50 years

University of Mining and Geology "St. Ivan Rilski" (1953 – 2003)

A N N U A L

OF THE UNIVERSITY OF MINING AND GEOLOGY "ST. IVAN RILSKI" – SOFIA

PART II: MINING AND MINERAL PROCESSING

Vol. 46 **2003**

Publishing House "St. Ivan Rilski"

Sofia
2003

ISSN 1312-1820

CONTENTS

50 years **University of Mining and Geology "St. Ivan Rilski"**
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.1-4

AN APPROACH TO DESIGNING A CLASSIFICATION OF THE UNDERGROUND ORE MINING METHODS

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ABSTRACT
An approach is proposed permitting every mining method to be expressed by a concrete logical formula. For the purpose, an alternative graph is constructed that takes into account the alternative elements of the mining technology. The general sequence of the elements supposed to construct the technical and economic model of each mining technology is determined by means of a logical formula. This is a decisive step in the procedure related to the technical and economic comparison and selection of an optimal mining technology for particular natural conditions.

A great number of classifications of mining methods and technologies are used the underground ore mining practice. In different regions and at different times different features have been used thus resulting in more than 50 known classifications. Each one formulates differently the set of similar elements eventually termed classes of mining methods. With improving the mechanization of the production processes a change can be noted in the tendencies for selecting a classification feature or features in designing a given classification. The large extent of variability of the natural factors and the low degree of mechanization during the 1930s presupposed the design of classifications combining over 200 various ways of mining the mineral deposits. The approach is the same, viz. building structures consisting of classes, groups, variants and subvariants of mining methods. Of the well-known classifications we should note M.I.Agoshkov's classification. It is based on the feature *stope state*. R.P.Kaplunov's classification is widely used. It is built up on the feature *rock pressure control*. V.R.Imenitov's classification is based on the *method of stope maintenance* during the ore extraction. In the western literature (mainly American, British and Australian reference sources) the concept of mining method is usually associated with the procedure for selecting an optimal variant of mining technology. Hartman's classification considers jointly the openpit and underground mining methods. It is based on the subdivision of the mining methods into classes and subclasses and for each one it gives the method of working and type of mineral mined. Morrison's classification is based on the geomechanical aspects, and in particular, the possibilities for accumulating the energy of the rock mass deformations. From this point of view we determine the method of maintaining the stope: without supports; with supports; with filling; with pillars.

Nicholas determines the applicable mining methods by using the weight coefficient He introduces this coefficient to account for various factors: orebody shape, rock characteristics in the hanging and foot walls, distribution of the valuable constituents in the ore.

D.M.Bronnikov elaborates on the question of the classification of mining methods and selection of an optimal mining technology by specifying the mining conditions of particular relevance in the form of a systematized table: *wallrock stability, ore stability, ore value*. These are the three features that determine the set of applicable mining methods for each combination between them. This is one of the modern perspectives for designing a classification that uses several features at the same time.

At least two are the characteristic features that should be taken into account in designing a classification of the mining methods at present.

First.

The designs of the machines used for the production processes at the extraction face are very sophisticated thus making them adaptive to the high degree of variability of the natural conditions (orebody thickness, slope angle, ore and wallrock stability). It is not necessary any more to design classifications consisting of many classes in order to achieve complete correspondence between the features on which they are based and the real natural factors.

Second.

Modern software and hardware products create practically unlimited possibilities for indetifying every mining technology. The problem is how to derive the factors of greatest weight so as not to burden unnecessarily the search for an optimal solution. From that point of view, the use of a graph with a suitable architecture can serve as a basis for ordering the features to be used in describing the respective mining technology.

The problem is reduced to determining the number of levels N and number of alternatives of each level M_N. Fig. 1 shows the construction of an alternative graph. It represents the structure of the elements constituting the core of the mining method. The alternative graph consists of six levels, i.e. N = 6. The six levels determine the classification features on the basis of which we design our classification of mining methods. The levels are ordered in the following sequence:

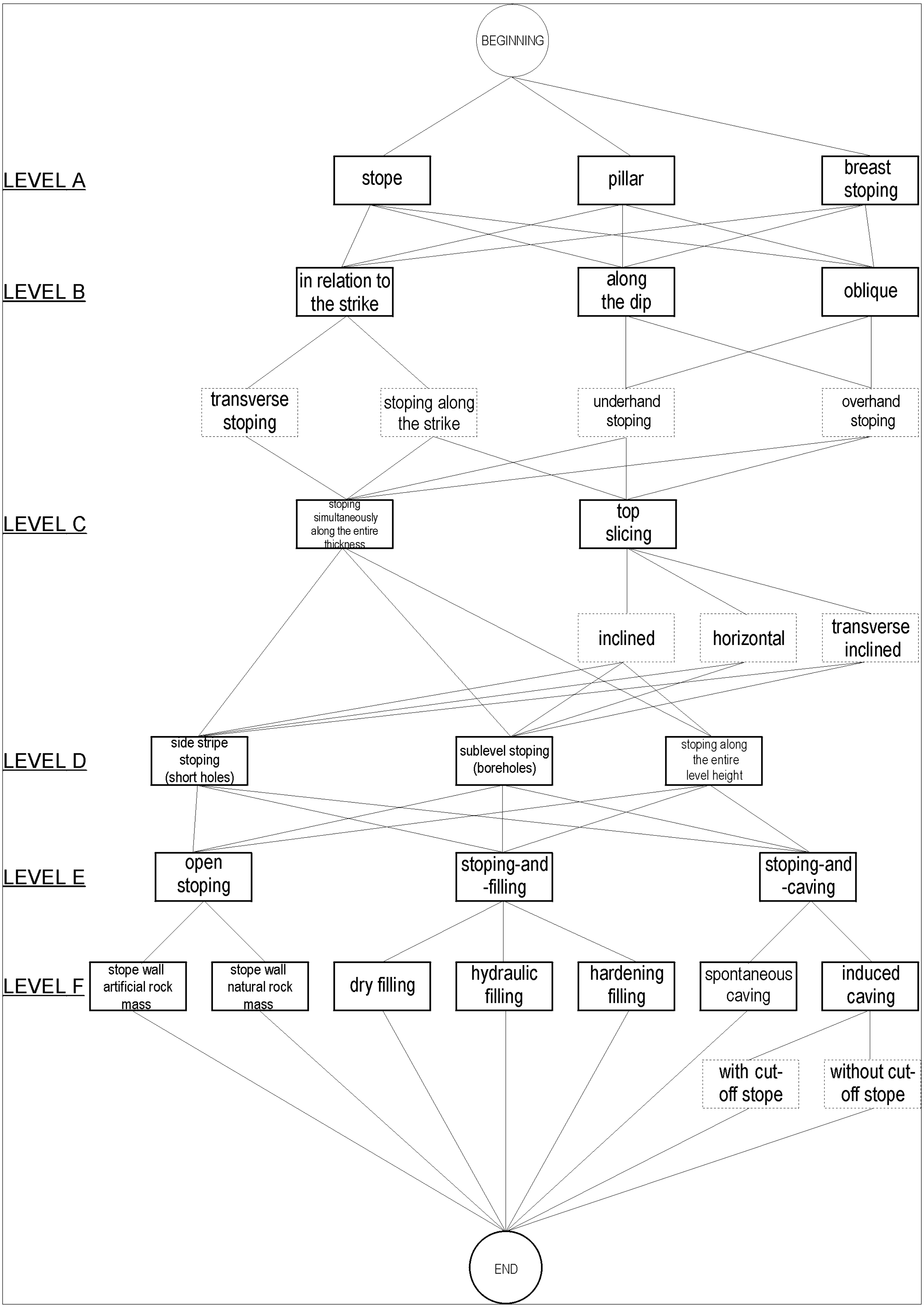
BEGINNING
LEVEL A
stope
pillar
breast stoping
LEVEL B
in relation to the strike
along the dip
oblique
transverse stoping
stoping along the strike
underhand stoping
overhand stoping
LEVEL C
stoping simultaneously along the entire thickness
top slicing
inclined
horizontal
transverse inclined
LEVEL D
side stripe stoping (short holes)
sublevel stoping (boreholes)
stoping along the entire level height
LEVEL E
open stoping
stoping-and -filling
stoping-and -caving
LEVEL F
stope wall artificial rock mass
stope wall natural rock mass
dry filling
hydraulic filling
hardening filling
spontaneous caving
induced caving
with cut-off stope
without cut-off stope
END

Figure 1.

Level A. Type of main production unit for performing the mining operations.

Level B. Direction of development of mining operations within the production unit.

Level C. Method of ore extraction with respect to orebody thickness.

Level D. Method of ore extraction with respect to the geometry of the production unit.

Level E. Stope state during ore extraction.

Level F. Computer hardware and software for designing the stope state.

The generalizing parameters of the alternative graph are presented in Table 1. The following rules are adopted for designating the alternatives and sublevels. The serial number of the alternatives for each level is found in the interval 1, M_N. Let us take, for example, **Level C. *Method of ore extraction with respect to the orebody thickness***, two alternatives are possible, i.e. $M_3 = 2$. Stoping simultaneously along the entire thickness, ***C1***; top slicing, ***C2***. The maximum number of alternatives on the sublevels is three (see Table 1). They are designated by symbols ***a, b, c***. Let us take the example with level C. In terms of the direction of development of the mining operations within the production unit ($M_2 = 3$) there is a sublevel that is described by the following alternatives: stoping along the dip in a descending order, ***B2a***; stoping along the dip in an ascending order, ***B2b***; oblique stoping in a descending order, ***B3a***; oblique stoping in an ascending order, ***B3b.***

Table 1

No	Designation of levels	Alternatives		No of alternatives in the sublevels
		No	Design-ation	
1	Level A $\Rightarrow$ N = 1	3	$M_1 = 3$	-
2	Level B $\Rightarrow$ N = 2	3	$M_2 = 3$	4
3	Level C $\Rightarrow$ N = 3	2	$M_3 = 2$	3
4	Level D $\Rightarrow$ N = 4	3	$M_4 = 3$	-
5	Level E $\Rightarrow$ N = 5	3	$M_5 = 3$	-
6	Level F $\Rightarrow$ N = 6	7	$M_6 = 7$	2

The levels of the alternative graph are the classification features on which the classification of the mining methods is based. In this case their number is N = 6. It should be noted that a similar approach was not used in designing such a classification. Another well-known classification is that of O.A.Baykonurov's. He adopts the matrix recording but the elements are two and correspond to the number of columns and rows in the designed table.

The methods of logical algebra are used to describe the forming set of possible combinations for the separate members of the alternative graph. They are used to translate into machine language the general equations of the production processes, engineering and technological solutions depending on their local parameters. Each system considered, according to the structure of the alternative graph, can be written by its logical formula. For example, the logical formula of stope filling with hydraulic filling, according to M.I.Agoshkov's classification, will be written as follows:

$$A1 \wedge B2b \wedge C1 \wedge D1 \wedge E2 \wedge F4$$

In this case the sign " $\wedge$ " means conjunction (logical multiplication) of the following logical variables:

- ***A1*** - Extraction within the stope boundaries;
- ***B2b*** - Overhand stoping along the dip;
- ***C1*** - Stoping simultaneously along the entire strike;
- ***D1*** - Side stripe stoping with short holes;
- ***E2*** - Stoping-and-filling;
- ***F4*** - Stoping-and-hydraulic filling;

The use of six classification features at the same time shows that most classifications known until now should be compatible with the one proposed in this paper. Taking into account the number of levels, sublevels, alternatives (local parameters of variables), incompatible connections, the number of methods involved in the classification amounts to 1000. This number covers the applicable mining methods under real natural conditions.

The proposed classification of mining methods has several advantages that can be reduced to:

- Prerequisites are created for generating new technological solutions that have not been developed yet, but with the development of mining mechanization they will inevitably be resorted to;
- Multi-stage mining technologies are practically considered and not only combined methods as is the case with most classifications known until now;
- By changing the structural elements of the mining methods it is possible to optimize the dimensions of the main production unit: stope, stope pillar, stope (exploitation) field;
- The architecture of the alternative graph by which the classification of the mining methods is described, enables us to use the same approach in the further detailed presentation of the basic technologies: open stoping, stoping-and-filling, stope caving.

The logical formula by which each mining method is written, enables us to move to the next step – constructing a suitable technical and economic model for evaluation of the efficiency in comparing a set of applicable technologies under certain conditions.

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Recommended for publication by Department of Underground mining , Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.5-10

VALUATION OF THE POSSIBILITIES FOR CHOOSING OF EFFECTIVE METHOD OF DEVELOPMENT FOR ORE BODIES WITH COMPLICATED MORPHOLOGICAL CHARACTERISTIC

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ABSTRACT
In development of ore bodies represented as separate geological blocks - poles, originate the problem with choosing of optimal method of development, when part of ore body is above ventilation horizon, respectively part of ore body remains below haulage horizon. In such cases is not expedience performance of full complex developing and cutting works, typical of the created mining technology because of limited volume of reserves in that zone. Object function is created, on the basis of that can be compared different variants of development and cutting of block with such complex morphological fact. Alternative graph is created, giving possibility for generating great number of variants, from among that should look for optimal.
Key words: ore, vain, minerals, mine field, development, graph.

INTRODUCTION

The design and practical implementation of mine fields developing is carried out under conditions of least reliability of information about the environment, when the mine field is determined in project stage and data are gathered during the process of driving prospecting workings. Often this information is based on data in driving or exploiting of mine workings in the mine field. This gives possibility to set up of more precise and more complete information for the space situation of the ore bodies in earth's womb.

The term of exploitation of the basic development and cutting workings often corresponds to the period of development of the mine field and every driving of needless workings in not paying ore zones has got serious economic consequences. This is showed in significant increasing of the investments for developing and cutting works, not retrieve of the investments subsequently.

BASE ON THE PROBLEM

When developing ore deposits the problem is further complicated by at least three additional, but at the same time essential factors published by G. Mihaylov and G. Georgiev (2002):

- Very often ore veins have got uneven distribution of useful components, they are represented by separate ore poles, which further on for convenience will be called *geological blocks*;
- The separate geological blocks in the ore veins have got clearly identifiable inclination, which in depth has considerable influence on the dimensions and boundaries of the mine field, i. e. inclination influences the topology of the network of extraction workings for the opening and preparation of the levels;
- The thickness of ore veins (geological blocks) is uneven, which means that the amount of loads corresponding to the reserves in the separate blocks on the separate levels will be different.

In this way premises are created for the development of a general algorithm based on multivariant approach for the choice of a technical solution and valuation of the variants for the developing of the mine field, based on the three-dimensional formulation of the problem. A reference coordinate system is introduced, oriented in such a way that the whole mine field is situated in the positive octant. The (y) axis coincides with the ore vein's strike line, while the (x) axis is oriented crosswise to the strike line, i.e. along the way of dipping.

When the geological blocks' coordinates on the separate levels are known it is possible to calculate the geometrical dimensions, the amount, the productivity and the location on the separate levels by means of three angles (fig.1):

- The angle of dip α of the vein or the geological block in the plane 0xz, α varying on the interval $0 \leq \alpha \leq 180°$;

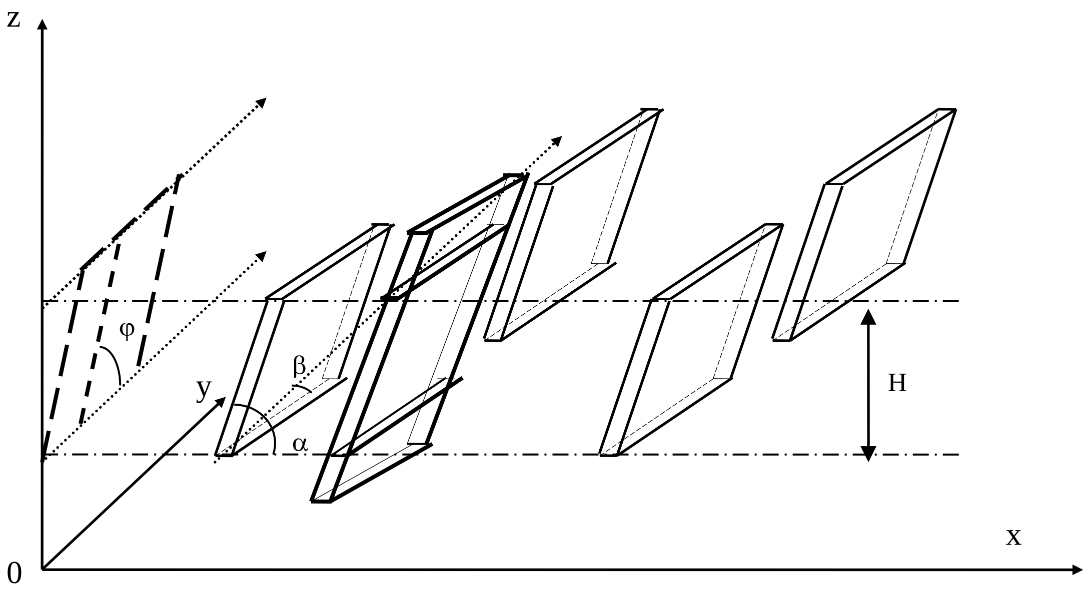

Figure 1. 3D base on the problem

- Angle of inclination φ of the geological block on the separate levels in the plane 0yz, φ varying on the interval $0 \le \varphi \le 180°$;

- The angle of azimuth β of the geological block on the separate levels in the plane 0xy, β varying on the interval $0 \le \beta \le 360°$.

The calculation of parameters such as geometrical dimensions, amount, productivity, takes place once the angles α, β, φ have been determined by means of the geological blocks' coordinates on the separate levels.

The problem thus formulated necessitates the introduction of triple indexation - i, j, k where:

i = 1..n – the index showing the sequential number of the geological blocks (poles), located on the level, and situated along the ore veins' strike line;

j =1..m – the index showing the location of the following vein, determined crosswise to the strike line;

k =1..t – the index showing the level's sequential number within the mine field.

In this case, when the stope block's geometry is predetermined (length L_{bl} and height $H_{bl} = H_l$, where H_l is the level's height) it is possible to calculate the amount of reserves, respectively amount of loads Q[ijk], to be transported to the surface and then to the consumer.

That base on the problem is possible only in this way, when geological pole (block) not has got non ore zones along way of dipping (whole height of level).

When geological pole (block) has got interrupting along way of dipping in limit of level or the ore zone goes up from haulage (ventilation) horizon reach some level, which located between both horizons forming one mine level and subsequently the ore zone is non ore to ventilation (haulage) horizon. Than is necessary introduction on fourth index (d):

d =1,2, ... p – the index showing the sequential number of the geological blocks (poles), and situated along the ore veins' dip line;

The problem is complicated after introduction on fourth index, but on the other hand more complexly solution is obtained giving variability of the form of the ore zone along the ore veins' dip line. New index (d) characterizing naturally datum of the deposits is different from number of the levels. New index (d) in the mine field can be in three dependence d = k, d > k и d < k. One of very often variants is in d = 1, when the ore zone is without interruption along the whole dip line. The ore zone is intersecting by ventilation and haulage horizons on levels. Very often, however, this condition is not performed than d > 1.

For facility's sake when we look at surface 0xz, instead three-dimension (3D) base on the problem (fig.1), from where we can see the scheme of the location of the ore body represented by fig. 2

In the boundary of one level between haulage and ventilation horizons can be formed from several ore zones. Of course, this precise making of contour of the ore zones can be performed in the driving of developing workings, when the mine field is in stage of development, and not making of contour which can be performed in process of driving prospecting works.

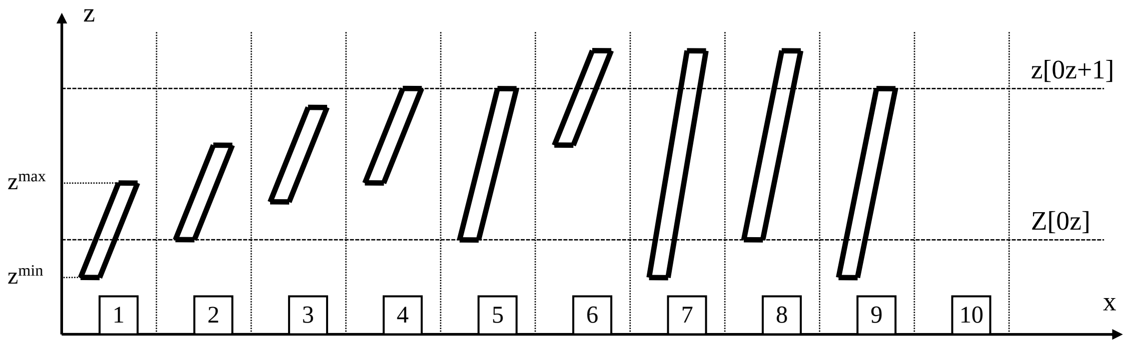

Figure 2. Location of the ore body in boundary of level

The situation of the ore body in the limit of level is shown in fig. 2. When we are look at Oxz, forming cross-section of the ore body can be make some marks:

- z[0z+1] is the axis of crosscut situated on ventilation horizon forming upper boundary of level.
- z[0z] is the axis of crosscut situated on haulage horizon forming bottom boundary of level.
- z^{max} – the maximum coordinate forming upper boundary of [i, j, d]th geological pole (block);
- z^{min} – the minimum coordinate forming bottom boundary of [i, j, d]th geological pole (block);

Last two are shown in fig.2 only in this case for variant 1, but in the other cases is on the analogy of all the rest variants.

MATHEMATICAL FORMULATION OF THE PROBLEM

Calculating the amount mineral body located in the ore zone between ventilation and haulage horizon Q[ijk]. For that purpose is necessary preliminary to be calculated the amount in a geological pole (block) – Q[ijd] in that complicated morphology of the ore bodies. This is represented by the formulas describing different variants:

- Variant 1:

$$Q[ijk] = (Q[ijd].Z_p[ijk]) / Z_o[ijd], t \quad (1)$$

where:

$Z_p[ijk]$ – the height along the (z) axis fig.2, located between maximum coordinate of the ore body and the axis of haulage crosscut and remaining in the boundary of level:

$$Z_p[ijk] = z^{max}[ijd] - z[0z], m; \quad (2)$$

$Z_o[ijd]$ – the height of ore body as difference between the maximum and minimum coordinate in each [ijd]th geological pole (block):

$$Z_o[ijd] = z^{max}[ijd] - z^{min}[ijd], m; \quad (3)$$

Q[ijd] – preliminary calculated amount ore mass in each [ijd]th geological pole (block),t:

- Variants 2,3,4 and 5:

$$Q[ijk] = Q[ijd], t. \quad (4)$$

- Variant 6:

$$Q[ijk] = (Q[ijd].Z_{pk}[ijk]) / Z_o[ijd], t \quad (5)$$

where:

$Z_{pk}[ijk]$ – the height along the (z) axis fig.2, located between the axis of ventilation crosscut and minimum coordinate of the ore body and remaining in the boundary of level:

$$Z_{pk}[ijk] = z[0z+1] - z^{min}[ijd], m; \quad (6)$$

$Z_o[ijd]$ – the height of ore body as difference between the maximum and minimum coordinate in each [ijd]th geological pole (block):

$$Z_o[ijd] = z^{max}[ijd] - z^{min}[ijd], m; \quad (7)$$

Q[ijd] – preliminary calculated amount ore mass in each [ijd]th geological pole (block),t:

- Variants 7,8 and 9:

$$Q[ijk] = (Q[ijd].Z_e[ijk]) / Z_o[ijd], t \quad (8)$$

where:

$Z_e[ijk]$ – vertical height of level, m:

$$Z_e[ijk] = z[0z+1] - z[0z], m; \quad (9)$$

$Z_o[ijd]$ – the height of ore body as difference between the maximum and minimum coordinate in each [ijd]th geological pole (block):

$$Z_o[ijd] = z^{max}[ijd] - z^{min}[ijd], m; \quad (10)$$

Q[ijd] – preliminary calculated amount ore mass in each [ijd]th geological pole (block),t:

- Variant 10 – not has got minerals in level. That variant can be obtained in large interruption of the ore zone along the ore veins' dip line. That can be obtained when ventilation and haulage horizons forming level not intersect [I, j, d] ore body.

Creating of an object function in the developing and cutting in each blocks in the mine field:

$$F = \sum_{l=1}^{r} C_d L_{d(l)} + \sum_{l=1}^{r} C_c L_{c(l)} \rightarrow \min, \quad \$ \qquad (11)$$

where:
C_d – the prime cost of one linear meter (1m) developing working, \$/m;
$L_{d(l)}$ – the general length of all developing mine workings in the mine field, m;
C_c – the prime cost of one linear meter (1m) cutting working, \$/m;
$L_{c(l)}$ – the general length of all cutting mine workings in the mine field, m;
l = 1,2, … , r – shows number of the variants depending on the indexes [i, j, k].
Existing the following relation between them:

$$r = n.m.t \qquad (12)$$

That object function shows the investments necessary to the developing and the cutting of the mine field. For that purpose to be expedience development of the deposit the object function must be minimum:

The investment for developing and cutting in each one of the extraction geological blocks (i), in each ore vein (j), forming from ventilation and haulage horizons in level (k) and it is expressed by formula:

$$F[i, j, k] = C_d.L_d[i, j, k] + C_c.L_c[i, j, k], \quad \$; \qquad (13)$$

where:
$L_d[i, j, k]$ – length of all developing mine workings in the boundary of $[i, j, k]^{th}$ extraction block, m;
$L_c[i, j, k]$ – length of all cutting mine workings in the boundary of $[i, j, k]^{th}$ extraction block, m;

The financial incomes for each one of the extraction geological blocks (i), in each ore vein (j), in each level (k) and it is expressed by formula:

$$Inc[i, j, k] = Q[i, j, k].W, \quad \$; \qquad (14)$$

where:
Q[i, j, k] – the calculating amount mineral located between ventilation and haulage horizons in each one of the variants shown by fig.2.
W – extracting value obtained through formula published by V. Shestakov, A. Dulin etc. (1984):

$$W = 0{,}01\sum_{1}^{n} c(1-S)\varepsilon R, \quad \$/t \qquad (15)$$

where:
c – the substance of mineral components (metal substance) in ore, %;
S – the dilution of ore in extraction;
ε - the extracting of useful mineral components in mineral processing of ore mass;
R – the price of 1t metal, \$;
n – number of mineral components in ore.

The value of the ore is formed through containing in the ore useful mineral components – through their amount, quality and market price. When the ore contains one component, as for example the iron ore, more value is that, which is with higher metal content. The value of polymetal ore is represented as sum through the value of the metals which contains published by Dr. Steffanov (1993).

The value of the ore influence as over choice of method of mining and technology of development, similarly in the opening and the developing of the mine field. In value ore can be applied expensive technologies and method of mining, if it is insure minimum losses of ore in extraction. As well, can be prepared and cut geological blocks with amount less located on the level. In ore with little value permit cheap methods of opening, developing and extraction and it permit increased losses too.

The profitableness of the developing in each [i, j, k] extraction block is defined as difference between Inc[i, j, k] and F[i, j, k] with giving an account of extraction worth of the ore. When the difference is positively, the preparing and the development of separate extraction block is economic expedience. The profitableness, defined according to extraction value of the ore, still must apprehend as relatively, as in many cases must not evaluate the metal as material, and the last made product.

CREATING OF AN ALTERNATIVE GRAPH

According to thereby made 10 variants about situation of the ore body in the boundary of level, we could see from fig.2, is created general alternative graph shown in fig.3.

S_n – beginning of alternative graph;
S_k – end of alternative graph;

- Level A – depending on number of the ore body located in the boundary of level;

1 – one ore body or part of ore body;
2 – many ore bodies, where the amount ore mass located on level is calculated as sum;
3 – There is not ore body in level.

- Level B – depending on the location of upper contour of $[i,j,d]^{-th}$ ore body z^{max} toward upper boundary of level, in this case ventilation horizon or the axis of crosscut located on ventilation horizon - z[0z+1] shown in fig.2:

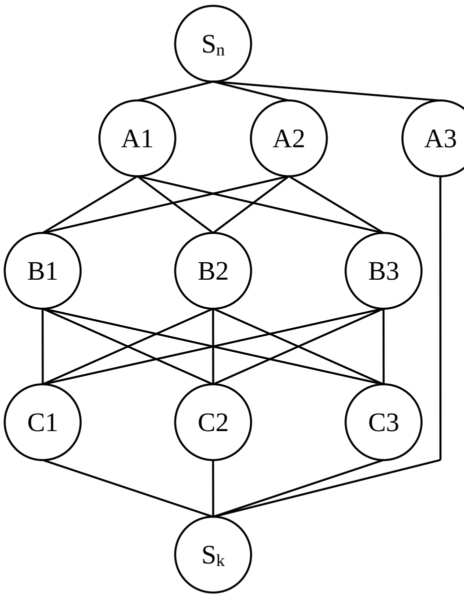

Figure 3. General alternative graph

1 – upper contour remains outside on level formed through ventilation and haulage horizon, and the axis of crosscut situated on ventilation horizon (fig.2) intersect the ore body;
2 – upper contour osculate ventilation horizon (the boundary of level) or the axis of crosscut situated on ventilation horizon (fig.2);
3 – upper contour remains inside between ventilation and haulage horizon (in the boundary of level) and the axis of crosscut situated on ventilation horizon (fig.2) not intersect the ore body.

- Level C – depending on the location of bottom contour of [i,j,d] $^{-th}$ ore body z^{max} toward bottom boundary of level, in this case haulage horizon or the axis of crosscut located on haulage horizon - z[0z] shown in fig.2:

1 – bottom contour remains outside on level formed through ventilation and haulage horizon, and the axis of crosscut situated on haulage horizon (fig.2) intersect the ore body;
2 – bottom contour osculate haulage horizon (the boundary of level) or the axis of crosscut situated on haulage horizon (fig.2);
3 – bottom contour remains inside between ventilation and haulage horizon (in the boundary of level) and the axis of crosscut situated on haulage horizon (fig.2) not intersect the re body.

The maximum common number of variants according to alternative graph can be shown by formula:

$$V = 10.n.m.t, \quad бр. \tag{16}$$

In this way a general problem is formed, the solution to which should be regarded as an example of the use of a comprehensive approach in the present day computer technology used in mining.

THE RISK OF THE INVESTMENT

According to Gotch, Zantrop and Eggert the risk of investments in field of development of mineral raw material is: “The risk in the true sense of the word is as measure of the rate of variability of the possible financial incomes and expense. Investments with low risk have got low variability of the possible financial incomes in comparison with that with high risk. The future incomes and expense, connected with investment for development of mineral deposit, are not sure, because is impossible the factors, which define them to be reliable well - known in the moment of investing process” published by M. Yordanov (1996).

The balance of incomes and expense of given business initiative is the extremely, solving factor in claim of solution for her realization. The reporting on the stability trend to drop of investments in the mining industry in worldwide importance especially actual are the efforts of some countries and governments to offer in most attractive project mineral raw material basis, published by M. Yordanov (1998).

The risk of investing is in a direct ratio of measure of the expectation a clear profit from the investment i.e. bigger risk – bigger clear profit, and of the other side – less risk less clear profit. That is important with same power for the investments in mining industry.

The risk of investing is provoked of factors of geological substance – incorrectly calculated (or not confirmed) reserves of ore and metal, from geoeconomic – changing of the price of an end produced product (metal) and political and macro economic – changing of law.

The first group risks is most often met and has got biggest significance because of the fatally consequences in not confirmed of the calculating reserves of ore and metal. Series of financial losses in a number of gold extraction and other companies in Australia, USA and Canada in the end during the past century are connected mainly with such risks. In the not confirmed reserves of ore and metal, which is under condition thoroughly of two group factors: applying of old procedures for example the method of geological cross-section and blocks for deposits with extremely an not even distribution of useful component (gold, silver etc.). As well, incorrect using of geostatistic methods and technologies.

The second group risks is connected with the changing of price of end product (metal). The changing of prices influence in significant rate over the extremely economic results. For example of that are the sudden changing of the price in copper during the past two years.

The third group risks is connected with the changing of law. That changing put under financial success in anyone investment project of large and often not predicted risk.

CONCLUSION

Thereby made evaluation of the variants for effective method of developing of the ore bodies with complicated morphological characteristic, is clearly, that in some of the cases is not expedience performing of full complex of developing and cutting works, typical of given extraction technology for ores with low value or limited amount of the reserves in that zone. The created object function will has minimum, when is not performed full complex of developing and cutting works, in not paying ore zones. The using of that complicated variant approach based on creating alternative graph can be owing to the present computer technologies in mining.

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Recommended for publication by Department of Underground mining , Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.11-13

A PROCEDURE FOR SELECTING AN OPTIMAL MINING TECHNOLOGY TAKING INTO ACCOUNT THE EXTRACTION AND QUALITY CHANGE RATIOS OF THE MINED-OUT ORE

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ABSTRACT
A general scheme of the necessary initial data on ore extraction and processing is presented: quantity of reserves and resources, extraction and quality change ratios and cost indices. A model has been developed based on compensation of the damages caused by ore losses and impoverishment in comparing two mining technologies. When generating more than two variants, a set of contrasted pairs is formed thus enabling us to determine the variant with the best technical and economic indices. A software package TWOTECH has been designed for solving problems related to both operative planning and a preliminary survey of technologies applicable under real natural conditions.

INTRODUCTION

Modern approaches to selecting an optimal mining technology require a technical and economic assessment because of the high degree of variability of the natural and mining factors.

Some of these factors enter the categories of known classifications. Others (mainly economic factors) allow for broad interpretation depending on the particular aims of the procedure. In the long run, the technical and economic assessment of the mining methods should proceed through several stages, one of these being the joint consideration of ore extraction and ore processing at the dressing plant, i.e. taking into account the end product (concentrate).

This stage is the subject of the present paper and the study is based on the principle of the technical and economic comparison of two mining technologies.

FORMULATION OF THE TASK

The selection of the mining method is based on criteria based on the ore extraction indices in applying a given mining technology. This approach is discussed in the works of M.I. Agoshkov. In this particular case we start with the natural and recoverable value of 1 t of balance reserves and 1 t of crude ore, respectively. The balance reserves **Z**$_{bal}$ and the metal content in the ore and wallrock, **A**$_m$ and **A**$_c$ respectively, are basic values. The initial data required for the computational procedure are classified in two separate groups:

First group of indices whose values depend on the type of mining technology:

- losses, a_e, %;
- contamination, b, %;
- prime cost of 1 t of extracted crude ore ex stope, C_{ex}, USD/t.

Second group of indices independent of the type of mining technology:

- metal price at the London Metal Exchange, P_{lme}, USD/t;
- geological exploration costs ensuring growth of reserves R_{ge}, USD/t; the geological exploration costs can be determined according to the expression $R_{gl} = 0{,}01 A_м \delta_2 P_{lme}$, where δ_i is the coefficient characterizing the relative share of the geological exploration works in the metal price. According to data from real practice δ_i varies within (0,01 - 0,05);
- costs for processing 1 t of crude ore at the dressing plant C_{pr}, USD/t;
- transportation costs per 1 t of crude ore to the dressing plant, C_{tr}, USD/t;
- recovery ratio in ore processing at the dressing plant α_0.

On the basis of the initial data we calculate the following indices characterizing the mining technology.

1. Metal content in the actually extracted crude ore A_a,

$$A_a = A_m - b(A_m - A_c), \quad \% \tag{1}$$

2. Ore extraction ratio during mining operations η,

$$\eta = 1 - \frac{a_e}{100} \tag{2}$$

3. Quality ratio of extracted ore, ρ,

$$\rho = 1 - \frac{b}{100} \qquad (3)$$

4. Quantity ratio of actually extracted ore, k_{qty},

$$k_{gty} = \frac{\eta}{\rho}\ ;\ k_{gty} > 1\ \ or\ k_{gty} < 1 \qquad (4)$$

5. Quantity of actually extracted crude ore Z_a,

$$Z_a = Z_{bal} k_{gty} = \frac{Z_{bal}(100 - a_e)}{100 - b}, \quad t \qquad (5)$$

6. Natural value of 1 t of balance reserves V_{bal},

$$V_{bal} = 0.01 A_m \delta_1 P_{lme}, \quad USD/t \qquad (6)$$

Where δ_1 is The coefficient characterizing the relative share of the metal price up to the metallurgical treatment stage; it is determined on the basis of the concentrate price and metal content in it.

7. Total prime cost of extraction, transportation and processing of 1 t of crude ore, C_{cr},

$$C_{cr} = C_{ex} + C_{tr} + C_{pr}, \quad USD/t \qquad (7)$$

The calculations from point 1 to point 7 are a prerequisite for the essential part of the technical and economic comparison. It is reduced to comparing two mining technologies and is carried out in the following sequence. First, the respective indexing for each technology is introduced ***i = 1, 2***. The mining technology that permits lower exploitation losses, assumes index ***i = 1,*** and the alternative one - ***i = 2***. Thus, according to (2), we have the condition $\eta_1 > \eta_2$. The analysis is based on the quantity of lost balance reserves Z_{lb}, which is determined by the expression:

$$Z_{lb} = Z_{bal}(\eta_1 - \eta_2), \quad t \qquad (8)$$

Taking into account the assumed impoverishment b_l, ρ_i respectively, according to (3) we determine the total prime cost of 1 t of balance reserves C_{bal_i} by the following formula:

$$C_{bal_i} = \frac{C_{cr_i}}{\rho_i};\ (i = 1, 2), \quad USD/t \qquad (9)$$

Then the value of 1 t of lost balance reserves V_{l_i}, taking into account the ore recovery at the dressing plant ε_0 will be:

$$V_{l_i} = \varepsilon_0 V_{bal} - C_{bal_i} + R_{ge};\ (i = 1, 2), \quad USD/t \qquad (10)$$

In expression (10) we assume that the lost balance reserves have not been turned into production costs. In this particular case we should emphasize it since such costs are likely to exist (e.g. for extracted but undelivered ore).

The damages caused by ore losses are compared with the compensations related to the lower costs for ore extraction, transportation, processing and recovery. The compensations related to the costs for extraction of 1 t of balance reserves K_{ex} will be:

$$K_{ex} = \frac{C_{ex_1}}{\rho_1} - \frac{C_{ex_2}}{\rho_2}, \quad USD/t \qquad (11)$$

The compensations related to the costs for transportation of 1 t of balance reserves to the dressing plant K_{tr} will be:

$$K_{tr} = \frac{C_{tr_1}}{\rho_1} - \frac{C_{tr_2}}{\rho_2}, \quad USD/t \qquad (12)$$

The compensations related to the costs for processing 1 t of balance reserves at the dressing plant K_{pr} will be:

$$K_{pr} = \frac{C_{pr_1}}{\rho_1} - \frac{C_{pr_2}}{\rho_2}, \quad USD/t \qquad (13)$$

The compensations related to the change in K_{re} will be:

$$K_{re} = V_{bal} - (\varepsilon_{01} - \varepsilon_{02}), \quad USD/t \qquad (14)$$

Therefore, the total compensation K_{sum} will be:

$$K_{sum} = \sum_{j=1}^{4} K_j, \quad USD/t \qquad (15)$$

where ***j*** = 1, 2, 3, 4 corresponds to the indices ***(ex), (tr), (pr), (re).***

In the expressions from (11) to (15) K_{ex}, K_{tr}, K_{pr}, K_{re}, K_{sum} we can have positive or negative numbers. If $K_{sum} < 0$, then the procedure should be stopped since there is actually no compensation for the damages caused by greater losses in applying the second technology (***i*** = 2). If $K_{sum} > 0$, then we have compensation of the damages caused by greater losses and the analytical procedure should continue.

The total compensation for damages caused by losses, in relation to 1 t of lost balance reserves K_{lb} will be:

$$K_{lb} = \frac{K_{sum} Z_{bal} \eta_2}{Z_{bal}(\eta_1 - \eta_2)} = \frac{K_{sum}\eta_2}{\eta_1 - \eta_2}, \quad USD/t \qquad (16)$$

Then the economic consequences of the losses caused in relation to 1 t of lost balance reserves ***S*** will be:

$$S = V_{l_1} - \frac{K_{sum}\eta_2}{\eta_1 - \eta_2}, \quad USD/t \qquad (17)$$

In expression (17) ***S*** can be a positive or negative number. If ***S*** > 0, then the compensations for the damages caused by losses are lower than the value of 1 t of lost balance reserves

V_{I_1} and preference should be given to the technology with an index ***i*** = 1. If ***S*** < 0, then the compensations for the damages caused by losses are higher that the value of 1 t of lost balance reserves $V_{заг_1}$. The technology with an index ***i*** = 2 should be accepted as a better one.

The determination of ***S*** in formula (17) is the essence of the technical and economic assessment in comparing the two mining technologies. The end results of the analysis made should be referred to the balance reserves $\boldsymbol{Z_{bal}}$ since they are the basic parameter. Then the economic consequences of the damages caused by losses, with respect to 1 t of balance reserves ***ΔS*** will be:

$$\Delta S = \frac{SZ_{bal}(\eta_1 - \eta_2)}{Z_{bal}} = S(\eta_1 - \eta_2)\,, \quad \text{USD/t} \qquad (18)$$

Obviously, the algebraic signs of ***S*** and ***ΔS*** according to (17) and (18) coincide. For ***ΔS*** < 0 we determine not only qualitatively but also quantitatively the additional profit with respect to 1 t of balance reserves if the technology with an index ***i*** = 2 is applied. In this form the procedure can be successfully applied in investigating the possibility to apply different variants of mining technology as well as the technical and economic feasibility of replacing the mining method.

COMPUTER IMPLEMENTATION

The practical implementation of the proposed procedure for a technical and economic assessment of a mining technology involves the development of an algorithm and a computer software package TWOTECH. The software program is written in the algorithmic language FORTRAN. It has been used for solving particular mining-engineering problems related to the justification for introducing efficient mining technologies for working polymetallic ores of non-ferrous metals.

Apart from the technical and economic comparison of two mining technologies with indices ***i*** =1, 2, the program TWOTECH permits the determination of the marginal value of the metal content in the ore $\boldsymbol{A_{mar}}$, for which the two technologies will be equally efficient, i.e. ***S*** = 0 according to expression (17). By means of TWOTECH it is possible to determine the ultimate metal price at the London Metal Exchange $\boldsymbol{P_{lme(min)}}$ up to which the mining technology will be profitable. It also enables us to calculate the losses suffered for a certain period of time with the view of the possibilities to compensate them when mining other mine districts. In case more than two technologies are applicable or a set of a given number of variants is considered, we form the matrix $\|\Delta S\|_{mn}$, where ***m*** and ***n*** are the columns and rows of the matrix. The columns correspond to the number of mining methods and ***n*** corresponds to the features by which the individual variants are formed.

The program TWOTECH determines the maximum value element of the matrix $\|\Delta S\|_{mn}$ that is used as a basis for further analyses of the applicable mining methods.

CONCLUSION

A procedure has been developed for a technical and economic assessment in comparing two mining technologies or their variants, which is based on the compensations for damages caused by losses. It depends directly on natural and economic indices such as metal prices, costs for extraction, transportation and processing, level of losses and impoverishment, recovery in ore treatment, intensity of mining operations, etc. The approach proposed permits the evaluation not only of the applicability of the mining technology but also the boundary values of the metal contents in the ore, ultimate metal exchange prices for which a given technology is efficient and effective.

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Recommended for publication by Department of Underground mining , Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.15-18

STUDYING THE METHOD OF MINING OF HIGHWALL MINING SYSTEMS (HMS)

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ABSTRACT
The so-called "Highwall Mining Systems" have been effectively applied for the last 10-15 years in coal mining practice in USA, Australia and the RSA. In general, those are systems of underground coal mining with mine entries directly communicating to the surface. However, they are almost unfamiliar under the above-mentioned name to a wide range of mining professionals and they have not been described in Bulgarian mining references yet. A comprehenive study of those methods of mining, technical and technological analysis allowed the presentation of the mining method and technology, technical and technological feasibility and requirements for effective application.

INTODUCTION

Methods of mining with mine entries, directly communicating to the surface were applied in the early stages of development of mining. Later, with the improve of technologies in mining, an extension of the field of application of opencast mining took place, mining operations went into the depth and those methods were forgotten.

Those methods were rediscovered 10–15 years ago. That was based on the extension of the application of opencast mining operations in wide areas and in depth, of modern devices for mechanization and automation, including means for positioning and control. One of the important preconditions for rediscovery of that method of mining was the coal mining practice in USA and Australia.

Mining operations are re-considered, when effective economic boundaries of the opencast mine are achieved and coal or other soft minerals are still available out of them. As a result opencast mining may be ceased, extended to new effective boundaries or transferred to underground mining.

The so-called "contour mining system" is applied in the mining practice of USA, Australia and the RSA. That means opencast development to certain final contours of the mine), and when the final contour is achieved, mining operations are ceased. Sometimes, systems and technologies for limited underground mining are applied. The underground mining is performed through mine entries directly communicating to the surface, and the main equipment is based on benches on the surface. That mining method is familiar in the mining science and practice of the USA, Australia and the RSA with the collective name of Highwall Mining. The system is based on an integrated mechanized technology, which is realized by a completed set of mining machines and devices.

Mutual interrelation and dependency of the system decisions (parameters of the mining method), parameters and feasibility of the integrated mining equipment, technology and organization of work in the recent practice is so strong that the common title of system and technology – Highwall Mining System (HMS) is quite popular.

MOST IMPORTANT FEATURES OF THE METHOD OF MINING

In general, the HMS is based on a working area (bench), where the coal seam opens, above the lower edge of the site (bench) and the stable bord, and the height is great (highwall). That site may be bench or ditch with a certain width and length.

The most important feature of the system consists in driving of a grid of mine entries with a direct connection to the bench, to the surface. The mine entries are blind, with different shape and dimensions of the cross-section. They are different orientation to the spread and dip of the seam. They are driven in a certain specific order and consecution, without any support during the driving and during the leaving. Technological stability of mine entry and roof control in the mined-out room are guaranteed by narrow bench ribs between entries. Main parameters of the mining method are: type and direction of mine entry; type and dimensions of cross-section, length of the entry; width of the narrow rib and dimensions of the operating area in length and width. Without reading the opportunities for creating different variants of the system, with respect its parameters, the qualitative variants of that method of mining are based on the type and direction of the blind entry. For example – adits along the line of spreading at different levels, adits at one and the same level, mining inclined shafts along the dip, along the rise or in a diagonal. In those variants, some of the parameters of the method of mining are determined by the specific natural conditions (thickness, inclination), others on the opportunities of mining equipment and a third one on the condition of opencast mining works during the final period.

MINING TECHNOLOGY

The mining technology of that method of mining consists of a mechanized mining complex. The complex equipment provides the drive of entries without the constant presence of people at the stope.

In general, the mining equipment consists of self-advancing controlled platform, cutter head, push-beam transfer system and a number of service devices, which generally form the mining complex.

The technological scheme of operation of the complex is as follows: the main platform is positioned on the operating site and is directed against the spot of the entrance of future mine entry; advance of the cutter head starts, which inserts into the massif and forms the initial cross-section of the entry; further driving of the entry is performed by pushing or dragging of the cyclically elongated pusher by the push-beam mechanism. When final length of the entry is achieved, the operation of the cutter head is ceased and it is pushed backword by shortening of the push-beam transfer mechanism. After getting out of the cutter head, the controlled platform, on its own drive, moves to a new position.

Parameters of the mining system, technology of work are based on specific mining conditions and final technical-economical results of mining are achieved for a specific scheme of organization.

A INVESTGATION ON HMS SYSTEMS AND TECHNOLOGIES

An investigation on web sites and publications related to HMS reveals that almost all of the information aims an advertisement. There are very few publications (Fiscor, St., Coal Age, 2002) showing the history, analysis, assessment and results of a specific practical application.

An investigation aiming a study of the opportunities for practical application of HMS in Bulgarian coal mines was carried out. (Dermendjiev, K., A, Slavov, 2003). It comprised the variants: Superior Highwall Miner (SHM); Nex Gen HMS (NHMS); Addcar HMS (AHMS); Racho HMS (RHMS) and Racho variants Racho Narrow Bench Mining System (RNBMS); Racho Skip Car System (RSCS) and Racho Conveyor System (RCS).

Results of the investigation showed that HMS have been known in the USA for long and that many variants and performances are familiar. Variants of HMS differ, depending on the completed set and dimensions of the controlled platform, type and the principle of operation of the cutter head, the push-beam transfer system, the auxiliary equipment of the powered support and control, loading and discharge mechanisms. Those differences bring to differences in the type, configuration and parameters of the entries, the order and consecution of mining, extraction and losses.

The analysis of technical and technological opportunities of the above HMS variants and mining conditions in some of the opencast coal mines in Bulgaria showed that the system Superior Highwall Miner may be applied. The selection of that system as a basis for assessing its feasibility and requirements for effective use are to a certain extent due to the more detailed information, which is available about it (Fiscor St., 2002).

The complex mining equipment of the SHM system, fig. 1 consists of a self-moving crawler platform. The four crawlers have their own driving systems and autonomous control. The control centre, the cable idler, the hydraulic pushing-beam mechanism, the transporting system for reloading and discharge of the mining mass are mounted on the platform..

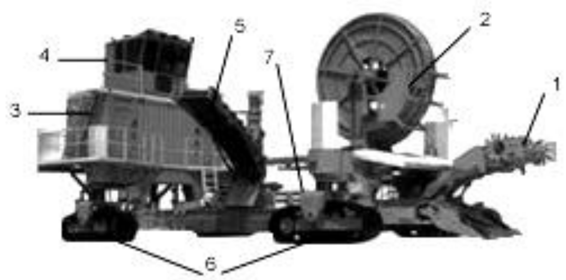

Figure 1. General view of a mining complex SHM-20
1. Cutter head; 2. Cable idler; 3. Power head; 4. Machine control centre; 5. Rear-discharge units; 6. Crawlers; 7. Push-beam transfer mechanism

The most important element of the complex is the cutter head, fig. 2. It represents a modified miner for narrow room stopes, the so-called "continuous miner" with a self-drive. It has a separate drive and individual hydraulic drive systems. The connection of the cutter head to the platform in the working move and maneuvering and transportation of mine mining mass is performed through engaging and disengaging two-auger panels of 6 m length. Two motors of 300kW each, positioned on the platform, drive the auger transportation system. That system may transport coal to a distance of 300 m.

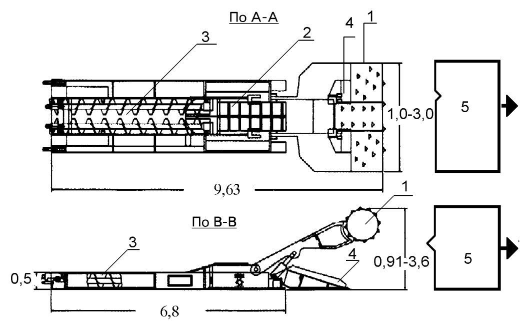

Figure 2. Cutter head: 1 – working instrument (cutter bit); 2 – short transporter; 3 –auger panel, 4 – loading device; 5 – mining room

The system is designed for mining of horizontal and slightly sloping seams of a slope not more than 12^0 and thickness from 0,91 to 3,6 m.

The simplified scheme of the method of mining, positioning of the mine equipment in the layout and in the cross—section is shown in fig. 3.

A detailed study and investigation is performed with the aim of assessing feasibility of the system and SHM mining technology andconditions for effective use.

The main parameters of the method of mining, parameters of main processes and final technical and economical characteristics of method o mining and mining technology are determined on the basis of variation of the values of a wide range of parameters, characterizing the geological, mining and technical conditions of operation.

The following parameters: strength of uni-axial pressure of the massif and the coal seam, shearing strength of the seam, depth of occurrence of the seam beyond the contour of the high bord, seam thickness are a subject of variation.

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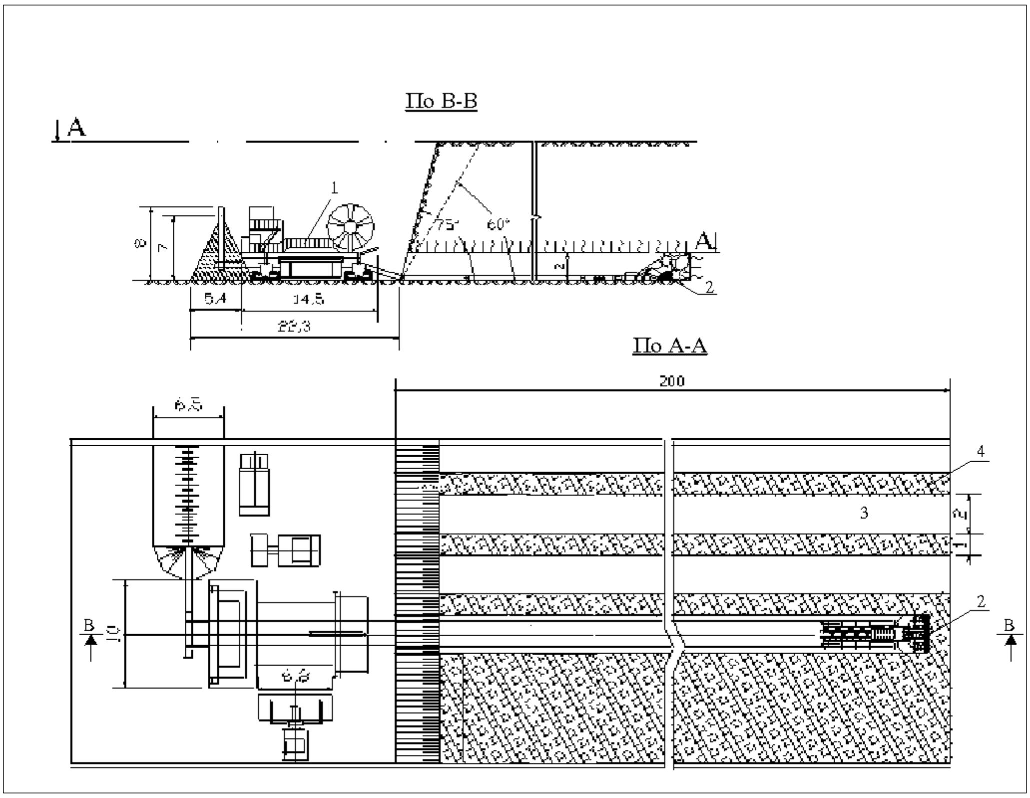

Figure 3. Scheme of system and technology Rectangular-shaped entries (rooms); 4 – narrow bench (rib)

The following are determined with the aim of analysis and synthesis of effective decisions for the method of mining and mining technology: ratio of width of the mining entry (room) and the narrow rib for different strengths of uni-axial pressure and depth of mining; working speed of advancing of the cutter head depending on the shearing strength, duration of working cycle. Those values allow the synthesis of a multitude of variants and feasibility characteristics. The rate of extraction of reserves and productivity of the complex are applied for basic parameters for evaluating and selection of decisions.

Approximate values of some of the economical parameters are determined in the final stage of the investigation. Those parameters are used for final decision-making. Various organizational schemes of operation are used to determine those values, shift and day-time loading of the mining complex, price of coal and equipment, effect of financial and conditions of work.

Results of the investigation showed that effective application of HMS and the SHM system, in particular, require a comparatively high strength and stability of the rock mass. When reserves of depth not more than 200 m are mined and stability of entries is guaranteed and gob area has the ratio of entry (room) width – rib – 1:1, the strength of uni-axial pressure of the coal seam should not be less than 20 MPa.

The economic efficiency of the system may be achieved, when the coefficient of extraction is not less than 50%; the average shift loading of the complex is more than 1000 t/d, and quantity of mined coal is not less than 1,5 million tones, when mining by a SHM is applied.

CONCLUSION

The Highwall Mining Systems belongs to the methods of mining with entries directly communicating to the surface. In the mining practice they are applied as different variants, under the title of mining technology, performed with different machines, combined in a system (complex). HSM and "SHM", in particular, is a very effective system and technology for underground coal mining. It allows a gradual transition of opencast toward underground mining and more complete extraction of the natural resource through extension of boundaries of extractable reserves in the area and in depth. The application of those technologies is limited by the strength and strain properties of the rock mass and due to the strata control of mine entries and gob area. For that reason effective application of those systems and technologies in some of the opencast mines in Bulgaria will not be possible.

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Recommended for publication by Department of Underground mining , Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.19-22

METHODS OF MINING AND EXTRACTION TECHNOLOGICAL SCHEMES WITH ROADHEADERS

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ABSTRACT
Transfer of extraction works from seam "B" to seam "A" of central section of coal mine "Bela Voda" requires change of method of mining and coal winning technological scheme. Taking into account the big difference between mining and geologic conditions in the above- mentioned seams and gathered experience and qualification of usage "4 flV" road heading machine in coal extraction operations, in the report are presented for discussions and recommended for application rational method of mining and coal extraction technology. Special attention is paid to working scheme of the roadheader when operating in different extraction workings in connection with support erection and roof control.

Application of hand or semi hand technologic schemes of coal extraction is typical when mining and geologic conditions are not favorable. The main problems upraise as a result of complicated hypsometry of coal seams, weak coal and host rocks, tectonic disturbances, high water and gas inflow, e. t. c.

When such conditions are at present, coal getting districts are with small dimensions and highly mechanized technologies are ineffective. For this conditions technologies must by highly productive too and with enough good standards of safety and appropriate level of mechanization of basic processes.

On this base before (Стоянчев, Анастасов, 1999) and after coal mine "Bela Voda" privatization, short wall method of mining with usage of "4 ПУ" road heading machine for development and extraction operations was applied in district "Central", seam "B".

In accordance with the Total Design for Coal Mine "Bela Voda" 2002, it is foreseen during the next two years extraction works to be transferred to seam "A" of the district with the same name.

Seam "A" has small occurrence and the productive section is 300 m long to the strike and 100 m to the depth. Its inclination is about 8°, it form is irregular. The thickness of the seam varies between 1,7 and 1,8 m and is intersected by thin clay bands. The immediate and nether roof is presented by clays and marls with enough good stability. Immediate bottom of the seam is presented by weak clays and their thickness is about 0, 8 m, and they cover thin not commercial coal seam with 0,35- 0,4 m. thickness.

Analysis of gathered experience with the appropriated method of mining and coal extraction technology proved, that "4 ny road-heading machine is worthy for the purpose. The machine is with small outer dimensions, good maneuverability and productivity, low energy consumption, requires low maintenance cost (see table 1). The previous estimations showed, that to begin work in seam, must be decided the problems as follows: to reduce length of development workings, coal losses for pillars; to reduce support costs and to arrange steel support for multiple usage; to find an appropriate decision for roof control with enough good synchronization between coal winning and strata control operations.

At extraction works in seam "B" blind breast workings are with defined form and dimensions. In this conditions the only limiting factor for roadheder is seam thickness. Such limitation could be avoided by some technological actions made in the mine pit.

Having in mind the thickness of seam "A" and technical parameters of coal winning machine (table 1) it is evident that it can take coal of all seam thickness in different kind of mine workings.

Table 1.

NO	Main technical parameters of "4 nV" roadheader	Dimensions	Values
1	Width at the loading platform	m	2,350
2	Maximal height	m	1,500
3	Reloading height	m	1,300
4	Length	m	5,900
5	Speed of movement	m/min	2,24
6	Parameters of mine workings in which machine can operate		
	Width	m	2,6-3,3
	Height	m	1,5-2,85
	Inclination	oO	±8
7	Total power installed	kW	63

So discussed problems could be decided by application of short wall method of mining with blind or open working faces. But at those methods of mining, working places directly contact with caved zones and the main requirement is to support minimal roof area. This area must correspond with parameters of coal getting machine, area for its maintenance and support structures placement. Parameters of this area at high degree define technological operations in the working, kind and support parameters, way of strata control, dynamics of main face processes and safe and economically effective coal extraction.

The passes of "4 ПУ" road heading machine and dimensions of working zone are graphically presented as plan and sections on fig.1. Dimensions are in centimeters and they correspond to technical parameters of the roadheader. For safe and free operation of workers, working zone with 0,7 m width is foreseen. Width of support zone is 0,2 m. On the figure is seen, that minimal width of the roadheader pass is 2,6 m and of the supported zone is 6,5 m.

Minimal working area for machine normal operation is 17 m^2

Within the boundaries of one pass (for a blind working) or split (open short wall face) the main processes are coal extraction, support of worked out area and roof control. These processes can flow consecutively or in parallel only in strictly defined conditions and its fulfillment in time and place is required by the technological scheme. For the coal mine "Bela Voda" the both, consecutive or in parallel technological flow sheets could be applied. But having in mind geometric conditions and district development, some limitations are defined.

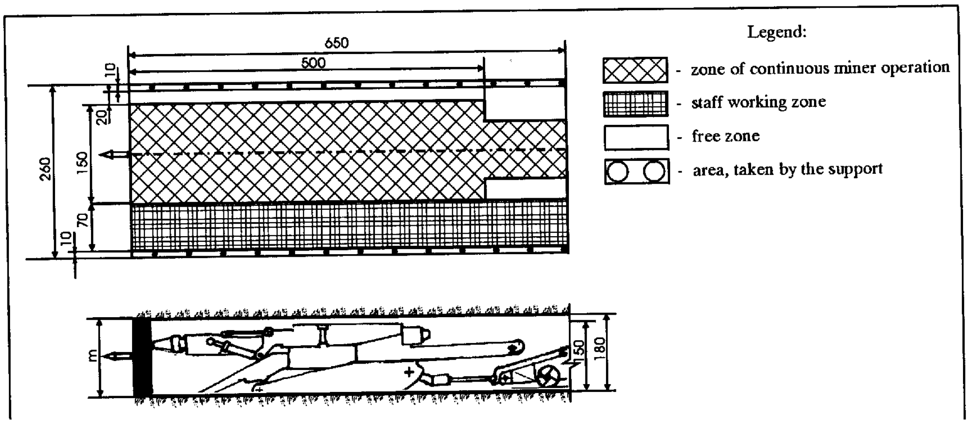

Figure 1. Determination of gabarits of roadheader working zone

Technological schemes discussed are for one sided machine operation without any turns. The mine working for coal transportation is one. Retreat order of district exploitation is appropriate in two variants: by blind workings and open ones, ventilated by main ventilation.

On the base of appropriate kind of extraction working, getting machine possibilities and above mentioned limitations on fig. 2 and 3 are presented discussed for application technological schemes. On them are pointed directions of extraction works, coal getting machine maneuvers, way of roof control, coal transportation, ventilation of the district and extraction workings.

On fig.2 are presented consecutive technological schemes of extraction works: a) with separate passes; b) with splits of the wall. On fig. 3 are presented technological schemes of extraction in parallel in three base variants: a) extraction and simultaneous caving of neighboring splits; b) extraction and simultaneous caving of neighboring splits behind a safety strip; c) extraction and simultaneous caving of neighboring splits behind a limited safety strip.

The consecutive technological scheme possesses some advantages: independence of extraction and caving operations. Supported area is small. But it has serious disadvantages too- complicated ventilation and coal transportation scheme, mounting and dismounting of supports and transportation means. Some of the work time is not productive and maneuvers of coal getting machine must be made.

Technological scheme with passes is with trough ventilation, but much bigger area must be supported for longer time, some greater is unproductive time in the extraction mine working.

Technological schemes in parallel assure reduction of unproductive period of time by overlapping extraction works and roof control. Working face is not blind and this requires support sets with three legs and two ceiling girders.

Three schemes discussed differ one another generally by the magnitude of supported area and degree of overlapping of extraction and strata control times.

From the three schemes the most advantageous is scheme "c", presented on fig. 3.

In accordance with the parameters of technological schemes discussed supporting plans and roof control designs were elaborated. Main variants were grouped in two groups. Support plans for support sets with one steel ceiling girder and two hydraulic legs and sets with two steel ceiling girders and three hydraulic legs. This support plans can be seen on fig. 4 and fig 5

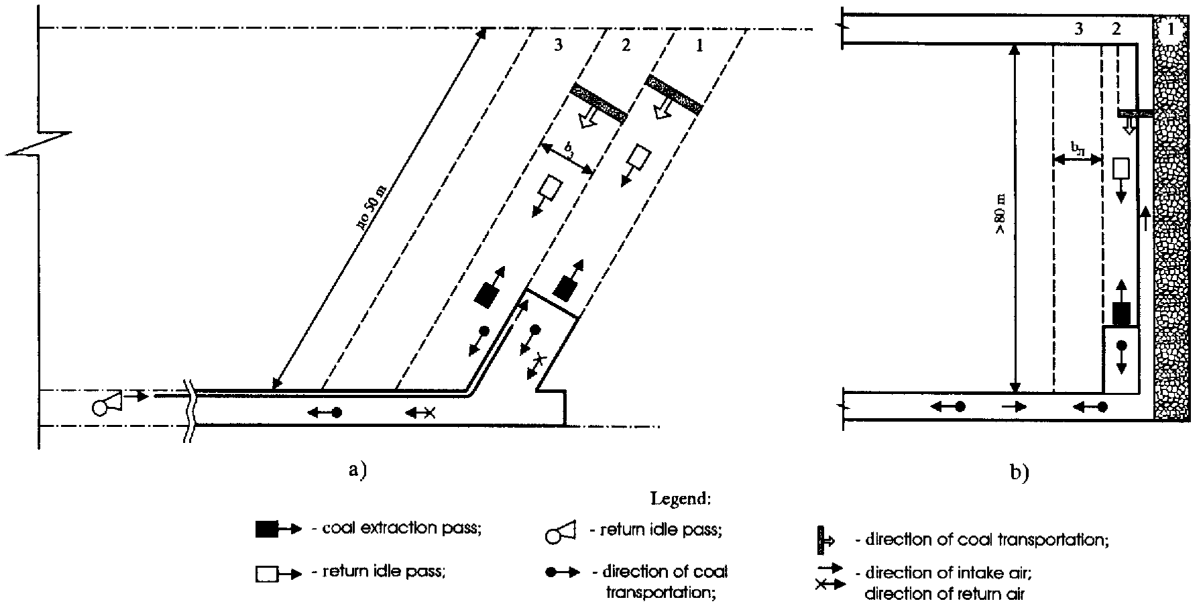

Figure 2. Consecutive technological extraction schemes: a) with separate passes; b) with splits of the wall

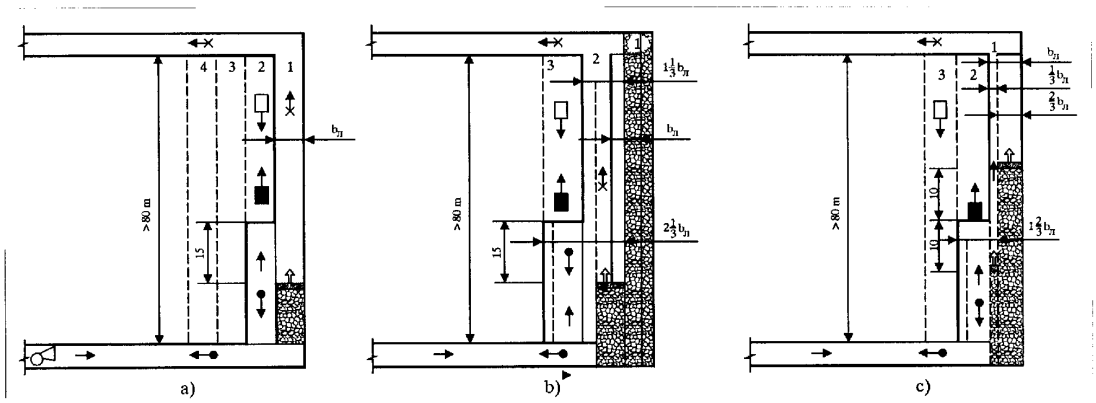

Figure 3. Technological schemes for extraction in parallel: a) strip to strip b) extraction and caving behind a safety strip; C) extraction and caving behind a safety strip with limited length.

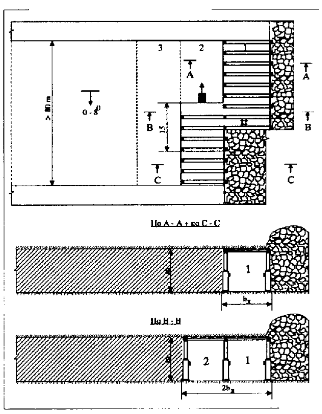

Figure 4

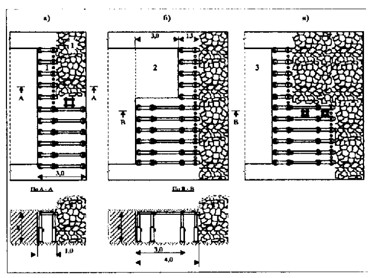

Figure 5

Analysis of all technological schemes and support plans show, that all of them are applicable in seam A extraction and could be tried in underground condition in coal mine "Bela Voda". The choice and its introduction in practice of the best of them will be made after analysis of results obtained.

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Recommended for publication by Department of Underground mining, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.23-25

ORE MINING RESTORATION AT DIMOV DOL MINE

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ABSTRACT
Dimov Dol Mine has been in the process of liquidation for 30 months. Mining has been discontinued and personnel dismissed. No maintenance has been undertaken of work places, machines or equipment, exclusive of drainage.
The paper presents a technology for repair, restoration, organizational and technical activities aiming at creation of a profitable mining company in the condition of transition to market economy.

INTRODUCTION

Dimov Dol Mine is one of the three mines of Gorubso-Rudozem Company in the city of Rudozem. By virtue of order of the Minister for Finance № 148 / 02.04.99, the mine was declared in liquidation, and in May all activity was discontinued and the personnel dismissed. For more than 30 months only drainage has been maintained in the mine. No funds are allocated or actions undertaken for maintenance and preservation of mining equipment. Mining and geological conditions and mine micro-climate adversely affect mine roads, machines and electrical equipment. Faces are falling, electric motors and cables are wetted, all mechanical equipment is corroding. Part of the tools and equipment in mining blocks is buried by self-caving rock.

Under these conditions, on 08.11.2001 the mine was privatized by Rudmetal Company in Rudozem. They set the following objectives:

- To complete restoration works and the mine to resume ore extraction as fast and as efficiently as possible;
- To undertake actions ensuring mine viability, i.e. the mine to become profitable.

RESTORATION TECHNOLOGY

Immediately after execution of privatization contract, restoration work commenced in order to make operable the major machines and equipment as soon as possible and to prepare faces for ore mining.

The first issue to be resolved is the type of organization that has to be established so that, after completion of certain activity, safe condition swill be ensured for commencing several new ones. On the basis of this organization, the hiring schedule was developed.

The work started with restoring of shaft winding. Inspection, repair and adjustment of all major components and systems were carried out, namely: lacing of driving drum, brake system, pneumatic pack, electric drive, G-D group, tiristor transducer, and associated lock and guard systems, mechanized car exchange, shaft signaling. At Golyam Palas – North shaft, which had been out of operation for 6 months, main ropes were replaced and 400 m of steel guides.

After ensuring normal access to the mining section, road repair started in order to access the blocks themselves. Thanks to the good organization, in two months over 2500 m of horizontal and 300 m vertical roads were re-supported Restoration in blocks proceeded in stages depending on the time needed to access them, the amount of work and available materials and workforce. This is why ore mining did not commence simultaneously on all faces but in stages depending on completion of restoration work and ensuring of safe working conditions. Restoration works started with 72 workers and finished with 147.

Simultaneously, a time schedule was developed for mining equipment repair and commissioning of major facilities – compressors, capital ventilation facility (CVF), Machines that could be taken out of the mine, were repaired in mechanical and electrical shops (MS and ES). Mine battery locomotives, pulling stations, trolley net, power and lighting installations were repaired on site after providing access thereto and ensuring safe conditions.
The time schedule of repair works is shown in Table 1.

Table 1. Time schedule of repair and restoration works

Activity	I^{-st} month	II^{-nd} month	III^{-rd} month	IV^{-th} month
Shaft winding				
Horizontal roads				
Vertical workings				
Mining blocks				
Compressors				
Ventilation facility				
Mining equipment				
Ancillary shops and units				

Along with restoration works in the mine, all ancillary shops and service units were also commissioned in compliance with the new conditions for normal mine activity. Timber support shop was put into operation and the workers' canteen. The communal building was reconstructed in view of expected number of personnel. A model heating system was built, shower and laundry rooms.

This organization of work allowed the mine to resume operation of all mining blocks in four months.

HOW TO MAKE THE LOSING MINE PROFITABLE?

During the last 10 years, before declaring liquidation, the mine had been incurring losses. Efficiency increase is additionally impaired by the following circumstances:

- There in no ore processing factory at the mine and a much higher price has to be paid for processing;
- Mining conditions are extremely difficult and require employing of high labor consuming and low productivity mining system, namely "layer caving";
- Drainage is difficult because water collection area is very large and mining in concentrated in a small section.

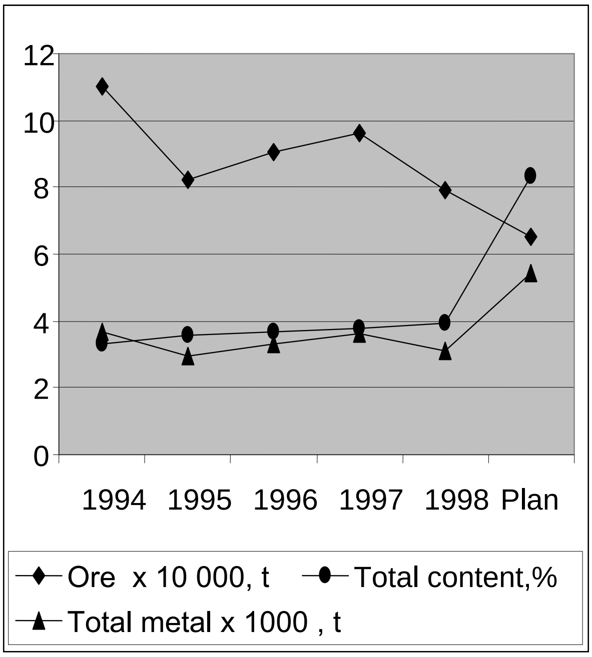
With these conditions, the planned measures have to radical. The paper focuses on the most important of those.

