appears as subsidence of different levels.

In the areas next to the small depression formed on the land surface, appear cracks and dips which forecast the extension of the compaction process.

This process is different from place to place according to the ore conditions (thickness and layers inclination, physical and mechanical properties of rocks, tectonics and hydrological situation, exploitation technologies etc.)

In this valley sector it can be seen the land deformation as a result of collapse of the mining works from the blocks III, IV, VII, layer 3 (Photo 5).

Defor Valley is a valley sector affected by the mining activities on an area of 12.56 ha, being located in the eastern side of Jieț brook.

As a result of closing the mining activities in the coal pit, in 1990, the Defor brook, which was crooked during the exploitation works, has renewed the old flow and the water has made a lake into the coal pit with the following dimensions: 250/170 m length and 20 - 25 m deep, water level +745. 92 m.

Photo 5 Subsidence in Lonea mining perimeter

Photo 6 Decline phenomena at Defor coal pit

Because the mining activities in this perimeter, the main areas with landslide risk are: Defor coal pit and Defor dump (Photo 6). The main factors of landslide are:

- cracks and ravines made up by rainfall (photo 7);
- the water from the coal pit;
- underground exploitation.

The rainfall causes a high risk potential by appearance of ravines of the coal pit's benches (Photo 7).

Photo 7 Products of erosion process, Defor coal pit

5. CONCLUSIONS

Field observations have led to the discovery of the essential aspects of activities related to coal mining activities in the Eastern region of Petrosani mountain valley:

- underground coal exploitation has special repercussions over the surface by causing falls, crashes or collapses;

- these phenomena, not only do not allow normal use of the land, for the original purpose, but severely affect the constructions in the area.

- land surface degradation is manifested depending on the thickness of the layers;

- exploiting thin layers of low inclination cause only small subsidence without any damages on the crops in the area;

- in the case of thick layers the subsidence is manifested in the form of steps with intense areas of cracks on the layers direction.

Other associated phenomena are: the advent of springs and wells, formation of permanent lakes.

The unstable land has affected almost 70 individual peasant households, and in some cases has necessitated the evacuation and demolition of residential blocks in the Petrila town.

Another specific situation for mining activities in the Eastern region of Petrosani mountain valley, in general, is represented by the existence of derelict coal pits because of unreliability.

Stopping the exploitation activities in the coal pits, the water courses have renewed the old flow, making up a few large lakes which are very deep. (Eg. Defor coal pit, Jieț West coal pit, Cimpa coal pit).

Another problem is the meteoric waters which flow on coal pit benches, resulting in erosion products as ravines. At the same time, this process is a factor of benches instability.

In the Eastern region of Petrosani mountain valley, the geomorphological environment affected by the coal exploitation activities, has an area of 136.09 ha.

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Scientific Reviewers:
Prof.Ph.D.Eng. IRIMIE Iulian

IDENTIFICATION OF ECOLOGICAL REHABILITATION POSSIBILITIES OF PETRILA MINE WASTE DUMP

FAUR G. FLORIN*[*]
LAZĂR MARIA[**]

Abstract: *E.M. Petrila waste dump consists of 5 branches arranged radially, from which branches I, II and III are already revegetated by spontaneous installed vegetation. Currently the sterile rocks are deposited on branch V which is partially united with branch IV. Because the closure of Petrila mine, in the near future, is in the question, it is required an analysis of the optimal alternatives of ecological rehabilitation and reuse of land affected by the waste dump. In this paper are analyzed on one hand natural conditions from areas adjacent to the waste dump and on the other the opportunities that can be provided by rehabilitation for local communities. Starting from the idea of transforming the Petrila mine surface installations into a museum and cultural center, several types of recovery and reuse of the waste dump are taken into account in order to identify the alternative that corresponds to this type of reuse of mining facilities.*

Keywords: *ecological rehabilitation, reuse of land, waste dump, Petrila mine*

1. INTRODUCTION

Petrila mine is one of the largest in the Jiu Valley, by the number of employees, operating depth and horizontal extension.

Hard coal extraction in the Petrila mining perimeter began in 1884, and shortly after, it became one of the most productive sectors in the Jiu Valley coal basin. The extracted coal was primary used for energetic purposes in the Paroseni thermo-energetic complex.

Today, is in question the closure of Petrila mine by 2014, and for now the closing variant involves beside securing and closing out underground works, demolition of buildings and installations from the site and grassing the perimeter. We believe that an important tradition and history for the local community is permanently and inadmissible deleted, wasting the opportunity to give a chance to use the surface installations (some buildings with special architectural value) and part of the underground installations for cultural, educational and touristic activities that would bring significant benefits to Petrila and even for the entire Jiu's Valley.

It should be added that in most European countries (and not only) in which mining was stopped, some closed mines and quarries have been transformed into museums or places of cultural and recreational activities, while having a component related to the so-called "technological archeology" in the sense of conservation for future generations of technological

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* *Assist. Prof. Ph.D. eng. at University of Petroşani*
** *Prof. Ph.D. eng. at University of Petroşani*

installations which mark different stages of industrial development. Mentioned here are only a few examples: Reiche Zeche silver mine in Freiberg, Germany, now a museum that attracts many tourists and also serves as a school for students from Bergakademie Freiberg, North Golpa lignite open pit, Germany, known today as Ferropolis, museum, industrial monument, sculpture exhibition and theme park, World Museum of Mining in Butte Montana, USA, arranged in a copper mine, where visitors have the opportunity to visit some of the underground work, and exhibitions arranged inside the buildings mining from the site.

Even if in Romania rehabilitation of mining areas by converting existing facilities and buildings intended for museums and cultural centers has not been seriously considered (few projects of this type, such as transforming Aninoasa mine in a museum mine, being abandoned) the experience in the field of the countries mentioned above shows that this type of ecological rehabilitation can be viable and able to generate profit.

In addition to the cultural purpose, the educational one is considered, in the sense that certain installations can be maintained in working order and can be used for practical training of students from specialties such as mining, environmental engineering, mine construction.

2. REHABILITATION OF PETRILA MINE AND FORMER COAL PREPARATION PLANT

The main site of Petrila mine (social buildings, personnel shaft, extraction shaft etc.) and the buildings belonging to the former coal preparation plant (photo 1) are situated in the proximity of Petrila town, on the left bank of East Jiu river, their location and easy access being another reason for which the ecological rehabilitation by turning them into cultural spaces may be considered.

Photo 1. Petrila coal preparation plant

Thus, it is considered to set up an open air museum where it can be exposed various types of equipments used over the years in coal mining activity in Jiu's Valley.

For this purpose the equipments removed from productive activity from Petrila mine and other mining units from Jiu's Valley should be restored and preserved, and as far as possible maintained in working order. Being an outdoor museum, a constant care should be taken in which means the conservation of the equipments, the most sensitive being preferably housed in buildings specially constructed for this purpose.

Only certain machines are suitable for outdoor exposure such as diesel locomotives, transport trolleys, combines, pumps, extraction machines, mechanized complexes etc. For equipments with high economic value or sensitive to environmental factors it is necessary the arrangement of exhibition halls, that can be set in the administrative building.

Also in the administrative building rooms can be arranged to host permanent exhibitions of photographs and paintings with themes related to coal mining activity in Jiu's Valley. Beside these facilities will be arranged film projection halls, conference halls that can accommodate different events and exhibition halls that can be made available to those who wish to organize such events.

To meet the requirements of the new destinations, the administrative building requires repartitioning works, interior features and exterior insulation and recovery.

The former coal preparation plant can be reassembled on the inside with specific machinery (by rehabilitating the remaining equipments and by bringing some others from the other mining units). Since one of the goals of the ecological rehabilitation is education, it is important that the technological flux to be a functional one, possibly to be used for student practice.

This building requires also rehabilitation works on the interior and exterior.

Because the technological flux will be functional, even for relatively short periods of time, it is necessary to maintain in service the waste water treatment plant currently available.

In the courtyard a small park can be arranged, with benches that will also function as a specially designed smoking area.

After the execution of modernization and re-equipment works, the mine existing cafeteria can become a restaurant, where the museum complex employees, tourists, students in training programs can dine and also at scientific or cultural events (conferences, book launches, exhibitions etc.) festive dinners can be organized.

Taking into account the dual role, museums and education it is envisaged setting up galleries from an upper horizon (250 m). In fact it is meant to be one of the attractions, the possibility of visiting galleries and the possibility of their use as school mine for training students from the University of Petrosani.

For this, the first step is to rehabilitate the personnel shaft (change the engine, rolls, transport wire and cage) and to obtain permits necessary to maintain the system functionally.

Underground transport equipment will be maintained functionally (conveyors, scraper conveyors, diesel locomotives), mechanical support systems, cutting combine, punching machines, automatic methane detection system, ventilation system, groundwater discharge system. To prevent accidents appropriate lighting should be provided, both visitors and students will wear specific protective equipment (helmets, rubber boots, overalls etc.) and visits will be made only in the presence of authorized personnel.

To ensure a higher degree of safety it is recommended that during visits the transport equipment should not be switched on, it will be turned on only during practice programs, after participants have completed an instruction program.

A very important aspect is related to galleries stability. In this regard is necessary a powerful movement tracking system, visual and specialized inspections carried out by trained and licensed persons.

3. REHABILITATION OF WASTE DUMP

The other phase of the rehabilitation program is related to the rehabilitation of waste dumps and the cable transportation installation (funicular). To identify opportunities for rehabilitation of waste dumps, and correlate them with rehabilitation option proposed for the mine site and coal preparation plant an investigation is necessary on the current status of the dump and surrounding areas.

3.1. Waste dump characterization

The terrain where the waste dump is located is an old plateau that includes the watershed area of the southern tributaries of Eastern Jiu River and northern tributaries of Maleia creek. The area designated for sterile deposits is about 86 ha.

Originally, the designated plateau for the construction of the sterile deposit had a smooth morphology, with small slopes not exceeding 10° (except for the southern area of Maleia village, where the slope exceeds 10°, but where, due to the influence of underground exploitation of coal layers, the morphology is completely altered).

The waste dump was analyzed on the branches I, II, III, located in the north eastern part of the angular station, with lengths of approx. 900 m. The maximum height of the dump is 14 m, falling from this point of view in the category of small deposits.

Branch I was built in 1933 and has a length of 800 m and covers an area of 7.5 ha. The width of the dump is 7-10 m at the top and 90-120 m at the bottom. The slope of the waste dump is steep on the southern side, 60°, and on the northern side is of 30 - 40°.

Branch II was built in 1950 and has a length of 850 m and covers an area of 8.5 ha. The width of the dump is 9-14 m at the top and 90-140 m at the bottom. The slope of the waste dump is steep on the southern side, 60°, and on the northern side is of 26 - 30°.

Branch III was built in 1958 and has a length of 900 m and covers an area of 9.5 ha. The width of the dump is 10-15 m at the top and 100-150 m at the bottom. The slope of the waste dump is steep on the southern side, 60°, and on the northern side is of 30°.

Branch R V has a length of 1560 m is the active one and is served by 11 pillars, located at variable distances from each other. The slopes height ranges between 16.5 and 19.4 m, with angles between 20 and 50°, the most common values ranging between 30-33°.

From the pedological point of view, the analyses carried out on the dumped material shows:

- soil reaction is normal (pH = 6.7-7.2);
- humus content: weak to medium;
- N content: weak to medium;
- P content: weak to medium;

These characteristics of the waste dump material can allow the development of spontaneous vegetation (already installed on branches I, II, III and partially IV).

3.2. Dump stability

Generally when we speak of the rehabilitation of a waste dump, a stability analysis is required to verify whether the minimum conditions required for the waste deposit are fulfilled and in order to proceed to the next steps.

As discussed above, the branches I, II, III and IV are in conservation and they are partially revegetated (spontaneous installed vegetation). It can be considered that due to the long time elapsed since their conversion to conservation, the material stored on these sectors suffered a settling and compaction that provides a relatively large reserve of stability. If it adds beneficial effects of reinforcement (on the stability) due to the action of vegetation and based on field observations that have not revealed the existence of sliding (superficial or in depth), these four branches can be considered stable.

The only branch that can be unstable is the branch V, in this regard being presented the results of a previous study.

Based on field observations, the following aspects may be highlighted:

- ✓ no major adverse geotechnical phenomena, such as active landslides or discharge of slopes and base terrain, are observed;
- ✓ however, it was found, the presence of gullies of different sizes, on the southern and northern slopes, formed by the action of runoff water (Photo 2);
- ✓ at the base of the dump, wetlands are formed (Photo 3), especially on the southern side, whose presence in the area may have adverse effects on stability due to water infiltration and deterioration of material strength characteristics. [3]

Photo 2. Gullies on the southern slope

Photo 3. Wetland at the base of southern slopes

Stability analyzes have revealed the following conclusions:

- ➢ the current geometry and the storage of drained rock provides the dump's stability, the safety coefficients being higher than 1.3. The smallest reserves of stability occur in areas with slope angles over 30 ° and heights over 20 m.

- circular failure surfaces are the most likely failure surfaces, which is justified both by the nature and consolidation of dump's material as well as the results of the stability analysis.
- in the presence of water the reserve of stability is reduced by approx. 33-43%, and safety coefficients indicate a limit of equilibrium in most examined sections, which can lead to sliding phenomena by the occurrence of other confounding factors (overload, freeze-thaw phenomena), or to instability phenomena. [3]

Given the current technical condition of the dump and stability analysis results, it is estimated that the dump is currently stable without major problems. Any stability problems may occur through improper modification of the dump geometry or amplification of the unfavorable factors. By respecting the designed geometry is considered that activity can continue safely and instability phenomena will not occur.

When the depositing works are stopped, for the final geometry of the waste dump, it should be carried out a new study on the stability of R V branch. If necessary measures to ensure stability should be taken so that the waste dump can retrieve its later function.

3.3. Establishing the ecological rehabilitation variant for the waste dump

Given in particular the educative role of the rehabilitation of Petrila mine and coal preparation plant premises, it is intended to maintain this role in establishing the rehabilitation variant of the waste dumps. Thus, it is possible to rehabilitate the waste dump for a double function: museum, keeping funicular line for educational and recreational purposes, by reinstalling vegetation and by constructing an infrastructure that enables recreational activities in the dump's area.

In this type of rehabilitation, the terrains that will have a recreational reuse must meet a number of criteria, including:

- absence of noise and air pollution;
- favorable climate, with long sunshine periods and no extreme temperature;
- the presence of water (seas, lakes, rivers, streams) as an essential element of the landscape;
- presence of forests and green areas (meadows, pastures);
- presence of natural forms of vegetation (fallow land with natural succession);
- presence of positive relief forms (mountains, hills);
- presence of characteristic landscape elements (rocks, gorges, valleys, typical trees). [2]

Near urban centers is appropriate to provide for recreational purposes recovered areas. It involves systematic interventions generally made simultaneously with naturalistic recovery, coupled with the installation of specific structures based on destination. The presence of such structures is modest for rest or picnic areas (alleys, basic equipment), more pronounced in areas designated for camping, powerful for sports areas (sports fields, basic structures, services etc.) and prevalent in recreation parks such as Disneyland.

Also, in this wide range of re-entering degraded land are lakes for fishing, urban and extra-urban parks and attractions like water or amusement parks.

Terrains that meet these criteria must be designed, configured and developed in a sustainable manner, to be used and to meet the needs of recreation and entertainment of the people from the region. The terrain must be connected to the means of transport and an adequate infrastructure must be constructed for recreation, most often consisting of paths for walking, cycling and horse riding trails, shelters, points of view, picnic sites and car parks. Terrains destined for recreational activities must be maintained in order to ensure their durability and quality. [2]

For all forms of recreation related to landscape, Turowski [4] systematized the recreation potential of a region and natural bases relevant to recreational activities (fig. 1).

Region's climate in the area where the dump is located is tempered with a maximum of 14.5° C in July and a minimum of -5.2° C in January. The annual relative air humidity regime has a maximum in December (93%) when air temperature is low and a minimum recorded mainly in March-April (77%).

Average rainfall decreases from west to east and increases with altitude. Rainfall has an irregular character, the total average rainfall is 822.6 mm, in Petrosani depression.

The months with the highest rainfall are June and July. Torrential rains frequency is low. There are no increased droughts. Atmospheric humidity is 4.59 l/mm^3 and wind regime presents a calm period (38%) and one with winds (62%). [1]

Analyzing the climatic conditions it can be concluded that the area under study allows both recreational and leisure activities for a period of time between May and October.

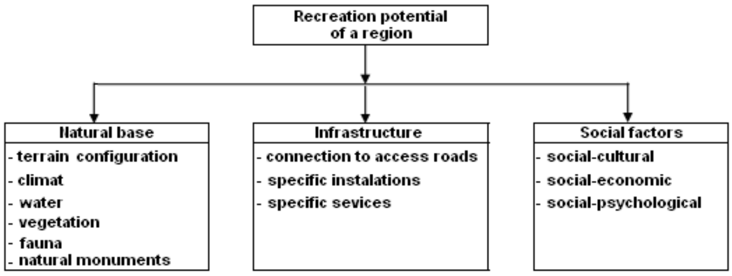


Fig. 1. Recreation potential of a region and relevant natural bases

Maleia plateau in terms of vegetation is located in the silvosteppe area with grasses and dicotyledonous (Stipa lessingiana, Festuca velesiaca, Delphidium consolida, Campanula macrostachya).

Meadows neighboring the waste dumps from Petrila, are dominated by grasses and perennials plants, developed on southern slopes, with soils having an A-C profile, on rocky substrate and sedimentary layers of loamy sands rich in gravel. These meadows have partly natural and partly anthropogenic origin.

Vegetation structure is made of species such as: Festuca valesiaca, Agropyron cristatum, Stipa capillata, Botriochloa ischaemum, Koeleria macrantha, Melica ciliata, Phleum phleoides, Medicago falcata, Astragalus Onobrychis, A. ponticus, Coronilla varia, Achillea setacea, Seseli tortuosum, Asperula cynanchica, Artemisia austriaca, Poa angustifolia, Tussilago farfara. All these species and others, less represented, form the upper layer of 40-45 cm in height.

Shorter species such as: Alyssum desertorum, A. alyssoides, Potentilla arenaria, Medicago minima, Trifolium arvense, Arenaria serpyllifolia, Scleranthus annuus, Taraxacum serotinum, form the lower layer of this meadow.

Vegetation is made up mainly of herbaceous plants and very few shrubs (hawthorn - Crataegus monogyna, blackthorn - Prunus spinosa, wild rose - Rosa canina).

Besides these species, in the adjacent areas and on the waste dump's branches are patches of forest where the dominant species is birch.

As regards the fauna, it has suffered from industrial activities carried out over time, a number of species characteristic of these areas disappeared and in others cases there is a decrease in the number considered normal for such regions.

However, in the adjacent areas of the waste dump, species such as foxes, wild boar, although they are characteristic for forests, squirrels, badgers, rabbits, hedgehogs and some rodents can be found.

Birds are represented by the woodpecker, dormouse, titmouse, blackbird, sparrows, crows and waterfowl species (especially coots).

In the lake formed between the two branches of the waste dump (III and V) species of fish (carp, perch, roach), turtles, newts, water snakes, etc. can be found.

For the area located between branches III and V, given the land configuration (a plateau fitted on the north-western and southern sides by the two branches of the waste dump) and that a considerable sized lake is present in the area, setting up a recreation area is proposed.

The lake allows fishing and small boat rides, not motorized (paddle boats, boats) and the eastern shore can be arranged as a picnic area.

For this purpose it is proposed the rehabilitation and maintenance of the cable transport installation (funicular) serving branch V. Rehabilitation and maintenance works necessary in order to keep the funicular in operation are related to rearrangement of pillars, changing transport cables, cages, reconstruction of buildings housing the angular and turning stations.

Branches I, II, III and IV can also be used for educational purposes, namely studies on the mode and speed of installation of spontaneous vegetation on waste dumps, the evolution in time of vegetation (qualitative and quantitative) soil studies and survey etc.

To facilitate the access of visitors and students entails setting an access road from the mine site to the angular station that can be traveled by public transport.

Because of the location of the waste dump, near an urban area, it may consider rehabilitation of both recreational and leisure purposes.

Also in the east, near the turning station, a car park can be arranged, the access to the car park is from the Petrila and from the access road to Jieț Gorge (besides explosives test site belonging to INCD INSEMEX). These two access roads should be properly constructed with gravel or asphalt.

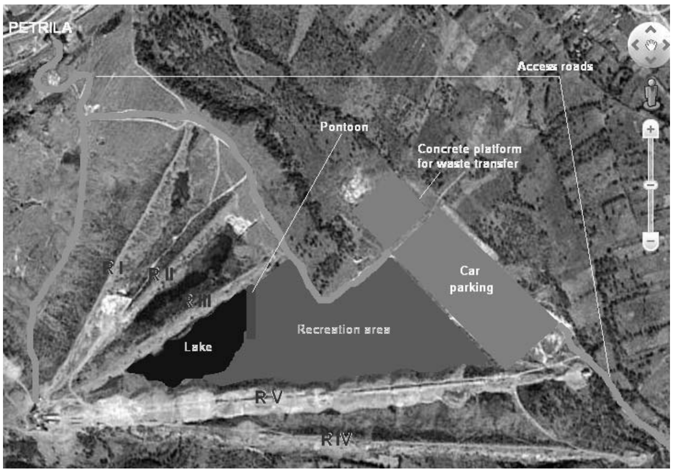
Otherwise access to the recreation area will be made on existing trails from Petrila and Petrosani.

Another objective required for a recreation and entertainment area is the collection areas for generated waste. For this purpose it is necessary, in addition to dumpsters provided with metal cover, a concrete platform where the waste can be transferred in specially designed trucks.

There is a problem with the arrangement of this area as a recreational one represented by the discomfort caused by the presence nearby of a waste landfill (between branches I and II). To remedy this inconvenience is imposed the closure of the landfill and ecological restoration.

Realization of such a project is able to generate a significant number of jobs (particularly important aspect for an area where mining ceased to be) both in the construction phase and the operational phase of the complex with multiple functions that will include former mine and coal preparation plant Petrila and Petrila waste dump.

In figure 2 is presented the final configuration of the recreation area situated between R III and R V braches of Petrila waste dump



4. CONCLUSIONS

The experience and successful transformation of closed mining objectives in museums, school mines and theme parks in countries in Western Europe and America is able to provide examples and know-how that can be successfully implemented in Romania.

Petrila mine closure offers an opportunity to rehabilitate the mine, the coal preparation plant and waste dump after western models and also the opportunity to develop a unique project currently in our country.

Surely such a project requires a more detailed study and feasibility one, but, as was shown in the paper such a complex with multiple functions is suitable to be realized in Petrila.

A complex with cultural, educational and recreational role can be regarded as a first step towards reviving the area, especially if we consider the shock that the cessation of productive activity will have on the local community.

Finally we must take into account that the realization and functioning of such a complex will generate jobs and income for those directly involved but also for the entire community of Petrila and hence the Jiu Valley, leading to a sustainable development of the region.

