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THE MODERNIZATION OF THE COAL PROCESSING INSTALLATION IN THE JIU VALLEY IN THE 30s. OF THE XXTH CENTURY

BARON MIRCEA *

Abstract: *The Jiu Valley was, during the inter-war period, Romania's most important coal basin providing over 50% of the country's coal production. Beyond these statistical figures one can speak about the quality of the Jiu Valley coal: coke pit-coal found in the central and Western area of the Jiu Valley; coal having energetic characteristics: energetic pit-coal and brown coal found in the Eastern area of the Jiu Valley. As regards the amount of the coal deposit, specialists consider that we deal with a geological reserve of the Jiu Valley coal basin counting 1.800.000.000 coal tons. In order to bring it to a semi-industrial shape allowing its subsequent direct or processed use, extracted coal is processed by dry or damp preparation equipments; from this point of view one can speak about two stages. The first stage covers the end of the 20th century 2nd decade when separations for coal's mechanical processing were used; nine such equipments functioned in the Jiu Valley. The 1929–1933 economic crisis that strongly affected coal industry too, on the one hand, and the natural evolution of techniques as well as buyers demands, especially those belonging to the Administration of the Romanian Rail - Road Company, on the other hand, were the main factors that determined the Jiu Valley coal companies to begin, according to a vast investment policy, the modernizing process, including that of the coal preparation equipments. This trend implied financial and technical measures that determined the Jiu Valley mining units of the 20th century 3rd decade, and especially "Petroşani" Company, to settle a complex of equipments, including seven groups, to mechanically prepare coal; the complex was unique in Romania. This complex of coal processing equipments allowed the Jiu Valley mining companies to provide their customers – Romanian Rail-Road Company, river and maritime ships, brick manufactures, gas plants, forging equipments, house consumers – various types of coal.*

Key words: *the Jiu Valley, mining companies, coal preparation equipments, the 30s' years of the 20th century*

The Jiu Valley was, during the inter-war period, Romania's most important coal basin providing the most part of the country's coal production. In order to exemplify, let's consider the following data: in 1919, out of Romania's 1.559.000 tons production, the Jiu Valley gave 1.028.934 tons (65.98%); in 1927, when the Jiu Valley provided the highest coal production of the whole inter-war period, namely 1.840.759 tons, it represented 57.10% of Romania's coal production (3.224.000 tons); in 1939 the Jiu Valley gave 60.74 % (1.497.211 tons of coal) of Romania's

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2.465.000 tons of coal[1]. Beyond these statistical figures one can speak about the quality of the Jiu Valley's coal. According to the petro-graphical analysis and to the physical, chemical, and industrial characteristics of the Jiu Valley coal, specialists have concluded that we deal with brown coal and pit-coal. The criterion of differentiating them is their caloric power within the limit of 5700 kcal/kg. Pit-coals are grouped, according to their characteristics and destination, in coke pit-coals and energetic pit-coals. Coke pit-coal is found in the central and Western area of the Jiu Valley: Câmpu lui Neag, Uricani, Bărbăteni, Lupeni, Paroşeni, Vulcan, and displays coke features ranging from poor ones to excessive ones as well as agglutination features ranging from poor ones to strong ones; the basin's mean is average. Energetic coals are the following ones: energetic pit-coals and brown coal found in the Eastern areas of the Jiu Valley: Aninoasa, Livezeni, Dâlja, Petrila, and Lonea[2]. As regards the size of the coal deposit, specialists consider that one can speak about a geological reserve of the Jiu Valley coal basin, representing 1.800.000.000 tons of coal[3].

Two important investors implied in the Jiu Valley. On the one hand, Hungarian State that, during the 19th century 6th decade began its geological research, and prospection, investing in opening coal deposits and Romanian State, after 1918. On the other hand, private capital settled certain mining companies that proved to be quite important for the economic and social destiny of the Jiu Valley: the Company of Mines and Furnace in Brasov (1858-1895); "Salgótarján" Company (1895-1921); "Uricani-Valea Jiului" Company (1892-1925); "Valea Jiului de Sus" Company (1900-1931); "Petroşani" Company (1921-1948); "Lupeni" Company (1925-1931).

Coal detained a leading position due to its various uses that were the result of the researches during the years: as a source of high temperature for industry[4] and of low temperature for urban and house heating; used as raw coal or as semi-coke coal in order to get the driving force used in transports or to produce electricity; changed into lightning gas or into synthetic petrol for hydrogenation; used in order to produce cement, for forging, in metallurgical furnaces with a view of removing impurities, to burn bricks and refractory materials; to produce perfumes and medicines, to get explosives and insulation in electro-technical industry.[5]. Extracted from the deposit, the coal was to be treated in order to bring it to a semi-industrial shape

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[1] Mircea Baron, *Cărbune şi societate în Valea Jiului. Perioada interbelică,* Editura Universitas, Petroşani, 1998, p. 237-238; V. Axenciuc, *Evoluţia economică a României. Cercetări statistico-istorice. 1859-1947*, Editura Academiei, Bucureşti, 1992, p. 218.

[2] Bujor Almăşan, *Exploatarea zăcămintelor minerale din România*, vol. I, Editura Tehnică, Bucureşti, 1964, p. 80-83. Gh. Giuglea, Gh. Mihuţ, Paki Ernest, Roman Petru, in *Centenarul exploatării industriale a cărbunelui în bazinul carbonifer Valea Jiului*, Petroşani, 1968, p. 9, showed that the Jiu Valley coal is, in fact, pit-coal. The pit-coal in the Eastern area of the basin belongs to the category of long flame and gas pit-coal; that in the Western area belongs to the category of gas pit-coal. The analysis of raw coal shows an increase of carbon from the Eastern area towards the Western area, namely from 77.96% to 81.36%, of volatile substances from 42.9% to 45.7%, of sulphur from 1.48% to 3.12%, of the caloric power from 7775 to 8.370 kcal/kg as well as a decrease of hygroscopic humidity from 4.7 to 1.9.

[3] *Ibidem*, p. 8.

[4] The attempts started by Dudley, in England, during the 18th century, in order to make cast iron out of pit-coal were continued by the metallurgists Abraham Darby I and John Thomas; they managed to melt cast iron owing to the coke contained by the pit-coal in a foundry, in 1711; in 1735 Abraham Darby II made the same thing in a high oven; the procedure became widespread in Western and Central Europe. Let's also remember Bessemer procedure of getting cast iron (1856) and Martin procedure of getting steel out of cast iron and recuperated iron (1864) which, as other inventions, determined the development of coal metallurgical industry (*Dicţionar cronologic al ştiinţei şi tehnicii universale*, Editura Ştiinţifică şi Enciclopedică, Bucureşti, 1979, p. 361-370).

[5] Ion E. Bujoiu, *Consideraţii asupra utilizării cărbunilor în România*, I.R.E., Bucureşti, 1933, p. 35-36; I. Simionescu, *Ce se scoate din cărbune?,* Gazeta Jiului, II, 1923, nr. 37, p. 3; xxx, *Ce se extrage din cărbune*, Calendarul minerului pe anul 1935, p. 117-119.

capable of subsequently allowing its direct or processed use. In order to do these coal companies created dry or damp preparation equipments; from this point of view one can speak about two stages.

The first period extends until the end of the 20th century 2nd decade when separations for coal's mechanical processing were used. Accordingly, the State's Mines of Lonea built, nearby Petroşani rail road station, instead of the old Separation of Northern Petroşani, constructed by the Mountaineering Treasury, and used by "Salgótarján" Company until 1908, the new Separation of Northern Petroşani, having an efficiency of 230 tons of coal per hour.

At Lupeni, "Uricani-Valea Jiului" Company built: the Eastern Separation (or the Northern one; 1896); the Southern Separation (1896); "Ştefan" Separation (1896); "Victoria" Separation (1910)[6]. Such mechanized equipments, containing sorting sieves, produced five types of coal: small coal = 0-10 mm, "semolina" coal = 10-18 mm, "nut" coal = 18-35 mm, cube coal = 35-120 mm, clods = > 120 mm; the coal was supplied in various compositions: sieved "salt mine" coal or "salt mine" I which was a mixture of cube coal and "nut" coal; dusted coal or "salt mine" II coal which was a mixture of cube coal, "nut" coal, and "semolina" coal; "salt mine" coal or "salt mine" III coal which was a mixture of cube coal, "nut" coal, "semolina" coal, and small coal.[7].

"Salgótarján" Company built at Vulcan a central mechanized separation having a capacity of 125 tons of coal per hour, operated by an electric motor of 40 HP; the separation took over the whole production of the three mines in Vulcan and, beginning with 1909, of the mine of "Valea Jiului de Sus" Company; it sorted out coal after removing the shale into five types: clod coal (>100 mm), cube coal (70-100 mm), "nut" coal I (30-70 mm), "nut" coal II (14-30 mm), and small coal (<14 mm)[8].

At Petroşani, "Salgótarján" Company built two separations. Initially, it used Northern Petroşani Separation that belonged to the Mountaineering Treasury; yet, after 1908, when it took over its properties, the company was forced to reassess its position and built Eastern Petroşani Separation, in 1910, nearby Eastern Petroşani Mine; it had a capacity of 70 tons per hour and was operated by a 26 HP electric motor. The separation was built in order to separate, owing to Westfalia type mobile sieves, the coal coming from the mines of Eastern Petroşani and Petrila into the following types: 0-10, 10-80, >80 mm. The second separation, the Western one, was situated on right bank of Eastern Jiu River. It was built during the years 1896 and 1897 and had a separating capacity of 90 tons per hour, being operated by a 124 HP electric motor. The separation produced, owing to Westfalia type sieves, coal sorted as follows 0-10, 10-80, >80 mm, and took over the production of the mines of Western Petroşani and Dâlja.[9].

The economic crisis during the years 1929 and 1933 that strongly affected coal industry too, on the one hand, the natural evolution of technique as well as the demands of the buyers, on the other hand, were the main factors that forced the coal companies of the Jiu Valley to respond the challenge and to begin, according to a policy of vast investments, the modernizing process that also included the coal preparation equipments.

It was a major component of the process of "rationalizing" that implied a constant preoccupation to improve the quality of the extracted coal both through mining methods (shooting reducing, selection, manipulation) and, especially, through coal's mechanic preparation. That trend determined financial and technical measures capable to respond the "challenge" of the coal buyers

[6] County Branch of the National Archives Hunedoara (DJANH), *Fond Societatea „Lupeni". Direcţia Minelor (D.M.)*, dos. 4/1925, f. 17-20.

[7] Krizko, Bohus, *Az Urikány-Zsilvölgyi magyar kőszénbanya-resz-tars. Lupényi bányatelepének rövid ismertetése*, Budapest, 1903, p. 28.

[8] *Monografia Societăţii „Petroşani",* Editura „Cartea Românească", Bucureşti, 1925, p. 120.

[9] DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 81/1931, f. 14-15.

and especially of the Romanian Rail-Road Company; at the same time, it aimed at providing coal's complete turning to good account at a convenient price. [10].

The coal companies of the Jiu Valley turned to good account extracted production both through using it for their own needs: power stations, steam boilers, coal allocation for employees, etc., and through selling it to private entities to be used for home needs or in industry; a significant part of it was sold to the administration of the Romanian Rail-Road Company in order to be used by locomotives. The Romanian Rail-Road Company was the main buyer of the coal produced in Romania and imposed its own policy regarding the quality and the quantity of the product it bought, including the coal coming from the Jiu Valley's mining companies. We consider we can firmly assert that the evolution of the Jiu Valley's mining companies was decisively influenced by the buying policy of the Romanian Rail-Road Company; at the same time, a part of the mining quantitative changes, and especially of the qualitative ones, were due to the demands made and imposed by the most important client. To this essential element, we can also add the understanding, by the leaders of those mining companies, that, as coal was not a regenerating source, it should be rationally exploited and especially efficiently turned to good account.

The most important answer of the Jiu Valley mining companies at the justified demands of the Administration of the Romanian Rail-Road Company regarding the quality of the supplied coal was exactly the preoccupation to strictly frame within imposed standards.

The mining companies and the mines that could not face the terms imposed left the competition; the most eloquent example was the situation determined by the application of the Fuel Protocol dating from May 9th, 1931: 41 out of the 58 mining companies and mines functioning after World War I that were nor properly exploited and had no solvability lost their right to supplies while earnest mining companies invested in order to improve coal's quality. [11].

Those mining companies and mines that had supplies contracts with the Romanian Rail-Road Company could be divided into four categories.

To a first category belonged the Jiu Valley's mining companies, especially "Petroşani" Company that made important investments beginning with the fourth decade; such investments allowed the foundation of a complex of equipments meant to mechanically process coal; the complex was unique in Romania[12]: the washing units, the drying units, and the flotation units in Petrila and Lupeni, the briquette unit in Petrila considerably improved vaporizing capacity and the opportunity of using small coal.

A second category included the mining companies that improved the quality only through selecting the coal within the stopes and through its straining. "Creditul Carbonifer" Company and Codlea mining unit took into account the introduction of briquetting as well as the increasing of the amount of small coal that could be turned to good account and the capacity of vaporization of the coal supplied to the Romanian Rail-Road Company. Codlea mining unit introduced in 1935 an equipment of pneumatic washing of the coal that, due to the removal of sterile layer, determined a new increase of vaporization capacity; Şorecani and Surduc mining units introduced simple straining, a procedure that did not determine an important increase of vaporization capacity. [13].

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[10] See, Mircea Baron, Oana Dobre-Baron, *Evoluţia preţului la cărbunele vândut Căilor Ferate Române în perioada interbelică*, Sargetia, XXXII, 2004, p. 609-622; Ioan M. Lăzăreanu, *Întrebuinţarea cărbunilor la C.F.R.*, Miniera, XVIII, 1943, nr. 3, p. 79-81.

[11] DJANH, *Fond Societatea „Petroşani". Direcţia Generală*, dos. 4/1940, f. 12-13.

[12] August Buttu, *Cerinţele consumatorilor de cărbuni din România şi adaptarea instalaţiilor de preparare la aceste cerinţe*, în, DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 63/1937, f. 1-26.

[13] Idem, *Vues retrospectives sur l'industrie extractive de charbon en Roumanie au cours des dernierés 25 années. Progrés realisés. Resultats. Perspectives d'avenir*, Analele Minelor din România (A.M.R.), XXVI, 1943, nr. 5, p. 85; A.M.R., XIV, 1931, nr. 7, p. 321; XV, 1932, nr. 1, p. 26-29.

To the third category belonged lignite exploitations that targeted the improvement of quality through de-hydration. In order to implement that principle, beginning with 1933, the mines belonging to Doiceşti basin as well a part of the mines belonging to Schitu Goleşti basin employed Filitti procedure; most of the mines belonging to Schitu Goleşti basin employed Fleissner procedure, a technology that determined the decrease of humidity, the increase of caloric power, and a using coefficient ranging between 0.59 and 0.63[14].

The last category included the coal exploitations that did nothing in order to improve coal's quality: Lapoş, Cozla, Sălătruc, Jibou, etc. that supplied coal as it was extracted from the mine during the whole inter-war period[15].

It seemed that, in the Jiu Valley, the preoccupations regarding the getting of a higher quality of coal had dated since 1923–1924 when, among other things, letters had been sent to three important German companies producing mining equipments: "Demag" in Duisburg, "Humboldt" in Köln and "Krupp" in Magdeburg, in order to receive data regarding the terms of building, in Petrila, a separation having a higher debit than the existing ones; yet, the project was abandoned[16].

Before World War I Western Europe possessed a mechanical damp method of preparing coal in washing units equipped with hydraulic or piston vats; the method was also employed by the washing unit of the Coke Plant in Lupeni. During the years 1914–1915 this method was replaced by a simpler and cheaper one providing, on the whole, the same results: mechanic preparation in an ascendant water current that determined a preparatory separation according to granule classes having the same falling speed; yet, it never allowed the definitive separation of raw coal into clean and sterile coal.

After World War I, two Belgians, France Fouquet and Habets built Rheo-washing units where the serial, parallel, and cascade arrangement of the washing equipments and pipes as well as the use of a horizontal water current and of an inferior ascendant current determined positive separation results; the procedure needed only one previous sorting[17]. The procedure was largely spread in Europe: Belgium, France, England, Germany, etc.

In the Jiu Valley they acted in order to settle two matters: the separation of shale from coal in order to increase coal's caloric capacity, and the turning to good account of small coal.

"Salgótarján" Company tried in 1912, at the Western Petroşani mining unit, an English equipment; the project was taken over during the years 1923–1924 by "Petroşani" Company that employed a prototype equipment conceived by a Romanian engineer, I. Lupascu; nevertheless it did not determined encouraging results; the water current separated the coal from heavy sterile but not from the shale that had a specific weight differing very little from that of the coal[18].

In 1926 "Petroşani" Company decided in favor of Rheo system[19]. A research was done on the basis of the visiting, in November 1928, of the equipments belonging to four companies in Northern France that functioned according to Rheo–laveur system. The research showed that the system was mostly employed in France and Belgium; in Northern France the whole production having a 10 to 40 mm granulation, most of the cases belonging to the class ranging from 0 to 50 mm, and, quite often those cases up to 80 mm granulation were washed accordingly. Further, the research displayed the reasons of the success of the coal preparing system. The research also showed that flotation could only be applied in case of the coal ranging from 0 to 0.5 mm or maximum 0–1.5 mm but, nevertheless, it allowed a better washing than Rheo–laveur system; as a

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[14] Gr. Socolescu, *Înnobilarea lignitului în România*, in vol., *Al II-lea Congres al Asociației Inginerilor Diplomați ai Scolii Politehnice Bucureşti. 10-12 noiembrie 1935*, Editura „Scrisul Românesc", Bucureşti, 1937, p. 391-414.

[15] DJANH, *Fond Societatea „Petroşani". Direcția Generală*, dos. 4/1940, f. 45.

[16] DJANH, *Fond Societatea „Petroşani". D.M. Secretariat*, dos. 7/1923, f. 827-835.

[17] R. Wüster, *Rheo spălatorul de cărbuni*, A.M.R. VIII, 1925, nr. 8, p. 280-288.

[18] DJANH, *Fond Societatea „Petroşani". D.M. Secretariat*, dos. 6/1923, f. 224.

[19] *Ibidem*, dos. 2/1926, f. 5.

conclusion they suggested that the whole production of coal ranging from 0 to 80 mm could be washed and proposed the functioning of a washing unit having a capacity of 100 tons per hour, in case of Petroşani and Lupeni Groups, and a washing unit having a capacity of 75 tons per hour, in case of Vulcan Group[20]. Those conclusions were completed by the data provided by a second trip to Belgium, Ruhr, and Lorena, on May 1929, which emphasized the progress registered within two years by the Rheo–laveur system, and, especially, its advantages as compared to piston devices. Comparing the costs of a ton of coal supplied by Ştefan Separation in Lupeni, representing 35.79 lei/ton and the costs at a new washing unit having a capacity of 200 tons per hour, representing 8 lei/ton, a difference of 27.79 lei resulted in favor of Rheo–laveur system; at the same time, the amortization of the equipment could be done within two years and a half. [21].

As regards the problem of small coal, the solution regarding its use – especially that the Administration of the Romanian Rail-Road Company decreased small coal's share within the amount of "salt mine" coal it bought – was settled through briquettes manufacturing. In 1923 a series of discussions took place with an engineer of "Gröppel" Company in Bochum (Germany) in order to create briquetting equipment that used petrol pitch as glue. On the basis of the demands of "Petroşani" Company, a series of companies, such as: "Gröppel", "Meguin" (France), "Demag" (Germany), sent their proposals and experiments were made in order to obtain briquettes and to determine their quality[22]; the conclusion was that out of unwashed coal no quality coke could be provided mainly due to its too high content of ashes; hence, the need of introducing Rheo–laveur washing system. Under those circumstances one should have also considered the contacts of "Lupeni" Company with "Indumine" Company in Paris with a view of carbonizing coal at low temperatures and its changing into semi-coke, gas, and tars that could be used for blowing scrapers and in order to get briquettes, using pit-coal pitch as glue[23].

In order to achieve the envisaged program they studied various equipments of separation, washing, and dust elimination made in Europe[24], and looked for the means of implementing that goal included within the ampler process of rationalizing. The 1930 exploiting programs and the program of special works during the years 1930-1931 and 1932 of "Petroşani" and "Lupeni" Companies envisaged the concentration of the activity of coal mechanical preparation within two complex and modern equipments, in Petrila and Lupeni capable to replace the old separations in Eastern Petroşani, Western Petroşani, Vulcan, and Lupeni: Victoria and Ştefan [25].

On August 18th, 1929 a contract is drawn with the Companie Internationale des Rheo-laveur "A. France" in Liege, according to which „Petroşani" Company required and obtained a project for the building, at Petroşani Group, of a washing unit capable of treating 150 tons/hour of "tout-venant" coal, in separation, 16 hours a day, and 270 tons/hour of 0-80 mm coal, through washing, 8 hours a day: 150 tons, 0-10 mm; 120 tons, 10-80 mm[26]. On October 21st, 1929, another contract was settled with the Czechoslovakian Company „Skoda" in Pilsen, with a view of building the separation and the washing units[27].

Contracts were settled with "Schmidt and Nicolau" Company, in Brasov, with a view of accomplishing the works of excavations, concreting, art, rail-road, etc.; at the same time, with "Astra" Company, in Arad, in order to build and assemble the metallic parts of the washing unit,

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[20] *Ibidem*, dos. 3/1928, f. 1-15.
[21] DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 26/1929-30, f. 23-29.
[22] *Ibidem*, dos. 6/1923, f. 1055-1057; dos. 3/1925, f. 135-136.
[23] *Ibidem*, dos. 15/1927, f. 1-8.
[24] DJANH, *Fond Societatea „Petroşani". D.M. Confidenţiale*, dos. 16/1926-27, f.f; *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 38/1929, f.f.; 43/1929, f.f.
[25] *Ibidem*, dos. 25/1932, f. 1-97.
[26] *Ibidem*, dos. 41/1929-30, f. 48-55.
[27] *Ibidem*, f. 56-80.

and of the cable-car belonging to Aninoasa Mining Unit and Petrila Washing Unit[28]. Besides these contracts that started the works envisaged end until April 15th, 1931, on September 23rd, 1929, engineer Ion E. Bujoiu asked the Boarding Committee of "Petroşani" Company to accept the building, at 250 m distance from the new well, of Petrila coal washing unit; its afferent costs represented 62.428.000 lei, of which: 22.5 million lei the reinforced concrete building, 31.955.000 the machines, and 7.973.000 lei other expenditures.

The washing unit included the following:

- The marshalling yard for separating the blocks over 80 mm, having a capacity of 150 tons/hour;
- A silo for storing the 0-80 mm coal extracted by a shift;
- The 8-80 mm, 0-8 mm, and the mixture of water and coal washing unit having a capacity of 270 tons/hour (one shift/day).

Washed coal was to be grouped as follows: 0-8 mm, 8-20 mm, 20-80 mm, with the possibility of either re-grouping the 0-80 mm class according to the desired proportions or over-separating the following sizes: 20-30 mm, 30-50 mm, and 50-80 mm. The estimated number of employees was 32 people, and exploiting expenditures were evaluated as representing 19.44 lei/ton of washed coal[29]. On November 15th, 1929, the 4th Mining Inspectorate of Petroşani was asked for approval in order to implement the whole rationalizing program that also included the building of Rheo-laveur system coal separation and washing units[30]; on August 12th, 1931, the same institution was informed that the building works including the mechanical preparation equipment in Petrila, the cable-cars, the maneuver circuits of the carriages, etc. were ready; it was also required the functioning authorization[31].

A similar route followed the building of the washing unit in Lupeni, placed nearby Ştefan Separation, evaluated at 44.060.000 lei. On December 12th, 1929 "Lupeni" Company handed in to the 4th Mining Inspectorate in Petroşani the building plans of the washing unit that was built, as in the case of Petrila, by "Skoda" Company in Pilsen according to the project drawn out by the Companie Internationale des Rheo-laveur "A. France" in Liege. The equipment had the same characteristics as those of Petrila unit; yet, its flow was different: 140 tons of coal/hour, in separation, during 16 hours/day and 200 tons/hour of 0-80 mm coal, in the washing unit, during 8 hours/day, of which: 50 tons/hour, 0-10 mm and 150 tons/hour, 10-80 mm[32].

On February 18th, 1931 "Lupeni" Company asked the approval of the 4th Mining Inspectorate of Petroşani in order to put into service the washing unit that included the following parts:

- The sorting equipment used in order to sort the clods >80 mm;
- The receiving silo of raw coal;
- The separation equipment used in order to separate 0-80 mm coal;
- "Rheo" washing equipment for 8-80 mm coal, for 0-8 mm coal; and for the mixture of water and coal;
- The separation equipment for 8-80 mm washed coal, for the following types: 8-20 mm; 20-30 mm; 30-50 mm; 50-80 mm;
- Coal silos for the above sizes;
- Mixing equipment for mixed coal;
- Coal silos to store mixed coal;

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[28] *Ibidem*, dos. 38/1930, f. 4-6; dos. 43/1930, f. 1-4.
[29] *Ibidem*, dos. 25/1929-32, f. 72-73.
[30] DJANH, *Fond Inspectoratul Minier Petroşani*, dos. 54/1931, f. 1-3.
[31] *Ibidem*, f. 32.
[32] ***Ibidem*,** dos. 82/1930, f. 1-10; *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 25/1929, f. 74.

- Equipment for loading the carriages[33].

The washing unit took over the whole production of coal belonging to Lupeni Group, beginning with October the 1st, 1932; it comprised 450000 tons/year; the sterile, resulting after manual sorting or hydro-mechanical procedures, and representing about 20% of the gross production, was to be transported by the 2294 m length cable-car that should have been built between the loading station at 644.2 m altitude, on the left bank of the Jiu River, and 887.5 m altitude; the cable-car was built by "Pohling" Company in Austria; it included two carrying cables and one traction no ending cable; it was moved by two 60 hp electrical motors and supported itself upon 19 iron pillars[34].

They went on building, and, accordingly, at the end of the inter-war period, "Petroşani" Company could take pride in a complex of equipments for the mechanical preparation of coal, unique in Romania, that included seven groups[35] and allowed to provide its customers a large variety of coal types[36].

1. The previously described *washing units* in Lupeni and Petrila. They witnessed during the years circuit changes through implanting new equipments: coal re-washing for forging, drying room, flotation. At the same time, according to the discussions, at the beginning of 1938, regarding the changes to be brought to Lupeni and Petrila Preparations such changes targeted:

a. The modification of the washing units in order to increase their capacity of retrieving, their opportunities of getting high quality products and of turning to good account intermediary products, or in order to clean waters;

b. The changes should integrate within the modifications that were to occur at the level of the preparations in order to adapt their capacity of processing to the planned growth of coal production; in 1938 the works having as a goal the accomplishing of the envisaged objectives started[37].

Within Lupeni Preparation an equipment of *re-washing the coal for forging* used to operate. The operation was done by a compositing machine with two compartments brought in 1933. The equipment came from an old washing unit and could process 7 tons of coal per hour with a content of 8-9% of ashes. The 16-20 mm coal resulting from the 0-80 mm coal washing was introduced in the compositing machine; due to this second washing the mixtures remaining after the first washing were extracted.

[33] DJANH, *Fond Inspectoratul Minier Petroşani*, dos. 29/1931, f. 1-4.

[34] *Ibidem*, dos. 84/1932, f. 1-7. At the end of 1939, under the predictable circumstances of the growth of coal production at Lupeni Mining Unit to 5000tons/day, and of the lack of safety of the old cable-car due to its instability, the building of a new cable-car was approved; it had 2288 m length, and connected the washing unit located at 643.47 m altitude with that located at 860 m altitude, Westwards from the old location of Ileana Mining Unit. Built before February 1943, by „J. Pohlig" Company, in Köln (Germany), the cable-car included a main line of 1578 m length, 18 metal pillars climbing up to 858.72 m altitude, and a downloading secondary line, of 710 m, with three pillars; the operating unit was located at 858.72 m altitude and was built according to an angular shape. It had a transport capacity of 155 tons/hour and carriages of 700 kg transported by two cables and drawn by a no-end cable. The place where sterile was to be stocked could receive 15 million tons of sterile = 540.000 tons of sterile yearly, during 28 years (DJANH, *Fond Mina Lupeni*, dos. 37-11/1940-1943, f. 47-48; dos. 34-14/1939-1947, f. 227-231).

[35] The description of the whole complex is done by engineer August Buttu (1902-1978), in the research, *Cerinţele consumatorilor de cărbuni din România şi adaptarea instalaţiilor de preparare la aceste cerinţe***,** displayed at the Congress of the Engineers and Techniciens of the Mining Industry, Bucharest, May 1937, to be found at DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 63/1937, f. 1-26.

[36] *Ibidem*, f. 4.

[37] DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 51/1938, f. 1-58; *Fond Societatea „Petroşani". D.M. Confidenţiale*, dos. 3/1940, f. 205-210.

Table no. 1

Consumer	Type	Ashes %	Humidity %	Volatile materials %	Caloric power kcal/kg	Agglutination	Coke	Granulation mm	Vaporization	Observations
Mechanized scraper pails, brick manufactures, spirit plants	semolina	10	4	38-40	7000-7400	-		0,5-10	-	Higher caloric power
Romanian Rail Road Company, ships, mechanized pails without scrapers	standard	12	4-5	38-40	6800	-		0,5-100	6,3	For the Romanian Rail Road Company 0,5-10 mm/28% 10-20 mm/22% 20-100 mm/50%
Romanian Rail Road Company, radiators, stoves, special scrapers	blocks	4-7	3-4	38-40	7200-7400	-		100	-	-
Sugar, starch plants	-	10	5-6	38-40	7200-7400	-		0,5-20	-	10-20 mm/25%
Gas plants	semolina, Lupeni	10	4-6	38-40	7200-7400	yes	yes	0-10	-	Gives quality coke
Forge workshops	semolina, Lupeni, forge	8-9	6	38-40	7200-7400	yes	yes	16-20	-	Twice washed
Stove, special scrapers, radiators	nut I	12	5	38-40	7000	-		20-30	-	-
Stoves, special scrapers, radiators	nut II	12	5	38-40	7000	-		30-50	-	-
Auto-calor	Auto-calor Petrila	9-10	4	38-40	6800	no	no	3-10	-	-
Stoves, special scrapers, radiators	mixed raw	10	4	40-42	6800	-		Ovoid prismatic cubic	-	Mixes within standard
Stoves, special scrapers, radiators	Mixed Semi-coke	10	3-4	20-25	6800	-		cubic	-	-
Carbonizing	semolina Petrila	10	4-5	38-40	6800	no	no	0-10	-	Semi-coke

2. The drying rooms. As washed semolina coal (0-10 mm) was not allowed to have a humidity under 10%, however long drying lasted (and the 1931-1932 winter determined the freezing of the coal supplied to the Romanian Rail-Road Company), a solution was looked for; they had to choose between the mechanical drying through centrifugation or thermal drying. The second solution was chosen; it consisted in using combustion gases in a direct draught; the heat exchange took place in cylindrical recipients where gases entered at 700-900°C and came out at 100-120°C[38].

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[38] Description of Humboldt drier in, DJANH, *Fond Mina Lupeni*, dos. 16-6/1932-33, f. 1-16. In 1939, centrifugal equipments able to decrease humidity were built at the two washing units in Lupeni and Petrila (August Buttu, *op. cit.*, p. 85).

Table no. 2

Preparation	ø/mm	Length mm	Flow tons/hour	Producer	Implementing year
Lupeni	1.800	12.000	22	Humboldt	1933
Lupeni	1.700	12.000	19	A.C.P.	1937
Petrila	2.500	10.500	35	Büttner	1933
Petrila	2.100	10.500	50	Babcock	1935

3. Flotation units. Within the washing units, the mixture of water and coal was washed in a Rheo battery, and the results were poor as the mixture was only made thin; the ashes resulting from the washed mixture represented 18-20% at Lupeni, and 15% at Petrila. The mixture was evacuated by the machines as shale and still contained enough coal; it was initially thrown into the sewerage system; then it was re-washed in order to retrieve a part of it, but the losses were quite important; the analysis of the 0-10 mm shale showed that the shale < 1 mm contained up to 26-28% coal with maximum 15% ashes. According to such data, during the period 1934–1935, certain studies began at the Laboratory of "Petroşani" Company in Lupeni; in 1936 Ekoff system flotation units were put into service having a processing capacity of 40 tons of coal per hour within each washing unit. Flotation cells, three batteries of 16 cells each, were built of wood and functioned according to the principle of air agitation at 0.2 atmospheres; agitation air was supplied by a blowing device having a flow of 170 m^3/ minute. The floating mixture was afterwards dried within two void filters, with discs; the water of the basin between the pump and the filter was evacuated into the sewerage system. The enlargement of such equipments after 1938 increased the capacity of treating the mixture up to 30 tons/ hour within each Preparation[39].

4. Water cleaning. Coal washing and used waters' evacuation into the Jiu River, both in Petrila, and Lupeni, determined an ecological disaster. An address of the Inspection of Plain Waters belonging to the Ministry of Agriculture and Estates, dated June 12th, 1937 accused "Petroşani" Company of having killed the fish in the Jiu River; it also asked the company to build the equipments necessary in order to clean the waters, to re-populate the Jiu River with fish, and to indemnify "Cerbul" Fishing Company in Târgu-Jiu[40].

Under such circumstances, "Petroşani" Company built the flotation equipments estimated to retain about 60 tons/day of coal dust[41], and, especially, the decantation units. The building of those units began in 1938 and ended in 1941. The washing units used 330 m^3/hour of clean water in case of Petrila Preparation, namely 300 m^3/hour at Lupeni Preparation; used water contained 50-100 g/liter of substances in suspension: fine coal, fine mixtures of water and coal, and argyles. In order to clean those waters both washing units were completed with the following equipments:

- Dorr system decanter, having ø = 20 m. From the decanter, the mud, thickened with a lot of coal = 450 g/liter, was pumped to the floating batteries; argyle water was then evacuated into the second decanter;
- Ekoff system flotation batteries;
- Dorr system decanter, having ø = 32 m, where all polluted waters gathered with sterile, the water decanted from the first decanter, and the flotation sterile. The content was treated with quick lime in order to accelerate decantation; a shale mud resulted and was evacuated into the place where sterile was stocked; decanted water was evacuated into the Jiu River, 263 m^3/hour, having a content of 0.6-1 g/liter at Lupeni Preparation, and 270 m^3/hour, and a content of 1-2 g/liter, at Petrila Preparation[42].

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[39] DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 63/1937, f. 12.
[40] DJANH, *Fond Mina Lupeni*, dos. 26-33/1936-38, f. 27.
[41] *Ibidem*, f. 26.
[42] DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 44/1936-46, f. 71.

5. Briquetting. The first briquetting equipment was put into service at Vulcan, in 1931 and consisted in an ovoid briquettes press, type Sahut Corneur, having a processing capacity of 10 tons/hour pressing the coal under 5 mm mixed with oil pitch[43]. The results obtained there determined the installing of more modern equipments at Petrila Preparation. Briquetting processed all the surplus of small coal (semolina coal) of "Petroşani" Company and used oil pitch and the pitch obtained from oxidizing tar during the process of carbonizing as glue. Briquettes were made of raw coal or semi-coke; their strength was similar to that of the foreign briquettes, and the Technical Unit of the Romanian Rail-Road Company obtained a vaporization of 6.99.

Beginning with 1933, when the equipment of "Fohr-Klein Schmidt" Company was installed, comprising two presses: an ovoid press having a processing capacity of 12 tons/hour and supplying 50-60 g briquettes, and a Conffinhal system press having a 7 tons/hour capacity and supplying 1.5 or 3 kg briquettes, briquetting constantly developed; during 1934–1935 and 1937 new units were added. Such developments made the briquetting unit comprise, at the end of the inter-war period, the following:

- Two mixing devices for raw coal and another one for semi-coke;
- Two disintegrators;
- Two main elevators;
- Four presses with their mixers:

a. a Schüchtermann & Kramer Barun press, supplying ovoid briquettes, having a processing capacity of 24 tons/hour;

b. a „Sahut-Corneur" press, supplying ovoid briquettes, having a processing capacity of 10 tons/hour;

c. a Schüchtermann & Kramer Barun press, supplying prismatic briquettes, having a processing capacity of 12 tons/hour;

d. a Schüchtermann & Kramer Barun press supplying cubic briquettes, having a processing capacity of 8 tons/hour.

6. Carbonizing. In March 1931, the doctor of Chemistry, I. L. Blum, made a report upon his visits in France and England, where he was sent by "Petroşani" Company in order to consider what the opportunities of making a superior fuel out of the Jiu Valley small coal were[44]. They started from the tendency of producing a fuel without smoke and from the need of using small coal, elements that imposed coal's low temperature distillation. According to the conclusions, they started to build the carbonizing and distilling equipments[45], built – until December 1932 – nearby the equipments of the briquetting unit at Petrila Preparation. The equipment of the 0-10 mm washed and dried coal processing included three sections:

*a. Carbonizing***,** made of two Ab-der-Halden furnaces – dating since 1939, 10 furnaces – owing to which a part of the 0-10 mm coal was distilled at low temperatures, decreasing accordingly the share of volatile substances and obtaining a semi-coke that burnt with less smoke. Each furnace had two round mobile hearths, one for drying and another one for distillation. The oven's flow was of 1200 kg of raw coal per hour; a mean of 930 kg of semi-coke, 140 kg of tar, 45 kg of water, and 55 kg of gases resulted.

b. Retrieving volatile materials determined the making of tar which was distilled; a ton of tar used to get: 28 kg of essences, 185 kg of oil, and 760 kg of pitch used for briquetting.

c. Briquetting (agglomeration)

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[43] DJANH, *Fond Inspectoratul Minier Petroşani*, dos. 94/1930, f.f.

[44] DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 30/1931, f. 8-15. La f. 1-7 there is a report with similar conclusions drawn out by an engineer of „Petroşani" Company, Grigore Schileru. See, I.L. Blum, *Distilarea cărbunilor*, Buletinul I.R.E., III, 1935, nr. 3, p. 3-54.

[45] DJANH, *Fond Inspectoratul Minier Petroşani*, dos. 7/1932, f. 11-12; *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 51/1932, f. 3-8.

*7. Self-agglomeration***.** A low temperature distillation furnace, a press, and equipments meant to retrieve the tar were installed in Lupeni, according to the project of the French and Belgium Company, entitled "Distillation à basse température des combustibles solides"[46].

One should notice the contribution of "Petroşani" Company to the improvement of the quality of the Romanian lignite, under the circumstances that imposed the minimum vaporizing power of the coal taken by the Administration of the Romanian Rail-Road Company stipulated by the Fuel Protocol, concluded on May 9th, 1931[47]. Lignite improvement appeared as an idea in Austria; the procedure of Professor Fleissner was industrially applied; in Romania, within the basins of Schitu Goleşti and Doiceşti, "Lignite" Company resorted to Fleissnet procedure; "Petroşani" Company resorted to Filitti procedure.

The first procedure treated lignite with vapors in order to eliminate a part of the water; the second one replaced vapor with fuel oil[48]. The mining engineer Grigore Filitti, who had studies lignite for over 30 years, experienced the procedure within a metal recipient using lignite coming from Mărgineanca Mining Unit belonging to Doiceşti basin. He observed that de-hydration can go until the total elimination of humidity, being the only procedure that managed to do such a thing until that moment. In 1932, Grigore Filitti got a patent for that procedure which he ceased, in April 1933, to "Petroşani" Company, during its whole duration, namely until 1943[49].

According to the obtained patent, at the beginning of 1933, they concluded a collaboration agreement with the Mining Cooperatives of Schitu Goleşti and Doiceşti mines comprising about ten mining units, in order to build lignite improving units at Doiceşti and Schitu Goleşti, by "Petroşani" Company, that cost 10 million each; the beneficiaries were to pay to "Petroşani" Company 23% of the price received from the Administration of the Romanian Rail-Road Company for the supplied coal[50]. According to such agreements, during the 1932-1933, a series of experiments were done in the Laboratory of "Petroşani" Company; in May, engineer Grigore Socolescu was assigned by engineer Ion E. Bujoiu to plan and build the first unit of lignite improving according to a Romanian patent. The first equipment was installed at Doiceşti and included two autoclaves; the second one was installed at Schitu Goleşti; in 1933 there were three autoclaves and two boilers. The operation of improvement lasted 120 minutes and treated, within an autoclave, 6000 kg of lignite; within 24 hours nine carriages of raw coal were introduced into the autoclave and seven carriages of improved lignite were produced. They generally worked with lignite having ø > 50 cm and a humidity of 38 %; they managed to extract 20-25 % of the water, resulting lignite having a humidity ranging between 8 and 19.5% and a caloric power ranging between 4900-5870 kcal/kg. As an average, humidity between 12 and 14 % was obtained, with an increase of the caloric power from 3400 to 5000 - 5200 kcal/kg, an increase of vaporization power of up to 4.4-4.8 and a using coefficient of 0.63 for the lignite at Schitu Goleşti and of 0.59 for the lignite at Doiceşti[51].

"Lupeni" Company and then "Petroşani Company also tried to start again the coke production at Lupeni, after the date of August 1925, when the coke unit had been closed as the coke did not match the demands of metallurgical industry. The researches made by "Petroşani"

[46] I.L. Blum, *op. cit.*, p. 53-54.
[47] Montanistică şi Metalurgie, V, 1926, nr. 7 şi 8, p. 8.
[48] Gr. Socolescu, *Innobilarea lignitului în România*, in vol., *Al II-lea Congres al Asociației Inginerilor Diplomați ai Scolii Politehnice Bucureşti*, p. 391-414.
[49] DJANH, *Fond Societatea "Petroşani". D.M. Serviciul Tehnic*, dos. 87/1933, f. 6. The interest of „Petroşani" Company for improving pit-coal was older, engineer Ion E. Bujoiu speaking in 1928 about briquetting and drying the lignite (DJANH, *Fond Societatea „Petroşani". D.M. Confidențiale*, dos. 1/1927, f. 16).
[50] DJANH, *Fond Societatea „Petroşani". D.G. Consiliul de Administrație*, dos. 2/1933, f. 82.
[51] Gr. Socolescu, *Înnobilarea lignitului*, Miniera, IX, 1934, nr. 4, p. 11-15; DJANH, *Fond Societatea „Petroşani". Direcția Generală*, dos. 4/1940, f. 42-43, 80.

Company based upon the developments in coking during the fourth decade of the 20th century, showed that the coal at Lupeni could provide coke capable of matching the physical qualities demanded by metallurgy, and having a caloric power of 6600 - 6800 kcal/kg, 10-11 % ashes, and 2-3 %, sometimes 1.7-1.8 %, sulphur. As the coke prepared out of the coal at Lupeni contained over 0.9-1.1% sulphur beyond the accepted margin of the coke used within high furnaces, the use of $Na_2 CO_3$ was required in order to eliminate it of the melted cast iron. The researches made at U.D.R. Coke Unit in Resita, in Essen (Germany) within Koppers furnaces, as well as at the pilot battery in Lupeni, built in the autumn of 1938, showed that coke obtained could be used within high furnaces having a capacity of 150 tons of cast iron per day and even up to 300 tons of cast iron per day, on condition that a mixture was obtained; the mixture was made of small coal in Lupeni and semi-coke containing 12% of volatile substances. According to the results obtained, "Petroşani" Company studied, together with "Koppers" Company in Germany (the one that also supported the building of the pilot station in Lupeni), a project with a view of building a coke unit in Lupeni, having a production capacity of 200 tons of coke during 24 hours[52].

The project was extended during World War II when a coke unit with a capacity of 250000 tons of coke per year was envisaged; of it 200000 tons represented metallurgy coke; the coke unit had annexed equipments for the retrieving of sub-products: benzyl, ammonia, sulphur, etc. Carbonizing equipment for briquettes and raw coal was also meant to be installed there, having a capacity of 100000 tons/year; they all cost over two milliard lei[53].

The company called "Cocseria Lupeni" was settled in 1944, under the control of "Petroşani" Company; to its capital, the following entities subscribed with an amount of 55.210.000 lei: Romanian Bank, the Bank of Romanian Credit, "Chrissoveloni" Bank, Count Bank; Romanian State also implied in the project, the payment of the machines that were to be imported from Germany being made out of the account of liquid assets Romania had there. B.N.R. gave, on December 1943, a loan of 19 million Rmk, guaranteed by the Ministry of Finance, in order to buy machines, equipments and materials; the credit was supplemented with 2.5 million Rmk according to the agreement ratified by the Decree-Law no. 289/May 23rd, 1944[54]. They hoped that the projected equipments that were to be imported from Germany could be put into service until April 1946.

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52 August Buttu, *Privire retrospectivă asupra industriei carbonifere a României a ultimilor 25 de ani*, A.M.R., XXVI, 1943, nr. 5, p. 87; DJANH, *Fond Societatea „Petroşani". D.M. Serviciul Tehnic*, dos. 79/1938, f. 2-5; Gr. Schileru, *Cocsul ca materie primă şi posibilităţile de aprovizionare*, in vol., *Contribuţiuni la problema materiilor prime în România*, vol. IV, part I, Bucureşti, 1941, p. 103-109.

53 August Buttu, *op. cit.*, p. 87.

54 Miniera, XIX, 1944, nr. 2, p. 61; nr. 6-7, p. 204. A more ambitious project existed, „Petroşani" Company projecting a group of metallurgical equipments for which the Ministry of National Economy gave its authorization in 1938. Romanian Metallurgical Company was founded; it ordered abroad machines for thin sheet iron rolling; it evaluated certain opportunities of buying iron ore from Bulgaria; in order to air the steel in electrical furnaces they envisaged to get the needed energy out of the Jiu Valley water sources. The whole metallurgical group that would have relied upon its own ores and the coke of „Petroşani" Company, representing 500 million lei and producing 55000 tons of steel per year, a project that was not accomplished, was to be installed in Livezeni area (DJANH, *Fond Societatea „Petroşani" D.M. Serviciul Tehnic*, dos. 79/1938, f. 6).

MODERN METHODS FOR DETERMINATION OF PHYSICS CHEMIC AND PETRO GRAPHICAL CHARACTERISTICS FOR COAL SLURRY FROM COROESTI SETTLE POUNDS

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Abstract: *The paper present a case study for coal slurry from Coroesti settles pounds and also a starting point for establish the correct processing methods for this coal slurry.*

Preparation bearing of coal and implicit of slurry coal depends of their characteristics (in carbonization grade, chemical and mineralogical compositions, physical and chemical proprieties for combustible substance, sterile nature, proportion between combustible substance and sterile, coal's granulometry, humidity, etc.).

Physic and chemical characteristics for slurry coal majorly influence the processing method:

1. Particles shape and dimensions (are depending of hard coal petrography structure)
2. granulometry compositions (is determined by granulometry analysis)

For each settle pounds were tacked slurry coal samples and calculated an average for partial refusal and obtain an average sample and also the cumulated refusals and passing for this sample. This data are presented in tables 1, 2 and 3.

Table 1 *Refusals and passing average for the 1A settle pound*

Granulometry class	Partial refusals for the average sample from 1A settle pound % (q_i)	Cumulated refusals $\sum q_i$ %	Cumulated passing's $100-\sum q_i$ %
3.15	3.97	3.97	96.03
3,15 – 1,25	11,26	15,23	84,77
1,25 – 0,8	7,40	22,63	77,37
0,8 – 0,4	16,46	39,09	60,91
0,4 – 0,125	21,39	60,47	39,53
0,125 – 0,071	5,27	65,74	34,26
0,071 – 0,040	3,71	69,45	30,55
0,040 – 0	30,56	100,01	0,0

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Table 2 *Refusals and passing's average for the 1B settle pound*

Granulometry class	Partial refusals for the average sample from 1B settle pound % (q_i)	Cumulated refusals $\sum q_i$ %	Cumulated passing's $100-\sum q_i$ %
3,15	2,10	2,1	97,9
3,15 – 1,25	8,30	10,40	89,60
1,25 – 0,8	7,30	17,70	82,30
0,8 – 0,4	18,00	35,70	64,30
0,4 – 0,125	21,90	57,60	42,40
0,125 – 0,071	7,00	64,60	35,40
0,071 – 0,040	4,95	69,55	30,45
0,040 – 0	30,45	100,00	0

Table 3 *Refusals and passing's average for the 2 settle pound*

Granulometry class	Partial refusals for the average sample from 1B settle pound % (q_i)	Cumulated refusals $\sum q_i$ %	Cumulated passing's $100-\sum q_i$ %
3,15	2,97	2,97	97,03
3,15 – 1,25	13,07	16,03	83,97
1,25 – 0,8	12,33	28,37	71,63
0,8 – 0,4	30,33	58,70	41,30
0,4 – 0,125	19,47	78,17	21,83
0,125 – 0,071	3,43	81,60	18,40
0,071 – 0,040	3,73	85,33	14,67
0,040 – 0	14,67	100,00	0

3. average dimension and specific surface (for 1A settle pound the calculated average dimension is 64 $*10^{-4}$ m and the calculated specific surface is 0,97 m^2/kg, for 1B the calculated average dimension is 53 $*10^{-4}$ m and the calculated specific surface is 1,17 m^2/kg , for the 2 settle pound he calculated average dimension is $76*10^{-4}$ m and the calculated specific surface is 0.82 m^2/kg)

4. ash quantity : for the coal slurry sampled from the two settle pounds was calculated an ahs average and obtain for each settle pound an average value for ash. The average samples resulted on granulometry classes and the weighted average ash for each settle pounds are presented in table 4.

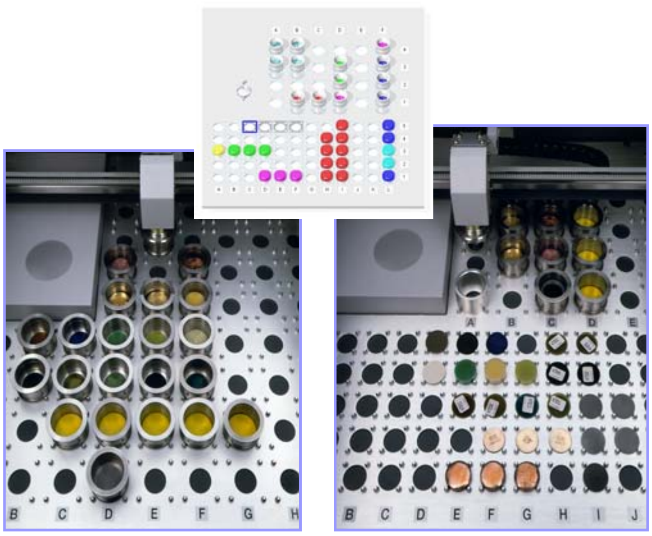
Table 4 Average content of ash for coal slurry from Coroesti settle pounds

Granulometry classes [mm]	Average content of ash %		
	1A settle pound	**1B settle pound**	**2 settle pound**
3,15	60,18	18,67	67,66
3,15 – 1,25	40,22	11,8	70,43
1,25 – 0,8	33,55	14,12	70,20
0,8 – 0,4	32,55	15,46	78,51
0,4 – 0,125	40,50	27,82	74,35
0,125 – 0,071	46,88	39,90	67,31
0,071 – 0,040	54,12	50,73	67,786
0,040 – 0	76,40	71,28	73,82
The weighted average ash **(a) %**	**51,24**	**38,28**	**73.82**
The weighted average ash without class 0,040-0 mm (a’) %	**27,89**	**15.58**	**62,99**
The weighted average ash without class 0,125-0 mm (a’’) %	**-**	**-**	**58,15**

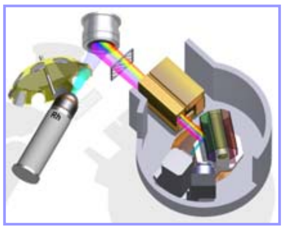
From table 4 it can be observed that ash content for 0,040-0 mm class granulometry is over 70%.

If from the coal slurry from 1 B settle pound is separated 0,040-0 mm class, and from the second (2) settle pound is separated 0,125-0 mm class, which has a high quantity of ash we can obtain a weighted average ash without class 0,040-0 mm (a’) for 1B settle pound and a weighted average ash without class 0,125-0 mm (a’’) for the second settle pound. The ash value after this process is much lower than the initial one. So it can be made a separation process of class under 0.040mm for 1B settle pound and under 0.125 for the second (2) settle pound. And the rest of coal slurry can be processed in flux and sold (because it has a lower ash quantity).

5. volatile matters are gases like CO_2, H_2S, CH_4, H_2, NH_3, etc



6. Thermal power
7. Chemical composition of coal slurry from Coroesti settle pounds is structured in: an organic mass (substances having a complex structure and made by carbon, oxygen, hydrogen, sulfur and nitrogen) and an inorganic mass (mineral substances which by roasting process became ash) and water. The organic mass is the thermal one. The coal slurry samples from Coroesti settle pounds were chemically analyzed using Xray spectrophotometry



The percentage quantities of organic and inorganic substance for coal slurry samples from Coroesti settle pounds are presented in tables 5, 6 and 7:

Table 5 *The percentage quantities coal slurry from 1A settle pound*

Component	**Sample 1**	**Sample 2**	**Sample 3**	**Sample 4**	**Sample 5**	**Sample 6**	**Sample 7**
Organic +water	48.91	34.56	41.89	43.06	71.25	59.24	64.83
Inorganic	51.09	65.44	58.11	56.94	28.75	40.76	35.17

Table 6 *The percentage quantities coal slurry from 1B settle pound*

Component	**Sample 8**	**Sample 9**
Organic + water	71,74	66,07
Inorganic	28,26	33,93

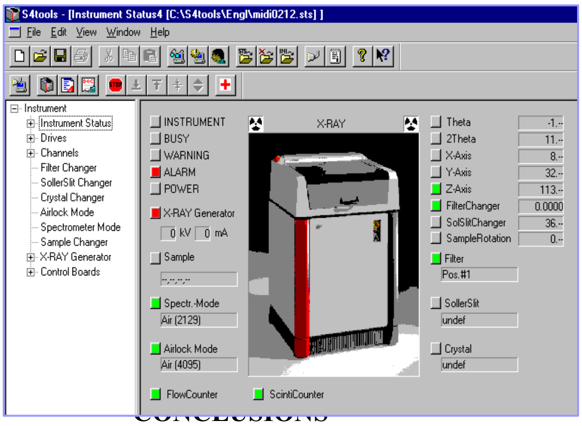
Table 7 *The percentage quantities coal slurry from the second (2) settle pound*

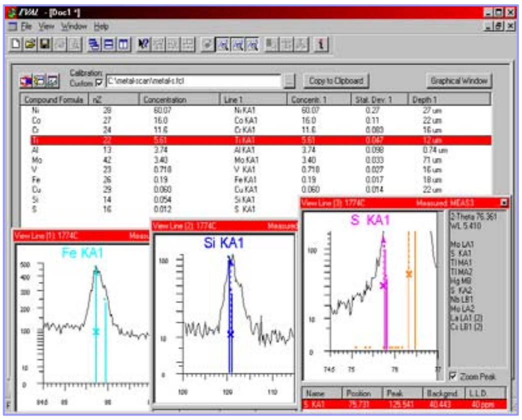
Component	**Sample 10**	**Sample 11**	**Sample 12**
Organic +water	**27,40**	**30,55**	**28,27**
Inorganic	**72,596**	**69,45**	**71,73**

The inorganic mass is making up mainly by 3 mineral types: silicates (especially aluminum silicates), abscesses (specially the iron ones) potassium carbonates, magnesium and iron. The chemical compositions determined by X-ray is presented in table 8

Table 8 *Chemical compositions of ash coal slurry from Coroesti*

Comp-ound	Sample 1	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6	Sample 7	Sample 8	Sample 9	Sample 10	Sample 11	Sample 12
SiO2	26,8	32	28,4	27,8	14	19,8	17,17	13,8	16,6	35,7	34,0	34,9
Al2O3	17	19,7	17,9	17,6	9,79	13,2	11,69	9,65	11,3	21,6	20,7	21,2
Fe2O3	2,37	4,8	4,12	3,97	1,54	2,46	2,022	1,5	1,92	5,60	5,23	5,66
K2O	1,79	2,45	2,12	2,07	0,892	1,35	1,139	0,874	1,27	2,81	2,65	2,77
SO3	0,875	2,78	2,41	2,35	1,05	1,56	1,324	1,02	1,09	3,17	2,99	3,11
MgO	0,652	0,731	0,664	0,656	0,375	0,498	0,445	0,370	0,431	0,795	0,766	0,755
TiO2	0,575	0,925	0,8	0,772	0,316	0,493	0,41	0,310	0,391	1,07	1,00	1,07
CaO	0,521	1,18	1,02	0,984	0,412	0,635	0,531	0,404	0,507	1,36	1,27	1,35
Na2O	0,341	0,454	0,412	0,409	0,238	0,314	0,281	0,235	0,273	0,491	0,474	0,475
BaO	0,0461	0,0799	0,0722	0,068	0,0288	0,0442	0,037	0,0282	0,0353	0,0913	0,0860	0,0993
P2O5	0,0395	0,0544	0,0469	0,046	0,0206	0,0307	0,026	0,0202	0,0249	0,0620	0,0585	0,0599
Cr2O3	0,0178	0,0291	0,0252	0,0243	0,00978	0,0154	0,0127	0,00957	0,0121	0,0337	0,0316	0,0342
MnO	0,0168	0,0307	0,0264	0,0254	0,00996	0,0159	0,0131	0,00973	0,0124	0,0358	0,0334	0,0360
V2O5	0,0167	0,0371	0,0306	0,0303	0,0119	0,0189	0,016	0,0116	0,0148	0,0434	0,0405	0,0399
SrO	0,0137	0,0205	0,0169	0,0164	0,00571	0,01	0,0077	0,00557	0,00731	0,0244	0,0226	0,0238
ZrO2	0,0136	0,0263	0,0216	0,0211	0,00738	0,0134	0,0099	0,00720	0,00942	0,0313	0,0290	0,0303
CuO	0,0114	0,0229	0,0192	0,0185	0,00656	0,011	0,0088	0,00640	0,00838	0,0271	0,0252	0,0272
ZnO	0,0111	0,0224	0,0187	0,0181	0,00634	0,0107	0,0085	0,00619	0,00812	0,0266	0,0247	0,0263
Rb2O	0,0105	0,0161	0,0133	0,0129	0,00448	0,00778	0,006	0,00436	0,00574	0,0192	0,0177	0,0187
NiO	0,00981	0,0217	0,0183	0,0176	0,00629	0,0105	0,0084	0,00613	0,00802	0,0257	0,0239	0,0259
Cl	0,00646	0	0	0	0	0	0	0	0	0	0	0
Os	0,00126	0	0	0	0	0	0	0	0	0	0	0





CONCLUSIONS

The determination of physic, chemical and petro graphic characteristics for coal slurry from Coroesti settle pounds is a starting point for establish the correct processing methods for this coal slurry.

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THE MATEMATICAL MODEL OF THE DISPERSION PHENOMENON OF TWO MISCIBLE FLUIDS IN A POROUS ENVIRONMENT

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EMILIA DUNCA**

Abstract: *In the specialty literature there are presented three categories of mathematical models of two miscible fluids' dispersion phenomenon in a porous environment: geometrical models, geometric – statistical models; probabilistic models. The geometrical models and the geometric – statistic models are based on the porous environment's presentation through a geometrical network, which allows the mathematical expression of the phenomenon. This type of models requires a high number of parameters, characteristic to the given geometry. The porous environment's representation through a geometrical network constitutes an idealization of the real conditions. The use of the geometrical models has given no satisfactory results. In order to make the smallest number of assumptions concerning the porous environment's geometry a general model, a general representation of the dispersion has been searched. This has lead to some probabilistic models (Scheidegger, 1963), based on the idea that the data concerning the porous environment are random and that the most adequate representation of a situation is the representation of the environment through a set of random variables.*

1. GENERAL DATA

The movement of the pollutant in the underground water is made due to a velocity field as well as a molecular difusion. The two phenomena are merging, creating a complex process, called the dispersion of the pollutant in the porous environment.

From the dispersion phenomenon's analysis of a pollutant in a porous environment results the existence of three main migration mechanisms of the pollutant substances: convection (advection); molecular diffusion; mechanical or kinematical dispersion.

By **convection** (**advection**) one understands the elements' engaging in the solution, in the moving fluid's movement. In a saturated porous environment, the water exists under two forms: tight water and free water (which can circulate under the action of some gradients of hydraulic burden). Actually, especially for the least permeable environments, the free water

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friction depends on the size of the burden gradients (that is the case of clays, in the availability area of Darcy's law). /2/

The molecular diffusion is a physical phenomenon connected to the molecular fuss. In a spell fluid, the Brownian movement causes the particles' movement in all the space's directions. If the concentration of the fluid is homogeny in space, two neighbor points send, on the average, the same number of particles one to the other, and the molecular fuss does not modify the solution's concentration. If there is a concentration gradient between two neighbor points, the point with higher concentration will send, on the average, more particles in every direction than the points with low concentration. The result of this molecular fuss will be a particles transfer from the high concentration area to the low concentration area.

In a porous environment, the molecular diffusion takes place in all fluid stage (tight + free water). The existence of the solid slows down extremely the Brownian movement of the particles that are transported in the fluid. As a consequence, the effective molecular diffusion quotient, d (m^2/s), in a spell fluid, situated in the pores of a porous environment, is smaller than d_0 (the molecular diffusion in the fluid quotient). The fraction of the two quotients is called tortuosity (Bear 1972) /4/:

In practice, **the tortuosity** d/d_0 varies between 0,1 for clays and 0,7 for sands. The molecular diffusion will lead to the pollutant's spread around the injection point, even in the absence of fluid's leak through the porous environment. One can show the distribution in space and time of a solution, having a concentration C_0, injected at time t_0, on a distance between (x-a), (x+a). After a time interval, the solution is spread on a distance higher than the initial space interval and the maximal value drops in time. The solution's concentration has a Gaussian distribution or a normal distribution./3/

The kinematical (mechanical) dispersion is a mixture phenomenon, connected to the microscopic velocities' heterogeneity. The kinematical dispersion could be resumed through the following aspects/6/:

- a faster spreading of the transported elements in pores' axe;
- a difference of the medium velocities between different pores;
- current lines are mixing, causing a non-uniform dilution of the concentration.

So, **the kinematical dispersion** is the result of the existence of a real velocity field, complex and unknown, which we neglect in the convection phenomena when we use medium, fictive, Darcy velocity.

Numerous researchers (Taylor, Schedegger, Bear, Bachmat, Fried) have developed the stochastic theory of the dispersion, starting from a random distribution of the pores which form the porous environment./5/

2. THE SCHEME OF THE DISPERSION

The dispersion phenomena of a pollutant in the water that circulated through a porous environment can be mathematically modeled through a system of differential equations with partial derivates, as follows /4/ /6/:

a) The dispersion equation:

$$div\left[\overline{\overline{D'}}\cdot\rho\cdot\left(grad\frac{C}{\rho}\right)\right]-div\left(\overline{v}\cdot C\right)+S_r=\frac{\partial C}{\partial t} \tag{1}$$

where: $\overline{\overline{D'}}$ is the dispersion quotient (m^2/s);

ρ – the mixture's density (kg/m^3),

$C = C(x, y, z, t)$ the pollutant's concentration, depending on the point's position and the time,

v – water's velocity in the pores (m/s),
t - time (s).

b) The continuity equation:

The continuity equation for a fluid's flow through a porous environment is a differential equation with partial derivates, which expresses water's preservation.

If the porous environment is nourished with an external source, the received or given massic debit received or given from the exterior being (ρq), the continuity equation is written /4/:

$$div\left[\rho \cdot \vec{U}\right] + \frac{\partial}{\partial \cdot t}\left[\rho \cdot n\right] \pm \rho \cdot q = 0 \quad (2)$$

Darcy's equation:

$$\vec{U} = -\frac{k_i}{\eta} \cdot (gradp + \rho \cdot g \cdot gradz) \quad (3)$$

where: k_i is the intrinsic permeability quotient (m^2),
η - the dynamic viscosity quotient (N·s/m^2),
n - porosity (%),
p – water pressure in the pores (N/m^2).

c) The mixture's state equations:

$$\rho = f(C)$$
$$\eta = g(C)$$
$$d_0 = d_0\,(C)$$

d_0 is the molecular diffusion quotient.

All sizes that come in these equations are functions of x, y, z, t. In the general form, the dispersion equation's quotients depend on concentration C. In order to obtain solutions with physical meaning, these equations are demanded initial conditions and frontier conditions, called oneness conditions. There are no direct methods that allow the solving of the mentioned equations system. It is necessary to introduce some simplifying hypothesis that allows a practical resolution of the problem. /6/

3. THE GENERAL METHOD FOR SOLVING SOME DISPERSION ISSUES

There are two cases for dealing with the problem:

a) the density and viscosity of the mixture are constant in space and time (trasors' case).

b) the density and viscosity of the mixture are variable in time and space, as concentration functions.

In the first case (η = ct, ρ = ct) the hydro-dynamic equations are independent of the dispersion's equation because, ρ and η don't depend on C.

The dispersion's model becomes an equations system with partial derivates. The time steps will be /2/ /3/:

1. The hydro-dynamic equations are solved (Darcy's equation and continuity equation), keeping count of the frontier conditions and the initial conditions. From that results the distribution of the velocity in space and time for all time steps of the experience.

2. The dispersion equation's quotients are expressed depending on the velocity, at each time step.

3. The dispersion equation is solved.

From that results the distribution of the pollutant's concentration in space and time, at all time steps.

In the general case, $\rho = \rho\ (C)$, $\eta = \eta(C)$, the velocity and quotients of the dispersion equation will be concentration functions. /5/

One can't obtain simultaneously solutions of all mentioned equations.

1. Simplifying hypothesis is being made:

a) the concentration's distribution is known at time *t,* the corresponded ρ and η are being calculated;

b) is assumed that the time period dt, C, ρ and η are constant.

2. The velocities' distribution is calculated for time t + dt.

3. The concentration's distribution is calculated for t + dt.

4. All parameters are calculated depending on the concentration from t + dt and then is continued starting with stage 2.

In theory, the tensor of the dispersion's quotients can be calculated through direct experiments, or from the other experimental determined quotients, with the help of some formulas.

Due to the fact that the dispersion's general equation is difficult to integrate, one writes the equation on the dispersion tensor's main direction by introducing the main dispersion quotients D_L and D_T.

A greater simplification of the dispersion scheme can be achieved through the tridimensional scheme's division in two bi-dimensional schemes, each being written on the plane of two dispersion's main directions.

The determination of the simplifying hypotheses is made based on the preliminary studies conducted for the actual pollution problems.

4. THE MATHEMATICAL MODEL OF THE TRANSFERS THAT OCCUR IN A POROUS ENVIRONMENT

In the mechanicist models the following simplifying hypotheses are generally made /3/:

• the porous matrix is rigid (many times it's considered an homogeny and isotope environment);

• the liquid phase is incomprehensible;

• the gaseous phase is continuous and at atmosphere's pressure;

• the leak is made at constant temperature;

• the different sizes that come in the transfer (flux, water content, velocity …) are being represented through medium value at macroscopic scale.