First: Complete change of extracted ore quality, mainly in terms of metal and moisture content. The main production (approximately 75) is meta-somatic ore with much higher metal content supplemented by one block in the thin ore section. Cumulative (lead + zinc) content exceeds 8 %, which is approximately 2.5 time higher than production in the last five years, prior to liquidation. This is shown in table 2.

Table 2. Production of lead-zinc ore

Year	Ore, thou t	Total content	Total metal
1994	110.2	3.33	3672
1995	82.3	3.56	2930
1996	90.5	3.65	3303
1997	96.2	3.76	3614
1998	79.2	3.93	3113
Plan	65.0	8.35	5427

Figure 1. Production of lead-zinc ore



Ore moisture content is also significantly less. Earlier ore was mine with moisture content from 10 to 14 %, now it is 4-5 %. This facilitates transport and reduces costs.

Second: Personnel optimization. Workforce is reduced by some 30 %, working on the lower limit. For the purpose of working time optimization, all ancillary workers were additionally qualified and assigned with minimum two professions.

Table 3. Production of t metal per man per year

Year	Total metal	Workforce	t metal/man
1994	3672	284	12.93
1995	2930	283	10.35
1996	3303	284	11.63
1997	3614	296	12.21
1998	3113	251	12.40
Plan	5427	171	31.74

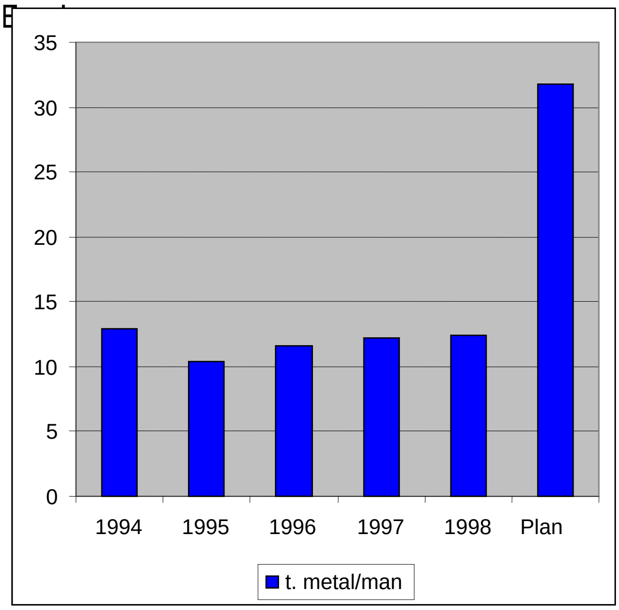


Figure 2. Production of t metal per man per year

As drainage costs are fixed, a continuous mining regime was introduced. Sunday is the only day off (first and second shifts) and then maintenance is carried out.

Third: New payment system. Salary is based on saleable production. Most volumetric indices were abandoned. The main index is total metal production.

The effect of these measures is seen in table 3 and on fig. 3. Earlier, 10 to 12 t metal was produced per man per year, not these are 31, i.e. nearly three times more.

Fourth: New technical solutions in all fields aiming at significant material cost reduction:

- Drainage. Reconstruction of old and useless mine workings into water collectors so that pumping could only proceed during night shifts when electricity price is the lowest;
- Compressors. Optimization of compressed air supply by eliminating one compressor facility and all excessive pipelines
- Ventilation. Efficiency increase of the main ventilation unit. Compaction of ventilation routes. Streamlining of ventilation flow by means of ventilation doors and barriers;
- Power consumption. Measures are undertaken ensuring reduction of both total power consumption and power redistribution in different zone in order to lower the average cost. Operation of big consumers is restricted in peak hours.

In addition to the above said, measure for increase of labor productivity are undertaken, for optimization of mining system, for reduction of all material costs, for transportation arrangement improvement, etc.

CONCLUSION

In the last ten years economic reforms are under way in our country for transition to market economy. Under these conditions, turning a losing mine into a profitable one is an extremely difficult task. On the one hand, economic conditions in the country are constantly changing, on the other hand – world economic trends have direct impact thereon. For example, non-ferrous metal price fall on London Metal Exchange has a direct impact on mining efficiency. Notwithstanding that, experience shows that where resources are put to maximum utilization and new solutions are employed, the objective can be reached.

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Recommended for publication by Department of Underground mining, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.27-29

A POSSIBLE ROUT TO DRIVING AN AIR-CONDITIONING RING GALLERY INTO THE TUMULUS OF THE FRESCOED THRACIAN TOMB IN THE VILLAGE OF ALEXANDROVO, HASKOVO REGION

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SUMMARY
The need to develop an underground mining technology for an access into the Thracian tombs comes as an alternative to the open exploration method, which leads to demolishing of the tumulus pile.
The realization of the design and engineering solutions for constructing a ring air-conditioning gallery inside the tumulus of the Thracian tomb in the village of Alexandrovo will be a continuation of the experience gained during the excavations of the Thracian temple in the village of Starosel.
This paper deals with the possibility to drive a gallery constructed entirely of prefabricated THN-16,5 and THN-29 metal units and supplied with box holes of impregnated timber.
The implementation of a mining technology aims at establishing control over the microclimate in the gallery and preserving the frescoes in the chambers of the Thracian tomb as well as at keeping the authentic mound pile.

INTRODUCTION

The development of a mining technology for access into the Thracian tumuli and their exploration is an important process. Being an alternative to the open exploration (implementing bulldozers and load-hauld-dumps) it will prevent the destruction of the mound pile and give more chances for air-conditioning and preservation of the original frescoes.

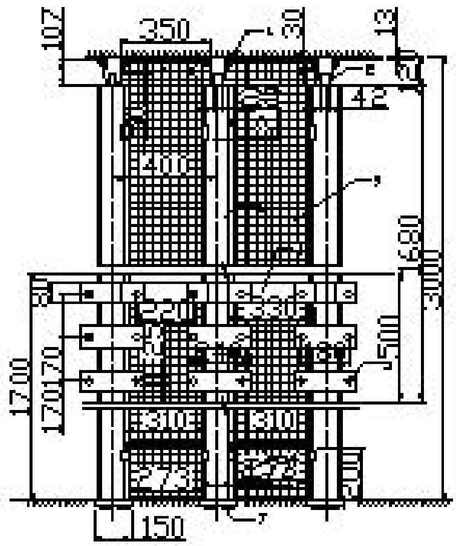
MINING CONSTRUCTIVE PART

The air-conditioning ring gallery is 41,90 m long. It has two straight sections - left and right (13,00 and 15,40 m long, respectively) and a circular 13,50 m long one.

The straight sections of the gallery are trapezium-shaped keeping the ceiling gradient of 10° for installing a hydrolyzing layer, which will drain the moisture from the mound section.

The dimensions of the air gate in its straight sections are the following (fig. 1 and fig. 2):

- height of the inside wall - 3,00 m;
- height of the outer wall - 2,68 m;
- width of the air gate at the ceiling - 1,41 m;
- width of the air gate at the floor - 1,44 m.



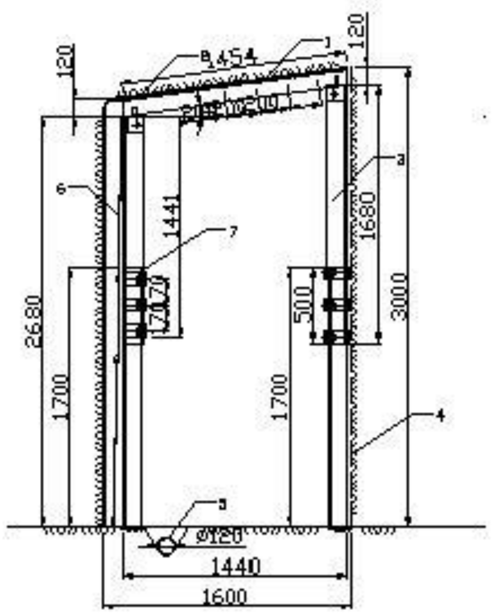
1. Hydroizolation layer
2. Steel bar
3. Fractional prop
4. Steel strut
5. Steel saddle
6. Steel П - like strut
7. Support foot pad
8. Slick sheet coating
9. Steel lacing

Figure 1. Longitudinal-section of the inner wall of the gallery in straight section of its inside wall.

At the beginning the earth from the mound pile is excavated with pick hammers. Then under the supervision of an archeologist the excavations continue manually with pickaxes regarding the cultural layers.

The loading of the excavated earth is manual. It is spaded into wheelbarrows and transported out of the site.

The section is lined with metal supports comprising 2 frictional steel THN-16,5 props. Each prop has an upper and lower part joined by 3 metal saddles. Their load capacity is 200 kN. The upper parts of the two props end with planks and cylindrical spikes 25 mm in diameter and 50 mm high. The steel bar, also made of THN-16.5, is fixed by the spike, which enters a round opening drilled into the bar.



1. Slick sheet coating d=8 mm
2. Hydrolization layer d=5 mm
3. Fractional prop
4. Drainage pipe
5. Steel lacing
6. Slick sheet coating d=5 mm
7. Steel saddle
8. Steel bar

Figure 2. Cross-section of the gallery in its straight section

The gallery ceiling is coated with steel sheets 8 mm thick. Seven headboards 20 cm wide and 50 cm long are put above each of the bars.

The inside wall is covered with steel lacing No 8. 10 cm is distance between the crosslinks of the lacing. Six 1,00 m x 0,40 m modules are used (fig. 3).

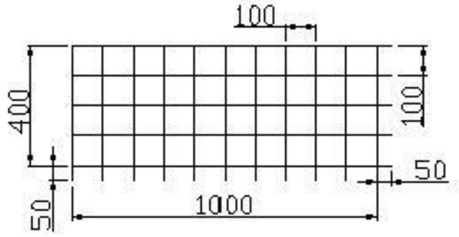


Figure 3. Steel lacing No 8 far the straight section

The outer wall is covered with steel sheets 5 mm thick. The six sheets are 0,4 m x 0,5 m at 3, 00 m height.

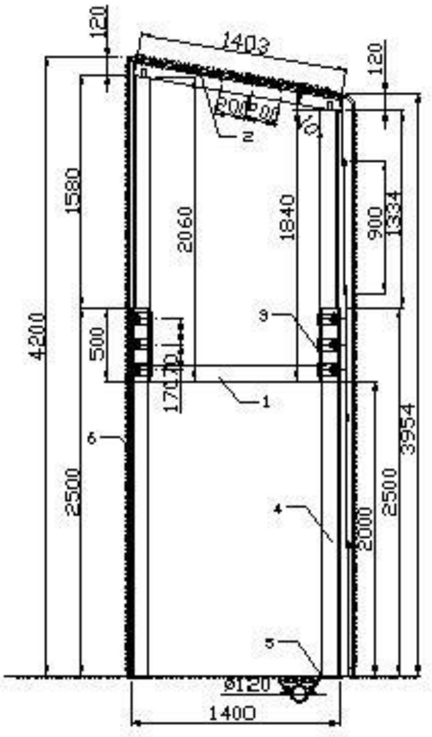
The steel bars are installed at every 0,4 m. They are stabilized by metal sleepers (2 for each frame) and metal hooks 24 mm in diameter and 0,32 m long.

The ceiling and the outer wall are coated with waterproof layer.

The 28,40 m long straight sections thus laid out need 71 frames placed in every 0,4 m.

THE SECTION OF THE GALLERY ROUNDING THE TOMB CHAMBER

The section of the gallery rounding the tomb chamber is 13,50 m long and is in the form of a semicircle with an inner radius of 2,90 m and outer radius of 4,30 m (fig. 4).



1. Lateral strut
2. Steel bar
3. Steel saddle
4. Prop
5. Support foot pad
6. Steel lacing

Figure 4. Cross-section of the support around the round tomb chamber

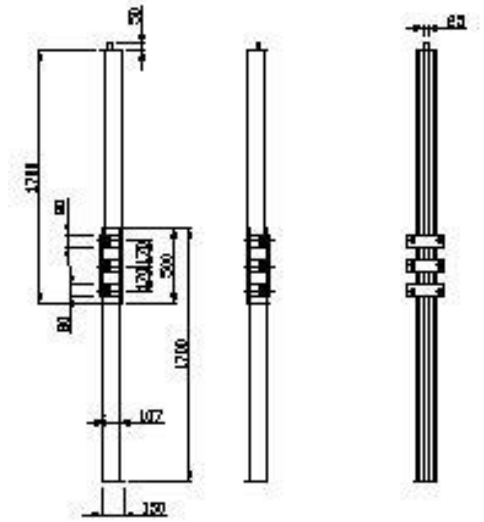
This configuration of the gate requires 25 THN-29 frames placed in every 0,6 m at the outer radius. The distance between the frames at the inner radius is 0,4 m.

The inside wall of the gate is 4,20 m high while the outer one is 3,95 m. The set gradient of the bar is 10°. But the two frictional props (fig. 5) and the bar a lateral strut is also foreseen. It is set at 2,0 m from the floor.

The driving of this section is combined with the driving of the box holes (fig 6), which ensure the pealing of the stone blockage of the mound. The box holes will be supplied PVC pipes for ventilation and conduct of the climate in the gate and in the frescoed chambers.

The box holes are closely lined with acacia timber elements, which were preliminary, impregnated.

The upper part of the box holes, the upper part of the frame and the outer wall of the section are isolated by a hydrolising layer.



Characteristics:
1. Working resistance - 200 kN
2, Maximum height - 2900 mm
3. Minimum height - 1800 mm
4. Weight-98 kg

Figure 5. Fractional prop

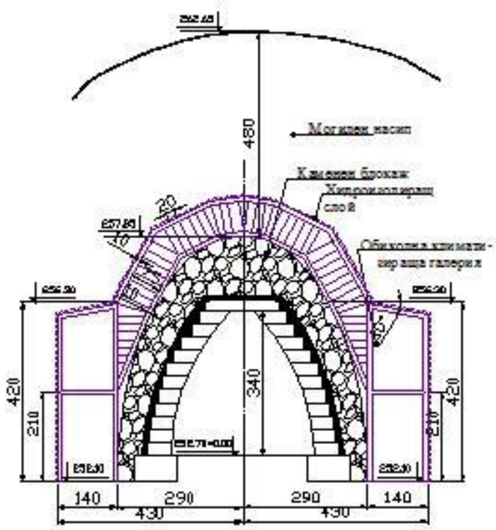


Figure 6. Cross-section of the round chamber. A variant with box holes with timber lining

CALCULATION PART OF THE SUPPORTS [1, 2,3]

The supportive frames, the metal coatings and the planks were calculated according to the technique of Prof. Tsimbarevich applied to weak and incompetent rocks. The load capacity of the supports was calculated 1.5 and 2 times higher in order to ensure their durability (set for 50 years). The choice of the corresponding profile is according to the resistance moment. The profiles THN-16,5 and THN-29 meet best the requirements of conditions.

The support units will be manufactured by the Single Owned Company Rockstal from Pernik.

COSTS CALCULATION

According to the preliminary estimations the project needs 5 months for realization. The funds needed are proportioned as follows:
1. Mechanization - 8%
2. Labour - 15%
3. Materials - 42%

Other expenses (transport, storage costs, travel costs, digging of a site, timber, geodesic tracing and control, etc.) - 35%.

CONCLUSION

In conclusion the authors would like to note that the support is composed entirely of prefabricated units (No welding works in the gate are foreseen). The metal elements are preliminary treated with anti-corrosion varnish.

This technology can provide an easy repair (when necessary) and maintenance of the particular elements of the supportive constructions.

We find that the air-conditioning of the two chambers in Thracian tomb in Alexandrovo will benefit from the suggested gate.

The preliminary calculations appraise the realization costs of the project to about 85 000 Euro. Its completion will take 5 months.

As a comparison the restoration of the Thracian tomb in Svesthari has lasted for 20 years. 2 500 000 Euro have been spent on the works.

The mining technology suggested herewith allows controlling the microclimate in the gallery. Thus the frescoes in the chambers of the Thracian tomb will be preserved and the authentic pile of the mould will not be destroyed.

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.31-35

THEORITIC-PRACTICAL APPROACHES OF ENERGETIC METHOD OF OPTIMIZED VARIANTS OF OPEN-MINING PROCESSES

Paun Ljubenov

ABSTRACT

Energetic approach lays on particular nature and mechanic-technologic factors of determinate and occasional (probable) character, as: quantities of mineral resources, openings (additionally supporting substances, useful lots, dead lots, tares of moving mechanisms.

This method accepts following of quality requirements to the useful mineral resource, lots in common, sequrity work conditions, even financial sources economy, indicated with their technic equivalent. It do not abolis, but suppose sizeshaping methods. After quantities of lots,energetic method optimizes distances of movement, on third place resistance of movement on the road of the lots, analizes "dispersion" of technologic process by supporting operations.

Forth, it analiyzes and indicates total coeficient of dead lots, as much as total coeficient of useful activity on the main movement direction of movement of useful lots.

Fifth, -It analizes times Тдв of movement, and this stges at the main decidion process for one cycle of lot moving separate or following connected facilities. Other parameters of mining source are derivativ of the mentioned above

$$k_{\text{от.i}}.Q_i.L_i.W_{\text{ср.i}},\left(\frac{tm}{t} \equiv \frac{tm}{\gamma m^3} \equiv \frac{1}{\gamma}\frac{t}{m^2}\right) \to \min,$$

I=1,2,3

Where:

Ei- total energyadsorbtion of I-variant of process

Qi- lot quantity in clear units- measure

K o t I-coeficient of total lot (useful and tare)

L пр.i- distance of lots movement

Wпрi-movement resistance of lot unit

Тпрi- time of movement of a cycle—movement. In accordance with reception of quantity of energy/ total and comparable/ for one production cycle.

$\prod_1^k k_{\text{п.д.к.i}}$ - total coefficient of useful activity of production process –basic direction.

INTRODUCTION INTO PROBLEM

As post-graduate student at LGI I started studying theory and practice of open mining of mineral sources by the book "Open mining of mineral resources sources"Уолстехиздат 1951, and the work "Bases of the theory of opening of quarry fields" M. 1953 (dissertation of E. F. Sheshco as Dr. of technical sciences), and his other printed works, as other Soviet authors. For solution

Me, as follower of dialectic materialism, and building engineer, as much, as private sciences, I was not fully satisfied of approaches, applied for solution of open-mining project tasks, and applied in management of exploitation of different stages of mining production.

Still at our first meeting with E.F. Sheshco in Moscow at the Department, I told him, that so much complicated system of mine, had or should have value of tension, similar to building construction.

At buildingmechanics buildingconstruction could tough and secure projected, and then used with the expression of warrantable tension. Of course, economically. Because, in my mind there was a physicomechanic analogy between building construction and mine: mine as complexed not static, but dynamic system.

I answered that I had such a vision. As a student at the scientific group "Building mechanics" at Sofia Polytechnic, I read the scientific report "Act of the Law of passing of quantity in quality at the Building (theoretic) Mechanics, in the presence of three professors static (mechanics), and it had been met with praises.

So, ahead of E. F. Sheshco, It became a formation of my scientific program "tensity" as law, craters of evaluation of the open-mining processes, basically connected to the topic of my dissertation "Rational schemes of opening of East Maritsa Lignite source".

Up to the mentioned moment, I have been concluded with the equalization exams of mining engineering and objects of candidate minimum. There was coming developing of formalized theory on the base of generalization of project and executive-mining practice.

Sometimes, somewhere, the great Russian composer M. I. Glinka, used to say: "People make music, and we-composers arrange it."

So, I had to study, to be introduced with, to assimilate project and open-mining practice after thinking and analyses, and to feed my corn of scientific and practice contributions. So, briefly, by the influence of my scientific manager Prof. B. V. Bokii, I connected my life with "Open mining" Department of Lengyproshacht, and "Investors direction "Maritsa East", which controlled complex studying of coke source, and began source of mine "Troianovo-1" with 3 mln t/year, and I worked on the project of this mining of 5 million tones/year, so I took an active participation.

By the indicators of total and comparative energyadsorbtion, which had (in the case of my dissertation), size of useful physic work, related to one tone of worked material, to find generalized indicator of the accepted in practice private criteria of optimization of the variant of project decisions. And when useful work have been referred not to tone of material, but

$\frac{\mathbf{1}}{\boldsymbol{\gamma}}t$ or to m^3, to $\frac{tm}{m^3} = \frac{t}{m^2}$,

Which tension have been increased γ times. So, my basic program, expressed ahead of Sheshco, has been completed. With it first stage of the theory of energetic optimization finished. It was a partual energetic appreciation of open mining lot flow by energy consumption of the useful physical work.

Second stage should take into consideration "the second beginning of thermodynamics, giving energy dispersion when making any useful work. We definated a generalized coefficient of useful work.

$\prod_{1}^{k} k_{\text{п.д.к.i}}$ Of mining lot flow.

Increasing with reciprocal value $\frac{1}{\prod_{1}^{k} k_{п.д.к.}}$ energyadsorbtion-general and comparative (Q=1) from the useful work of lots flow, and variant optimization by choosing lowest energyadsorbtion, mentioned in the Law in the resume.

I should stop on this second stage of my mathematic aparate. But history of mechanics shows, that scientists tried to get a summary of the mechanic laws (after Galilei, Newton), and to develop It from united begging. Mopertui have offered the principle of activity-i.e. Multiplication mass by speed by path of division- all the three are observed parameters of the moving body.

According Mopertui division is a law, by what GodFather have ordered movements of bodies to be made.

By vibration task L. Oiler generalized "Law of smallest activity, called so lately. Lagrange have improved, that variations of smallest activity give conditions (differential) ones of leading three Newton Laws of movement of a point or body.

There are formulas of the smallest activity of different physic processes- Lagrange, Hamilton, Einstein, going to Plank constant, showing that energy has a structure and it is transmitted in microspace processes in quantity parts, called "energy quant " and "quant of smallest activity", or constant of Plank.

In my long-year studies of open-mining processes, Idiscovered analogues between change and dispersion of energies in micro and macrospace.

In this case- transformation of specific portions of energy. Icalled it still before my pension in 1975 smallest activity and macroquant energies in open-mining processes.

Actually, they are an energyadsorbtion of process for unit of mass, for unit of path, and unit of time cycle.

Or, from my point of view, there exists Quant mechanics, except for the microworld, but for the macroworld with it macroquants of energy and activity.

Quants of energy will exist then, when unit of mass microobjects, and unit of path of lot flow are real distance and microspace and time is considered as frequency (cycles) of energetic activity.

With it, I think, I prove correctness of Einstein, that "God do not occupy in playing in dies".

Quant phenomenon, as I'll show above by formulas, have much more complexed determination of this, which follows of the logic and aparate of classic Newton mechanics. It's, maybe for nowadays, an end of my most difficult scientific-energetic program.

2. Basic energetic exchange and its application for choosing of optimal open-mining lot flows

Our energy aparate method lays on particular nature mechanic-technologic factors of determined and of a probable character. We will name with capital Latin letters macroparameters of open-mining variants off lot flow. And with differential differences, as much as shunting quantities. Macro quantities of the lot flow are $Q_и$, $Q_о$, $Q_д$ - lot weight and useful mineral resources indexed "o", opening and useful mineral resource, "д" –additional nature mineral lots.

Machines and their moving organs have dead weight-coefficient of tare weight.

Length of lot flow have two macro quantities- L' and L – respectfully distance between centers of weight of starting off field (fig 1 and fig 2), or the distance of the trace on, the road of lot flow, $k_{р.т}$ = L / L-coefficient of developing of the trace;

$L = \sum_{1}^{k} l_k$; l_k – section with particular strength of movement w_k, for which engine, auto traction-engine and etc. is defined the speed k_V;

$$V_{\text{дв.i}} = \sum_{1}^{k} V_k ; T_{\text{дв.i}} = \frac{L}{\sum_{1}^{k} V_k} ; W_{\text{ср.i}} = \frac{\sum_{1}^{k} W_k l_k}{\sum_{1}^{k} l_k}$$

- avarage strength of movement on the trace of the road $L_{i..}$ …
$P_{л.i}$ –weight of engine or auto-traction-engine; q_o –weight of the waggon

q –volume of the wagon; $k_т = \dfrac{q_o}{q}$ -coefficient of tare;

$k_{от.i} = 1 + 2\dfrac{q_o + P_л}{q}$ - coefficient of total tare of the autotrain

i = 1, 2, 3,… - number of lot flow variant

So, making such an analyses in my dissertation of movement of mine lots by (auto) train, conveirs and hydrotransport (for generalization of the analyzes), I got to formula an optimality criteria of lot flow by energy equal to the necessary useful physics work, in the type of dimension as energyadsorbtion and tension. My first (not full Law (criteria) is exactly:

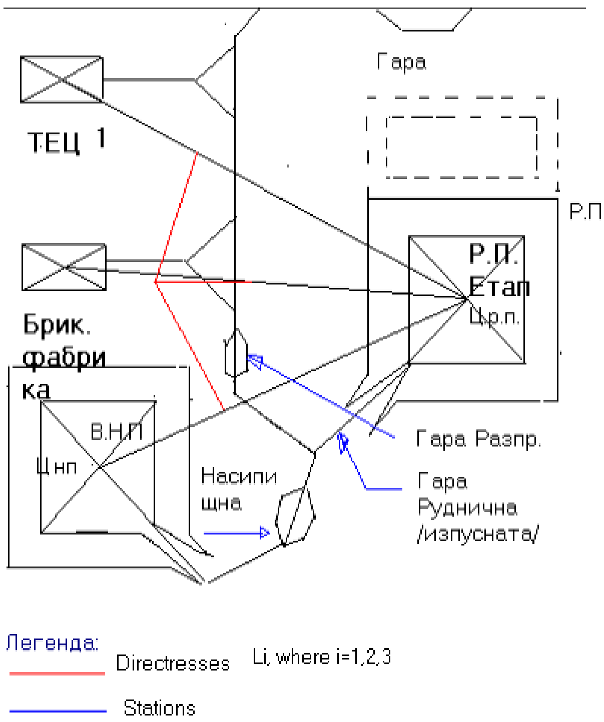

Fig.1 "Maritsa East" -First Industrial Complex

$$k_{от.i}.Q_i.L_i.W_{ср.i}, (\frac{tm}{t} \equiv \frac{tm}{\gamma m^3} \equiv \frac{1}{\gamma}\frac{t}{m^2}) \to \min \tag{I.1}$$

With this generalizing criteria, as I told, finished the first stage of my program. There have been laid private criteria for different occasions:

H₀ – min (minimal power of opening in the ranges of dispose of transport and section trnch

Criteria Qo, t —- min -или Qo, куб. m ----- min –applied for choice of variant of transporting and cutting trenches when opening the mine

Criterias L or L / L` - min –--, Крт = L / L-----min - of the coefficient of developing of the trace for the choice of spoil, or given-received field for users.

When Q I = 1, total enegyadsorbtion for the usefull phisics work of lot flow passes to comparable one. It becomes

$$k_{om.i}.L_i.W_{cp.i}, (\frac{tm}{t} \equiv \frac{1}{\gamma}\frac{t}{m^2}) \to \min;\ here\ (Q_i = 1) \tag{I.2}$$

With this first stage the program finished. Defense of my dissertation passed with encouragement from

Scientific Union of LGI- Leningrad Mining Institute – to continue my air energetic development.

In Bulgaria, at MGU. Due to heavy family illnesses, and death of my wife of stomach cancer, and I was ill of strong hepatitis, I was in a big difficulty. So my scientific program have been almost interrupted, without being fully forgotten.

Second stage have been realized by introducement of criteria of total energyadsorbtion –total and comparable, of the type:

$$\frac{k_{о.т.i}.Q_i.L_i.W_{ср.i}}{\prod_1^n k_{п.д.к.i}} \to \min \tag{II.1}$$

here $\prod_1^n k_{п.д.к.i}$ -general coefficient of useful activity of lot float, as multiplication of private ones.

or when Q=1

$$\frac{k_{о.т.i}.L_i.W_{ср.i}}{\prod_1^n k_{п.д.к.i}} \to \min \tag{II.2}$$

Here $К_{п.д.к}$ – private coeficient of useful activity, component of the total KUA (coeficient of useful activity).

So, It was kept the requirement of indication of energy of dispersion, when executing lot flow in variants I= 1, 2, 3…

The Third, generalizing, but new again, at the same time concluding stage of optimization of open-mining lot floats, by now, lays on:

Universalization of “The second beginning of thermodynamics of all the macroparamaters of energyadsorbtion- total and comparable”, as it have been accepted, but this parameters disperse also.

Dispersion is considered and indicated by private subparameters of every macroparameter type. Subparameter types and its empiric coefficients- correlation of dispersion – written by us in small Greek letters. Most corresponding letters of the capital Latin letters of the macroparameters.

2. Second base of my generalized Law of optimization lays on the generalization of the Law of classic Newton Mechanics, as I said above, by the magnitude of “smallest activity”.

Generalizations made by L. Oiler, Lagrange and other great scientist. Without more details, our basic Law (Criteria) of optimization of open-mining lot flows is of the type:

$$\frac{V_{ср.i}^2}{Q_i.L_i} = \frac{\gamma_i k_{о.т.i}\mu_i\lambda_i}{\upsilon_i\tau_i}\frac{1}{T_{д.в.i}} = \frac{\gamma_i k_{о.т.i}\mu_i\lambda_i}{\upsilon_i\tau_i}\varphi_i \rightarrow \min \qquad \text{(III.1)}$$

$\varphi_i = \frac{1}{T_{д.в.i}}$ - Frequency (frecvention) of open-mining lot flow, substitution of time movement, without indicating of technologic and sudden stoppings –under-parameters.

But multiplied with L, full formula (III. 1) is the comparative cinematic energy (alive power), what I-variant of open-mining lot flow radiates (disperses) for unit of useful lot. And L length is real quantity of lot flow, it may be macro or micro amount. First right fraction and middle parts are a macroquant of activity of mine lot flow.

Consequently, left part is “ macroquant of energy” of lot flow, for which its names I have been ironized by lots of “omniscents”.

As much as lot flows-variants, have been timely constantly optimized ago, made so by their microparameters at time, it follows from formula (III.1), and its multiple L (III. 2), that it is not enough.

It is known, that every constant in variation makes zero, so main practice interest should be aimed to the casual quantities (magnitudes – correlation), in the attitude (magnitudes-correlation), in the attitude of useful lot length of the road, change of volume of lot moving resources, to values – fluctuations of the speed. My major theoretic conclusion Is, that basic law of macro-quant dynamics (III. 1 and 2) is more common, and it includes in it Law of quant mechanics, where когато Q_I is a microobject, and L_I is some real distance, - and $T_{дв.i} \neq 0$, но $\rightarrow 0$.

I’ll point down one more our fundamental result of developments by now:

It is the proved by us (formula I. 1 and I. 2. fact), that there exists in nature reality “pressure of movement” at the material part, except for the wild strains (pressures) famous: elastic, plastic and distructing pressures, proved by test examples of material in testing stations.

So, three Optimality Laws could sound so:

1. Optimal variant of open-mining lot flow is this one, that has lowest tension (comparative energyadsorbtion) of making useful physic work, i.e. for the main energy expense (Stage I of my candidate dissertation in Leningrad).
2. Optimal lot flow is this one , that has lowest tension of lot movement and lowest energyadsorbtion.
3. Optimal project or produce execution of open mining lot flow is that one, in which together with smallest energyadsorbtion (lowest tension of movement of the lot) by the determined microparameters of lot flow , should be indicated lowest tension of the lot flow (and of determined macroparameters of flow should be indicated “lowest macroactivity” and “lowest macroquant of energy” of dispersion of subparameters of macroparameters of lot flow, in accordance to “Second beginning of thermo-dynamics”.

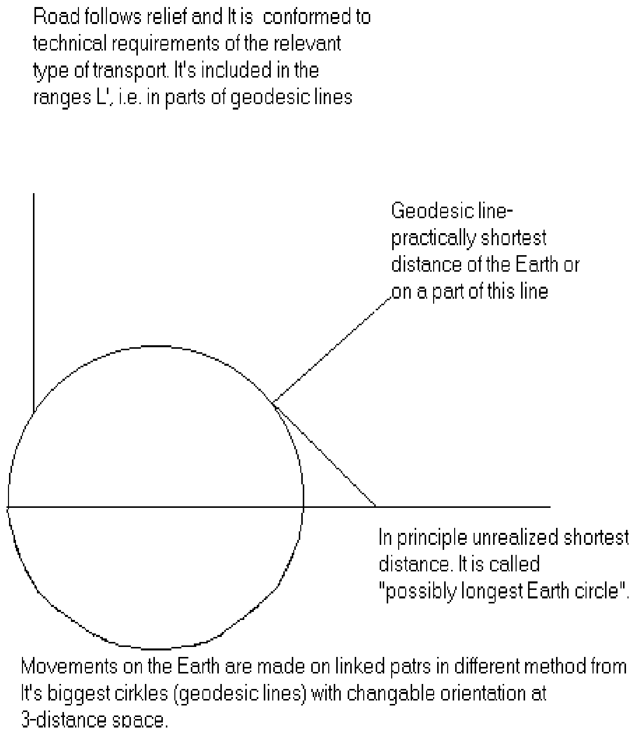

Figure 2. To the change of influence of the Earth shape and It's releif on the shortest movement distances of burdons and any surface movements.

4. In the end, we will present the choice of optimal open-mining lot flow according criteria of optimal psycho-physic and creativity-inventors workability , on which base we differ phisic, psycho-fiziologic and creativity-inventors workabilities. Analitic equation of optimal criteria is:

J = 15 j – pfisic workability of person angered at the working process.

Nфр – number of workers only with physic workability

Кпс.Nпс – coefficient of psycho-fiziologic workability of workers group of the increased quallification staff.

Кти.Nти –Coefficient of creative-inventors ability of workers group of flow serving staff.

Nпс – nomber of workers with increased psychic workability.

Nти –nomber of workers with creativity inventors ability,

Кпс > coefficient of correlation of higher price of psychic workability of person of the following workers group only with physic workability

Кти > 1 coefficient of correlation of indicating of the highest valueof creative-inventors ability of this worker, engaged with their physic workability at the moment.

Main Law (criteria) критерий of optimality of variant of open-mining lot flow , according condition of psychophysics and creative-inventors workability of the person engaged in the process is:

Э = 15 / Nфр + Кпс.Nпс + Кти.Nти /, j ----- min, -/IV.1/

а при Q .= 1 t, follows:
э = Э / Q i = 15 / Nфр + Кпс.Nпс + Кти.Nти / : Q i, j, ---min, /IV.2/

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Recommended for publication by Department of Underground mining , Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.37-38

ANALYSIS OF MAJOR TECHNOLOGICAL PARAMETERS OF QUARRYING OF BLOCKS OF MARBLE BY "DIAMOND" ROPE STONE CUTTERS

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ABSTRACT
An analysis is done of the technological parameters of quarrying of blocks of marble "Pelegrini" TDD 100 rope stone cutters. Parameters of mined blocks, number of stopes and stone cutters are determined for working with different heights of benches for the conditions of the "Mura – 8" quarry.

Marble blocks in our country are quarried by "diamond" rope stone cutters of the "Pelegrini" TDD-100 type. Quarrying of blocks is carried out from benches of 2,5 m height, from several lamellas, located one by another along the longitudinal or cross axes of the quarrying field.

Height of the bench ($H_{ст}$) is determined depending on length of rope ($l_в$) and parameters of quarried marble blocks under the condition that 2/3 of the rope length is located in the section.
When vertical longitudinal cuts are done the maximum height of the bench is:

$$H_{ст} \le \frac{2}{3} l_в - 2 l_{бл}, m, \qquad (1)$$

where $l_{бл}$ is the length of quarried block, m
When horizontal cuts are done the maximum length of the rope is:

$$\frac{2}{3} l_в = (2 l_{бл} + b_{бл}), m, \qquad (2)$$

or

$$\frac{2}{3} l_в = (2 b_{бл} + l_{бл}), m, \qquad (2)$$

where $b_{бл}$ is the width of quarried blocks, m
When vertical cross cuts are done the height of bench is the highest

$$H_{ст} \le \frac{2}{3} l_в - 2 b_{бл}, m, \qquad (3)$$

The lowest value of bench height is determined by formulas (1) and (3) for standard lengths of rope 20 and 25 m and approved parameters of quarried blocks and it is approved as a designed height of the bench. The required length of rope (table 1) - $l_{в\ необх.}$ Is determined for a given height of the benches and different parameters of quarried blocks and a check is done by formula (2).

Table 1.

$H_{ст}$, m	2,5	2,5	2,5	5	5	5
$b_{бл}$, m	1,5	2	2,5	1,5	2	2,5
$l_{бл}$, m	5,5	5,5	5,5	6	6	6
$l_{в\ необх}$, m	18,75	19,5	20,25	20,25	21	21,75
$l_{в.ст}$, m	20	20	20	25	25	25
$l_{от}$, m	1,24	1,08	0,98	1,034	0,866	0,766
$n_з$, бр	2,48	2,16	2,0	2,05	1,73	1,53
$S_{вн}$, m^2/см	9,099	9,09	9,09	8,33	8,33	8,33
$S_в$, m^2/см	33,3	25	20	33,3	25	20
$S_х$, m^2/см	20	20	20	10	10	10
$S_{об}$, m^2/см	62,4	54,1	49,09	51,63	43,33	38,33
$S_{об}$, %	100	87,6	78,7	82,7	69,4	61,4
$N_{км}$, бр	2,5	2,16	1,96	2,06	1,73	1,53
$V_{бл}$, m^3	20,6	27,5	34,4	45	60	75

The relative area of cuts for quarrying of 1 m^3 rock mass is

$$l_{от} = \frac{1}{b_{бл}} + \frac{1}{H_{ст}} + \frac{1}{l_{бл}}, m^2/m^3 \qquad (4)$$

The following parameters are determined for the conditions of "Mura-8" quarry, when the average shift productivity of rope stone cutters "Pelegrini" TDD-100 is $Q_{см}$=25 m^2/cm and duration of shift is $T_{см}$=11 h, for the quarrying of 17,5 m^3/cm marble blocks or 50 m^3/cm marble mass for shift:

1. Required number of simultaneously mined stopes (blocks) in the quarry

$$n_з = \frac{Q_к . l_о}{k_и . Q_{см}}, броя \qquad (5)$$

where:
$Q_к$ is the shift productivity of rock blocks of the quarry, m^3/cm;
$Q_{см}$ – the average shift productivity of stone cutters, m^3/cm;
$К_и$ – coefficient of mined stone blocks (recovery).

Results are shown in table 1 for different parameters of mined blocks ($b_{бл}$, $l_{бл}$ and $H_{ст}$).

2. The needed area for vertical cross cuts for a shift is:

$$S_{вн} = \frac{Q_k}{k_и . l_{бл}}, m^2 / см \qquad (6)$$

3. The needed area for vertical longitudinal cross cuts for shift is

$$S_в = \frac{Q_k}{k_и . b_{бл}}, m^2 / см \qquad (7)$$

4. The needed are for horizontal cuts for shift is:

$$S_x = \frac{Q_k}{k_и . H_{ст}}, m^2 / см \qquad (8)$$

5. The total area for cuts for shift is

$$S_{об}=S_{вн}+S_в+S_x,\ m^2/cm \qquad (9)$$

6. The needed number of cutting machines for the cuts of one shift is:

$$N_k = \frac{S_{об}}{Q_{см}} \qquad (10)$$

Results from the analysis of those parameters are presented in table 1.

In case it is accepted that the cost for driving the cuts is equal, table 1 shows that the most profitable mining of marble blocks will be realized for $H_{ст}$=2,5 m and $l_{бл}$=5,5 m, $b_{бл}$=2,5 m and respectively for $H_{ст}$=5 m for $l_{бл}$=6 m, for $b_{бл}$=2 m and 2,5 m. For that cases the relative area of cuts is the lowest ($l_{от}$, m^2/m^3) and two cutting machines are required.

When the height of bench is $H_{ст}$=2,5 m the use of ropes of length of 20 m is recommended, and for $H_{ст}$=5 m – ropes are recommended to be 25 m long.

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STUDY ON FACTORS HAVING INFLUENCE UPON THE PRODUCTIVITY OF CYCLIC-FLOW-LINE TECHNOLOGY APPLIED IN OPEN PIT "ASAREL"

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ABSTRACT
The cyclic-flow-line-technology (CFLT) for mining of overburden has been introduced since 2000 in "Asarel-Medet". It's working is characterized by often interrupts with different continuance.
The stays of technological line has been studied. A review of factors having an influence upon the productivity has been made. A statistic working of data is used for a determination of the degree of significance of each of the investigated characteristics.
A comparative analysis has been used for the determination of the indexes having the most important influence upon the work of CFLT. A mathematical model has been developed with describing the connection between the studied indexes and the CFLT' s productivity.

INTRODUCTION

The cyclic–flow-line technology (CFLT) is most extensively applied at Asarel Medet Open Pit. The mine is the primary producer of copper and pyrite concentrates in Bulgaria, and one of the main producers in Europe. The processing complex is located on an area of 20 000 da close to the city of Panagyurishte (Bulgaira).

CFLT was introduced in the production process in 2000. This technology involves great variety of mining and transportation equipment and it was introduced for the first time in our country. The following machines are used: excavators "ЕКГ"-8И, "EKG"-5A, "RH"-120C, "R"-994, front loader "CAT"- 992D, drills "SBS" – 250, trucks "BelAZ"-7519, "BelAZ"-75125, "BelAZ"-7522, "BelAZ"-7523, "CAT"-777C, "CAT"-777D, "O&K"-K95, "TR"-100, a stationary re-loading point equipped with screen and cone crusher "KRUPP", belt conveyors and spreader "VOEST-ALPINE".

The cyclic-flow-line is used for overburden haulage from "Asarel" Open Pit to the Western Dumpsite (fig. 1).

Factors influencing upon productivity of the CFLT

The mechanization in open pit "Asarel-Medet" works in various and variables conditions. For that reason the work of the mining machinery is characterized with often interrupts. Because of that there is a special interest in analyzing the work time structure, the kind of stays and their influence upon the machinery productivity using the CFLT for output of overburden.

The following are the main reasons for the interruption of the production process in an open pit area: the planned running repairs (PRR), the moving of mining machines, the different kinds of damages, blasting, weather conditions, physical-mechanical properties, rock compositions, the interruption of the work in next section and etc. All that allows that it could be accepted that the productivity of mining mechanization is influenced by numerous accidental factors. Therefore it has a probable character. The planned repairs have not been taken into account in this paper, because of the fact that they are known in advance, and they don't influence upon the productivity of CFLT.

Figure 1.

The main factors influencing upon the productivity of CFLT are given in a table 1. The values for 24 hours stays of mechanization in "Asarel-Medet" in 2001 have been used.

Statistical working of data

Some formulas of (Toncov, 1984) have been used for solving the concrete task of statistical working of data. The calculations have been done by "Statgraphics Plus" program. The used symbols are as follows:

- $\overline{X}$ - average;
- σ^2 - variance;
- σ - standard deviation;
- X_{min} – minimum value;

- X_{max} – maximum value;
- X_l – lower limit (95% confidence intervals);
- X_{up} - upper limit (95% confidence intervals);
- A_s – standard skewness;
- E_x – standard kurtosis;
- r - correlation;
- R – rank correlation;

The final results of data statistic working are given in tables 1 and 2. The analysis shows that average overburden 24 h productivity of CFLT is 33079 t.

The mining quantities of overburden for 24 hours are in interval from 31507,7 t to 34651,7 t in 95 % of the cases.

The correlation analysis is used to study connection between a productivity of CFLT and the factors given above. The calculations have been done by "Statgraphics Plus" program using the entrance data. The results are given in tables 1 and 2. The found correlations and rank correlations are given in table 3.

It can be seen from the made correlation analysis that every studied index has a weak influence upon a productivity of CFLT. On the other side it can be seen the accumulation of influence by the different factors upon the system's work. That is why the further efforts are directed to the investigation of the joint influence between given indexes and productivity of CFLT.

Determination of the degree of importance of the studied indexes

The correlation analysis has been used for determination of the multiple correlation between indexes given in table 3. The calculations are made by "Statgraphics Plus" program using the entrance data. The receiving multiple correlation is 0,665.

Table 1 Found statistical indexes of data.

Studied indexes	Characteristic				
	Number of data	X_{min}	X_{max}	$\overline{X}$	σ
Productivity of CFLT, t per 24 h	290	2800	65500	33079,7	13601,6
Stays for material, h	290	0	12,45	1,328	1,916
Stays for rocks with bigger size than crusher size in hopper, h	290	0	2,43	0,19	0,4
Stays for metal detector switched off, h	290	0	4,38	0,261	0,539
Stays for voltage falls, h	290	0	3,98	0,153	0,522
Stays for weather, h	290	0	8,97	0,214	0,99
Stays for blasting, h	290	0	2,47	0,247	0,605
Stays for personnel change, h	290	0	5,54	0,552	1,172
Stays for crusher damage, h	290	0	11,6	0,991	2,135
Stays for screen choked and overfilled conveyer belt, h	290	0	3,35	0,146	0,459
Stays for overfilled delivery chute, h	290	0	13,79	0,457	1,774
Stays for conveyer belt vulcanization, h	290	0	21,9	0,514	2,619
Stays for conveyer belts 1, 2 and 3 removal and conveyer belt's disconector switched off, h	290	0	21,1	1,191	2,717
Stays for spreader damage, h	290	0	17,87	0,475	2,043

Table 2 Found statistical indexes of data.

Studied indexes	Characteristic				
	σ^2	$X_{ляв}$	$X_{десен}$	A_s	E_x
Productivity of CFLT, t per 24 h	185003522,6	31507,7	34651,7	- 1,069	- 2,223
Stays for material, h	3,672	1,107	1,55	22,507	40,405
Stays for rocks with bigger size than crusher size in hopper, h	0,16	0,144	0,236	23,747	45,898
Stays for metal detector switched off, h	0,291	0,199	0,323	32,724	92,61
Stays for voltage falls, h	0,273	0,093	0,214	29,505	68,46
Stays for weather, h	0,98	0,099	0,328	42,75	148,387
Stays for blasting, h	0,366	0,177	0,317	15,121	10,668
Stays for personnel change, h	1,374	0,417	0,688	16,956	17,57
Stays for crusher damage, h	4,577	0,774	1,238	22,713	38,771
Stays for screen choked and overfilled conveyer belt, h	0,211	0,093	0,199	29,375	68,986
Stays for overfilled delivery chute, h	3,147	0,252	0,662	39,204	122,311
Stays for conveyer belt vulcanization, h	6,859	0,211	0,817	40,08	120,21
Stays for conveyer belts 1, 2 and 3 removal and conveyer belt's disconector switched off, h	7,384	0,877	1,506	33,672	90,334
Stays for spreader damage, h	4,172	0,239	0,712	43,999	148,601

Table 3 Correlation, rank correlation and β coefficient of the studied indexes.

Studied indexes	Symbol	Productivity of CFLT, t per 24 h		
		r	R	β
Stays for material, h	X_1	-0,192	0,149	-0,348
Stays for rocks with bigger size than crusher size in hopper, h	X_2	0,097	0,27	0,009
Stays for metal detector switched off, h	X_3	0,053	0,185	0,012
Stays for voltage falls, h	X_4	-0,032	0,058	-0,132
Stays for weather, h	X_5	--0,104	-0,135	-0,173
Stays for blasting, h	X_6	-0,11	-0,123	-0,148
Stays for personnel change, h	X_7	-0,008	-0,173	-0,121
Stays for crusher damage, h	X_8	-0,173	0,012	0,257
Stays for screen choked and overfilled conveyer belt, h	X_9	0,048	0,093	0,021
Stays for overfilled delivery chute, h	X_{10}	-0,18	-0,114	0,227
Stays for conveyer belt vulcanization, h	X_{11}	-0,277	-0,213	0,362
Stays for conveyer belts 1, 2 and 3 removal and conveyer belt's disconector switched off, h	X_{12}	-0,264	-0,058	0,35
Stays for spreader damage, h	X_{13}	-0,2	0,058	0,284

This shows that there is an expressive linear dependence between studied indexes. It is express by multiple regression equation.

$$Y = 45325,6 – 2473,82X_1 + 309,737X_2 + 295,25X_3 – 3433,61X_4 – 2381,64X_5 – 3321,52X_6 – 1411,16X_7 – 1638,37X_8 – 611,662X_9 – 1739,39X_{10} – 1881,38X_{11} –1749,95X_{12} – 1892,58X_{13} \quad (1)$$

where:
Y – productivity of CFLT, t per 24 h;
X_i – indexes noticed in table 3;

Multiple regression allows influence of different factors upon productivity of CFLT to be measured. For this purpose it is used so called β coefficients (Четыркин, 1982).

They are found by the following equation:

$$\beta_I = a_i . \left(\frac{\sigma_{X_i}}{\sigma_Y} \right) \quad (2)$$

where:
a_i – coefficient in correlation equation;
β_I – correlation received after calculation;
σ_{X_i} - standard deviation of variable X_i;
σ_Y - standard deviation of Y for each factor;

The calculated coefficients β_I for each of the factors are given in table 3. Using them the following arrangement of the indexes according to their degree of influence upon the productivity of CFLT has been used:

1. Stays for conveyer belt vulcanization – 100%;
2. Stays for conveyer belts 1, 2 and 3 removal and conveyer belt's disconector switched off – 97%;
3. Stays for material – 96 %;
4. Stays for spreader damage – 78 %;
5. Stays for crusher damage – 71 %;
6. Stays for overfilled delivery chute – 63 %;
7. Stays for weather – 48 %;
8. Stays for blasting – 41 %;
9. Stays for voltage falls – 36 %;
10. Stays for personnel change – 33 %;
11. Stays for screen choked and overfilled conveyer belt – 6 %;
12. Stays for metal detector switched off – 3 %;
13. Stays for rocks with bigger size than crusher size in hopper – 2 %;

It is accepted that index having the heist value of coefficient β has the relative weight 100 %. The relative weight of the other factors is expressed through it.

A mathematical model describing connection between study indexes and productivity of CFLT

Figure 2 describes change of multiple regression in depending on number of factors having influence upon productivity. For example the influence of the first six indexes arranged by a degree of significance is determined (table 3).

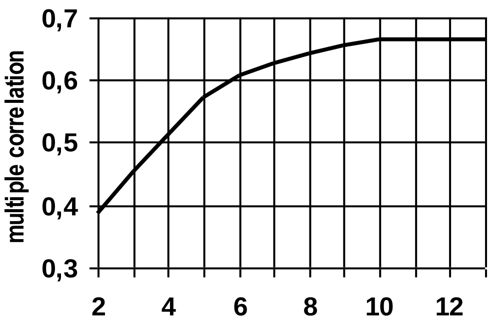

Figure 2. The change of multiple correlation depending on number of factors

The follow conclusions are done by chart from fig. 2:

- The first six factors have the most important influence upon the multiple correlation;
- The influence of the other four indexes is less;
- The last three factors almost don't influence upon multiple correlation;

The received model by determining the above – mention 13 factors according to their grouping from 1 to 13 will be difficult to use. That is why the most important factors from 1 to 6 will be used.

The correlation analysis is used for describing a connection between productivity and chosen factors. The calculations are done by a “STATGRAPHICS PLUS” program using the entrance data. The received model is:

$$Y = 41879{,}7 – 2118{,}63X_1 – 1750X_{11} — 1570{,}29X_{12} – 1679{,}35X_{13} - 1722{,}19X_8 - 1552{,}45X_{10} \quad (3)$$

Multiple correlation is 0,606. This shows that there is an obviously expressed linear dependence between the productivity of technological line for output of overburden and the studied indexes.

Forecasting the change of productivity by the received model

The studied model is used for forecasting of productivity of CFLT by indexes influencing upon it.

The forecasted results, received by the model and the results, observed for the period January – March 2001 are given in figure 3.

The evaluation of the developed model can be done by absolute percentage mistake ($e_{pr.m.}$) (Shim.J. 2000). It is calculated by a formula:

$$E_{pr.m.} = \frac{100}{n}\sum_{i=1}^{n}\frac{|Y_i - f(x_i)|}{Y_i}, \% \quad (4)$$

where:

Y_i – observe value;
$f(X_i)$ – forecasted value;
n – number of data;

The model including the all 13 factors is used for comparison. The forecasted results received by equation (1) and the observed values for the period January – March are shown on figure 4.

The percentage mistake for each of the models is calculated for the whole studied period. It is 34 % for equation (1) and 36 % for equation (2).

The analysis shows that a model (2) forecasts productivity of CFLT has almost the same accuracy as a model (1), in spite of the less number of used factors in model (2). Therefore the equation (2) is better for practical calculation.

CONCLUSION

The basic factors influencing upon the mining of overburden in open pit “Asarel-Medet” are studied in this paper. The significance of each of the indexes influencing upon the productivity of CFLT is determined by a correlation analysis. Six are the most significant factors received after suitable arrange of all 13 factors. Therefore the mathematical model describing the studied connection is received by them. The absolute percentage mistake of the model is 36%.

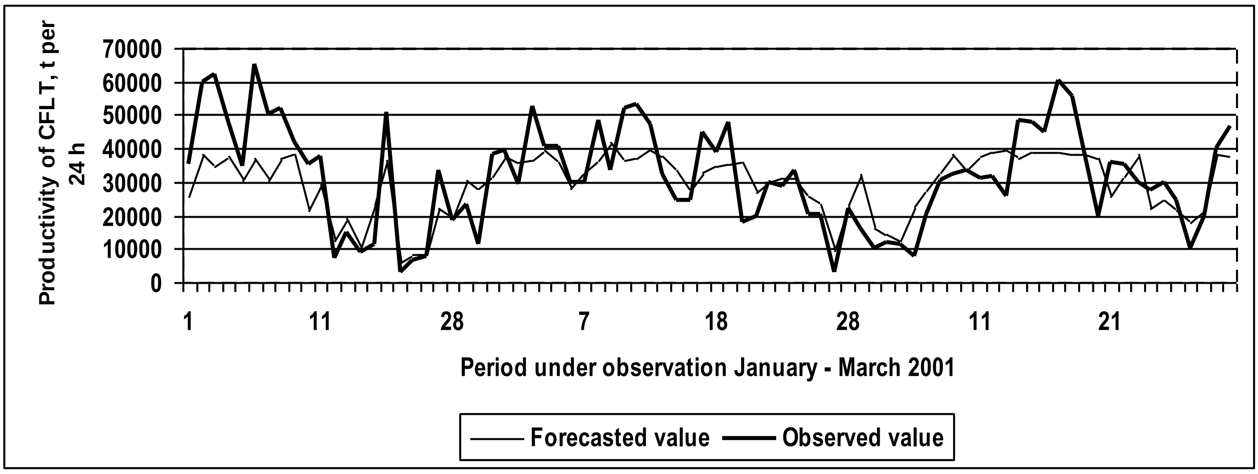

Figure 3. A change of open pit productivity during the period January – March 2001

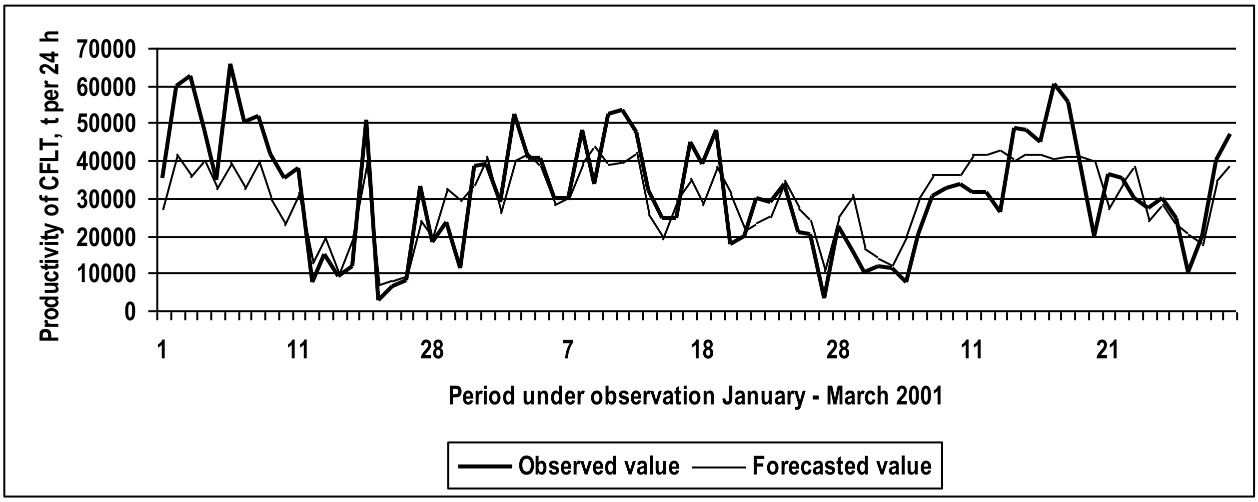

Figure 4. A change of open pit productivity during the period January – March 2001

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Recommended for publication by Department of Opencast mining and blasting technologies, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.45-49

TECHNICAL CONDITION OF MOVABLE RAILROADS AND EFFECT ON TECHNOLOGICAL PARAMETERS OF COMBINED OPERATION OF RAILROAD HAULAGE AND SINGLE-BUCKET EXCAVATORS

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ABSTARCT
The effect of technological condition of movable railroads on some of the technological parameters of combined mining operation of railroad transport and single-bucket excavators is studied by means applied to the processes and technology of opencast mining.

INTRODUCTION

Technical condition of movable railroads is determined by the real deviation of rails from their design postioned. The most important factors to determine deviation are as follows:

- visible and hidden settlements of the vertical plane;
- visible and hidden deformations in a horizontal plane;
- torsion of road round its axis.

The above factors bring to mudding of the gravel ballast bed, breaking of railroad in the zone of joint connections, loosening and breaking of joints between rails and sleepers, breaking of rails bringing to visible or invisible settlements.

The first type of deformations are shown in fig. 1, and the third type – in fig. 2.

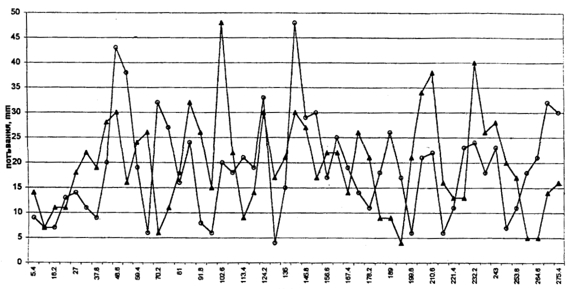

Figure 1. Response of invisible settlements of spreader railroad of inner dumping areas at the AS-1600, No 3, "Troyanovo – 1' mine

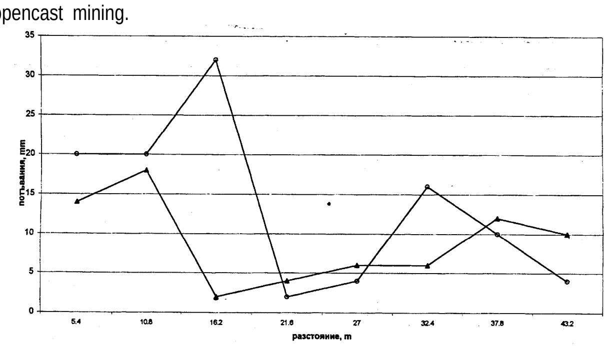

Figure 2. Response of invisible settlements of railroads at the area of spreader AS-1600, No 6, "Troyanovo north"

The first figure is shared by a project, managed by Prof. D. Stoyanov "Preventive control and repair and maintenance of movable railroads at the "Troyanovo" opencast mine of the "Maritza-East" EAD mines [1], and the second – by a project of the same author and identical title [2], but studying the same issues at the "Troyanovo-sever" mine. Measurements were carried out in October 1998.

On the other hand, the real technical condition of movable rialrods results in the admissible speed of train cars on them. For example, similar issue is treated at the "Kremikovtsi" mine by Atanas Smilianov in the project entitled "Expert assessment of road and railroad network at the "Kremikovtsi" mine [3], which is the reason for limiting the speeds of motion within their safe speeds. However, geometrical characteristics in the plane and section and haulage characteristics of locomotives allow a speed up to 45km/h, however the project of Atanas Smilianov "Problems of the moving, maintenance and repair of movable railroads at the "Kremikovtsi" mine [4], recommends speeds of moving along them limited as follows:

- in the mine – not more than 5 km/h along the southern line;
- in the dumping area and the special dumping site:
  - not more than 12 km/h in direction "empty;
  - not more than 8 km/h in direction "full"

Considering the above mentioned, technical condition of movable railroads effects significantly on major operating parameters of the mines, shown below.

EFFECT OF TECHNICAL CONDITION ON MAJOR TECHNOLOGICCAL PARAMETERS

The concept consists in presenting the effect by formulas, applied in technology of opencast mining. This is implemented by showing the consecution and interrelation of commonly applied formulas, describing the movable railroads in a more or less evident type.

Below presented cconsiderations are shown in the example of the scheme of railroad transport of the “Kremikovtsi” mine, and even more, to simplify the model, only mine – dumping area routes are treated – fig. 3.

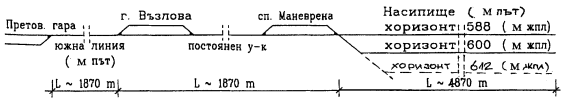

Figure 3. General haulage scheme for transportation away of overburden from the “Kremikovtsi” mine

It is worth mentioning that considerations and conclusions refer to other routes as well, including movable railroads in the specialized depot with different ways for barite ore and iron ore.

The scheme in fig. 3 is subordinated to the idea of minimum idle time of excavator at the re-loading station, i.e. the mine is supposed to work with its maximum design capacity. That suggests 100 % use of the capacities of the illustrated haulage scheme.

That requires a minimum idle time of the excavator to guarantee the performance of production capacities. In that case, if admitted that excavator works only for overburden, the number of trains, serving the excavator for a unit time is a function of time for moving of train in both directions and time for loading and unloading and expecting the next composition. It is shown by (1).

$$N_{вл} = \frac{t_к}{t_{тов} + t_о} \qquad (1)$$

where:

$N_в$ –number of trains, operating in one and the same route, and served by one excavator during a shift;

$t_{тов}$ – time for loading of train. It depends on number and volume of cars and actual technical productivity of excavator. The expression (2) is accepted as precise enough to simplify below the presented calculations

$$t_{тов} = \frac{n_в . V_в}{Q_{тех}} \qquad (2)$$

where:

$n_в$ – number of cars in a train. It depends on geometrical characteristics in layout and profile (R_{min} [m], i_{max} [‰]) of movable railroads in the mine and haulage characteristics of electric locomotives.

$V_в$ –volume of mass (in that case overburden) in one car;

$Q_{техн}$ – technical productivity of excavator.

The product $n_в.V_в$ represents the useful load, transported by the train. It depends on the maximum slope of longitudinal sections of movable and constant railroads, radiuses of curve and additional effect on the maximum value of slope within the specific portion of the railroad.

The locomotives used at the mine are E type “Bo – Bo”. The haulage calculations, carried out (by method of Lipetsk MPS, shown in [5] according to a reduced longitudinal section (reading the additional effect of resistance of curves) in [3] show values of time for moving on the southern line of the mine as follows – table 1.

Table 1. Results of haulage calculations, carried out by [3].

One locomotive				Two locomotives				Three locomotives			
7 в.	8в	9 в.	10 в.	7 в.	8в	9 в.	10 в.	7 в.	8в	9 в.	10 в.
24,3	-	-	-	19,6	27,2	-	-	19,6	23,2	26,1	29,0

When values of slopes are reduced, the effect of radiuses of the curve within that portion of the railroad is read by the formula (3) according to [5].

$$W_r = \left(\frac{200}{R} + 1,5.\tau \right) . \frac{l_k}{l_{вл}} \qquad (3)$$

where:

W_r – resistance from the curve in ‰;

R – radius of the curve in m;

τ – absolute value of non-compensated centrifugal acceleration in m/s^2, determined by formula (4).

$$\tau = \left(\frac{V^2}{13.R} - \frac{h}{S} \right) . g \qquad (4)$$

where:

V – speed of train m/s;

h – exceed of railroad transport in the curve mm;

S = 1500 mm – axial distance between rails;

G = 9,81 m/s^2 – earth acceleraion;

l_k, m – length of curve

$l_{вл}$, m – length of train.

The cars used at the mine are of dumpcar type, both Bulgarian and Russian manufacture, with a volume of 40 m^3, four axes.

Bulk weight of overburden, volume of car and own weight of car are used for calculating the total weight of cars of the train. The principal resistance of train is read as follows:

- for cars – by formula (5) – according to [5].

$$W_o^{''} = \frac{V+65}{12+0,55q} \qquad (5)$$

where:
$W_o^{''}$ – principal resistance of mition of the four cars;
V – speed of train km/h;
q – gross weight of car in t;

- for locomotives in ghaulage and non-haulage mode of operation, respectively by formulas (6) and (7), according to [5].

$$W_o^{'} = 1,9 + 0,01.V + 0,0003.V^2 \qquad (6)$$

$$W_{ов}^{'} = 2,4 + 0,11.V + 0,00035.V^2 \qquad (7)$$

where:
V – speed of moving in km/h.

In the other two portions of the railroad, according to the scheme in fig. 3, the maximum slopes are within the zones of roads towars benches of dumping area and combined to the effect of curved portions they do not exceed 9 %. For that reason speed is limited only by the geometry and is considered by the formula (8)

$$V = c\sqrt{R},\ km/h \qquad (8)$$

where:
R, m – radius of curves;
c – constant. It depends on width between rails and value of overheight of the outer rail.

A value of c=3,4 is recommended for values from 150 to 200 meters, according to S. Trendaphilov in “Construction of opencast mines” [6].

The maximum achievable speed in those portions of the railroad, not reading their technical condition, is about 45 km/h (according to geometrical characteristics of the layout and section and haulage characteristics of locomotives applied). The admissible maximum sped of trains in he mineis limited by in-company orders, reasoned by [4].

t_0 – (according to formula (1))is time for expecting the train by excavator. The thransportation scheme for re-oading station allows reducing that time to the minimum, i. e. time for maneuvres for in-coming of the empty train after out-going of the full one.
t_k – (according to formula (1)) is the time for a complete route and it is determined by (9)

$$t_k = t_m + t_{мп} + t_{пост.п} + t_p + t_{техн.п} + t_{др.з} \qquad (9)$$

where:
t_T – time for loading, according to (2)
$t_{мп}$ – time for movement of train along movable railroads of specific route and it is determined according to formula (10)

$$t_{мп} = \frac{2.\sum_{i=1}^{n} l_{i\,мп}}{V_{i\,мп}} \qquad (10)$$

where:
the numerator shows the total length of all portions of the railroad, where the trains moves along movable railroads in both directions of specific route;
$V_i(м\ п)$ – speed of train along the movable roads in km/h and according to [4]'
$t_{пост.\ п}$ – time for motion of train along the constant railroads of the same route and determined according to (11)

$$t_{пост.п} = \sum_{j=1}^{m}\left(\frac{l_j}{V_п} + \frac{l_j}{V_{пп}} + 0,0025\right),\ h \qquad (11)$$

where:
$\sum l_j$ - total length of portions of constant railroad for a specific route;
$V_п$, km/h – speed of full train;
$V_{пр}$, km/h– speed of empty train.
t_p – time for unloading the train. It depends on power and faultless work of compressors in locomotives, faultless work and power of air-conductive system and pneumatic unloading systems of cars. In the case of a train of constant number of cars and locomotives that time is a rather constant value;
$t_{техн..п}$ – depends on the route, railroad track development, installations for opening and closing of railroad switching arrows, type of track transportation schemes. For the specific case it also is a rather constant value.
$t_{др.з}$ – time for other retains of any kind. The highest is the weight of time for repairing faults of some of he sub-systems of railroad transport: railroad itself, vehicle etc. Options for reducing that time to the minimum are consist in well-dimensioned activities of scheduled repair of vehicles and maintenance of movable and constant roads in a good working condition.

It is evident that for the specific route from the example in fig. 3 the movable railroads have the most significant relative share in the algebraic total for t_K. It is evident that time for movement itself reverse proportional to speed, limited by the order, according to [4] .

It is evident that if recommendable engineering activities are suggested that may effectively and reliably counteract to to intensity of accumulation of residual deformations the technical condition will be significantly improved. That will allow speed acting according to [4] to be changed with higher ones in the denominator of (10). When an increase of the above speeds of 5/8/12 km/h to only 15 km/h, due to the high relative share of time for moving along the moveable portions of the railroad, number of routes by one and the same train for serving one and the excavator, is significantly increased.

Thus a consecutive effect on a number of important technological parameters occures. For example:

- Number of routes, done by one train;

$$r = \frac{T_{см} - t_{регл}}{t_к},\ бр \qquad (12)$$

where:
$T_{СМ}$ – duration of shift in hours h;
$t_{регл}$ – regulated time for review and revision of cars and locomotives before and after the shift;
$t_к$ – time for total route of train.

- Productivity of train:

$$Q_{вл.(см)} = r.n_в.V_в \qquad (13)$$

- Productivity of car:

$$Q_{в.(см)} = \frac{Q_{вл.(см)}}{n_в} \qquad (14)$$

The effect of movable railroads on the following technological parameters may be shown in the same way:

- Total number of routes for shift:

$$R = \frac{f.W}{n_в.V_в} \qquad (15)$$

where:
W – shift loading of volume of transported overburden according to the example;
f = 1,2 – 1,25 – recommended coefficient of reserve aiming to guarantee the shift loading.

- Total number of trains in the mine:

$$N_{вл} = \frac{R}{r},\ бр \qquad (16)$$

- Total number of cars in the mine:

$$N_в = N_{вл}.n_в,\ бр \qquad (17)$$

Determination of inventory number of cars and locomotives is realized by formulas, applied in opencast mining and according to approved organization of work in the opencast mine, approved system for planned repairs, qualification of machine-operators, reviewers and other repairing staff, condition of equipment etc.

At the second place, movable railroads effect on the time-schedule of train movement and therefor on permission and haulage capabilities of the mine railroad network. Different versions of time-schedules are applied in opencast mines. The main types of time-schedules are the parallel ones and the package one.

For example, in the case of parallel time-schedule, time necessary for permitting of a couple of trains through a certain portion of the railroad (in the example of fig. 3 oror certain distance between two stations) along a single line is:

$$T = t_1 + t_2 + 2.\tau,\ \min \qquad (18)$$

where:
T – period of the time-schedule of train movement;
t_1 and t_2 – time for moving of the train in direction “full” or direction ”empty”, respectively;
τ – time for maneuvers for permitting the train in both directions.

In the case of a single line the permissible ability is:

$$N = \frac{T - t_{регл}}{t_1 + t_2 + 2.\tau} \qquad (19)$$

where:
N – ability of the portion to permit trains;;
T – duration of shift;
$t_{регл}$ – regulated idle time. For a 24 hour day the idle time is from 180 min to 300 min.

The importance of time for moving of trains is evident related to portions of movable railroads from (19). It depends on the speed, and speed depends on technical condition.

Thus an effect is exerted on haulae ability “M” of different portions and combinations between them, crossing the capital trench.

$$M = \frac{N}{f}.n_в.q_м \qquad (20)$$

where:
f – coefficient of reserve
$q_м$ – useful loading of the train;

The following approaches are applied to increase the haulage ability of any route:

- increase the speed of train movement. In the portion , depending on movable railroads that may be achieved only by realization of engineering activities, which will significantly incrase their stability;
- increase of weight of train. It may be achieved (in the portion, depending on movable railroads) by the realization of activities, icreasing the stability of lines, i.e. the upper case or increasing the number of locomotives (coupled haulage capability) and aplication of motor-car integrated sections;
- improvement of the means of moving, opening and closing of railroad switching arrows for formation of specific routes. That is applicable to our conditions, however, each further improvement of those systems is expensive and senseless in case all other activities for improvement of stability of movable railroads have not been performed before.

The entire above presentation shows that technical condition of movable railroads effects synthetically through the admissible speed. For the example in fig. 3 It may easily be shown that if speed is increased to 15 km/h (compared to admissible speeds – see the initial page), then the time for one route will be reduced with 40 % and numbr of routes done by one train will be increased with nearly 50 % etc.

This may be digitally shown for all the technological parameters of haulage system, shown by the dependencies (12), (13), (14), (19), (16), (17), (19) and (20). In a reverse aspect the effect on important technological parameters of mine equipment – technical, shift, week productivity of single bucket excavators may be revealed.

CONCLUSION

The dependencies are presented, where speed of motion of trains on movable railroads, directly or indirectly, effects on the technological parameters of haulage.

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Recommended for publication by Department of
Opencast mining and blasting technologies, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.51-55

SPECIFIC MOMENTS OF HISTORY OF OPENCAST MINE "BELI BREG"

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ABSTRACT
After nationalisation in year 1907 to the end of 50-th years and early 60-th years the mine "Beli breg" (then "Bolshevik" was one of the biggest mines in Bulgaria for coal mining. The most interesting moments of discovery and exploitation of lignite deposit "Beli breg" and "Pedeliste" are treated in the report.
The approach of development method is very interesting – first underground and next – opencast. The production capacity of the mine from the beginning to year 2000 is shown. The low experience in development of this type of deposits, where are faults and ancient landslides, calls three times for change of development direction in the western section of the Mine "Beli breg" in order to prevent the catastrophic landslides. The technological solutions, realised for the first time in Bulgaira, are described. All described in the paper will enter into the history of opencast coal mines of the country.

During development of mine "Beli Breg" are separated three main periods: the first one is the opening, coal prospecting and its exploration up to the nationalisation in 1947; the second one - its exploitation after nationalisation and the third one – after privatisation which just started.

It was established that coals in the mine are typically lignite ones with sufficient hardness and toughness. Its average ash content is about 30% and the moisture – about 50%. The structure is inhomogeneous, because of significant quantity of clay bands and accretions. The average thickness without bands is 17.62 m and with bands – 18.66 m. As it can be seen form the stratigraphic range (fig. 1) in the eastern part the number of band achieved 5-6, the thickness – from 0.2 to 0.6m. In the rest locations their number is 1-2.

FIRST PERIOD OF MINE DEVELOPMENT

In the fateful for Bulgaria year 1885 the energetic villager Alexi Mitanov from the village of Tsatsarovtsi listen that in the field of Velin Bozilov form the village of Jalbotina near the river Chakanets (Jablotina "the earth smokes" without burning of straw or wood.

