# *CONTENTS*

# CYANIDE BETWEEN NECESSITY AND RISK

*Ioan BUD*, Simona DUMA*, Irina SMICAL*, Ioan DENUȚ*, Vasile OROS**

***Abstract**: Cyanide is a substance that becomes toxic under certain conditions. However, the general population regards it as a strong poison, irrespective of the conditions in which it is used. This opinion which has been taken over and enhanced by the mass media which turned it against the mining industry which has a long history of using cyanides. However, there is very little evidence that cyanide is a major risk factor for the health of the staff handling it or for the environment, particularly when all the measures provided by occupational safety regulations have been met. The hostile attitude towards the use of cyanide in the processes of precious metal extraction is at present artificial and exaggeratedly supported and encouraged by some non-governmental environmental organisations and by some mass-media trusts which invoke the cyanide-related mining incidents. As it results from the official findings, such incidents were caused either by designing errors or by errors in operating the mining tailings.*

***Key words:** cyanide, risk factor, powerful poison, gold, environment, accident*

## 1. Introduction

The term *cyanide* has a strong impact on the population when used in connection with mining activities, since it is perceived as and associated with strong poison. Only 13% of the world cyanide production is used by the mining industry; the rest is used for other activities which are not affected by its bad reputation. An important quantity of cyanide is released into the environment resulting from exhaust gases, vegetation, burning of biomass, the chemical industry, cigarette smoking, etc. We can therefore conclude that cyanide is obtained industrially in a number of different forms, but it can also be generated by various other human activities as well as plants.

Cyanide is a substance that becomes toxic under certain conditions. However, the general population regards it as a strong poison, irrespective of the conditions in which it is used. This opinion which has been taken over and enhanced by the mass media which turned it against the mining industry which has a long history of using cyanides. However, there is very little evidence that cyanide is a major risk factor for the health of the staff handling it or for the environment, particularly when all the measures provided by occupational safety regulations have been met.

The human body can come into contact with cyanide by breathing, drinking water or eating contaminated foods, as well as by touching it directly (cutaneous contact).

Gold is one of the water-insoluble noble metals. For the dissolution of gold are required a complexant, such a cyanide, which stabilises the gold species in solutions, and an oxidant, such as oxygen. The cyanide content in the solution necessary in order to assure the dissolution can be of 350 mg/l or 0.035 % (as NaCN) [15], or 0.01%...0.05% in very diluted solutions [32].

Alternatives complex agents for gold, such as chlorides, bromides, thiourea and triosulphate, which are less stable, require oxidants and more aggressive conditions for the dissolution of gold. These reactives pose risks for both health and the environment and are also more expensive. All these account for the prevalence of cyanide as a chief reactive in the extraction of gold from ores ever since it was first used industrially, in the late 19th century [15] (studies on the solubility of gold in cyanide solutions were performed in 1783 and the process of cyanide use was patented in 1887).

## 2. Chemism

The term *cyanide* defines the chemical substances containing the cyano anion ($CN^-$) which consists of one carbon atom triple-bonded to a nitrogen atom. Cyanide can be either produced naturally or manufactured industrially and under certain quantitative conditions becomes a poison, depending on the dose and exposure time, like many other chemical substances. However, what makes cyanide special is that it is one of the most active and quick-acting toxic substances. The best known substances included in the generic name of cyanide are cyanhidric acid (HCN), a gas, and two simple salts – sodium cyanide and potassium cyanide – used for the extraction of precious metals.

Cyanhidric acid and the ($CN^-$) ion are free cyanides.

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Thiocyanate (sulphocyanide or rhodanide) is a chemical compound found in many foods and plants. It results from the reaction of a free cyanide with sulphur. This reaction occurs in the industrial wastes containing cyanide (the mining tailings ponds) and in the human body after the ingestion of cyanide. Thiocyanate has a lower toxicity than free cyanide. The anthropogenic sources of thiocyanate are the herbicides and the industrial processes, while the natural sources are represented by the decomposition of certain plants (the ones belonging to the Brassica genus, such as cabbage, mustard, etc.). Thiocyanate is biodegradable in both aerobic and anaerobic environments, as well as in water and soil.

Table 1 explains the notions of *total cyanide*, *weak-acid dissociable (WAD) cyanide* and *free cyanide*.

<table>
<tr><th colspan="4">Table 1 Species of cyanide [15]</th></tr>
<tr><td rowspan="4">Total cyanide</td><td colspan="3">Complexes very stable in medium-acidity environments – with Au, Hg, Co and Fe</td></tr>
<tr><td rowspan="3">WAD cyanide (dissociable in weak-acid environments)</td><td colspan="2">Low- or medium- stability complexes – Cd, Cu, Zn cyanide</td></tr>
<tr><td rowspan="2">Free cyanide (the most toxic form)</td><td>$CN^-$</td></tr>
<tr><td>$HCN^-$</td></tr>
</table>

The toxicity of cyanide is generated by the release of the ($CN^-$) ion. Discharged into the environment, this anion has a low affinity to alkaline metals, but a high affinity to the ferric ion ($Fe^{3+}$) and other metals. Within the tailings ponds there are large quantities of ferric ions and metals, which creates the conditions for the generation of cyanide compounds which are less stable and more toxic.

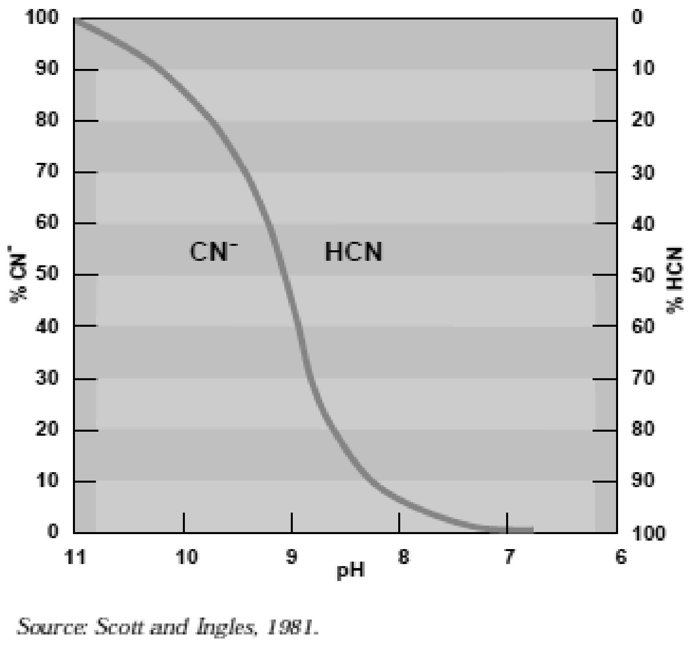


**Figure 1** *Chemism of cyanide in the ore preparation process*
*$CN^-$/HCN balance by pH [32]*

Solid sodium cyanide dissolved in water generates $Na^+$ ions and $CN^-$ anions. $CN^-$ anions combine with hydrogen, resulting cyanhidric acid (HCN), a toxic component which can volatilise and disperse into the air ().

According to Figure 1, almost all the free cyanide is present as HCN when the pH of the solution is 8 or below. At a pH = 10.5 nearly all the free cyanide is $CN^-$ and the technological process must be conducted in such a way as to keep the pH lower than this value. In normal pressure and temperature conditions the HCN and $CN^-$ concentrations are equal for pH = 9.4 [32].

## 3. Exposure to cyanide and risks

The most common sources of cyanide exposure for the population are active and passive smoking as well as the combustion of all kinds of materials, such as wood, plastic, biomass, etc. In Romania biomass is burnt frequently and, which is even more serious, people burn plastic, rubber, etc. in their households. Another important source of exposure to cyanide is represented by exhaust gases from vehicles.

People can also risk cyanide intoxication if they eat big amounts of certain fruit seeds and stones, such as apricots, apples, peaches, etc., as well as certain vegetables, such as white beans or tapioca roots, etc. The cyanide content of certain varieties of white beans amounts to 100 to 3120 mg/kg [37]. Other sources mention values ranging from 100 to 170 mg/kg which are common in the white beans in the U.S. ([8] from [1]). Concentrations of 660 mg/kg sulphocyanide were found in plants from the Brassica genus ([29] from [1]), while in the soil treated with rapeseed residues the sulphocyanide concentration was 6 mg/kg ([3] from [1]).

The cyanide contents of the cassava- or manioc-rich diet of the Tukanoan Indians of north-western Amazonia is estimated to be in excess of 20 mg/day ([6] from [1]).

Many studies, among which *Toxicological Profile for Cyanide* [1], published in the U.S.A. in 2006, and *Study on Assessment of the Health Risk for the Population* [16], published in Romania in 2005, conclude that the highest risk of cyanide intoxication is represented by the exposure to exhaust gases and smoking, both active and passive. 9209 cases of occupational disease were reported in Maramureş county in the period 1972-2011. Occupational morbidity was represented chiefly by silicosis. However, such cases of occupational disease due to cyanide intoxication were recorded [27].

The risk of disease increases greatly with the presence of non-rehabilitated tailings pond which continuously generate silicogenic dust rich in heavy metals and acid draining, i.e. acid waters and heavy metals.

A contrastive risk analysis published in the *Annual Environmental Report* for the years 2010 and 2011 – *The Situation of the Environmental Factors in Romania, Chapter 8. Environment, Health and Quality of Life* (www.anpm.ro) – reveals a disturbing situation. In 2010 there were 260 cases of acute occupational intoxication with pesticides, of which 15 deaths. In the year 2011, according to the records, there were 185 cases of intoxication with pesticides, of which 19 deaths. But these figures don't seem to vex anyone.

The risk of death or serious disease due to cyanide exposure is smaller than that of being killed by lightning [22].

A study of the risk of disease due to exposure to industrial population, issued by "Iuliu Moldovan" Institute of Public Health of Cluj Napoca for the period 2000-2004 (a period during which the company SC Transgold SA Baia Mare was operating) emphasises that the normal concentrations of thiocyanate in urine were somewhat exceeded. Thus, the figure for non-smokers was 5 mg/l, while for smokers it was 16 mg/l. The significant difference between the two categories draws attention on the risk of exposure to cyanide from other sources than industrial activities. In percentage, in a small number of persons from the studied groups the normal values were increased (4.8% of Bozânta-Săsar group and 5.2% of the Baia Mare group) [16]. Another series of studies published in *Toxicological Profile of Cyanide* [1] reveals significant differences between the level of thiocyanates in the plasma, saliva and urine of smokers vs. non-smokers (about three times higher).

Cyanide gets into or out of the bodies of animals and humans in the same ways. Also, cyanide has not been reported to cause cancer [1].

Based on many studies on the metabolism of cyanides carried out on animals it has been concluded that, once in the body, cyanide rapidly gets into the blood circuit. An important part of it (80%) converts into thiocyanate, which is less toxic and is eliminated through urine. The rest combines with hydroxocobalamin, carbon dioxide, cyanhidric acid and formic acid (HCOOH) and converts into cyanocobalamin or vitamin B12.

## 4. Anthropogenic sources of cyanide

American scientists [1] identified the following main sources of human-generated cyanide discharge:

- into the water: metal processing industry, steelworks, organic chemical industry. The contribution of mining to water pollution is considered negligible, although the total content of cyanides in the exfiltration of tailings ponds ranges between 0.3 and 310 mg/l ([12], [25] of [1]). Cyanide is treated with hydroxide and sodium hypochloride and converted into less toxic compounds. The non-punctual cyanide sources are generated by farming and road maintenance (the salt used as antiskid). It was estimated that the quantity of ferrocyanide used as antiskid in wintertime in the north-eastern part of the U.S. amounts to 907 t ([10], [13] of [1]).
- into the air: the chief source of cyanide are the exhaust gases from vehiclest, followed by the

plastic manufacturing and cyanhidric acid manufacturing industries, then the steel manufacturing industry, coal burning, oil refining, municipal waste burning, cigarette smoking, the deposits of household wastes, the pesticides used for farming, etc. It is assessed that 20 t of cyanhidric acid are generated by the mining industry, while cigarette smoking generates 340 t, pesticides 62 t and incineration 81.6 t ([10] of [1]). Worldwide, the emissions from the mining industry are assessed at 20000 t ([18] of [1]), while the emissions generated by biomass burning – cyanhydric acid and acetonitrile – amount to 1.678 million tons ([6] of [1]), i.e. 83.9 times higher. In 2003 the American industry monitored for cyanide emissions reported an estimate 517 t of cyanhidric acid and 142 t of cyanide compounds discharged into the atmosphere [1].

- into the soil: the household waste deposits and the salt used as antiskid on roads. A total of 1340 t cyanide compounds and one ton of cyanhidric acid were estimated to have been discharged into the soil in the U.S. in 2003. [1].

## 5. Alternatives to cyanide

For the gold and silver processing industries cyanide represented an alternative to amalgamation, allowing bigger productions, protection of staff's health and lowering the environmental impact. The first pilot plant – a demonstration station – operated in Queensland, Australia, in 1888, and the first plant was built in New Zealand. In South Africa the gold production rose from 300 ounces in 1890 to 300000 in 1893. In the U.S. the production rose from 1.7 million ounces in 1892 to 4.5 million in 1905. Cyanide marked a significant progress in gold processing, transforming it into a much more economical process than gravitational separation or amalgamation. [20].

The cyanidation technology has been improved and, at the same time, many processing methods have been considered as alternatives to cyanidation. Among these, the use of sodium thiosulphate, which was applied commercially in the period 1875-1895 for silver ore processing. In the same context the risks and costs of such technologies have been analysed and assessed. Based on these studies, the Relative Risk Scores obtained indicated that the cyanidation technology poses the smallest risk of all technologies. The thiosulphate technology is far riskier than the cyanide technology. (Table 2)

**Table 2** *The risk by types of lixiviation [20]*

| | Lixiviation system | Relative Risk Score |
|---|---|---|
| 1 | Sodium cyanide/lime | 41 |
| 2 | Bromine/bromide/sulphuric acid | 159 |
| 3 | Chlorine/chloride | 193 |
| 4 | Ammonium thiosulphate/ammonia/copper | 219 |
| 5 | Thiourea/Ferric sulphate/sulphuric acid | 1574 |

In the 1980s and 1990s many academic and industrial studies focused on the ammonium thiosulphate technology. This is a much more complex but more efficient technology than the thiosulphate technology [20].

Some alternatives based on thiourea, thiosulphates, thiocyanates, etc. used for the extraction of precious metals from gold and silver concentrates involve both costs, and environmental and health risks that are far bigger than the cyanide technology [14].

## 6. Cyanide in mining and specific standards

In the year 2000, when the Baia Mare cyanide spill happened, 460 mining plants worldwide were using cyanide and 90% of the world gold production was being obtained through cyanidation [22]. The worldwide production of cyanhydric acid was estimated to be 1.4 million t/year, of which 18% was converted into sodium cyanide for the mining industry. The rest of 87% was used for other industries (adhesives, electronic parts, products for fire control, cosmetics, paints, varnishes, nylon, pharmaceutical products, plexiglas, table salt and salt for road management, etc.).

The worldwide production of NaCN in the period 1998-2012 and the one envisaged for 2015 are illustrated in the table and chart below. These show that the constantly increasing production has tripled as a result of the market's demand.

**Table 3** *Worldwide production of sodium cyanide*

| Year | 1998 | 2006 | 2009 | 2012 | 2015 |
|---|---|---|---|---|---|
| NaCN production in tons | 280000 | 500000 | 760000 | 797000 | 915000 |

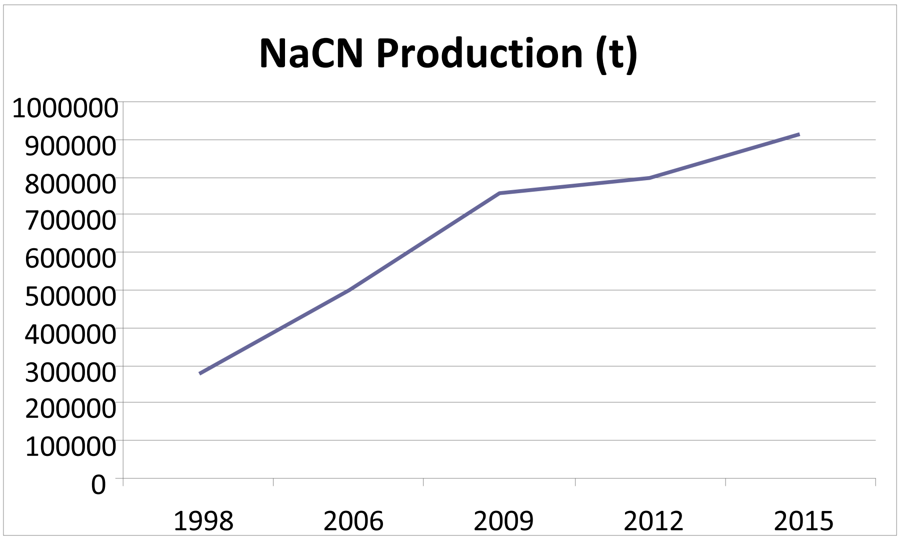


**Figure 2** *The NaCN production in tons in the period 1998-2013 and estimates until 2015*

Biodegradation is a very important process for the cyanide in the surface waters and depends on many factors, such as: the cyanide concentration, the pH, the temperature, the existence of nutrients and the acclimatization of microbes. The cyano ion is toxic for microorganisms at concentrations of 5...10 mg/l ([17]; [19] of [1]); through acclimatization their tolerance to this compound increases ([24] of [1]). Certain pure microorganism cultures can degrade the cyanide solutions with low concentrations in aerobic or anaerobic media ([8], [9],[11]of [1]).