The material or energy transfers in a soil, whatever their nature is (water, gas, solutions, heat) consist in the overlay of two processes

• **an open movement through a dynamic law** (the movement of the particle's position depending on the solid matrix);

• **a stock's variation in time (gain or loss).**

This variation takes place due to the external influences (precipitations, evaporation, radiations), local consumes (the roots necessary taken) or the changes with other phases (freezing, evaporation, condensation). The stock's variations are being described quantitively through the low of matter's preservation (the continuity equation). The global description of the transfers is being obtained through **the association of a dynamic law with the continuity equation.**

The dynamic laws used in the mechanic of the porous environments are /2/:

• **Darcy's law** which expresses the fact that that the water flow is proportional with the gradient of hydraulic potential;

• **Fourier's law** expresses the proportionality between the heat and temperature flow;

• **Fick's law** translates the proportionality between he gas or solution flow and the concentration gradient.

One can see experimentally that the water's movement in a porous environment can be produced by the existence of some gradients (different from the burden gradient). Thus, the water moves from the high voltage area to the low voltage one. This principle has been used before for the electro-kinetic drainage of the less permeable soils (Terzaghi and Peck 1967). Also the water moves from the high concentration areas to the low concentration areas and from the high temperature areas to the low temperature areas.

The flow in a porous environment can be: **uniform** (the flow's characteristics are invariable in time and space); **permanent** (constant in time); **non-permanent**.

From the velocity's regime point of view, the flow can be:

- **laminar** (the flow is slow and takes place in parallel layers, with no mixture of mass and energy between them);
- **turbulent** (flow with high velocities, the mass and energy transfer between he layers taking place also). The determination of the flow regime is made on the Reynolds number.

Most models express the bilance, the considered quantities (water mass, pollutant mass, heat).

The first modelling step is the building of a <u>conceptual model</u> of the aquiferous field. This consists in a set of hypotheses which reduce the real problem and the real field, at a simplified version, acceptable from the objectives point of view and the associated management problems' point of view./3/The conceptual representation of the dispersion through the equations that describe the dynamic law and the mass transfer, schemed under the shape called *"Dispersion Scheme"*, has a high generalization degree. /2/

In order to solve a concrete, real problem, one must take into account the oneness conditions for the differential equations and the real values of the parameters that come in the equations. Thus, one must know: the medium velocity in the aquifer's pores, the dispersion quotients, the initial conditions for the concentration, in all analyzed fields, the conditions on the field's frontiers.

As a consequence, the experimental research has as main objective the determination of the dispersion quotients and the validation of the used mathematical models for the resolution of some real pollution cases.

Between the solution of the classical dispersion equation and the experimental results there are systematical discrepancies. In order to determine them, any research of the dispersion phenomenon must include a measurements program. For example, when measuring the concentration, we must assure that the value we found is independent of the measurement method.

Generally, measurements have two purposes /5/:

- to check and improve the different dispersion models;
- to determine the influence of the different parameters (such as the fluid and the porous environment's characteristics).

4. CONCLUSIONS

➢ The solving of a pollutant's dispersion problem in a porous environment requires the simultaneous integration of the equations that describe the movement of the water in the soil's pores and the dispersion equation.

- The analytical integration of the dispersion equation in its general form, with initial and frontier conditions demanded by the actual pollution cases, is practically impossible.
- The dispersion equation's quotients are non-linear functions, variable in time and space.
- The dispersion quotient depends on the flow regime and is, in the limits of a laminar flow, a linear velocity function.
- The hydraulic conductivity quotient (permeability) is a non-linear function, and the dynamic density and viscosity of the solution (water + pollutant) depend on the concentration.
- Thus, the solving in practical purposes of some pollution problems, involves mainly a simplification of the dispersion scheme, in the way of preserving the elements that influence the best the phenomenon. This resolution requires:
- The preliminary study of the existent data for the estimation of the flow and the possible pollution type (local, global, horizontal, vertical). The permeability, soil's morphologic parameters, the pollutant's concentration are being analyzed.
- The formulation of the work hypotheses resulted from the preliminary study and the choice of a mathematical representation depending on the phenomenon's scale (field, pollutant, flow).
- Usually, one chooses a hydro-convective or and hydro-dispersive model.
- If the chosen model is hydro-convective, the pollution problem becomes a classical hydrological problem and only the movement of the underground water must be determined.
- In order to avoid inherent approximations for the numerical models, sometimes the analytical approach of some dispersive problems is preferred, which allows a simplification of the dispersion equation in one of the cases: permanent dispersion, one-dimensional dispersion or bi-dimensional dispersion.
- For the hydro-convective pollution problems, starting from the "piston effect" hypothesis of the phreatic water on the pollutants, one can obtain simplification of the dispersion's scheme.
- Thus, in order to accomplish some prognosis, it's enough to analyze the water's movement in the aquifer.
- For this, one can calculate through analytical, numerical or graph-analytical methods, the time the pollutant takes to cover the aquifer, the pollutant concentration's variation in time and space.

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THE USE OF MONTE CARLO SIMULATION TECHNIQUE IN HUMAN HEALTH AND ECOLOGICAL RISK ASSESSMENT

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Abstract: *A human health or ecological risk assessment using probabilistic methods is very similar in concept and approach to the traditional point estimate method, with the main difference being the methods used to incorporate variability and uncertainty in the risk estimation. A variety of modelling techniques can be employed to characterize variability and uncertainty, one of the most common ones being the Monte Carlo simulation technique. The paper has as purpose to present a general framework and broad set of principles, important to ensuring good scientific practices when applying Monte Carlo simulation technique in human health and ecological risk assessment.*

Key words: *risk, assessment, human health, ecological, Monte Carlo, simulation*

1. INTRODUCTION

The basic goal of a Monte Carlo analysis is to characterize, quantitatively, the uncertainty and variability in estimates of exposure or risk. A secondary goal is to identify key sources of variability and uncertainty and to quantify the relative contribution of these sources to the overall variance and range of model results.

An analysis of variability and uncertainty should provide its audience with clear and concise information on the variability in individual exposures and risks; it should provide information on population risk (extent of harm in the exposed population); it should provide information on the distribution of exposures and risks to highly exposed or highly susceptible populations; it should describe qualitatively and quantitatively the scientific uncertainty in the models applied, the data utilized, and the specific risk estimates that are used.

Ultimately, the most important aspect of a quantitative variability and uncertainty analysis may well be the process of interaction between the risk assessors, risk manager and other interested parties that makes risk assessment into a dynamic rather than a static process. Questions for the risk assessor and risk manager to consider at the initiation of a quantitative variability and uncertainty analysis include:

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- Will the quantitative analysis of uncertainty and variability improve the risk assessment?
- What are the major sources of variability and uncertainty? How will variability and uncertainty be kept separate in the analysis?
- Are there time and resources to complete a complex analysis?
- Does the project warrant this level of effort?
- Will a quantitative estimate of uncertainty improve the decision? How will the regulatory decision be affected by this variability and uncertainty analysis?
- What types of skills and experience are needed to perform the analysis?
- Have the weaknesses and strengths of the methods been evaluated?
- How will the variability and uncertainty analysis be communicated to the public and decision makers?

One of the most important challenges facing the risk assessor is to communicate, effectively, the insights an analysis of variability and uncertainty provides. It is important for the risk assessor to remember that insights will generally be qualitative in nature even though the models they derive from are quantitative. Insights can include:

- an appreciation of the overall degree of variability and uncertainty and the confidence that can be placed in the analysis and its findings;
- an understanding of the key sources of variability and key sources of uncertainty and their impacts on the analysis;
- an understanding of the critical assumptions and their importance to the analysis and findings;
- an understanding of the unimportant assumptions and why they are unimportant;
- an understanding of the extent to which plausible alternative assumptions or models could affect any conclusions;
- an understanding of key scientific controversies related to the assessment and a sense of what difference they might make regarding the conclusions.

2. THE VALUE OF MONTE CARLO ANALYSIS IN QUANTITATIVE RISK ASSESSMENT

Not every assessment requires or warrants a quantitative characterization of variability and uncertainty. For example, it may be unnecessary to perform a Monte Carlo analysis when screening calculations show exposures or risks to be clearly below levels of concern (and the screening technique is known to significantly over-estimate exposure). As another example, it may be unnecessary to perform a Monte Carlo analysis when the costs of remediation are low.

On the other hand, there may be a number of situations in which a Monte Carlo analysis may be useful. Other situations could include when it is necessary to disclose the degree of bias associated with point estimates of exposure; when it is necessary to rank exposures, exposure pathways, sites or contaminants; when the cost of regulatory or remedial action is high and the exposures are marginal. Often, a “tiered approach” may be helpful in deciding whether or not a Monte Carlo analysis can add value to the assessment and decision. In a tiered approach, one begins with a fairly simple screening level model and progresses to more sophisticated and realistic (and usually more complex) models only as warranted by the findings and value added to the decision. Ultimately, whether or not a Monte Carlo analysis should be conducted is a matter of judgment, based on consideration of the intended use, the importance of the exposure assessment and the value and insights it provides to the risk assessor, risk manager, and other affected individuals or groups.

3. BASIC ISSUES IN HUMAN HEALTH AND ECOLOGICAL RISK ASSESSMENT

3.1. Defining the assessment questions

The critical first step in any exposure assessment is to develop a clear and unambiguous statement of the purpose and scope of the assessment. A clear understanding of the purpose will help to define and bound the analysis. Generally, the exposure assessment should be made as simple as possible while still including all-important sources of risk. Finding the optimum match between the sophistication of the analysis and the assessment problem may be best achieved using a “tiered approach” to the analysis that is, starting as simply as possible and sequentially employing increasingly sophisticated analyses but only as warranted by the value added to the analysis and decision process.

3.2. Selection and development of the conceptual and mathematical models

To help identify and select plausible models, the risk assessor should develop selection criteria tailored to each assessment question. The application of these criteria may dictate that different models be used for different subpopulations under study (e.g., highly exposed individuals vs. the general population). In developing these criteria, the risk assessor should consider all significant assumptions, be explicit about the uncertainties, including technical and scientific uncertainties about specific quantities, modelling uncertainties, uncertainties about functional forms, and should identify significant scientific issues about which there is uncertainty.

The main considerations which should be taken into account in the selection of models are:

- appropriateness of the model's assumptions vis-à-vis the analysis objectives;
- compatibility of the model input/output and linkages to other models used in the analysis;
- the theoretical basis for the model;
- level of aggregation, spatial and temporal scales;
- resolution limits;
- sensitivity to input variability and input uncertainty;
- reliability of the model and code, including peer review of the theory and computer code;
- verification studies, relevant field tests;
- degree of acceptance by the user community;
- friendliness, speed and accuracy;
- staff and computer resources required.

3.3. Selection and evaluation of available data

After the assessment questions have been defined and conceptual models have been developed, it is necessary to compile and evaluate existing data (e.g., site specific or surrogate data) on variables important to the assessment. It is important to evaluate data quality and the extent to which the data are representative of the population under study.

4. GUIDING PRINCIPLES FOR MONTE CARLO ANALYSIS

In this section we are discussing the principles of good practice for Monte Carlo simulation as it may be applied to human health and ecological risk assessments. It is not intended to use as detailed technical guidance on how to conduct or evaluate an analysis of variability and uncertainty.

4.1. Selection of input data for use in Monte Carlo analysis

The capabilities of current desktop computers allow for a number of "what if" scenarios to be examined to provide insight into the effects on the analysis of selecting a particular model, including or excluding specific exposure pathways, and making certain assumptions with respect to model input parameters. The output of an analysis may be sensitive to the structure of the exposure model. Alternative plausible models should be examined to determine if structural differences have important effects on the output distribution (in both the region of central tendency and in the tails).

Numerical experiments or sensitivity analysis also should be used to identify exposure pathways that contribute significantly to or even dominate total exposure. Resources might be saved by excluding unimportant exposure pathways (e.g., those that do not contribute appreciably to the total exposure) from full probabilistic analyses or from further analyses altogether. For important pathways, the model input parameters that contribute the most to overall variability and uncertainty should be identified. Once again, numerical experiments should be conducted to determine the sensitivity of the output to different assumptions with respect to the distributional forms of the input parameters. Identifying important pathways and parameters where assumptions about distributional form contribute significantly to overall uncertainty may aid in focusing data gathering efforts.

Dependencies or correlations between model parameters also may have a significant influence on the outcome of the analysis. Conducting a systematic sensitivity study may not be a trivial undertaking, involving significant effort on the part of the risk assessor. Risk assessors should exercise great care not to prematurely or unjustifiably eliminate pathways or parameters from full probabilistic treatment. Although specifying distributions for all or most variables in a Monte Carlo analysis is useful for exploring and characterizing the full range of variability and uncertainty, it is often unnecessary and not cost effective. If a systematic preliminary sensitivity analysis (that includes examining the effects of various assumptions about distributions) was undertaken and documented, and exposure pathways and parameters that contribute little to the assessment endpoint and its overall uncertainty and variability were identified, the risk assessor may simplify the Monte Carlo analysis by focusing on those pathways and parameters identified as significant. However, the risk assessor and risk manager should continually review the basis for "fixing" certain parameters as point values to avoid the perception that these are indeed constants that are not subject to change.

The choice of input distribution should always be based on all information (both qualitative and quantitative) available for a parameter. In selecting a distributional form, the risk assessor should consider the quality of the information in the database and ask a series of questions including (but not limited to):

- Is there any mechanistic basis for choosing a distributional family?
- Is the shape of the distribution likely to be dictated by physical or biological properties or other mechanisms?
- Is the variable discrete or continuous?
- What are the bounds of the variable?
- Is the distribution skewed or symmetric?
- If the distribution is thought to be skewed, in which direction?
- What other aspects of the shape of the distribution are known?

When data for an important parameter are limited, it may be useful to define plausible alternative scenarios to incorporate some information on the impact of that variable in the overall assessment (as done in the sensitivity analysis). In doing this, the risk assessor should select the widest distributional family consistent with the state of knowledge and should, for important parameters, test the sensitivity of the findings and conclusions to changes in distributional shape.

The risk assessor should always seek representative data of the highest quality available. However, the question of how representative the available data are is often a serious issue. The assessor should identify and evaluate the factors that introduce uncertainty into the assessment. In particular, attention should be given to potential biases that may exist in surrogate data and their implications for the representativeness of the fitted distributions.

Whenever possible, collect site or case specific data (even in limited quantities) to help justify the use of the distribution based on surrogate data. The use of surrogate data to develop distributions can be made more defensible when case-specific data are obtained to check the reasonableness of the distribution.

As a general rule, the development of data for use in distributions should be carried out using the basic principles employed for exposure assessments. For example,

- receptor-based sampling in which data are obtained on the receptor or on the exposure fields relative to the receptor;
- sampling at appropriate spatial or temporal scales using an appropriate stratified random sampling methodology;
- using two-stage sampling to determine and evaluate the degree of error, statistical power, and subsequent sampling needs;
- establishing data quality objectives.

In addition, the quality of information at the tails of input distributions often is not as good as the central values. The assessor should pay particular attention to this issue when devising data collection strategies.

Expert judgment is used, to some extent, throughout all exposure assessments. However, debatable issues arise when applying expert opinions to input distributions for Monte Carlo analyses. Using expert judgment to derive a distribution for an input parameter can reflect bounds on the state of knowledge and provide insights into the overall uncertainty. This may be particularly useful during the sensitivity analysis to help identify important variables for which additional data may be needed. However, distributions based exclusively or primarily on expert judgment reflect the opinion of individuals or groups and, therefore, may be subject to considerable bias. Further, without explicit documentation of the use of expert opinions, the distributions based on these judgments might be erroneously viewed as equivalent to those based on hard data. When distributions based on expert judgement have an appreciable effect on the outcome of an analysis, it is critical to highlight this in the uncertainty characterization.

4.2. Evaluation of variability and uncertainty

Variability represents the true heterogeneity or diversity inherent in a well-characterized population. As such, it is not reducible through further study. Uncertainty represents a lack of knowledge about the population. It is sometimes reducible through further study. Therefore, separating variability and uncertainty during the analysis is necessary to identify parameters for which additional data are needed. There can be uncertainty about the variability within a population. For example, if only a subset of the population is measured or if the population is otherwise under-sampled, the resulting measure of variability may differ from the true population variability. This situation may also indicate the need for additional data collection.

There are formal approaches for distinguishing between and evaluating variability and uncertainty. When deciding on methods for evaluating variability and uncertainty, the assessor should consider the following issues:

- variability depends on the averaging time, averaging space, or other dimensions in which the data are aggregated;
- standard data analysis tends to understate uncertainty by focusing solely on random error within a data set; conversely, standard data analysis tends to overstate variability by implicitly including measurement errors;

- various types of model errors can represent important sources of uncertainty; alternative conceptual or mathematical models are a potentially important source of uncertainty; a major threat to the accuracy of a variability analysis is a lack of representativeness of the data.

Numerical stability refers to observed numerical changes in the characteristics (i.e., mean, variance, percentiles) of the Monte Carlo simulation output distribution as the number of simulations increases. Depending on the algebraic structure of the model and the exact distributional forms used to characterize the input parameters, some outputs will stabilize quickly, that is, the output mean and variance tend to reach more or less constant values after relatively few sampling iterations and exhibit only relatively minor fluctuations as the number of simulations increases. On the other hand, some model outputs may take longer to stabilize. Ideally, Monte Carlo simulations should be repeated using several non-overlapping subsequences to check for stability and repeatability. Typically, the analyst has the least information about the input tails. These suggest two points:

- data gathering efforts should be structured to provide adequate coverage at the tails of the input distributions;
- the assessment should include a narrative and qualitative discussion of the quality of information at the tails of the input distributions.

Accounting for the important sources of uncertainty should be a key objective in Monte Carlo analysis. However, it is not possible to characterize all the uncertainties associated with the models and data. The analyst should attempt to identify the full range of types of uncertainty impinging on an analysis and clearly disclose what set of uncertainties the analysis attempts to represent and what it does not. Qualitative evaluations of uncertainty including relative ranking of the sources of uncertainty may be an acceptable approach to uncertainty evaluation, especially when objective quantitative measures are not available.

4.3. Presentation of the results of a Monte Carlo analysis

In view of presenting the results it have to be provided a complete and thorough description of the exposure model and its equations, including a discussion of the limitations of the methods and the results.

It is important to document thoroughly and convey critical data and methods that provide an important context for understanding and interpreting the results of the assessment. This detailed information should distinguish between variability and uncertainty and should include graphs and charts to visually convey written information.

The probability density function (PDF) and cumulative distribution function (CDF) graphs provide different, but equally important insights. A plot of a PDF shows possible values of a random variable on the horizontal axis and their respective probabilities (technically, their densities) on the vertical axis. This plot is useful for displaying:

- the relative probability of values;
- the most likely values (e.g., modes);
- the shape of the distribution;
- small changes in probability density.

A plot of the cumulative distribution function shows the probability that the value of a random variable is less than a specific value. These plots are good for displaying:

- fractiles, including the median;
- probability intervals, including confidence intervals;
- stochastic dominance; and
- mixed, continuous, and discrete distributions.

Risk assessors should never depend solely on the results of goodness-of-fit tests to select the analytic form for a distribution. Goodness-of-fit tests have low discriminatory power and are generally best for rejecting poor distribution fits rather than for identifying good fit. For small to medium sample sizes, goodness-of-fit tests are not very sensitive to small differences between the observed and fitted distributions. On the other hand, for large data sets, even small and unimportant differences between the observed and fitted distributions may lead to rejection of the null hypothesis. For small to medium sample sizes, goodness-of-fit tests should best be viewed as a systematic approach to detecting gross differences. The risk assessor should never let differences in goodness-of-fit test results be the sole factor for determining the analytic form of a distribution.

Graphical methods for assessing fit provide visual comparisons between the experimental data and the fitted distribution. Despite the fact that they are non-quantitative, graphical methods often can be most persuasive in supporting the selection of a particular distribution or in rejecting the fit of a distribution. This persuasive power derives from the inherent weaknesses in numerical goodness-of-fit tests. Such graphical methods as probability-probability (P-P) and quantile - quantile (Q-Q) plots can provide clear and intuitive indications of goodness-of-fit.

In a fashion similar to that for the input distributions, the risk assessor should provide plots of both the PDF and CDF for each output distribution, with one above the other on the same page, using identical horizontal scales. The location of the mean should clearly be indicated on both curves. Graphs should be accompanied by a summary table of the relevant data.

Covariance among the input variables can significantly affect the analysis output. It is important to consider covariance among the model's most sensitive variables. It is particularly important to consider covariance when the focus of the analysis is on the high end (i.e., upper end) of the distribution. Traditional deterministic (point) estimates should be calculated using established protocols. Clearly identify the mathematical model used as well as the values used for each input parameter in this calculation. Indicate in the discussion (and graphically) where the point estimate falls on the distribution generated by the Monte Carlo analysis. Discuss the model and parameter assumptions that have the most influence on the point estimate's position in the distribution. The most important issue in comparing point estimates and Monte Carlo results is whether the data and exposure methods employed in the two are comparable. Usually, when a major difference between point estimates and Monte Carlo results is observed, there has been a fundamental change in data or methods. Comparisons need to call attention to such differences and determine their impact.

Entirely different types of reports are needed for scientific and non-scientific audiences. Scientists generally will want more detail than non-scientists. Risk managers may need more detail than the public. Reports for the scientific community are usually very detailed. Descriptive, less detailed summary presentations and key statistics with their uncertainty intervals (e.g., box and whisker plots) are generally more appropriate for non-scientists.

The following are some recommendations that may prove useful for effective graphic presentation:

- Avoid excessively complicated graphs. Keep graphs intended for a glance (e.g., overhead or slide presentations) relatively simple and uncluttered. Graphs intended for publication can include more complexity.
- Avoid pie charts, perspective charts (3-dimensional bar and pie charts, ribbon charts), pseudo-perspective charts (2-dimensional bar or line charts).
- Colour and shading can create visual biases and are very difficult to use effectively. Use colour or shading only when necessary and then, only very carefully. Consult references on the use of colour and shading in graphics.
- When possible in publications and reports, graphs should be accompanied by a table of the relevant data.

• If probability density or cumulative probability plots are presented, present both, with one above the other on the same page, with identical horizontal scales and with the location of the mean clearly indicated on both curves with a solid point.

• Do not depend on the audience to correctly interpret any visual display of data. Always provide a narrative in the report interpreting the important aspects of the graph.

• Descriptive statistics and box plots generally serve the less technically oriented audience well. Probability density and cumulative probability plots are generally more meaningful to risk assessors and uncertainty analysts.

5. CONCLUSIONS

The purpose and scope of any human health or ecological risk assessment should be clearly articulated in a "problem formulation" section that includes a full discussion of any highly exposed or highly susceptible subpopulations evaluated (e.g. children, elderly etc.) The methods employed, including all models, data and assumptions that have significant impact in Monte Carlo simulation technique, should comprise the names of models and software used to generate the analysis.

Probabilistic techniques should be applied to the compounds, pathways and factors of importance to the assessment, as determined by sensitivity analyses or other basic requirements of the assessment.

Providing values obtained by calculation of exposures and risks using simulation techniques such as Monte Carlo, will allow comparisons between the probabilistic analysis and past or screening level risk assessments. Further, deterministic estimates may be used to answer scenario specific questions and to facilitate risk communication. When comparisons are made, it is important to explain the similarities and differences in the underlying data, assumptions and models.

The general framework and broad set of principles presented in the paper are important for ensuring good scientific practices in the use of Monte Carlo simulation technique a frequently encountered tool in human health and ecological risk assessment.

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HAZARD ASSESSMENT BENCHMARKS AND METHODS IN POST-MINING PERIOD

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Abstract: *The closure of mining operations does not lead to the complete and permanent elimination of risks and harmful effects are likely to affect the surface within the geographical limits of the old mine workings. Therefore, during the period following the extraction, traditionally known as the „post-mining" period, several kinds of problems may develop, sometimes just after the closure process but also, much later. These phenomena may generate major consequences for people, ground, water, atmosphere (gas) and infrastructure. They are also likely to have a major influence on regional development in mining areas. Therefore the elaboration of evaluation methods able to identify and assess the residual risks that may affect people and properties after closure is of great interest. Emphasizing the basics and benchmarks specific for post-mining hazard assessment process the paper develops a primary classification of applicable methods, with respect to main methods selection parameters such as data availability, economic considerations and regulatory considerations.*

Key words: *mining, closure, hazard, assessment, benchmark*

1. OBJECTIVES IN POST-MINING HAZARD ASSESSMENT

The majority of experts agree on the definition of a hazard. Generically, a hazard corresponds to a condition that has a potential for causing an undesirable consequence.

A specific hazard can be defined by the possibility of its occurrence and on the possibility of a specific magnitude at a specific location.

Hazard = probability of occurrence x possible physical magnitude

Considering surface instabilities, it may for example be characterized by diameter of a crater, horizontal deformations along a subsidence though, volume of unstable rock mass.

Predicting a-priori, the specific magnitude of a hazard, even from well-defined data, may be difficult. However, the types of failure mechanisms and their geometrical occurrences are well defined. Given a site's discontinuity feature and shallow working geometry and depth, the nature and the magnitude of a particular failure mechanism can be identified. This narrows

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considerably the range of probability values for specific magnitudes and simplifies the identification of the worst case. In some cases there can be only one failure mechanism. The specific failure mechanism case studies would be helpful for a practical understanding of the different site behaviors.

The probability of occurrence, reflecting a site's sensitivity to be affected by any of the events analyzed, is generally more difficult to quantify than the magnitude [7]. No matter what type of events are feared, the complexity of the mechanisms involved, the heterogeneous environment and the very partial available data that it is generally very difficult to assess quantitative probabilities (x % risk of a given event during a defined time period).

Priority may therefore be given to qualitative classification and characterization of a site's „predisposal" to suffer a particular type of damage or nuisance. Evaluation of this predisposition depends on the combination of different factors, which are favorable or unfavorable to the initiation and development of a given mechanism.

The „worst physical magnitude concept" can help in defining the highest level of consequence. The consequence can be quantified for a particular category or a range of features, e.g. specific on-site infrastructure, possible economic impacts (e.g. infrastructure, land value) or an all-encompassing consideration (e.g. public aspects, economic aspects, environmental aspects). Although monetary value is usually the basis for evaluation, a point system can be used when different subjects are jointly considered. Hazard and consequence combine to provide a quantified evaluation of risk.

Qualitative risk assessment will yield a qualitative scale ranging from negligible to high with the number of levels to be decided by the user. Quantification of risk refers to specific numerical values. For the purposes of risk prevention and regional development, the reference period for identifying risk levels is generally the long term typically human life-time (and not geological), by the date of mine closure.

Considering the mine closure process or post-mining management, the hazard evaluation, characterization and mapping phase is the main task of the experts. The objective is to identify potential dangerous areas that could affect safety of people and the integrity of infrastructure or property.

The following step (risk assessment phase) concerns local and national authorities rather than technical experts. It would have been difficult to provide an in-depth coverage of abandoned mine risk and related issues regarding the broad range of impacts, interests and jurisdictions particular to the many countries.

Nevertheless, some references to the risk concept should be used in practice, notably concerning the selection of the most adapted hazard assessment methods.

2. HAZARD ASSESSMENT SPECIFICALLY TO POST MINING

Post-mining hazard assessment studies are performed in specific contexts strongly influenced by the general principles defining hazard assessment methods.

First of all, the experts are confronted with very old mining structures, some of them having been abandoned several decades or centuries ago. Most of experts had to face the „collective memory loss" syndrome. Even if a mine has been active during a very long period, very rapidly after its closure (not more than one or two generations), many inhabitants are not aware of the existence of the closed mine, especially if all disused surface structures have disappeared.

This partly results from the progressive mixing of local population (some leaving the area, others settling there) but there is also a kind of a psychological „repressing process" that consists in forgetting most information that could potentially be inconvenient for the future. However, after appearance of the first problems, it is common that the local memory comes back. People then begin to remember data that they have never mentioned before, even if they have been asked previously. This contributes obviously to the unease in the collection of available data.

Besides old archive management processes (e.g. collecting, referencing) has not always been very well performed. Moreover, due to the very difficult social climate inherent to mine closure, most precious data (e.g. technical notes, maps) have been lost as a result of miners' anger and demonstrations of despair (e.g. documents thrown in shafts).

Mining methods and geological contexts are various. Potential problems may thus vary from almost undetectable subsidence effect to major collapses that may be very dangerous to people and property. Assessment methods must thus be either very adaptive or previously selected so as to fit properly the presupposed phenomena or mechanisms.

Surface occupation contexts can, also vary significantly. In many extended countries, many small-scale disused mines are located in arid and desert areas with no people or property on the surface. Assessment methods, the best adapted to mining sites will generally differ from those dedicated to large mines located in very urban areas. In this case, the hazard identification and location have to be as precise as possible.

It has to be reminded that post-mining hazard assessment studies are generally performed in a delicate socio-economical context. The concerned areas already had to face the closure of the most important local industrial activity (unemployment increase, need for re-industrialization). In this context, when properties are seriously affected, economical and psychological trauma may be severe.

In the same way, condemning surface occupation in large areas may limit severely the possibility of attracting new organizations in order to develop new industrial activities. This may contribute to the impoverishment of the area and the progressive departure of its inhabitants, especially young population. Risk assessment and management processes have to integrate this delicate issue.

3. SELECTION OF BEST ADAPTED HAZARD ASSESSMENT METHODS

3.1. Main parameters influencing the method selection

A very large variety of methods is available and can contribute to evaluate, characterize and describe post mining hazards. [1, 4, 5].

Some methods are explicit. The others are implicit. Some methods combine qualitative criteria, others are restricted to quantitative values. Among those methods, one may quote rating, upgrading, multi-criteria hierarchisation, cross-tables, empirical approaches, analytical or numerical modeling.

Some hazard assessment methods require a large amount of precise data and need much time for the analysis. They may then be quite expensive but provide very precise information concerning hazard levels and zoning. Other assessment methods require less data and are much simpler and quick to apply. They are thus much less expensive but generate a much higher level of uncertainty than the first ones. The first step of a hazard assessment process consists generally of the selection of the best adapted method to the local context.

Figure 1 illustrates the main stages of a hazard assessment process. Considering the definition of hazard, economic and regulatory considerations are not supposed to influence the assignment of hazard classes. Nevertheless, those socio-economical parameters may strongly influence the selection of the hazard assessment method. As mentioned earlier, the best adapted methods to small-scale isolated sectors are very different compared to those needed for very sensitive areas. Figure 1 lists 4 main parameters to be taken into account: economical considerations, regulatory issues, identification of hazards and availability of data.

Economic considerations

The nature of surface occupation and, more precisely, the nature of future surface utilization are usually the most important parameters in the selection process of the best adapted hazard assessment method. The objective is to optimize the cost and time period of the assessment process to the sensitivity of a given context (presence of buildings or infrastructure on surface, urbanism development). It is clear that more investment is needed for prevention in the areas with dense surface occupation with respect to the areas, where no people or property are concerned.

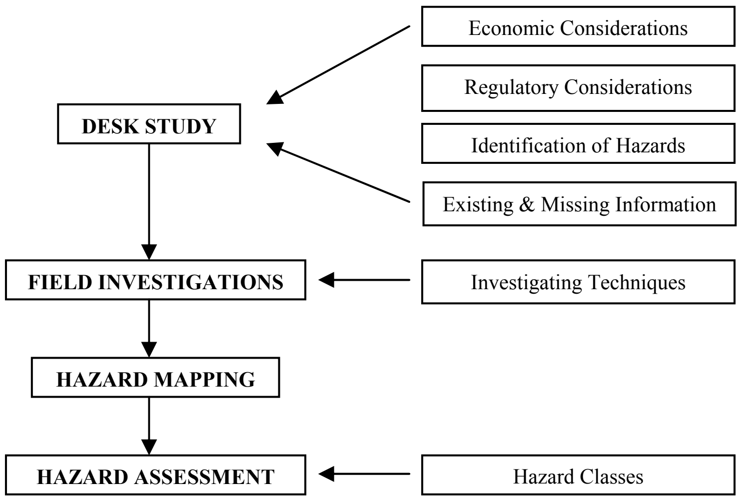

Figure 1. Detailed outline of the hazard assessment process stages

Normally, data collection within the process of a full hazard evaluation is significantly more expensive than the cost of assessing the hazard per se. Thus, one will use in the non-sensitive sectors, methods that minimize the exhaustiveness and precision and of requested data, even if the results of the hazard assessment process induces high levels of uncertainty. On the other hand, where important infrastructure is concerned by the supposed hazards, sufficient funding is required to collect large amount of information. Much more detailed and precise hazard assessment methods can thus be selected in order to limit, as far as possible, the uncertainties affecting the location and the characteristics of the feared event.

In some cases, the hazard assessment project may only have a limited objective such as developing a preliminary site evaluation. In this case, representative preliminary and sometimes empirical evaluations can be carried out using an initial cross-section of data [2].

Regulatory considerations

At the initial stages of hazard assessment, it must also be ascertained which jurisdiction, and national, provincial, state or local regulations will guide the process and influence on the selection of the hazard assessment method. Regulatory issues are presented in mining and environmental codes, adopted best practices/national guidelines.

In many countries, there are also legal professional associations of engineers, geophysicists and geologists that require the application of professional obligations. Those obligations may concern due diligence for known hazardous sites, and the application of codes of responsibility in all work, such as the application of best methods and communication of all findings.

Identification of phenomenon

Every assessment method has its specificity. None is adapted to every phenomenon, context and objective that can be encountered in the post mining management process. Thus, it is very important to identify the pre-supposed phenomena that can develop on a given site in order to properly select the hazard assessment method.

As an example, in the case of sinkhole development, simple analytical or empirical approaches appear more adapted than complex and expensive numerical modeling. Most of the analysis will then consist in collecting data on geometry of residual voids (e.g. underground visits, analysis of maps, boreholes visual investigation).

On the contrary, in a complex geo-mechanical context (e.g. incorrect superimposition of pillars, non-homogeneous stress distribution within the rock mass, presence of major discontinuities such as faults, influence of water or time), advanced and powerful numerical approaches may be developed to enhance the understanding of the triggering mechanisms.

Available data

The extension of the disused mining works and the amount of available data may also influence the selection of the assessment method. When only a few data are available, qualitative methods may be used taking benefit, of the expert judgement.

On the contrary, when large amount of data is available (very large mining field with different mining methods and contexts), one will consider the interest of the decision-making aid methods. Those methods were developed to assist experts in taking benefit of large amounts of data.

By pairing those pre-supposed parameters, it is possible to evaluate the probable consequences of the feared phenomena (in terms of predisposition and severity). Table 1 gives an example of a matrix that can be used as a help to define the best-adapted class of hazard assessment methods depending on the „sensitivity” of a given site (from very simplified to precise qualitative approaches) [3].

Table 1. *Example of a matrix helping to identify hazard assessment method classes*

Frequency of occurrence	**Severity**			
	Catastrophic	**Major**	**Minor**	**Negligible**
Frequent	A	A	A	C
Probable	A	A	B	C
Occasional	A	B	B	D
Remote	A	B	C	D
Improbable	B	C	C	D

Action Guide

Risk level	**Analysis**
A	Detailed quantitative
B	Semiquantitative
C	Qualitative
D	Not required

3.2. Hazard assessment methods adapted to low consequence (non-sensitive) cases

Some abandoned mining sites are located in areas situated far from a significant by developed zone on the surface.

Where the areas are uninhabited and not adopted for future surface occupation development (e.g. arid or marshy zones, mountainous areas), it is generally not necessary to perform specific hazard assessment analyses. One will then identify and localize the mining sites and draw global hazardous envelopes on cartographic documents for surface occupation management. This process ensures memory preservation for future generations.

Such a configuration is principally met in very large geographical territories with a low density of population (e.g. Canada, Australia). For highly populated countries (e.g. France, Germany) or mining areas where towns have grown up over the workings, mining works are rarely encountered in areas without surface infrastructure in the surroundings. Such a configuration requires a limited budget for the assessment process

3.3. Hazard assessment methods adapted to moderate consequence (semi-sensitive) cases

Semi-sensitive case configuration (average hazard class in sectors characterized by moderate surface occupation) is probably the most traditional and widespread situation. Relatively simple, fast and mainly qualitative hazard assessment methods are well adapted to this context.

Qualitative expert analysis

Expert analysis consists in using the knowledge of an expert in order to define the nature of the hazards likely to develop at long term above an old mining site. The analysis is mainly qualitative and attempts to take benefit of the available data (mining maps, past problems, visits of mine workings). Often, the analysis is based on the experience feedback, considering that previously occurred events on the same (or a similar) site constitute a reference index for the prediction of phenomena likely to develop in the future.

Even purely qualitative, the analysis should be as formalized as possible. Several tools can contribute to this. One will quote for example the principle of the matrix given in table 2.

Table 2. *Example of Hazard Class identification*

<table>
<tr><th>Event Probability / Magnitude</th><th>Very low</th><th>Low</th><th>Moderate</th><th>High</th></tr>
<tr><td>Very limited</td><td rowspan="4">Low hazard level</td><td colspan="3"></td></tr>
<tr><td>Limited</td><td rowspan="3">Moderate hazard level</td><td colspan="2"></td></tr>
<tr><td>Moderate</td><td rowspan="2" colspan="2">High hazard level</td></tr>
<tr><td>High</td></tr>
</table>

Analytical modeling

Analytical modeling is based on explicit mathematical equations used for the analysis of the mechanical stability. It requires important simplifications of the real problem and provides thus information that has to be interpreted in terms of trends. The simplicity of calculations makes possible to do many simulations and to perform a sensitivity study of the effect of various parameters.

Empirical approaches

The empirical approaches are based on the experience feedback analyses. They take benefit of previous observations, relatively simple mathematical relationships between parameters characterizing the site and a criterion to determine.

One may quote for example empirical laws established to characterize safety factors for mine pillars (e.g. according to the dimension of the pillars, of the nature of material, depth of work). The „10 x H law", often quoted, constitutes another example. It considers that a sinkhole hazard becomes negligible, in where specific geological configurations, where the overburden thickness is higher than 10 times the height of the opening forming the mine workings [9].

3.4. Hazard assessment methods adapted to high consequence (sensitive) cases

The configurations corresponding to very unstable mines or very urbanized areas require precise and detailed hazard assessment methods. Because those methods need quantitative data, detailed field investigations (e.g. core drillings) are often performed depending on the requested information.

Numerical modeling

The context, the failure mechanisms or the constitutive laws are too complicated to make possible analytical modeling, numerical models may be used. There are many codes available. They have been developed based on different methods (e.g. Finite Element Methods, Distinct Element Methods, Finite Difference Method, etc.). Each method has its specificity and requires a high level of expertise [8].

Numerical modeling does not constitute, by itself, a hazard qualification method. It contributes to a better understanding of the complex instability mechanisms. It thus assist the expert in the validation (or the invalidation) of assumptions.

Decision-making aid methods

Decision-making aid methods applied to post mining issues consist in comparing a great number of mining areas between each other in order to formalize comparison criteria as well as rules that help the expert to identify the most critical areas and the zones of the least risk. Several techniques can contribute to this classification. One will quote the multi-criteria methods by hierarchisation [6], the methods by ranking or neural networks. Such methods are mainly justified when the quantity of information is very significant and the criteria likely to intervene in classification are numerous.

Probabilistic approaches

Probabilistic approaches in post-mining are not well developed yet. Indeed, a realistic probability quantification requires a good knowledge of the problem and the application of a quantifying model (analytical, numerical) representing the probability of occurrence of the problem [7]. Semiquantification of the occurrence probability of a hazardous event can be based on case studies representing stable, limit and unstable cases. This can lead itself to a statistical failure probability.

In practice it is difficult to have sufficiently quantified variables related to the initiation and continuation of the hazard. A simplified qualification of hazards can be achieved if an appropriate scale is used. This is matched to the nature and the extent of the problem understudy.

4. CONCLUSIONS

The quality of a hazard assessment is closely related to that of available data. The first (if not the only) stage of the assessment is a desk study. This step aims to gather existing data and applicable knowledge from which pertinent engineering information can be extracted and used in order to identify and evaluate the hazard. Desk study may be complemented by a field study, with the objective to look for supporting information concerning mining works, previous problems and general environmental considerations.

In the case of very old mine workings (serious lack of data) if the surface occupation is very sensitive to potential disorders (urban environment, important infrastructure), complementary information may be necessary. This then leads to a field complementary investigation stage. Due to the difficulty in collecting abandoned mining data, practitioners and organizations of record must be aware of the trend of basic loosing and site information with time. Even for contemporary mining companies, very often, only a few mine plans and drawings survive mine closure. Most of them have been lost or have deteriorated over the years. Often, a mine site no longer has surface structures marking its location and the presence of underground mine openings is unknown. Some information may not be reliable or is very approximate.

Hence, there is a need to identify, evaluate, catalogue and store site information as a first step in keeping accurate records and data. This will be necessary for supporting abandoned site management and projects such as hazard assessment and problem mitigation, as well as active mine decommissioning requirements. Furthermore, well-sorted archives facilitate many other applications such as: standardization of the type of information required by national organizations, 2D and 3D graphical site reconstruction, storage of new site information from subsequent projects, and the development and support of regulatory requirements.

While information currently is easier to handle and transmit in electronic format, the original information must be properly stored in case problems arise from electronic storage, such as future incompatibility of software's and lost databases.

Analysis of aerial photos taken at various times provides useful information on the evolution over time of the old workings and the environmental conditions.

There is a wide range of information and sources that a practitioner involved in abandoned mine geotechnical issues should review. It is also at this stage that required but missing information is identified. A post mining hazard assessment process needs to use, as frequently as possible, a detailed inspection on the site. Only the knowledge of the site and its history produces a satisfactory degree of accuracy in the hazard assessment process. This data gathering also provides a strong basis for the expert credibility and his judgment with respect to the local population.

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THE POTENTIAL FUEL OF DEBRIS HEAP OF PETRILA

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Abstract: *The debris heap of Petrila contains the coal mass that can be used energetically. The work presents the potential of the researched heaping branches and the ways of practical usage and of ecological rehabilitation after exploitation.*

1. DESCRIPTION OF THE DEBRIS HEAP OF PETRILA

In the North East of Petroşani depression, in the near vicinity of Petrila town, it is placed the heaping perimeter that is developed close to the mining space and it continues up to the South slope of Rusalin brook and up to the North slope of Maleia brook. The both brooks are affluents of Jiul de Est.

The debris heap is made up of a mixture of hard rocks and soft rocks, resulted from the technological processes from the underground and from the preparation of the coal, which have been deposited by certain one step funicular plants, from 8-10 m to 30 m in height.

The petrographic nature of the heaped material highlights the presence of argillaceous rocks, of marls, of microsandstone and of argillaceous sandstones, of coal shale and of **coal fragments**, generally rocks easily breakable.

The direct foundation of the heap is made up of dust and argillaceous rocks whose resistance features are influenced by the humidity as a result of the high content of pelitic material. It can be drawn the conclusion that there are no big differences between the two types of samples (foundation and heaped mixture) from the point of view of granularity.

Its deposition was done discordantly, by manual or automatic overturning of the cups on the funicular direction, after which the deposited material was pushed by the bulldozer and the top surface of the heap turned to a relatively flat form.

Branch I was built in 1933 when Preparation Plant of Petrila was set in motion and the debris material that occupies approximately 7.5 ha was deposited.

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Branch II was built around 1950 and a 8.5 ha surface was covered by heaping.

In 1958, the **branch III** was built, branch that occupied an even larger surface (9.5 ha). The heaping ceased on the branch III in 1973.

The branch IV was set in motion in 1973 and it presents a higher deposit capacity, with 1700 m in length and having a funicular plant with 12 posts.

The branch V was set in motion in 1977. This funicular plant has 1560 m in length and it has 11 posts.

The **branch VI** had been designed and its construction began; only the service posts of the plant were mounted and then the work was abandoned.

The distribution of the five heaping branches from the North to the South is: the branches (III), (I), (II), (V), then the most southern branch (IV). Placed as a fan, with angles between them of 9º, 14º, 16º and 24º, the draining branches start from the angular station and they totally occupy approximately 47.5 ha of ground.

From the observations made on the field, it results that when building the first three branches, water sources were covered by the heaped material without draining them initially.

The existing lakes from the North of the branch V are especially remarked, from which the one on the area of P_2 post extends on approximately 100 m in length and 10-30 m in width and the one from the P_3, P_4 post extends on approximately 345 m in length on South-West direction and on a 80-100 m in width. To these lakes, the lakes formed next to the P_9 and P_{10} post at South of the branch V of the heap with a surface of approximately 1800 – 3000 m² are also added. The surfaces of these lakes depend on the season and on the presence of the rainfalls in the area. From these reasons, the presence of certain swampy areas downstream the branch IV in the area of P_3 and P_4 posts and between these branches is also remarked.

The ground on which the heap was initially placed was a table land with quiet morphology, with small slopes that did not exceed 10^o, generally tilted from South to North, the area including the bubble level between the southern affluents collected by Jiul de Est River and the northern affluents of Maleia brook.

The southern area, from Maleia village, presents a morphology slightly modified by the influence of the underground exploitation of the coal layers, the slope becoming more tilted than 10^o and while exploiting the branch no. V, in certain areas, land slides towards the valley of Maleia brook occurred. But in the studied area, the vegetation is represented especially by poor pasture lands and brushes, among which it is mentioned especially the birch tree, the willow tree and the locust tree, and the area from Maleia, where the village houses are placed, there are fruit trees and plots of land under crop.

It was estimated that due to the big extensions of the heap, the meteorological factors and especially the presence of the rainfalls, influence the stability of the heap by reducing the resistance features and by increasing the volumetric weight of the rocks or by forming the water pressure from the pores and of hydrodynamic pressure.

2. PRESENT SITUATION OF THE DEBRIS HEAP

Beginning from the angular station placed in the Westside of the heaping table land, the five heaping lines are disposed as a fan, from which the branches (I), (II), (III) and (IV) are preserved at present, and the branch (V) is in functioning state.

These heaps were formed by using the funiculars for transporting the debris material resulted from the digging processes of the mining works from E.M.Petrila and the debris resulted from preparing the cola in the Preparation Plant of Petrila.

The press cakes resulted from the preparation technological process (from the filters), the debris slimes impregned more or less with flotation reagents, light oils with or without phenols, resins, gas, Diesel fuel, wood parts, etc. were also deposited here.

In the Table no. 1, it can be observed the percentage of different types of rocks deposited in the heap of the Preparation Plant of Petrila

Table no. 1 Types of rocks from the heap of Petrila:

Crt. no.	**Lithologic types of rocks**	**Percentage [%]**
1	clays, marl clays and sandy clays	30
2	marls, argillaceous and sandy marls	6.6
3	sandstones, marl sandstones, argillaceous sandstones	59
4	conglomerated argillaceous shales	4.4

As for the soils from the branches of the heap of Preparation Plant of Petrila, there are more types of bad land presented in the Table no. 2.

Table no. 2. Types of bad lands from the heap area:

Crt. no.	**Type of bad land**	**Surface (ha)**
1	Bad land with crumbling phenomena	0.281
2	Bad land with erosion phenomena in depth (ravines)	0.1
3	Surface of naturally regenerated heap with well adapted vegetation	3.47
4	Surface of naturally regenerated heap with poor adapted vegetation	9.513
5	Land with excessive surface erosion	7.008
6	Bad land with partially grassed surface	4.513
7	Submersed bad lands	6.21
8	Table lands	3.025
9	Bad land with swamping phenomena	1.48
	TOTAL	36.6

The debris heaps negatively influence the environment regarding the quality of surface and underground water, the quality of the air, of the vegetation and of the general aspect of the area and of the comfort.

For protecting the environmental factors from the areas affected by the presence of the heaps, a series of measures must be taken, among which the following are mentioned:

a. Measures by which the pollution of underground and surface water with noxious substances should be prevented. It is not the case of heap of Petrila as the infiltration waters are in reduced quantities, they are flown in the rivers on the area and they do not contain noxious substances.

The infestation problems of the ground waters occur only regarding the abandoned waste deposit without preserving measures, deposit that is placed close to the returning station of the funicular on the branch R-V.

b. Measures by which the dislodging of the material heaped by the running waters should be prevented. Due to the fact that there are no water courses in the area of the five branches, this possibility does not exist. The only dislodging of the material are caused by the flow waters from the natural slopes through the ravines formed.

Closing these ravines and assuring certain slopes of the heaping platforms towards North will lead to avoiding these dislodging of the material.

c. Measures for avoiding the picking-up of dust particles by the predominant winds. It is not the case of these heaping branches, as the deposited material has a big granularity in mixture with the small and fine one. Yet, dusty sources are created by disintegrating and altering the rocks from the natural slopes and, for this reason, the execution of some grassing works of the natural slopes in the areas in which the depositing works are not effectuated anymore and in the ones where this was not naturally effectuated is recommended.

All these measures regarding the assurance of heap stability and the environment protection are, in the same time, preventing measures regarding the management of the emergency situations caused by dangerous meteorological and geotechnical phenomena, accidents to the engineer land constructions and accidental pollutions, stipulated in the Directive no. 638/12.05.2005 of the Ministry of Administration and Interior and the Ministry of Environment and Water.

3. RESEARCHES REGARDING THE EXTRACTION AND PRACTICAL USAGE POSSIBILITIES OF THE FUEL'S MASS

The study had in view the establishment of the fuel's mass and the quantity of the coal that can be recovered, following a processing flow that should lead to a product that can be used energetically in thermal plants, at a heating power of 3600 kca/kg.

The analyses samples have been made up by their compounding from the samples taken from the heaping branches no. I, II and III and the following were determined: ashes content, humidity, volatile matters, the inferior heating power and the fixed carbon. The results of the analyses are synthetically presented in the Table no. 3

Table no. 3 Results of the analyses of the fuel's and debris mass from the heap of Petrila

Feature	Branch I			Branch II			Branch III		
	Value			Value			Value		
	Min.	Med.	Max.	Min.	Med.	Max.	Min.	Med.	Max.
Ashes to anhydrous [%]	74.03	76.56	78.21	72.86	76.04	79.61	74.29	76.19	78.09
Volatile matters [%]	17.903	18.357	18.754	17.388	18.371	18.792	17.877	18.589	20.479
Inferior heating power [kcal/kg]	1055	1213	1450	1030	1262	1480	1090	1121	1380
Fixed carbon [%]	3.2282	5.0803	7.8727	3.0034	5.5887	8.3961	3.7441	5.2237	7.0112

It is observed a certain uniformity of the average results by branches and that were established based on the analysis of 11 samples taken from each heaping branch. The most homogeneous samples are observed to be on the heaping branch no. III. The smallest value of the heating power was found on a sample from the branch no. I, and the best heating power on a sample collected from the heaping branch no. II. From the analysis of these results, it cannot be localized an area with a high content of fuel's mass which could make the object of a selective exploitation. If we pass to the extraction of the fuel's mass, this one must be made by exploiting the hole deposit, possibly in the order II, III, I of the branches for which these analyses were done.

The granulometric analyses have been done for the samples taken from the branches of the heap. For each granulometric class, the ashes content has been also determined. The results are presented in the Table no. 4

On the old heaping branches, no. I, II and II it is observed a high percentage of the small and fine material, the result of the granulometric degradation under the influence of the natural factors, while on the last branches for depositing the debris, IV – V, the granulometric distribution is more homogenous with a higher percentage of the coarse material.

Table no. 4 Granulometric and ashes distribution by classes of the debris from the heap of Petrila

Granulometric class, mm	Average R I		Average R II		Average R III		Average R IV-V	
	Quantity [%]	ashes content [%]	Quantity [%]	ashes content [%]	Quantity [%]	ashes content [%]	Quantity [%]	ashes content [%]
+80	4.1	69.20	5.12	80.81	7.89	76.43	5.94	85.71
40 – 80	6.6	67.13	8.92	78.13	12.00	71.40	22.01	78.73
20 – 40	9.7	69.06	14.99	75.71	15.75	65.42	21.05	73.07
10 – 20	30.3	63.02	18.35	66.07	13.86	73.53	11.66	74.85
6.3 – 10	5.5	72.52	10.84	78.58	13.22	73.27	9.21	75.24
2.5 – 6.3	11.9	77.22	15.14	79.22	14.67	73.52	11.15	76.23
0.63 – 2.5	19.3	79.6	18.18	79.96	15.01	73.29	12.06	77.04
– 0.63	12.6	82.02	8.46	82.55	7.60	80.25	6.92	75.93
TOTAL/ AVERAGE	100.00	72.05	100.00	76.57	100.00	72,77	100.00	76.50

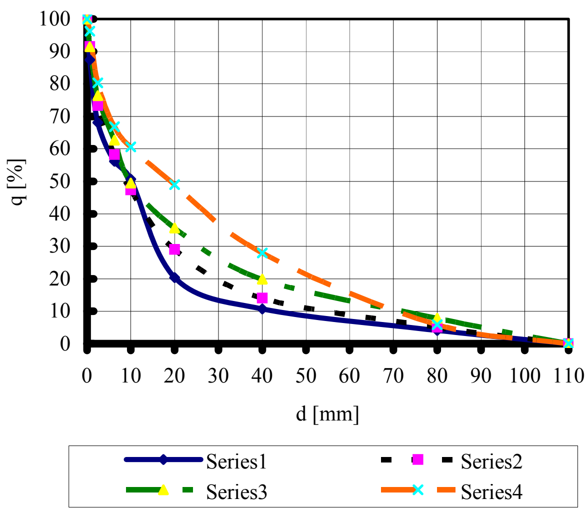

Fig. no. 1 Granulometric curves of the debris accumulated for the debris on the branches of the heap of Petrila (the series number corresponds to the heaping branches)

The distribution by granulometric classes of the ashes from the debris is graphically presented in Fig. no. 2.

The granulometric class under 10 mm has an increasing ashes content, over 75% that could be a reason that, by classifying, it should be removed from the reprocessing operation. The class 0 – 0.63 mm has a high ashes content due to the argillaceous material in this class and which can encourage obtaining the autogenous dense environment necessary to reprocessing.

For the branches I and III, the class that can be treated for recovering the fuel's mass is 10 – 80 mm, while for the branches II and IV-V, only the class 10 – 40 mm contains a fuel's mass that can be extracted even by a classification.

The ashes content by classes varies within reduced limited and, consequently, the separation of a class that has a significant percentage of fuel's mass and that does not need reprocessing operations is not justified.

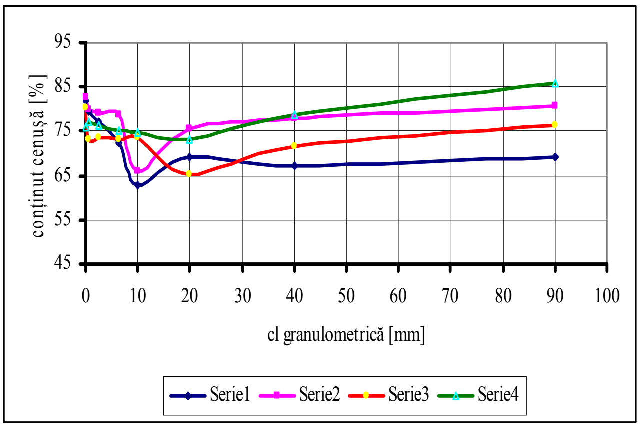

Fig. no. 2 Variation of the ashes content by granulometric classes
(the series number correspond to the heaping branches)

From the tests made on the samples collected on the branches IV – V, it has been observed that by breaking the coarse classes, it does not result a significant quantity of coal components and, consequently, it cannot be recommended such an operation for the other 3 heaping branches.

The class +80 mm in present in a proportion between 4 – 7%, and the raised ashes of this granulometric type removes the variant of breaking and only an inverse hand picking can be done, when the presence of the coal mass is observed as a result of the visual observations.

4. PREPARATION CURVES AND ESTIMATION OF RESPONSE TO REPROCESSING

For estimating the response way when preparing the material from the researched heap structure, it was used the analysis by densimetric fractions of the collected samples and whose results are presented centralized in the Table no. 4, from the point of view of the extractions and of the ashes content.

These primary data have formed the basis of calculation for tracing Henry – Reinhardts (HR) curves used for determining the main parameters and the theoretical preparation indicators.

For estimating the preparation, we used the method of static variation coefficient of reprocessing and that is based on the static variation indicator for reprocessing "$\boldsymbol{i_p}$", also known as washability value, with the relation:

$$i_p = 100\frac{\sigma}{a}$$

where:

$$\sigma = \sqrt{\frac{\sum (c_i - a)^2 \cdot v_i}{100}}$$

c_i – the ashes content of each densimetric fraction that floats;

a – the ashes content from raw debris;

v_i – extraction in weight of each densimetric fraction;

The relation between the static variation coefficient of reprocessing and the reprocessing capacity of the coals is presented in the Table no. 2.4.

Based on these reasons, we used to the application of the method for the debris that is to be subjected to reprocessing for recovering the fuel's mass that is still contained.

Table no. 5 Method of static variation coefficient for reprocessing:

Value of static reprocessing indicator, i_p	**Group**	**Estimation of response to reprocessing**
175	I	Very easy preparation
150 – 175	II	Easy preparation
125 – 150	III	Pretty easy preparation
100 – 125	IV	Moderate preparation
75 – 100	V	Submoderate preparation
50 – 75	VI	Difficult preparation
< 50	VII	Very difficult preparation

The aspect of the HR curves confirms the presence of the associated particles in high percentage, even in the conditions of a breaking operation of the class +40 mm (in the case of the debris from the branches IV – V) and, consequently a difficult response to preparation.

Applying the formulas presented at the densimetric analysis of the debris from each heaping branch that is to be subjected to reprocessing, an **indicator of variation response to reprocessing** results, that is presented in the Table no. 6.

Table no. 6 Estimation of the preparation of the coal debris:

Heaping branch	**Square average deviation [s]**	**Preparation indicator [i_p]**
R I	14.40	20.07
R II	14.21	18.96
R III	14.94	20.26
R IV - V	16.95	22.02

The washability value has values within the 7th group corresponding to a very difficult preparation, aspect approved also by the estimation of the preparation based on the aspect of the HR curves. There are no significant differences between the heaped debris on the 5 branches of the deposit of Petrila.

The curves aspect confirms the presence of the associated particles in high percentage, even in the conditions of a breaking operation of the class + 40 mm, in the same way in which it was proceeded on the samples taken from the branches IV-V, so, a difficult response to preparation, that can be reduced by a higher degree of breaking or by concentrating in more technological steps, in the same way in which it is proceeded when preparing the green coal. The coarse debris material will be transported in the construction area of the placing barrier of the final debris heap after extracting the coal mass that can be placed between the existing branches III – IV.

The probable results when using the energetic potential by obtaining a fuel with a heating power of 3600 kcal/kg are presented in the Table no. 7.

5. PRACTICAL PROBLEMS RAISED BY THE PROCESSING OF THE DEBRIS MATERIAL DEPOSITS

The practical usage of the secondary energetic resources presented in the mass of the debris deposits formed over the time at the Preparation Plant of Petrila, requires a detailed knowledge of the areas that could make the object of extracting and processing, taking into account the possibility of accessing with the equipment for extracting the raw material and for transporting that concentrate of the final debris, as most of the heaps are placed on rough grounds, and the existing roads cannot be used in the conditions of a higher level of rainfalls.

The extraction of the fuel's components from the debris heaps will need modifications of the existing geo-topometrical configurations of the heaping branches, occasion for estimating works of ecological rehabilitation, activities that are encouraged by governmental programs.

Table no. 7 Estimated quality and quantity parameters

Parameter	Measurement unit	Estimated value			
		R I	R II	R III	R IV - V
Average ashes of sample:	[%]	76.56	76.04	76.19	76.49
Average heating power of raw debris	[kcal/kg]	1213	1262	1121	1250
Theoretical extraction (at δ_s = 1.92 [kg/dm^3]	[%]	18.2	13.9	16.4	17.7
Average ashes of recovered product:	[%]	40.1	42.5	41.8	40.18
Product heating power	[kcal/kg]	3600	3600	3600	3600
Theoretical annual quantity extracted from the stock on the branch IV - V	[t]				54 500
Total quantity of fuel's mass that can be used	[t]				450 000

Redepositing the final debris will cause specific problems due to the even higher content of argillaceous minerals than it can be currently found in the heaps, to which it can be added the finer granularity, as the wet processing will encourage the continuous degradation of the component mineral material.

It is expected that the product resulted from exploiting the heaps could cause additional problems when burning it in the fireplaces for producing the energetic or thermal agent.

The extraction of the fuel's components from the debris heaps will need modifications of the existing geo-topometrical configurations of the deposits, occasion for estimating the works of ecological rehabilitation, activities that are encouraged by governmental programs.

The purchase or the design and the local production of certain mobile plants for extracting and processing the debris supposes investments for which financing sources are necessary, sources that must be identified especially in facilities areas for ecologic rehabilitation.

CONCLUSIONS

✓ The presence of a significant potential of fuel in the deposited debris mass justifies the research of recovering possibilities, at least its partial recovering and the practical usage in energetic along with the current production

✓ The debris that is to be subjected to the preparation for recovering the fuel's mass has a very difficult response to reprocessing;

✓ The possible technological options (classification and extraction, class 10 – 40 mm; washing type 10 – 80 mm; washing class 10 – 40 mm, with or without breaking the class + 40 mm, etc.) do not present major differences regarding the response to reprocessing, even in the conditions of breaking a part from the debris;

✓ The decision of choosing the design variant will be made on the criterion of the minimum consumption of energy, so on avoiding the breaking operation

✓ The presence in high percentage of the clay in the debris mass encourages the option for reprocessing in autogenous dense environment, formed by thickening the washing waters and the symptotic classification for obtaining the clay-based suspension.

✓ We estimate that redepositing the final debris will cause specific problems due to the even higher content of argillaceous minerals than it can be currently found in the heaps, to which it can be added the finer granularity, as the wet processing will encourage continuous degradation of the component mineral material.

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INDELAND – LARGE LAKE IN THE RHINELAND AS A SUSTAINABLE LIGNITE FOLLOW-UP LANDSCAPE

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Abstract: *The mining of lignite in the western German surface mine of Inden will end in the year 2030. Some syndicates' aim is to realise that by the year 2030 no external material from the Hambach surface mine should be dumped into the residual pit in Inden, but a large lake is to be created. With a surface of 1,100 ha it would be Germany's fourth largest lake and it would add a new component to the profile of the region -the so-called 'Indeland'- as an innovative economic area with new leisure qualities.*

1 INTRODUCTION

First, after the termination of any surface mining of a deposit cavities remain, which have to be transferred to a useful future utilisation. By the year 2030 the German large-scale surface mine of Inden will be confronting this issue. The Inden lignite surface mine is one of three large surface mines in the Rhenish District in West Germany within the city triangle of Aachen, Mönchengladbach and Cologne (see fig. 1)

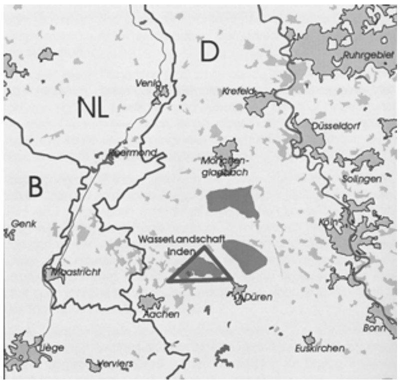

Figure 1. Planned layout of the Inden surface mine [1]

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In the Inden surface mine, the operator RWE Power reaches a yearly output of approx. 20 – 25 million tons of lignite, which is exclusively used for power generating in the Weisweiler power plant. In the process, 80 – 85 million tons of waste occur, which is dumped in the internal spoil tip. The operator owns a licence to extract coal deposits of approx. 510 million tons of lignite until the year 2030. On the other side, there are 1,600 million m³ of waste that will have to be moved to win the coal. Continuous mining technology exclusively consists of the combination bucket wheel excavator –conveyor belt – spreader. The lignite seams are as thick as 45m and are excavated up to a depth of 230m [2].

60 % of the population of the community of Inden have been or will be affected by relocation as the mine progresses. In the next years to come the resettlement of the village of Pier will be advanced (see fig. 2).

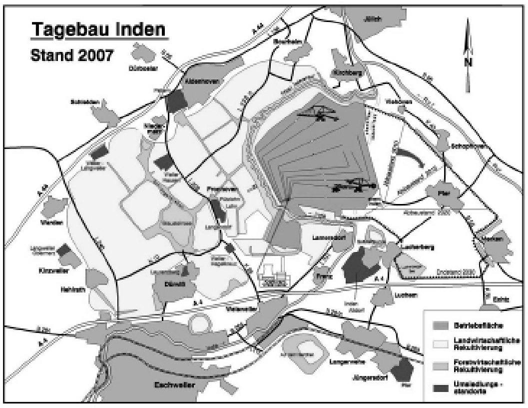

Figure 2: Area utilisation of the Inden surface mine [3]

2 INTENDED FUTURE UTILISATION OF THE INDEN SURFACE MINE

As the subsequent predominant future utilisation of the remaining pit of the Inden surface mine located in the area of the community of Inden the backfilling with overburden is planned. In the authorised lignite plan, spatial partial section Inden II, a small residual lake with a surface of approx. ha is planned (see fig. 3).

With an open operational surface area of 1,300 ha and an authorised total surface of 4,500 ha it is obvious that the waste accruing in the Inden surface mine cannot be used for the final backfilling. The resulting deficit of material of a total of 1,400 million tons of waste is to be transported on a conveyor belt yet to be built from the Hambach lignite mine from 2030 [2].

The main future utilisation is to be agricultural according to the lignite plan Inden II. Areas for forestry and water surfaces have a rather small part in the planning (see fig. 3). This unequal distribution of the ways of future utilisation is also an expression of regional political aims at the beginning of the 1980s. The chamber of agriculture as the political representative of the agrarian economy was at the time above all trying to secure as many areas as possible for agriculture.

3 CHANGE OF BASIC CONDITIONS

In present, the political intention of agriculture to cultivate as many areas as possible is no longer so predominant. The number of active farmers has meanwhile significantly dropped, which is also due to the fact that primary production in Europe in the era of globalisation has come under strong economic pressure.

Presently, the people in the region between Aachen and Cologne have become more aware of the significance of nature, recreation and leisure structures. Some reference projects were able to prove that a restructuring of a site formerly shaped by industry into an attractive landscape entails some advantages.

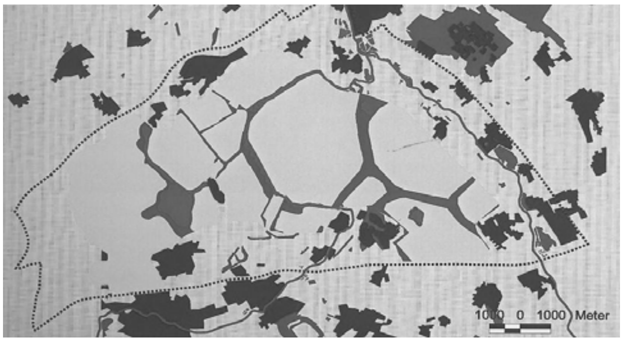

Figure 3: Lignite plan Inden II – completely recultivated surface mine [1]

Some follow-up projects of lignite mining, too, have come under the direct sphere of influence of the Inden surface mine. The residual lake of the predecessor surface mine Zukunft, the Blausteinsee lake designed for leisure use was very well accepted by the population and the surrounding municipalities. The Blausteinsee lake has ever since been regarded as a model recultivation project.

The relocation of the Inde river flowing through the Inden II surface mine area over a length of 4 km was embraced by the great majority of the adjacent owners and the neighbouring municipalities. Since 2005, the Inde has flowed over a length of approx. 12 km through former surface mine area on the dump side of the Inde surface mine. In the layout of the new Inde bed course the idea of a natural river with a far spreading floodplain and meadow landscape was successfully realised (see fig. 4).