The whole 25 years Alexi Mitanov tenaciously searched contractors and researchers for exploration of this "burning earth". Only in 1910 shortly before the Balkan war he found the lawyer Todor Ahtarov from Sofia and succeeded to persuade him to visit Burela and to see himself Burela's lignite. As a lawyer of experience, before the visit, Ahtarov put in touch with the than Ministry of Trade, Industry and Labour (MTIL) and from there recommended Eng. Bogomil Radoslavov as an expert on coal problems. Together they visited and surveyed the outcropped bed and they assumed that Mitanov have been right and there ware excellent conditions for mining of lignite coal. After this visit, in the beginning of 1911, MTIL issued to lawyer Ahtarov licence to "search" coal and perimeter which spreaded the both sectors – eastern (Tsatsarovtsi) and western (Jablotina). In 1915 lawyer Ahtarov got the second licence, which enlarged the perimeter for coal exploration and it included the section Nedelishte.

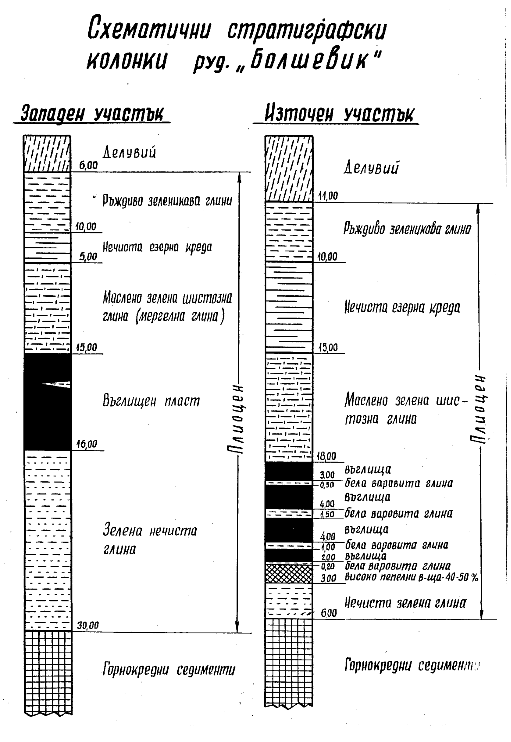

Figure 1. The stratigraphic range of Bolshevik mine

That time Krum Ganchev became their partner. In 1915 started lignite underground mining by using of galleries towards the bed in the locality Mechkaritsa – on the southwest of today's separation. Than the mine was called "Beli breg".

A new period of mine life is driving of trench for opencast mining in 1919 in the locality Mechkaritsa (fig.2) by the Krum Ganchev workgroup, but it is undoubtedly proved, that it was Todor Ahtarov's idea.

The opencast mining is made in this mine for the first time in Bulgaria. As a result fields, meadows and other private areas were liquidated and Ahtarov was forced to ask the Mininstry for a new licence. In this year started making of statistics of mined coal quantities (table 1) and was mined and distributed 490 tones coal. From fig. 2 is seen, that the size of coal pieces in the cars was from 120 to 150 mm and more and it was clear coal. Coal was mined by hand, through the niches with length from 4 to 6 m and width form 1.5 to 2.5 m. The niches were worked off form above downwards on layer. The work instrument was ell sharpened pick – with cutting part and a prick. Besides different steel chisels, axes, pricks etc. were used for crushing the obtained plate coal and for separation from the massive. The pick undertake the coal towards the bedding with the thickness about 5 to 8 cm, than were crushed to the size about 150-200 mm and were pushed aside on the vertical slope of the bench. When coal fallen down part of them were crushed. Parallel to the bench, on the distance form the niche in its bottom end was mounted a railway with width 500 mm) for cars used in the underground mine "Beli breg". The distance between the front wall of the niche and the axis of the railway was about 1.5 to 2 m.

On this ground were deposed fallen coals from the upper parts of the niche. The coals were loaded by hand with spades. When the cars were loaded, in the mining part of the niche work was stopped. At last were loaded small-sized coals, which were used for fuel in the power station. When the work surface of the niche became half a meter, the coals were loaded direct in the cars. They worked this way up to the bottom of the horizon. On fig. 2 were shown two mining niches, which were reached the bottom of the horizon – one on the left part of section to the right – where were seen large-sized coals. Loaded with coals cars were pushed to the beginning of the skip trench, were suspended on the pulled rope of the winch and were pulled on the ground and there were unloaded by special wipers in bunkers. Most of them mechanically were feed to the storage for drying. The storage has volume 700 tones. This mean storage provided work of the mine in the whole year, even in the winter. So the dried coals with low ash content reached caloricity of 2200 to 2500 kilocalories of the kilogram flow. After burning there were a little wood ash.

In 1938 the trader Boris Galabov obtained licence mainly for investigation of section Nedelishte, but he explored the both sections of the mine "Beli breg" So the competition appeared. Galabov droved 40 m gallery towards the bed, in the section western of Nedeliste village. He made investigations up to 1947. In 1942 he drived another gallery in the bed section Nedeliste and distributed about 300 t coals. The new created small mine in the section Needieste later was called "Galab". Near to the entrance was constructed sawmill, too. In 1945 the "Galab" mine distributed 665 tones, and in 1946 – 227 t.

Figure 2.

Table 1

Year	Annual production t	Year	Annual production t	Year	Annual production t
1911	*	1941	1,174	1971	1016,246
1912	*	1942	5,886	1972	1043,976
1913	*	1943	6,550	1973	1135,016
1914	*	1944	3,325	1974	963,027
1915	**	1945	4,000	1975	830,056
1916	**	1946	**	1976	697,214
1917	**	1947	**	1977	662,800
1918	**	1948	26,413	1978	459,381
1919	490 t	1949	75062	1979	522,384
1920	1360 t	1950	122,413	1980	670,288
1921	345 t	1951	232,066	1981	724,045
1922	59 t	1952	310,068	1982	798,771
1923	68 t	1953	475,879	1983	566,559
1924	50 t	1954	321,976	1984	616,902
1925	40 t	1955	483,521	1985	755,961
1926	376 t	1956	611,435	1986	698,189
1927	1,210 t	1957	876,984	1987	588,771
1928	3,202 t	1958	1324,685	1988	658,687
1929	1,552 t	1959	1673,831	1989	626,468
1930	1,652 t	1960	2014,144	1990	554,500
1931	1,240 t	1961	1744,529	1991	405,972
1932	2,346 t	1962	1669,072	1992	570,072
1933	1,696 t	1963	1779,652	1993	510,336
1934	2,216 t	1964	1941,507	1994	513,403
1935	1,937 t	1965	1358,805	1995	461,915
1936	1,597 t	1966	1246,113	1996	425,317
1937	1,154 t	1967	1245,490	1997	360,349

1938	1,705 t	1968	1330,372	1998	417,106
1939	1,650 t	1969	1357,556	1999	473,018
1940	1,932	1970	1270,473	2000	443,024

So, in conditions of loyal competition between the concessionaires – Krum Ganchev and Boris Galabov enlarged coal mining - in mine "Beli breg" opencast mining and in "Galab" – underground mining. Year after year the necessary lignite coals were mined from both mines (opencast and underground) and were well distributed on the market and realised good profits.

Form 1919, when was registered the first serious coal mining to the end of 1947 – the year of nationalisation, were mined 48 812 t coals, which is 0.12 from the total quantity mined coals (44100000 t) for the period 1919-2000 (table 1).

THE SECOND PERIOD OF MINE DEVELOPMENT

The second period started after nationalisation, so called period of state management of the mine, which was about 50 years.

For 1946 and 1947 there were no data about mined quantities of coal (Table 1). We could only suppose that the mining was carried out by hand. In 1947 Krum Ganchev and his son Boris Ganchev left the mine. The mine "Galab"in section Nedelishte stopped work, too.

In 1948 coal mining in mine "Beli breg was mechanised and overburden removal was made by our technology and the technology was applied in the opencast sections of mine "Pernik", the Check electrical excavators "Shkoda" were used, loading steel bunkers and belt conveyors (BC) with wooden construction and width of the belt 800 mm – for coal; for the overburden were used the same excavators and railway transport (tracks spacing 900 mm) and carriages (type Pernik) with small steam locomotives made in GDR. The year productivity in 1948q 1949 and 1950 were respectively 26413 t, 75062 t and 122413 t.

The company was developed according to the general project made by Projecting Institute "Lekgiproshaht", Leningrad, USSR. The head of the project was Maria Nikolaevna Demidova. This project concerned only Eastern and Western sections, not the section Nedelishte.

In the beginning of the 50 years the managing and working staff was mine technician and practice workers, in the beginning of the 1958 there were new personnel – mining engineers, who have been completed the Institute of mining and geology. To the end of 60 years all sections were full of mining engineers and mine electromechanics.

The more important moment in 1958 is unification of most of sections in a common Western section, managed by one chef of section. Besides the gravelling of the temporary roads in the overburden levels were made in the Western section. The premium- progressive system of payment was introduced for each worker. This way the section started to fulfil its planes. Eng. Borislav Zahov, Georgi Petrov etc., together with the Director Koev, carried out this reorganisation.

In the mine was used moment blasting as well as millisecond blasting technologies with shaking crashing of coal beds and good work of the excavators. The blasting works were carried out in the section "Drilling and blasting works".

Organisation of mining of overburden and coal is the following: the front of mine works is divided in two flanks – northern and southern, each with length about 300 m. For example, in the northern flank form the middle of the front, excavator ESH-4/40 mines the overburden on one bench throw it across in the worked area and backfilled the front of the coal layer. When the excavator reached the end of the flank, it moves to the internal dump in return travel to the centre of the front, handled the overburden in the worked area and cleaned the front of the coal layer up to the bottom. This way the excavator opened the necessary coal reserves for mining.

In the mine was realised the system of without transport mining (fig.3) with whole backfilling of the front of layer (scheme 2B on the fig.3) with overburden.

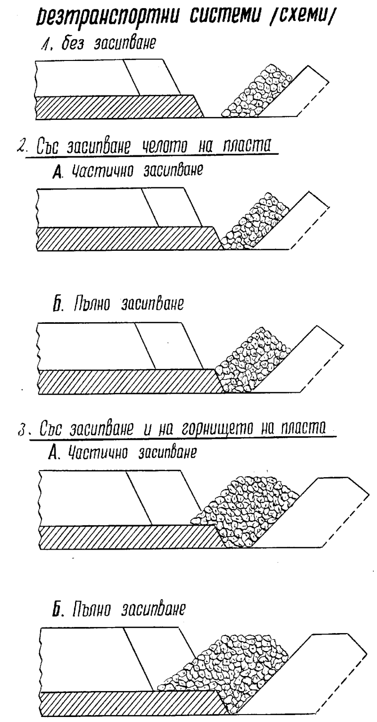

Figure 3. Without transport system of opencast mining

Organisation of mining of overburden and coal is the following: the front of mine works is divided in two flanks – northern and southern, each with length about 300 m. For example, in the northern flank form the middle of the front, excavator ESH-4/40 mines the overburden on one bench throw it across in the worked area and backfilled the front of the coal layer. When the excavator reached the end of the flank, it moves to the internal dump in return travel to the centre of the front, handled the overburden in the worked area and cleaned the front of the coal layer up to the bottom. This way the excavator opened the necessary coal reserves for mining.

In 1958 the stope belt conveyors and the main transport in the Western section were replaced with metal construction with belt width 1200 mm. In 1959 entered second mine excavator in the western part. The technology is the same as in the Eastern section. The coal bed is about 20 m and was mined on two benches by both excavators Shkoda E 25. As it could be seen in table 1, in 1959 the yearly coal mining increases to 1637,831 thousand tones, i.e. when work two excavators in the Western section and one in the Eastern. In 1960 the same number of excavators were used, but with better organisation of works in both section, were mined 2014,144 thousand tones coals. This is the maximal quantity of mined coals during the whole history of the mine. It should be mentioned that in the middle of 1960 the new governing body of the mine organises the works in the mine successfully in order to obtain the mentioned above results.

In the same year started the filling of the internal spoil heaps of the Western section. Loading of trains full of overburden was made through the satiation Dispecherna.

In 1962 in the Western section happened the largest landslide in the mine history in the working slope. Slide down about 5 mil. Tones coals and rocks. The opened coal reserves in the large areas were put aside by the overburden to the big fault. The landslide opened all coal reserves and the Western section worked on it two years.

According to the project "Lengriproshaht" the Western section finishes to the bud fault and the coals after the fault could be left of mined opencast.

The governing body of he mine decided to mine these coal reserves and this way it signed its verdict. For opening of the out-of-balance coal reserves (30-40 m deeper occurred) is necessary additional mechanisation and time.

In order to compensate the governing body of the mine began mining of pillar under correction of Checanska River, the road to the village Jalbotina and 5th burrow line. The line was dismounted quickly and the road for village Jablotina was closed.

The Western section looked like a large mined are and in June 1964 the river Checanska passed through the section, and in November the water began to flow in the channel, made in advance, to the river Goljama. On the east of the pillar and on the west of the deep coals, the overburden was transported and filled in the lake by tip-lorries.

For about two years from the pillar were mined 600 thousand tones coals with the best quality. Than the mined pillar of the section was so filled that the river Checanska returned to its previous bed.

The new technological conditions in 1964, the volume of opening works increased 4 times – from 2 mil. m^3 to 8-9 mil. m^3. The mine finished the year with non-completed planes. To the middle of the 1965 the situation is the same.

In June, on the mine territory was carried out a meeting of the association of the Ministry of Energetic, Bureau of OC of BKP, Bureau of CC of miners, The governing body of the Company of the First in the Industry. On this meeting were considered the biggest problems of Mine. All was at one with the necessity to help the governing body of the mine. For the following six months the plane was corrected (decreased with 400 thousands t) and was granted funds for the new excavators and tip-lorries.

The new moment in the development of mine started. The new excavators and tip-lorries began to work. Nevertheless the Western section went to his end and it should be though about new perspectives, i.e. to prepare the mine Nedelishte.

A meeting was made and Minproject started projecting of a mine. Economically a part of bed of Nachevska river was corrected, the people of village Nedelishte, situated on the coal layer, was moved. The belt conveyor from separation to the cutting trench was made. In 1968 the mine Nedelishte started his work.

In underground mining the problems were firedamp and fires, and in opencast – landslides and flooding. In the western (Tsatsaritsa) section occurred the permanent landslides, in eastern - flooding, and in the Nedelishte mine – both of them. A lot of problems caused transport distances for tip-lorries, which transported overburden that consists of wet and sticky clays.

The most of works in mine Nedelishte were made by own automobile transport.

In 1970-1975 the governing body of the main paid attention to the mine Nedeliste , where the coal reserves were bigger. The correction of river Goljama started. It was moved to the north about 3.5 km from its bed together with the road Dragoman-Tran.

Than were mined the coals northern the big fault, in the Western section – about 3.5 mln. tones. The mining in Tsatsaritsa section decreased, when on 12.02.1974 was occurred the landslide. Later, in 1990 the mining in the section Tsatsaritsa was stopped.

The main consumers of energetic coals began to use masout and natural gas. The coals form the mines "Bolshevik", "Katina" and "Stanjantsi" were directed to the TEPS "Maritsa istok 2". In 1973 the mine "Katina" stopped.

In 1978 the in the mine "Trojanovo 2" occurred two landslides of the working slope. This is the appropriate moment instead to stop mining in the mines "Bolshevik and "Stanjantsi" to transport the coals to the TEPS " Maritsa istok". The transportation was made to 1996. This time the coals were used in the TEPS "Bobov dol", too. The expensive coals from Bobov dol were mixed with cheep lignite coals. Form 1996 the coal form both mines are used only in TEPS "Bobov dol".

THE THIRD PERIOD OF MINE DEVELOPMENT

On February 2000 is registered the Joint stock company "Beli breg-Burel". In the same year started the privatisation procedure. There were two candidates on the auction – the "Beli breg –Burel" and "Minstroi Holding". The contract was obtained by "Beli breg – Burel".

On April 2002 the Corporate Trade Bank sold the "Beli breg – Burel" to company "LM Impex", represented by Christo Kavachki.

Eng. Asen Goranov- Director and Eng. Gancho Ganchev – head engineer manage the mine now.

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Recommended for publication by Department of Opencast mining and blasting technologies, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.57-58

MINE SURVEYING CONTROL OF THE CONCESSION GRANTER

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ABSTRACT
Subsoil protection and rational use of ores and minerals are basic responsibilities of the concession granter. These obligations can be fulfilled only by spatial-geometrical fixation of the exploration and extraction operations in accordance with the designation of the concession. The spatial-geometrical fixation is carried out only by mine surveys. Law does not regulate the mine surveying control of the concession granter. In this connection, in 2002 a team of experts from the University of Mining and Geology was commissioned to draw up a draft proposal for effective mine surveying control on the part of the concession granter.

The concession for extraction of ores and minerals, known since 1891, was renewed several years ago – Art. 4 (2) of the Ores and Minerals Act (OMA). Two legal persons again came into being: a concession granter (the Council of Ministers or the respective authorized ministry) and a concessionaire (the person carrying out the extraction of the ores and minerals). The control of the concession granter is multi-directional but concerned mainly with the subsoil protection and rational use of ores and minerals for the following reasons:

- The non-renewable nature of ores and minerals given the present mining and primary processing operations;
- The ores and minerals belong both to the present and future generations so that human development will be disrupted if we disregard the public benefits from the subsoil;
- Huge investments are demanded to prepare the extraction of ores and minerals whose return requires a long period of time;
- In the ex-Soviet bloc countries the money for exploration, development and preparation for extraction was provided by all taxpayers that presupposes maximum dividends from the development of a given mineral deposit;
- The degree of technogenic transformation of a certain geogenic deposit determines the risk of anthropogenic and ecological disasters.

The inspection agencies of the concession granter responsible for subsoil protection and rational use of ores and minerals are obliged to establish the facts objectively and report the result s of the inspection under Art. 92(1) of OMA.

Practice dictates and Art. 81 (1), (2), (3) and (6) of OMA specify the documents (surveying maps, cross-sections, records and statistical data) that have to be kept for each mine and quarry with respect to subsoil protection and rational use of ores and minerals. These documents serve as a basis for the inspection agencies of the concession granter to fulfil their control functions in respect of the rational use of ores and minerals. The documents contain concepts related to two groups of categories – the current output report (statistical data) and the mine surveying report (shortly mine surveying control).

These categories have to be specified. The current output report presents data on the ores, minerals or rock mined per shift, day, month determined by the number of transport vessels and the mass in each vessel or directly weighed during the transportation from the mine excavation. Mine surveying is a complex of surveys of the mine excavations, spoil heaps, loaded transport vessels, etc. The survey results are used for preparing the mine surveying report on the state and movement of the industrial reserves, losses and contamination in the subsoil, the mineral extracted and the volume of mining operations performed for a definite period of time.

Practice dictates and the normative base specifies that the mine surveying control is a basis for determining the state and movement of the industrial reserves and the volume of mining operations. The following can also be adduced to support the statement about the advantage of the mine surveying control over the current output report (control):

- Mine surveying reflects the actual state of the industrial reserves. It is objective and permits the storage and reproduction of objective data;
- Surveying maps and graphic documents have acquired the force of legal evidence both in the Bulgarian and international practice due to their objectivity.
- The quantitative and qualitative parameters of the mine surveying control are a proper basis for determining the level of utilization of ores and minerals and subsoil protection.

The mine surveying control of the concession granter concerning the industrial reserves in a chronological sequence requires the implementation of the following stages:

- <u>Stage A.</u> Determining the geological, geodetic-mine surveying and mining-technological status of the site.

- Stage B. Establishing the norms used by the concessionaire (sector and specific norms set by the concession granter).

- Stage C. Assessment of the mine surveying operations on the site.

- Stage D. Check surveys, calculations and graphic constructions.

- Stage E. Drawing conclusions about and evaluation of the concessionaire's activities.

The concessionaires, according to Art. 22 of OMA shall provide to the Ministry of Environment and Water information about the state and changes in the reserves and resources in the granted areas as well as the geological and technical documentation required for inspecting their activity every year or at request but not more than twice per year.

The conclusions about the concessionaire's activity shall involve all activities related to the granted concession under the Ores and Minerals Act and the concession contract signed. They shall express an opinion on the methodological execution of the mine surveying operations and observance of specifications and instructions.

The mine surveying control shall result in preparing documentation that contains quantitative and qualitative characterisation of the control activities performed, description of the inspections carried out by types of operations and the omissions, inadequacies and errors found. If necessary, proposals are made to eliminate the established defects.

Over the last decade no control has been exercised on the mine surveying activities by administrative bodies. This resulted in interested companies and organisations commissioning control determination of volumes of extracted and/or transported subsoil material in case of dispute as a rule. Except for two pilot sites, no mine surveying control has been exercised on concessionaires by a concession granter. On the basis of experience, the following conclusions can be drawn:

1. The structural changes in the mining sector eliminated the three-level organisation of the mine surveying activities and their control. The lack of agencies that can exercise control on the activities of the mine surveyors in the mining companies resulted in a number of negative consequences, the most important being the concessionaire's activity outside the concession area.
2. Conflicts have occurred as a result of slow updating of the normative base for performing mine surveying activities in compliance with the newly passed laws and regulations in the field of exploration and extraction of ores and minerals. This creates problems for the mine surveys, the control on the rational use of ores and minerals, the protection of buildings and structures from the harmful effect of mining operations, the environmental conservation as well as the coordination of activities with other organisations and departments.
3. The efficient control on the concessionaire for carrying out the mining operations in space and time, the extracted ores and minerals and their rational use is impossible without the complete, accurate and timely elaborated mine surveying documentation.
4. The international and national mining practice accepts the results of the mine surveys, calculations and graphic constructions as a basis for accounting and determining objectively the actual quantities of mined material and redeemed reserves for a definite period of the concession contract. They also serve as a basis for fixing volumes of activities and payments and are legal evidence in solving disputes.
5. Before the existing and future legislation has been united in a special System (Codification) regulating the prospecting, exploration, extraction and primary processing of the ores and minerals, it is more appropriate to subject certain opinions and proposals to public discussions. The participation of specialists from the mine surveyors' guild in these discussions is motivated and advisable because there is a great number of university disciplines that they study during their training. These include the scope and aims of mine surveying – determination of the conditions for extraction, use and protection of the subsoil resources and environment based on modern, reliable and objective information about the spatial and temporal position of a certain element in exploring and/or mining the mineral deposit.

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Recommended for publication by Department of Mine surveying and geodesy, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.59-64

INFLUENCE OF THE PILLAR GEOMETRIC FORM OVER THE LOAD BEARING CAPACITY

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SUMMARY

The weak points arising upon parameter design of the open stope and pillars give us the challenge to overcome them developing new scientific–based, up-dated and precise methods of analysis and design. Consequently such methods are developed on the basis of profound theoretic and experimental study of the problem on the proper room and pillar measuring under specified mining and geological conditions.

Specifying the mining and geological parameters of the system, such as:width of room, width of pillar, height of pillar and the respective correlation at each separate step of the mining process, makes it possible to reach and to provide the mechanical completeness of the supporting pillar and maximal ratio volumetric.

For that purpose an adequate method of measuring of the gypsum pillars upon analysis of the geometric parameters of pillars, and particularly applying the method of the subordinated area has been suggested.

In the present study has been demonstrated a principal procedure how such a task could be accomplished under the conditions in the "Koshava" mine plant.

INTRODUCTION

Last decades are described as to have marked a considerable boom regarding the clarification of the pillar stability mechanism. Relevant to that, the problem of the proper measuring of the pillars could formally be treated as a complex decision including the solution of 2 problems.

The purpose of the first complex problem is to define in a theoretical way the pillar load bearing, going out from the accepted hypothesis.

The second purpose is related to specified empiric estimation of parameters of the collapse of pillar. To find the proper solution, numerous factors such as: strength properties of pillar, geometric parameters of pillar, flow properties, kind of contacts etc. could be the outgoing subjects.

In the present report the problem is referred as to the geometrical parameters of pillars in the open stope and pillar of exploitation.

GROUNDS OF THE PROBLEM

The exploitation of the fields and the underground equipment suppose that various pillars acting as natural support, barrier, wall of some peculiar units of the underground constructions, providing the normal functions of the mine plant. are left in the bowels of the earth.

The principal design of the pillars is to guarantee the maintenance of the works within not limited long period of time, in order to get use of them for the second time, as well as to perform a main function in the management with the rock pressure. Indeed the experience has proved that the bad influence on state of the rock massif due to blasting works, weathering and flow processes might come up upon caving after 30-years, or sometimes even earlier.

To supply a full economic evaluation of the negative consequences of such sudden collapses would hardly be possible. However in all those cases the losses of the mining plants sustained in the respective countries are reported to be quite big.

At present the underground gypsum extraction as a rule is made using the versions of the open stope and pillar of exploitation, mainly upon leaving rib pillar room fender and keeping maintenance of the mined area within not limited long period of time.

The decision on the exploitation version of the gypsum fields is believed to result from the choice of 3 factors:

- Relatively wide spread of the gypsum fields, bedding not quite deep;
- Big thickness (10-30 m) and more;
- Slope bedding of the gypsum layers;
- High resistance of the open-mining areas.

The small volume of the development and undermining cut operations, the gypsum extraction in works of big section, comparatively not difficult mining extraction /yield/ related only to the explosion breaking and carrying away the mined mass mainly by road transport, the absence of some developments of the rock pressure during the exploitation in the works – are known to be the main reasons determinant the comparatively low cost of the mined gypsum mineral.

Indeed the applied technology in the practice is going along with a number of disadvantages. As a main disadvantage are considered the big losses sustained of the mineral in the pillars (sometime reaching up to 70% of the balanced stocks) and the inevitable caving of the mined area.

The main design of the gypsum pillars left is to provide the maintenance of the works within not limited long period of time in order to get use of them for the second time. However the examinations in the practice have demonstrated that due to the bad influence of drilling and blasting operations, the weathering and the flow processes, rock caving is known to occur as early or as late. As examples could be pointed out the caving in the mine fields of Nikitovsko, Dekonsko, in Donbas, Port-Maron and Chante Le Vou in France, Koshava in Bulgaria.

The purpose is to exploit the gypsum mine in a proper way and to provide comparatively long lifetime of the extracted areas with the open stope and pillar of exploitation and to do it in a way to prevent from uncontrolled caving of the earth surface through out the years. Such an uncontrolled earth caving could last uncertain period of time while the gypsum reserves left in the pillars are in such a shape that makes the repeated stopping either not possible or economic not efficient.

All stated above proves the necessity to study the machinery and technologies of the gypsum mining and the relevant methods on parameter calculations of rooms and pillars.

SUBJECT OF THE SCIENTIFIC RESEARCH

The subject of scientific research in the present study is referred as to the analysis of the geometric parameters of pillars in “Koshava” gypsum field, by means of theoretically study of the problem on the pillar load bearing capacity and to find out an adequate method of calculation.

The gypsum environment has clearly expressed flow properties, which are to be found mainly in the behavior of pillars, put under long lasting and very often changing exertion. Several cases of considerable flow-shifts and deformations have been registered. The long –term experience of Koshava mining exploitation has pointed out that upon building of solid enough pillars,is being created a core of confinement, which can guarantee the stability of “room-supporting pillars” system. However in that case the extraction ratio drops considerably. That leads to inefficient exploitation, which finally cuts the life – time of the mine.

The technical and economic efficiency of the open stope and pillar of exploitation is closely linked to the dimensions of the pillars. The economic factors require that the technical concepts assure the most efficient extraction of the mine stocks. However the economic factors are against the requirements of maximal safety of labor, which force to work at higher safety ratio, and that provoke the enlarging of the pillar dimensions.

In table № 1 is shown the relation between the parameters of the open stope and pillar of exploitation and the appropriate losses of gypsum, left in the pillars and the protective top and floor benches in the rooms. It is obviously that in cases of gypsum extraction at the full sickness of the layer, the extract ratio is going to be increased and reaches 60-70% (the mine fields of Kamsko-Ustinsko, Gorazubovsko), (Usachenko, B.M., 1985).

The recent 100 years are rich of a number of analytic and experimental explorations carried out on the matters concerning the rock pressure in the open stope and pillar of exploitation. Some authors have developed analytic methods for evaluation of the loading and load bearing capacity of the pillars for the conditions of horizontal and sloped mine- fields. The more profound the objective law of development of the rock pressure is known, the more precise and real the evaluation of the pillar parameters will be. The investigations in the recent years based on scientific experiments in labs and industrial conditions have concentrated mostly on the digital modeling and universal study of the physic-mechanical and geological properties of the rocks. The loading and deformations of the pillars as a function of the rock pressure is said to be dependant on number of factors, such as:

- The depth of mining works;
- The correlation between the surface of pillars and rooms ;
- The composition of the covering rock layers;
- Their bending resistance;
- The composition, strength, and deformation-properties of the mineral;
- The flow properties of the rocks;
- The pillars location in the mined area.

The latest examinations on the load bearing of pillars are to be found in different works of various contemporary explorers, which are not completely generalized.

One of the most precise empiric relation expressing the effect of the volume and geometric form of the pillar toward the pillar strength is that of Hardi and Agapito(1977):

$$S = S_0 .V^a.(W_p / h)^b = S_0 .V^a .R^b$$

Referred to as follows:
V – volume of pillar, m^3;
W_p - width of pillar, m;
h - height of pillar, m;
S_0 - parameter of strength, representing the mined area, MPa;
R – correlation width of pillar toward height of pillar., (Brady, B.H.G., E.T. Brown, 1993)

The shape effect arises from three possible sources:

- Confinement which develops in the body of a pillar due to constraint on its lateral dilation, imposed by the abutting country rock;
- Change in pillar failure mode with change in aspect (i.e. width / height) ratio;
- Redistribution of field stress components other than the component parallel to the pillar axis, into the pillar domain.

ANALYSIS OF THE GEOMETRIC PARAMETERS OF THE ROOMS AND PILLARS IN "KOSHAVA" MINE PLANT

The open stope and pillar is the principal one, which the works in Koshava mine exploitations have followed from the very beginning till now. It should be highlighted that the system has been applied in different versions and that is believed to determine different behaviour and different operation conditions of the composition units: room – pillar – roof - floor. For different versions of the open stope and pillar, the versions applied are as follows:

- Various forms and geometric parameters of pillars.
- Various forms and geometric parameters of rooms.
- Various conditions of the extracted area after finishing their exploitation.
- Various sequence in extraction of the room mine stores.

The variety of applied versions of the open stope and pillar exploitation system, has resulted in certain kinds of pillars, divided according to the form and geometric size, which are to be found in the mine as follows:

- Square pillars, with dimensions of 48x48 m;
- Square pillars, with dimensions of 20x20 m;
- Rectangular pillars, with dimensions of 20-30 m;
- Rectangular pillars, with dimensions of 19-27 m;
- Continuos pillars, with length 110 - 120 m, width 16 - 18 m;
- Continuos comb-shaped pillars, with length 110 - 120 m, width 18 - 19 m and narrowing - 6 m;
- Continuos comb-shaped pillars, with length 110 - 120 m, width 18 - 20 m and narrowing – 12 m;
- Barrier pillars with dimensions 120 -120 m.

According to the condition of the extracted area after finishing the exploitation, the rooms in the mine plant are known to be with filling and without filling.

Parameters of the open stope and pillar upon gypsum exploitation
Table № 1

Mine field	Depth of Bedding	Thickness of layer	Angle of dipping	Length Of Room	Width Of Room	Thick-ness Of safety bench	Pillar dimensions	Extract ratio
	m	m	°	m	m	m	m	η
Koshava Bulgaria	297	30	18	12	7	3	15-17	0,36-0,40
Shoals- USA	105-158	30	6	3,6-5,2	7,5-9	-	6 x 9	-
Stemp Hil-England	30-270	24,4-36,6	6	9,14	4,88	-	7,2 x 7,2 6,4 x 12	0,5-0,75
Britling-England	40	2,1-6	-	1,8-4,8	6,4	-	6,4 x 6,4	0,56-0,79
Taverni-France	45-80	9-12	0	7,5-9,5	8	2	16 x 16 4 x 4	0,40-0,65
Port-Maron-France	70-80	8	1	6	7,5	2	6,5 x 6,5	0,70
Gorazobaerk -Ukraine	150	10-16	5-8	9-14	8	0	5 x 150	0,61
Olekminsk-Russia	35-65	6-11	8-10	6-7	10	1-2	6 x 100	0,42-0,46
Beblovska-Russia	90	4,4	0-3	3-4	8	1	4 x 10	0,38-0,40
Artemovska-Russia	100	25	3-7	15-18	8-11	5,5	12 x 30 12 x 100	0,32-0,42
Neomoskovska-Russia	120-130	17	0-2	9-11	10-11	6,5-8,5	9 x 50 10 x 50	0,34-0,36
Kamsko Ustinskaja-Russia	100-130	12	0-1	10-12	15	0	12 x 20	0,62-0,70
Dambo Main-Canada	116	1,5-2,4	5	-	6	-	6 x 6	-
Obrigeim Germany	-	12-13	2	11,5-12,5	10	-	8 x 15	0,74

To be in position to follow easier the analysis of the geometric forms of pillars, an experimental task to divide the mine plant in districts has been undertaken. Consequently 3 districts have been established which on turn according to secondary attributes have been subdivided in few districts. There are 2 main factors which are known to be taken into consideration upon split of the mine: creeps through out the years, and the appropriate version of the applied open stope and pillar (with or without backfilling).

Analysis of the geometric parameters of pillars in the area of creeps fields (district 1)

The district is situated in the northeast part of the mined area. Three creeps have been registered in it as follows:

- In 1976 sudden caving of rectangular pillars have appeared, (1A);
- In 1991, in the district of bore – hole № 24 arised a self – caving, known as chimney-like heading which has reached the surface (1B);
- In 1992 a mass creep of square pillars in the district of drift № 14 has come up, (1C);
- On 05.03.1992 has been registered a creep in the extraction area of the district known as "Bermudian triangle" (in the north part of the mine field). The creep arised along with self - caving known as chimney-like heading. To prevent from further development of that chimney - like heading to the surface, certain walls of reinforced concrete have been built and a partial backfilling with dry sand has been made in the extracted areas, (1D).

Moulds of subsidence spread out wide and deeply in the area are believed to be the result from the creep at the surface. It is necessary to stress on the fact that waters from quaterner water-carrying horizont did not penetrate in the mine -field. The land did not become boggy and the soil productivity has been preserved. There will be no more extraction works in the district and it should be liquidated by backfilling the not caved existing rooms left under the village of Koshava. The observations for assessment of the surface conditions in "Koshava" mine should go one. The necessity to study the deformation processes has a principal importance for assuring a trouble free development of the mine works as long as the mine exist.

District 1A

It is situated to the north of the main extract haulage drift 3. The working parameters are given in table 2.

Basic geometric parameters of district 1A
Table 2

PARAMETERS	VALUE
Thickness of gypsum seam	18,6 m
Depth of mine works	263 m
Specific weight of gypsum	0,0223 MN / m^3
Width of pillars	16 m; 18 m
Height of pillars	14 m
Width of rooms	7 m
Height of rooms	14 m
Length of rooms	200 – 240 m

The ratio of width toward height R =W_p / h_p for the specified dimensions is as follows:

R = 16 / 14 = 1,14
R = 18 / 12 = 1,5

District 1B

It is situated round a bore – hole № 24 and rooms № 55, 56, 1/14, 1'/14, 1"/14. Rectangular pillars of dimensions 20 x 31 m and 19 x 27 m are available.

Basic geometric parameters of district 1B
Table 3

PARAMETERS	VALUE
Thickness of gypsum seam	19,6 m
Depth of mine works	260 m
Specific weight of gypsum	0,0223 MN / m^3
Width of pillars	19 m; 20 m
Height of pillars	15 m
Width of rooms	7 m
Height of rooms	15 m
Length of rooms	20 m

The ratio of width toward height R =W_p / h_p for the specified dimensions is as follows:

R =19 / 15 = 1,27
R = 20 / 15 = 1,33

District 1C

It is situated round a bore – hole № 14. Square pillars of dimensions 20 x 20 m are predominated.

Basic geometric parameters of district 1C
Table 4

PARAMETERS	VALUE
Thickness of gypsum seam	19,6 m
Depth of mine works	260 m
Specific weight of gypsum	0,0223 MN / m^3
Width of pillars	18 m; 22 m; 24 m
Height of pillars	15 m
Width of rooms	7 m
Height of rooms	15 m
Length of rooms	18 m; 20 m; 30 m

The ratio of width toward height R =W_p / h_p for the specified dimensions is as follows:

R = 18 / 15 = 1,2
R = 22 / 15 = 1,47
R = 24 / 15 = 1,6

District 1D

It is situated in the middle part to the north of a panel drift 3.The registered creep of the extracted area is going along with a self-caving structure, known as chimney-like heading. To prevent from further development of that chimney-like heading to the surface, certain walls of reinforced concrete have been built in the extraction rooms in the area of creep and a partial backfilling with dry sand has been made.

There will be no more extraction works in the district and the remaining part of the mine stores should be registered as items out of the balance sheet.

Basic geometric parameters of district 1D
Table 5

PARAMETERS	VALUE
Thickness of gypsum seam	19,6 m
Depth of mine works	210 m
Specific weight of gypsum	0,0221 MN / m^3
Width of pillars	18 m; 20 m
Height of pillars	13 m
Width of rooms	6 m
Height of rooms	13 m
Length of rooms	20 m

The ratio of width toward height R $=W_p / h_p$ for the specified dimensions is as follows:

$$R = 18 / 13 = 1{,}38$$
$$R = 20 / 13 = 1{,}53$$

Analysis of the geometric parameters of pillar in the open stope and pillar without backfilling the rooms (district 2)

The district comprises the central part of the mine plant. In the west and in the north the district borders coincide with the natural borders of the mine field. The outlines of the safety pillar placed under the rail - way and the pillar of the industrial site serve as borders in the south and in the east. Most of the old exploitation fields, which have been exploited before 1973, are also supposed to get into the district. Those fields have been purposely insulated by forced caving of the panel drifts, assuring the access to them, because of danger of sudden collapse of pillars. The condition of rooms inside both fields is not known, but it is quite probable that some amount of accumulated waters might be available. The complicated geo-mechanical conditions are attributed by the availability of some self-caving structures, known as chimney-like headings. To prevent from further development of those chimney-like headings in the works, certain protective barriers of reinforced concrete have been built and backed up by dry sand. Extraction works are known to have taken place within 1976-1980. A open stope and pillar of exploitation with rib pillars without further room backfilling has been applied. The basic parameters of the system are given in table № 6.

Basic geometric parameters of district 2
Table 6

PARAMETERS	VALUE
Thickness of gypsum seam	19,6 m
Depth of mine works	230 – 260 m
Specific weight of gypsum	0,0225 MN / m³
Width of pillars	16 m; 17 m; 19 m
Height of pillars	11 – 12 m
Width of rooms	7 m
Height of rooms	11 – 12 m
Length of rooms	102 - 128 m

The ratio of width toward height R $=W_p / h_p$ for the specified dimensions is as follows:

$$R = 16 / 11 = 1{,}45$$
$$R = 17 / 11 = 1{,}55$$
$$R = 19 / 11 = 1{,}73$$

Analysis of the geometric parameters of pillar in the open stope and pillar with a follow up backfilling the rooms (district 3)

The district is located in the central part of the mine plant. The extraction works in the district are carried out by partial- or entire- backfilling of the extracted rooms. Some old extracted rooms are to be found in the district. For that reason it is necessary to backfill them as an indispensable condition for extraction of the mine stores left, du to the fact they have been locked in rib pillar room fender.

The parameters referring as to the version of backfilling, is shown in table № 7.

Basic geometric parameters of district 3
Table 7

PARAMETERS	VALUE
Thickness of gypsum seam	19,6 m
Depth of mine works	290 – 330 m
Specific weight of gypsum	0,0225MN / m³
Width of pillars	16 m; 17 m
Height of pillars	12 - 13 m
Width of rooms	5 – 8 m
Height of rooms	12 – 13 m
Length of rooms	110 - 132 m

The ratio of width toward height R $=W_p / h_p$ for the specified dimensions as follows:

$$R = 16 / 12 = 1{,}33$$
$$R = 17 / 12 = 1{,}42$$

Going through the above analysis on the pillar works, following results are supposed to be concluded: the pillar strength is closely related to the volume and the geometric form. The reached correlation R vary in the range of 1,14 - 1,73. The comparison made in the present study has demonstrated that the version with long rib pillars in the open stope and pillar without backfilling has substantial advantages which can be categorized as bigger stability of the supporting pillars. The general results from the comparative analysis are specified in table № 8

Results from the comparative analisis
Table 8

DISTRICT	R	S_0, MPa	% of the total pillar quantity in the district
1A	1,14 – 1,15	17,5	66
1B	1,27 – 1,33	17	83,3
1C	1,2 – 1,47 – 1,6	12,5	60,5
1D	1,38 – 1,53	16,3	75
2	1,45 – 1,55 – 1,73	16,5	47
3	1,33 - 1,42	16,2	91

CONCLUSION

It turned out to be quite complicated issue to draw analysis on the geometric parameters of the pillars in Koshava mine field, having in mind the great variety of forms and dimensions of pillars and the applied versions of the open stope and pillar system of exploitation. The collapse processes which are to be found in different part of the mine demonstrate that one of the main problems to be solved is the estimation of the stability of the pillars and rooms which are varied in types, geometric forms, location and sequence of building.

A second possible source to become more representative could be the pillar strength, defined in a empiric way, as well as the safety ratio, provided that they have been included in the terms of the problem.

Thus there are appropriate replies to the questions of substantial importance for the further development of the mine works in mine plant and they could be found.

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Recommended for publication by Department of Underground mining , Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.65-68

PARAMETERS OF DEFORMATIONS OF GYPSUM MASSIFFE IN CONDITIONS OF "KOSHAVA" MINE

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ABSTRACT
The results of an experimental research of deformation parameters of gypsum massiffe in mine "Koshava", determined by the use of speeds of elastic waves. These results are compared to the data received for the same parameters in laboratory conditions with the help of a static and dynamic method. The established ratio enable to estimate a degree of destruction of the certain volumes of a massiffe. The deduced dependences can be used for the decision of practical tasks connected with supervision over stability of pillars and mine workings.

INTRODUCTION

The necessity of operative management of rocky pressure in underground mines requires regular reception of the information about the strain and deformation condition of a rocky massiffes near mine workings. There are various physical methods for this purpose. First of all - acoustic, based on change of parameters of distribution of elastic waves in rocks dependent of their strain condition. The basic characteristics determining distribution of elastic waves in environment are their speeds. They in the large degree depend from deformation of parameters of environment. These parameters of rocks in the most cases are determined in laboratory conditions. For this purpose use the rather not broken samples with the limited sizes. The deformation characteristic of rocks in a massiffes differs from the characteristics of samples used in laboratory, mainly because of presence of infringements in a massiffes, first of all of cracks. In order to get information about values of deformation parameters in the gypsum layer, investigation works in mine "Koshava" were carried out.

RESULTS OF EXPERIMENTAL RESEARCHES.

For definition of deformation parameters (module of elasticity and Poison ratio) gypsum massiffes static measurement by pressiometter ППБ-76 and measurement of speeds of elastic waves were made. According to the data of pressiometric measurements (Daskalov, Iv. etc. 1969) on depth *0,5 - 0,9 m* from a wall of pillars the static module of elasticity has average value $E^{stat} = 15,3$ *GPa*, and on depth *5 m* - $E^{stat} = 21,4$ *GPa*.
For the same conditions the average values of dynamic module of elasticity determined by the use of seismic-log method are the next : for an external zone of development $E^{dyn} = 19,6$ *GPa*, and in depth of massiffes $E^{dyn} = 25,0$ *GPa*.

If it is impossible to drill, then either the method of seismic profiling of walls of mine workings or the method seismic ring out of a massiffes, located between two various developments is used .

The method of seismic profiling allows to define a situation of external border of a zone of basic pressure and to receive an estimation of dynamic parameters of rocks in the weakened zone and in a massiffes. In area 103 galleries on a method seismic profiling the following results are received: for an external part $E^{dyn} = 5,0$ *GPa*, and for internal - $E^{dyn} = 14,5$ *GPa*. The depth of border between both structural types varies from *1,5* up to *2,5 m* and its contrast changes considerably.

Integrated estimation of parameters of deformation of pillars turns out on a method seismic ring out. At ring out of the not broken part of pillars, located between by galleries 103, 109 and 110 for the dynamic module of elasticity the value E^{dyn} = 31,9 Gpa is received. At ring out barrier pillars, crossed by several cracks filled by clay: if one crack - $E^{dyn} = 19,1$ *GPa*, if some cracks, located by a fan, - $E^{dyn} = 17,0$ *GPa*.

Poison ratio in a massiffes has rather high values. For a strong massiffes its value is about $\mu = 0.30$, and for broken massiffes as a result of influence the mining works, or tectonics processes $\mu = 0,32 - 0,34$.

The significant interest represents comparison of the received results with deformations parameters of gypsum determined in laboratory conditions with use of drilling samples. For this purpose the significant volumes of test works were carried out. Both static and dynamic method Were used. Deformation parameters were determined at the various strained conditions. Some of the received results are shown in the publication of the authors in 2002.

The similar dependences describing change of deformation parameters of gypsum, determined were established both at static loading, and through measurement of speeds longitudinal V_p and transversal V_s of a wave and at change of the intense condition, namely: significant increase Poison ratio μ at pressure, making *60-80 %* from durability on one-axial compression, and reduction of the module of elasticity at pressure reaching *80-90 %* from durability of a material (figure 1).

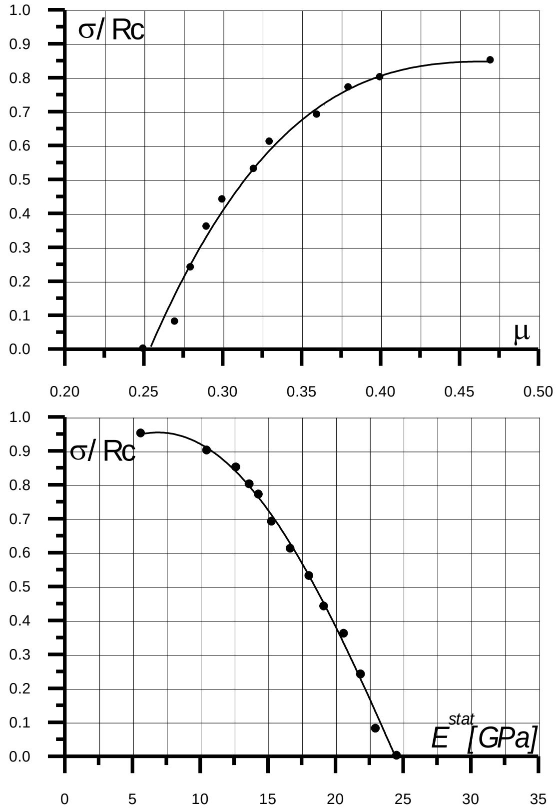

Figure 1. Dependences between: a) Poison ratio μ and pressure, per limit to compression Rc; b) static module E^{stat} of elasticity and strain σ to limit of compression Rc.

To compare given deformation parameters of gypsum massiffes with received at research of separate samples from boreholes 3473 and 3474, located near to a researched site (crossing between 103 gallery and gallery 205) the in addition experimental works in laboratory conditions for definition of the module of elasticity and Poison ratio by speeds of elastic waves in samples and static method were carried out, at pressure close to expected in a massiffes. Average results of experimental researches in laboratory and miner conditions are shown in the table 1.

Table 1. Average values of the module of elasticity *E* and Poison ratio μ for samples of gypsum and gypsum massif's

	V_p m/s	V_s m/s	μ	E^{dyn} GPa	E^{stat} GPa	$\dfrac{V_p^{mass}}{V_p^{lab}}$	A_i
Samples	3100	1660	0.30	21.6	20.0	-	
massiffes (interior)	2950	1570	0.31	14.5		0.95	0.90
massiffes (periphery)	1850	910	0.34	5.0		0.60	0.36

The given results show, that dynamic module of elasticity of a gypsum massiffes is less what characterizes samples, it is more essential in the broken zone (peripheral part). Thus, on change E^{dyn} it is possible to estimate coming changes in a condition of a massiffes and first of all processes of infringement of its solidity. For a quantitative estimation of a degree of cracks of a massiffes an acoustic parameter of cracks, A_i (Турчанинов И.А etc. 1989) often is used.

$$A_i = \left(\frac{V_p^{mass}}{V_p^{lab}} \right)^2 , \qquad (1)$$

Where: V_p^{mass} is a speed of a longitudinal wave in a massiffes,

V_p^{lab} - is a speed of a wave in a sample measured in laboratory conditions.

The values of this parameter for a concrete case are given in the table 1. With the help A_i factor of structural easing of rocks in a massiffes is defined also.

The substantial growth of Poison ratio and reduction of the module of elasticity at the certain values of strain established in laboratory conditions (a figure 1), in determined of a degree is ascertained and at definition of these parameters in a massiffes. And their essential change is connected to course of separate stages of process of destruction.

The opportunities of operative use of acoustic methods for definition of a condition of various volumes in amassiffes are illustrated by results received at research of separate parts is whole, located between the chamber 44/1 and gallery 194. The chambers are filled with sand. On the part of panel gallery in pillar the niche is made out. The acting parts of pillar have the sizes 3.8 m and 10.5 m. The smaller part of pillar is located on the part of the chamber 44/1 is strongly destroyed. For measurements four items of excitation of elastic waves are used and four items of reception (pulse generated in each of items of excitation are accepted in each of items of reception) three - dimensional by a geobackground made of devices "CB-30". In the tables 2, 3 and 4 the results ring out of these parts and seismic profiling of pillar from the party Gal. 194 (figure 2) are given.

Table 2. Results of definition of dynamic parameters of gypsum in pillar, located between chambers 45/1 and 46/1.

Parameters	V_p ***[m/s]***	V_s ***[m/s]***	μ	E^{dyn} ***[Gpa]***
Average value	1910	939	0.335	5.3
Deviation	481	221	0.037	2.5
Range of change	1120-2580	610-1220	0.28-0.37	1.9-8.7

Table 3. Results of definition of dynamic parameters of gypsum in massiffes between the chamber by 45/1 and panel gallery 1.

Parameters	V_p ***[m/s]***	V_s ***[m/s]***	μ	E^{dyn} ***[Gpa]***
Average value	2495	1172	0.330	9,0
Deviation	576	435	0.063	4,7
Range of change	1660-4150	870-1480	0.23-0.40	4,2-16,1

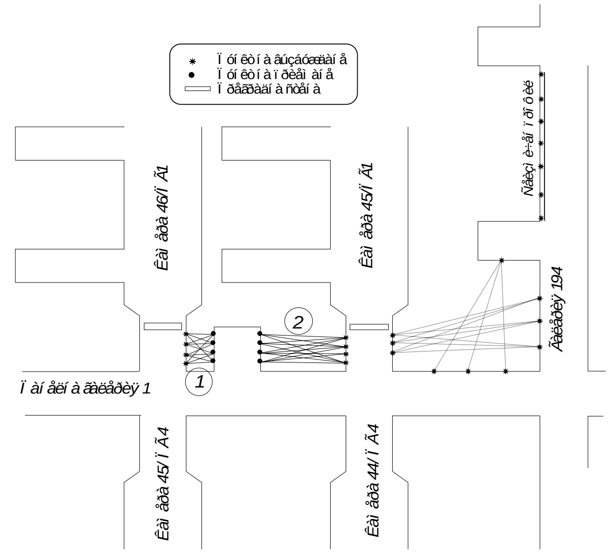

Figure 2. Scheme of a researched site.

Table 4. Average values of dynamic parameters of gypsum determined seismic profiling of pillar along gallery 194.

Parameters	***Vp [m/s]***	***Vs [m/s]***	μ	***E^{dyn} [Gpa]***
Walls	2164	998	0.360	6,0
Internal part	2735	1327	0.335	11,0

The results of researches show that the processes of destructions in between chamber pillars are shown more intensively on the part of panel galleries, and the central parts remain are more saved. Presence it is enough large (width 10 m), monolithic pillars not broken by miner works protects from destruction of crossings between galleries.

The impression makes a significant range of changes of speeds of elastic waves received at measurements. It is possible to explain it by the fact, that ring out are spent in various directions, in which directions of property and the condition of a massiffes differ essentially. It testifies to high sensitivity of a method and assumes its use for detailed research of structural features of a massiffes about miner developments.

From the given data it is visible, that between results received in laboratory conditions and at direct research in mining conditions the essential distinctions are observed. For understanding and practical use of this fact follow to pay attention to physics of interaction and condition of measurements. The elastic waves cooperate with commensurable with their lengths not by uniformity. Seismic of a wave cooperate with macrodefects of environment - layers different litology of structure, borders between them, tectonic cracks and zones of basic pressure, while ultrasound waves cooperate with mezo and micro not uniformity, commensurable on the sizes with the sizes of crystals, microcracks and zones of the raised permeability. The distinctions in structural levels of interaction define various mechanisms of transfer elastic energy and absorption by its environment. Therefore it is necessary to pick up conditions of similarity - scale factors between lengths of waves, sizes not uniformity base of measurements and thermodynamic interactions.

One of features of registered speeds of elastic waves and deformation of parameters, in particular of module of elasticity, is their basic nonlinearity and natural variability with removal from walls of development.

In a vertical direction of value of modules is described monotonous exponental by a curve and dispersion of parameters it is possible to explain by change of material structure. In horizontal chinks of curve spatial distribution essentially more complex, similar on sinusoid. The characteristic maximum of modules of elasticity determined as with the help static is observed, and dynamic methods, after which the local minimum and following increase to an interior целика follows. The generalized curves of the module of elasticity are shown on a figure 3 a, b.

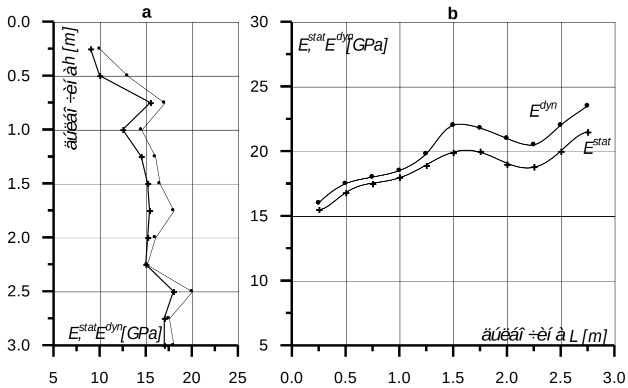

Figure 3. The generalized diagrams of the module of elasticity a- in vertical boreholes b -in horizontal boreholes.

It is possible to explain this distribution of parameters of deformation by representations zonal disintegration of rocks about mining developments (Шемякин Е.И and др, 1986).

Zonal disintegration is obliged to formation of compression, alternating by zones, and stretching caused by character of a wave of deformation. This implies, that in various zones of value of an intermediate main pressure σ_2 will change not linearly and not monotonously. Same concerns and to distribution of parameter Nadai-Lode, intense condition, determining a kind, and mechanism of destruction.

The analysis of distribution of deformation parameters in pillar shows, that the maximal destructions in pillars are shown at $\sigma_3 = 0$, in the field of the generalized shift, and the zone of fragile destruction is formed in depth of a massiffe, practically simultaneously in several consecutive zones, with decreasing intensity. The research of these processes is especially important in conditions of the greater intensity of production or transition to use of system with minimization of volume of filling.

CONCLUSIONS

1. The seismic and acoustic methods can be used for operative definition of deformation parameters of rocks in massiffes, which differ from parameters measured in samples, in laboratory conditions, in the various degrees dependent on strained and deformed condition of a researched massiffes.

2. The continuous measurements of changes in deformation of parameters of a rocky massiffes provide the information about development of processes of deformation and destruction.

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Recommended for publication by Department of Underground mining , Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.69-71

ON THE KINEMATICS OF DESTRUCTION OF ROCK MASSES AS A RESULT OF LANDSLIDES AND MINING OPERATIONS

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ABSTRACT
Usually a little attention is paid to kinematics of particles of destruction on processes and mainly dynamic and physical topics are referred. The paper reveals that this matter referred to landslides is rather important for complete assessment of phenomena. Some final conclusions and recommendations are presented.

It is well known that cylindrical surfaces belonging to circular cylinder are the only curvilinear surfaces on which rock mass slidings as plane systems are possible. Great number examples of such slidings are observed. On fig 1 is presented such a case, taken from the reality. For many important building sites had been possible to think about such movements or heavy results from such movements to be carried.

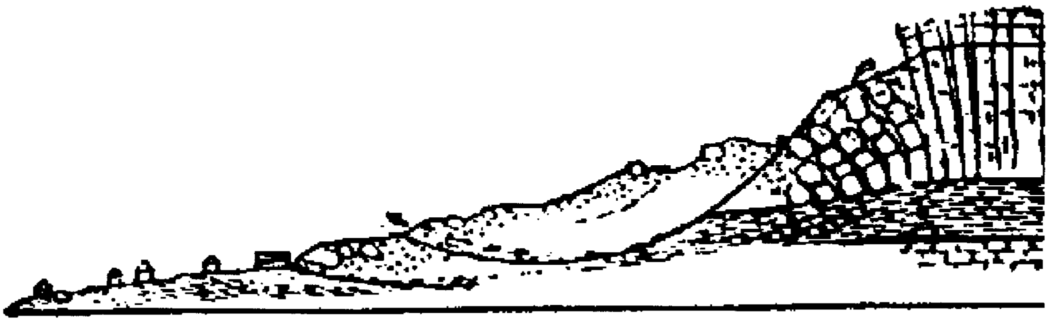

Figure 1

And naturally arises the question how could landsliding movement be presented, how stress and strain status of rock mass to be described in general, and what is the way such movements to be foreseen.

Here we will note that we are not interested of sliding movement itself. It is too complex to be described mathematically correct. In some cases it develops in an avalanche way and it is really not possible to be described mathematically correct. We only strive to discover forces interaction at the moment just before sliding movements start.

In another words we try to discover the forces active or retaining just before sliding movements start. Also phisicomathematical parameters generally factor of friction and factor of cohesion must be known.

To decide this difficult mathematical problem, we must have in our disposal some rock mass parameters such as its structure, data about its loosening, water circulation, applied (outer) forces from useful loads, vehicles eventual blasting works, seismic waves, piles driving in and other dynamic influences.

But our attention will be concentrated generally towards one side of the phenomenon, very rarely discussed with necessary completeness and correctness. Very often the fact, that sliding movement together with all other sides, very strictly defined kinematic properties. Here we have in mind the kinematical geometry, leading to definite position function and to infinitesimal properties, which follows from the fact that for state of equilibrium only ordinary equation for free rigid body are not enough, but equation, coming from the principal of virtual replacements must be taken into account *(Г.К Клейн,* Стр.Механика Сыпучих тел М.1977, стр.91).

Thee principal of virtual replacements requires introduction in the most cases of a decart coordinate system and demands rather heavy way of virtual replacements description by using partial derivatives of functions in decart coordinates. Besides, there has not thought for observation the rock mass problem as an elastic, or elastoplastic, homogeneous or heterogeneous body. Correct, well pointed and practically verified investigations in this directions are made very long ago (В.В.Соколовски, Н.Н.Маслов, Г.Л.Фисенко, Robert Schuster, Raymond Krizek,С.С.Голушкевич), but all of them require very laborious operations.

It is practically necessary methods to be used, which permit some procedures to be repeated hudredth or thousands times. And such populations of results to be selected from which convergence to only one defined numerical result is evident. These are the only cases when the result obtained leads to correctly foreseen events. A team of scientists from University of Minig & Geology from which this article comes, is pointed to such results and usage of such relevant methods.

Method, proposed in this article is connected with special kind of modeling by systems of n- angles polygons (n is equal or bigger than 2). When system moves, multyangle polygons are moving too. But for equilibrium problems is necessary to use only virtual displacements. Virtual displacements are isochronous (not time dependant). All applied on the system forces rest unchangeable during movement. Systems consists of blocks and very often relative block displacements are discussed. Moving multiangle polygons bordered intermediary surfaces with small areas between neighbouring blocks.

Because of the small area of intermediary surfaces and small angles (only parts of the degree) and angles, very near to 180^0 correct description of intermediary surfaces is very difficult. Their drowing is practically impossible. Because there is not such precise devices for drowing only parts of the degree angles. In spite of that , correct decision is possible and necessary. In many cases big number of polygons must be composed. This is the reason a particular principally new in method to be created, described below.

What is the essence of the method. Instead of the small difficult to be presented polygons, big polygons are composed and then reduced keeping the principal of mechanics for all active and retaining forces for defined points of polygons. As all displacements are brought to virtual displacements, all active and retaining forces will rest unchangeable at any movement, i. e. they will receive only translations and will neither change their values nor their directions. And of course neither convergences nor work of the forces will have physical sense. Especially displacements of polygons and connected to them rock blocks are unbelievable or even impossible on the first look. But these unbelievable pictures will disappear when defined, specially chosen parameters- by nature- generalized forces begin to tend to zero and this small values of the angles and ither dimensions will be reached which we have in reality. The benefit of this scheme of work is, that all polygons and the expressions for forces work can be described mathematically and calculations to be done with arbitrary correctness. Than all final results can be followed and took as correct at this boundary translation.

It will be demonstrated on simple examples how this method is applied. As a most simple block scheme will be taken this one on fig 2.

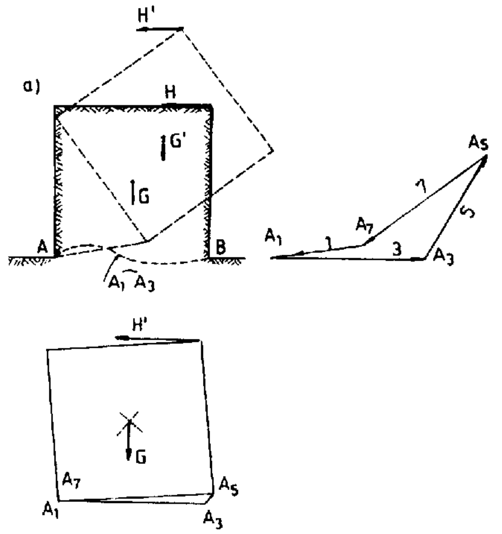

Figure 2

It consists of one single block, which is prismatic and on scheme a) its base can be seen. Many forces are applied on the block. If connections between particles of curve A and B vanish, under the action of forces presented- active and retaining, block will converge and on scheme b) is presented the new position. It is not physically possible. And calculated results of forces work are useless for any purpose. They are obtained just to compose correctly expressions for works and trough them, reducing the parameter nearly with zero, we obtain an expression very near to the sum of virtual works. But from it is very easy to obtain equilibrium expression, which is the final aim. And this equation can be used to go to investigation of stressed and strained block status. On the next schemes on fig. 3 are schematically presented many well known from the practice cases. It can be seen a slope after removal of part of the rock mass, restricted by the straight line AE.

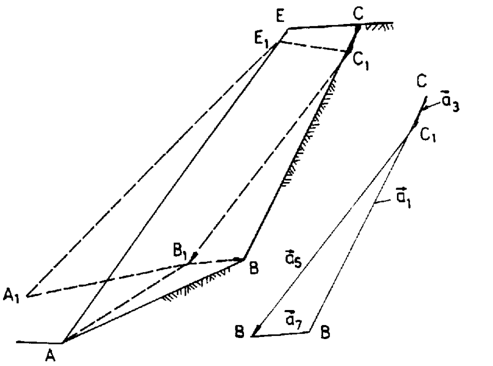

Figure 3

Under the action of active and retaining forces, the block realizes imaginative displacements and from position ABCE, comes to position A1, B1, C1, E1. With the aid of quadrangle BCC1B working forces are calculated and after the translation equilibrium equations are obtained, accepted as real ones. On fig. 4 is presented a rock mass, supposing that it is bordered by circular- cylindrical surface, divided to several elements. Trough the same procedure, for a defined element a procedure of imaginative movement of all rock mass is made and after calculations this leads to real force interaction between elements. Procedure proposed has the advantage, that it responds the question, concerning the interaction on vertical walls of the elements. This important disadvantage of previous methods is pointed out by some authors (mentioned above Клейн, Цытович).

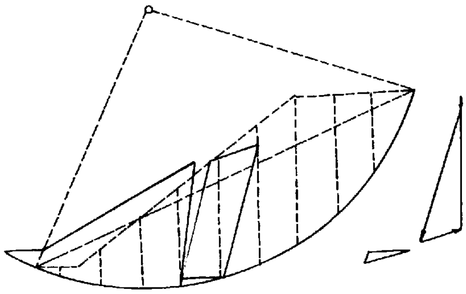

Figure 4

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Recommended for publication by Department of Opencast mining and blasting technologies, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.73-76

SEALING OF CAVITIES BEHIND THE CONCRETE LINING OF THE KINGA SHAFT IN THE WIELICZKA SALT MINE

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The paper was prepared as a part of statutory researches of AGH University of Science and Technology no.11.11.190.01 and 11.11.140.408

ABSTRACT
Long-term mining activities in the Wieliczka Salt Mine resulted in a fresh water flux of varying intensity to the salt formation behind leached caverns in the salt rocks. Safety hazards of the Wieliczka Salt Mine have been discussed with a focus on the existing quality of the Kinga shaft concrete lining, as well as the disturbed geological, hydrogeological and mining conditions in the shaft vicinity.
The injection method, applied for the strength improvement of the concrete lining and for sealing of the surrounding salt formation, has been described.

INTRODUCTION

Technical-engineering conditions in concrete constructions, especially in shaft lining, belong to factors limiting the correct and economic performance of sealing with borehole injection methods.

The aim of injection works in shaft lining may be the following:
- direct improvement of physico-chemical properties of the lining;
- lowering water permeability of the lining.

The above objectives can be obtained only when
- technical design of injection procedure;
- design of injection technology;
- selection of recipes and technological parameters of sealing slurries;
- injection works

are performed correctly.

All kinds of physical discontinuities appearing in the shaft lining in the immediate neighbourhood play a crucial role in water permeability.

Physical discontinuities may form complex networks of fractures, which can be characterized by means of a number of properties, the so-called fracturing parameters, e.g.
- spatial orientation of fractures;
- linear dimension of fractures;
- degree of cracking;
- degree of divisibility.

From mathematical-engineering the point of view these parameters can be determined by means of measuring or calculation methods, e.g. unit water absorptivity index, permeability, porosity, filtration coefficients.

The aim of the paper is to present the technology of liquidation of voids behind the lining of the Kinga shaft in the Wieliczka Salt Mine with the use of clayey-cement slurries and borehole injection methods. Injection procedures carried out in the above mentioned object liquidated water leakages through the lining, creating conditions for uniform distribution of hydrostatic pressures acting on the shaft lining, and generated conditions in which the lining could co-operate with the rock mass.

TECHNOLOGIES OF WORKS FOR SEALING THE SHAFT STRUCTURE

Sealing of the shaft lining was realized by the borehole injection method.

This solution lied in making boreholes in the planned sealing area of the shaft and neighbouring rock mass followed by injection of sealing slurry under pressure.

Due to the varying technical and mining-geological conditions, grids of injection wells were made, at arbitrary length and order.

The wells were made with the use of varying techniques [1]:
1. order of injection works;
2. location of drilling wells;
3. rheological properties of sealing slurry;
4. injection pressure.

As a principle, the Kinga shaft and rock mass were sealed from the lowermost level towards the shaft site. The sealing operation was divided into a few stages concentrating on different parts of the shaft passing through the specific levels of the Wieliczka Salt Mine. And so, in line with the above principles, the lining between the floor and the level VII sealing was sealed in the first order. This was followed by making:

- the shaft lining below the level VIII.

At a depth of 197.97 to 296.53 m the shaft lining was made of bentinite bricks 0.5 m thick. First, vertical boreholes were drilled from gangways sited at the east and west part of the Kinga shaft, from the level VIII. At each side, 3 boreholes were drilled at about 1.5 m distance from the lining. Two boreholes in the gangway were located 0.5 m from the side walls, and the third one was in the gangway axis. They were shown in the Figure 1 as P8I, where 8 denoted that the holes were drilled from the level VIII and I signified that it was a vertical borehole from 1 to 6. The length of each of the boreholes was 11.7 m.

The second stage of sealing the shaft lining below the level VIII lied in making five sealing rings at 2.25 m distance from one another. Injection wells 2 m long and 60 mm diameter were made from inside of the shaft:

- shaft lining between levels VII and VIII.

First, two horizontal sealing rings were made, followed by vertical injection wells drilled from the level VII

- three sealing rings consisting of five horizontal boreholes, each 2.0 m long;
- three sealing rings consisting of five horizontal boreholes, each 0.7 m long;
- three vertical boreholes on the east side of the gangway, 1.5 m from the inside of the lining and 11.0 m long.

Further works were so scheduled as to seal the shaft and the neighbouring rock mass in compliance with applied technologies.

SELECTION OF SEALING SLURRIES FOR SECURING THE KINGA SHAFT

To efficiently seal and reinforce the Kinga shaft lining and the neighbouring rock mass, it was necessary to elaborate recipes for slurries adjusted to the variable geological conditions in the rock mass and also material making up the shaft lining. When working out recipes of sealing slurries, attention was paid to the good co-operation with the rock mass and increased durability in corrosion-aggressive conditions.

To elaborate sealing slurries for securing the Kinga shaft, the following solutions were applied:

- metallurgical cement CEM III/A – 32.5;
- alkaline activator – Na_2CO_3;
- mineral additives in the form of drilling bentonite silt;
- fully saturated Wieliczka brine.

Bentonite silt was added to the sealing slurry in order to:

- lower filtration of sealing slurry;
- lower sedimentation and increase stability, especially sealing slurries with increased water-mixing properties;
- adjust mechanical parameters of hardened sealing slurry to the parameters of sealed and reinforced rock mass and shaft lining;
- increase plasticity of hardened sealing slurry;
- lower permeability of hardened sealing slurry as a result of silt particles blocking the pores.

Sodium carbonate was used as an:

- activator of bonding time;
- plastifier;
- component securing good co-operation with the rock mass, especially of the clayey type.

Brine 1200 kg/m^3 from Wieliczka Salt Mine was used for making sealing slurries. Its chemical composition was as follows:

NaCl	305 g/l
Ca^{2+}	1.05 g/l
Mg^{2+}	0.22 g/l
NH_3	0.02 g/l
HCO_3^-	0.21 g/l
pH	7.5

Wieliczka brine is very aggressive to concrete and hardened cement slurries.

The type of injected sealing slurry through injection wells depended on:

- assumed range of penetration of the slurry beyond the lining;
- admissible slurry injection pressure;
- type and technical condition of shaft lining (porosity, permeability, filtration coefficient, fractures and hydraulic connections with external surfaces of the lining);
- technological parameters of fresh and hardened sealing slurry (density, rheological parameters, bonding time of slurry, mechanical parameters of hardened slurry).
- Injection works connected with sealing of the rock mass beyond the shaft lining and the lining itself will require minimum three recipes for saline sealing slurries (Table 1)
- basic slurry (SK1);
- slurry liquidating escapes from behind the lining and side walls (SK2);
- sealing-filling slurry (SK3).

Basic technological parameters of sealing slurries used for sealing and reinforcing the Kinga shaft lining and the surrounding rock mass are presented in Table 2.

Table 1 Mass of components used for making 1 m³ of saline sealing slurry.

No.	Components of slurry					Coefficients		Calculated average density of dry components of slurry, kg/m³	Calculated average mass of dry components of slurry, kg	Calculated density of sealing slurry, kg/m³
	Symbol of sealing slurry	Base slurry– fully saturated brine, m³	Metallurgical cement CEM III/A 32.5 NA, kg	Bentonite clay	Na_2CO_3 kg	Water-mixture w/m, -	Mixture -water m/w, -			
1	SK1	591	1158	23.64	0	0.600	1.660	2890	1182	1891
2	SK2	629	1027	20.69	20.96	0.706	1.416	2800	1069	1824
3	SK3	69	936.9	19.12	0	0.840	1.194	2890	956	1759

Table 2 Technical parameters of saline cement-bentonite slurry determined in 20 ($\pm$2) °C (293 K).