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3D CONCEPTUAL MODEL OF A POTENTIAL TRAP FOR CARBON DIOXIDE STORAGE. CASE STUDY: OIL STRUCTURE VALCELE, ROMANIA

PAGNEJER MIHAELA*[*]

Abstract: *The objective of this paper is the construction of a 3D model of the main characteristics (rocks and fluids) of a potential trap for carbon dioxide. This model must provide the necessary information in order to select an optimal location for carbon dioxide injection and fluid dynamics modeling. The 3D parametric model is based on data collected from drillings and seismic investigations using geostatistical methodology. The uncertainty of the 3D model was evaluated using multivariate geostatistical tools (across-semivariogram for structural analysis and cokriging) in order to reduce estimation variances in a specific situation where is a cross-correlation between a variable and one or more variables that are under sampled.*

Keywords: *trap for carbon dioxide storage, 3D stratigraphic model, indicator kriging, cokriging, hydrodynamic model*

1. INTRODUCTION

The available geological structures for carbon dioxide storage are deep structures, with depths between 800 and 2800 m. This depth interval ensures the storage security and its economic efficiency.

A 3D conceptual model of a potential trap for carbon dioxide storage should be the basis for evaluation of:

- storage ***space*** (horizontal and vertical ***expansion***, available ***volume*** for storage);
- ***parameters*** of storage space (***geomechanical***: porosity, volumetric weight, shear strength, ***hydrogeological***: permeability, hydraulic conductivity, storage coefficient);
- existing ***hydrostructure*** in the storage space or in hydraulic communication with the storage space, using like description:
  - top and bottom characteristics of storage formations (impermeable, semi-permeable or permeable);
  - type of ***aquifer boundaries*** (*Dirichlet* - where the head is prescribed, *Neumann* - with a prescribed flux, *Cauchy* - where the flux is equal to the product of a leakage coefficient and the head difference);
  - ***fluid dynamics*** in storage space (hydraulic gradient, flow directions, flow rates etc.)

[*] *Eng., Ph.D. at University of Bucharest, mihaelapag@yahoo.com*

The 3D conceptual model is used in the second stage to simulate the environmental impact of the carbon dioxide, specifically for:

- mechanical stability of the formations (cracking, deformation, instability);
- fluid dynamics (leakage, overpressure).

Romania has adopted a national target of stabilizing the carbon dioxide (CO_2) emissions by 2015 at 1989 level. The disruption of the industrial production has dramatically lowered both fuel and electricity consumption in industry, while the population consumption increased slightly. This led to an overall reduction of CO_2 emissions of about 25%. After the restructuring process of Romanian economy in the last years Gross Domestic Product (GDP) has increased, so it's expected an increasing of CO_2 emissions in the future, but to a lower level than in 1989. In this context, Romania is interested in storing carbon dioxide in ***depleted oil structure***.

Valcele area represents a Paleogene anticline covered with Badenians and Sarmatians deposits, as well as with Pliocene formations, all broadly molding and borrowing the structural shape of the Oligocene. Now, on this structure, 24 boreholes are producing oil and associated gases, 11 boreholes are producing associated gazes (from the head of the gas) and 7 boreholes are producing free gases. In the near future, this structure, after oil depletion, will be a perfect trap for carbon dioxide storage.

2. METHODOLOGY

The achievement of the ***3D conceptual model*** is based on ***topo-probabilistic methods*** of interpolation between *alphanumeric values* (stratigraphic units) and *numeric values* (rocks parameters) obtained from a network of observation points with irregular distribution. To this effect, I used two specific methods:

- *indicator kriging* for alphanumeric variables;
- *punctual kringing* (ordinary/universal) for numeric variables.

The application of kriging in the two variants was preceded by an analysis of primary data regarding the following:

- *the normality* of distribution for the numeric values;
- *the stationarity* of spatial distribution for the numeric values;
- *the anisotropy analysis* for the alphanumeric and numeric values.

It has been necessary to normalize and to eliminate regional trends of polynomial type (level two) for the stratigraphic elevation.

The ***anisotropy analysis*** was realized using as the main tool ***semivariogram*** **[6]**:

$$\gamma(\vec{d}) = \frac{1}{2 \cdot N(\vec{d})} \sum_{(i,j)\mid d_{ij}=\vec{d}}^{N(\vec{d})} (v_i - v_j)^2$$

where:

$N(\vec{d})$ – the number of values pairs situated at $\vec{d}$ distance;

$\vec{d}$ – the oriented distance between the points i and j where the v_i, v_j values are determined;

v_i, v_j – the values measured in the points i and j.

The estimation of values and errors of the ***3D conceptual model*** is based on minimizing the variance of the estimation errors synthesized in kriging systems **[7]**:

a) for ordinary kriging:

$$\begin{cases} \sum_{j=1}^{n} w_i \cdot \tilde{\gamma}_{ij} + \mu = \tilde{\gamma}_{i0}, \forall\, i = 1,2,\ldots,n \\ \sum_{i=1}^{n} w_i = 1 \end{cases}$$

b) for universal kriging

$$\begin{cases} \sum_{j=1}^{n} w_j \cdot \tilde{\gamma}_{ij} + \sum_{l=1}^{k} \mu_l \cdot f_l(p_i) = \gamma_{i_0}, i = 1,2,\ldots,n \\ \sum_{i=1}^{n} w_i \cdot f_l(p_i) = f_l(p_0), l = 1,2,\ldots,k \end{cases}$$

where:

$\tilde{\gamma}_{ij}$ - the value of variogram for the pair of values v_i, v_j situated at distance d_{ij};

w_i - the weight of v_i values;

μ - Lagrange's parameter;

f_l - the regional trend function in the p_i observation points; ($i = 1,2,...,n$);

n - the number of observation points.

The ***estimated value*** is calculated as follows **[8]**:

$$v_{p_0}^{*} = \sum_{i=1}^{n} w_i \cdot v_i$$

The minimum variance of error estimation has two calculations:

a) for ordinary kriging:

$$\tilde{\sigma}_R^2 = \sum_{i=1}^{n} w_i \cdot \tilde{\gamma}_{i0} + \mu$$

b) for universal kriging:

$$\tilde{\sigma}_R^2 = \sum_{i=1}^{n} w_i \cdot \tilde{\gamma}_{i0} + \sum_{j=1}^{k} \mu_j \cdot f_j(p_0)$$

The ***error estimation*** of the ***3D conceptual model*** was realized for an *assumed risk* $\alpha = 5\%$ using the relationship: $\varepsilon(x_0, y_0, \alpha) = \pm 2 \cdot \tilde{\sigma}_R$

The ***kriging*** allows optimizing the monitoring network by the *fictive point method* **[6]** that uses the variogram model of the structure (model that incorporates the anisotropy parameters).

The calculation of ***anisotropy parameters*** (orientation for ellipsoid of anisotropy θ and anisotropy ratio $\eta = \frac{R}{r}$) is done using the *surface variogram* for each component of the ***3D conceptual model*** (stratigraphic surfaces elevation, permeability, hydraulic conductivity etc).

On the basis of the maximum permissible errors, the monitoring network is improved by filling it with additional investigation points **[3]**.

The succession stages of building the model assumes:

1) separation of oil and gas productive complexes, cap-rocks intervals and aquifers;
2) identification of major structural faults;
3) evaluation of parameters distribution in each lithological units;
4) study of uncertainty in two 3D models (kriging and simulation models).

The main steps of preliminary data processing were: statistical analysis of data, de-clustering for representative distribution and trend analysis. A special attention was give to the study of the anisotropy made with surface variograms. Directional variograms had a convenient tool to calculate a series of orthogonal directions and finding the major and minor directions of anisotropy.

3. DATA DESCRIBING THE VALCELE SITE, ROMANIA

In Valcele structure there are **241 boreholes** drilled (the deepest borehole F5205 has a depth of 4656 m) and now, on this structure:

- 24 boreholes are producing oil and associated gases;
- 11 boreholes are producing associated gazes (from the head of the gas);
- 7 boreholes are producing free gases.

In 2009 was realized a 2D seismic investigation and in 2010 was developed a new 3D seismic model of Valcele structure. There are more details about the extension and the position of the faults and the distribution of the oil and gas productive complexes.

A comprehensive summary of the data sets is given in **table 2.1**.

All components of the ***3D conceptual model*** were achieved with **70** selected boreholes from the 241 available boreholes. In each borehole were selected two series for CO_2 storage from ***OLIGOCENE*** and ***BADENIAN*** formations:

- ***OLIGOCENE***, the most important oil complex (500 – 1500 m):
  - **LOWER SERIES** (sandstones, sands, conglomerates, with marlstone intercalations: Ol1b7, Ol1b6, …,**OLb1-b2**);
  - **SUPERIOR SERIES** (argillaceous series with marlstone, clay with few sandstone intercalations: Ol1a1, …, Ol1a10.
- ***BADENIAN*** (HELVETIAN-old name; 20-1000 m):
  - **LOWER SERIES** (argillaceous series with thin intercalations of fine sand and calcareous sandstones: He III undivided, He III 4,…, He III 1)
  - **MEDIUM SERIES** (sand and sandstones: He II): **He II A (Ba5)**, He II B2, He II B1;
  - **SUPERIOR SERIES** (sandy series with marl and argillaceous intercalations: He I): He I b1+b2, He I c+d.

Table 2.1. **The information source for the primary data used**

Data set	Data type	Format	Host and maintenance	Contact	Availability
Surface data					
Geological maps	Analogue and digital	Arc GIS shape tiff, jpg. pdf	IGR	www.igr.ro	Public
Geographic data	Analogue and digital	Arc GIS shape Tiff, jpg. pdf	OMV-PETROM-PITESTI	-	Public
Seismic and structural data and 3D information					
Site dimensions, thicknesses and area extent	Template Table	Excel and word	OMV-PETROM-PITESTI		Public, presented in D023
3D surfaces for identified layers	Digital	grid formats	OMV-PETROM-PITESTI	-	Restricted to MUSTANG project
Well reports and wire-line logs					
Wire-line logs from the older 4 wells	Analogue	Tiff, jpg	OMV-PETROM-PITESTI		Public

Well reports from the older 4 wells	Digital	LAS	OMV-PETROM-PITESTI		-“-
Physical and chemical parameters					
Cap rock properties, Numerical data	Template Tables	Excel and Ascii	OMV-PETROM-PITESTI		Public, presented in D023
Reservoir properties, 3D Numerical model	Template Rkw database	Excel and Ascii Rkw files Jpg, tiff	OMV-PETROM-PITESTI UB-FGG		Public, presented in D023
Geological setting – key publications					
3D parametric model of the potential trap for carbon dioxide storage. Case study: oil Valcele , Romania, 2011, EGU, Vienna	Analogue	pdf	UB, Faculty of Geology and Geophysics.	D.Scradeanu daniel.scradeanu@g.unibuc.ro M.Pagnejer mihaelapag@yahoo.com	Public presented in DO23

3. STRATIGRAFIC MODEL

Valcele oil structure belongs to Getic Depression. The stratigraphic model of the oil structure is realized from ten stratigraphic units separated in each borehole from the 241 boreholes available (the deepest borehole F5205, with a depth of 4656 m).

3.1. Geology of Valcele structure

Valcele oil structure has ***29 oil complexes,*** associated gases and free gases in Oligocene, Badenian (Helvetian) and Sarmatian. For my reasons, only Oligocene and Badenian (Helvetian) deposits are important for the storage of CO_2.

Fig. 1. Valcele site

OLIGOCENE is the most important oil complex (thickness between 500 m in the east part of the structure and 1500 m in the west part.

- **LOWER SERIES**:
  - sandstones, sands, conglomerates, with marlstone intercalations;
  - seven complexes (from bottom to top): O1b7, O1b6, …,O1b1.
- **SUPERIOR SERIES:**
  - argillaceous series with marlstone, clay with few sandstone intercalations;
  - ten complexes (from bottom to top): O1a1, …, O1a10.

BADENIAN (HELVETIAN-old name) has thickness between 1000 m, in the central part of the structure, and 20-80 m in the west part of the Valcele structure.

- **LOWER SERIES (He III):**
    - argillaceous series with thin intercalations of fine sand and calcareous sandstones;
    - five complexes (from bottom to top):He III undivided, He III 4,…, He III 1.
- **MEDIUM SERIES (He II)**:
    - sandy series of 350-400 m, represented by sand and sandstones;
    - three complexes (from bottom to top).
    - **He II A** (150-200 m): gray-yellow sand with thin intercalation of marlstone;
    - **He II B2** (40-50 m) two sandy layers of 20-30 m separated by a marlstone layer of 20 meters;
    - **He II B1** (50-60 m) argillaceous series with thin intercalation of sand and marlstone;
    - on the top of He II is a cap rock of 40-60 m of marl deposits.
- **SUPERIOR SERIES (He I)**
    - Sandy series with marl and argillaceous intercalations;
    - Two complexes:
        - He I b1+b2, 80-100 m of sand with grey and green marlstones;
        - He I c+d, sandy series with marlstone intercalations.

3.2. Tectonic of Valcele site

The 3D seismic investigations made in 2010 have brought details for West-East anticline Valcele structure:

- The upper Oligocene is a erosion surface filled with Neogene deposits;
- There are two distinct systems of faults for the **Neogene** deposits and **Oligocene** deposits
    - In the **Oligocene** deposits there are only three longitudinal faults (Great Faults, F1 and F2) with slopes between 5 and 45 degrees and tightness controlled by the thickness of marlstones and clay intercalation:
        - Great Fault is in general a leaky faults (GF is waterproof for some Neogene complexes;
        - F1 and F2 are entirely waterproof for all Oligocene deposits.
    - In the new model of ***Neogene*** deposits there are two longitudinal faults - F1 and GF and two transverse faults - f1 and f2.

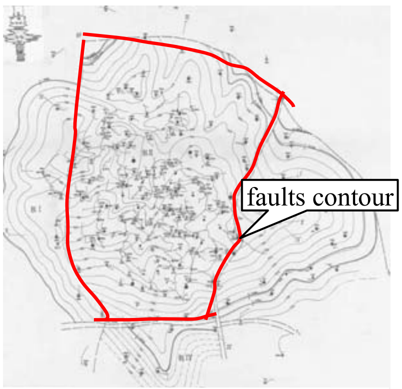


Fig. 2. He I b1+ b2 complexe

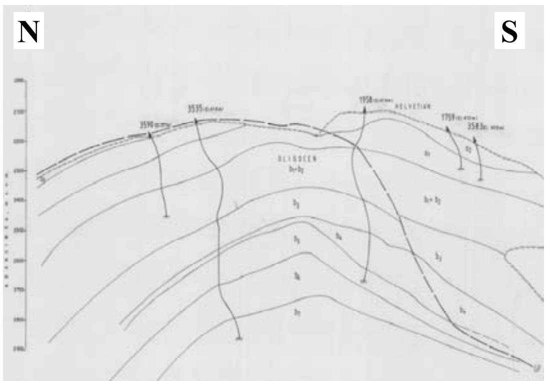


Fig. 3. Oligocene longitudinal section

3.3. The model of Valcele structure

The stratigrafic model in the frame of the main fault of the structure, with the ten separated stratigraphic units (**Fig. 4.**), indicates a good continuity for the Badenian (Helvetian_A; b5) units and Oligocene (b1-b2) units. These two complexes were chosen to simulate the injection of carbon dioxide in Valcele structure.

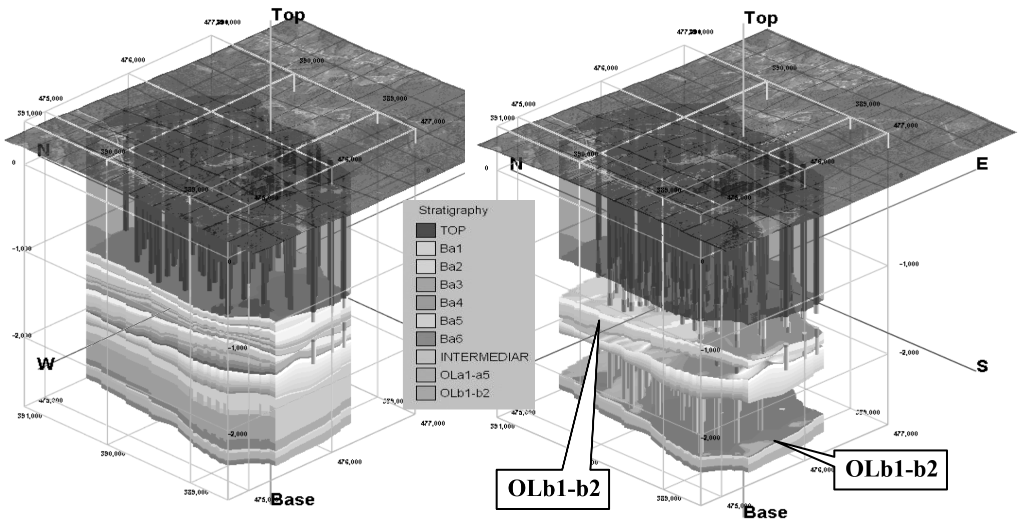


Fig. 4. The 3D stratigraphic model

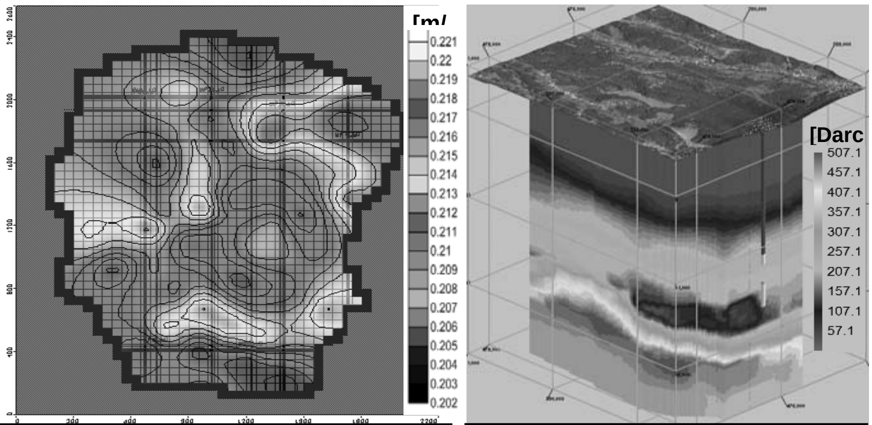
4. PARAMETRIC MODEL

The analysis of distribution and uncertainty for multilayered parametric model of Valcele structure is based on multivariate geostatistical tools: cross-variogram, cokriging **[4]**.

The results of the ***anisotropy*** study (made with surface variograms and cross-variogram) indicate small differences (2-5%) between structural direction of anisotropy and parametrical anisotropy **[5]**.

Applied to **four parameters** (porosity, permeability, hydraulic conductivity, storage coefficient) and for the **two selected series** (**Ba5** and **OLb1-b2**) cokriging reduces estimation errors by 10-15 % for under sampled parameters.

The parametric models of conductivity (**Fig. 5a**) and permeability (**Fig. 5b**) are presented as examples. All parametric models are used to achieve the finite difference hydrodynamic model for the two confined aquifers of Valcele structure.



a) 2D Conductivity model

b) 3D Permeability model

Fig. 5. The 3D parametric model

5. HYDRODINAMIC MODEL

The **hydrodynamic** model of Valcele structure (**Fig. 6.**) is a finite difference numerical model based on **3D stratigraphic model** with **two** selected stratigraphic units and a **3D multiparameter model [9].**

The hydrodynamic model is delimited by impermeable faults and the flow is steady state and conservative. The distribution of pressure is conditioned by the morphology of aquifer boundary and by the parametric models (permeability and conductivity model).

An **injection test** was simulated on hydrodynamic model of Valcele structure using 6 boreholes and a constant flow of 120 mc/day for each borehole and for each interval. The simulation was performed for a period of ten years (**Fig. 7.**).

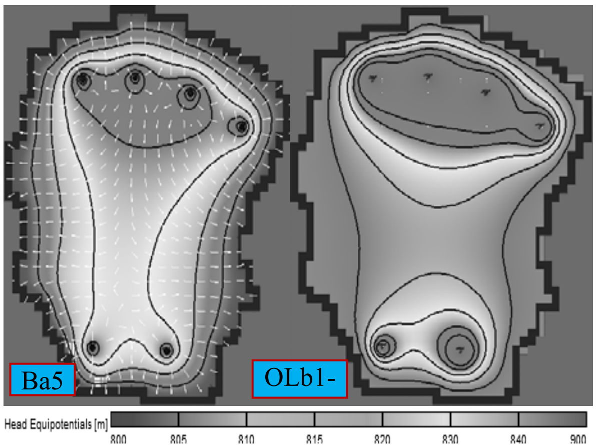


Fig. 6. The hydrodinamic model

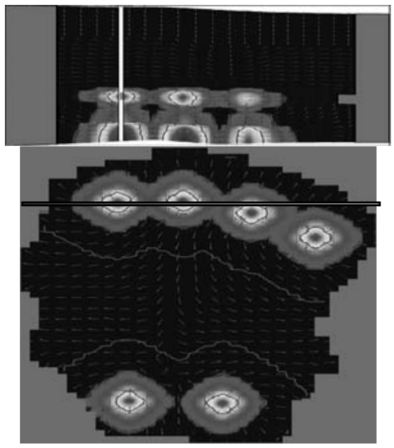


Fig. 7. Injection test

In addition to the injection test was performed a **transport model** using a longitudinal dispersivity between 100 and 1000 m **[1, 2]**. The main aim of the transport model is to evaluate the affected area by injection.

The ***preliminary*** hydrodynamic and transport models will be fitted in the future stage on the data that will be obtained in an additional experimental injection test.

CONCLUSIONS

The 3D conceptual model achieved using topo-probabilistic methodology for Valcele site allows optimization of carbon dioxide storage by:

1. minimisation of uncertainty (by cokriging: 10-15 %) of the parametric models transferred to the hydrodynamic model;
2. additional filtering of pressures distribution generated by injection test using the hydrodynamic numerical model sensitivity;
3. fitting of the two models using a complete set of experimental data (regarding additional chemical and mechanical aspects) will control the final uncertainty of the evaluation of the carbon dioxide storage.

The optimization results can be maximized based on successive (water/carbon dioxide) injection test conducted in oil exploitation drillings.

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Prof.Ph.D.Eng. Mircea GEORGESCU

RESEARCHES CONCERNING THE ECOEFFICIENCY OF THE LIQUEFIED LAYER COMBUSTION OF VARIOUS SOLID FUELS

CIOLEA DANIELA IONELA*
DUNCA EMILIA**

Abstract: *The current paper goals to optimize, from environment protection point of view, the combustion in liquefied layer of various solid fuels and the achievement of a CLLC (Circulatory Liquefied Layer Combustion) laboratory installation with pyro-gaseous possibilities, which lead to a 99.9% - 100% proficiency pollutant retain.*

Key words: *reduction, emissions, atmospheric pollutants, combustion, coal*

1. INTRODUCTION

The achievement of the CLLC laboratory installation will allow the reproduction of the combustion and pyro-gasification processes of the solid fuels with monitoring and controlling of all process parameters, gas, liquid and solid effluents, as well as the determination of the energetic efficiency. The originality and complexity of the proposed them consists in the methods and techniques for coil (Jiu Valley pit coal and Olteny lignite), biomass, cellulosic wastes and other solid fuels' co-incineration in order to optimize their combustion from an environment protection point of view. In order to solve this theme, the following steps will be taken: the documentation concerning the current stage of the combustion processes of different solid fuels; the theoretical establishment of chimney emissions and of atmospheric noxes' dispersion; the modeling of the atmospheric pollutants' dispersion; the influence of the emissions from the thermal plants from Romania on the environmental factors' quality analysis; the projection, execution, testing and implementing of a laboratory CLLC installation for performing production and use of the solid fuels' energy systems; the achievement of experimental models; the processing of the experimental data; the proposal of solution and the selection of the most efficient from the solid fuels combustion's eco-optimization point of view.