Figure 4: River Inde (aerial photo) – relocated section along the internal spoil tip of the Inden surface mine [3]

4 INITIATIVE OF THE INDEN LOCAL AUTHORITIES

The Inden local authorities have financially long been dependent on tax revenues the mine operator RWE Power as the biggest industrial resident has paid into the community coffers. This dependency was further compounded when the German federal government enacted a new trade tax act [4].

Approximate calculations of the Inden local authorities resulted in a financially difficult scenario for the time after 2030, when there are only low or no tax revenues at all to expected from RWE Power. Since the municipality administration was not expecting that in the region marked by agricultural a similarly important company would invest intensive considerations were made whether or how it would be possible to compensate the threatening financial problems [4].

The future analyses of the Inden local authorities showed that after the termination of lignite excavation the value of the region regarding leisure utilisation and tourism would not be to high if the kind of recultivation planned so far were to be materialised as planned. In the process, it was ascertained that there is the possibility to considerably increase the value of the region for leisure utilisation by creating one of the largest lakes of Germany [4].

This idea could technically be realised relatively easily by filling the residual pit of the Inden surface mine with water straight away in contrast to the originally planned backfilling with material from the Hambach surface mine (see fig. 5).

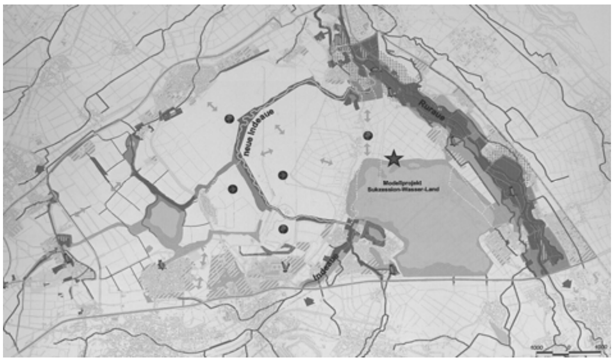

Figure 5: Alternative planning of the recultivated Inden surface mine with large residual lake– „Indeland“ [1

The newly proposed residual lake the „Indesee” would be Germany’s fourth lake with a planned surface of 1,100 ha and would add a new component to the profile of the region – the so-called „Indeland“- as an innovative economic area with new leisure qualities.

Figure 6: The „Indesee“ [1]

Topics such as employment, value creation, waste avoidance and preservation of resources, environmental sustainability have been expanded on in the planning by the Inden local authorities. In addition, this new form of the design of a lignite follow-up landscape in the Rhineland should be a model for the longer operated adjacent lignite mines of Hambach and Garzweiler.

5 REGIONAL INTEGRATION OF INDELAND

The Inden local authorities were campaigning with other municipalities and political and social bodies their idea of the Indeland and it was well received, e.g. with the concerned neighbouring communities/municipalities of Eschweiler, Aldenhoven, Jülich and Langerwehe. Even the chamber of agriculture was not principally against the idea. However, one of the few neighbouring municipalities – the town of Düren- spoke out against this planning.

By border-crossing contacts into the adjacent three-country-triangle Germany/ Belgium/ Netherlands (Euregio Maas-Rhein) it became possible to integrate the "Indeland" project into the trinational touristic overall concept of "Grünmetropole" (green metropolis), which is funded by the regional EU programme Interreg IIIa. As a consequence, Indeland became an integral part of the structural political development programme 'Euregionale 2008' of the federal state of Nordrhein-Westfalen (see fig. 7).

Figure 7: Green Metropolis of the Euregio Maas-Rhine [5]

For the mining operator of the Inden surface mine - RWE Power –the idea of 'Indeland' would mean that it does not have to set up a conveyor belt to transport the waste from Hambach to Inden and that the waste could be dumped immediately into the internal spoil tip of Hambach. After verifying the circumstances RWE Power agreed to support the idea of 'Indeland'.

Presumably by 2008, the last legal framework for the realisation of ‚Indeland' will be laid down with the current lignite plan being modified by the responsible lignite commission.

6 PLANNING AND REALISING THE FIRST PARTIAL PROJECTS BY 2008

Numerous elements were proposed in the run-up of the realisation of the concept and are now in their realisation phase. Some of them are named in the following:

On the Goltsteinkuppe, an old lignite overburden tip, which was piled in the centre point of the slewing area of the Inden surface mine, the so-called 'Indeman', a beacon-like steel construction in the shape of a human of approx. 45m of height and which can be seen from the far will be erected (see fig. 8). From the viewing platform of the tall 'Indeman', which will be illuminated at night, an impressive view on the ever-changing region should be possible. In the belly area of the tall 'Indeman' visitors should be able to inform themselves about the development of the surface mine and Indeland with the help of an adventure exhibition.

Figure 8: The „Indeman“ on the Goltsteinkuppe [1]

The tall one should also correspond with two small ‘Indemen’ (each of a approx. 12m of height) that are to be built at the foot of the Goltsteinkuppe at the belt junction of the surface mine and at the opposite edge of the mine in the North.

To the Indeland also belongs the 1.6 km long green corridor in the recultivated excavation area, which is to evoke two trains standing next to each other (see fig. 9). Here, on a width of 35m cultivations of renewable primary products are to be laid, e.g. switch grass, sun flowers, rape and fruit trees.

Figure 9: Renewable primary products in the green corridor [1]

Near the adjacent Blausteinsee, which is also part of the Indeland, a few attractions for the population and tourists alike are already projected for the very near future. At the moment, a modern Lake Centre is being set up, which is supposed to comply with the numerous demands of the population and tourists regarding diving, water sport, swimming, hiking, cycling, catering, hotels, service etc. (see fig. 10).

Very soon, a so-called lake window is to be installed on the water surface to allow among other things simulcasts of football tournaments. From visitor stands built on shore viewers should be able follow these video events if necessary.

In the near future floating islands are to be launched, which can be illuminated in different colours and where performances will be offered.

Figure10: Lake Centre at the Blausteinsee [1]

Next to this short selection numerous possible projects should further increase the attractiveness of the Indeland that is forming. The current planners are absolutely aware that they themselves will only live to see the final realisation of Indeland- if at all- only in its beginnings due to the long period to time. Still, they consider it – and surely rightfully so- a goal that is worth the effort to strive for the sustainable overall concept of a regionally important Indeland.

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LIGNITE MINING IN GREECE: ENVIRONMENTAL MANAGEMENT AND SOCIAL ISSUES

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Abstract: *The present contribution outlines the various environmental impacts of lignite surface mining in West Macedonia, Northern Greece, as well as the preventive and mitigation measures that are applied, according to the environmental permits, the National and European legislation and the best available technologies proposed by mining and environmental protection experts. It also presents the main land reclamation and environmental protection components of the applied Environmental Management System, which has been certified according to ISO 14001 standards. This system determines all decision-making, implementation, monitoring - auditing and reviewing procedures applicable to every environmental project. Furthermore, some critical issues that affect the relations between the mining operator, supervising authorities and local communities are discussed, such as emissions of pollutants, land expropriations, resettlements of villages, compensational works, etc. Taking into consideration these facts, it is concluded that the experience gained from the systematic implementation of environmental management activities during the last 25 years provides a concrete base for developing in common with local authorities an innovative, sustainable strategy that guarantees public welfare.*

1 INTRODUCTION

West Macedonia Lignite Centre (WMLC) is a complex of four open-pit mines operated by Public Power Corporation SA. It is equipped with continuous excavation - transportation and stacking systems and approximately 1,000 pieces of conventional earth moving equipment. On an annual basis, it excavates more than 320 Mm^3 of earth materials and produces ca. 50Mt of poor quality lignite, which is supplied to thermal power plants located in the vicinity of the mines. These plants correspond to 36.5% of the total installed power-producing capacity of Greece and meet ca. 50% of the country's electricity demand.

This mining operation has strategic importance for the portfolio of energy businesses in Greece. It has significant advantages, such as low cost of extraction and relatively stable and

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easily predictable fuel prices. It offers also a secure fuel supply option while, at the same time, the lignite extraction and utilisation industry provides thousands of jobs in rural areas, where high rates of unemployment prevail. Therefore, lignite has contributed significantly to the growth of the Greek National Product.

The operation of the first lignite mine in West Macedonia started in 50s. The remaining exploitable lignite reserves exceed 1.8 billion tones and are enough for supplying thermal power plants of equal capacity as today for more than 35 years.

Up to now an area of 16,000 ha has been affected by mining operations. As it is expected, a mining operation of such a large-scale is a potential threat for the environment. Although 3,700 ha have already been rehabilitated, there is no doubt that new ideas and considerable effort is needed in order to combine ecological integrity and economical development. These two dimensions are critical for ensuring equity between generations and equity between those who develop the deposit, those who work at the mine and those whose landscape and community are affected (Kavouridis et al., 2002).

Today, the environmental management strategy of WMLC is based on a series of Environmental Permits, which cover all mining activities and auxiliary operations located in the mine land. The Environmental Permits consist of several terms and conditions that reflect the National and European policy for the environmental protection in minerals extraction industry, as this is described in laws, presidential decrees, ministerial decisions and European directives, mainly the Regulation of Mining and Quarrying Works of Greece. Moreover, guidelines of international organizations that promote the implementation of best available techniques in environmental protection have been also taken into consideration, together with know-how and experience coming from abroad and from the implementation of similar environmental protection strategies in other sectors of Greek economy.

2 ENVIRONMENTAL IMPACTS

In general, open-pit lignite mines are complex operations that can affect numerous environmental constituents in various ways. Often, mining activities led to severe environmental degradation just because of their enormous size. Nevertheless, these activities last for many decades. Otherwise there is not enough pay-back time for the capital invested for mine development. As a consequence, mining activities affect irreversibly the local communities, which adjust their life-style and depend their economic prosperity on the development plans of the mining company.

Following the methodology of Life Cycle Analysis, the environmental impact mechanisms can be easily assessed after having defined the boundaries of a production system. In the examined case, these boundaries include both lignite mining and power production operations, since they belong to the same corporation, are located in the same area and, in many cases, impact cumulatively the environment (e.g. release of particulates to the atmosphere).

Figure 1 presents schematically the main operations and the boundaries of the lignite extraction – combustion system. The most important "outputs" of this system to the environment of the greater mining area are (Pavloudakis & Roumpos, 2004):

(i) the reclaimed mine land: The surfaces of pits, backfills and waste heaps are characterized as final when they cannot be used by any mean for mining works (excavations, waste dumping). According to a Law voted recently, after the entrance of Public Power Corporation in the stock-exchange market, the reclaimed mine land remains property of the mine operator, which has the right to rent it to farmers, to donate it to local authorities or to develop its own development plans (e.g. installation of photovoltaics, energy crops, recreational parks).

(ii) water and wastewater discharges: for every ton of lignite produced about 1 m3 of water is pumped to aquatic receivers either from the sedimentation ponds that operate in-pit or from wells, which have been bored around the pit in order to depress the water table and to reduce groundwater intrusion in the open pits.

(iii) particulates, mainly fly ash released from the stacks of thermal power plants and fine particles of lignite, ash and overburden material coming into suspension during their transportation on belt conveyors or conventional diesel-engine earth moving equipment.

(iv) waste collected and disposed of by authorized contractors: this particular "waste stream" includes recyclable material produced by repair & maintenance workshops, which operate within the system boundaries, as well as by worksites that dismantle equipment of closing mines, which cannot be utilized by the remaining mining activities.

Furthermore, Table 1 correlates the main environmental impacts caused by the large scale surface mining operations with potential social and economic problems that can deteriorate the life quality standards of the local communities. It is worth noticing that these impacts do not threaten the environment to an equal magnitude. Some of them can be eliminated by applying certain preventive or mitigation measures. The adverse impacts, in such cases, are related to abnormal situations (e.g. a wastewater treatment unit is out of operation) and can be analysed based on risk analysis methods.

It is obvious that Table 1 excludes all positive impacts of mining to the development of the local, regional and National economy since they are not related to some environmental impacts.

3 ENVIRONMENTAL MANAGEMENT

Environmental management in WMLC is responsibility of the Sector of Environmental Protection and Land Reclamation (SEP&LR). The Section is organised in three sub-sectors: (i)

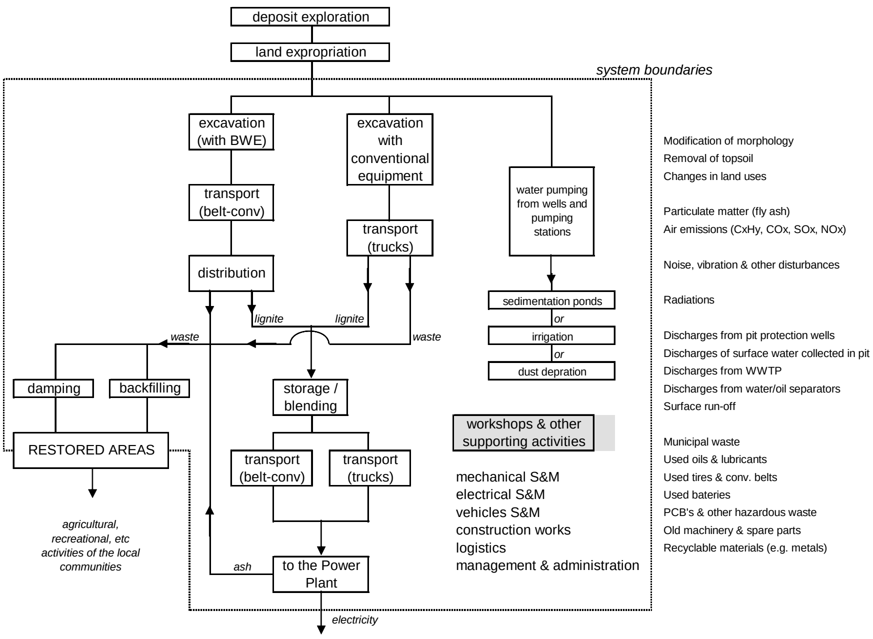

Figure 1. Main operations of an open-pit lignite mine and their outflows to the environment (Pavloudakis & Roumpos, 2004).

Land Reclamation, (ii) Environmental Protection, and (iii) Water Quality Monitoring. It has now 45 employees of various scientific backgrounds and technical skills and an annual budget of 2.5-3.0 million Euro.

From June 2007 the Environmental Management System (EMS) applied in WMLC has been certified according to ISO 14001 standards. EMS is based on a series of terms referred to the Environmental Permits of all mining activities and to the legislation that are currently in force.

Table 1. *Correlation between environmental impacts and social and economic problems caused by surface mining activities in WMLC*

Environmental impacts	**Social and economic problems**
• Occupation of large areas for long time periods for developing the mine pits and waste heaps	• Resettlement of communities, damage of roads and other public utilities' infrastructures
• Alteration of morphology → Degradation of landscape value	• Reduction of property value
• Alteration of morphology → Changes in hydrological pattern	• Alteration of water balance & floods
• Topsoil removal → increased surface run-off & water pollution	• Loss of agricultural land & → Loss of incomes
• Particulate matter → air pollution	• Degradation of potential water uses
• Air emissions (C_xH_y, CO_x, SO_x, NO_x) → air pollution	• Life quality degradation & health problems • Life quality degradation & contribution to greenhouse effect
• Discharges from pit protection wells → drop of water table & increased flow in streams	• Insufficient water quantities for irrigation
• Surface run-off and discharges from surface water collected in pit → suspended solids in streams → aquatic life disturbance	• Insufficient water quantities for irrigation & floods & degradation of potential water uses
• Discharges from waste water treatment plants and water/oil separators → pollution of aquatic receivers → aquatic life disturbance	• Degradation of potential water uses
• Improper management of solid and hazardous waste → increased concentration of toxic substances in soils/waters & aesthetic problems	• Degradation of the potential land and water uses → Life quality degradation & danger for human health
• Loss of wild animal habitat	• Life quality degradation
• Noise, vibrations, heavy traffic, etc.	• Disturbance of human activities

The key element in the implementation of this EMS is the formal and documented procedure that is followed for keeping the different levels of the company's hierarchy informed about the overall effectiveness and the potential 'weaknesses' of the applied environmental protection and land reclamation practices. Taking advantage of this process, the hierarchy is capable of participating in the decision-making and of setting the priorities of the environmental protection policy. In this way, the selected mitigation measures are part of a uniform and long-term strategy with unambiguous objectives, which does not favour spending money on remedial actions not included in the specified conditions. Figure 2 describes schematically the continuous process that supports the aforementioned decision-making approach.

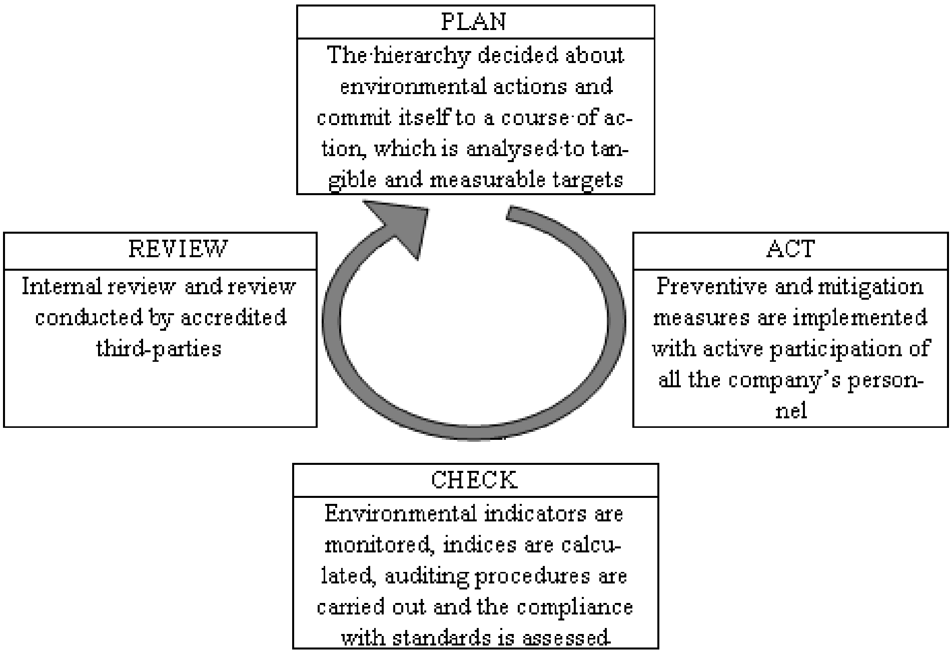

Figure 2. Schematic description of the continuous process that supports environmental decision-making

Another critical advantage of the certified EMS derives from the requirement of active participation of all the company's personnel to environmental protection actions (e.g. waste recycling). This participation is achieved through the entrusting of specific responsibilities and the realization of seminars and other training programs (Pavloudakis & Agioutantis, 2008).

4 LAND RECLAMATION

4.1 Selection of land uses after land reclamation

Development of farming land in horizontal areas on the top of the waste damps and reforestation of the slopes of the heaps' margins are the main choices when land uses of reclaimed mine land is of concern (Figure 3).

For the selection of the land use that fits better to every piece of mine land the following criteria are examined:

- The morphology of the final surface
- The position of the area in relation to already reclaimed areas and neighbouring communities

– The type of material stacked on the surface
– The altitude of the area
– The exposure to sunlight and winds

Most of the cultivated land is seeded with wheat. The productivities achieved are varying considerably from site to site as a result of the large fluctuations in the chemical and mechanical composition of the earth material that covers the top surfaces. In general, the productivity of the reclaimed lands varies from 1,000 kg/ha to 4,000 kg/ha (average value is 2,200kg/ha) and is comparable to this reported for cultivations in the vicinity of the mines.

According to sampling and lab analyses conducted systematically the pH values are relatively constant (7.45-7.85) but CaCO3 content varies from 12.1% in silty sands to 42.2% in clay-silts and the K2O content varies from 12.7 mg/100g in sandy clays to 54.6mg/100g in clays. In addition, the loose structure of the new soils developed on the dumping sites allows rapid change in nutrients concentration due to the washing action of the water.

Nevertheless, the spreading of topsoil, the weather conditions, the use of fertilisers and the age of the reclaimed lands are also critical parameters for the achievement of high productivity.

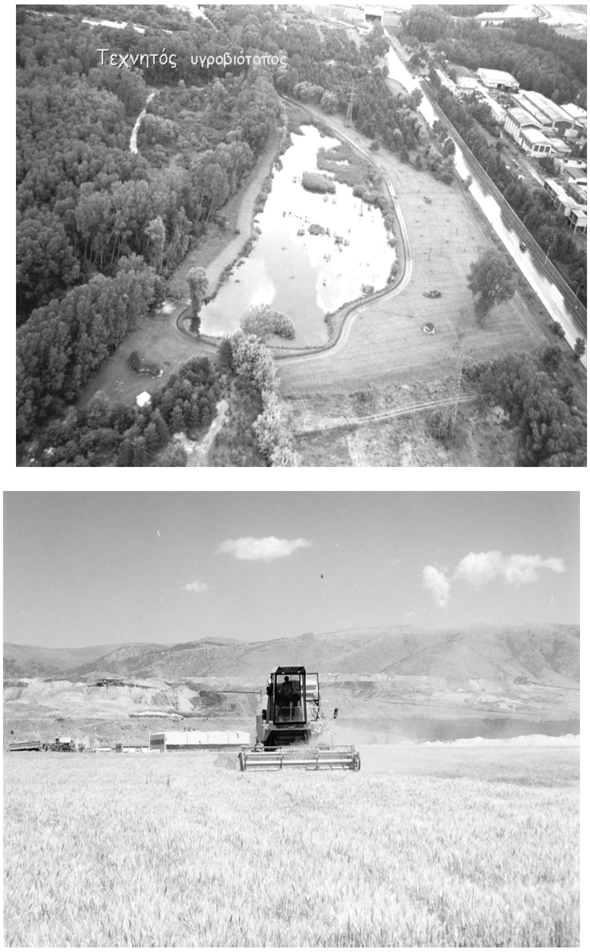

Figure 3. Recreational and agricultural activities in reclaimed mine areas of WMLC

Regarding forestation of sloped surfaces, more than 7 million trees have been planted so far. During the last 10 years the annual rate of planting has considerably increased exceeding 400,000 trees. The tree species that are usually planted are acacias, rushes, pine trees, poplars, cypresses and ornamental trees.

According to the records of local hunting clubs and silviculture institutes, the population density of some endangered mammals and birds in the forested mine land is higher than this counted in the surrounding land, which has not been affected by the mining operations. This is probably explained by the controlled admission in the mine area for all people that are not working in it (Vlachantonis et al., 2003).

4.2 Phases of land reclamation works

The reclamation works that are usually taken place in mine areas are organised in 4 separated groups of activities: (a) grading of final surfaces, (b) spreading of topsoil, (c) planting, and (d) development of infrastructures, such as roads and irrigation networks (Kavouridis et al., 2002).

Figure 4. Planting of trees with the conventional method and with the method of ripper

Grading of final surfaces developed on the top of the dumping sites is necessary because continuous mining systems stack the wasted material in series of piles that form a completely irregular surface, whereupon the driving of any vehicles apart from bulldozers is impossible. Grading is also necessary in foothills of dumping sites, where the steep slopes must be smoothened in order to avoid landslides.

Topsoil spreading is required only in cases that cultivated land of high productivity rate must be developed. This is necessary because dumping of waste rocks is carried out in such a way that there is no control on the origin and type of the material that is stacked on the top of the dumps. This fact makes impossible the prediction of the physical, chemical and mechanical characteristics, and consequently the productivity, of each piece of reclaimed land.

Planting and growth of forest trees is mostly depended on irrigation rather than topsoil characteristics. For the planting of forest trees, apart from the conventional labour-intensive method (Figure 4), the following methods patented by SEP&LR personnel are used:

– Method of ripper: The use of a modified ripper, which is mounted on a dozer or tractor, allows planting of more than 1,000 trees per hour

– Method of roots transfer: According to this method the entire roots ecosystem that has been developed in front of the mine face is transferred together with the topsoil to the final surfaces of the reclaimed areas. By this method, from the root of a single tree tens of new trees can be grown.

The development of infrastructures is necessary for securing the normal operation of all agricultural and recreational activities taking place in the reclaimed areas. Roads, water pumping stations and reservoirs, irrigation networks, electricity supply networks, firebreak zones, and fencing are some of the infrastructures usually developed in reclaimed areas.

5 ENVIRONMENTAL PROTECTION

5.1 Protection of water resources

The collection of surface water in sedimentation ponds, the diversion of streams and rivers, the treatment of wastewater in activated sludge units and oil separators are some of the measures that are implemented in order to protect the quality of the aquatic bodies that receive the mine discharges. Until now only sporadic deviations from the standards set by laws and regulations of the Prefectures of Kozani and Florina have been detected in certain sampling points. These deviations regard in most of the cases suspended solids and total dissolved solids concentrations.

The groundwater intrusion within the open pits and the groundwater pumping from protection wells bored in front of the exploitation face has as unavoidable impact the drop of groundwater table in the vicinity of the mines. Research conducted in collaboration with university institutes proved that the local hydrogeological conditions do not allow the expansion of the depression cone far from the mine-pits margins.

In addition, the entire energy production activities of PPC in Ptolemais area contribute positively to the water balance of the watershed. This is because large water quantities consumed by the thermal power plants are coming from another watershed, this of Aliakmon River, and are partly released as steam to the atmosphere and partly discharged into nearby streams.

However, the cumulative actions of (a) WMLC’s mine protection works, (b) the continuous pumping from hundreds of water supply and irrigation wells bored in the greater area and (c) the relatively dry winter seasons of the last years resulted to considerable drop of the groundwater table.

5.2 Control of atmospheric emissions

Dust is the principal air pollutant related to surface mining operations. According to the data collected from the existing monitoring network of ambient air quality in the greater area of WMLC mines, the particulate matter concentration is a major environmental threat, which is related to the operation of both mines and thermal power plants.

The following sources of particulates are considered to contribute principally to the dust emissions problem:

- traffic of auxiliary vehicles on unpaved roads within the mines,
- traffic of overloaded trucks (the material that drops on the road surface is coming into suspension when the next truck passes) or trucks that do not respect the speed limit,
- traffic of lignite of fly ash transporting trucks that do not use their covers,
- belt conveyors that transport fly ash,
- belt conveyors that transport lignite and/or waste material, which is coming into suspension or falls from the belt due to abnormal operating conditions,
- belt conveyors connection points
- wind blow across the large surfaces of mine benches and waste dumps that have not been reclaimed yet,
- lignite stockyards,
- overburden stackers dropping material from height.

In order to reduce dust emissions the main transportation routes of the mine complex have been paved. Moreover, dust depression is achieved by constructing permanent water sprinkling networks in roads with heavy traffic and by using more than 30 tanker trucks in secondary unpaved roads.

Additional measures for reducing dust emissions are the construction of covers over belt conveyors that transport fly ash and the installation of water sprinkling systems at the belt conveyors connection points that are close to wells or irrigation pipelines.

5.3 Waste management

A mining activity of the type and scale of WMLC produces large quantities of various wastes that require special management and disposal. Metal scraps are collected and sold by auction. Used lubricants and batteries are also collected in special containers and they are delivered to recycling systems certified by Presidential Decree. Used tires of diameter less than 140 cm and containers of lubricants are also included in the implementation field of the solid waste recycling legislation and are delivered to certified recycling companies. Redundant equipment and used tires of diameter larger than 140 cm are stored in temporary storage yards and they are delivered to authorized recyclers selected after auction. Paper and small batteries are recycled in cooperation with the regional enterprise for solid waste management and disposal.

WMLC tries to meet all the environmental regulations set by national and EU legislation regarding the management and/or disposal of building materials containing asbestos.

Municipal solid waste collected from the mine facilities (ca. 150tn/yr) and from all the towns, villages and small communities of West Macedonia Region, which have a population of about 200,000 people, is disposed of in a sanitary landfill located on an old backfilling area of Kardia mine. The landfill incorporates the state-of-the-art of leachate and biogas containment systems and has a capacity adequate to receive the solid waste of the served area for the next 15 years. Moreover, a separate sanitary landfill has been constructed in the same area for the disposal of building materials containing asbestos, which are dismantled from the cooling towers of the thermal power plants. This activity is carried out under the supervision of Energy Production Division of Public Power Corporation aiming at the enhancement of the efficiency of the power generation process and at the substitution of asbestos from a new plastic, more environmental friendly material.

6 SOCIOECONOMIC ISSUES

For a mine operator, the development of public relations on the basis of mutual respect and understanding with local authorities is a critical parameter for the acceptance of a surface mining activity. In this context, WMLC takes various compensation measures in collaboration with local authorities aiming at establishing good community relations and avoid delays in mine development. These measures include roads construction, earth moving works, snow removal during the winter, demolition of old buildings, forestation, donation of various activities, etc.

Nevertheless, the employment of people who live in the greater mining area is the kind of compensation that always is requested by local politicians and community representatives. Although new job opportunities in PPC SA are limited, due to the programme for reduction of the company’s personnel, the situation has been improved in the last years after the vote of a Law that gives extra bonus to people coming from the villages that neighbour to the lignite mining activities (Pavloudakis et al., 2006).

6.1 Resettlement of communities

Mine development necessitates expropriation of private properties and resettlement of villages that are located over the lignite deposit. In the case of WMLC the resettlement of communities is considered as an alternative for surface mines development only if lignite deposits are located in inhabited areas. In these cases the mining company is obliged to pay compensations for every private property and for public buildings and infrastructures (churches, streets, utility networks, etc). Moreover, the company must cooperate with local authorities for the selection of the site, where the community will be resettled, and for the development of new infrastructures that will allow early move of people to the new village.

Up to now, three small communities with more than 2,700 inhabitants and 750 dwellings have been settled in new sites outside the mining area. In the next years, the resettlement procedures of two more villages will have been completed (buildings construction has already been started at “New Kleist’s” site), while, until 2020, another two villages will be moved for developing the new mines at West and South-west fields.

Furthermore, there is a continuously growing pressure from communities that claim for resettlement due to the violation of their life standards caused by the various impacts of surface mining. Even if the mining operator meets the regulatory standards as far as the minimum distance from inhabited area is concerned, local authorities request their resettlement having as basic argument the large size of coal surface mining operators, which makes unavoidable the spreading of several pollutants to a wide area and reduces the availability of land for activities other than mining.

6.2 Land expropriation

The expropriation of the land required for the development of a new surface mine is a time-consuming procedure that is normally come to an end in the Court, which determines the unit prices for the different components of private properties (land, trees, fences, irrigation wells, etc) that will be destructed due to excavations or waste dumping. After a long lasting and systematic presentation of arguments, PPC succeed to have Court judgments that calculate the compensations of the land owners taking into consideration not only the property items that are found in each piece of land, but also the compliance with agronomical practices (i.e. a farmer is not allowed to ask full compensation for trees planted just a few months before expropriation using a very dense tree-spacing net).

6.3 Levies

For compensating the adverse impacts of lignite surface mining on the environment and on other economic activities, Public Power Corporation SA, which produces more than 95% of the lignite used for electricity generation in Greece, pays a levy, which corresponds to 0.4% of the company’s turnover. This levy is distributed to the three Prefectures, where lignite mining activities are carried out, accordingly to the lignite quantities produced. The local authorities have taken full responsibility for using this money for financing various development projects.

7 PERSPECTIVES – DEVELOPING A NEW ENVIRONMENTAL ACTION PLAN

The more and more stringent environmental standards regarding the quality of life and the rational management of the different waste streams that are met in the lignite mining operation necessitate further improve of the environmental protection and land reclamation works carried out from WMLC.

Taking into account:

– the competitive conditions of the global energy market, and

– the further activation of the pollutant-pay-principle, (e.g. implementation of CO_2 emissions trading system and realisation of more frequent and stringent inspections at local and national level),

– the compliance with all environmental laws and regulations will be directly connected with the viability of many lignite mining companies and lignite-fired power plants.

WMLC is ready to face these challenges being in a strong position that provides many opportunities, providing that a few weaknesses will be not underestimated.

Strengths:

– The experience gained from the implementation of surface mining activities for more than five decades and from the realization of a systematic land reclamation programme in the last 25 years.

– The development of a reliable regulatory framework, which is based on the environmental permits and refers to all mining and auxiliary activities carried out within WMLC. The series of environmental terms listed in these permits simplify the planning of all land reclamation and waste treatment – disposal activities setting site-specific goals that overcome ambiguities relevant to legal and regulatory aspects.

– The implementation of a modern Environmental Management System, which has been certified according to ISO14001 standards, and provides a reliable basis for planning, applying, monitoring and reviewing environmental management activities contributing, consequently, to continuously improving environmental results.

Weaknesses:

– The lack of an updated master-plan that will set environmental targets, priorities and working standards taking into consideration the new mine development plans, which have been modified according to the new technical and economic conditions, mainly the considerable increase of the prices of all fossil fuels.

– The lack of a reliable, computer-based system (e.g. based on a GIS platform or on a mine planning software), which will establish a common “language” for communicating and plotting ideas and proposals relevant to land reclamation works and land uses selection - planning. This system will be possible to incorporate programmed decision support tools.

– The limited experience in making decisions regarding the priorities of the mines’ environmental management strategy in collaboration with local authorities and the citizens. In spite of the new legal framework, which determines that PPC will keep the ownership of the reclaimed mine land and the rights to exploit it in ways irrelevant to electricity generation, local communities must participate or, at least, must be informed about every decision concerning the socioeconomic development of the area.

Opportunities:

– The possibility of substituting old, inefficient thermal power plants with new ones, which incorporate innovative technologies (e.g. fluidised bed combustion) capable of reducing both fuel consumption and atmospheric emissions, contributing in this way to a significant reduction of the emitted quantities of green house gases.

– The possibility of using innovative technologies for waste treatment and disposal, minimising in this way the risk for the future generations as far as the need for applying soil or water decontamination measures is concerned.

– The full support of the vast majority of PPC’s personnel syndicates and local authorities to the lignite, recognising its vital role for the development of the National and regional economy, the reduction of the dependency from imported energy resources and the creation of new job opportunities, most of them in rural areas that suffer from high unemployment percentages.

Threats:

– The new EU Directives related to soil protection, mine waste management, etc, which will probably result to a considerable increase of the overall cost of lignite mining, for instance if ash and overburden material will be characterised as waste that must be disposed of with an appropriate manner.

– The probable closure or rapid declining of lignite exploitation activities, as a result of the new, stringent emissions trading policy of the European Union.

The above-mentioned threats may lead to sudden interruption of the land reclamation works and to the abundance of all the preparatory activities that support the smooth transition to the post-mining era. However, all stakeholders consider that this will be not happened due to the following reasons:

– According to the Environmental Permits, the mining operator has already deposited to a special account the amount of money required for funding environmental protection and land reclamation works.

– The increased cost of energy produced from lignite-fired thermal power plants, due to the implementation of the emissions trading scheme proposed by the European Union, cannot equalise lignite advantages connected with the fact that it is the only abundant domestic energy resource, which reduces the strong dependency of Greece from imported fuels.

8 CONCLUSIONS

Lignite mining and power production activities in Western Macedonia meet a remarkable percentage of Greece's electricity demand. Due to the type and size of these operations, severe environmental damages will be caused unless site-specific preventive and mitigation measures will be implemented aiming at the reclamation of the mine land and the protection of sensitive environmental components.

The Environmental Permits that are in force for all mining activities, workshops and waste treatment facilities located within WMLC set the performance standards for every environmental project and the certified Environmental Management System provides the "tool" for achieving continuous improvement of all environmental indicators.

Facing the dramatic changes in the global markets of electric energy and fossil fuels, the Greek Public Power Corporation has to deal with many critical issues regarding lignite competitiveness in relation to hard coal, natural gas and renewable energy sources. The implementation of 'clean coal technologies' for constructing new thermal power plants, modernising existing power producing capacity, and reducing emissions of greenhouse gases is the critical parameter that will determine lignite share in the energy mix for the next decades. Moreover, lignite mines must eliminate environmental damage by applying effective measures for preparing high-yield and ecologically valuable land at post-mining areas and for reducing the escape of pollutants to soils, aquatic bodies and the atmosphere.

In this context, the experience gained from the systematic implementation of environmental management activities during the last 25 years provides a concrete base for developing in common with local authorities an innovative, sustainable strategy that guarantees the welfare of the public.

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WORK FROM WASTE

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Abstract: *Employment is, in the words of the World Employment Conference of 2000 „one of the most effective means of ensuring a just and equitable distribution of income and of raising the standard of living of the majority of the population." The only objection to its widespread use as a solution to economic and development problems is that there is rarely enough of it, especially in those countries that need it most. Unemployment and underemployment (employment for too little reward or in a 'marginal' activity such as crime or scavenging) are features of underdevelopment as common as poverty, illiteracy, bad housing or disease. Worse, it is in the countries that lack the resources for social security that unemployment means not just boredom and spiritual demoralization but, in addition, abject poverty, destitution and even starvation.*

This paper is about one field of employment opportunity: the exploitation of waste. Waste is one of the world's largest industries, although you could not discover this from any book of statistics, because its activities cut across the normal divisions into which industries are placed. If you buy a bottle of medicine it may have a metal top and be protected by plastic loam padding, in a cardboard box. To recycle these parts after the medicine has been taken you will need to sell the bottle to a glassmaker, the top to a foundry, the plastic to moulder and the cardboard to a paperboard mill.

Recycled materials are only a small part of the materials used by most of those factories; yet add up all the bottles, tops, packaging and cardboard cartons and you have an enormous quantity of material.

Then add wastes from agriculture, animal and meat industries, mining and quarrying, industries that make iron and steel and other metals, textiles, rubber, chemicals and oils and it is even bigger.

Why, when employment is so difficult to create, should it be easier to do so in the waste business, an activity in which hundreds of thousands of people are already scratching a living. There are several reasons and they will be discussed in some detail. By persuading the reader that waste offers unique opportunities for creating jobs, it is hoped to encourage him to read further and then to consider whether he can apply some of the ideas to his own business or to the group of poor people with whom he is working or just to pass on the ideas or the book to others.

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This paper is of no value until someone, one day, uses it and finds work and earns money when previously they were idle.

The reasons are these:

Waste is plentiful. In most towns and cities of the world it is not only heaped in huge quantities on refuse dumps but also lies in piles around the streets and in small illegal dumps on any piece of waste ground. Most Third World cities are worse than those in industrialized countries which have the money, the technical abilities and the public attitudes to control their waste to some degree. They are usually growing more rapidly with an increasing population of middle and upper social classes. These are the people who can afford packaged goods, processed goods, new furniture or a car or clothing, who take a daily newspaper and cultivate a garden.

Waste is free or if not tree then very cheap. It is thrown away because it is either impossible or not worth the time or trouble to sell. Any process that uses materials to make a product has to pay for those materials and, worse, has to pay for them before being paid for the product itself. This need to finance raw a material stock with working capital is, as much as any other, a requirement that hinders the setting up of small industries. If the materials are free or very cheap this hindrance is removed.

Waste is flexible. Even if the material is free, there are other expenses in making a product to sell for money; even if only that of feeding one's family while doing so. Waste is flexible in the amount of work it needs. A weaver cannot sell a halfsell it immediately to someone else for the price of his next meal.

Waste is labor-intensive. It needs people to collect and sort it. This is because it comes from so many different places; from thousands of different homes, each eating slightly different foods, wearing different clothes, living in different kinds of houses down different sorts of streets. Although it can be collected and sorted by machines, no one has yet produced a machine that can do these two things as cheaply and effectively as a human with eyes, hands and legs. In the industrialized countries, where wages are high and fewer and fewer people are willing to do dirty jobs, machines are being used where possible to collect and sort waste but often the result is a cost so high that the attempt to recycle or re-use the waste is abandoned altogether. This will not happen in the Third World while so many people earn a very low income. Waste lends itself to the kind of sub-contracting by big firms to small that helped the 'economic miracles' of Japan, Singapore and Hong Kong.

Waste needs little capital. It can be collected and sorted and sometimes processed with very little equipment, buildings or supplies. Of the many industries that need only simple tools and equipment, few lend themselves to such a range of different levels of capitalization, as does waste. To collect waste paper, if you have no lorry you can use a van; if you have no van you can use a donkey cart; if you have no donkey you use a hand-cart; if you have no hand-cart then use a sack and if you cannot even buy or find a sack you can tie it in a bundle. If you have nothing whatever you can carry it loose. However, the less the capital or equipment used the smaller the profits that can be made and the more the effort needed to earn them. You can collect a tonne of paper in an hour with a lorry. With a van it may take two hours, with a donkey cart half a day, a hand cart, a day, with a sack two days, four days with bundles, and loose in your hands it will take a week!

Waste sells for cash which, once the collector and his family have eaten, *may* be used to buy the equipment needed. The first bundle of paper sold can buy half a loaf and a length of string; the next, larger because it is tied, can buy half a loaf and a sack; a week's collections with the sack can yield the price of four second-hand wheels and some scrap wood with which to make a simple cart. If the collector is lucky or sensible or hardworking he can (in theory) continue until he has bought the lorry.

Waste is familiar. Even if ways of processing it are technical or complicated, the fact that the simplest person knows what paper or glass is, what it is used for, whether it burns, where it is bought and sold, helps him to develop the confidence to work with it. No one who is unfamiliar with farm crops will try and make his living as a farmer; no one who grew up away from sea or rivers will suddenly seek his employment as a fisherman; yet there is no mystery about waste; we are all brought up among it, wherever we live. The most important requirement to start a small business is the confidence that you will be able to do it and succeed, and familiarity with the raw material helps to achieve this.

Recycling of waste is approving. If you decide to obtain your living as a pickpocket there will be many people trying to prevent you succeeding, for reasons that have nothing to do with whether you are good at it or not! If you try to become a cleaner of shoes only your clients are at all interested in your work and if you are ill tomorrow they will go to someone else. If you collect waste, however, a whole range of people and organizations benefit and, if they are wise they will help you or at least not obstruct you. Your customers need the material for their business. The householder or factory that produces the waste needs to get rid of it. The local council will have to collect what you do not, they will have to find somewhere to put it. The government may have to find foreign money to import raw materials if local waste is not collected and used instead. More imported coal or oil or electricity will be needed to process those raw materials than are needed to recycle the waste you collect. Finally, the wastes you do not collect may drift around the streets as litter and spoil the town for tourists, attract flies or rats and block up the drains if there are any. So the community has an interest in your success; provided always that you do not create more litter than you save, or cover the neighbour's washing with thick black smoke or do other things that will quickly lose everyone's sympathy, but ready to be suspended in water for making paper, is known as 'pulp'.

Softwood or coniferous pulps are used for tough wrapping and packaging papers because of their long fibres; deciduous or hardwood pulps provide 'filler' for printing and writing papers. There are three principle types of 'primary wood-pulp' used in paper-making; mechanical, 'Kraft' sulphate, and sulphite pulp.

Mechanical pulp

This is made by pounding or grinding cellulosic material such as wood. It is used for printing newspapers. Newsprint is very weak and loses its strength altogether if wetted - this is characteristic of mechanical pulp. It is ideal for newspapers because it is highly absorbent and liquids, such as printing ink, are soaked up and dry very quickly. Mechanical pulp often contains tiny particles of wood which have not been reduced to fibre and are visible to the naked eye. Thus paper made from mechanical pulp is often described as 'woody'.

Chemical pulp

A strong paper product is most cheaply achieved by pulping cellulose fibres so that they are not weakened by mechanical damage. The wood or stalks are mechanically reduced to small chips and then cooked at high pressure with certain chemicals that attack the bonds between the fibres and release them to form pulp. The most common chemicals used are:

✓ Caustic soda and sodium sulphate which produce coarse, very strong fibres known as Kraft, suitable for sacks and boxes to hold heavy weights, and

✓ Various sulphites (such as ammonium and calcium) which produce fine fibres suitable for making high quality printing and writing papers (usually bleached white); these are fine and strong but expensive. There are many variations to pulping processes, but only one point needs to be made here: all types of pulp start out the colour of the cellulose (usually a wood-like colour) and can be bleached white and later tinted (coloured) to other colours.

Use of waste ***paper in paper-making***

The phrase primary wood-pulp was used above to distinguish it from 'secondary pulp' which is made by vigorously stirring waste paper in water (usually in a 'hydra-pulper', a cylinder containing rotating blades), to separate the fibres bonded during the original paper-making process. As these bonds are weaker than those of the original cellulose plant, hydra-pulping is a more gentle process than wood-pulping and consumes less energy. Even so, paper cannot be repulped an indefinite number of times without becoming much weaker. Secondary pulp can never, therefore, have as high a quality as the primary fibre from which it was made, although it can close *provided* pure waste paper of the same type is used. For example, pulp made by hydra-pulping clean Kraft sacks will make new sacks of only slightly lower quality, particularly if mixed with a proportion of primary Kraft pulp. If, however, the secondary pulp is made from material which contains newspapers or some other weak mechanical pulp product or dirt, dust or clay, it will not be strong enough to use instead of primary Kraft; its value to the paper-maker (and the price he is prepared to pay for it) will be much reduced. This leads to the first important rule of waste paper collecting: pure, clean material is of far higher value than dirty or impure waste paper.

Coated papers

In some cases the matted, absorbent, dull surface of the paper is coated with a material which makes it appear brighter, shinier and harder, and makes printing on it sharper and more contrasting. Coated papers are particularly used in magazines that are financed by the advertisements they print. In the process of hydra-pulping waste coated paper, the coating is washed out: thus the weight of fibre obtained from a tonne of coated paper is less (often by 20%) than that obtained from a tonne of uncoated. The value to the paper-mill and the selling price will also be lower. Moreover, if the coating is plastic, tar or other material which will not dissolve in water, then the waste paper is not suitable for paper-making and has no value at all. Indeed, it may even reduce the value of other, good paper with which it is mixed. The same is true of polythene film, 'cellophane', glued papers, string, and any material which will not break down in water.

Printed and coloured papers

As well as variations in how the paper is made, waste paper may be printed or tinted. Both reduce its value: print because it makes both the pulp and the paper made from it dull grey in colour unless bleached (which is expensive); tinting because the tints colour the pulp which must then either be used for a limited range of similarly coloured products (or cheap, grey products) or must be expensively bleached. Therefore it is important to remember that white waste paper is more valuable than similar material which is coloured, and unprinted waste paper is of greater value than the same material printed.

The paper-making industry

The manufacture and use of paper is one of the world's biggest industries; there is little hand-making or small-scale manufacture. The following types of factories are involved:

- ✓*Pulp mills* that process timber or other materials to make pulp.
- ✓*Paper mills (and board mills)* which use pulp or waste paper to produce finished paper and board.
- ✓*Paper converters* that use paper or board and produce items from them such as boxes, tubes, rolls of toilet tissue, boxes of blank office paper, stacks of printing paper cut to standard sizes, etc.
- ✓*Printers* who usually buy from converters, although larger firms such as newspaper presses may buy direct from the paper mills.
- ✓*Integrated mills* are those that make pulp and then use it to make paper themselves.

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A NEW VISION OF THE URBAN ECOLOGY. EXAMPLE: DEVELOPMENT OF PETROŞANI CITY

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***Abstract**: A correct understanding of the concept of* **"sustainable development"** *involves an analysis of the development of the human socio-economic system in its historical perspective. This process is analogue to ecological succession, but, differently, is dictated by economic and social laws. The engine is the exponential dynamic of human population, and, correlated with this, the exponential dynamic of the needs of humans. In a first step, human populations were integrated in natural communities, and the man felt himself as a part of the environment.*

When industrialization started, humanity passed to a new step- the "divorce" between man and nature, ending in the 70's with the energy crisis. Human society realized at this point that uncontrolled development is joined by phenomena as:

- Overexploitation of the resources of the natural capital
- Pollution
- Loss of biodiversity
- Fragmentation of habitats
- Introduction of new species
- Genetic manipulation

Execution of large constructions on fresh waters (such as dams etc.) united under the common name of **"environmental deterioration".**

At this point, an unrealistic solution was proposed: **preservation of the natural capital,** opposed to the development of the human society. The solution to this apparent conflict was **sustainable development,** marking a new step, the era of the reconciliation between man and nature.

Theoretically, sustainable development is the development that satisfies the needs of the actual generation without compromising the chances of the future generations to satisfy their needs in the same manner, and in practice it involves the integration of environmental protection regulations in the policies and strategies for development.

The implementation of the sustainable development solution assumes the elaboration of models for sustainable development. This involves a holistic approach to the problems, is based on public participation and includes ecological reconstruction.

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The problem of sustainable development represents a real problem, and, at the same time, a priority, proved by the large number of papers on this topic, and the amplitude of workshops, conferences, symposia and other similar meetings on this matter.

The elaboration of models for the sustainable development of cities involves approaching the cities as ecosystems. This approach led to the apparition of a new branch of systemic ecology: the **urban ecology.**

The essential difference between urban and natural ecosystems is represented by the presence of the human species as a dominant species in the first case; the other differences represent, in fact, consequences of it.

Urban biotope includes stationary and man-generated elements (last being generated throughout human activities). The first category includes geographical, hydrological, geological, and soil-related elements. However, each of them differs through a series of parameters by its homologous in a natural ecosystem, as a consequence of human actions. Man-generated elements are produced as well by the urban structures and by human activities. Urban structures reflect the satisfaction of some human needs, of a biological nature, related to socio-economic organization and purely economic. Human activities generate different aspects of environmental deterioration, the most well known one being pollution, manifested at the level of atmosphere, water, and soil (see figure 1).

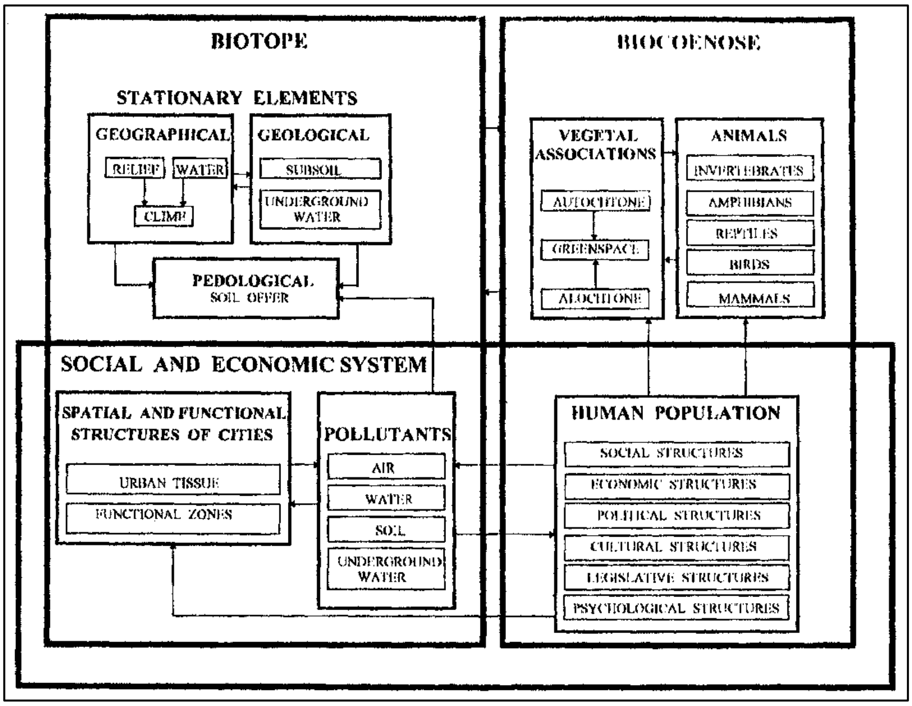

Figure 1 Elements of urban ecology

The organization of urban communities is based on ecological niches, spatial or food-related. Vegetal communities are subject to the modeling action of the man, being found especially as green spaces or urban forests. Animal communities are poorer, being represented especially by eusynantropic species.

From the common features viewpoint, urban ecosystems have a lesser diversity compared with natural ecosystems; functionally, it is concretized in a lesser self-regulatory capacity, a reduced stability and dependence on human actions. Moreover, biological laws act differently in these ecosystems, and socio-economic laws are predominant.

With respect to the functions of ecosystems, essential differences appear. Urban ecosystems are energetic parasites on natural ecosystems, bio-geo-chemical circuits tend to become linear, and stability decreases. Furthermore, urban ecosystems have new functions, related to their socio-economic specific.

From a historical perspective, the impact on the environment increased gradually since the very beginning of humanity until nowadays. At the same time, people started thinking over the elaboration of the model for the city of the future. After a series of sectional essays, in the United States of America a holistic model was elaborated to account for the legislative frame, theoretical base of systemic ecology, economic frame, accumulated experience, and participation of the public, specialists, and decisional factors. Meanwhile, this model resolves the problem of energy management, and controls the flux of urban wastes, liquid or solid.

The national legislative, administrative and institutional frames in this field rely on national legislation, built on the universal principles of environmental legislation. It comprises regulations for environmental protection *{Constitution, Law of Environmental Protection)* and for other legislative fields (related to activities inherent to the development of cities: *General Regulatory of Urbanism, Law of Urbanism and Urban Planning.* At the international level, the most important documents in this field are the *Convention on Biological Diversity, Chart of Torremolinos,* and *Agenda 21.* The responsibility belongs in our country to the **Ministry of Environmental Protection,** and to the **Ministry of Public Works and Urban Planning,** and at the international level to the **Organization of the United Nations,** through its programs.

A first category comprises models elaborated by urban planners. Historically, there can be distinguished pre-urban and urban models.

The main pre-urban models are:

❖Progressive model, focused on the needs of humans;

❖Cultural model, based on the needs of human communities.

None of these models offers a satisfactory solution with respect to the impacts on the environment. Urban models are:

➢New progressive model, that promotes the functional city;

➢New cultural model, similar to the previous one;

➢Naturalist model, corresponding to organic architecture.

The naturalist model and, somewhat, the new cultural model decrease substantially the impacts on the environment, even though they are the result of a sectional approach.

• *"Technotopia"* promotes the model of the technical city, involving a complete elimination of the natural ecosystems;

• *"Anthropopolis"* accounts mainly for the needs of humans, involving public participation. All the urban models represent the result of a sectional and anthropocentric approach. Economic models involve performing an impact study for each proposed model for the development of the city. Problems are related to the identifying all components and assigning a market value to environmental goods and services. Mathematical models are synthetic models, resulting from a holistic approach; they assume the participation of a team of specialists, and approach mostly the holistic model for the development of cities.

The development of Hunedoara County, that includes Petrosani city, is mostly related to the development of mining activities and, during our century, to the development of various industrial sectors. The landscape is predominantly mountainous, and the clime supports western influences. The vegetation follows the levels of the landscape and clime, and the fauna corresponds to different vegetal associations.

The main problems at the county level relate to population increase and its consequences, manifested mainly though the overexploitation of natural resources, and pollution. Environmental protection regulations are sporadic and result from a sectional approach.

Petrosani appeared as a city in the period of mining industry development. From the biotope and community viewpoint, it lies within the range characteristic to Hunedoara County. A particular aspect is represented by the insufficient reserves of water.

The main problems at the level of Petrosani city are related to the elevated pollution, degradation of lands, supply of water and energy reserves, unwise management of urban and industrial wastes, and economic problems, related to the specific of the area and to the transition period crossed by the Romanian economy in general.

The model is focused on the satisfaction of human community needs (biological, social, and spiritual needs), and proposes three major strategic objectives:

- Optima! living condition for everybody- year 2020
- An ecological city- year 2093
- A sustainable city- undetermined period.

An economic model does not exist yet, but economic criteria can be presented to constitute the base of a possible model, through the analysis of existent problems and possible solutions. All these problems are related to the increased density of the population in Petrosani area and refer to:

- ✓ Waste management
- ✓ Water resources supply
- ✓ Misuse of lands
- ✓ Structure of productive activities and services
- ✓ Elevated consumption of energy
- ✓ Maintaining the autochthon population
- ✓ Increased impact on the environment.

Based on the economic and urban models for the development of Petrosani conurbation, a mathematical model could be elaborated to establish the optimum of a **Lagrange** function for consumer's utility.

CONCLUSIONS

- 1. In the conditions where the only chance for the survival of human socio-economic system is the implementation of the sustainable development solution, the elaboration of models for sustainable development represents a real problem and a priority as well.
- 2. An analysis of the theoretical base existent in the field reveals the presence of sectional approaches; this is why the only solution is to implement the sustainable development solution.
- 3. This approach involves approaching cities as ecological systems and identifying them according to the system model.
- 4. A comparison between urban and natural ecosystems evidences the existence of essential structural and functional differences between them, as a result of the presence of the human species as a dominant species in the first case.
- 5. The urban, economic, or mathematical models already elaborated, even if they represent the result of a sectional approach, can offer criteria necessary to construct a holistic model for the sustainable development of cities.
- 6. An analysis of the case of Petrosani conurbation proves that there are premises in

our country to implement the solution of sustainable development in the activities related to the development of cities.

▪ 7. An analysis of the theoretical base, proving the existence of uncertainties and lacks, sustains the necessity to develop the knowledge in the field of sustainable development of cities.

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ARGILACEOS SUSPENSION DESTABILIZATION TESTS OF WASTE WATER WITH SLURRIES

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Abstract: *Jiu Valley is the most important checkable hard coal field from Romania. At the same time this area has great touristic resources, bat most part of them is not useful because of environmental pollution due to mining activities. In order to develop the tourism in this area it is necessary first of all to rehabilitate the environment. Mining industry use and return to the environment great quantities of water resulted both during coal processing and coal mining.*

Key words: *waste water, coagulation, flocculation, ferric chloride, turbidity, sedimentation velocity*

1. WASTE WATER PHYSICAL-CHEMICAL CHARACTERISTICS

The most important water polluting sources are the coal processing plants. From raw coal processed, 14-18% represent sludge with particles, size less then 0,5 mm that it is found entirely in the processing water.

These facts impose the existence of a complex processing water circulation. Processing water quality and waste water flowed in emissary characteristics depend on a good water circulation management.

Waste water physical – chemical characteristics are show in table no.1.

From presented data in table 1 we may draw the following conclusions:

⇒ biological oxygen demand value are under the admissible values for ground water, third utility range (maximal 12 mg/dm^3) .This fact show that organic water do not represent a proper pollutant for Jiu water;

⇒ chemical oxygen demand has relatively great values but this thing is due to countryside farms placed at the extremities of coal field.

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⇒ solid particles that have excessive high values. These values are because of coal processing plants and Paroseni power station also.

We may conclude that coal mining industry is responsible for water pollution with solid suspensions and in a smaller extent for pollution with organic mater.

Table 1. *Waste water physical-chemical characteristics*

Characteristics	U.M.	Value	Admissible value
Hardness	° Hardness	2.92-5.04	-
Oxygen dissolved	mg/dm^3	7.8-8.7	4 min.
Oxygen chemical demand	mg/dm^3	20.2-45.5	25 max.
Oxygen biological demand	mg/dm^3	0.6-2.0	12 max.
Cl^-	mg/dm^3	27.5-33.2	1.0 max.
Fe^{2+}	mg/dm^3	0	0.5 max.
CN^-	mg/dm^3	0	0.5 max.
Suspension	mg/dm^3	500-11.000	80 max.
pH	unit. pH	7.2-8.4	6.5 –8.5

2. EXPERIMENTAL STUDY REGARDING WASTE WATER CLEANING EFFICIENT IMPROVEMENT

Coal argillaceous suspension flowed by the mining industry has a very great gravitational suspension because by granulometric point of view are placed in colloidal or semicolloidal domain. By this reason, excluding specific flora and fauna disappearance, river Jiu is excluded from touristic circuit, even if it passes an excellent canyon.

The cleaning of such waste water involve to important aspects:

a) Waste water cleaning by argillaceous colloidal suspension destabilization;
b) Sterile sludge resulted after waste water cleaning drying and depositing;

2.1 Cleaning technology tested for waste water with solid suspensions

The reagents used by coal mining industry for waste water cleaning may be divided in three classes:

- electrolytes or chemical salts;
- natural polymers;
- synthetic polymers.

These reagents are used single or mixed. Cleaning reagents tested were: plaster , lime, ferric chloride , sodium chloride, calcium chloride, ferrous sulphate , aluminum sulphate with specific consumption of 100-600 g/m^3; caustificated starch with specific consumption of 30-100 g/m^3, polyacrylamide , polias 330, solacril, magnafloc, separam, polifin with specific consumption of 1-6 g/m^3.

Based on tests made with cleaning reagents mentioned before we rdaw the following conclusions:

- for first class substances (plaster, lime, $FeCl_3$, NaCl, $CaCl_2$, $Al_2(SO_4)_3$) is obtained a clear water with a very small turbidity, with a solid particles content of 0.01-0.3 g/l but sedimentation velocity is very small- 0.2-0.4 m/h and sludge settle rate is very small also ;

- second class substances (starch derivates) provide coal slurries flocculation together with a part of colloidal clay. Cleaning tests realized with caustificated starch provides good sedimentation velocity between 0.6-1.5 m/h but cleaned water turbidity is great enough (>1g/l);

- for third class substances (polyacrilamide, polifin, etc.) the cleaning process develops with great velocities but cleaned water has a very high content of solid particles (>3g/l) because reagent act mainly on coal particles , surface , clay being very little affected . Sludge obtained is well settled.

Taking into account the fact that tested reagents regimes did not prove their efficiency because flowed waste water quality exceed much legal limits, we proposed to realize a systematic study that to analyze separately the two processes involved in water cleaning: coagulation and flocculation using trivalent electrolytes (Badulescu, 1998). The tested electrolytes are : ferric chloride and aluminum sulphate.

Comparing this two electrolytes we draw the following conclusions:

- ferric chloride forms flocules with greater density than aluminum sulphate:
- aluminum sulphate form hydrolysis products just in a limited pH domain and do not hydrolyse at temperatures less that 10˚C;
- ferric chlorides formed products more insoluble than aluminum sulphate;
- aluminum sulphate is not colored an excess the water as ferric chloride colored water in yellow.

2.2. Coagulant concentration influence

Taking into account the consideration presented above we decide that is better to use ferric chloride in our study. Cleaning process using only ferric chloride did not give good results. This reagent realizes just a good coagulation, this fact being proved by the very small turbidity of cleaned water (figure 1) but the sedimentation velocity is very small.

We note that cleaned water turbidity decreased when coagulant reagent concentration increase and sludge settlement decrease. Coagulant reagent concentration increasing over a critical concentration will reduce the sedimentation velocity but the cleaned water quality remains the same.

Sludge settlement depends on coagulant reagent concentration just in the measure that affects cleaned water transparency.

2.3. Flocculants concentration influence

Flocculent reagent concentration is not involved directly in parameters that described the cleaning process. Because flocculation has like effect just floccules size increasing, the influence of this parameters will consist first of all in sedimentation velocity increasing.

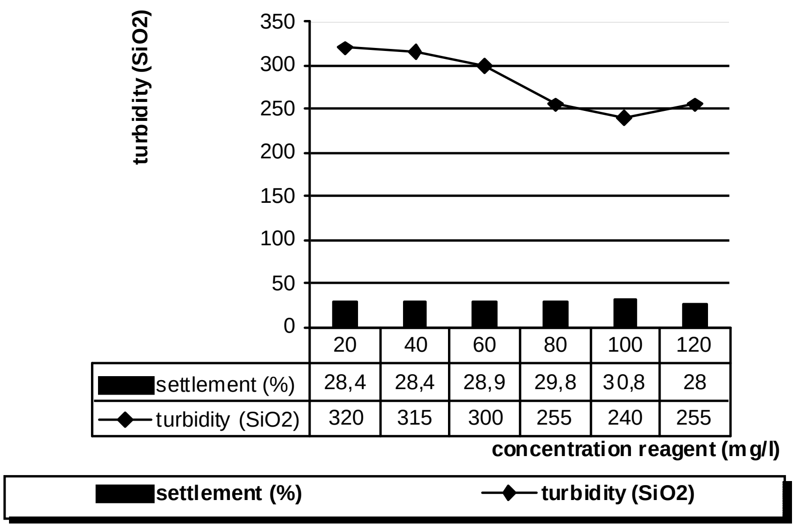


Fig.1 Reagent concentration influence on cleaned water turbidity

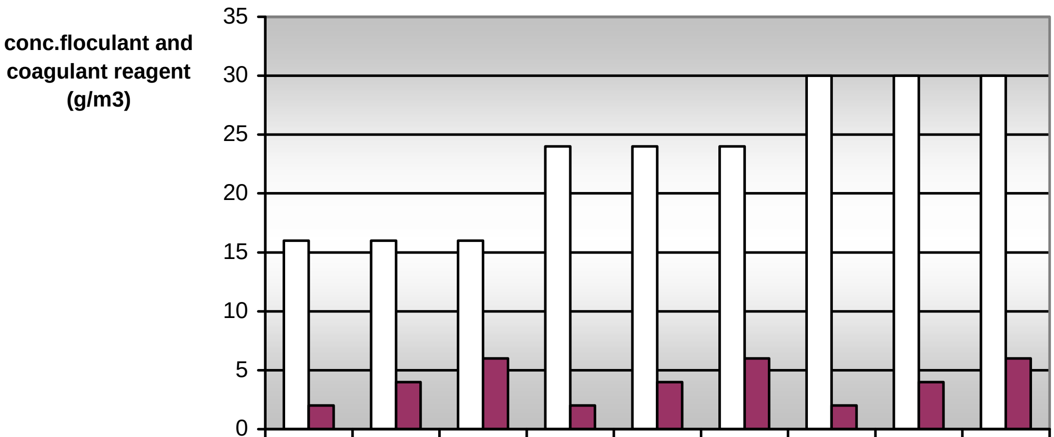
Like flocculant reagent, we use polyacrylamide. Floculant reagent concentration effect on cleaned water clarity is show in figure 2.

By graph presented in figure 2 we note the strong influence of floculant reagent concentration on cleaned water turbidity.

Like in the case of coagulation reagents, this is an optimal concentration of floculant reagent when we obtain the smallest supernatant turbidity.

The turbidity increasing at great concentration of floculant reagent is because particles surfaces covering rate by polymer molecules increase and in this way they are not other possibility to create bridges between them.

To a floculant concentration great enough then is not more sedimentation process water volume being entirely gelified. Because these two cleaning reagent are used together ($FeCl_3$ and polyacrylamide) it is very important to evaluate the cumulate action on cleaned water turbidity and sludge settlement.Figure 3 and 4 present the dependence of cleaned water turbidity at different floculant reagent concentration on coagulant reagent concentration (figure 3) respectively the dependence of cleaned water turbidity at different coagulant reagent concentration on floculant reagent concentration.



turbidity (SiO2)	850	1200	1300	560	1500	590	300	200	40
□ conc.coagulant reagent(g/m3)	16	16	16	24	24	24	30	30	30
■ conc.floculant reagent(g/m3)	2	4	6	2	4	6	2	4	6

□ conc.coagulant reagent(g/m3) ■ conc.floculant reagent(g/m3)

Fig. 2. Floculant concentration influence on cleaned water turbidity

Figure 4 show that coagulant reagent concentration increasing has like effect cleaned water turbidity decreasing indifferent by floculant reagent concentration. On the other hand floculant reagent concentration increasing has different effects depending on coagulant reagent concentration. These facts prove that floculation depend strongly on coagulation those action is just completed.

2.4. Solide particles concentration and electrokinetic potential influence on sedimentation velocity

On waste water from coal processing plants cleaning particles, important issues consist on variation of solid particles concentration.

In table 2 and figure 5 is presented solide phase concentration influence on waste water sedimentation velocity.