In waste waters, such as the ones from the gold tailings ponds the concentration of cyanide ions and sulphocyanides is naturally lower due to the acclimatization of the local microflora in the ponds. ([2],[23],[28]of[1]). A number of microorganisms which have been identified, such as Actinomyces, Neisseria, Pseudomonas, Thiobacillus, etc., are capable of absorbing, converting and/or precipitating the cyano ion, thecyanate and the thiocyanate. Some of these species, for example Pseudomonas, can use the cyano ion and the thiocyanate as a basic source of carbon and nitrogen. Therefore they are very efficient for cyanide degradation. In fact the commercial applications for the degradation of cyanide from the mining waste waters is based on Pseudomas ([2] of [1]).

Cyanide regulations include a very large range and differ from one country to the next, depending on the level of the knowledge about this phenomenon and various psychological aspects. For example, in Romania, according to Governmental Resolution (HG) 730/1997, in January 2000 when the Baia Mare spill occurred, the acceptable maximum total cyanide contents for the waters discharged in the emissary was 0.05% mg/l, and the maximum acceptable content of free cyanides in drinking water was, according to STAS 1324/1991, 0.01 mg/l. Both these regulations were still valid in 2000 [5]. The laws became somewhat more permissible with the introduction of Law no. 458/2002, modified through Law no. 311/2004, which translates into Romanian legislation Directive 98/83/CE on the quality of water intended for human consumption, which provides for drinking water a maximum admissible content of total cyanides of 0.05 mg/l, and of free cyanide of 0.01 mg/l. For the waste waters HG 352/2005 provides an admissible content of total cyanides of 0.1 mg/l in the waters discharged into a NTPA-001emissary. In the U.S.A. the maximum admissible concentration of cyanides for drinking water is 0.2 mg/l, 20 times higher than in Romania.

In 2006, Directive 2006/21/CE recommended that the concentration of dissolvable cyanides in a weakly acid medium in tailings ponds should be reduced to the lowest possible level by using the best available techniques (BAT's). For the tailings ponds which have obtained occupancy permits and were operational in 2008, this. Directive provides a gradual reduction of the concentrations of the respective cyanides at the discharge of the waste in the pond, as follows:

- as of 1 May 2008 – maximum 50 mg/l;
- as of 1 Mai 2013 – maximum 25 mg/l;
- as of 1 May 2018 – maximum 10 mg/l.

The maximum admissible value of 10 mg/l is also imposed for the plants which were to obtain their environmental permits after May 1, 2009. These measures are applied nationwide through the translation of the Directive into Government Resolution (HG) no. 856/2008.

The comparison of the cyanide concentrations to the maximum admissible values provided by the national and international regulations has generated much controversy since 2000 when the cyanide spill at the Aurul tailings pond of Baia Mare occurred. The maximum cyanide concentrations measured by the Public Health Direction of Maramureş County in the wells in Bozânta village, downstream from the tailings pond, on 11.02.2000 were of 0.66 mg/l [5]. This, according to the Romanian standards, means 66 times higher than the maximum admissible concentration, which is alarming. However, by the American standards this value would be only 3.3 times higher than the maximum admissible value. If we compare things further, we can equate the consumption of two litres of water with a cyanide content of 0.66 ml/l with smoking three cigarettes or eating 100 grams of white beans.

According to UNEP [36], the cyanide concentration in wells was 80 times higher than the maximum admissible value on February 10th, but on February 20th it dropped within admissible limits.

According to the tests performed by SGA Maramureş, the free cyanide content in the slime pulp discharged in the period 31.01-01.02.2000, right after the leak, was 126 mg/l, whereas the total cyanide content was 405 mg/l. A concentration of 19.14 mg/l was detected in the Lăpuş river, while in the Someş river, at Cicîrlău, the concentration amounted to values between 13.26 and 0.082 mg/l [5]. The water treatment station of Szolnok, Hungary was closely monitored; in such conditions no exceeding of the Hungarian cyanide contents was found (in 2000 Hungarian standards provided a maximum of 0.1 mg/l, which was 10 times more permissive than the Romanian standards). The cyanide pollution incident did not endanger the Hungarian public system of water distribution [36].

According to the information presented by SC Romaltyn Mining SRL, until 2006 the cyanide content in the tailings pond was between 200 and 300 mg/l. At present within Romaltyn plant there is a plant which destroys the cyanide from the slime pulp resulting from processing, which uses the INCO ($SO_2$-Air) procedure, through which the concentration of dissociable cyanides in mildly acidic medium will be reduced to the lowest possible level – below 10 mg/l, by using the best available techniques (BAT). The detoxification tests of the slime pulp with a total cyanide content of 358 mg/l and a WAD of 328 mg/l, showed the reduction of the cyanide concentration to the value of 1.43 mg/l total cyanide, 0.0705 mg/l WAD and 0.129 mg/l free cyanide [33].

As a result of treating the water in the Aurul tailings pond in the station of Romaltyn Mining, the waste waters discharged in the emissary are going to meet the quality standards provided by the regulations in force, i.e. NTPA 001 (0.1 mg/l). The previous technology, used in 2006, did not include destroying the cyanide in the slime pulp or treating the pure water from the tailings pond within the treatment station before being discharged in the emissary.

In our opinion, in consideration of all the facts presented above, the whole outlook on the 2000 Baia Mare spill changes completely. Presented as a very serious, even catastrophic, incident, it brought about a serious prejudice to Romania's image, the mining activity and Baia Mare's image. We must emphasize the evolution of the cyanide concentrations presented by UNEP 2000 [36]. The figures indicate that upon discharge in Aurul tailings pond the cyanide concentration was 19.4 mg/l, after which the concentration decreased to 7.8 mg/l, to suddenly soar again to 32.6 mg/l in Csenger, Hungary. It seems that nobody has tried to find an explanation for these spectacular differences.

In the same context there is the „mystery" of the dead fish in Hungary. On the internet ostentatious photos of the dead fish are still associated with the name of our city, thus continuing to manipulate the public opinion. Nobody bothers to ask the simple question: why did the fish die only in Hungary and up to there, in the Lăpuş and the Someş nothing like this happened? This incident generated hysterics against cyanide due to the denaturation of facts by both the mass media and the authorities. The incident was serious indeed, but the errors in connection with its causes and the way in which it was managee are even more serious. Anywhere in the world there would be a penal trial and the people found responsible or guilty for these errors would be punished. Today this spill is considered even more than a serious incident, a catastrophe, a disaster, etc., even without a court order which, based on technical expert's reports, should prove responsibilities and guilt, and establish the level of damage. Which makes any opinion ungrounded, a mere speculation made by unauthorised people.

The current situation of the cyanide in the Aurul tailings pond shows significantly lower cyanide concentrations, decreasing from 250 mg/l in December 2005 (when the last quantities of waste were deposited in the pond) to WAD cyanide values of 0.155 mg/l and total cyanide values of 0.228 mg/l, which are average values. These values were measured on samples of water drawn from the tailings pond in January-February 2014 from 23 piezometres all around the dam. After drawing the samples, the chemical analysis was performed by Romaltyn laboratory. These tests confirm the capacity of cyanide degradation and infirm the opinions according to which cyanide continues to exist for longer periods of time and pose a major risk for the environment.

**7. The history of cyanide use in Baia Mare**

In February 1939, based on the research studies carried out by the Research Laboratory of Lupeni on the gold and silver ore from Săsar mine, was opened, close to the current premises of SC Romaltyn Mining SA, a cyanidation plant with a capacity of 100 t/day, with tools delivered by two companies, Humboldt from Germany and Dorr Oliver from Holland. Subsequently this plant was expanded to 220 t/day in 1958, 550 t/day in 1959 and 990 t/day in 1960 [4]

In 1962 the cyanidation capacity was increased to 60,000 t/year and gold-rich pyrite concentrates from IM Brad were received for processing as well. The Săsar Preparation Plant, which in 1999 became part of Aurum Baia Mare Mining Exploitation, processed both ore and gold-and silver-rich concentrates using classic cyanidation, while Transgold company used the “CIP” technology [4].

Processing the ore from Săsar Mine using direct cyanidation:

**Table 4** *Processing the ore from Săsar mine using direct cyanidation [4]*

| Year | Sasar ore (t) | Specific NaCN consumption (g/t) | Total NaCN consumption (kg) |
|---|---|---|---|
| 1939 | 28144 | 1412 | 39739.3 |
| 1940 | 29887 | 1201 | 35894.3 |
| 1945 | 5226 | 1302 | 6804.3 |
| 1955 | 94717 | 2115 | 200326.5 |

Processing concentrates using cyanidation:

**Table 5** *Processing gold-and silver-rich concentrates using cyanidation [4]*

| Year | Săsar (t) | Nistru, Ilba, (t) | Apuseni (t) | Total Concentrates (t) | Specific NaCN consumption (g/t) | Total NaCN consumption (Kg) |
|---|---|---|---|---|---|---|
| 1955 | 629 | - | - | 629 | 2115 | 1330.3 |
| 1965 | 14587 | - | 24621 | 39208 | 10600 | 415604.8 |
| 1970 | 18725 | - | 19799 | 38524 | 8346 | 321521.3 |
| 1975 | 17980 | 26890 | 27300 | 72170 | 6120 | 441680.4 |
| 1980 | 18900 | 7210 | 32100 | 58210 | 6041 | 351646.6 |
| 1985 | 18100 | 2570 | 35400 | 56070 | 6140 | 344269.8 |
| 1990 | 14650 | 5790 | 33320 | 53760 | 7208 | 387502.1 |
| 1991 | 13485 | 9465 | 17835 | 40785 | 6369 | 259759.7 |
| 1992 | 13285 | 12380 | 15691 | 41356 | 6518 | 269558.4 |
| Year | Sasar, ore (t) | Meda tailings pond (t) | Concentrates (t) | Total (t) | Specific NaCN consumption (g/t) | Total NaCN consumption (Kg) |
| 1999 | 84228 | 1370160 | 145179 | 1599567 | 1400 | 2239393.8 |
| 2000 | 142539 | 1073022 | 36641 | 1252202 | 1590 | 1991001.2 |
| 2001 | 272800 | 1031068 | 14477 | 1318345 | 1350 | 1779765.8 |
| 2002 | 268530 | 104796 | 49036 | 422362 | 1680 | 709568.2 |
| 2003 | 248623 | - | 32951 | 281574 | 2320 | 653251.7 |
| 2004 | 215056 | - | 63974 | 279030 | 2260 | 630607.8 |
| 2005 | 277439 | - | - | 277439 | 2300 | 638109.7 |

Based on the figures in Tables 4 and 5, one can note that significant quantities of cyanide were used for the mining activity, amounting to 300 to 400 t cyanide/year in the 1960s, 2000 t in the years 1999 and 2000, and 600 to 700 t cyanide/year in the period 2002-2005. The use and handling of such large quantities of cyanide involves risks as well, but these did not become apparent either in the production activity concerning the staff or for the population or the environment. This assertion is supported by the answers received from the authorities who are in charge of risk exposure.

The real environmental risk for Baia Mare has to be associated with the acid draining generated by the acid waters loaded with heavy metals which lead destroy the water life and load the soil. The accumulation of heavy metals in plants and animals and implicitly in the products derived from plants and animals is an important risk factor for people's health and causes the degradation of the environment.

The mines were closed down but the risk entailed by the mining activity was completely ignored.

The existence of the non-rehabilitated tailings ponds, closed-down mines or mines which have not been conserved properly poses real short-, medium- and long-term risks. The Central Pond, located upstream from Baia Mare city, is an important source of pollution with acid waters, heavy metals and dust. All these sources of long-term risk exceed by far the risk of using cyanide over a definite period of time, in a controlled system.

## 8. Conclusions

Cyanhydric acid has existed since life appeared and developed on Earth, as a precursor of amino acids. Carbon and nitrogen are present in the molecules which form the basis of all forms of life. Unlike synthetic chemical substances, which do not decompose easily and accumulate in the trophic chain, cyanides naturally convert into less toxic substances as a result of a number of physical, chemical and biological processes.

In the last years, the interest of the public, the mass media and the political class in the environmental implications of using cyanide for mining has grown immensely. In fact, the use of cyanide worldwide has a very long history. Although it is a highly toxic substance, its impact on the environment as a result of the mining industry has been smaller than that of other chemical substances. The studies on the incidents which occurred in Australia, Canada, New Zealand and the U.S. concluded that only three deadly accidents were caused by cyanide over a period of over 100 years. Following the investigation of these cases, the suspicions were inconclusive [32].

The hostile attitude towards the use of cyanide in the processes of precious metal extraction is at present artificial and exaggeratedly supported and encouraged by some non-governmental environmental organisations and by some mass-media trusts which invoke the cyanide-related mining incidents. As it results from the official findings, such incidents were caused either by designing errors or by errors in operating the mining tailings.

The information about the persistence of cyanide in the mining waste deposits for longer periods of time has been circulated vehemently lately, creating the impression that there is an imminent risk in any situation where cyanide is used for mining activities. The tests on the evolution of the cyanide content in Aurul tailings pond over the period 2006 – 2013 shows a significant decrease of this content, of 1250 times.

It is remarkable that nowadays in all European countries any precious metal extraction process using cyanidation can be carried out only if the rules and regulations in force are observed. Also, regulation documents for such activities can only be issued if all the safety and protection rules concerning the environment and health have been met.

The development of any human society takes place in close connection with the development of its economy if this can be sustained in safe conditions for both the environment and the population.

After the 2000 Baia Mare cyanide spill the international interest in the risks involved by using cyanide has become very strong, leading to the issuance, by many organisations including governments, NGOs, academic organisations, consultants, industries, financial institutions, between the years 2000 and 2002, of a Cyanide Code for the mining industry (http://www.cyanidecode.org). In 2005 this document was signed by 14 companies, and in May 2013 by 135 companies based in 47 countries (38 mining companies, 16 cyanide manufacturers and 81 cyanide transporters). 95 gold mines were certified according to Cyanide Code in the year 2013. The Code was recognised as well by G8, a group of countries including Canada, France, Germany, Italy, Japan, Russia, United Kingdom and U.S.A. In any business it is the investor, i.e. the one who pays and assumes the highest risk, who has biggest interest in diminishing any risk. To this end the banks, which take the best precautions when they finance mining projects (which involve high investments), have adopted and applied the Cyanide Code as a good practice rule. Thus, IFC (International Finance Corporation) as part of World Bank, as well as BERD (the European Bank for Reconstruction and Development) and other financial organisations apply the Cyanide Code in their operations. The Code recommends a maximum value of 50 mg/l cyanide in the tailings ponds, compared to 10 mg/l recommended by the European Directive at present.

The management of the risk associated with the use of cyanide for mining activities has been studied extensively and it involves an engineering approach, close monitoring and the implementation of the best practices in order to prevent and minimise any potential discharge of cyanide into the environment.