No.	Symbol of sealing slurry	Mixture-water coefficient m/w, -	Density ρ, kg/m³	Cone fluidity AzNII, mm	Relative viscosity, acc. Ford cup No. 4	Sedimentation %	Specific filtration ΔP=0.7MPa	Plastic viscisity ηa, Pa s	Apparent viscosity at 600 rpm ηa, Pa s	Flow limit τ_y, Pa
1	SK1	0.600	1870	130	-	0	25/300	0.0963	0.117	21.434
2	SK2	0.706	1810	210	35	0.2	32/200	0.0618	0.073	11.445
3	SK3	0.840	1730	260	16	0.4	42/125	0.0290	0.036	7.051

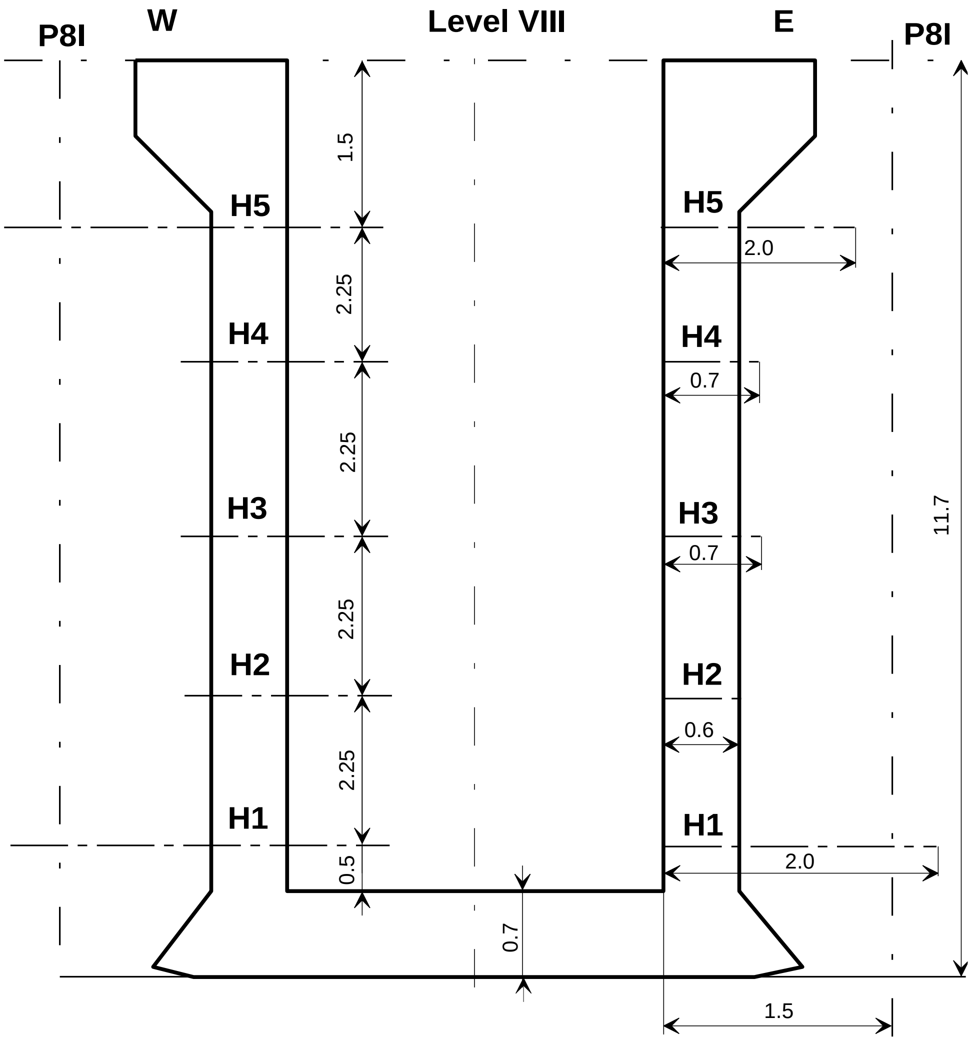

Figure 1. Location of injection wells below the level VIII.

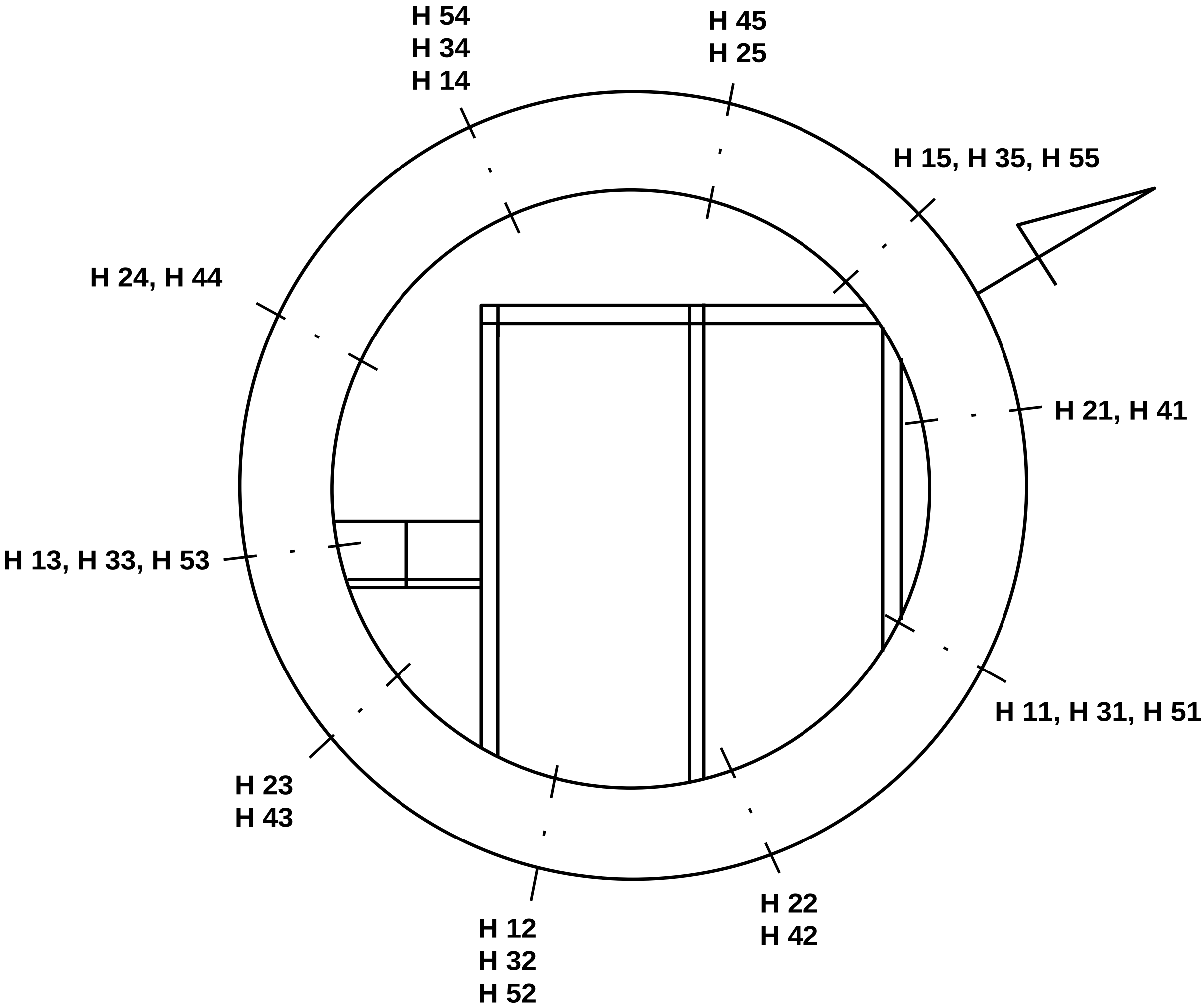

Figure 2. Location of injection wells below the level VIII in the Kinga shaft.

CONCLUSIONS

1. Many years exploitation of shaft linings under the influence of the rock mass and the surrounding waters often necessitates reinforcement and sealing of the lining.
2. Rheological properties of slurry, location of injection wells, order of injection works and final pressure at which slurry is injected are important parameters in the injection method used for improving the tectonic state of the shaft lining.
3. The applied slurries were made on the basis of fully saturated brine proved to be very useful in the conditions of the Wieliczka Salt Mine.

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50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.77-82

STUDIES ON THE SELECTIVITY AND FINANCIAL IMPROVEMENT IN THE PROCESS OF TIN FLOTATION BY CHANGES OF THE pH, FLOTATION TIME AND CONCENTRATION OF COLLECTOR

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ABSTRACT
The main purpose of the present investigation is determining in which order the test results concerning the adsorption mechanism of the Aerosol 22 on the oxides could be used in tin flotation practice. Tests were performed with tin hematite, tourmaline and quartz. The minerals are studied separately in respect to their behavior in the flotation process in conditions of pH change, concentrations of the collector and slurry temperature. Differing from the ferrous minerals, hematite and tourmaline tin shows strong flotation properties at pH 2. From their own hand, the tourmaline and the hematite show a good flotation at pH 6. It was established that the increasing is the collector consumption the increasing monotonically is the mineral recovery. These results could be proved only by flotation tests of mineral mixtures to be found whether the selectivity is influenced as a result of an interaction among the minerals. The difference in the adsorption at different slurry temperatures is studied additionally. The calculations for financial profit show that the increase of the Sn recovery on the base of the reagents used will lead to a nominal profitableness of 1,2 DM per annum in a case of plant productivity 300 t/d.
Key words: flotation, selectivity, temperature, pH, collector.

INTRODUCTION

The primary tin ore flotation attain more and more significance as a result of the increasing complexity of the ore mined and the related to it necessity of a further ore grinding.

In spite of the intensive research activity in the last years, a general flotation method has not been found yet (Arbiter, 1977; Wottgen, 1980). It concerns most of all the type of the collectors used. For example, in Bolivia - the greatest producer of primary tin ores they use Aerosol 22 in all their flotation plants (Savvidis, 1997, 2001). The infrared spectroscope studies (Berger, 1980) of the Aerosol 22 deposition on the tin and the other by-minerals such as hematite, tourmaline and quartz have been used as an explanation of the adsorption mechanism of those reagents on the oxidized mineral surfaces. An attempt by using of appropriate reagent regimes was made aiming a flotation process selectivity improvement (Savvidis, 1997, 2001). Some directions regarding the optimal conditions of the collector accumulation were obtained as a result of those investigations which however do not fit totally with the values obtained in the flotation practice.

It should be found in the frame of the investigation performed in which order the results of the qualitative and quantitative studies of the adsorption mechanism are applicable to the tin flotation. Using different minerals performed the flotation tests. A priority in our investigations was the tin flotation improvement by founding an appropriate and most of all selective reagent collectors. The main aim is prevented in some order by the following factors:

1. Mineralogical content of the tin ore in the different deposits is highly variable;
2. A big amount of the by-products manifest similar flotation abilities;
3. The facts characterizing the region and most of all the very high concentration of the metal ions disturbs strongly the tin flotation process.

The carboxylates used in the past act unselectively and react very sensitively in respect to the ions in the slurry (Siebert, 1977). A relative improvement is observed by Siebert (1971) and Serano (1975) in the case of a substitution of mono-, di- and polycarbonic acids with brome. The last one leads to an improvement of the selectivity.

The phosphoric and the arsenic acids have showed good parameters as specifically operating collectors when they were used for a flotation of the Althenberg ores and for the tin ores in Tasmania. They dissolve fully the carboxylates. It should be noted that they are not so appropriate for the flotation of the Bolivian tin ore. The both type acids react in a poor acid media with the tin and ferrous minerals by a hemi-sorption (Dietze, 1975). The reason for the proportionate high selectivity take place in the fact that the adsorption on the tin minerals is considerably faster than on the other rock minerals.

EXPERIMENT SETTINGS

Tin flotation with Aerosol 22

After introducing of the sulphosuccianamates as a collector for the tin ore (Arbiter, 1977) a reagent was found by which even in a case of a presence of large amount of alien ions good flotation results could be achieved. In the case of Aerosol 22 the matter goes to the tetrasodium N – (1, 2 dicarboxyilethyl – N – octadecilsulphosuccianamate – a product of the

American company Cyanamid. The structural formula of that reagent is shown in the Figure 1.

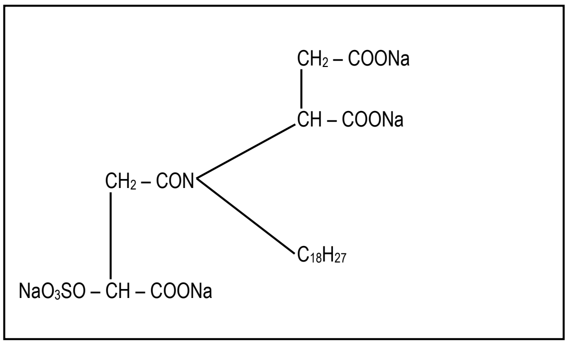


Figure 1. Structural formula of the Aerosol 22

Berger (1981) has an essential contribution to the clarification of the adsorption mechanism of the Aerosol 22 on the mineral surface. According to the quantitative adsorption measurements the following suitable flotation conditions are obtained: An adsorption minimum is obtained in the case of the ordinary for the practice concentrates as of 10^{-4} mol/l. A higher quantity of the collector is adsorbed at a slurry temperature of 35°C than at temperatures of 20°C. Analogically, an increased collector adsorption is observed when the hydrogen ion concentration and the activity increase as well as with the polyvalent metal ion concentration increase. It should be noted that the influence of the collector concentration, pH as well as the temperature have to be studied in some flotation experiments as well.

Description of the material studied

In the present experimental study tin ore, hematite, tourmaline and quartz were used. The sample of tin ore was prepared as a tin concentrate from a secondary deposit in Spain located near by Malaga. It has a content of 69% of Sn and about 2% of Fe. The rest ore impurities are quartz and some other silicates.

The hematite is extracted from the ore body of Minas Gerais and has a content of Fe as of 66%. The ore contains also a small quantity of impurities of quartz, magnetite and limonite.
The shorl that is also well-known as "elzen-tourmaline" has coarse accretions with the quartz. The material is extracted from Kaatiala in Finland and has a Fe content of 9,9%.

The quartz is extracted from the Frechen quartz deposits and contains 99,9% of silica as well as small quantities of iron impurities.

The flotation experiments were carried out in a pneumatic micro flotation cell of 100 ccm. The slurry solid content in the process of the agitation was 100 g/l and at the beginning of the flotation process it was 50 g/l. The agitation time was 5 min. The foam product is carried out in 4 concentrates. The times of the separate flotation processes for obtaining of the respective concentrates are e 1; 1,5; 3 and 5 min respectively. The Sn and Fe contents were determined by the Quantitative X-ray Fluorescent Analysis.

Influence of the pH values

It is well-known from the flotation practice as well as from a number of investigations that the Aerosol 22 shows best collector properties in respect to the tin minerals when the process is carried out in the strongly acid area at pH 2,5 (Breuer, 1960; Barbery et al., 1977; Savvidis, 1997).

That fact is confirmed also on the base of some experiments which are graphically expressed in the Figure 2a.

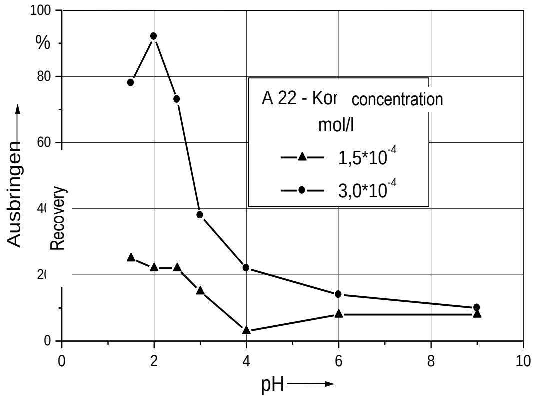


Figure 2a. Tin recovery depending on pH.

Herein, the tin recovery is presented at two different collector concentrations and depending on pH. The chosen concentrations of 1,5 и 3,0.10^{-4} mol/l correspond to the most used ones in the practice. Recovery maximums are obtained for the both collector concentrations at pH values of 1,5 и 2,5 respectively. The quite high pH values decrease strongly the recovery and it decreases with more than 10%. The amount of the recovery is highest, i.e. - 90% at collector concentrations as of 3.10^{-4} mol/l and pH 2. There is a waste product with 78% at pH 1,5. The investigation of the collector concentration influence is considered once more in the further investigations. The flotation behavior of the rest of the minerals in the same conditions is shown in the Figures 2b, 2c and 2d.

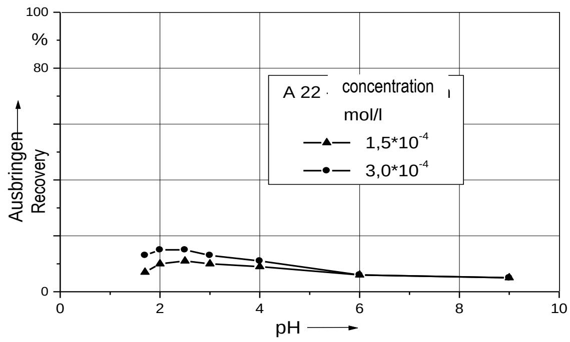


Figure 2b. Recovery results depending on pH.
в) regarding the quartz;

FLOTATION EXPERIMENT RESULTS

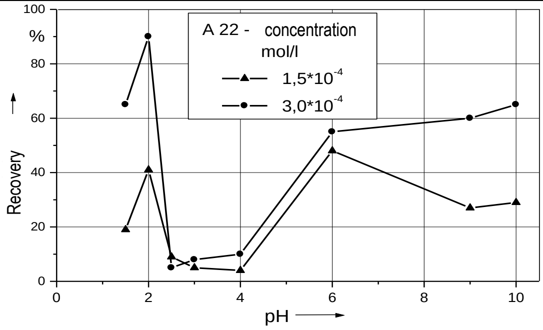


Figure 2c. Recovery results depending on pH. c) regarding the hematite

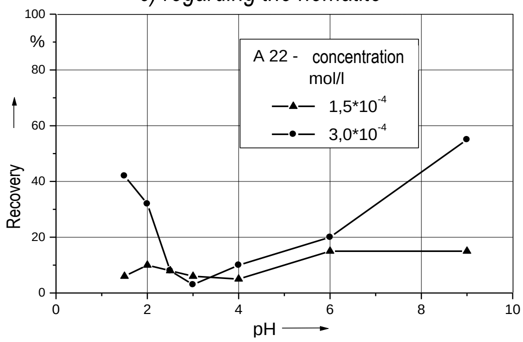


Figure 2d. Recovery results depending on pH. d) regarding the tourmaline

The quartz recovery in the whole pH area and for the both concentrations was not more than 20%. A slight increase was found in the pH acidic area between 1,5 and 3,0. That could be explained by the increased quantity of the physically attached collector. A hemi-sorption on the clear quartz is not possible to be observed. The tourmaline and the hematite showed a similar behavior in the whole pH area. The hematite showed a good recovery of about 90% at pH 2 and its recovery at pH 1,5 and 2,5 decreases rapidly.

The recovery of the four minerals depending on the flotation time at pH 2,5 is shown in Figure 5.

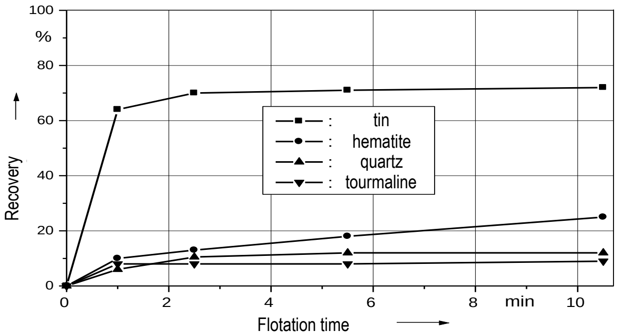


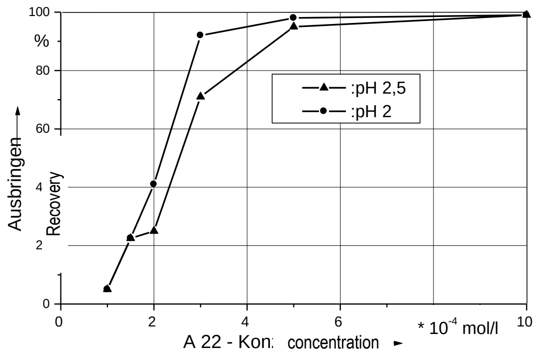
Figure 5. Recovery of the tin, hematite, quartz and the tourmaline depending on the flotation time at pH 2,5

The hematite showed extremely bad flotation properties in that pH area. The tourmaline and hematite recovery increased slightly of about 10% at pH 4 and at pH 6 decreased to 50% and the tourmaline one – to 20%. In that case, the iron fragments in the minerals react in a hemi-sorption manner with the carboxylate groups of the collector. Most probably, hydrate compounds are formed in that pH area. The solubility of the iron composing the tourmaline and the hematite increases with the pH decrease and that leads to an additional influence on the flotation results.

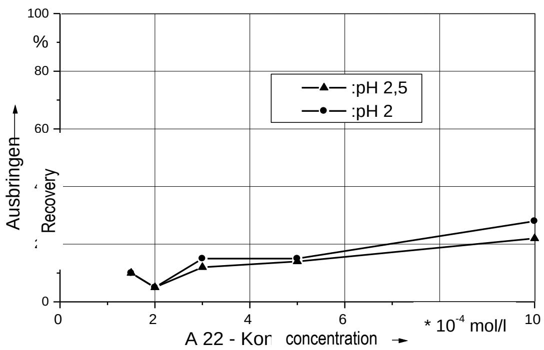
Influence of the collector concentration

The Aerosol 22 consumption varies between 500 and 1000 g/t. The last one corresponds to a collector molar solution from 1,53 up to 3,06 10^{-4} mol/l at a slurry density of 200 g/l. As a result of the Berger's (1980) investigations it was found a decrease of the collector quantity taken by the adsorption in the range from 10^{-4} up to 10^{-3} mol/l. The flotation behavior of the four minerals depending on the Aerosol 22 concentration in the range from 10^{-4} up to 10^{-3} mol/l is shown in Figures 3a, 3b, 3c and 3d. The experiments were performed at pH 2 and 2,5.

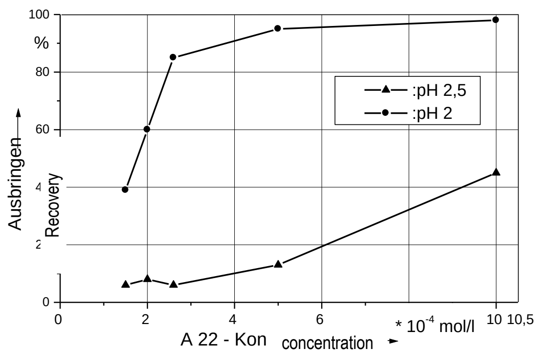
In the tin flotation case it is observed a recovery increase for the both pH values and the highest recovery is found at a concentration of 5.10^{-4} mol/l. The tin flotation in conditions of an increasing collector concentration could be seen in the Figure 3b. A maximum recovery of 22% is achieved at pH 2 and a collector concentration of 10^{-3} mol/l.



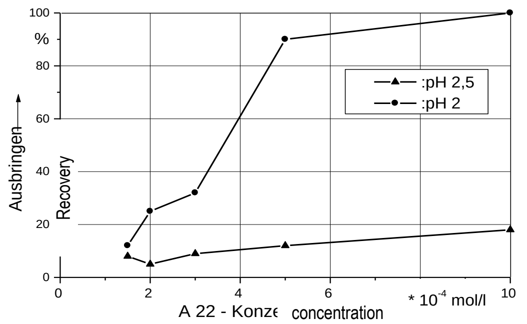
(a)



(b)



(c)



(d)

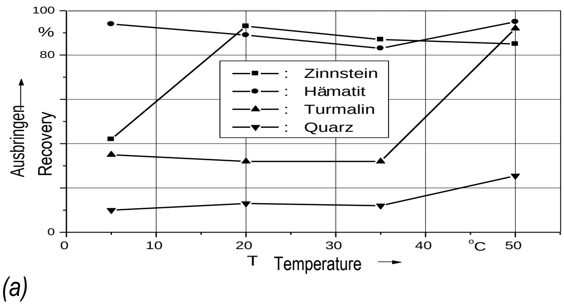
Figure 3. Recovery results depending on the collector concentration
a)regarding the tin; в)з regarding the quartz; c regarding the hematite; d) regarding the tourmaline;

A low recovery is observed in a similar way at a high collector concentration and relatively worst rates of the linkages between the collector molecules and the mineral surface. Most probably, a collector physical accumulation is observed depending on the concentration as well as its adsorption on defects of the crystal lattice. Similar flotation results are found for the tourmaline and the hematite. The both minerals are undergone to a full flotation at pH 2 and 5.10^{-4} mol/l. A slight recovery increase is observed at pH 2,5 as well as at 10^{-3} mol/l. Surprisingly, the adsorbate decrease described by Berger (1980) do not influence negatively on the flotation behavior of the four minerals. Most probably, there are an existing relation between the dimmer formation described by him and the flotation results. There is an increase of the quantity of the adsorbed collector on the base of the higher potential of the collision between the minerals and the collector dimmer molecules and in the same time the layered molecules strengthen the hydrophobic properties. That effect of the increasing collector hydrophobic ability properties could be compared to the action of the long-chained non-polar molecules.

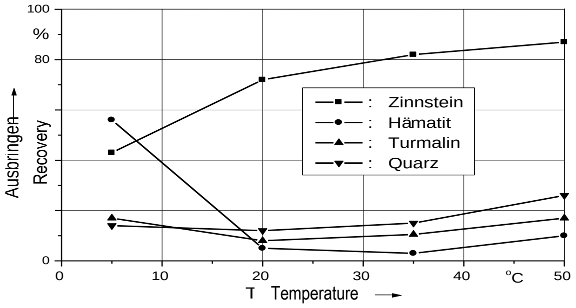
As a consequence of the results described it could be concluded that the selective tin flotation can pass at pH 2,5. The additional experiments are carried out at a collector concentration of 10^{-4} mol/l.

Influence of the slurry temperature

There is a well-known relation of the adsorption processes from the slurry temperature. It could be seen in the Figures 4a and 4b that in all of the cases the temperature influence on the flotation results. The four minerals recovery is shown depending on the temperature and at pH 2, 2,5, 4 and 6

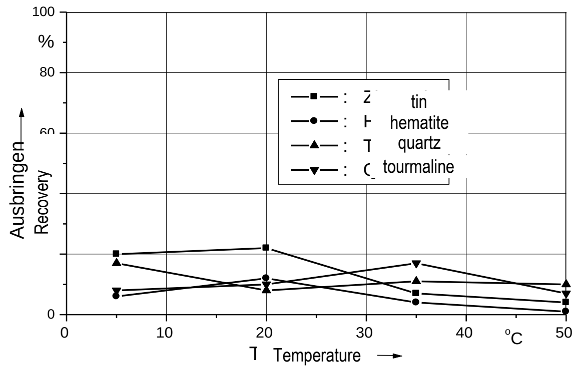


(a)

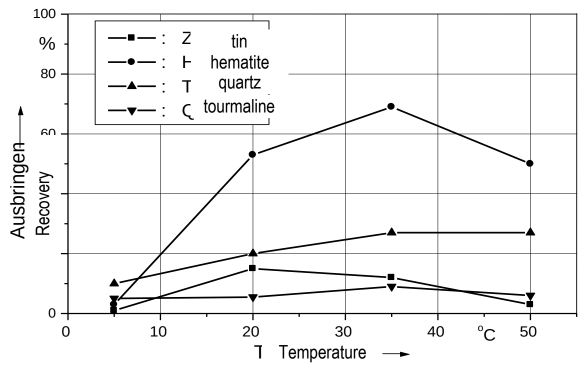


(b)

The agitation tank and the flotation cell were set in an additional tank full with water to regulate their temperature. The water temperature regulation was done by a thermostat. The cooling of the water to 5°C was based on the same principle. The water was cooled to 1 - 2°C by ice bits in such way that a temperature of 5°C was achieved in the volumes of the cell or Becker glass. The results show that a selective separation of the tin minerals from the ferrous ones in the acid pH range and especially at a pH 2,5 is possible by a temperature regulation. As it could be seen in the Figure 4b, the tin minerals recovery increases monotonically with the temperature increase. Improved flotation results for the rest of the minerals are found only at 1m of the temperature range from 35 up to 50°C. An entire activation as a result from the high temperatures is observed probably in the acid pH range. There are negative results observed as it is shown in the Figure 4d at higher pH values and a temperature increase of 50°C.



(c)



(d)

Figure 4. Recovery results depending on the temperature.
a) at pH 2 ; в) at pH 2,5; c) at pH 4; d) at pH 6

Most probably, the collector-mineral link is destroyed at high temperatures and in the same pH range. The tin minerals recovery at temperatures of 5, 20, 35 and 50°C as well as at pH from 2 up to 6 is presented in the Figure 6.

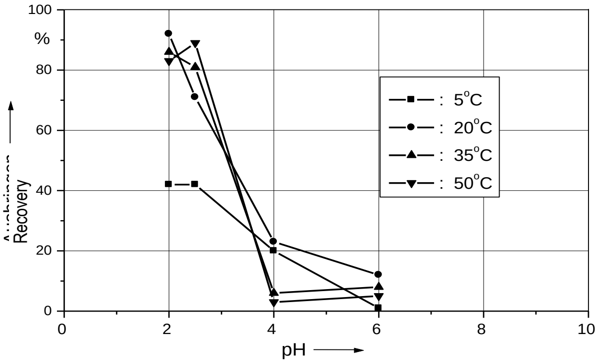


Figure 6. Recovery results depending on the pH at 5, 20, 35 u 50 °C.

The effect of the increasing temperature on the type of the collector accumulation is shown in the figure. The recovery is low (42%) at temperature 5°C and pH 2 and pH 2,5. As it is shown in the Figure 6 the results are worst at pH 6 and a maintaining of the same other conditions. Most probably, the chemical activity is not enough for a hemi-sorption on the tin minerals surface at the low temperatures. In that case, the physical adsorption mechanisms are more strongly expressed and in that way the linkages stability is not enough.

The tin minerals flotation behavior at 20°C is similar to that at 5°C. It could be noted here that usually a reaction behavior activation of the mineral surface and or collector molecules are aimed by the temperature increase. It is expressed by a double increase of the recovery at pH 2 and 20°C in a comparison with that at 5°C. The high recovery values are identical to those in the conditions of tourmaline and hematite flotation as it could be seen in the Figures 7 and 8.

A considerable increase of the Aerosol 22 selective accumulation on the tin minerals is possible at 35°C and pH 2,5. There is even a slight 10% increase of the tin recovery found at that temperature and at pH 6. That effect could be explained by tin hydroxide ion activation.

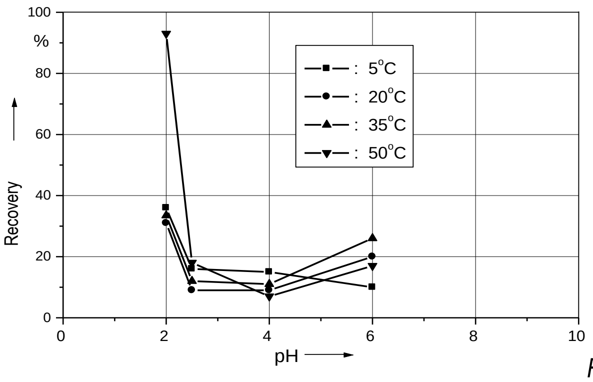


Figure 7. Recovery of tourmaline depending on pH at 5, 20, 35 and 50 °C.

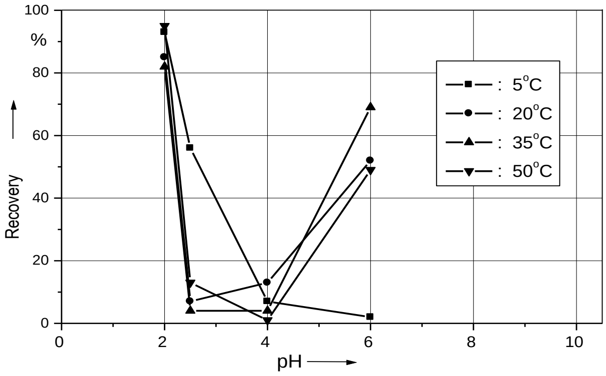


Figure 8. Recovery of hematite depending on pH at 5, 20, 35 u 50 °C.

The results of the flotation of the tourmaline and hematite shown in the Figures 7 and 8 are similar. The flotation of the both minerals is good at pH 2 and pH 6. Activation by increasing the temperature is aimed in the acid range of the pH and a temperature at 35 °C. It is obviously seen especially at pH 6 at which we have activation by a ferrous oxide formation. It is not valid in conditions of low temperatures as it could be seen from the tourmaline and hematite curves. The temperature influences slightly on the quartz flotation behavior and there is an increase of 37% observed only at very high temperatures (about 50°C) and an acid pH.

Consideration of the financial profit

An attempt is made for a calculation of the nominal profit by a calculation of the additional profit in the case of an increased productivity of the tin flotation with 5% of a recovery processing of the slimes of a grind size less than 20 μm

Bases of the calculation

1. Initial ore material: 300 t/d
- Sn content – 0,75%
- Concentrate price: 3,25 US $/lb, or 18 DM/kg
- Price of the reagent Aerosol 22: 6,75 DM/kg
- Additional costs for reagents:
  300 g/t Aerosol 22: 2,025 DM/t or 607,5 DM/d
- Nominal profit following from the extraction increase: 5%
- Съдържание на Sn в концентрата: 20%

2. Additional concentrate quantity: 0,5625 t/d
3. Tin content: 0,1125 t
4. Nominal profit: 4.214,25 DM/d
- Nominal profit – reagent costs: 4.214,25 - 607,5 = 3.516,75 (300 work days/a: 1.237,275 DM/a)

DISSCUSIONN OF RESULTS

A surfactant reagent was investigated by the Aerosol 22, which thanks to its complex molecular structure and its four polar groups finds a large-scale application in the practice.

The ferrous minerals as hematite and tourmaline have a highly reduced flotation properties at pH 2,5 in compared to the tin minerals. The ferrous minerals have good flotation ability at pH6. It is a consequence of the high affinity of the carboxylate group to the iron and it clearly proven by the adsorption experiments. There is a recovery increase at pH 6 while there is not a difference in its value at pH 4. Most probably, the linkage between the ferrous hydroxide and the collector plays at pH 4 a high order role which one opposes to the strongly mechanic interactions in the flotation process in the higher pH ranges.

There is no found a strong relation between the concentration in the range 10^{-4} - 10^{-3} mol/l and the adsorbed quantity of the collector. That could be compared to the increased collector consumption and the monotonic increase of the minerals recovery. It should be expected in the process of the tin flotation that the adsorption improvement and thus the valuable component content in the concentrate will be not achieved by the collector consumption increase. There are insignificant differences found in the adsorption at the different temperatures.

It could be seen from the economical evaluation that the increased recovery that is rendered in an account in the reagent usage would led to a nominal profit increase up to 1,2 millions of DM annually in the case of the operating of an plant with a productivity of 300 t per day.

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Recommended for publication by Department
, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.83-86

INVESTIGATIONS REGARDING THE IMPROVEMENT OF THE SELECTIVITY AND THE PROFITABLENESS IN THE TIN FLOTATION PROCESSING OF FINE CLASSES

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TEI of W Macedonis Greece	TEI of W Macedonia Greece	TEI of W Macedoni Greece	TEI of W Macedonia Greece	TEI of W Macedonia Greece	TEI of Kavala Greece

ABSTRACT
The principle of multistage de-sliming to be separated the fine class is used up to now in all flotation plants in the process of the fine tin ore dressing. In this case the losses of tin amounts of up to 40% from the used initial ore material. A decrease of these losses of tin by using of slim flotation is the main aim in the present investigations. Flotation tests in washer and rod flotation plants were performed to optimize the flotation conditions. The recovery values in the flotation process performed in the second type of the plants mentioned above vary in the range about 60%. They are between 65-80% in the case of washer impeller. Good results in the recovery process could be expected when washer impellers with a high speed (5,65 m/s) are used during the preliminary flotation stage and after that low revolutions are used during the following stages of purification activities. Similar results could be expected also in usage of rod impellers when during the preliminary flotation stages is used a low speed and during the purification stages – a high speed. It should be mentioned that a heavy minerals dissolving and tendency of the slurry to a coagulation is observed especially in the case of water flotation. In this case an intensive agitation of the slurry is necessary by using of a high rotor speed. The high revolutions in the case of finger impeller usage lead to low recovery losses. The introduction of modified impellers of that type is very promising in respect to the low grinding size ores. It could be concluded from the performed calculations for the profitableness that a flotation plant will be effective only in the case when the concentrate contents are higher than 7,5% Sn and the recovery is more than 70%.

INTRODUCTION

Wetting gravimetric methods of grading are used very often in the process of dressing which ones depending on the density and the density differences of the minerals to be separated give the best results in processing of the classes grading of 20 – 100 μm and are economically profitable respectively. Other methods are used in principle in the processing of a material of a lower size than the mentioned above but usually they used the flotation. A premise for establishing of the flotation method is the variable character of the ore materials.

The most of the present deposits are characterized by a fine accretion of the ore and rock minerals and which requires by itself a fine grinding to be guaranteed a good outcropping of the minerals. As the necessary grade of outcropping of the minerals is often less than the grinding achieved by using of density grinding the flotation methods find a great application as of a grinding method for the classes of size less than 100 μm.

There are problems arising very often in the process of the primary tin ores flotation and which are a subject to the present paper. In spite of the fact that there are appropriate flotation plants existing in Bolivia as well as in England, Australia, South Africa and the former DDR (Moncrief et al. 1977; Arbiter, N., 1977) the problem of the tin flotation is not fully solved yet. The last one is proved by the fact that independently of the intensive investigations performed and the made new proposals about usage of new methods and reagents (Savvidis 1996), up to the moment there is not a tin ore flotation conception developed yet which uses a specific and selectively acting collector. It is like that because of the specific properties and the different forms of the tin.
It is well-known that the friableness of the tin facilitates the fine classes formation during the grinding, but for example in Bolivia could be to meet very needle-shaped and crossed forms of the crystal structures, noted as needle-shaped or wood tin. The needle-shaped tin could be easily transversally in respect to its length axis during the processing and therefore it forms a big amount of fine classes. The "wood" tin being formed by weathering of sulphostanates forms in the most of the cases fine complex aggregates which are difficult to be separated. The colloidal tin as well as the needle-shaped one have a tendency to over-grinding.

It is established that the most part of the Bolivian tin ores tends to possibility for tin enrichment for the fine classes (Savvidis 1996, Savvidis et al., 2001). For the moment the rich of tin fine slimes in Bolivia were predominantly thrown out, because there were not an existing method of grading.

The accents of the present investigation are aimed on the investigations of a fine classes extraction by the flotation aiming an extraction improvement as whole as well as the tin extraction profitableness.

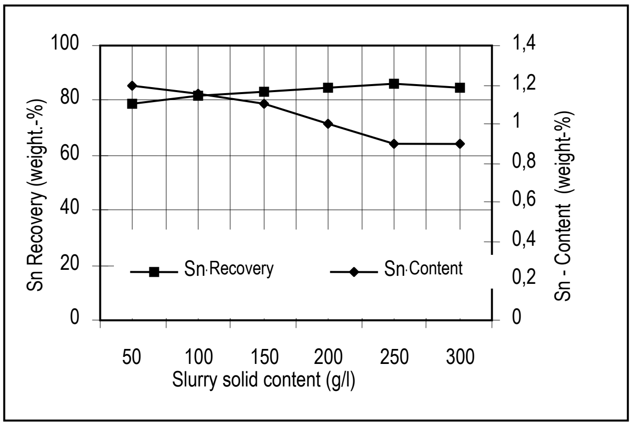
EXPERIMENT CONDITION

Influence of the slurry density

The experiments were performed by using of a sample of the cyclone weir form Huanuny aiming an optimization of the flotation conditions in the flotation cells. There are different opinions about the optimal slurry solid content. Imhof (1975) realizes that in the process of the sulphide ore slimes flotation the solid content should not exceed 25 g/l. Töpfer (1964) to the opposite of that proposes the fine fragments to float in conditions of high slurry density. The very high solid fragment contents in the slurry should be avoided as the slurry viscosity becomes very high. The following conditions were established for the investigations in respect of the slurry density influence:
Flotation cell: 1,4l – laboratorial, with a washer impeller
Air consumption: 30 cm^3/cm^2min
Solid content in the slurry: 30 – 500 g/l
Collector: Phosphonic acid P-184 (750 g/t)
Foam generator: Flotigol CS/MK, 50 mg/ l
pH: 5,8
Conditioning time: 20 min
Flotation time: 15 min (5 concentrates)

The collector feed was made on the base of each stage. The variant of a continuous reagent feeding was rejected to be estimated how the slurry density is influenced by the other factors

The flotation results are shown in the figure 1.



1Figuire 1. Sn content and Sn-recovery in dependence on the slurry density, collector: P-184, 750 g/t, pH 5.8.

As it could be seen in the figure, the slurry density influence on the Sn recovery and content is small. The tin recovery is about 80% and in case of an average slurry solid content– 250 g/l a slight maximum is observed. The tin contents in the concentrate are low and at low slurry solid content - 50 ÷ 150 g/l they have a highest value of 1,2%. Probably, the slime is recovered unselectively. It is supposed that in that case in flotation tests the following effects are competitive to each other: the foaming consistence is slightly stabile at at its low concentration; the floated out valued minerals leave partly in the foaming product; the high solid contents increase the slurry viscosity; the turbulence needed for the mineralization could be not achieved. The tin concentrates contents at different slurry density are presented in respect to the flotation times are presented in the Figure 2.

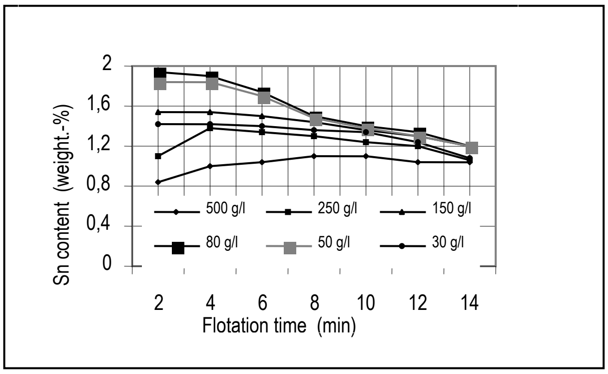


Figure 2. Sn content in respect to the flotation times*,collector: P-184, 750 g/t, pH 5.8*

As it is obviously seen the tin flotation is optimal only at low solid fraction content (up to 50 g/l) as in conditions of a normal processing of the test the selectivity is low. Initially, the most of the fine fragments are floated out at high slurry solid content ant that is why the selectivity of the process is delayed. Wollmann (1981) has established that at high slurry density the finest fragments emerge with the air upward and a only a small number of them go back to the slurry. Therefore, the unselective slime recovery is stimulated.

The rotor speed influence

The slurry solid content was not changed in those tests (50 g/l); Most probably, the selectivity is influenced in the same way.
Initially, the tests were performed in conditions of different speed of mixing by washer impeller. The tin content and recovery in dependence on the rotor revolutions are presented in Figure 3

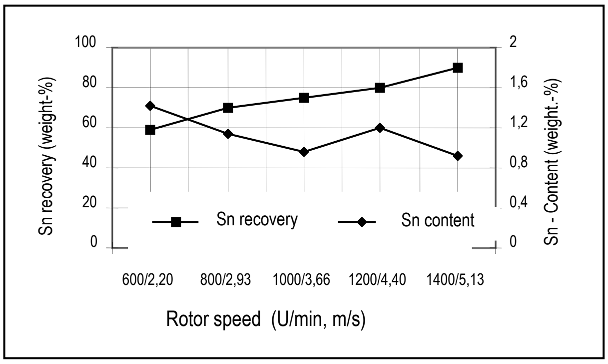


Figure 3. Sn content and u Sn-recovery depending on the rotor speed of washer impeller, collector: P-184, 750 g/t, pH 5.

Initially, the recovery values were scientifically increased by increasing of the speed. It was 60% at 2,4 m/s and at 5,5 m/s – 90%. Simultaneously, the concentrate content decreased from1,4 to 0,8%.

As it could be seen in the figure 3 the best conditions are obtained at 1200 min. In that case the achieved Sn content is 1,2% at 80% of recovery. It appears that a significant improvement of the selectivity is not possible to achieve as by solid content change as by rotor revolutions adjustment.

In contrast of that , the better results were obtained by specific air consumption decrease – 20 cm^3/cm^2min and the parallel collector concentration increases up to 1250 g/t, accompanied by a tangible selectivity increase.

Tests were performed in the same conditions for establishing of the rotor speed influence by using of a pin mixing system. The results are shown in the Figure

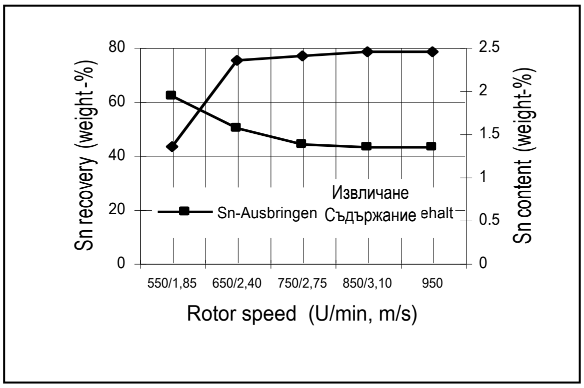


Figure 4. Sn content and Sn recovery depending on the rotor speed of pin impeller, collector: P-184, 750 g/t, pH 5.8

Here, for a difference of the results obtained for a washer impeller, now the Sn recovery from the concentrate decreases when the rotor speed increases. In the same time, a Sn content increase is observed up to 2,5% at revolutions of 850 min^{-1} but the recovery here is only 45%.

The result differences obtained for the different impellers could be explained by the different turbulence degree. According to Shubert (1977, 1979, 1979, 1982) exactly the turbulence is an important factor in the processing of the flotation with mechanical flotation devices. The investigations by different impellers show Shubert (1982) that in the process of the flotation of fine fragments of silvine parallel to the increasing the thickness in the cell at high rotor speed the valuable component content in the concentrate increases. From the other hand, it is established for the fine tin ore classes the increased recovery values accompanied by lowering of the content are observed with the increasing of the rotor speed Shubert (1977).

The authors find the conclusion that a specific low power is necessary for the coarse fragments flotation, and a high speed is necessary for the fine fragments flotation for achieving of high recovery values.
The investigation results and the conclusions do not fit the separate results as whole.

It could be observed during the washer impeller tests that there is not a strong agitation of the slurry surface at rotor speed up to 5,5 m/s. There is strong slurry agitation is running at the very near of the rotor which helps the fed air and solid material dispersion as well as an intensive collision in the solid – bubble system.

In spite of the high power it is possible a calm foam layer to be formed at the slurry surface because directly on the stator runs a negligible influence of mineralized air bubbles. The speed increase of a washer impeller leads to an increased possibility of collision between the bubbles and solids caused by the increased power. The influence on the mineralized bubbles strongly decreases in the upper part of the cell because of a significant turbulence decrease observed in the area over the stator. It could favour the rock solids extraction in the foam and which are very slightly attached to the air bubbles.

In a contrast of that the slurry is is mixed by means of strong turbulence not only in the stator area but also on the surface in the case of using of the finger impeller. A strong interaction of the mineralized bubbles runs in the whole cell volume and as a consequence it increases with the mixing speed.

Te last leads to an increased coalescence of the fine solids on the mineralized air bubbles and to an extraction increase. There are also slightly attached rock fragments and therefore a selectivity increase is observed.

Profitableness consideration

The flotation profitableness of fine tin solids is investigated in a plant with capacity of 100 t/d.

Investment costs:	**млн.DM**
1. Machines and apparatuses	0,875
2. Electrical equipment Including measurement devices	0,26
3. Pipes and water pipes	0,085
4. Buildings and steel constructions	0,485
5. Erection and internal industrial supervision	0,221
6. Engineering	0,26
7. Contingencies	0,085
Installation costs	**2,271**
8. Packing, transport and Insurance (5% of 1÷4)	0,085
Total costs	**2,356**

The production costs calculations are made on the base of 300 work days per annum in the following capital conditions (15% of the investment costs), i.e. 11,78 DM/t.

Energy consumption

The necessary electrical energy power for the flotation plant, including agitators, pumps and dehydratation is about 170 KW
The specific costs for the energy consumption are calculated on the base of an initial current price of 0,15 DM/KWh and they amount of 14 KWh/t, i.e. 2,1 DM/t.

Reagent costs

Type of regent	Reagent consumption, g/t	Price, DM/kg	Costs, DM/t
P-184	750	13,5	10,13
H_2SO_4	15.000	0,12	1,80

Specific costs for reagents: 11,93 DM/t

Total industrial costs:

Capital costs:	11,78 DM/t
Energy costs:	2,10 DM/t
Reagent costs:	11,93 DM/t
Total:	25,81 DM/t

DISCUSSION OF THE RESULTS

The multistage de-sliming for a fine solid separation in the process of the tin flotation is up to now applied in the all still working flotation plants in the process of fine tin ores dressing.

It is accompanied by losses which amount of about 40% of the initial ore material. In the present investigation an attempt was made to be decreased flotation losses of the slimes
For the purpose were performed tests with flotation machines having finger and pin impellers for the optimization of the flotation conditions. The values of the recovery in the pin impeller flotation were about 60%. The same values amount of 65 – 80% if a washer impeller is used. The high recovery values could be achieved if the washer impeller flotation is performed at high rotor speed – 5,65 m/s, and the impurities removal operations run at low revolutions.

Similar results could be expected also by using of pin impeller when for the basic flotation are used low revolutions and the impurities removal operations are performed at high revolutions.

A fact more should be taken into account that the heavy metal ions content increases if the water flotation is used and they stimulate the coagulation of the slurry.
An intensive agitation of the slurry by rotor speed increasing is required in this case. The finger impeller high revolutions lead to (as it is seen from the results) decreased recovery values. A modified pin impeller could be applied more successfully in the process of fine-grained ore flotation.

As it could be seen from the performed economic estimation of the profitableness that the flotation plant existing is economically reasonable only in the case if the tin concentrate content is more than 7,5% and recovery is more tham 70%.

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Antoaneta Boteva, Hristina Petrova: Influence of the pulp density by pulp oxid processes, University of Mining and Geology “St. Ivan Rilski” Sofia, Bulgaria, 2002

Препоръчана за публикуване от
катедра “ ”, МТФ

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.87-89

INTERRELATION BETWEEN THE CONSTITUTION OF THE THREE-PHASE FROTH AND THE DENSITY OF THE FLOTATION PULP

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ABSTRACT
The constitution of the three-phase froth is decisive for the selectivity of the running flotation process. The influence of the density of the flotation pulp is investigated as on the process of secondary enrichment of the concentrate also on the selectivity of the process of precipitation of the minerals form the three-phase troth with the drainage waters. This connection of secondary enrichment is investigated as contrary to the hydrophobicity of the minerals and thus with the size of the mineral particles. Key words: three-phase froth, solid mass, lamella liquid.

INTRODUCTION

The flotation like a process running at the boundary of three phases in contrast to a number of other technological processes at which a fundamental role plays the two-phase froth, prevails over the properties and configuration of the three-phase froth. This is the froth that except some air and water contains a further solid particles, stabilized over the air bubbles and forming the three-phase froth. Its important parameters have a direct relation to the process of selection of the minerals and they are:

- modification of the constitution of the three-phase froth as a whole and the form of the bubbles in particular on the height of the froth layer;
- the quantity of solid mass arrested in the three-phase froth;
- the quantity of some lamella liquid and the velocity of its drainage;
- kinetics of breaking of the three-phase froth;
- the modification of the composition of the solid mass on height in section.

From everything said above in follows that between the parameters characterizing the three-phase froth and the selectivity of the froth process exists a relationship. From this relationship follows the possibility for an existance of a relation between the density of the flotation pulp and the constitution of the three-phase froth. Such relationship in a technological aspect examines like a component of the relationship of the selectivity of the flotation from the density of the flotation pulp.

TECHNIQUE USED IN THE INVESTIGATION

A special technique is prepared for a realization of the investigation described below. Some elements were completed for it like these:

- a glass pipe of height 1 m and diameter 25 mm. In the lower end of this pipe is fixed shot filter which feeds some air under pressure from a compressor. The compressor is supplied of a precise manometer and a flowmeter for the air under pressure;
- the glass pipe uses like a flotation machine for a cleaning operation while the flotation runs at a variable content of the solid phase in the pulp. The litre weight of the flotation pulp measures when is fed for cleaning;
- additional reagents do not feed;
- a constant residence concentration of the flotation reagents holds up while the dilution for a change of the density of the pulp accomplishes with some filtrational water from rougher flotation;
- the concentrate for each experiment extracts by means of flotation of copper ore in the scheme of the figure 1;
- the glass pipe in the upper end is graduated in 5 mm where the correspondent froth layer extracts by means of siphoning.

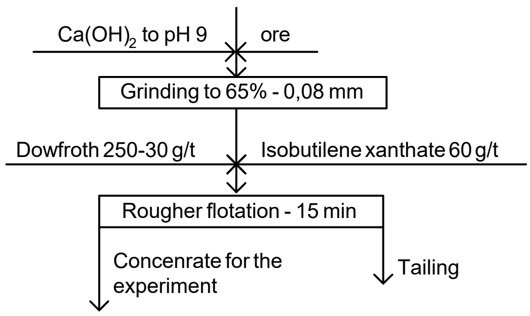

Figure 1. Flowsheet of the experiments for receiving of collective copper concentrate

EXPERIMENTAL PART

Three series of experiments are carried out. The first serie of experiments had for a purpose to check the relation between the height of the froth layer and the content of solid by weight in the pulp. Received results are given in figure 2. They show that the relationship between the thickness of the froth layer and the content of solid measured in percents, is not proportional and has exponential character.

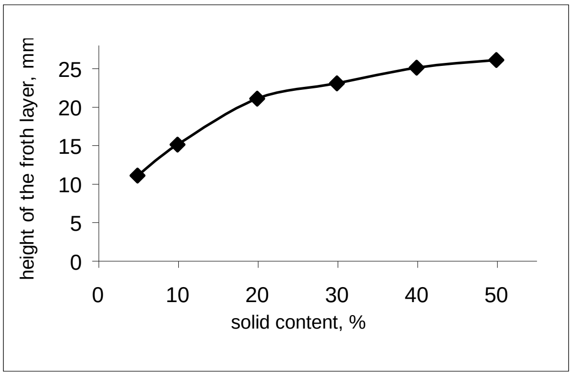

Figure 2. Relationship of the thickness of the froth layer from the percent solid in the flotation pulp

The second serie of experiments investigates the relationship between the content of solid in the pulp and the change of the content of copper by height of the froth layer. The layers were taken down with a thickness 5 mm by means of siphoning. Sum probe of 10 experiments was given for an analyses. The received results are given in figure from 3 to 5.

The kinetics of the shrinkage of the three-phase froth in a relationship of the content of a solid mass in the flotation pulp was followed by means of the third serie. The received results are given in figures 6 and 7.

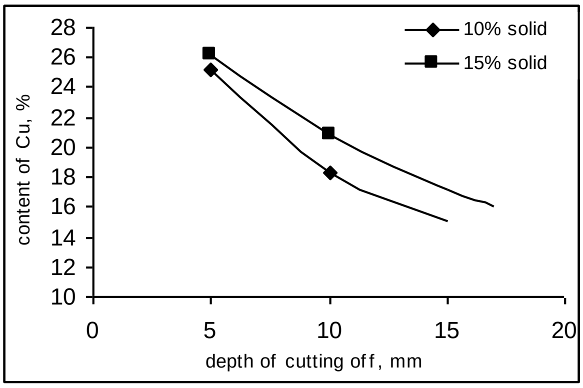

Figure 3. Modification of the content of copper by height of the froth layer at content of solid in pulp 10 and 15%

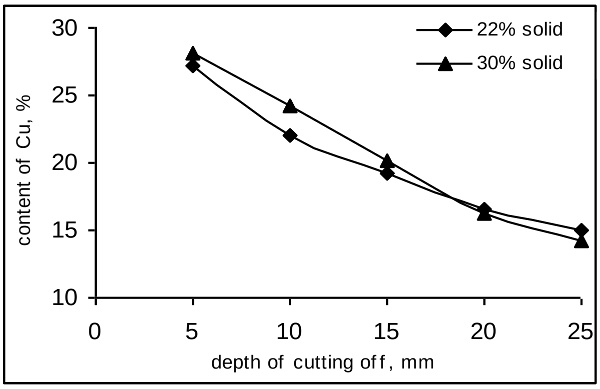

Figure 4. Modification of the content of copper by height of the froth layer at content of solid in pulp 22 and 30%

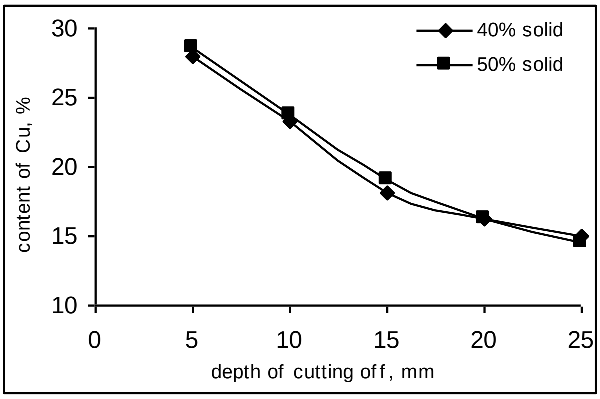

Figure 5. Modification of the content of copper by height of the froth layer at content of solid in pulp 40 and 50%

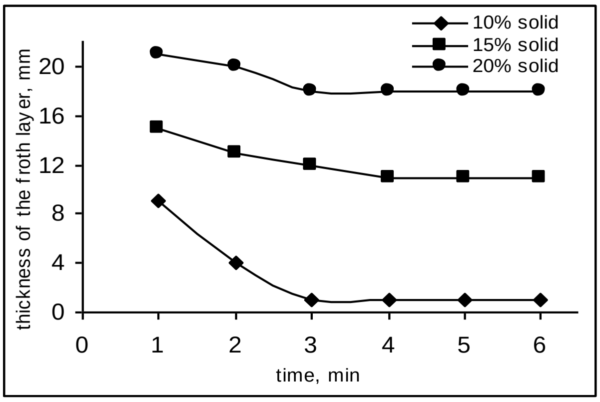

Figure 6. Kinetics of shrinkage of the froth layers at 10, 15 and 20% solid phase in the flotation pulp

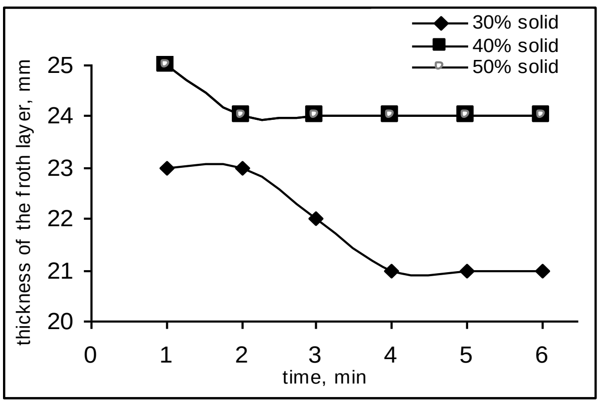

Figure 7. Kinetics of shrinkage of the froth layers at 30, 40 and 50% solid phase in the flotation pulp

DISCUSSION OF RESULTS AND CONCLUSION

The results received from the conducted investigation show the following:

1. The density of the flotation pulp influences thoroughly on the process of the secondary enrichment of the concentrates in the froth layer.
2. The secondary enrichment of the concentrates is caused from the differences in the hydrophobicity of the mineral particles participating in them. The drainage waters at the coalescence of the air bubbles in the froth layer carry away with themselves more hydrophilic particles which do not succeed in forming an three-phase perimeter of watering after the coalescence of the bubbles. This succeed to realize only sufficiently hydrophobic particles.
3. The secondary enrichment is more strongly expressed at flotations with lower density of the pulp. The degree of the enrichment decreases when the density increases. The most probably reason is higher degree of filling of the boundary water-air with flotation particles still in the process of flotation. This competition after a definite density of the flotation pulp allows the flotation only to sufficiently hydrophobic mineral particles therefore the selectivity of the process increases.
4. The faster breaking of the froth at the low density of the pulp most probably is connected with lower degree of mineralization. According to the Rebbinder's theory, the mineral particles in the three-phase froth mineralizes it.
5. The practice in leading of the flotation into cleaning operations at low density of the flotation pulp is wrong. The flotation front of the cleaning operations has to calculates at a density that depends on the density of the concentrate in rougher flotation at a minimum expense of some transport water. It is advisable for that purpose the cascade arrangement of the cleaning operations independently on the type of the flotation machnes.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp. 91-93

ROLE OF CLEANING OPERATIONS AT THE SELECTIVE FLOTATION FLOWSHEET

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ABSTRACT
The importance of cleaning operations at the selective flowsheet of a processing of mineral raw materials defines thoroughly from their decisive meaning for the receiving of the concentrates of the necessary consumer properties. The preparing of norms for the applying of the cleaning operations allows their applying in an optimum extent while the significant decreasing of the extraction of the fundamental components of interest to us prevents. The necessity of an additional milling of respective intermediate products examines in the article. Key words: cleaning operation, flowsheet of flotation.

INTRODUCTION.

The construction of the flowsheet by which a given raw material will be processed is a key moment at the dicing of the ore dressing plant. The flowsheet is a sequence of operations whom ought to put under the processed raw material. The cleaning operations are an important constructive part of the flotation flowsheet. The right inclusion requires precisely understanding of the role, which they can build in the common sequence of operations realized at the processing of raw materials.

VARIANTS OF AN UTILIZATION OF THE CLEANING OPERATIONS IN THE FLOTATION FLOWSHEET.

The fundamental variants for an utilization of the cleaning operations in the flotation flowsheet are given in the figures 1-6 but the factual variety is much greater.

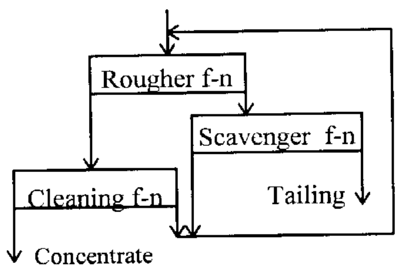

Figure 1. Flowsheet with rougher concentrate cleaning.

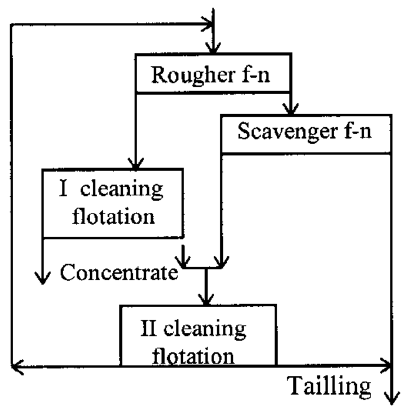

Figure 2. Flowsheet with middle products cleaning.

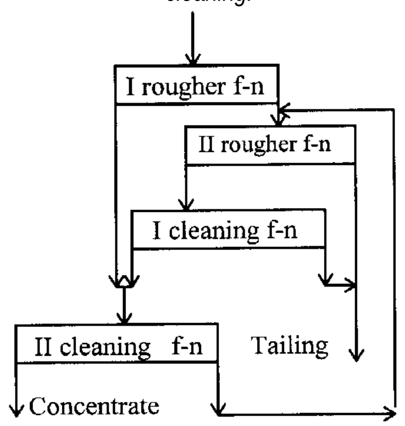

Figure 3. Flowsheet with separate cleaning of the concentrate from I and II rougher flotation's.

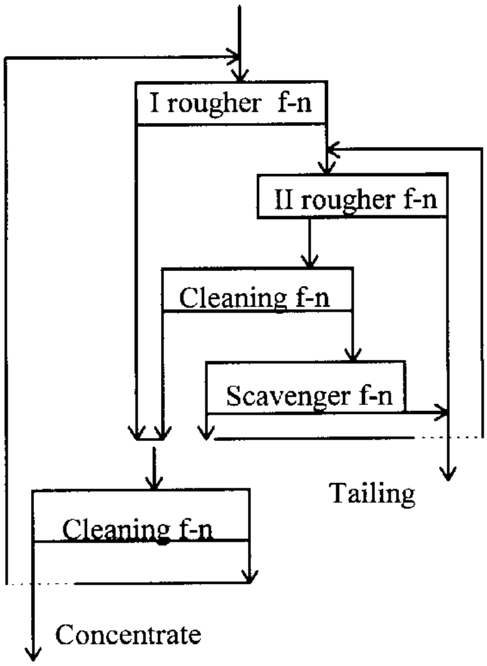

Figure 4. Flowsheet with two rougher operations and with scavenger of the middling products.

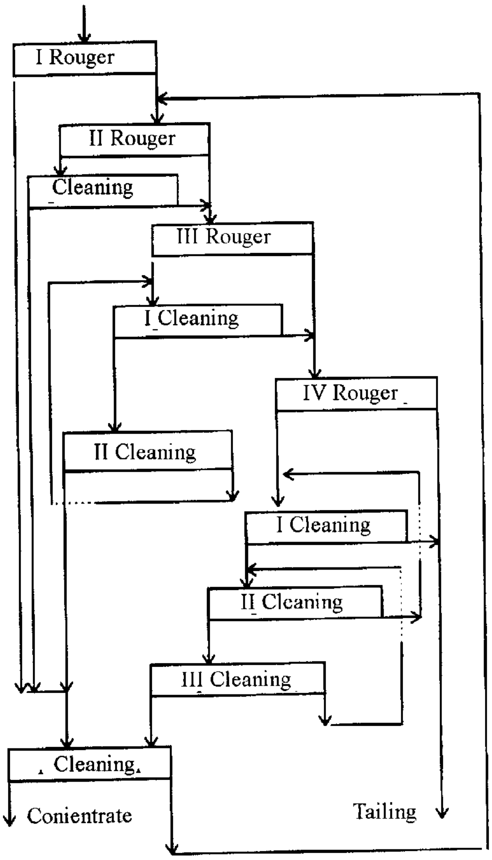

Figure 6. Flowsheet with more rougher flotation operations.

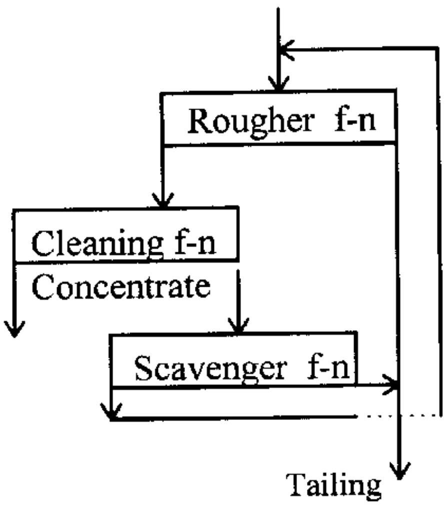

Figure 5. Flowsheet with scavenger of the middle from first cleaning.

ANALYSIS OF THE VARIANTS FOR AN UTILIZATION OF THE CLEANING OPERATIONS IN THE FLOTATION FLOWSHEET.

The utilization of the cleaning operation realizes in two fundamental variants:
1. Without preliminary additional milling of the product fed for a cleaning.
2. After an additional revealing by means of an additional milling of the twins in products fed for a cleaning.

Two pointed out variants can realize with an additional feeding flotation reagents and without an additional feeding of flotation reagents. Each one of the pointed out variants has its own special features of a realization. The accepting of the variant that includes about a given cleaning operation ought to realize only after an investigation in detail of the material constitution of the cleaned product. The practice has proved the following relationships of the efficiency of the cleaning operation from the material constitution of the products:
1. When in the product for cleaning twins are between component of interest to us and the other components participant in the ore, then an additional milling is not necessary. In this case ought to be divided grains sharply distinguishing by their hydrophobicity . Then an utilization of additional depressors is not necessary.
2. When in the product exist twins of the component of interest to us but it is easy regrading mineral, then more

suitable the cleaning operation leads without additional milling and reagents while twins remains in the chamber product whom after control floatation separates a concentrate. This concentrate puts under an additional milling. In this case runs away the possibility for depression of the grains of the mineral of interest to from slime coatings of the slime mineral.

3. When twins in the product before cleaning are between ore minerals and minerals of the inserting rocks then preliminarily additional milling of the product put under cleaning forces by all means.

4. The cleaning of the concentrates without decreases the common extraction from the ore of the component of interest to us is possible only in the cases of:

- a selective disclosure of the mineral grains before cleaning;
- a reducing to minimum of the slime forming;
- preliminarily rubbing in of the mineral grains in order to an increasing of flotabitility of the grains of the components of interest to us;
- a removal of the residue concentration of collectors and activators in the flotation pulp.

5. The increasing of numbers of the cleaning operations in many cases leads to losses that could escape if flotation machines ensuring better secondary enrichment of the concentrates apply in the frame of the froth layer.

6. In an utilization of cleaning operations in the floatation flowsheet , its place and the number of the cleaning operations ought to select according to the kinetics of a floatation of the basic minerals.

CONCLUSION

The right choice of the variant, the place and the number of the used cleaning operations decides to a great extent the possibility of receiving qualitative concentrates at high degree of an extraction of the component of interest to us.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp. 95-100

SULFATE-INDUCED DEGRADATION OF CEMENT STONE

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ABSTRACT
The study treats sulfate ion transport in cement stone and resulting material degradation. Those phenomena are essential for the safety assessment of concrete containers for nuclear waste disposal, for the rehabilitation of civil and industrial constructions, for preserving ancient cultural heritage etc. The process is studied theoretically and experimentally, and a survey of various experimental approaches is made. A diffusion model is designed, considering change of the material structure- void filling with ions and precipitants, liquid push-out of capillaries etc. Long-term experiments of cement stone saturation with sulfate ions are performed and material degradation is registered. Specimen degradation is explained, considering liquid motion within the material pores, which accompanies the transfer process.

INTRODUCTION

Cement-based composites have been always in researchers' focus, since they constitute the basic material for building industrial and civil constructions. Consider the unfavorable effect of air, water and soil polutants on ecology and on structure durability. Regard also the strict requirements for safety of nuclear waste containers, means of protecting industrial and civil constructions, means of preserving ancient cultural heritage etc. Then, the assessment of the mechanism of chemical corrosion of cement-based composite materials becomes crucial. The work presents a survey of some experimental and theoretical studies of sulfate attack on cement stone, performed by our research group. The interest to sulfate aggression is verified by the fact that sulfates are common polutants of environment, and local mineral waters are sulfate-containing, too. The results seem useful in following damage of cement-based structures which undergo sulfate attack and in assessing structure durability.

EXPERIMENTAL STUDIES OF SULFATE ATTACK ON CEMENT STONE

Consider sulfate degradation of cement stone. Sulfate attack develops in time and to model it, one needs to perform long-term experiments of saturating cement stone specimens with sulfate ions. The scheme of specimen saturation is shown in Fig. 1. Cement-stone prisms and/or plates are immersed in water solution of Na_2SO_4 and kept there for a definite period of time. Note also that the concentration of the solution is different, so that its effect on material degradation can be assessed. Material is sulfate resistant cement 35, type "Devnya", whose chemical composition is given in Table 1. Material compression strength is found after keeping the specimens in solutions of different sulfate concentration. The results are shown in Fig. 2 and discussed in detail by Gospodinov, Kazandjiev et al (1996).

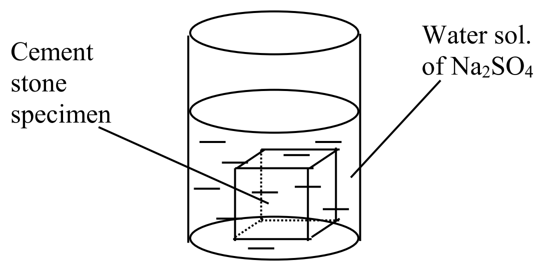


Figure 1. Scheme of specimen immersion in sulfate solution

Table 1. Mineral composition of SRP cement, type 35 Devnya, (% by weight)

C_3S	42.83
C_2S	27.75
C_3A	2.81
C_4AF	16.29
Na_2O	0.64
K_2O	1.46
MgO	0.80
Others	7.42

As said, Fig. 2 illustrates change of the compression strength in time, which displays local maximums and minimums. One can explain this effect by material saturation with ions and chemical products, resulting in formation of micro-voids, i.e. in strength decrease. Then, subsequent void filling with compounds, i.e. subsequent material strengthening follows etc. The process seems to be periodical and reflects the change of material structure due to the sulfate attack. As said, the results are important for the assessment of the modification of cement stone structure. It is assumed that due to the sulfate attack, the specimen divides into two layers - an external, corrupted one and an intact internal core of cement stone. Moreover, using the approach given in (Mironova, 1997; Gospodinov et al 1998; Gospodinov et al 1999), one can find the elasticity modulus of the corroded material, knowing in advance that of the cement stone. Fig. 3 shows specimen division into layers.

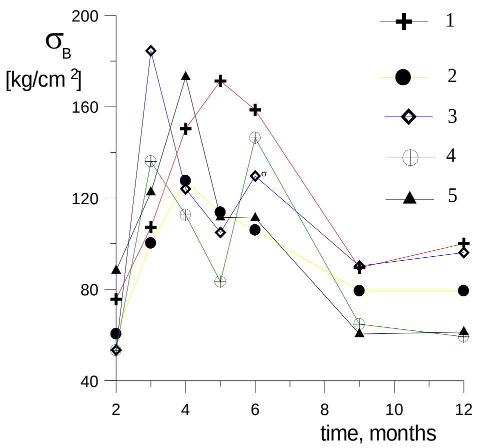


*Figure 2 Change of material compression strength in time, depending on the solution concentration – **1**-0.5 % Na_2SO_4, **2** – 1% Na_2SO_4, **3** – 3% Na_2SO_4, **4**- 5% Na_2SO_4, **5** – 10% Na_2SO_4,*

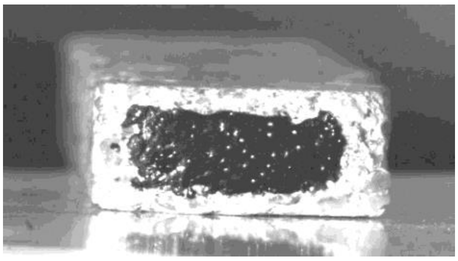


Figure 3. Specimen division into layers. Solution concentration-3%; immersion time – 9 months.

Another effect of structure modification of the cement stone is material swelling due to the sulfate aggression. Data on the change of specimen weight and density are given in Table 2 and are shown in Fig. 4 and Fig. 5, as depending on the solution concentration, while immersion time is 5 years. As seen, material density decreases with the concentration increase. This is an important experimental fact, attributing to the clarification of the modification of the cement stone structure and proving that although voids are filled with chemical products and ions, void formation is prevailing over their filling. Hence, material is subject to total degradation in some cases, as the experiments show.