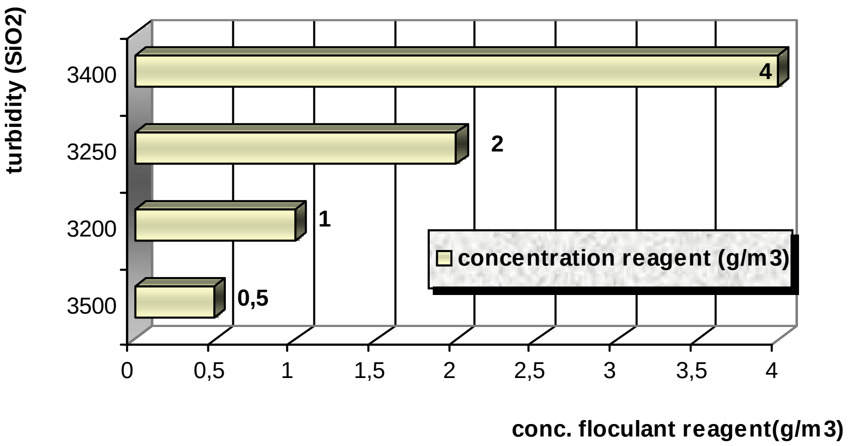


Fig. 3. Cleaned water turbidity dependence at different floculant reagent concentration on coagulant reagent concentration

Table 2. *Solide phase concentration influence on waste water sedimentation velocity*

Suspensions (g/l)	$FeCl_3$ (mg/l)	PAA (mg/l)	Sedimentation velocity (m/h)
20	32	6	5,41
40	32	6	2,00
60	32	10	0,73
200	32	4	2,10

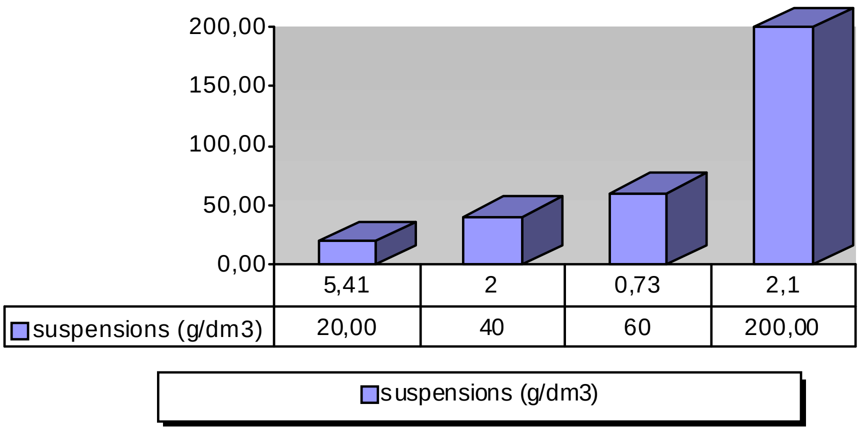


Fig. 4. Cleaned water turbidity dependence at different coagulant reagent concentration on floculant reagent concentration

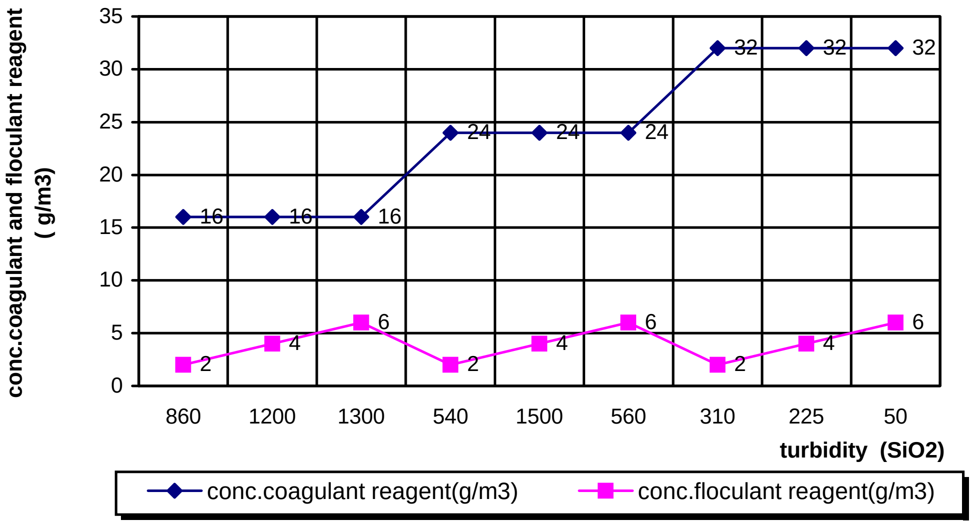


Fig. 5. Solide phase concentration influence on waste water sedimentation velocity

From data presented in table 2 and figure 5 we observe that the optimum reagent concentration (determined above 32 g/l) it is not influenced by solid suspension concentration. This fact can be explicated if we take into account the fact that electrokinetic potential depend first of all by electrolites concentration.

By this reason, even if a solid phase concentration differs, at the same electrokinetic potential optimum coagulation concentration will be the same.

In the case of floculant reagent, besides optimum concentration it is necessary to provide the requested amount of reagent that will be adsorbed to the particle surfaces in order to create flocules bridges.

For this reason its concentration depends on solid particles concentration also.

In table 3 is show the electrokinetic potential zeta influence on cleaning reagent consumption.

Table 3. *Electrokinetic potential influence on cleaning reagent consumption*

Electrokinetic potential ξ (mV)	$FeCl_3$ (mg/l)	PAA (mg/l)	Sedimentation velocity (m/h)
40	100	20	0,73
60	40	4	1,80
140	32	2	2,50
200	16	6	2,93

From data presented in table 3 we observe that to smaller potential is necessary a greater coagulant consumption, fact that seems to be inn contradiction with expected results.

Ferric ions adsorption is controlled by zeta potential. The electrokinetic's potential is greater the adsorption process will develop easier.

The dependence of optimum ferric chloride consumption by electrokinetic's potential is exponential one:

$$C_{FeCl3} = 2374 \cdot \xi^{-0,95} \quad (1)$$

The optimal flocculants concentration increase with electrokinetic's potential decreasing too. This dependence is exponential also:

$$C_{PAA} = 190 \cdot \xi^{-0,78} \quad (2)$$

The relations presented above aim to compute the reagents regimes correlated with suspension nature respectively the souses of processed coal.

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THE TOPOGRAPHICAL ELEMENTS' JUNCTION MECHANICS TRANSMISSION PIT ON A VERTICAL

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LARISA FILIP **

In current practice there are known and applied various methods such as junction: radial, triangle linking, forced alignment, intersections, etc.

The goal is to translate the topographic (coordinate, orientation) near the pit area, and remained in the ramp of pit (underground) (fig.1).

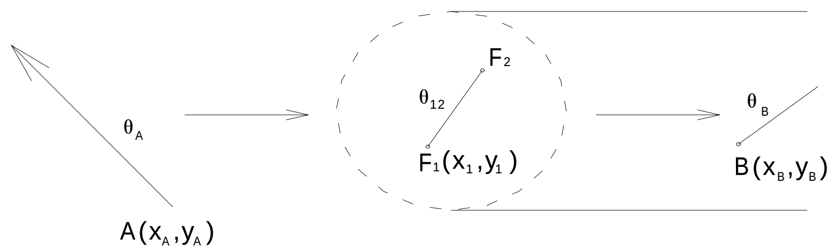

Fig. 1

So:
It is known:

$$x_{A,}\, y_A \text{ si } \theta_A$$

The cause:.

a) $x_{1,}\, y_1$ si θ_{12}

b) $x_{B,}\, y_B$ si θ_B

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The methods known for the translation of the topographical elements (land, mineral) use the angles and distances sizes measured or calculated, without a rigorous processing in advance.

On the other hand, the measured angles that result from the measured directions have large errors because of their small lengths.

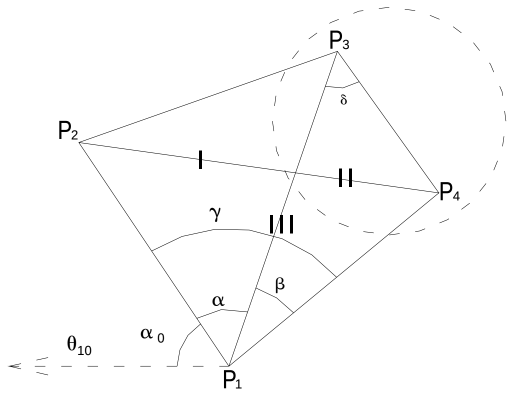

Fig. 2

Anyway, it can be done at a quadrilateral area (fig. 2) (linking) defined by topographical elements:

- coordinates x_1, y_1 point of P_1;
- orient θ_{10} of the direction P_1P_0;
- data and quadrilateral sides α_0, measured.

The measured lines can be taken using the theory of smaller squares, writing:

$$(\alpha)+(\beta)-(\gamma)=0 \tag{1}$$

Or:

$$\alpha + v_\alpha + \beta + v_\beta - \gamma - v_\gamma = 0 \tag{2}$$

The angles α, β and δ are obtained from the edge triangles I, II and III with the general relationship:

$$tg\frac{x}{2} = \sqrt{\frac{(p-b)(p-c)}{p(p-a)}} \tag{3}$$

$$p = \frac{a+b+c}{2}$$

ω being α, β or δ..

And then:

$$v_\alpha + v_\beta - v_\gamma + \omega = 0 \tag{4}$$

$$\omega = \alpha + \beta - \gamma$$

The correlations $v_\alpha, v_\beta and v_\gamma$ are obtained with the relationships:

$$\begin{aligned} v''_\alpha &= \left(A_1 v_{23} + B_1 v_{13} + C_1 v_{12}\right)\rho'' \\ v''_\beta &= \left(A_2 v_{34} + B_2 v_{14} + C_2 v_{13}\right)\rho'' \\ v''_\gamma &= \left(A_3 v_{24} + B_3 v_{14} + C_3 v_{12}\right)\rho'' \end{aligned} \tag{5}$$

Introducing in relation (4) results:

$$\begin{aligned} &\left(C_1 - C_3\right)v_{12} + A_1 v_{23} + A_2 v_{34} + \left(B_2 - B_3\right)v_{14} + \\ &+ \left(B_1 + C_2\right)v_{13} + A_3 v_{24} + \frac{\omega''}{\rho''} = 0 \end{aligned} \tag{6}$$

Correction coefficients are obtained with normal relations:

$$\begin{aligned} A &= \frac{\sin\alpha}{4}\left(\frac{1}{p-b} + \frac{1}{p-c} - \frac{1}{p} + \frac{1}{p-a}\right) \\ B &= \frac{\sin\beta}{4}\left(-\frac{1}{p-b} + \frac{1}{p-c} - \frac{1}{p} - \frac{1}{p-a}\right) \\ C &= \frac{\sin\gamma}{4}\left(\frac{1}{p-b} - \frac{1}{p-c} - \frac{1}{p} - \frac{1}{p-a}\right) \end{aligned} \tag{7}$$

With parameters 1, 2 and 3 corresponding to the triangles I, II and III, equation (6) has the form:

$$[av] + \omega_0 = 0 \tag{8}$$

Correspondent to the normal equation:

$$[aa]k + \omega_0 = 0 \tag{9}$$

Resolving it results:

$$k = -\frac{\omega_0}{[aa]} \tag{10}$$

While:

$$v_{12} = \rho"\left(C_1 - C_3\right)k$$

$$v_{23} = \rho" A_1 k \tag{11}$$

$$v_{24} = \rho" A_3 k$$

The estimated values of the sides are obtained from the corrected measures, corrections resulted from equation (11). For example:

$$\left(P_1P_2\right) = P_1P_2 + v_{12}$$

$$\left(P_2P_3\right) = P_2P_3 + v_{23}$$

…………………..

In the last stage, using the estimated values of the sides we calculate the angles α, γ and the survey elements from the clear area of the shaft.

$$x_3 = x_1 + \left(P_1P_3\right)\cos\left(\theta_{10} + \alpha_0 + \alpha\right)$$

$$y_3 = x_1 + \left(P_1P_3\right)\sin\left(\theta_{10} + \alpha_0 + \alpha\right)$$

$$\theta_{34} = \theta_{10} + \alpha_0 + \alpha - \delta \pm 200$$

For underground method applies similar.

CONCLUSIONS

The method of presentation contains a rigorous process of processing of the measured measurements.

The measurements are determined more precisely, and the precisions can be assessed. The calculus volume does not constitute a disadvantage by using current techniques.

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MODERN TECHNOLOGIES FOR MONITORING THE SUBSIDENCE PHENOMENON

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Abstract: *The mining sinking field begins in raw rock from the exploitation horizon and ends at the surface of the covering bulked layer where there are the constructions, the transport network, art works and agricultural cultures. The artificial gap created by exploiting the rock clears out the natural support of the covering layers so that the covering layers bend themselves over the exploited space under the influence of their weight and the underground hole is closed more or less. The ampleness of the movement into the superior layers depends on this closure in time of the underground gap (the convergence of the exploited space). So this is the basic size of calculating the movement of the rock and the surface field. During the last years, the appearance of the calculation technique was felt by its many advantages. Through its facilities that are offered by the computers we have as follows: great speed of calculation, great capacity of stocking the information, less errors of calculation. By using the above mentioned facilities we could stimulate on many examples the mechanism of forming the sinking beds, graphic representation – plan and level representation and also graphic representations of the displacement and deforming parameters based on some concrete measurements. The results obtained on different programs will be presented as follows making references only to the qualitative aspect of the problems.*

Key words: *superposition effects, displacement and deforming, underground gaps*

1. CHECKING THE PRINCIPLE OF SUPERPOSITION EFFECTS

One of the principles used for developing the theory of displacement and deforming the field surface under the influence of the exploited space is the principle of the superposition effects. This principle specifies that the effect of displacement represents a sum of individual effects generated by total of caused generated by the displacement, sum respected on each current point of the surface.

One of the most important causes is represented by the existence of underground gaps that by using the mechanism of forming the pressures into the rock will generate on the terrain surface vectors of the displacement convergent to the geometric center of the gap. From the

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point of view of the nuances it is important not to be confounded the causes that generate the displacement and deforming that are the underground gaps with the causes that favor this process and they are as follows:

The system of fissure fracture
Terrain bent
Solicitations generated by the constructions
Great shoots
Regime of underground waters

We can say that the principle of effects superposition appears as a qualitative principle that satisfies the mechanism of displacement in a first approximation. The aleators factors whose effects cannot be measured will give the exceptions with bed shape. If the aleators factors have a great ponder in comparison with the balance given by the displacements caused by the existence of the underground gaps we stimulated on the computer this principle.

If we consider G1, G2, … Gn gaps resulted after exploiting the rock corps and A1, A2 … An final and individual sinking beds for the gaps G1, G2, … Gn (Fig. 1)

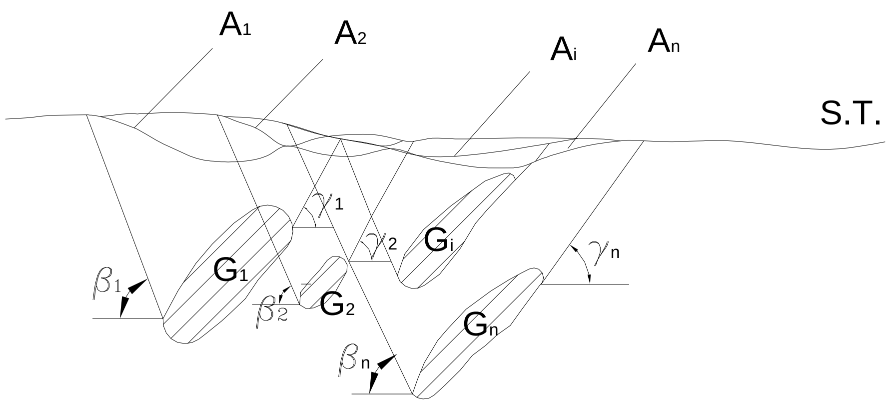

Fig. 1 Individual effects into the coal face

If we consider the S.T.(ITS) as being the initial topographical surface of the field and S.T.F. (FTS) the final topographical surface after stopping the phenomenon and establishing completely the field (Fig. 2)

From fig. 2 we can notice that the final shape of the field surface on the studied profile is not framed in no known shapes for the layering coal for small declination (<19grd) and also for great declination.

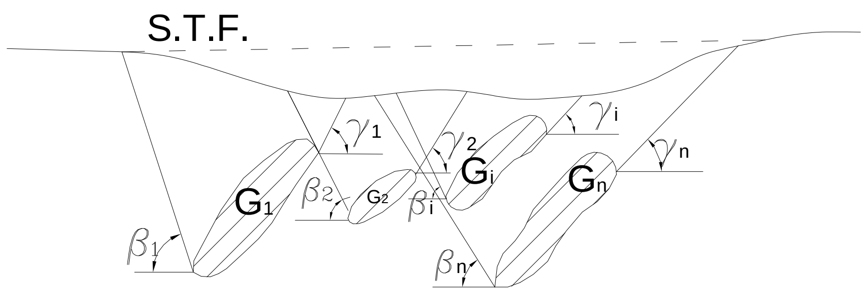

Fig. 2 Shape of sinking bed after establishing the surface

The mathematical modeling will be made on settled individual beds so independent of time, being followed only the final shape of the bed on a certain profile.

It was accepted that for a gap Gi situated at Hi deepness the distribution of the sinking made by this gap has a shape of a Gauss distribution, this shape existing also into the literature (Martoş). In particular for the case from fig. 3 by choosing the coordination system into zero point we have as follows:

$$f(x) = a\exp(-b(x-c)^2) \qquad (1)$$

where: a., b, c – parameters having the following significances:
a – maximum sinking (m) ;
b – declination on function b<1;
c – distance from 0 to the point of maximum sinking / m/ ;
x – distance from 0 (m);
f (x) - sinking (mm) .

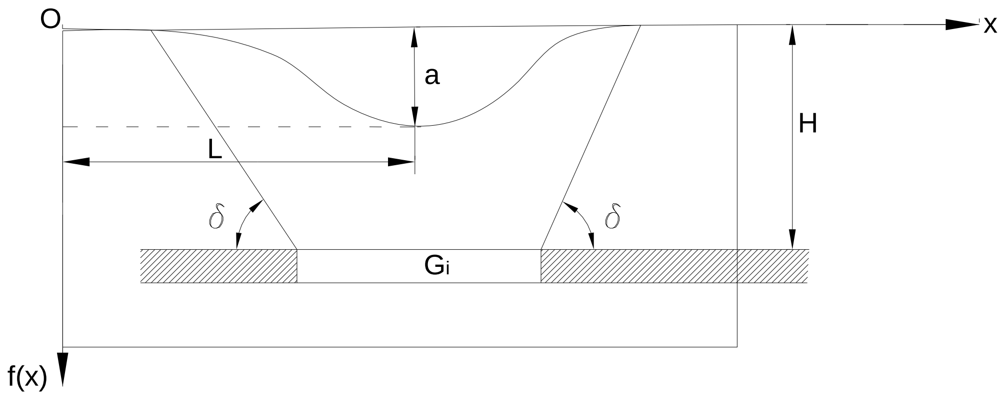

Fig. 3 GAUSS Distribution for horizontal rocks

By using the computer program for designing and mathematical calculation (MCAD-2) it was generated a number of examples like the one from fig. 3 and where the exploitation deepness had variations. (H).

2. QUALITATIVE ANALYSE DEPENDING ON THE EXPLOITATION DEEPNESS

So we could represent in a spatial shape the dependence of the sinking of I point (maximum point) of exploitation deepness (H).

In fig. 4 we have as follows:
Ot – time axis (years);
OW-axis of sinking speeds (cm./year)
OH – axis of exploitation deepness (m)

CURVE (1) – represents the function that adjusts the sinking speed of the central mark I depending on the time and for an exploitation deep of H=100 m. We can notice that for this deep, at the volume of considered gap (v) it is provoked the phenomenon of landfall till the surface into R point from where it is formed the sinking bell.

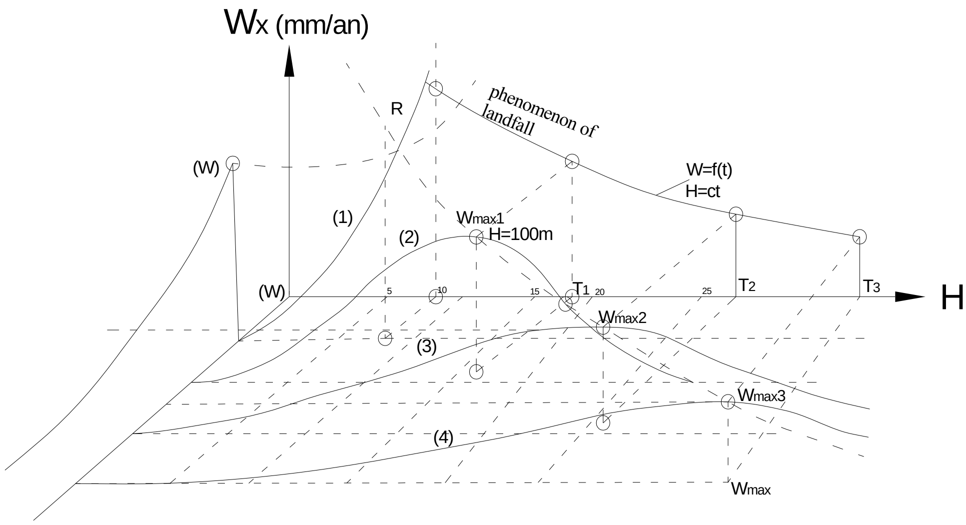

Fig. 4 Dependence W=f(V, H, T)

CURVE (2) - represents the function that adjusts the sinking speed of the central mark I depending on the time and for an exploitation deep of H=200m at the same volume of considered gap.

In this case the landfall phenomenon does not appear by obtaining a value T1 for the time in years for which the phenomenon is ended. Correspondent to it we have the value of Wmax1 for the maximum sinking of the mark I.

CURVE(3) – same function but for H=300m and same gap volume (v).

Analogy with the curves 1, 2, 3 there will be obtained the curves 4 and 5. Finally it will be obtained the following relations:

$$W_{max1} >>> W_{max2} > W_{max3} > W_{max2} > W_{max1}$$ - relation between sinking

$$T_5 < T_4 > T_3 > T_2 >>> T_1$$ - relation between periods (2)

$$H_5 < H_4 < H_3 < H_2 < H_1$$ - relation between deepness

By excepting the relation between the deepness which had been imposed by the chosen model as being linear, the relations between the maximum sinking (Wmax(1)) and periods of phenomena T(i) are non-linear and cannot be established only based on the real measurements.

The conclusion that comes from this simulation mentioned above is that the maximum sinking of the central mark I for a volume of imposed gap to be constant (v) is decreased non-linear with increasing the deep, the time of developing the phenomenon is decreased non-linear with increasing the deep and the time is increased with increasing the deep.

These curves are presented into the fig. (5).

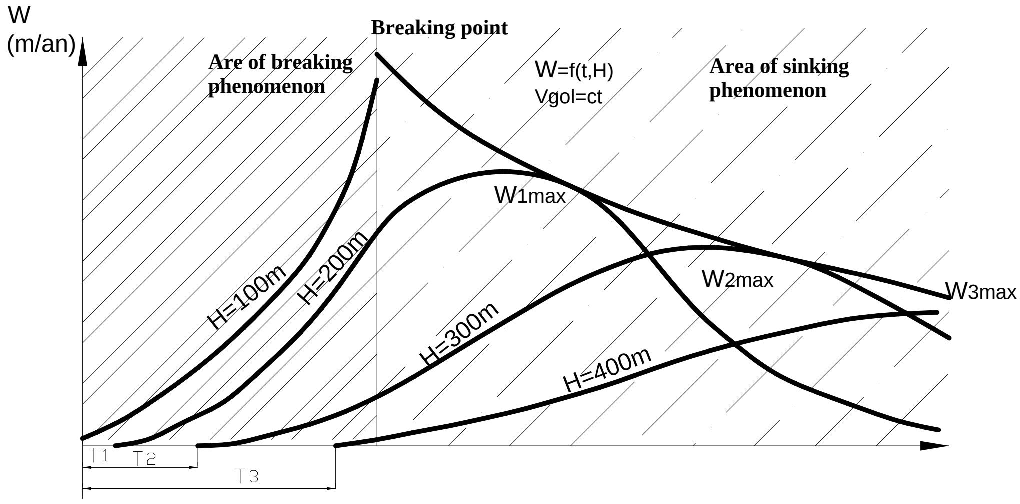

Fig. 5

3. QUALITATIVE ANALYSE IN REPORT WITH GAPS VOLUME

By using the simulation method it was created an example where it is evidenced the singular action of the variation of the underground gap. In fig. 6, there are presented the individual effects of three volumes of different gaps but situated on same deep. Reporting the effects to a tri-axis system and projecting the points into plans WOV it will be obtained the curve that reflects the dependence of the maximum sinking Wmax of gaps volume Vg.

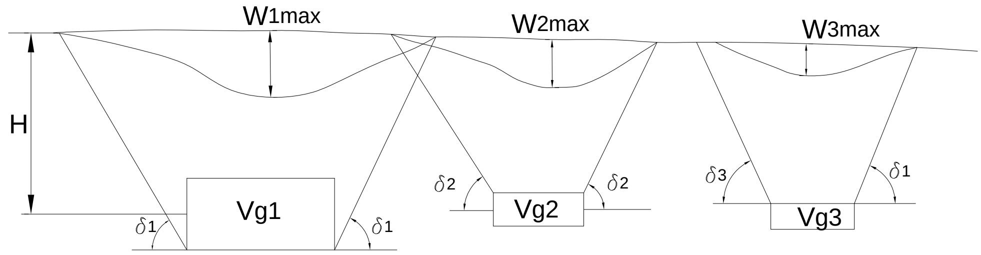

Fig. 6 Analyses in report with gaps volume

From fig. 7 it is noticed the non-linear dependence of the sinking of gaps volume. The quantitative appreciation and shape of the non-linear dependence can be done only with real measurements.

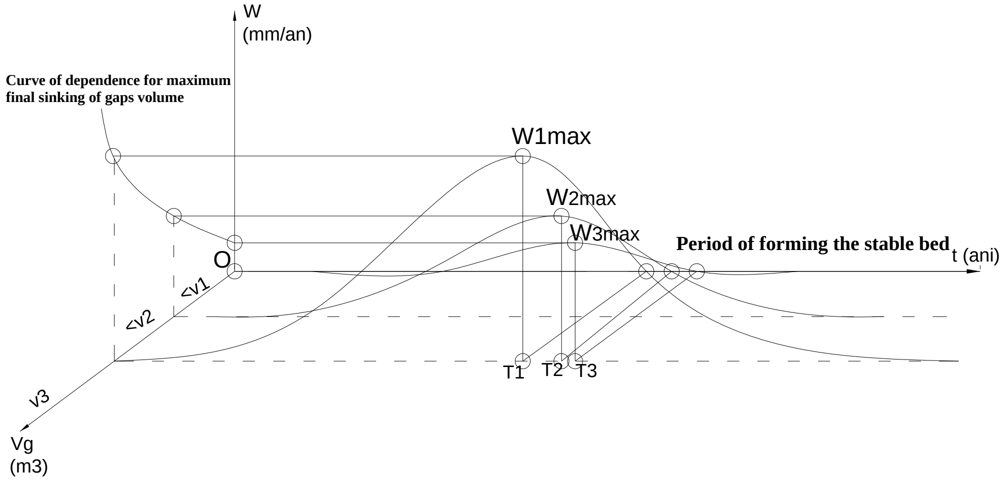

Fig. 7 Dependence W=f(V)

4. QUALITATIVE ANALYSE IN REPORT WITH THE GEOLOGICAL FEATURES OF THE ROCK

The displacement and deformation phenomenon of the terrain surface is generated by the tension state created by the existence of the gaps provoked by the underground exploitation. Under this state of supplementary tension, the rock enters into movement of creeping until it reaches an equilibrium state when the displacement stops.

If we consider a mark from the sinking bed. Taking the sinking recorded on this mark and reporting them at the height of stratigraphic column (H), it is formed the report W/H, which has a physical signification of deformation if we accept the idea that the sinking represent to short the stratigraphic column at the value of H(0) initial at the value of H(t(i)) at t moment t. (shorting represents the height at first measurement – height at current measurement < = > shorting = sinking (w)

$$\frac{\text{Shorting (w)}}{\text{stratigrap hic column (H)}} = \text{deformatio n(\%)} \tag{3}$$

By evaluating the report between the sinking and the height of stratigraphic column at successive points of time we can do the following graphic representation:

deformation(%)=F(time) – that represents the creeping function of the surface into point R(x.y) materialised with the topographic mark.

The graphic representation was made into the fig. 8.

It had been analysed the slimy and plastic model of the rocks for which the creeping function is given as follows:

$$E(t) = E_{\max}(1 - \exp(-t\frac{E_e}{\eta})) \tag{4}$$

E(t) – deformation at moment t (%) ;

Emax - maximum deformation at moment $t \to \infty)$;

Ee – elasticity module of the rock (daN/cm.);
η - dynamic slim of the rock (poise) ;
T – time.

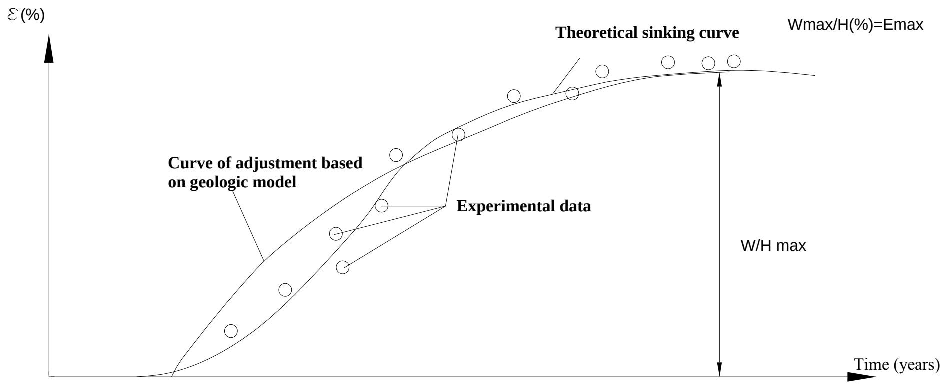

Fig. 8 Creeping function

This function does not adjust the experimental data, but it was chosen in order to attract the attention over the fact that the phenomena of displacement and deforming of the surface enters into the category of creeping phenomena and it should be studied into the research groups that should comprise also the specialists in rock mechanics.

If the qualitative analyse of the dependence of the sinking of a mark by a mechanic feature of the rock presume the analyse of different sinking beds and existence of a great number of data with high grade of certitude, the solution applied at studying it in report with the geological features was also the stimulation on the computer by varying the viscidity parameter η and parameter Wmax.

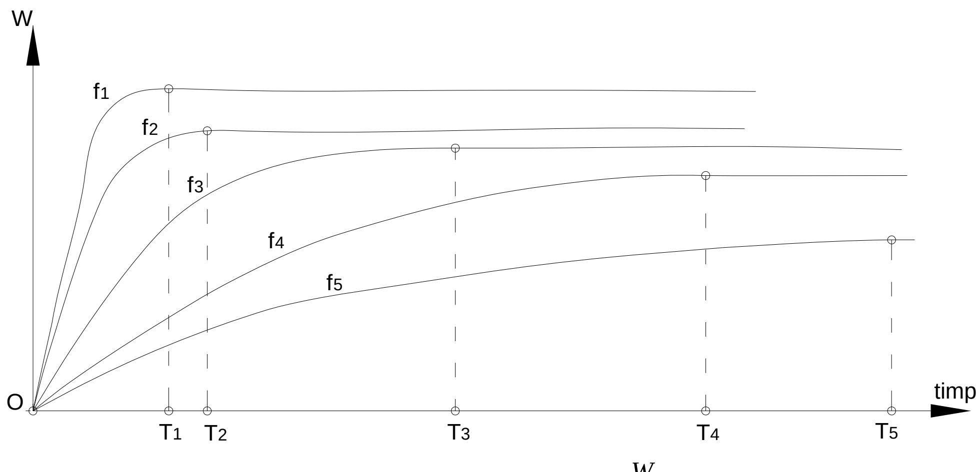

Fig. 9 Dependency of viscidity of the rock ($\frac{W_{\max i}}{H} = ct$)

5. CONCLUSIONS

From the analyse of the results it will be resulted the following conclusions:
The function used to do the adjustment was the time:

$$W(t) = W_{\max}(1 - \exp(-ct)) \tag{5}$$

where: W max and c were determined by regression by using the lost squares and obtaining the following values:

$$c_5 = 0.0054321$$
$$c_3 = 0.0074312$$
$$\ldots\ldots\ldots\ldots\ldots\ldots\ldots$$
$$c_1 > c_2 > c_3 > c_4 > c_5 \quad \text{restrictive relation} \tag{6}$$

By presuming the possibility to apply the geologic model we have as follows:

$$c(i) = \frac{Ee}{\eta(i)} \tag{7}$$

From (6) results:

$$\eta_1 < \eta_2 < \eta_3 < \eta_4 < \eta_5 \quad \text{viscidities} \tag{8}$$

The resulted conclusion is that it is a concordance between the geologic model chosen and the behaviour reflected by the sinking in sense that the smaller the rock viscidity is, the faster is the sinking phenomenon and the sinking time is smaller.

From the data used on different marks from the studied sinking bed, it was noticed a regression and correlation calculation that the numeric parameter c has the limits of 100-5 and 100-3 which requests a continuous research in this direction, because by taking as value the elasticity modul (Ee) and the dynamic viscidity η for same type rocks, making the report of these measures, there are obtained numerical values with order of the results of premaking the field data for c value.

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THE QUALITATIVE ANALYZE OF THE CARTOGRAPHICAL PROJECTIONS USED IN OUR COUNTRY

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MIHAI HERBEI[**]

Abstract: *The basic documents made after the geodesic and topographical measurements for accomplishing different projects are the topographical plans and maps. The accomplishment of these documents must represent very good of the terrestrial area with representative elements. Using the cartographic projections that must be made depending on the quality of the representation so that the plans and maps, which are obtained, should be made with high precision and high quality makes these documents. The elements that are deformed are as follows: angles, distances and areas.*

Key words: *Gauss–Kruger, UTM, Stereographic, projection, deformations*

1. INTRODUCTION

To choose a projection system is made depending on the size and shape of the area that should be represented. So, for the mid territories, whose shape is almost round we can use the stereographic projections, and for the big ones, as continents, we can use the cylindrical and transversal projection Gauss-Kruger. The military and engineering needs are satisfied by the conform projections, and for surveying needs we can use the equivalent projections, and for the maritime and air navigation it is used the Mercator projection. In Romania there were used along the time the following projection systems: Conic projection *Bonne*, on the *Clarke* ellipsoid, between1916-1930, Stereographic projection on the *Hayford* ellipsoid, between 930-1951, *Gauss-Kruger* projection, on the *Krasowsky* ellipsoid, between1951-1970, Stereographic projection 70, on the *Krasowsky* ellipsoid, starting with 1971. For the military applications in Romania it is used the **UTM** projection **(*U*niversal *T*ransverse *M*ercator).**

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2. THE ANALYZE OF THE DEFORMATIONS FOR THE PROJECTIONS USED IN ROMANIA

2.1 Cylindrical transversal Gauss–Kruger projection

2.1.1 The geometric elements of this representation

This projection is characterized that a certain area of the terrestrial surface is represented on a cylinder tangent or transversal to the surface of reference which is considered a sphere (Fig. 1)

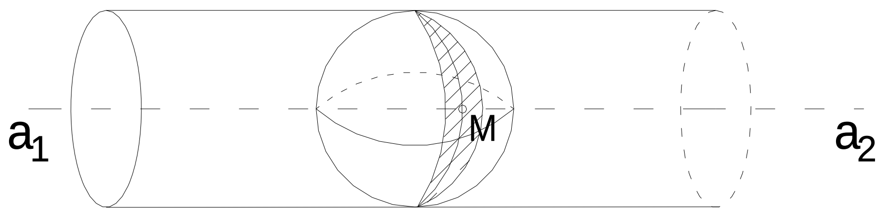

Fig. 1

In a certain position of the reference surface inside the cylinder this is tangent after the median line. This line after it is developed into the plane of the cylinder remains unreformed. It is deformed any element of length situated on a side and the other side of the tangent meridian if the distance is big.

By being imposed the limit of 1/2500, it results that the representing area is limited to 6^o and it is named shaft of representation. By imaginary rotting of the reference surface inside the cylinder it can be obtained 60 shafts numbered from 1 to 60 over the meridian 180^o. In these conditions our country belongs to the shafts 34 and 35 limited by the meridians 18^o and 24^o respective 24^o and 30^o. The reference surface is divided in bends having the width of 4^0 numbered for each hemisphere with letters from A to V. being situated at the latitude of 45^o of the north hemisphere, our country belongs to the bend L, with small regions in bends K and M, limited by the parallels 44^o and 48^o.

By such deviation it results trapezes having the sizes of 4^oX4^o noted for our country with L-34 and L-35.

The reference system which establishes the positions in plane of the points represented on the developed surface of the cylinder is xOy. (Fig. 2)

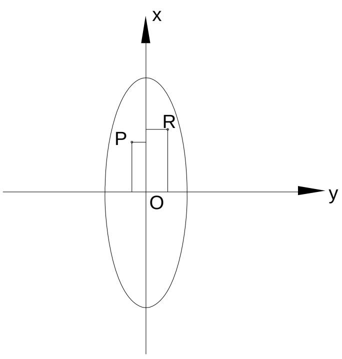

Fig. 2 The rectangular reference system in Gauss – Kruger projection

The x ax is oriented on the direction of the axial meridian of the shaft, and the y ax is on the direction of the equator. In order to have the points inside the shaft to have the coordinates positive, the x ax is moved to West with 500 km so that the origin has the coordinates 0 km and 500 km.

The writing of the coordinates is made as follows:

For R point:
$$\begin{aligned} x_1 &= x_1^C \\ y_1 &= n(500Km + y_1^C) \end{aligned} \tag{1}$$

For P point:
$$\begin{aligned} x_2 &= x_2^C \\ y_2 &= n(500Km - y_2^C) \end{aligned} \tag{2}$$

In which:

- x_1^C , x_2^C - is the calculated value and represents the distance between the points R and P to the equator
- y_1^C , y_2^C - is the calculated value and represents the distance between the points R and P and the axial meridian
- n – is the number of the shaft to the Greenwich (4 or 5 for our country); n does not multiply the parenthesis

2.1.2 Distance Deformations

In order to establish the deformations in Gauss – Kruger deformations it is used:

The module of linear deformation:
$$\mu = 1 + \frac{y^2}{2R_m^2} = \frac{dS}{ds} \tag{3}$$

where: - dS - is the linear element in plan
- ds - is the linear element on the ellipsoid

It results :

Unitary deformation
$$\Delta = \frac{dS - ds}{ds} = \frac{y^2}{2R^2} \tag{4}$$

If at medium latitude of our country the shaft of 1^0 has the width of 80 km, in the limitation area of the shaft that means at 3^0 over the axial meridian the deformation on km may reach 0.70m/km. for the representations at great scales the deformation is so great that we can use the shafts of 3^0 when at the limit the deformation on km is only of 0.18 m/km.

We can notice from the following diagram (Fig. 3) that into the Gauss projection the linear relative deformations are positive and proportional with the distance over the axial meridian.

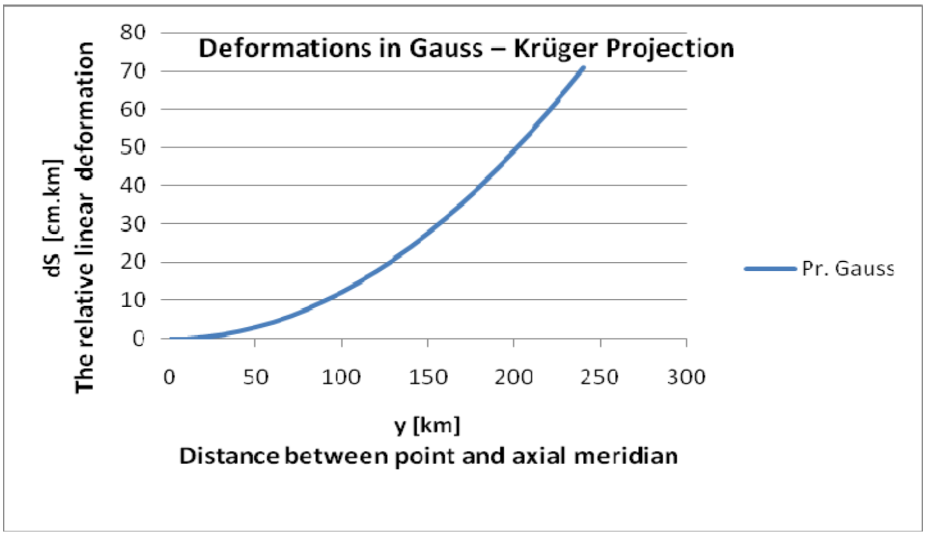

Fig. 3 Diagram of deformations in Gauss-Kruger projection

2.2 Stereographic Projection 1970

2.2.1 Geometric elements of this representation

The stereographic representation (projection) is characterized that a certain part of terrestrial surface is represented on the surface of a plan tangent or secant on the reference surface (fig. 4).

Geometric elements of the representation are:

- H – projection plan tangent or secant to the reference surface;
- C – projection center;
- O_1 – point of view where start the projection rays situated on the reference surface opposed to C point;
- P – point which is represented;
- P' – projection of P point on H plan.
- x ax on the direction of the C point meridian;
- y ax on the direction of the C point parallel;

In order that all coordinates should be positive the origin of axes that are moved and becomes O (500 Km ; 500 Km).

The geographic coordinates of C point are:

$$\Phi_0 = 46^0; \qquad \lambda_0 = 25^0$$

The C point is situated near Făgăraş town.

For a certain point P situated on the reference surface having the plan coordinates x and y in *stereographic projection* with tangent plan are as follows:

$$\begin{aligned} x &= a_{10}\Delta\varphi + a_{20}\Delta\varphi^2 + a_0 2\beta + \ldots \\ y &= b_{01} l + b_{11}\Delta\varphi l + b_{21}\Delta\varphi^2 l + \ldots; \end{aligned} \tag{5}$$

Where:

a_{10}, a_{20}, ...,b_{01}, b_{11}, ... are coefficients calculated for the latitude of C point;

l – are differences between the latitude and longitude of P point and C point.

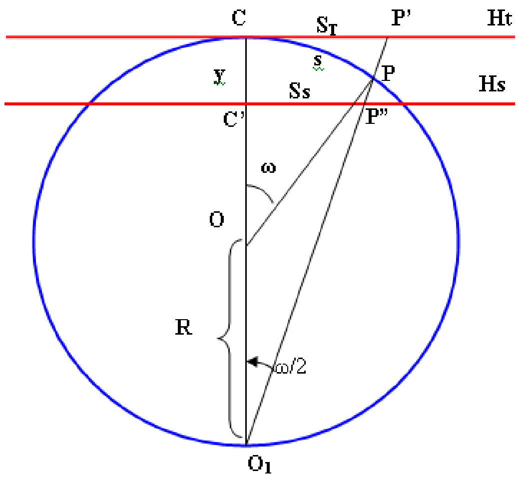

Fig. 4 Stereographic projection with tangent and secant plan

2.2.2 Distances deformations

From fig. 4 it can be written as follows:

$$S_T = 2R\,tg\frac{\omega}{2}; \qquad \text{is developed in Taylor series} \qquad (6)$$

By differencing it can result: $$dS = \left(1 + \frac{s^2}{4R^2}\right)ds; \qquad (7)$$

Unitary deformation $$\Delta = \frac{dS - ds}{ds} = \frac{s^2}{4R^2}. \qquad (8)$$

By comparing the mentioned relations it results that the unitary deformations in stereographic projection with tangent plan has values smaller than the deformations from cylindrical projection. So it is qualitatively superior to this.

For the stereographic projection with secant plan we have the following relations:

$$S_S = R\omega + \frac{(R\omega)^3}{12R^2} - \frac{R\omega}{2R}y = s + \frac{s^3}{12R^2} - \frac{s}{2R}y. \qquad (9)$$

Unitary deformation $$\Delta = \frac{dS - ds}{ds} = \frac{s^2}{4R^2} - \frac{y}{2R}. \qquad (10)$$

By comparing the mentioned relations we can notice that the unitary deformations in stereographic projection with secant plan are smaller with the quantity $\frac{y}{2R}$ over the ones from the stereographic projection with tangent plan.

The value of this deformation is established as follows:

In stereographic projection with tangent plan the maxim unitary deformations result for s = 275 Km, and the minim ones for s = 0.

So: for s = 0; $\Delta = 0$ and for s_{max} = 275 Km; $\Delta_{max} = \frac{1}{2000}$.

There are reduced with $\frac{\Delta_{max}}{2}$ the deformations from above and result: $\Delta_{max} = \frac{1}{4000}$

So: $\frac{y}{2R} = \frac{1}{4000}$ and y = 3.189 Km; (11)

There will be a circle of null deformations with the following ray:

$$\frac{s^2}{4R^2} - \frac{y}{2R} = 0; \qquad s = \sqrt{2Ry} = 201{,}718\,\text{Km}; \tag{12}$$

2.2.3 Transformation of the coordinates from tangent plan into secant plan

If for certain point situated on the reference surface, the coordinates in the two systems of projections we can write with x_t, y_t and x_s, y_s as follows:

$$\begin{aligned} X_S &= X_T - \frac{X_T}{4000}; Y_S = Y_T - \frac{Y_Y}{4000} \\ X_S &= (1 - \frac{1}{4000})X_T = KX_T; Y_S = (1 - \frac{1}{4000})Y_T = KY_T \\ K &= (1 - \frac{1}{4000}) = 0.999750 \end{aligned} \tag{13}$$

From the above mentioned equations it can also be made the inverse transformation as follows:

$$X_T = \frac{1}{K} X_S; Y_T = \frac{1}{K} Y_S; \frac{1}{K} = 1.000250063 \tag{14}$$

The linear local deformations depending on the distance over the central point of the Stereo 1970 projection are shown in the table 1 (R_0=66378956,681 m)

Table 1 *The distance over the central point of the Stereo 1970 projection*

S [km]	Region deformation [cm/km]	Relative deformation
0	-25	1/4000
1	-25	1/4000
10	-24.94	1/4010
20	-24.75	1/4040
30	-24.45	1/4090
---------	-------------------------	----------------------------
200	-0.42	1/38000
201.718	**0**	**-**
210	2.09	1/47800
220	4.74	1/21000
230	7.5	1/13000
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430	88.6	1/1100

The curve of regional (local) deformations on secant plan is shown in the following diagram (Fig. 5)

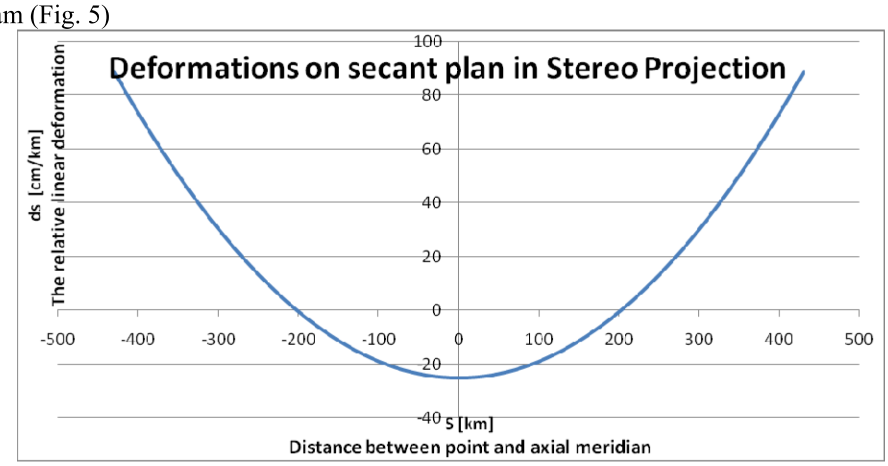

Fig. 5 Diagram of deformations in Stereo70 projection

2.3. UTM projection (Universal Transversal Mercator)

This projection is a variant of the *Gauss – Krüger* projection, used in United States of America and in other countries, being important also for Romania due to our integration in new political and military structure.

The cartographical representation is made on fuses of 6° longitude, into the interval delimited by the parallels of 80° latitude south and 84° latitude north. The reference ellipsoid is the international ellipsoid called *WGS – 84*(Fig. 3), for which:

Great semi ax: $a = 6378137{,}000\ m$

Geometrical flattening: $f = 1/298.257223563$

The projection cylinder (Fig. 4) is modified by reducing its dimensions and bringing it into secant with the ellipsoid along of 2 lines parallel with the central meridian. This means that in an area of 6 degrees there are 2 line of secant situated of almost 180 000 m E and V from the axial meridian. In order avoid to use the negative coordinates, to the central meridian it is attributed a false value of the east of 500 000 m E, this leading to the values of 320 000m E and respective 680 000 m V for the two secant lines.

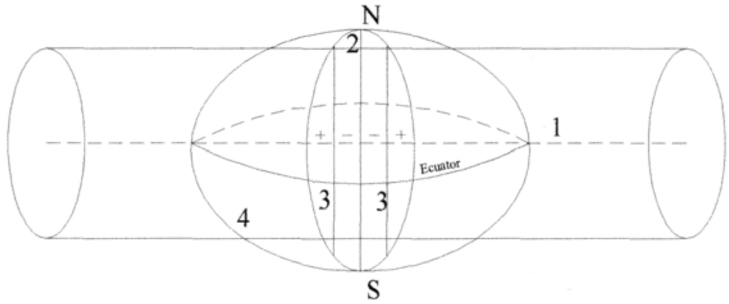

Fig. 6 – Projection cylinder

1 – ax of the cylinder situated in equatorial plan;
2 – axial meridian;
3 – secant meridians;
4 – margin meridian of the fuse of 6 ;
- flexion ray of the meridian ellipse of φ latitude
- meridian arc β between 2 parallels φ_1 and φ_2

Adopting the representing system on fuses of 6° longitude, the plan representation is almost fiddled. Using the Mercator projection which is conform projection because the angles and the module u of linear deformation are not deformed very much it leads to a precise representation of the earth. It disadvantages are that the calculation is very difficult but due to the modern technology of calculation this is not observing.

2.3.1 The deformations in UTM projection

The relative linear deformation has the following formulas**:**

$$D_{UTM} = k(D_{Gauss} + 1) - 1 = k(L^2 / 2R^2 + L^4 / 24R^4 + 1) - 1 \ \text{[cm/km]} \tag{15}$$

where:

- D_{UTM} relative linear deformation in UTM projection;
- D_{Gauss} relative linear deformation in Gauss projection;
- R is the average ray of flexion into the considered point;
- $y=(y-y_0)$ is distance between the point and axial meridian;
- k is the constant report between the distances from the UTM projection plan and the ones from the Gauss projection plan.

By using this formula for the relative linear deformation in UTM projection there are obtained values which are directly proportional with the distance against the axial meridian and they increase starting with the negative value -40 cm/km (Fig. 7)

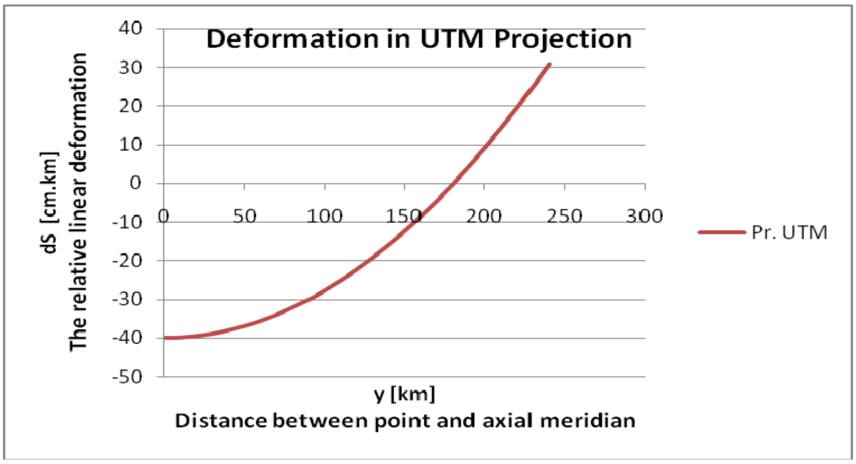

Fig. 7 The diagram of relative linear deformation in UTM projection

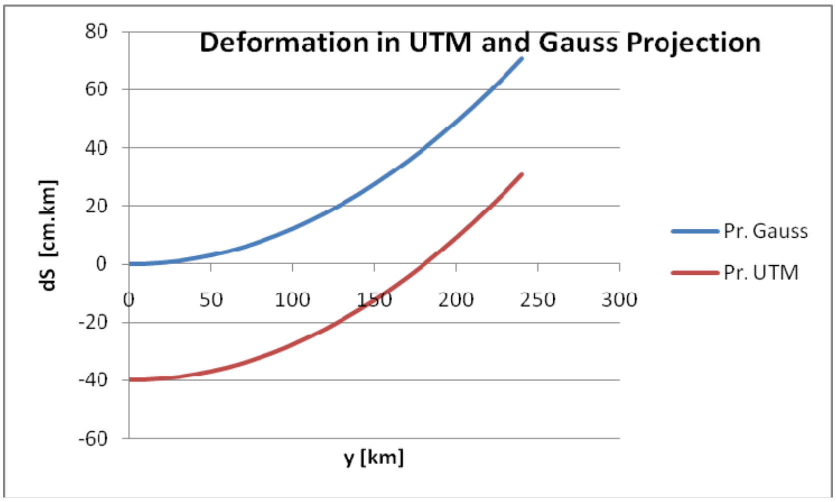

Fig. 8 Comparative Diagrams of the relative linear deformation in UTM and Gauss projection

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ASPECTS CONCERNING THE EVOLUTION OF THE SUBSIDENCE PHENOMENON AT LIVEZENI M.E.

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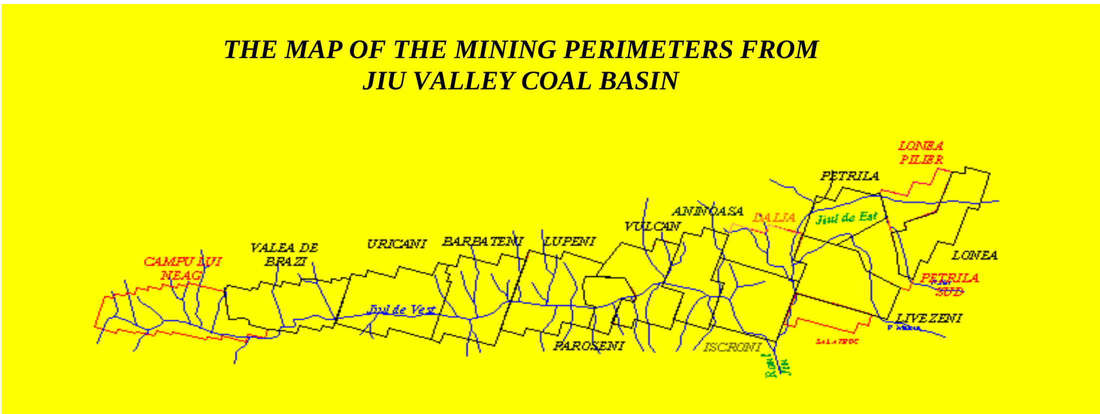
Abstract: *This paper presents the prognosis of the surface's deformation and movement phenomenon by using the influence functions method for Livezeni M.E. All these prognosis methods offer good results in the parameters which characterize the phenomenon are correctly chosen*.

Key words: *subsidence, deformation, movement, influence, method, parameters.*

1. A GENERAL PRESENTATION OF THE JIU VALLEY COAL DEPOSIT AND THE LIVEZENI MINING PERIMETER

The Jiu Valley coal deposit, situated in the S-V of the Hunedoara County has a surface of 163 km^2 and it's presented as a triangular depression.



The Jiu Valley coal basin has been divided in 16 mining perimeters of which 15 have had mining activity.

CNH Petrosani has negotiated and obtained concession licenses for exploiting in 8 perimeters (Lonea, Petrila, *Livezeni*, Vulcan, Paroseni, Lupeni, Barbateni and Uricani), these having smaller surfaces than the old mining perimeters, some of these areas being preserved and closed.

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In the nearby future, we expect the recession of the exploiting parameters, in the way of giving up at the areas proposed for preservation and closure from the current concessioned exploiting parameters and their deliverance to ANRM.

The perimeter of the Livezeni mine is situated in the eastern side of the Petrosani depression and it's sided by the Petrila and Petrila South perimeters at north, Salatruc at south, Dilja and Iscroni at west, Lonea at east.

Administratively, Livezeni mine's perimeter belongs to Petrosani city. The access in the mining perimeter is insured through electrified railway Filiasi – Targu Jiu – Petrosani – Simeria and the national road Targu Jiu – Petrosani – Simeria.

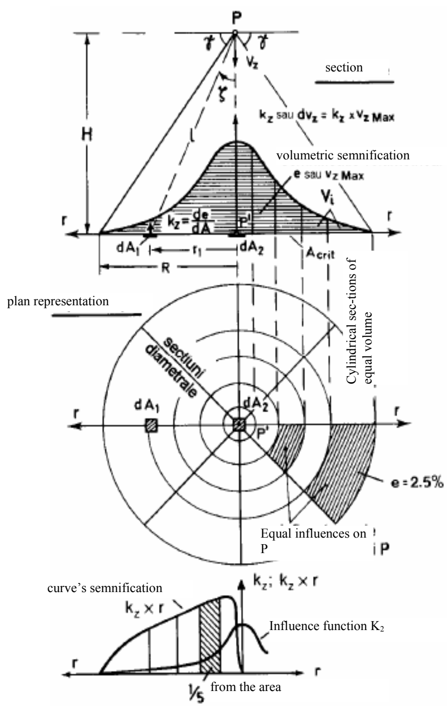
The relief of the Livezeni perimeter is hilly, with deep valleys and heights that reach 850 m, except the terrace areas and the Jiu meadow, with heights below 600 m.

By analyzing the possibility of implementing the prognosis methods of the subsidence phenomena in the Jiu Valley's conditions, we applied the influence functions method for the study case – Livezeni M.E.

2. THE INFLUENCE FUNCTIONS METHOD

The influence functions method appeared in 1932 and was elaborated by Bals.

It has the advantage of not being constrained by rectangular extracted areas, some parts of the extracted area can be evaluated separately and admit the hypothesis that a surface point is drawn by the exploited elementary voids with a force that is reverse proportional with distance's square.



The construction of theintegration network by dividing the influence graphic based on the equal contents and influences

The method is based on 7 principles:

- **the principle of** linearity
- **the principle of** equivalence and limit angle
- **the principle of** rotation's symmetry
- **the principle of** superposition
- **the principle of** transitivity
- **the principle of** volume's constancy
- **the principle of** reciprocity

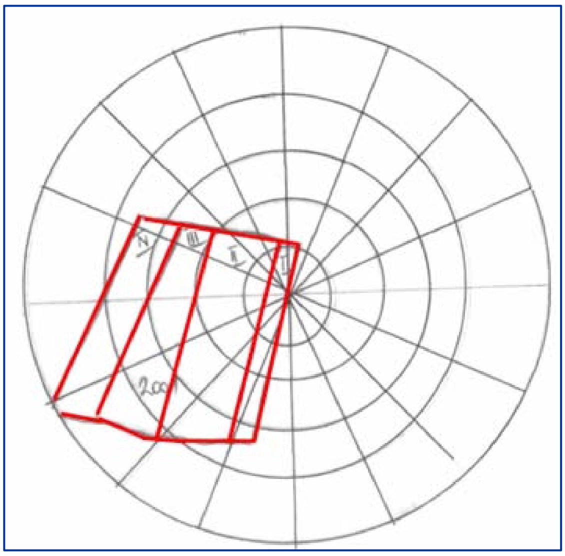
3. THE DEVELOPMENT AND VERIFICATION OF THE HYPOTHESIS CONCERNING THE LAND SURFACE'S MOVEMENT FROM LIVEZENI M.E. – THE MALEIA AREA, BY USING THE INFLUENCE FUNCTIONS METHOD

The Livezeni station has been materialized in April 1976 and has been formed from a transversal alignment with two stable heads, with a length of 852,89 m and 44 bench-marks. The observations have been effectuated until 1979 by the Mining Topography Cathedra and continued by I.C.P.M.C. Petrosani.

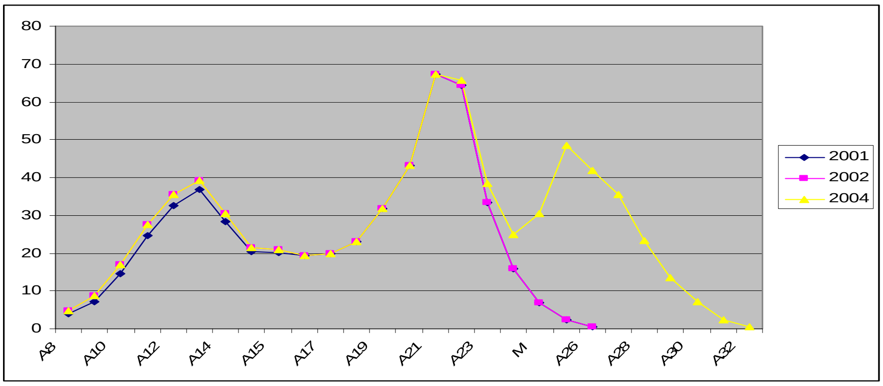
The Livezeni deposit has been sectioned in 9 tectonic blocks, numbered from E to W and from N to S. The exploitation of the third layer in the WIA sector will continue until 2012, when it's foreseen that the coal reserve that's been billeted in this block will be finished.

For the development and the verification of the hypothesis concerning the land surface's movement at Livezeni M.E. - Maleia area, by using the influence forces method, a integration network containing 5 sectors and 16 quadrants has been chosen, resulting a total of 80 surfaces of equal influence.

When building the network we have taken into consideration the medium values of the layers' depth and inclinations from M.E. Livezeni mining field, respectively of the influence angle resulted from the measurements, by using AUTOCAD.



Over the *surface – underground situation plan*, we proposed the integration scale (as in the above figure) in every point of the alignment, according to the Bals method for inclined layers



The graphical representation of exploitation's influence on the surface – Maleia area

The data obtained due to integration's scale overlap on the situation plan, in every point of the alignment of the topographical tracking unit Livezeni M.E. – Maleia area, have been processed with the help of the Excel application.

Next we present synthetically the data resulted from the processing.

In order to elaborate the submersion's prognosis method, we must:

- analyze the dynamics of the submersion's evolution in representative points;
- estimate two factors that are connected from the duration and the exploitation period of a panel, that is: *the response time, T1* – the duration from the beginning of a panel's exploitation and submersion speed's modification and *time constant, T2* – the duration from the beginning of the submersion until it's amortization.

We propose a *submersion's prognosis equation* for a point in the exploitation's influence point (for example, the A27 bench-mark):

$$S= K1 \exp(T/T0)\, D^{\beta} A^{\alpha} \qquad (1)$$

Where: S is the submersion, in mm;

K1, α and β – empirical constants;

T_0 – time constant, in trimesters;

D – the medium horizontal distance from the point considered to be the mass center of the exploited surface, in m;

A – The value of the exploited surface.

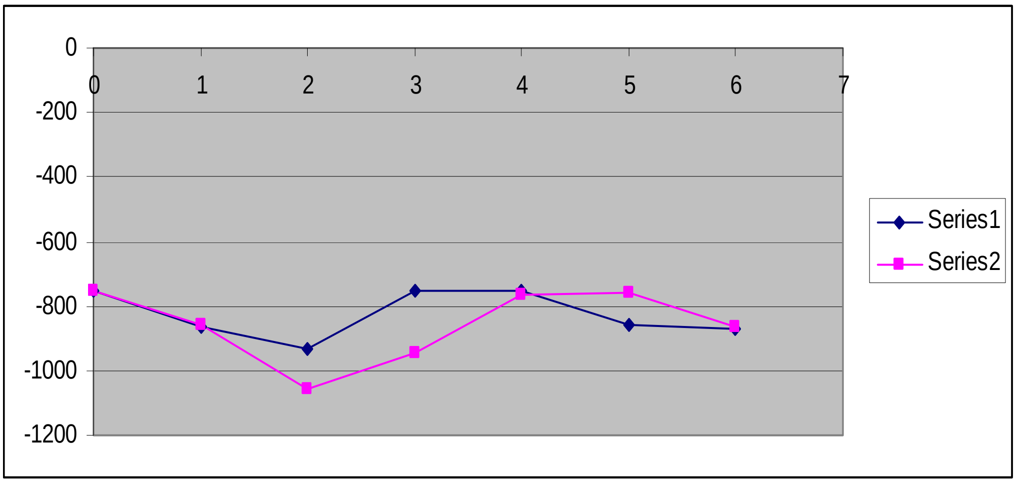
3. GENERAL CONCLUSIONS

❖ the whole mining activity produces, because of it's specific, multiple and various negative effects on the environment;

❖ the crumbling of the rocks that cover the underground excavations leads to the subsidence phenomenon, which has a negative influence that can extent to all covering rocks thickness, up to surface, and it manifests for a very long period of time, even after total ceasing of the productive activity in the area;

❖ currently, a series of prognosis relations are established worldwide, with the help of which one can determine, in a relative short time, the parameters of rock's movement under the influence of underground exploitation.

❖ these calculations aren't always applicable for different deposit conditions, and they must be continued in order to find the definitive solution for this problem.



The measured and prognozed values for the A27 bench-mark

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HORIZONTAL WELLS INSTALLED BY HORIZONTAL DIRECTIONAL DRILLING (HDD) AS AN ALTERNATIVE MINE DEWATERING TECHNOLOGY

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C. DREBENSTEDT*

Abstract: *Horizontal wells, installed by using the Horizontal Directional Drilling (HDD) – Technology potentially are more efficiently in dewatering loose rock aquifers within mining or building projects compared with the commonly used vertical wells. Before this kind of horizontal wells can be operated as a primarily used dewatering element, deficiencies in calculation methods and drilling technology need to be eliminated. Furthermore the evidences of economical and environmental advantageousness must be provided. Therefore laboratory tests are accomplished. The tests go along with numerical simulation to precalculate the tests, to reproduce the test sequences and to implement the improved calculation methods. The implementation of improved calculation methods is essential to get planning tools, which are exactly enough to presimulate a complex mine dewatering process.*

1. INTRODUCTION
1.1 General considerations

Many mining and underground engineering projects, which are realized in open pit mines and building pits, demand dewatering of aquifers to ensure safe and efficient excavation, transportation and dumping of loose rock material.

Today the state of the technology for dewatering projects in open pit mines and building pits are vertical wells. Especially in thin aquifers vertical wells have a short screen section and the active screen section is continuously reduced by dewatering progress. Vertical wells in average show a pumping rate up to 1 to 2 m³/min in maximum. To ensure technological necessary dewatering progress a great many of vertical wells need to be operated. All this wells are provided with pumps of lower capacity and therefore lower degree of efficiency, they all separately are provided with electrical equipment and penstock as well as with control and feedback control systems.