2. THE IMPORTANCE AND RELEVANCE OF SCIENTIFIC

Over time have done research to reduce the harm resulting from the combustion of solid, so there are different processes and a variety of equipment and techniques.

All environmental quality control tasks are designed to eliminate the consequences of it. It is not fully satisfactory measure certain pollutants. It is necessary to establish a correlation

* *Assist.Prof.PhD.Eng. at University of Petrosani, Romania*
** *Lecture PhD.Eng. at University of Petrosani, Romania*

between the results of these measurements and the effects observed. On the other hand, is well known correlation between the concentration of atmospheric pollutant at ground level emission flows and propagation conditions, which signifies the existence of a relationship type because - result in the source, spread and effects of pollution. [4]

According to several estimates, global energy demand will grow by about 1.8% per year until 2030. This leads to the need to increase energy production in direct proportion to demand. If we correlate this with the estimate according to which nuclear energy, hydroelectric and other clean energy will remain at a weight of more than 25% of energy production will result in two directions. [2]

The first direction is by increasing consumption of conventional fuel, which will mean a considerable increase of noxious emissions into the environment. In conditions which are not working to research and development of clean fossil fuel technologies that direction is not acceptable interms of environment.

The second direction is the first step to be made in the fight against pollutants, namely research, development and final use of facilities with high efficiency to produce electricity and heat, increasing efficiency and translated by reducing fuel consumption for the same amount of energy produced thus harm. In this direction should we respect and ascending cost of natural gas and liquid fuel, in proportion to their depletion, which surely will guide us to use mainly solid fossil fuels (coal).

Power generation generally uses a variety of combustion technologies. For solid fuel combustion, pulverized combustion, fluidized bed combustion and burning on a grill are considered best available techniques (BAT). The technologies used in thermal power plants involve combustion or gasification of solid fuels to produce electricity. [5]

After firing, the vast majority of solid fuels emit sulfur oxides (SO_2 and SO_3), nitrogen oxides (NO and NO_2), carbon monoxide and carbon dioxide (CO and CO_2), dust (fly ash, particles of unburned coal, slag and earth) and in smaller quantities: tars, hydrocarbons, soot, sulfates, organic acids etc.

There are different methods and a variety of equipment and techniques that can be used to reduce emissions from combustion plants for energy production. Reducing emissions from large combustion plants can be realized in various ways, but generally the measures envisaged can be divided into two categories, namely primary measures (integrated measures to reduce emissions at source or during combustion, which include measures supply, fuel and combustion modification) and secondary measures (measures the path of combustion gases (fine boiler), such as those that regulate air emissions, water and soil).

3. TWO ENVIRONMENTAL DIRECTIVES OF THE EU

Two environmental directives of the European Union on large combustion plants in the electricity and heat production:

1. Large Combustion Plant Directive **(LCP, 2001/80/EC);**

2. Directive on the prevention and control of industrial pollution (**IPPC, 96/61/EC)** and for air quality, was developed Directive (96/62/EC).

Transposition of Directive 2001/80/EC (LCP) in the national legislation aims to achieve by Romania of environmental performance in line with the European Union to improve air quality and the objectives are:

a) fulfillment of the accession to international conventions in the field of air quality protection;

b) combating phenomena of acidification, eutrophication and tropospheric ozone formation;

c) reducing concentrations and critical levels of major pollutants responsible for the phenomenon of acidification (sulfur oxides and nitrogen) concentrations and the levels necessary to protect human health and the environment. [10]

Directive 96/61/EC (IPPC) is one that focuses on industry and deals with both air and water and waste. Its transposition into Romanian legislation aims are:

a) prevention and control of pollution caused by emissions into air, water and soil, emissions from industrial activities;

b) implement measures to prevent¬ or reduce emissions into the atmosphere, water and soil, including waste management measures for industrial activities;

c) achieve a high level of protection of the environment as a whole. [11]

d) establish the legal framework for issuing agreements / integrated environmental permits for industrial activities referred to in Appendix. 1 of Directive 96/61/EC, in compliance with legislation in force and the obligations of international conventions in this area, to which Romania is party.

4. THE MAIN COMBUSTION TECHNOLOGIES CURRENTLY

The main combustion technologies currently in place are:

1. Burning-layer fluid

Combustion in the fluidized layer theory was developed in 1922 by researchers Winkler (Germany) and Cidell (U.S.), but was applied to burning coal in steam generators outbreaks only in the last 30 years. In principle, the combustion process in fluidized layer consists of burning solid fuel particles in suspension. In an oxidizing current situation different second limit, determined by the value of instilling speed air: combustion in stationary or dense layer fluid (ASF); combustion circulating fluidized layer (ASFC). Circulating fluidized combustion layer is a relatively new technology; the first steam generator based on this technology was developed in 1979, when the company changed Ahlstrom a 15 MWt steam generator running on fuel oil to switch to burning in a fluid layer circulating peat and wood waste.

The many advantages that we offer have led to a rapid development, currently being operated by steam generators contracted to 250 MWe and 460 MWe. [1]

In Romania, this technology was applied to a first ASFC CAF (hot water boiler) of 120 MW, whose design began in 1990 [8], and the world at that time the situation looks like in Table 1.

Table 1. Boilers equipped with global outbreaks ASFC

Country	Energy Groups	Thermal Power
	Number	MW_t
SUA	120	15 188
Germania	32	4 000
Finlanda	16	1 265
Japonia	10	1 136
Corea	8	1 030
Suedia	20	927
Franta	6	561
Canada	2	479

Austria	4	307
URSS	2	265
India	3	265
Italia	1	187
Polonia	3	310
Anglia	3	178
Spania	1	140
Other countries	9	432
Total	240	26 647

A fluid layer is a system in which a gas, distributed through a distribution grid (grilled or insufflations' jets), is expelled, from bottom to top, a layer of solid particles, so particles floating in the stream of gas and is in a constant turmoil. [6]

The behavior of the two-phase medium, in which solid particles can move with respect to each other, is comparable to that of a boiling liquid, hence the name fluid layer.

Dependence of pressure drop in the fuel layer and lower transfusion speed up air was, presents four distinct areas (I, II, III, IV) characterizing as many types of outbreaks for burning coal. [8]

Basically, the process consists of burning coal particles suspended in a stream oxidant differs two limited circumstances, determined by the value of instilling air speed:

- Combustion in stationary fluid layer) and
- Combustion in circulating fluidized layer).

1.1. Combustion in stationary fluid layer (ASF)

The minimum fluidization velocity field and the layers with large particle size (as happens in case of crushed coal) segregation occurs, characterized by the fact that fine particles gather at the top of the layer, and large based.

If streamlining process takes place in an outbreak that is inserted properly and air solid fuel and ignition and combustion conditions are met, combustion takes place intensively, and the process is known as combustion in stationary fluid layer (ASF).

Abroad have been made and boilers equipped with combustion in pressurized fluidized layer (ASFP), which have a much more efficient than boilers with ASF, but the execution and operating conditions are more severe, why have not proliferated. [8]

Inspired air in the outbreak, whether it's primary air, secondary or tertiary, you should be reheated to temperatures around (200 - 250) ^{0}C. The ratio of total air flow introduced in the outbreak and the minimum necessary for combustion perfect dedicated called the coefficient of excess air and is denoted by. For combustion in the fluidized layer is recommended = 1,1-1,3.

1.2. Combustion circulating fluidized layer (ASFC)

Circulating fluidized combustion layer - the area denoted by III - in Figure 1 is characterized by the fact that, on leaving the layer, agent streamline train a quantity of increasingly large particles, which grows as you approach the speed wa Want speed drive value when solid particles are involved in their entirety. Very little pressure drop increases with the increasing speed of fluidization agent WFC, because turbulent flow and solid particle involvement. [8]

Recently, this technique is used increasingly more in technology enhanced combustion of solid fuels, due to facilities it offers in comparison with the burning of such fuels. It is primarily the burning rate increase, to enhance the exchange of heat convection but downtime and increase the fuel particles in the outbreak. [5]

Burning coal particles in large part on the time elapsed since that is in focus, but leave it still have a high content of unburned which first reduces the energy efficiency of the process, and on the other hand, the source importance of industrial pollution of the boiler heating surface, the erosion of their mechanical and last but not least, a source of environment pollution. Therefore, the exit of the outbreak, the flue gas with the particles involved (ash, coke flying) enters into a cyclone where the separation takes place largely gaseous phase solid phase. [8]

ASFC technology has increased worldwide over the past 20 years. Remarkable tendency to build boilers fluid combustion systems, which is an evolved version of boilers fired in circulating fluidized layer. [9]

Events within the fluid layer

Inside the fluid layer, which reaches heights of (3-4) m, coal particles are drawn into a chaotic motion, colliding with each other, and heat exchanger pipe or adjacent walls of the outbreak. Following these clashes shook the ash layer formed behind the combustion of the particle surface, thus strands oxygen access to surface reaction, which favors the growth of the particulate burning rate and convection coefficient increasing importance of gas to wall pipe heat exchanger.

When using a higher quality coal, so the higher calorific value, the danger of temperature increase in the fluid layer above the ash softening temperature, which leads to the appearance of ash danger, so the disappearance of the flow of coal and the conversion layer in the fixed layer with all its adverse consequences of this situation. Temperature control fluid layer can not be ensured only by providing a gas-water heat exchanger immersed, but need to resort to achieve combustion unstoichiometric while ash recirculation. His appearance came as the installation of solid fuel combustion in circulating fluidized layer. [6]

Coal particles with dimensions up to (15 ÷ 20) mm, are placed in focus with the powder of calcium carbonate and are placed under the primary air, instilled through grate bars or special nozzles. Flow takes place layer of fresh coal particles mix with other gases and particles in the fluid layer ignite and burn in suspension. In the first stage combustion takes place in unstoichiometric regime as the primary air is brought about (70 ÷ 90) % of that required for combustion. The remaining air is instilling in the form of secondary air chamber in the center of the outbreak through holes arranged at several levels.

1.3. The best of technology ASFC

1. Ability to use different qualities of solid, from the lower to the senior, the mixtelor coal or waste or other waste. Ignition and combustion stability is high, and unburned losses do not exceed (1 ÷ 2) %.

2. Pollutant emissions are minimal because of the possibility of desulphurization forward (up to about 80 ÷ 85%) of the combustion gases by absorbing substances ($CaCO_3$, CaO, $CaCO_3$, $MgCO_3$) and heat-blocking mechanism of production of nitrogen oxides due to temperature relatively low in the outbreak (below 900 ^{0}C);

3. Preparation of fuel solid is brief and is accomplished simply by crushing it. Therefore is an important energy saving fuel grinding sound, but lower operating and investment costs related to non-existent and coal. [4]

1.4. Combustion air fluidized layer (AFBC)

The technology consists in maintaining the particles of coal with a grain of order millimeters or tens of millimeters, in an upward airflow.

Due to density modification by burning, coal bed will stay put until the surface of the bed and as we will burn down to the bottom, the movement of particles creates the visual sensation of boiling of a fluid, where the technology has also received the name of bed fluidized boiler. Airflow velocity is ensuring a balance between weight and strength Archimedes coal particles created by it.

Efficiencies are about 40%, but the big advantage comes from the possibility of reducing emissions of sulfur oxides and nitrogen due to temperature of approx. 900 °C (compared to 1800 K to burn pulverized) in the outbreak, thus reducing thermal NO_x formation is favorable reaction of dolomite and / calcium carbonate and sulfur dioxide.

In 1200 the world are central circulating fluidized bed combustion, with a total thermal power of 65 GWT, distributed as follows: Asia 52% North America 26%, other 1%.

Leading companies in this market are detached Foster Wheeler / Ahlstrom (about 180 units in service) and Lurgi Lentjes Babok (about 90 units), other companies are Ahlstrom Power, Kvaerner and Babcock & Wilcox. [14]

Currently there is a noticeable tendency to develop this technology, reaching powers as high as 2020 can be produced 150 GW because of the positive evolution of the market.

AFBC technologies are:

- adaptable to both new and to existing installations;
- suitable for refurbishment (replacement of existing boiler with an AFBC);
- suitable for boiler conversion (replacement of a portion of the boiler with an AFBC) in various applications
- can burn low quality coal (e.g. lignite of low calorific value, waste left over from washing coal, petroleum coke and other waste materials. [1]

Boilers / AFBC type systems have these processes and features:

- lime is injected into the furnace to capture sulfur and removing the dried product;
- the gas temperature in the boiler is between 820 and 840 °C, which affect the whole construction and arrangement of the boiler heating surfaces.

AFBC technology can also be combined with other technologies to meet the specific needs of each site. Special interest it presents the combination of cleaning coal, pulverized coal and AFBC technologies. Physical coal cleaning may be used to provide clean coal for power / pulverized coal plants, while waste from cleaning coal and raw coal can be burned in the AFBC. The ability to burn waste coal AFBC offers significant flexibility in terms of how the carbon washing plants. They can be operated so as to produce a coal with less sulfur, without regard to generating high value waste heat. [12]

AFBC technology has proven effective and is available commercially for power modules larger than 300 MW. In North America are in operation more than 600 installations (installed capacity of 30 GW), works similar capacity in Europe, and China has more than 2,000 small bubbling AFBC boilers in operation.

Several projects are planned or are currently being implemented in the field from 250 to 350 MW. Development Corporation of Japan has made power plants convert a 350 MW PC boiler Takehara, bubbling AFBC technology. EDF of France to build a 250 MW AFBC circulation (Lurgi technology). In general, projects over 300 MW have higher technological risk and therefore should thoroughly analyze the data for each project. [5]

As in developed countries demands the removal of SO_2 pollutants are high (usually over 90% of removed), most recent projects with the option AFBC circulation.

1.5. Burning layer fluid pressure (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 the 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 (mostly removal of pollutants SO_2, NO_x emissions low, the capability to burn fuel of low quality and flexibility in choosing fuels) and has in addition:

- compact and modular design. Re-engineering is easier than for AFBC existing power plants due to reduced space requirements,
- potential to achieve a greater output of power (over 45%) than conventional power plant with pulverized coal or AFBC (efficiency 36.5%) and
- lower capital costs than IGCC technology for coal or gas sprayed from scruberele wet.

Proven performance of PFBC technology are:

- more than 90% SO_2 is removed with sulphide of calcium (Ca / S) - ratio of 1.5 to 3.0 molecular;
- NO_x emissions from 100 to 200 ppm;
- NO_x emissions can be minimized by using technologies of selective non-catalytic reduction or SCR;
- efficient 40 or 42% in combined cycle regulation. [14]

Fields which must be given special attention if you consider the PFBC technology:

- hot gas technology final degassing / cleaning, especially performance and reliability demonstrated that cleaning technology;
- coal and sorbent preparation and feed systems;
- effects of a water heater / boiler PFBC gas contaminants on gas turbine performance, reliability and expected lifetime.

The latest and most advanced plants with circulating fluidized bed combustion pressure built by the world leader in this field Alstom Power, are these [5]:

- Quantity Turkey. The power plant has an installed power of 2 x 160 MWe and running on lignite. Each steam generator has 4 cyclones.
- Red Hills (USA). It is a plant with an output of 500 MWe (2 x 250 MWe), commissioned in 2002, and the burning lignite.
- Guyana (Puerto Rico Power Authority). Commissioned in 2002, with a power of 2 x 250 MWe. Due to stringent emission limits, the plant was equipped with denoxis and desulphurization.

2. Burning spray

Coal powder by spraying in outbreak steam generators, is the most used technology currently burning in the world and certainly the most used combustion technology in Romania. [9]

For plants in which coal is by spraying, the most used to increase overall efficiency in rising the gross average temperature higher thermodynamic cycle, specifically by increasing the live steam parameters. Today is November fired steam generators built by Benson type, with crossed forced only, which ensures the overcritical parameters of live steam turbines to be used in specially adapted for legally constructive raise these parameters. It is known that the only limitation in the growth parameters of live steam is a material. For the same operating mode of the plant and the same weather conditions, a 10% increase in efficiency means a reduction in fuel consumption by, say, 8% (because the increased live steam parameters increase the thermal efficiency of the thermodynamic cycle, but consumption own technology in general remain the same), which means a reduction in CO_2 emissions by approximately 80 000 t CO_2/an (about 8%).

By a simple calculation we can see that the same technology for combustion, for the same fuel and the same time, power efficiencies achieved today by modern fossil fuel combustion ground, running at overcritical of live steam parameters (temperature around 600 ° C and pressure of approx. 250 bar) are between 40 and 52%. Efficient plants in Romania are in the best cases of 37%.

For comparison, the AD700 project funded by the EU is to achieve ultra-overcritical parameters of live steam temperature of 700 ° C and pressures of 375 bar, leading to efficient 52 to 55%. [13]

3. Gasification

Gasification technology (combined cycle with coal gasification - IGCC) is obtaining synthetic gas from solid fossil fuel. [5]

There is general tendency not to consider this technology as already commercial, especially because of cost by 10 -20% higher than power plants burning pulverized. But most times was not taken into account for conventional power plants and installation cost of reducing emissions of SO_x, NO_x, particulates and CO_2.

Considering a reduction in CO_2 emissions by 85 - 90% for both technologies, the price differences is reduced or change their meaning, IGCC become cheaper. [13]

The main advantages and disadvantages: the investment of 0: 1400 - $ 1600 per kW (indeed 10% higher compared to a combustion plant, but without any measures to reduce pollutant emissions), availability of technology 90%; effective: at least 40% NET can run on coal, biomass, waste and especially can work on petroleum residues, is facilitated CO_2 capture, as for other emissions of pollutants, they are reduced or not result from processes plant, IGCC performance may be negatively affected by high moisture content (WI) and/or high ash content (you) calorific value synthetic gas, which is composed mainly of hydrogen and carbon monoxide, is about . 4 times smaller than that of natural gas, which means that a gas turbine will require 4 times the amount of synthetic gas to produce the same power as the operation on natural gas, gasification can be oxygen or air. The investment in oxygen gasification plant producing oxygen is about 15% of the total, and electricity is about lost. 15% of GDP.

The gasification air, due to the large price increases synthetic gas cooling and cleaning it, and the calorific value is half that of synthetic gas obtained from gasification with oxygen.

Following those listed can be concluded that gasification is a technology that in the coming years is likely to become a feasible solution increasingly used in the construction of new ones, which would have all environmental systems included, particularly those limiting air emissions.

From analysis a show that is not yet solved the problem of environment protection, the combustion of solid fuels. But a defining technology in terms of denoxis flue gas is burning in circulating fluidized layer (ASFC). This allows the fulfillment of the rules on limited concentration of nitrogen oxides and sulfur gases discharged into the environment, with minimum investment and operating costs compared with other combustion processes. And take account of its advantages, namely: preparation of coal is brief, opportunity to use different qualities of coal, combustion efficiency is improved (98-99%), burning is autoterma and emissions are minimal.

5. CONCLUSIONS

Laboratory ASFC an installation proposal for experimental tests. A potential laboratory facility ASFC is shown in Figure 1.

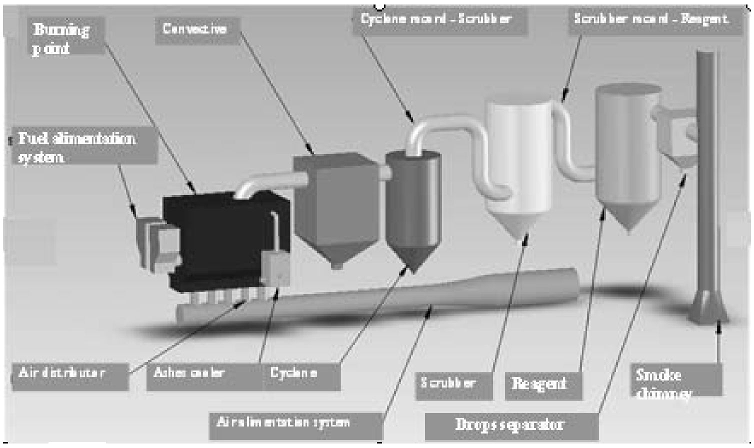

Fig.1. Proposed draft - plant laboratory for experimental tests

For putting the experimental model plant design along the laboratory is necessary to ensure adequate utilities, related equipment and systems of measurement and adjustment of operating parameters.

Necessary utilities operation of facilities/experimental model is: electricity, natural gas, water cooling, solid fuels, and combustion gas purification reagents.

Related equipment that are practically part of the installation and can be changed/provided by the reserve are mainly: Fans, pumps, thermocouples, Worm Motor, Compressor drive membrane pumps, current sources, manometers, pressure measurement sensors, moisture.

Systems of measurement and adjustment of operating parameters: data acquisition system - temperature (and possibly pressure, humidity, concentration of noxious) gas meter, water meter cooling water flow measurement roamers washing scrubber / reactor, chain measurement of particulate concentrations gases, thermocouples, thermometers, manometers, valves / flow adjustment valves for air / water cooled / liquid gas purification system.

For the operation is important to know to make performance and calibration of equipment where appropriate. For process control in the facility is very important to know thermal regime.

Also, the data acquisition system which should include: computer system, data acquisition board, sensors read the temperature, data acquisition program was.

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EFECTS OF NATURAL ABSORBANT TO THE OIL, GASOLINE AND DIESEL SOIL POLUTION

CHIS TIMUR *
POPESCU RAMONA**

Abstract: *Recovery of crude oils and petroleum products and mineral oils and accidentally spilled their waste into the environment (water, soil / subsoil) when they are in the form of thin films iridescent or water surface and on / in soil / subsoil, the final stage is achieved by using sinks. Most used sorbents are the type that are based absorbatilor biodegradable peat. This paper presents the mechanisms of biodegradation of crude oils and various absorbents effect on accidental oil pollution. It also focuses on the biodegradation of crude oils and petroleum products using peat.*

Keywords: *oil pollution, natural absorbent*

1. MECHANISMS OF DEGRADATION OF OIL POLLUANTS

Transportation of crude oil and petroleum products is an activity that can produce and accidental pollution on the ground (basement) and water (surface and groundwater). To recover these pollutants affect the environment (water, soil, sub soil) were used absorbent substances and skimmers.

To limit the effects were used absorbent dams or cessation of flow and dispersants. Sorbantii (ad / absorbed) [1] can be used:

- natural organic, such as peat, sawdust, straw chopped carbon based products,
- synthetic polyurethane, polypropylene, polyethylene,
- natural inorganic volcanic tuff guy, clay, rock wool, perlite.

Best type using organic sorbents based on turba. Dupa that occurs due to leakage of oil hydrometeorological conditions, the pollutant volatile compound evaporates in a very short time. The rough degrades in a long time.