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# DISTRIBUTION AND IMPLEMENTATION OF DRILLING MACHINES AT THE QUARRY BENCHES

*Mostafa M. ELBEBLAWI*, Mohamed A. SAYED*, Mostafa T. MOHAMED*, Mohamed E.I. ABDELRASOUL***

***Abstract*** *In many times, a good distribution and implementation of equipment and machines along the quarry faces is considered a challenge to the mining engineer or the quarry operator. Quarry equipment includes drilling machines, air compressors, loading equipment including loaders and/or excavators, in addition to the transportation fleet of high-tonnage trucks. These equipment cost large investments in the production processes and need high skill to be efficiently distributed and operated. In this investigation, study of the optimum distribution of the available drilling machines of different specifications and capacities along the faces of the lime stone quarry of Assiut Cement Company has been carried out. A mathematical model has been used to find several alternatives and to choose the best alternative. Calculation of the minimum number of drilling machines that satisfy the annual production plan is an important step to decrease production costs.*

***Key Words:*** *drilling machines, modeling, quarry.*

## 1. Introduction

Production cycle in most of the quarries includes the processes of drilling, blasting, loading and transportation. Rock drilling is the first operation carried out. Blast holes are drilled with adequate geometry and distribution within the rock mass to accommodate the suitable explosive charges and their accessories [1,7,8].

Considering drilling equipment, most of the drilling cost is time dependent rather than product cost dependent. Hence, higher –priced drilling equipment based on new technologies that save time are economically justified. Rate of penetration (ROP) includes the hours after a bit reaches hole bottom divided by the distance drilled until the time of tripping the bit out of the hole. By this definition, ROP includes actual drilling time plus time spent in reaming, surveying, and making connections. This means that, this drilling time includes time spent in actual drilling (rotating time) in addition to non-drilling (non-productive) time. This is of great importance to the drilling engineer and operation management [2].

There are three basic elements, which must be considered in evaluating a drilling system, they are [3, 4]:

1- Production schedules, operating conditions, and rock types encountered.

2- Equipment productive capacities including pattern size, tons of material effected per hole, drill production rate in meters drilled per shift or hour, and drill availability and utilization %.

3- Capital costs and operating costs including repair, maintenance, and related storage costs of spare parts and drill steel.

It has been advocated that operations research facilities in describing and analyzing the behavior of a system by constructing appropriate models and predicting future behavior using these models. Studies on the Queuing, Markov and reliability models lead to the conclusion that with the help of operations research, an appropriate mathematical model for situations, processes and systems can be developed. Then, the model can be tested and operated by changing variable values to implement optimization of the parameters. In the present era, optimal use of resources is essential and operations research can facilitate taking proactive decisions to make the system profitable and competitive [5].

In this investigation, study of the optimum distribution of the available drilling machines of different specifications and capacities along the faces of the lime stone quarry of Assiut Cement Company has been carried out. A mathematical model has been used to find several alternatives and to choose the best alternative. Calculation of the minimum number of drilling machines that satisfy the annual production plan is an important step to decrease production costs[7,8].

## 2. Formulation of the problem

In order to formulate any problem as a model, the following parameters must be defined [4, 6]:

1- The variables in the model,

2- The constraints,

3- The objective function.

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***1- The variables***

The variables are defined as follows:

TPY is production rate or material mined per year (ton per year)

MPH is material produced per hole, ton/hole

N is the number of holes drilled

MDH is meters drilled per hole = bench height + sub-drilling, m

$T_m$ is total meters drilled = N × MDH, m

DR is drilling rate, m/hr

TDH is total drilling hours in one bench per year

Oh is operating hours

PAT is possible available time (total calendar time – holidays/year), days

AOT is available operating time = PAT – holidays, days

OR are operational restrictions

LDM is long drill moves

PT is personal time

OND is other non-drilling time

NDT is net drilling time

NDT = AOT – OR – LDM – PT – OND

***2- The constraints***

The model has several constraints in form of equations. The equations are formulated basically for the purpose of calculations.

$$\%\text{Availability} = \frac{AOT}{PAT} \times 100 \qquad (1)$$

$$\%\ \text{Utilization} = \frac{AOT - OR - LDM - PT - OND}{AOT} \times 100 \qquad (2)$$

$$\%\ \text{Operated of total time} = \text{Availability} \times \text{Utilization} \qquad (3)$$

$$\text{Oh/shift} = \text{Utilization} \times \text{hr/shift} \qquad (4)$$

***3- The objective function***

The objective function of the model is to minimize the number of drills realizing the production plan required. The minimum number of drills needed is expressed as follows:

$$\textit{Minimum number of drills (MND)} = \frac{\sum_{i=1}^{I} (TDH)_i}{PAT / drill \times Avail. \times Utili.} \times 100 \qquad (5)$$

where:

$(TDH)_i$ is total drilling time in one bench, hrs., where i = 1,2,3,…. $I$, in this case i = 3 benches.

Avail. is the availability %

Utili. is the utilization %

No. of hours per day = 8 hours

**3 Collection of field data**

The present study has been carried out on limestone quarry of the Assiut Cement Company that operates 300 days per year, one drilling shift per day. It is required to produce 5 million tons of limestone per year to be used in the cement factory. To identify the production plan requirements; the quarry is divided into three benches upper, middle and lower bench. Each bench produces one third of the annual production plan, but they have different bench heights. The height of the upper bench is 26 m, the height of the middle bench is 30 m and the height of the lower bench is 35 m.

The evaluation will start with review of some of the drilling machines available as grouped into three classes based on the drill hole size capability and model of the machine. Table 1, presents rotary percussive blast hole drills that are available and some of pertinent information for the drills listed. Calculations of the drilling patterns, the material effected by each borehole according to the different bench heights and the average drilling rate for the three drills are illustrated in Table 2. Average density of limestone is given to be 2.2 $t/m^3$.

In case of this quarry, the annual production is distributed between the three classes of drills as follows: 33.76% by Compare, 16.59% by Atlas Copco 660 and 49.65% by Atlas Copco 460. 33% of the annual production is required from each of the upper and middle bench and 34% from the lower bench.

**Table 1** *Representative blast hole drills by class.*

| Class | Name and model | Typical bit size, mm (in) | Tube diameter, mm | Tube length, m |
|---|---|---|---|---|
| C-1 | Compair Holman | 111 (4.5) | 76 | 3 |
| C-2 | Atlas Copco Roc 606 | 152 (6) | 89 | 3 |
| C-3 | Atlas Copco Roc 460 HF | 152 (6) | 89 | 3 |

**Table 2** *Drilling patterns and material effected by each borehole*

| Class | Pattern size, m | Material per hole, ton | | | Average drilling rate, m/hr |
|---|---|---|---|---|---|
| | | Upper Bench | Middle Bench | Lower Bench | |
| C-1 | 6.5 × 4.5 | 1673 | 1931 | 2252 | 20 |
| C-2 | 8.5 × 5.5 | 2674 | 3086 | 3600 | 15 |
| C-3 | 8.5 × 6 | 2917 | 3366 | 3927 | 30 |

## 4 Application of the model

The procedure for determining drill availability and utilization is outlined in Table 3 and based on the information of a typical operation as used in the present quarry. Equations (1) and (2) are used to obtain the availability and utilization percentages for the drills to be evaluated. Percentage of operation time in relation to total possible time is calculated by using equation (3).

**Table 3** *Determination of drill availability and utilization, according to operating time of the present quarry working one 8 hrs-shift/day.*

| Description | Hours | Days |
|---|---|---|
| Total calendar time | 2920 | 365 |
| Less holidays per year | 80 | 10 |
| Possible available time (PAT) | 2840 | 355 |
| Less outages | 440 | 55 |
| Available operating time (AOT) | 2400 | 300 |
| Less operational restrictions (OR) | 208 | 26 |
| Less drill moves and other major interruptions | 72 | 9 |
| Less personal time (PT): | | |
| Travel time | 48 | 6 |
| Lunch | 48 | 6 |
| Other | 24 | 3 |
| Less other non-drilling time (OND): | | |
| Lubrication and inspection | 80 | 10 |
| Short moves | 24 | 3 |
| Running repairs | 40 | 5 |
| Other | 40 | 5 |
| Net drilling time (NDT) | 1816 | 227 |

$$\%\text{Availability}=\frac{PAT-outages}{PAT}*100=\frac{AOT}{PAT}*100=\frac{355-55}{355}*100=84.5\%$$

$$\%\text{Utilization}=\frac{AOT-OR-LDM-PT-OND}{AOT}*100=\frac{300-26-9-15-23}{300}*100=75.5\%$$

%Operated of total time = Availability x Utilization = 84.5% x 75.5% = 64%

Then, operating hours per shift = $8\times\frac{227}{300}=6$ hrs / shift

In actual practice, the amount of time lost due to operational restrictions can vary from 5 to 40%of the available operating time. In this study, the amount of time lost is about 25%, the amount of time lost is between the lower and higher limit of the possible loss in time [3].

## 5 Results and discussion

### Calculation of minimum number of drills

For the initial evaluation, it is assumed that all of the three drills have the same availability and utilization values. All the drill productive capacities are given in (Table 2). The minimum number of drills needed in each drill class to maintain supply of broken material according to the annual production plan, can be calculated by using equation (5). The minimum number of drills needed is calculated as shown in Table 4.

From Table 4, it can be seen that using three drills one of each class leads to very high ratio of standby time. The ratio of standby time is 0.23, 0.57, and 0.69 for class1, class2, and class3 respectively. This is a great loss of operating time which if used can produce 5,702,859 tons annually i.e. greater than the current annual production by about 14%. Hence, the current practice distribution of drills is not suitable and needs improvement.

Now, there are six alternatives suggested, in addition to the case actually applied in the present practice of the quarry as follows:

If we only use one class of the drilling machines, we can calculate the minimum number of drills of this class required to fulfill the five million tons annual production. Hence, we have three alternatives: alternative 1, only using class1; alternative 2, only using class2; and alternative three, only using class3. Table 5 presents calculations of the minimum number of drills for each class that satisfies the annual production.

If we use two drill classes at a time to fulfill the annual production, we have three additional alternatives: alternative 4, using class1 and class2; alternative 5, using class1 and class3; alternative 6, using class2 and class3. Summary of calculations of these alternatives is presented in Table 6.

**Table 4** *Calculation of minimum number of drills according to the current practice in the quarry*

| Class of Drills | Bench | TPY, ton | MPH, ton/ hole | N | MDH, m/hole | $T_m$, m | DR, m/h | TDH, Hrs | MND | Lack(-) - stand by (+) |
|---|---|---|---|---|---|---|---|---|---|---|
| C-1 | Upper | 557040 | 1673 | 333 | 28 | 9324 | 20 | 466.2 | | |
| | Middle | 557040 | 1931 | 289 | 32 | 9248 | 20 | 462.4 | | |
| | Lower | 573920 | 2252 | 255 | 37 | 9435 | 20 | 471.8 | | |
| Total | | 1688000 | | 877 | | 28007 | | 1400.4 | 0.77 | +0.23 |
| C-2 | Upper | 273735 | 2674 | 103 | 27.5 | 2833 | 15 | 188.9 | | |
| | Middle | 273735 | 3086 | 89 | 31.5 | 2804 | 15 | 186.9 | | |
| | Lower | 282030 | 3600 | 79 | 36.5 | 2884 | 15 | 192.3 | | |
| Total | | 829500 | | 271 | | 8521 | | 568.1 | 0.31 | +0.69 |
| C-3 | Upper | 819225 | 2917 | 281 | 27.5 | 7728 | 30 | 257.6 | | |
| | Middle | 819225 | 3366 | 244 | 31.5 | 7686 | 30 | 256.2 | | |
| | Lower | 844050 | 3927 | 215 | 36.5 | 7848 | 30 | 261.6 | | |
| Total | | 2482500 | | 740 | | 23262 | | 775.4 | 0.43 | +0.57 |
| T.Grand | | 5000000 | | 1888 | | 59790 | | 2760 | | |

**Table 5** *Calculations of minimum number of drills for each class that satisfies the annual production of 5 million ton*

| Class of Drills | Bench | TPY, ton | MPH, ton/ hole | N | MDH, m/hole | $T_m$, m | DR, m/h | TDH, Hrs | MND | Lack (-) - stand by (+) |
|---|---|---|---|---|---|---|---|---|---|---|
| Alternative1 | | | | | | | | | | |
| C-1 | Upper | 1650000 | 1673 | 987 | 28 | 27636 | 20 | 1381.8 | | |
| | Middle | 1650000 | 1931 | 855 | 32 | 27360 | 20 | 1368 | | |
| | Lower | 1700000 | 2252 | 755 | 37 | 27935 | 20 | 1396.8 | | |
| Total | | 5000000 | | 2597 | | 82931 | | 4146.6 | 2.28 | -0.14/1 |
| Alternative2 | | | | | | | | | | |
| C-2 | Upper | 1650000 | 2674 | 617 | 27.5 | 16968 | 15 | 1131.2 | | |
| | Middle | 1650000 | 3086 | 535 | 31.5 | 16853 | 15 | 1123.5 | | |
| | Lower | 1700000 | 3600 | 473 | 36.5 | 17265 | 15 | 1151 | | |
| Total | | 5000000 | | 1625 | | 51086 | | 3405.7 | 1.88 | +0.06/1 |
| Alternative3 | | | | | | | | | | |
| C-3 | Upper | 1650000 | 2917 | 566 | 27.5 | 15565 | 30 | 518.8 | | |
| | Middle | 1650000 | 3366 | 491 | 31.5 | 15467 | 30 | 515.6 | | |
| | Lower | 1700000 | 3927 | 433 | 36.5 | 15805 | 30 | 526.8 | | |
| Total | | 5000000 | | 1490 | | 46837 | | 1561.2 | 0.86 | +0.14 |

From Table 5:

Alternative1 suggests using two drills of class1. However, there is a lack of operating time about 14% which can be accounted for by allowing overtime for operating the two drills (about one hour/shift). The quarry has only one drill of class1 at the present time, hence this alternative cannot be applied.

Alternative2 suggests using two drills of class2. There would be standby time about 6% which is an acceptable percentage. However, the alternative is not acceptable because at present the quarry has only one drill of class2.

Alternative3 is very attractive. It suggests one drill of class3 with 14% of standby time. However there is a great potential risk if sudden breakdown happens to the drill. This will lead to a complete stop not only for the quarry operations but also for the subsequent whole system of cement production. This would be much greater economic loss than the saving in the drilling process. Accordingly alternative3 is rejected.

**Table 6** *Minimum number of drills considering alternative 4, 5 and 6, using two drill classes at a time to produce the annual production*

| Class of Drills | Bench | TPY, ton | MPH, ton/ hole | N | MDH, m/hole | $T_m$, m | DR, m/h | TDh, Hrs | MND | Lack (-) - stand by (+) |
|---|---|---|---|---|---|---|---|---|---|---|
| Alternative 4: Using class 1 and class 2 drills | | | | | | | | | | |
| C-1 | Upper | 825000 | 1673 | 494 | 28 | 13832 | 20 | 691.6 | | |
| | Middle | 825000 | 1931 | 428 | 32 | 13696 | 20 | 684.8 | | |
| | Lower | 850000 | 2252 | 378 | 37 | 13986 | 20 | 699.3 | | |
| Total | | 2500000 | | 1300 | | 41514 | | 2075.7 | 1.14 | -0.14 |
| C-2 | Upper | 825000 | 2674 | 309 | 27.5 | 8498 | 15 | 566.5 | | |
| | Middle | 825000 | 3086 | 268 | 31.5 | 8442 | 15 | 562.8 | | |
| | Lower | 850000 | 3600 | 237 | 36.5 | 8651 | 15 | 576.7 | | |
| Total | | 2500000 | | 814 | | 25591 | | 1706 | 0.94 | +0.06 |
| T.Grand | | 5000000 | | 2114 | | 67105 | | 3804 | | |
| Alternative 5: Using class 1 and class 3 drills | | | | | | | | | | |
| Class of Drills | Bench | TPY, ton | MPH, ton/ hole | N | MDH, m/hole | $T_m$, m | DR, m/h | TDh, Hrs | MND | Lack (-) - stand by (+) |
| C-1 | Upper | 825000 | 1673 | 494 | 28 | 13832 | 20 | 691.6 | | |
| | Middle | 825000 | 1931 | 428 | 32 | 13696 | 20 | 684.8 | | |
| | Lower | 850000 | 2252 | 378 | 37 | 13986 | 20 | 699.3 | | |
| Total | | 2500000 | | 1300 | | 41514 | | 2075.7 | 1.14 | -0.14 |
| C-3 | Upper | 825000 | 2917 | 283 | 27.5 | 7783 | 30 | 259.4 | | |
| | Middle | 825000 | 3366 | 246 | 31.5 | 7749 | 30 | 258.3 | | |
| | Lower | 850000 | 3927 | 217 | 36.5 | 7921 | 30 | 264 | | |
| Total | | 2500000 | | 746 | | 23453 | | 781.7 | 0.43 | +0.57 |
| T.Grand | | 5000000 | | 2046 | | 64967 | | 2874 | | |
| Alternative 6: Using class 2 and class 3 drills | | | | | | | | | | |
| Class of Drills | Bench | TPY, ton | MPH, ton/ hole | N | MDH, m/hole | $T_m$, m | DR, m/h | TDh, Hrs | MND | Lack (-) - stand by (+) |
| C-2 | Upper | 825000 | 2674 | 309 | 27.5 | 8498 | 15 | 566.5 | | |
| | Middle | 825000 | 3086 | 268 | 31.5 | 8442 | 15 | 562.8 | | |
| | Lower | 850000 | 3600 | 237 | 36.5 | 8651 | 15 | 576.7 | | |
| Total | | 2500000 | | 814 | | 25591 | | 1706 | 0.94 | +0.06 |
| C-3 | Upper | 825000 | 2917 | 283 | 27.5 | 7783 | 30 | 259.4 | | |
| | Middle | 825000 | 3366 | 246 | 31.5 | 7749 | 30 | 258.3 | | |
| | Lower | 850000 | 3927 | 217 | 36.5 | 7921 | 30 | 264 | | |
| Total | | 2500000 | | 746 | | 23453 | | 781.7 | 0.43 | +0.57 |
| T.Grand | | 5000000 | | 1560 | | 49044 | | 2502 | | |

From Table 6:

Alternative4 using one drill of class1 and one drill from class2 is very acceptable. A very short operating overtime (about 4%) will be needed to fulfill the annual production plan. The two drills are available in the quarry. This alternative will lead to saving one drill of class3 for rent or sale which is a good economic gain.