The horizontal well technology offers the possibility to dewater the aquifers (at least partly) by using the free gradient along the wells. The water is given to the open pit, to collect the water in sump drainages and pump it out of the pit with bigger pumps and higher degree of

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efficiency. Because horizontal wells show long screen sections it is considered to be possible to increase the pumping rate up to 5 m³/min per well and more. Therefore the goal of dewatering process should be obtained with fewer wells in operation in comparison with dewatering systems based on vertical wells (Figure 1). [1]

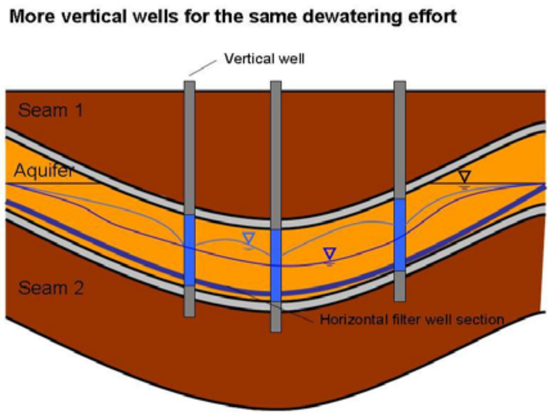

Figure 1. Comparison between vertical wells and horizontal well

A changeover in dewatering from vertical to horizontal wells as primarily used dewatering elements promises savings in the need for energy and material, it promises the ability to protect the mining field from impacts longer before it is actually excavated and it promises a better protection of the resource groundwater by performing a technological necessary dewatering focused on mine safety.

But there are expected not only hydrological advantages and advantages in the need for energy and material. There are also technological advantages for some cases. Figure 2 and figure 3 for example show the differences between vertical and horizontal wells by operating field wells. While the operation of vertical wells need to be interrupted, if the cutting face reaches the field wells, going along with an arising water level near the slope, horizontal wells with free discharge are working without interruption, ensuring a stable dewatering process.

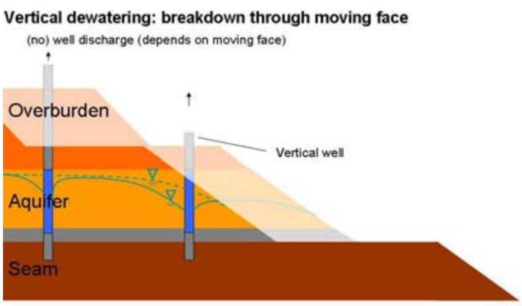

Figure 2. Operating interruption of vertical field wells

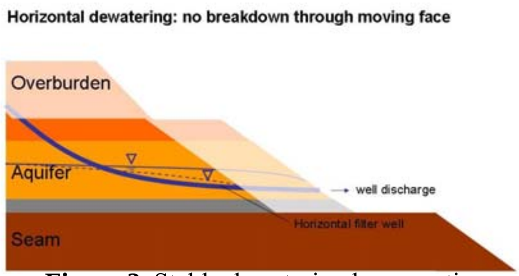

Figure 3. Stable dewatering by operating horizontal wells as field wells with free discharge

1.2 Present problems in HDD – well operations

The HDD - technology is a widely used system for trenchless installation of product pipes. [2],[3] The general use of horizontal wells made by HDD also is approved as far as it is a problem of land reclamation or of ensuring geotechnical conditions in slopes through draining or even if it is a problem of water recovery in hydro-geological optimal structures.

But several unanswered questions avoid a utilization of HDD-wells for use as a basic dewatering element in open pit mines and building pits.

- unsatisfying methods for numerical modelling of horizontal wells with incomplete flow to well and insteady and insymmetric flow to a zylindic hole
- the ability to ensure vertical dewatering with horizontal elements in heterogeneously sedimented aquifers in order to lower the water table
- choice of drilling methods, drilling fluids and methods of bore hole treatment in order to minimize costs and ensure dewatering effort having great vertical depth, long filter sections with low inclination, rhythmic stratification of aquicludes and aquifers leading to problems with cuttings removal
- the need for coordination of dewatering technology with mining technology

Exemplified can be the mentioned questions with a project in the German lignite mine “Vereinigtes Schleenhain” of MIBRAG mbH in 2002. There a horizontal well was drilled and installed by using HDD (Figure 4) to bring out the residual flow, which was left over by vertical dewatering, out of a small scale dell in a sandy aquifer.

Especially each time after the bucket wheel excavator, which operated on that level, just had completed a block, problematically high volumes of water flew out of the slope, causing instable slope, wet plane, and erosion and lowering in excavator efficiency because of sludgy material. The horizontal well, which was installed to solve the problems, showed the attributes like described in Table 1.

The analysis of construction details and dewatering effects of this well [5] and additionally made literature and patent enquiries bared unsolved problems with horizontal wells drilled by HDD relating to calculation methods and drilling technology.

Especially problems with the removal of cuttings from bore holes with great vertical depth and long horizontal filter sections, which additionally intersect different layers of clay, sand, gravel and coal, turned out to be the worst problem due to drilling technology. Cuttings filled the annulus space and showed a lower conductivity then the surrounding aquifer material (Figure 5). [4]

Table 1. Technical data of the HDD-well in the open pit mine “Vereinigtes Schleenhain”

total length	605 m
length of effective screen	360 m
maximum vertical depht	60 m
well diameter	100 mm
screen type:	slotted PEHD – screen

Figure 4. Mine section dewatered by HDD-well and drill rig at open pit mine „Vereinigtes Schleenhain“

Figure 5. Borehole filling with material of low conductivity caused by problems to remove the cuttings

2 METHODS

A number of fluidic basic conditions by using HDD – wells under mining conditions are solved unsatisfactory and therefore calculations and numeric simulations are insufficient exactly.

A horizontal well used for dewatering under mining conditions generally shows the following attributes:

(1) filter screen is located next to the bottom of the aquifer

(2) with continuity of dewatering the ground water level decreases (theoretically no steady state will be obtained)

(3) during the final stage of dewatering lowering of ground water table next to filter screen down to filter screen level, which leads to incomplete incident flow

(4) with continuity of dewatering the effective filter screen length decreases at inclined horizontal wells

(5) changeover from false water table to free water level

The horizontal well equations, which commonly are used for calculation and which are also implemented in numerical software consider the attributes mentioned above unsatisfactory. To test their adaptability to HDD-wells in mine dewatering, different known calculation methods has been discussed on practical examples [5]. The comparison showed that all the discussed methods can not be easily applied to HDD-dewatering wells and further researches on that issue should be done.

This resulted in the preparation of laboratory tests, which are run currently. Therefore a big laboratory test station at the Institute for mining at TU Bergakademie Freiberg is operated at the moment. The test station got the size 6 m x 6 m x 2.5 m (length x width x height). The simulated aquifer in the test station has a volume of approximately 70 m³. Three sides of the test station are equipped with cells to define border conditions in the line to choose different water levels. Towards the forth side a horizontal filter screen dewaters the test station. The water level is monitored by 46 observation wells, which are equipped with readings recorder. The measuring of the flow rate of the water flow towards the border condition cells is done by 2 electromagnetic flowmeters.

The parameters to vary in the several tests are:

(1) the model material (2 different sandy materials)

(2) the type of filter screen and filter screen diameter (based on commercially available material)

(3) annulus space fillings, inclination of filter well and location of well above ground

(4) water level of the border condition cells, the effective filter length

Below the test station is shown (Figure 6).

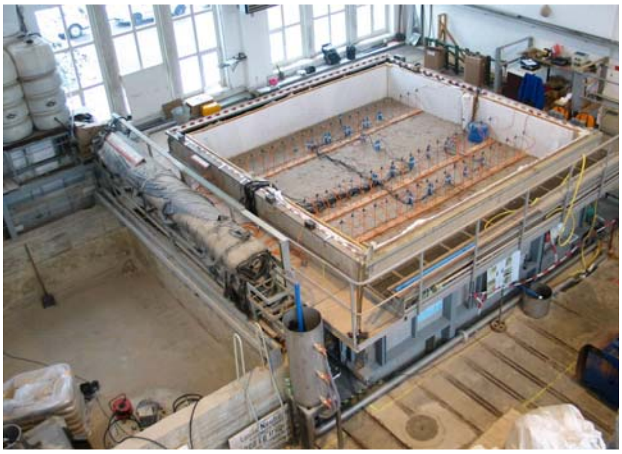

Figure 6. The laboratory test station in bird view

The laboratory tests as well as the field tests go along with PCGEOFIM® numerical simulations. The numerical simulation was used for planning and pre-calculating the laboratory tests and to help choose the best suitable test equipment [6], [9]. Now it is used for proof the test results and developing improved calculation basics.
Below the general layout of the finite volume element model is illustrated (Figure 7).

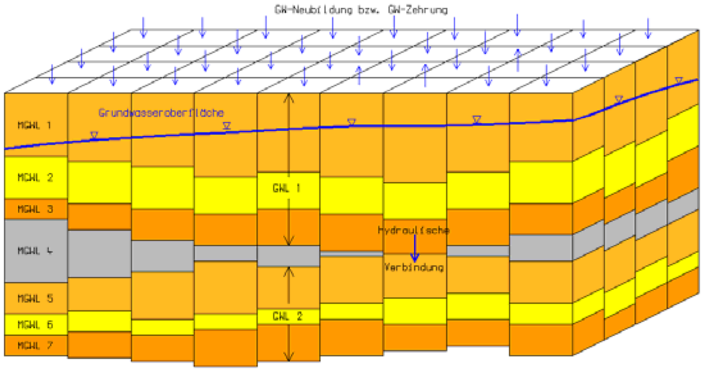

Figure 7. General layout of a PCGEOFIM® - model

To ensure, the test results, which are obtained in the laboratory test are valid and reliable, an extensive pre-test was made. The pre-test period included 17 single tests.

Because the necessity to achieve the same test results by repeating one test, always the same boundary conditions were adjusted. The only variation was in the test running technology changing the time between two tests and the technology to restore the water level after ending the one and before beginning the next test. That was important, because the biggest problem was to ensure a constant and homogeneous saturation before each test. Comparing the first 10 tests (pre-test period 1) for example the cumulated net flow rate differed up to 500 l in summary (120% to the average) what could be reduced to about 80 l (< 10 % to the average) comparing the last 5 tests (pre-test periods 2a, 2b) like seen on Figure 8.

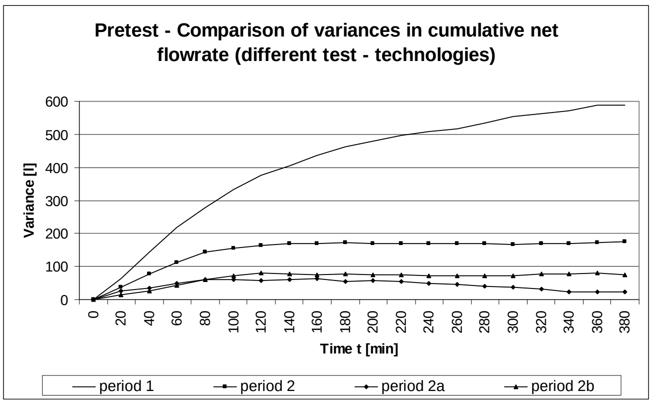

Figure 8. Pre-test – Comparison of cumulative net flow rate

During the pre-test period geo-electrical measurements in the test station helped to assess the saturation and the homogeneity of the model material in the test station.

The pre-test resulted in the determination of parameters for the test running technology such as test runtime, saturation time, saturation technology, measurement technology for which the reliability, validity and accuracy of the tests are as well as possible. [8]

Figure 9. Slug and Bail tests

The necessary material parameters like compactness of packing, pore content and others are achieved by taking samples and doing the standard tests. The conductivity is achieved by doing slug and bail tests on each single observation well in the test station (Figure 9).

At this time about 30 of 100 single tests are finished.

3 CONCLUSION

A dewatering system based on horizontal wells, which are installed by HDD promises savings in the need for energy and material, compared with vertical wells, especially in thin aquifers.

Problems with the installation of those horizontal wells under mining conditions and deficiencies in calculation methods resulted in laboratory tests running at the moment. The tests shall clarify and verify improved calculation methods for horizontal wells, which are operated under mining conditions. The results will be integrated in numerical simulations. Thereby the laboratory tests take place in a 6 m x 6 m x 2.5 m test station. The most important parameters, which are influencing horizontal wells, will be varied. The laboratory tests are planned to be finished in spring 2009, and results can be expected by then.

Shortly also field tests will start in a German lignite mine. The results of both methods will be compared to be able to adapt the calculation basics obtained from laboratory tests running under defined conditions to in situ conditions.

By using improved planning tools, which will be available after the tests ended successfully, a dewatering planning will be made and the economical and environmental effects will be evaluated.

4. ACKNOWLEDGEMENTS

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THE SEISMIC EFFECT OF QUARRY BLASTING OPERATIONS ON UNDERGROUND MINE WORKINGS IN THE CASE OF SIMULTANEOUS SURFACE-UNDERGROUND MINING

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Abstract: *Blasting operations seismology has a great deal of importance if we consider their seismic effect on slope stability in quarries and underground mine workings if simultaneous surface-underground mining is done. That is why the present paper deals almost exclusively with the reduction techniques of this effect and the methods employed for seismic control of blasting generated vibrations. In the paper, several issues are synthesized which are leading to blasting operation optimization and slope stability anti-seismic design improvement, in order to achieve maximal breaking effect without any adverse consequences.*

This issue rises for simultaneous mining operations in surface (quarries) and underground, when a safety floor should be designed, together with the maximum allowable explosive charge in surface operations, not affecting the underground mine workings.

The means to solve the problem are those recommended in literature, using the two relationships below:

$$v = \frac{37{,}5\left(v_P^2 - \frac{4}{3}v_S^2\right)(\eta^{8/8} - 1)}{V_{P^\eta}} \tag{1}$$

$$Q = \left[\frac{vp(r_{oriz} + h\sqrt{h})}{70}\right]^3, \tag{2}$$

Where: $\eta = 1 + (1 - 2v)\varepsilon$; v – oscillation velocity (particle velocity), cm/s;

V_P şi V_S – P and S waves propagation velocities, m/s;

v - Poisson's coefficient;

a - deformation;

Q – explosive charge, kg TNT;

r_{oriz} – horizontal distance between the blasting point and the protected objective;

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h – the level difference between the blasting point and the measurement station, or between the blasting point and the protected objective, or:

h – floor's width;

p – coefficient depending on the blasting treads number, n, as it follows:

$$p = \frac{n}{0,445n + 0,45} \tag{3}$$

Obviously, for instantaneous blasting, n = l.

First of all, for the studied mining exploitation, a practical check-up of the following relationship validity is done:

$$v = \frac{70\sqrt[3]{Q}}{p(r_{oriz} + h\sqrt{h})} \tag{4}$$

In order to do this it is required to gather underground registration data for several (3-5) blasts operated in the quarry, with relatively small charges (assessed by formula (4)), generating no danger at all. Maximum oscillation velocities v is measured from registered seismograms, being then compared to the velocity values determined employing formula (4). If gaps between observed and computed values are not significant, relationship (4) can be further applied to compute the maximal allowable charge Q corresponding maximal allowable vibration level (expressed by particle's velocity). Maximal allowable deformation levels (expressed through ε rock deformation) as a function of protected mining structure's class are highlighted in table 1.

The related maximal allowable vibration levels, expressed by particle's velocity, v, is determined employing relationship (1), which requires the knowledge of velocity values V_P and V_S. These two values can be assessed by measurements of estimated according to the rock type specific for the open-pit and underground mine workings. Within this scope, table 2 depicts the maximal allowable levels for vibration, expressed by particle velocity, v, as a function of mine structure's class and the rock type.

Using relationship (4) the maximal allowable charge can be determined for any values of r_{oriz}, h and n. It is strongly recommended that the results to be represented as by-logarithm graphs. Also, it is recommended to consider the worst-case scenario, when open-cast mining is done/ would be done exactly above the underground mining area (r_{oriz} = 0), namely:

$$Q = \left[\frac{vph\sqrt{h}}{70}\right]^3 \tag{5}$$

Of course, in by-logarithm co-ordinates (log h and log Q) equation (5) represents, for a certain vibration level, a straight line with n (or p) as parameters, allowing the maximal allowable charge determination for a given situation. These graphical representations or equation (5) written as it follows:

$$h = \left(\frac{70\sqrt[3]{Q}}{pv}\right)^{2/8} \tag{6}$$

are also facilitating the computation of minimal allowable floor width h (corresponding to maximal allowable vibration level, v), for any charge Q and any number of delay treads, n. It is recommended to consider the minimal allowable floor thickness value which corresponds to the worst-case scenario (meaning that besides the $r_{oriz} = 0$ condition, the n=1 condition is added).

Table 1 *Maximal rock deformation allowable levels*

Mining structure's class	Working's description and utilization span	Maximal allowable deformation
I	Very important mine structures, having long life cycle span (over 10-15 years) such as: hydro-technical tunnels, main shafts and entries, underground chambers.	0,0001
II	Important mining structures, having a life cycle span of over 10-15 years: channels and transportation tunnels, hydro-technical structures, safety pillars, cross-sectional galleries, boundary quarry benches.	0,0002
III	Short life cycle span mine structures (1-5) years: rooms, entries, headings, benches.	0,0003
IV	Lower importance mine workings, with less than 1 year life span: blocks (panels), faces, quarry's working bench etc	0,0005

Table2 *Maximal allowable levels for vibration expressed by particle velocity*

Rock type	Strength coefficient, f	V_P [km/s]	Particle's maximal velocity allowable , v, for each mine			
			I	II	III	IV
Sedimentary, detrital, fine and alluvial	0,5 - 1	1 - 2	4,1	8,2	12,2	20,4
Sedimentary cracked, clay and high porousity	1 - 3	2 - 3	6,8	13,6	20,3	34,0
Rocky, with pronounced natural fissuration	3 - 5	3 - 4	9,5	19,0	28,4	47,3
Quite monolithic, less fractured	5 - 9	4 - 5	12,2	24,4	36,7	60.0
Monolithic, very low level of fissuration	9 - 14	5 - 6	14,9	29,8	44,6	74,5
Very strong and monolithic, no fissuration	14 - 20	6 - 7	17,8	35,6	53,3	89,0

Another alternative to solve this problem consists in a series of operations based on the so-called reduced distance method. Based on registered data the constant values H, α and β are computed from the propagation law. If α is adopted (α = 1/3; α = 1/2 etc.), significant simplification of the problem is obtained, while it reduces to determination of H it β constant values. In our case, r expresses the distances from blasting point within the quarry to the underground measuring stations, or between the blasting point and the underground located protected mining structure. In the case when α = 1/3, (the situation of combined surface and underground mine operations) there were determined experimentally, both for the particle's velocity v and for particle's acceleration a.

After the propagation law is established, certain equations are used in order to compute the maximal allowable charge for any distance r between the blasting point and the protected mining structure, or to determine the minimal allowable distance between the blasting point and the protected objective, for any charge value Q. Basically, the procedure is the same, not depending on α values (α = l/2; α = l/3 etc.).

Again it is recommended to consider the worst –case scenario, when the distance between she blasting point and the protected structure exactly equals the floors width, and the charge Q corresponds to instantaneous blasting. Obviously, for this second way it is also required the knowledge of vibration maximal allowable levels. Those presented in table 2 are rather appropriate for rough estimations. Because the issue concerning the floors thickness in the considered case is very important , and the vibration maximal allowable levels given in the literature are very different and corresponds to different conditions, it is more suitable to determine for each location the maximal allowable vibration level. Within this goal, seismic registration should be done (in specific underground locations) for blasting operations carried out in the quarry. It should be started with small charges, and from a blasting to another the chare is gradually increased. Before starting the experimental field tests, the underground rock fractures are covered with glass pieces, employing a resin. After each blast the deterioration degree is analyzed, in correlation with the effects exerted on the underground rocks As a function of detected deterioration degree and correlated with registered oscillation velocities (if needed, seismic movement acceleration is also registered), the maximal allowable vibration level (expressed by velocity v and/or acceleration a) can be assessed. The seismic registrations done in this purpose are also used to determine the propagation law, as shown before. It is suggested that the underground seismic measurements to be carried out in at least 3-4 points, on all the three directions (radial, transversal and vertical). When selecting the registration locations, consideration should be given to the fact that seismic vibration level depends, beyond the well-known factors, of the mine working's shape, position and direction

As a consequence of blasting operations in quarries, the strength of rocks within the floor can reduce. That is why, it is recommended that values obtained by any of the procedures described above, for floor minimal allowable thickness and maximal allowable charge, to be controlled at least twice using seismic measurements, basically when, as an effect of quarry's exploitation depth increase, the floor thickness became higher with 50% and respectively 25% than the limit value thickness, obtained by initial calculus.

Seismic effect of blasting operations on slopes

The blasting operations performed nearby stable slopes should be done very carefully and in a controlled manner, so that the seismic effect does not affect their integrity.

Table 3 *Maximal allowable vibration levels for slopes*

Slope's life cycle span	**Oscillation velocity v [cm/s] for the following values of the strength coefficient, f, of blasted rock**					
	1	**1 - 3**	**3 - 5**	**5 - 9**	**9 - 14**	**14 - 20**
> 5 ani	8,2	13,6	19,0	24,4	29,8	35,6
<5 ani	12,2	20,3	28,4	36,7	44,6	53,3

The maximal allowable vibration level depends upon the blasted rock strength and slope's life cycle span (table 3). The link between velocity v [cm/s], total explosive charge, Q

[kg], blasted block length, L [m], number of delay stages, n, and r [m] distance from blasted block limit to the protected slope is given by the relationship below:

$$v = \frac{K_{tal}}{\sqrt[3]{n}} \sqrt{\frac{Q}{L}} \frac{e^{-0,08r}}{r} \tag{7}$$

Relationship (7) is valid for r<70 m. The constant K_{tal}, depends on rocks type within the slope and strength of blasted rock. (Table 4)

For any vibration limit and, implicitly, for any maximal allowable vibration level (Table 3), the corresponding value Q/L can be calculated for each blast, using relation (7), written as below:

$$\frac{Q}{L} = \left(\frac{vre^{0,08r} \sqrt[3]{n}}{K_{tal}} \right)^2 \tag{8}$$

Consequently, prior to blasting it should be seen if the Q/L ratio to be applied is not exceeding the Q/L ratio obtained with relationship (8). If it is higher, the explosive charge Q should be adequately reduced.

Table 4 *Values for K_{tal} constant in relationship (7)*

Rock type within the slope	K_{tal} constant for the following values of blasted rock's strength f		
	f≥14	6≤f<M	f<6
Rocky; f≥6	300 - 310	390 - 415	560 - 585
Semi-rocky; f<6	310 - 315	415 - 425	585 - 620

Basically, relationship (8) enables the total maximal allowable charge determination for a given situation and, obviously, for a given maximal allowable vibration level (Table 3).

For a given Q total charge, equation (8) also can serve to compute the blasted block's minimal allowable length corresponding to the given maximal allowable vibration level (table3). Computations should be carried out either for every blasting operation, or for the worst-case scenario. (See relation 8).

When higher explosive charges are required at lower distances from slopes, several measures can be applied in order to reduce the seismic effect, among them the most effective being outlined as it follows:

- blasted faces can be re-directed so that the stable slope line should be parallel with the blasted block's mean line. This way, the seismic activity at the blasted block's margins diminishes and, consequently, the seismic effect decreases on the entire operated and mined bench;

- achieving a prior fissuration (a seismic shield) at the stable slope side. The shield is developed by detonation of holes from one or two rows basted using the same delay time. After 70-100 milliseconds', the explosive charges located in other boreholes are ignited. The parameters or those specific for common blasting operations. The charges should be discontinuous ones, and the gap between the shield and slope's margin ranges from 0 to 8m.

Massif rock blasting operations (both carried in underground and in quarries located at surface above underground mines) can also represent a factor of rock bursts and mine stroke generation. The techniques presented before for reduction and control of blasting operations effects on mine structures can also be applied, to a certain extent, the rock bursts initiated by blasting. Apparently seen as a paradox, this triggering role together with the rock fracturing effect, are transforming the underground blasting operations (performed in a well thought and designed system) one of the most effective techniques in rock bursts prevention.

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METHODS AND TECHNOLOGICAL IMPROVEMENTS FOR THE EFFICIENT REMOVAL OF THE OVERBURDEN HARD ROCK FORMATIONS AT SOUTH FIELD LIGNITE MINE, PTOLEMAIS, GREECE

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Abstract: *Greece, mining 65-70 Mt annually, is the second lignite producer in the EU, fifth largest in the world. South Field Mine (SFM) is the biggest mine in Greece with an overall production exceeding 20 Mt per year. The SFM can be considered unique regarding the mining conditions and the methods used to remove the overburden material. Approximately 30% of the overburden formations consist of hard and semihard material that cannot be excavated by BWE continuous mining system. Therefore, conventional mining methods (drilling, blasting and shovel-truck operations) need to be applied for the overburden removal, in parallel to continuous mining operations. The combination of the two mining systems at the SFM could be considered as a non-conventional process in open pit mining. Significant achievements and large experience, which have been accumulated during many years of mining operations for hard formation removal, place SFM in a leading position worldwide in terms of development and utilization of both conventional and continuous excavation techniques. This paper presents the methods, technological improvements and achievements for the efficient removal of the overburden hard formation at the SFM.*

1. INTRODUCTION

Lignite in Greece is the main significant domestic energy resource and accounts for approximately 58% of total electricity produced in the Greek power generation system. Greece, mining 65-70Mt annually, is the second lignite producer in the EU, fifth largest in the world.

Lignite stays as the major and main pillar of the electricity generation for Public Power Corporation SA (PPC), which is the dominant lignite producer in Greece. Approximately 96% of the lignite used to supply the lignite fired power plants is mined by PPC.

PPC has more than 50 year's in house experience in exploration and open-cast mining. At present PPC operates five open-cast lignite mines. The four mines (Main Field, Kardia Field,

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South Field and Amynteon – Florina Fields) constitute the Lignite Centre of Western Macedonia (Northern Greece) with an annual production of about 55 Mt, while the fifth mine in Peloponese region under the name of Lignite Centre of Megalopolis produces approximately 14Mt per year. Currently, Greece's total generation capacity amounts to 12,695MW, of which 5,288MW is lignite capacity (Kavouridis, 2008).

Lignite is mainly extracted by continuous operation of bucket wheel excavators, belt conveyors and spreaders. In addition to the continuous mining method, conventional mining equipment including off-highway trucks, front-end loaders, electric rope and hydraulic shovels and dozers are utilized for mining the hard and semi-hard rock formations, which are encountered in the overburden strata (Pereti, 1982). The combination of the two above-mentioned systems for removing the overburden formations could be considered as a non-conventional process in open pit mining. Hard formations are mainly found in the overburden strata of SFM, where 30% of overburden consists of hard and semi-hard material.

SFM is the biggest mine in Greece with an overall annual production exceeding 20Mt, covering an area of 24 km^2. The original lignite deposit of SFM was estimated at 1.2 billion tones of lignite with an average stripping ratio of 4.96/FM^3 of waste material per ton of lignite. Mine opening works started in August 1979 and completed at the end of 1988. The opening of the mine lasted unusually long due to extraordinary hard rock formations found in overburden, which caused serious delays to the BWE operation. From the exploration data, the location and the quantity of hard rock formations have been underestimated. Currently SFM operates on ten benches, five for overburden strata and five for lignite seams. In each corresponding bench one BWE is located. Sixty five belt conveyors with a total length in excess of 100km are installed in SFM. There are three spreaders in the outer dump and three in the inner dump and also a pair a stacker and reclaimer operating at the lignite bunker of the mine. Figure 1 shows the present mining position and configuration of SFM. Mining depth is expected to reach 200m.

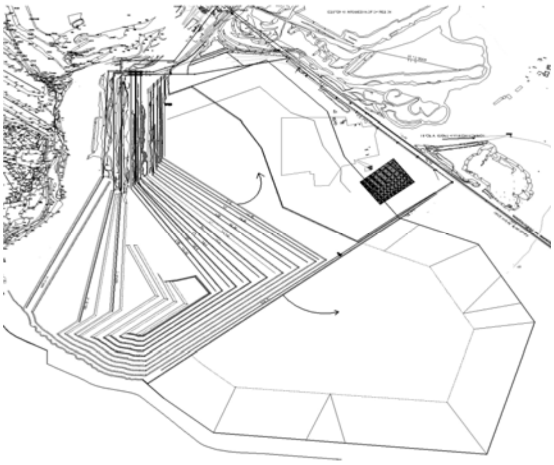

Figure 1. Present position of South Field Mine excavations and backfilling areas.

From the beginning of the mining operations until the end of 2006, 1592Mm^3 of total earth material was removed and 330,2Mt of lignite was produced. For the next five years the scheduled annual lignite production at the SFM ranges from 18-22 Mt and the volume of the total annual excavation targeted to the level of 100-110 Mm^3. The overburden removal ranges from 47-53Mm^3 out of which 20-25 Mm3 are hard and semi-hard formations.

The personnel involved directly in SFM's activities is 1,400 employees. Among them 280 work for the hard rock formations removal. The SFM can be considered unique regarding the mining conditions and the methods used to exploit the lignite deposit.

Significant achievements and large experience, which has been gained during many years of mining operations for hard formations removal place SFM in the leading position worldwide in the development and utilization of both conventional and continuous excavation techniques.

2. DESCRIPTION OF OVERBURDEN

The overburden material consists mainly of clay, sand, mixed formations of sand and gravels, marl and layers of semi-hard to hard rocks (sandstones, marls and conglomerates) (Kavouridis & Agioutantis 2006). The average specific weight of the overburden is 19.62 kN/m^3 (2 tn/m^3) and the bulking factor ranges from 1.4 to 1.5. The average thickness of overburden material is 90m.

The series of the overburden rocks of the SFM can be distinguished according to the rock coloration and way of formation into the following groups:

– Red to brown clastic sediments: This group consists mainly of clays, conglomerates and limestone gravels. The thickness of the whole formation is about 25m, while the average thickness of hard material is approximately 10m.

– Gray to yellow clastic sediments: This group consists of clay, sandstones, sand, conglomerates and siltstones. The average thickness of the whole formation is 25m, while the average thickness of the hard material is 11m.

– Green-grey clay, sand and silt sediments: The thickness of this formation varies from 25m to 50m. No hard material has been found in this formation.

The extent and the distribution of hard and semi-hard formations in the overburden strata is presented in the geological section of Figure 2.

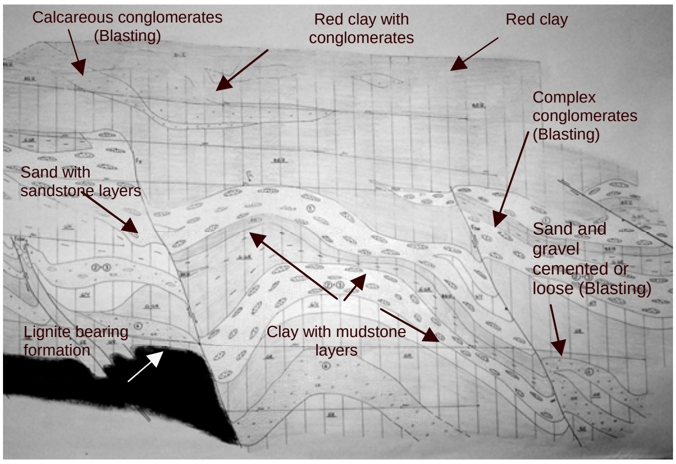

Figure 2. Extent and the distribution of hard and semi-hard formations in the overburden strata of South Field Mine.

Table 1 (Papageorgiou & Pakas 1997) summarises the range of the mechanical and physical properties of the hard rock formations, while Table 2 presents the approximate extent of each geological formation at the SFM.

From the data of Table 2 it is calculated that approximately 25-30% of the overburden consists of hard and semi-hard formations which cannot be excavated by BWE. Therefore, conventional mining methods must be applied (drilling-blasting and shovel-trucks operations).

Table 1. *Range of mechanical and physical properties of the hard rock formations.*

Parameter	Range
Uniaxial compressive strength (MPa)	15-143
Tensile strength (MPa)	2.4-11.2
Density (kN/m^3)	23.5-26.5
Density (tn/m^3)	2.4-2.7
Bulking Factor	1.4-1.5

Table 2. *Extent and dimensions of each overburden formation.*

Rock type	Rock description	Area (km^2)	Average thickness (m)	Volume (Mm^3)	Percent (%)
Hard material	Conglomerates, sandstone	24	11	264	13.0
	Conglomerates, breccia	10.5	10	105	5.2
Semi-hard material	Clay, sandstone, gravel	10.5	10	105	5.2
Loose material	Clay	24	61	1,464	72.3
	Sand, gravel	22	4	88	4.3
Total		24		2,026	100.0

It has to be pointed out that the hard rock formations calculated from the drilling exploration data were much less than the actual quantity due to the significant loss of coring (58%) in the overburden strata during the initial drilling research work. That explains why the initially selected methods and equipment for overburden removal were unable to face the actual demand creating serious delays in the opening and development of SFM.

3. OVERBURDEN REMOVAL OPERATIONS – METHODS FOR REMOVING HARD ROCKS

After one year of mine operation it was clear that the extent of hard and semi-hard rock formations is much larger than the expected based on the results of the deposit exploration study. Hard rocks caused significant problems to the operation and efficiency of the BWEs.

The first method that was applied for solving the hard rock removal problems was the use of bulldozers for ripping, the use of blasting, where necessary, and finally the loading of hard rocks on belt conveyors using BWEs. However, the transport of hard rocks with continuous mining systems caused significant failures to the cutting edges of BWEs as well as reduction of their operating time and efficiency. As a consequence, the method was considered as inadequate and was stopped after one year of operation.

For determining the optimal solution for the removal of hard rocks and other overburden material, a more precise knowledge of the extent and distribution of hard rocks in the entire deposit was necessary. For this purpose, additional boreholes for overburden sampling were drilled using a mesh of 200mx200m. Furthermore, additional data deriving from the inspection of the open excavation faces of the mine benches was taken into consideration in order to review all information regarding hard rocks. Based on the results of this process, the enormous larger extent of hard rock formations was confirmed. This fact led to the following modifications in the development study and operational plans of the mine as far as the overburden transport is concerned:

– Splitting the overburden strata into five benches of relatively lower height; development of four benches of 14m and one of 28m, instead of three benches of 28m, which was proposed in the initial study. This development simplified the positioning of hard rock formations and increased the efficiency of the mine operation. In order to realize the new development plans, the mine was necessary to purchase and install two new BWEs and the corresponding belt conveyors, which was not included in the initial mining study.

– Purchase of diesel-engine earth moving equipment of non-continuous operation for the haulage of hard rocks independently from the continuous mining systems. The later would be responsible only for the excavation and transport of loose rock formations. The initial fleet of diesel engine equipment was purchased gradually from 1981 to 1985. This machinery was of lower capacity compared to the trucks and excavators being in operation today.

Specifically, the diesel engine equipment at that time consisted of:

- Drilling rigs of 3'' diameter (TOYO KOYO, SULLAIR)
- Bulldozers D9 (Komatsu 355)
- Loaders (Komatsu D155)
- Wheel-loaders (Terex 72-71B)
- Hydraulic shovels (Liebherr 982 HD)
- Trucks of 40 tons payload (BELAZ)

The above mentioned equipment, combined with additional auxiliary machinery (graders, shovels, trucks, vibrators, etc), was in operation during the mine opening phase, achieving annual earth moving rates of 2-2.5 Mm^3 when the average overburden excavation annual rate at that period (1981-85) was 18.2 Mm^3. From the early stages of implementation of this method, it was obvious that equipment of higher capacity is required in order to maximise the efficiency of BWEs systems that were excavating the loose rocks of the overburden strata. In this period the efficiency of BWEs was kept to very low levels due to the delays cause from the hard rocks removal worksites that were operating in front of the BWEs. The BWEs achieved in this period annual excavation rates varying form 3.9-5.8 Nm^3 when the mining study had predicted rates of 7.0 Mm^3.

Based on these operational results and the experience gained during the mine development period, it was decided the purchase of modern, high capacity earth moving equipment.

– While the above mentioned modifications were in progress, the use of a shovel – crusher system was also tested (Rheinbraun Consulting GmbH, 1981). The hard rocks were excavated, crushed in–situ and loaded on the belt conveyors together with the loose rocks excavated by the BWEs. In the period 1988-89 two systems of shovel and crusher were installed, each one with a capacity of 2.5 Mm^3/yr (shovels P&H 2100 BLE with bucket capacity of 15m^3 and crushers with double rolls (KSK Strojexport, CZ) with a capacity of 2,200 t/h.

The annual production rates of these two systems in the period 1990-1997 (period of operation in full-scale) varied from 0.65-2.20 Mm^3 when the targeted production was 5.0 Mm^3. In spite of the efforts for upgrading this operating scheme it was finally proven inefficient and expensive for the site-specific conditions of SFM (i.e. type of rocks, extent and distribution of hard rocks to many faces that necessitated frequent changes of crushers' installation points). Moreover, these systems caused considerable delays to the BWEs production due to problems related to the damages of belts used for transporting the crushed hard-rocks.

From 1989 to 1996, 13.0 Mm^3 of hard rocks were processed using the shovel – crusher systems.

Since 1990, modern, high capacity earth moving equipment has been gradually purchased. This development contributed to significant increase of the overall efficiency of the mine. Today, the in-time and low-cost haulage of hard rocks has been achieved, while, at the same time, the continuous and efficient operation of BWEs systems is a fact.

The equipment used today for hard rock excavation and haulage is presented in Table 3 (Bozinis & Triantafyllou, 2007).

Using the equipment presented in Table 3 and other auxiliary machinery, SFM achieves annual hard rocks transportation rates above 15 Mm^3. However, due to the higher demand for hard rocks removal, private contractors are also used. As a result, the total transport of hard rocks varies between 25-30 Mm^3/yr and corresponds to 50% of the total overburden material transport.

4. DESCRIPTION OF TODAY'S WORKING METHOD – KEY STATISTICS

The location of hard rock formations and the way of earth moving equipment installation on each bench is determined based on the data gathered during the drilling programme and the maps of the excavation faces of the benches, which are plotted after every transfer of the belt conveyors axis. In the following paragraphs, the sequence of hard rocks excavation works and the results obtained so far are presented and discussed.

Table 3. *Basic equipment used for excavation and haulage of hard rock formations (31.12.2006).*

Type of equipment	No of pieces	Average hours of operations since purchase per machine unit
Drillhole rigs		
Tamrock C50K3L	6	29,403
Bulldozers		
Komatsu D475 A2 (D11 size)	4	20,390
Komatsu D475 (D11 size)	4[(1)]	46,748
Komatsu D275 AX-5	1	14,115
Graders		
Komatsu GD 825	3	37,416
Trucks (payload)		
Euclid/Hitachi-3500 (193t)	8	17,511
Unit Rig (136t)	6	54,360
Terex (85t)	10[(2)]	58,598
Terex (50t)	11[(3)]	26,668
Shovels		
O&K-RH170, 21m^3 bucket	2	18,169
P&H BLE2100, elect. - 15m^3	2	37,385
Liebherr 994, 13.5m^3	1	36,778
Liebherr 999, 13.5m^3	2	53,551
Demag H185, 13.5m^3	1	44,431
Liebherr 984, 5.5m^3	1	49,089
Loaders		
Marathon, bucket of 10m^3	2	34,278
Terex, 5m^3	1	19,885

[(1)] In 2007 four bulldozers (Caterpillar D11) was purchased for substituting the old ones

(2) The 85t trucks operate mainly in areas where there are limitations that does not allow the operation of high capacity trucks, e.g. height limit in roads that pass below belt conveyor bridges

(3) The 50t trucks operate in auxiliary workshops, e.g. haulage of sand and gravel for road construction

a. Ripping

It has been proved in practice that individual layers of hard rocks of thickness less than 1m, which have been uncovered by the BWEs, are possible to be excavated using the ripper of a Komatsu D475-A2 dozer. Then, the excavated material is hauled using shovels and trucks. In cases of thin layers of semi-hard rocks, it is possible to use a BWE for loading the excavated material to belt conveyors. In general, it is evident that Komatsu D475-A2 dozers are reliable and productive in various working conditions in SFM, such as ripping, material spreading at the dumping area and support of shovels at the loading point.

In Table 4, the availability, the average operating time per dozer and the fuel consumption for the D475-A2 dozers that were installed in SFM after 2001 are given for the five-year period 2002-06.

Table 4. *Availability, operating hours and fuel consumption of Komatsu D475-A2 dozers.*

Year	Availability (%)	Average operation hours per dozer	Fuel consumption (lt/h)
2002	63	5,056	94.0
2003	74	3,878	97.0
2004	87	5,610	92.0
2005	74	4,642	86.0
2006	62	3,453	93.0
Average 2002-2006	72	4,528	92.4
Average 2002-2006 for old Komatsu D475 dozers	40	1,844	77.7

The Table also presents the same parameters for the old D475 dozers, which were installed in 1992 and have completed until the end of the year 2006 an average of 46,748 hrs of operation per dozer.

The fuel consumption per hour depends on the type of work (excavation, transport, ripping, etc). The old dozers exhibit lower consumption because they are used under easier working conditions.

Both old and new dozers operate in three shifts per day during the whole year.

b. Drilling

Today, six rotating drills Tamrock C50K3L operate in SFM. Five have been purchased in 1990 and the 6th in 1995. The usual diameter of drills is $7^{7/8}$´´ and 9´´, while the rods are 8m long. After many years of trials, it has been concluded that the optimal distance between blastholes varies between 3-8m while the optimal distance between successive series of blastholes varies between 4-6m (Konya, 1997).

The blastholes length covers the entire bench height even in areas where hard and loose rocks are intercalated. In these cases loose materials are excavated and transported together with hard rocks (Agioutantis & Kavouridis,1998).

Drilling equipment operates during the whole year in three shifts per day. Until the end of 2006 it has completed an average of 29,403 hrs of operation per rig. Additional data related to the operation of blasthole rigs are presented in Table 5.

Table 5. Operational data and efficiency of blasthole rigs for the period 2002-06.

Year	Availability (%)	Efficiency (m/h)	Fuel consumption (lt/h)
2002	42.4	28.9	45.0
2003	52.6	34.1	48.0
2004	49.0	31.6	50.0
2005	48.2	32.9	49.6
2006	40.7	32.3	46.2

The availability of blasthole rigs is low due to aging. In their first period of operation, the availability of rigs was 75-85%. Depending on the hardness of the excavated formations, the life span of the tricon heads varies from 4,000-12,000m of drill while for the rods the life span is 24,000-30,000m.

During the last years the mine demands a total of 450,000m of blastholes per annum. Private contractors meet only 9% of this demand.

c. Blasting

Originally, blasting was accomplished using ANFO mixes with either ammonia dynamite products or gelatin dynamite (30% weight strength) for boosters. Wet holes were loaded with 30% weight strength gelatin dynamite.

The ANFO mixes were prepared in-situ using two ammonia nitrate and fuel oil bulk mixing trucks, which have been purchased in 1993. Each truck is able to carry 4tn of NH_4NO_3, 5.5tn of emulsion (nitrate salts dispersed as small droplets in a continuous oil base) and 2,000 lt of fuel oil (Agioutantis et al., 2000).

For the first time in 1995 SFM engineers used a special mixture of ANFO and an emulsion called Heavy ANFO (H-ANFO), which also was prepared in-situ by mixing the emulsion and porous ammonium nitrate. Several mixing ratios were tested before selecting the optimum one. This mixture has relatively high bulk strength, a higher critical diameter (over 150mm) but good blasting characteristics in wet holes. Hence, ANFO is currently used in dry holes and H-ANFO in wet holes. Ammonium dynamite is used as a booster to ANFO and gelatin dynamite as a booster to H-ANFO. The use of water resistant explosives has increased in recent years. The in-situ preparation of ANFO mixes and H-ANFO and the mechanization of the explosives treatment result in a more safe and cost-effective basting procedure (Agioutantis et al., 2001).

It has been concluded that in 2006 the cost of in-situ preparation of ANFO mixes and H-ANFO was lower of about 3.5 MEuro compared to the market prices.

During the last five years the explosives consumption was 88% H-ANFO, 7% ANFO and 5% boosters. In every blasting 1-6tn of explosives are used depending on the number of holes and the mining equipment that operates in the vicinity of the blast area. Statistical distributions of blast load per blast show that, currently, the most common configuration is a 3-3.5tn blast. SFM, in cooperation with Technical Universities are trying to establish a ground vibration monitoring system in the whole mining area.

It should be noticed that blasting conditions at SFM are unique due to the variability of the hard rock formations and the fact that the hard rock removal has to accommodate the needs and advance rates of the BWE systems that excavate the majority of the overburden material. Based on the experience gained till today it is preferable and more economical to remove the soft cover of a bench and then to blast the hard lenses rather than to blast the full height of the bench using complicated blastholes loading techniques, such as decking.

During the last five years the material moved by conventional equipment, which needs to be loosened by blasting, varied from 88-53% (average 58.5%).

Table 6 presents the total explosives consumption per annum and the corresponding total drilling effort in the SFM for the last 10 years. In the same Table the total volume of overburden handled by conventional equipment is also shown.

According to Table 6, the explosives consumption, the drilling and the hard rock formations removal are continuously increasing for the last 10 years, but with different rates. It should be noted that, since the hard rock lenses are not uniformly distributed within the overburden and considering also the variability of the hard rock formations mechanical and physical properties, drilling and blasting data may experience fluctuations for a given time period. During the last two years (2005-06) the specific consumption of explosives was 284 kg/m^3, while in the period 1997-2000 it was 250kg/m^3. For the period 2002-06 the total cost of blasting works (including drilling) varied from 0.42-0.48 Euro/m^3 of loosened material (Bozinis, 2000).

Table 6. *Variation of explosives consumption, length of blastholes and hard rocks excavations in the period 1997-2006.*

Year	Explosives consumption (tn)	Length of drilling[(1)] (m)	Hard rocks haulage[(1)] (10^6m^3)
1997	1428	306,976	15.3
1998	1533	325,936	14.6
1999	1293	357,482	16.5
2000	1249	350,000	14.6
2001	2280	385,000	12.0
2002	2414	437,265	15.2
2003	2453	428,858	19.3
2004	2460	412,765	24.9
2005	3988	480,290	27.6
2006	3332	443,884	26.2

[(1)] Including work carried out by private contractors

d. Shovel-truck loading and haulage system

i. Shovel-loading system

During the last decade the main loading equipment of the large DUMPERS (85-193 t) were the hydraulic shovels with bucket capacity from 13,5 m^3 (3 units are in operation in the year 2006) to 21 m^3 (2 units were installed in the year 2004) and electric rope shovels with bucket capacity of 15 m^3 (2 units were purchased in the year 1989). Front-end loaders are used only in cases of firm surfaces for loading sand and gravel material, while the hydraulic hoe with bucket capacity of 5,5 m^3 is used in working places of intensively water-bearing surfaces as well as in supporting activities, in cooperation with dumpers of 50t payload.

Hydraulic shovels have been extensively used because they were proven very reliable and flexible equipment for the mining conditions of SFM (often changes in working positions, high percentage of plastic clay, water-bearing areas etc.). Electric rope shovels, which are compact machinery, were proven efficient only in work places that don’t need often transfers and in excavation faces with height more than 3m.

The shovel type or the loading equipment selection in the complicated conditions of SFM depends on the position of the working area along the bench, on the possibility of approaching the mining truck of suitable capacity (crossing with BWE and belt conveyor position), on the type of the material that will be loaded, on the prevailing mining conditions and dimensions of loading area, on the life time of the working activities etc. This selection is based on a combination of many factors that are evaluated almost daily utilizing the advantages of various types of machinery in the suitable loading positions. Figure 3 presents the availability of various types of hydraulic shovels that are used in SFM in relation to total operating hours per machine for the period 1997-2006.

From the diagrams of Figure 3 it is derived that the hydraulic shovels presented high availability and operational efficiency at least for the first 5 years of their operation. More specifically, for the operation period of first 5 years the following data are observed:

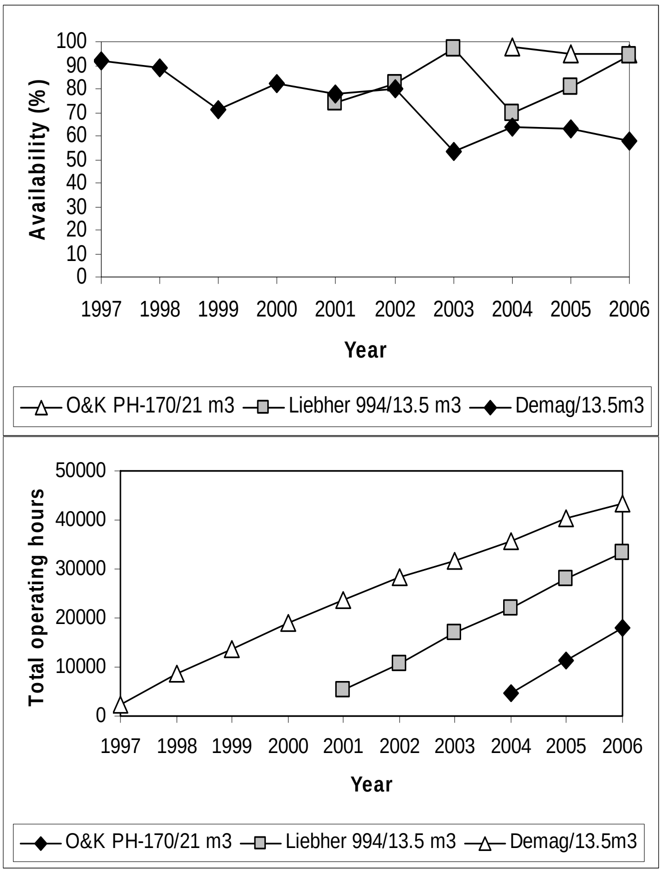

Figure 3. Availability of various types of hydraulic shovels in relation to total operating hours per machine for the period 1997-2006

- Availability (%): 95-70
- Annual operating hours per machine (average of 5 years): 6150-5250
- Operating hours per machine and shift: ~5.7
- Fuel consumption:

\- Shovels with bucket capacity of 13.5 m^3: 90-120 lt/h

\- Shovels with bucket capacity of 21 m^3: 160-172 lt/h

For the year 2006, the contribution of basic shovels in the overburden material removal as well as the data of their productivity are presented in the following Table 7.

Table 7: *Productivity of basic excavation-loading machines for the year 2006*

Shovel type	No of units	Operating hours per machinery	Hourly Production (m^3/h)	Annual production (Mm^3)
O&K-Terex/RX170 ($21m^3$) *Year of purchase: 2004*	2	6,762	806	10.90
Liebherr 994 ($13.5m^3$) *Year of purchase: 2000*	1	5,410	385	2.08
Demac H185.5 ($13.5m^3$) *Year of purchase: 1997*	1	3,185	268	0.85
P&H BLE2100 ($15m^3$) *Year of purchase:1989*	2	3,200	598	1.91
TOTAL	6			15.74

ii. Haulage

The mining trucks that were used for the haulage of hard rock material have been already presented in Table 3. With these trucks, 16.64 million m^3 of hard rock material were transported in the year 2006, while private contractors transported ca. 10 million m^3 for achieving 26.61 million m^3 that was the total productive volume of hard rocks.

The purchase time of various types of trucks as well as their contribution in the hard rocks haulage data for the year 2006 are presented in Table 8.

Table 8. *Contribution of various types of mining trucks to the haulage of hard rocks in the year 2006.*

Truck type	Number of units	Year of purchase	Hard rocks haulage	
			m^3x10^6	Percentage (%)
EUCLID/HITACHI-3500/193t	8	2004	10.03	60.3
UNIT RIG/136t	6	1997	3.57	21.5
TEREX/85t	10	1992	2.16	13.0
TEREX/50t	11	1993	0.88*	5.3
TOTAL	35		16.64	100.0

* It is excluded the not productive operation of TEREX 50t (e.g. haulage of road construction material).

For obtaining the above-mentioned removal, important infrastructure works are required and a significant number of auxiliary equipment is utilized including: Hydraulic hammers for secondary crushing (2 units), Dozers D9 (3 units), Loaders with bucket capacity of $5m^3$ (3 units), Graders (2 units), Vibrators (1 unit), Fuel trucks (3 units), Water trucks (2 units) and Vehicles for transport and supervision (9 units).

The quality of haulage roads as well as the surface conditions of loading and dumping areas constitutes an important factor for the dumpers efficient operation. For the infrastructure works, construction sand gravel of good quality is used systematically (it is excavated from

SFM overburden strata). With the infrastructure works, the annual operation of mining trucks is practically ensured, in a system of 3 working shifts, independently of the weather conditions.

The need for continuous operation of the BWE system, in combination with the fact that hard rocks are dispersed in all excavation benches, imposes the parallel, continuous operation (365 days the year, with a system of 3 working shifts) of the conventional hard rocks removal equipment.

For the evaluation of the productivity data that are given below, it should be noticed that, in the present conditions (2005/2006), the haulage route of dumpers consists of a segment of approximate length 1000m inside the excavation area, that is changing with the mine development, of a 2000m segment (the main ramp, descending with a gradient 5%) and finally of a segment 1100-1200m in the dumping area. During the decade 1997-2006 the average haulage route length to the inside dump area was increased from 3.3 km to 5.2 km.

The high capacity trucks have obtained high efficiencies, especially in the first years of their operation, in spite of the difficult working conditions of SFM. Characteristic data of mining trucks operation and efficiency are presented in Table 9.

Table 9. *Operational and productivity data for DUMPERS with payload ≥ 85t.*

	EUCLID/HITACHI -3500 193t (2005-2006)*	**UNIT RIG 136t (1998-2002)***	**TEREX 85t (1993-1997)***
Annual operation hours per truck unit	6368-6506 (6437)**	5368-6361 (5830)	3600-4795 (4353)
Availability	89,1-90,0 (89,6)	71,2-80,2 (77,8)	60,4-89,3 (73,6)
Productivity m^3/truck unit-shift	1270-1285 (1277)	951-993 (973)	560-615 (595)
Dielel consumption (l/$m^{3)}$	0,47-0,50 (0,48)	0,55-0,60 (0,59)	0,58-0,62 (0,62)
Dielel consumption (l/h)	90,3-99,3 (94,8)	78,8-88,5 (83,3)	60,4-66,1 (63,5)

* First 5 years of operation. The EUCLID/HITACHI were installed in the middle of the year 2004.
** Number in brackets show the mean of the corresponding values for the 5 years of operation. (For EUCLID/HITACHI, mean values refer to 2 years of operation).

The evolution of productivity (m^3/truck and shift) as well as of the annual operating hours per truck during the 10-years-period 1997-2006, are presented in Figures 4 & 5, respectively. Based on these figures the productivity has been increased, taking into consideration that the average haulage distance has been increased from 3.3km (1997) to 5.0km (2006).

Based on the above-presented calculations it is derived that the overall performance of the shovel-truck systems is very satisfactory, approaching the international best available practices, especially if the following adverse factors are taken into account:

- Adverse weather conditions in the winter period (4-5 months)
- Conjunction of the hard rocks removal equipment with the operation of BWEs
- High percentage of plastic clay (>70%) and extensive water-bearing areas.
- Difficulties in blasting because of the existence of BWEs and belt conveyors.

For the last years (2005-2006), the average total cost of hard rocks removal was approximately 2.5 €/m3. The percentage contribution of the each parameter is presented in Figure 6.

5. OVERALL MINE PERFORMANCE

The successful removal of hard rocks as result of the installation of high capacity earth moving equipment and the high rates of hard rocks transport that were achieved, especially during the last decade, contributed to a considerable efficiency increase of BWEs operating both in the overburden and lignite bearing strata. The total excavations of SFM, the total overburden transport (including hard and loose material) and the hard rocks haulage for the period 1997-2006 are shown in Figure 7. The increase of total mine excavations during this 10 years period was 38.8% while the increases for overburden and hard rocks transport were 24.4% and 71.9%, respectively.

Moreover, the efficiency of BWE also increased as a consequence of an improvement in hourly efficiency and operating time. Figure 8 presents the material quantities transported by the BWE systems installed in the overburden and lignite bearing strata of SFM for the period 1997-2006. From the above data, it is obvious that the progress in hard rocks removal from the equipment of the overburden strata affects directly the performance of BWEs and consequently of the entire mine. It worth noticing that lignite production in 2005-06 was 20Mt when 10 years ago it was 16.1Mt.

6. CONCLUSIONS

The initially limited information about the size and distribution of hard rock formations in the overburden strata of SFM resulted to decisions, regarding mine development plans and equipment selection that did not meet the demanded production and did not fit to the real operating conditions. This fact caused considerable delays in mine opening and development. Although mining operations started in 1979, the pit bottom, in a depth of 150m, was reached in 1988.

The additional data gathered in the early period of mine operation from a special programme of exploratory drills for plotting the hard rock formations, combined with information derived from the observation of the open mine excavation faces, led to the revision of the mine development plans regarding the following parameters:

– Number of benches in overburden strata (i.e. five benches of relatively low height, instead of three benches that were initially designed)

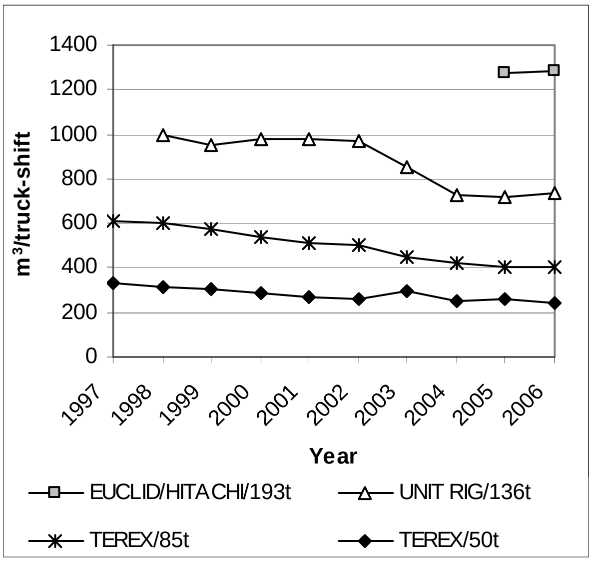

Figure 4. Evolution of productivity per truck type during the period 1997-2006

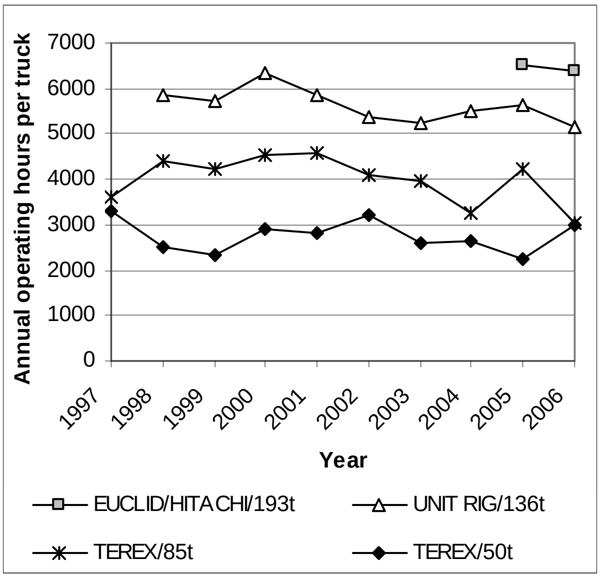

Figure 5. Evolution of the annual operating hours per truck type during the period 1997-2006

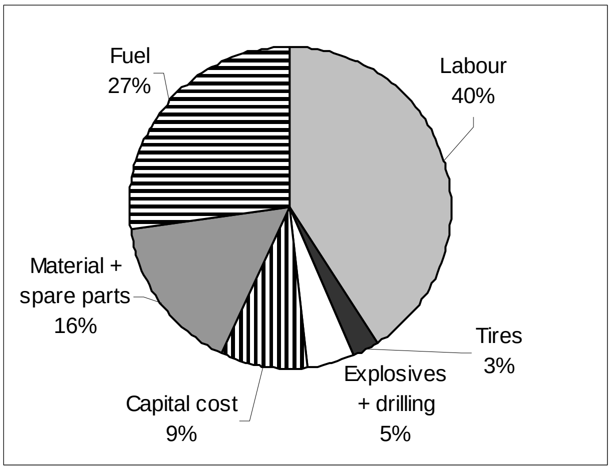

Figure 6. Contribution of cost parameters to the total cost of hard rocks removal.

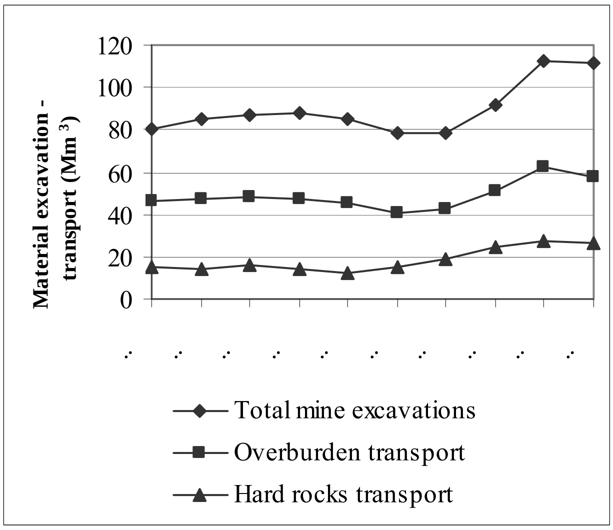

Figure 7. Evolution of total excavations, overburden and hard rocks transport at South Field Mine during the 10-years-period 1997-2006.

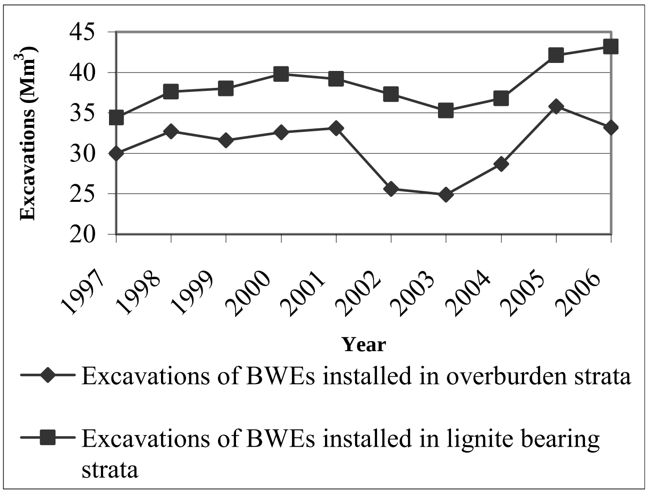

Figure 8. Material quantities transported by the BWE systems installed in the overburden and lignite bearing strata of SFM for the period 1997-2006

– Purchase of new BWEs

– Purchase of conventional earth moving equipment for hard rocks haulage, which would operate separately from the continuous mining systems.

If the problems related to the abundance and the distribution of hard rocks were known during the elaboration of the first mining study of SFM, a non-continuous excavation and transport system would be selected for removing both the hard and loose rocks of the overburden strata. Nevertheless, based on the achieved results up to now, it is concluded that the combined use of BWE systems, for loose material transport, and conventional earth moving equipment, for hard rocks haulage, is a realistic and low-cost solution, which can be improved further in the future.

In addition, the complex and continuously varying conditions in SFM require a close collaboration between continuous and conventional mining systems. Because of the dynamic character of these variations it is also necessary to inspect and modify the operating parameters of the mine on a daily basis.

SFM lignite deposit is unique as far as its technical characteristics and exploitation conditions are concerned. As a consequence, it is a challenge facing Technical Universities and equipment manufacturers the development and pilot-scale implementation of innovative technologies and new earth moving equipment for surface mining sites worldwide.

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AN EXPLOITATION PROCESS OF A CHUTE SYSTEM WITH PRIORITY

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Abstract: *The theory of operation of mine continuous machinery systems developed in Poland 1970s has been verified and proved its usefulness in mine practice. Many general as well as particular dilemmas in this regard have been considered, however some challenging problems remained intact. In the paper the problem of exploitation characteristics of the chute type node machinery system is considered. In the presented case of study discussion is orientated on a system consisting of two conveyors delivering material onto one unit. For this elementary system different methods of servicing can be employed giving different system characteristics. The method of analysis of this problem is presented in the paper.*

1 INTRODUCTION

Mining engineer considering problems connected with continuous type machinery systems is usually in extremely convenient situation. The set of mathematical tools for modelling, analysis and calculation of this type of machinery systems is rich enough to make it properly and comprehensively. The state of art in this field is much better comparing to mining cycling systems or readiness systems.

First papers concerning analysis of operation of belt conveyor systems appeared at the early 1960s: witness Rist (1961), Teicholtz (1963), Teicholtz ane Douglas (1963), Bishele et al. (1964), Harvey (1964). They related to the application of simulation technique to investigate operation of belt conveyor systems with some orientation on design problems. Generally, in 1960s many publications occurred in several countries, just treating with the application of statistical experiments for imitation of operation of continuous mining machinery systems. Simultaneously, in Poland researches were involved in analysis and calculation of mine conveyors systems from the analytical point of view. First elaborations occurred in 1962-63, first significant paper – Gladysz (1964). After field investigations times of reliability states have been identified as the exponential ones and the whole considerations have been placed in the Markov theory field. Therefore, majority of characteristics and indices were easy to obtain and calculate. In late 1960s and early 1970s many papers were published in this regard in Poland and the theory of mine machinery systems of continuous operation was developed. A bit later

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some textbooks appeared on academic shelves, e.g. Czaplicki and Lutynski (1987), Sajkiewicz (1979). Conveniently, at the beginning of 1970s some papers described results of investigations confirming accordance outcomes obtained from simulation and analytical considerations. Generally, the theory of mine machinery systems of continuous operation developed in mature form in 1970s remained in almost unchanged form till now. It was no special need to conduct further theoretical investigations, especially because of continuous progress in the field of simulation technique. Regularly, every few years a new enhanced method of simulation, just for mine purposes, came into being giving some new possibilities in this regard, e.g. Mutmansky and Mwasinga (1988), Sturgul (1989), Tan, S and Ramani (1988).

Both ways of modelling, analysis and calculation have their own merits and demerits but – what is important – they complement each other. Nevertheless, some problems are more convenient to discuss analytically, other problems using simulation.

Problems of servicing, making repair are more convenient to consider methodically considering transitions between states of a given system if the system is an elementary one or simple, not complicated. For complex systems matrix of transition between states and the corresponding exploitation graph is complicated, expanded and usually difficulty legible. Application of the simulation is more convenient for analysis in such a case.

It seems that more comprehensive analysis of different methods of servicing of elements of continuous systems has been missing in general theory of this type of systems. It is worth to realize that the developed method allows analysing that type of problems.

The purpose of this paper is to draw attention to subtle problems of different methods of servicing, possibility to analyse analytically such problems and to present a method of calculation of effects of presumed scheme of maintenance.

2 EXPLOITATION OF THE SYSTEM AND THE NUMBER OF REPAIR TEAMS

Let us commence our considerations recalling definition of exploitation because this term will be repeated many times in this paper.

*The exploitation of a technical object is a set of intentional actions of a technical, economical and organizational nature directed at this object, as well as mutual relationships existing between them (people cooperating and the object) from the moment of the object's first usage until its withdrawal and disposa*l (Polish Standard PN-82/N-04001).