Table 2. Change of specimen weight and material density

Solution concentr. C, %	Cement stone desnity, [g/cm³]	Specimen weight G, [g]	Weight difference ΔG, [g]
0	2.317	1.733	0
3	1.762	1.938	0.205
5	1.683	2.658	0.925
10	1.673	4.978	3.244

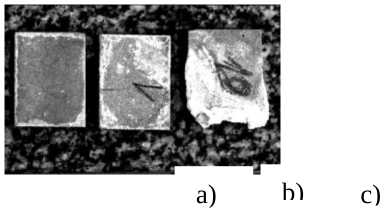


Figure 4. Material swelling and degradation due to the sulfate attack: a)-3% solution of Na_2SO_4; b)-5% solution of Na_2SO_4; c) 10% solution of Na_2SO_4;

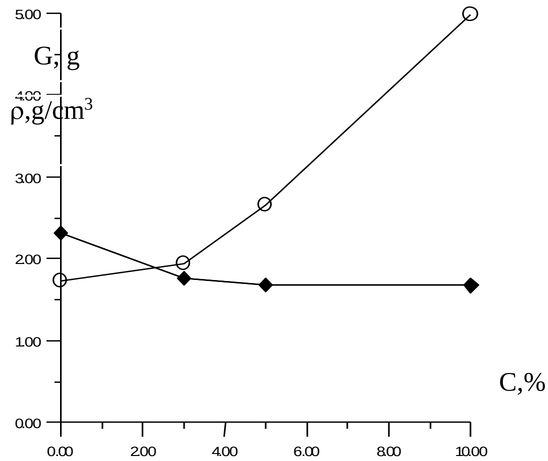


Figure 5. Change of specimen weight G and cement stone density ρ: o - G and ♦- ρ

Another important phenomenon resulting from material degradation is the formation of disturbances (cracks, voids, pores etc.) within the specimen and on its surface-Fig. 6.

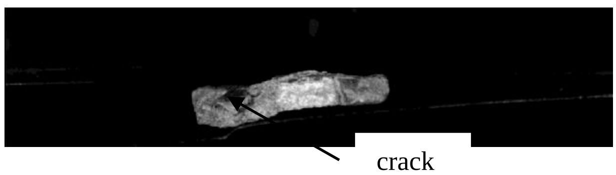
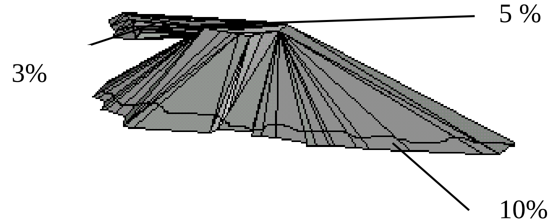


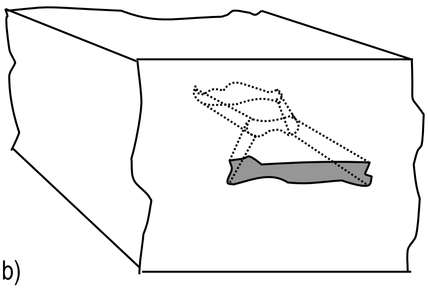
Figure 6. Crack formed on the specimen surface.

Their registration is important, since one can thus follow material degradation and change of the construction durability. To estimate the degradation mechanism, Kazandjiev et al (2002) propose a model of reconstructing material cracks, following cement stone immersion in solutions of different concentration.

Fig. 7.a shows a reconstructed crack, accounting for the different concentration of Na_2SO_4, while Fig. 7.b –its location within the cement stone specimen. The estimation is qualitative but enables one to assess the mechanism of cement stone corruption. The reconstruction is done, involving cracks formed under solution sulfate concentration of 3%, 5% and 10% and using 3D pattern recognition.



a)



b)

Figure 7. Reconstruction of a crack in cement stone specimen regarding the solution sulfate concentration; a) actual crack reconstruction; b) crack location

THEORETICAL MODEL OF SULFATE DIFFUSION

Modelling of corrosion due to ion aggression on cement composite is studied in (Atkinson, Haxby *et al*, 1988) and (Ratinov and Ivanov, 1977). Theoretical models are based on the diffusion equation, disregarding the heterogeneous chemical reaction occurring between the ions and the composite matrix. Diffusion of weak electrolytes in a porous medium is considered in (Sherwood, Pigford et al, 1982) and (Zaykov, Yordanskiy *et al*, 1984). The 1D steady case of pore filling in a catalyst and its effect on the reaction rate is considered in (Matros, 1982). A 2D model of the diffusion of sulfate ions in cement composite is proposed in (Gospodinov, Kazandjiev *et al*, 1999) and (Mironova, Gospodinov *et al*, 2002). We refine the model of ion transport, following several successive steps - account for material porosity, account for ion transport and accompanying chemical reactions and further account for the liquid push out of the cement stone capillaries. This is shown in (Mironova, Gospodinov et al, 2002) and (Kazandjiev, Gospodinov et al, 2002). Hence, we consider here 3D model of sulfate ion diffusion in a prismatic specimen of cement stone, which gives plausible results and generalizes 1D and 2D models, designed so far. Before immersion, the specimen has been kept for some time in drinking water. The vessel volume is assumed to be large enough, and diffusion does not change the concentration of sulfate ions in the solution, which remains constant. Owing to the concentration driving force, ions enter the liquid that fills material pores. They are formally treated as capillaries, shaped as straight circular cylinders and with symmetry axes parallel to the coordinate axes. As a result of the occurring heterogeneous chemical reaction, chemical products precipitate on the walls of the capillaries, partially filling them. Capillary filling and volume decrease yields liquid push out of the capillaries. This process is directed from the specimen internal area to the solution, i.e. its direction is opposite to that of diffusion.

Thus, the following balance equation for an elementary volume of the area is considered, which generalizes the 1D and 2D transfer processes:

$$\frac{\partial c}{\partial t} = \operatorname{div}\left(D_{\mathrm{eff}} \operatorname{grad} c\right) - \operatorname{div}(\mathbf{V}c) - k(1-k_z q)^2 c \tag{1}$$

where the spatial operators $\operatorname{div}(\ )$ and $\operatorname{grad}(\ )$ read

$$\operatorname{div}(\ ) \equiv \sum_{i=1}^{R} \frac{\partial}{\partial x_i}(\ )_i \qquad \operatorname{grad}(\ ) \equiv \sum_{i=1}^{R} \mathbf{j}_i \frac{\partial}{\partial x_i}(\ )_i \tag{2}$$

and $R \in [1,2,3]$ is the spatial dimesions of the area considered. $x_1 = x;\ x_2 = y;\ x_3 = z$. The last term in the RHS of equation (1) is a source term, which models the heterogeneous chemical reaction between the capillary wall material and the sulfate ions in the solution. The term accounts also for the change of the reacting surface, due to capillary filling. Quantity $c(x_1,...,x_R,t)$ in equations (1)-(2) is the concentration current value, while $q(x_1,...,x_R,t)$ is the quantity of chemically reacted ions at point $(x_1,...,x_R)$ and at moment *t*. *k* is the rate constant of the heterogeneous chemical reaction and k_z denotes the coefficient of capillary filling. The effective coefficient of ion diffusion in the whole volume, accounting for pore filling is:

$$D_{\mathrm{eff}} = D(1 - k_z q)^2 \quad , \tag{3}$$

where

$$D = k_{\mathrm{diff}} \exp[\beta(c(x_1,..., x_R, t) - 0.5c_0)] \quad . \tag{4}$$

The denominator k_{diff} in Eq. (4) is the coefficient of diffusion of sulfate ions in the whole solid volume, consisting of cement matrix and cavities filled with liquid. It accounts for the material porosity and grain structure, and for the capillary shape. Constant β is a fitting parameter, while c_0 is a concentration characteristic value - the concentration of the solution where the specimen is immersed and kept for a definite period of time. Projections V_i, $i = \overline{1, R}$, of velocity *V* in eq. (2) denote the average velocity of the liquid flow in the capillary, along axes x_i, $i = \overline{1, R}$.

The value of the solution concentration is given as a boundary condition on the interface surface "water solution – cement stone". Due to symmetry of the areas considered, symmetry conditions are given on boundaries $x_i = L_i / 2$, $i = \overline{1, R}$.

The initial ion concentration within the specimen volume is taken to be zero, since the specimen is previously kept in drinking water.

The quantity of chemically reacted ions *q*, at a moment *t* and at a point with fixed coordinates $(x_1,..., x_R)$, can be found by integrating the concentration value at that point, taken as function of time *t*.

$$q(x_1,..., x_R, t) = \int_o^t k\, c(x_1,..., x_R, \tau) d\tau \quad , \tag{5}$$

Consider the velocity component V_i, for fixed values of the other coordinates x_m, $m \neq i$. Then, the following integral along the capillary should be solved in the interval $[0, x_i]$.

$$V_i = \int_{\eta=U}^{\eta=G} -2kk_z(1 - k_z q) c\, d\eta \; , \tag{6}$$

$$U = L_i / 2 - x_i \;, \; G = L_i / 2, \; i = \overline{1, R}$$

Since the origin of the coordinate system is the symmetry center of the specimen, the velocity is zero for $x_i = 0$ and maximal for $x_i = L_i / 2$. The velocity direction coincides here with the positive direction of axis Ox_i. V_i is found by calculating the integral (6). Note that the lower integration limit here is $L_i / 2 - x_i$, and the upper integration limit is $L_i / 2$. Thus, one can get the velocity $V_i(x_1,..., x_R, t)$ at each point of the area under consideration. Velocity components $V_m(x_1,..., x_R, t), \quad m \neq i, \quad m \in [1,..., R]$ are obtained in the same manner.

NUMERICAL SOLUTION

The equation of transfer (1)-(2), together with the boundary conditions and the initial condition, pose the initial non-steady boundary-value problem. It is completed by the integral relations (5) needed to find the quantity of chemically reacted ions, as well as by integral relations of type (6), needed to find liquid velocity field in the capillaries. An implicit difference scheme to solve numerically the formulated diffusion problem is used. The difference value problem, for a given time *t*, is reduced to the solution of a linearized system of algebraic equations, which has a diagonal and weakly filled matrix. The algorithm enables one to model numerically volume sub-areas with completely different conductivity – inert filler, inclusions, reinforcement etc. Due to problem non-linearity, an internal iteration process is used.

The numerical results are found for the following values of the dimension and dimensionless constants, as given in (Mironova, Gospodinov et al, 2002):

-coefficient of ion diffusion in the water solution

k_{diff} = 0.361 x 10^{-9} m^2/s;

-fitting parameter participating in Eq. (11):

β = 0.2917 m^3/kg;

-constant of the chemical reaction rate:

k = 0.305 x 10^{-7} s^{-1} ;

- coefficient of pore filling:

k_z = 0.05 m^3/kg

Those values are obtained for specimens, molded of cement paste of sulfate resistant Portland cement 35, type "Devnya" - see Table 1.

RESULTS AND DISCUSSION

As outlined above, the experimental evidence proves a number of mechanisms that develop in the cement stone – change of the material structure due to pore filling with ions and precipitants, liquid push out of the material capillaries, cement stone loosening due to the subsequent formation of material voids, resulting change of the material compression strength etc. One can plausibly account for these effects by using the model of ion transport (1)-(4) and relations (5) and (6).

The basic numerical results are found for a prismatic specimen, kept in 3% water solution of sodium sulfate. The content of the sulfate ions in the solution is 20.282 kg/m^2. The solution sulfate concentration is assumed to remain constant in time. We consider at first ion transport in a cement stone bulk without and with an arbitrary inclusion. The calculated isolines of sulfate concentration are given in Fig. 8 and Fig. 9.

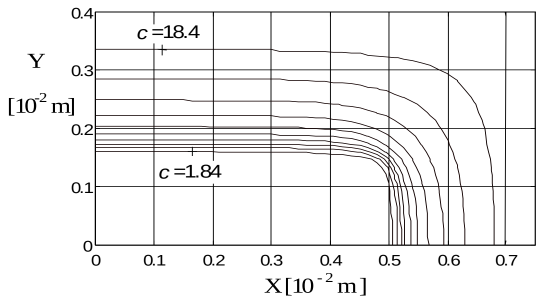


Figure 8. Isolines of sulfate concentration in a cement stone specimen without an inclusion

The comparison of both figures shows that the isolines deform due to the presence of an inclusion. Hence, some effects resulting from the sulfate attack can be established near the inclusion surface, where isolines compact as seen in Fig. 9.

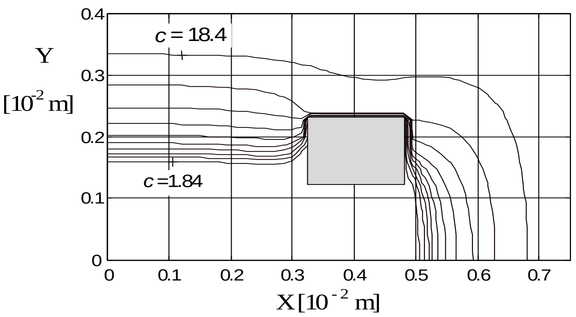


Figure 9. Isolines of sulfate concentration in a cement stone specimen with an inclusion.

Next, to illustrate the capabilities of the 3D model and those of the numerical algorithm, it is assumed that two cylindrical bodies (inert filler or reinforcement) are located in 1/8th of the specimen volume – Fig. 10.

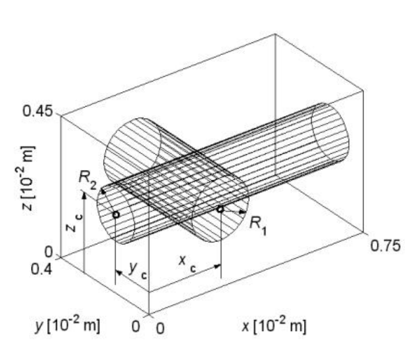


Figure 10. Schematic representation of the inclusion sub-area in 1/8th of the specimen volume

The calculations are performed for 3 and 6 months of specimen immersion in the solution,. Fig. 11.a and Fig. 11.b show the isosurface corresponding to ion concentration within the specimen c_{const} = 5 [kg/m^3]. The effect of the sub-

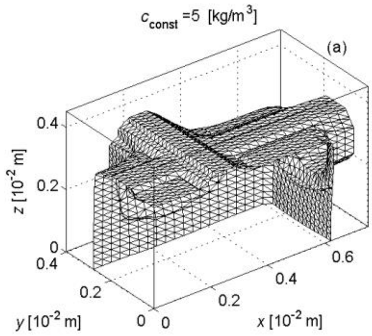


Figure 11.a. Isosurfaces for concentration c_{const} = 5 [kg/m^3] after 3 months of specimen immersion

area of inert filler on the shape of the isosurface is clearly visible.

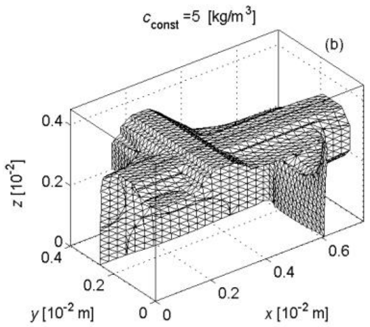


Figure 11.b. Isosurfaces for concentration c_{const} = 5 [kg/m^3] after 6 months of specimen immersion

CONCLUSIONS

The presented study outlines a mathematical model of ion transfer which comprises simultaneous effects, such as pore filling and liquid push out of the capillaries. It enables one to study processes of transfer of sulfate ions in cement stone, giving adequate explanation of some experimental data and allowing for a better account of the real conditions of mass transfer. The numerical algorithm developed is effective to solve different practical problems and to investigate real processes, which take place in structures or structural elements. The calculation results are in satisfactory agreement with the experimental evidence.

ACKNOWLEDGEMENT

This work was sponsored by the National Council “Scientific Research”, Grant No 1106/20001.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.101-104

CARBONATE MINERAL POWDERS - PROBLEMS AND SOLUTIONS

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ABSTRACT
Carbonate mineral powders are obtained by grinding of several terrestrial occurrence minerals - limestones, marbles, dolomites, chalk. Fine milling in ordinary ball mils is practically impossible without adding of surfactants, which are not always safe. The process of fine milling is violated of aggregation of particles and particles sticking to the surface of the working bodies as well as surface of the working chamber of the mill. An investigation for fine milling of dolomitezed limestones from the area of village Balsha in vibratory mills with horizontally placed working chambers was carried out. Suitable working medias and regimes without adding of surfactants were defined for the production of fine carbonate powder under normal conditions.

INTRODUCTION

Carbonate rocks are terrestrial occurrence in the nature. They are used as basic building material for centuries. Currently carbonate minerals on base of $CaCO_3$ as limestone, marble, chalk, dolomite and different transitions between them found wide application in the metallurgy chemical industry, agriculture, sugar industry, buildings, glass industry, ecology and so on.

Marketing investigations and the practice shows that market demand of carbonate powders are high. It is used in different industries according to particles’ size mainly as a filler:

- in the paper industry with size up to several microns;
- in the production of plastics (after modification with reagens)
- in the tire and rubber industry;
- in the pharmaceutics industry;
- in asphalt mixtures for tarmac road constructions.

The main requirements of the customers are in two directions:

- Granulometric content of the product;
- Minimal content of impurities;

Requirements for the granulometric composition are satisfied by the processes of milling, crushing, sieving and classification. Technological flowsheets and the specific apparatus define the final price of the product. Decreasing the size of the particles cause difficulties not only to the process of grinding but also in the processes of classification and sieving.

TECHNOLOGICAL BASES

Carbonate powder used for production of tarmac road mixtures for road constructions according to BDS should fit the following conditions for granulometric composition given in Table 1.

Table 1.

Index	Value, %
Granulometric content , %	
Particles passed trough sieve 1,25 mm	100
Particles passed trough sieve 0,315 mm	> 90
Particles passed trough sieve 0,071 mm	> 70
Moisture, %	< 1

Changes in the economic conditions in Bulgaria and applying the world standards in the area of road building change the requirements given in Table 1.

Road construction managed by foreign firms require applying of different standards, to guarantee better quality of the road coverings.

Mineral powder as an ingredient of tarmac road mixtures consist of fine particles obtained from milled limestones. It should be well dried. It should not contain lumps and should fit the requirements AASHTO T 37 given in Table 2.

Table 2.

Index	Value, %
Granulometric content, %	
Particles passed trough sieve 0,6 mm	100
Particles passed trough sieve 0,15 mm	85
Particles passed trough sieve 0,075 mm	75

Activated mineral powder is very often used as mineral filler in the tarmac road mixtures. It is made by milling of carbonate minerals and modification with activated mixture of bitumen and surfactants.

Activated mineral powder should fit the requirements for hidrophobicity and to be homogeneous. It should not get compressed with drying and should fit the granulometric composition defined by AASXTO T 37 given in Table 3.

Table 3.

Index	Value, %
Granulometric content, %	
Particles passed trough sieve 0,6 mm	100
Particles passed trough sieve 0,15 mm	90
Particles passed trough sieve 0,075 mm	80

The referred as illustration standards are not only ones used at the moment but the others foreign standards also have strict requirements to the granulometric characteristics of the carbonate fillers applied in the tarmac road mixtures. Obviously the Bulgarian manufacturers should take into consideration new realities to answer on the increased requirements.

"Ljuliacite" deposit characteristic

Ljuliacite deposit is situated in the south slopes of "Stara Planina" about 7 km near the village Balsha, Sofia region. It has occupied area "Kamiko" and "Machnov vrah" and lands on the north and east-north. The rocks comprise Jura and Triassic sediments. Middle and bottom Treas as well as upper Jura are represented. Two units are formed in the deposit region.

- Carbonate one comprises compact limestones, dolomitized limestones and limestonized dolomites.
- Limestone-mergel one comprises mergels, alevrite limestones and limestones.

Limestones as well as dolomites are rarely in pure form.

As percentage the distribution has the following description:

Limestones -	39%
Dolomitized limestones -	42,7%
Limestonized dolomites -	11%
Dolomites -	1,8%
Limestonized mergels -	5,3%

MDZ "Balsha" produces classes of crushed stone for the building purposes. The offered classes are 0 – 4 mm, 4 – 8 mm, 8 – 12 mm, 12 – 16 mm, 16 – 20 mm, 20 – 32 mm. A powder like product is collected from the aspiration of the working crusher and sieved equipment.

Despite that this powder product does not fit the requirements even of the BDS but it is successfully sold as mineral carbonate powder. This fact shows the existing lack of carbonate powders – fillers.

METHODOLOGY AND EQUIPMENT

The methodology for defining of suitable conditions for fine milling of carbonate rocks being treated in MDZ "Balsha" under the laboratory conditions in vibratory mill with horizontally placed working chamber include the following moments:

- Defining of the optimal quantity of material for milling in interrupted regime with one working media;
- Defining of optimal duration of milling with constant quantity of the milled material;
- Defining of suitable working media under optimal output of the working chamber with material.

The results of the milling are controlled by measurement of the yield of minus class 0,063 mm by dry sieving in vibratory devise.

The class of 4 - 8 mm was used for the purpose of investigation.

The chemical content of the sample defined by AES – ICP is: CaO - 54,1%, SiO_2 - 0,94%, Fe_2O_3 - 0,24%, Al_2O_3 - 0,32%, MgO - 0,34%, K_2O - 0,34%, Na_2O - 0,20%, ЗПН - 42,60%.

The sample for investigation is additionally crushed in roll crusher and the granulometric composition is shown at Figure1.

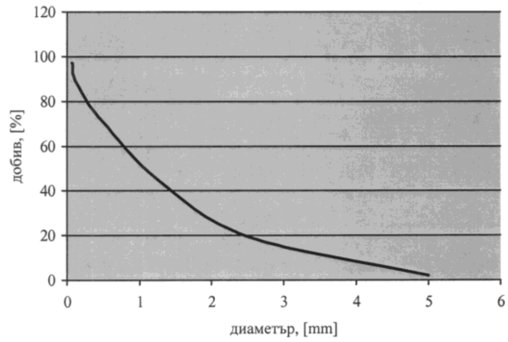

Figure 1. Granulometric content of the investigated sample.

The investigation is carried out with laboratory vibratory devise with working chambers with volume 350 cm^3. Vibratory parameters were kept constant: f – 1430 min^{-1} and A – 3,4 mm.

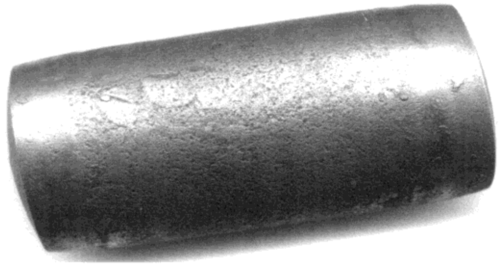

Figure 2. Working media 1 – smooth steel rod

The working medias are shown at figure 2 and Figure 3. Figure 2 shows smooth steel rod with weight of 1040 g and length 94 mm. Two working media are given at Figure 3 - smooth steel rod with weight of 155 g and length 94 mm and riffle steel rod with weight of 120 g and length 94 mm.

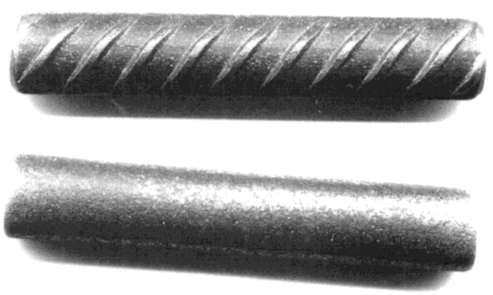

Figure 3. Working media 2.

RESULTS

The investigation for defining of the optimal quantity of the sample for milling in working volume of 350 cm^3 and working media of smooth steel rod with weight of 1040 g is carried out for the interval of 30 to 110 g with step of 20 g and duration of the milling – 3 min. The obtained results are given in Table 4.

Table 4.

Sample quantity, g	30	50	70	90	110
Observed class, (-0,063 mm),%	10,34	13,00	22,86	21,82	18,89

The investigation for defining of optimal milling duration with quality of the sample 70 g with smooth steel rod with weight of 1040 g gave the results shown at Table 5. The interval of 1 to 7 min. was observed.

Table 5.

Treatment duration, min	1	2	3	4	5	7
Observed class, (-0,063 mm),%	15,71	24,29	22,86	22,57	20,71	18,57

The investigation for different working media includes:

- Smooth steel rod with weight of 1040 g;
- Two smooth steel rods (weight of one rod – 155 g) and riffle rod with weight 120 g – working media 1;
- Three smooth steel rods (weight of one rod – 155 g) and riffle rod with weight 120 g – working media 2;

The investigation of the different working media shows the results:

- The results obtained by the milling with one smooth steel rod with weight of 1040 g and 70 g quantity of the sample are shown at Table 5.
- Vibration milling with milling media 1 (two smooth rod and one non-smooth) and media 2 (tree smooth rods and one riffle rod) with quantity of the milled sample 70 g is investigated in the interval 3 – 19 min. with step 2 min. Results are given in table form – Table 6.

Table 6.

Treatment duration, min	3	5	7	9	11	13	15	17	19
Working media 1									
Observed class, (-0,063 mm),%	18,57	22,86	30,0	32,86	39,29	46,43	51,43	55,0	58,57
Working media 2									
Observed class, (-0,063 mm),%	23,98	34,42	47,13	59,29	63,55	66,21	70,0	75,71	81,43

DISCUSSION

The effect of decreasing of the process efficiency after reaching definite stage of milling is known for dry milling of some mineral raw materials – carbonate rocks. An opposite process of aggregation and sticking of the fine milled particles on working bodies and the surface of the working chamber in observed. This process appears in different size of the different materials and depends on physicochemical characteristics, milling conditions, moisture and so on. Structural features of the mineral particles represent as defects, relief, sectility, crystal size, the character of the coalescence of the crystals an so on defines the character of the crystal destruction. The crystal destruction depends also on physicomechanical properties as hardness. The experiments for defining of correlation between these parameters and the size of the milled material does not lead to generally valid answer.

The existing practice of milling of carbonate rocks is multifarious. Combination of hit crusher and sieving, milling in two-chamber pipe mill with working media in the first chamber – steel balls and in the second chamber cilpebs combined with air classification are used under dry conditions.

Obtaining of high percentage (over 80 %) class – 0.074 mm is possible by stage milling and classification but the technological flowsheet become very difficult for maintaining and the obtained product expensive.

The investigation of vibration milling of carbonate rocks for obtaining of powder suitable for tarmac road mixtures for the road infrastructure is realized in vibratory mill with horizontally placed working chamber. The milling in the vibration mill is several time quicker than the milling in the ball mil but the process of aggregation and sticking of the fine milled carbonates is also quicker.

Possibilities for fine milling in vibration mills are hidden the chousing the proper milling medias. Smooth and riffle steel rods with different sizes are investigated as milling media in interrupted cycle. The obtained results of the milling represented as observed class – 0,063 mm give reasons for the following conclusions:

- Milling with smooth rods is impossible due to the aggregation of the fine carbonate particles;
- Milling with riffle rod is possible but the efficiency of the process is low;
- Milling with combination of smooth and riffle rods gives possibilities for obtaining of high efficiency of the process and reaching of the observed class over 80 % without usage of surfactants, without sieving or classification.

This fact could be probably explained by the impact of the riffle rod expressed by the scoured effect of the riffle rod over the working surface. Probably mechanical is the impact over the smooth surface of the steel rods. This process is possible due to the features of movement of the working media in the volume of the chamber under vibrations. The morking media take part at the same time in two movements:

- Vibration and rotation movements around the long axis of the rod;
- Vibration and rotation movements of the whole working media opposite to the direction of the vibrations around the common axis passing to the length of the working media.

From practically point of view fine milling of carbonate rocks are possible by usage of combined working media. The treatment duration could be optimized. Carbonate powder suitable for tarmac road mixtures could be obtained by optimization of the treatment duration of continuous technological process without usage of additional sieving or classification.

CONCLUSION

Vibration milling of carbonate rocks in vibration mill with horizontally placed working chambers with mixed working media of smooth and riffle rods is a technological solution of the problem of aggregation and sticking of fine carbonate particles on the surface of the working chamber and working bodies. Vibration milling with combined working media simplify the technological flowsheet for production of carbonate powders because the processes of sieving and classification becomes redundant.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.105-110

AIR BUBBLES CREATION AND BEHAVIOR IN A VIBRATORY-ACOUSTIC FIELD

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ABSTRACT
The paper presents the mechanism of air bubbles creation in a vibratory column flotation machine at the aerator vibration in vertical direction and the influence of its constructive parameters on the gas phase dispersion. The vibratory field distribution along the water column according to the applied to the aerator oscillations in water medium and air bubbles presence is researched. A hypothesis explaining the deceleration of the gas bubbles emerging velocity in a vibratory field with a vibrations energy absorption of the bubbles at their vibration as a result of the changeable sound wave pressure and the change in the interaction between the bubbles and the liquid is expressed.

INTRODUCTION

The air bubbles creation in liquids is a wide used method for different technological processes realization and it is a basic process at the mineral products processing floatation.

Regardless the aeration process application its efficiency is determined by the shared surface at the bound gas-liquid, for which formation a definite energy has to be used up. Besides, conditions for the obtained bubbles size, which determines their ability to transport a definite number of solid particles to the froth layer are laid down by the process application at flotation processing.

Mechanical, pneumatic, hydraulic and other combined methods for the gas phase dispersion by different energy types are used for the process realization. Because of the continuous contact time of the dispersed gas phase and solid particles it turns out that there are mechanically transported particles in the froth layer so an additional water sprinkling of the froth layer is used for the concentrate quality improvement.

In the University of Mining and Geology "St. Ivan Rilski" – department "Mineral technologies" a disperser at which the gas phase is realized by the use of vertical vibrations with defined frequency and amplitude has been developed. The dispersion method created as a result of the long work in the department on the use of the vibratory-acoustic technologies for the technological processes intensification gives the opportunity to affect not only the bubble formation process but the tree phases in the entire work chamber volume as well.

GAS PHASE DISPERSION

The gas phase dispersion in liquid medium is realized by its transmission through an annular slot in the upper disperser part fig. 1. The air quantity (fig.1 - 1) depends on the pressure difference in the supplied air and the water column in the flotation machine

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When the disperser does not vibrate the air is dispersed in big bubbles, which are obtained at random places on the annular slot (fig. 1 - 2) due to its inability to limit bubbles of a certain size because of the slot type. At the disperser vibration the bubble formation process character considerably changes – when the vibrations frequency and amplitude increase there is a decrease in the obtained bubbles size and an increase in their number. By the experiments and surveillances it has been ascertained that the bubbles number and respectively – their size depend on the vibrations velocity $u = \frac{dx}{dt} = A\omega\cos(\omega t + \varphi)$, which could be identified with the expression u_V=Af – a product of the amplitude and frequency oscillations.

At the disperser vibration the growing bubble from a random annular slot sector is affected by two forces: the lifting force of the liquid removed by the bubble and the resistance force caused by the liquid flow circumfluent the bubble and moving on the inclined disperser face. The former has always had up direction, while the latter is alternating –- depending on the oscillations phase it could be directed as the lifting force is or in the opposite direction. There is an increase in the lifting force with the bubble diameter growth. There is also an increase in the resistance force but it changes in accordance with a sinusoidal law.

The bubbles formation itself is not definite in coordinates because it is not formed by nozzles but by an annular slot as a result of the determining factors accidental alterations. The above mentioned predetermines the complex character of bubbles formation at a vibratory effect at a certain disperser construction.

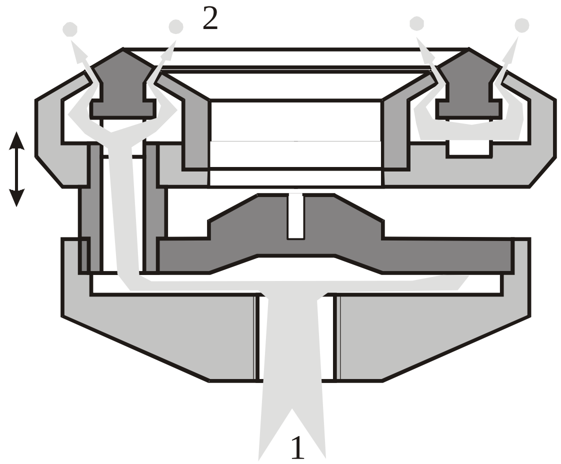

Figure 1. The vibratory air-dispersing unit

The determination of the forces, which affect a bubble formation at applied vibratory effect, is based on the fact that at a bubble formation by a nozzle it detaches when the lifting force becomes bigger than the surface tension force:

$$F_{отк} = \frac{1}{6} \square p d^3 \qquad (1)$$

Where: F_{OTK} – the force detaching the bubble from the nozzle;

d – the bubble diameter;

r – liquid density.

The circumfluence Robt resistance force is determined by the formula:

$$F_{обт} = cS\frac{\rho v^2}{2} \qquad (2)$$

c – resistance coefficient;

S – the bubble cross-section in a plane perpendicular to the liquid movement direction;

V – liquid velocity.

The liquid velocity changes according to a sinusoidal law. If the slot plane is parallel to the vibrations direction the circumfluence velocity is equal to the vibratory velocity:

$$V_{обт} = A\omega \cos \omega t \qquad (3)$$

At the other bound case – the slot plane is perpendicular to the vibrations direction – the circumfluence velocity is equal to zero (in the ideal case). In the general case – when the slot plane and the vibrations direction form a certain angle ß the velocity of the bubble circumfluence by the liquid is:

$$V_{обт} = 2\pi Af \cos \omega t \sin \beta \qquad (4)$$

The resistance force at circumfluence is:

$$F_{обт} = \frac{1}{2} c\rho\pi^3 d^2 A^2 f^2 \cos^2 \omega t \sin^2 \beta \qquad (5)$$

As it was mentioned, the force F_{OTK} needed for the bubble to be detached from the nozzle refers to a bubble formation in calm liquid and a determined nozzle size. In this way the force needed for the surface tension surmounting is determined. It is preliminarily accepted that the application of the same force to a bubble forming at the same nozzle the diameter d_1 of the obtained bubble will be different, i. e. by $F_{отк} = F_{обт}$

$$kd^3 = kd_1^2 A^2 f^2 \cos^2 \omega t \qquad (6)$$

and the diameter d_1 of the newly obtained bubble will be:

$$d_1 = \frac{d\sqrt{2}}{A^2 f^2 \cos^2 \omega t} \qquad (7)$$

b

The value of the expression $A^2f^2\cos^2\square t$ is in the limit:

$$A^2f^2 \geq A^2f^2\cos^2\omega t \geq 0 \qquad (8)$$

At the disperser oscillations upper and the lower points, when the movement direction changes the vibratory velocity $u_{виб}=0$.and the forming bubbles have the opportunity to grow to a bigger diameter. From this point the vibratory velocity increases and reaches the maximal value at a distance equal to the vibrations amplitude A. After that it again decreases to zero. The obtained bubbles diameters alter according to this harmonic law.

With the disperser vibration the resistance that the air flow experiences increases because of the faces circumfluence by the water layer. As a result of the increased resistance for the air at the annular slot outlet begins bubble formation on its entire perimeter. The air pressure pulsations are synchronous with the oscilations – the pressure decrease at the upper and lower points of the disperser movement (at increase of the slot flow capacity due to resistance decrease) and increase at acceleration.

According to the above worked out dependences the vibrations use gives the opportunity for gas bubbles diameters regulation in accordance with the vibratory effect, which allows the flotation process optimization by the gas phase dispersion improvement. This positively influences the air dispersion in a column flotation machine, where the water column height definitely stipulates conditions about the initial gas bubbles diameter.

EXPERIMENTAL RESEARCHES

Series of experimental researches are carried out in order the vibratory-acoustic influence on the air bubbles formation at vertical holes with different diameters at a vertical vibrations direction and different air flow to be determined. Measurements of the gas bubble emerging at different water column height and gas bubble diameter in a model of a column flotation machine are carried out. The vibratory field values are determined according the water column height for monophase and diphase systems.

Table1 A gas bubble emerge velocity with and without vibrations

F, [Hz]	A, mm	V, cm/s	V, cm/s	%	V, cm/s	V, cm/s	%
		without vibrations	with vibrations		without vibrations	with vibrations	
		S = 120 cm; d = 4,0 mm			S = 90 cm; d = 4,0 mm		
20	0,5	26,43	25,21	3,7	26,55	25,86	2,6
	1,00	25,92	25,26	3,5	26,87	25,71	3,1
	1,5	26,43	25,26	3,5	26,47	25,28	4,7
	2,00	26,2	25,00	4,5	26,39	24,66	7,1
	2,5	26,3	24,84	5,1	26,39	25,00	5,8
25	0,5	26,14	25,42	2,9	26,47	25,42	4,2
	1,00	26,09	25,1	4,1	26,63	25,42	4,2
	1,5	26,32	25,00	4,5	26,55	24,66	7,1
	2,00	26,26	24,74	5,5	26,47	25,14	5,3
	2,5	26,03	25,1	4,1	26,32	25,21	5,00
30	0,5	26,49	25,59	2,2	26,55	24,66	7,1
	1,00	26,2	24,84	5,1	26,63	24,19	8,9
	1,5	26,2	24,44	6,6	26,55	23,94	9,8
	2,00	25,97	23,81	9,00	26,55	23,56	11,20
	2,5	26,09	23,12	11,7	26,39	23,32	12,1
35	0,5	26,61	25,81	1,4	26,87	26,16	1,4
	1,00	26,32	25,26	3,5	26,71	26,01	2,00
	1,5	25,97	24,39	6,8	26,55	25,35	4,5
	2,00	26,03	24,59	6,00	26,55	25,00	5,80
	2,5	26,03	24,69	5,7	26,47	25,07	5,5
40	0,5	26,32	25,37	3,1	26,71	25,77	2,9
	1,00	26,26	24,9	4,9	26,39	25,07	5,5
	1,5	26,14	24,05	8,1	26,63	24,73	6,1
	2,00	26,2	24,29	7,2	26,47	24,32	8,4
	2,5	25,97	24,29	7,2	26,47	23,75	10,5
45	0,5	26,55	25,53	2,4	26,87	25,94	2,3
	1,00	26,14	24,54	6,2	26,63	24,39	8,1
	1,5	26,09	23,81	9,00	26,63	24,00	9,60
	2,00	25,92	23,72	9,4	26,32	23,20	12,6
	2,5	26,03	23,76	9,2	26,32	22,84	13,9

DETERMINATION OF THE VIBRATIONS EFFECT ON THE GAS BUBBLES EMERGING VELOCITY

At the gas phase dispersion the bubble is affected by the vibrations at two moments: at the formation, when the bubble shape and diameter is altered by the vibrations effect; at the emerging, when the emerging velocity is decreased.

Measurements in two regimes – with and without vibrations at air bubbles different size and different water column height have been carried out to determinate the vibrations effect on the gas bubble emerging velocity. The module for singular bubbles creation provides the opportunity for micrometric air supply through changeable nozzles with a certain hole diameter in order gas bubbles with determined initial diameter to be obtained. The vibratory effect on the gas bubble is applied after its detaching from the nozzle. The measurements results are presented in the table 1, where the emerging velocity difference with and without vibrations is percent computed.

The gas bubble emerging velocity without vibrations is higher than that with vibrations. Vibrations contribute to the gas bubble emerging velocity decrease, i. e. extend its stay in the column flotation machine.

DETERMINATION OF THE VIBRATIONS AMPLITUDE ACCORDING TO THE WATER COLUMN HEIGHT

The dispersed vibration at the column flotation machine lower part causes oscilations distribution in the water column depending on the pipe elasticity and the water column height as well as on the determined vibrations frequency and amplitude. The amplitude value according the height is a variable quantity.

Table 2. The vibrations amplitude along the water column height (H=150cm)

A1 - amplitude measured by the sensor on the vibrator

A2 amplitude measured by the sensor in the water

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h - the sensor height in the water

№	Frequency	The vibrations amplitude along the water column height																					
		h=20cm		h=30cm		h=40cm		h=50cm		h=60cm		h=70cm		h=80cm		h=90cm		h=100cm		h=110cm		h=120cm	
		A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2
	Hz	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm
1.	25	0,50	0,3	0,50	0,4	0,50	0,3	0,50	0,4	0,50	0,4	0,50	0,3	0,50	0,3	0,50	0,4	0,50	0,6	0,50	0,6	0,50	0,7
		1,00	0,5	1,00	0,7	1,00	0,5	1,00	0,6	1,00	0,7	1,00	0,5	1,00	0,5	1,00	0,9	1,00	1,4	1,00	1,2	1,00	1,5
		1,50	0,6	1,50	1	1,50	0,6	1,50	0,8	1,50	1,00	1,50	0,8	1,50	0,8	1,50	1,3	1,50	2,5	1,50	2,00	1,50	2,4
		2,00	0,7	2,00	1,2	2,00	0,9	2,00	1,00	2,00	1,3	2,00	1,3	2,00	1,3	2,00	2,00	2,00	4,6	2,00		2,00	3,8
2.	30	0,50	1,4	0,50	0,6	0,50	2,4	0,50	2,3	0,50	2,5	0,50	2,7	0,50	2,3	0,50	2,3	0,50	2,00	0,50	1,4	0,50	1,5
		1,00	1,5	1,00	1,2	1,00	3,2	1,00	3,7	1,00	3,4	1,00	4,00	1,00	2,9	1,00	3,4	1,00	1,8	1,00	2,6	1,00	2,6
		1,50	1,6	1,50	2,2	1,50	2,8	1,50	4,1	1,50	3,3	1,50	4,3	1,50	3,5	1,50	4,4	1,50	3,6	1,50	4,4	1,50	4,2
		2,00	1,7	2,00	2,4	2,00	2,4	2,00	2,8	2,00	2,4	2,00	4,2	2,00	4,6	2,00	4,8	2,00		2,00		2,00	
3.	35	0,5	0,9	0,50	0,8	0,50	0,9	0,50	0,6	0,50	1,00	0,50	0,70	0,50	0,7	0,50	0,9	0,50	1,00	0,50	1,1	0,50	1,1
		1,00	1,5	1,00	1,5	1,00	1,6	1,00	1,4	1,00	1,8	1,00	1,40	1,00	1,4	1,00	1,7	1,00	1,9	1,00	2,1	1,00	2,00
		1,50	2	1,50	2	1,50	2,2	1,50	2,00	1,50	2,6	1,50	2,1	1,50	2,1	1,50	2,4	1,50	2,8	1,50	2,9	1,50	2,8
		2,00	2,5	2,00	2,6	2,00	2,6	2,00	2,5	2,00	3,4	2,00	2,80	2,00	2,8	2,00		2,00		2,00		2,00	
4.	40	0,50	0,4	0,50	0,5	0,50	0,6	0,50	0,4	0,50	0,5	0,50	0,5	0,50	0,5	0,50	0,5	0,50	0,8	0,50	0,8	0,50	0,8
		1,00	0,9	1,00	1	1,00	1,2	1,00	0,9	1,00	1,00	1,00	0,9	1,00	0,8	1,00	1,1	1,00	1,6	1,00	1,7	1,00	1,6
		1,50	1,2	1,50	1,3	1,50	1,8	1,50	1,40	1,50	1,6	1,50	1,4	1,50	1,4	1,50	1,9	1,50	2,2	1,50	2,4	1,50	2,2
		2,00	1,6	2,00	1,7	2,00	2,4	2,00	2,2	2,00	2,2	2,00	1,9	2,00	2,00	2,00		2,00	2,9	2,00		2,00	

Measurements researches with a vibratory sensor have been done along the water layer for the amplitude values according the water column determination. In order the influence of the water densities difference and the vibratory sensor the latter was put into a water-isolated Styrofoam sphere with a diameter making the sensor density equal to that of the water. In this way the difference between the real and the measured values of the amplitude of the phases oscillations difference is avoided.

The measurement has been carried out at different water column height in the column flotation machine and in two regimes: without air bubbles (table 2) and with air bubbles (table 3). A part of the obtained results are presented in tables 2 and 3.

The results from the measurement of the vibrations amplitude along the water column height provide the opportunity conclusions about the vibratory-acoustic influence distribution in the flotation machine volume to be made. The fact that the vibrations distribution is influenced by the column geometric dimensions and the resonance phenomena connected with its diameter, the faces thickness and the measurement distance from the vibratory disperser (fig. 2) is definitely corroborated. At the disperser oscillation frequency 40Hz and amplitude 1,0 mm, the liquid vibrations alter from 0,8 to1,7 mm with a clearly expressed corrugated character of the change along the height due to the resonance phenomena along the height.

The liquid amplitude measurement in presence of dispersed in the form of bubbles air shows the amplitude decrease in the bounds from 0,2 to 0,5 mm , which is also with a determined corrugated character, but its values are several times lower than those measured without air bubbles and even lower than the vibratory disperser amplitude.

This decrease of the liquid oscillations amplitude in a gas phase presence explains the air bubbles emerging velocity decrease. The energy dissipation, which is realized at a gas phase presence, causes gas bubbles fluctuation –decrease and increase in their diameter is obtained periodically, synchronous with the supplied vibrations frequency.

Due to the water layer pressure decrease the emerging gas bubble diameter continuously increases. As it is well known that the emerging bubble saves its spherical shape only at very small sizes. At the diameter increase the bubble shape changes providing the smallest moving resistance.

Table 3. The vibrations amplitude along the water column height at an air flow 200 l/h

h - the sensor height in the water

A1 - amplitude measured by the sensor on the vibrator

A2 - amplitude measured by the sensor in the water

№	Frequency	The vibrations amplitudes along the water column height																			
		h=20cm		h=30cm		h=40cm		h=50cm		h=60cm		h=70cm		h=80cm		h=90cm		h=100cm		h=110cm	
		A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2	A1	A2
	Hz	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm	mm
1.	20	1,00	0,3	1,00	0,4	1,00	0,4	1,00	0,4	1,00	0,3	1,00	0,2	1,00	0,2	1,00	0,3	1,00	0,3	1,00	0,4
		2,00	0,3	2,00	0,4	2,00	0,4	2,00	0,4	2,00	0,3	2,00	0,3	2,00	0,3	2,00	0,4	2,00	0,4	2,00	0,5
		3,00	0,4	3,00	0,4	3,00	0,4	3,00	0,4	3,00	0,3	3,00	0,4	3,00	0,4	3,00	0,4	3,00	0,4	3,00	0,5
2.	25	1,00	0,3	1,00	0,4	1,00	0,4	1,00	0,4	1,00	0,4	1,00	0,2	1,00	0,3	1,00	0,3	1,00	0,4	1,00	0,4
		2,00	0,4	2,00	0,4	2,00	0,4	2,00	0,4	2,00	0,4	2,00	0,3	2,00	0,3	2,00	0,3	2,00	0,4	2,00	0,4
		3,00	0,4	3,00	0,4	3,00	0,4	3,00	0,5	3,00	0,4	3,00	0,4	3,00	0,3	3,00	0,4	3,00	0,4	3,00	0,4
3.	30	1,00	0,8	1,00	0,8	1,00	1,2	1,00	1,5	1,00	0,8	1,00	1,00	1,00	0,4	1,00	0,5	1,00	0,4	1,00	0,9
		2,00	1,9	2,00	1,8	2,00	2,1	2,00	1,9	2,00	1,6	2,00	1,7	2,00	1,2	2,00	0,8	2,00	0,6	2,00	1,5
		3,00	2,5	3,00	2,5	3,00	2,3	3,00	2,1	3,00	1,7	3,00	2,00	3,00	1,6	3,00	0,4	3,00	0,8	3,00	2,1
4.	35	1,00	0,7	1,00	0,7	1,00	0,4	1,00	0,6	1,00	0,4	1,00	0,4	1,00	0,3	1,00	0,4	1,00	0,3	1,00	0,4
		2,00	1,1	2,00	1,2	2,00	0,8	2,00	1,00	2,00	0,6	2,00	0,8	2,00	0,3	2,00	0,4	2,00	0,4	2,00	0,4
		3,00	1,1	3,00	1,4	3,00	1,4	3,00	1,3	3,00	1,2	3,00	1,3	3,00	0,7	3,00	0,6	3,00	0,7	3,00	0,6
5.	40	1,00	0,3	1,00	0,4	1,00	0,4	1,00	0,4	1,00	0,4	1,00	0,4	1,00	0,2	1,00	0,4	1,00	0,4	1,00	0,3
		2,00	0,4	2,00	0,6	2,00	0,6	2,00	0,7	2,00	0,6	2,00	0,5	2,00	0,3	2,00	0,6	2,00	0,5	2,00	0,6
		3,00	0,5	3,00	0,5	3,00	0,6	3,00	0,9	3,00	0,8	3,00	0,6	3,00	0,4	3,00	0,6	3,00	0,8	3,00	0,7
6.	45	1,00	0,4	1,00	0,4	1,00	0,5	1,00	0,3	1,00	0,3	1,00	0,2	1,00	0,3	1,00	0,4	1,00	0,4	1,00	0,4
		2,00	0,7	2,00	0,6	2,00	0,6	2,00	0,4	2,00	0,3	2,00	0,3	2,00	0,4	2,00	0,5	2,00	0,7	2,00	0,8
		3,00	1,00	3,00	0,9	3,00	0,9	3,00	0,5	3,00	0,5	3,00	0,4	3,00	0,5	3,00	0,5	3,00	0,7	3,00	1,00
7.	50	1,00	0,3	1,00	0,3	1,00	0,4	1,00	0,3	1,00	0,3	1,00	0,2	1,00	0,2	1,00	0,2	1,00	0,3	1,00	0,3
		2,00	0,5	2,00	0,4	2,00	0,4	2,00	0,4	2,00	0,4	2,00	0,3	2,00	0,3	2,00	0,2	2,00	0,4	2,00	0,4

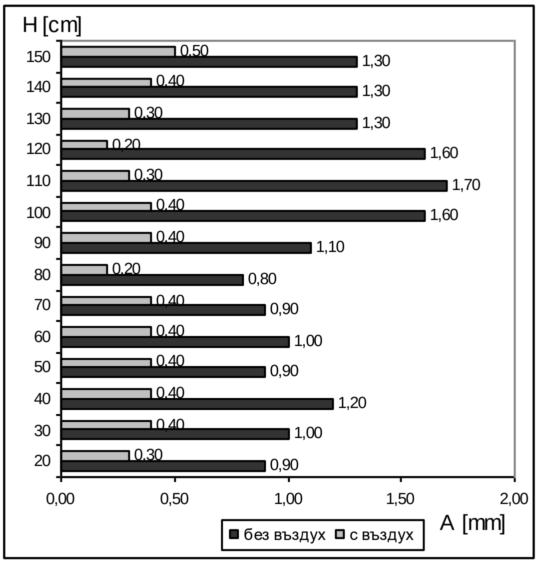

Figure 2. The vibrations amplitude along the water column height with and without gas bubbles

Because of the gas bubble vibration the hydrate layer is not allowed to accept that resistance decreasing shape so the emerging is realized with a shape close to the spherical, which under the defined circumstances creates higher resistance. That resistance is expressed in the bubble emerging velocity decrease.

The velocity decrease gives the opportunity for the bubble stay in the mineral particles contact zone time protraction. The hydrate layer vibrations give the opportunity for the elementary flotation act realization by providing higher relative velocities at a meeting of the bubble and the solid phase and because of the layer thickness as a result of the continuous increase and decrease of the bubble diameter.

CONCLUSION

The carried out vibratory-acoustic researches in the vibrations influence on the air bubbles creation, emerging and behavior in the flotation machine volume provide the opportunity definite interactions to be clarified as well as the question about the vibrations operation on the gas, liquid and solid phase in the column vibratory flotation machine. The results will be used in the final research stage – the technological tests by real products flotation and constructive and technological tests combination on order the vibratory-acoustic processes influence on the flotation process in a column flotation machine to be entirely clarified.

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Recommended for publication by Department , Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.111-115

TECHNOLOGY FOR MIO PRODUCTION

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ABSTRACT
MIO is natural iron oxide pigment with specific flaky or plate forms of the particles (Micaceous Iron Oxide). It is widely used in anticorrosive coverings. It has no competition against aggressive mediums and ultraviolet radiation.
Raw material for production of MIO is sized in Kremikovtzi mineral deposit. Characteristic parameters for production of such pigment were set by laboratory experiments. A technological flowsheet for obtaining of quality MIO type pigment according to the international standards were drawn up.

MIO - ESSENCE AND USEAGE

The name MIO leads its beginning from the abbreviation of Micaceous Iron Oxide. As it was underlined by John Benbow the name does not have connection with the mica. Micaceous is used to emphasize the plate shape of the different particles which are usually in size of 10 µm to 100 µm with thickness about 5 µm. Taking into account the morphology of these particles, used as a covering they align parallel to the surface they are laid. This is the main characteristic of the MIO type pigments. Fe_2O_3 is not soluble in water, organic solvents, alkali. It is not toxic. It is stable under temperature variations, not corrodible. These facts indisputably show that these pigments based on Fe_2O_3 are priceless as anticorrosive coverings.

MIO is a crystalline form of Fe_2O_3 that differs from the more popular red, yellow and brown pigments. The mineral is known as specular hematite and has the same chemical content as hematite but crystallizes in different crystal form. The growth of the crystalline forms of the hematite is clearly connected to the crystallization conditions: contactometasomathic hematite crystals are similar to cubs or lenses with strong developed rhomboidal forms, volcanogenetic crystals are platy, while metamorphic and hydrothermal ones could be flaky - specular hematite - Figure 1 (Kostov, 1973).

Hematite crystal is without sectility_but with separation in (0001) and (1011) due to the epitacsial layers of goethite. The energy needed for crystal destruction in different directions is shown in Figure 2.

Temperature

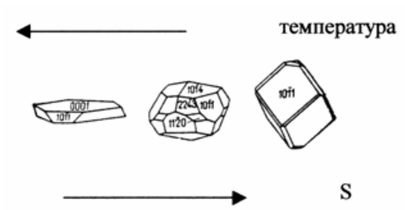

Figure 1. Crystalogenetic trends of hematite

Main usage of MIO type pigments are for anticorrosive coverings. Arrangement the particles parallel to each other results in barrier against access of aggressive fluids, oxygen, UV radiation and different ions to the surface onto which the dye is laid. The barrier effect is illustrated at Figure 3 (Kartner Montanindustrie Ges. mbH, 2001).

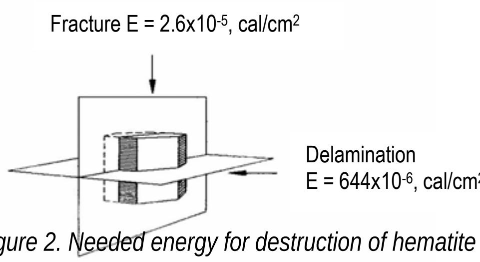

Figure 2. Needed energy for destruction of hematite crystal

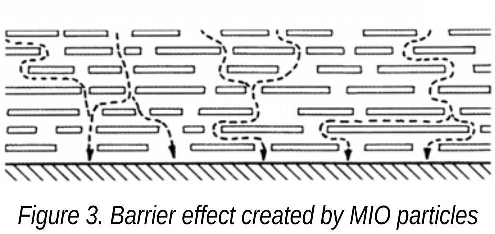

Figure 3. Barrier effect created by MIO particles

Another special feature of the pigment is almost complete absorption of waves with length in the ultraviolet spectrum. UV radiation is dangerous for wood and plastic materials. It has destructive effect onto lignin in the wood and changes the structure of the plastics, which leads to undesired changes in their properties.

At Figure 4 is shown electron microscope picture of cross-section of covering with MIO type pigment.

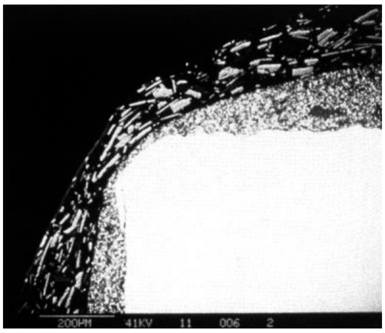

Figure 4. Cross-section of covering of MIO. Electron microscopic picture.

It is known usage of special type pigment - transparent iron oxide (Benbow, 1989). Such pigment has particles with size in range of 0,01 μm (in this size and plate forms they are transparent). Only 2 g/m^2 of this pigment are completely sufficient for 100 % absorption of the UV lights fallen on the surface.

Usage of such type pigment with combination with fine milled mica gives pearl effect of the surface due to the different light-breaking ability of the minerals (Benbow, 1989).

Usage the pigment in the plastic industry is aimed in several directions. Using the painting ability and using the plate forms of the particles - resulting in two subdirections: improving the gasbarrier ability of the plastics and improving its fire resistance.

In spite of continuously increasing application of MIO type pigments the base usage is in anticorrosive coverings of structural constructions including: bridges, pylons, petrol platforms, and other marine outfits as submarines and boats, in fact, everywhere where a good protection from the atmospheric influence and influence of the aggressive medias is needed.

REQUERMANTS TO THE MIO

Several standards were created for the usage of the pigment in the lacquerdye industry. ISO 10601 and ASTM D5532 are valid at the moment. Requirements accordingly the percentage of the flaky particles in the sample of the pigment are defined according ISO 10601. According to this requirement the pigment is categorized. The division is in three grades, where: the first grade should content more than 50 % particles with flaky shape. The other requirements are divided in two categories: essential and conditional. In the group of the essential are:

- Fe_2O_3 content – min 85%;
- Volatile matter at temp. 105°C max. 0,5 %;
- Matter soluble in water max. 0,5 %;
- Granulometric composition:
  - + 0,065 mm – 5, 15 and 35 % according to the grade of the pigment;
  - + 0.105 mm max 0,1 %.

The defined in the standard conditional requirements, which are not obligatory but are object of agreement between the interested bogies are: pH of the solution, oil absorption and CaO content.

According to the American standard ASTM D5532 the pigments are divided in two grades, where the quality of the first grade is increased in respect of flaky particle content, that should not be bellow 65 % (according the ISO it is 50 %). The other requirements are analogical.

MINING AND PRODUCTION OF MIO

Deposits of quality MIO pigments in the world are limited. MIO is mined in Austria, South Africa, Spain and Morocco in small extends. Austria is the world leader in the production and export of such pigment. Kartner Montanindustrie Ges.mbH mines in Waldenstein deposit in Austria and probably posses 90 % of the world market. Their output is 10 000 t per year while the firm poses capacity to twice the production. The pigment is offered as MIOX trademark and 97 % of it is used in the paint industry. Technology includes – selective mining, separation of the impurities, drying and milling and classification for achievement of the desired from the consumers particle size.

Romero Hermanos SA offers MIO from La Aparacida deposit in Sierra Nevada. G & W Base & Industrial Minerals (PTY) Ltd offers MIO from South Africa.

Attempts for production of synthetic MIO are not achieved yet technological solution adequate to the market requirements. An attempt for production of synthetic MIO is made by Cookson PLS I Magnesium International Copr. They offer product in high purity and high content of Fe_2O_3 under the trademark Laminox.

KREMIKOVTZI AS SOURCE OF MIO

Part of the currently mined iron ore in Kremikovtzi deposit could be used as source for production of iron oxide pigments with wide color range (I. Kuzev, et al. 2001, Atanasov, 1999). More interesting is the fact that specular hematite is found in the deposit. Unfortunately coarse-flaked fraction occurs rarely and it is not restricted in defined areas. Sufficient quantity of fine-flaked fraction could be mined by selective collection.

The coarse-flaked fraction in difference from the fine-flaked does not posses properties to be disintegrated easily. This is the main obstruction for obtaining of MIO from Kremikovtzi deposit. . A special type of treatment that generates predominantly tangential exertion in the milling apparatus is needed. The process of delamination of separate flakes is more probable under such conditions in difference of ordinary process of comminution where the fraction of the particles is more possible.

Practically, the pigment obtained by this method completely fit the conditions of standards defining the quality of pigments type MIO. In the standards is not pointed minimal size or defined granulometric characteristic of the particles. In difference of popular pigment MIO which particles are in range up to 100 μm the maximal size of the particles of the pigment obtained from Kremikovtzi deposit is 10 μm.

METHODOLOGY AND EQUIPMENT

Investigated raw materials is selectively collected from the Kremikovtzi mineral deposit. It includes several types of almost equal in chemical content and with difference in its mineralogy ores. Comparative investigations were made with the following raw materials:

- Coarse-flaked specular hematite;
- Hematite;
- Fine-flaked specular hematite;
- Transition between Hematite and Fine-flaked specular hematite;

The methodology of treatment includes:

- Crushing to 10 mm and 3 mm sizes suitable for the following treatment;
- Vibration treatment – vibration milling and vibration attrition under different conditions:
  - Dry treatment;
  - Wet treatment;
  - Autogenous treatment.
- Sieving or classifying by hydrocyclones according to the desired size.

The investigated technological parameters are:

- Treatment duration;
- The slurry density;
- Filling percentage of the mill;
- Granulometric characteristic of the input;
- Adding of surfactants.

The influence of the different milling medias was investigated. Special working bodies (lenses of tungsten carbide) were used to transform the vibration milling in vibration attrition.

Laboratory vibration devise with volume 350 cm^3 and constant vibration parameters: frequency 24 Hz and amplitude 3,5 mm are used for the purposes of the investigation.

Working medias include rods of different materials – Fe, tungsten carbide, as well as lenses of tungsten carbide for creation of attrition conditions. Lenses of tungsten carbide are shown at Figure 5.

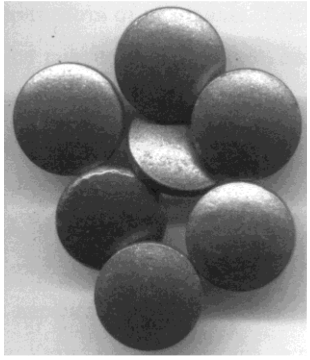

Figure 5. Working media of tungsten carbide

The results were observed by:

- Observation of class (-0,063 mm);
- Defining of the granulometric characteristics by laser devise Analizette 22 made by Fritsch, Germany;
- Optical microscopy with microscope МБИ 15;
- Electron microscopy with microscope Philips 515;
- Image analyze with GALAI CIS 100 carried out in Germany.

Samples of anticorrosive dyes were produced at "Lakprom" research lab Svetovrachane with methodology of the current production applying pigments produced by our technology.

RESULTS AND DISCUSSION

Two methods of treatment were used for obtaining the optimal granulometric characteristic of the input of the vibration attrition stage.

- Consecutive crushing in jaw and roll crusher with outlet of 10 and 3 mm.
- I stage crushing in jaw crusher with outlet of 10 mm followed by II stage vibration treatment with crushing media of steel rod.

Granulometric characteristics of the obtained products are shown at Table 1.

Table 1.

Fraction, mm	Yield, %		
	Hematite	Specular hematite	Fine-flaked specular
+ 1,6	3,0	9,0	2,0
- 1,6 + 1,0	13,0	14,0	10,0
- 1,0 + 0,5	22,0	18,0	20,0
- 0,5 + 0,2	24,0	14,0	23,0
- 0,2 + 0,071	19,0	14,0	22,0
- 0,071	19,0	31,0	23,0
Sum	100,0	100,0	100,0

The second method gives higher yield of fine classes that's way it is preferred in the investigation.

Obtained results from the dry autogenous milling of material with feed size of 14 - 20 mm gives results shown in Table 2.

Table 2.

Treatment duration, min	Yield of observed class – 0,063 mm, %	
	Specular hematite	Hematite
5	11.19	8.33
10	13.05	11.48
20	15.59	13.15
30	15.93	15.19
60	16.10	22.96

Dry autogenous milling of fine-flaked specular hematite has turned out impossible due to generation of agglomerates hampering the process of comminution.

Adding of surfactants does not improve the process even more steppes the process of comminution of hematite and coarse-flaked specular hematite.

Milling by usage of different milling media in dry conditions leads to the following results, shown in Table 3.

Table 3.

Milling media	Yield of observed class – 0,063 mm, %	
	Specular hematite	Hematite
Tungsten carbide lenses	94.50	90.0
Al rod	41.0	28.0
Fe rod	91.0	64.0
WC rod	92.0	70.0

Vibration attrition with milling media of tungsten carbide lenses leads to the biggest yield of observed class of coarse-flaked specular hematite and hematite. Dry vibration attrition of fine-flaked specular hematite has practically turned out impossible due to the fact that fine classes stick to the working bodies and stops the process.

Wet technological process substantially differs from the dry one. Vibration attrition with solid content in the slurry of 71 % leads to 99 % yield of observed class from the coarse-flaked specular hematite, over 67 % for the hematite and 100 % for the fine-flaked specular hematite. With the purpose of comparison treatment with rod working media dives yield of observed class as follows: coarse-flaked specular hematite - 97,5 %, hematite 47 %, fine-flaked specular hematite - 99 %. It seems that the differences in the yields of observed class obtained by vibration milling and vibration attrition is not so high but taking into account the aim of producing particles with flake morphology vibration attrition no doubt has priority (Hristov et al., 2202).

Granulometric characteristics of the products obtained by vibration attrition are shown at Figure 5 and Figure 6.

The sample obtained of fine-flaked specular hematite is in very small size and goes to the zone of unsatisfactory accurateness of the laser devise. That's why it was sent to Germany and analyzed with Image analyzer GALAI CIS 100. Results are shown in Table 4.

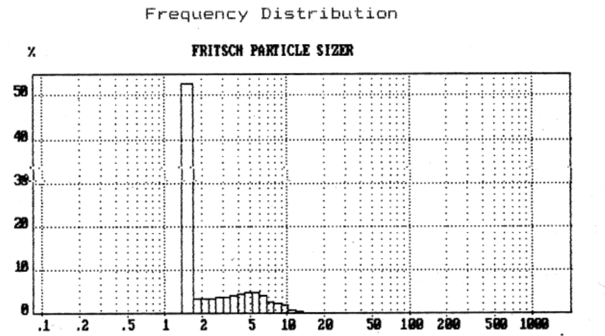

Figure 5. Granulometric characteristic of specular hematite.

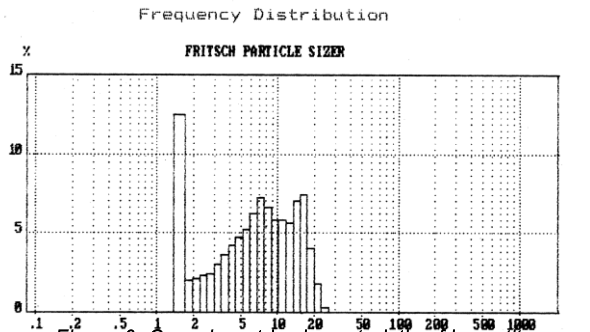

Figure 6. Granulometric characteristic of hematite.

Images from CCD camera were used for the purpose of the analysis. 400 particles were analyzed in tree series of counting. Table 4 represents computer calculated averages of the investigated parameters.

Table 4.

Shape factor	Average Ferret, μm	Min Ferret, μm	Max Ferret, μm	Aspect ratio
0,72	1.18	0.93	1.34	0.43

Obtained pigment was analyzed in AES-ICP and Table 5 shows the results.

Table 5.

Contend, %					
SiO_2	Fe_2O_3	TiO_2	Al_2O_3	MnO	CaO
1,92	77,08	<0,01	1,30	0,05	5,33
Contend, %					
MgO	Na_2O	K_2O	P_2O_5	З.П.Н.	H_2O
1,77	3,85	4,66	0,05	<0,05	0,55

The shape of the particles obtained by vibration attrition was observed by scanning electron microscope. A picture is shown at Figure 7.

The analysis of the pigment applied in anticorrosive dye carried out in "Lacprom" shows excellent results. In fact the pigment shows better quality than the pigments used in the current production of the firm. Probably this is due to the flaky shapes of the particles obtained by vibration attrition and barrier effect generated by these plate particles.

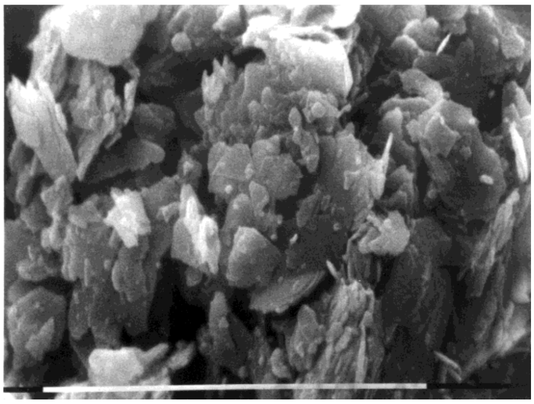

Figure 7. Particles of fine-flaked specular hematite

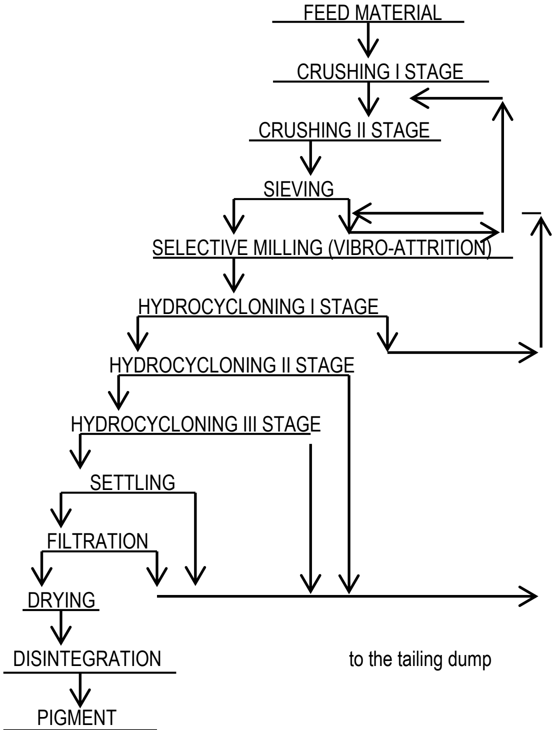

Figure 8 Flowsheet for MIO production

Analyzing the obtained results and aiming covering of the requirements to the pigments type MIO we suggest the following technological flowsheet. Figure 8 shows the flowsheet for MIO production from the ore mined in Kremikovtzi mineral deposit.

CONCLISION

Characteristic parameters for production of pigment Micro MIO from selective run of mine raw materials from Kremikovtzi mineral deposit were defined by carried out investigation. Technological flowsheet for industrial production are presented. In fact, the obtained pigment fit to the all requirements of standards ISO 10601 and ASTM D 5532. Anticorrosive dye “ПФ – 025” produced by the laboratory obtained pigment fit the recuirements of the technikal sheets (ЛП-ТЦ-ППХ-029 for Fe_2O_3) excluding point - hardness of the film.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.117-121

A PYROELECTRIC MATERIAL FOR A SENSITIVE ELEMENT IN GAS ANALYZERS

Slaveiko Tsvetkov

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ABSTRACT

The use of high performance pyroelectric materials as a sensitive element of catarometers in gas chromatographs enables us to decrease the gas detection limit to 10^{-5} - 10^{-6} vol.%. The pyroelectric and dielectric properties of ferroelectric ceramic samples made of pure calcium leaden titanate as well as those modified by nickel and manganese have been studied. These results show that most of the parameters are affected both on the calcium quantity in the initial substance and on the nature and quantity of the modifying agent. The synthesized piezoelectric materials on that type of substances ensure improved physical parameters and stabilized optimal characteristics of the sensitive element of gas analyzers.

INTRODUCTION

Strengthened work in searching for possibilities and methods for increasing of the sensitivity of gas chromatographs has began since 60th years of the XXth century. Studies have been directed mainly in two directions: developing of an express chromatographic method (Leithe, 1980) and introducing of new perspective pyroelectric materials as sensitive elements (Novik, 1979). Because the signal of the pyroelectric catarometer is proportional to the speed of the change of the deposed mater concentration it is possible express methods for analyzing of the metallurgic, hole and other gases to be developed and combine them with step chromatography (Guglya et al., 1973; Guglya et al., 1984). Using of high effective pyroelectric materials in the catarometer sensitive elements ensures very low concentrations of the substances in the mix to be determined. In some cases they reach values of the order of magnitude of $10^{-5} – 10^{-6}$ vol. %. Because of these reasons the choice and synthesizing of suitable material are a stage of substantial meaning in creating of transformers that determine their transforming properties.

Pyroelectricity is a phenomenon observed for a group of piezoelectric crystals with low space symmetry (pyroelectrics). It is expressed in appearing of spontaneous polarization in dielectric crystals when there is no outer electric field. As the temperature varies the magnitude of the spontaneous polarization also changes. For difference from the ferroelectrics the sign of the polarization of pyroelectrics does not change when an outer electric field is applied.

Usually polarized charges, created as a result of the polarization, are compensated because of trapping of electrical charged motes and aerial ions as well as a result of pyroelectric own conductivity.

At regular heating or cooling of the pyroelectric crystals, however, change of the pyroelectric charge density, respectively spontaneous polarization $\boldsymbol{P}_s$ of the pyroelectric, can be observed. A direct proportional dependency exists between the temperature change ΔT and the spontaneous polarization change $\Delta \boldsymbol{P}_s$.

In dependence on the purpose of the transforming element the requirements to the materials are oriented to either the complex of physical properties, which ensure maximal exploitation characteristics to be achieved, or improving of their constructive and technological capabilities. Keeping and reproducing of the properties of the pyroelectric transformer at the work conditions (hits, concussions, irradiation, climatic factors etc.) depend strongly on the properties of the chosen material.

Among the existing three groups pyroelectric materials (monocrystals, polycrystals and organic polymers) the polycrystal pyroelectrics appeared to possess the best properties. They give possibility for flexible managing of their properties, 100 % using of the material, a low working cost (6 - 8 times less than the monocrystals working cost), a large specific surface (up to some thousands cm^2/g). On the other hand, they suppose widely varying of their properties by changing their chemical composition and polarization regime. In these reasons the experiments considered in this work, were restricted only in studying of the pyroelectric ceramic polycrystal materials.

EXPERIMENTAL

Pyroelectric and dielectric properties of basic ferroelectric ceramic material made only by calcium leaden titanate as well as the samples on this basic material with additives of nickel or manganese in given constant quantities were studied. The stoichiometric composition of the basic material expressed in at. % is $(Pb_{0.75}Ca_{0.25})[(Co_{0.5}W_{0.5})_{0.04}Ti_{0.06}]O_3$. In comparison with other atomic ratios of the elemental composition the shown one possess the highest pyroelectric activity, the least dielectric permeability and the highest thermostability.