Alternative 5 suggests that using one drill of class1 and one drill of class3. This will lead to saving one drill of class2 for rent or sale providing an economic gain for the quarry. This alternative will provide a plenty of standby time (about 44%) even after sharing drill1 to satisfy its production share. This can be a good choice if there is a plan to expand the cement production in the near future.

Alternative6 suggests using one drill of class2 and one drill of class3 and saving one drill of class1. It is acceptable because of the economic gain by saving that drill of class1. However there is a plenty of standby time (about 60%) which is not exploited now. It is available for future expansion.

From the above, alternative 4 (using one drill of class1 and one drill of class2) provides the best option for the distribution of the drilling machines. It provides a complete use of the two drills and save the most efficient drill of class3 for rent or sale. The subsequent options are alt.5 and alt.6 respectively.

**Table 7** *Summary*

<table>
<tr><th>Alternatives</th><th>Class of drills</th><th>MND</th><th>Overtime required(+)/Stand by, hr/year</th><th>Remarks</th></tr>
<tr><td>1</td><td>C-1</td><td>2</td><td>####</td><td rowspan="2">Inapplicable</td></tr>
<tr><td>2</td><td>C-2</td><td>2</td><td>####</td></tr>
<tr><td>3</td><td>C-3</td><td>1</td><td>254</td><td>Rejected for risk</td></tr>
<tr><td rowspan="2">4</td><td>C-1</td><td>1</td><td>+ 62</td><td rowspan="2">The production plan can be accomplished with 113hr overtime, with one drill spare for rent.</td></tr>
<tr><td>C-2</td><td>1</td><td>+ 51</td></tr>
<tr><td rowspan="2">5</td><td>C-1</td><td>1</td><td>0</td><td rowspan="2">The production plan will be accomplished. The C-3 drill will be stand by for nearly half of the net drilling time which consider waste of resources.</td></tr>
<tr><td>C-3</td><td>1</td><td>940</td></tr>
<tr><td rowspan="2">6</td><td>C-2</td><td>1</td><td>108</td><td rowspan="2">The production plan will be accomplished. The C-3 drill will be stand by for more than half of the net drilling time which consider waste of resources.</td></tr>
<tr><td>C-3</td><td>1</td><td>1035</td></tr>
</table>

## 6. Conclusions

This study has been carried out on the lime stone quarry of Assiut Cement Company (CEMEX). The quarry has three benches and three drilling machines. Namely one drill of Compare (class31), one drill of Atlas Copco Roc 606(class2), and one drill of Atlas Copco Roc 460 HF (class3). A simple mathematical model has been used to calculate the minimum number of drilling machines and their types for six alternatives in addition to the current practice. The following conclusions and recommendations have been drawn.

1- Using one drilling machine of class1 and one drilling machine of class2 (alternative4) is the best option for the quarry. That is because the two drills are operating almost all the time and the drill of class3 is saved for rent or sale.

2- Using one drill of class1 and one drill of class3 (alternative5) will save one drill of class2. Whereas using one drill of class2 and one drill of class3 (alternative6) will save one drill of class1. However, there will be a plenty of operating standby time, 44% and 60% respectively.

3- Using two drills of class1 (alternative1) and using two drill of class2 (alternative2) are technically acceptable. However, they are rejected because the quarry actually has only one drill of each type.

4- Using one drill of class3 (alternative3) is technically the best option. However, it is rejected because there is a great potential of risk if a sudden break down of the drill takes place. The whole operations in the quarry and the cement producing lines will stop causing great economic loss.

## 7. Recommendations

1- Alternatives one and two could be reconsidered if new drilling machines will be bought.

2- Alternatives five and six could be reconsidered if expansion of the production is expected.

3- The methodology used in this investigation can be adopted to calculate the minimum number and distribution of other pieces of equipment such as loading units and transportation trucks if their capacities and production targets are known.


## Aknowledegments

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# THE OPPORTUNITY OF THE USE OF ANCHORED SUPPORTS AT THE UNDERGROUND EXCAVATIONS

*Sorin RADU*, Valeriu PLEȘEA***

**Abstract:** *As an objective requirement of the a confinement of the application domain of the metallic support presently used for the execution of underground excavations, it results from the non-correlation of the symmetric construction of the support system having an irregular behaviour of the vector of maximum pressure, and on economic considerations, generated by increased costs of metal and production costs of laminated profiles, by the application of the suppliers of additional enhancement treatments, it becomes more and more necessary to introduce and generalise anchored supports for the consolidation of rocks. The paper analyses the possibility to generalise anchored supports for the execution of underground excavations, as well as the particularities of this kind of support system and its interaction with the surrounding rock, the purpose being to fundament the choice of this rock consolidation procedure.*

**Key words:** *underground excavation, traction tension, expansion bond, cement mortar, rock massif, support interaction*

## 1. Introduction

Analysed considering the reliability and safety criteria, the theoretical and experimental researches carried out during the past years prove that for difficult placement conditions of the underground excavations, the metallic supports are irregularly kept under stress with increased intensities. Therefore, they lose their bearing capacity after a relatively short period of time after having been installed, due to the local tensions forming in certain spots on the outline of the work, tensions which exceed the limited flow frequency of execution steels. It is therefore considered that the deformation of the support is generated not only by the burden of the rock massif, but especially by the discordance of the construction of the support system used to the direction of which the pressure vector has a maximum value. It is also known that the classical usual metallic support, with an elastic-sliding operation, considering its construction may carry only vertical loads generated by the vertical pressure of the ceiling, for limited convergences of the rock of 150-200 mm. In the case in which the lateral pressure manifests itself with intensity, and the ratio $P_x/P_y > 1$, the metallic props give in, due to their bearing capacity and their limited reaction to the load of the rock massif, with all the unfavourable consequences regarding the optimum operation of the support. Consequently, considering the present economic crisis, during which the iron and steel plants are reducing their activity of producing support laminated profiles, for the insurance of the stability of underground excavations, internationally they are oriented to introducing and generalising anchored supports for the consolidation of rocks, aspect which is imposed more and more profound in the practice of underground constructions in our country as well.

## 2. Appreciations regarding the effect of stress on the rocks surrounding the underground excavations

As the result of the theoretical and experimental researches on the geomechanical processes show it the most intense rock movements around the underground excavations are produced after the excavation activity and the advance of the face. The rocks with increased mechanical resistances may ensure a new balance, based on their self-bearing capacity, while the rocks with medium and reduced resistances, such as sediments, undergo an active movement process which is finalised through the deformation of the entire profile of the excavation (Figure 1) consequently this kind of rock needs to be supported.

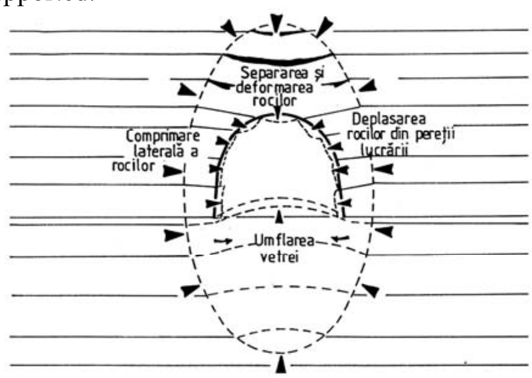


**Figure 1** *The effect of stress on the rock surrounding the underground excavations*

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The classical usual support, such as the metallic sliding one, with a symmetric construction and installation of the composing elements, is designed to bear the maximum loads which come down preponderantly from the ceiling of the excavation and, consequently, by asymmetric distribution of the state of tension on the outline of the work, the present system no longer corresponds to the given requirements [1],[2],[3],[4].

Therefore, the conditions of non-uniform manifestation of the underground pressure, when the lateral stress ($Pp_1$ , $Pp_2$) of the excavation records values which exceed the vertical ones ($P_t$), results in accentuated deformations of the pillars with the loss of balance of the "rock-support" system and the decrease of the stability of the work (Figure 2).

The same non-uniform deformation aspect is recorded in the case of rigid arch bricks masonry supports as well, respectively monolith concrete or pre-fabricates, which, although have superior bearing capacities, following the asymmetric stress manifested on the outline of the excavations and the non-corresponding filling degree of the voids in the back, both in the walls and in the ceiling (Figure 3).

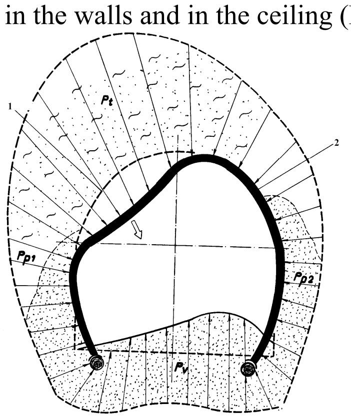


**Figure 2** *The irregular deformation of the mine work supported with metallic reinforcements, considering the taking over of maximum stress from the wall next to the layer: **1**-initial profile; **2**-profile resulted after the stabilisation of the stress*

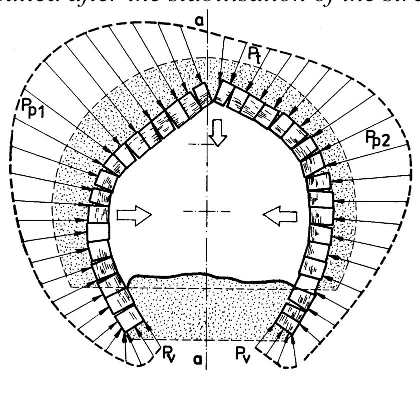


**Figure 3** *Asymmetric stress and deformation mode of the mine opening works supported by masonry with arch bricks (case $P_t < Pp_1, Pp_2$)*

As a consequence of the incorrelation between the construction of the present classical supports and the direction in which the maximum pressure vector manifest itself, the theoretical and experimental researches of the geomechanical processes have highlighted the need to orient the actions to benefit to a maximum of the bearing surface of rocks in taking over the stress, by consolidating them with the help of anchored supports. Consequently, the increase of the resistance of rocks to the shearing and traction stress becomes possible, with the formation and the implication of the consolidated area, at a very short period of time in taking over the pressures. this type of mechanism of the interaction in the "anchor-rock" system is accepted by most of the specialists in the field, constituting the main argument regarding the anchored support for various kinds of rocks.

Considering this context, most of the countries with experience in this field observe the diversification and extension of system of anchoring the rocks, phenomenon appreciated as one of the most important realisations in the technique and technology of mine works supports during the post-war period. The increasing interest manifested by the mining industry of the Western Europe, USA, Australia, South Africa, and others, towards the anchored supports, results mainly from the economic situations of mines and the efforts which are undertaken to make the extraction activities of useful mineral substances more efficient.

In order to give examples concerning the interest manifested in the field of anchored supports it is highlighted that in England, the year 2000 meant coming back to its senses, namely using this kind of support for the execution of over 238 km of underground mines, representing 78% of the total network system of opening and preparing, made for the coal mining industry, effectively contributing to reducing the exploitation expenses by 2.5 from the initial situation. Also, during the past 10 years, the coal sector of the USA used 100 million differently constructed anchors per year, while in South Africa over 65% of underground mines are supported by anchors based on the principle of friction with the surrounding rock.

**3. Particularities of anchored supports and their interaction with the surrounding rock**

The option of using the anchored support is practically exemplified by its following extremely important particularities, namely [3], [4]:

**a** – consolidates the rock, forming an increased bearing pillar which is involved in staking over the pressure (Figure 4)

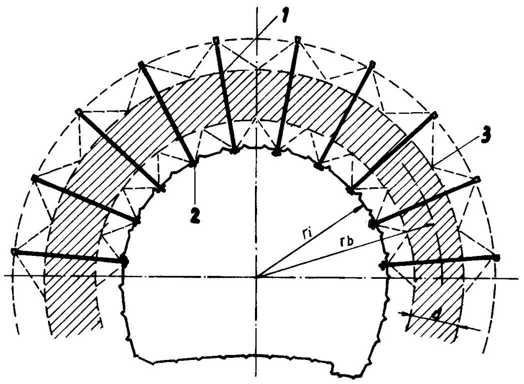


**Figure 4** *The creation of the consolidated rock pillar: **1**-metalli anchor (anchoring rod); **2**-pretensioning plate; **3**-consolidated rock pillar; **d**-the thickness of the consolidated rock pillar*

**b –** develops forces in different spots on the outline of the excavations, which through their action generate the redistribution of tensions and limit the rock deformation process (Figure 5).

**c –** creates a consolidated arch which amplifies the resistance of rocks and ensures their stability in the conditions of increased traction tensions which are developed within the walls of the excavation (Figure 6).

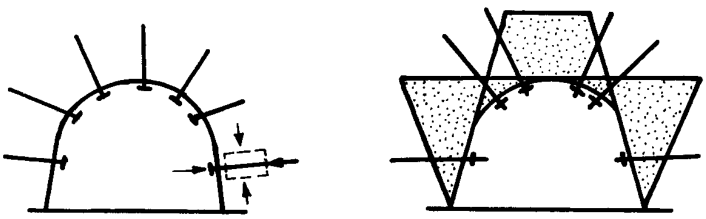

**Figure 5** *The influence of the anchored support when developing the additional consolidation forces on the outline of the excavation*

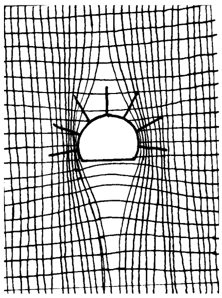

**Figure 6** *Redistribution and concentration of traction tensions in the walls of the mine work*

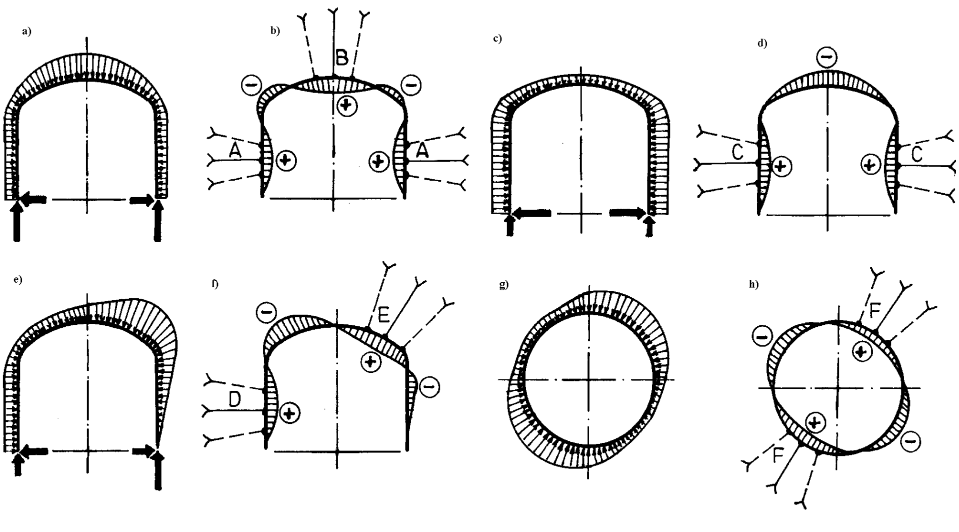


**Figure 7** *The formation areas of the maximum arching momentums around the underground excavations for different types of profiles and their loads: **a, b**–the case of the arched profile and the maximum load of the ceiling; **c, d**–the case of the arched profile and the maximum load of the walls; **e, f**–the case of the arched profile and the asymmetric load of the from the ceiling; **g, h**–the case of the circular profile and the maximum vertical load (ceiling and floor)*

The basic principle in placing and using the anchors needs to be represented by the knowledge of the place where the maximum arching momentums are formed, the proportion of which is recorded to be 90% of the total of the main tensions. Consequently, if the actual metallic supports take over most of the loads only from the ceiling of the mine work (Figure 7a), then the maximum arching momentums will be created corresponding to graphical representation in Figure 7b

Therefore, in the action area of the maximum momentums A and B, the most intense rock movements and deformations are observed, scenario in which the placement of the anchors in these areas becomes rational, namely in the ceiling and the wall of the work. The installation of anchors in the walls ensure not only the decrease of the deformations but also decrease of the maximum arching momentums which manifest within the support pillars, favouring the formation and development of additional internal forces which improve the interaction of the "rock-support" system, taking part in the increase and insurance of the stability of the mine work.