In the Central and East Europe engineers adopted a French word *exploitation*, which means usage. In connection with engineering problems, exploitation is based on the statement that it is *usage of something in a rational way*. But in the English-speaking world the term *exploitation* possesses many negative connotations and is almost unacceptable in that meaning. In common practice the term *operation* is being used; however this word possesses almost fifty different meanings (Thesaurus.com). It is a picklock word (More details on relation between operation, exploitation and terotechnology one can find in Czaplicki's monograph 2008 chapter 5). Instead of theory of exploitation the word terotechnology is being used (mainly in the UK and India), nevertheless these words mean roughly the same but they are not equivalent to each other.

Generally, an exploitation process consists of two processes: utilization and maintenance, interlacing with each other. If we shift our considerations into the reliability field the exploitation process converts into the process of changes of states, reliability states. They are: work, repair, standstill and reserve. If the continuous mine machinery systems are concern, times of states occurring in random way are exponential. Thus, exploitation processes of these systems can be satisfactory described by the Markov processes.

In the regular analysis of a Markov process the term repair has one meaning and the only difference – when system is considered – is connected with the number of elements in failure and which elements are in repair. When different methods of servicing will be introduced effects of such introduction will be visible in transitions between states and values of appropriate intensities of transitions.

To illustrate the above statement consider the well-known node of machinery system – a chute. This elementary system is shown schematically on Figure 1.

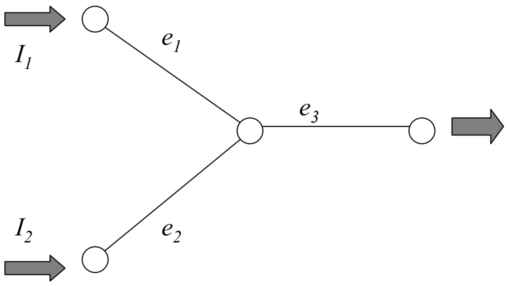

Figure 1. The structure of considered system – a scheme of the machinery node of chute type

It can be determined as follows:

$$S : < e_i,\ i = 1, 2, 3;\ \lambda_i,\ \mu_i;\ O >$$

Let us read this notation. The system S consists of *3* elements, e_1, e_2 and e_3. Each element is characterized by the intensity of failures λ_i and intensity of repair μ_i. These elements create a system of a structure given in Figure 1. Functioning of the system relies on receiving the stream I_1 by element e_1 and element e_2 obtains the stream I_2. Element e_3 collects both streams.

Element e_1 can be a single one or it can represent a series system after reduction. The same supposition can be formulated towards elements e_2 and e_3. Therefore this system describes well – for instance – two continuously excavating machines operating in two different working faces and delivering extracted rock to one gathering unit that can consists of one or more elements (conveyors) connected in series.

The last component of the notation O describes maintenance of the system elements. Here several different information is needed to specify the exploitation process of the system.

Firstly, information on the number of repair teams should be given. If the number of repair teams is concern, all systems can be divided into two categories:

- systems with the *sufficient number* of repair teams, i.e. the number of teams equals the maximum number of failures that can occur in the system,
- system with the *insufficient number* of repair teams – otherwise.

For the system considered the maximum number of repair that can be cleared in the system is *2* if ordinary assumption will be given stating that element is a standstill state can't fail (or saying more precisely, the intensity of failures of an element in a standstill is negligible). For the system having sufficient number of repair teams we can assume that all repair commence at once if we include time to rich the failure place by repair team to the time of repair action.

If the number of repair teams is insufficient we have to determine a policy of maintenance. For the system considered we have to decide what to do when during repair (the only repair team is busy) a new failure appears. A question is: to continue started work or to stop, go to a new failure place, make repair and return to finish previous repair? All cases with more than one repair in a given time should be considered carefully and appropriate policy should be formulated.

3 SOME SERVICING METHODS AND THEIR REPERCUSSIONS

We can make use of the analysis procedure given in Czaplicki and Lutynski textbook 1987 taking these parts from it that are useful in our consideration. Let us specify the exploitation repertoire for the system. It is presented in Table 1, where: W – work, R – repair, S – standstill. The number of theoretically possible states is obviously *8*, however the eighth state – all elements in failure is impossible if we presume that element in a standstill will not fail.

Table 1. *The exploitation repertoire of the system*

State of system	Element		
	e_1	e_2	e_3
s_1	W	W	W
s_2	R	W	W
s_3	W	R	W
s_4	S	S	R
s_5	R	R	S
s_6	S	R	R
s_7	R	S	R

Let us now construct the exploitation graph. It illustrates transitions between all technically possible states. Its graphical construction depends exactly on the presumed policy on maintenance.

Let us start from a case when is no problem with number of repair teams (*2*) and is no delay in this regard. Figure 2 shows the corresponding exploitation graph.

Let us consider what is going to change in the graph when we have only *1* repair team for disposal. As it was stated we have to consider carefully all states with two elements in failure, i.e. states s_5, s_6 and s_7. Additionally, we may presume that elements e_1 and e_2 are equally important but element e_3 should have priority because if it is in failure no production can be conducted. If so, we may presume that:

(a) If failure appears in third element while element e_1 is being repaired or element e_2 is being repaired, the current repair is stopped, team repairs the element with priority (e_3) and then the team finishes stopped repair

(b) If failure appears in element e1 while element e2 is being repaired or vice versa repair team does not halt its action.

Observe that one item in our consideration may be changed. If one stream of mineral is significantly greater than the second one a new priority can be formulated stating:

(c) Element e3 is the most important

(d) Element transporting greater mass (e1 or e2) is more important that the other (e2 or e1).

One additional assumption is required, yet. An answer on the following question must be given: What kind of repercussions is connected with the fact that a repair is stopped? The same question is formulated considering the general model in mass servicing theory.

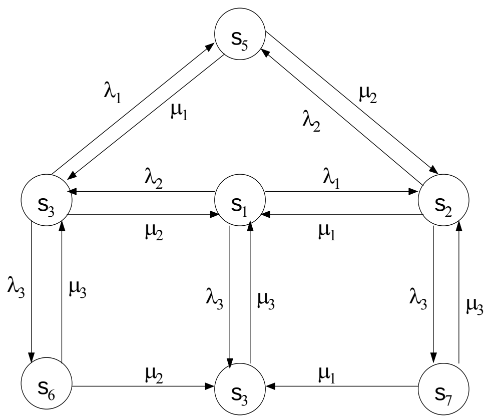

Figure 2. The exploitation graph for the chute system with two repair teams

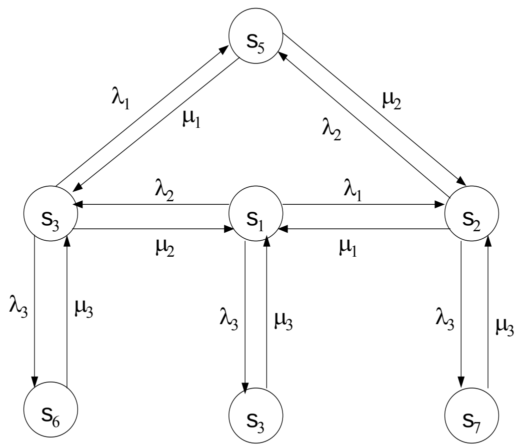

Figure 3. The exploitation graph for the chute system with one repair team and determined policy of maintenance

Here a few different answers can be formulated, namely:

(e) No repercussions: the repair after stoppage is continued, the time of repair – no change

(f) There is a repercussion: the repair must start from the beginning,

(g) There is a repercussion: the repair is continued but it lasts a bit longer than without stoppage.

Case (f) in elements of continuous mining systems does not hold.

The most probable case is (g), however extension of time of repair – if any – is usually in practice, small. Calculation is more convenient if this increment is neglected. Therefore, the most frequently case (e) is being assumed.

Selecting given set of assumption we obtain different course of the exploitation process of the system. Basing on the ordinary logic we can construct the exploitation graph of interest. Figure 3 illustrates the exploitation graph for the chute system presuming – for example – set: A = { (a), (b), (e) }.

Let us now find effects of our presumption.

Let us construct two matrixes of transitions between states. They are presented just below; the left one is for the system with full service, the right one concerns the case when set A is assumed.

$$\begin{pmatrix} \Delta_1 & \lambda_1 & \lambda_2 & \lambda_3 & 0 & 0 & 0 \\ \mu_1 & \Delta_2 & 0 & 0 & \lambda_2 & 0 & \lambda_3 \\ \mu_2 & 0 & \Delta_3 & 0 & \lambda_1 & \lambda_3 & 0 \\ \mu_3 & 0 & 0 & \Delta_4 & 0 & 0 & 0 \\ 0 & \mu_2 & \mu_1 & 0 & \Delta_5 & 0 & 0 \\ 0 & 0 & \mu_3 & 0 & 0 & \Delta_6 & 0 \\ 0 & \mu_3 & 0 & 0 & 0 & 0 & \Delta_7 \end{pmatrix} \qquad \begin{pmatrix} \Delta_1 & \lambda_1 & \lambda_2 & \lambda_3 & 0 & 0 & 0 \\ \mu_1 & \Delta_2 & 0 & 0 & \lambda_2 & 0 & \lambda_3 \\ \mu_2 & 0 & \Delta_3 & 0 & \lambda_1 & \lambda_3 & 0 \\ \mu_3 & 0 & 0 & \Delta_4 & 0 & 0 & 0 \\ 0 & \mu_2 & \mu_1 & 0 & \Delta_5 & 0 & 0 \\ 0 & 0 & \mu_3 & \mu_2 & 0 & \Delta_6 & 0 \\ 0 & \mu_3 & 0 & \mu_1 & 0 & 0 & \Delta_7 \end{pmatrix}$$

Elements on the main diagonal are easily calculated having in mind the theorem stating that the sum of all intensities in a raw must be closed to *0*.

Let us go further with our consideration. Looking at columns of the above matrixes we can construct equations determining the steady-state probabilities (see for instance Kopocinski 1973, Czaplicki and Lutynski 1987). Rejecting one equation and replacing it by the equation that the sum of all probabilities must be *1*, we have a set of equations allowing to find searching probabilities. Presuming the following intensities in h^{-1}:

$\lambda_1 = 85 \times 10^{-3}$ $\lambda_2 = 110 \times 10^{-3}$ $\lambda_3 = 38 \times 10^{-3}$

$\mu_1 = 0.88$ $\mu_2 = 0.72$ $\mu_3 = 0.90$

We obtain the following estimates for the steady-state probabilities (looking more carefully at values of these intensities is easy to notice that they are obtained after reduction):

$$\begin{pmatrix} P_1 \\ P_2 \\ P_3 \\ P_4 \\ P_5 \\ P_6 \\ P_7 \end{pmatrix} \qquad \begin{pmatrix} 0.763 \\ 0.072 \\ 0.114 \\ 0.036 \\ 0.011 \\ 2.672 \times 10^{-3} \\ 1.54 \times 10^{-3} \end{pmatrix} \qquad \begin{pmatrix} 0.759 \\ 0.073 \\ 0.116 \\ 0.032 \\ 0.011 \\ 4.898 \times 10^{-3} \\ 3.097 \times 10^{-3} \end{pmatrix}$$

4 GENERAL REMARK

Observe that the exploitation repertoire remains intact. The whole difference here is connected with the changing sequence of occurring states. This change has influence on a structure of the matrix of transitions between states. This matrix, in turn, is a base for construction of equations determining the steady-state probabilities for the Markov process (fee for instance Kopocinski 1973, Czaplicki and Lutynski 1987). Hence, changing a policy of maintenance of the system we make change values of steady state probabilities. Being in possession of estimates of these probabilities we are able to make proper decision if policy of maintenance is concern.

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COLLAPSE RISK ASSESSMENT OF ANTON MINE, TURDA SALINE

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Abstract: *This paper presents the stability analysis of Anton mine, Turda saline, taking into consideration an analytical model of calculus for a cloche shape of excavation. The results of the calculus confirm that the Anton mine rests stable.*

Key words: *collapse risk assessment, rock salt, rock salt strength, room stability, cloche shape*

1. INTRODUCTION

To answer to the requests of the research, it is necessary to know, in the first stage, the geomechanical characteristics of the rock salt and the overburden rocks of Turda rock salt deposit (Table 1).

Table 1. *Rock salt characteristics of Turda mine (after S.C. ICPM Cluj-Napoca)*

Cons. nr.	Specification	U.M.	Average value
1	Apparent specific weight	$N/m^3 \cdot 10^4$	2.06
2	Uniaxial compression strength	daN/cm^2	175
3	Tensile strength	daN/cm^2	6.75
4	Bending strength	daN/cm^2	15.48
5	Poisson's ratio	-	0.24
6	Failure specific shortening	%	2.6
7	Internal angle of friction	0	28.4
8	Apparent cohesion	daN/cm^2	50
9	Insoluble	%	0.87
10	Specific weight	$N/m^3 \cdot 10^4$	2.14
11	Shear strength	daN/cm^2	13.67 – 14
12	Real cohesion (laboratory)	daN/cm^2	36

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From the outset, we specify that we haven't had the possibility of determining the geo mechanical characteristics, therefore the calculus will take into account the laboratory determinations performed by S.C.MINESA-ICPM S.A. Cluj-Napoca, shown in the table no.1.

Because the determinations provides by S.C.MINESA-ICPM S.A. Cluj-Napoca are missing some necessary characteristics for further dimensioning, these were estimated by us, taking into account the relation between the geomechanical characteristics [1,3] and which, for the rock salt, are the following:

Table no.2

Type of rock	$\frac{\sigma_{rt}}{\sigma_{rc}}$	$\frac{\sigma_{rf}}{\sigma_{rc}}$	$\frac{\sigma_{inc}}{\sigma_{rc}}$
Rock salt	1/19.7	1/12.8	1/9.7

σ_{rt} is failure tensile strength;

σ_{n} -failure compression strength;

σ_{rf} -failure shear strength;

$\sigma_{r\,inc}$ - failure bending strength.

Taking into account these rapports, result that the failure shear strength is approximately 14 daN/cm^2.

2. THE CHECKING OF ANTON ROOM (MINE) STABILITY FROM TURDA SALINE

The Anton room or mine is a room with a cloche shape (fig. 1), with 75m height and approximately 77m diameter. It was closed in 1862, because of the high content of clay in the rock salt.

During and after the Second World War the Anton room was used as waste deposit, especially with animal origin. In order to diminish the unpleasant smell of these organic wastes, the access towards this room was closed with a dam, constructed from bricks and clay, in the Franz Iosif gallery.

In the present (year 2008), the entire Turda saline is the object of an ample arrangement programme, in view of enlarging the balneoclimatic touristic base. Because the Anton mine is a part of Turda saline – outside of the touristic circuit – and its stability could influence the ground surface and other underground voids' stability and safety, it is necessary to check the failure risk of the Anton room.

For checking the stability of this room, the room's shape is approximated with a trapezoidal profile (fig.2).

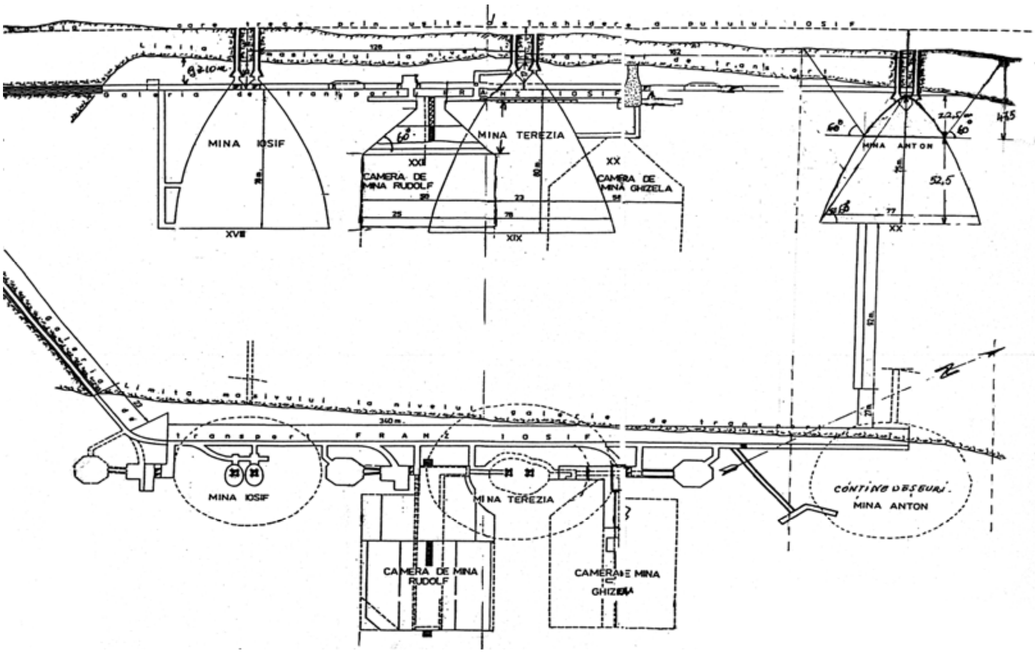

Fig.1. Turda saline representation

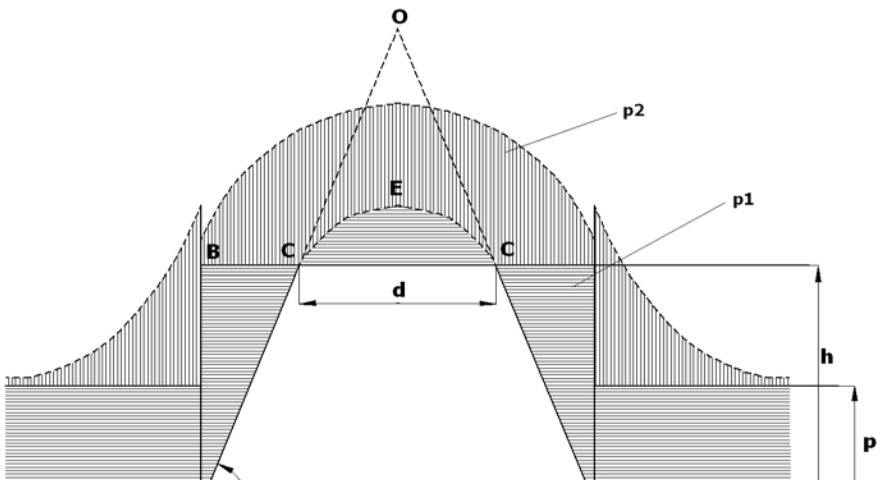

Fig.2. The pressure development surrounding the mining room with a trapezoidal shape

According to the scheme represented in figure 2, for stress equilibrium to exist, the following condition is made:

$$p_1 + \frac{1}{2} \cdot p_2 \le \sigma_f \cdot h \qquad (1)$$

where:

$$p_1 = \frac{1}{2} \cdot \gamma_a \cdot h^2 \cdot ctg\alpha \text{ and } p_2 = \gamma_a \cdot D \cdot \left(\frac{D^2}{48 \cdot u} - u \right) \qquad (2)$$

2.1. The first calculus hypothesis

In the case of $D \le u \cdot \sqrt{48}$, replacing the values of p_1 and p_2, results:

$$\frac{1}{2} \cdot \gamma_a \cdot h^2 \cdot ctg\,\alpha + \frac{1}{2} \cdot \gamma_a \cdot D \cdot \left(\frac{D^2}{48 \cdot u} - u \right) \le \sigma_f \cdot h \qquad (3)$$

where: σ_f is the shear strength of rock salt

Replacing:

$$D = d + 2 \cdot h \cdot ctg\alpha \qquad (4)$$

results that:

$$\begin{aligned} &8 \cdot ctg^3\alpha \cdot h^3 + 12 \cdot ctg\alpha \cdot (d \cdot ctg\,\alpha + 4 \cdot u) \cdot h^2 + \\ &+ 6 \cdot \left[\left(d^2 - 16 \cdot u^2 \right) \cdot ctg\alpha - 16 \cdot u \cdot v \right] + d \cdot \left(d^2 - 48 \cdot u^2 \right) = 0 \end{aligned} \qquad (5)$$

The cloche rooms are assimilated to the trapezoidal rooms, with a profile resulting from the prolongation of the trapeze sides (fig. 2). Making $d = 0$ in the relation (5) results:

$$8 \cdot ctg^3\alpha \cdot h^3 + 48 \cdot u \cdot ctg\alpha \cdot h^2 - 96 \cdot u^2 \cdot ctg\alpha \cdot h - 96 \cdot u \cdot v \cdot h = 0 \qquad (6)$$

or:

$$8 \cdot h \cdot \left[ctg^3\alpha \cdot h^2 + 6 \cdot u \cdot ctg\alpha \cdot h - 12 \cdot u \cdot ctg\alpha \cdot (u \cdot ctg\alpha + v) \right] = 0 \qquad (7)$$

The roots of this equation are the following:

$$h_1 = 0 \text{ and } h_{2,3} = h = \frac{-3 \cdot u \pm \sqrt{9 \cdot u^2 + 12 \cdot u \cdot ctg\alpha \cdot (u \cdot ctg\alpha + v)}}{ctg^2\alpha} \qquad (8)$$

In this relation:

$$u = \frac{\sigma_t}{n \cdot \gamma_a} \quad \text{and} \quad v = \frac{\sigma_f}{n \cdot \gamma_a} \qquad (9)$$

σ_t - failure tensile strength;
σ_f - failure shear strength;
γ_a - apparent specific weight;
n – security coefficient;
α - angle between horizontal and the triangle sides.
For the rock salt of Turda saline, from the table 1 result:
$\sigma_t = 6.75$ daN/cm^2; $\sigma_f = 14$ daN/cm^2; $\gamma_a = 2.06 \cdot 10^4$ N/m^3 ;
$\alpha \cong 60^0$ (really, $\alpha = 58^0 \div 62^0$).
Taking into account different values for the security coefficient *n,* it is obtained:

n = 1.5 ; u = 22.5 ; v = 46.7 ; h = 152 m;
n = 2 ; u = 16.8 ; v = 35 ; h = 112 m;
n = 3; u = 10.92; v = 22.65; $h \cong$ 75 m.

It is appreciated that, for the security coefficient $n \le 3$, the room is stable because the calculated height is approximately equal to the height of the real room.

For n=3, in relation (4), results that $D = 2 \cdot 75 \cdot 0.577 \cong 86$ m and the Anton room, having the diameter of 77m, rests stable.

2.2. The second calculus hypothesis

In this case the limit condition is the following:

$$D > u\sqrt{48} \qquad (10)$$

Consequently, results the following equation:

$$8 \cdot ctg^3\alpha \cdot h^3 + 12 \cdot ctg\alpha \cdot (d \cdot ctg\alpha + 2 \cdot u) \cdot h^2 + 6 \cdot (ctg\alpha \cdot d^2 - 8 \cdot u \cdot v) \cdot h + d^3 = 0 \qquad (11)$$

The equation (11), for d = 0 becomes:

$$8 \cdot ctg^3\alpha \cdot h^3 + 24 \cdot u \cdot ctg\alpha \cdot h^2 - 48 \cdot u \cdot v \cdot h = 0 \qquad (12)$$

or

$$8 \cdot h \cdot \left[ctg^3\alpha \cdot h^2 + 3 \cdot u \cdot ctg\alpha \cdot h - 6 \cdot u \cdot v\right] = 0 \qquad (13)$$

The roots of this equation are the following:

$$h_1 = 0 \quad \text{and} \quad h_{2,3} = \frac{-3 \cdot u \pm \sqrt{9 \cdot u^2 + 24 \cdot u \cdot v \cdot ctg\alpha}}{2 \cdot ctg\alpha} \qquad (14)$$

In the same case, taking into consideration the safety coefficient $n = 2$, results: u = 16.4; v = 34; $h \cong 45$ m and by consequence the stability condition of the room is not accomplished; but for n=1.2, results u=27.32, v = 58.68 and h=77m > 75m (real height) and from the point of view of the room's height, the excavation is stable.

In the same previous conditions, the room diameter is $D = 2 \cdot h \cdot ctg\alpha = 2 \cdot 77 \cdot 0.577 \cong 89$ m and after this criterion the Anton room stability is satisfied.

3. CONCLUSION

If the second calculus hypothesis is accepted, which is more unfavourable, the Anton room rests stable for a safety coefficient n=1.2 (without the influence of other disturbance factors) and, theoretically, an information that confirms the collapse risk of the Anton mine does not exist.

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VIRTUAL REALITY – NEW OPPORTUNITIES FOR MINE PLANNING

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Abstract: *The paper discusses the use of virtual reality (VR) as a tool to improve the mine planning process. Considering the short term production planning VR can be used furthermore for an operator assistant system. For a bucket wheel excavator an example is given.*

Keywords: *Surface Mining, production control, virtual reality, bucket wheel excavator*

1. DEFINITION

Virtual reality (VR) can be defined as computer simulated environment in which the user can view something or interact with different objects. For technical processes VR is used to visualize difficult technical processes. The main object is to planning task more transparent where models and assumptions can be verified and evaluated. The visualization of the results strengthens the user's confidence in the results.

2. VIRTUAL REALITY IN THE MINING INDUSTRY

Nowadays, in contradiction to other branches, there is only a minor and partly underdeveloped use of VR technologies throughout the mining industry [1]. The first steps were made in the fields of underground mining, where a lot of processes are not visible (e.g. full automated long wall without workers). A high noise and dust concentration as well as low illumination underground reduces perception so that e.g. education and training of workers is a difficult task.

In open pit mining the use of VR technologies is currently limited to visualize GPS measurements in order to show multiple data in a simple manner [2, 3]. Otherwise different studies showed that using VR in planning and production could lead to significant savings. References to these studies are given in [1].

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3. PRINCIPAL USES OF VR IN THE MINING INDUSTRY

Using the VR technology as a core component for interactive 3D graphics can be an innovative approach for the mining sector to combine the streams of geological, planning, production and economic data in a simple system [6]. In this manner, the VR system can combine planning data with real time process measurement (e.g. position, production, quality). For the practical use in the mining industry it has to be considered which applications and advantages a VR system can have. In connection with a calibrated calculation model, which includes all relevant data of the main processes, VR can be used for the following tasks [4, 5, 6]:

- *Production planning*

It can be simulated how the excavator will work in future geological conditions. Critical assumptions can be evaluated and technological (block) parameters may be changed afterwards. VR gives the advantage that many alternatives can be visually compared which can be interesting, especially in the consulting business.

- *Continuous model improvement*

In contradiction to current programs, where a lot of manual work for updating has to be done, the production data can be directly compared with the underlying models. If there are other than the predicted geological conditions, the model can be updated continuously.

- *Surveying*

Real time GPS data can be converted by the VR system in order to show the progress of the mining machines and pit outline which is necessary for planning as well as for accounting and the government. In dependence of legal questions frequent surveying of the pit outline may be obsolete.

- *Operator assistant system*

A calibrated VR simulation can calculate beforehand which geology has to be expected and which technical and technological parameters are most suitable for effective mining. These optimal parameters can be displayed at the operator’s cabin and help to ensure a high production during the block.

- *Operator training*

The VR system can be used to display the operator’s cabin and simulate the movements of the excavator. For several decades such a simulation is common for airplane pilots. New personnel can be trained already during the assembling of the equipment. This saves a considerable amount of time, because for large machines such an operator training takes up to several months. Additionally disasters and accidents can be safely simulated which may increase the working safety.

- *Data availability*

Combining the VR system with up to date computer connections every engineer or specialist can login into the system from everywhere to take a look on the actual situation in the mine. Therefore a specialist has not to travel to the mine sight or call for data in order to get a quick overview over the situation. This may lead to savings in travel expenses, especially during international projects.

- *Quality control*

While comparing the position of the excavator’s bucket and the geological model it can be calculated which material and volume will be soon on the conveyor belts. With a dumping model a continuous database for the overburden can be established. Even after years the user knows which material and volume is situated in a certain part of the former mine. This may of interest for future construction projects.

- *Education*

During the education a VR system can be used to quickly show the students different technologies. Avoiding long explanations, the lecturer has more time to provide additional information or answer questions. This approach may of extraordinary interest for distance learning.

- *Designing of the post mining landscape*

Main goal of designing the post mining is to restore the landscape close to the former state. VR may help the engineer to plan the future landscape. It can be simulated e.g. how the forest will look like in 20 years and if the power plant will be still visible.

- *Public relations*

In today mining business a lot of stakeholders are involved during planning and permission. Not everybody has the technical background to understand the mining operation and it's alternatives. A visual demonstration of the operation may increase transparency and confidence.

4. APPLICATION OF VR IN MINE PLANNING

4.1. Excavation model for bucket wheel excavator

For the development of the VR system the "open source" software „OPENSCENEGRAPH“ was used, which is freely available in the internet and can be independently altered. The program is based on the standard VRML 2.0. In a first approach a bucket wheel excavator (BWE) SRs 6300.50.15 was modelled. While combining the core components geology, machinery and technological parameters a simulation was established which calculates the output for random block shapes and cutting parameters [5]. Figure 1 illustrates the elements of the VR concept.

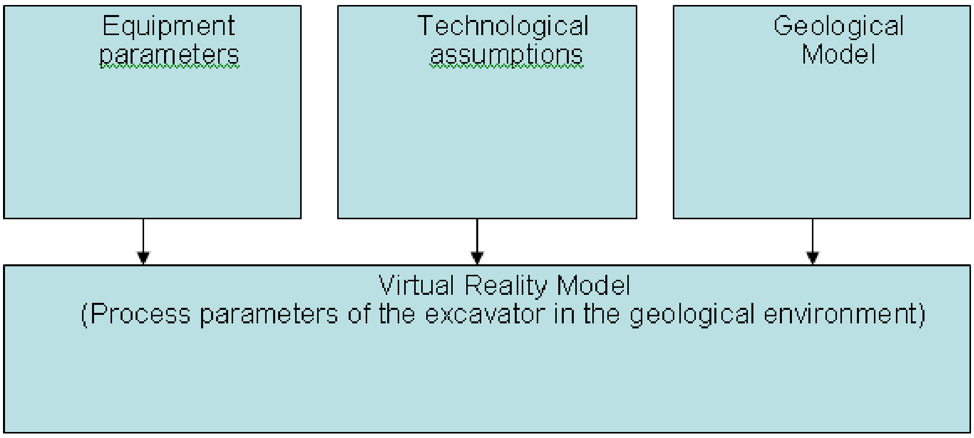

Figure 1 VR simulation model of a bucket wheel excavator

The mathematical model of excavator output control based on the calculation of the possible chip width for the usable power of the bucket wheel drive. The calculation considers the technical, technological (chip) and geological (cutting resistance) parameters. For an bucket wheel excavator the chip width is than defined us [7, 8]:

$$b_{Sp}(\varphi) = \frac{3600\left(P_M \cdot \eta_M \cdot \eta_{ausl} - P_V - \dfrac{2F'_{Gr\,l}\, a_{Sp}(\varphi)\cdot h_{Sp}(\varphi)\cdot v_R}{D_a \cdot \psi}\right)}{\dfrac{3600 \cdot F'_{Gr\,l} \cdot \varphi_H \cdot v_R}{\psi} + a_{Sp}(\varphi)\cdot h_{Sp}(\varphi)\cdot n \cdot 60 \cdot \rho_R\left(gh_m + v_R^2\right)}$$

with:

F'_{Grl}	*specific digging resistance [kN/m]*
P_V	*mechanical power losses [kW]*
η_M	*electrical efficiency of bucket wheel engine*
v_R	*cutting speed [m/s]*
ρ_R	*density*
h_m	*average transportation high*
ψ	*angle between bucketsl [°]*
h_{Sp}	*chip hight [m]*
a_{Sp}	*chip depth [m]*
D_a =	*bucket wheel diamete [m]*
n	*discharge trate [1/min]*

The results of the calculation were compared with practical measurements at the German lignite mine Nochten in order to calibrate the parameters [9].

Figure 2 shows a screen shot of the VR simulation model for the bucket wheel excavator 1510 SRs 6300 in Nochten mine.

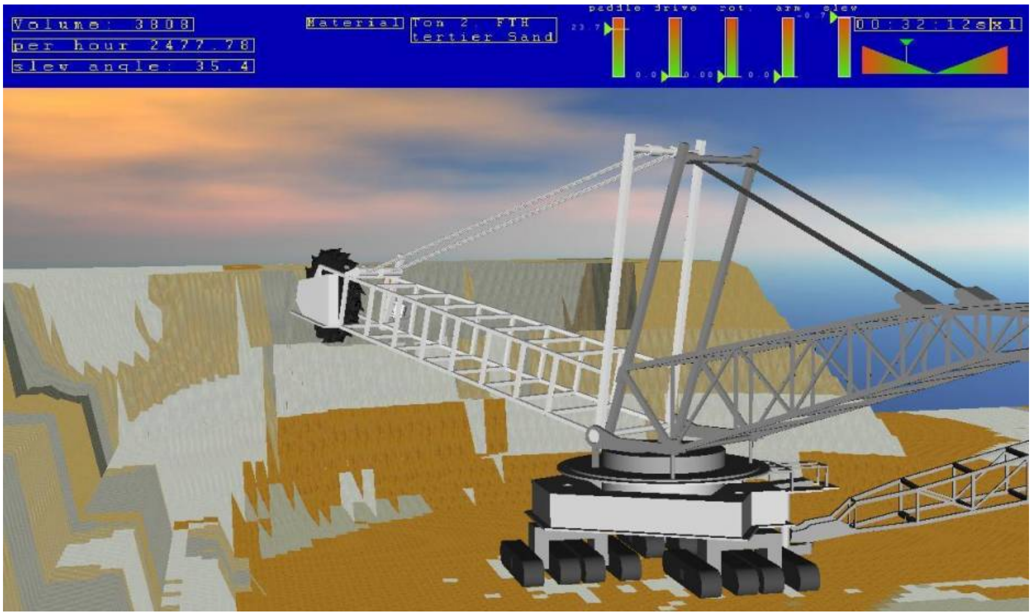

Figure 2 Narrow realistic bucket wheel excavator work simulation in geological environment

The excavator model fulfils all technical parameters of the existing excavator and controls the output automatically by the calculation model. This means that during the slewing process the excavator continuously get the information about the geological parameters and regulate the slewing velocity depending on the power consumption.

4.2 Integrated mine planning and control system

The complex information flow in an integrated mine planning and production system can be structured in the following main modules (Figure 3):

- exploration data (e.g. one-dimensional from drill logs ort two-dimensional from geophysical prospection),
- geological model (three-dimensional processing of the exploration data),

- mine modelling (four-dimensional chronological mine development),
- economic model based on time schedules and mass flow from the mine planning,
- data of the production control to verify the planning assumptions.

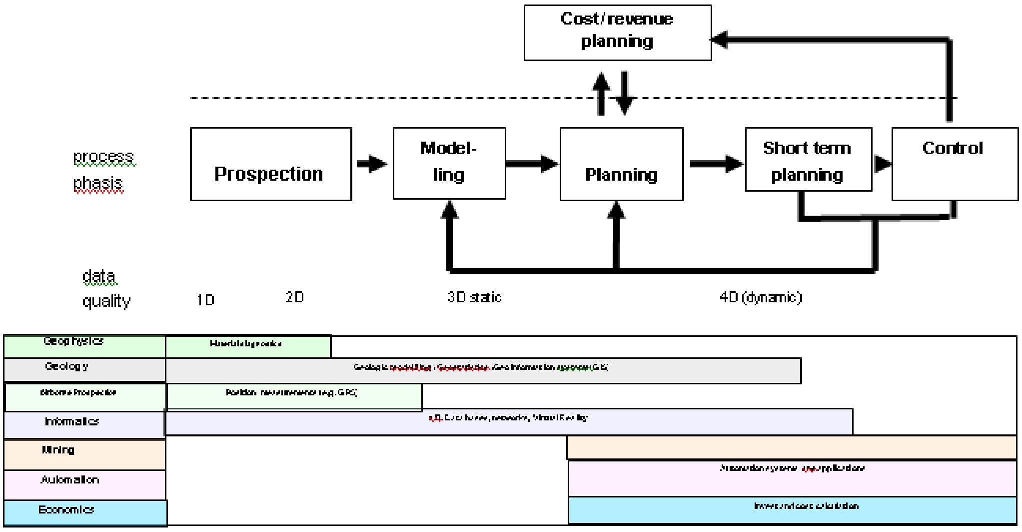

Figure 3 Basic structure of the modular concept for a dynamic mine planning and production control system [1]

The modular concept is based on the logical connection of the modules via object functions. Together they form control cycles in which a constant comparison between planned and production data takes place. With accordingly automated data acquisition and statistical interpretation algorithms a self-learning mine planning and production control system can be configured. The control cycles of the system are:

- geological modelling – mine planning (static),
- geological modelling – production control (dynamic),
- mine planning – production control (dynamic),
- mine planning – cost and revenue planning (static),
- production control – cost and revenue planning (dynamic).

The static control cycles interact with each other during the planning stage. For instance the mine planning may demand further requirements from the exploration or the economic cost and revenue planning requests for another machinery selection or mining technology.

The dynamic control cycles are based on the data acquisition from the production control. These data, consisting of a combined material identification and position measurements, can be used e.g. for the automated verification of the geologic model or the build-up of a dump model. Additionally, complex performance measurements contribute data for improving the performance planning depending on geological, technical and technological factors. Finally, the factual costs (manpower, material and energy usage) give information about the mine's situation (e.g. geological conditions, mine organisation, maintenance schedule).

The implementation of this modular and dynamic concept demands a complex collaboration of different specialists e.g. (Fig. 1):

- **geophysics:** online forecast of the material,
- **geology:** geostatistics of the deposit and build-up of GIS systems,

- **surveying:** position measurements (GPS, laser scanning …),
- **mining:** machinery selection and planning,
- **economics:** invest, cost and revenue planning,
- **automation:** data acquisition devices and technology,
- **informatics:** data storage, preparation and distribution.

The mine planning, as an example for one module of the dynamic mine planning and production control system, will be furthermore discussed, where virtual reality acts as an innovative approach in the mining sector.

Using the VR technology as a core component for interactive 3D graphics can be an innovative approach for the mining sector to combine the streams of geological, planning, production and economic data in a simple system [10]. In this manner, the VR system can combine planning data with real time process measurement (e.g. position, production, quality, Figure 4).

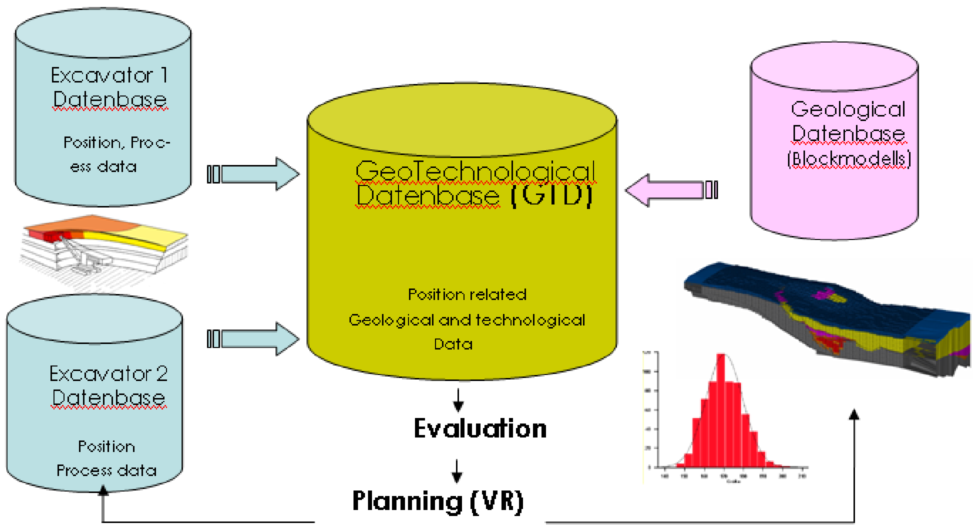

Figure 4 Concept of position based geological, technological and process database

Key elements for improving the performance and the efficiency of the mine production, as well as saving costs, are a high level of automation and the usage of computer-aided systems for data acquisition, data analysis, data processing and mine planning.

For the implication of automated planning and production control systems, suitable requirements have to be meet for data acquisition (e.g. measuring equipment), data transport (e.g. communication infrastructure) data analysis and data processing (e.g. data bases, mathematical models, user interfaces). In this case, data and information management are becoming more and more important. Therefore the data streams in complex systems have to be structured and categorized according to the control cycles.

For efficient and cost-effective mining process, the entire transportation chain from excavation, to transportation and dumping has to be analysed. In Fig. 5 an example is given for pre-cut in the Nochten lignite mine where two BWEs feeding one conveyor and one spreader dump the waste rock.

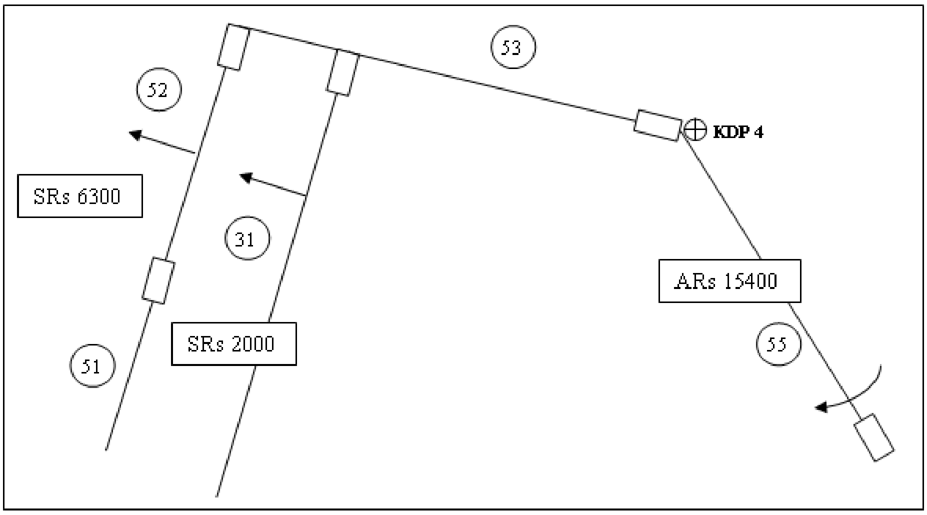

Figure 5 Schematic drawing of two BWEs feeding one conveyor [9]

The main goal in this kind of working scheme is to fully use the current capacity without overloading. Additionally, a certain material quality has to be send to the spreader in order to fulfil geotechnical and reclamation requirements. In order to do so the following prerequisites have to be met:

1. precise performance forecast and mass distribution,
2. measurement of each mass flow according to it's amount and rock characteristics,
3. accurate machinery positioning,
4. quick comparison of the positioning and the geological model for the determination of discrepancies in rock characteristics.

The main objectives of the concept are:

- efficient production control,
- automated production log,
- quality control,
- development of a planning tool/gathering of planning figures (e.g. expected machinery performance),
- display of performance losses → optimisation approaches,
- statistical data logging (e.g. distribution of the production rate → conveyor load and dimensioning).

4.3 Ergonomics / Training

If the model of the excavator is available, further investigations are possible, e.g. the visibility of the working area of the equipment or the ergonomics of the control station.

Figure 6 shows the model of the control station in the control cabin of the bucket wheel excavator SRs 6300. On this model the ergonomic of the working place can be investigated. This is very important because the value of the excavator or another single mining equipment can rich several ten thousand $US and an operation with high productivity and without damages is necessary. Can the operator see all relevant information? The control instruments are within reach? Is the working place comfortable to work a shift with high attention?

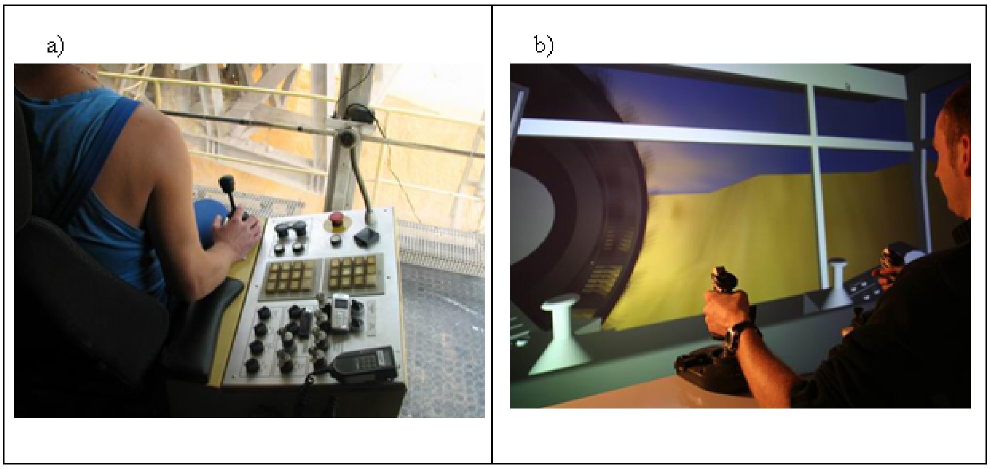

Figure 6 Visibility of the working area a) real situation b) virtual control station with simulation of the excavation process

To train operators and students the virtual excavation model can be used widely. It is possible to simulate different geological, technical and technological situations and visualize in real time the results of decision making. This training is effective before the operator or engineer work under field conditions. They understand the possibilities and boundaries of the equipment to optimize the process. Figure 7 shows a 3D projection of the excavator with possibility of interaction.

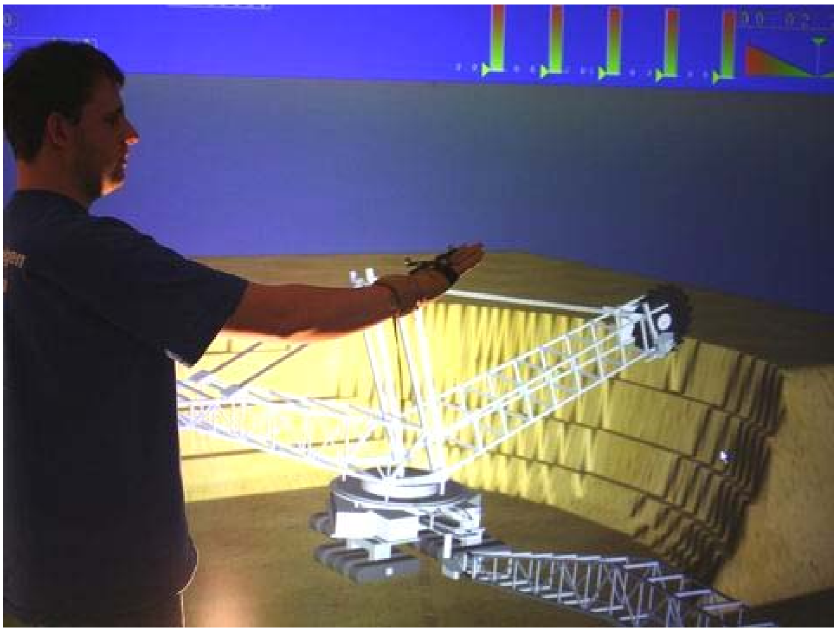

Figure 7 Virtual 3D projection of a bucket wheel excavator with all real functions in real time modus

5 SUMMARY AND OUTLOOK

Regardless the fact that Virtual Reality is not widely used yet, it becomes more and more important for optimizing processes and therefore cost saving. In the present paper principal applications of the VR in the mining industry were discussed. As a first example a simulation of a bucket wheel excavator was presented which allows the user to calculate the output for random block and cutting parameters.

In order to achieve an efficient and cost-saving mining process an interdisciplinary modular concept for a dynamic mine planning and production control is presented.

The connection between the modules geologic model, mine planning and production control is made by using virtual reality to establish a self-learning data acquisition and planning system. The results are being visualized to improve transparency and control the results.

A sufficient calibrated system can be used for detailed planning tasks. One of the most important components is the identification of the geologic structure which is strongly influencing the hole process and has to be considered as one of the main research realizing a self-learning system for dynamic mine planning and production control.

The next steps will be to enlarge the simulation with the other components of the transportation chain, such as other excavators, belt conveyors and spreaders, in order to analyse the influence of downstream equipment on the extraction and transportation process.

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MINE FIRES – SECURITY RISK FROM MINING

TUDOR GOLDAN*

Abstract: *Mine fires are much more common than most people realize. Most fires in underground mines are small and quickly put out. Disasters caused by mine fires are less frequent. Any mine fire could, however, become a major disaster if not quickly brought under control. Everyone who works underground in a mine should have a basic understanding of what fire is and how fires are best controlled. Knowledge of fire and the hazards of mine fires will encourage every underground worker to do his part to prevent fires. Workers must be trained to take the appropriate action if they discover a fire. The health and safety of workers cannot be left to chance.*

Keywords: *fires, risks, underground mine.*

1. INTRODUCTION

A number of accidents with severe consequences that have happened recently in coal mines of Romania make us again and again review the accident analysis using various methods and approaches/ Unfortunately the terms "risk analysis" and the related "risk assessment" are little known to our domestic experts. However, they have been used as a basis to develop reliability criteria and safety requirements for complicated ergatic systems.

Numeric safety analysis of complicated multifunctional systems was introduced in 1962 after disastrous accidents in four ICBM underground launching complexes in the USA. To all appearance, the time is ripe for practical realization of the above approaches to assess and standardize potential risks for such complicated ergatic systems as a coal mine, defining *risk as a probability of human casualties and material damages or losses*. Referring to various sources [1, 2], it is possible to say that our public, as well as abroad, are not anxious if the risk is 10-6/year and down, and therefore countermeasures to lower the level are seldom considered. Table 1 shows individual risks of premature fatality due to various causes.

Assuming the permissible fatal risk at 10^{-6}/year, as estimated by preliminary analysis, it should be noted that beyond the scope of this discussion are voluntary risks, for instance in mountaineering, that can be as high as 10^{-2}/year, since the values are in no way related to safety at a job site. It follows from the above that the approaches mentioned can be used to describe relevant situations in other industries, i.e. metallurgy, coking, though structurally, an emergency-contributing situation will be quite different.

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Table 1. Individual fatal risks values [2].

Accident causes and locations	Risk level per annum
Motor transport	$3\cdot10^{-4}$
Falls	$9\cdot10^{-5}$
Fire and burns	$4\cdot10^{-5}$
Drowning	$3\cdot10^{-5}$
Poisoning	$2\cdot10^{-5}$
Firearms	$1\cdot10^{-5}$
Machine tools	$1\cdot10^{-5}$
Water transport	$9\cdot10^{-6}$
Air transport	$9\cdot10^{-6}$
Falling objects	$6\cdot10^{-6}$
Electricity	$6\cdot10^{-6}$
Railway transport	$4\cdot10^{-6}$
Lightning	$5\cdot10^{-7}$
Tornado	$4\cdot10^{-7}$
Hurricane	$4\cdot10^{-7}$
Others	$4\cdot10^{-5}$
Nuclear accidents and disasters (100 reactors)	$2\cdot10^{-10}$

2. CLASSIFICATION OF MINING INDUSTRY RISKS

There are risks in all walks of life and in every activity we undertake. From getting up in the morning (for example, the risk of falling out of bed) or the ability to arrive to work on time (for example, the risk of heavy congestion) to more complex and perhaps higher value risks such as interest rate variations on ones mortgage. In the mining industry there are numerous risks that can be addressed and risk analysis benefits from clear and concise classifications. The risk in the mining sector has benefited from the following categorization [3]:

- geological risk;
- geotechnical risk;
- project risk;
- operational risk;
- environmental risk;
- security risk;
- market risk;
- macroeconomic risk;
- political risk;
- counterparty risk.

Geological risk generally depends on the probability of the actual ore body being different from that which was predicted in the business plan. Risks in this category relate to the inherent properties of the ore. The ore grade (Cu, Mo, Ag, Au, As etc.) could vary and so could the work index, which influences the grinding product line in the concentrator. Furthermore, humidity and CO_2 contents vary, which affect the recovery in the copper leaching process. In general, risk analysts also include the resource and reserve uncertainty risk in this category. This relates to the accuracy of the planned size of the ore base that the geologists measured. Commonly, there is greater risk associated with the size of the resource share of the ore body as opposed to the reserve share.

Geotechnical risk, on the other hand, typically involves slope failures in open pit mines and major collapses or rock bursts in underground mines. Such incidences affect the composition of the ore processed at a plant level.

Mining companies often engage in new projects in order to improve operations, increase output capacity or replace ore bodies. They can be carried out at the mine level (for example, greenfield deposit development), plant level (for example, increased treatment capacity or mill replacement) and at the smelter/refining level (for example, emissions reduction at the smelter). Project risk occurs whenever such projects are implemented. Especially, the result of variations from planned resource employment, involving time or money, are very relevant as deviation from budget could result in significant losses. For example, the cash flow could be seriously affected if the start-up of a relevant expansion is delayed or if the costs of such project were incorrectly estimated.

Operational risks result in variable production and recovery as a result of significant negative risks such as a train failure, poor shift management, lack of spare parts for haulage trucks, and so forth.

These risks are necessarily different from operational uncertainty; that is, output variability which relates to other risks such as geotechnical, geological or environmental risks.

Over the past few decades environmental risk has become more important as regulations have tightened and regulators have increased their monitoring and control of mining operations. Risks falling in this category relate to the compliance with environmental standards and regulations. Noncompliance could result in anything from a minor sanction (for example, a cash penalty) to a major shut down. The latter, especially, will have a considerable negative impact on shareholder value.

Environmental risks often strike the least suspecting and usually have a significant impact on value.

Market risks generally concern commodity prices, freight costs and production costs such as energy, treatment charges and refining charges. Although management's ability to influence market risk is highly restricted, the risk must be measured in order for management to make informed decisions. For example, the price of copper is relevant when a new or ongoing project is valued. An unsuspected drop in prices could be the reason for not going ahead with the project or foregoing additional investment to extend existing operations.

Political risk is the risk that the return on an investment suffers as a result of political instability or significant political adjustments in a country or region. Instability affecting investment returns could stem from a change of government, legislative bodies, foreign affairs and military control. Political risk is notoriously difficult to quantify because there are limited sample sizes and case studies when discussing individual nations. Nevertheless, some political risk can be insured against
through international agencies and other government organizations. Political risk can be so severe that investment returns suffer and even develop to such an extent that the capacity to withdraw capital from an investment is removed.

Counterparty risk is the risk of one or multiple parties failing to honor the commitments stipulated in a contract. This is commonly known as default risk. In the mining sector this generally refers to the commercialization of output, for example the inability of clients to make payments.

With this classification in mind, risk analysis can move on to a conceptual framework for risk adjusted decision-making.

3. SOURCES OF FIRE

Fire or burning is a form of rapid oxidation of a substance that produces much heat and light energy. The release of heat energy in a fire may be so rapid as to cause an explosion (a violent expansion of the gases produced).

Oxidation is the chemical reaction combining oxygen with another element or compound. This reaction is almost invariably accompanied by a release of heat energy (exothermic reaction). The amount of heat energy released depends on the oxidizing (burning) compounds. Among the hottest heat energy releases are those occurring when oxygen combines with carbon, hydrogen, or a compound of both elements.

If the chemical combination of carbon and oxygen is complete, carbon dioxide, a colourless gas, is produced. If hydrogen and oxygen combine, water vapour or steam is produced.

If the chemical combination includes both carbon and hydrogen and the reaction is complete, then carbon dioxide and water vapour are produced and the resulting smoke is white. If the combustion is incomplete, the products of combustion are carbon monoxide, carbon dioxide, water vapour, and particles of free carbon, and the resulting smoke is grey or black.

Heat, as energy, is a measure of molecular motion in a material. Because molecules are constantly moving, all matter contains some heat regardless of how low the temperature. The speed of the molecules increases when any matter is heated. Anything that sets the molecules of a substance in motion is producing heat in that substance. There are five general sources of heat energy:

- chemical;
- electrical;
- mechanical;
- solar;
- nuclear.

Chemical heat energy is rapid oxidation or combustion. Substances capable of oxidizing rapidly are known as combustibles. The most common of these substances contain significant amounts of carbon and hydrogen.

Sufficient heat for combustion is normally achieved when combustible material absorbs heat from an adjacent substance acting as a source of ignition. Some combustibles are capable of selfgenerating temperatures which increase to a point where ignition can occur. This is known as spontaneous ignition. While most organic or carbon-based substances do oxidize and release heat, this process is usually slow enough to dissipate the heat before combustion takes place. Spontaneous ignition occurs when combustion heat is not sufficiently dissipated.

Electrical energy can produce enough heat to start fires through arcing, dielectric heating, induction heating or through heat generated by resistance to the current flow. This last process may be intentional heating (e.g., filaments or heating elements) or accidental heating (e.g., electrical “shorts” or overloading).

Static electricity causes an arcing effect between a positively and a negatively charged body when frictional electricity becomes great enough so that a spark is discharged from body to body. This spark may not be hot enough or last long enough to ignite ordinary combustibles. However, it may ignite flammable liquid, vapour or gases.

Lightning has an action similar to that of static electricity. It occurs when one cloud arcs to the ground or to another cloud with an opposite charge. The magnitude of a lightning charge often generates sufficient heat to ignite combustible materials. The high amperage and high voltage potential, although of short duration, can do much structural damage even though fire may not occur.

One source of mechanical heat energy is friction or the resistance to motion of two bodies rubbing together. Another source is produced by the compression of gases. When a gas is compressed, its temperature increases. This can be demonstrated by pumping compressed air into a car tire or tube. As the pressure builds, the tube valve and pump fitting heat up. This can easily be felt by the hand.

In mines, a more common occurrence of mechanical heating can be found when the bearings seize or the brakes lock on a moving vehicle. Small fires from such sources are quite common.

The energy transmitted from the sun in the form of electromagnetic radiation is known as solar heat energy. Typically, solar energy is distributed fairly evenly over the face of the earth and, in itself, is not really capable of starting a fire. However, when solar energy is concentrated on a particular point, as through the use of a lens, it may ignite combustible materials.

The release of very large quantities of energy from the nucleus of an atom is known as nuclear heat energy. Nuclear heat energy can be released from the atom by two methods. Nuclear fission is the splitting of the nucleus of an atom. Nuclear fusion is the fusion of the nuclei of two atoms.

In reviewing the rapid oxidation process known as combustion, we note that three factors are necessary for a fire:

- a combustible material;
- the presence of oxygen or an oxidizing agent;
- enough heat to increase the temperature of the combustible material to its ignition temperature.

Fire burns in two ways:

- smoldering (surface);
- flaming combustion.

The smoldering (surface) mode of combustion is represented by the fire triangle (fuel, heat and oxygen). The flaming mode of combustion, such as the burning of logs in the fireplace, is represented by the fire tetrahedron (fuel, temperature, oxygen and the uninhibited chemical chain reaction).

These three factors, fuel, oxygen, and heat, have been incorporated into the simple fire triangle model (Figure 1).

Once combustion has begun, with ample fuel and oxygen, a fire can become self-supporting. As the fuel burns, it creates more heat. The increase in heat raises more fuel to its ignition temperature. As the need for more oxygen arises to support combustion, it is drawn into the fire zone. The oxygen, in turn, increases the heat and more fuel becomes involved. Combustion will continue as long as the factors from the three sides of the fire triangle are present.

While oxidation is speeding up to the combustion stage, another process is occurring that helps combustion. A chemical decomposition process occurs when a substance is exposed to heat. As chemical decomposition takes place, the substance emits vapours and gases that can form flammable mixtures with air at certain temperatures (pyrolysis).

This chain reaction and interaction continues until either all the fuel has been consumed, all the oxygen has been used up or the heat has been dissipated so that the temperature of the fuel drops below its ignition temperature. This, in essence, states the fundamental method of fire extinguishment – removal of one side of the triangle by:

Cooling: Cooling reduces the temperature of the fuel to below its ignition temperature.

One of the most common ways to put out fire is by cooling it with water. The process of extinguishing by cooling depends on cooling the fuel to a point where it does not produce sufficient vapour to burn. Solid and liquid fuels with high flash points can be extinguished by cooling. Low flash point liquids and flammable gases cannot be extinguished by cooling with water as vapour production cannot be reduced sufficiently. Lowering the temperature is dependent on the application of enough flow in proper form to establish a negative heat balance.

Smothering: Smothering is used to prevent oxygen from reaching the fire by:

- displacing the air with an inert gas;
- sealing the fire off within an inert blanket of foam;
- smothering the fire in some other way.

Extinguishment by oxygen dilution means reducing the oxygen concentration in the fire area. This can be done by introducing an inert gas into the fire or separating the oxygen and the fuel. This method of extinguishment will not work on self-oxidizing materials or on certain metals that are oxidized by carbon dioxide or nitrogen (the two most common extinguishing agents).

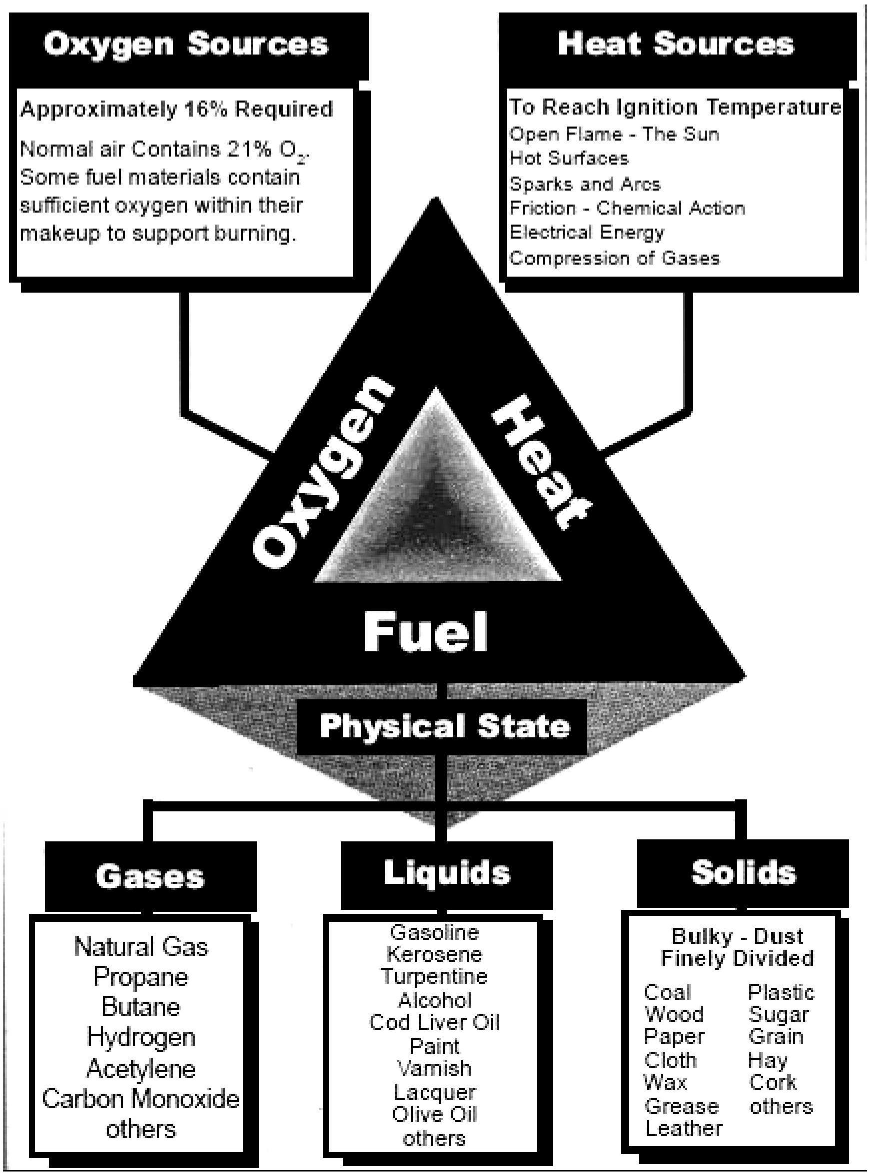

Figure 1. The fire triangle [1].

Separation: In some cases, a fire is effectively extinguished by removing the fuel source. This may be accomplished by stopping the flow of liquid or gaseous fuel, or by removing solid fuel in the path of the fire. Another method of fuel removal is to allow the fire to burn until all fuel is consumed.

In addition to the fire triangle, the fire tetrahedron is a four-sided figure, similar to a pyramid, with the four sides representing fuel, heat, oxygen and uninhibited chemical chain reaction (Figure 2).

There are many by-products from fire. These can include carbon monoxide (CO), carbon dioxide (CO_2) and sulphur dioxide (SO_2). The flammable by-products can combine with oxygen and burn, thus feeding the chemical chain reaction of combustion and contributing to the chain that expands the fire. The vapours that are produced in a fire may also be combustible and contribute to the fire.

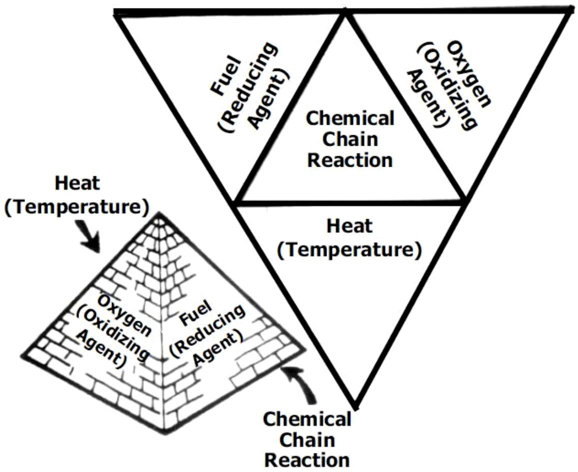

Figure 2. The fire tetrahedron.

The health hazard from exposure to the thermal decomposition (burning) process depends on the particular material involved and the decomposition temperature. These materials could include such things as tires, conveyor belting, electrical equipment and cables, styrofoam, brattice. Gases and smoke produced in fires involving material can be acutely toxic or severely irritating to the respiratory tract. Decomposition products may include hydrogen cyanide, hydrogen chloride,
aldehydes, nitrogen oxides, phosgene and heavy smoke (particulate).

Vapour density is the density of gas or vapour in relation to air. Vapour density is of concern with volatile liquids and gaseous fuels. Gases tend to assume the shape of their container, but have no specific volume. If a vapour is less dense than air (air has a vapour density of one), it will rise and tend to dissipate. If a gas or vapour is more dense than air, it will tend to hug the ground and travel, as directed, by terrain and wind.

It is important for all firefighters to know that every hydrocarbon except the lightest one, methane, has a vapors density greater than one and will sink and hug the ground, flowing into low lying areas. Hydrocarbons are very dangerous for that reason. Common gases such as ethane, propane and butane are examples of hydrocarbons that are heavier than air.

Smoke consists of gases and finely divided solids. It may be combustible and even explosive under some conditions (e.g., a sudden inrush of air from opening of a door). During a fire, smoke and gases rise, therefore air is more breathable closer to the floor.

Of the various gases associated with fire, you will probably be most concerned with carbon monoxide (CO), a product of incomplete combustion. Common usage of polyvinyl chloride (PVC), polyurethanes and plastics mean precautions may have to be taken for phosgene and hydrogen cyanide gas as well.

Most fires occurring underground are caused by the following:

- electricity;
- human intervention;
- manmade (deliberate or accidental);
- friction.

Electricity. Some mine fires are caused by the use or misuse of electricity on battery locomotives, power cables, trolley wires, motors, electric heaters and even light bulbs. Worn insulation on live wires is a common source of fires in mobile equipment. Overloaded electrical circuits can cause electrical cables to overheat.

Circuit breakers or fuses provide protection against overloaded electrical circuits, but if someone tampers with fuses or circuit breakers, then this protection is lost and overheating can take place. Electrical circuit protection devices are fire prevention devices and tampering with one can cause electrical cables or motors to burst into flames.