The leaden titanate ($PbTiO_3$) was used for synthesizing of the basic material. The structure was stabilized by substitution of some leaden atoms from the composition with calcium. The samples with calcium quantity between 23 and 27 at. % were studied - this means that the general formula of the basic material is $(Pb_{1-x}Ca_x)[(Co_{0.5}W_{0.5})_{0.04}Ti_{0.06}]O_3$, where x = 0.23, 0.24, 0.25, 0.26, 0.27. The samples were obtained using a standard method: hydraulic milling of the raw material in the ball mill, thermal synthesis at 900 °C for two hours, second milling of the product till its specific surface achieved a value $\varphi_v \geq 4000$ cm^2 / g, double plasticizing using 10 % solvent of polyvinyl alcohol, pressing in the disc form with diameter of 20 mm and thickness of 2 mm and 2 hours baking at the temperature 1170 - 1210 °C with the rate of 200 °C / h for reaching of the temperature given. The sample composition was made in three variants: basic (without any modificator); with 1 mol. % modifying agent of NiO and with modifying additive of 1 mol. % MnO_2. After cooling the discs down their thickness was going down to 1 mm. Then their surface was smeared with silver paste and baked at a temperature of 950 °C with an aim stabilizing of the silver layer on the sample surface. The polarization of the sample was performed in silicon oil at the temperature of approximately 100 °C and a field intensity of 6 kV / mm for 1.5 h. 24 h later the following dielectric parameters were measured:

Relative dielectric permeability ε_{33}^T, dielectric looses tgδ, resonant frequency trough the diameter f_r, antiresonant frequency trough the diameter f_a, resonant resistivity R_r. resonant frequency trough the depth f_r' and antiresonant frequency trough the diameter f_a'. The samples were designated as follow: the first number shows the calcium amount in at. % and the figure after the dash - the type of the sample modification – "1" for nickel modification, "2" for manganese modification and "0" for unmodified samples (for example, "25 - 1" means that the sample is modified with NiO and contains 25 at. % calcium).

EXPERIMENTAL RESULTS AND DISCUSSION

The mass looses were determined two hours after the thermal synthesis at 900 °C. The results show that they were not big and they are in the limits of 2 %.

In Table 1 the results of the relative dielectric permeability of the disc samples before polarization $(\varepsilon_{33}^T)_0$, its change after polarization relative to its initial value $\varepsilon_{33}^T/(\varepsilon_{33}^T)_0$ and the dielectric looses before (tgδ_0) and after (tgδ) polarization were shown.

Table 1. Effect of calcium Ca, nickel Ni and manganese Mn contents of the structure on the relative dielectric permeability and dielectric loses of the pyroelectrics. The designation of the sample is as follow: the first number shows the calcium quantity in the sample. The modification of the samples was shown by the figure after the dash: „0"' for row material, "1" – for material with nickel modificator and "2" - for material with manganese modificator. $(\varepsilon_{33}^T)_0$ and ε_{33}^T are relative dielectric permeability of the material before and after polarisation, and tgδ_o and tgδ - the material dielectric loses before and after polarisation, respectively.

Sample	Baking temperature	Relative dielectric permeability		Dielectric loses	
	[° C]	Before polarisation $(\varepsilon_{33}^T)_0$, x10^14	After polarisation $\varepsilon_{33}^T/(\varepsilon_{33}^T)_0$	Before polarisation tgδ_o	After polarisation tgδ
24-0	1210	208	163	4.71	3.35
25-0	"	232	201	8.06	2.89
26-0	"	221	194	8.27	2.69
27-0	"	195	152	4.62	2.60
23-1	1170	196	166	6.45	6.08
24-1	"	225	208	6.14	6.18
25-1	"	227	216	4.27	6.10
26-1	"	225	214	5.30	7.93
27-1	"	235	225	4.75	723
23-2	1200	186	179	2.46	1.98
24-2	"	192	182	2.35	1.75
25-2	"	185	181	1.85	1.59
26-2	"	194	185	2.42	1.79
27-2	"	191	186	2.11	1.71

Increasing of $\varepsilon_{33}^T/(\varepsilon_{33}^T)_0$ till and over 200 (for basic $PbTiO_3$ crystal, for example, $\varepsilon_{33}^T/(\varepsilon_{33}^T)_0$ is 78-80) is probably due, on one hand, to the different calcium quantity in the samples and on the other - to the type of the modifying agent. After substitution of the leaden atoms in the crystal cell with calcium ones, decrease of the tetragonal bending took places in the basic material, that is accompanied with the decreasing of the Curie temperature. When the ratio of $PbTiO_3$ to $CaTiO_3$ was 1 : 1, the T_k became 80 °C and $\varepsilon_{33}^T/(\varepsilon_{33}^T)_0$ increased till 800. As the main purpose of this studying was producing of material possessing high sensitivity (that is material with low permeability) and better piezoelectric properties, introduction of calcium atoms should be limited to 27 at. %. The results for chosen range of calcium amount presented in Table 1 show that the change of the dielectric permeability ε_{33}^T is in the range of 5 – 25 % for every at. % calcium quantity, depending on the nature of the modifying agent.

Slight growing of the ratio ε_{33}^T for the sample with nickel additive could be seen, while ε_{33}^T for the manganese modified samples decreased and stabilized. Dielectric looses, tgδ, for the sample with manganese modifier are very low and almost did not depend on the calcium quantity, while for these one with nickel modifier they grew up with increase of the calcium quantity in the materials.

Table 2. Effect of the temperature on the deviation of the relative dielectric permeability and temperature coefficient of the dielectric permeability for different modifying additives.

Sample	Baking temperature [°C]	Deviation of the relative dielectric permeability, ε_{33}^T, %			Temperature coefficient of the dielectric permeability, TK ε_{33}^T, x10^{-3}, [K^{-1}]		
		-25÷15 °C	15÷70 °C	-25÷70 °C	-25÷15 °C	15÷70 °C	-25÷70 °C
23-1	1210	9.48	5.07	14.55	2.37	0.90	1.53
24-1	1210	15.50	9.46	24.96	3.88	1.72	2.63
26-1	1170	13.45	13.89	27.34	3.36	2.52	2.88
27-1	1190	14.09	12.76	26.85	3.52	2.32	2.83
25-2	1200	8.28	19.26	27.54	2.07	3.50	2.90
27-2	1200	8.48	21.56	30.04	2.12	3.92	3.16

The ratio $\varepsilon_{33}^T/(\varepsilon_{33}^T)_0$ in the different temperature intervals and the values of the temperature coefficient of the dielectric permeability, TKε_{33}^T, are shown in Table 2. It was found destabilization of the ratio $\varepsilon_{33}^T/(\varepsilon_{33}^T)_0$ for the nickel-modified material when the calcium quantity changes. According us, this was due to the decrease of the Curie temperature when the calcium amount increased. Stabilization of this ratio was established for unmodified samples (not shown in the Table 2) and for the manganese-modified samples. The temperature coefficients of the dielectric permeability, different for the three kinds of samples were comparatively more stable.

Table 3. Effect of the temperature on the value and sign of the resonant frequency deviation and the frequency temperature coefficient for manganese modified samples.

Sample	Deviation of the resonant frequency, f_r, %			Temperature coefficient of the resonant frequencyTKf_r, x10^{-6}, [K^{-1}]		
	-25÷15 °C	15÷70 °C	-25÷70 °C	-25÷15 °C	15÷70 °C	-25÷70 °C
25-2	0.425	0.159	0.159	1.062	0.404	0.631
27-2	-0.492	-0.768	-1.250	-1.231	-1.402	-1.316

In Table 3 data for the temperature coefficient of the radial vibrations for two of manganese-modified samples were presented. Similar values were found for nickel-modified samples as well. With increase of the temperature the values of f_r decreased. When the calcium amount was below 25 at. %, the resonant frequency f_r moved to the higher values, but at calcium quantity of 27 at. % it moved to the opposite direction (f_r decreased).

The results for the coefficient of the electromechanical connection for radial K_r and depth K_f vibrations at the baking temperature chosen were shown in Table 4.

Nature of the modifying additive as well as the number of the calcium atoms in the crystal lattice affected the material anisotropy. Adding of NiO and MnO_2 into the basic matter caused qualitatively different properties of the ceramics. It was supposed that when nickel atoms displaced titan ones from their positions in the crystal lattice leaden vacations appeared that caused increase of the dielectric permeability and dielectric looses (see Table 1) and coefficient of the

Table 4. Effect of the electromechanical action on the radial K_r and depth K_t vibration.

Sample	Baking temperature	K_r	K_t	K_t/K_r
23-0	1210	0.270	0.485	1.80

24-0	“	0.137	0.450	3.28
25-0	“	0.096	0.450	4.62
26-0	“	0.096	0.438	4.56
27-0	“	0.110	0.483	4.39
23-1	1170	0.094	0.522	5.55
24-1	“	0.153	0.502	3.28
25-1	“	0.176	0.498	2.83
26-1	“	0.116	0.483	4.16
27-1	“	0.136	0.462	3.40
23-2	1200	0.142	0.352	2.48
24-2	“	0.125	0.428	3.42
25-2	“	0.125	0.442	3.54
26-2	“	0.129	0.434	3.36
27-2	“	0.197	0.466	4.31

electromechanical connection (see Table 4). The effect of manganese additive was not clear yet because manganese changed its valence in dependence on the conditions of the technological operations. Generally, one could conclude that the modified materials possessed less dielectric permeability, low dielectric looses and negligible changes of K_r and K_f. Bigger decrease of K_r and increase of K_f values could be expected if the technology for material producing improves and perfects and also if the nature and amount of the modifying agent change. The non-ferroelectric $Pb(Co_{0.5}W_{0.5})0_3$ additive, compensating manganese valence, probably created additional leaden vacations and in that way the structure stabilized. The $Pb(Co_{0.5}W_{0.5})0_3$ additive was the same in all samples studied.

The comparative characteristics' for the piezoelectric ceramic materials synthesized by us and for such, produced in Japan were shown in Table 5.One could see that as far as parameters used are concerned, materials synthesized by us achieve the Japanese ones. This conclusion is an indication for their eventually future application as sensitive elements in the catarometry. An object of the next experiment is studying of the piezoelectric ceramic materials as detectors for few inorganic gas components.

Table 5. Comparative characteristics between Bulgarian and Japanese piezoelectric materials

Parameter	Bulgarien			Japanese		
				Toshiba		Hitachi
	23-1	25-2	27-2	C-10	C-12	PC-11
$\varepsilon_{33}^T/\varepsilon_o$	166	181	186	250	180	170
tgδ	6.08	1.59	1.71	-	-	1.00
K_r	0.094	0.125	0.197	0.14	0.06	0.05
K_t	0.522	0.422	0.466	0.45	0.47	0.5
N_t , Hzm	1874	2470	2022	2454	2358	2150
$d_{33}.10^{-12}$, C/N	80	90	95	61.6	55.6	52.8
$g_{33}.10^{-3}$, Ym/N	54.40	56.18	57.70	27.11	34.62	35.13
σ, N/m^2	0.219	0.220	0.220	-	-	0.220
T_k, °C	325	308	309	360	340	355

CONCLUSIONS

1. Three types of samples made by ferroelectric material based on the leaden-calcium titanate with nickel and manganese modifiers were synthesized and their dielectric and pyroelectric parameters were studied;

2. It is shown that in most of the cases the values of the parameters studied depended on calcium amount in the basic material as well as on the quantity and nature of the modifying additive;

3. It is established that dielectric looses of the samples are function of the nature of the modifying atom. For nickel additive they were negligible, while for manganese additive they depended on the calcium amount in the structure of the basic material;

4. Results from our studying of the piezoelectric materials produced indicated that improving and stabilizing of the specific physical parameters for these substances are possible to be obtained and optimal characteristics' for the sensitive elements of the chromatographic catarometer to be found.

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Recommended for publication by Department of Chemistry, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.123-125

ON THE POSSIBILITY OF PHOSPHATES REMOVAL FROM WASTE WATERS AND OBTAINING OF MIXED FERTILIZERS WITH NATURAL ZEOLITES

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ABSTRACT
The possibility of utilization of phosphates' saturated zeolites as raw materials for the obtaining of mixed phosphorous fertilizers has been substantiated. Taking into consideration that the regeneration of the sorbing agent is economically baseless, the proposed alternative solution presents an interest.
It has been demonstrated that the adsorbed in the structure of the zeolite phosphate ions are in the form of P_2O_5, able to be assimilated. The cation less and modified forms of the different clinoptilolites' samples preserve their structure and adsorption capacity. The quantities of the total P_2O_5 and of the citrate's soluble P_2O_5 in the calcium and potassium modified forms are more important that in the natural zeolites. In spite of the lower content of able to be assimilated phosphorous in the potassium form in comparison with the calcium modified, the first one is recommended as a raw material for obtaining mixed phosphorous fertilizers, due to the introduction in the ground of the essential for the plants potassium.

INTRODUCTION

The widespread natural zeolites, including in our country, find a polyvalent application in several areas of the industry and the agriculture. Today predominates their use as catalysts or catalysts' matrix in petrochemical industry and oil refining processing, as adsorbing agents in order to solve ecological problems, as nutritional additives to insure useful and essential microelements for the breeding and the plant-growing.

Their unique adsorption's and molecularly sieve's properties initiated the studies on the natural clinoptilolites, beginning from the 60th years of the XX century, in order to remove phosphate ions from waste and washing waters from phosphates' productions. The economically baseless regeneration of phosphates' saturated adsorbing agents represents a prerequisite for the search of an alternative solution for their posterior utilization. It is known that the mechanic mixture of clinoptilolite and super phosphate (Lian, et al., 1978) ensures the more complete phosphorous assimilation by the plants and the possibility of supplementary ground's enrichment with useful microelements, as well as the amelioration of its structure.

For this purpose, the synthesis of phosphorous containing zeolites has been tested, by substitution of Al from the network with phosphorous, applying a controlled copolymerization and coprecipitation in homogeneous phase. However, the characteristics of the obtained products are their reduced thermic stability and adsorption capacity, that sometimes reach 50 % of that of the non-containing phosphorous zeolites (Flanigen, et al., 1971).

That why the investigations on the phosphates' removal by adsorption with natural zeolites and the use of the enriched sorbing agents as a raw material for the obtaining of mixed mineral fertilizers represent an interest.
Currently phosphates' removal is carried out by neutralization with lime or by intermediary flocculation with polyelectrolytes (0,20 - 0,25 mg/cm^3) with pH $\approx$ 11. The degree of extraction reaches 90 %.

RESULTS AND DISCUSSIONS

In order to study the sorption's mechanism of the phosphate ions on clinoptilolote, comparative researches on the sorption capacity of samples coming from some bulgarian deposits and one from Georgia, whose chemical composition is presented in table 1, have been accomplished.

Table 1. Chemical composition of the tested clinoptilolites' samples

№	Sample	Chemical composition, %							Silicate module	Mineral content in the sample, %
		SiO_2	Al_2O_3	Na_2O	K_2O	CaO	MgO	Fe_2O_3	SiO_2/Al_2O_3	
1	BG-green	66.60	11.32	1.58	3.37	2.16	0.23	0.88	9.98	90
2	BG-pink	68.90	11.63	1.93	3.76	1.85	0.37	0.88	10.05	90
3	BG-white	66.40	12.30	1.87	3.50	2.00	0.52	2.21	9.16	70
4	Georgia-Dzegvi	68.04	14.41	2.08	1.80	6.87	2.20	3.93	8.30	65

Admixtures of β-quartz, β-tridymite, orthoclase, albite, anorthite, biotite, chrysotile (in the green clinoptilolite BG-green) have been established.

The preliminary researches concern studies on the influence of particle dimensions on the adsorption capacity of the samples. The experiments have been carried out using three samples of each type with particles' dimensions in the interval respectively 0.2 – 0.5 mm, 0.8 – 1.0 mm, 1.2 – 1.5 mm. The effect of the granulation composition of the clinoptilolite on the adsorption is presented in fig. 1. The obtained results allow to admit that the following tests have to continue by using samples with particles' dimensions 0.2 – 0.5 mm.

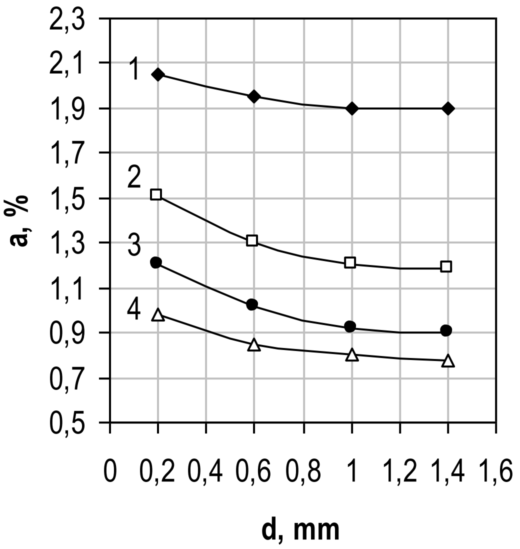

Figure 1. Influence of the clinoptilolite particles'dimensions on the sorption capacity, %:
1) BG-green; 2) BG-pink; 3) BG-white; 4) Dzegvi.

In fig. 2 are presented the adsorption isotherms obtained at ambient temperature and with particles' dimensions of clinoptilolite 0.2 – 0.5 mm. A direct relationship between the adsorption capacity and the stability towards acids has been observed, as well as of the purity of the mineral. The important quantity of limestone in the zeolite from Dsegvi - Georgia explains its higher capacity against P_2O_5, that will be mentioned again below.

It is known that an important characteristic of the phosphorous fertilizers represents the quantity of the able to be assimilated P_2O_5, i.e. phosphorous soluble in water and in citrates.

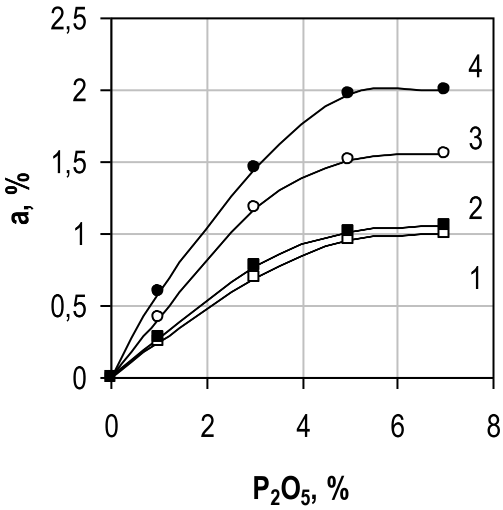

Figure 2. Adsorption isotherms of P_2O_5 at 20^oC on clinoptilolite with particles'dimensions 0.2 – 0.5 mm:
1) BG-green; 2) BG-pink; 3) BG-white; 4) Dzegvi.

It is interesting to test under what form is the adsorbed phosphate. For this purpose an experiment has been accomplished, during which samples of clinoptilolite with similar particles' dimensions have been saturated with an aqueous solution of P_2O_5 with a concentration of 3%. Then the samples have been treated respectively with Petermann reagent (ammoniac solution of ammonium citrate), with distilled water and with HCl 20 %. The quantities of the different forms of phosphates in the filtrate have been established, as shown in table 2.

Table 2. Solubility of the adsorbed by the clinoptilolite phosphorous

№	Clinoptilolite	SiO_2/Al_2O_3	Water soluble		Citrate soluble		Acid soluble		Total P_2O_5, %
			%	% total	%	% total	%	% total	
1	BG-green	9.98	2.09	91	0.07	3.00	0.14	6.00	2.30
2	BG-pink	10.05	2.46	88	0.10	5.50	0.24	8.50	2.80
3	BG-white	9.16	9.55	77	0.37	3.00	2.48	20.00	12,40
4	Dsegvi	8.32	9.76	63	0.78	5.00	4.65	30.00	15.19

The obtained results allow concluding the following:

1. As much the silicate module is great, i.e. as much the quantity of aluminum in the structure is small, as much the adsorption capacity of the clinoptilolite expressed via P_2O_5 is low and as much is the relative content of its soluble in water form.
2. The soluble in citrates phosphorous that, as it is known, is a phosphate of alkaline earth elements and especially of the calcium is the best to the plants form of P_2O_5, able to be assimilated. Its constant quantity in all samples with the exception of that from Dsegvi, in which it is approximately 1.5 times greater represents again a proof for the higher quantity of limestone in this sample, who's chemical and Rö-structural analysis confirm that the $CaCO_3$ content reaches 40 %.

3. In order to increase the sorption activity of the clinoptilolite's natural tuff, expressed via P_2O_5 and taking into consideration data in table 2, experiments using aluminum less and modified

samples have been carried out. For this purpose, the zeolite BG-green has been treated with HCl 4 mol/L at 25°C by applying a standard methodology [3] and a cation exchange has been accomplished, transforming it in a hydrogenated form. The partial Al substitution in this case increases the silicate module and leads to a supplementary widening of the structural pores. This insures a higher adsorption capacity. Parts of the obtained dehydrogenated form are transformed in calcium and potassium forms by treatment with $CaCl_2$ 1 mol/L and with KCl 1 mol/L respectively. This kind of modification is chosen in order to increase the quantity of the citrates' soluble phosphorous and to ameliorate the nutritional properties of the mixed fertilizer by the addition of a third important for the plants element, the potassium. Aside this, it is known, that when in the structure of the zeolite are introduced cations with greater dimensions than those in the natural zeolites, its thermic stability and elements' arrangement in its structure are ameliorated. Thermodynamic and Rö-structural studies show that the skeleton is conserved, while the results of the chemical analysis are given in table 3.

Table 3. Chemical composition of the modified clinoptilolites' samples

№	Sample	Components, %							SiO_2/Al_2O_3	Degree of aluminum exchange, %
		SiO_2	Al_2O_3	Na_2O	K_2O	CaO	MgO	Fe_2O_3		
1	BG-green	66.60	11.32	1.58	3.37	2.16	0.23	0.88	9.98	-
2	H-BG-gree	70.30	9.14	0.46	2.86	0.56	0.19	0.55	1.,08	19.25
3	Ca-BG-gree	71.01	9.20	0.65	2.98	4.20	0.18	0.68	13.11	18.72
4	K-BG-gree	70.85	9.08	0.55	5.25	1.16	0.20	0.86	13.26	19.78

The obtained modified and cations less forms have been saturated with P_2O_5 in static conditions and at ambient temperature. Each of the samples, in quantity of 1.0 g, has been treated during 1.5 – 2 hours with H_3PO_4, containing 4 % of P_2O_5. It has been established after analysis, that the calcium form contains 4.76 % of total P_2O_5 and 0.15 % of citrate soluble P_2O_5, while in the potassium form these values are 4.07 % of total P_2O_5 and 0.13 % of citrate soluble P_2O_5, what is approximately 50 % more than of P_2O_5 forms in the natural clinoptilolite.

In spite of the lower adsorption capacity of the potassium form, it is recommended as a raw material for obtaining mineral mixed fertilizers, because of the introduction of the third essential for the plants' grow element, the potassium.

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Recommended for publication by Department of Chemistry, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.127-129

BRIQUETTING OF BROWN COALS WITH A BINDING AGENT MODIFIED AMYLUM WITH SOLUBLE COLOPHONY

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ABSTRACT
Possibilities of modified amylum with soluble colophony are investidated at briquetting of brown coals. Performance indices of briquettes are expressed by the strenght indices of briquettes and their moisture resistance. A combined action of the binding agent modified amylum with soluble colophony with an additive of hydrated lime in quantity of 2 to 6% and aluminium sulphate in quantiti of 2 to 6 % is investigated. Practical interest represents the investigation of briquettes for compressive strenght of one week period.

INTRODUCTION

Briquetting without binding agents of fine-grained coal waste of brown coals and coals of more advanced degree of coalification does not apply in practice.
Qualities of the binding agents, as their market value, are decisive at use of one or other binding agent.

Popular used binding agents are :
- organic matters : coal-tar pitch, oil bitumens, sulphite-cellulose liquor etc.
- inorganic matters : cement, gypsum, lime, water glass etc.

The inorganic matters like binding agents allmost do not apply in briquetting as they increase the ash content and decrease moisture resistance of the briquettes and their heat value. [1], [2]
.

About some organic binding matters used in the past it is established that they are dangerous of cancer and they do not reply of the ecological reguirements.

The advanced trends of search and use of organic matters like binding agents are mainly to pitches, dextrin, molasses, amylum, synthetic received organic compounds etc. [3].

It is known from the literature sources that briquettes with a binding agent like amylum and molasses are not moisture resistant and briquettes with a binding agent like peat and sugar solution have low streght indices.

THEORETICAL PART

The modified amylum with soluble colophony (MASC) uses like a binding agent in paper industry like 10 % water solution. The colophony is hydrophobic and insoluble in water. It is used for hydrophobization of paper and cardboard surfaces, known in plactice like a sizing process. The use of the colophony for sizing is impossible in its natural appearance.
Its leading to the suitable state for the sizing process of paper becomes by saponification with sodium hydroxide (NaOH) or a fused soda ash (Na_2CO_2) according to the reactions :

$$C_{19}H_{29}COOH + NaOH = C_{19}H_{29}COONa + H_2O \qquad (1)$$

$$2C_{19}H_{29}COOH + Na_2CO_3 = 2C_{19}H_{29}COONa + H_2O + CO_2 \qquad (2)$$

In both cases as result of saponification of the insoluble abietic acid form sodaic resinate. Their fixing are made by addition of aluminium sulphate in excess.

The aluminium sulphate influences positively on the sizing by means of its own metalions.
The residual sulphuric acid holds up necessary pH of the medium in an interval 4,5 – 5,5.

EXPERIMENTAL PART

Technique and materials

For holding up of investigation in laboratory conditions about briquetting of brown coals from the town of Pernik with a binding agent like modified amylum with soluble colophony is used a sample of the following characteristic :

-Initial dampness, %	20,3
- ash content of dry matter,%	26,3
- sulphur content in dry matter,%	1,2
- caloricity, kcal/kg	4200

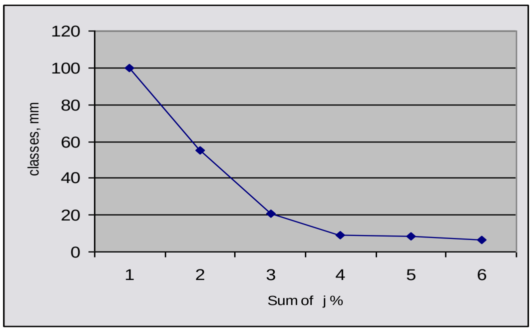

Figure 1. Granulometric characteristic of the coals of a Pernik's mine

The technique for holding up of the investigation is next :
Certain quantity of the coals (45 g) mixes with modified amylum with soluble colophony (MASC). the influence of the hydrated lime is investigated joint with MASC. The quantity of the additive of hydratedn lime is changed in an interval from 2 to 10 %. The mixture homogenizes well by stirring and it dumps in a pressform. It closes by the upper fixed piston and presses at pressure to 138 kg/cm^2.

The received briquette releases of the pressform after taking down of the upper pistin and the lower piston moves through an additional element. So the briquette pushes out from the cylindrical form. The strenght indices define after staying 2 hours. In orderto have compatibility between the received briquettes with a different content of binding agents, briquettes are prepared without a binding agent.

The defining of the strenght indices of the received briquettes accomplishes by following technique. Ten nombers briquettes are taken and weighed out to an accuracy of 0.1 kg. Every briquette falls from height 2 m. over a metal plate with thickness 15 mm. and side boards high 200 mm. it the briquettes after the first falling down do not destroy, they fall down to destruction one more time, twice and so on. The throwing does not repeat in case, that the received pieces are smaller than 25 mm. after destruction at the first falling down.

Experimental equipment

The investigation on briquetting is realised by means of a laboratory type press. It is manufactured in the technical laboratory of the University. The press form a briquette at one-sided pressure feeding in a vertical direction and it ensures a pressure to 138 kg/cm^2.
The pressform is cylindrically formed by an upper fixed piston and a lower mobile piston and give a possibility to receive briquettes with a diameter 45 mm and a height 65 mm.

Experimental results

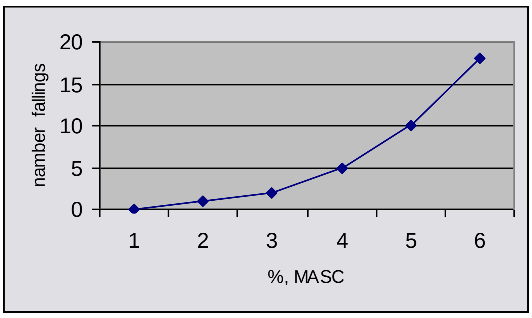

Figure 2. MASC, % + 2% hydrated lime

In figure 2 are given the experimental results reflecting the relationship between the content of MASC and a constant additive of 2% hydrated lime and numbers of fallings.
The optimal quantity of MASC is 6% at which the received briquettes are thick, without cracks, with increased "green " strenght. They are not loose and they are economic advantageous.

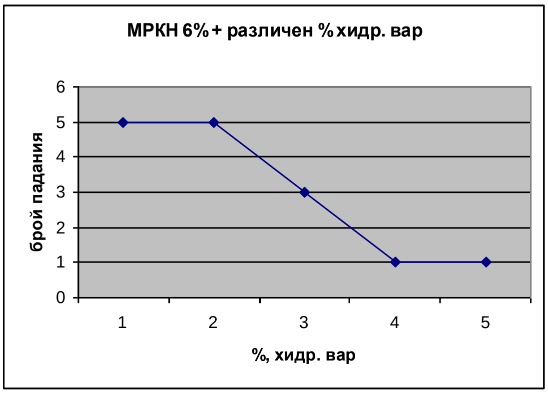

Figure 3. MASC 6% + hydrated lime,%

In figure 3 is displayed the relationship, between the consumption of hydrated lime with constant quantity MASC 6% and the numbers of fallings.

The received results show that the optimal content of hydrated lime is 2-4 %. Then the briquettes are mechanical strong and moisture resistant.
In figure 4 is displayed the relationship between strenght indices of briquettes established at usage of a binding agent MASC 6% and 2% hydrated lime and the time of outage of briquettes.

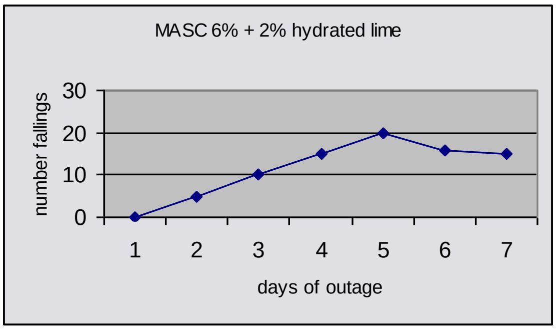

Figure 4. An investigation of briquettes of outage MASC 6% + 2 % hydrated lime

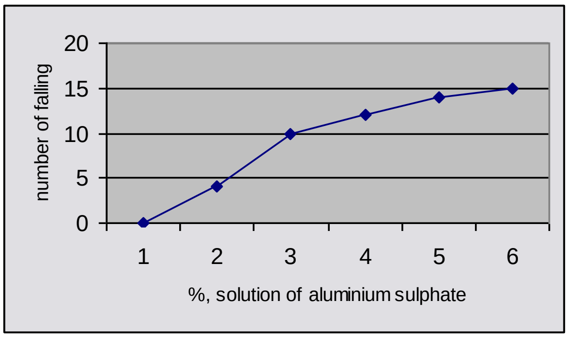

Figure 5. MASC 6% + $Al_2(SO_4)_3$, % + 3% hydrated lime

In figure 5 in a graphic mode is expressed the relationship between the used binding agents 6% MASC, 2% hydrated lime and aluminium sulphate ($Al_2(SO_4)_3$) of 3 to 6 % and mechanical characteristics of the briquettes.

The laboratory experiments investigate the influence of modified amylum with soluble colophony and hydrated lime upon the qualitative indices of briquettes.

DISCUSSION

Results which are received after laboratory investigation on briquetting wich a binding agent modified amylum with soluble colophony of brown coals from the town of Pernik proof, that the briquetting ensures the receiving of of briquettes with improved mechanical characteristicsand moisture resistance.
The optimal quantity of a binding agent from the point of view of the technology is 6%. Briquettes with MASC under 4% do not receive which is due of partially and obviously insufficiently soaking of the surfaces of coal particles. The utilization of MASC about 6% is unprofitable from an economical point of view of the advanced marcet prices.

The utilization of hydrated lime in the process of briquetting has an positive effect which expresses in warming of the briquettes at their reaction with the available moisture from solution of the binding agent. Bonds between coal grains reinforce at the carbonization. The decrease of the free moisture between the particles in the briquette also is not less important.

The best results as regards to the mechanical strenght give briquettes formed with 2 – 4 % hydrated lime. The moisture resistance is the highest at these values. The increase of the quantity of the hydrated lime in briquettes to 8-10 % makes worse their solidity because of presence of a residual calcium base which reacts subsequently with the atmospheric CO_2.
Briquettes received at content of MASC 6% and hydrated lime are investigated for a change of mechanical characteristics after outage at a room temperature 10^0C. After 4 days outage according to the graphic relationship 4 reach maximum number fallings from a height 1,5 m according to an Bulgarian standard over a hard surface.The trend to decreasing of the mechanical characteristics of briquettes is available after the fourth day.

The received briquettes with a binding agent MASC reply to the requirements of a Bulgarian standard about dampness which is in the interval from 10,9 to 13% and ash which is from 20,8 to 22,9 %. The compressive strenght is between 7,3 and 8,6 Mpa and the strenght of falling is in the limits 80-91%.

Analogue investigation are accomplished with briquettes formed without a binding agent.

The briquettes without a binding agent have very low values of strenght indices. The combined action of MASC and hydrated lime creates good conditions for receiving of briquettes from the Pernik's brown coals. They have high moisture resistance and high strenght of falling. An additive positive effect give the presenceof hydrated lime in the briquettes in the process of the burning. This effect expresses in neutralization of the received sulphur oxides.

CONCLUSION

The action of modified amylum with soluble colophony is investigated like a binding agent at briquetting of brown coals. The received briquettes are with good mechanical characteristics and moisture resistance. The optimal quantity MASC is from 6%.

The combined action of MASC and hydrated lime is investigated too. An established optimal quantity is 2-4% hydrated lime. The received briquettes from this combination have improved moisture resistance and strenght of falling.
The addition of aluminium sulphate improves the mechanical characteristics of briquettes which are received with a binding agent MASC and hydrated lime.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.131-135

PLASMA-CHEMICAL PROCESSING OF MINERALS AND INDUSTRIAL WASTE

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ABSTRACT

Results are presented of the authors' investigations on plasma-chemical processing, under conditions of electric arc thermal plasma, of minerals (zircon, natural phosphates, bulgarite, serpentinite, pyrite), industrial concentrates (manganese oxide, molybdenum, cobalt cake) and industrial waste – solid (phosphogypsum, and refuse from Medet Company), liquid (waste of organics production) and gaseous (chimney gas from Thermo-Electric Power Stations, chemical and metallurgical enterprises, etc.). Processes of plasma-chemical production of nano-dispersed powders (NDP) are reviewed, namely plasma re-condensation (zircon, corundum), plasma destruction and/or reduction of zircon, manganese oxides, molybdenum compounds (molybdenum concentrate, molybdenum oxides), cobalt cake, iron-bearing compounds, etc. Design of electric-arc plasmotrons, plasma-chemical reactors and mixers, quenching devices and powder collectors is presented. A mechanism has been proposed for the plasma-chemical production of nano-dispersed powders (NDP) under conditions of high-enthalpy plasma jets. Production of NDP (metals – Mn, Mo, Co, Fe, oxides, nitrides, pigments, catalysts, etc.) is reviewed, which NDP are of practical interest (for example – in different contactors production, in powder metallurgy, chemical industry, machine-building, etc.).

INTRODUCTION

Plasma-chemical processing of different minerals, concentrates and industrial waste has been increased recently (Vissokov, 1984, 1987, 1998). The interest of investigators and users to this processing method is predetermined by the indisputable advantages of the plasma-chemical processes (PCP): 1. These processes proceed with high rate (contact time of $10^{-5} \div 10^{-3}$ s) at high specific enthalpies and high temperatures – all these factors lead to a miniaturization of plasma-technology equipment; 2. Most of the interesting from the practical point of view PCP are one-stage processes; PCP are practically insensitive to the admixtures in the raw materials; 4. PCP can be used to process difficult for processing but widely spread raw materials (air, ores, industrial waste, etc.); 5. PCP can be used to utilize valuable components from inorganic and organic chemistry production, from metallurgy at simultaneous waste destruction to non-harmful products that do not pollute the environment; 6. PCP can be easily modeled, optimized and controlled. Gas and electrodynamical controlling methods can be applied. This type of control leads to decreased thermal stability requirements for the plasma-chemical (PC) equipment; 7. All PCP can be organized in a similar technological scheme. The principal scheme of the PC equipment, independently on the PCP used, consists of low-temperature plasma (LTP) generator (plasmotron), mixer, plasma-chemical reactor (PCR), quenching device and a device for collection and separation of final products. 8. The plasma-chemical treatment does nor require big capital investment; 9. Uniform treatment can be achieved.

Goals of the present work were:

1. To summarize our investigations on plasma-chemical processing, under conditions of electric arc thermal plasma, of minerals, industrial concentrates and industrial waste – solid, liquid and gaseous.

2. To summarize the results of studies on plasma-chemical synthesis of nano-dispersed powders (NDP) by plasma re-condensation and plasma destruction and/or reduction.

3. To review the equipment (our design and preparation) for different electric-arc plasmotrons, plasma-chemical reactors and mixers, quenching devices and powder collectors.

EXPERIMENTAL EQUIPMENT

The following equipment has been designed and constructed (Vissokov, 1984):

- Electric arc plasmotrons – with two and many sections, with the capacity from several to 15 kW;
- Plasma-chemical mixers – cylindrical, truncated cone, etc.;
- Plasma-chemical reactors (metal and ceramic) – with "cold walls" (CW) – temperature of the wall T_{cw} = 500 K and with "warm walls" (WW) – temperature of the wall T_{ww} = 1500 K;
- Powder collecting devices: cyclones, sleeve-filters, electrical filters, etc.

A principal scheme of a plasma installation used for processing of minerals and industrial waste is presented in Fig.

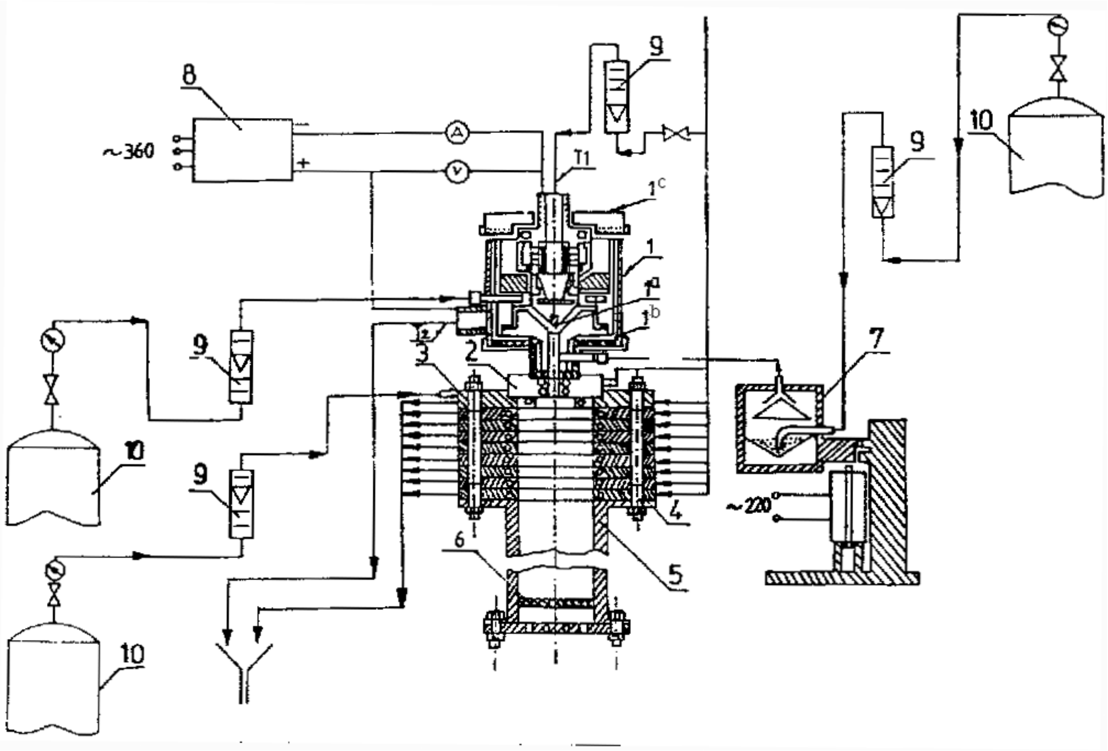

Figure 1. Schematic drawing of the plasma-chemical installation
1 - electric-arc D.C. plasmatron, 1ª – thoriated tungsten cathode, 1ᵇ – copper water-cooled anode, 1ᶜ – plastic adjusting ring, 2 – CW PCR, 3 – quenching device, 4 – copper water-cooled sections for the quenching device, 5 – powder-trapping chamber, 6 – filter, 7 – vibration powder-feeding device (if necessary, a piston type vibration powder-feeding device can also be used), 8 – current rectifier, 9 – flow-meters, 10 – bottles with plasma-forming, powder carrying and quenching gases, T_1 – temperature of inlet water, T_2 – temperature of outlet water.

RESULTS AND DISCUSSION

Model and thermodynamic calculations

Three-dimensional models of the motion, heating, melting, and evaporation (thermal destruction) of micron-size particles in an axial-symmetric PCR were developed. The models give the opportunity to determine the optimal parameters of the PCR: dimensions of the reactor, residence time of particles in the reactor, temperature in the PCR (CW) and PCR (WW), changes in particles size along the PCR, etc. The models allowed also obtaining the rate, density and temperature profiles of the plasma (gas) in the PCR, as well as parameters characterizing the motion, heating, melting and evaporation / thermal destruction of micron-sized particles in the PCR. Good correlation was found between model calculations and experimental results (Vissokov, 1998).

Thermodynamic equilibria calculations in multi-component systems are based on the fact that the Gibbs free energy reaches an extremum at the equilibrium. An isolated system (not exchanging mass and energy with the surroundings) was considered. Thermodynamic calculations were made at pressure of 0.1 MPa within the 1000÷3700 K interval (step of 300 K) at ratio solid phase / gaseous phase = 100 g/m^3. The following materials were considered as raw materials: natural phosphate, phosphogypsum, phosphite from Marocco, pyrite concentrate, refuse from Medet Company, bulgarite, serpentinite. Different mass ratios of ingredients were applied (Vissokov, 1984, 1998). The following components were considered in the solid phase (depending on the composition and ratio of ingredients): CaO, $CaSO_4$, Ca_2SiO_4, $Ca_2Al_2SiO_7$, CaF_2, $Ca_{10}F_2(PO_4)_6$, $Ca_2F_2O_5$, $Ca_3(PO_4)_2$, FeO, Al_2O_3, $CaMgSiO_4$, etc. The following components were considered in gaseous phase: S, SO, SO_2, SiO, Ca, CaO, CO, CO_2, PO, PO_2, F, MgF_2.

Experimental investigations on the plasma-chemical processing of minerals and industrial waste

Plasma destruction of minerals and concentrates

Thermal destruction of zircon was carried out (Vissokov, 1998) using the installation shown in Fig. 1. The crystal lattice of zircon was destroyed at temperature higher than 2050 K and a mixture of oxides was obtained:

$$ZrSiO_4 \Leftrightarrow ZrO_2 + SiO_2 \quad (1).$$

The mixture was treated with 50 % solution of NaOH at temperature of 530 K. As a result, the amorphous SiO_2 was transformed into soluble salt:

$$SiO_2 + 2NaOH \Leftrightarrow Na_2SiO_3 + H_2O \quad (2),$$

and ZrO_2 (grade of 95-96 %) was left in the precipitate. Even without SiO_2 leaching, the mixture of ZrO_2 and SiO_2 can be used for coloring the thermally resistant ceramics, as refractory and/or abrasive material, for chemicals production, etc. Materials made of ZrO_2 are heat-resistant up to 3000 K and those made of ZrO_2 produced by destruction – up to 2000 K.

Plasma destruction of Mn and Fe minerals (wustite, magnetite, hematite, and manganese oxide concentrate) turned to be an effective method for their transformation to inferior compounds (oxides, for example). Destruction of the lowest-valence oxides (MnO, FeO) to the corresponding chemical element and oxygen represents the limiting stage of the thermal destruction of the minerals.

Plasma-chemical re-condensation of high-melting-point substances is a method for producing nano-dispersed powders (NDP) (Vissokov, 1998). Nano-dispersed Al_2O_3 with specific surface of 30 ÷ 50 m^2/g (depending on the plasma-process conditions) was obtained by re-condensation of corundum (α-Al_2O_3) with particles size less than 50 μm.

Plasma-chemical reduction of ores and concentrates

The plasma-chemical reduction of metal oxide and metal sulfide concentrates (cakes) represents a new direction in the technology for processing metal-bearing raw materials and residues. Nano-dispersed metal powders (Mn, Mo, Co) have been obtained for the first time worldwide by reduction of corresponding concentrates (manganese oxide, molybdenum, cobalt cake) (Vissokov, 1998) under conditions of electric arc LTP at laboratory and/or pilot scale with use of gaseous reductants (H_2, C_4H_{10}, NH_3). Nano-dispersed Fe (with specific surface up to 160 m^2/g) was obtained by reduction of Fe_2O_3 with H_2, and the ratio α-Fe/γ-Fe can be controlled by changing the quenching rate.

Investigations on reduction of Co cake are presented here as an example. The Co cake is residue by-product of the Zn production by means of electrolysis of lead-zinc concentrates. The cake usually contains the following metals (in mass %): Co 2÷6, Zn 9÷11.5, Ni 0.0007÷0.0009, Cu 0.1÷1.0, and Cd 0.16÷0.5. Cake with the following composition was considered in our investigations aimed at increasing the Co amount in the cake (in mass %): Co 4.02, Zn 11.2, Ni 0.0007, Cu 0.72, Cd 0.16, and S 25.5. The cake enrichment with Co by means of plasma chemical reduction is based on the reaction:

$$CoS + H_2 \Leftrightarrow Co + H_2S \qquad (3).$$

Thermodynamic calculations for the reaction were made. Standard enthalpies of the substances participating in the reaction were used. Their aggregate state at corresponding temperatures was used in calculations. Results are presented in Table 1.

Based on the results shown in Table 1, it can be stated that from the thermodynamics point of view, the reduction of CoS with H_2 under the conditions of LTP has to be carried out in the temperature range of 1500 ÷ 2500 K.

Table 1. Temperature dependencies of the Gibbs free energy and equilibrium constant for the reaction (3)

T, K	1000	1500	2000	2500	2000	3500	4000
$\Delta G_T.10^3$, J/mol	7.53	-64.4	-33.8	-3.17	4.25	16.8	29.4
lgK_p	-1.64	9.39	3.69	0.277	-0.309	-1.05	-1.61
K_p	$2.3.10^{-2}$	$2.55.10^9$	$4.9.10^3$	1.9	0.49	$8.9.10^{-2}$	$4.05.10^{-2}$

Similar calculations were made for other components of the cake, which components can be reduced more easily. By drying at 500 K the cake was transformed into black powder with the following composition (in mass %): Co 6.08, Zn 16.9, Ni 0.0011, Cu 1.08, Cd 0.24, and S 19.9. The powder was plasma-chemically treated in the installation shown in Fig. 1. Results from chemical analysis (Co content, degree of reduction) and specific surface determination are presented in Figs. 2 ÷4.

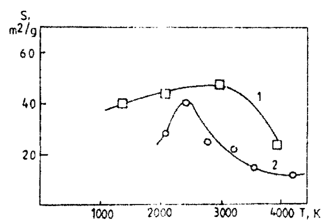

Figure 2. Specific surface (dispersity) as function of the average mass temperature of the plasma: 1 – at H_2 flow of 1540 L/h, 2 – at H_2 flow of 770 L/h.

The maximum in Figs. 2 ÷ 4 could be explained with the influence of opposite processes. The maximum observed in Fig. 2 could be assigned to the action of the following factors: a) as the share of the process taking place in gaseous phase

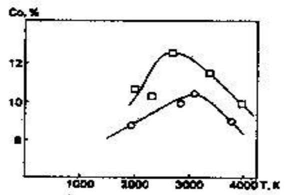

Figure 3. Co content in the product obtained as function of the average mass temperature of the plasma: 1 – at H_2 flow of 1540 L/h, 2 – at H_2 flow of 770 L/h.

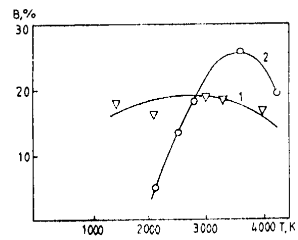

Figure 4. Reduction degree as function of the average mass temperature of the plasma: 1 – at H_2 flow of 1540 L/h, 2 – at H_2 flow of 770 L/h.

increases with the temperature increase, the specific surface of the product obtained also increases, and b) as the temperature increases, the process of NDP sintering in the dust-collecting chamber is enhanced and that leads to a decrease in the specific surface. The maximum observed
increases with the temperature increase, the specific surface of the product obtained also increases, and b) as the temperature increases, the process of NDP sintering in the dust-collecting chamber is enhanced and that leads to a decrease in the specific surface. The maximum observed for the dependence "Co content / average mass temperature in the reactor" is moved to higher temperature at increasing the reductant flow (Fig. 3), because the reductant is used as powder-carrying gas and its movement decreases the average mass temperature in the PCR. The maximum observed for the dependence "degree of reduction of Co cake/ average mass temperature" depends also on the reductant flow (Fig. 4). At higher flow, the temperature in the reaction zone of the PCR decreases and as a result the equilibrium is moved to the left (towards sulfide). A mechanism with the following stages can be proposed to describe the process of Co cake reduction in H_2 medium: 1) heating the cake particles (predominately sulfides of Co, Zn, Cu, Cd and Ni) up to the melting point – heterogeneous reduction of sulfides takes place at that stage; 2) spheroidization of nearly-melted particles; 3) particles melting and formation of spherical drops; the drops are formed at constant temperature (melting temperature) and their size is gradually decreasing due to the evaporation; 4) entire evaporation of particles and substance transition to gaseous phase (under the form of atoms and radicals) – at temperature of 4000 K; processes of homogeneous reduction take place; 5) particles condensation in different phases during the quenching. Intensive reduction takes place during the first four stages (up to 4000 K). Conditions for the reverse reaction, described by equation (3) are created in the fifth stage, depending on the quenching effectiveness.

X-ray analysis of the products obtained showed that the Co is under the form of elemental Co, CoO and CoS. This finding shows the high chemical activity of Co powder obtained by means of reduction under the conditions of LTP – the air oxygen easily oxidizes the metal.

Experiments have been carried out to reduce the Co cake with Ar-H_2 plasma using Ar as powder-carrier. Results are presented in Table 2.

Table 2. Technological parameters for the plasma-chemical reduction of Co cake

I A	U V	W kW	Ar_{pl} L/h	$H_{2_{pl}}$ L/h	Ar_{pc} L/h	Co %	S m^2/g	$S_{residual}$ %	B %
200	54	10.8	2550	390	340	11	36	18.26	8
300	55	16.5	2550	390	340	10.2	28	14.49	27.1

As it can be seen in the Table 2, compared to Figs. 2 ÷ 4, results for Co cake reduction by means of H_2 and Ar – H_2 plasma do not differ considerably. It has to be pointed out that Co powder concentrate containing about 20 mass % of Co and practically non-bearing S was obtained in pilot-plant installation using Ar – H_2 mixture and plasma-forming gas (Vissokov, 1998). This concentrate can be used to produce pure Co or Co compounds.

The main properties (such as specific surface, dispersity, degree of reduction, pyrophority, etc.) of the NDP, obtained by reduction, can be explained with the influence of the plasma-chemical process parameters and structural and chemical changes in the treated raw materials. Nano-dispersed metal powders (NDMP) obtained by plasma reduction of metal oxide and/or metal sulfide concentrates are characterized with their high specific surface (from several to several tens m^2/g), respectively - size of their predominately spherical particles – from several to 100 nm. The high dispersity of NDMP depends mainly on two factors: degree of raw material evaporation in the PCR and quenching effectiveness. The degree of raw material reduction depends on: material composition, temperature, kinetics of the corresponding reduction reaction (residence time in the PCR), quenching rate, effectiveness of the inhibition of the high chemical activity of the plasma-chemically synthesized NDMP. Physicochemical parameters of these powders can be controlled in a wide range by varying the parameters of the PCP.

The plasma-chemical production of NDMP (Mn, Mo, Co, Fe) can be of significant practical interest (for powder metallurgy, production of different Mo contactors, etc.), if raw materials with suitable composition are used and effective measures for NDMP passivation are taken. Suitable passivators were found (Vissokov, 1998) for

decreasing the high chemical activity (pyrophority) of NDMP: N_2 – for Mn powder, CO – for Mo powder, and N_2 + 0.5% O_2 – for Fe powder.

Plasma-chemical oxidation of chemical compounds, ores and concentrates

High-energy parameters of the electric arc LTP represent an effective factor for obtaining NDP of different materials, such as: SiO_2 (by hydrolysis and oxidation of $SiCl_4$ or by oxidation and destructive evaporation of quartz sand), Al_2O_3 (by oxidation of elemental Al with O_2), Fe oxide pigments (by oxidation of coarse Fe and pyrite dross), metal oxides (by oxidation of corresponding metal concentrates, such as manganese oxide concentrate, etc.). To find the suitable temperature range of investigations, values of reaction enthalpy and equilibrium constant were preliminarily calculated.

Optimal conditions (temperature range, consumed power, flow, ratio of amounts of reagents used, place of reagents introduction into the reactor, shape of the reactor, type of the reactor – CW or WW etc.) of the PCP were determined in each case in order to ensure production of NDP and oxides with pre-determined characteristics.

Physicochemical properties (bulk mass, specific surface, chemical and phase composition, particles shape and size, purity, availability of admixtures, etc.) of the plasma-chemically-obtained NDP of SiO_2, Al_2O_3 and iron oxides have been studied. It has been found that a) the particles specific surface can be controlled in a wide range – from several to several hundreds m^2/g – depending on the PCP parameters; b) ND particles are with spherical or nearly spherical shape, their size is between 5 and 100 nm, the degree of monodispersity depends on the PCP parameters; c) the bulk mass, that depends on the dispersity of ND oxides, is in the range from several to several hundreds kg/m^3; d) admixtures in the ND oxide powders are due to admixtures in raw materials or (in some cases) to the erosion of electrodes or of PCR lining.

The phase analysis showed predominately availability of high-temperature thermodynamically unstable modifications of the corresponding oxides, except of the SiO_2 that was obtained as amorphous phase. The chemical composition of the nano-dispersed oxide powders (in the case when more than one oxide was obtained, iron oxides – as an example) is a function of the average mass temperature in the PCR (temperature profile of the reactor), over-stoichiometric amount of oxidizer and the reaction duration. The degree of raw material oxidation is determined by the thermodynamic and kinetic parameters of the corresponding process and by the process parameters, such as temperature, oxidizer concentration, time of the contact between the raw material and the oxidizer.

A mechanism has been proposed to explain the availability of high temperature thermodynamically unstable modifications of oxides that form NDP depending on the plasma process parameters and quenching effectiveness (Vissokov, 1998).

A technology has been developed for plasma-chemical synthesis of ND SiO_2 by reduction-oxidation process of quartz sand evaporation. A pilot plant (with capacity of 5 t/annum) was built on the territory of Himko-Vratza (Vissokov, 1998).

The plasma-chemically-synthesized ND oxide powders can be applied widely, as it follows: ND SiO_2 – as filler in pharmaceutical, cosmetical, food, and rubber industry, etc.; ND Al_2O_3 – as inert filler, adsorbent, heat- and corrosion-resistant material, in powder metallurgy, etc.; ND iron oxide pigments – for mineral pigments production (black, dark brown, brown, yellow, violet); and ND metal oxide concentrates – for producing the corresponding metals (Co, Zn, Mn) and their oxides and other compounds.

Plasma-chemical deactivation and utilization of solid, liquid and gaseous waste

Literature survey and experimental studies have been carried out on the plasma-chemical destruction (deactivation) of solid (expired pesticides), liquid organic (petroleum residues, pitch organic residues) and gaseous (CO_2, NO_x, SO_2, H_2S, industrial smoke, etc.) waste (Vissokov, 1984, 1987).

Solid and liquid organic waste is destroyed mainly under conditions of electric arc LTP. High-calorific combustible gases are obtained at that. Pesticides are rendered harmless under conditions of water electric arc LTP. Gaseous toxic oxides are plasma-chemically treated under conditions of corona discharge of different types of electrical filters. Nitrogen fertilizers are obtained as final product.

CONCLUSION

Summarizing the results of our investigations on the plasma-chemical processing of minerals and industrial waste, it could be pointed out that the plasma-chemical method proved to be effective in two main cases: a) when it is impossible products with specific characteristics (ND powders and/or oxides with high purity) to be obtained by conventional methods, and b) highly toxic solid, liquid and gaseous waste is to be deactivated and/or destroyed in a safe way.

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Recommended for publication by Department of Chemistry, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.137-139

ENVIRONMENTALLY FRIENDLY TREATMENT OF WASHING ACIDS

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ABSTRACT
Copper recovering from washing effluents of copper smelting plants is discussed. Possibility for copper electrowinning at avoiding AsH_3 release in the atmosphere is presented from theoretical and experimental point of view.

INTRODUCTION

Copper smelting plants release highly acidic (pH = 0.3 ÷0.5) washing solutions of flue gas. The solutions usually contain As (2÷10 g/L) and Cu (30÷40 g/L). Elevated acidity and high Cu concentration represent environmental danger but high As concentrations pose the major danger related with these effluents. That is why the main attention is paid to As removal. The choice of a method for As removal from waste effluent is highly dependable on the As concentration. Use of sorption (including filtration trough adsorptive media), ion exchange, membrane processes, and biological treatment alone or in combination with methods for As(III) species oxidation to As(V), are technologies more suitable to remove As from drinking water or to treat wastewater with relatively low As content (up to 500÷1000 mg/L). Precipitation methods turned out to be more effective and economically feasible for As removal from wastewater bearing high As amount. Scorodite formation, sulfidization, use of ferrous or ferric salts, mixture of aluminum and iron salts, mixture of iron and lanthanum salts are among attempted methods. Liming still remains the method most widely spread worldwide for treating effluents containing high As concentrations.

Besides shortcomings connected with necessity to depose voluminous (in the most cases) precipitates, that some times are in equilibrium with high As concentrations and represent a possibility for secondary pollution of water contacting with the precipitates, precipitation methods show the disadvantage of impossibility to recover Cu from the precipitate formed.

Number of works dealing with the electrochemical treatment of acidic effluents, bearing Cu and As, has been increased recently (Peck, 1993; Zhang, 1997; Mochida, 1999; Panayotov et. al., 2000). In the most cases the treatment is based on electrolysis with dissoluble iron electrodes. Electro-generated iron ions act as precipitant and/or coagulant. Impossibility to recover Cu in economically feasible manner can be pointed out as shortage of this method.

To recover Cu from washing effluents avoiding release in atmosphere of hazardous substances (e.g. AsH_3) at simultaneous rendering the effluents environmentally harmless present a real challenge from economic, environmental and scientific point of view. Present paper is devoted to the above mentioned tasks.

THEORETICAL BACKGROUND

Different species of As(III) and As(V) have been determined in acidic media: $H_3AsO_{3(aq)}$, $HAsO_{2(aq)}$, AsO_3^{3-}, $H_3AsO_{3(aq)}$, AsO_4^{3-}, but considerable concentrations of positive ions, such as As^{3+} and AsO^+ were not detected (Cotton and Wilkinson, 1977).

While dealing with As species, one should take into account the danger of AsH_3 formation. Possible ways for AsH_3 formation are as it follows:

- Reduction of elemental As with Zn or Al in acidic medium, for example:

$$2As + 3Zn + 6H^{\circ} \rightarrow 2AsH_3 + 3Zn^{2+} \quad (1);$$

- Acidic hydrolysis of arsenides, for example,

$$Zn_3As_2 + 3H_2SO_4 \rightarrow 2AsH_3 + 3ZnSO_4 \quad (2);$$

- Electrochemical reduction of elemental As:

$$As^o + 3H^+ + 3e^- \Leftrightarrow AsH_{3(g)}\ ,\ {}^oE = -0.225\ V \quad (3);$$

- Reaction of As compounds with nascent (atomic) hydrogen.

Having in mind the above-said, it could be stated that possible ways to avoid AsH_3 formation are the following:
- To avoid conditions, leading to formation of elemental As;
- To avoid conditions favoring the reaction (3);
- To create conditions suppressing the eventual electrochemical formation of nascent hydrogen.

Possible way to avoid release of AsH_3 in atmosphere is to destroy it thermocatalytically.

The electrochemical reactions in which the As species (available in the washing effluents) could participate are, as it follows (Bard et al.,1985):

$AsO_4^{3-} + 8H^+ + 5e^- \Leftrightarrow As^o + 4H_2O$, $^oE = 0.648$ V (4)

$2H_3AsO_{4(aq)} + 4H^+ + 4e^- \Leftrightarrow As_2O_3 + 5H_2O$, $^oE = 0.580$ V (5)

$H_3AsO_{4(aq)} + 2H^+ + 2e^- \Leftrightarrow HAsO_{2(aq)} + 2H_2O$, $^oE = 0.560$ V (6)

$H_3AsO_{4(aq)} + 3H^+ + 2e^- \Leftrightarrow AsO^+ + 3H_2O$, $^oE = 0.550$ V (7)

$As^{3+} + 3e^- \Leftrightarrow As^o$, $^oE = 0.300$ V (8)

$AsO^+ + 2H^+ + 3e^- \Leftrightarrow As^o + H_2O$, $^oE = 0.254$ V (9)

$HAsO_{2(aq)} + 3H^+ + 3e^- \Leftrightarrow As^o + 2H_2O$, $^oE = 0.248$ V (10).

Copper ions' reduction is based on the reaction:

$Cu^{2+} + 2e^- \Leftrightarrow Cu^o$, $^oE = 0.338$ V (11).

Reactions (8) ÷ (10) can be suppressed by carrying out the process at properly chosen conditions (voltage applied). Carrying out the electrolysis under voltage control (in the range of potentials 0.25 ÷0.55 V[1]) is expected to lead to Cu extraction from washing effluent at avoiding / decreasing to a minimum the danger of AsH_3 formation.

The mechanism of H_2 evolution on the cathode (with a probable stage of atomic hydrogen formation) is very complicated and not very well established. Hydrogen evolution on cathode could decrease the adhesion of electrowinned copper to cathode. That is why, carrying out the reaction at avoiding/decreasing to a minimum the H_2 evolution would benefit Cu electrowinning and decrease the probability for AsH_3 formation. The possibility for H_2 evolution on cathode could be decreased by the following measures:

- Choice of a proper electrode material (with high H_2 overvoltage) and its proper surface preparation;
- Addition of small amount of $NaNO_3$ ($^oE_{NO3-/NO2-} = 0.010$ V);
- Leading the process at low temperature.

EXPERIMENTAL

Experiments were carried out with real washing effluent with the following parameters: pH = 0.4, Cu concentration 36 g/L, As concentration 7 g/L. Investigations were carried out in a electrolysis cell with working volume of 2 L, with specially chosen (in terms of material and surface finishing) electrodes. Small amount of $NaNO_3$ was added. Experiments were carried out under voltage control. Influence of applied potential, current density, electrode material and its surface preparation, as well as treatment time on the Cu electrowinning process effectiveness was studied. Standard laboratory equipment was used to measure the current and solution pH value. Copper concentration was determined by ICP-AES. Availability of AsH_3 was checked by the Gutzait method. Possibility was ensured to collect separately gases formed at cathode and anode. Gases collected at cathode passed through a special chamber where a possibility for thermocatalytic decomposition of eventually formed AsH_3 to As^o and H_2 was ensured.

[1] The influence of overvoltage due to Cu crystals' formation is taken into account.

RESULTS AND DISCUSSION

It has been found that Cu can be extracted at significant extent from the washing effluents. Depending on the working regime parameters Cu forms layer well adhering to cathode or part of the formed elemental Cu falls onto the bottom of the vessel. Amount of the electrowinned Cu and specific energy consumption (consequently – the energy cost) vary with working conditions – Table 1.

Table 1. Energy cost and Cu extraction efficiency

Working regime	Cu extraction from the solution, %	Cost (USD) of energy consumption per ton of electrowinned Cu
A	70.7	90
B	75.5	84
C	79.3	92
D	82.2	96
E	84.0	101
F	80.5	90

As it can be seen, working regime B, C and F ensures the optimal conditions.

In all experiments AsH_3 was not detected at the exit of the thermocatalytic chamber and around the electrolysis cell by use of Gutzait method.

When the electrolysis is carried out with indissoluble anode, what is preferable in order to avoid solution contamination with additional ions, the main anodic process is:

$2H_2O \Leftrightarrow O_{2(g)} + 4H^+ + 4e^-$, $^oE = 1.23$ V (12).

Practically, the solution pH value was not changed.

After the extraction of Cu, the washing effluent can be treated with soluble (iron) anode, as described in our earlier works (Panayotov et al., 2000, POCMIW; Panayotov et al., 2000, Balkema). Effluent neutralization, removal of As and other heavy metals can be achieved in this way.

CONCLUSION

Conditions could be found for Cu electrowinning from washing effluents of Cu smelting plants, at avoiding AsH_3 release in the atmosphere. This is achieved by suppressing AsH_3 formation by use of specially chosen and surface finished electrodes and by application of a thermocatalytic chamber able to destroy the eventually formed AsH_3.

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Recommended for publication by Department of Chemistry, Faculty of Mining Technology

50 years University of Mining dnd Geology "St. Ivan Rilski"
Annual, Vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.141-145

AIR DISTRIBUTION IN LARGE VOLUME VENTILATION OBJECTS

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ABSTRACT
Main task of ventilation is to ensure safety environment for people and equipment via pollutants' liberation and distribution control. Processes of emissions and distribution by air currents in ventilation objects with one directional flow are well studied and controlled though they are problem in ventilation objects with large volumes. In such objects different aero-dynamical zones exist. This paper comments parameters of ventilation objects with large volumes and presents main expressions, describing pollutants and air flows distribution.

INTRODUCTION

Directed flows of air currents in ventilation systems assumes well controlled ventilation paths with clearly defined aero-dynamical characteristics. Majority of mine workings can be described as the above-mentioned objects. There exists though suck configurations, object to ventilation, where ventilation current is not clearly defined, zones with free and semi-restricted jets, stagnation and recirculation ones can be obeserved.

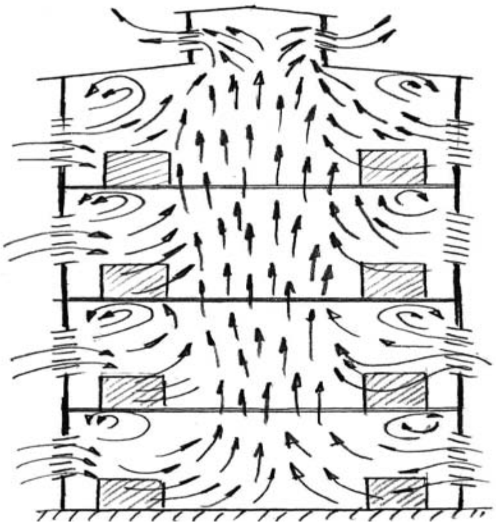

Figure 1. Industrial plant

Similar ventilation configurations are many of industrial ventilation objects [2] – industrial plants and halls, with different in type, action and consistence heat and mass sources, surface facilities of underground mine, camera type and blind mine workings, metro stations, car parks etc. Classification of one object as Ventilation Object with Large Volume (VOLV) assumes definition of critical relationships between dimensions, ventilation in and out flows, aero-dynamical characteristics, and similarity criteria.

One possible approach for analysis of air flows distribution is physical and mathematical modelling, which include:

- Based on analysis of technological processes clarification of all potential sources;
- Description of sources and their influence;
- Modelling of their distribution;
- Zones description where safety conditions are violated;
- Planning of ventilation measures;
- Modelling of their influence;
- Results analysis.

This paper deals with part of these problems, namely – mathematical expressions, presenting main processes in ventilation objects with large volumes.

VENTILATION OBJECTS WITH LARGE VOLUMES

As already mentioned in the introduction, ventilation object with uniform flow modes in different zones and directions of ventilated volume are analysed in this paper. Figures 1, 2 and 5 present such objects. Depending on purpose of different sections of objects, controllable, uncontrollable and isolated zones can be defined (figure 3). From other side, in regard to way of air inflow and distribution zones with free and semi-restricted jets, main air current, stagnantion and recirculation zones exist.

These general determinations for analysed types of objects depend on geometrical, aero-dynamical and ventilation characteristics. Objects' dimensions are presented by length (L), width (B) and height (H), as well as inflow/outflow holes, serving as supply and exhaust air paths. One example of industrial plant with large volume is represented by& main hall with length 180 m; width - 45 m; height – 20.95 m (15.50 to the roof and 17.30 to фонара).

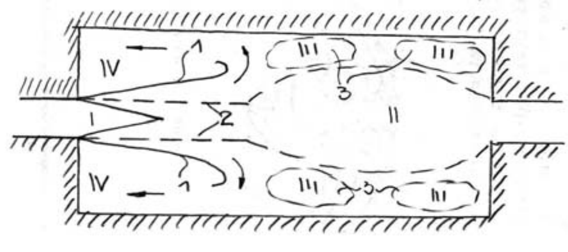

Figure 2. Camera type mine working

The roof is with фонар, having windows along the whole hall length. Plant volume is 137 214 m^3, which assignes it as VOLV. Inflow/outflow supply/exhaust openings are located at levels 0.00, 4.50, 10.30 and 17.30. Their surfaces (not all of the open able) are approximately 2000 m^2. One impeding factor is existence of great holes at level 4.50 with whole surface 716 m^2. They create local circulation currents, which prevent proper distribution of available air.

Other VOLV class are camera type mine workings. Their dimensions vary in ranges 150 x 10 x 7 m. Suppply/ecxaust oppenings are also variable. For this types of objects are valid following relationships [7]:

- If air supply working height equals to the chamber height, while its width is incomparable small to its width then flat free jet is formed in the camera.
- If height and width of air supply working are incomparable in dimensions with corresponding camera dimensions then round free jet or very similar to it is formed.

Initial configuration of ventilated areas can be done based on the following expressions:

- *Short premises*: Fresh air is reflects in the opposite wall. Beyond the jet zone re-circulation zone is composed, where fresh and exhaust air is mixed. Supply air is more than exhaust one. In order to ventilate one premises by directed through one supply place flow, its length should comply with the expression:

$L \leq 0,62\, k_1 \sqrt{B\,H}$, where:

k_1 – air jet disintegration coefficient,

$$\frac{u_{\max}}{u_0} = k_1 \frac{\sqrt{A_0}}{x}$$

u_{max} – maximal outflow velocity of ventilation tag, m/s;
u_o – average velocity of air jet, m/s;
Ao – area of jet inflow, m^2;
x – distance from inflow location, m

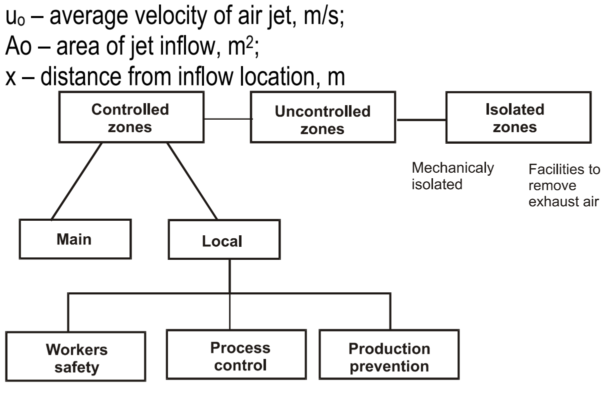

Figure 3. Zones of ventilated object

- *Long premises*: if $L > 0,62\, k_1 \sqrt{BH}$. Inflowing fresh air jet disintegrates and reversed flows become dominated. Critical length, for which jet is shortened, is defined ($x_{\max} > L_{крит}$). According to Norwegian researchers [4] $x_{\max} \approx 3{,}3\sqrt{BH}$.
- *Wide premises*: In order to ventilated one premises through fresh air, inflowing from one hole, the following expression should be valid: $B \leq 3H$. If $B > 3H$ premises is classified as long one.

MODELLING OF POLLUTANTS' LIBERATION AND DISTRIBUTION

Pollutant sources according to the place of their liberation are classified as:

- external;
- internal;
- linked with ventilation system.

Outside air is normally less polluted, compared with the exhaust air out flowing from ventilated object unless due to mistake in ventilation the exhaust air become a supply one. Sources, associated with ventilation system usually are as a result either of improper operation or bad project.

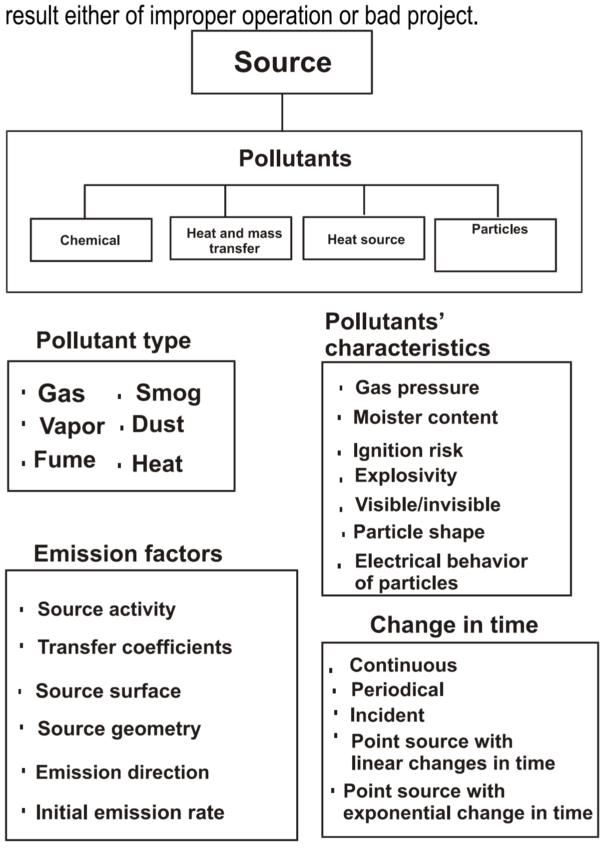

Figure 4. Source characteristics

Internal sources can be:

- chemical (particles, gas, aerosols, vapors, smog);
- biological (bacteria, viruses);
- physical – heat parameters (temperature, velocity, moisture) leading to human discomfort beyond normal parameters and causing radiation, noise, vibrations and other negative phenomena.