On the other hand, during the experimental researches carried out on site confirmed that the installation of the anchors only in the lateral walls of the mine work, where the largest movements and deformations of rocks take place, it is observed a more uniform distribution of tensions on the entire outline and also a decrease of the effort of traction. Therefore, when the rock movement in the wall is predominant, with the development of lateral pressures (Figure 7c), the anchors need to be installed in spot C (Figure 7d). In the case of galleries found under the influence of the coal face, exposed asymmetrically to the stress of the vector of maximum pressure (Figure 7e), the representation of the maximum arching momentums indicates correctly the spot to install the anchors, respectively spots D and E (Figure 7f), namely the efficiency of the obtained stabilisation to be maximum, with a reduced consumption of materials.

For the circular profiles of the mine work, which also bear the asymmetric stress of the pressure (Figure 7g) is rational to place the anchors in spots F (Figure 7h), namely on those sectors on the outline of the work where the deformation tendency of the rocks is going to reach the maximum value.

**d** – influences the stress regime which is formed during the rock excavation process, which is favourably modified. Therefore, the laboratory experimental researches, carried out through photoelastic methods, have shown a comparative situation between an unsupported ceiling excavation and the models supported by anchors (Figure 8a, b and c).

In the first case (Figure 8a) the stress is focused in the middle of the opening, making the ceiling to expand and to deform. In the model presented in Figure 8b there is a contact-point anchor fixed and therefore the stress is spread around the rod which compresses the area between the two fixing points keeping the ceiling horizontally during the exploitation period of the work. For the model presented in Figure 8c, the anchor fixed integrally redistributes the compression stress around the rod, with favourable stabilisation influences of the rocks of the ceiling. Based on the obtained results it is emphasised that in the case of anchored sections the surface of the rocks is being supported with a normal effort the maximum value of which is found in front of the rods and the support plates, while the minimum value situated in the middle of the interval between two anchors. Around the anchoring rods, the form of the force lines is presented in Figure 9.

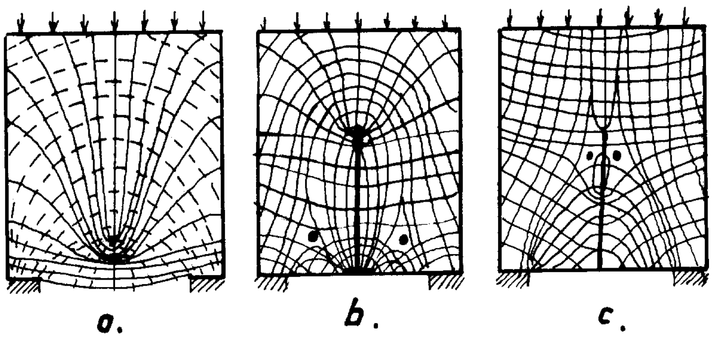


**Figure 8** *The representation of efforts considering the ceiling of an excavation: **a**–the case of an unsupported ceiling; **b**–supported ceiling by contact-point anchors; **c**–ceiling supported by plug anchors on its entire length*

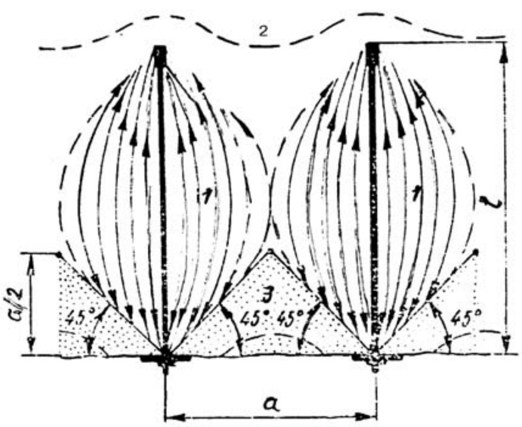


**Figure 9** *The repartition of force lines in the package of anchored rocks;* ***1****–the force lines area;* ***2****–the balanced rocks area;* ***3****–uncompressed area*

The rock package is compressed with the value of the force of closure of the rod of its axle which decreases gradually, as it goes further away from the place where the anchor was installed. Therefore, between the rods of the anchors there is an uncompressed area, extending towards the surface of the work, representing a form of cones with the generator tilted at 45° and the height a/2.

The influence of the anchors on the tensions of the massif is conditioned by time as well. As the period of time spend between the excavation to the anchorage is smaller, the rocks expand less and the effect of the anchors is better. When the rocks already record an expansion process, the anchors present only a limited action, increasing their resistance to shearing.

**e** – compared to the traditional supports, the consolidation of the rock using anchors ensures a more efficient balance in the "rock-support" system, explained by the formation of the consolidate pillar and involving the anchored support in taking over the pressure in a short period of time from its installation (Figure 10).

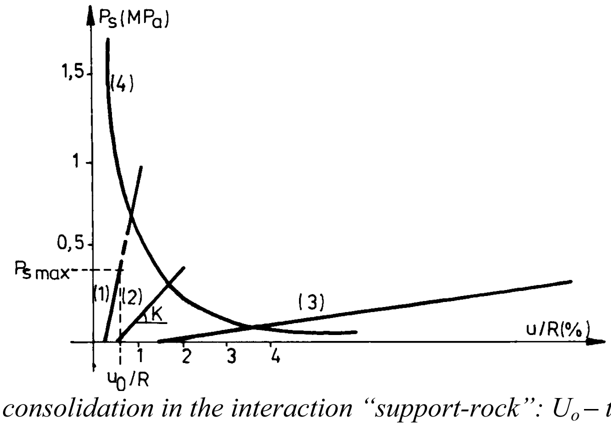


**Figure 10** *The role of the consolidation in the interaction "support-rock": $U_o$ – the deformation of the rock around the underground excavation; R-the radius of the minework*

Based on the presented graph, it is observed that for average values of stress, i.e. 0.4 MPa (40 t/m$^2$), the working characteristic of the concrete or arch brick rigid support (Curve 1) indicates that this type of support suffers total deformation, their characteristic curve being unable to intersect the deformation curve of the rocks around the underground excavation (Curve 4).

In the case of anchored supports (Curve 2), the consolidated rock pillar is formed with an increased bearing capacity and a characteristic almost identical (Curve 1), but, which, due to a higher value of the developed bearing capacity, ensures the corresponding balance with the rock massif.

In the case of elastic supports comprised by the sliding reinforcements, the characteristic working curve (Curve 3) intersects the relaxation curve of the rock after a relatively large interval of time, but for larger deformations of the rock, respectively repeated sliding of elements and reduced values of stress, case in which the balance of the support with the rock and the stability of the mine work are produced for large reductions of the profile of the excavation, the conditions of efficiency not being fully meet.

## 4. Conclusions

The options for choosing a support adequate to the conditions with a diversified manifestation of the loads around the underground excavation, along with the need to allocate reduced material and labour expenses, they shall also consider the value of the bearing surface of the surrounding rock, possible to realise by the reinforcing them with the help anchored supports.

Corresponding to the constructive and operational characteristics of this type of support, as application steps underground it is imposed to perform the wholes, radial on the outline of the excavation, followed by the implant of the anchoring rods and the installation of the pre-tensioning plates, immediately after unveiling the rocks, disposing them in distant layers on the direction of the work depending on the type of rock met and the type of anchoring system used.

Considering the difficult geomechanical conditions, the anchored support may be associated with concrete supports, having as purpose the protection of the rocks against alteration, which are designed in 2-3 succesive layers with a 2-5cm thickness, above the metallic net foreseen for bandaging.

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# THE SITUATION OF DUMPS AND DEGRADED LANDS OF BERBEŞTI MINING BASIN

*Nicolae DICAN**

***Abstract:*** *In the Berbeşti mining basin was a continues concern of land reclamation into the economic circuit which were used as sterile dumps. The reclamation works executed, generally have achieved the goal for which they were executed, except for some small areas. The land which inclination prevents the achievement of mechanized agriculture and which are prone to erosion or landslides have been reused, so far being successfully planted Acacia species only.*
***Key words:*** *dumping, stability, accumulated material, cohesion, passive resistance.*

## 1. Introduction

Berbeşti mining basin is located in the structural unit of Dacian Depression, in the extreme East of the Oltenia lignite deposit. Of the four coal pits with technology of excavation, transport and dumping in continuous flow, Ruget, Oltet, Berbeşti and Panga in over 30 years of activity were exploited 85 million tonnes of coal that have necessitated a stripping of 513 million $m^3$ sterile [7].

## 2. Dumping of sterile rocks

Sterile rocks belonging to the Pliocenului and Quaternary, heterogeneous and with wide particle size, resulting from ripping of useful minerals, are transported and stored in dumps. At all the coal pits in the basin, in the opening phase but also in the exploitation phase of ore at great depths, were used the outer dumps which were located in lowland areas or on slopes with gradients of less than 15 °, aiming to use degraded lands or less productive [5].

Only when the situation required it, were affected lands with a high agricultural potential, such as those located in Olteț meadow (Jigăi dump), or partly in the Tărâiei meadow (for outer dumps Panga North and Panga South) which are located on very little horizontal land or inclined. The transport distance of rocks over 7 km away, as is the case of outer dump of Oltet coal pit, where for excavation, transport and dumping of sterile is used a specific electricity consumption of more than 4,5 kW/m³, and the need for additional occupation of large areas of land, leading to the conclusion that these are uneconomic, but absolutely necessary to ensure the transition to the dumping conditions within this coal pit.

The inner dumps are most economical, due to short circuits transport of sterile and the use of land already affected by the extraction of coal pit activities. It is ensured the maximum productivity and is the simplest of the organizational point of view. The dumping should provide a number of requirements needed for the effective activities namely: storage of the entire volume of sterile excavated, complete security of work for staff, equipment and surrounding attractions, low costs, high productivity, etc. [6]

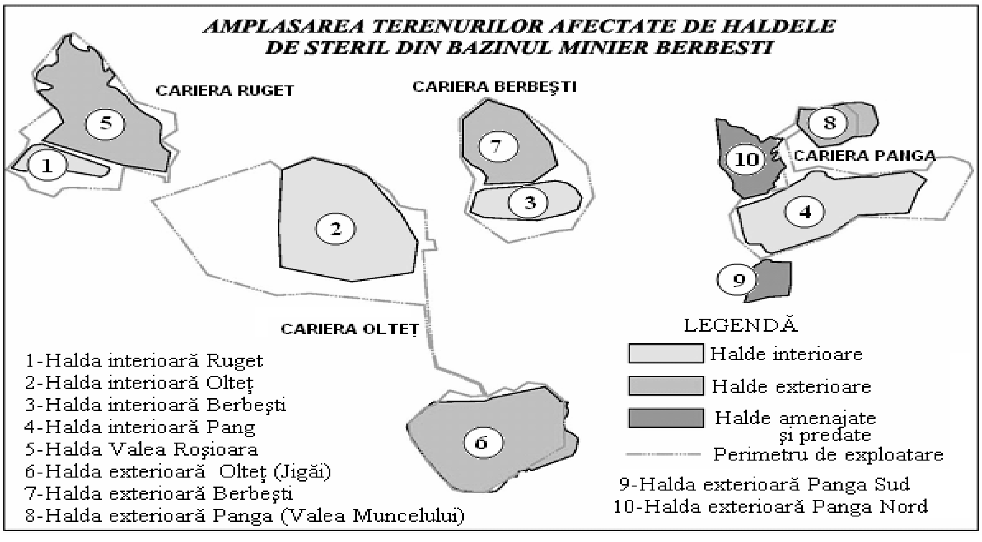


**Figure 1** *Location of sterile dumps from Berbeşti mining basin*

* *Eng. Ph.D UMC Berbeşti*

### 3. Location of sterile dumps

Every coal pit from the mining basin have used one or more external dumps and open up to the possibilities of creating the dumping possibilities within them. Location of the dumps from the Berbeşti mining basin is shown in Fig. 1.

It will be made a brief presentation of dumps, from West to East, highlighting in particular the areas showing the importance of reclamation in the economic circuit, volume and geometric elements.

*Valea Roşioara outer dump* is located in the vicinity of Ruget coal pit, in the northern part of it in a valley area, which name it bears. External dump is built in five benches, with maximum heights of 15 m, jointed with the natural terrain on the right slope of the valley, to the final rate of + 470m, the height of it dropping to 40 m in the final results of the eastern area located in the middle of the valley. In general, the designed geometry have been breached, the slopes (1:2), equivalent to a slope angle $\alpha = 26$ °, and a width of at least 90 m, and the front berm also has a height of 15 m. The amount deposited in the landfill is 82 million m3 and the area of occupied land and not given to the economic circuit is 217 ha.

*Ruget inner dump* is located in the vicinity of the coal pit, being at present actually assisted in the final slope of the coal pit, as can be seen in Figure 2. Due to the height of the front bench of the dump of over 35 m , the rock flow software [2]. The storage ability of the Ruget inner dump is over 30 million m³ and the further inside dumping works is possible either by exploiting the Bustuchin perimeter either in the North Oltet expanded perimeter. The surface of the land reclamation of Ruget inner dump is 56 hectares.


**Figure 2.** *Sterile dumping in the inner dump of Ruget coal pit assisted in the final slope*

Exterior dump of Olteț coal pit (Jigăi dump) is located at a distance of 7 km South-East from the coal pit, in the area of confluence of the Oltet river and Tărâia. Choice of location from a so far distance, was conditioned by the fact that there is a high density construction near the coal pit and expropriation thereof would have produced an impact too heavily on the local community. Being the greatest distance transportation from the Mining Division Branch, plus the great share of sterile posted in the exterior dump vs. inner dump, the specific electricity consumption exceeds 4,5 kW/m³ excavation sterile, loaded, transported and stored in the exterior dump, representing the great disadvantage with direct implications for the price of production.

Morphological point of view, the dump is located on a relatively level plateau (lowland area), made up of the alluvial deposits that are part of the lower terrace of the Oltet river.

The volume of sterile dumped into the outer dump is 101 million m³ in an area of 250 hectares, and to the finish of dumping the total volume put away will be 140 million m³, occupying a total area of 303,5 hectares.

Olteț inner dump is located in the northern part of Oltet coal pit, with the shape and dimensions of the coal pit's floor. From tectonic point of view takes the form of a system of steps with the E-W direction and tilt N-S between 4°-5°, although it is made up of one, maximum two, Figure 3.

The volume of sterile deposited in inner dump on 01 January 2012 is 47.5 million $m^3$ on an area of 200 hectares, and the exploitation of coal in the total volume of stored will be 210 million m³, occupying a total area of 490,5 ha.

Berbeşti outer dump is located on the West, Northwest side of the Berbeşti coal pit and is geographically located within the Berbeşti and Alunu localities. Its foundation is made up of weak cohesive rocks arranged in a slope that tilts East-West with 6-8 °, over which the waste rock has been made from E-2 and E-01 excavators located on the 1st and 2nd bench of the coal pit. The surface of the outer dump is 127 hectares and contains a volume of 64.3 million m³, and upon completion of filing, its surface will be 139 hectares, with a total volume of sterile storage of 68,5 million m³.

Berbeşti inner dump is located in the northern part of the coal pit, having a horizontal footprint identical to the excavations carried out by the excavator situated on the floor of the coal pit. Has a stable foundation of grey clays with coal inclusions, with a slope of 4-5 ° from North to South. The occupied area is 129 ha and the amount deposited is 46 million m³. The depletion of the reserve, the total area of inner dump will be 188 hectares with a total volume of deposited sterile of 76.8 million m³.