Other common causes of mine fires are the overfusing and shorting out of deteriorating wiring on vehicle control panels and faulty battery cables.

Manmade (deliberate or accidental). Welding and burning, and smoking and blasting operations, are among the many causes of fires. Strict control and patrol procedures must be observed whenever welding is done in any place where the welder may bring the three sides of the fire triangle together.

Active burning can be delayed for long periods of time by a slow smouldering or oxidation of wood started by the hot slag. An active fire can break out many hours after the hot work is finished.

Friction. Friction causes overheating of brake bands or clutches on slushers, transmission gear boxes and v-belt drives. Two of the most common friction-caused fires are the result of forgetting to release vehicle parking brakes and clutch slippage. Conveyor belts slipping, overheated bearings or rubbing against flammable items have also caused fires.

Spontaneous combustion. Spontaneous combustion occurs when ventilation is not sufficient to carry away the heat of oxidation. Slow oxidation of coal can generate enough heat to cause burning to start without any outside source of heat because the material is highly combustible. As oxidation starts, heat is produced which causes the oxidation to speed up which, in turn, creates more heat. This chain reaction eventually causes the material being oxidized to burst into flame.

This type of fire can best be understood by examining its three progressive phases. A firefighter may be confronted by one or all of the following phases of fire at any time. Knowledge of these phases is important for understanding ventilation and firefighting principles.

Stage 1: Incipient phase. In this first phase, the oxygen in the air has not been reduced significantly. The fire is producing water vapour, carbon dioxide, sulphur dioxide, carbon monoxide and other gases. Heat is being generated and the amount will increase as the fire progresses. Although the temperature in the area may be only slightly increased, the fire may be producing a flame temperature in excess of 537°C [4]. Incipient fires generate heat, smoke and flame damage.

Stage 2: Free-burning phase (steady state burning phase). During the second phase of burning, oxygen-rich air (+16.25% oxygen) is drawn into the flame as convection carries the heat to the top of the enclosed area. From the top downward, the heated gases expand laterally, forcing the cooler air to lower levels and eventually igniting the combustible material in the upper levels of the area. The first indication of a fire may be the discovery of smoke in air currents at some distance away or even on surface.

Stage 3: Too hot to proceed phase. In the third phase, there may be no detectable flame if the area is sufficiently airtight. Burning is reduced to glowing embers. The area becomes completely filled with dense smoke and gas. Smoke and gas may be forced by pressure through any openings and cracks. The fire will continue to smolder at a temperature well over 537ºC [4].

Stage 4: Out of control phase. It is not always possible to control a mine fire by conventional methods. This condition is called the fourth stage. A fire in this stage can only be controlled by sealing it off on the surface.

CONCLUSIONS

Fires constantly produce deadly gases. Workers must not be exposed to these gases or other hazards associated with fires, such as explosions, weakening timber or deteriorating ground.

Every fire, regardless of how small, must be reported at once because it may have released deadly gases into the mine's air. Once put out, the fire area must be watched until re-ignition is impossible.

After a fire extinguisher is used, it must always be returned for recharging and its use recorded.

Any unusual occurrences in the mine should be noted and reported at once. An unusual occurrence could be: the odours of smoke or other contaminants, clouds of dust, sudden changes in ventilation, interruption of normal services such as power failures, unusual noises or explosions.

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IMPACT OF SMOKE AND CO TO ESCAPING MINERS FROM UNDERGROUND MINE FIRES

TUDOR GOLDAN*

Abstract: *The smoke density and the physical irritants produced in underground fires pose a greater threat to escaping miners than the levels of CO and other gases, which do not reach toxic levels when the critical optical density is reached. In the mine fires, miners could have been in danger had CO levels been high enough. Air loss is due to a variety of reasons, including frictional resistance in the air courses and leakage across stopping and overcasts. In a mine fire, air leakage across ventilation devices can result in significant amounts of smoke making its way into escapeways and other entries.*

Keywords: *coal mines, fires, smoke, underground*.

1. INTRODUCTION

An underground coal mine is a sociotechnical system, with workers and machines organized in particular ways during production. The physical environment of an operation is a powerful factor in the work life of miners.

A coal mine is a complex system. It is defined as all parts of a mining plant's property (both underground and surface) that contribute, under one management, to the extraction of coal. As suggested, many functions that must be carried out at an operation are only indirectly related to coal mining and processing. Even the jobs that are directly related tend to be numerous and varied.

Long-range planning is needed to ensure that the mine producers coal in a cost-effective manner. During long-range planning there is a general focus on such essentials as equipment type, deployment, utilization, and haulage. Laying out a mine also involves auxiliary factors including ventilation arrangements, roof and support plans, power distribution, and communications. All of these planned systems are incorporated into a "projection map" that is developed by a team of technical specialists. This team will include, at various times, mining engineers, electrical engineers, industrial engineers, and company geologists, among others.

To extract coal, miners must operate large machines under extremely conditions. Out by support personnel are scattered through the labyrinth of underground entries. They are needed to help maintain the many auxiliary subsystems found in the mine. Work done by these miners includes building and maintaining air stopping, installing supplemental roof supports,

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cleaning coal spills around or under conveyor haulage belts, moving supplies, maintaining electrical installations, and conducting hazard inspections. Generally, these support workers do their tasks singly or in small crews, usually without direct contact with other miners, supervisors, or the outside world. They also have to deal with poor footing due to uneven or muddy bottom. In sum, all miners must do their jobs in an environment that is harsh and potentially dangerous.

Increased mechanization and the introduction of greater numbers of electrical machines have resulted in mine fires being ranked just behind explosions as a major cause of mine disasters.

2. SMOKE AND VISIBILITY

In general, smoke consists of hundreds of thousands of very small particles. These particles have some "size", usually expressed in terms of their diameters, and they have some concentrations, usually expressed either in the number of these particles per unit volume or the total mass of the particles per unit volume.

Humans cannot see individual smoke particles because they are too small. Similarly, "umber concentrations" and "mass concentrations" of smoke particles do not have much meaning to people unless they are trained technically.

The eye is only sensitive to light in the wavelength region from about 400 nm to about 700 nm. The maximum sensitivity of the human eye is to light that has a wavelength of about 555 nm. It is important to know how the eye responds to light because if its response as the eye. Such a detector can then be used to quantify the visible characteristics of smoke because it responds in the same manner as the human eye.

Smoke is visible because it either scatters or attenuates (diminishes) light. In some instances, smoke is visible because the smoke particles reflect light which is then detected by the eye. The eye actually "sees" an intensity of light that has been reflected from a cloud of smoke particles. Imagine shining a flashlight into a cloud of smoke. Someone off to the side can actually "see" the beam of light as it traverses the smoke cloud. This is called scattering. Smoke is also visible because it attenuates light. Imagine having someone shine a flashlight into your eyes. As smoke begins to build up along the beam of the flashlight, the light begins to dim. The smoke is visible because it is now reducing the intensity of light that upon the eye. As the smoke level increases, it is said to be 100%. In other words, none of the light energy from the flashlight makes its way through the cloud of smoke. Another way of saying that the obscuration is 100% is to say that the transmission of light through the cloud is zero.

Although it is possible to measure the light that is scattered by smoke, most studies usually measure the amount of light that is transmitted through a cloud of smoke. There are three basic reasons for measuring light transmissions rather than the amount of light that is scattered. First, the intensity of scattered light depends on many factors, such as the size of the smoke particles, the angle at which one measures the scattered light in the space between the beam and the light detector. Second, the amount that smoke obscures light is a direct measure of a visibility hazard. Obscuration by smoke is one hazard that is clearly evident in mines. Imagine a 100-watt light bulb 3 m away. If the smoke is dense enough so that the effective power of the bulb is only 1 watt, then the obscuration would be 99%. If the cloud of smoke is so dense that obscuration is total, then it becomes impossible to see. Smoke from an unwanted fire that reaches this level of obscuration represents a critical, life-threatening situation because it becomes impossible to use one's eye to escape. Finally, the measurement of light transmission allows for characterization of smoke by single parameter. This parameter is called the "optical density" and is derived from the amount of light that is transmitted (T), at a given intensity, through a smoke cloud over some path length (L):

$$D = \frac{1}{L} log\left(\frac{1}{T}\right) \quad (1)$$

Optical density is used to assess hazards of smoke and levels of detectability.

The chemical composition of smoke particles depends, in part, on the material that is burning. Some materials may produce gas, or gases that attach to smoke particles, which can cause the eye to tear, even at moderate levels of obscuration. Smoke from a fire is also breathed into the lungs, where some of the smoke is deposited before it can be exhaled. The smoke and its chemical composition can irritate the respiratory system and also contain elevated levels of toxic gases or compounds that attach to the smoke particles. All of these effects are difficult, if not impossible, to quantify because of the many combustibles that can burn and produce adverse effects.

Several studies [6] have been conducted to assess the effect of smoke on humans, especially with regard to ability to escape from smoke-filled environments. Results of these studies indicated that, for the general public, most individuals began to experience emotional effects when the smoke optical density reached 0.044 m^{-1}. In contrast, most subjects in the group of researchers began to show emotional fluctuation at smoke densities of 0.15 to 0.24 m^{-1}. Studies concluded that:

- For a person unfamiliar with the escapeways and exits, that individual's ability to escape safely from a fire is severely reduced when the smoke optical density exceeds 0.066 m^{-1}.
- If an individual is familiar with the escapeways and exits of a building, that person's ability to escape safely is severely reduced when the smoke optical density exceeds 0.22 m^{-1}.

During these tests, the levels of CO were continuously measured, reaching a peak value of a 50 ppm at the end of each test (D=0.305 m^{-1}), which equates to an optical density/CO ratio (D/CO) of 6.10×10^{-3} ppm·m^{-1}. At these levels of optical density, smoke obscuration is severe enough to reduce visibility to near zero levels. For instance, at D=0.066 m^{-1}, the range of visibility is about 13 m while at D=0.22 m^{-1}, it is approximately 4 m. Because of this, authors referred to these optical densities as critical values at which the smoke becomes untenable due to the total impact of the smoke on the human response, which includes reduction in visibility and other physiological and psychological effects.

Heyn [5] obtained similar results when measuring the relation between smoke density and visibility at the Tremonia Experimental Mine in Germany. For these experiments, Heyn conducted tests using small conveyor belt fires which resulted in visibilities of only a few decimetres.

Jin [6] concluded that a person's ability to escape by an unfamiliar route is severely reduced when the smoke optical density equates to a sight distance of about 13 m. Persons familiar with escapeways have their ability hampered when the smoke optical density equates to about 4 m of visibility. Based on these conclusions, it is reasonable to expect that miners who were not familiar with their escapeways would have been at a disadvantage compared to those who knew the travel routes.

In Jin's experiment, critical levels of optical density were measured for wood smoke. Depending on the actual material burning and the resultant characteristics (both physical and chemical) of the smoke produced, these critical values could increase or decrease. Clearly, a critical level of optical density at which the range of visibility is reduced to 1 m represents an upper limit. The range of visibility is defined as the distance at which the light obscuration exceeds 86% (or, the transmission is less than 0.14). At a 1 m visibility range, the critical level of optical density is 0.92 m^{-1}. This value should be considered as an absolute maximum value based solely on reduced visibility.

Table 1. Visibility as a function of CO level in smoldering fires [8].

Combustible	**ppm CO** at D=0.22 m^{-1}	**ppm CO** at D=0.92 m^{-1}
Wood	36.0	150.0
Coal	9.0	38.0
SBR conveyor belt	2.5	10.5
PVC conveyor belt	3.0	12.5
Neoprene conveyor	7.0	29.0
PVC line brattice	11.0	46.0

Table 2. Visibility as a function of CO level in flaming fires [8].

Combustible	**ppm CO** at D=0.22 m^{-1}	**ppm CO** at D=0.92 m^{-1}
Wood	56.0	234.0
Coal	17.0	71.0
SBR conveyor belt	19.0	79.0
PVC conveyor belt	42.0	176.0
Neoprene conveyor	32.0	143.0
Transformer fluid	7.5	31.0

Using the data of Litton for the ratio of CO to smoke optical density [8], it is interesting to compare the expected levels of CO that would be present at the values of optical density equal to 0.22 m^{-1} (the maximum critical level reported by Jin) and 0.92 m^{-1} (the absolute maximum), discussed above. These levels are shown in table 1 for smoldering fires and in table 2 for flaming fires.

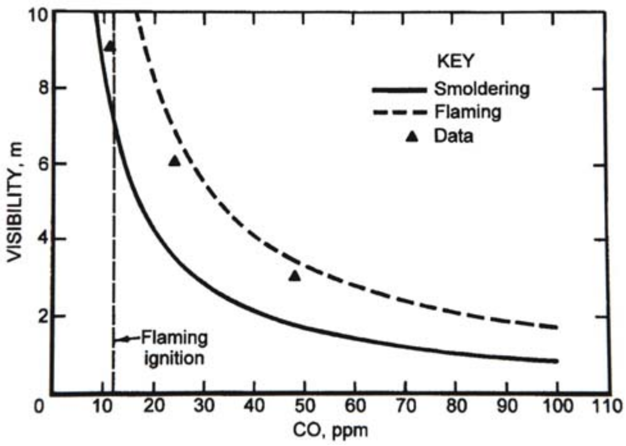

Figure 1. Visibility measured as a function of CO level.

Figure 1 indicates the visibility (in meters) measured as a function of the CO level (in parts per million). The solid line indicates the level of visibility predicted from smoldering coal fires, while the dashed line indicates the level of visibility predicted from flaming coal and styrene-butadiene belt fires. It is important to note that during the large-scale experiments, the initial levels of CO and smoke come from a smoldering coal fire while the later levels come from a flaming coal and conveyor belt fire. In figure 1, the level of CO at which the coal fire ceases to smoulder and begins to flame is indicated by the arrow. The importance of these data is apparent: significant reductions in visibility occur at relatively low levels of CO (10-20 ppm).

Depending on the material burning, other toxic and irritant elements can be produced. For conveyor belts, in particular, the generation of HCl vapours due to chlorine in the belt, as either a component of the base polymer or as an additive to make the belt more flame-resistant, is an example of such an irritant and also represents a potential toxic hazard in addition to the CO produced.

3. JUDGMENT AND DECISION-MAKING MODEL IN MINE FIRES

The notion of a model is introduced because growing research interest in the subjective aspects of group and individual behaviour has led to a debate over whether judgment is a skill that can be understood scientifically. A related point of contention is whether such an understanding could lead to the development of methods for estimating people's ability to make good decisions during an emergency. There is some literature that supports the potential usefulness of this approach. However, little agreement seems to have been reached on how to define and operationalize even those basic concepts necessary to assess the soundness of decisions from within their environmental and group contexts [1, 2].

From a cognitive perspective, any person engaged in decision making (either alone or in a group) is actively involved in a process characterized by certain elements. These is: (1) detection of a problem, (2) definition or diagnosis, (3) consideration of available options, (4) choice of what is perceived as the best option given recognized needs, and (5) execution of the choice based on what has transpired [1]. At any moment in this process, there are factors at play that have a large impact on one's ability to solve complex problems in a limited time: (1) an internal state is the sum of a person's psychomotor skills, knowledge, attitudes, etc.; (2) uncertainty is caused by faulty or incomplete information received from the external environment; (3) stress is generated both by the problem at hand and any background problem that may exist; (4) complexity, as it is used here, refers to the number of elements involved that must be attended to. These variables are depicted in figure 2, and their relationship to each other and to an outcome is indicated. This schema is designed to suggest interaction, because while the judgment and decision-making process may be conceptualized as discrete stages, experience tells us that this is not the way people function in real-world situations.

The interactive model reflects underlying demands on decision-maker in most life or death situations.

Throughout each episode, workers engaged in an ongoing series of activities, some of which seem to have been well thought out and others that (in hindsight) do not seem so logical. Yet, all the while, they were attempting to solve the problem that confronted them. Such behaviour is in line with much of the recent literature dealing with human actions in fires, which advances the argument that people engage in adaptive behaviour based on choices made from among those perceived to be available at any particular time during the occurrence.

People seem to exercise judgment and make decisions during a fire, although they oftentimes fail to perceive the fundamental problem adequately. This is especially true if they are focused on a task, or are having some type of difficulty. Unfortunately for those interested in reaching a more objective understanding of the quality of those decisions, choices are usually judged ex post facto depending on their outcomes. Accordingly, if a person survives, he or she is credited with making sufficient correct decisions and little attention is paid to poor choices; if a victim dies, most second-guessing focuses on what he or she might have done wrong and there is not much analysis of any good decisions that were made.

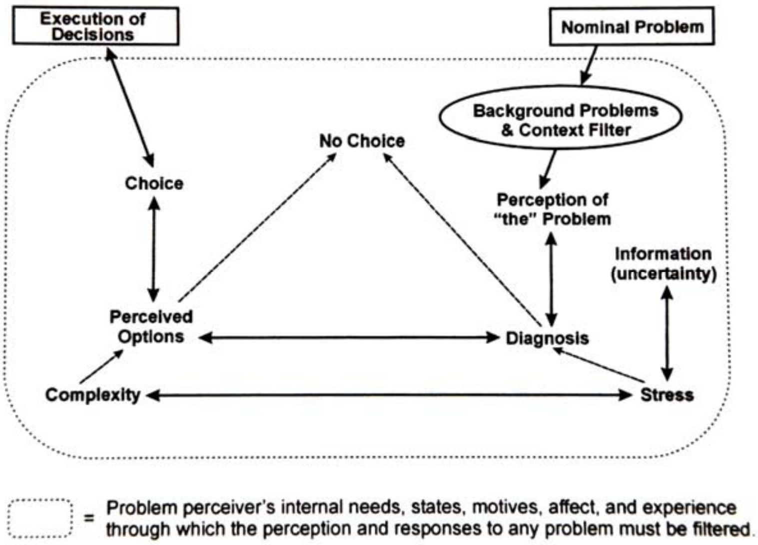

Figure 2. A model of judgment and decision-making.

After completing the diagnosis of a problem, a person must decide which actions, if any, must be taken. This part of the decision-making process calls for recognizing and evaluating available options and then choosing an action that is determined to be best given the circumstances. A number of variables impact a person's perception of particular choices and their appropriateness to his or her situation. Analyses of decision-making therefore must focus not only on the objective outcome of each action, but also (and perhaps more importantly) on choices that were made given the impact of elements influencing the decision-maker.

CONCLUSIONS

Research on human behaviour in smoke has shown that (1) people not familiar with escapeways tend to experience higher levels of emotional instability, and their ability to escape from a fire is severely reduced when the visibility falls below 13 m and (2) subjects familiar with escapeways experience relatively more problems with physiological effects of smoke, and their escape ability becomes hampered when the visibility falls below 4 m. Fire research data indicate that smoke reaches levels of untenability significantly earlier than it takes the fire to generate a toxic environment due to its product gases. It is only when the levels of smoke begin to totally obscure visibility that the toxicity of the combustion products begins to play a role in the question of escape and survivability.

The psychological effects of smoke, in some instances, inhibited workers' ability to think clearly, make correct choices, and take proper action during their escape. In addition to suffering emotional upset during their escape, a number of miners also experienced some physiological effects of smoke, including smoke inhalation and eye irritation. In short, miners' ordeals in smoke when escaping mine fires confirm the conclusions.

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ASPECTS REGARDING THE GENERATION AND MANAGEMENT OF THE EXTRACTIVE INDUSTRY WASTE

MIHAELA PETRE*

Abstract: *A major environment problem in Romania is the management of waste according to the international policies in the field, in a manner in which, they must not affect the environment or the health of the population. The present paper shows aspects related to the generating of industrial waste and more exactly the generating of extractive industry waste, in Romania, between the years 2003-2006. The waste generated by the extractive industry represent 75% of the generated total, from which we can count more than 99.5% as being non-hazardous (inert) waste and the rest, represented by 0.5%, as being hazardous waste. The non-hazardous industrial wastes resulted from the extractive industry are almost totally eliminated by storing them in dumping sites and mud-setting ponds. The impact of stored industrial waste on the environment, resides first of all in soil pollution, depth water and surface water pollution, the removal from the agricultural circuit of some wide fertile terrains. The primary objectives of wastes management, according to the most recent European Union Directives are the prevention and reduction of waste production and of their injuriousness degree, their material capitalization, with their eventual transformation in secondary raw material or their utilization as an energy source. It is hoped that future times will bring reduced industrial waste quantities, with the gradual employment of "clean technologies", and that the principles of pollution prevention, reduction and integrated control, will be applied.*

Key words: *industrial waste, pollution, elimination, storage, management.*

1. INTRODUCTION

Of all the environment problems that humanity finds itself against today, a major one is surely considered that of the wastes. In our society, waste generation is an inevitable consequence of human industrial and domestic activities. The European Union produces yearly 3.5 tones of waste per inhabitant and despite the present recycling techniques, this quantity increases [1]. The efficient management of residual waste constitutes the key-aspect of every environment policy. At this moment there are over 20 European directives and regulations regarding waste management.

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Romania has concentrated its efforts in the field of waste management, consisting in the transposition and implementation of the European standards and legislation as national laws. According with the EU legislation requirements, the national strategic documents on waste management are made of two main components, that is to say [2]:

- *The National Waste Management Strategy* (NWMS) - is the organism that regulates the objectives of the Romanian government in the field of wastes management. This has been elaborated for the period between the years 2003-2013, and it would be periodically revised, in accordance with technical progress and with the environment protection requirements.

- *The National Waste Management Plan* (NWMP) - represents the Strategy implementation plan - it contains details regarding the actions that must be enacted in order to meet the objectives of the strategy, the deployment of these actions, including deadlines and responsibilities. The *National Waste Management Plan* is renewed and revised once every five years.

The National Waste Management Plan is applied for all types of residual waste, solid or liquid, as it follows:

- public waste (domestic and resulted from commerce, institutions and public services),
- mud resulted from the public water treatment plants,
- waste resulted from demolitions and constructions,
- non-hazardous and hazardous production industrial wastes.

The Regional Waste Management Plans (RWMP) has been elaborated by each Regional Environment Protection Agency in cooperation with the representatives of the local and county environment protection authorities. The regional waste management plans are revised once every five years or when it is considered to be necessary [2].

The purpose of the regional waste management plans is: creating the necessary conditions for meeting the demands of waste management objectives.

In order to achieve the European and national objectives in the field of waste management, it is necessary that society involves itself, members represented by local and central public authorities, waste generators, specialized associations, research institutes and of course, the civil society.

Waste management contains all activities of collecting, transportation, treating, capitalization and waste elimination. The responsibility of waste management activities belongs to the generators, according to the principle "he who pollutes, pays", or to the producers, according to the principle "the responsibility of the producer".

The purpose of this paper is to present the aspects related to the generating, capitalization, elimination and management of industrial wastes in Romania, between the years 2003-2006, as well as presenting the objectives and primary measures that should be applied in the field. The industrial wastes dumps affect the environment through pollution of the soil, depth and surface waters, the removal of some wide fertile terrains from the agricultural circuit and through their unpleasant visual impact. At the end of this paper, we will show the main objectives of every European country in the field of industrial waste management.

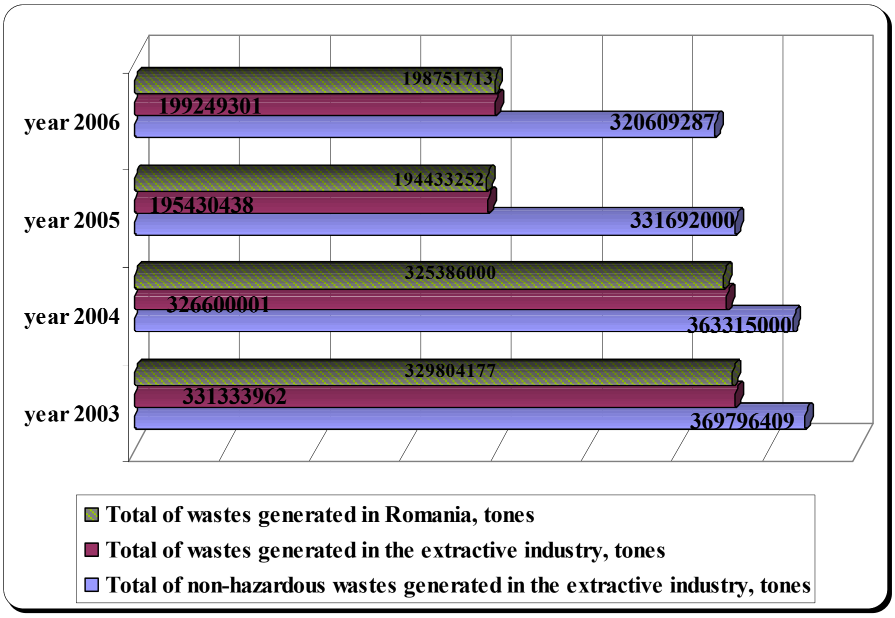
2. THE GENERATED WASTE IN ROMANIA

In Romania, large quantities of waste are generated every year, these representing one of the most important problems linked to environment protection. The waste management activities can become a potential risk for the ambient, because the different management methods imply releasing some polluting agents into the environment. The inadequate waste management leads to numerous soil and depth water contamination, thus threatening the human health.

According to the *Regional Waste Management Plans*, the categories of wastes in Romania refers to the ones generated from: the extractive industry, other industrial activities (the extractive industry exclusive) and municipal waste.

The extractive industry includes the extraction of the minerals existing in nature as solids (coal and ores), liquids (petroleum) or gaseous (natural gases). The extraction process can be achieved through the process of subterraneous of surface mining, or by well exploiting. This economic sector includes supplementary activities necessary for the transportation and commercialization of the mineral products, for example: ore crushing, triturating, cleaning, drying, sorting, and concentrating the ores, natural gas liquefaction and also congestion of solid fuels.

According to the data presented in statistics, in the period 2003-2006, the residual wastes generated from the extractive industry represents, as average, 75% of the generated wastes in Romania. Figure 1 shows the quantities of non-hazardous waste generated from the activities performed in the extractive industry, as compared to the total of waste generated in this economical sector and the total of residual waste generated in Romania in the period 2003-2006.



Source: Processed data from Statistical Publications regarding the generation and wastes management and from *The Regional Waste Management Plans* [2]

Fig.1. The total quantities of waste generated in Romania in the period 2003-2006.

According to the data presented in figure 1, the wastes generated from the extractive industry represent, 99.5% non-hazardous waste (inert), the rest, of 0.5% being hazardous waste.

In the year 2004, the quantity of waste generated in Romania has reached a total of 363.315 thousand tones, from which, 99.4% are non-hazardous and 0.6% are hazardous wastes. The quantity of waste generated in Romania in the year 2005, has reached a total of 331.692 thousand tones, from which, 99.5% are non-hazardous. The hazardous wastes generated represent approximately 0.5% of the total quantity of wastes. In the year 2006, the quantity of waste generated in Romania has reached a total of 320.609 thousand tones, from which, 99.7%

are non-hazardous. The hazardous wastes generated, represent approximately 0.3% of the total quantity of wastes [2].

2.1 Aspects regarding generating industrial wastes

After 1990, Romania has started a long industrial transformation process which has directly influenced the total of generated industrial wastes, as well as, their composition. But, even as we speak, the effects of inadequate resource management in the communist regime are felt, effects that have contributed to excessive levels of pollution, as well as to the generation of an enormous amount of industrial wastes. The transition from the central planning to free enterprise has introduced complex macroeconomic reforms, which have generated positive changes in the structure of the industry branch with direct consequences in the decrease of the generated wastes level.

Some industry branches are responsible for the generating of the largest amount of industrial waste. Here, we can name: coal extraction and processing, ore extraction and processing, energy production, chemical industry, metallurgy etc.

The wastes generated from the extractive industry are non-hazardous and hazardous. The non-hazardous ones are generated from the processes of: coal extraction and processing, hydrocarbons extraction, metalliferous ore extraction and processing and other extractive activities. Non-hazardous residual waste are generated also from other economical activities, such as: the processing industry, production, the electrical and thermal energy transport and distribution, gases and water, water catching, treatment and distribution, construction sites etc.

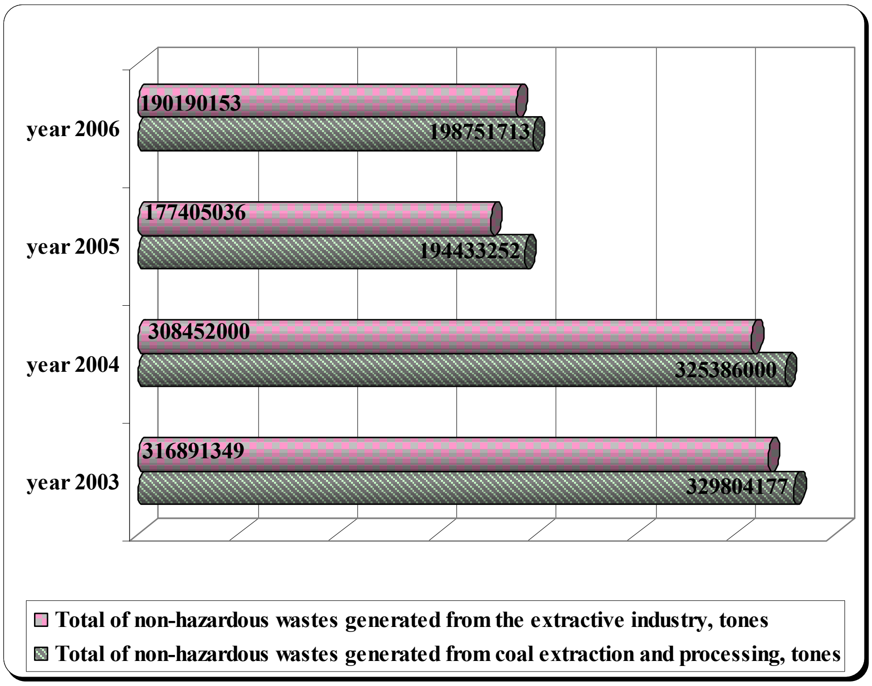
The economical activities that generate hazardous wastes are: the extractive industry, wood and wood related products processing, petroleum processing industry, coal carbonization, manufacturing of substances and chemical products, metallurgy, metal constructions and metal constructions related products industry, machines and equipments industry, optical and electrical equipment industry, automotive industry etc. Coal extraction and processing is the industrial activity from which the largest amount of wastes is generated. In table 1 we can observe the percentage of generated wastes from coal extraction and processing from the total amount of generated residual waste in the period 2003-2006.

Table 1. The statistical data by quantities of industrial waste generated in the period 2003-2006, in Romania.

Year	**2003**	**2004**	**2005**	**2006**
Total of wastes generated in Romania, tones	369796409	363315000	331692000	320609287
Total of wastes generated from the extractive industry, tones	331333962	326600001	195430438	199249301
Total of wastes generated from coal extraction and processing, tones	316891570	308452000	177405257	190190210
The percentage of wastes generated from coal extraction and processing, from the total amount of wastes generated in Romania, %.	85.69	84.90	53.48	59.32
The percentage of wastes generated from coal extraction and processing, from the total amount of wastes generated from the extractive industry %.	95.64	94.44	90.78	95.45

Source: Processed data from Statistical Publications regarding the generation and wastes management and from *The Regional Waste Management Plans* [2]

Fortunately, if one may say so, the highest percentage of generated wastes from coal extraction and processing are almost totally non-hazardous, inert waste, as it can be seen in the graphically depicted data in figure 2.



Source: Processed data from Statistical Publications regarding the generation and wastes management and from *The Regional Waste Management Plans* [2]

Fig. 2. The total of quantities of non-hazardous waste generated from coal extraction and processing and from the extractive industry in Romania, in the period 2003-2006

In the year 2003, the most important quantities of production wastes generated from the extractive industry have been registered in Gorj, Vâlcea and Hunedoara counties. In the south-west Oltenia development region (which includes Gorj, Vâlcea and Dolj counties), the industrial wastes generated from the extractive industry (represented mostly by coal extraction and processing), raised more than 94% from the total quantity of regenerated residual waste, resulted from the entire extractive industry branch in Romania [2].

In 2004, in the field of extractive industry, coal extraction and processing activities have generated 94.4% of the total wastes amount. The other activities that have generated large quantity of industrial waste were: the processing industry; production, the electrical and thermal energy transport and distribution, gases and water; water catching, treatment and distribution, construction sites [2].

The extractive industry generated in 2005 almost 60% from the total amount of non-hazardous wastes generated in Romania, while the coal extraction and processing activities generated more than 90% from these [2].

In 2006, in the field of extractive industry was generated the highest amount of non-hazardous industrial wastes, approximately 62% from the total amount of non-hazardous wastes generated in Romania, from which the coal extraction and processing represented more than 95% [2].

2.2 The capitalization and elimination of industrial wastes

The industrial waste, represent, on account of their composition, extremely high risk levels for the flora and wildlife. A lot of the present collecting, transport, treatment and storage techniques, as well as the existence of industrial waste "historical depots", have a strong negative impact on the environment and on the public health.

The largest amount of industrial waste is eliminated through the process of storage. The impact of industrial wastes depots on the ambient surroundings resides in the release of toxic chemical substances in the environment, substances existing in the waste or formed during the storage process, in the release of dust and powders resulted from the activities developed in the storage depot.

The responsibility for the industrial wastes belongs to the producer but it can be transferred to a waste operator. This way, the dumpsites can be set "in situ", near the waste generating installations, on the producer's property or "ex situ", the storage depot belonging to the operator in charge with its disposal [3]. At this time, in Romania, there aren't any central stations for wastes storage, and in this manner the producer is obliged to build his own site, for which the building, operating, shutdown and post-shutdown costs exceed his financial capacity.

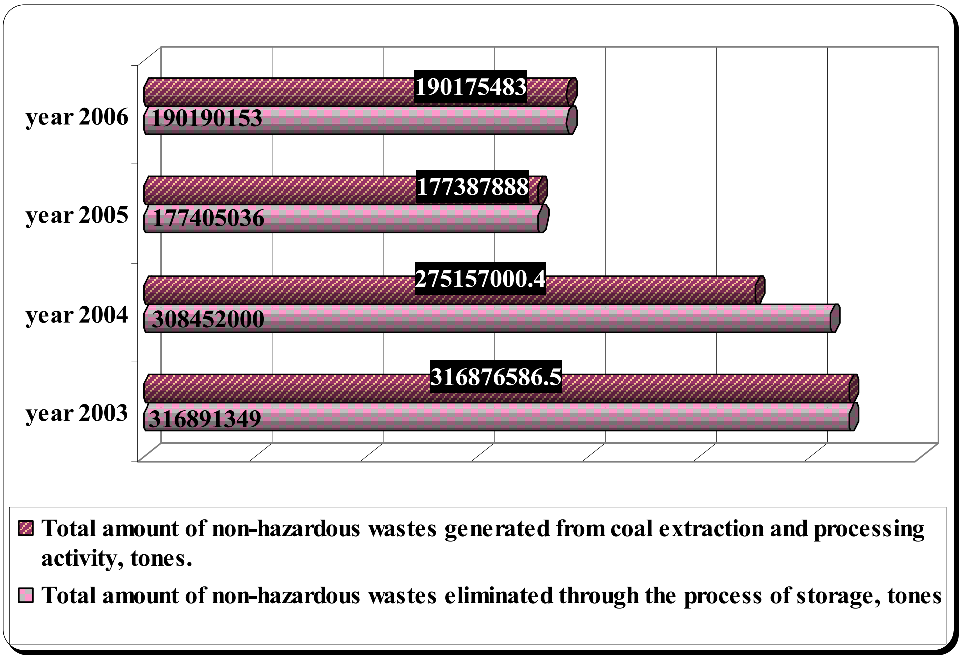
The industrial waste resulted from the extractive industry are almost totally eliminated through the process of storing them in dumpsites and mud-setting ponds, 99% of them being non-hazardous, inert wastes (table 2).

Table 2. The statistical data by wastes generated and capitalized from coal extraction and processing, in the period 2003-2006, in Romania

Year	**2003**	**2004**	**2005**	**2006**
Total of non-hazardous wastes generated from coal extraction and processing, tones	316891349	308452000	177405036	190190153
Total of wastes capitalized derived from coal extraction and processing (hazardous and non-hazardous), tones	14936.32	5000.4	17148	16990
The percentage of non-hazardous wastes generated from coal extraction and processing, eliminated through the process of storage, %.	99.99	89.21	99.99	99.99

Source: Processed data from Statistical Publications regarding the generation and wastes management and from *The Regional Waste Management Plans* [2]

The quantities of non-hazardous wastes generated from coal extraction and processing that are eliminated through the process of storage, in the period 2003-2006, can be observed from the graphically presented data in figure 3.



Source: Processed data from Statistical Publications regarding the generation and wastes management and from *The Regional Waste Management Plans* [2]

Fig. 3. The quantities of non-hazardous waste eliminated by storage compared with the quantities generated from coal extraction and processing activity, in the period 2003-2006

A dirt-heap is the surface storage place and the picked-off dirt extracted from the mine or the dirt resulted from mechanical processing. The dirt-heaps contain a large variety of bedrocks and soil depending on the geology and mine type. Formally, the wastes stored in dirt-heaps can be categorized, as it follows [4]:

- useless material and rocks eliminated from the surface quarries;
- useless material and rocks from the subterraneous mines, fruitless ores that are not processed;
- processing wastes – waste dried in the initial processing, like small or large material, coarse material from the watering process, other wastes.

The quantity and characteristics of the wastes stored in the dirt-heaps can vary according to the exploited ore type and to the specific way of the process development. A problem of great importance is constituted by the stability of the dirt-heaps, taking into account their design characteristics, the type of the stored materials and specially, the fact that, in some cases, people store on the same dirt-heaps, different types of wastes, resulted from the respective economical unit, besides the ones that came from mining activities.

The mud-setting ponds constitute "on earth" building structures (normally, no excavations are necessary, but the natural characteristics of the earth surface are used). These are used in order to separate the water sedentary fraction, used in technological processes. After the process of waste water clarification, the water is recycled of it is cleansed in view of its evacuation. These mud-setting ponds are usually filled with process waste (from the process of gravitational separation, flotation etc). In some cases, these mud-setting ponds are also used for the release of other types of waters that do not come directly from the production processes (draining waters, waters collected from mountains).

The quantity and characteristics of the wastes downloaded in the mud-setting ponds varies according to the exploited ore type and according to the special way of activity development. The solid fraction contains inert material, salts and other metal compounds. The liquid fraction contains different types of dissolved chemical substances, among which, some are toxic (for example: cyanides resulted from the processing of valuable metals) or can have a negative impact on the environment (for example: pH acid or base, with a sulfate and carbonate content) [4].

The storage of wastes from mining activities in dirt-heaps and in mud-setting ponds generate different impact manners on the environment, from which, the most important are:

- the affecting of wide terrains that can not be used for other purposes for long periods of time;
- soil pollution, depth and surface water pollution with different solubilizated compounds through the influence of meteoric waters;
- in the case of exploiting sulphide containing materials, the phenomenon of acid drainage is amplified;
- the visual impact.

A very important problem is constituted by the risk of damage in the case of various installations related to the mud-setting ponds, risks that can be increased through the occurrence of unusual natural phenomena (earthquakes, floods, earth slides).

From a geographical point of view, the industrial waste depositories are placed close to industrial zones, quarries, mines etc.

In the year 2003, there have been inventoried a total of 21 mud-setting ponds currently in use. From this total of 21 mud-setting ponds in the extractive industry, 16 have ceased storage until to 31 December 2006, and in the case of 5 mud-setting ponds, transition periods have been obtained until 31 December 2011 the latest: 2 for conformation (Valea Sesei and Valea Stefancei) and 3 for ceasing (Baita Stei, Aurul Recea and Ostra – Valea Straja) [5].

Besides the impact on the environment, the storage of industrial waste also affects the landscape, creates a visual discomfort and removes the terrains on which they are deposited from the natural or economical circuit.

3. PERSPECTIVES AND PRIORITIES REGARDING THE INDUSTRIAL WASTES MANAGEMENT

In The European Union Adhesion Treaty, Romania took on the responsibility of correlating its environment politics to the one of the European Union and of acting according to its creed. For the wastes management sector, Romania held forth to achieve an integrated wastes management system and to deallocate the wastes storage facilities the contravened the regulations of the EU until the year 2017 [6].

The most important regulation in the field of wastes was the Directive regarding wastes management resulted from the extractive industry (Directive no.2006/21/CE) [7]. The Directive introduced communitarian rules for the prevention of soil and water pollution, on account of the long term storage of wastes in mud-setting ponds or dirt-heaps. The stability of such temporary storage facilities must be warranted in order to decrease possible accident consequences. Implementing the Directive assured an efficient extractive industry wastes management in the European Union (through accident and damage prevention to the environment and to the public health, produced through the process of treatment and elimination of mining residual waste). It is for the first time that the wastes resulted from the extractive industry fall under a specific regulation.

The European Commission suggested a new Frame Directive no. 2008/98 regarding wastes that stipulates the purpose and objectives on the environmental impact given to residue generating and management, taking into account the life cycle of the resources and the hierarchy of wastes. Through this suggestion, they consider a “better regulation” in the field of wastes. The Directive is in full accordance with the principle of prevention and minimization of residual waste, as it follows [7]:

- introduces environment objectives and clarifies the capitalization and elimination concepts of the wastes;
- establishes minimal standards and a procedure for the establishing of these standards, for a certain wastes management operation number.
- demands the development of national residue generating prevention programs.

The wastes management primary objectives are the prevention and decrease of residual waste production and the decrease of their injuriousness degree through:

- the development of “clean technologies”, with a reduced natural resource consumption;
- the development of technology and the commercialization of products that, through their manufacturing, utilization or elimination way, do not have a serious impact or have the smallest possible impact on the increase of the amount or danger of the wastes or on the pollution risk.
- the development of adequate technologies for the final elimination of hazardous substances from the wastes destined for capitalization;
- the material and energetically capitalization of the wastes, with their transformation in secondary raw materials or with the use of residual waste as energy sources.

The necessary investments for the treatment/elimination of hazardous wastes will be performed by the institutions that generate the wastes or, in a private sector, will be performed by the specialized operators that will carry on these actions, for third parties, receiving remuneration. The shutting down of some industrial wastes storage facilities, in accordance to the legal European and national requirements, is an activity that necessitates relatively large amounts of financial funds. The viable economical units will be compelled to invest so that they should solve the problems of the wastes storage facilities that lack confirmation of conformity with the EU regulations. A great financial support comes from the state which will be allowed in the period 2009 – 2012 over 1 billion RON.

Although it is very difficult to achieve a residue generating prognosis, because it is directly influenced by the industrial development prognosis, we can expect that the industrial wastes generating coefficient would drop, as “clean technologies” will be implemented and because the principles of pollution prevention, reduction and integrated control, will be applied.

4. CONCLUSIONS

From the industrial activities in Romania it is generated a very important quantities by wastes, both non-hazardous and hazardous. The extractive industry from Romania, and especially the coal extraction and processing activity by the enormous waste quantities eliminated by storage, affect all the environmental fields.

The paper shows aspects related to the generating, capitalization, elimination and management of industrial wastes in Romania, between the years 2003-2006. The processing of statistical data presented in the literature, indicated that in the evaluated period, the waste generated by the extractive industry represent 75% of the generated total, from which we can count more than 99.5% as being non-hazardous (inert) waste and the rest, represented by 0.5%, as being hazardous waste. The almost part from those industrial wastes are eliminated by storing them in dumping sites and mud-setting ponds. The impact of stored industrial waste on the environment, resides first of all in soil pollution, depth water and surface water pollution.

The responsibility of industrial waste management activities belongs to the generators, which is obliged to shutting down those storage facilities, in accordance to the legal European and national requirements.

Romania has concentrated its efforts in the field of waste management, consisting in the transposition in national laws the European directives and regulations.

At the end of this paper, we will show the main objectives in the field of industrial waste management, according to the most recent European Union Directives (Directive no.2006/21/CE regarding the wastes resulted from the extractive industry, and the Frame Directive no. 2008/98 regarding wastes).

The priorities in waste management are: the rehabilitation of the damaged landfill, the material and energetically capitalization of the wastes and the prevention and reduction of waste production. The viable economical units will be compelled to invest so that they should solve the problems of the wastes storage facilities that lack confirmation of conformity with the EU regulations. In this context, it is expected in the future will bring reduced industrial waste quantities.

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RISK MANAGEMENT AND ITS IMPLICATIONS FOR MINE SAFETY

Tudor GOLDAN*

Abstract: *Advancement in technology and management systems too often outstrip the ability of experts to provide exacting community standards for the safe and effective operation of management systems and equipment. The ideal workplace would have fit for purpose equipment, competent personnel, management systems in place, all within a known environment. In reality inherent hazards associated with technology and management of technology within the mining environment requires a process to be utilized not only to reduce hazards to an acceptable level but also produce management systems appropriate for the business. This demands the adoption of a structured process for the identification of hazards and evaluation and control of work related risks.*

Key words: *accident, management of risks, risk assessment.*

1. INTRODUCTION

Because of the inherent hazards of mining as an activity, and the complexity of mining machinery and equipment and the associated systems, procedures and methods, it is not possible to be inherently safe. Regardless of how well machinery or methods are designed, there will always be the potential for serious accident. It is therefore not possible for any external agency to ensure the safety of an organization such as a mining company, nor of the machinery or methods it uses. The principal responsibility for the safety of any particular mine, and the manner in which it is operated, rests with the management of that mine.

It is now widely accepted within industry in general that the various techniques of risk assessment contribute greatly toward improvements in the safety of complex operations and equipment. In many industries there is a legislative requirement for risk assessment to be undertaken of all hazardous equipment, machinery and operations, taking account of the procedures used for operation, maintenance, supervision and management.

Risk management aims to reduce the likelihood and impact of mishaps of all kinds. In the mining industry, with its inherent potential for major accidents which could injure or kill many people, damage the environment, cause serious loss of production and hence profit, there is a particular need for a sound approach to the process of risk management.

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Risk assessment aims to assist in effective management of risks, by identifying:

- which risks are most in need of reduction, and the options for achieving that risk reduction;
- which risks need careful on-going management, and the nature of that on-going attention.

2. THE REQUIREMENTS FOR MANAGEMENT OF RISKS

The essential requirements for effective management of risks for the mining industry can be defined in various ways:

1. The hazards (i.e. the potential for different types of accident) must be understood by everyone involved, including an understanding of how the accidents may arise, how serious they could be, and the nature of the preventive and protective "barriers".
2. The appropriate facilities, machinery and equipment must be provided to match the hazards.
3. The appropriate systems and procedures must exist to match the hazards and the facilities, machinery and equipment. These systems fall into several classes:
   - those for operating to high standards, including selecting, operating and maintaining equipment correctly;
   - those for monitoring performance, including supervising and managing the performance of machinery and people;
   - those for progressing improvements;
   - those for auditing the systems for monitoring, supervising and managing the operations, and for progressing improvements.
4. The appropriate organization should be provided, with appropriate staffing levels, communication systems and training.
5. There should be a high level of emergency preparedness, including means of detecting the onset of an emergency early and responding to it effectively and promptly.
6. Safety and risk management must be actively and visibly promoted by management.

The above requirements have a number of important implications for mine safety.

1. Risk assessment is not a once-off activity which, once done, can be put aside so that mining operations can continue as before. Risk assessment is one part of an on-going process of risk management. It aims to improve the understanding of the potential for accidents, their possible consequences, and the adequacy (or inadequacy) of the various safeguards.
2. The *foundations* of effective risk management are:
   - understanding, by all those involved, of the possible causes of accident or other mishap;
   - real and visible commitment to managing the risks by staff at all levels.
3. The *structure* of an effective risk management programme comprises:
   - physical facilities, equipment, and other "hardware";
   - systems and procedures of many kinds;
   - suitably trained and experienced people, working in an organized manner, with good formal and informal communications;

- emergency capability, comprising understanding of the potential for emergency, together with appropriate emergency equipment, procedures, staffing, training etc.

4. If a mine manager wishes to review the adequacy of the risk management approach being used, it is helpful to structure the review about the six requirements set. They are all equally important.
5. It is also helpful to remember that management comprises:
   - planning;
   - organizing;
   - controlling;
   - leading and motivating.

Thus any manager who is managing risks will have stated objectives and a plan, will have organized for it to be implemented, will be monitoring progress and taking corrective action where it is not going according to plan, and will be showing active leadership and commitment to achieving the plan. Where these are missing, the manager is *not managing risks*, but (at best) reacting to risks, i.e. being *managed by the risks.*

3. THE STEPS IN RISK MANAGEMENT

The steps of risk management are:

- establish the context;
- identify risks;
- analyse risks;
- assess and prioritize risks;
- treat risks;
- monitor and review.

This form a continuing process is illustrated in Figure 1.

While the process of risk management is not difficult in concept, it is often very valuable for initial studies to be facilitated by someone with previous experience in risk assessment and risk management programmes, and with an awareness of the special nature of mining operations and the risks which arise from them.

3.1. Establish the context

Strategic context. This entails considering the relationship between the mine's operations and its environment. By environment is meant the physical environment, the stakeholders of various kinds, and the political environment.

- physical environment (e.g. natural environment, neighbours whether agricultural, residential or industrial etc.);
- stakeholders (e.g. the owners of the organization, employees at all levels, customers, suppliers, local community, interest groups and society as a whole including the various levels of government);
- political environment (e.g. relationship between the particular mine or the mining industry in general with local government, state or federal governments etc.).

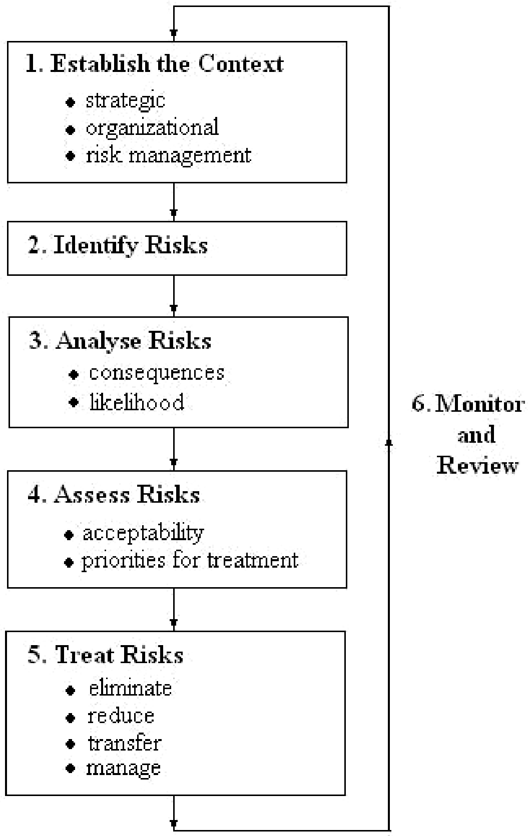

Figure 1. The steps of risk management.

Organizational context. This entails considering both the goals and objectives of the organization in relation to risk management, and as its capabilities. For example, the organization may not know whether its risks are well controlled or not; it may regard its performance as quite good, and needing only a tune-up; or it may recognize a need for a rapid and major improvement. It may, or may not, have staff able to undertake the steps in the risk assessment and the risk management programme. It may see its priorities for risk management as principally relating to safety, rather than the environment, property damage or loss of production.

These all affect the objectives, scope and methods of the risk assessment to be undertaken and of the risk management programme of which the risk assessment study is part.

Risk management context. This entails defining and recording:

- the goals of the risk assessment study, and of the risk management programme;
- the scope and limits of the study (in time, location, depth, breadth etc.);
- the specific studies to be undertaken.

3.2. Identify risks

This is perhaps the critical step of a risk assessment. A risk which is not identified cannot be actively managed.

There are many techniques available for identifying risks, but none of them can be expected to identify all the risks, large and small. The aim, in risk identification, is to be *confident* (not *certain*) that no significant risks have been overlooked.

Having recognized the existence of risks, it is necessary to consider possible causes and scenarios. It is said that *"time spent planning is seldom wasted"*, because the usual fault in planning is that too little is done; rarely is too much done. Similarly, in risk assessment, time spent identifying hazards is seldom wasted.

There are many techniques available for identifying risks. Some of these are listed below.

- Action Error Analysis - a method of postulating and analyzing possible human errors, by considering each step in a procedure against a checklist of possible errors, such as: error of omission; error of time; extraneous act; transposition; error of selection; error of sequence; miscommunication, qualitative errors; etc.
- Failure Mode and Effects Analysis (FMEA) - a systematic review of the effects of different types of failure of each component of a machine or item of equipment;
- Failure Mode and Effects Criticality Analysis (FMECA) - similar to FMEA, but with addition of estimates of the severity of the effects and the likelihood of occurrence;
- Fault Tree Analysis (FTA) - a method of analyzing possible causes of defined unwanted events, by starting with the defined unwanted event, identifying possible causes, then analyzing the factors leading to those causes etc, until "root causes" have been identified;
- Hazard and Operability Studies (Hazop) - a systematic review of the consequences and likelihood of different process or system abnormalities, such as excessively high or low flow, pressure, temperature, etc. It can be adapted to a wide range of types of industry and operation. Its general approach is the foundation for other methods such as MHI and PHEI.
- Machinery Hazard Identification (MHI), in which plant or machinery is considered, by sections or components, considering the potential for a range of injuries to people by studying a checklist of possible causes.
- Potential Human Error Identification (PHEI), which is a variation on Action Error Analysis.
- Rapid Ranking - a method developed initially for ranking identified risks, but which has been developed to include hazard identification.
- Workplace Risk Assessment and Control (WRAC), which is a process for identifying potential production or maintenance incidents and losses, and uses a matrix approach to define risk levels from estimates of consequence and likelihood.
- What-If? Analysis - a method of examination of the consequences of a wide range of types of occurrence, drawn from a comprehensive checklist designed to suit the particular type of operation.

3.3. Analyse risks

As set out earlier, risk analysis is *"A systematic use of available information to determine how often specified events may occur and the magnitude of their likely consequences."*

Analysis of risks entails understanding the nature of the risks which exist, the nature of the existing controls and "barriers", and assessing the *likelihood of occurrence* of mishaps, and the *severity of the consequences* of those mishaps.

It is often thought that the likelihood and severity must be expressed in numbers. This is not correct. Certainly, where they can be quantified this should be done, but in the mining industry there are many factors affecting the likelihood and the consequences which cannot be expressed in numbers, and which need to be expressed in words.

In some industries, such as the oil and chemical industries, it is possible to calculate the size of the possible accidents, such as the heat radiated from a jet of burning gas, or the toxicity of a poisonous gas at a distance downwind from a leak.

In the case of risks arising from the operations and equipment at a mine, it is likely that a significant proportion of the identified risks will not be suitable for mathematical analysis. In most cases, the likelihood of accidents and the severity of their possible consequences can only be estimated, using experienced judgment, and drawing on mine site and industry-wide accident and incident data whenever appropriate and available. These estimates of consequences and likelihood may be expressed in numbers on a scale, with different scales being used for risks of different types.

For example, the *consequences* of risks can be estimated in terms such as those below:

- *risks to people:* the numbers of injuries of different severities;
- *risks to property and production:* the monetary value of the damage or production lost;
- *risks to the environment:* the extent and severity of the environmental damage, or the extent of public reaction.

When estimating consequences, it is important to take account of both tangible and intangible consequences. For example a major accident which resulted in many fatalities would be expected to have a major impact on the future operations of the mine, but there the nature and extent of that impact would have both tangible and intangible dimensions, such as the increased costs of additional safeguards (tangible) and the difficulty of recruiting miners afterwards (intangible).

As the basis for deciding the priority to be given to treating the various identified risks will be decided on the basis of the assessed likelihood and consequences, it is important that the estimates be made by people with sufficient experience for their judgment to be reliable, i.e. as good as anyone else, and with good credibility within the organization.

3.4. Assess and prioritize risks

Risk assessment is *"the process used to determine risk management priorities by evaluating and comparing the level of risk against predetermined standards, target risk levels or other criteria."*

This explicitly requires the existence of *"predetermined standards, target risk levels or other criteria"*.

Implicit in the definition of risk assessment is the assumption that it is not possible fully to treat all risks at once, and that there is a need to sort out those which will be treated at once from those which will be treated later, and from those which are too small to need treatment.

When deciding the degree of effort to be put into reducing risks, one of two possible approaches must be selected. They are:

- comparing the assessed magnitude of the risks (e.g. as expressed above) with a statutory requirement, or a organizational policy on safety, or with a consensus view of what "good practice" or "due diligence" would require (with one test of due diligence being whether one would be comfortable explaining the approach to a court!);

- deciding on an amount of capital or staff effort which can be allocated each year to risk reduction and risk management.

There are a number of methods by which the risks can be prioritized for treatment. But first it is important to recognize that there are two main classes of treatment which may be needed. They are:

- ***risk reduction*** - needed for risks which individually or in total are excessive when compared with some standards or criteria;
- ***risk control*** - needed for risks which could have very serious consequences if they were realized, but which are not regarded as high risks because of their low likelihood. (If they are not monitored and controlled, they may be expected to increase in likelihood and become high risks).

3.5. Treat risks

Risk treatment is *"selection and implementation of appropriate options for dealing with risk."*

Typically the options comprise:

- acceptance (particularly applicable to low risks); i.e. deciding to do nothing to reduce the risk or to control it;
- reduction (particularly applicable to high risks); i.e. acting to reduce the likelihood or the consequence of the potential mishap – or both – (by changing designs, procedures, management methods etc.);
- transfer (particularly applicable to risks with serious consequences and low likelihood); by such means as insurance or contractual arrangements;
- on-going management (particularly applicable to risks with serious consequences and a low likelihood which could increase unless actively managed);
- retain the risk (particularly applicable to residual risks, left after risk reduction, which may require financing).

The full value of a risk assessment becomes evident when consideration is given to the actions needed to treat the risks.

The risks from any particular machine or equipment will have been found to take a number of forms, to arise from various features of design and operation, and to have a number of possible causes. In the analysis, the likelihood and consequences have each been assessed, either in numbers or in words, and the causes and safeguards understood.

The main reasons for any individual risk needing treatment are thus displayed for examination, together with their contribution toward the total risk. What is needed is selection of the most cost-effective combination of actions to reduce the total risk, and a work plan for undertaking those actions. The options can be developed by consideration of the six principles: understanding, equipment, procedures, staffing and training, emergency preparedness, and promotion.

This leads to another requirement, without which any risk assessment is a sterile document:

"A risk assessment should include:

- *a list of the actions planned:*
  - *to reduce high risks;*
  - *to control the likelihood of potentially serious accidents even if the likelihood is seen as low;*
- *a programme for undertaking at least the first steps of implementation."*

3.6. Monitor and review

As set out earlier, a well-recognized role of management is *control.* Control, in turn, entails:

- setting a standard;
- monitoring or measuring the actual performance;
- comparing the actual performance with the standard;
- acting to correct any deviation from the standard.

Thus an essential part of any risk management programme is monitoring of performance compared with the standards and the plans, and review of the standards themselves, the plans, the way work is organized and monitored, and the nature of and reasons for the problems or shortcomings identified.

4. CIRCUMSTANCES IN WHICH A RISK STUDY MAY BE APPROPRIATE

In everyday life everyone assesses risks. Fortunately these are mostly minor, such that inadequate assessment, and the consequent mishaps, does not matter greatly.

In principle, a systematic risk assessment should be undertaken wherever there is:

- potential for a mishap which could have serious consequences;
- a large number of risks with varying consequences and likelihood, where the priority of them is not clear.

Examples of such situations include:

- a mine which has not previously undertaken a formal risk management programme wishes to do so, but has limited resources (financial, staff) to make dramatic improvements at once;
- a “near miss” has occurred, and management wish both to act effectively to prevent a recurrence leading to an accident, and to broaden the approach to prevent other serious mishaps which have not yet given “early warning” by means of a near miss;
- a modification is planned to a procedure which could affect some important mining safeguard;
- a new type of equipment or machine, or equipment or machine which has been extensively modified, is to be purchased;
- equipment or a machine is to be modified;
- an established method of undertaking a mining operation is to be modified.

These can be summarized as

1. wherever a mine has not had a formal risk management programme to date, a broad brush risk assessment should be undertaken to identify the principal priorities (e.g. improvements in understanding, equipment, procedures, organization, emergency preparedness, commitment etc.);
2. wherever there are so many risks that it is important that they be treated with an appropriate priority and in an organized way, to achieve most rapid overall risk reduction with the limited resources;
3. wherever there is a particular risk which could have serious consequences and where the causes and adequacy of safeguards are not entirely clear or understood;
4. wherever there is a change planned to equipment, machinery, procedure, manner of working etc. with the potential to affect the magnitude or likelihood of some inherent mining hazard, or the effectiveness of some safeguard or “barrier”.

CONCLUSIONS

Risk analysis, risk assessment and risk management can be undertaken very effectively even where it is not possible to put numbers onto the severity and the frequency of occurrence of hazardous incidents.

It is possible to ascribe a number to a judgment (or even a "gut feeling") about the magnitude or likelihood of a hazardous incident, by preparing lists of descriptions of effects and of likelihoods, and putting numbers to them.

Of course, those numbers are no more valid than the judgment which derived them, but important decisions are made every day about matters which cannot be quantified, relying solely on judgment.

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THE LOAD OF HAZARDOUS ELEMENTS IN SPOIL BANKS OF THE NORTH BOHEMIAN BROWN COAL BASIN AFTER FOREST RECLAMATION

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Abstract: *In 2004 – 2006 in the North Bohemian Brown Coal Basin the load of hazardous elements was evaluated in the oldest and pedologically most important Anthroposols of spoil banks that had undergone forest reclamation. Compared to the limit background values (Decree No. 13/1994 of Ministry of Environment of the CR) the load (of geogenic and air pollution origin) in these new soils can be considered as insignificant; some profile anomalies (on very small areas) with the above-limit content of some elements (Co, Cr, Cu, Ni, V) were observed only in Anthroposols influenced by bentonites or in substrates of the coal seam where As becomes a problematic element.*

1 INTRODUCTION

The concentrated surface mining of brown coal and developed industrial activities in the North Bohemian Brown Coal Basin are accompanied by extensive devastation of landscape on an area up to 29 000 ha and by the impaired environment. To remove the consequences of such mining activities purposeful reclamations are carried out that result in the formation of new soils – Anthroposols (NĚMEČEK 2001) exploitable for agricultural and forestry production.

The formation of Anthroposols on different geological substrates is connected with a risk of their potential contamination by undesirable substances from substrates and amendments used for reclamations; so it is necessary to continually pay an increased attention to their contamination (CHRENEKOVÁ 1992; TOMÁŠEK 1994; KOHEL et al. 1995; ŘEHOŘ 1997; ČERMÁK et al. 2003). In an industrially intensively used countryside air pollutants generated by power engineering and chemical industries are other negative factors (ŠANDROCH et al. 1989; FACEK 1990; BENEŠ 1994).

Tertiary clays are the most frequently used rock for forestry purposes in this territorial region, and also selectively stripped loess loams and marlites; their geochemical composition is usually below the limits of hazardous elements in soils belonging to farm land resources

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(BENEŠ 1994, PODLEŠÁKOVÁ et al. 1994). Exceptions in spoil bank conditions are only substrates of the coal seam with an increased content of coal and pyrite or where ash from the burning of brown coal in thermal power plants was used for reclamations (TOMÁŠEK 1994, KOHEL 1994, ŘEHOŘ et al. 2004).

2 MATERIAL AND METHOD

2.1 Reclamation characteristic of evaluated localities

The evaluated Anthroposols in the North Bohemian Brown Coal Basin, which underwent forest reclamation, represent a characteristic of the most important (direct) method of reclamation of overburden rocks deposited on the spoil bank surface for these purposes while only the slopes of Střimice I spoil bank are an exception where the reclamation of spoil-bank soils was more labour-intensive as a consequence of the application of bentonites (due to the landslide):

Merkur – pelitic Anthroposol, montmorillonitic-illitic-kaolinitic Tertiary clays with the content of physical clay 75 – 80% are represented, reclamation age 30 years

Velebudice – pelitic Anthroposol, illitic-kaolinitic Tertiary clays with the content of physical clay 40 – 50% are represented, reclamation age 45 years

Šverma – pelitic Anthroposol, illitic-kaolinitic Tertiary clays with the content of physical clay 40 – 50% are represented, reclamation age 25 years

Střimice I slopes – overlaid Anthroposol, reclamation of lighter-textured spoil-bank substrates with the application of bentonites (3 000 t/ha), reclamation age 20 years

Střimice I – Anthroposol with frequent transitions to arenose Anthroposol, heterogeneous substrate composed of sandy to sand-loamy spoil bank soils, frequent admixture of sintered clays and substrates of the coal seam, reclamation age 20 years

Větrák – pelitic Anthroposol, heterogeneous substrate composed of clays, loess loams, sintered clays, bentonites, including substrates of the coal seam, reclamation age 45 years

Lotta Marie – Anthroposol, heterogeneous mixture of clayey sands and loess loams, reclamation age 50 years

Václav – pelitic Anthroposol, heterogeneous mixture of Tertiary clays and clayey sands, reclamation age 40 years

Úžín – Anthroposol, montmorillonitic-illitic-kaolinitic Tertiary clays with the content of physical clay 20 – 25% are represented, reclamation age 45 years

2.2 Experimental works

The evaluated spoil banks represent a set of localities characterising the entire territorial region of the North Bohemian Brown Coal Basin, and in pedological terms the oldest and the most important overburden rocks used for forestry purposes. The collected disturbed soil samples in each evaluated locality represent the profile characteristic in three replications of the condition of humus horizon, initial cambic horizon and original spoil bank substrate not influenced by the soil-forming process to a larger extent. The analyses of soil samples (total decomposition by HCl, HNO_3 and HF in accordance with standard ISO 14 861 – 1 and final measurement on AAS VARIAN 240) were done in a central laboratory of the Research Institute for Soil and Water Conservation and compared with the limit background values of admissible contents of hazardous elements in soils belonging to the farm land resources (Decree No. 13/1994 of the Ministry of Environment of the CR), with the values in natural forest soil in the central part of the evaluated territorial region (Cambisol on loess, Kaňkov locality) and in the extracted raw material – bentonite for industrial purposes in Braňany area. Tabs. 1 and 2 show the results of soil analyses and their comparison with the used criteria of the load of hazardous elements in different soil conditions.