Figure 4 shows main parameters, defining processes of pollutants' formation, liberation and distribution. Modeling and analysis of distribution should take into account all interacting factors and to give proofs for neglection and/or inclusion into model one or other phenometa, depending on the actual problem examined. Polution type and its behavior in time are parameters, which are defined approximately well. Specific attention should be paid to emission rate. Some expressions for its evaluation are given below:

Process with clear pollutant formation parameters

$M = R_1 * T_{process}$

where:

M – total emission rate, kg/h;

R_1 –t liberated amount of gas, fume or vapor, kg/min;

$T_{process}$ – average time duration for pollutant liberation within one hour, min/h;

Pressure differences inside/outside the pipeline

$$M = k\,C\,V\sqrt{\frac{m}{T}} \tag{1}$$

where:

к – coefficient $k \in (1 \div 2)$;

C – coefficient depending on pressure, given in the table below:

Pressure, atm	<2	2	7	17	41
Coefficient C	0,121	0,166	0,182	0,189	0,192

V – inside pipeline diameter, m³;

m – molecular weight of gas/vapors

T – temperature in the pipeline, °C

Gas emissions from open spaces (reservoir, spots on the floor)

Nuselt, Prandl and Grashov criteria numbers are applied:

$$Nu = \frac{u\,d}{D}; \quad \Pr = \frac{\nu}{D}; \quad Gr = \frac{ud^3(\rho_0 - \rho_1)}{\nu^2 \rho_1}$$

where:

d – typical dimension, m;

u – liberation velocity from the taken surface,m/s;

D – diffusion coefficient, m²/s;

ν - kinematic viscosity, m²/s;

ρ_o – density of air environment, kg/m³;

ρ_1 – density of air environment, closed to evaporating surface, kg/m³;

- _Emission rate under laminar flow mode_ $(2.10^2 < Gr\Pr < 2,3.10^8)$

$$M = F.A.d^{-1/4} D^{1/2} (C_1 - C_0)^{5/4} \left|(M_{air}/M_l - 1)\right|^{1/4} \text{ g/s} \tag{2}$$

where:

$$F = \begin{cases} 0,334 \ when\ M_{air} > M_l \\ 0,184 \ when\ M_{air} < M_l \\ 0,224 \ for\ wet\ vertical\ surfaces \end{cases}$$

A – evaporating surface, m²;

M_{air} – relative molecular weight of air;

M_l - – relative molecular weight of evaporating substance

- _Emission rate under turbulent flow mode_

$$M = F.A.D^{1/3} (C_1 - C_0)^{4/3} \left|(M_{air}/M_l - 1)\right|^{1/3} \text{ , g/s} \tag{3}$$

where:

$$F = \begin{cases} 150 \ when\ M_{air} > M_l \\ 75 \ when\ M_{air} < M_l \\ 113 \ for\ wet\ vertidcal\ surfaces \end{cases}$$

Evaporation rate:

$$M = 7.4(a + 0.017V)(P_2 - P_1)101.3\frac{A}{P_B} \tag{4}$$

where: M is evaporation rate $[kg/h]$

V – air velocity across the surface, $[m/s]$

a - coefficient reflecting the influence of air movement, given in [4]

P_2 - saturated vapours pressure over the liquid, кPa

P_1 - water vapour pressure above the surface, кPa

A – evaporation surface, m^2

P_B - barometric pressure, кPa.

Gas, vapours and dust particles are distributed in air space due to convection flows of ventilation and turbulent and molecular diffusion. Main equation, which presents process of pollutant with concentration C_A distribution, is convection diffusion equation:

$$\frac{\partial C_A}{\partial t} + u_x\frac{\partial C_A}{\partial x} + u_y\frac{\partial C_A}{\partial y} + u_z\frac{\partial C_A}{\partial z} = D_{AB}\left(\frac{\partial^2 C_A}{\partial x^2} + \frac{\partial^2 C_A}{\partial y^2}\frac{\partial^2 C_A}{\partial z^2}\right) + q_c \tag{5}$$

Initial and boundary conditions are given in table 1. They should be particularly specified according to concrete problem.

One simplified expression for concentration C_x evaluation at distance x from the source:

$$C_x = C_0\, e^{-\frac{u}{D}x} \tag{6}$$

where:

Co – source concentration, mg/m³;

u – air velocity, m/s;

x – distance from source, m;

D – diffusion coefficient, m²/s.

Value of D depends on air flow velocity and on the way of it inflowing and distribution into the premises. In incoming zone (the jet has not been disintegrated) it is calculated under well known formula [Taylor, Laigna etc.]. Outside this zone the following formula should be applied [4]:

$$D = c\,\varepsilon^{\frac{1}{3}}\ell^{\frac{4}{3}} \text{ , m}^2\text{/s} \tag{7}$$

where:

c = 0,25 ± Δ - coefficient under reliability interval Δ;

$$\ell = \begin{cases} premisses\,height \\ ventilation\ instalation\ diameter \end{cases}$$

$\varepsilon = \dfrac{E_{prem}}{M\,\tau}$ kinetic energy, lost in air mass M[kg] for time τ [s].

$$E_{prem.} = \sum E_{jets} + \sum E_{conv.} + \sum E_{movingobjects}$$

$$E_{jets} = \frac{1}{2} \rho\, V u^2$$

$E_{conv} = \dfrac{g\, W_{conv.}\, H}{1{,}8\, c_p\, T_0}$, $W_{conv.}$ – heat source convection component

$$E_{movingobjects} = \frac{1}{2} kAu^2 \rho\, t$$

Table 1. Boundary conditions

Boundary condition	Flows values	Scalars
Fresh air inflow	Air volume, velocity, pressure	T(temperature), C_i (concentration of impurity i); Turbulence degree
Exhaust air	Air volume, velocity, pressure	Т(температура), C_i (концентрация на вредност i);
Large holes	$P_{\text{пълно}}$	T(temperature), C_i (concentration of impurity i); Turbulence degree of inflowing air into zone
Walls	$\vec{v} = 0$ outside heat	T (temperature),
Pollution sources	S_c – pollutant emission Heat flow	

AIR DISTRIBUTION MODELLING

In ventilation objects with great volumes in respect to air flows behaviour following types of zones exist:

- Main air flow;
- Free or semi-restricted jets, generated by air supply equipments;
- Re-circulation zonesрециркулационни зони;
- Stand still zones.

Mathematical modelling [5,3] can help in zone evaluation and definition as well as transition between modes – fully turbulent, transitional, laminar. Classical model in mechanics, presenting turbulent air flow, consists of mass and momentum conservation (Navie-Stocks):

$$\frac{\partial \rho}{\partial t} + \frac{\partial(\rho u_1)}{\partial x_1} + \frac{\partial(\rho u_2)}{\partial x_2} + \frac{\partial(\rho u_3)}{\partial x_3} = 0 \qquad (8,9)$$

$$\frac{\partial u_i}{\partial t} + \frac{\partial}{\partial x_j}(u_j u_i) = -\frac{1}{\rho}\frac{\partial P}{\partial x_i} + \frac{\partial}{\partial x_j}\left[\nu \left(\frac{\partial u_i}{\partial x_j} + \frac{\partial u_j}{\partial x_i} \right) - \overline{u_i u_j} \right] + \rho \beta \delta_{i2} \left(T - T_{ref} \right)$$

where: $\vec{u} = u(x_1, x_2, x_3; t)$

$\rho = \rho(x_1, x_2, x_3; t)$

Indexes i and j are used, instead of traditional x,y,z in order to transfer easily mathematical model into its numerical analogue.

Simplified turbulent model can be applied, namely:

$$\frac{\partial u_i}{\partial t} + \frac{\partial}{\partial x_j}(u_j u_i) = -\frac{1}{\rho}\frac{\partial P}{\partial x_i} + \frac{\partial}{\partial x_j}\left[(\nu + \nu_t) \left(\frac{\partial u_i}{\partial x_j} + \frac{\partial u_j}{\partial x_i} \right) \right] + \rho \beta \delta_{i2} \left(T - T_{ref} \right) \qquad (10)$$

Most widely applied turbulent models to solve Navie-Stocks equations are $k - \varepsilon$ and $k - \omega$. Turbulent viscosity is presented in the way:

$\mu_t = C_\mu\, \rho \dfrac{k^2}{\varepsilon}$ for $k - \varepsilon$ model and $\mu_t = C_\mu\, \rho \dfrac{k}{\omega}$ for $k - \omega$ model.

Turbulent model application is obligatory due to the need to model different turbulent modes [1,4] and also transition from one mode in another. $k - \varepsilon$ model can create numerical problems in zones with weak turbulence. The reason is that when $k \to 0$ expression $\dfrac{\varepsilon^2}{k}$ representing turbulent diffusion in ε equation tent to infinity. ε equation is:

$$\frac{\partial}{\partial x_j}(\rho u_j \varepsilon) = \frac{\partial}{\partial x_j}\left[(\mu + \frac{\mu_t}{\sigma_\varepsilon}) \left(\frac{\partial \varepsilon}{\partial x_j} \right) \right] + \frac{\varepsilon}{k}(C_{\varepsilon 1} P_k - C_{\varepsilon 2} \rho \varepsilon) \qquad (11)$$

Such kind of problem doesn't exist in $k - \omega$ model:

$$\frac{\partial}{\partial x_j}(\rho u_j \omega) = \frac{\partial}{\partial x_j}\left[(\mu + \frac{\mu_t}{\sigma_\varpi}) \left(\frac{\partial \omega}{\partial x_j} \right) \right] + \frac{\omega}{k}(C_{\varpi 1} P_k - C_{\varpi 2} \rho \omega)$$

In zones with weak turbulence diffusion term tent to 0 and no numerical problems cause. Special attention in airflows distribution in large volumes should be paid to air jets. They predefine ventilation measures effectiveness and ventilation strategy [1].

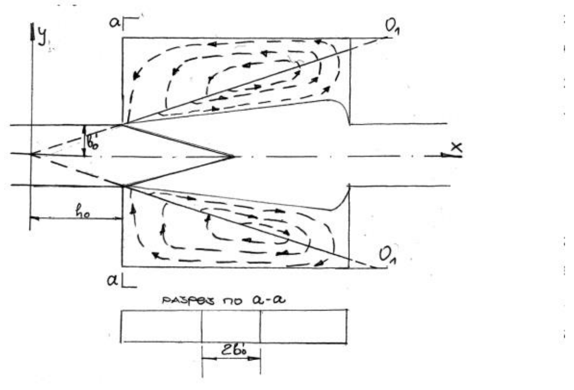

Figure 5. Free jet elements

Air jets are formed as a result of directed air inflow through ventilation devices. In case no influence of walls, ceilings and other jets exist, the jet can be considered as a free jet. Very often its identity depends on reverse flows and other currents in ventilated space with different temperature, which lead to origination and action of Archimedes forces. Free jets are classified according to ventilation outlet to: compact, linear, radial, swirling. Four zones are observed in jet development and existence (figure 5)

- Zone 1 (jet kernel) – short zone $(2 \div 6d)$. Centerline velocity remains nearly equal to supply velocity;
- Zone 2 (transition zone) – Depend on diffuser type. For a compact jets it is from 8 to 10d. Maximal velocity may vary inversely with the square root of the distance from the outlet;

- Zone 3 (fully established turbulence). Its length depends on the air jet shape, type and size of supply diffuser, initial velocity, turbulence characteristics of ambient air. It has major engineering importance since this is the place where jet enters the occupied region;
- Zone 4 (terminal/decay zone) or the place where jet looses its identity.

Expressions below can be used to evaluate jet velocity in arbitrary place in jet. Diffuser type is assumed rectangular with dimensions 2L x 2B. In point with coordinates x,y,z against jet center, velocity is defined under the following expression:

$$u(x,y,z)=\frac{u_0}{2}\sqrt{\left(erf\frac{L-y}{cx}+erf\frac{L+y}{cx}\right)\left(erf\frac{B-z}{cx}+erf\frac{B+z}{cx}\right)} \quad (12)$$

Centerline velocity (y=0, z=0) is:

$$u(x)=u_0\sqrt{erf\frac{L}{cx}+erf\frac{B}{cx}} \quad (13)$$

Above expressions are valid for zones 1 to 3.

In case diffusion type is rectangular with great length ($L\to\infty$) and width 2B (linear diffuser) velocity is evaluated under the expression:

$$u(x,y,z)=\frac{u_0}{2}\sqrt{erf\frac{B-z}{cx}+erf\frac{B+z}{cx}} \quad (14)$$

while on the centerline it is:

$$u(x)=u_0\sqrt{erf\frac{B}{cx}} \quad (15)$$

Centerline velocity for radial diffuser type is [4]:

$$\frac{u(x)}{u_0}=1-\exp\left(-\frac{d^2}{4c^2x^2}\right) \quad (16)$$

Known expressions for flows formation in large volume premises [7] are given below:

- In case Re number, related to free jet initial outlet area is in range 1900-2500, only turbulent jet is formed in the premises, initiated from outlet diffuser;
- In case Re number is less than 1900 only laminar or laminar-turbulent jets can be presented in the premises, last of them forming when Re ≅ 1800.

CONCLUSION

Air flows and pollutant distribution can be made by applying the following approaches:

- Experimental measurements and calculations;
- Mathematical and computer modeling;
- Combination of the above.

Mathematical and computer modeling include:

- Mathematical and physical model description (equations, expressions, coefficients);
- Equations' transitions into computer model;
- Numerical solution of discrete model.

Mathematical and computer modeling [3,6] shortened the path for investigation of a given problems to practical results. This approach is extremely useful in the following cases:

- Lack of data and possibility for real measurements;
- Complex problem – in our case different in number and structure interrelated zones;
- Known initial and boundary conditions – supply and exhaust air volumes;
- Need to optimize ventilation parameters in order to choose more effective scheme for ventilation.

Additional advantage of simulation approaches is that it can give hints where and what to measure so as to obtain close to reality model thus leading to realistic models' results.

Mathematical models (general and simplified), presented in this paper will serve as a basis for velocity field evaluation and furthermore – to important fro ventilation point of view zones in ventilated premises.

More accurately definition of models' parameters, experiments on models' adequacy, applicable mathematical models to transform into discrete analogue, the results themselves are the object of following works. .

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Recommended for publication by Department of Mine ventilation and occupational safety, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, Vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.147-152

FORCING OF NATURAL VENTILATION IN ELECTROLYTIC SHOPS

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ABSTRACT
The paper presents an idea for improving ventilation of big industrial units containing a large number of surface sources of acid vapors. Analysis has been made of incoming air quantities for shop ventilation, location of ventilation points, and distribution of acid vapor cresset. Parameters are explained of existing ventilation system with focus on its problems. For proper direction of contaminated air, it is appropriate to use motor-less, wind-driven roof fans, which are new in world practice. They facilitate natural convection by creating additional draught to direct contaminated air outside the shop and far from working spaces. Parameters of these fans are show and their possible applications. Comparative analysis of their efficiency is presented.

INTRODUCTION

Electrolytic shops of metallurgical works contain a number of acid vapor sources area-distributed and of significant density. These create hazards in the shop and in the working space and give rise to specific ways of their safe extraction. Because of the large emitting surface area, use of local ventilation systems presents technological difficulties for normal operation of electrolytic baths. Local exhaust ventilation adversely impacts temperature regime of electrolytic baths, particularly in cold weather. Vapor acidity requires extraction along paths that do not cross the working space or allow condensation. Such suitable passageway can be ensured along the paths of natural convective flows. However, natural ventilation depends both on specific parameters of sources and premises and of building location and weather conditions, thus not every combination of those is efficient enough.

The paper presents analysis of natural ventilation parameters at a specific site with active sources of heat and contamination, ventilation openings, convective flows and their distribution. The analysis shows that in warm weather natural convective ventilation is not efficient enough and requires significant enhancement.

One possibility of enhancement of natural ventilation whilst preserving heat flows' direction is found in the suction of air contaminated with acid vapors by means of wind-driven roof fans. Basic parameters of such facilities are presented, and their effective application field. Their efficiency is shown in directed extraction of air contaminated with sulfur acid vapors. Simultaneously, wind ventilation improves microclimatic conditions in the shop whilst avoiding additional power consumption and new sources of noise.

PROBLEM DESCRIPTION

Investigation of ventilation conditions is based on the following parameters:

- Production process analysis from the point of view of emission mechanism, propagation and carrying out of harmful substances;
- Analysis of shop geometric dimensions and of ways forf fresh air intake and contaminated air extraction;
- Climatic conditions in shop location – temperatures, prevailing directions and magnitudes of air flows.

The paper presents the problems of natural ventilation and the ideas for its improvement on the example of a shop for electrolytic refining of anode copper to a purity of 99, 99%. Daily metal production is approximately 100 t with planned increase of 30%. One technological cycle lasts 21 days – removing of electrodes, draining of electrolyte, cleaning of baths and another loading.

Refining process is accompanied by emissions of vapor and gas – 2 051 m^3/hour. Total evaporation area of baths is 1536 m^2 and this will be increased by approximately 32% on reaching design capacity. Peak values of air contaminants in the shop are, as follows:

- Moisture content – 22 g/kg;
- Sulfur acid vapors – 5 mg/ m^3;
- Dust – 13 mg/ m^3.

In winter, sulfur acid vapor concentration exceeds by 1.2 to 1.8 times the permissible Exposure Level (PEL) of 1 mg/m^3 during the working shift. Average shift excess value for January was 1.58 ± 0.17. In June excess over MAV during the shift at various working locations was 2.2 to 3.2 times PEL. Average shift excess is 2.7 ± 0.27 times MAV. In could be expected that during the hottest days of July and August MAV will be exceeded move than 4 times.

The shop has a typical structure – main hall and roof glass cubicle with windows located along the building length. Dimensions are, as follows:

- Length 180 m;
- Width 45 m;
- height – 20.95 m (15.50 to roof and 17.30 to cubicle).

Shop volume is 137 214 m^3, which makes it a large-volume ventilation site. The openings that can be used for fresh air intake and for contaminated air extraction are located on levels 0.00, 4.50, 10.30 and 17.30. Their areas (not all of them can be opened) are, as follows:

- Doors on level 0.00 – 45.8 m^2;
- Windows on level 0.00 – 315 m^2;
- Windows on level 4.50 – 540 m^2;
- Windows on level 10.30 – 360 m^2;
- Cubicle windows on level 17.30 – 700 m^2.

A fact adversely impacting natural ventilation is the presence of technological openings in the floor on level 4.50 of total area 716 m^2.

INVESTIGATION OF NATURAL VENTILATION IN THE SHOP

Building natural ventilation parameters were investigated by series of measurements under the following three scenarios:

- Scenario 1 – current state of the building;
- Scenario 2 – partially closed windows on level +4.50;
- Scenario 3 – partially closed doors on level +0.00

Incoming fresh air quantities and extraction of contaminated air quantities for the three scenarios are presented on figures 1a, 1b and 1c, respectively.

During measurement, no changes were made in the state of active mechanical draught sources. The difference between incoming and outgoing flows on level +20.00, shown on fig. 1a, b, is due to:

- Operation of two fans in the aspiration system;
- Vapors inflow, which represents internal flow source;
- Recirculation contours between levels.

Existing situation (Scenario 1) shows that the main fresh air source is terrain level, or level +0.00. This means that, as at any ventilation entry, harmful emissions should be reduced to the technical minimum, and where this is not possible, local ventilation should be installed. Presently on level +0.00, there are zones with excessively high content of harmful emissions, mostly around electrolyte and water tanks. Air exchange in the zone from level +0.00 to level+4.50, with existing ventilation mode involving door opening on level 0.00, ensures big ventilation flow through the building. However, this flow is not properly distributed by contamination source because of structural and technological openings in the floor on level +4.50 with total area exceeding 700 м^2. Location of floor openings on level +4.50 opposite northern and western doors on level +0.00 causes air flow to pass along the contours "**ground atmosphere – door on level +0.00 – floor opening on level +4.50- vertical jet – window openings in roof cubicle – atmosphere around level +20.00**." Airflows through these contours do not ventilate contamination zone where people work. As a result of improper air distribution, huge areas wherein people move and work (between levels +4.50 and +8.40) remain insufficiently ventilated with contaminant concentrations exceeding MAV.

Changes in ventilation conditions in Scenario 2 include the following: door opening on terrain level; window closing on level +4.50. The result is increased temperature around baths with adverse effect on vapor emissions from electrolytic baths due to inhibited heat evacuation.

The essence of the changes in Scenario 3 is the regulated air intake on terrain level and increased flow from level +4.50 in intensive vapor emission zones. This scenario shows the best results for re-distribution of ventilation flows. The positive result will be enhanced further if additional draught is provided to direct flows to roof outlets. Authors suggest roof wind fans, discussed below, to serve as such additional draught.

Choice of additional means for regulation airflows in the shop should also be based on another two factors whereon intake air quantity and quality depend, namely:

- Wind speed and direction;
- Evaporation rate and expansion of evaporation cresset.

During the three measurement cycles, wind speed on the shop roof varied from 1.9 m/s to 2.6 m/s. Data submitted by nearby weather station were used to develop the wind rose (figure 2). Northeast mountain winds are the most frequent, followed by west winds. This conclusion should be taken into account in location of intake and outlet openings in the building. South and southeast winds are with the smallest frequencies and speeds. West winds have the highest speed.

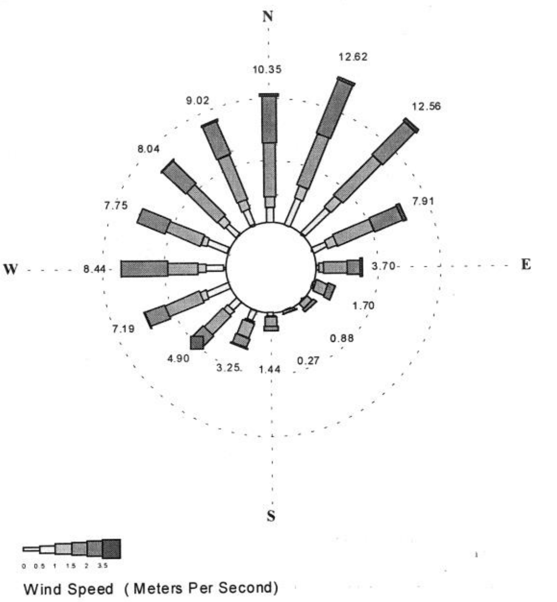

Figure 2. Wind Rose

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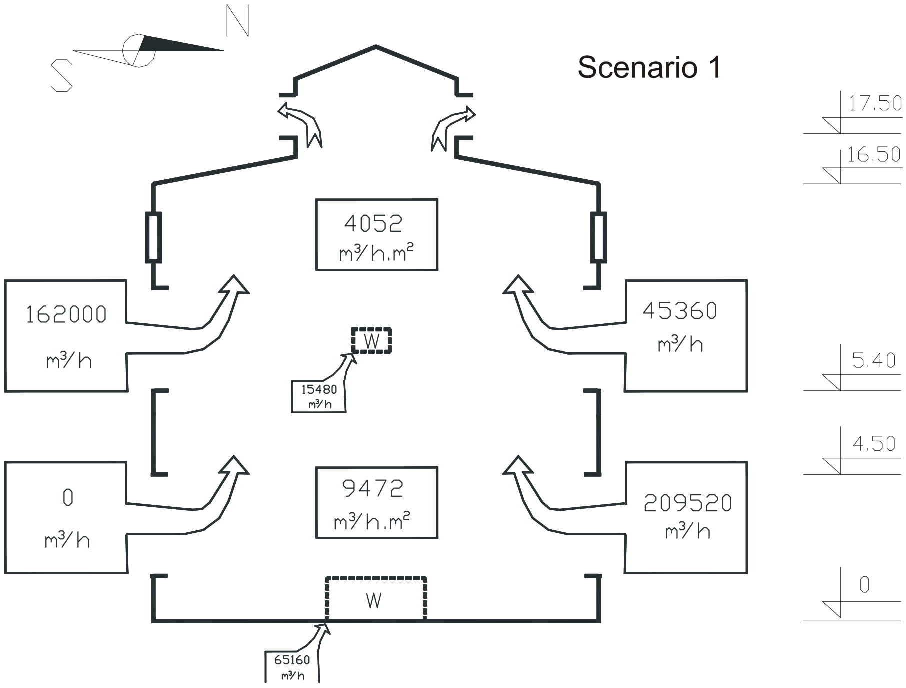


Figure 1a.

Figure 1b.

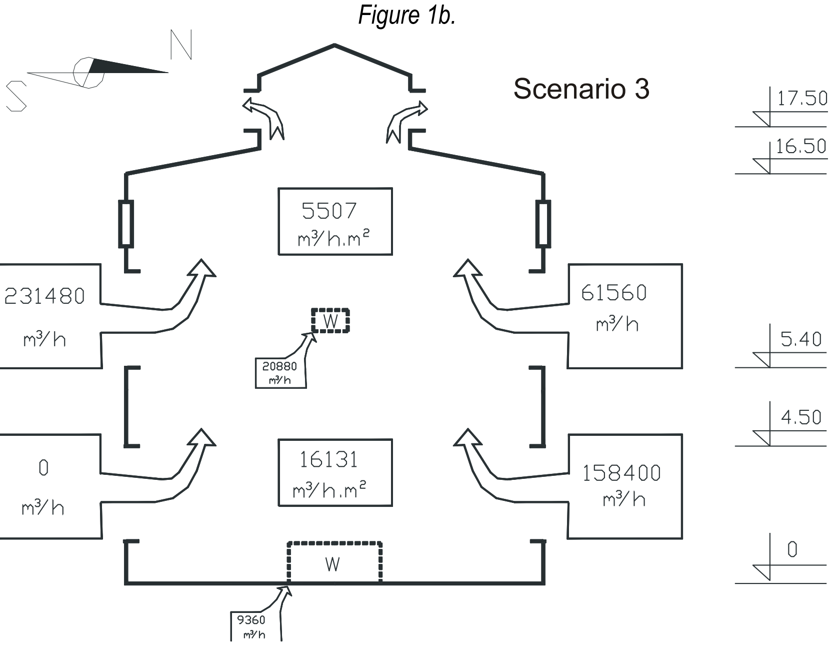


Figure 1c.

Evaporation rate can be calculated, according to [1], by the equation:

$$M = 7.4\left(a + 0.017V\right)\left(P_2 - P_1\right)101.3\frac{A}{P_B} \qquad (1)$$

where:M is evaporation rate $[kg/h]$

V – air velocity component perpendicular to evaporation, $[m/s]$

a - air mobility factor. Calculated with the help of equations shown in [1]

P_2 - pressure of saturated water vapors over fluid , кPa

P_1 - water vapor pressure over fluid, кPa

A – open surface area of fluid , m^2

P_B - barometric pressure, кPa.

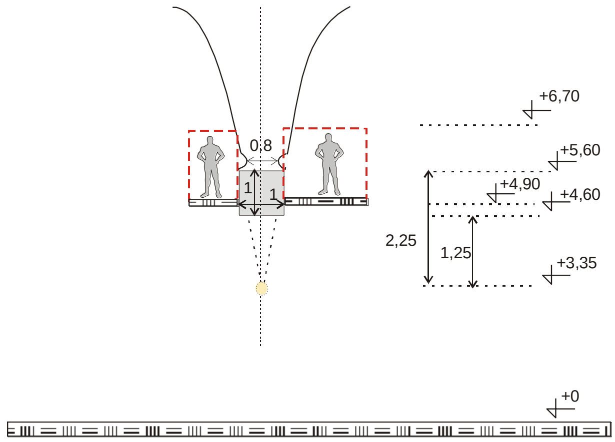


Figure 3a. Evaporation cresset at the short side of the bath

Results obtained with formula (1) and with measurement of evaporation in real conditions were quite consistent (difference 5%).

Evaporation cresset is important for the general picture of people's movement and work. Modeling was carried out using the methodology published in [1]. Worker's position at the narrow side of a single bath is shown on fig. 2a, and the position at the long bath side – on fig 3b.

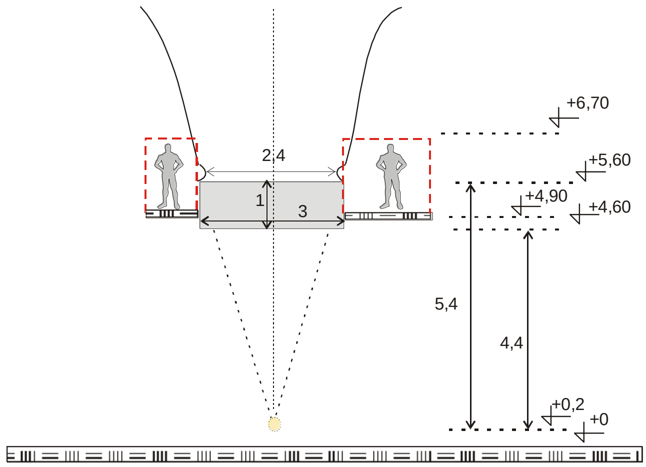


Figure 3b. Evaporation cresset at the long side of the bath

The extreme closeness of vapor cresset to the movement and work zone is clearly seen. The greater density of sulfur acid vapors and the fast cooling of in-cresset flow cause heavier vapors to flow into the work area. This picture was confirmed by smoke-tracing of flow. In order to prevent contamination of the work zone, vertical speed component should be increased in this zone or local exhaust devices should be used.

WIND DRIVEN FANS

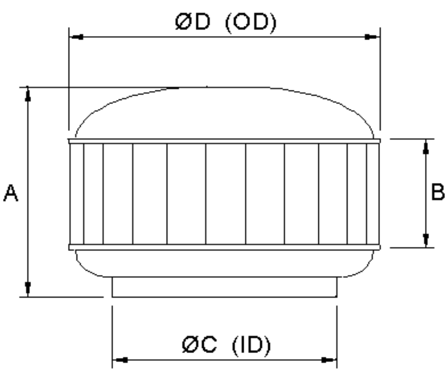
Wind fans are turbine, motor-less devices driven by wind energy. They are mounted on the roof of the building to be ventilated. The appearance of such fan is presented on figure 4, its structure – on figure 5. The fan comprises:

- turbine;
- throat ;
- a varipitch base that suits all roof slopes,
- vertical vanes for improved torque at low wind speed,
- double row ball bearing system,
- cord, remote control, manual, or electric dampers.

Figure 4. Outside view k of turbine fan

Operation is simple and sure. When the slightest breeze touches the scientific blade construction it causes the turbine to rotate. The centrifugal force caused by the revolving turbine creates a partial vacuum within the turbine. This vacuum is then replaced by a strong upward draft of air. A powerful exhaust is thus achieved. The fan works automatically, continuously and silently without operating or maintenance costs. Ball bearings are made from acetal polymer with caged 316 stainless steel precision ground ball bearings. Acetal is unique polymer that retains the lubricant within its molecular structure to ensure long life and smooth operation. Major manufacturers of such fans are shown in table 1.

Fabrication materials depend on intended application of the fan. Most often these are corrosion-proof aluminum, stainless steel or with the TGIC polyester powder coating of O'Brien Powder Products Inc., the latter, during tests, has shown resistance of 60% to sulfur acid and other harmful substances.



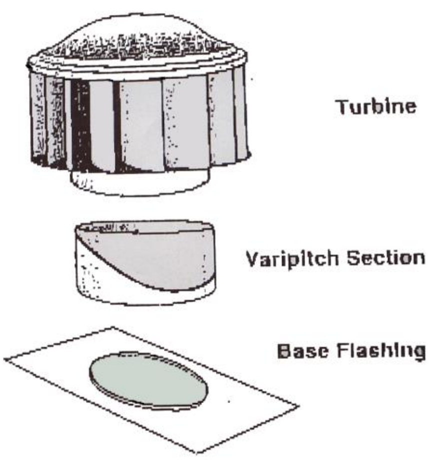


Figure 5. Structure of turbine fan

Table 1 . Major manufacturers of wind driven fans

Manufacturer Country	Fan type	Wind speed [m/s]	Fan capacity [m^3/s]
Edmonds Pty Ltd Australia [2]	H600	1.67 ÷3.33	0.6 ÷1.5
	H700	1.67 ÷3.33	0.85 ÷1.9
	H800	1.67 ÷3.33	1.1 ÷2.7
	H900	1.67 ÷3.33	1.55 ÷3.4
Metallic Products Corporation [3] USA	T500	1.8	0.8
	T600	1.8	1.1
	T900	1.8	2.1
Chin Foh Berhad Malaysia [4]	BS460	1.67 ÷3.33	0.4 ÷1.0
	BS600	1.67 ÷3.33	0.55 ÷1.35
McMaster-Carr USA [5]	1992K54	1.67 ÷3.33	0.4 ÷1.0
	1992K55	1.67 ÷3.33	0.6 ÷1.5
	1992K56	1.67 ÷3.33	1.05 ÷2.4
Ampelite Fiberglass Pty Ltd Australia [6]	AS450	1.67 ÷3.33	0.38 ÷1.0

Digits in fan names denote opening dimension - for instance H800 means Hurricane model with opening of 800 mm. McMaster Carr fans have dimensions from 500 to 762 mm. The height of fans in table 1 varies from 484to 990 mm.

Wind roof fans are very light (their weight varies from 9 to 27 kg), bur they are capable of withstanding very strong winds – of some 240 km/h. They are increasingly employed throughout the world in residential and public buildings and in industrial productions with high temperatures and contaminants. Many schools and administrative buildings in New Zealand and Australia use this type of ventilation for air conditioning. In summer the fans exhaust the hot air, and in winter they prevent condensation of hot vapors in the under-roof area. Figure 6 shows actual building equipped with wind fans.

COMPARATIVE RESULTS FROM WIND FAN OPERATION

As noted in the natural ventilation analysis and as seen from figures 1 a, b, c, ventilation and climatic conditions can be improved by increasing outlet open cross sectional area above level +15.50. This can not be accomplished solely by adjustment of cubicle openings as additional power is required to direct airflows. Such additional power should be consistent with working conditions in the shop – release of hot acid vapors, which involve specialized requirements in respect of equipment.. Therefore, the authors have chosen roof fans resistant to exhausted contaminants.

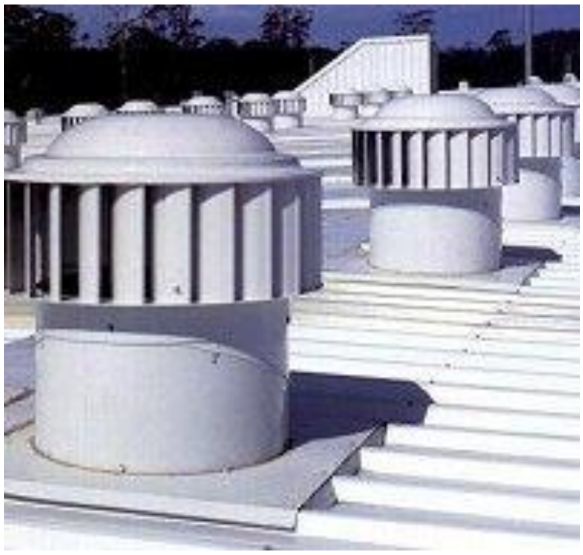


Figure 6. Actual building with roof fans

Table 2. Comparative data on additional capacity

Fan	Wind speed, m/s	Exhaust capacity (1 fan), m^3/s	Exhaust capacity of 16 fans	Air exchange divisibility , h-1
H800	1.67	1.1	17.6	0.46
	2.22	1.4	22.4	0.59
	2.78	1.8	28.8	0.76
	3.33	2.1	33.6	0.88
1992K56	1.67	1.05	16.8	0.44
	2.22	1.3	20.8	0.55
	2.78	1.6	25.6	0.67
	3.33	1.9	30.4	0.8

Table 2 shows summary information on two proposed fans - H800 fan of Edmonds Pty Ltd and 1992K56 fan of McMaster Carr. The additional flow that will be provided by the operation of 16 fans is within the range of 16 to 34 m^3/s. This means compensation of the difference between incoming and outgoing flows, with the additional capacity on level +17.30 intensifying airflows and increasing contaminant exhaust rate. This is very important in summer because temperature and air mobility are outside the admissible range for the working environment.

CONCLUSION

The above-discussed idea of forcing natural ventilation present a comparatively new approach in the world practice, the application whereof is increasing due to the following obvious advantages:

- no additional power consumption;

- can be used in various buildings – residential, public, industrial;
- can be employed in conditions of acid vapors, high temperature of exhaust gases and other contaminants;
- renewable environmentally – friendly power source;
- reliable facility with low operation costs.

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Recommended for publication by Department of
Mine ventilation and occupational safety, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.153-154

ABOUT THE GENERALISATION POSSIBILITIES IN A NEW STOCHSTICAL MODEL OF MINING SUBSIDENCE

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ABSTRACT
This paper presents the mining subsidence trough from the viewpoint of the stochastic processes. Taking into account the laws of the chance processes and the elastic properties of the particles of the loose media a generalized nonlinear equation for calculating the mine subsidence trough is obtained. The new obtained equation generalizes the existing stochastic linear and nonlinear models in geomechanics and may be used as an universal basis for deeper understanding and analyzing the subsidence phenomenon.

When creating the stochastic geomechanical theory of mining subsidence J. Litwiniszyn (1956) considered the many time underworked rock mass as an accumulation of loose stone blocks (Fig.1). This system of bodies has such a degree of freedom that is seems logical to apply statistical methods in calculating its displacements.

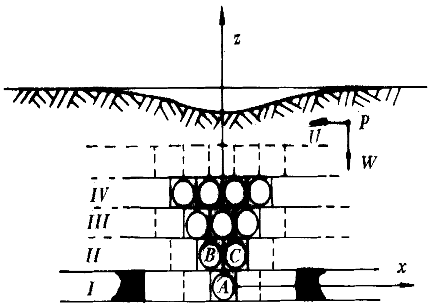

Figure 1

In this work we study the possibility to use a generalized nonlinear model of stochastic medium with application in the mining subsidence.

For calculating the displacements in rock mass and on the earth's surface, caused by underground mining works, let us assume that the following presumptions are fulfilled:

- the displacements of the rock mass are regarded from the view point of the stochastic processes;
- the geomaterial is stochastic medium built from elastic particles;
- the characteristics of the rock mass allow to use the model of the new stochastic medium, presented from M. Vulkov (1997).

In order to sketch this consideration let us assume a plane rhombic packing of particles in the Cartesian system of coordinates (x, z), as in Fig.2.

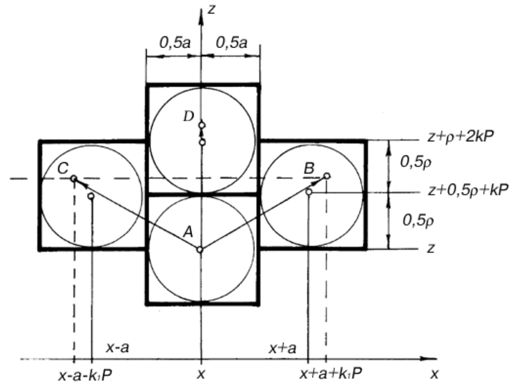

Figure 2

If the particle in field A is removed, the vacant space can only be filled by a particle from the neighboring higher fields B, C or D.

When an elastic particle migrates from higher to lower level of the loose media, this causes appearance of elastic displacement changes, as a result of the external load changes.

Let P(x, z) be the probability of a void appearance in a cage with a center-point coordinates (x, z). Following the assumption of J. Litwiniszyn (1974) for correspondence of that probability with the vertical displacements of the rock mass points for the stochastic medium presented above the following relationship could be written:

$$w(x,, z) = w_1 + P(x, z)w_2 \qquad (1),$$

where $\boldsymbol{w_1}$ is the classical stochastic displacement, by determining of which the rock mass particles are assumed for solid bodies after Litwiniszyn (1974);

$\boldsymbol{w_2}$ is the increase of the elastic vertical displacement, caused by the change of places among particles in different levels i.e. by the change of the external load. Its value is proportional to the probability for removing at lower horizon.

Let p, q, and c be the chances of a void migration from cage A to B, C or D respectively. Then the following equality exists:

$$p + q + c = 1 \qquad (2),$$

i.e. if a particle in field A is removed, the vacant space can only be filled by a particle from cages B, C or D.

In accordance with the mechanism of chance processes the following equation may be written:

$$P(x, z) = pP(x - a - k_1P, z + 0{,}5\rho + kP) + qP(x + a + k_1P, z + 0{,}5\rho + kP) + cP(x, z + \rho + 2kP). \qquad (3),$$

where $\boldsymbol{k_1 = \Delta u (x,, z)}$ is the change of the horizontal elastic relative displacement of the particle's center-point (Fig.2); $\boldsymbol{k = \Delta w (x, z)}$ is the increase of the vertical elastic relative displacement of the particle's center-point.

The relationship (3) is the starting point for obtaining the generalized non linear equation for mining subsidence determining. After developing the right-side-terms of equation (3) in Tailor's series for x, z – point area, after substituting the obtained expressions in relationship (3), following M. Vulkov (2001) and after limiting process $\boldsymbol{a \to 0, \rho \to 0}$ by corresponding choice of the measurements, the following equation is constructed:

$$[A_{11}(P)P_x]_x + 2[A_{12}(P)P_x]_z + [A_{22}(P)P_z]_z + B_1(P)P_x + B_2(P)P_z = 0 \qquad (4),$$

where $P_x = \partial P/\partial x$, $P_z = \partial P/\partial z$,; $A_{11}(P)$, $A_{12}(P)$, $A_{22}(P)$, $B_1(P)$ and $B_2(P)$ are rock mass characteristics.

By obtaining the equation (4) we take into consideration the fact that when passing to the limit expressions of higher order disappear and is assumed that appropriate limits such as by M. Vulkov (2001) exist.

The newly constructed equation (4) is a generalization of the classical linear models of J. Litwiniszyn (1956, 1974, 1974) and of the M. Vulkov's (1988, 1997)models.

So by $A_{12}(P) = A_{22}(P) = B_1(P) = 0$ from (4) follows the nonlinear parabolic equation

$$[A_{11}(P)P_x]_x + B_2(P)P_z = 0 \qquad (5),$$

whose interesting solutions and their applications for mining subsidence engineering were presented from M. Vulkov (1989).

The newly obtained generalized nonlinear equation (4) may be used as a universal basis for the interpretation of the mining subsidence in different conditions and by different kinds of rock mass characteristics.

After A Tichonov (1972), it can be shown that the equation (4) can change its type from hyperbolic through parabolic into elliptic one.

In conclusion it can be said that the generalized nonlinear equation (4) obtained in this paper, offers interesting possibilities for formulating and solving strata movement problems caused by underground mining works. The last equation certainly enables us to complete the visions about the stochastic theory in mining subsidence and to enrich the models and methods for strata movement calculations.

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Recommended for publication by Department of Theoretical and applied mechanics, Faculty of Mining Technology

50 years **University of Mining and Geology "St. Ivan Rilski"**
Annual, Vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.155-162

UNTIGHT PIPELINE VENTILATION SYSTEMS' CALCULATIONS

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ABSTRACT
A review of pressure and air volumes in un-tight pipelines predictive methods has been done. Two main approaches, utilized to work out distribution functions for hx and Qx, are compared. Reasons have been given for virtual substitution of integral model with algebraic system of equations for recurrent calculations. Paper present so called passports *Rx-Px* for typical characteristics of pipelines. Such approach can avoid repeatable prediction of hx and Qx under different boundary conditions in calculation sectors. Pressure and air volumes distribution is defined under graphical and polynomial approximation, given in the passports. A comprehensive algorithm for the method, worked out by authors, has been presented. Its application is demonstrated on real examples.

INTRODUCTION

Fan operation in tight pipelines is described by exact mathematical model. The design of such pipeline systems is a routine work. In un-tight pipelines mathematical description is harden due to existence of air leakages towards and from pipes thus forming complex network from mutually interacted transit and filtration flows.

Mine ventilation pipelines are normally un-tight for heavy natural and technological conditions in the process of their construction and maintenance. Two physical models are utilized to describe their un-tightness:

- **Network** (fig. 1) – fixed along the pipeline un-tightness (flanges or other conjunctions);
- **Continuous** (fig. 2) – randomly distributed outlets along the pipeline walls.

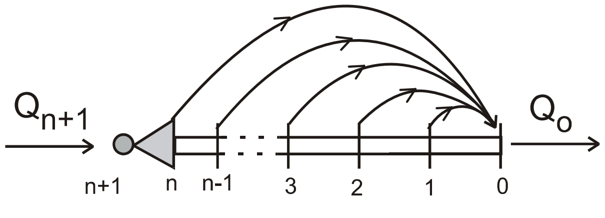
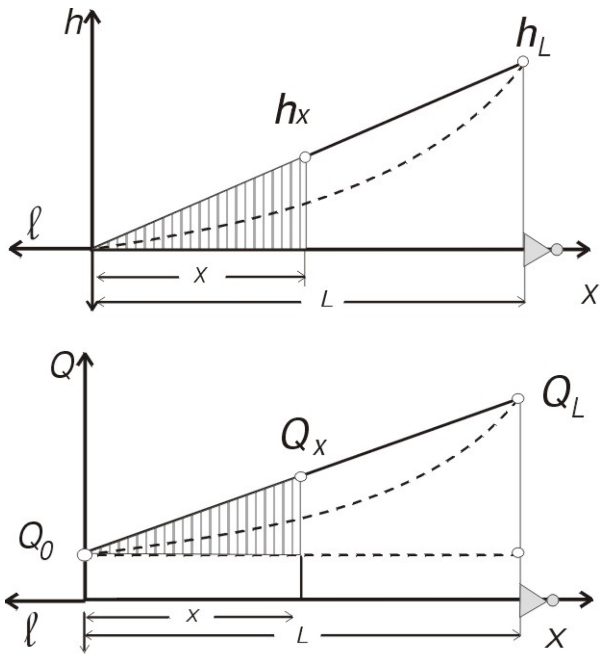


Figure 1. Network model of pipeline

Air distribution in such pipelines is described by mass and energy conservation equations, which are solved under following assumptions:

- Turbulent flow mode;
- Non changeable air density;
- Momentum conservation is not affected by filtration;
- Local resistances are taken into account by increased value of friction factor;
- Air resistance of main flow is neglected or added to pipeline friction factor.



.Figure *2. Q_x, h_x distribution functions*

Conservation equations under above written assumptions are as follows:

1. **network distribution** (fig.1):

$$\begin{cases} Q_i = Q_{i-1} + k_f \sqrt{h_{i-1}} \\ h_i = h_{i-1} + r\,\ell_f\,Q_i^2 \end{cases} \tag{1}$$

where : $i = 1,2,3,...,n; n = L / l_f$ is junction number and its adjoining out flowing branch; l_f - distance between flanges, m; L – pipeline length, m.

Quantities Q_i and h_i represent air volumes (m3/s) and pressure (N/m2) distribution in separate junctions, where $Q_1 = Q_0;\ h_L = 0;\ Q_{n+1} = Q_L; h_n = h_L$.

2. **continuous distribution** (fig. 2):

$$\begin{cases} \dfrac{dQ_x}{dx} = -k_x \sqrt{h_x} & (2a) \\ \dfrac{dh_x}{dx} = -r\,Q_x^2 & (2b) \end{cases} \qquad (2)$$

where Q_x and h_x are continuous functions along the transit flow direction x in boundaries:

$$Q_L \geq Q_x \geq Q_0; h_L \geq h_x \geq 0; 0 \leq x \leq L \ .$$

Functions Q_x and h_x are transformed for convenience into Q_ℓ and h_ℓ along reciprocate axe ℓ (fig.2), where: $x = L - \ell; Q_L \geq Q_\ell \geq Q_0; h_L \geq h_\ell \geq 0.$

Distribution functions, obtained from solutions of either (1) or from (2) can be applied in the following engineering calculations:

- Fan selection under given Q_O;
- Evaluation of initial air quantity Q_O under given fan;
- Location of two and more fans with given characteristics $h_{Fi}(Q_{Fi})$ evaluation.

These problems motivated continuous interest in solution of models (1) or (2) and distribution functions evaluation.

DISTRIBUTION FUNCTIONS

Recurrent formulae (1) define exact numerical values of air volumes and pressure (Q_i, h_i) in the simplified model (fig. 1) under following input data - L, r, l_f, k_f и Q_o. Such approach is applied more than hundred years to solve simple parallel networks. Stefanov T.P., V.V. Tomov, I.S. Velchev (1975) utilized iteration solution under H.Kross method, presenting local ventilation system as a complex diagonal network with ventilation tubes (transit flows), diagonal branches, arbitrary number and place of fans, different resistance factor of branches etc.

Model (2) utilization in mine ventilation is initiated in the works of Loisson R. и J.Ulmo, (1950); Holdsworth J.E., M.A. Pritchard и W.N.Walton, (1951); Воронин В.Н., (1956). During the second half of 20th century series of new solutions are published: Simode E., (1976) ; Pawinski J., J. Roszkowski и J.Strzeminski (1979); Robertson R. и P.B.Wharton (1980), Browning E.J., (1983); Vutukuri V.S. (1983), Kertikov (1994). This is a stage of analytical treatment of the model and trials for engineering expressions deduction. Different simplifications are applied in the process of solution of (2), leading afterwards to different Q_L and h_L, in some cases reaching serious deviations from real values. Satisfying results give an integral solution (3), obtained under $h_x(m)$ approximation:

$$Q_L = Q_0 + \frac{2k_x L}{m+2}\sqrt{h_L(m)} \qquad (3)$$

Voronin's (1956) formulae are very useful and give good agreement with reality for *(L<750 m)*:

$$h_L = r\,L\,Q_0\,Q_L \ ; \ P_L = \frac{Q_L}{Q_0} = \left(\frac{k_x}{3}\sqrt{r\,L^3} + 1\right)^2 \qquad (4)$$

Similar is Browning's E.J. (1983) approach.

Authors of this paper apply sequent iterative integration of (2) to required accuracy via polynomial approximation (Vlasseva 2001). First approximation for h_x is: $h_x^I = r\,L\,Q_0^2\,\dfrac{\ell}{L}$. This expression is substituted into (2a), which solution is first approximation for Q_x^I :

$$\int_{Q_0}^{Q_x^I} dQ_x = -k\int_0^x \sqrt{h_x^I} = Q_x^I$$

This is a starting point for polynomial approximation, developed by the authors of this paper, namely: Q_x^I is approximated by 3-rd degree polynomial (functions Q_x and h_x analysis show that such degree polynomial describes very well their behavior):

$$Q_x^I \approx P_3(x) = a_0 + a_1x + a_2x^2 + a_3x^3$$

This approximation is then substituted into (2b), giving:

$$\frac{dh_x}{dx} = -\,r\left(a_0 + a_1x + a_2x^2 + a_3x^3\right)^2$$

Above written equation is the second iteration of h_x^{II} . Numerical values of $\sqrt{h_x^{II}}$ are approximated by 3-rd degree polynomial, which polynomial is substituted into (2a): $\dfrac{dQ_x}{dx} = -\,k\left(b_0 + b_1x + b_2x^2 + b_3x^3\right)$. Solution os this equation is second iteration for Q_x^{II} . Iteration procedure continue till preliminary given accuracy is reached, namely, 10^{-2} in regard for Q.

Above described algorithm is transferred into computer code giving distribution functions h_x and Q_x under given input data – diameters, filtration factor, required air quantity.

As a result of above described procedure continuous functions for pressure h_x and air volumes Q_x are obtained. In this way the authors have achieved continuation of R.Loisson и J.Ulmo's (1950) solution, made analytically till the second iteration and is similar to the approach, applied by Vutukuri V.S. (1983) and Gillies, 1999

Insufficient information from empirical tests of solution to the model (2) lead to results comparison with proven calculation methods.

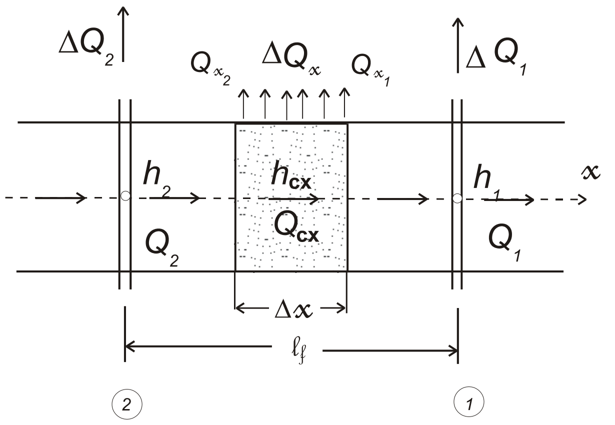


Figure 3.Filtration outflows

MODELS' COMPARISON

Difference between models (1) and (2) comes from filtration inflows determination (fig. 3). In model (1) each outflow is calculated separately by its friction factor $R_f = 1/k_f^2$, kg/m[7] on the basis of averaged friction factor of holes (pores), located along the pipe wall surface $\pi \ell_p d$, namely: $R_p = 1/(\ell_p k_x)^2$, kg/m[7]. Thus dimension of the two filtration factors is determined: concentrated k_f, $(m^7/kg)^{1/2}$ and distributed along surface k_x, $(m^5/kg)^{1/2}$. According to (1) and (2) these fators are defined (fig. 3) as ratio between filtration flow (*ΔQ*) from the hole or from filtration surface to pressure ($\sqrt{h}$) in the pipeline:

$$k_f = \frac{Q_2 - Q_1}{\sqrt{h_1}} = \frac{\Delta Q_1}{\sqrt{h_1}}; \quad \ell_p k_x = \frac{Q_{x2} - Q_{x1}}{\sqrt{h_{cx}}} = \frac{\Delta Q_x}{\sqrt{h_{cx}}} \quad (5)$$

where: $h_1 = h_f = R_f \Delta Q_1^2$ and $h_{cx} = R_p \Delta Q_x^2$, while h_{cx} and Q_{cx} are integral expressions under (2) for the sector $\ell_p = \Delta x$ (fig. 3).

Expressions (5) give ground to draw the following conclusions for pipelines with equal diameters (d_1=d_2) and with concentrated and distributed surface filtration:

1. when $\ell_f \to \ell_p$ and $k_f \to \ell_p k_x$, functions h_i, Q_i are close to h_x, Q_x;
2. best approximation is achieved when $\ell_f = \ell p$
3. full coincidence of distribution functions, obtained by the two models could not be achieved due to initial differences in the models – in tight (ℓ_f) and un-tight (ℓ_p) sector, when $\Delta Q_i < \Delta Q_x \ and \ \Delta h_i > \Delta h_{ni}$;
4. the above unevens become negligible when $\ell_f = \ell p = \Delta x < 10m$.

Convergence criteria from model (1) to (2) can be presented in the following way:

$$\ell_V = \ell_p \ u \ k_v = \ell_v k_x \quad (6)$$

where distance ℓ_v and factor k_v are virtual values of $\ell_f \ and \ k_f$. Under these conditions recurrent calculations made by (1) can be taken into engineering calculations as equivalent to the integral model (2).

Vutukuri V.S. (1983) transforms model (2) into (1) by splitting of air flow in given sector ($\ell_p = \Delta x$) into two parallel flows – transit in tight pipe $\left(h = r\ell_p Q_d^2\right)$ and filtration – conditional pore with resistance $R_p = 1/\left(\ell_p k_x\right)^2$, equivalent to filtration resistance along sector walls $\left(h = R_p Q_p^2\right)$. Vutukuri A.S. solves the problem by introduction of R_p into (2), eliminates Q_x and integrates thus achieved second order differential equation by approximation proposed by Holdworth et al. (1951):

$$\left(h_x'\right)^{1/2}\left(h_x''\right)^2 = \left(\frac{r}{R_p}\right)^{1/2} 2\,h_x'\left(h_x\right)^{1/2}$$

Table 1 presents comparative solutions, obtained by Vutukuri A.S. transformated model (2), Vlasseva E. – polynomial iteration approxiмation of (2) and virtual model (1). The problem example №3 in Vutukuri A.S. (1983) with the following input data: *d=1,00m, r=0,02464kg/m[8]; k_x=0,00005*$(m^5/kg)^{1/2}$*, Q_o=10,00m[3]/s, l_v=20m, L=2000m*.

Table 1. Comparative results

L	Vutukuri 1983		Vlasseva 2001		Virtual model	
100	--	--	10.06	247.103	10.06	247.35
500	10.48	1258.35	10.58	1291.71	10.61	1290.75
1000	11.64	2785.29	11.71	2813.05	11.74	2813.87
1500	13.30	4765.39	13.23	4722.21	13.27	4722.10
1840	14.59	6416.90	14.49	6331.12	14.53	6328.30
2000	--	--	15.15	7195.41	15.19	7104.30

Factors $k_x \ and \ k_f$ are of great importance to the results obtained under (1) or (2). Their real values can be experimentally proved by measurements of *ΔQ* and *Δh* in control pipelines sectors as shown in Simode E., 1976; Selezniov A.C., 1992; Gillies A.D.S., H.W.Wu, 1992.

Pipeline junction tightness (k_f) is defined by laboratory modeling and similarity.

Factor k_x results from un-tightness, locate along the whole pipeline length L. Its average value should reflect measured values ΔQ and Δh in comparatively long sector $\Delta L = L_1 - L_2$. When utilizing same technology for tightness achievement and developed turbulence in it, location and length of control section (fig. 3) are chosen based only on accuracy of measurements and representatives of averization. Formulae for k_x evaluation is obtained on the basis of Q_1-Q_2 under expression (3):

$$k_x = \frac{Q_2 - Q_1}{2\left(\frac{L_2}{m_2+1}\sqrt{h_2} - \frac{L_1}{m_1+1}\sqrt{h_1}\right)} \qquad \left[\frac{m^3/s}{m\sqrt{N/m^2}}\right] \qquad (7)$$

or via integration in boundaries ΔL (fig. 3) when m=1 (linear approximation):

$$k_x = \frac{3(Q_2 - Q_1)(h_2 - h_1)}{2\,\Delta L\left(\sqrt{h_2^3} - \sqrt{h_1^3}\right)} \qquad (8)$$

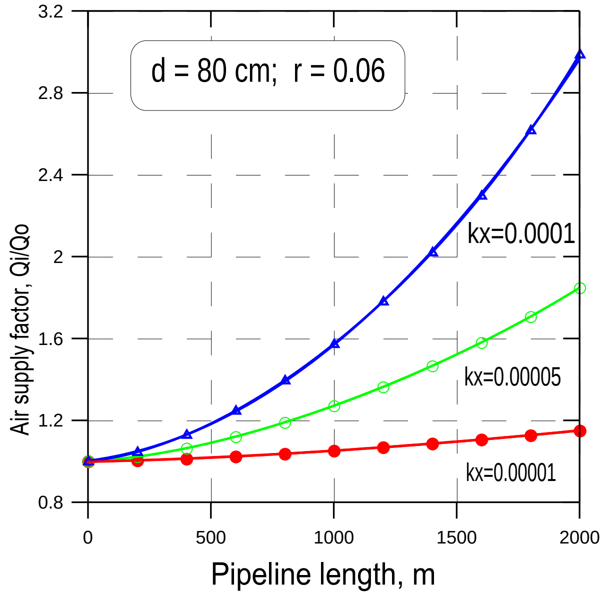


Figure 4a. Air supply factor (d=80 cm)

◀ $P = 1.00247 + 0.000151\ X + 4.1491\,E - 7X^2$
○ $P = 0.9922 + 0.0001278\ X + 1.5005\,E - 7X^2$
● $P = 0.9974 + 3.1876\,E - 5\,X + 2.2565\,E - 7X^2$
d= 100 cm; r= 0.02
◀ $P = 0.99241 + 0.000139\ X + 1.82596\ E - 7X^2$
○ $P = 0.99376 + 8.40209\ E - 5\ X + 7.47475\ E - 8X^2$
● $P = 0.9984 + 1.875895\ E - 5\ X + 1.260837\ E - 8X^2$

For the same purpose can serve expression (4). English equivalent of k_x is $L_c = k_x 100\sqrt{1000}$.

Above described measurements and calculations can be used to obtain real values for k_x and resistance R_p, reflecting pipe diameter, i.e. filtration surface $F = \ell p\ d$ and aero dynamical leakage conditions. When filtration intensity in equal by length sectors $(\ell_{P_1} = \ell_{P_2})$ with different diameters $(d_1 \neq d_2)$ is equal ($FILQ_1$ $=FILQ_2$), the following expressions are obtained:

$$k_{x1}\,d_1 = k_{x2}\,d_2 \quad or \quad \frac{R_{p1}}{d_1^2} = \frac{R_{p2}}{d_2^2} \qquad (9)$$

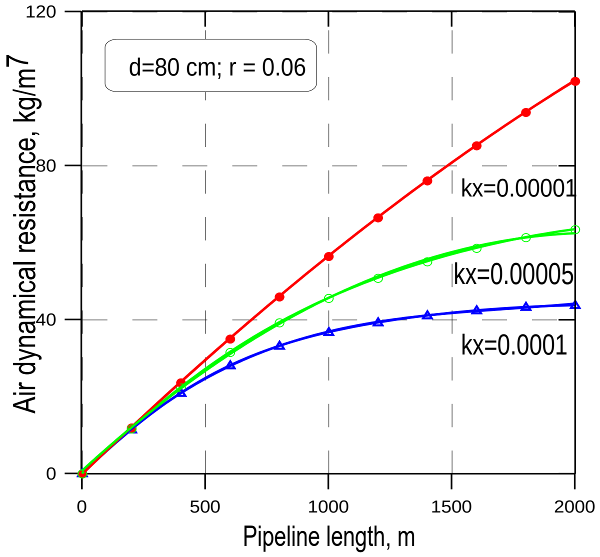


Figure 4b. Pipeline resistance (d=80 cm)

◀ $P = 0.26634 + 0.06223\ X + 5.51496\ E - 6X^2$
○ $P = 0.680196 + 0.05889\ X + 1.40185\ E - 5X^2$
● $P = 0.24751 + 0.06559\ X + 3.5308\,E - 5X^2 + 6.8006\,E - 9X^3$
d= 100 cm; r= 0.02
◀ $P = 0.068201 + 0.02052\ X - 1.15546\ E - 6X^2$
○ $P = 0.680196 + 0.05889\ X + 1.40185\ E - 5X^2$
● $P = 0.1336 + 0.02195\ X - 8.34636\ E - 6X^2 + 1.17097\ E - 9X^3$

These are criteria for equal pipeline tightness degree dith different diameters. Following indexes can be written: $K_d = k_x d$ or $Z_d = L_c d$ and $R_d = R_p / d^2$, in order to compare pipelines in regard to tightness.

Table 2 presents classification based on Vutukuri V.S. (1983) of un-tight pipelines for different diameters but equal tightness conditions k_x and R_p for ℓ_p=100 m.

Table 2. Comparison of tightness factors

d	kx	Rp	Kd	Ld	Rd	tightness
0.25	0.0002	2500	0.00005	0.158114	40000	Very good
0.50	0.0002	2500	0.0001	0.316228	10000	Good
0.75	0.0002	2500	0.00015	0.474342	4444.4	Average
1.00	0.0002	2500	0.0002	0.632456	2500.0	Poor
2.00	0.0002	2500	0.0004	1.264912	625.0	Very bad
5.00	0.0002	2500	0.001	3.162280	100.0	

Tight factors of E. Simode (1976) and Pawinski (1979) are for coefficient kx and refer to equal pipelines diameters.

SPECIFIC CHARACTERISTICS OF UN-TIGHT PIPELINES

Main purpose in design process of pipeline ventilation systems is to achieve advisable air flows and pressure distribution for variety of facilities and boundary conditions such as: length of pipeline (L), length of sectors (ΔL) and composition tubes (ℓ_f); diameters and resistances *(d,r)*; degree of tightness (k_x, k_f); number, location *(i)* and fans' pressure characteristics (h_F-Q_F), different *Qo* etc. Calculations are conducted for each subset of data by utilization of distribution functions Q_i and h_i or *Qx* and *hx*, obtained either under (1) or (2).

Such repeatable calculations can be avoided by application of specific characteristics for the pipeline *Rx* and *Px*, which depend only on its length (x), friction (*r*) and filtration (R_f or *Rp*) factors:

$$R_x = \frac{h_x}{Q_x^2} = R(x;r,k) \quad (10)$$

$$P_x = \frac{Q_x}{Q_0} = p(x,r,k) \quad (11)$$

Functions (10) and (11) are developed once by solution of (1) or (2) for arbitrary L and Qo (for istance L=2000 m Q_0 =1 m³/s). They are permanent characteristic (passport) for the taken pipeline (*r, kx* or *kf*). Catalogue of *R-P* passports and corresponding $h_F(Q_F)$ fan characteristics give sufficient information for engineering calculations in un-tight ventilation systems. Table 3 presents R-P passports for two widely used pipelines with three degrees of tightness. Figures 4a and 4b show in graphical way resistance and air supply coefficients for *d=80 cm and r=0.06*. Polynomials, approximating these characteristics for the two pipelines with three types of tightness are shown under the graph.

Table 3. P-R passports for two pipelines with different degrees of tightness

x	D = 80 cm; r = 0.06						D = 100 cm; r = 0.02					
	kx=0.00001		kx=0.00005		kx=0.0001		kx=0.00001		kx=0.00005		kx=0.0001	
	P	R	P	R	P	R	P	R	P	R	P	R
0	1.00	0.00	1.00	0.00	1.00	0.00	1.00	0.00	1.00	0.00	1.00	0.00
200	1.00	11.93	1.02	11.67	1.05	11.35	1.00	3.99	1.01	3.94	1.03	3.87
400	1.01	23.65	1.06	22.33	1.13	20.89	1.01	7.93	1.04	7.67	1.07	7.37
600	1.02	35.01	1.12	31.54	1.25	28.05	1.01	11.81	1.07	11.10	1.14	10.32
800	1.04	45.96	1.19	39.24	1.40	33.13	1.02	15.60	1.11	14.18	1.22	12.72
1000	1.05	56.47	1.27	45.58	1.57	36.68	1.03	19.30	1.15	16.93	1.32	14.65
1200	1.07	66.51	1.36	50.80	1.78	39.20	1.04	22.91	1.20	19.36	1.42	16.19
1400	1.09	76.11	1.47	55.10	2.02	41.03	1.05	26.42	1.26	21.51	1.55	17.43
1600	1.11	85.23	1.58	58.59	2.30	42.34	1.06	29.83	1.32	23.40	1.68	18.42
1800	1.13	93.88	1.71	61.37	2.62	43.23	1.07	33.13	1.39	25.04	1.83	19.18
2000	1.15	101.97	1.85	63.44	2.99	43.70	1.09	36.30	1.46	26.41	2.00	19.73

ENGINEERING CALCILATIONS

One fan in pipeline

Pipeline and fan are set by their specific characteristics $R-P$ and $h_F(Q_F)$. Calculations are performed for the whole pipeline length, divided into separate sectors ΔL (fig. 6), thus defining junctions i – fans' location and place of intermediate calculations. To each of such junction are tied down sector's growths of ΔR_i and ΔP_i along the axes *X* и ℓ.

When fan is installed at the beginning $(x=0;\ \ell=L)$ of transit flow (fig. 5), it operated in compressed regime along the whole pipeline length. On the contrary, when it is installed at the end of pipeline $(x=L;\ \ell=0)$, its action is forced. Air current parameters in both cases are equal, but with opposite meaning. Air resistance overcome equals to R_L, filtration is one directional $(+FILQ_x\ or\ -FILQ_\ell)$ and maximal, no re-circulation is presented $(RECQ=0)$. Водеща цел на проекта е да осигури достатъчно Project's main goal is to ensure required air volume for the ventilated object Q_0, pre-defined or calculated in advance on limitating factors.

Aero-dynamical calculations for such system are performed in two ways:

- under required Q_0 work regime of fan is evaluated by the following expressions:
-

$$Q_F = Q_L = Q_0 P_\ell \text{ и } h_F = h_L = R_\ell\, Q_0^2 \quad (12)$$

- fan or fan aggregate is selected $h_F(Q_F)$ and its resulting regime under R_L and air volumes distribution are then evaluated by the system of equations:

$$h_F(Q_F) \quad u \quad h_L = R_L Q_L^2 \quad (13)$$

$$Q_0 = Q_L / P_L\ ;\ Q_i = Q_0 P_i\ ;\ h_i = R_i Q_i^2 \quad (14)$$

In case obtained value of Q_0 is inacceptable, solution is repeated with other fan or pipeline.

In case fan is located along the transit current (fig. 5), two pressure zones are formed in the pipeline: forced by x and compressed by ℓ with corresponding filtration $\left(+FILQ_x \ and \ -FILQ_\ell\right)$ and re-circulation $\left(RECQ_{\ell x}\right)$. Aero-dynamical pipeline resistance is divided into two independent branches $R_{ix} \ and \ R_{i\ell}$, where:

$$R_i = R_{ix} + R_{i\ell} \ ; \ h_i = R_i Q_i^2 \tag{15}$$

Fan work regime $\left(h_i, Q_i\right)$ is evaluated under (15), while pressure and air volumes distribution – on the expressions written below:

$$Q_{0x} = Q_i / P_{ix} \ , \ h_{ix} = R_{ix} Q_{ix}^2 \tag{16}$$

$$Q_{0\ell} = Q_i / P_{i\ell} \ , \ h_{i\ell} = R_{i\ell} Q_{i\ell}^2 \tag{17}$$

The above described calculations are repeated for the selected $R-P$ pipeline for $L = L_{\max}$ and for $L < L_{\max}$. In this way need for one or more fans for pipeline installation construction and maintenance is estimated. Calculations might be repeated with other input data in case results doesn't satisfy either by technological or by economical reasons.

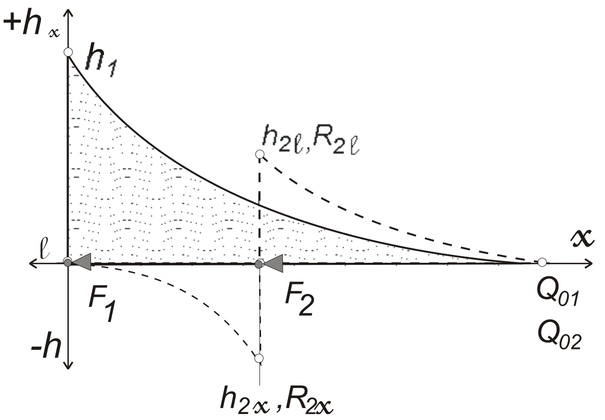


Figure 5.Fan in the beginning of pipeline

Two and more fans in the pipeline

Normally one of the fans is installed at one end of the pipeline – compressed along x or forced along ℓ (fig. 6). The rest of them are located along the air current. Their interaction is evaluated by summing up their individual functions $h_{ix} \ and \ h_{i\ell}$ for each pipeline junction:

$$\overline{h}_i = \sum h_{i\ell} - \sum h_{ix} \tag{18}$$

and by re-calculation of resulting air flow:

$$\overline{Q}_i = \sqrt{\overline{h}_i / R_i} \tag{19}$$

Resulting values $\overline{h}_i$ and $\overline{Q}_i$ describe pressure and air volumes variation along the transit flow direction (fig. 6). Function $\overline{h}_i$ forms behind each fan junction with one of the following pressure:

- N – null ($\overline{h}_i = 0$);
- K - compressed ($\overline{h}_i > 0$);
- D – depressed ($\overline{h}_i < 0$);

Function $\overline{Q}_i$ describes continuity of transit air current - descending by x and ascending by ℓ.

Between two adjacent junctions N-N zone is formed, where pressure loss is restored only by fan located within. Zone N-N parameters result from individual regime of this fan $\left(R_i, P_i, h_i, Q_i\right)$.

Compression in junction К defines the degree of sequential aggregating of interacting fans. Increase in the distance between them lead to appearance of junction N and then to D. Decrease in distance between fans lead to increase of compression to 100% (fans operate in one junction consecutively).

Depression sector N-D causes re-circulation in the pipeline, which should be avoided by default. It can be reduced and overcame by decreasing of distance between interacting fans.

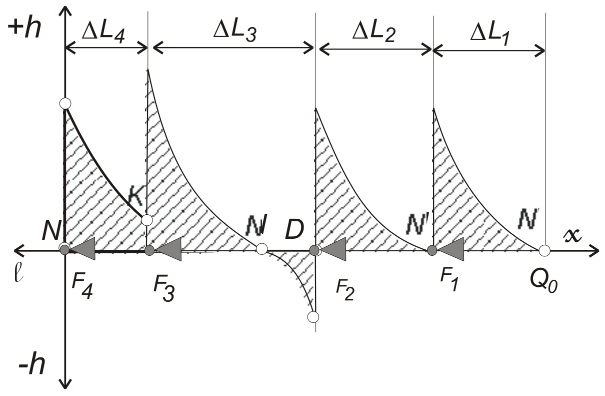


Figure 6. Two and more fans

Desired operation of given (available) fans is searched through variant calculations in order to find the best distance between fans.

Ventilation system without re-circulation is achieved assuming N-N scheme under required Q_0 (fig. 6) by the following procedure:

- Maximal pipeline length *L* is divided into sectors ΔL_i. Under required at the end of transit flow value Q_0 functions $h_i \ and \ Q_i$ are evaluated either by P_i , R_i passport or under solution of (2).
- Selection of Fan №1 (of fan aggregate composed from smaller fans) is performed for the first N-N zone, which include several sectors. Thus selected fans should satisfied sectors' characteristics h_i ; Q_i for each pipeline lengthen. This could be achieved by increase of motor rotation or change of fans' blades angle.
- In the same way next fans (aggregates) №2, №3 etc. are selected for the following zones with required air quantity Q_0,

equals to fan operation efficiency in the preceeding zone $(Q_{02} = Q_1\ ,\ Q_{03} = Q_2)$ etc.