Outer dumps from Panga coal pit, in number three, have been used successively in order of presentation, from opening until the achievement of a sufficient space in inner dump, for putting away all of the remaining excavated volume from the coal reserve. These conditions were created in 2009, a year in which it was brought into the inner the A-01 dumper and its associated transport circuit. Panga South outer dump, with an area of 46 hectares and a volume of sterile of 18 million m³ and the North outer dump, with an area of 92 hectares and a volume of 52 million m³, have a common feature in terms of the nature of the land, being located on hillsides located on the left side of the Tărâiadownstream, upstream, respectively. Both dumps have been reclaimed in the economic circuit and given to the local communities, Figure 4.

**Figure 3.** *Olteț inner dump*


**Figure 4.***Wheat crop from the reclaimed land of Panga South dump*


**Figure 5.** *Crops of wheat, maze and alfalfa on the West slope of Panga inner dump*

Muncelului Valley outer dump is located in the north-eastern part of Panga coal pit within Mateeşti and Berbeşti localities. The occupied land is 95 ha and the sterile volume of 44.2 million m³ is dumped in a relatively flat valley lying between two hillsides that is coming to the South, has facilitated the construction of a stable deposit. With the completion of sterile dumping, in 2009, have begun the forestry works for re-use.

Panga inner dump is located to the left of Tărâia brook and is bordered in the South by the end slope of the coal pit and to the North of the outcrop of coal layer I. At the present time, the 53,5 million m³of sterile occupies an area of 99.5 hectares, at the completion of the work, the area of occupied land will reach 171 hectares and the volume of sterile deposited at 95.5 million m³. Still in construction phase of dump, after the release of technological tasks of land located on the West end, the slope was re-shaped and arranged for reclamation in the agricultural circuit, being handed over to the local community, figure 5.

In table 1 is given the situation in the occupied areas and the volumes of stored sterile in every dump in the present day and at the completion of dumping works.

**Table 1.** *Current situation of sterile dumps from the basin and from the copletion of works*

| Coal pit | Dump | Ocuppied area [ha ] | | Volume of dump [mil.m³] | | Current situation |
|---|---|---|---|---|---|---|
| | | Current | Final | Current | final | |
| Olteț | Inner dump | 198 | 490,5 | 47.5 | 210 | Under construction |
| | Outer dump | 248 | 303,5 | 101 | 140 | Under construction |
| Berbeşti | Inner dump | 129 | 188 | 21.1 | 76,8 | Under construction |
| | Outer dump | 127 | 139 | 64,3 | 68,5 | Under construction |
| Panga | Inner dump | 99,5 | 171 | 53.5 | 95,5 | Under construction |
| | Panga North dump | 92 | 92 | 52.0 | 52.0 | Restore |
| | Panga South dump | 46 | 46 | 18.0 | 18.0 | Restore |
| | Valea Muncel dump | 95 | 95 | 44.3 | 44.3 | Partial restore |
| Ruget | Inner dump | 15 | 56 | 29,9 | 50 | Under construction |
| | Outer dump | 240,5 | 240,5 | 82 | 82 | Partial restore |
| **Total Berbeşti basin** | | **1290** | **1821.5** | **513.6** | **837.1** | |

The stored volumes in dumps of the tens of millions of cubic meters, or more than 100 million m³ in case of Oltet outer dump, have imposed the construction of sterile deposits with great height between 30 and 75 m, with classification index (2.2.2.), consisting of two or more benches with a height between 10-15 m each, corresponding to an index (2.1.2.) for the classification of dumps [8], as shown in table 2. In terms of surface topography on which were built the outer dumps, they were located on slopes with gradients of less than 20° (2.3.2), except Jigăi dump which is located on the flat ground or slightly inclined (2.3.1.).

**Table 2** *Dumps classification after their geometry*

| Classification criteria | | Index | Classification groups |
|---|---|---|---|
| | | | **Characterization** |
| 1 | 2 | 3 | 4 |
| **After dump's geometry** | 2.1.Number of benches | 2.1.1. | One bench dumps |
| | | 2.1.2. | Dumps withe more benches |
| | 2.2. Height of dump | 2.2.1. | Dumps with small height (<30m) |
| | | 2.2.2. | Dumps with big height (30÷100m) |
| | | 2.2.3. | Dumps with very big height (>100m) |
| | 2.3.Land topography | 2.3.1. | Dumps on flat land |
| | | 2.3.2. | Dumps on incline land ($<20^0$) |
| | | 2.3.3. | Dumps on incline land ($>20^0$) |

## 4.Technical condition of dumps

The field observations and analyses conducted at each dump in part, highlighted a few general conclusions and others with particular character.

In Berbeşti mining basin was a continues concern for reclamation of land awarded by technological tasks which were used for dumping. At the present time, the outer dumps Panga South, Panga North, Muncelului Valley were rendered entirely in forestry or agricultural circuit and given to the local communities or Romsilva.

Also, to the dumps: inner Panga, outer Berbeşti, outer Olteț and outer Ruget, on small surfaces, work was done to the economic circuit, others being a work in progress.

The reclamation works executed on these land, generally have achieved the goal for which they were executed, except for some small areas, where subsequently occurred landslides which have affected the access road from the South Panga, or the houses at the bottom of the Panga North dump, figure 6.

**Figure 6** *Landslide near Panga North dump*

The dumps with technological tasks have geometry contained in the admitted limits to the design of the building, there is a tendency to maintain the slope angle of the individual steps at the maximum permissible limit, and on the small stage lengths there is definitive undersized berms. In their great majority, the upper rungs have a poor or non-existent, leveling that allows rainwater infiltration into their body. All the dumps there are manifestations of mining negative phenomena, such as: abnormal bulging local subsidence, pressing of slope or terrain, superficial landslides and erosions or clough. The presence of accumulations of water dumps is found on surfaces inside the Panga inner dump and Helen outer dump. The small size of the zones in which manifests itself occasionally these phenomena, they are not affecting the stability of the overall system of the dumps.

A large landslide took place in the inner dump of Oltet coal pit in the year 2005 and has affected the bench system on the base [10], which had spread over an area of 60 hectares, and the material has been leaning on the excavation bench slope of the coal pit floor.

After lansliding and restoring balance in the new conditions, with the excavation of the block on the coal pit's floor along with a part of waste rock that is supported by it, the first bench was reconstructed in terms of stability, thus forcing the greater caution, but the and the adoption of additional measures to increase the reserve stability.

Analyzing the causes that have led to the loss of system stability, we determined the stability factor in the hypothesis of slipping on a polygonal surface, based on the theory of active resistance and knockback liabilities [4,8].

Fs1 stability factor was calculated under the conditions existing at the time of the landslide, in which the rocks were deposited directly into a floor constructed by excavating with the inverse buckets, with an angle β = 4 ° towards the outside of the dump, using the formula:

$$F_{s1} = \frac{\gamma_a \cdot h_1^2 \cdot \lambda_p \cdot \cos\beta + 4 \cdot c \cdot h_1 \cdot \sqrt{\lambda_p} \cdot \cos\beta + \gamma_a \cdot (h_2^2 - h_1^2) ctg\alpha \cdot \cos\beta \cdot tg\varphi_c}{\gamma_a \cdot h_2^2 \cdot \lambda_a \cdot \cos\beta - 4 \cdot c \cdot h_2 \cdot \sqrt{\lambda_a} \cdot \cos\beta + \gamma_a \cdot (h_2^2 - h_1^2) ctg\alpha \cdot \sin\beta} + \frac{2 \cdot c_c \cdot \dfrac{(h_2 - h_1) ctg\alpha}{\cos\beta}}{\gamma_a \cdot h_2^2 \cdot \lambda_a \cdot \cos\beta - 4 \cdot c \cdot h_2 \cdot \sqrt{\lambda_a} \cdot \cos\beta + \gamma_a \cdot (h_2^2 - h_1^2) ctg\alpha \cdot \sin\beta} \quad (1)$$

where:

α – slope angle unghiul of the individual dump (α=25 °);
$h_1$- height of the forward bench of dump ($h_1$=11 m);
$h_2$- total height of the system of two dump benches ($h_2$=26 m) ;
$\gamma_a$- volumetric weight of the dumped rocks ( $\gamma_a$ =19,2 kN/m$^3$ );
φ- the angle of internal friction of the dumped rocks ( φ=18 ° );
$\varphi_c$- the angle of internal friction on the contact surface ($\varphi_c$=23 °);
c- cohesion of dumped rocks (c=24 kN/ m$^2$);
$c_c$- cohesion of rocks on the contact surface with the floor ($c_c$=33 kN/m$^2$);
$\lambda a=tg^2(45-\varphi/2)$ coefficient of the active pushing ( λa=0,52 );
$\lambda p = tg^2(45+\varphi/2)$ passive resistance coefficient ( λp =1,89);

$$Fs_1 = \frac{19{,}2\cdot121\cdot1{,}89\cos4^\circ+4\cdot24\cdot11\cdot\sqrt{1{,}89}\cdot\cos4^\circ+19{,}2\cdot(676-121)ctg25^\circ\cdot\cos4^\circ\cdot tg23^\circ+2\cdot33\frac{(26-11)\cdot ctg25^\circ}{\cos4^\circ}}{19{,}2\cdot676\cdot0{,}52\cdot\cos4^\circ-4\cdot24\cdot26\sqrt{0.52}\cdot\cos4^\circ+19{,}2\cdot(676-121)ctg25^\circ\cdot\sin4^\circ} \quad (2)$$

**$Fs_1$= 2.60**, the stability factor is below the minimum limit specified by this theory.

It is mentioned that the stability factor Fs > 3, is a necessary condition to ensure the stability of the bench system when the stability analyses are carried out on the basis of this theory.

The made excavation on the coal pit's floor with a slight inclination towards the dump (ß1 = 0.5 °), as can be seen from Figure 7, perfectly possible with the right cups due to the lack of an important aquifer in the coal layer is bunk what would generate the risk of fragmentation of the screen protector and flooding her career, contribute to increasing stability factor of the speed of the dump. For the same humidity conditions, and friction angle, cohesion of dumped rocks, the same geometry of the treads of the landfill, an increase in stability factor of at $Fs_1$=2,60, to **$Fs_2$=3,18** [2].

The $Fs_2$ stability factor was calculated for those conditions, using the formula:

$$F_{s2} = \frac{\gamma_a\cdot h_1^2\cdot\lambda_p\cdot\cos\beta_1 + 4\cdot c\cdot h_1\cdot\sqrt{\lambda_p}\cos\beta_1 + \gamma_a\cdot(h_2^2-h_1^2)(\cdot\cos\beta_1\cdot tg\varphi_c+\sin\beta_1)ctg\alpha}{\gamma_a\cdot h_2^2\cdot\lambda_a\cdot\cos\beta - 4\cdot c\cdot h_2\cdot\sqrt{\lambda_a}\cos\beta} +$$

$$+\frac{2\cdot c_c\cdot\frac{(h_2-h_1)ctg\alpha}{\cos\beta_1}}{\gamma_a\cdot h_2^2\cdot\lambda_a\cdot\cos\beta_1 - 4\cdot c\cdot h_2\cdot\sqrt{\lambda_a}\cdot\cos\beta_1} \quad (3)$$

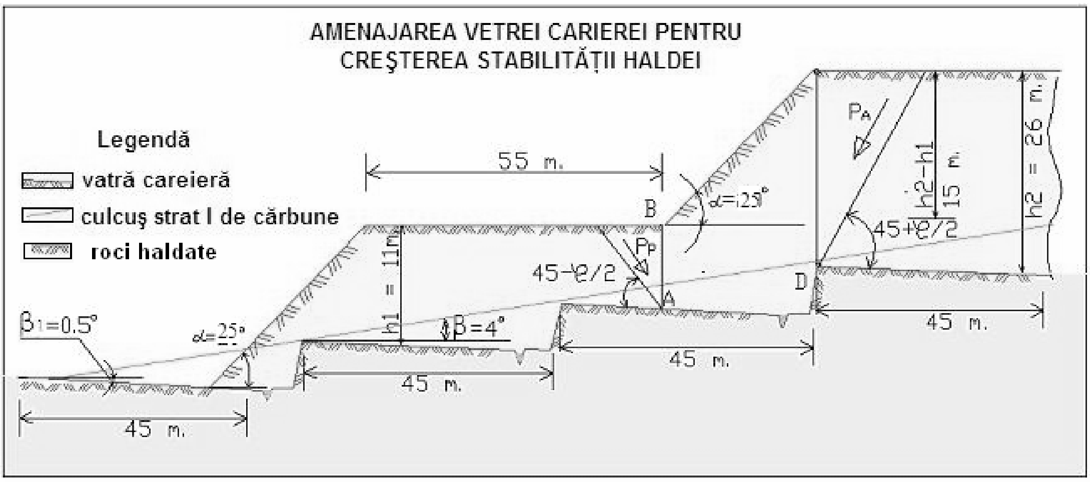


**Figure 7.** *Arranging the coal pit's floor from the excavation phase to ensure the stability of dump reserve*

**5. Degraded land for different purposes and their characterization.**

The exploitation of stratiforme lignite deposits, particularly in mining, involves the employment of large surfaces, but also qualitatively and quantitatively, of land which originally met the interests of local communities. At the end of 1989, when was made the maximum output of over 6 million tonnes, the organisation that encompassed all Horezu Mining mines and coal pits of the Gilort and up to the Bistrita had a total surface occupied by 1814 hectares of land. Further exploitation of coal in the basin and commissioning of new production capacity, it required an additional purchase of other land areas, so that in 1997 it was recorded maximum surface occupied by 2209 hectares, of which 1695 hectares farmland respectively 77% and 514 hectares of land, with a share of 23%, as shown in Figure 8.

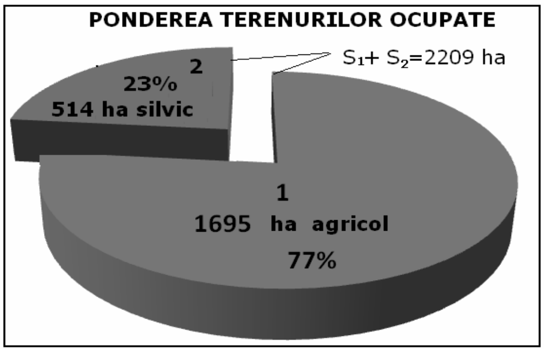


**Figure 8.** *Percentage of occupied land in 1997*

After 1997, went into preservation and then were closed one by one the mines from Cuceşti, Armăşeşti, Cerna, Berbeşti, Alunu, Copaceni and Albeni, small coal pits from Oteșani, Cernişoara, Cerna, Valea Mare, Panga and Seciuri. These closed mines and small coal pits in the Berbeşti mining basin have completed a list of 52 goals of national society of mining of lignite Oltenia, 550 respectively, at national level, which have been shut down in 1998-2008 through 10 successive decisions by the Government [1].

The influence of the subsidence to surface and underground works, preparation and service was trivial thanks to the great depth at which the operation took place and small thickness of overlying strata of coal and rock from the roof direct aeration during was tapped to fill the space. Thus, it was not necessary to use tools for leveling, it could have a more destructive effect as a result of the degradation of soil, crop or vegetable plantations or forests. The occupied land by the small coal pits basin were subjected to substandard facilities and after seeding with meadow and inorganic fertilization and, were handed over to local communities. On the final slope did not intervene, so that the more than 15 years since the execution of the work, these areas have partially seeded, and currently there is a natural slope as a result of environmental factors.

A special situation is the occupied lands from Bustuchin coal pit of 48 ha. In full development in 1997, with an EsRc-1400 excavator and a dumping installation into service, and other excavators of the same range almost completed in mounting platform, due to the impossibility of acquisition of land required for the continuation of the activity of extraction at the time, went into preservation.

On these lands has not stepped in to reclamation,Bustuchin coal pit being an alternative continuity of work in the western part of the basin, given the imminent closure of Ruget coal pit.

As it has been awarded by technological tasks, the outer dumps and a small portion of the inner dumps were also arranged and given in the economic circuit.

Beside the occupied lands for the dumping process, a significant area of land remains occupied by the excavation tehnlogical lines and transport.

Intrinsic operational activity in the Berbeşti mining basin, conducted to a production of 2.2 million t/year, in terms of productivity of a thickness of 8-10 t/m², lead to additional annual occupation of an area of minimum 30 ha/year, to which must be added and the necessary expansion of the dump of Oltet coal pit.