For example, Miller [2] found these changes in the properties of the Oklahama oil, over time and increase biodegradation.

∗ *Lecturer, Ovidius University Constanta*
∗∗ *Masterand, Ovidius University Constanta*

	Gravity API	Sulf %	Vanadium Mg/kg	Nichel Mg/kg
OIL	32	0,6	30,6	16,4
Moderately degraded oil	12	1,6	224	75,1
Heavy fraction degraded	4	1,5	137,5	68,5

It is found that if the oil pollution biodegradation of these products does not lead to total elimination of pollution [3,4] because of the coarse fractions which degrades appear hard and heavy metals. Larter and Aplin [3] showed that crude oil biodegradation rates are between:

a. 10^{-6} - 10^{-7}/an. (For crude oils at temperatures degraded anaerobically 600 º C)

b. 10^{-2} - 10^{-1} kg oil / oil degradation m^2/year to the surface.

Also Larter and [3] demonstrate that the rates of biodegradation of crude oil in the range 10^{-3} - 10^{-4} kg oil/m^2/an to contact water / oil for fresh oil from closed stores.

Also at temperatures above 40 º C increase oil biodegradation [4]. Analysis of oil degradation Petromar type showed increased and decreased polar compounds asphalt saturated and aromatic compounds [6].

	Saturate	Aromatice	Polare	Asfaltice
oil	55 %	23 %	21 %	2 %
Moderately degraded oil	25 %	21 %	39 %	14 %
Heavy fraction degraded	20 %	21 %	41 %	21 %

Also sorbantilor analysis [1.7] demostrates absorbing oil product oil in the proportion established mg/mg sorbate.

Natural inorganic sorbates

Sorbant	absorption capacity product with high viscosity oil	absorption capacity of low-viscosity oil	product buoyancy
Vermulit	4	3	Sink
Volcanic ash	20	6	buoyant
Glasswool	4	3	buoyant

Organic sorbates

Sorbant	absorption capacity product with high viscosity oil	absorption capacity of low-viscosity oil	product buoyancy
chopped straw	6	5	Sink
Cellulose fibers	18	10	Sink
Peat	4	7	Sink

Synthetic organic sorbents

Sorbantul	Capacitatea de absortie a produsului petrolier cu viscozitate mare	Capacitatea de absortie a produsului petrolier cu viscozitate mica	Flotabilitatea sorbantului
foam	70	60	Flotabil
Urea formaldehyde foam	60	50	buoyant
polyethylene fiber	35	30	buoyant
polypropylene fibers	20	7	buoyant
polystyrene powder	20	20	buoyant

2. ANALYSIS TYPE PEAT SORBENT

Peat has a high content of organic matter is composed of organic substances nehumice (carbohydrates, proteins, peptides, amino acids, etc.).

And humic orgaanice substances (humic acids and fluvici). Also present factories is confirmed in nonpolar organic substances on the surface of colloidal humus particles (molecular sortie or apolar).

To study the effects of sorbent type oil to use peat peat over Maramures with the following features:

a. It is a 100% natural product, is biodegradable,

b. is composed of peat moss (sphagnum Spagnum) 90% water and 10%,

c. It is soluble in water,

d. The solid product resulting from the decomposition of peat moss,

e. has a high porosity (80%) with a fibrous appearance with a high sorption capacity for liquids

f. reddish-brown in dry and wet brown,

g. Non-toxic,

h. has a high content of humic acids which can support microbiological activity in the biodegradation of petroleum products.

The physico-chemical properties of peat

feature	Value
Residue on drying,%	18,20
Total nitrogen,%	0,22
pH of the aqueous filtrate	4,2-5,2
Relative humidity,%	11,8
Absolute humidity,%	12,9
Free Bulk density, kg / m	76
Bulk density compact, kg / m	110
Slope angle	40
Surface area, m / w	222
Total pore volume, cc / g	0,63
Ash,%	3,46
Combustible mass,%	85,40
Lower calorific value, kcal / kg	2690

Testing the retention capacity of peat from different pollutants

The test was performed for 4 pollutants and 3 types of absorbers.

Pollutant /Density at 20 ° C	**peat**	**briquettes**	**straw**
Gasoline 0,473	4,1-4,9	2,7-3,7	5,1-5,4
Diesel 0,840	6,1-7,2	3,8-4,7	6,3-7,6
Oil 0,900	12,1-13,1	5,3-6,4	12,5-14,1
CLU 0,928	7,2-9,8	3,8-4,6	8,0-10,3

Capillary rise test

This test determined the rise of petroleum products through a capillary tube filled with absorb. Works by adsortie adsortie sorbents and operating as a sponge and collecting the product by capillary action and suction. Sorbents are useful for lighter petroleum products and adsorbents for heavy oil products. The study conducted by us show that diesel and gasoline are retained instantaneous (less than 6 seconds), oil and CLU are retained between 1-10 seconds and the water is absorbed up to 2 hours.

	1 s	5 s	10 s	20 s
Water	2	5	10	15
Crude oil	10	30	100	
Diesel	60	100		
Gasoline	70	100		
CLU	15	35	100	

3. CRUDE OIL BIODEGRADATION WITH PEAT SORBENT

Many microorganisms are designed to use the hydrocarbon gases, solids and liquids series aliphatic, aromatic and asphalt as carbon and energy source, descompundu them into compounds with lower molecular weight, water and CO_2. The first observations of these biological decomposition processes they study Myoshi (1895) and Rahm (1906) that have established that the soil microfugii (Penicillium glaucum) attack paraffin. Microbioana degradation of petroleum products is a particular case of microorganisms such hidrocarbonoclastice (Ahearn, 1973) [8]. It is said that petroleum products can be beneficial for soil fertility, treating them with small amounts of toluene temporary lowering of the number of microorganisms, followed by an increase from baseline of $5 * 10^6$ / g phenomenon is due to the use of toluene, xylene, hexane and chlorobenzene as carbon and energy source.

Discharge of petroleum products on soil brings numerical growth of bacteria populations by reducing their diversity (grow bacteria that destroy hydrocarbons).

Noting that the massive oil pollution destroy vegetation but occurs aerobically degradation and return to normal.

Peat studies showed these cultures of microorganisms:

Total microbiological	indicator organism / g peat
Total microflora	900×10^7
Celulozotici	320×10^5
Amonificatori	168×10^{18}
Denitrified	140×10^2
Proteolytic	920×10^{12}
Nitrous	420×10^5
Nitrites	380×10^6
Aerobic nitrogen fixing	180×10^2
Anaerobic nitrogen fixers	220×10^2

Also on oil polluted soil with peat treatment was performed and watched the behavior of microorganisms.

Total microbiological	indicator organism / g peat
Total microflora	970×10^{8}
Celulozotici	280×10^{5}
Amonificatori	435×10^{18}
Denitrified	190×10^{1}
Proteolytic	979×10^{12}
Nitrous	220×10^{5}
Nitrites	270×10^{6}
Aerobic nitrogen fixing	290×10^{1}
Anaerobic nitrogen fixers	180×10^{1}

4. CONCLUSIONS

Oil pollution and petroleum products can be treated to restore polluted environment using natural or artificial sorbents.

Use peat increases by 10% the level of biological decontamination of soils polluted.

Oil doses did not result in inhibition of microbial growth.

Microbiology was determined by isolating strains that oil polluted soils behave best Gram negative cultures (Pseudonomas).

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EFFICIENT SOLUTION FOR HARD-COAL EXPLOITATION WITH UNDERMINED COAL BED USING THE LONG BLASTING BOREHOLES METHOD FOR WEAKENING AND PRE-CRUSHING THE ROCK AND COAL MASSIF

PLESEA VALERIU[*]
DRAGOESCU PETRE[**]
STOCHITOIU DANIELA[***]
TOMESCU CRISTIAN[****]
VLASIN NICOLAE - IOAN[*****]

Abstract: *Once with the application of the undermined bed exploitation method of Jiu Valley's coal, which increased up to 3 times the thickness of extraction slices compared with classical exploitation methods, appeared the need of directional weakening and pre-crushing of coal, in order of gravity trip and coal discharge form the undermined bed. Also, directional weakening and pre-crushing of coal it is necessary for unloading and discharge areas of the bed layer, where the tilt is below the natural slope angle, and dragging of breaking line at stope start from a natural alignment. All these requirements of operation require the use of solution with mine shooting long holes of "coal – rock" massif, through security explosive cargo with great length. In the paper there is presented the necessity and opportunity of application of undermined bed coal pre-crushing works, as well as the technique of mine long holes shooting, rendering of safety explosive loads construction and results of laboratory tests performed for various simulated mines atmospheres.*

Keywords: *coal, undermined coal bed, drilling and blasting, pre-crushing, mining holes.*

1. INTRODUCTION

According to the design method applied to the particular conditions of underground mining, the extracting coal layer in undermined front method provides the basic slice of stope, which height is h slice = 2.5 m, followed by gravitational evacuation unloading of coal from undermined coal bed (h_{layer} = max. 10m) behind the working coal face.

∗ *1st degree Scientific Researcher,University "Constantin Brancusi "Tg-Jiu*
∗∗ *Eng., National Coal Company Petrosani*
∗∗∗ *PhD., Eng., Assoc. Prof., University of Petrosani*
∗∗∗∗ *PhD. Student, Eng., INCD INSEMEX Petrosani*
∗∗∗∗∗ Eng., *INCD INSEMEX Petrosani*

Separation of coal from existed sterile rock from the roof occurs naturally, when this is unloaded during the control of mining operation pressure by taking out the last row of beams behind the stope. Separation at this level of coal from sterile rock in the existing bed roof is made visual, the "dumping - discharge" cycle being stopped when the sterile rock reach a higher percentage than the discharged coal.

In this percentage the mixed coal with the sterile rock is considered to be a waste, reducing it, provided at maximum of 15%, characterize the profitable method.

Complete dumping and discharging of the coal from the undermined coal layer, followed by filling the created hole with sterile is a great technical and economical issue; the evolution of this process in optimal working conditions and safety deposit by performing the additional work of pre-crushing the coal and sterile rock massif, representing the overall efficiency of the exploitation method.

2. CAUSALITY AND OPPORTUNITY OF COAL AND COVERING ROCKS PRE-CRUSHING WORKINGS

At the level of long breast with undermined coal bed and adjacent areas, for long breast with undermined coal bed at their level and adjacent areas, the original strain is modified and a new state appears, more complex and with a higher intensity, as the coal layer thickness and the abruption of stope is bigger, the volume of coal and rock that is moving is bigger.

In this case, an area will be made into the interior of the massif where it forms a tracing and cracking of the coal and during this, the bearing stress development occurs [1]. The destruction of coal's degree of compaction around the stope due to its transition from uni status request in a biaxial or triaxial, favors the reducing of grading and increasing the flow velocity / discharging of coal to the evacuation operation, which is more marked and the time if the coal is much more rotten and the standstill of stope is higher, that means that the stope cycle of 1.25 m is longer than 3 days and the working speed for advancement of the working face is below than 7 - 8 m a month. In the specific coal features: strength and high fall-in rate (σ_{rc} <15 MPa, IS = 2-3), the bearing pressures make possible the alteration and the transition of elastic-plastic behavior to powder, making possible the movement of the breast under a strong split and loosened coal horizon creating favorable conditions to the discharge of coal from the undermined bed.

Reticence about the coal discharge operation can appear if the thickness of coal bed increases to a maximum of 10m, Reticence about the coal discharge operation can appear if the thickness of coal bed increases to a maximum of 10 m, imposed by the adjustment of costs to the ton of extracted coal and according to this situation the naturally extend of the splits can partly cause the fragmentation of coal and rocks from the stope roof into blocks.

In such cases, the situation becomes more difficult for the operation of coal discharging when the resistance and compactness is higher ($\sigma_{rc} \geq$ 15M Pa) and rocks from the natural stope roof (for operations of low inclination layer) has $\sigma_{rc} \geq$ 60 MPa. For such conditions, approx. 30% of the coal located at the top of the bed is pensile and not discharged, looking like an outrigger.

As a result, after the mining pressure is directed by the incomplete fail of the coal and covering rocks it generates some gaps in the operated area, with all the adverse consequences on airflow and increased tendency to self-ignition of coal or large accumulations of gas (methane) and coal dust, with a possibility of mine explosions.

On the other hand, as the advancing of breast, the length of coal and rocks collapse area increases significantly (L_n 1> 15m, fig. 1), favoring an instant mechanical phenomenon of uncontrolled detachment of coal and covering rocks with free vertically movement, of oversized blocks to the exploited area, requesting a dynamically support under the influence of gravitational forces compromising the safety working of personnel in stope, with temporarily stop of working front.

In cases of higher inclines of exploitable bed coal layer (α_{layer} > 45 °), where slices / horizontal undergrowth operating method is applied, the diverting of the mining pressure behind the stope, generates the split and alteration of coal more pronounced in the area of roof layer making possible the discharge of this faster, if the gradually discharge operation in stages is not implemented properly. In such cases there are gaps into exploited area, with all the adverse consequences shown before.

Trend of creating the exploitation gaps is in the areas of coal bed, for average inclines (α_{layer} = 25-45°) and higher, being below the natural slope angle, and low inclines (α_{layer} < 25°) along the directional roadway made for coal bed.

For all these adverse situations, by design and application documentation for the exploitation method of undermined layer shall be provided prior work of coal and sterile rocks pre-grounding by drilling and blasting the long mining holes, which are made from the working front ,adjacent existing mining works or from the preparation roadways [2,3].

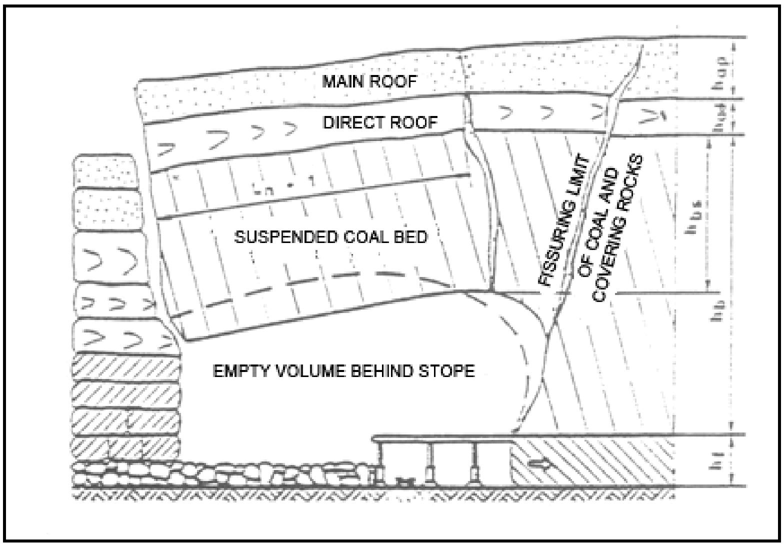

Figure 1. The formation of exploitation gap at the undermined coal bed method:
h_f – thickness of basic slice of stope, h_b - total thickness of undermined coal bed;
h_{bs} – part of the hanging layer, h_{ad}, h_{ap} - thickness of the directly and primarily roof.

3. SAFETY EXPLOSIVE CHARGE FOR LARGE PRE-GROUNDING LENGTH

For making a safety explosive charge with a large length can be taken into account two alternatives [1, 3]:

- use of high safety explosives made on the principle of salts with ion exchange;
- use of discontinuous charges into the mining holes, consisting on classical safety explosives and gas-proof detonating fuse, with or without added salt coolers.

The second variety is used by most countries that apply the techniques of coal exploitation by long gas-proof mining holes (South Africa, Ukraine, Spain and Poland).

This variety of mixing the blasting cartridges with salts coolers was adopted in the Jiu Valley mining practice, taking into account that then, for the application blasting method of long mining holes, safety gas-proof explosives (such as AGP) were produced in Romania (Nitramonia – Făgăraş), according to the physical-chemical, thermo-dynamics, ballistics and safety parameters similar to the products made by prestigious producers in the world.

On principle, according to this blasting technique for coal pre-grounding, long holes are made by boring machines similar to those of WD-02EA type (Polish origin) used in the mines, which have the possibility of perforation holes in diameters up to 66 mm and lengths of over 30 m.

Long discontinuous charge are made by an alternation of 1-3 safety blasting cartridges and one cartridge with rock salt to absorb the heat given, so having the duty of cooling. The salt rock used has 0,1 mm granulation with a cartridge weight of 100 g.

In Jiu Valley, the detonation conduction to the discontinuous exploder is made using a Spanish blasting fuse.

For blasting are used gas-proof electrical squibs, one or two for a charge/ hole. Explosive charge made by explosive cartridges, cooler salts and blasting fuse, before placing into the mining hole, are made in special Omega tubes (made in Spanish), which have an outside diameter of 32 mm and 3 m lengths.

Placing explosive charges into holes is made by pushing the Omega tubes containing the charge which are assembled together according to the length of holes, by overlap to a minimum length of 200mm and joint by boring and bounded through with string or wire. An explosive charge is made by placing over the blasting fuse the blasting cartridges and salt rock cartridges in a sequence of one salt rock cartridge and three blasting cartridges (figure 2). The electrical blasting squib / squibs are placed to at least 100 mm from the end of the blasting fuse, by close binding with tape (figure 2, 3).

In the situation when it should be connected two segments of gas-proof blasting fuse will only proceed by abut them, using a tube connector (figure 4).

Based on the results of laboratory tests made in the explosive charges test area of INSEMEX Petroşani on discontinuous explosive charges consisting on AGP blasting cartridges and cooler salt cartridges (sodium chloride, calcium carbonate), in air-methane and air-methane-coal dust atmosphere, the following conclusions were stated (table 1):

Table 1. Test results for long discontinuous charges

Cons. no	**Weight of explosive charge, gr.**		**Weight of salt charge, gr.**		**Methane or coal dust concentration**		**No. of blasting /no. of tests**
	section	**total, for test**	**section**	**total, for test**	**CH_4, % vol.**	**coal dust, gr/m^3**	
1.	900	3600	100	500	970- 980	-	1/2
2.	700	3500	100	600	840- 990	-	1/78
3.	600	3000	100	600	860- 980	-	1/7
4.	500	3000	100	700	810- 980	-	1/13
5.	400	3200	100	900	840- 970	-	1/24
6.	300	3000	100	1100	830- 930	-	0/100
7.	500	3500	100	800	-	330	0/71

- making the explosive charge by alternative sections of three AGP cartridges (300 grams) and sodium chloride cartridge (100 grams), blasted with the a gas-proof electrical squib and RIOCORD blasting fuse 6g / m Antigrisu, provide the safety blasting of the explosive charge in the air-methane atmosphere 9.0 ± 1.0% vol.;

- discontinuous combined explosive charge, blasted by an electrical squib and blasting fuse, both gas-proof type, ensure safety for air-methane-coal dust atmosphere.

On the applicability of this solution of long blasting holes and discontinuous explosive charges for explosive for real underground mining conditions, it is estimated to be a good solution compared with the usual method of short mining holes and continuous explosive charges previously applied, being recorded a 80-85% coal recovery rate from undermined coal bed, the decrease of technology dilution up to 10 to 15% and creating good conditions for safety work and protection of the deposit [1].

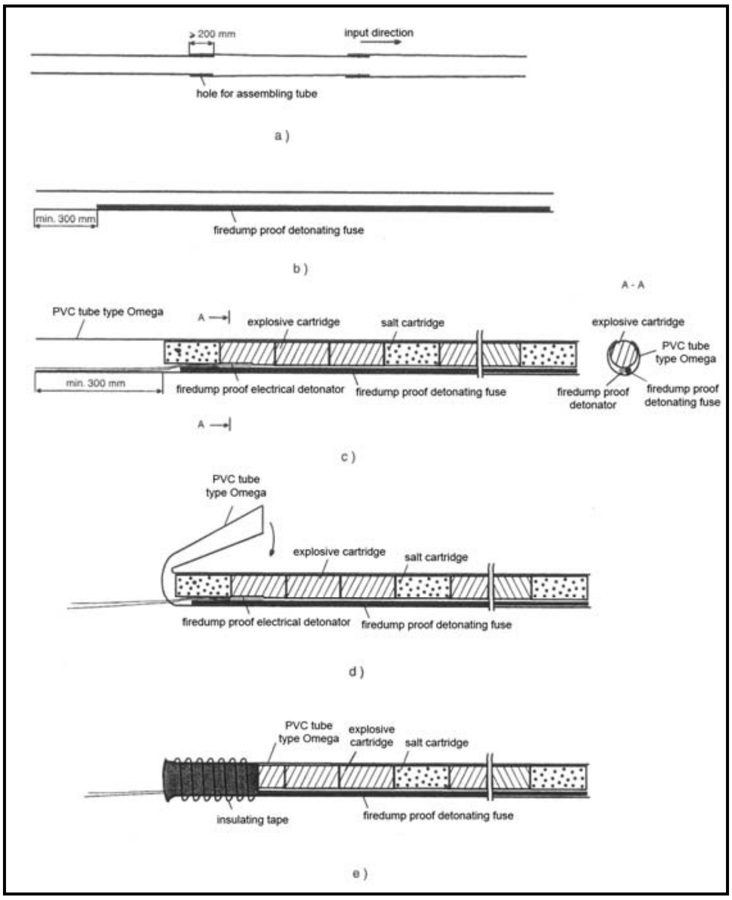

Figure 2. How is made a discontinuous long explosive charge

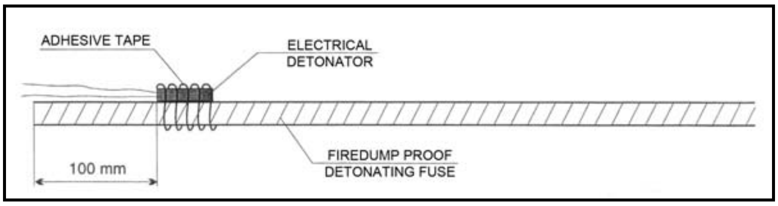

Figure 3. How is placed a squib to a blasting fuse

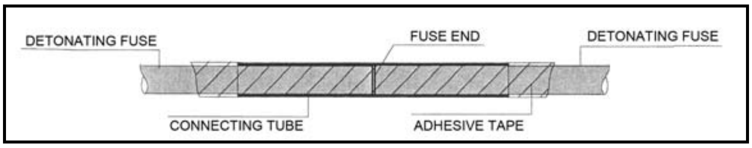

Figure 4. The connection between sections and blasting fuses by connecting tubes

4. CONCLUSIONS

1. For a productive technological processes of coal dumping and discharging to the undermined coal bed method, and over the whole stope flow, the insertion and application of coal pre-grounding technique by long mining holes blasting, using a discontinuous charges, it proves to be a good technology solution in terms of technical and economical profitability, and to ensure a safe working conditions and protection of the deposit.

2. Nowadays, when the production of NITRAMONIA Făgăraş for gas-proof explosives has stopped and started the purchase of Slovakian explosive SLAVIT V for discontinuous charges used for coal pre-grounding (125 gr. per cartridge), it is recommended to use this cartridge in an alternation of two blasting cartridges and one salt cartridge for one mining hole.