3 RESULTS AND DISCUSSION

As – the highest contents were measured identically in all evaluated spoil bank substrates, particularly in the humus horizon (to 0.01 m), exceeding the limit background values max. by 10%. The profile load of this element is characteristic only of localities with an increased proportion of substrates of the coal seam, and is of geogenic and air-pollution origin

Be – profile contents in all spoil-bank substrates are markedly below the limit value (Větrák locality is the only exception). The characteristic of the load of this element in Anthroposols is mostly geogenic, which is also applicable to natural forest soil

Cd – a very slight increase in all spoil bank substrates was observed only in the soil horizon to 0.2 m. The increase in the load of this element in Anthroposols is caused by air pollution, which is also applicable to natural forest soil

Co – only spoil-bank substrates influenced by bentonites have profile increased contents exceeding the limit background value by up to 63%. The load of this element in Anthroposols is of geogenic origin, which is also applicable to extracted bentonites

Cr – only spoil-bank substrates influenced by bentonites have profile increased contents exceeding the limit background value by up to 88%. The load of this element in Anthroposols is of geogenic origin, which is also applicable to extracted bentonites

Cu – only spoil-bank substrates influenced by bentonites have profile increased contents exceeding the limit background value by up to 76%. The load of this element in Anthroposols is of geogenic origin, which is also applicable to extracted bentonites

Mo – profile contents are markedly below the limit value in all spoil bank substrates. The load of this element in Anthroposols is of geogenic origin, which is also applicable to natural forest soil

Ni – only spoil-bank substrates influenced by bentonites have profile increased contents exceeding the limit background value by up to 223%. The load of this element in Anthroposols is of geogenic origin, which is also applicable to natural forest soil

Pb – increased contents in all spoil bank substrates were measured only in humus horizons (to 0.1 m), amounting to 24% of the limit background value. The load of this element in Anthroposols is caused by air pollution, which is also applicable to natural forest soil

V – only spoil-bank substrates influenced by bentonites have profile increased contents reaching up to 95% of the limit background value. The load of this element in Anthroposols is of geogenic origin, which is also applicable to extracted bentonites

Zn – only spoil-bank substrates influenced by bentonites have profile increased contents reaching up to 100% of the limit background value. The present character of the load of this element in Anthroposols is rather a geogenic problem. But it cannot be excluded that the situation in the surface soil horizon of Anthroposols has slightly been influenced by air pollutants (similarly like the situation in natural forest soil)

Hg – only surface soil horizons to 0.2 m have increased contents in all spoil bank substrates, but the increase amounts only to 15% of the limit background value. The load of this element in Anthroposols is caused by air pollution, which is also applicable to natural forest soil.

4 CONCLUSION

The load of hazardous elements in new soils that underwent forest reclamations – Anthroposols of spoil banks in the North Bohemian Brown Coal Basin is insignificant in comparison with the used limit background values (Decree No. 13/1994 of Ministry of Environment of the CR). Some profile increased contents of some hazardous elements (*Co, Cr, Cu, Ni, V*) in Anthroposols influenced by bentonites can be considered as an environmental curiosity in relation to the area (max. 1%) out of the total area of all evaluated localities. As for *As*, a potential increase in its content is a geogenic problem connected with random influence of substrates of the coal seam on spoil bank substrates, and it is also a consequence of air pollution caused by the longer-term load of industrial activities in the evaluated territorial region. A further very slight increase in their contents in surface horizons of all Anthroposols was observed in *Cd*, *Pb* and *Hg*. The presence of the other hazardous elements (*Be, Mo, Zn*) in spoil bank Anthroposols is a common characteristic of the geogenic load of spoil banks with stripped overburden rocks.

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Table 1. *Average total contents of hazardous elements in spoil banks after forest reclamation in the North Bohemian Brown Coal Basin*

Spoil bank	Depth (cm)	Total content (mg/kg)												
		As	**Be**	**Cd**	**Co**	**Cr**	**Cu**	**Mn**	**Mo**	**Ni**	**Pb**	**V**	**Zn**	**Hg**
Merkur	0-10	21.7	2.0	0.41	16.4	43.6	31.2	951	0.62	34.7	29,7	76,3	129,3	0,07
	10-20	13.3	2.2	0.12	17.4	56.5	23.1	843	0.42	39.2	24,8	96,3	102,6	0,05
	50-70	13.6	2.4	0.11	15.4	54.9	25.9	828	0.34	41.5	22,8	98,3	104,7	0,04
Šverma	0-10	65.7	2.2	0.58	15.0	46.6	28.9	631	0.81	29.4	37,1	88,1	104,8	0,08
	10-20	13.8	1.7	0.54	17.0	55.5	25.2	529	0.54	32.7	27,0	84,6	79,1	0,05
	50-70	11.6	2.0	0.21	26.4	41.4	19.6	1259	0.47	47.1	22,1	70,2	110,5	0,05
Velebudice	0-10	23.7	2.0	0.24	10.7	99.9	21.1	414	1.46	30.2	22,0	77,6	65,1	0,22
	10-20	27.7	1.2	0.66	12.5	111.6	22.5	188	1.95	32.3	26,9	105,2	76,5	0,19
	50-70	20.9	1.0	0.19	27.5	148.8	23.3	747	1.71	58.5	30,9	140,6	132,5	0,09
Střimice slopea	0-10	10.6	2.1	0.26	57.5	421.5	61.4	538	0.69	294.9	<5,0	173,2	131,6	0,07
	10-20	6.8	1.8	0.23	60.1	452.6	70.3	490	0.71	309.1	<5,0	190,9	129,6	0,04
	50-70	5.6	2.0	0.32	74.9	495.2	66.5	496	0.45	357.3	<5,0	180,7	135,0	0,05
Střimice I	0-10	9.0	1.7	0.08	8.0	51.1	13.6	103	0.77	20.8	13,5	59,3	45,9	0,05
	10-20	4.9	1.1	0.05	5.6	39.8	8.6	66	0.57	12.5	9,5	42,2	35,7	0,04
	50-70	3.7	1.2	0.06	7.1	34.9	6.8	135	0.57	13.9	7,9	32,6	38,4	0,03
Větrák	0-10	11.7	6.5	0.42	37.7	180.3	65.6	2286	1.0	104.1	15,3	168,6	253,0	0,11
	10-20	8.2	2.5	0.22	61.5	240.5	77.0	3164	1.5	128.9	<5,0	224,9	268,7	0,03
	50-70	6.6	2.4	0.24	87.6	255.4	85.1	3303	1.5	159.6	<5,0	223,6	259,3	0,02
Lotta Marie	0-10	9.4	1.9	0.50	12.8	64.6	18.1	846	0.69	27.0	17,5	59,4	119,1	0,08
	10-20	9.4	1.8	0.50	14.2	66.5	17.1	734	0.84	20.9	17,3	61,8	122,9	0,06
	50-70	9.3	3.0	0.15	21.7	79.9	23.0	889	0.78	39.9	13,8	80,1	80,2	0,04
Václav	0-10	12.1	2.4	0.25	14.9	78.7	18.9	375	0.83	35.4	24,8	92,0	85,4	0,09
	10-20	8.8	2.7	0.15	17.2	93.5	17.8	399	0.91	42.2	21,0	104,7	75,6	0,07
	50-70	7.9	2.0	0.11	13.6	66.5	14.1	359	0.58	28.8	15,7	70,7	63,3	0,04
Úžín	0-10	19.6	1.5	0.65	16.0	33.4	35.8	763	1.13	30.4	44,5	65,5	139,3	0,11
	10-20	14.8	1.7	0.73	16.9	34.1	31.7	655	0.87	30.9	31,2	59,5	124,4	0,07
	50-70	8.3	2.0	0.21	24.6	33.6	28.0	802	0.56	49.5	22,7	45,5	113,2	0,04
Cambisol (Kaňkov)	0-10	27.2	2.4	0.45	41.6	115.0	54.6	1807	0.55	76.5	48,5	160,2	210,5	0,09
	10-20	11.5	2.9	0.57	55.9	136.5	55.5	3341	0.58	108.8	26,0	175,3	223,7	0,08
	50-70	11.1	2.5	0.26	32.5	60.6	28.7	3618	0.55	43.1	17,7	86,4	115,4	0,04
Bentonit mine (Braňany)		5.9	<0.1	0.14	69.2	359.2	192.7	1769	0.74	217.0	<5,0	197,9	112,6	0,01
Decree No. 13/1994		30.0	7.0	1.0	50.0	200.0	100.0	-	5.0	80.0	140,0	220,0	200,0	0,8

Table 2. Comparison of the total content of hazardous elements in spoil banks after forest reclamation with the limit background values (Decree No. 13/1994 of ME of the CR)

Soil characteristic	**Depth (cm)**	**Total content (mg/kg) in %**												
		As	**Be**	**Cd**	**Co**	**Cr**	**Cu**	**Mn**	**Mo**	**Ni**	**Pb**	**V**	**Zn**	**Hg**
Tertiary clays	0-10	109	27	47	29	29	29	-	20	39	24	35	55	15
	10-20	58	24	41	32	32	26	-	19	42	20	39	48	11
	50-70	45	27	18	47	35	24	-	15	61	18	40	58	8
Substrates influenced by bentonites	0-10	37	61	34	95	150	64	-	17	249	7	78	96	11
	10-20	25	31	23	122	173	74	-	22	274	4	95	100	5
	50-70	20	31	28	163	188	76	-	20	323	4	92	99	4
Substrates of Quaternary and Tertiary origin	0-10	34	29	28	24	32	17	-	15	35	13	32	42	9
	10-20	26	28	23	25	33	15		15	32	11	32	39	8
	50-70	23	30	11	28	30	15	-	9	34	9	20	30	5
Cambisol on loess	0-10	91	34	45	83	58	55	-	11	96	35	73	105	11
	10-20	38	41	57	112	68	55	-	12	136	19	80	112	10
	50-70	38	36	26	65	30	29	-	11	54	13	39	58	5
Bentonite mine		20	1	14	138	180	192	-	15	271	4	90	56	1
Decree No. 13/1994		100	100	100	100	100	100	-	100	100	100	100	100	100

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VENTILATION PLANT TO COLLECT AND RETAIN THE CONTAMINANTS EXHAUSTED AT STEEL MANUFACTURING IN ELECTRIC ARC KILNS

DANIEL-ONUT BADEA *

Abstract: *The equipment is a system for collecting the contaminants directly from the kiln arch made of cylindrical fitting that crosses the kiln arch with a sucking pipe at the end of it at an adjustable distance from the fitting, maintaining a pressure of 3-4 mm H_2O in the kiln, a heat exchanger to cool the gases with a fitting of automatic adjustment valve to maintain the exhaust temperature under 120^0C, a hood to collect the contaminants during the kiln loading and discharging operations, a filter to retain dusts and a centrifugal fan.*

Key words: *Ventilation, toxic gas, dusts, collecting, filtration, health*

Metal melting in electric arc kilns is used to obtain steel in metallurgy. Besides the advantages of rapid melting of metals and alloys the steel manufacturing in electric arc kilns has the disadvantage of a significant quantity of contaminants (toxic gas and dusts) exhausted during the melting technological process. The contaminants are evacuated out of the kiln by the openings in the kiln arch meant for electrodes passage, by the top and discharge pan for melted metal resulting in pollution of the work and general environment.

Together with the contaminant release, significant exhaust of convection and radiation heat takes place.

The most dangerous exhausted gas is carbon oxide which during the oxygen or argon blower can reach 90% of the exhausted gas volume.

Other gases exhausted at steel manufacturing in the electric arc kiln are sulphur dioxide, carbon dioxide, nitrogen oxides, various metal oxides in vapour form and gases resulted from burning various auxiliary products introduced in the kiln together with the materials representing the charge submitted to melting: oils, paints, organic materials etc.

Dusts from the electric arc kilns are fumes (having sizes of 0.1 – 1 μm) and dusts entrained by gases and fumes of sizes up to 50μm.

The chemical composition of the entrained gas particles is variable and depends on the kind of materials included in the charge. Generally, the dust components are iron, calcium, silica, manganese, zinc, chrome, copper, aluminium etc. oxides.

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The quantity of gas exhausted from an electric arc kiln depends on the charge content, work system, quantity of blast oxygen and the blast period, degree of aeration, kiln structure etc.

The gases come out the kiln having a temperature of about 1300^0C during the melting time and about 1500^0C when the oxygen aeration is made; increase in gas temperature is also caused by the ignition of carbon oxide exhausted from the kiln when it comes in contact with air at the opening of the kiln arch.

Measurements made at the electric arc kilns when the ventilation equipment is not operating make evident excess of contaminant concentrations in the work area near the kiln and also in the room where the kiln is placed as they are entrained by air drought.

The gases and dusts are evacuated through skylights into the environment thus polluting it intensively.

The values of contaminant concentrations in the work environment during various stages of the technological process overpass about 3-15 times the admitted concentration values.

The suspension particle granulometric spectrums show that it includes 32-45% submicronic particles, 53-63% particles of sizes between 1–3 μm and under 1% particles of 3 - 5μm.

The researches made revealed the variation of the contaminants resulted from the technological processes; thus, during the steel manufacturing in a kiln of 3 tons a quantity of about 600 kg dust of sizes between 0.1 and 12 μm is exhausted in 24 hours. Consequently, the pollution due to the contaminants exhausted in the atmosphere (dusts, metallic oxides, nitrogen oxides, carbon oxide, sulphur dioxide etc.) represents an important aggression against the environment.

The laboratory and on site researches resulted in a new method of collecting contaminants with a reduced air flow and high efficiency.

Contaminant collection is made as it follows:

- directly from the kiln arch for the contaminants exhausted during the manufacturing technological process;
- through a hood placed above the kiln for the contaminants exhausted during feeding and discharging the kiln.

Contaminant collection directly from the kiln is made in the following system:

- an cylindrical fitting across the kiln arch. It's diameter is calculated according to the kiln capacity;
- a pipe of adjustable end placed as an extension of fitting; the sucking up of contaminants in the kiln is developed by the variation of distance between the fitting and the pipe in accordance with the requirements of the technological process, maintaining a pressure of 3-4 mm H_2O in the kiln so that the charge does not oxidize.

Before being collected in the filter, the sucked gases from the electric arc kiln must be cooled to protect the filtering sacks in the filter from ignition. The gases have temperature of about 1100^0C when entering the sucking fitting in the kiln arch and must be cooled to a temperature of maximum 120^0C (at which they can operate permanently without the filtering sacks being in danger of ignition).

Cooling of gases is made in two steps:

- the first step develops the cooling of the gas having an initial temperature of 1100^0C down to a final temperature of 350^0C by mixing it with the atmospheric air sucked from the room;
- the second step develops the cooling of the gas loaded with dust and having an initial temperature of 350^0C down to a final temperature of 120^0 (temperature of entering in the filter) with the help of a heat exchanger.

The heat exchanger can be used as a heat recovery to be used at heating the work environment or water for domestic use.

The heat exchanger is provided with a sucking fitting for the false air with an automatic adjustment valve. It has the following functions:

- maintain a pressure of 3-4 mm H_2O in the kiln that is necessary for the charge not to oxidize, by sucking false air to adjust the flow of the air exhausted;
- maintain a temperature for entering the filter under 120^0C by mixing the air sucked from the room.

The ventilation equipment providing the collection of contaminant from electric arc kiln is presented in Fig. 1. and includes the following:

- a fitting (2) that crosses the kiln arch (1) to suck the contaminants;
- a pipe (3) with adjustable end placed at the end of the fitting;
- a hood (4) that sucks the gases leaked from the kiln during discharging and feeding the kiln;

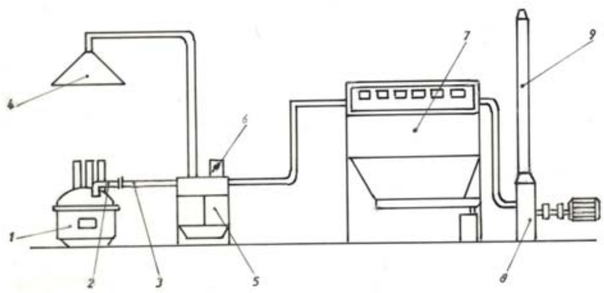

Fig. 1. Ventilation of electric arc kiln

- heating exchanger for gas cooling (5) where the temperature is reduced down to 120^0C, a temperature that permits filtering without damaging the filtering material the sacks are made of;
- a fitting having an automatic adjustable valve (6);
- a thermocouple for measuring the entering temperature in the filter;
- sack filter (7) to retain the solid particle from the gas with pneumatic shaking by inverted blow of automatic command. The sacks are made of synthetic material – polyester fibres – resisting at the temperature of maximum 140^0C;
- fan of centrifugal type (8);
- exhaust chimney (9)

The ventilation system meant to collect and retain the contaminant exhausted by the kiln is automated and provided with adjustment controls for air flow.

The present ventilation equipment applied to an electric arc kiln of 8 tons capacity.

In order to assess the efficiency of the ventilation equipment measurements of the contaminant concentrations in the work environment, emissions and filter retaining efficiency were made.

The results are presented in Table 1.

Serial no.	**Contaminant concentration (mg/m^3)**						**Filtering efficiency (%)**
	Work environment				**Ventilation pipe**		
	Without operating ventilation equipment		**With operating ventilation equipment**		**Before the filter**	**After the filter**	
	CO	**Total dusts**	**CO**	**Total dusts**	**Total dusts**	**Total dusts**	
1.	140	18	4	0,25	250	4,5	98,2
2.	160	20	5	0,27	241	4,8	98,0
3.	170	21	6	0,26	261	4,9	98,1
4.	140	19	5	0,25	255	4,8	98,1
5.	150	20	5	0,24	245	4,7	98,0

The analysis of the measurements of the contaminant concentration revealed that they are under the limits imposed by the legislation in force.

The system of collecting the contaminants directly from the kiln arch and cooling the gases using a heat exchanger has the following advantages:

- provide the total collection of contaminants resulted from the electric arc kilns with a minimum flow of air;
- avoid the hazard of occupational ill health;
- using the heat exchanger the air flow in the equipment is reduced and so is the fan capacity and filter;
- as compared to any other ventilation equipment the present equipment needs low investment and exploitation costs due to the air flow and low consumption of electric power and materials.

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THE REHABILITATION OF THE OLD ANINOASA MINE PRECINCT FOR THE PURPOSE OF TRANSFORMATION INTO MUSEUM

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Abstract. *Through the closure of Aninoasa mine, as a restructuring and efficiency activities of mining industry from the Jiu Valley, there has been created the favorable frame for transforming its precinct into a museum and its giving back into the touristic circuit of the area. In the purpose of mining activity remodeling practiced not long ago in this area, the designing of the objective „Aninoasa Mining Museum" consisted in rehabilitation of the Aninoasa North – Piscu tunnel and above-ground constructions, with entire utilities, and rehabilitation of available terrain rehabilitation.*

1. INTRODUCTION

Through HG 644 from 20 of June 2007 there had been approved the transformation of Aninoasa mine into a museum, being created this way the legal frame for starting the specific designing activity.

Designed to be placed in the old Aninoasa mine main precinct, the museum will belong to Aninoasa town, placed on the southern part of Hunedoara County, on north-east being adjoined with Petrosani municipality, on south with Gorj County, and on west side being adjoined with Vulcan municipality.

The access in the region it's accomplish by rail road Filiasi - Tg. Jiu - Livezeni - Simeria, route Petrosani - Livezeni - Iscroni - Vulcan and by road on E 79 Tg. Jiu - Petrosani - Simeria - Arad, variant DN 66A Petrosani - Aninoasa - Câmpu lui Neag.

Until the elaboration date of the documentation for „Technical Project for Aninoasa mining museum– Rehabilitation of constructions, land and utilities", it has been finalized the stage of elaboration of the documentation for mine closure and rehabilitation of the land, and underground and above-ground works closure, dismantling and demolition of structures, constructions and utilities as well as the rehabilitation of the area. Also, has been finalized the stage of underground and aboveground connecting works physical closure, being to that along the museum works to be precede at buildings dismantling, demolition and rehabilitation, including the ecological rehabilitation of the area.

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From the existent buildings inside Aninoasa mine precinct, as fix assets (facilities) that served the exploitation activity in the perimeter, following assets were settled to constitute the objectives of the museum as a system, respectively (fig.1):

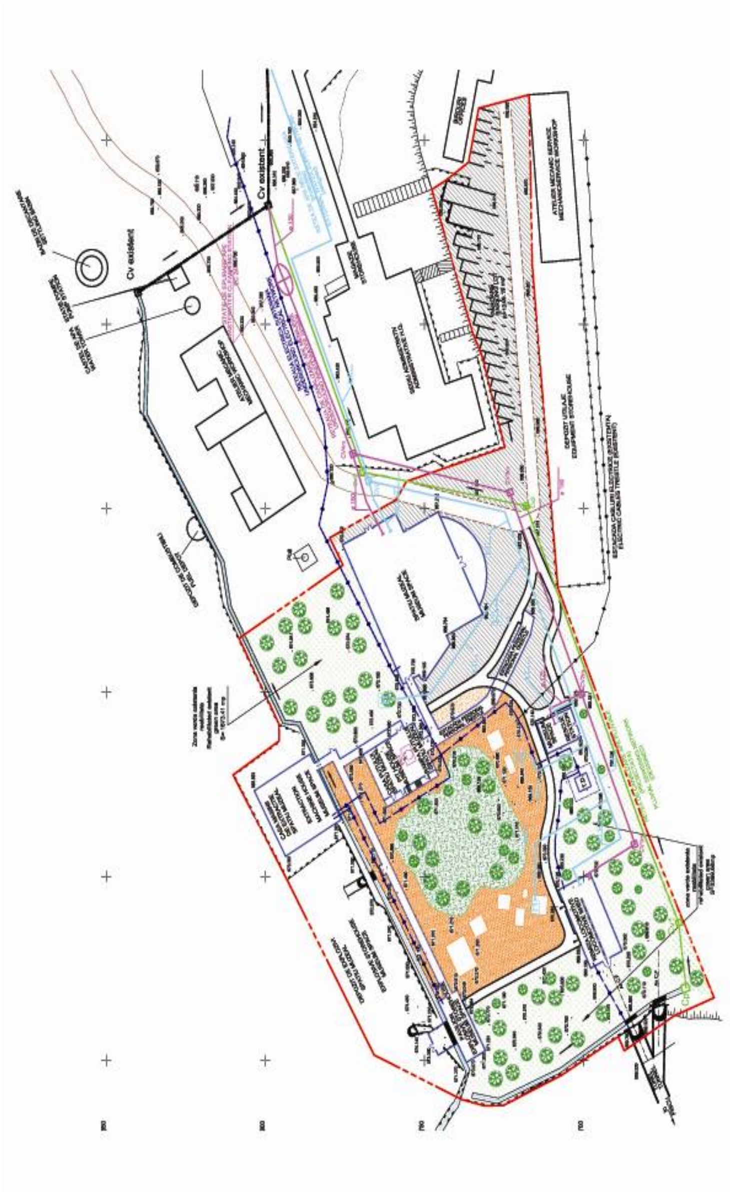

Fig.1. General plan of museum objectives inside old Aninoasa mine precinct

- Aninoasa North – Piscu tunnel;
- Electro-mechanical workshop, consist of electrical workshop house, mechanical workshop house and tool house;

- Hoisting – engine house of the North Main;
- Hauling shaft; hauling shaft house;
- Explosive warehouse;
- Locomotive shed;
- Rescue station.

In order to use the above mentioned as a museum objective, the buildings are to be rehabilitated and rearranged as fallows.

2. THE REHABILITATION OF UNDERGROUND MINING CONSTRUCTIONS

The rehabilitation of Aninoasa North – Piscu tunnel

Along mine functioning, the tunnel served as connection between Aninoasa North precinct and Piscu precinct, having the purpose of materials provision, waste transport and people traffic.

The tunnel it's situated on a maximum depth of 80m and measures 670m, excluding the 20m that measure the two double-entries with role of access into the tunnel (fig. 2).

In accordance with the rehabilitation works of the tunnel supporting, and the double entry, the following types of timbering will consist in sights seeing:

- Guss concrete timbering, along the length of 282m, inclusive the length of 20m as measures the double-entry;
- Prefabricated blocks masonry timbering, unpolished – L=269m and polished – L=40m, equivalent to the total length of the tunnel of 309m;
- Brick walls timbering, along the length of 54m, from which 40m unpolished and 14m polished;
- Combined timbering from SG armatures – wire mesh – anchors – jetcrete, equivalent to the length of 25m.

In case of tunnel work circuit, there is appointed the execution of present, to G.S.B. (Z)-5,2 profile, on the length of 20m in case of section 1 (area b), respective Rb-1a-3,1 profile, on the length of 43m in case of section 3 (areas a and b).

For exhibition of other types of timbering, applied in the current mining practice for preparation galleries execution, as well as different working technologies used for coal exploitation, the following equipments are appointed as exponents in the working circuit rigging, namely:

- Rolled section bead core SG.23 and wire mesh, half round timber set lagging in case of section 1 (area a), respective 1 and 2 (areas b);
- Trapezoidal frames of round timber and boards set lagging in case of section 2 (area a);
- Wooden frames with rectangular profile and binding with bulk heads/ board in case of section 3 (area a), for presentation of an exploitation strip of the chamber working face method;
- SJV 2500 piles, articulated metallic beams GSA 1250 and short beams GS 570, equivalent to two rows of beams in case of section 3 (area b), for the presentation of a longwall working face with individual timbering.

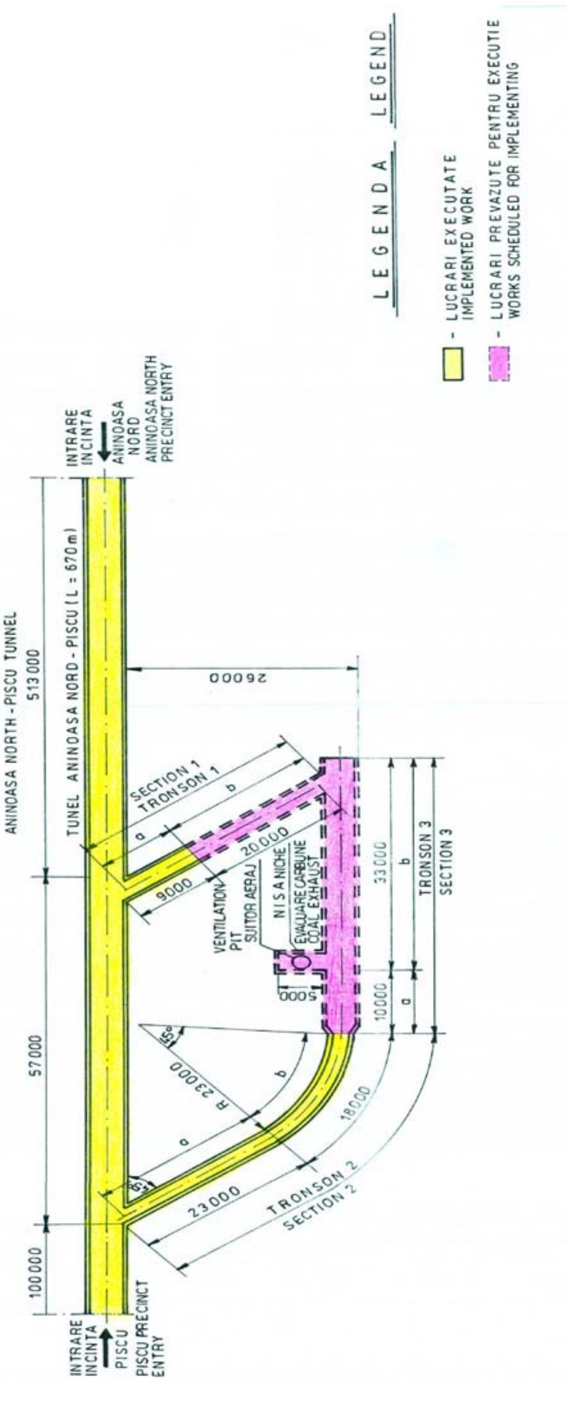

Fig.2. Aninoasa – North – Piscu tunnel and works circuit – volumes appointed for execution

The exploitation areas of section 3 inside the workings circuit are appointed to be fitted with scrappers for simulation of coal evacuation, TP-2 in case of chamber working face, respective TR-3 in case of longwall working face.

Also, for the simulation of coal evacuation system, at intersection of area a and b of section 3 of working circuit, there is appointed the execution of a 5m long bay, to GSB (Z) - 1a - 5,2 profile , and as a continuation of gravitational transportation flow, it is considered the execution on the depth of 1m, of a rectangular chute, fitted with one compartment, of transportation. Along the concreted bay will be mounted, at 0,7m, SG.23 laminated metallic beams, and for simulation of coal transportation a TR-3 conveyor, until the mouth of the wooden chute Ø170mm, with section of the profile of 1x1m.

On the period of tunnel functioning as a touristic objective, the airway of underground works will be done by a system "general depression", trained for this purpose by a tube airway raise Ø1000mm. The airway raise will communicate with "coal evacuation bay" and will be fitted with a centrifugal ventilator placed at its surface. For visiting the hole working and exhibit equipments assembly, as well as for facilitating the transition of the tunnel to Piscu precinct by the visitors, the tunnel will be fitted with a 4 cars train for people transport, towed by an accumulator locomotive, type LA – 4N.

Explosive warehouse

Designed for keeping the explosive materials and afferent equipment, the warehouse it's composed from 5 rooms, from which two for AG explosive storage, one for dynamite, one for fuse caps storage and a room for explosive manipulation.

The reconversion of the warehouse will consist of isolation by embankment of the two blind drifts, existent for warehouse access, reducing of the number of the rooms to 3, for presentation and keeping of the explosive materials – room 1, fuse caps storage – room 2, respective manipulation of the explosive materials– room 3.

3. THE REHABILITATION OF ABOVE-GROUND CONSTRUCTION (BUILDINGS REHABILITATION)

The buildings proposed for rehabilitation and keeping as touristic objectives are into an advance stage of dilapidation, requiring the consolidation of the supporting structure – hauling shaft house, or renovation of interior and exterior finishing, for consolidation and rehabilitation, as well as with alignment to the conditions imposed by the new destination of those inside the museum complex.

In the case of the location appointed for rehabilitation, the situation it's presenting as follows:

Electromechanical workshop

It is built by annexation of two different houses, the electrical workshop house and mechanical workshop house, having as supporting structure prefabricated concrete piles, which sustain prefabricated crane beams with "T" section, type GRA – 60A. The workshop is considered as starting point in the museum location line. The rehabilitation of the museum consists in accomplishing next activities groups and sub-groups, for accomplishing the ulterior facilities, namely:

- **rehabilitation of the building:**
    - o demolition of interior compartments, which destination was that of workshops, storage rooms, etc.;
    - o making access walls holes between workshops; arrangement of access doors for personnel and materials, compounds, products provision;

- consolidation of bad bearing walls, through STM 4mm welded wire coating and mortar M50-T;
- remaking of interior and exterior coatings;
- remaking of roof hydrofuge and thermal insulation, replacement of plateworks, chutes and stove-pipes – in case of electromechanical workshop;
- remaking of the wooden frame type roof and replacement with roof boarding.

- **re-compartmentalizing execution for museum functioning:**
  - making of access and guest receiving halls, fitted with dressing room and tourist information and souvenirs and self-adhesive distribution;
  - offices;
  - women, men and locomotor – handicap people rest rooms;
  - exponents hall;
  - snack-bar, with kitchen, storage spaces, rest rooms for the personnel, unisex rest room;
  - dinning hall;
  - open and enclosed terrace.

Hoisting – engine house

Having role of housing the hoisting – engine necessary to the transportation of Aninoasa North main shaft, the height being underground + ground floor, with two openings and 6 bays, the hoisting – engine house has a supporting structure from prefabricated reinforce concrete, with marginal closures made of brick masonry, and reinforced concrete walls at the underground floor level. Its roof it's a hydro-insulation ridged roof cast plate.

Its destination as a museum objective of the hoisting – engine house will be the use of the afferent rooms for exponents, its rehabilitation needing:

- consolidation of bearing walls by wire mesh coating;
- remaking of interior walls and ceilings coatings, by applying interior washable paint; remaking of exterior walls and sockets coatings, by applying exterior washable paint;
- remaking of roof hydrofuge and thermal insulation, replacement of plateworks, (chutes and stove-pipes);
- replacement of the windows and doors carpentry with PVC carpentry and termopan windows.

Rescue station

It's an older building, ground + first floor, the supporting structure is made up of bearing brick walls, an intermediary reinforced concrete floor and a ridged frame wooden roof, with marginal closures made of brick masonry; the infrastructure of the building it is done from continuous concrete under walls foundations.

The functioning of the rescue station will be the one of museum space – where will be presented the development plan of the mining area, respective consultancy for professional re-orientation for the population in this area.

Therefore, the building will use for exponents halls, traffic ways, offices, stores, rest rooms.

The rehabilitation works consist in:

- remaking of interior and exterior walls and ceilings coatings; replacement of the windows and doors carpentry with PVC carpentry and termopan windows; applying of interior and exterior washable paint;
- replacement of the present insulation with roof boarding and protection polyethylene foil; replacement of plateworks;

- replacement of the paving with vitrified non-skid gritstone in the exponent's halls and traffic ways, cast-mosaic floor in the ground-floor store rooms, stoneware floor in the rest rooms and parquetry in the offices.

Hauling shaft house

Having role of housing the hauling main shaft Aninoasa it is a one floor height building, the resistance structure is made up of prestressed reinforced concrete frames, poles, and cross and longitudinal reinforced concrete beams, on which it's a ridged roof cast plate. The marginal closures are made of brick masonry, the infrastructure of the building is an under poles foundation, made of plain concrete block, that integrates the pillows too; the roof floor it's fitted with a bordering hollow of secondary beams, through it's raising the hoist tower.

The functioning of this building will ensure the access to the rooms of the shaft circuit; the rehabilitation will consist in execution of a stairhead with access stairs. Also, the rooms will have role of storage, and not being strictly part of the museum circuit.

In the same time, there will be done consolidation works, replacement works of interior and exterior walls and ceilings coatings, by applying exterior washable paint. At the roof will be replaced the hydrofuge and thermal insulation, and plateworks.

Locomotive shed

It's a ground floor building, having the resistance structure made of bearing concrete walls, wooden floors and a ridged frame wooden roof, with metal sheet covering; the infrastructure of the building it is done from continuous concrete under walls foundations.

The building is in an advanced phase of degradation, therefore it will be demolished and rebuilt entirely, at the same height and placed at ca. 20m from the tunnel, with CFI ax of locomotive shed to coincide with the tunnel one. This way, the locomotive shed along its initial functioning will have be the point of tourist embarking for underground transportation through the cars train for people transport.

Toward its destination, the locomotive shed will be compartmented with effective embarking space of the tourists, storage room for equipments and women and men dressing rooms, with the possibility of tourists' access from other exponent halls, offices, workshops and outside too.

Beside the rehabilitation of the existent buildings, Aninoasa mining museum precinct supposes the execution of **pavilion – explosive warehouse**, as a new ground floor building, with the role of protect tourists' underground access. Also, it supposes the execution of a **covered trestle**, with connection role between the main museum objectives, namely: electromechanical workshop, existent access walkway, hauling shaft house, hoisting – engine house, and afferent pavilion of the explosive warehouse.

CONCLUSIONS

Entering into the rehabilitation actions of mining objectives affected by mine closure, for Aninoasa mine precinct, there were designed the transformation works of the this one into a museum.

With the purpose of keeping and giving back through different addressing forms of exploitation activity within the mine and not only, a series of the present buildings will necessitate partial or integral rebuilding (Aninoasa North – Piscu tunnel, electromechanical workshop, hauling shaft and hoisting – engine house of the Aninoasa main shaft, explosive warehouse, locomotive shed, rescue station).

Beside the main activity of mining exponents presentation, a part of the buildings inside the precinct will house information activities – professional documentation and orientation for the people in this area (rescue station), and for touristic personnel will be installed a preparation point and snack-bar (electromechanical workshop).

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TEACHING STRATEGIES IN EARTH SCIENCE EDUCATION - THE PROBLEM-BASED LEARNING METHOD

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LORINT CSABA***

Abstract: *The present study explores a new teaching strategy, about how university students perform with the Problem-Based Learning method in a Computer-Based Learning environment. During the study, technology (as algorithmical and parallel processing computer based codes) was used as a catalyst for university students. One of the most important aspects was related to the way in which educators can effectively address the problem of passive learning among geoscience students. The study examined and compared students' experiences in a Computer-Based Learning environment, specifically customized for the third year Geophysics course. The results of the study showed that, overall, a technology-enhanced learning environment has the potential to help students in changing their views about what teaching and learning science means, with direct repercussions on their individual accomplishments.*

1. INTRODUCTION

Despite delays in the implementation of computer based courses and practical assignments (as part of Computer Based Learning strategy), during the past decade educators have dedicated significant resources and time in introducing computers as a sine-qua-non asset during the conjugated effort of mentor teaching and student learning. However, the current trend in teaching at university level consists of the use of technology as a secondary asset, following the chalkboard or projector. Moreover, the traditional student testing and evaluating methodologies seem to prevail over methodologies relying on technology.

Teaching strategies at university level focused unilaterally on teacher-directed activities (Barr & Tagg, 1995) are expressed by the delay in the implementation of more advanced pedagogical concepts. The use of the Computer Based Learning strategy assumes that instructors themselves understand how technology can be best used to improve student learning. Without an in-depth understanding and use of technology as a teaching asset, educators end up

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by simply reproducing the old curriculum via the new medium of information technology (Harris, 1999). As a response, the students do not get engaged in mutual communication with the instructors and absorb information by means of bulk assimilation of concepts.

A case study of how technology was used in the Geology Department at the University of Petrosani, Romania to teach the third year course of Geophysics (with special focus on the lab activities) is presented in this paper. The study aims to offer a broader understanding of how technology can act as a fundament for initiating changes in teaching and learning at university level.

2. TEACHING WITHIN A COMPUTER BASED LEARNING ENVIRONMENT

Practical assignments give the students the chance to learn by doing while promoting scientific thinking. As part of the practical assignments, laboratory instruction represents a primary vehicle for learning. Without labs, student learning would be impeded, as would their personal development. At the time when the present study was performed (2004) computers have been used for the Geophysics classes for about 3 years. Additionally to the traditional use of computers for presentations during the lectures, lab assignments were partially computer-assisted for learning purposes. The main learning stream used data selection, processing and interpretation.

2.1 Teaching Borehole Geophysics on specific case studies

In the particular case of the Geophysics topic, borehole geophysical information comes as a complex response of the investigated lithology to electrical, electromagnetic, radioactive and acoustic stimuli. Wire logging and data acquisition have been used for a long time for a wide range of lithological characterization, from oil reservoirs to mineral deposits. Particularly good results of lithology classification were obtained based on borehole geophysical signal (Busch et al, 1997, Baldwin et al 1990, Rogers et al, 1992, Wong et al, 1995, Gonclaves et al 1995).

Borehole geophysics, performed in the Petrosani basin since 1956, is a crucial support to the geological interpretation of the basin. Extensive wireline geophysical investigations offer a significant amount of information from individual wells to rely upon. The result is a comprehensive database of geophysical data, spanning several decades of investigation. This database was used as part of the teaching process, during laboratory assignments. Assignments consisted of interpretation of lithology based on analysis of the geophysical signal.

The essential aspect of the challenge during the study was to engage the students in a competitive activity, where creativity, initiative and will of success would play the major role. The focus was placed on the Problem-Based Learning (Hmelo-Silver, 2004, Merrill, 2007) strategy, as part of the Computer Based Learning training.

The interactive and challenging competition involved cooperation between students from the Earth Sciences and Computers Science Departments. Initially, two strategies of data analysis and lithology interpretation were chosen: algorithmical data processing versus parallel data processing. The algorithmical data processing of the geophysical signal was performed with a proprietary code licensed as educational resource for student teaching purpose. For the parallel data processing, a computer code which we later called PROCoal was created. After comparing the versatility of each strategy as an option of data processing, the student's choice was to use the parallel data processing. Further changes and tuning of the PROCoal were performed, while enhancing the students' ability to select, analyze and interpret geophysical data.

3. BOREHOLE GEOPHYSICS ASSIGNMENTS AND THE PROBLEM-BASED LEARNING METHOD

The problems in Problem-Based Learning are typically in the form of cases as real-world challenges. The Problem-Based Learning is both critical and creative, as it gives the student the ability to analyze, synthesize, and evaluate information. Through Problem-Based Learning, students learn how to apply that information appropriate to a given context.

The method consists of giving students a real world problem – in this case the interpretation of the geophysical plots from the Petrosani coal basin. Students have to determine what they need to know to address the case. Then, they go searching for the information.

The students, after the lecture classes, begin to familiarize with the geophysical plots at the labs. By applying the theory, they have to distinguish the different types of rocks on the plots, according to their features. This is done firstly by visual observation.

The next stage is represented by the search for characteristic plots (belonging to boreholes from certain mining perimeters, grouped along specific alignments able to reveal important geological features). This task is performed in small groups, each group analyzing a different mining perimeter in the basin. Once the students chose the representative borehole logs, they digitize the selected paper copies and aquire the data nedded for further interpretation. At this stage, they have to choose between two strategies of data processing and interpretation:

- the use of a propietary algorithmic software, which assesses the litology based on a previous model:
- the use of parallel processing with neural networks, as an artificial intelligence system, able to learn litologies by itself and recognize them further. Parallel processing is a highly customized approach, related directed to the type of data characteristic for a certain geological setting. As part of the learning process, students are incouraged to seek resources for developing a specific processing code for the data to be analyzed.

Initially, both models are chosen by different groups of students. The obtained results are compared at the end of the project with the response offered by the cores of rocks.

3.1 Design of a Back Propagation Neural Network code

While mimicking the brain structure mechanisms and functions (Golden, 1996), artificial neural networks (ANN) use parallel processing during computation to generate answers usually regarded as data classification. The artificial neural networks are useful for classifications, pattern recognition and optimization problems which tolerate a certain imprecision, based on important amount of training data, unable to be grouped according to complex and rapid rules (Nigrin, 1993, Haykin, 1994, Chandrasekaran and Josephson, 1994, Bishop, 1995). In the case of classification problems, the most popular learning algorithm is the back-propagation (Rumelhart et al., 1986, LeCun, 1985).

The artificial neural networks are extremely versatile when analyzing, interpreting and recognizing on the geophysical logs types of lithology and certain marker seams. Given the complex sedimentology and tectonics in the Petrosani Basin, the utilization of ANN to determine the relationships between geophysical data and lithology becomes a valuable tool. The predicted relationship can be used further to delineate marker seams and correlate the geological layers in the basin.

As part of the Problem-Based Learning strategy program, PROCoal was designed by students from the Computers Science Department. PROCoal is a computer code based on a Back Propagation Neural Network, BPNN, feed forward with supervised learning (also known as training). The important features of a neural network are the basic computing elements (nodes or neurons), the network structure describing the connections between the computing elements and the training algorithm used to adjust the connection weights between nodes in order to solve a particular problem. The computational capabilities of such a network are suitable for learning the input-output mapping of patterns, while the network's response is insensitive to noise or distorted patterns.

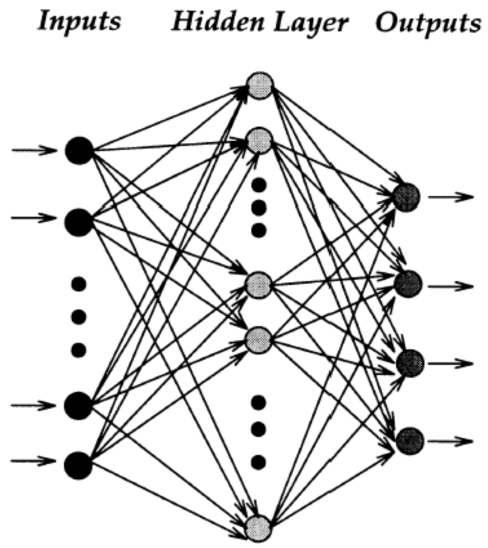

Figure 1 Multilayer neural network structure with one layer of hidden nodes.

Figure 1 represents the conceptualized model of a multilayer neural network structure with one layer of hidden nodes. The output computation of a single computing element is presented in Figure 2.

The structure of the network used for the lithology recognition in the present study was organized into five layers: one input layer, three hidden layers and an output layer.

The input layer consisted of five nodes, corresponding to five geophysical parameters (self potential, resistivity, natural-gamma radiation, neutronic-gamma radiation, gamma-gamma radiation for density) able to characterize the selected types of rocks. The input layer constitutes the input data for the nodes in the second layer, which is also the first hidden layer.

The three hidden layers of sigmoidally activated neurons had, respectively 15, 7 and 5 nodes, fully interconnected to all the input nodes, among the hidden layers and to the output nodes. There was no connection between nodes in the same layer. The output data of each hidden layer were input for the next layer. Although there is no theoretical limit to the number of hidden layers in a neural network, practice has demonstrated that a maximum of three hidden layers suffice to solve even complex classification problems (Benaouda et al., 1999).

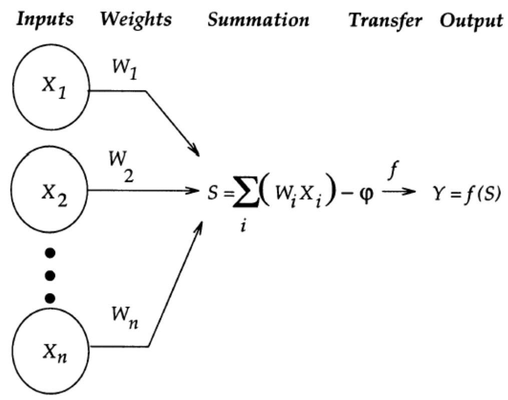

Figure 2 Output computation of a single computing element (after Benaouda, 1999).

The output represents the overall response of the network to a given data for the input (first layer). As a result, this type of neural network is called a feed-forward network. The output layer had 5 nodes representing 5 categories of lithofacies. Five digital classes were separated on the geophysical logs as coal, sandstone (0-25% clay), argyle (75-100% clay) and gradients defined as argillaceous sandstone (25-50% clay) and sandy argyle (50-75% clay).

3.2 Why PROCoal?

At present, the majority of ANN research studies are based on analysis of numerical data (such as for porosity) and less are used for lithological classification. In addition, no previous attempt of algorithmically or parallel data processing of wire logging signal was made in the Petrosani Basin. The PROCoal code integrates geophysical signals and drill core data with additional geological information to determine the lithology in the basin and the presence of marker seams. The lithology and marker seams were assessed by parallel processing of the geophysical data, using a Back Propagation Neural Network (BPNN), feed forward neural network with supervised learning.

4. PROBLEM BASED LEARNING INTERACTION DURING BOREHOLE GEOPHYSICS ASSIGNMENTS

The main goals of using the PROCoal code as part of the learning process through assignments were: to emphasize the relation between practical facts and theories; to provide knowledge about the geology of the Petrosani basin and ability in recognizing of the lithological character from the geophysical logs and to use in a creative manner computer skills combined with research skills.

Through the study, the students were expected to perform at a broad range of levels, encompassed by:

- Knowledge, as ability to recall together geological and geophysical facts. Lithology and specific marker seams within the basin had to be recognized first visually on the plots and than by implementation of computer codes.
- Application, as ability to use ideas in concrete situations (why is necessary to recognize the lithology and markers on the plots?)

• Evaluation, as ability to judge the chosen procedure (parallel/algorithmic data processing)

The class activities consisted on a diversified array of tasks, such as:

• Case study (each team assesses a certain mining perimeter in the Petrosani basin)

• Laboratory exercise (gathering the data - transforming old paper plots into digital data by scanning)

• Library research (documentation about lithology, geological structure of the deposits, marker seams in the area, as observed and described by various authors)

• Field research (re-examination of the drill cores, to validate/invalidate/compare the lithology obtained by geophysical investigation with the real core.

The learning process was supported by access to references and bibliography, practice sessions, models and demonstrations. The assignments were evaluated for each group through oral presentations spaced during the successive stages of the experiment and written projects at the end of the term. An immediate positive outcome of the experiment was the class atmosphere, which became competitive between the different teams of students, while cooperative within the same team.

The instructor's input, as leader of the experiment, was estimated at about 25% of the learning process. The students were expected to provide the rest of 75% as hands-on problem-solving, through consistent completion of assignments and responsibility shared with the team.

5. RESULTS

One step of the learning exercise is achieved when the students compare the results of the two approaches and draw up their own conclusions regarding the use of the parallel versus algorithmic data processing.

This approach places explicit responsibility on the students' shoulders for their own learning. Creating assignments and activities that require student input increases the likelihood of students being motivated to learn.

The use of the ANN in the Problem Based Learning approach challenges the students with stimulating activities and develops critical thinking abilities, based on the following:

• Cognition and problem solving: students learn to organize and retain information, and to use strategies to solve the lithological correlation problem;

• Apprentice in borehole geophysics: students learn through laboratory work and computer use; discuss their findings, then write up the obtained results in teams. Like professionals, they exchange work in progress for suggestions and mutual learning;

• Geophysics-related intellectual skills: the methods of inquiry, categories of analysis;

• Purpose, motivation: students discover why the lithological correlation problem is worthy of their attention as further professionals.

6. CONCLUSIONS

After two years of implementation of the Problem Based Learning as strategy, the results were positive. The teaching/learning process benefits from a new approach of problem solving and the versatility of the choices enables the students to think more critically. The lithological interpretation it is not seen any more as a difficult visual task or, at the other end, as a simple computer solving situation. Borehole geophysics was chosen for this experiment based on the previous experience of the instructor with teaching Geophysics. Usually, geophysical branches as gravimetry, magnetometry, electrometry, seismometry benefit from more dissemination through well-documented and widely attractive case studies. The borehole is somehow a less attractive topic and the students are usually not very enthusiastic about exploring it. The use of Problem Based Learning sparked a more focused and enthusiastic answer from the students, with visible benefits on the assimilation of information during the learning process.

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NEW METHODS IN GEO TECHNIQUE STUDY OF OPEN PIT AND WASTE DUMP

CSABA R. LORINȚ*

Abstract: *The diversification of modern techniques and technologies let them to be utilized in geo techniques domain. This study tries to put in practice some modern techniques like GPS data acquisition, virtual 3D modeling using Surpac Vision program and finally, application of GeoTecB software (geo technique software), all useful in open pit and waste dump stability studying. Also, this research checks up the proposed methods by one case study, made on Comarnic Poieni interior waste dump.*

WORK METHODOLOGY

Geotechnical studies have on base data like:

- area's situation plan and field morphology
- geometrical parameters of the waste dump (height, embankment angles)
- mechanical and physical parameters for dump's rocks, for the fundamental rocks and for the vegetal soil

All these information are processed with different computing algorithms for stability.

Because for obtain these information we need laborious operations and techniques which lead to height materials charges and also needed time, in this study we tried to use cheaper and faster ways.

Because in more case the information referring to area's situation plan and field morphology from technical documentation were inadequate (they do not contain actual information regarding instability phenomena), we can update this using modern and easy techniques like GPS data acquisition, virtual 3D modeling using Surpac Vision program and finally, application of GeoTecB software (geo technique software).

THE GLOBAL POSITIONING SYSTEM DATA ACQUISITION

The Global Positioning System (GPS) is a U.S. space-based radio navigation system that provides reliable positioning, navigation, and timing services to civilian users on a continuous worldwide basis freely available to all. For anyone with a GPS receiver, the system will provide location and time. GPS provides accurate location and time information for an unlimited number of people in all weather, day and night, anywhere in the world. The GPS is made up of three parts: satellites orbiting the Earth; control and monitoring stations on Earth;

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and the GPS receivers owned by users. GPS satellites broadcast signals from space that are picked up and identified by GPS receivers.

Each GPS receiver then provides three-dimensional location (latitude, longitude, and altitude) plus the time. Individuals may purchase GPS handsets that are readily available through commercial retailers. Equipped with these GPS receivers, users can accurately locate where they are. Disaster relief and emergency services depend upon GPS for location and timing capabilities in their life-saving missions. Surveyors, geologists and countless others perform their work more efficiently, safely, economically, and accurately using the free and open GPS signals.

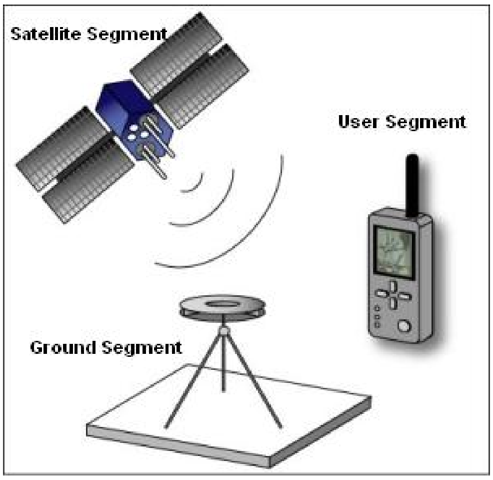


Figure 1. The three components of the GPS

Data collection systems provide decision makers with descriptive information and accurate positional data about items that are spread across many kilometers of terrain. By connecting position information with other types of data, it is possible to analyze many environmental problems from a new perspective. Position data collected through GPS can be imported into geographic information system (GIS) software, allowing spatial aspects to be analyzed with other information to create a far more complete understanding of a particular situation than might be possible through conventional means.

Benefits of GPS using:

- GPS data collection systems complemented with GIS packages provides a means for comprehensive analysis of environmental concerns.
- Environmental patterns and trends can be efficiently recognized with GPS/GIS data collection systems, and thematic maps can be easily created.
- GPS data can be quickly analyzed without the preliminary requirement for field data transcription into a digitized form.
- Accurate tracking of environmental disasters can be conducted more efficiently.
- Precise positional data from GPS can assist scientists in crustal and seismic monitoring.
- Monitoring and preservation of endangered species can be facilitated through GPS tracking and mapping.

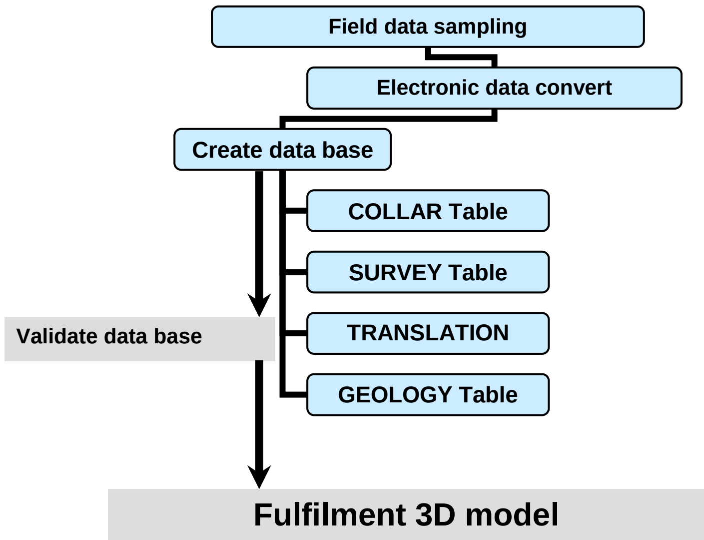


Figure 2. 3D model fulfillment algorithm

SURFACE MODELING

The surface modeling and map updating are possible using 3D modeling program „Surpac Vision”. Surpac Vision is the flagship product of Surpac Software International. It is a 3D Mine Design and Geology package used in the mining industry for such tasks as surveying, mine design (open pit & underground), blast design (open pit & underground), exploration, and tailings simulation.

Usually, Surpac require 2 mandatory tables: collar and survey.

The information stored in the collar table describes the location of the drill hole collar or name/number of measurements point.

The survey table stores the drill hole or measured point name survey information.

So, using this package we can get tridimensional images for the topographic area and also cross sections with the actual morphology of field and geological parameters of open pit or waste dump. Without these programs for obtain all these date we would need enlarged topographic measurements.

STABILITY ANALYSIS

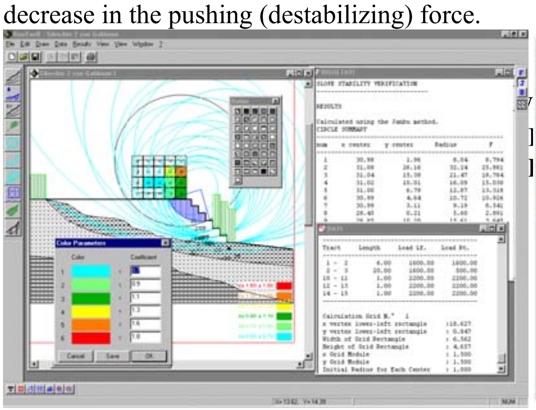
For stability analysis can us a geotechnical soft like GeoTecB, based on Fellenius, Jambu & Bishop methods. Thus the iterative processes we established the minimum value for the stability factor and the position of the critical landfall surface.

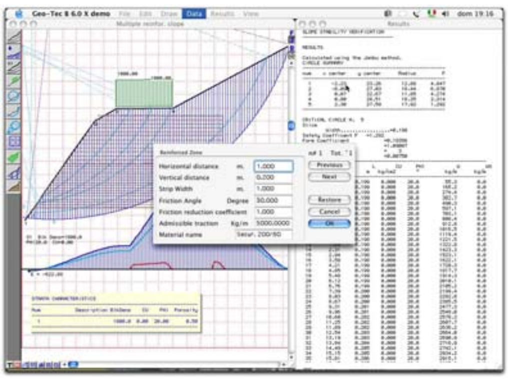
GeoTecB allows the verification of stratified slopes in the presence of water beds and loads.

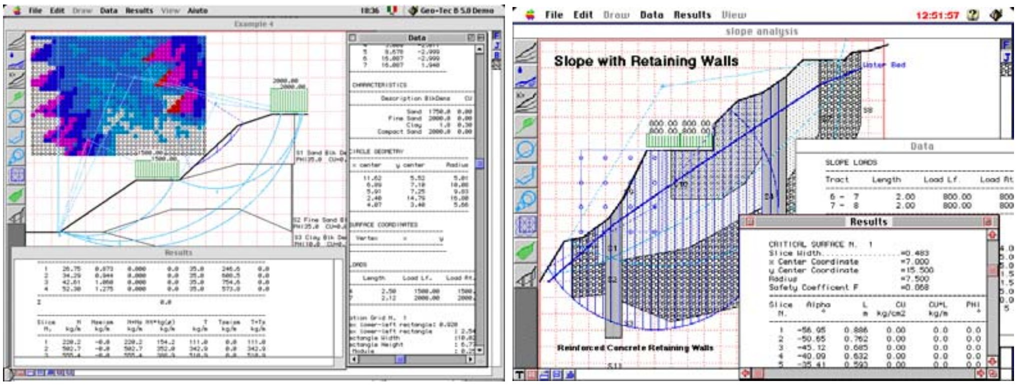
Work begins by inserting geotechnical parameters. Following phases involve the definition of slope and strata geometry. Circular or polygonal slide surfaces can be inserted and modified using the mouse. GeoTecB can automatically calculate a series of circles through an interpolation between two selected extreme circles.

GeoTecB calculates the safety coefficient for each surface using Janbu, Bishop and Fellenius methods and graphically identifies that having the lowest coefficient as the critical surface. Strata dimensions are defined by the user but GeoTecB is able to perfect dimensions according to the theoretic foundations of the adopted method. GeoTecB also allows an immediate verification for seismic zones using a pseudo-static method.

When using the Janbu calculation method, it is possible to apply an external force to the slope. This force is considered an increase in the resisting force (stabilizing) and not a decrease in the pushing (destabilizing) force.







GeoTecB simplifies a goetechnician's most complex calculations by reducing them to a series of very simple operations. It produces a large number of safety coefficients from which GeoTecB selects the most important one, allowing an almost statistic verification very different from that produced manually or with other programs.

CASE STUDY FOR THE INTERIOR WASTE DUMP COMARNIC POIENI

Physic and mechanical parameters for quarry’s rocks, for base fundament rocks and for vegetal soil were established through national standards in the Geo mechanical laboratory from Petrosani University. To make the stability analysis we might choose as values for geotechnical parameters, the medium values or processed valued which eliminate the big variation limit for geotechnical parameters. For this reason after we estimate we consider the values from Tab. 1.

Table 1. *Physical and mechanical characteristic used in stability analysis*

Rock type	Natural Humidity		
	Volumetric weight. γ_{nat} (cN/cm^3)	**Cohesion c, (daN/cm^2)**	**Interior friction angle φ (grade)**
Bauxite	2,08	31,06	40,49
Material from dump	1,83	0,28	26
Base field	1,90	0,50	29,0
Vegetal soil	1,78	0,39	21,5

Then, were collected 34 data sets from minimum 4 satellites (a necessary thing for measurement precision) using GPS Magellan Spor Trak Map.

Table 2. *The 34 point coordinates measured with „Magellan SporTrak™ Map" GPS*

Point	Coordinates			Point	Coordinates		
	x	y	z		x	y	z
1	449154	357121	779	**18**	449244	357187	784
2	449175	357116	781	**19**	449263	357187	778
3	449217	357113	782	**20**	449288	357190	778
4	449235	357128	781	**21**	449329	357201	794
5	449243	357146	781	**22**	449298	357172	794
6	449257	357161	779	**23**	449293	357153	800
7	449274	357174	780	**24**	449244	357084	803
8	449253	357172	780	**25**	449247	357054	814
9	449232	357159	780	**26**	449264	357078	813
10	449210	357138	778	**27**	449327	357131	814
11	449182	357129	777	**28**	449373	357210	802
12	449160	357129	773	**29**	449400	357215	803
13	449141	357126	772	**30**	449426	357236	800
14	449151	357149	778	**31**	449397	357267	798
15	449171	357151	784	**32**	449358	357263	801
16	449205	357162	789	**33**	449348	357258	805
17	449224	357178	786	**34**	449374	357237	793

Figure 3. GPS Magellan Spor Trak Map

These data were converted in an agreeable system by virtual tridimensional modeling program Surpac Vision. The obtained topographic surface was divided in 8 transversal sections witch will serve for the stability calculation.

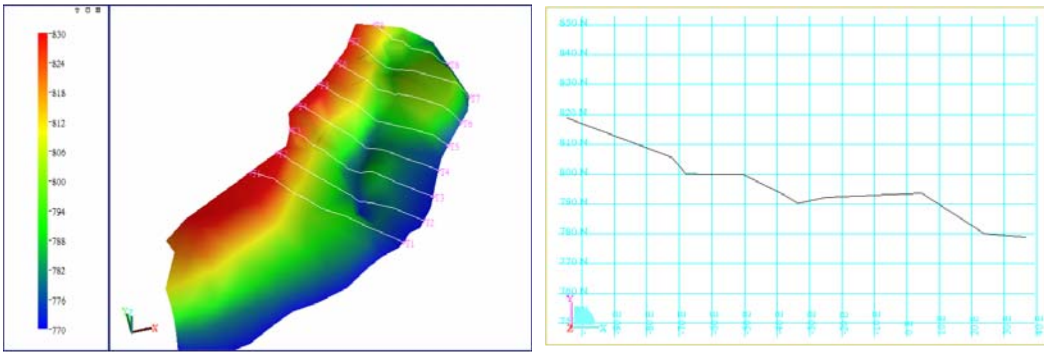


Figure 4. Example of topographic surface and cross sections obtained whit 3D modeling program „Surpac Vision"

STABILITY ESTIMATION

Stability analysis for the quarry and the waste dump Comarnic Poieni was made using a geotechnical soft Geo Tec B, and using Fellenius, Jambu and Bishop methods and the iterative processes. Was established the minimum value for the stability factor and the position of the critical landfall surface. The graphical materials used (the topographic surface map and 8 cross sections $T_1 - T_1$, $T_2 - T_2$, $T_3 - T_3$, $T_4 - T_4$, $T_5 - T_5$ $T_6 - T_{6,}$ $T_7 - T_7$ and $T_8 - T_8$) were obtained on some acquisition data with GPS, graphical processed with "Surpac Vision" obtaining a total surface over 16 ha, with 50% bigger than the surface specified in the anterior technical documentation.

The stability calculation were made for two sections T5-T5 and T7-T7 because in these areas were identified big problems and the stability calculation consequences the determination of stability reserve for dump's embankment in natural humidity condition and also in saturation condition. The numerical results were centralized in table 3 and the graphics in figure 5-7.

Table 3. The values for the critical landfall surface of the interior dump section T5-T5 and T7-T7 computed with GeoTechB soft

Section	**T5**		**T7**	
	Natural Humidity	**Saturation Humidity**	**Natural Humidity**	**Saturation Humidity**
Fellenius	1,49	1,15	1,15	0,89
Jambu	1,55	1,20	1,22	0,95
Bishop	1,52	1,17	1,21	0,93
H [m]	26		24	
α [^{0}]	40		45	

The two sections were choosing also because the embankments geometrical elements (the height and the embankment angle) have the highest values.

It can be observed that the stability factor for the studied areas from Comarnic Poieni interior dump is between "the normal" and "at stability limit".

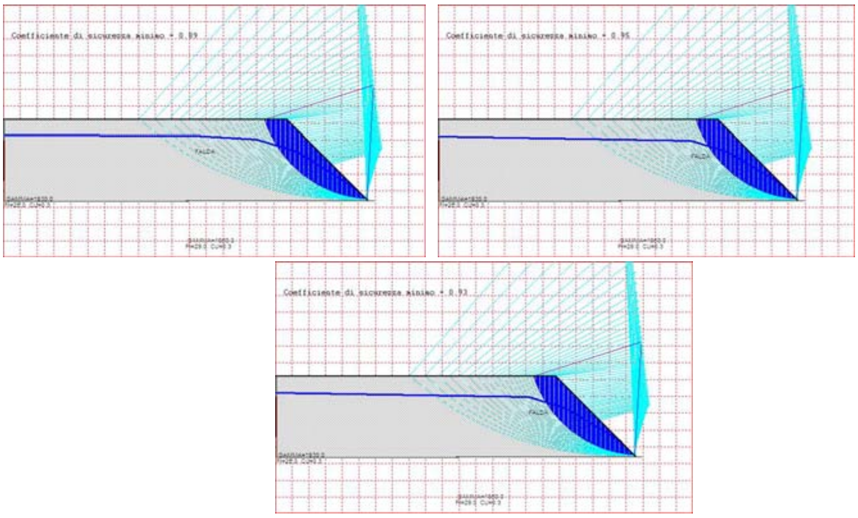


Figure 5-7. Example by graphical model of stability estimation for saturation humidity in T7-T7 section

Also it can be point out the precarious technical stage of embankment from section T7, even for the natural humidity conditions when the stability factor is approaching to the equilibrium limit and for the saturation conditions they appear instability problems.

CONCLUSIONS

The fact that the determination were made for the natural humidity W=15,90% and a saturation coefficient of S=0,56 (for the dump material) and W=22,43% and S=0,70 (for the base field) denote the possibility of increasing the humidity until saturation which lead to the modification of physical mechanical parameters, a negative influence for the stability factor. In that direction there are recommended leveling, compaction measurements for the interior dump for obtaining parameters which can assure a satisfactory stability factor. Also conforming to the rehabilitation variant of waste dump, are recommended stabilization and re vegetation works using naturalistic engineering techniques.

The obtained results allow making the conclusion that using these techniques, the stability calculation for the quarry and waste dump can be made with minimum investments and preparing works as topographical detailed works, making the situation plans, etc.

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THE TITLE OF THE PAPER WILL BE WRITTEN WITH CAPITAL LETTERS, CENTERED, AT 7.0 cm FROM THE UPPER EDGE OF A4 FORMAT, ALONG THE ENTIRE WIDTH, TIMES NEW ROMAN, 14 POINTS, BOLD.

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* *Prof., Ph.D., University of Petroşani, Romania*

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1.1 Subchapter. Subchapters can be used in the text, numbered with the number of the chapter and a number showing the number of the subtitle within the chapter. The subchapters are Times New Roman, 11 points, bold.

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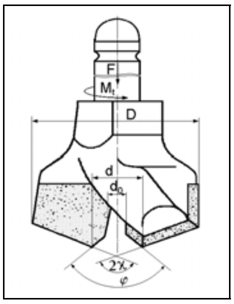

Figure 1. Detachable bit

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$$X^2 + Y^2 = Z^2 \qquad (1)$$

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[1]. Dardac, N.; Barbu, T. - *Monedă, bănci şi politici monetare*, Editura Didactică şi Pedagogică, Bucureşti, 2005

[2]. Greuning, H.; Bratanovic, S. - *Analyzing and Managing Banking Risk*, The World Bank, Washington, 2003

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