And location of the main premises, roads and coal and materials deposits, where the transport is carried out mostly by rail, it required removal of set-aside land with a good productive potential. Last but not least it should be stressed that the installation of mining mass transportation network, electricity supply, mounting platforms, fuel filling stations, pools of water and others, contribute a significant proportion of the sum of the areas of land that are taken out of the forest, set aside or for a longer time.

The situation of the occupied lands by the mining activity, and the order in which they are used is shown in table 3 [3].

By the end of the mining works of Berbeşti mining basin, will be affected another 809 ha of land with different types of use, as shown in the table 4.

It is revealed that both the expansion and agricultural productive potential, Oltet coal pit produce the maximum impact of anthropogenic impacts on the environment.

This is due to both the size of the coal pit, as well as placing the dump in an area outside of the meadow, situated at the confluence of the Tărâia and Oltet brooks what changes the landscape giving it a good appearance.

Table 5 it is shown the situation in the occupied lands in the present Oltet coal pit, the purposes for which they are used and the initial categories of use.

**Table 3.** *Waste land made by the mining activity according to the licence exploitation*

| Name of coal pit | Bad land areas [ha] | | | | | Area from licence perimeter [ha] |
|---|---|---|---|---|---|---|
| | Total day [ha] | Where | | | | |
| | | Excavations | Dumps | Buildings and premises | Roads and railways | |
| Olteț | 548 | 62 | 443 | 34 | 9 | 1099 |
| Berbeşti | 351 | 62 | 256 | 29 | 4 | 480 |
| Panga | 385.5 | 48 | 332.5 | 5 | 5 | 430 |
| Ruget | 404 | 112.5 | 255.5 | 28 | 8 | 536 |
| Bustuchin | 48 | | | | | * |
| **Total** | 1736.5 | 284.5 | 1287 | 96 | 26 | 2545 |

*- waiting for the final solution

**Table 4.** *Type of land use for waste land*

| Type of use | Area [ha] | Percentage in the use type [%] |
|---|---|---|
| Arable | 278,23 | 8,06 |
| Orcharding | 10,19 | 1,64 |
| Vineyard | 1,46 | 4,8 |
| Meadow | 67,02 | 5,39 |
| Forestry | 123,13 | 1,81 |
| Yards, constructions | 24,04 | 5,14 |
| Waters | 0,19 | 0,12 |
| Roads | 9,91 | 3,58 |
| Unproductive | 23,87 | 2,67 |
| Total area | 809 | |

**Table 5.** *The main use of the lands and their purpose*

| Olteț coal pit | Land use category [ha] | | | | | | Total area[ha] |
|---|---|---|---|---|---|---|---|
| | Arable | Orchard | Meadow | Yards | Unprod. | Forests | |
| Outer dump | 245 | | | | | | 245 |
| Inner dump | 36,1 | 4,8 | 134,7 | 2,2 | 15.4 | 4,8 | 198 |
| Coal pit | 12,2 | 2,6 | 33,7 | 1,8 | 9,5 | 2,2 | 62 |
| Coal yard | 4,4 | 0,6 | | | | | 5 |
| Dump flow | 13,8 | 1 | | | | 0,2 | 15 |
| Premises etc | 13,7 | 1,5 | 6 ,4 | 0,4 | 2,6 | 1,4 | 26 |
| Total area | 324,2 | 10,5 | 174,8 | 4,4 | 27,5 | 8,6 | 548 |

## Conclusions

- The development of mining recorded in the past 30 years needed an occupation of over 2200 ha of land which had served prior to obtain agricultural products or forestry;
- Restricting the mining activity started in 1997 left free of large technological tasks expanses of land that together with heaps of external surfaces, they could also be given in the economoic productive circuit;
- Landslides from Oltet and Panga coal pit, even many years after the completion of the works getting into the economic circuit, as a result of human factors, have drawn attention to the risk of significant damage to property, embodied by the adoption of additional measures to increase the stability reserve;
- As a result of the fact that the average of 1975 m²/inhabitant of the affected area is much lower than the national average of 4,500 m² and taking into account the request of the local communities is a priority for re-use of degraded lands of the mining activity, the specialists concerns were highlighted by the attainment of this objective on the grounds of horizontal surfaces and on incline lands resulted from the re-shape of the first bench of the inner dump of Panga coal pit;
- The land which inclination prevents the achievement of mechanized agriculture and which are prone to erosion or landslides have been reused, so far being successfully planted Acacia species only;

The reduction or depletion of coal reserves inmany coal fields in the basin requires the enhancement ofreclamations in the productive economy circuit.

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# ADAPTING 2K-52MU CUTTING AND LOADING MACHINE TO TR-5 SCRAPER CONVEYER

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***Abstract:*** *The necessity of mechanization of coal extraction in Jiu Valley mines, in the conditions of the present economic crisis, involved the adaptation of the existing equipments to the new conditions of reequipping of certain faces. Thus, within the program of reequipping of a longwall in Lonea Mine, the adaptation of 2K-52MU cutting and loading machine to TR-5 scraper conveyer was required. The main problem of adaptation of the two equipments was in the design and development of systems for affixing the ends of the feeding gear chain of the machine to the driving and returning stations of the TR-5 scraper conveyer. In solving this problem, the focus was on ensuring for all the elements of the attaching system of the ends of the machine chain to resist to the machine's maximum haulage force of 250 kN, and this strain not to be transmitted to the metal structure of the stations and be taken over by the hydraulic prop of anchorage of the conveyer station*

***Keywords***: *adaptation, cutting and loading machine, scraper conveyer*

**1. Introduction**

To adapt the existing equipments in view of reequipping the faces, one should be deeply familiar with the structure and technical characteristics of the equipments that are used and of the way they are correlated with the mining method, the geological-mining conditions specific to the layer exploited, and with observing the occupational health and safety norms.

By the implementation of a longwall reequipping program in Lonea Mine in the fourth trimester 2013, the classical method of extraction with a productivity of 6,8 ton/job was abandoned in favor of longwall extraction method with individual support and cut with a machine with an estimated productivity of 12 ton/job. A technical documentation was required to be drawn up in this sense, regarding the adaptation of the affixing program for the 2K-52MU machine to TR-5 scraper conveyer stations. In order to see the correlation of the equipments, an analysis of the framework extraction method was made for longwalls with cutting and loading machines. Fig. 1 shows the working stages for a maximum 3,2 m high longwall and 1,8 m high cut with the machine, as it follows:

I – machine in the bay; II – cutting the first strip; III – manual fixing of the place for mounting the articulated beam; IV – machine advance in the bay and mounting of the fourth prop; V – cutting the second strip; VI – manual cleaning up of the face and machine advance in the bay; VII – moving the back prop under the first beam; VIII – ripping of the back prop and beam.

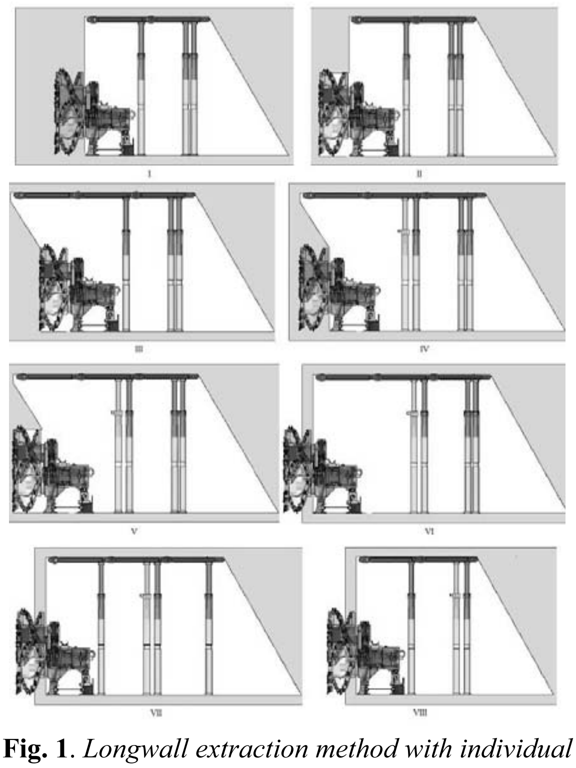


**Fig. 1**. *Longwall extraction method with individual support and machine cutting*

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From the above, it results that an analysis of economic efficiency should be made, regarding the correlation of the longwall height with the cutting height of the machine, with the geological mining conditions and the occupational health and safety norms. The price of the 2K-52MU machine, recovered from cassation should be considered, scrap iron – 2500 Euro, rehabilitated and put to operate with maximum 5000 Euro.

The technical characteristics of the 2K-52MU machine were not correlated with those of the TR-5 scraper conveyer, especially considering that the latter had not been designed for faces, and they had been manufactured by different companies in different countries. The two equipments are nevertheless compatible, at least from the following points of view:

- More than half of the theoretical cutting capacity of the machine, which is not achieved in practice due to the correlation of the advance speed with the geological-mining conditions, can be handled by the scraper conveyer;
- Size-wise, the machine can be mounted on the conveyer by adopting suitable modifications, adapting TR-5 stations to chutes, TR-6 with the use of TR-7 laterals, and modification of machine shoes without diminishing the characteristics of their resistance;
- The conveyer's robust design supports the approx. 12 …14 ton mass of the machine;

The design of the scraper conveyer allows coal to be loaded in good conditions by the auger drums.

The main lack of correlation between the two equipments consists in the fact that the actuating and return stations for the TR-5 conveyer are not foreseen with attaching hangers/plates of the affixing device of the machine haulage chain end.

## 2. Affixing system to the actuating station

Fig. 2 shows the design solution for the positioning of the attachment system of the end of the chain calibrated with 26x92 links of the 2K-52MU machine advance mechanism, reference point 2, in the metal construction of the actuating station, reference point 1.

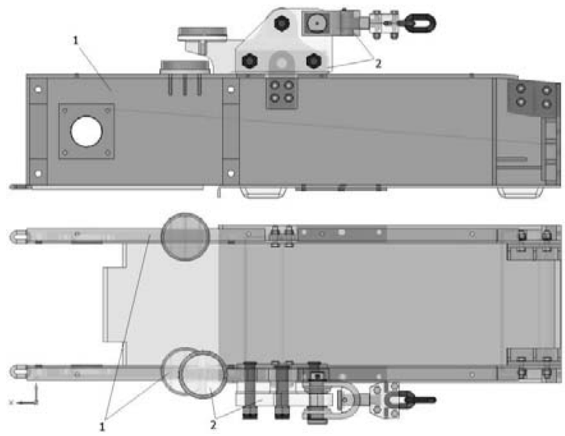

**Fig. 2.** *Movement of the affixing system on the actuating station [3]*

The design solution of the affixing system is shown in Fig. 3, which is made up of 1 – hanger fixed on the station; 2 – lateral connecting plate; 3 – lateral blocking plate; 4 – blocking bolt Φ50; 5 – blocking bolt Φ60; 6 – articulation bolt Φ60; 7 – device hanger; 8 – haulage rod; 9 – wear ring; 10 – wear ring support; 11 – chain fixing strap; 12 – centering pin; 13 – chain calibrate with 26x92 links; 14 – M20x100 screw; 15 – M10 screw for greasing; 16 – distance ring; 17 – M42 low washer; 18 – M42 washer; 19 – M24x100 screw.

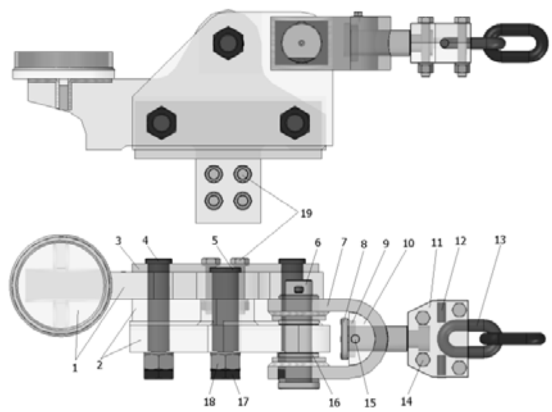

**Fig. 3.** *Design solution for the affixing system of the chain end to the TR-5 actuating station[3]*

The affixing system for the chain end to the TR-5 actuating station shows the following design improvements:

- the hanger fixed to the station, instead of the lifting hanger, leans on the framework of the station by two distanced shoes and has its own support for the SVJ station anchoring hydraulic prop, which improves stability of hanger and the way forces are transmitted to the station;

- overlapping the prop supports avoids the error possibility in placing the anchoring prop and allows interchangeability of the actuating station;
- parallel positioning of the lateral linking plate, reference point 2(Fig. 3), and rigid fixing on the hanger through the three bolts Φ50, reference point 4, and Φ60, reference point 5, made a movement outward the chain axis by 120 mm, which improved the movement of the machine on the conveyer in the area of the actuation station;
- the distance between the wings of the device hanger, reference point 7, by inside welding of the of two 20 mm thick attachments plates, which improved the strain on the articulation bolt, reference point 6;
- axial rolling contact bearing with balls 51112 was replaced with a wear ring of cast iron or bronze, reference point 9, its part being to take over the twist of the chain, and a screw was provided, reference point 15, for greasing the contact area between the wear ring and the ring support, reference point 10;
- transversal pins were provided, reference point 12, to center the two bridles in view of improving the haulage force transmission between the haulage rod, reference point 8, and the chain link, reference point 13;
- only 20 mm and 60 mm thick sheets were used, which were in stock in Lonea Mine;

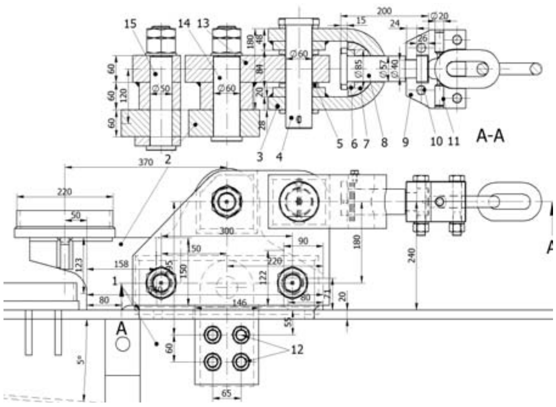

**Fig. 4.** *Calculation model for the connection system at the actuation station[3]*

Based on the design solution in Fig. 3, the calculation model was drawn up, with constructive dimensions in view of dimensional verification of its elements given in Fig. 4, where: 1 – partition of actuation station; 2 – hanger fixed to the station; 3 – device hanger; 4 – articulation bolt Φ60x270; 5 – distance ring; 6 – wear ring; 8 – haulage rod Φ57x200; 9 – bridle; 10 - M20x100 screw; 11 – centering pin Φ20x45; 12 - M24x100 screw; 13 – lateral connecting plate; 14 – blocking bolt Φ60; 15 – blocking bolt Φ50.

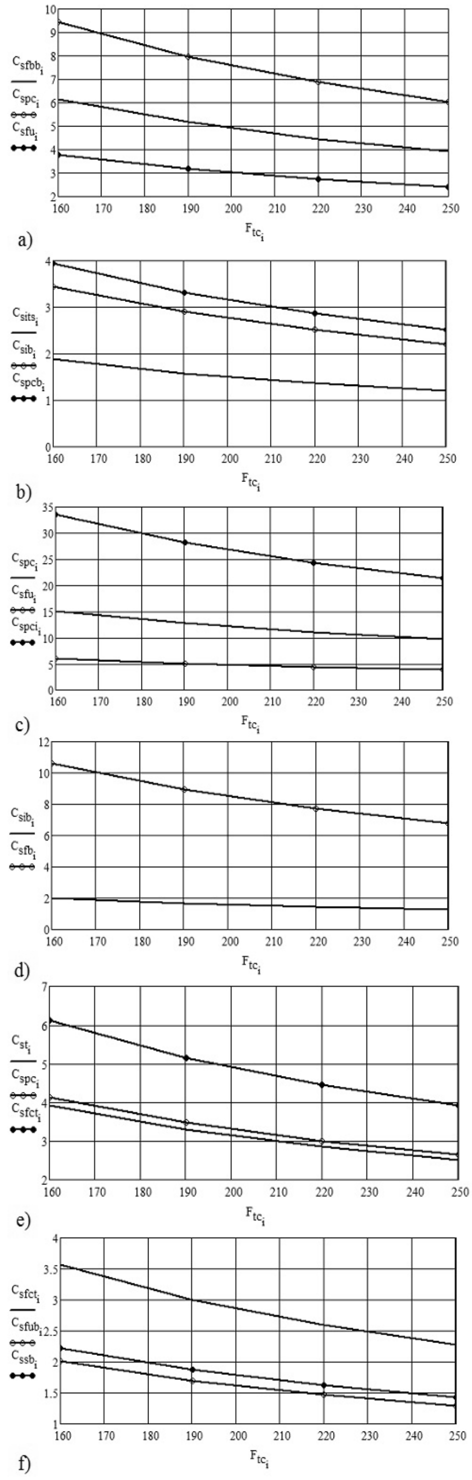

**Fig. 5.** *Variation of safety coefficients of connection system to the anchoring station[3]*

Based on the calculation model in Fig. 4, a calculation breviary was drawn up in Math CAD for the variation of the machine's haulage force $F_{tc}$ between 160 and 250 kN, and the value of safety coefficients are graphically shown in Fig. 5 for the following constructive elements

- lateral connecting plate(Fig. 5a);
- hanger fixed to the station, 2, (Fig. 5b);
- hanger of the connecting device, 3,(Fig. 5c);
- articulation bolt Φ60x245, 4, (Fig. 5d);
- haulage rod Φ57x200, 8, (Fig. 5e);
- attachment bridle for the chain link, 9, (Fig. 5f). Safety coefficients resulted by relating to mechanical characteristics of OL 37, 210 N/mm$^2$ flow limit for sheets and improved OLC 45, 500 N/mm$^2$ flow limit for bolts. The lowest values are with bending for bolts, reference point 4, $C_{sib}$=1,28, and with shear of the fixing bridle hanger of the chain, reference point 9, $C_{sfub}$=1,29, these values can be amplified 1,7 times, if the relating is made at tear resistance.