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CONTAMINATION /RESPECTIVELY POLLUTION WITH PESTICIDES OF THE SOILS IN THE PLANT PROTECTION CENTER OF MIERCUREA CIUC

BORBOLY CSABA[*]
SZÉP ROBERT[*]

Abstract: *This study aims to present the enviromnental problems, which exist more than 20 years in former plant protection areas, which are listed in the county council's administration. In this areas were stored unused pesticides for more then 15 years, and in this period the storage tanks were deteriorated, thus the soil layer and the groundwater became excesively polluted. The stored materials are organochloride pesticides, and their derivates are extremely dangerous (carcinogenic) for the human body. The materials determined through gas chromatography were the following: 4,41-DDT1, 4,41-DDD1, 4,41-DDE1, DDT/DDD/DDE (3)1, Aldrin1, Dieldrin1, Endrin1, Total drin (3)1, alfa-HCH1, beta-HCH1, delta-HCH1, gamma-HCH (Lindan)1, Total HCH (4)1, Heptaclor1, Heptaclor-epoxid1, Metoxiclor1, alfa-Endosulfan1, beta-Endosulfan1, Endosulfan-sulfat1, Endrin-aldehid1, Hexaclorbenzen1, other organochloride pesticides (8)1. Many of these substances exceed in some cases the admitted level for more hundred times, thus represent a real danger for human health, due to the fact that through Decree no. 756/1997 their neighborhood areas were classified in sensitive use category.*

Keywords: *pesticides, organochloride pesticides, sensitive use cathegory, organocloride derivates,*

1. INTRODUCTION

The objective is situated in the industrial area of Miercurea-Ciuc municipality, in the North-East part of the city, Progresului street, no. 18. The lands situated in the close neighbourhood of the objective, according to Decree no. 756/1997 are "less sensitive" and apart from this area in all directions are sensitive areas.

The surface where the building lies is of 8.449 mp, of which 547 mp is arable land, 1585 mp orchard and 6317 mp constructed area of which 1.192 mp buildings. According to Land Registry no. 1811N/2005, the owner of the building is Harghita County Council.

The activity profile of the emplacement was the distribution and the storage of phyto-sanitary products, packaging and repackaging of these products, preparation of phyto-sanitary equipments in order to intervent with pesticides in the activity area of the plant protection center.

∗ *Sapientia University, Miercurea Ciuc, Faculty of Sciences, 530104 Miercurea Ciuc, Piața Libertății no. 1*

The buildings of the former plant protection center of Miercurea-Ciuc were used between 1965-1972, and in the period of 1993-1995 the unused pesticides, collected from the former agriculture cooperatives were stored in the basement of the buildings, in total of 9036 kg respectively 9455 l. Around 90% of these are organochloride pesticides. These pesticides were stored here until 2007, when through a governmental programme the pesticides were collected and neutralised. Unfortunately until the period of neutralisation, the storage tanks of the pesticides were damaged, thus the inferior soil layers respectively the underground water were polluted. For the present study we have determined the soil type, we undertaken 11 drillings of which 3 of hydrochemical research, and we determined the main active comtent of the organochloride pesticides and of their derivates. The analysed components were the following: 4,4[1]-DDT[1],4,4[1]-DDD[1],4,4[1]-DDE[1], Aldrin[1], Dieldrin[1], Endrin[1], alfa-HCH[1], beta-HCH[1], delta-HCH[1], gamma-HCH (Lindane)[1], Heptachloride[1], Heptachloride-epoxide[1], Metoxichloride[1], alfa-Endosulfan[1], beta-Endosulfan[1], Endosulfan-sulphate[1], Endrin-aldehide[1], Hexachlorbenzene[1].

2. MATHERIAL AND METHODS

The undertaken analyses and interpretations were realised according to laboratory examinations and field studies. Thus were analysed 11 samples of which 3 hydrogeochemical (figure 1).

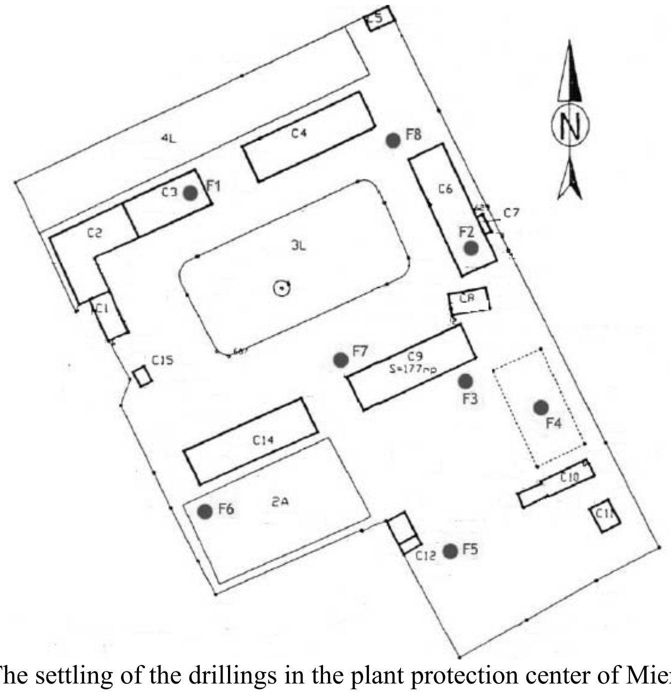


Figure 1 The settling of the drillings in the plant protection center of Miercurea Ciuc

The chemical analyses were undertaken in an acredited laboratory in Hungary - Wessling laboratory – project 2011-364 (2011/K/3993). The chemical analyses were realised with gaschromatographic method with an equipment LP-6890-GCMS_03-5973.

The soil samples were determined macroscopical on the field through drillings and were confronted with the pedologic maps in the area of Miercurea Ciuc municipality.

3. RESULTS AND DISCUSSIONS:

The undertaken pedogeological studies revealed a tipfaeoziom greic soil, which has the following sequence of horizons:***Am–Ame–Bt-Cca***.

The horizon ***Am*** has a dark brown-grey colour of a thickness of 0-40 cm.

The horizon ***Ame*** has a lighter brown-grey colour then the above horizon, due to the partial eluvial of the colloide particles and residual enrichment with quartz grains stripped of colloidal film. The thickness of Ame horizon is of approximately 10-30 cm.

The horizon ***Bt*** **(**60-140 cm) has a dark yellow colour, which on the lower part of the horizon has a brown-yellow, almost red colour, with fine texture and prismathic structure.

The horizon ***Cca*** is situated in the lower part of the profile. In the profile are biogenic neoformations like coprolite, cervotocine, bunks and/or places of organisms, especially on the level of bioaccumulation horizon and ***Cca*** are chemical neoformations of $CaCO_3$.

The horizon ***Bt*** has frequent iron oxide spots and iron/mangane concretions, and also thin or moderate thick clay films.

The horizon ***Ame*** has residual neoformations, represented by isolated quartz grains stripped of colloidal film or other mineral particles resistent to alteration.

The faeoziomous texture on the ***Am*** horizon is of sandy-clay and clay loam with glomelural structure. In the underlying ***Ame*** horizon the texture is more rough and the structure is of subangular polyhedra. On the level of this horizon are frequently present the sandy particles stripped of colloidal films.

The percent of the clay in ***Ame*** registers a slightly decrease. The pH values are neutral in ***Am*** (6,7) and strongly moderated in acide in ***Ame***(5,3). The horizon ***Bt*** presents a growth of the clay fraction.

The laboratory analyses on the soil and water samples identified the following concentrations (tab. 1, 2).

Table 1 Concentrations of the pesticides in the analysed soil depths

Component	**Sample code** mg/kg				
	F1 4-4,5	**F2 4-4,5**	**F3 4-4,5**	**F4 4-4,5**	**F5 3-3,5**
4,4[1]-DDT[1]	<0,01	<0,01	<0,01	<0,01	7,33
4,4[1]-DDD[1]	<0,01	<0,01	<0,01	<0,01	3,95
4,4[1]-DDE[1]	<0,01	<0,01	<0,01	<0,01	<0,01
DDT/DDD/DDE	<0,03	<0,03	<0,03	<0,03	11,3
Aldrin[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Dieldrin[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Endrin[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Total drin	<0,03	<0,03	<0,03	<0,03	<0,03
alfa-HCH[1]	0,003	<0,002	<0,002	<0,002	2,63
beta-HCH[1]	<0,002	<0,002	<0,002	<0,002	<0,002
delta-HCH[1]	<0,002	<0,002	<0,002	<0,002	1,12
gamma-HCH	<0,002	<0,002	<0,002	<0,002	11,0
Total HCH	<0,008	<0,008	<0,008	<0,008	14,8

Heptachloride[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Heptachloride-epoxid[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Metoxichloride[1]	<0,01	<0,01	<0,01	<0,01	<0,01
alfa-Endosulfan[1]	<0,01	<0,01	<0,01	<0,01	<0,01
beta-Endosulfan[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Endosulfan-sulphate[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Endrin-aldehide[1]	<0,01	<0,01	<0,01	<0,01	<0,01
Hexachlorbenzene[1]	<0,01	<0,01	<0,01	<0,01	0,01
Other organochloride pesticides	<0,07	<0,07	<0,07	<0,07	<0,07

Component	**F5 5-5,5**	**F6 0-0,5**	**F6 1-1,5**	**F6 3-3,5**	**F6 4-4,5**	**F7 4-4,5**	**F8 4,5-5**
4,4[1]-DDT[1]	<0,01	0,22	0,19	<0,01	<0,01	0,09	0,01
4,4[1]-DDD[1]	<0,01	0,12	0,02	<0,01	<0,01	0,06	<0,01
4,4[1]-DDE[1]	<0,01	1,22	<0,01	<0,01	<0,01	0,05	<0,01
DDT/DDD/DDE	<0,03	1,56	0,21	<0,03	<0,03	0,20	<0,03
Aldrin[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Dieldrin[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Endrin[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Total drin	<0,03	<0,03	<0,03	<0,03	<0,03	<0,03	<0,03
alfa-HCH[1]	<0,002	<0,002	0,004	<0,002	<0,002	0,020	<0,002
beta-HCH[1]	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
delta-HCH[1]	<0,002	<0,002	0,003	<0,002	<0,002	0,015	<0,002
gamma-HCH	<0,008	<0,002	0,020	<0,002	<0,002	0,078	<0,002
Total HCH	<0,008	<0,008	0,027	<0,008	<0,008	0,113	<0,008
Heptachloride[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Heptachloride-epoxid[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Metoxiclor[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
alfa-Endosulfan[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
beta-Endosulfan[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Endosulfan-sulphate[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Endrin-aldehide[1]	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01	<0,01
Hexachlorbenzene[1]	<0,01	<0,01	<0,01	0,01	<0,01	<0,01	<0,01
Other organochloride pesticides	<0,07	<0,07	<0,07	<0,07	<0,07	<0,07	<0,07

Table 2 Analysis of the water samples F6, F8, Well

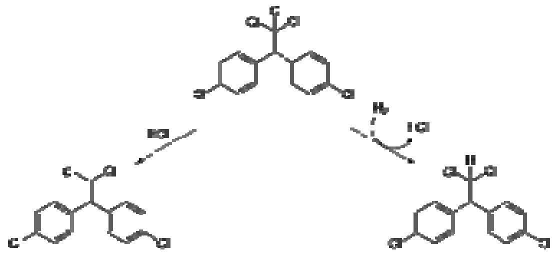
Component	**Sample code** µg/dm^3			
	F5	**F6**	**F8**	**WELL**
4,4[1]-DDT[1]	33,3	<0,0002	0,0691	0,0108
4,4[1]-DDD[1]	18,3	<0,0002	0,0696	0,0142
4,4[1]-DDE[1]	<0,0002	<0,0002	<0,0002	0,0167
DDT/DDD/DDE (3)[1]	51,6	<0,0006	0,139	0,0417
Aldrin[1]	<0,001	<0,001	<0,001	<0,001
Dieldrin[1]	<0,001	<0,001	<0,001	<0,001
Endrin[1]	<0,001	<0,001	<0,001	<0,001
Total drin (3)[1]	<0,003	<0,003	<0,003	<0,003
alfa-HCH[1]	23,2	0,131	2,45	0,015
beta-HCH[1]	3,26	0,338	0,693	0,599
delta-HCH[1]	15,5	0,061	1,87	<0,001
gamma-HCH (Lindane)[1]	159	0,049	16,2	0,026
Total HCH (4)[1]	201	0,579	21,2	0,640
Heptachloride[1]	<0,001	<0,001	<0,001	<0,001
Heptachloride-epoxid[1]	<0,001	<0,001	<0,001	<0,001
Metoxichloride[1]	<0,001	<0,001	<0,001	<0,001
alfa-Endosulfan[1]	<0,001	<0,001	<0,001	<0,001
beta-Endosulfan[1]	<0,001	<0,001	<0,001	<0,001
Endosulfan-sulphate[1]	<0,001	<0,001	<0,001	<0,001
Endrin-aldehide[1]	<0,001	<0,001	<0,001	<0,001
Hexachlorbenzene[1]	0,060	<0,001	<0,001	<0,001
Other organochloride pesticides (8)[1]	0,060	<0,0008	<0,008	<0,008

Studying the results of the chemical analyses, we can observe that two groupe of pesticides are resistent in the soil: grupa DDT (**d**ichloro**d**iphenyl**t**richloroethane) şi HCH (Technical HCH is a mixture of 60%–70% α-HCH, 2%–12% β-HCH, 10%–15% γ-HCH, and other minor isomers şi derivații acestuia[3].

DDT is similar in structure to the insecticide methoxychlor and the acaricidedicofol. It is a highly hydrophobic, nearly insoluble in water but has a good solubility in most organicsolvents, fats, and oils. DDT does not occur naturally, but is produced by the reaction of chloral (CCl_3CHO) with chlorobenzene (C_6H_5Cl) in the presence of sulfuric acid, which acts as a catalyst.

4. ENVIRONMENTAL IMPACT

Degradation of DDT to form DDE (by elimination of HCl, left) and DDD (by reductive dechlorination, right)



DDT is a persistent organic pollutant that is readily adsorbed to soils and sediments, which can act both as sinks and as long-term sources of exposure contributing to terrestrial organisms [2]. Depending on conditions, its soil half life can range from 22 days to 30 years. Routes of loss and degradation include runoff, volatilization, photolysis and aerobic and anaerobicbiodegradation. Due to hydrophobic properties, in aquatic ecosystems DDT and its metabolites are absorbed by aquatic organisms and adsorbed on suspended particles, leaving little DDT dissolved in the water itself. Its breakdown products and metabolites, DDE and DDD, are also highly persistent and have similar chemical and physical properties.[1] DDT and its breakdown products are transported from warmer regions of the world to the Arctic by the phenomenon of global distillation, where they then accumulate in the region's food web.[4]

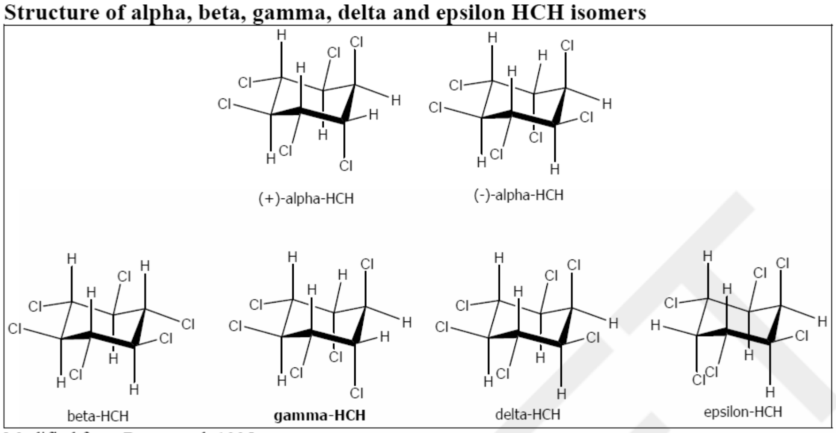
4.1. HCH lindane. Environmental impact

Lindane is a persistent organic pollutant: it is relatively long-lived in the environment; it is transported long distances by natural processes like global distillation.[2]

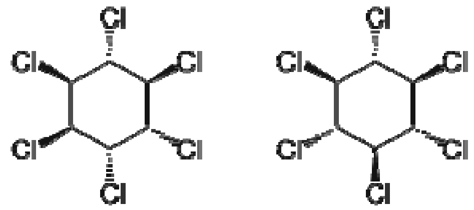
4.2. Isomers

Lindane is the gamma isomer of hexachlorocyclohexane ("γ-HCH"). In addition to the issue of lindane pollution are concerns related to the other isomers of HCH, namely alpha-HCH and beta-HCH, which are notably more toxic than lindane, lack its insecticidal properties, and are byproducts of lindane production. [5]

Structure of alpha, beta, gamma, delta and epsilon HCH isomers



Modified from Buser et al, 1995.



α-hexachlorocyclohexane β-hexachlorocyclohexane

5. CONCLUSIONS

The analysed soil and water samples show the highly resistence of these pesticides in soil, their accumulation and dispersion, and the pollution respectively contamination of the underground water as well.

The major part of the pollution reached the underground water through infiltration from the soil and then through its transportation to the underground water due to the storm water.

The underground water is situated at a very close level, the buffer layer is frequently discontinuous or is also missing. Thus i tis permitted the direct entrance of the pesticides in the underground water, their blocking possibility in the clay layers is almost impossible.

Due to the slight solubility of the pesticides in water, the major part of the polluting factors (organochloride pesticides) is located in the underground water oscillation area, at a depth of approximately 2-3 m, where come in contact with the soil and slightly accumulates.

At the moment the polluted soil area is considered being the main polluting source of the underground water, because the pesticide quantity of the soil is high and can easily reach the underground water.

We consider that the spatial extension of the underground water pollution occupies the major part of the emplacement especially in the oscillation area of the underground water level, emphasized on the polluted soil area.

Suggestions for soil decontamination:

F5 - 3 -3,5 m – is proposed the elimination of the soil layer until 3,8 -4,0 m in an area of 5-6 m around sample area

F6 – 0- 0,5m – is proposed the elimination of the soil layer from the superior horizon (A0) in an area of 6-7 m around sample area

Water decontamination:

All the samples are contamined, the polluting factors dispersion was done radially through the mixture of the soil water and their infusion (near the buildings C9-C14) directly in the lower layers of the soil. The lack of the organic meterial in the lower soil layers and huminic and fulvic acides has determined the considerable quantity migration in water layer.

High concentrations can be reached in the flow direction of the aquifer towards Șumuleu stream. Having in view the drillings we can observe that under the aquifer limit exists a buffer layer composed of compact clay, which makes waterproof the captive aquifer layers.

In these sense it is not suggested the intervention of the ground water bacause this is naturally regenerated through the elimination of the above situated source.

At the same time there are suggested the following:

- introduction in the PUG of the water utilisation restrictions for household and agriculture purposes on a surface of approximately 500-700 m in the neighbourhood of the Plant Protection Center;
- monitorization of the site after the decontamination in each third year;
- transportation of the dangerous waste with special cars, according to the legislation
- cultivation of fitoaccumulatory herbaceous plants and trees, which will be anually burnt according to the regulations (in case of the trees the leaf and branches resulted of the cleaning), and the ash obtained can be reintroduced in the soil circuit;
- utilisation as ornamental plants of some energetic species, these plants will remove from the soil the remaining pesticides, which remained after decontamination.
- it is forbidden the plant cultivation destinated to any kind of consumption.

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THE ANALYSIS OF VARIOUS ACTIVE AND PASSIVE TEHNOLOGIES FOR HEAVY METALS REMOVAL USED IN MINE WATER TREATMENT

CĂPUŞAN CORINA CRISTINA*

Abstract: *Heavy metal pollution has become one of the most serious environmental problems today. Treatment of mine water has a special concern due to the environmental degradation and the persistence of heavy metals over time. In recent years, various technologies for heavy metal removal from mine water have been extensively studied. These technologies can be active or passive. The category of active (traditional) technologies include the well known methods to increase the pH or to insure favorable conditions for the occurrence of redox reactions, followed by application of separation processes. Passive technologies include natural and biological processes that occur in different types of locations, without or with minimal electrical or mechanical assistance, which can be controlled during their course. They can be considered attractive to mine water treatment due to low construction costs, operation and maintenance and by the ability to operate in remote locations with limited operating requirements.*

***Keywords**: mine water, treatment, technologies, heavy metals*

1. ACTIVE TREATMENT TECHNOLOGIES

Active treatment involves neutralizing acid-polluted waters with alkaline chemicals. However, the chemicals can be expensive and the treatment facility is expensive to construct and operate.

1.1. Pond Treatment

The treatment system involves the addition and mixing of lime in water flow and the possibility to settle the precipitates in a pond. The pond is often divided into two sections. Primary pond serves for the accumulation of precipitated sludge and fills up fast enough. Frequent dredging and a storage area are required. Secondary pond is larger and requires a long retention time with laminar conditions which ensure a clean effluent.

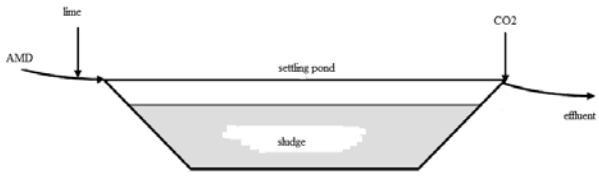

Figure 1: Pond treatment

* *student PhD.Eng. at University of Petrosani*

One of the greatest advantage is that the pond treatment is a simple low-cost construction but there are some big disadvantages such as the facts that they occupy large areas of land, they depend on weather conditions, they don't allow any controlling systems (input flow, flow velocity, mixing, sedimentation) and they realize a low lime efficiency due to the absence of mechanical mixing process.

In order to ensure proper treatment in a pond system, the pH set point is often brought up much higher than necessary for the targeted metal(s).

In order to improve both lime efficiency and treatment efficiency, a sludge recycle system can be installed. Tests completed at Falconbridge, Kidd Mining Division showed that recycling sludge using an automated dredge would decrease overall treatment costs (Aubé, 2004c).

1.2. Systems for mine water treatment in tanks

The treatment in tanks is similar with the one in ponds but requires an additional pumping of the water.

The treatment is completed by pumping the drainage to a tank, where pH is controlled by adding lime suspension until around the value of 11. For a better separation of solid - liquid ferric sulfate and a flocculant's for coagulation and flocculation are added. The treated slurry is then discharged underwater at one end of the tank to promote settling of the fresh particles. A clear supernatant is pumped out from the opposite end of the tank. The effluent is then neutralised to pH 9.2 using sulphuric acid prior to release (Aubé and Arseneault, 2003).

A feature of this system is that the treated water is pumped from the tank using a floating barge in order to ensure that only clean surface water is pumped. If pumping speeds in and out of the tang ranges it will produce a variation of the level in the tank (Aube B. et al., 2003).

As advantages can be mentioned the improvement of the mixing, a significant efficiency increase of lime action and the possibility to stop the discharge.

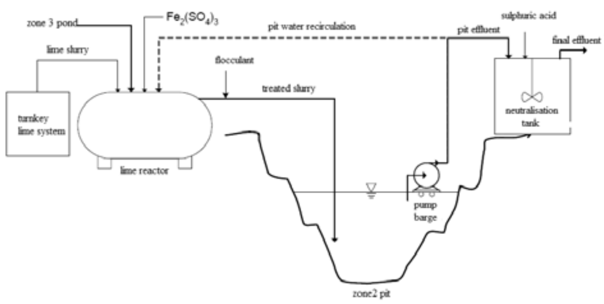

Figure 2: Tank treatment system

As disadvantages can be mentioned the possibility of reforming of the suspension in windy conditions also the potential for uncontrolled addition of contaminated water from the tank walls and from seepages entering the tank and the presence of colloidal particles raise problems in sedimentation.

1.3. Conventional plant treatment of mine water

It applies the neutralization of mine water in a mix tank by controlled addition of lime to achieve the desired pH set point. In the slurry a flocculant's is dosed and in this state it supplies a clarifier for solid – liquid separator. The sludge is collected from the clarifier and can be pumped into a storage area or it can be subjected first to a filtration in the pressure filter in order to increase density before shipment. The clarifier overflow can be directly discharged but more often a passage through a sand filter or a pond is used to reduce residual suspended solids content. If upsets occurs in the clarifier, power can be stopped avoiding improper discharge of water.

An advantage is that the sludge formed by a conventional treatment plant is of less than 5% solids, while an HDS process will often create sludge of more than 20% solids. The sludge of such low density requires significant pumping and storage, particularly if the metal concentrations in the mine water are high. High metal concentrations result in high solids formation and increased sludge production. Another advantage is a greater efficiency of lime.

1.4. Simple sludge recirculation systems

In such systems, the sludge recirculation is made from the base of the clarifier to the point of neutralization. (Figure 4). The system has a number of advantages over conventional treatment system: reduced scaling oxidation in reactors, improved solid – liquid separation, lower lime consumption, sludge with higher density

In this case, the particles grow, but the precipitates formed are similar to those obtained in conventional processes. It is easy to apply this system for plants with conventional treatment to improve the performance of filtering. This requires a recycling pump and pipes.

While pond or conventional treatment processes will form a sludge of less than 1% solids to 3% solids, the Simple Recycle can form sludge of up to 15% solids. This is a significant advantage, but if sludge storage space is critical, the HDS or Geco Process should be considered.

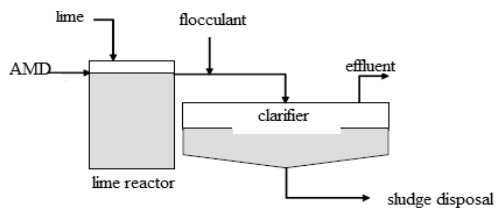

Figure 3: Conventional Treatment Plant

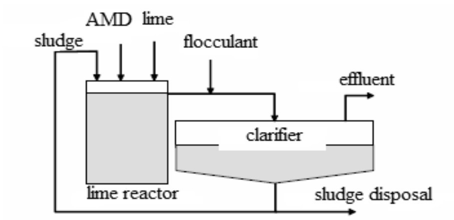

Figure 4: Simple sludge Recirculation Process

1.5. HDS systems for mine water treatment

Lime is dosed in the recycled sludge. This mixture is then discharged into a fast mixing tank where a tight control of pH is realized. The neutralized mixture feeds the lime reactor where the precipitation reactions are completed. Aeration is applied in this stage in order to produce the oxidation of ferrous iron to the ferric iron. Further the flow is passed through a flock tank where in the presence of a flocculants; the agglomeration of all precipitates takes place and promotes efficient settling in the clarifier. The obtained sludge may be downloaded or recycled, and the clarified water can be discharged.

The particularity of the system consists in the mixing of the recycled sludge with lime before neutralization. The fact that the calcium hydroxide comes in contact with recycled particle favors the precipitation on the surface of the existing particles with positive impact on their size and density. Precipitates formed in this case are different from those resulted in the previous cases, fact that can be observed microscopically. (Aube B. et al., 2004)

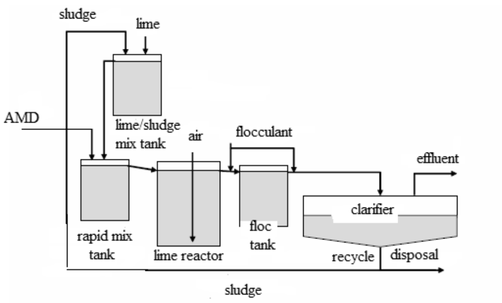

Figure 5: HDS Process

1.6. The Geco system for mine water treatment

In this system mine water treatment, the sludge from the clarifier is recycled to the first reactor where it comes in contact with the mine water (Figure 6). This reactor has a retention time of at least 30 minutes to allow dispersion of solid particles and precipitated in the newly powered mine water. A pH control is achieved in a rapid mixing tank by dosing lime. Reactor 2 has a retention time of 40 minutes needed to produce precipitation reactions and the oxidation of ferrous iron to ferric iron. After flocculant's dosing, the slurry reaches the clarifier in order to realize the solid – liquid separation.

Analyzes on the solutions presented in this first reactor showed that most heavy metals present in mine water precipitates almost completely (95%). Since all reactions occur on the particle surfaces, the particles will tend to increase at every contact with mine water mine and therefore increase their size and density. This may be the explanation to obtain sludge over 20% solid content, the denser particles that can efficiently compact.

An advantage of the system is that it doesn't include the sludge-lime tank.

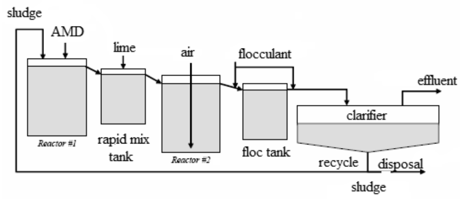

Figure 6 : Geco Process

1.7 Mine water treatment systems with neutralizations in successive stages

The Staged-Neutralization (S-N) process applies crystallization principles to enhance sludge crystallinity and reduce sludge volume (Demopoulos, 1995). This process has been patented in the both the United States and Canada (Demopoulos et al., 1997; Zinck et al., 2001).

The system ensures the neutralization in a series of steps, by controlling the level of super saturation during metal precipitation. The system uses recalculated sludge in the first two reactors for the partial neutralization of mine water. The sludge addition rate should be controlled to reach a desired pH. In reactors 3 and 4 lime is dosed to increase pH to the specified value. To increase sedimentation, a flocculant's is added in order to raise solid particle size. The number of reactors required and the tagged pH value in each reactor are based on the type and concentration of metals existing in the mine water and the required quality for the sludge in terms of density.

Neutralization system in successive steps allows obtaining an excellent sludge in terms of quality and low lime consumption but requires higher investment costs due to the presence of several reactors.

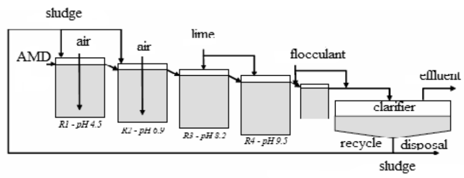

Figure 7: Staged-Neutralisation Process

2. WATER TREATMENT SYSTEMS WHICH USE OTHER PROCEDURES

2.1 Mine water treatment systems based on adsorption

Adsorption process provides flexibility in design and operation, and in most cases it will produce a quality effluent. Even more adsorption by showing reversibility, the adsorbents can be regenerated. A column with a fixed bed may allow contact with activated carbon.

In order to avoid clogging problems for such columns were developed expanded or moving bed columns. In the expanded-bed system, the influent is introduced at the bottom of the column and is allowed to expand. In the moving-bed system, spent carbon is continuously replaced with fresh carbon.

Adsorption on activated carbon, although effective is considered impractical due to high costs of production and regeneration of this type of adsorbent. An option could be the use of composite materials containing this adsorbent.

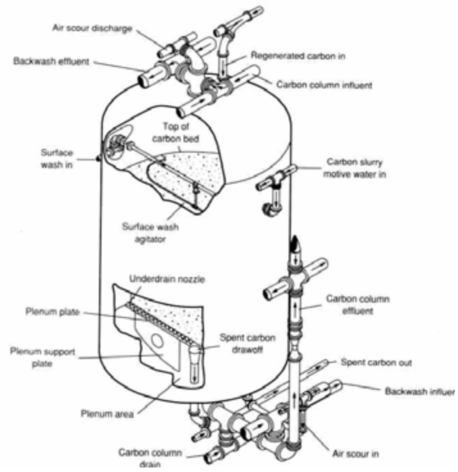

Figure 8: A typical granular activated carbon contactor *(Source*: Metcalf and Eddy, *Wastewater Engineering*, 3rd edition.)

2.2 Water treatment systems for mine based on membrane filtration

Membrane filtration technologies with different types of membranes show great promise for heavy metal removal for their high efficiency, easy operation and space saving. The membrane processes used to remove metals from the wastewater are ultrafiltration, reverse osmosis, nanofiltration and electrodialysis.

Improved ultrafiltration was studied using the micellar process due to the addition of surfactants or polymers. It is recommended the use of surfactants with opposite electric charges to those of the metal ions. As the cost of surfactants can be quite high for technological feasibility it is essential that the surfactant can be recovered and reused.

Water-soluble polymers can be able to complex metal ions and form macromolecular aggregates that can be easily retained on the ultrafiltered membranes. Such complexion agents used in order to retain metal ions are polyacrylic acid (PAA) (Labanda et al., 2009), polyethyleneimine (PEI) (Aroua et al., 2007; Molinari et al., 2008), diethylaminoethyl cellulose (Trivunac and Stevanovic, 2006) and humic acid (Kim et al., 2005). The main parameters influencing this process are metal and polymer type, the ratio metal - polymer, pH, the presence of other metal ions in the water filtered.

The reverse osmosis process uses semi-permeable membranes that allows only the passage of the fluid that is being purified. The main disadvantage of this method consists in a very high energy consumption required to ensure pressure pumping and reconditioning of membranes. Electrodialysis is another membrane process for the separation of ions across charged membranes from one solution to another using an electric field as the driving force.

Electrochemical heavy metal wastewater treatment techniques are regarded as rapid and well-controlled that require fewer chemicals, provide good reduction yields and produce less sludge. However, they require high investment and operating costs, which contributed to a restriction of their applicability to industrial scale.

2.3 Galvanochemical method

Galvanocoagulation relies on the action of a galvanic element found in a solution, such as iron – coke couple. In this case, the iron will tend to dissolve and loads to negative charge (anode) by passing into the solution of its ions, while coke is loaded to positive charge (cathode). The galvanocoagulator is actually a drum which rotates during the operation. Alternant contact between galvanic pairs on one side and the oxygen in solution or air on the other side during drum rotation increase the iron speed of dissolution and facilitates oxidation of Fe^{2+} to Fe^{3+}, ensuring the conditions of formation of iron compounds in higher state of oxidation: hydroxides such as lepidocrocit and ghoetit or oxides such as magnetite Fe_3O_4 and Fe_2O_3 hematite. Method initiators found that ferrous ions are extracted during such treatment by forming metal ferrites and also by cathodic deposition under certain conditions. Ferrites are formed by replacing one atom of iron with one from a nonferrous metal in its cubic structure.

Galvanochemical treatment has a number of advantages such as: ensures the removal of most toxic components; salt content decreases; ensures increasing the amount of water that can be recycled; provides the possibility to recover useful elements from the obtained precipitated.

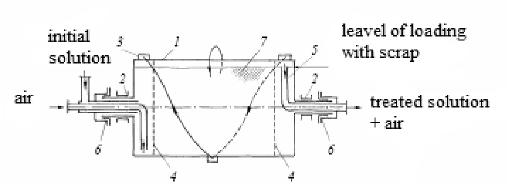

Figure 9: Galvanocoagulator design: *1*) drum; *2*) supporting bearing; *3*) circulation device (inverted loop); *4*) gridded baffle; *5*) rotating tube for adjusting filling level; *6*) seals; *7*) scrap. (Source L. P. Sokolova,1 et al. Accelerating galvanic coagulation in effluents *Chemical and Petroleum Engineering, Vol. 39, Nos. 11–12, 2003)*

3. PASSIVE TREATMENT TECHNOLOGIES

Passive treatment involves the construction of a treatment system that employs naturally occurring chemical and biological reactions that aid acid rock drainage treatment and which require little maintenance. Passive control measures include anoxic drains, limestone rock channels, alkaline recharges of groundwater, and the diversion of drainage through man-made wetlands or other settling structures.

3.1 Anoxic Limestone Drains

Anoxic limestone drains (ALD) are a passive form of alkalinity addition for AMD having a net acidity. ALD's are essentially underground limestone beds through which an unaerated effluent stream, such as a waste rock seep, flows by gravity. As the effluent flows through the system, limestone is dissolved, and calcium and bicarbonate are introduced, adding alkalinity and increasing the pH of the stream. Key to the performance of the drain is the exclusion of oxygen. In the presence of oxygen, metal hydroxides are formed which may armor the surface of the limestone or plug spaces between limestone, making the drain ineffective and subject to failure.

An anoxic limestone drain is essentially a trench filled with crushed limestone, sealed under plastic and geotechnical fabric, and covered by soil through which a contaminated effluent stream flows by gravity.

In general, the size of crushed limestone chosen to construct the drain should be a compromise between allowing free flow and sufficient surface area for dissolution to occur. In most of the ALD's reviewed, limestone crushed to between 2 cm to 4 cm was used.

The dimensions of anoxic limestone drains vary from site to site. ALD's are generally shallow in depth, and contain an effective limestone thickness of 1 m to 2 m which is covered by a minimum of 0.6 m of soil.

Traditionally, ALD's have been narrow in width (0.6 m to 1.0 m), with sufficient length to provide the retention time required to reach chemical equilibrium based on the predicted flow regime. Drains of up to 20 m wide have also been shown to be effective and produced alkalinity concentrations similar to more conventionally shaped systems (Hedin *et al.* 1994).

3.2 Natural and constructed wetlands

By their efficiency wetlands were increasingly required due to: low cost of construction and operation, economy of energy, simple monitoring over time , advanced technology landscape remediation and attractiveness of the area for wildlife.

Wetlands both natural and artificial (constructed Wetlands), operates through biological mechanisms, physical and chemical, supported by elements of the system such as: aquatic plants, microorganisms or soil types or particular substrates used for plant growth. Processes include oxidation, reduction, precipitation, chelating, adsorption, complexation, sedimentation, filtration, active absorption plant, bacterial mechanisms. Wetland plants grow and die continually providing degradable organic matter, organic carbon source.

When the mine water is passing through the wetland heavy metals are progressively released and neutralized. Metals are removed by precipitation, chelating and ion exchange, while neutralization is done primarily by sulfate-reducing bacteria activity and the increase of alkalinity through microbial or chemical reactions, including dissolution of limestone.

Two objectives are taken in consideration:

- Increasing the pH
- Removal of metals: sulphides (CuS, PbS, ZnS, CdS, NiS, FeS_2), hydroxides ($Fe(OH)_3$, $Al(OH)_3$, $Mn(OH)_2$), carbonates ($FeCO_3$, $MnCO_3$, $ZnCO_3$)

There are two types of wetlands: aerobic and anaerobic. Usually a passive treatment system incorporates both environments.

Aerobic systems are designed to maximize the efficiency of the oxidation reactions. Aerobic processes are able to significantly reduce the metals, but these reactions also produce an increase in acidity due to the release of hydrogen ions and/or carbonate consumption.

In contrast, anaerobic wetlands are based on the lack of oxygen and the promotion of chemical and biochemical reactions that generate alkalinity. The reactions occur between saturated water and the substrate depleted in oxygen and not in the thin column of water. Typically, the substrate is made out of material with high organic content as manure, sawdust or compost which acts as a source of nutrients for sulphate reducing bacteria, and as carbonates source, usually limestone (Hedin et al., 1994).

An effective wetland is a battery capable of storing metals and other undesirable products. Such a system may be able to work between 20 and 30 years without interventions to restore.

3.3 Bioreactors

Passive bioreactors are used for more than 20 years in mine water treatment without energy consumption and extreme weather conditions. The system offers advantages such as good efficiency in removing metals at low pH, obtaining stable sludge, low operating costs, minimal power consumption. In addition, building materials are available, ordinary and generally cheap. They are based on sulfate-reducing bacteria found in natural environments where anoxic conditions overbear. Sulfate-reducing bacteria oxidize organic material, produces bicarbonate which increases the pH and alkalinity of water and reduces sulphates present to sulphides in anaerobic conditions. Sulfur ions give insoluble sulfides with present metal ions that can precipitate.

In passive bioreactors mine water is fed horizontally or vertically across a solid reactive mixture (contained in a container or tank) and is then released into the environment. Reactive mixture composition is crucial to the effectiveness of treatment.

Effective reactive mixtures generally contain an organic carbon source, a source of bacteria or a bacterial inoculation possibility, a porous medium, a source of nitrogen and a neutralizing agent. The most important component of the mixture is the source of organic carbon.

In general, sulfate-reducing bacteria use best short chain organic molecules (eg methanol, ethanol, lactic compounds).

The main mechanism for removal of metals from mine water in passive bioreactors is the precipitation of sulfides (Pb^{2+}, Co^{2+}, Cd^{2+}, Cu^{2+}, Ni^{2+}, Fe^{2+}, Zn^{2+}), hydroxide (Fe^{3+}, Cr^{3+} and Al^{3+}) and carbonate (Fe^{2+}, Mn^{2+}).

Sorption mechanisms that occur are adsorption, precipitation of surfaces, polymerization on inorganic or organic support, bacteria and metal precipitates. Co-precipitation with (or adsorption) Fe and Mn oxides and metal sulphides produced by bacteria is another mechanism for removing metals from mine water. (Jong and Parry, 2004) Mine water quality is improved by filtering the particulate matter is removed even colloids.

The first generation of bioreactors generally used as a substrate animal manure or compost mushrooms, to ensure sufficient alkalinity (Dvorak et al., 1992, URS Report 2003). Newer generations of bioreactors are using a mixture of limestone, sawdust and alfalfa instead of animal manure as it provides alkalinity, higher hydraulic conductivity and also seem to be better energy sources for bacteria (URS Report, 2003).

4. CONCLUSIONS

From the technologies presented, we can see that the active technologies provide higher efficiency for the treatment process but with significantly higher costs than passive technologies, which used properly, can provide an acceptable level of decontamination, but large areas of land are needed to ensure appropriate bios and follow-up treatment activities for areas where heavy metal accumulation is achieved.

To choose one of the methods it is necessary for a comparative study to be made, depending on the composition of mine water that needs to be treated, the level of neutralization that can be achieved and the concrete conditions existing in sites.

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STUDY OF SO_2 EMISSIONS RESULTING FROM COMBUSTION OF JIU VALLEY COALREALIZED BY USING FBC TECHNOLOGY

SBIRNA LIANA SIMONA[*]
MOLDOVAN CLEMENTINA SABINA[**]
CODRESI TECLU[***]
IONESCU CLEMENT[****]
SBIRNA SEBASTIAN[*****]
MATEI ARONEL[****]

Abstract: *This paper reports the results obtained for the emissions of SO_2 from a large coal under staged combustion without any additive. The experiments were carried out at fluidizing velocities of 1 and 2 m/s, bed temperatures of 1000-1100 K, 20% and 40% excess air, at a primary air staging of 70:30 and a secondary air staging of 60:40, by using bed particle sizes of 0.2 and 0.7 mm. The effect of each of these work parameters on SO_2 emissions was investigated, in order to draw the conclusions of the study.*

Keywords: *Jiu Valley coal, sulfur dioxide emissions, FBC technology*

INTRODUCTION

Coal, which is primarily used for the generation of electricity, is one of the largest domestic contributors to sulfur dioxide emissions. The public has become more concerned about global warming which has led to new legislation.

The coal industry has responded by running advertising touting clean coal in an effort to counter negative perceptions.

Fluidized bed combustion (FBC) is a combustion technology that is based on suspended solid fuels in upward-blowing jets of air during the combustion process. The result is a turbulent mixing of gas and solids [5].

The mixing action of the fluidized bed brings the flue gases into contact with a sulfur-absorbing chemical, such as limestone or dolomite.

* *Ph.D., University of Craiova*
** *Ph.D., University of Petrosani*
*** *Pr. Math., "St. Stephen" Ecological School of Craiova*
**** *Ph.D., University of Petrosani*
***** *Ph.D., "St. Stephen" Ecological School of Craiova*

More than 95% of the sulfur pollutants in the fuel can be captured inside the boiler by the sorbent. The sorbent also captures some heavy metals, though not as effectively as do the much cooler wet scrubbers on conventional units.

The literature reports that FBC technology involves either two beds in series with two distributors or a simple air staging technique.

In the staged operation mode, the combustion air is separated into a primary air stream.

This constitutes the fluidizing air supply to the bed and a secondary air stream that is injected higher up in the bed or freeboard [3].

Combustion is then completed following the introduction of secondary air. However, overall excess air conditions are maintained in a similar way as in conventional operation.

Air-staging is a proven technique to reduce NO_x emissions, but is known to increase in SO_2 emissions.

This increment is due to the presence of secondary air in the freeboard that allows further combustion (it seems that at high staging this increment is caused by the carryover of unburned fuel sulfur into the freeboard).

The objective of this study is to determine the effects of different work parameters on SO_2 emissions.

One can compare the efficiency of SO_2 reduction with the one obtained if a sorbent is added to the bed (as previously mentioned, a sorbent denotes here a sulfur-absorbing chemical, such as limestone or dolomite).

Bed temperature exhibits a significant effect on SO_2 emissions which indicates that the rate of formation of SO_2 in the freeboard is affected by staging and changes in the temperature profile of the freeboard [1].

In this study, the simple air staging technique was adopted where most part of the total air is injected through the distributor and the remainder is injected in the freeboard of the fluidized combustor.

EXPERIMENTAL

A stainless steel combustor, 0.3x0.3 m in cross section and 2 m high was used.

Fluidizing air was supplied through a multi-hole distributor. An adjustable secondary air injector was used along the vertical axis of the combustor to introduce secondary air in the freeboard.

Investigations were carried out at 20% and 40% of total air injected in the freeboard above the bed, at a primary air staging of 70:30 and a secondary air staging (primary/secondary air) of 60:40, whereas the fluidizing velocities were of 1 and 2 m/s, bed temperatures of 1000 and 1100 K. The bed particle sizes were of 0.2 and 0.4 mm [2].

The fluidized bed was preheated by a propane burner, fixed above the bed, and the fluidizing air flow rate was set at the lowest level to minimize the heating time.

The secondary air was injected through a secondary air injector consisting of a stainless steel pipe with a 15 mm inside diameter, containing twelve holes of 3 mm diameter. This injector was located on the vertical axis of the combustor and its position above the bed or in the freeboard was adjustable. The bed temperature was maintained by using an adjustable cooling coil [4].

Coal sampled from Jiu Valley was used in the experiments.

RESULTS AND DISCUSSION

Coal feeding started when the bed temperature reached 1000 K.
The proximate and ultimate analyses of coal is given in Tables 1 and 2.