Based on the documentation of execution the system of connection of the machine chain to the actuating station was carried out in Lonea Mine, Fig. 6

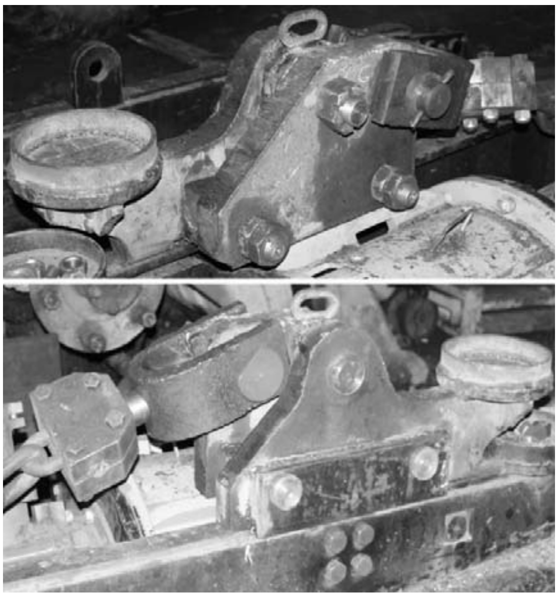

**Fig. 6.** *Chain connecting system to the actuating station carried out in Lonea Mine[3]*

### 3. Connecting system to the return station

Fig. 7 shows the constructive positioning solution of the connecting system of the chain end calibrated with 26x92 links of the advance mechanism of the 2K-52MY machine, reference point 2, to the metal structure of the return station, reference point 1.

Fig. 8 shows the constructive solution of the connecting syste, made up of: 1 – hanger fixed to the station; 2 – articulation bolt Φ60; 3 – chain connecting device; 4 - M30x100 screw; 5 – M30x200 screw.

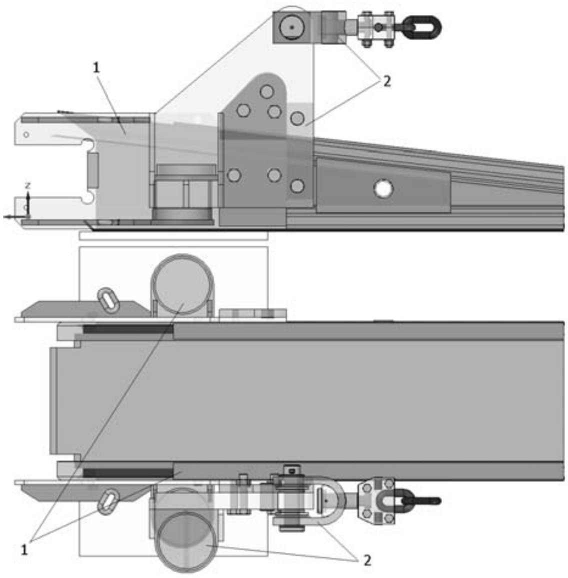


**Fig. 7.** *Connecting system positioning to the return station[3]*

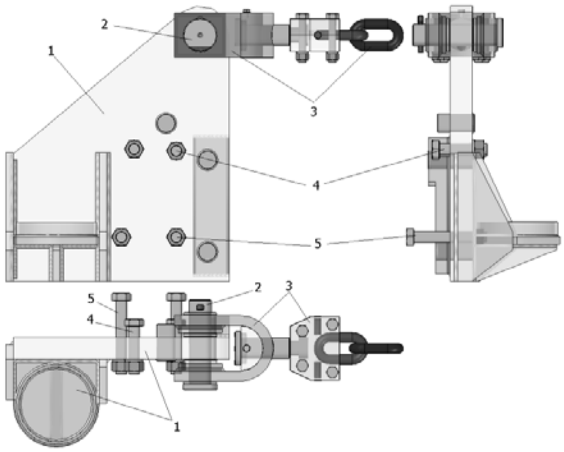


**Fig. 8.** *Design solution of the chain end connecting system to the TR-5return station [3]*

The chain end connecting system to the TR-5 return station shows the following design improvements:

- the hanger fixed to the station, to the lifting bolt by a bolt Φ50 and four screws M30, has in front a rigid threshold by two bolts Φ50, blocking the turning of the hanger against the station partition

- on the outside of the hanger the support for the station anchoring hydraulic prop SVJ was placed, in a welded structure, solider than the one on the return station;
- by overlapping the prop supports, the error of wrong positioning of the anchoring prop can be avoided;
- in case of changing the return station, part of the gussets for hardening the prop support should be discharged with oxyacetylene flame, on the return station, on the hanger's mounting part, and the two holes for screws M30x100 should be made in the lifting hanger.

Based on the design solution in Fig. 8, the calculation model with constructive dimensions in view of dimensional verification of its elements, shown in Fig. 9, was drawn up, where: 1 – station lifting hanger ; 2 – connecting system hanger; 3 – bolt Φ50x100; 4 - bolt Φ50x120; 5 - screw M30x100; 6 – chain connecting device.

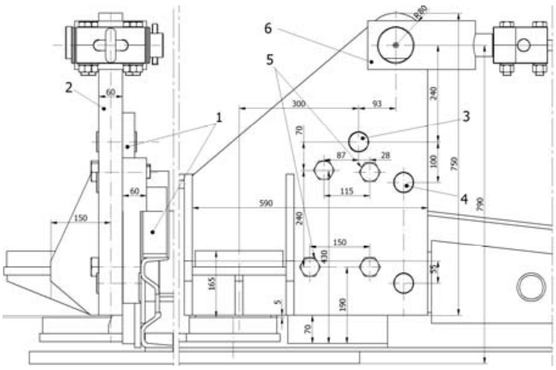

**Fig. 9.** *Calculation model for connecting system to return station[3]*

Based on the calculation model in Fig. 9, a calculation breviary in MathCAD was drawn up for $F_{tc}$ haulage force variation of the machine, between 160 and 250 kN, and the values of the safety coefficients are graphically shown in Fig. 10 for the following design elements:
- assembling by bolt Φ50x80, 4, and screws M30x100, 5, of the hanger on the station partition, (Fig. 10a);
- station lifting hanger, 1, (fig. 10b).

Since the values of sliding safety coefficient $Csa_i$ are low, an additional blocking of the hanger was carried out against the return station lifting hanger by bolts Φ50x120, reference point 4, and a vertical rule.

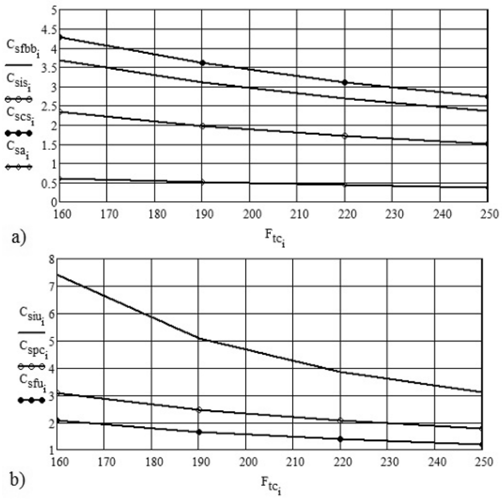


**Fig. 10.** *Safety coefficients variation of the connecting system to the return station[3]*

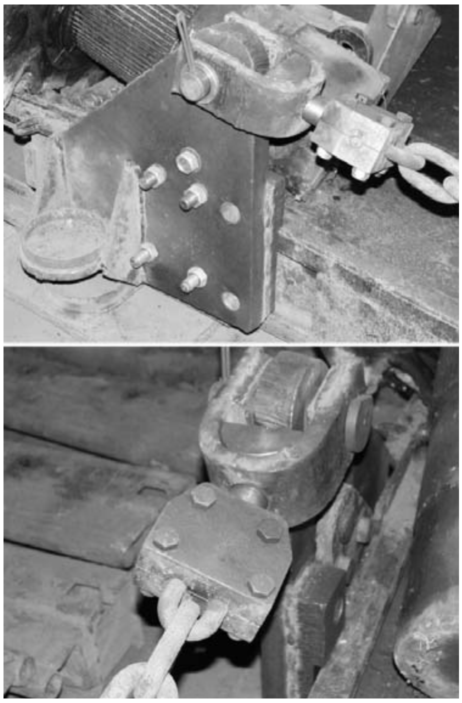

**Fig. 11.** *Connecting system of the chain to the return station carried out in Lonea Mine[3]*

Based on the execution documents, a connecting system of the machine chain to the return station was carried out in Lonea Mine fig. 11.

To check the way of positioning and movement of the 2K-52MU machine on the conveyer carried out of TR-5 stations, TR-5 chutes and lateral of TR-7A , the assembly in Fig. 12 was carried out in Lonea Mine Shops, with new systems of connecting the chain ends to the machines' advance mechanism.

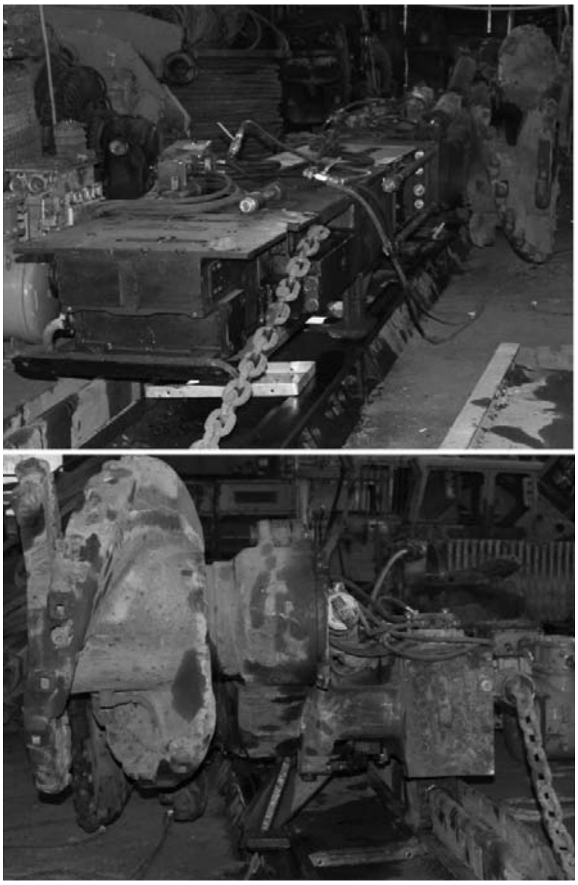

**Fig. 12.** *Mounting the machine to the conveyer[3]*

## 4. Conclusions

Even if the cutting height of the machine is not correlated with the height of the face, an improvement of work conditions results for a minimum reequipping of the face.

By adapting the 2K-52MU machine to a hybrid scraper conveyer, TR-5 stations, TR-6 chutes and TR-7A laterals, experience was gained in the use of these equipments in the exploitation of longwalls of individual support ant machine cut.

In an endeavor to move the machine on the conveyer, the connecting systems of the chain ends of the advance mechanism to the conveyer stations were checked.

The information obtained as a result of face reequipping in Lonea Mine, with the advantages and disadvantages of the method application, will allow in the future to achieve an optimization of the equipments correlation in a shortwall working.

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# IN MEMORIAM
# Prof. eng. Ph.D VASILE POPA

On 24 July 2013, prof.eng. Ph.D Vasile Popa, important personality of the romanian mining industry and mining high education, passed away.

Professor Vasile Popa was born on 28 July 1936 in Totoreni, Bihor county on the left bank of the Crisul Negru river, 22 km from the hem of Apuseni Mountains. He takes the primary school in this village, afterwards the „Samuil Vulcan" High School in Beius.

After getting his Baccalaureate degree he follows the Petrosani Mining Institute, and in 1959 he gets his Engineer License as class leader from the Mining Faculty, specialization Mining.

For his marks and results and his university behavior and leadership, he has been elected as President of the Students Association in the Petrosani Mining Institute.

Since 1959, due to the government decision of production repartition, he became stagiary engineer for a year, then 4 years as department head of Sasar Mining Enterprise, one of the most important mining units in Maramures.

Due to the results in his department, to his production and work management capacity, to his concerns for mining technology improvements, he was promoted to Baia Mare Mining Trust as head of the Technical Department. On 1st January 1970 he was promoted as Development Chief Engineer, and he coordinated the development and operation of multiple industrial and social objectives, with the purpose of increasing the romanian metal ore production and enhancing the life and work conditions for the miners in Maramures. The opening of some new mining sectors in Baia Mare region, the expansion and construction of new processing plants, the development of work technologies from mining underground, the development of social and education buildings are connected with the name of prof.eng. Ph.D Popa Vasile. Prof. His most important achievement is the development in Baia Mare of a modern high education institute with library, amphitheaters, lecture and laboratory rooms, accomodation spaces and dining rooms for students.

Starting in 1967, for 4 years, he follows the extramural Ph.D program, taking all the stages from the individual training plan under the guidance of prof.eng.doc. Ph.D Stefan Covaci. In March 1972 he holds the Ph.D paper with the title „Contributions to enhancing the vein deposit preparation methods from the CMN Baia-Mare mines, with the purpose of reducing the pillar losses", gaining the title of „Ph.D engineer" in Technic Sciences.

Due to his outstanding production results, the Mining Ministry management enrolls him to the Central Institute for Economic and State Administration Management Training, where he graduates after two years the specialization „The organization and management of industrial activity".

Together with the department head and chief engineer activity at the Ore Center in Baia Mare, prof.eng. Ph.D Vasile Popa worked as associate professor at the Mining High Education Institute in Baia Mare.

In 1973 he gets the universitary associate professor title, takes the education work as his main activity and gets the vice-rector title for the High Education Institute in Baia Mare.

Starting with 1974 until 1981, prof.eng. Ph.D Vasile Popa was rector of the High Education Institute in Baia Mare, followed by other positions of dean and department head until 1997 in the same instituion, where he worked for more than 25 years.

In 1994 the Education Ministry granted the position of Ph.D coordinator, so he practiced his scientific competence and rigor in coordinating young specialists for finishing their Ph.D papers in the domains of mining industry and geology.

Together with his production and education activities he held an intense scientific and publishing activity, solving and supporting various scientific papers connected to the technology improvement in mining and non-ferrous metalurgy and publishing a significant number of articles, universitary courses and speciality treaties which are useful guidelines for the following mining engineers promotions.

Beside his professional qualities, Vasile Popa was also a kind person, who understood and helped everyone who asked for his advice and support, from the simple student to specialist, reason why I am sure we will keep for him a sense of esteem and gratitude.

In sign of appreciation for his production, education and social activity, for his valuable results in training the specialists from mining industry, in solving technical problems for mining enterprises and for the scientific organization of production and education activities, he was decorated with various romanian orders and medals.

Through the loss of prof.eng. Ph.D Vasile Popa, the romanian science and high education suffered the loss of a high moral and professional specialist, in which memory the romanian mining specialists will keep an eternal esteem and consideration.

Prof.eng. Ph.D *Dumitru FODOR*
Rector of the Petrosani Mining Institute (1978-1989)
Active Member of the Technical Sciences Academy in Romania