Table 1. Proximate analysis of coal used

	Proximate analysis (dry basis)	Weight (%)
Ash	6.23	6.22
Volatile matter	33	33.00
Fixed carbon	60	60.78

Table 2. Proximate analysis of coal used

	Ultimate analysis (dry basis)	Weight %
Carbon	70	77.51
Hydrogen	5	4.8
Oxygen	8	8.5
Nitrogen	1	1 .43
Sulfur	11	15
Moisture	5	5.0

Values of SO_2 emissions were continuously recorded.

More specifically, the SO_2 in the flue and the axial concentration profiles of SO_2 through the combustor were measured for a fluidizing velocity of 1 and 2 m/s; the bed material was sand, of about 0.2 and 0.7 mm size respectively, staging 70:30 and 60:40, excess air 20 and 40%, at bed temperatures between 1000 and 1100 K (the Ca:S molar ratio was found to be approximately 3:1).

The results (*i.e.*, the values of SO_2 emissions – measured in ppm) are summarized in Tables 3 and 4 for the two particle types that were used. As is obviously that the particle size does not practically affect the results, one may reduce the study to the one involving the particles of 0.7 mm.

Table 3. The values of SO_2 emissions (ppm)
for the case of 0.2 diameter sand particles in fluidized bed

	70:30	60:40	70:30	60:40	
1 m/s	627	670	653	700	40%
1 m/s	689	719	703	823	20%
2 m/s	731	781	833	904	40%
2 m/s	818	885	1020	1204	20%
	1000 K	1000 K	1100 K	1100 K	

Table 3. The values of SO_2 emissions (ppm) for the case of 0.7 diameter sand particles in fluidized bed

	70:30	60:40	70:30	60:40	
1 m/s	628	670	653	700	40%
1 m/s	690	720	703	823	20%
2 m/s	732	780	834	905	40%
2 m/s	818	885	1020	1205	20%
	1000 K	1000 K	1100 K	1100 K	

EFFECT OF DIFFERENT WORK PARAMETERS ON SO_2 EMISSIONS

It shows that the SO_2 emissions decreased when the amount of the secondary air was increased and the fluidizing velocity decreased. One may notice that the SO_2 emissions are sensitive to bed temperature, increasing as it increases.

The SO_2 emissions at different secondary air ratios appear to be affected by combustion efficiency.

They exhibit higher values at higher secondary air (which equals to lower air staging) are due to increased combustion of sulfur in the bed and freeboard.

Figures 1-4 illustrate the two charts representing SO_2 emissions as functions of bed temperature at a particular air staging (either 70:30 or 60:40) for a fixed fluidizing velocity (either 1 or 2 m/s) and also for a fixed excess air (either 20% or 40%).

The results demonstrate that the extent of SO_2 emission during staged combustion is influenced by the amount of secondary air and by the bed temperature. More specifically, it increases as bed temperature increases and the primary/secondary air increases (*i.e.*, at higher secondary air).

One can observe that the trends obtained are alike for these charts.

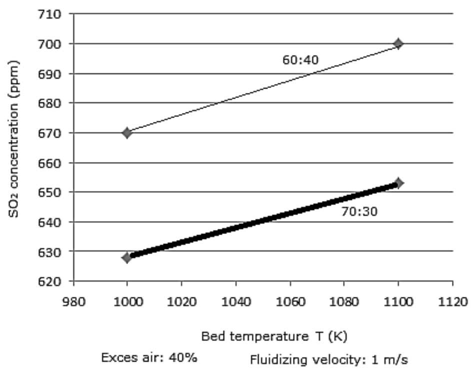

Figure 1. SO_2 emissions as functions of bed temperature at two particular air staging levels (the fluidizing velocity is fixed at 1 m/s, whereas the excess air is fixed at 40%)

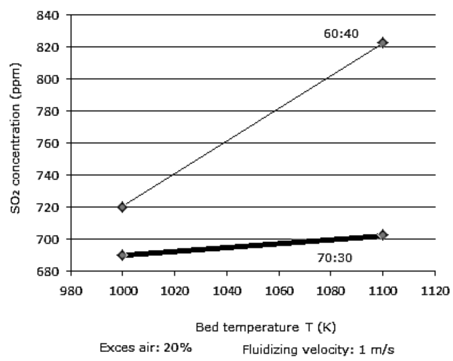

Figure 2. SO_2 emissions as functions of bed temperature at two particular air staging levels (the fluidizing velocity is fixed at 1 m/s, and the excess air is fixed at 20%).

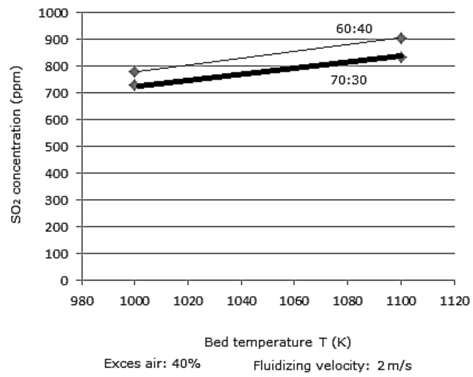

Figure 3. SO_2 emissions as functions of bed temperature at two particular air staging levels (the fluidizing velocity is fixed at 2 m/s, whereas the excess air is fixed at 40%)

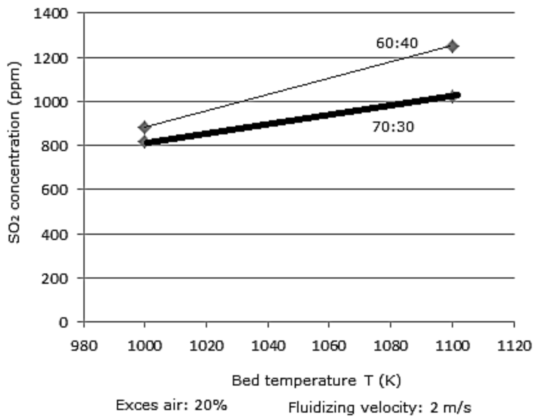

Figure 4. SO_2 emissions as functions of bed temperature at two particular air staging levels (the fluidizing velocity is fixed at 2 m/s, whereas the excess air is fixed at 20%)

The primary air to coal ratio (air/fuel ratio) is defined as the ratio of primary air supplied to the stoichiometric air required, calculated from the coal composition (at 40% staging, an excess air level of 40% resulted in a primary air to coal ratio of 0.8, and 20% excess air resulted in a primary air to coal ratio of 7:10, *etc*.).

At a low primary air to coal ratio, an increase in carryover of unburned fuel sulfur species into the freeboard where it subsequently oxidizes also increase SO_2 emissions.

This indicates that oxidation of some of the sulfur bearing compounds to SO_2 cannot be ignored in the second stage (above the bed).

The air: fuel ratio has significant influence on the rate of sulfur release from the coal and on the ratio of H_2S formed during fuel-rich combustion.

CONCLUSION

The results indicate that SO_2 emissions increase with a rise in bed temperature. The extent of SO_2 emission during staged combustion is strongly influenced by the amount of secondary air and bed temperature.

For a given bed temperature and excess air level, increasing the level of air staging or lowering the primary air to coal ratio causes an increase in SO_2 emissions.

An increase in SO_2 emissions as excess air is reduced was observed at both fluidizing velocities and at all secondary air ratios.

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Scientific Reviewers:
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GIS ANALYSIS FOR SUSTAINABLE DEVELOPMENT OF PETROŞANI CITY, HUNEDOARA COUNTY

AVRAM (BARBUTA) CIPRIANA*
ULAR ROXANA**
MATEI ARONEL***
MOLDOVAN CLEMENTINA****

Abstract: *The sustainable development follows and tries to find a stable theoretical framework for taking the decisions in any situations where it can be found a report of type man/environment, even if we talk about the surrounding environment, or the social or economical one. Known as GIS (Geographical Informatics Systems), the instruments of visualizing and analyzing the geographical information constitutes today a field with a spectaculars evolution.*

1. INTRODUCTION

The sustainable development process of an area implies the existence of an efficient informatics management system implemented by the local community. This system allows the management analysis of special data related to studied area.

In this paper we presented the process of implementation and interpretation process of a GIS in the city area of Petrosani, Hunedoara County, a system on which can visualize, query, manage, search certain areas from different perspectives related to the reality in the field.

2. STUDIED AREA

Petosani city is located in the center area of Romania, in the south of Hunedoara County, to the confluence of East and West of Jiu river. The administrative territory of the city has an area of 19.556 hectares, containing also a few villages: Slătinioara, Peştera Bolii, Dâlja Mare and Dâlja Mică. The GPS coordinates of the city are located between 45 024' 44" N and 23 022' 24" E.

The landscape of Petrosani is mainly depression being surrounded by mountains. To the East Parang mountain is located, Godeanu mountain to the West, Retezat mountain to the North and to the South we have Valcan mountain. The altitude of the city is between 615-620 meters, the highest limit of the area being reached by Parangul Mare peak: 2.519 m.

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* *Student PhD. at University of Petroșani*
** *Assist. PhD. eng. University of Petroșani*
*** *Assoc. Prof. PhD. ch. University of Petroșani*
**** *Lecturer PhD. ch. University of Petroșani*

On the administrative territory of the city there are 2 nature protected areas: "Momarlani" in Jiu Valley and "Piatra Crinului" having a surface of 0.5 hectares located on the South-West side of the Parangul Mic peak on the altitude limit of 1.757m being the only reservation in the country where the *Potentilla haynaldiana* species exists.

There are two categories of natural resources as follows: coal and wood. The mining perimeter of the city has a reserve of 430 million tones, not taking into account the unexploited reserves. The wood stock of the territory covers 166.570mc.

3. OBJECTIVES

The objectives of this study are: the data acquisition from the field (raster and vector), the DEM and contour map generation, the land analysis through maps of the slope, its aspect and gradient, creating and interpreting certain theme maps (watershed).

4. METHODOLOGY

Different types of data were used and stored in the specific database. Thus were used: satellite data, data from the field, data from analog support followed by the process of georeference and digitize.

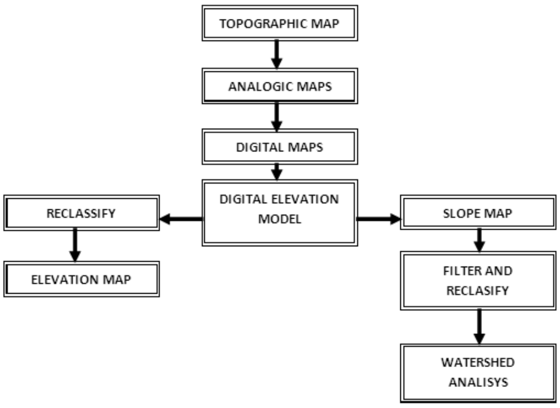


Fig. 1 The methodology scheme

In this application, there were defined a series of thematic layers of which the most important are the following: land covering (Corine Landcover2000), altitude (DEM), land configuration, constructions, roads, rivers, inside and outside built – up areas, contours, watershed, surveying data, raster images etc.

The methodology followed was:

- **Collecting and representing the data from the field** in CAD format and exporting them in shapefile format in ArcGIS**;**
- **Scanning, georeferencing and digitizing** of the collected data from analog format in CAD and exporting them in shapfile format in ArcGIS;

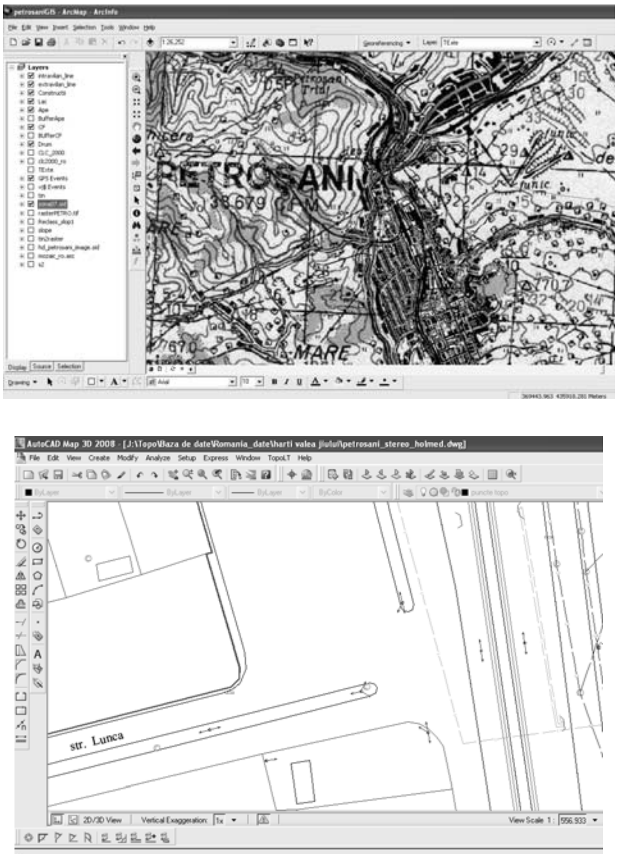
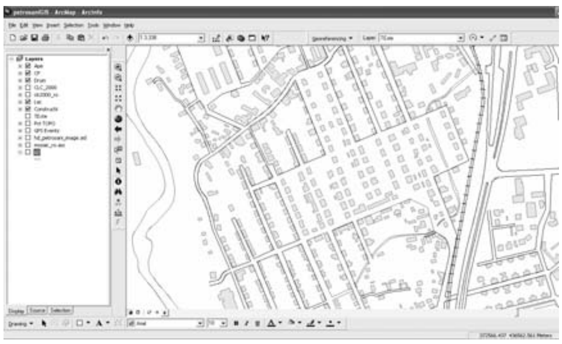


Fig. 2 The acquisition of analog data and field data

- **Executing the thematic layers based on vector and raster data;**



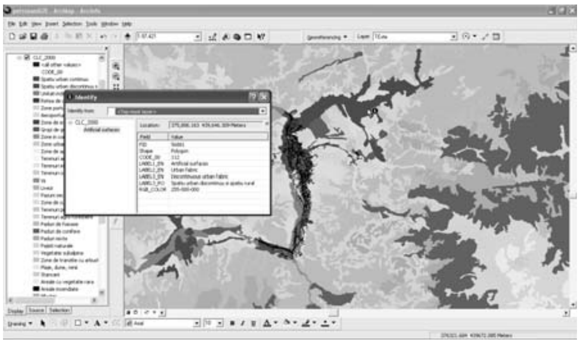


Fig. 3 The Vector and Raster formats imported in ArcGIS

- **The classification and representation of the categories of land use**

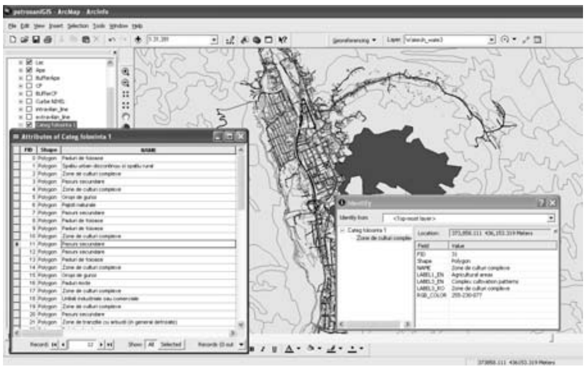


Fig. 4 Example of connecting the raster data and non graphic data

- Collecting and representing of satellite data in GlobalMapper software and exporting them to shapefile format in ArcGIS**;**

- Generating and importing the digital model of the terrain in ArcGIS which also contains the digital elevation model of the terrain;

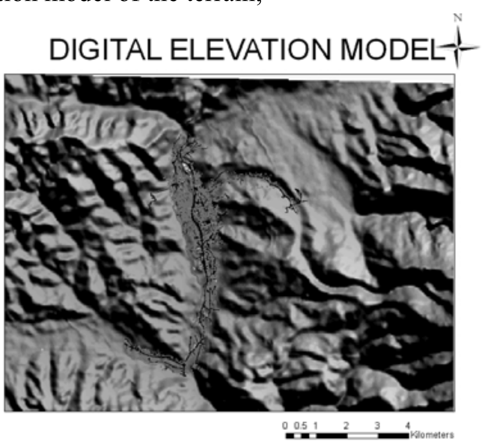


Fig. 5 The digital elevation model

- Generating the contour map;

From the imported image with a resolution of 65,69×93,00 m, contour lines were generated with 10m equidistance in GlobalMapper with the instrument *Generate Contour* then being exported to shapefile format in ArcGIS.

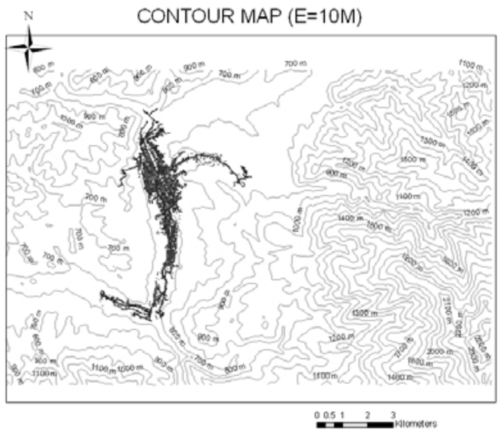


Fig. 6 The contour map in ArcGIS

- The surface analysis using different thematic maps (map of slopes, aspect and gradient);

In order to represent DTM model, a TIN (Triangulated Irregular Network) model will be created by merging the closest points, and on these lines interpolate the values of the elevations with the purpose of approaching the reality in the field.

Thus, in order to create a TIN layer we have to choose the height indicator which is the elevation layer.

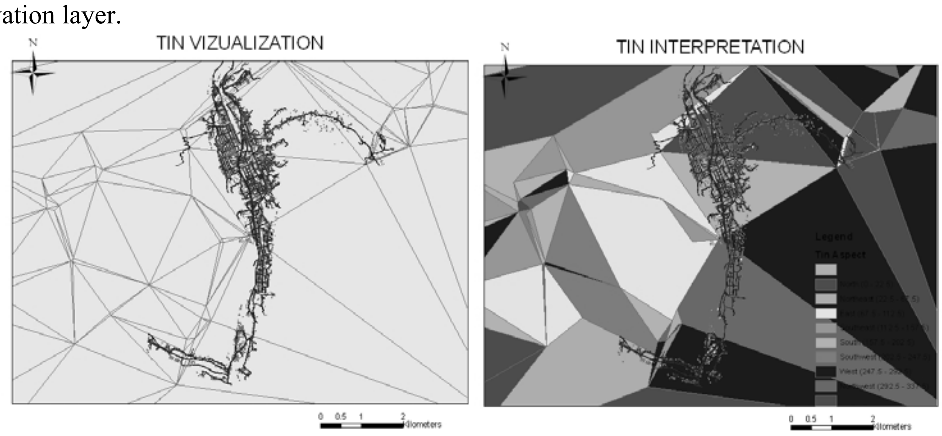


Fig. 7 Creating and interpreting the TIN model

The analysis of DTMs 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.

Slope is normally expressed in planning as a percent slope which is the tangent (slope) multiplied by 100.

Percent Slope = Height / Base * 100

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.

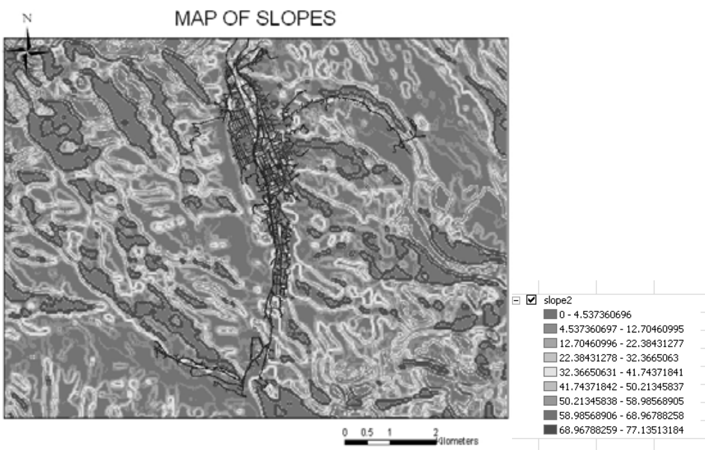
The formulas are:

$$Gradient = \sqrt{\left(\frac{\Delta z_x}{\Delta x}\right)^2 + \left(\frac{\Delta z_y}{\Delta y}\right)^2} \qquad Aspect = \tan^{-1}\left(\frac{\frac{\partial f}{\partial x}}{\frac{\partial f}{\partial y}}\right) \qquad Curvature = \sqrt{\left(\frac{\partial^2 f}{\partial x^2}\right)^2 + \left(\frac{\partial^2 f}{\partial y^2}\right)^2}$$

An example of terrain analysis is **watershed analyses** based on Digital Terrain Model in order to establish the water flow direction and the flow areas towards certain studied area. The calculation of the water flow is based on the flow algorithm of the water in 8 points (D8) in every location correlated with a downside-up approach in order to establish the flow in flat areas and an algorithm of automatic filling of the depressions in the terrain.

A *watershed* is used for this purpose in order to determine the catchments area, by identifying the steepest downhill path extending from an area of interest.

This analysis can be done in ArcGIS with the spatial analyses tool *Hydrology – Watershed* from Spatial Analyst Tool menu.



Descriere	Clasa (%)
Aproape plat	<1
Panta foarte mica	1--3
Panta mica	3--7
Panta moderata	7--15
Abrupt	15--25
Foarte abrupt	25--45
Foarte foarte abrupt	>45

Fig. 8 Map of slopes

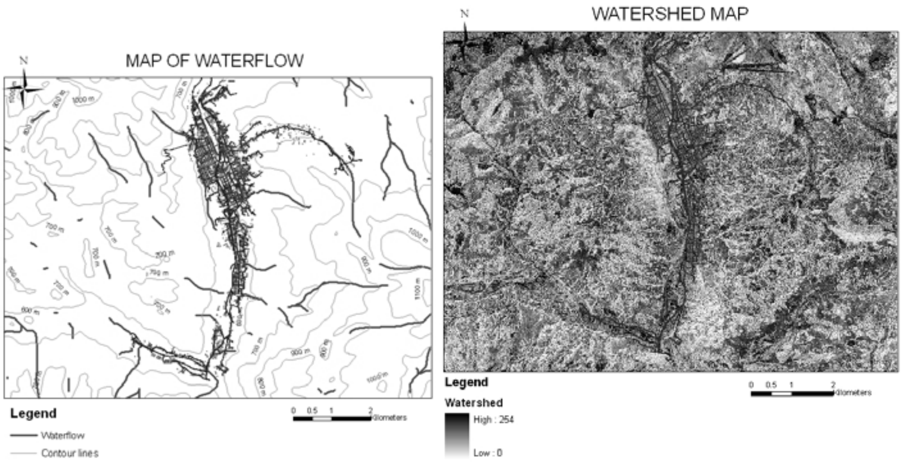


Fig. 9 Watershed map

5. CONCLUSIONS

Many of the uses of DTMs are implicit in their modeling and interpretation techniques. DTMs represents the starting point for calculating some morph metric elements of the relief and making the digital geo-morphological maps and also for spatial and mathematical analyses, methods specific for GIS in order to solve some theoretical and practical problems of geography and not only of it.

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Scientific Reviewers:
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SOLUTIONS OF FERTILIZATION ON DEGRADED SOILS DUMP BOHORELU

DUNCA EMILIA[*]
CIOLEA DANIELA-IONELA[**]

Abstract: *Tailings resulting from excavation of Roşiuţa Quarry was transported and stored in external dump Bohorelu Valley with the help of the dump. From field and laboratory studies were identified undeveloped and degraded soil types typical mining perimeters. In this paper the based on studies conducted to research the possibility of restoring the quality of these soils using green and organic fertilizers.*

Keywords: *degraded soil, restoring, mining perimeters, organic fertilizers*

1. GENERAL FEATURES DUMP

Bohorelu Valley, has a very irregular configuration as for the main thread of the valley, valleys converge more auxiliary located on the left side and on the right. The general slope of the valley is about Bohorelu. 7.5 to 10%, its slopes and valleys of side slope with 8-12o.

External dump tailings deposit in Bohorelu Valley was made from the bottom up and from upstream to downstream, as the specific technological process dump construction in valleys.

Tailings dump was done with two lines of dump, a dump outside line and a line of dump Bohorelu inside.

The design flow Bohorelu technological Valley dump to take account of conditions and configuration that presents this valley. Figure 1

Geometrical parameters considered when designing external dump tailings are: total height of pile: 90 m dump height steps: 15 m, number of dump steps: 6-speed general downstream slope angle of the dump: 6^o; natural slope angle gear dump: 33^o.

Area occupied by Bohorelu dump is about 552 hectares.

2. DETERMINATION OF MATERIAL FROM DUMB QUALITY

For morphological and physico-chemical dump accomplished a number of 7 soil profiles from which samples were collected. [5]

Current analyzes to characterize soil (pH, humus content, nitrogen, phosphorus, potassium, sieve analysis), determination of heavy metals has been carried out in accordance with current rules and interpretation of analyzes was based on the methodology execution soil studies developed by ICPA Bucharest. Following the field trial was considered that the deposits

∗ *Lecturer Ph.D.Eng. at University of Petroşani*
∗∗ *Assist.Prof. Ph.D.Eng. at University of Petroşani*

of waste dumps in conservation are very heterogeneous in age, composition, size and structure. However, deposition of tailings dumps is very uneven in that area and volume.
Dump material consists of sand, loamy sand, clay, clay sandstone, clays, sandy clays.

Due to high heterogeneity, deposits of waste dumps have deep erosion resistance, which makes torrential phenomena installed on slopes are rapidly evolving.

Existing coal fragments in these deposits have an influence on humus content determined. Morphological and physico - chemical properties are determined by the nature of the deposits, their thickness and stage of transformation.

Land filled materials analyzed are characterized by medium-coarse texture, rich in skeleton and low fertility elements. Larger amounts of humus and nitrogen reported in some samples are caused by fragments of charcoal found in the warehouse.

In the analyzed soil samples are presented in Table 1, morphological and physico - chemical characteristics for both Bohorelu Valley dump in conservation.

Chemical and physical analyzes were also established, and the contents of nitrogen, potassium, the rock dumps, because some of these elements have a great influence on plant growth and crop. [5]

Soil sampling was done to a depth of 0-20 cm (from profiles made of 0.2 m); samples from each stockpile (2 kg sterile material from each sample) were mixed together in a sample and in the sample were extracted by sieving 2 kg material with a particle size less than 20 mm.

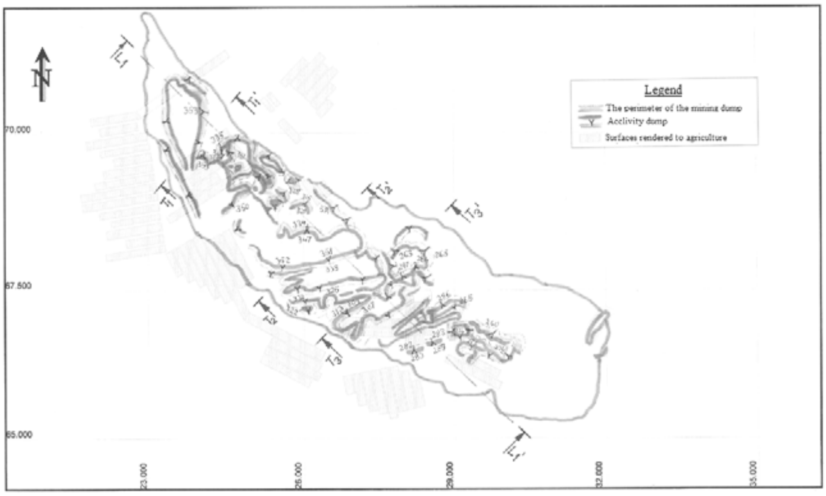

Figure 1 Valley dump Bohorelu

3. STUDIES AND RESEARCH FOR SOIL FERTILIZATION

Depreciation processes of land degradation are produced from natural factors may have an important influence on the onset and intensification of these processes. [6]

The destruction of forest vegetation in areas with high slopes, followed by deposition of tailings, favoring the production of high discharge storm water on slopes and that the onset and development of intense erosion processes f the land affected by erosion processes runs counter work, especially fixing soil with vegetation on slopes favoring water retention, and reduces leakage rate decreases to halt the erosion.

Table 1 Physico - chemical analyzes U.S. 1

Profile	Horizon and depth (cm)		pH	Humus	N total	P2O2 mobile mg/100mg	K2O mobile mg/100mg	Granulometric analysis					Texture	Hygroscopic
								Coarse sand	Sand	Powder I	Powder II	Clay		
US 1	Ao	0-20	5,3	2,6	0,11	4,5	12,6	35	32,1	6,5	10,5	15,9	LN	3,26
	El	20-40	5,4			1,5	8,8	35	38,8	7,3	11,2	15,7	LN	3,46
	Bt	40-55	5,3					26	26,1	6,8	10,6	30,5	L	6,05

Table 2 Soil sampling was done to a depth of 0-55 cm

Profile	Horizon and depth (cm)		SB me	SH me	T me	V %
US 1	Ao	0-20	7,6	5,0	12,6	60,3
	El	20-40	7,4	5,0	12,4	59,6
	Bt	40-55	7,6	7,0	14,6	52,0

3.1. Application of organic fertilizers

Through the use of manure as fertilizer, studies show that when the application on the ground is particularly important.

Litter fall is given to the work of the soil (by plowing the furrow return) in favorable weather conditions, especially during rain and little wind. Table 3

As the waste is spread, the land is plowed plow, which mix and incorporate well trash. Incorporation is deeper by 30 cm, light land dry areas.

3.2. Conditions of application

Quality work soil manure management is considered good when the ground is covered evenly and given material remains in aggregate greater than 4-6 cm. [7] For the application of manure mechanized machines are used to apply manure.

When applying manure mechanized material should be well mixed during loading, free of impurities and foreign matter and waste bunker coat machine used is uniform in thickness.

Manure Efficiency is increased when given with mineral fertilizers, especially with the phosphate. This allows dose reduction of nitrogen by 20 - 50% without increase production to decline. Table 2

During administration, should be avoided as the material used to reach water sources for this purpose is necessary to prevent fertilization on field parts 5-6 m wide, in the vicinity of canals, rivers or other water bodies, to consider the weather conditions and soil moisture status.

3.3. Improvement soil by green manure

Green manure can be applied on any type of soil, but the increased efficiency and sandy soils luvic. Incorporation depth is between 18-25 cm, depending on soil moisture, plant mass volume, etc.

To facilitate incorporation is recommended threshing culture, and when the table is very rich vegetable and stems are long, you should be crushing plant with a discussion table.

It is better to take into account the timing of incorporation and recommendations on the optimal stage of growth of the culture used as green manure. Eg lupines and peas, the best time of incorporation in soil coincides with the stage when pods are formed.

At vetch, sweet clover, mustard, rapeseed, buckwheat, clover finely optimal moment of incorporation in soil coincides with that of flowering, the time is optimal gray rye and sunflower to form the head.

Table 3 Doses recommended manure application in soil annual (t/ha)

Culture	The degree of fermentation	The steppe zone			Forest steppe zone			Forest area		
		Soil texture								
		easy	middle	heavy	easy	middle	heavy	easy	middle	heavy
Vegetable crops	Well fermented	30-40	35-40	40	30-35	30-40	35-40	30-35	30-35	35-40

Table 4 Use coefficients (%) of nitrogen, phosphorus and potassium in manure (with litter)

Year	N	P_2O_5	K_2O
Year 1	0,35	0,45	0,65
Year 2	0,25	0,15	0,15
Year 3	0,10	0,05	0,00
Effect of total	0,70	0,65	0,80

3.3.1. Green fertilizer crop

Green fertilizer crop is an ecological fattening and soil protection, and information about the role, importance, how management culture, etc. are abundant in the literature. (Figure 2)

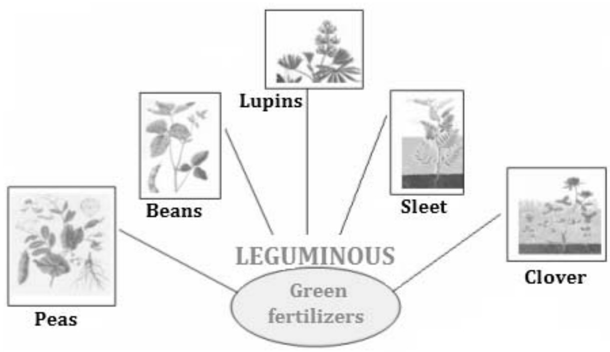

Figure 2 Green fertilizers

Green fertilizers are special culture, which is incorporated into the soil to reach a maximum dry matter accumulated in vegetative parts, often in flower, to improve the soil.

In addition, these crops are designed to protect its surface, which, in between two cultures, remains uncovered by vegetal cover crop and exposed climatic factors, especially weather (E.g. heavy rain at speeds exceeding the infiltration rate of the soil and causes erosion) and wind (wind erosion, especially in less structured soils).

A green manure crop has many advantages:

- *help increase the structural stability of the soil*, incorporating fresh organic matter rich in carbohydrates, allows quickly promote proliferation of microorganisms in the soil which increases soil resistance aggregates; they will better withstand the impact of weather as rain, especially heavy rain;

- *do to increase or at least stabilize soil organic matter content*: green manure in soil contributes about 15% dry matter stable (humus), is production of 2.5 tones dry matter / ha resulting 375 kg humus / ha;

- *maintain soil moisture*;

- *reduce the phenomena of surface runoff or leakage*;

- *protect fragile soils susceptible to compaction or erosion, surface runoff and crust*; during the growing season of crops as green manure, vegetal cover format they protect soil against direct impact of raindrops and the roots contributes to the fixation and retention of aggregates and soil particles;

- *limited drainage and nitrogen leaching into groundwater aquifers sails*; nitrogen unused by the preceding crop is partially recovered and returned to the soil at the time of incorporation of green manure; nitrogen absorbed and assimilated by plants grown as green manure depends on many factors, such as green manure crop species, soil nitrogen availability, nitrogen fertilization possibly stage green manure.

It can be estimated that the amount of nitrogen absorbed between 30 and 50 kg/ha. [7]

Depending on the periods in which these green manure crops can be established encountered the following situations:

- *summer green manure*, occupations a short period of time during the warm term. These cultures appear as a niche in the crop rotation and help improve soil conditions, especially for the poor and to prepare the soil to settle perennial crops.

- *winter green manure or cover crops that are set up in late summer or autumn to protect sensitive soils*, especially in regions with precipitation as rain in winter. They are often chosen for this purpose legumes and grasses (clover, vetch, ryegrass and wheat). Often these crops serve as "traps" for nutrients likely to be leached to groundwater aquifers sails.

Living mulch or hidden cultures under annual or perennial crop. These crops are designed to suppress the growth of weeds, reduce soil erosion phenomenon and improve water infiltration into the soil. An example of living *mulch* may be the last growing vetch over seeding with a corn crop. In perennial crops often use culture of such species in orchards and vineyards, or nurseries. [7]

Some forage crops can act as protective cultures cultivated area during employment. To determine the maximum benefits for soil, crops, mowed or grazed before not to gain maximum biomass.

In table 5 are some species used in the production of green manure crops, soil protection or trap crops, many of which are also very valuable forage plants:

Table 5 Species used in the production of green manure crops, soil or crop protection trick

Legume family	
Lucerne *Medicago sativa*	Fodder plant, forming a rich vegetative mass on the surface of the soil, the root develops a caster that can permeate the soil up to about 2 m. Fixes atmospheric nitrogen by bacteria of the genus *Rhizobium* that lives in symbiosis.
Grain *Vicia faba*	Fodder plant that performs a significant amount of biomass. Fix atmospheric nitrogen in symbiosis with bacteria of nitrogen fixers.
Case *Lupinus* sp.	Fodder plant and green fertilizer in main culture or as a second crop, sown directly into stubble, light-textured soils, poor in organic matter, with a poor structural stability and slightly acidic reaction.
Red clover *Trifolium pratense*	Fodder plant and improvement of soil fertility by fixing catches (about 120-200 kg/ha/year); the root system contributes to the structuring of the soil.

Crimson clover *Trifolium incarnatum*	Annual or perennial plant, which can produce about 4.5-5.0 t/ha on average biomass inflorescences of red-maroon beautify sole grown and meadows, contributing to the plus-value landscape; very good ameliorative for soils.
Medick *Medicago lupulina*	Plant that can produce a significant amount of biomass; as a green manure is incorporated into the soil at blossom; It is preferable to hidden cultures.
Chickpea *Vicia sp.*	It has forms of spring and autumn; has the fast-growing and produce large quantities of biomass; is improving soil fertility as a result of nitrogen fixation, like other leguminous.
Family Hydrophyllaceae	
California Bluebell *Phacelia tanacetifolia*	Rich in protein, plant produces a large amount of biomass in a very short time; It is a very good plant and honey bee; It is often used as a fertilizer; You can incorporate into the soil or you can chop and composted; the flowers are fragrant; contributing to the beautification of the landscape.
Family Gramineae	
Rye-grass ryegrass *Lolium multiflorum*	Can be grown in pure culture and mixed with red clover or clover, when the role of ameliorative soil plant grows. Has the fast-growing and produce a large amount of vegetative mass.

3.3.2. Role of leguminous plants

The most important nitrogen source for obtaining natural way is the cultivation of legumes, the roots of which are formed by nodule bacteria living in symbiosis with leguminous plants (*Rhizobium*). [7]

Initially the plant parasitic bacteria live using its hydrocarbon substances. Later, she gives the plant the nitrogen absorbed from the soil air. The bacteria enters the early roots where multiply, forming colonies. Around each colony, root cells multiply extensively, forming nodule.

Thus, after a leguminous crop, the soil is rich in organic nitrogen from these nodules. The soil analyzes established that after peas main in soil over 50 kg/ha nitrogen as beans over 65 kg/ha, after soy than 100 kg/ha, and after perennial legumes like alfalfa, soil remain 200-300 kg nitrogen/ha.

It follows that a land use properly applied with a good soil aeration and moisture, with the introduction of legumes in the rotation, you can take in large quantities of nitrogen to obtain organic crops, without appeal to the chemical industry is polluting and expensive. It is important that not to lose nitrogen through leaching in deep layers in the atmosphere or through volatilization.

4. IMPROVEMENT OF THE SOIL THROUGH CROP ROTATION SYSTEM

The dump Bohorelu it is proposed to improve the system with tree field that led to crop rotation. In simple rotation have succeeded or temped and drills, and each of these double rotation crops occupied the two years. Yields and weed control were higher in the double rotation.

Table 6 Crop rotation scheme with improving sole for soil degraded Valley Bohorelu

I. Crop rotation of 3-year	Crop rotation of 3-year
Wheat + clover	Wheat+ clover
Clover	Clover
Corn	Wheat + barley
Wheat+ barley	Corn
Corn	Annual technical plant legumes (peas, soybeans, muslin, flax)

Production capacity of the dump Bohorelu depended on the rotation correctly planned and executed in a manner consistent. Because the surface is a very large heap has opted for the system with three fields. In such holding plots were grouped close together in order to achieve a rotation in the intensive rotation pulses, potatoes, tech plant. The plots on the most fertile soils are grouped in a rotation of the plant which grows best in stationary conditions, lands that are required and that have an accessible outlet. [3]

Sole quality of the soil with weeds and with an attack of diseases and pests has been grouped in a special field, applying a rotation in which he introduced the culture of plants ameliorative soil. Ameliorative are plants like the second culture (green field) usually in combination: *autumn rape and vetch autumn; sweet lupines and seradela; seradela and grasses; white melilot and grasses; white clover and grasses; red clover and grasses.* Of the grains used in mixtures were rye and oats.

The second culture (green field) was set up as follows: The wheat was harvested with a combine equipped with straw chopper. Work the soil to the depth of 15 cm with strong grower growing total. He left a field of Black 2-3 weeks for the germination of weeds. Volunteer plants was destroyed with the disc, and in the end it was sowing the mixture of improved crops. [3]

The advantages of our green land are multiple. In the first main crops production increases with 20%.

Another advantage is that there is no need to pull the plow and sloughing so fuel consumption per hectare by 30%. And that due to soil loosening and corrupting the land of green field.

To restore the quality of the soil on the dumps In Bohorelu, has proposed a rotating model with improved plant without having to work with the plow:

Year I: *beet sowing;* growing; worked with the shows; sowing culture.

Year II: processed soil by growing; *sowed corn*; hacked haulms; processed soil by growing; *sowing wheat*.

Year III: harvesting wheat, processed total soil by cultivation, worked with polydisc, sowing the second culture of mustard or radish fodder left over the winter culture.

Year IV: chopped mustard drive dry time in order not to leave traces; the total growing to 4-5 cm; *sowing sunflower*; haulms minced of the flower; worked with the multiplies clutch; *sowing wheat*.

Year V: wheat harvesting; growing; worked with the shows; *sowing California Bluebell*.

Year VI: chopped California Bluebell; growing; *sowing maize*.

It is important to know the principles developed by science and confirmed by practice rotation in crop plants.

In the rotation the following types of cultures meet: first culture; culture prior to the secondary or successive crop.

Main crop provides main production intended for marketing. It may be early crop varieties and hybrids early); late (varieties or hybrids tardily); double (after a crop to be harvested early); the autumn or spring. *Secondary culture* is a culture is done between two main crops and is used as fertilizer or green.

Culture pre-before is a culture that the basic crops are grown on the same plot.

Successive culture is the culture to be grown as soon as the basic culture on the same plot. It is preferable to cultivate crops which are successively stimulates and complement each other. [3]

-nitrogen consuming crops after crop nitrogen fixers;

- consuming hummus since plants that increase soil humus reserve;

- herbs that reduce the leavening, the plants that contribute to the rising;
- plants with shallow rooting culture after reaching deep.

4.1. Effect of crop rotation on soil

Crop rotation has direct effects on the *physical* and *chemical* properties and *biological* characteristics of the soil.

In terms of the effect on *physical* properties particularly the influence on soil structure and structural stability.

The structure of soils is influenced by the type of rotation and especially plants that follow each other in rotation. At the same time depends on the type of soil where its structure will be influenced more or less.

Alfalfa, which is a perennial plant for the production of fodder, has a very large impact on the level of structuring (% aggregated establish hydric) the soil in the upper layer, i.e. 26.3-26.4%. This is followed by the monoculture of wheat with values of 21,3%-19.3 for 4 year rotation of wheat and maize, with roughly equal amounts, between 18.6 and 19.4% respectively, 18.4-19,1 and only 17.4-18.5% maize monoculture.

Research carried out on the ground showed that luvic structuring of the soil after 10 years of experimentation, the highest percentage of aggregate hidrostabile in 4-year rotation, the peas (an agrofond 55,9% unfertilized and 54,3% an agro fond fertilized).

In the rotation for 2 years, after corn, hidro-stability aggregates were lowest (48.3% on agro fond unfertilized and 54,3% agro fond fertilized). The lower structure of the soil after the corn was the work of mobilization of soil during the growing season, causing a spray, and the degradation of the structure, compared with legumes or drills that are characterized to restore and protect the soil aggregates. Depending on the physical properties of soil properties and form of relief, crop rotation can act in favor of preserving the productive capacity of the soils and their conservation.

So in the slope lands, often exhibited the phenomenon of water erosion, crop rotation for five years with the sole amelioratoare is one of the important links of the complex antierozionale measures conducive to achieving production increases without effort and at the same time contributes to the reduction of water leakage and the soil on hillsides.

Many researchers say the role of crop rotation in preventing soil erosion and ecological stability of the landscape. Water erosion is the main factor of main agricultural areas, affecting 56% of total arable land worldwide and which has already led to the removal of agricultural output to 430 million hectares or 30 percent of the total arable land available.

Ability to limit soil erosion through crop rotation depends on some morphological properties of plants grown and how they are sown or planted. Of course, this is related to the interval between the rows of plants and the plants of the time, which makes the plant density and, therefore, their resistance to the phenomenon of erosion. The vegetal carpet is denser, much the opposite of erosion resistance will be higher.

Crop rotation can also influence the chemical properties of the soil. Cultivated plants have different requirements in terms of consumption of nutrients so that some can impoverish the soil nutrients, and others can enrich the soil the considerable quantities of nutrients that can be taken into account for the next crop.

Leguminous plants, as mentioned previously, due to symbioses with bacteria from the genus *Rhizobium*, let the ground large quantities of nitrogen. Also, temporary meadows, after their abolition, provide significant quantities of soil organic matter and inorganic slightly significant quantities of nutrients.

5. CONCLUSIONS

In this work we analyzed the possibilities of restoration of soil quality on the Bohorelu Valley dump by using green fertilizers and organic fertilization.

Following field studies and laboratory to identify the types of soils are degraded lands as well as their chemical composition. The analyses were performed on seven profiles, we indicate a poor quality soil, and interpretation of results shows the following:

- oil reaction is moderately to strongly alkaline to acidic,
- the content of humus is the dropped down to middle,
- the level of total nitrogen supply range from average to normal,
- the supply of phosphorus has seen the evolution of cell values from the middle to the good,
- the level of potassium supply digestible variety from the poor to the rich,
- the degree of saturation with bases indicates a ground state evolves from the moderate to strong saturate

Green fertilizers culture is a means of fattening and ecological protection of the soil, these crops should be successfully on sterile Bohorelu Valley dump due to soil type luvic they prefer.

Cultures of green fertilisers present many advantages:

- contribute to increasing the structural stability of the soil;
- to increase or, at least, stabilizes organic matter content of soil;
- green fertilizers in soil with about 15% dry matter (humus), a production of 2.5 tones dry matter/1ha resulting 375 kg/1ha hummus;
- reduce the phenomena of flow or leaking to the surface of soil;
- protects fragile soils, or attuned to press against erosion, run-off to surface and formation of the crust;
- limited leaching of nitrogen and water drainage to the underground aquifers sails;

In situations in which N, P and K are not found in normal amounts in the soil and organic fertilization after achieving targets with chemical fertilizers for crop development and achieve satisfactory yields.

Due to the large variability of crop and soil nutrient balance must be submitted for each parcel or group of parcels of relatively uniform.

Through properly applied to agricultural machinery, with a good aeration of the soil and moisture, with the introduction of leguminous plants in the rotation, you can take in large quantities of soil nitrogen for crops environmental.

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