- Project N-N should be achieved with 5-10% increase of calculated fan pressure for fans №2, №3 etc., which ensures reserves for avoiding re-circulation sectors. presents numerical solution for 2000 m pipeline, composed from two consecutive sectors with characteristics written below:

$$\Delta L_1 = 1000; d_1 = 80; r = 0.060; k_x = 0.0001$$

$$\Delta L_1 = 1000; d_2 = 100; r = 0.020; k_x = 0.0001$$

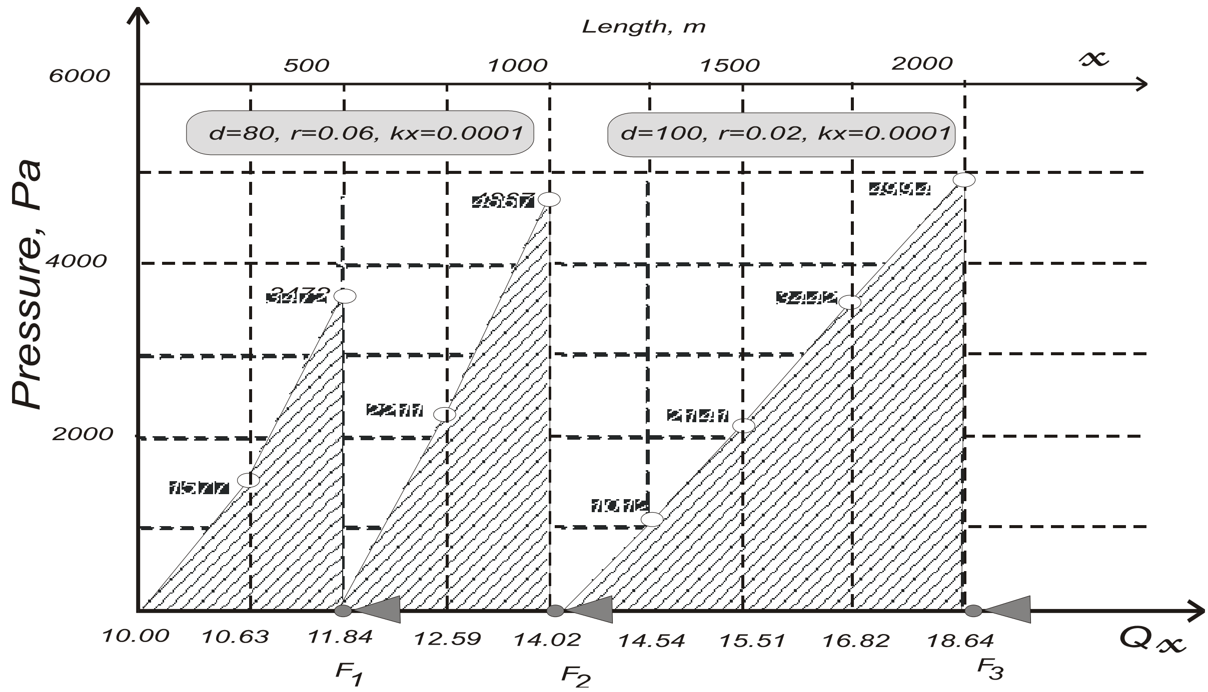


Figure 7.Numerical solution for 200 m pipeline

Final stage of designed forced ventilation system is shown on the figure. Fans selected are one type with possible changing of their blades' angel:

$$(\mathrm{F}_1, \beta = 15^\circ; \mathrm{F}_2, \beta = 25^\circ;\ \mathrm{F}_3, \beta = 30^\circ)\ .$$

Intermediate stages (regimes) after each pipeline lengthen are shown at each $\Delta L = 250\,m$. Total theoretical fans' power equals to 202,36 kW.

CONCLUSION

Comparative calculations for typical un-tight pipelines are performed based on known solutions and approaches. They give ground for the following conclusions and achievements:

- Computer program calculating distribution functions $h_x\ and\ Q_x$ is developed. It reflects algorithm based on consecutive polynomial approximation and integration of (2);
- By utilization if the above described program passports $R-P$ for two types of pipelines with characteristic parameters are created and presented in graphical and numerical way;
- Numerical algorithm for "un-tight pipeline – fans" ventilation system design is presented which allows to optimize types and fans location.

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Recommended for publication by Department of
Mine ventilation and occupational safety, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.163-169

INTELIGENT COMPUTER SYSTEM FOR CONTINUOUS RISK EVALUATION AND DECISION SUPPORT OF SAFETY MANAGEMENT IN MINING

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ABSTRACT
The structure and the theoretical basis of Bayesian network - Mine Accident Risk dot Net (MAR.NET) for decision support in mining safety are presented. The network is composed from 22 nodes described with specific states designed for description of mining risk factors. In the heart of the network is conditional probability distribution of the chance node "Type of the accident" containing 37800 state configurations. A new formula for evaluation of safety risk level is proposed. For the purpose of decision making the network is extended to influence diagram. The possibilities of learning and adoption of MAR.NET to specific object in uncertainty and some application of simulated safety cases are discussed.

INTRODUCTION

The information support of the safety management required processing of a large amount of data both of quantity and quality type. It is well known from the practice that taking into account only quantificators of safety risk like coefficients and indexes of frequency and weight are not sufficient for characterizing and control of safety level. The quality system for risk evaluation is needed. The new generic ISO 18000 series devoted to quality safety management is an eloquent fact about the importance of the problem.

Today the investigation and registering of an accident are documented in minimum 60 data fields of different formats. The scrutinize of safety risks in the large mining companies ordinary includes more than 3000 massive of data for description of 50 and more accidents per year. Every one accident is classified in 21 indicators (tab. 1) any of which described with 2 up to 26 characteristics (states) (Michaylov and Petrov (1997), Michaylov and collective (2002)).

The psychology and cognitive science are ascertain the fact that the human mind cannot effectively manipulate such kind of data structures and meet serious difficulties to make an inference when the possible decisions have more than 3 alternatives. Such informational overload of the consciousness leads to ignorance of information and heuristic deciding of variants. The risk of bad decisions increasing and the safety become pursuit rather than achieved purpose. The problem is of particular interest in time critical decision-making.

Another important feature of safety management is the probability and fuzzy uncertainty attending decision-making. In practice this mean that the expected effects of actions for risk reducing also will be connected with some likelihood. What is the degree of certainty that one of the expected effects is most likely is the correct question. Rendering the expert opinions is everyday activity in safety management. The successive implementing of acquired data and expert experience in decision-making is of vital importance.

A typical approach for safety risks investigation by separately studying of the impact of isolated factor groups inevitably relevant to loses of information about the mutuality in the safety system. In terms of information such a "disjoint" is irreversible process. In other words the model we use to support our decisions those not reflect the real world because the interdependences in the safety system are very diligence removed since on methodological level.

New synergetic approach should apply for the purposes of risk investigation and decision support for improvement of safety. A model representing the mutual influence of dangers, human and control over the safety is needed.

The structure of intelligent computer system MAR.NET (My Accident Risk dot Net) for information fusion of databases and expert opinion is presented. The system is designed for practical use from safety managers in mining companies.

INDICATORS OF SAFETY RISKS IN MAR.NET

In the best practice exist pursuit of an integrated system of indicators characterizing safety risks. An integrated system of safety indicators for mining industry was developed in University of Mining and Geology "St. Ivan Rilsky" in the beginning of 90 and last updated in Michaylov and collective (2002) (tab. 1). Every indicator has a set of characteristics structured in hierarchical groups. The number of groups in stets are between 2 and 26. Because of lack of space the characteristic sets are not present in this paper.

The defined indicators in table 1 can be used for quality investigation of safety risks in any other industrial branch. The characteristic sets of some of the indicators will be different. But the principle of studying and data manipulation remains the same. This is great advantage for implementation in practice and for software development.

The shown indicator system can be used for risk investigation in all industries. The worldwide practice shows that successive computer systems using artificial intelligence methodologies are developed for a local domain irrespective of some universality. MAR.NET is developed for investigation of safety risks in mining industry. The sub domains in knowledge base for mining branches - coal mining, metal and nonmetal mining both for underground and open pit, required different sets of characteristics for some indicators.

Table 1. Safety Risk Indicators.

Name	Short label
Time of occurrence	01.Hour
Occupation groups	02.Occupation
Degree of education	03.Education
Length of service	04.Practice
Length of service in entertainment	05.Practice_Co
Length of service in profession	06.Practice_Pro
Day after last rest (weekend)	07.Day_after
Hours after start of job	08.Hour_after
Place of accident	09.Place
Kind of job during the incident	10.Job
Kind of incident leading the accident	11.Incident
Human factor in cause of accident	12.Human_Factor
Material factor in cause of accident	13.Material_Factor
The dangers of the environment	14.Environment
Deviation from ordinary actions and conditions	15.1.Deviation_A 15.2.Deviation_C
Severity of the accident	16.Severity
Harmed parts of body	17.Body
Kind of injury	18.Injury
Period of health restore	19.Recover_Period
Safety precautions (risk redusing measures)	20.Measure
Machines related with the accident	21.Machinery

Place of accident is a typical indicator for which the characteristic set needs to be overwritten for the different mining objects. On the other hand the learning of MAR.NET system will be much more adequate for specific branch and adoption – for specific objects. The convergence of the systems is the next step.

DRAWING OF INFERENCE FOR SAFETY RISKS LEVEL

Reporting the safety risk level is the important end result of risk evaluation. It is hard to define all the different aspects of risks in notion of one safety level. For the purpose the following definitions are accepted:
The drawing of inference for safety risk level is a process of statistical conclusion for synergetic influence of the risk factors on the accident severity in uncertainty. The risk factors under review are shown in table 1.

In the terms of safety a classical definition for risk is the production:

RISK = PROBABILITY x CONSIQUENCES

In the description of safety level all risks must be take into account. After execution of safety programs the object of evaluation is the remaining (current) risk. - R_c. The level of safety can be useful quantificator for comparison of objects and branches with one value. But the dimension of risk is specified from the dimension of consequences. Usually the consequences are classified as economical and human and social. In capacity of quantitative link can be used the count of loosed working hours. It is not necessary to be human-hours.
The indicator 19.Recover_Period described the consequences of the accidents with 10 discrete intervals – “A. Up to 3 days”, “B. 4 to 17 days”, ... , “J. 6000 days (means irrecoverable accidents)”.

The following expression for calculating the safety risk level are proposed:
If we accept that the threshold for sensitivity of risk evaluation is in probability of $1/10^8$ and 3 loosed working days as a consequence, than the minimal safety risk is evaluated on $R_0 = 3_x 10^{-8}$. In that case the level of safety risk L_s can be calculated as a function of current risk - R_c and the threshold risk - R_0 by (1).

$$L_s = \log\left(R_c / R_0\right) \quad (1)$$

The L_s posses some properties which makes it useful for calculating of risk level. First, since the risks are always positive quantity and $R_c \ge R_0 > 0$ the value of level will be always calculable and $L_s > 0$. Second, when the current risk aligns with the threshold $R_c = R_0$, the safety risks level $L_s = 0$. And last but not least the human perceptions are determined from logarithmic levels as stated of generalized psychophysical low of Veber-Fehner. Take into these considerations the 10th basis of logarithm is recommended.
Besides the one-value quantification of safety risk the management is needed of detailed quality investigation based on the available knowledge. The data collection for risks and safety level of a specific object is made by excerption (registry of accidents, failures, protocols of inspections etc.). It leads inexorable to statistical evaluation of excerpted conclusion for the real state of the safety system, which obviously is richer of properties (*Вентцель (2001)).* Nonlinear dynamics of manifestation of the incidents with possibility the safety system to pass in chaotic regime (Guastello (1997); Stengers and Prigogine (1997); Petrov (1999)) puts the question outside of the application range of well-developed methodologies for reducing of the problem dimension like deterministic factor analysis and classical statistical averages.
(Трухаев, Горшков (1985)).

Bayesian approach for statistical inference

The frequency interpretation of probability is called objective ore classical point of view in statistic theory. In the statistical decision theory the Bayesian approach is used to draw of conclusions in uncertainty. The approach offers a different interpretation of probability called subjective point of view. The idea of conditional probability takes a main place in Bayesian approach. From the investigator is required to use subjective probability as a measure of belief for the state of observed object (*Hines, (2000)*). This is more intuitive perception of probability, which means rather than chance than frequency. The level of belief is specified with the probability distribution for a given unknown parameter. This procedure is completely different of any other statistical approaches, where the uncertain parameters are treated as unknown constants. The Bayes approach required from investigator to think for unknown parameters as random variables.

The low for complete probability and Bayes Theorem

The evaluation of impact of co incidents on the accidents is in the base of detailed risk investigation. The mathematical fundament of the MAR.NET model is based on the following major dependencies:

Let *A1, A2, …* is an enumerated collection of events sharing a space of realization – *S.* The events in collection are mutually independent with union *S*. Let *B* is another event. If the probabilities $P(A_i)$ and $P(B \mid A_i)$ are known for $i \in I$ (*I* is the set of indexes of events) than it can be shown that:

$$P(B) = P(B \mid A_1)P(A_1) + P(B \mid A_2)P(A_2) + \ldots, \quad (2)$$

a result known as low for complete probability;

$$P(A_j \mid B) = \frac{P(B \mid A_j)P(A_j)}{P(B \mid A_1)P(A_1) + P(B \mid A_2)P(A_2) + \ldots}, \quad (3)$$

a result known as Bayes Theorem and;

$$P(A_1, A_2, \ldots, A_n) = P(A_1 \mid A_2, A_3, \ldots, A_n)P(A_2 \mid A_3, A_4, \ldots, A_n) \ldots P(A_{n-1} \mid A_n)P(A_n), \quad (4)$$

a result known as chain rule, with significant importance in Bayesian networks.

Bayesian belief network - BBN

The Bayesian network is presented with directed acyclic graph with the following elements: the chance nodes representing random variables and the edges – probability independencies between the variables. The nodes have conditional probability tables assigned to describe the independencies. The nodes without the parent have unconditional (marginal) distribution.

One of most powerful feature of Bayesian network is the global treatment of local uncertainty. In other words – the changes in probability distribution in one chance node are propagated to all the nodes linked with edges following the directions in the network. The propagation of probability against the directions is possible due to Bayes Theorem. For statistical description of net and propagation of probability the results (2, 3 and 4) are used.

Unlike of classical statistical inferences (which work rather than with confidence intervals than statement of probability) is, that the Bayesian inference completely described the fact, that the expectation alone cannot predict the probability of unexpected events. Prior information for unexpected events is needed. The necessity of prior opinion is the key part of Bayesian inference.

Of course, this requirement is a weakness. Not always is easy to obtain the prior information, except of experts. But in lack of data for safety in an entertainment (newly created or thinly proficient) the opportunity to use experience from similar objects and from experts is great advantage. A well-learned Bayesian network can be of great benefit for newly appointed safety personnel. In objects with complicated behavior of safety system, even not large, the advantages of such technology will stay clear for a short time.

Drawing of inference in Bayesian network

Drawing of inference or making conclusions in Bayesian networks means calculating of conditional probability for some variables to be given information from the others. It is ease when all indications are lead from predecessors (parents) to child variables (nodes) of interests ($B_i \rightarrow A$, fig. 1). But when an indication is gived from child to parent, the network must draw a conclusion against the direction of edges. The Bayes Theorem (3) is used for such a back propagation of probability.

Decision-making and Influence diagram

Computer models for decision support can be developed on the base of pure BBN, but the conceptions of utility functions and the decisions are not clearly formulated and fully cover. The extension of BBN with two tapes of nodes represents an influence diagram (fig. 1). The influence diagrams are used fore evaluation of different variants of decisions by calculating of expected utility of launched actions.

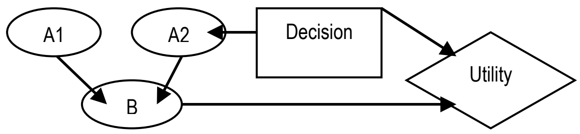

Figure 1. An example of Bayesian network extended to influence diagram with node "Decision" and node "Utility"

The decision nodes must be linked in chain in the logical consequence of independency between variables in the model. The nodes from which the decision is depend, must be with known state before the decision is made.

STRUCTURE OF MAR.NET

MAR.NET is designed as Bayesian belief network extended to influence diagram (fig. 2). The main purpose of MAR.NET is to support decisions for increasing of safety level in industrial objects, entertainment and whole branch. Currently the states of the nodes are adapted for the specific of coal mining. The nodes in the model correspond with the indicator variables described in tab.1. The states of indicators are labeled like the characteristics of the indicators. The conditional dependency between the variables can directly be read from the graph on fig. 2. The probability of different configurations of states in the net described the subjective point of view to happen an accident according to conditions given by states of parent nodes.

Marginal nodes in the root of the net are 14.Environment, 10.Job and 02.Occupation. The initial state of MAR.NET are uniform distribution of probability of the state configurations. The zero probability assigned to a state or configuration of states means striking off the possibility of occurrence of this state. The probability tables assigned to the marginal nodes are shown in table 2.

Social-human severity of accidents is evaluated of 16.Sevirity node. Economical risk of accident consequences is evaluated from utility node "Loses". The risk measured in loosed working days is evaluated from chance node 19.Recover_Period. All the 3 nodes are child of parent nodes and have conditional probability distributions.

Every state configuration (column) is an independent group with complete probability of 1.

In the heart of the MAR.NET is a chance node 11.Incident – "Kind of incident leading the accident". In current realization the node is a child of five predecessors – four chance nodes and one decision node respectively 12,13, 15.1, 15.2 и 20 (see tab.1 for full names). The conditional probability table of node

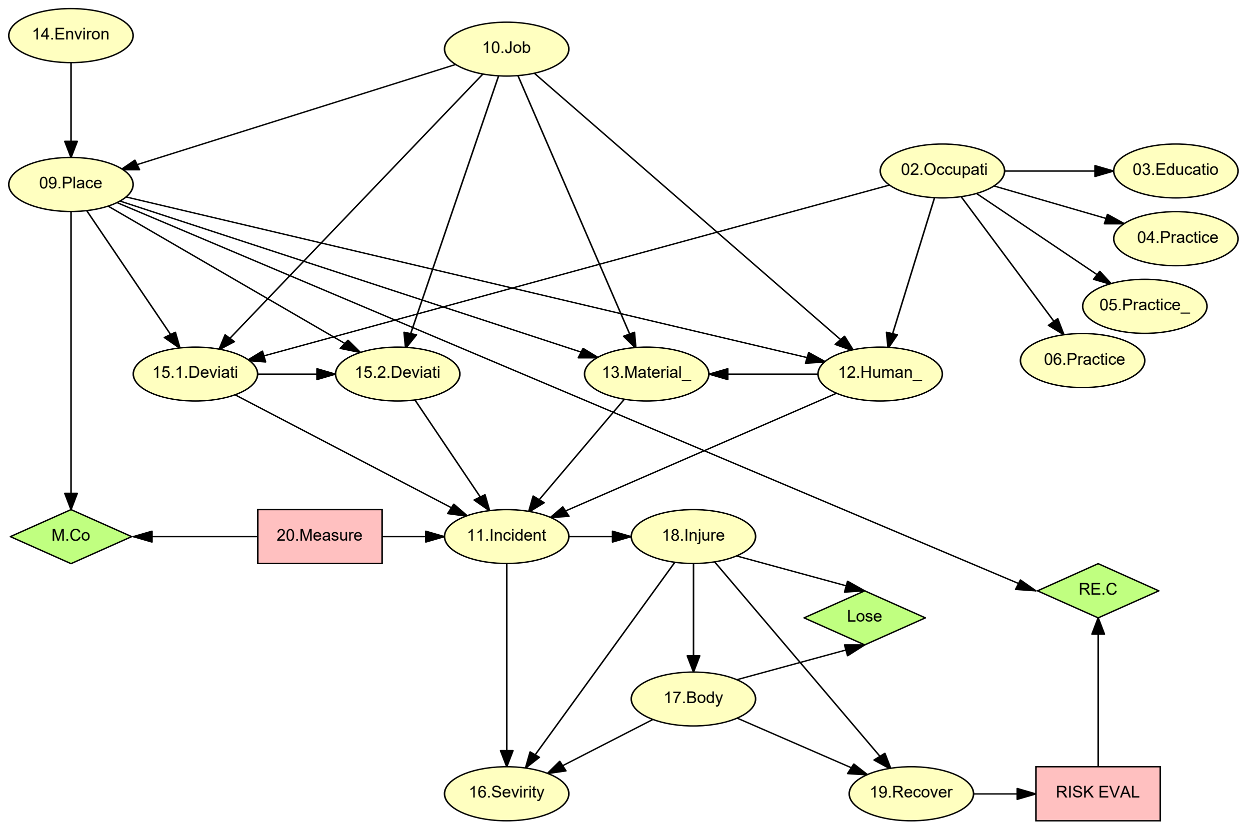

Figure 2. MAR.NET decision support system for improvement of safety level

Table 2. Initial probability table of chance node 10.Job.

State	Probability
A. Transport and load	0.2
B. Ordinary exploatation	0.2
…	…
E. Other	0.2

The impact of nodes 01.Hower, 07.Day_after and, 08.Hower_after (tab. 1) are object of additional study and not connected in the MAR.NET model on the present stage of development. The rest of the nodes have conditional probability distributions of their states (tab. 3).

Table 3. Initial conditional probability table P(17.Body|18.Injure)

18.Injure / 17.Body	A	B	…	Z
A. Head	0.25	0.25	0.25	0.25
B. Hands	0.25	0.25	0.25	0.25
C. Legs	0.25	0.25	0.25	0.25
D. Others	0.25	0.25	0.25	0.25
Total	*1*	*1*	*1*	*1*

11.Incident is a multidimensional massive given by probability distribution (5).

$$P(11.Incident|12.Human\_Factor, 13.Material\_Factor, 15.1.Deviation\_Environment, 15.2.Deviatin\_Action, 20.Measure) \quad (5)$$

The massive dimension is 18*14*15*5*2 = 37800. It is clear why for the experts is impossible to take into account all known configurations of conditional states.

The looses caused by the accident are described in utility node "Loses" (tab. 4)

The decision node 20.Measure – "Safety precautions (risk redusing measures)" is used for continuous evaluation of actions provided in safety programs.

Table 4. Utility table of node "Loses"

Body	A			B …	
Injure	A	...	P	A	…
Loses	-90	…	-100000	-150	

The effects of actions are propagated in MAR.NET through chance node 11.Incident. A simple question can be given by defining of two states of decision node 20.Measure: Action 0 and Safety Program. It means to do nothing or to execute a safety program. The cost of actions are specified in utility node

"M.Cost" (tab. 5). The conditional independency of the nodes 09.Place and 20.Measure can be read of the graph in fig. 2.

Table 5. Utility table of the node "M.Cost"

Place	A		B		C
Measure	Action 0	Safety Program	Action 0	Safety Program	…
M.Cost	0	-5000	0	-10000	…

With the decision node "RISK EVALUATION" the expected utility of risk evaluation procedures are evaluated. For example if two state "Yes" and "No" are assigned as the decision the utility function will calculate expected utility of both actions. The pressure for starting of risk evaluation renders the increasing of loosed working days – node 19. The expenses related with the procedures of evaluation are given from utility function by node "RE.Cost" as conditional distribution determined of predecessor nodes "RISK EVALUATION" и "09.Place" (see fig. 2 and tab. 6).
The choice of alternative decision is make on principle of maximal expected utility. The global utility function U is a total of all expected local utility (5).

Table 6. Utility table of the node "RE.Cost"

Place	A		B		C
Risk Evaluation	YES	NO	YES	NO	...
RE.Cost	-1000	0	-2500	0	…

$$U = \sum_i u_i \quad (5)$$

$$u_i = \sum_j p_j u_j \quad (6)$$

where p_j is the conditional probability for occurrence of state configuration c_j, and u_j - is the value of utility related with this realization. For example the distribution of utility node Loses contains $j = 104$ probability. But in calculation of U the local utility expected in nodes RE.Cost and M.Cost are taking into account, i.e. $i = 3$.

INFERENCE OF SAFETY LEVEL WITH MAR.NET

If a new prior information is available the procedure for statistical inference must be started in order to update the probability distributions in the net. The propagation of known probability according to the independency between the variable (given by the edges) is executed. The posterior probability distribution for all the nodes is as a result. The expert can see the probability of any configuration of the node states and evaluated utility of alternative decision actions.
An example for node 10.Job -" Kind of job during the incident" is given on figure 3.

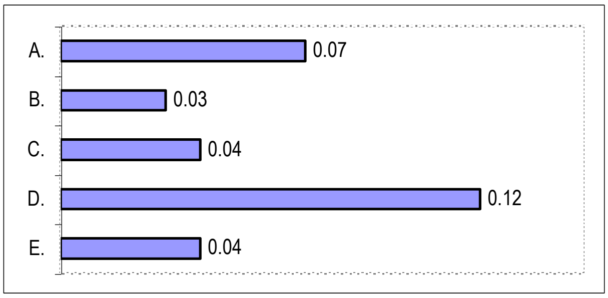

Figure 3. Posterior probability distribution of node 10.Job.

LEARNINIG OF MAR.NET

The learning of the model can be done by 6 different ways. The first – imputing the known probability by hand for given configurations of states. According to dimensions of distributions this is a hard task. For the purpose of machine learning the special algorithms are developed. When the database for safety in the object is accessible, union of SQL queries can prepare the initialization of probability distributions. For the purpose of learning, the structure of MAR.NET is described in manner of (5) from the roots of network to the end child nodes. The SQL queries in the union follow described consequence. As a result the probability tables are formed.

Learning from data cases
This type of learning is appropriate for initializing of probability distribution after structure defining of the net. A set of variables for which the prior information is available is specified. For any of the nodes related to the specified set the experience table is assigned. The experience tables count the number of realization of any specified configuration of states in the set. Learning set of variables (nodes) in MAR.NET envelope all indicators (tab. 7).

Table 7. Example of learning data cases for MAR.NET

N01	N02	N03	N04	N05	N06	…	N21
A	A	D	Q	A	B	…	N/A
C	I	D	D	N/A	C	…	C
...							

Where N01..N21 is the internal name of the nodes.
When thee is no information about manifestation of some variable in the case the missing data is marked with "N/A" symbol. The codes A, B etc. corresponding with the labels of states of variables. The nodes represent the indicators (tab.1). The states are the characteristics of the indicators. The nodes and states of MAR.NET are compatible with databases of the software product for registering and analyzing of accidents Mine Accident Risk version 2002 (MAR). The product Mine Accident Risk are developed since 1995 in the department of "Mine Ventilation and Labour Safety" in MGU "St Ivan Rilsky".
The learning metod of MAR.NET is known as EM-algorithm commonly used in Bayesian network for graphical associated models with missing data (Cowell and Dawid, 1992; Lauritzen, 1995). The target of algorithm is enriching of conditional probability tables assigned to the nodes of network. For this purpose the algorithm performed a number of iteration. In any iteration logarithm of probability the given example to produce the current probability distribution is calculated. The EM-algorithm tries to maximize this log-probability. The iterations

stop when the deferens between log-probabilities obtained of two successive iterations became sufficiently small (for example of the order of 10^{-4}). The EM-algorithm cannot learn the conditional distributions for continuous nodes. In MAR.NET there are no continuous nodes.

Learning adoption from data cases

Learning adoption of MAR.NET is necessary when a new accident is registered or new information from inspection, investigation or observation is available. The adoption of knowledge about the safety through consequtively updating of probability distributions in the net on the base of available experience is performed. The experience about a given discrete node is present as a set of counts for evidence $Alpha_0,...,Alpha_{n-1},$ where *n is a count of configurations of the parent nodes.* $Alpha_i$ *means the number of times a parent node to fail in* i^{th} *state configuration* конфигурация. The count has a sense of frequency and is a nonnegative real number. $Alpha_i$ is stored in experience table assigned of the nodes determined for learning adoption. The nodes for which there are no experiences are adopted according the rules of probability propagation in the net as discussed above.

Entering expert opinions

The notion of experience in Bayesian networks can be introduced as a quantitative memory which can be based both on quantitative expert judgment and past cases. *Dissemination of experience* refers to the process of computing prior conditional distributions for the variables in the network. *Retrieval of experience* refers to the process of computing updated distributions for the parameters that determine the conditional distributions for the variables in the network (*Spiegelhalter and. Lauritzen, (1990)).*

The used in MAR.NET algorithm for entering the expert opinions allows control of the actuality of learned experience through special fading tables for reducing the impact of past. The fading factor $Delta_i$ is used for reducing the experience count $Alpha_i$. The fading factor $Delta_{i.}$, is a nonnegative real number between 0 and 1 but typically close to 1. The detailed description of the algorithm is given in *Spiegelhalter and. Lauritzen, (1990).*

Structure learning

A possibility to extract structure of the net from data cases is an exceptionally interesting feature of BBN. The data cases are structured in manner shown in tab. 7. The algorithms for structure learning of BBN are known as PC-algorithms (*Spirtes, C. Glymour and. Scheines (2000); Pearl (2000))*. The independency tests for variables in the model is performed. The test statistic is approximately χ^2 distributed and allows conditional independency. The recommended value of level of confidence in which the zero hypothesis for independence is rejected is *LC = 0.05*.

Some interesting results were obtained in structure learning of MAR.NEY with 122 data cases for registered accidents in coal mine of "Babino". The conditional dependency of following variables where accepted in *LC = 0.05:* Occupation $\rightarrow$ Time of accident occurrence, Length of service $\rightarrow$ Human factor, Education level $\rightarrow$ Day after weekend $\rightarrow$ Deviation from ordinary actions. In *LC=0.1* new dependence between Time of accident occurrence $\rightarrow$ Length of service in entertainment is accepted".

The structure learning gives an alternative way the experts to reconsider his conceptions for safety in given object using artificial intelligence. When the understanding of safety risks manifestation is changed, the model of MAR.NET on structural level also can be changed.

Simulation

Three approaches for obtaining simulated experience will be discussed. The first is by generating of simulated data cases and learning MAR.NET with EM-algorithm. The simulations are based on variations of the current prior distribution. The result of simulations must be in the format given in tab.7 in order to be useful from the learning algorithm. The simulation can be set to give a percent of missing data. The missing data imitate unknown probabilities for configuration of variable states and simulate uncertainty in the safety system. A more efficient approach is generating data cases by simulation models of real subsystems of the object. In both approaches the fixedness of safety system in case of occurrence of unregistered cases.

The third way of simulation is based on the powerful feature of Bayesian networks to derive conclusion against the direction of the edges. It can be simulate increasing of severity of accidents and after propagation of probability according Byes Theorem to obtain posterior distribution of the predecessor nodes of interest.

CONCLUSIONS

A system for decision support in mining safety MAR.NET is proposed. The system can be adopted for other branches saving the type nodes and the proposed structure. The states of the part the nodes can be different. It is recommended the learning of MAR.NET to realize on different copy of the system for open pit and underground coal mining and for metal and non-metal mining and quarries. Adoption of MAR.NET is adequate to realize for different objects on the learned instances of branch models.

There are not hidden layers in the MAR.NET. The structure is clear and ease to modify according to changes of expert opinions. The inference of safety level can be done in uncertainty, which is the usual case in safety management.

The well learned MAR.NET model can be used for education and training. The contemporary technologies allow the .NET models learned and adopted for different objects to communicate each other including via the Internet. Such a super-BBN in which the nodes are other BBNs can constitute intelligent net with distributed calculation and possibilities of knowledge exchange.

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Recommended for publication by Department of
Mine ventilation and occupational safety, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.171-173

DYNAMIC MODEL OF CAGE HOIST

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ABSTRACT
The article treats a dynamic model for investigation of the force load in the rope system of cage hoists. For this purpose, a bi-mass mathematical model of a dissipative system has been used, proceeding from the structure specifics and the technological process, obtaining solutions satisfactory for the investigation task.

Control tests and investigations of mine hoists are inseparable part of their operation. These activities are directly orientated to the operating security of mine hoists.

In the last years, non-destructive control methods have gained increasingly wider practical application which, no doubt, involves theoretical site investigation by modeling of the dynamic processes.

For complex multi-mass vibrating systems, where the vibrations of some masses are described by linear coordinates, and others by angular coordinates, the most convenient method is Lagrange's method. For a non-conservative system (with attenuation, damping), Lagrange's equation:

$$\frac{d}{dt}\left(\frac{\partial T}{\partial \dot{q}_i}\right)-\frac{\partial T}{\partial \dot{q}_i}+\frac{\partial D}{\partial q_i}+\frac{\partial \Pi}{\partial q_i}=Q \qquad (1)$$

where:

T and Π are the kinetik and potential energy of the system:;

D – energy of dissipation (distance);

q_i – generalized coordinates;

Q – generalized forces;

t – time.

Knowing the structure of a given type of mine shaft system, a number of dynamic models can be composed in connection with the investigation task preliminarily set.

Thus, for example, of particular importance for mine hoist is the force load in the rope system. According to [2], when the object of investigation is the force load in the rope system of a hoist-conveyor mechanism, the dynamic model is reduced to a single bi-mass vibrating system, which is described with the mass inertia moment of all rotating parts, reduced to the drum shaft, and with the equivalent hardness of the whole mechanism.

In the most general case of a cage hoist (Fig.1), the mathematical description of the dynamic force load in the rope system according to Lagrange's method, will be:

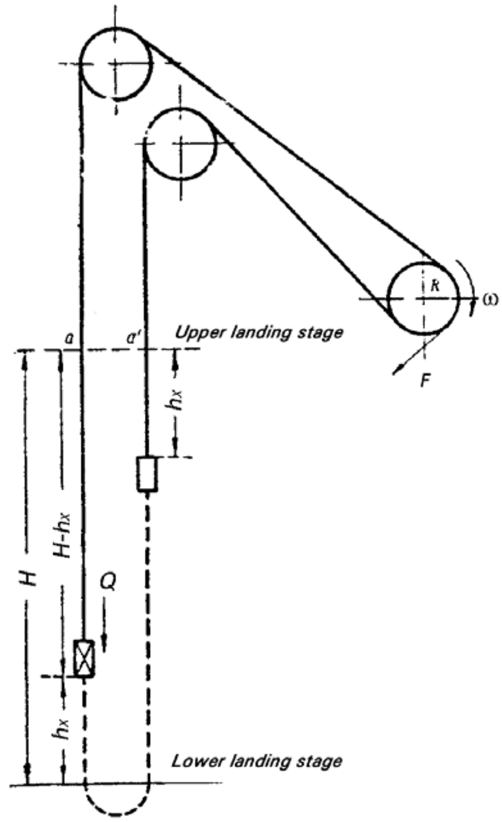

Figure 1.

$$\left|\begin{array}{l} m\ddot{x}+d\dot{x}+kx-d\dfrac{R}{i}\dot{\varphi}-k\dfrac{R}{i}\varphi=Q \\ J\ddot{\varphi}+d\left(\dfrac{R}{i}\right)^2\dot{\varphi}+k\left(\dfrac{R}{i}\right)^2\varphi-d\dfrac{R}{i}\dot{x}-k\dfrac{R}{i}x=M \end{array}\right. \qquad (2)$$

where:

x, φ - are the generalized coordinates;

m – load mass;

J – mass inertia moment of all rotating parts reduced to the drum shaft;

κ – equivalent hardness of the system;

Q – generalized force of the weight (+Q for lowering, -Q for hoisting);

M – moment of the engine (brakes) reduced to the drum axle;

R – drum radius;

d – coefficient of damping, resp. for progressively moving and rotating masses [2].

In the bi-mass model, the generalized force Q is assumed as equal to the maximal one for dimensioning of the hoist force, determined in [1], and the el.motor moment M as proportional to the angular speed of the rotor.

The system of linear differential equations (2) is solved for different initial conditions whereby the specific peculiarities of the physical process are introduced in the mathematical model. These peculiarities in the cage hoists must reflect the characteristic periods of the technological cycle, and namely:

First period t_1 – equi-accellerating movement with speed increase from *0* up to V_{max} by linear law, with constant accelleration a;

Second period t_2 – movement with constant speed V_{max} for accelleration $a = 0$;

Third period t_3 – equi-retarding movement with speed decrease from V_{max} down to *0* for constant accelleration a.

During the period of movement of hoist vessels with constant speed V_{max} = const., only static forces act in the hoist system, determined by the weight of the hoisted loads, the weight of the rope with the engaging units and the friction forces. In the periods of equi-accellerating and equi-retarding movement, besides the static forces, there also act inertia forces of the moving masses. Consequently, in this specific case, of interest are the dynamic loads upon acceleration and stopping of the load Q. If in the theoretical investigations it is assumed that the starting torque of the engine and the brakes torque are constant, and also that the washer block is undeformable, the solution of system (2) is of type [2]:

$$\left| \begin{aligned} x &= \frac{A_1}{2}t^2 + B_1 t + C_1 + D_1 e^{-\lambda t}\left(\cos\omega t + \frac{E_1 - D_1\lambda}{\omega D_1}\sin\omega t\right) \\ \varphi &= \frac{A_2}{2}t^2 + B_2 t + C_2 + D_2 e^{-\lambda t}\left(\cos\omega t + \frac{E_2 - D_2\lambda}{\omega D_2}\sin\omega t\right) \end{aligned} \right. \qquad (3)$$

where:

ω is the frequency of vibrations, which depends on the parameters of the bi-mass model and is determined by:

$$\omega = \sqrt{k\left(\frac{1}{m} + \frac{R^2}{Ji^2}\right) - \frac{d^2}{4}\left(\frac{1}{m} + \frac{R^2}{Ji^2}\right)^2}$$

λ is coefficient of attenuation, determined by:

$$\lambda = \frac{d}{2}\left(\frac{1}{m} + \frac{R^2}{Ji^2}\right)$$

A_{12}, B_{12}, C_{12}, D_{12}, E_{12} are coefficients depending on the system parameters and the initial conditions;

t – time.

The velocities $\dot{x}$ and $\dot{\varphi}$ can be determined through differentiation of system (3).

To solve the problem, it is necessary to determine the constants A_{12}, B_{12}, C_{12}, D_{12}, E_{12}, in accordance with [2], for the respective initial conditions, which for the period of load acceleration, are: $t = 0$, $x_0 = 0$, $\dot{x}_0 = 0$, $\varphi_0 = 0$, $\dot{\varphi}_0 = 0$

$$\left| \begin{aligned} A_1 &= \left(\frac{M}{m} \pm g\frac{R}{i}\right)\frac{k}{J}.\frac{R}{i} / \left(\lambda^2 + \omega^2\right) \\ B_1 &= \left[\left(\frac{M}{J}.\frac{d}{m} \pm g\frac{d}{J}\frac{R}{i}\right)\frac{R}{i} - 2A_1\lambda\right] / \left(\lambda^2 + \omega^2\right) \\ C_1 &= \left(\pm g - A_1 - 2B_1\lambda\right) / \left(\lambda^2 + \omega^2\right) \\ E_1 &= -B_1 - 2C_1\lambda \\ D_1 &= -C_1 \end{aligned} \right. \qquad (4)$$

$$\left| \begin{aligned} A_2 &= \left[\frac{k}{J}\left(\frac{M}{m} \pm g\frac{R}{i}\right)\right] / \left(\lambda^2 + \omega^2\right) \\ B_2 &= \left[\frac{M}{J}\frac{d}{m} \pm g\frac{d}{J}\frac{R}{i} - 2A_2\lambda\right] / \left(\lambda^2 + \omega^2\right) \\ C_2 &= \left(\frac{M}{J} - A_2 - 2B_2\lambda\right) / \left(\lambda^2 + \omega^2\right) \\ E_2 &= -B_2 - 2C_2\lambda \\ D_2 &= -C_2 \end{aligned} \right. \qquad (5)$$

For the period of load stopping for initial conditions $t_0 = T - t_c$, (t_c – time of the braking process), $x_0 = H - h_x$ (from Fig.1), $\dot{x}_0 = V_{\max} = const$, $\varphi_0 = \frac{2\pi R}{i}$ и $\dot{\varphi}_0 = \dot{\varphi}_{\max}$, the constants A_{12}, B_{12}, C_{12}, D_{12}, E_{12}, are determined by systems of equations (6) and (7):

$$\left| \begin{aligned} & A_1\left(\lambda^2 + \omega^2\right) = \left(\frac{M}{m} \pm g\frac{R}{i}\right)\frac{k}{J}.\frac{R}{i} \\ & 2A_1\lambda + B_1\left(\lambda^2 + \omega^2\right) = \left(\dot{x}_0\frac{k}{J}\frac{R}{i} + \varphi_0\frac{k}{m} + \frac{M}{J}\frac{d}{m} \pm g\frac{d}{J}\frac{R}{i}\right)\frac{R}{i} \\ & A_1 + 2B_1\lambda + C_1\left(\lambda^2 + \omega^2\right) = \dot{x}_0\frac{d}{J}\frac{R^2}{i^2} + x_0\frac{k}{J}\frac{R^2}{i^2} + \dot{\varphi}_0\frac{d}{m}\frac{R}{i} + \varphi_0\frac{k}{m}\frac{R}{i} \pm g \\ & B_1 + 2C_1\lambda + E_1 = \dot{x}_0 + x_0 d\left(\frac{1}{m} + \frac{1}{J}\frac{R^2}{i^2}\right) \\ & C_1 + D_1 = x_0 \end{aligned} \right. \qquad (6)$$

$$\left| \begin{aligned} & A_2\left(\lambda^2 + \omega^2\right) = \frac{k}{J}\left(\frac{M}{m} \pm g\frac{R}{i}\right) \\ & 2A_2\lambda + B_2\left(\lambda^2 + \omega^2\right) = \frac{M}{J}\frac{d}{m} + \dot{\varphi}_0\frac{k}{m} + \frac{k}{J}\frac{R}{i}x_0 \pm g\frac{d}{J}\frac{R}{i} \\ & A_2 + 2B_2\lambda + C_2\left(\lambda^2 + \omega^2\right) = \frac{M}{J} + \frac{d}{m}\dot{\varphi}_0 + \frac{k}{m}\varphi_0 + \frac{d}{J}\frac{R}{i}\dot{x}_0 + \frac{k}{J}\frac{R}{i}x_0 \\ & B_2 + 2C_2\lambda + E_2 = \dot{\varphi} + \varphi_0 d\left(\frac{1}{m} + \frac{1}{J}\frac{R^2}{i^2}\right) \\ & C_2 + D_2 = \varphi_0 \end{aligned} \right. \qquad (7)$$

The sign (-) refers to a case of IQ load lifting, and the sign (+) for a case of its lowering.

Thus, for known height of the hoist *H* and duration of the movement of the hoist vessel *T*, the force in the rope *F* can be determined as a difference between the coordinates of the moving masses, multiplied by the hardness *к*, and namely:

$$F = \left[\frac{R}{i}\varphi(t) - x(t)\right]k \qquad (8)$$

The modeling of dynamic processes and the composing of adequate models for investigation of real objects is a theoretical approach where the volume of natural experimental measurements for evaluation of the functional and operational fitness of the machines is optimized.

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Recommended for publication by Department of Underground construction, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.175-177

LABORATORY STUDIES OF THE LOAD ON TZ-38 TANGENTIAL PICKS

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ABSTRACT
The article summarizes the results of experimental studies related to determining the cutting forces of three types of TZ-38 picks. The test samples, test stand and operating conditions are described and the results obtained have been processed. Possibilities are presented for their practical use on the basis of the conclusions made.

The shearing drums of the Aykov continuous miners used at Babino Mine are equipped with tangential cutting tools of the TZ-38 type (Fig. 1). The loads these picks take up during operation have a considerable effect on the general technical condition of the miners. In order to determine the magnitude of this load under lab conditions, we carried out long-term studies. They aimed at determining the distribution of the resultant cutting force between the hard alloy plate (pin) and the head (holder) of the cutting tool. For the purpose, special picks with elongated heads were prepared (Fig. 1b), in which only the hard alloy plate participated in the cutting process as well as a pick without a hard alloy plate (Fig. 1b) where the cutting was performed only by the holder. Representative fragments from the Babino Mine were used in the experiment in the form of specially prepared rock samples. The tests were carried out on the test stand shown in Fig. 2.

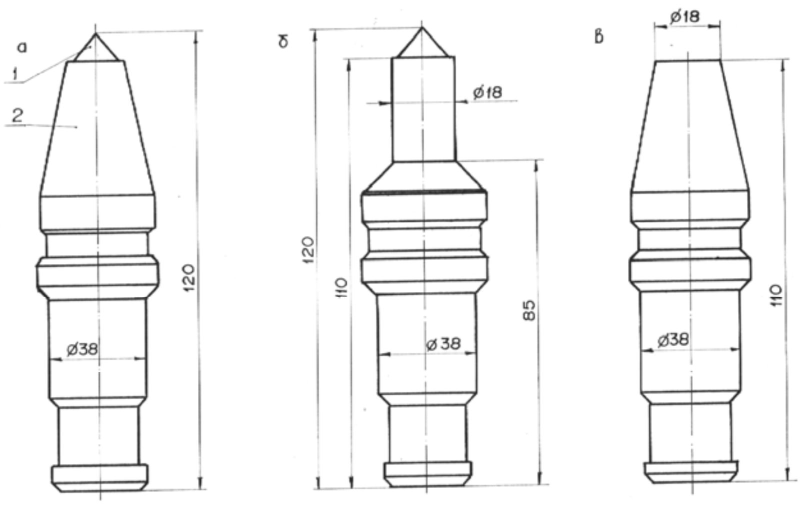

Figure 1. Experimental TZ-38 picks
a – original head; b – pick with elongated head;
c – pick from hard alloy plate

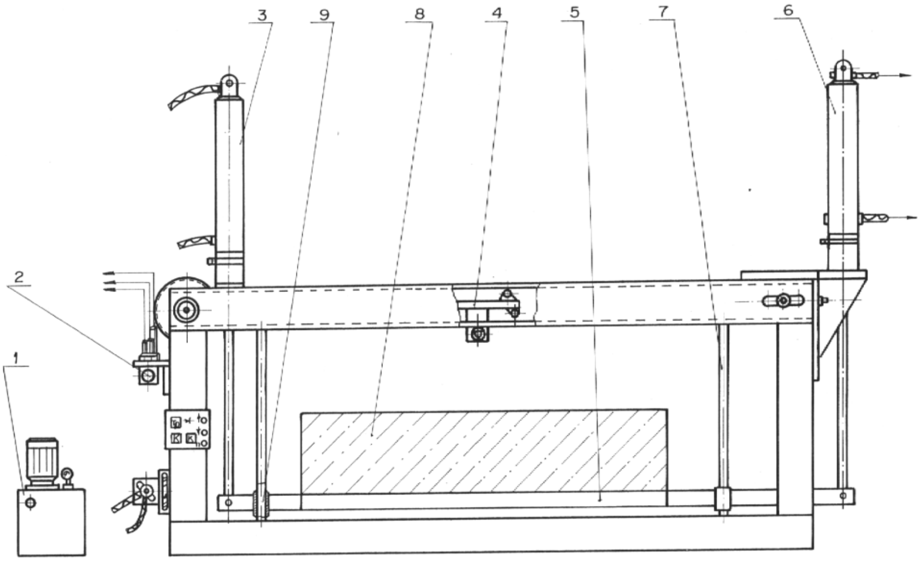

Figure 2.

It enables us to determine the resultant cutting force P_{res} and its three components oriented along the main axes X, Y and Z of the spatial coordinate system, P_x, P_y and P_z, respectively. The stand (Fig. 2) is equipped with a special strain-measuring head 4, driven by the hydraulic motor 2 and the hydraulic oil station 1. The rock sample 1 is moved vertically by the hydraulic cylinders 3 and 6, the platform 5 and the guides 7 and 9. Distributors with regulators were connected to the hydraulic system thus achieving automation of the full-load and no-load run as well as infinitely variable regulation of their rate. The new pick brands enable us to determine the loads taken up by the holder and the cutting part.

The tests were performed under the following operating conditions: blocked longitudinal scraping cutting along smooth surfaces; thickness of sickles – h = 10mm, angle of cutting tool β = 70°-110° changing through 10°; hardness of broken material f = 2.5-3 (according to Protodyakov) and feed rate of shearing drum V_p = 20mm/s.

The system for recording the test results enables us to control and record simultaneously the resultant cutting force and its components. The data obtained from the measuring head are stored in the primary memory of the computer every 0.02 s and during the reverse run are sent to the magnetic memory. The information, presented in a binary code, is processed by programs and the data are retrieved by files and fed to the printer for recording. Along with the printing, the results obtained can be observed on the computer screen. The analog-digital converter, which is used, enhances the metrological qualities of the system, increases its dynamic range and the error is determined by the computer digit capacity being 1/1256 for a 16-bit CPU.

The results of the summarized values of the controlled variables obtained during the experiments are shown in Table 1. In the same table, in percentages, are determined the ratios between the resultant cutting forces for the original and separate types of tested picks.

Table 1

Cutting angle β	Cutting forces, N												Ratio between resultant cutting forces, %	
	Original pick				Pick without head				Pick with elongated head					
	P_x	P_y	P_z	p_{res}	P_x	P_y	P_z	p_{res}	P_x	P_y	P_z	p_{res}	P_{wh}/P_{or}	P_{eh}/P_{or}
70	0.29	1.37	1.67	2.18	0.18	1.02	1.47	1.8	0.11	0.66	0.69	0.96	0.82	0.53
80	0.31	1.39	1.7	2.21	0.21	1.05	1.5	1.84	0.13	0.68	0.73	1.01	0.83	0.54
90	0.32	1.42	1.73	2.26	0.23	1.08	1.52	1.88	0.17	0.71	0.75	1.05	0.83	0.55
100	0.34	1.44	1.76	2.29	0.26	1.11	1.56	1.93	0.19	0.74	0.78	1.09	0.84	0.56
110	0.36	1.47	1.79	2.34	0.28	1.14	1.61	2.04	0.21	0.77	0.81	1.12	0.87	0.55

From the experimental studies carried out it was found that the cutting force is distributed between the hard alloy plate and the pick head. The experiment was necessitated by the processes occurring in the cutting tools used under the conditions of the Babino Mine, where the plate is broken and the entire cutting force is taken up by the pick head.

On the basis of the data obtained we can draw the following conclusions:

1. The resultant cutting force for the original picks is determined by the forces acting on the hard alloy plate and the body.
2. The hard alloy pin takes up approx. 80% of the cutting force and the remaining part (approx. 20%) is taken up by the pick body.
3. The vertical and side forces have a considerably weaker effect on the two pick elements since their participation in the formation of the resultant cutting force is insignificant.
4. The change in the angle of cutting from 70° to 110° has a weak effect on the ratio between the individual components.

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Recommended for publication by Department of Mine mechanization, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.179-183

APPLICATION OF GROUND PENETRATING RADAR IN A MINING ENVIRONMENT

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ABSTRACT
The possibilities and limitations of the ground penetrating radar in quarries and mines are illustrated, using several examples. First, the radar was tested in limestone quarries and in a salt mine to detect karsts (in limestone) and fractures (in both media). In the limestone quarry, the medium velocity could be estimated using a common-mid-plot acquisition. Second, an inversion method based on the frequency content of the radar reflection was developed to characterise the fracture properties (filling material and opening). It was first tested on experiments using limestone blocks with controlled fracture properties. The radar signal analysis showed that both the opening and the filling material could be estimated, if the openings are not too small compared to the wavelength. The inversion method was then applied to in-situ data registered in a salt mine to estimate the opening of open fractures present in the roof of the drifts. Boreholes were available to compare the estimated opening with the real opening. Most of the results were satisfactory, although some poor results were also obtained. It can be explained by geological factors or it can also be due to the method itself.

INTRODUCTION

The radar method is based on the propagation and reflection of high frequency electromagnetic waves (20MHz-2GHz). The technique is widely described in Daniels (1996). The radar method provides images of non-conductive media at shallow depths. The reflections are due to a contrast in the dielectric properties. The radar pulse has a large bandwidth to achieve a high resolution. The penetration, which can reach several tens of meters in resistive environments, decreases when the used frequency increases. The quality of the profiles is well improved by using processing techniques like filtering, automatic gain control, background removal and migration.

The principle of the method used at the surface is presented in Figure 1. A transmitter antenna and a receiver antenna are moved along a profile with constant transmitter-to-receiver offset. A planar reflector will give a planar event on the radar profile. Heterogeneities (e.g. a hole) with smaller dimensions than the wavelength result in a hyperbolic event on the radar profile. The method can also be used in reflection from a borehole or in transmission between two boreholes. In the first case, the analysis provides an image of the discontinuities around the borehole. In the second case, the analysis provides a map of the velocities and attenuation between the two boreholes. The velocity of the electromagnetic waves can be estimated by permittivity analysis of samples in laboratory. In-situ the medium velocity can be estimated by a common-mid-point (cmp) acquisition, whereby the distance between the transmitter and the receiver increases with regular steps during the acquisition. Discontinuities parallel to the surface will result in a hyperbolic event on the cmp plot. The shape of the hyperbolic events is related to the propagation medium velocity. Flat hyperbolas correspond to media with higher velocity. For a same velocity, deeper events are also more flat. Using those considerations, it is possible to estimate the mean propagation velocity with a satisfying accuracy. With the knowledge of the velocity, the time scale can be converted in a depth scale. The electromagnetic waves are attenuated by the medium. For very conductive media, the method becomes non-effective (e.g. in clay, loam).

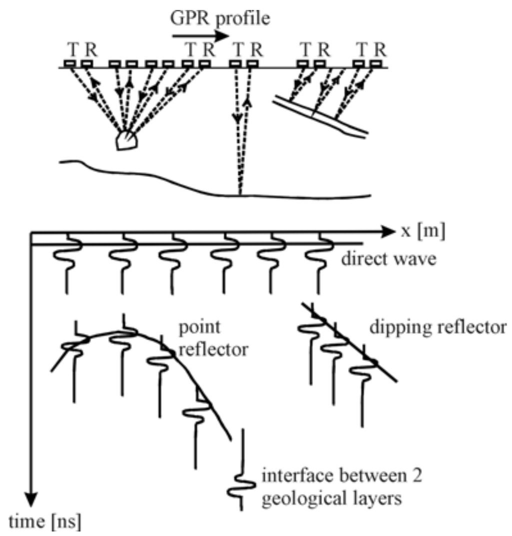

Figure 1. Principle of the radar method.

APPLICATIONS

The radar method is effective to detect discontinuities (fractures, bedding planes, lithologic contacts,…) in resistive media such as limestone, granite, salt,… (Dubois, 1995; Halleux, *et al*., 2000). The penetration and the resolution obtained depend on the used frequency.

In a first test, radar data were collected in limestone to detect and locate open fractures (Grégoire and Halleux, 2002b). The 100MHz radar antennas were used to insure a good penetration. The profile is presented in Figure 2. A penetration of about 20m could be obtained. A first fractured area was located between 5m and 10m from the surface. The individual fractures within this zone could not be differentiated due to the resolution. A reflection corresponding to an isolated fracture appeared at about 350ns on the radar profile corresponding to a depth of about 17.5m.

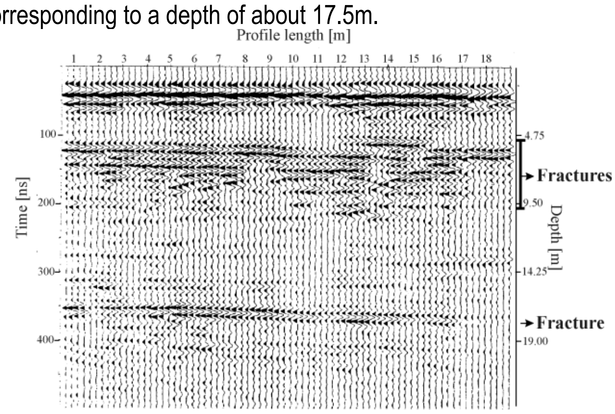

Figure 2. Radar profile carried out in limestone. The frequency used is 100MHz.

The velocity of the electromagnetic waves in the limestone (100m/μs) was established using common-mid-point measurements (Figure 3). On Figure 3, the first event corresponds to the direct wave propagating from the transmitter to the receiver without penetration in the investigated medium. Two groups of hyperbolas can be observed on the cmp plot. The first group corresponds to reflectors in limestone. The second group corresponds to obstacles at the surface. The hyperbolas are flat due to a higher propagation velocity in air. The shape of the first family of hyperbolas is used to determine the velocity of the electromagnetic waves in limestone (in this quarry 95-100 m/μs).

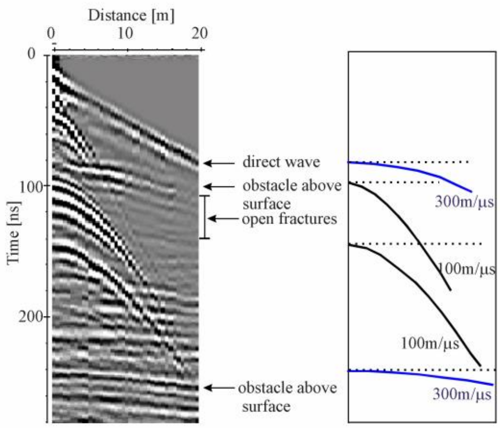

Figure 3. Common-mid-point plot in limestone (100MHz).

Another example shows the ability of GPR to detect and locate karstic areas in a working limestone quarry of ornamental stone. The stone is locally called 'Petit Granit'. The 'Petit Granit' is a compact encrinidic limestone from the Tournaisian serie. The operator in the limestone quarry was interested in the detection of possible karsts prior to the excavation (Grégoire and Halleux, 2002a; Grégoire and Halleux, 2002b). Such analysis allows to estimate the volume of good limestone which can be extracted. Several areas in the quarry were investigated. The frequency which is chosen for the measurements depends on the location in the quarry.

The first investigated area was characterised by the presence of a thin clay layer at 4m depth. The penetration below this layer was not possible. The 500MHz antennas were used because they insure a penetration of 4m and to improve the resolution in the investigated area. The first profile (Figure 4) was registered in an area where the limestone was good. The radar profile was compared to the different limestone banks present in the quarry. These banks are characterised by different limestone quality (Figure 4 at right) and are separated by very thin bedding planes. The horizontal events on the radar profile correspond to these bedding planes. Although these reflectors are very thin (order of size of millimetres), they were detected by the radar. The event located at 50ns does not correspond to a real event in the ground. It is a multiple of the reflection at 25ns.

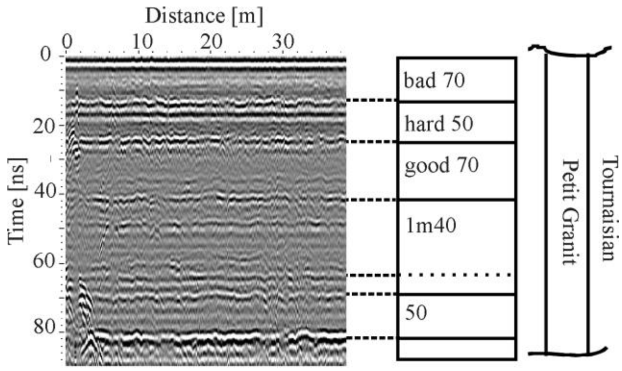

Figure 4. 500MHz radar profile carried out in limestone (left). Comparison with the stratigraphy of 'Petit Granit' (right) and various banks present in the quarry.

The second radar profile (Figure 5) is carried out in the same quarry but in a different area where karsts are present. The reflections are due to contrast in permittivity between the limestone and the altered areas. The shapes of the reflection are due to the irregular interfaces of the discontinuities. From the radar profile, it can be deduced that the karstic areas are filled with a conductive filling material (mixing of clay and sand). Indeed, the signal is very attenuated below the reflections. This signal attenuation can also be observed on the discontinuity of the reflection corresponding to the clay layer.

Radar profiles were carried out in two perpendicular directions in order to get a good coverage of the investigated area. After processing and analysing of the profiles, the location of the karsts was mapped in 2D with information about the extension over depth. The analysis of a radar profile allows the determination of extension of a karstic area with depth but not a decrease of the size of the karstic area. This interpretation could be verified after extraction. It was quite satisfactory, except for an overestimation of the karstic areas.

Another example of the use of the radar technique is situated in a salt mine. The radar is efficient in salt to image the fractures or geological discontinuities (Halleux, *et al.,* 2000).

Two years ago, a trial was conducted in the Werra mine of K+S KALI GmbH to use the ground penetrating radar as a possible method to detect fractures in the roof of the drifts. These fractures sometimes occur in horizontal bedded salt up to approximately 3 m above the roof because of stress relief due to mining. When fractures are detected, the roof has to be scaled mechanically or anchors have to be installed to prevent blocks falling. The early detection of these fractures is therefore very important. A car is equipped with a control unit (4 channels) and a hydraulic arm to lift a maximum of 4 antennas near the roof of a drift. Two 900 MHz Gssi antennas are installed. A working distance of 0.5 to 1m from the roof gives a satisfactory signal quality. The survey speed is about 15km/h so that an extensive survey can be realised in a relatively short time period.

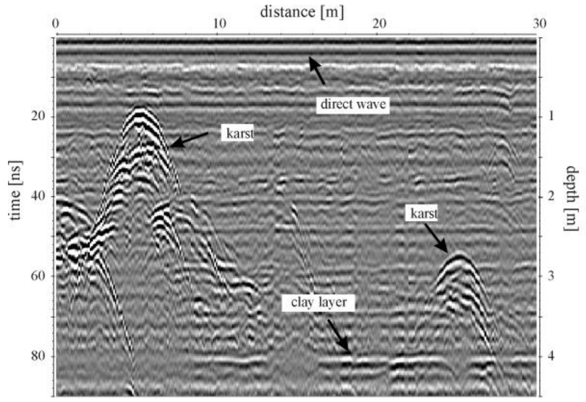

Figure 5. 500MHz radar profile carried out in limestone.

Figure 6 shows an example of radar profile registered in the salt mine. A penetration of 2 or 3m is obtained in salt. The first reflection corresponds to the reflection of the roof of the drift. The reflections which are indicated by an arrow correspond to fractures present in the salt.

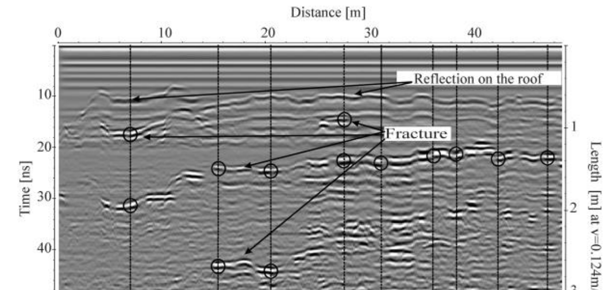

Figure 6. Radar profile carried out in a salt mine at 1GHz. The antenna is not located against the roof of the drift.

These examples have shown that the radar method was effective to detect and locate discontinuities like fractures, bedding planes, or karsts in limestone and salt. Nevertheless, the analysis of the profile does not give information on the fracture properties (opening and filling material).

ESTIMATION OF FRACTURE OPENING

In order to characterise the fractures, a more detailed analysis of the radar signal is required.

Theory

The reflection coefficient at the interface of two semi-infinite media (1) and (2) is defined in function of the wavenumber *k:*

$$R_{12} = \frac{k_1 - k_2}{k_1 + k_2} \qquad (1)$$

where k_1, k_2 are the wavenumbers related to medium (1) and medium (2).

The wavenumber k is related to the electric and dielectric properties of the medium by:

$$k = \omega\sqrt{\varepsilon_e \mu} \qquad (2)$$

where μ the magnetic permeability
ε_e the effective permittivity

The effective permittivity is frequency dependent. The signal radar being broadband, it is important to take into account this frequency dependence. The Jonscher model with 3 parameters is used to characterise the frequency dependence of the effective permittivity (Jonscher, 1977). It means that 3 parameters are required to describe the dielectric behaviour of a filling material. Due to the frequency dependence of the effective permittivity, the reflection coefficient is also frequency dependent.

A thin layer is characterised by a thickness smaller than the vertical resolution. It is not possible to differentiate the reflections on both interfaces. The reflection coefficient due to a thin layer (<λ/2) corresponds to the sum of the reflections on each interface. This is illustrated on Figure 7 (Hollender and Tillard, 1998).

$$r_{12} = \frac{R_{12} + R_{21}.e^{-i\varphi}}{1 + R_{12}.R_{21}.e^{-i\varphi}} \qquad (3)$$

where $\varphi = 2k_2 d$ and d the fracture width.

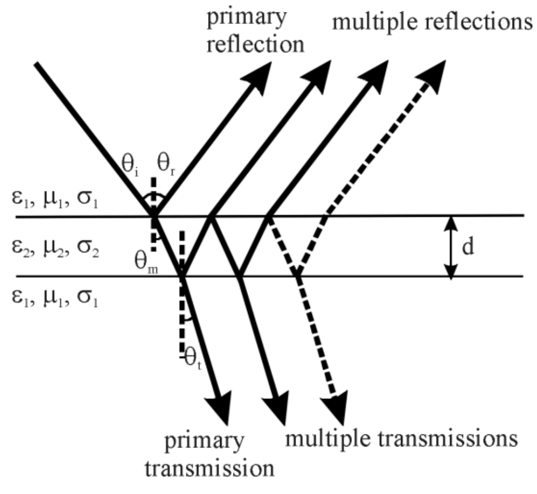

Figure 7. Reflection coefficient due to a thin layer.

The reflection coefficient is represented on Figure 8 and on Figure 9 respectively for a fracture filled with dry sand (perfect dielectric material, relative permittivity equal to 2.5) and filled with clay (conductive material, relative permittivity equal to 55). The wavelength in clay is smaller than the wavelength in sand (at 1GHz, the wavelengths in sand and clay are respectively equal to 0.19m and 0.04m). The different curves correspond to several openings (1mm, 10mm, 50mm and 100mm). In the case of dry sand as filling material, the shape of the reflection coefficient is linear as a function of the frequency for fracture openings of 1mm and 10mm. The reflection coefficient presents extremes for larger openings. In the case of a fracture filled with clay, only the curve related to a fracture of 1mm is linear in function of the frequency. The curves corresponding to larger openings are smoothed due to the high conductivity of the clay.

The figures show clearly that the reflection coefficient in the frequency domain is very sensitive to the fracture properties (opening and filling material characterised by the permittivity).

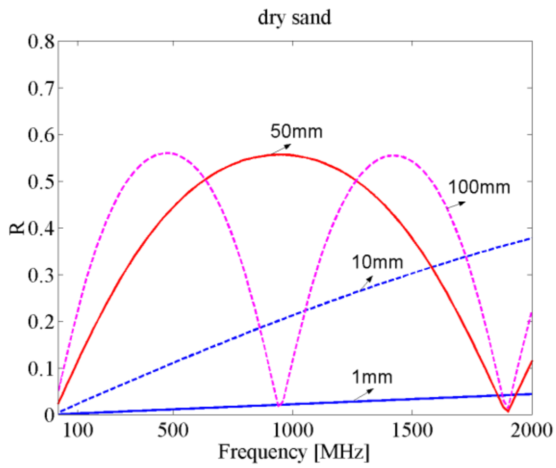

Figure 8. Reflection coefficient in the frequency domain due to a thin layer filled with dry sand.

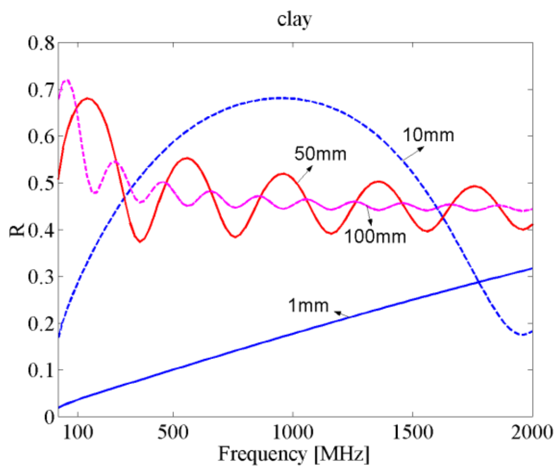

Figure 9. Reflection coefficient in the frequency domain due to a thin layer filled with clay.

It was proposed to study the radar reflections on fractures in the frequency domain in order to characterise the fractures (Grégoire, 2001a; Grégoire, 2001b).

Application of the inversion method to real data

Experiments were carried out to simulate fractures with controlled properties between two limestone blocks. Fractures with various openings and various filling materials were simulated. These measurements aimed to test the theoretical formulation of the reflection coefficient. From this analysis, an inversion method was developed to be applied to the radar reflection to estimate the fracture opening and the filling material (effective permittivity characterised by the 3 Jonscher parameters).

To test the inversion method on in-situ data, complementary testing measurements were carried out in the Werra mine of K+S KALI GmbH. The inversion method was applied to the radar reflections to estimate the opening of fractures present in the roof of the drifts (Grégoire and Halleux, 2002a). These measurements aimed also to see the influence of the acquisition speed and to evaluate the effect of the antenna-roof distance on the data quality.

Static (antenna fixed at one position) and dynamic data (moving antenna) were collected at two sites of the mine. Two radar systems were used (a 900MHz Gssi antenna and a 1GHz Ramac antenna). Fifteen boreholes were available on the two sites to compare the estimated fracture openings with the measured fracture openings. The fracture openings vary from 1mm up to 90mm. Complex fracture systems, characterised by fractures located very close to each other, could be observed in several boreholes. The minimal distance to separate the fractures is the vertical resolution (62mm in salt at 1GHz).

A dynamic profile is represented in Figure 6. The location of the boreholes is indicated. The first energetic reflection corresponds to the reflection on the roof. The other reflections correspond to fractures. It can be observed on the profile that the reflection amplitude can vary very fast with the distance. The penetration in salt is about 3m.

The inversion method is applied to the radar measurements to determine the opening of the first individual fracture in the roof. As reference signal, the reflection on the roof is chosen. This signal has a high signal-to-noise ratio and is quite stable, except in areas with irregularities in the roof. The dielectric parameters of the salt being known (ε= 5.85, σ= 0.5 mS/m), the corrections related to the propagation in air and salt can be applied to the signal. When the first fracture is very thin and not detectable, the second fracture is analysed.

The results of the inversion method are presented in Figure 10. Most of the results are satisfactory although poor results are also obtained. It can be explained by the fast spatial variation of the fractures (the position of the profiles was not accurately defined), the irregularity of the fracture, the complex fracture systems composed of several fractures or the quality of the reference signal. It can also be due to the method itself for opening ranges where the reflectivity is less sensitive to opening variations (Grégoire, 2001a). The reliability of the results should be improved using a statistical analysis. Although the method is not perfect, the knowledge of the fracture is improved.

CONCLUSION

The possibilities and limitations of the radar technique in quarries and mines are illustrated, using several examples. The method is efficient in media such as limestone and salt to detect fractures, bedding planes or altered areas. Geological discontinuities can also be imaged. The penetration depends on the used frequency and is particular to each site.

The frequency content of the radar reflection was analysed and compared to a reference signal to characterise the fracture properties. An inversion method was developed and first tested on laboratory experiments with controlled fracture properties.

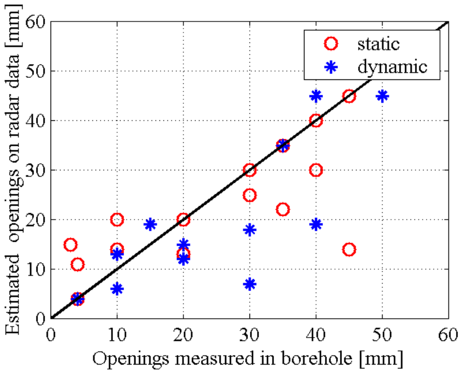

Figure 10. Results of the inversion method applied to the radar data collected in the Werra mine.

Both the opening and the filling material can not be estimated if the openings are too small compared to the wavelength. The ratio depends on the used frequency and on the filling material.

The inversion method was applied to in-situ data registered in a salt mine to estimate the opening of open fractures present in the roof of the drifts. Most of the results are satisfactory although poor results are also obtained. It can be explained by geological factors or it can also be due to the method itself. The reliability of the results should be improved using a statistical analysis.

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ACKNOWLEDGEMENTS

We would like to thank the quarry of Clypot in Neufvilles (Belgium) for the help with the limestone block experiments and especially Francis Kirten for his interest and the organisation of the measurements.

We would also like to thank Dr. Lukas (K+S KALI GmbH) for giving us the authorisation to publish the results of the radar measurements in the Werra mine.

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50 years **University of Mining and Geology "St. Ivan Rilski"**
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.185-187

ON SOME METHODS FOR SURFACE EROSION CONTROL ON TAILINGS PONDS AND WASTE FLY-ASH PILES

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ABSTRACT
An important issue for the mining, metallurgical and power supply industries is the control of fugitive dust emissions. This paper intends to demonstrate the outcomes of researches, assessment of potential and effectiveness of some dust control methods applied on fine-grained tailings deposited by mineral processors, waste fly-ash resulted from coal-firing at power stations, slimes discarded by steelworks. The attention is paid essentially on the cases concerning creation of protective covering films on industrial waste sites by application of acetate polymer dispersion, sold under trademark Terra-Control SC 823. The long-lasting protective film was formed at a reagent dosage rate 30-50 g/m^2 and was stable exposed up to 20 m/s speed to wind erosion during the period of 6-8 months. Terra-Control SC 823 was used also as a surface layer stabilizer within vegetation of tailings. Terra-Control SC 823 has formed initially three-dimensional permeable matrix at a dosage rate 15-20 g/m^2, allowing germination of tolerant grass species straight on deposited wastes. The seed material was distributed onto tailings at dosage rate 15-20 g/m^2. Terra-Control SC 823 reduces moisture loss and protects plants and wasteland from drying out, along with allowing water and oxygen to penetrate. Terra-Control SC 823 is readily biodegradable, thus enhancing nutritious medium for the species.
The results of conducted modeling trials for preparing long and short-term dust suppression projects are presented in this contribution, and as well conditions and the effect of implementation of erosion control projects on two industrial wastelands.

INTRODUCTION

Exposed surface layers, containing fine grain particles, of wastes discarded from mineral processors, steelworks, coal-fired power stations and other production units, are subject to regular erosion by wind and water, causing in this way the environmental pollution. There are many techniques controlling fugitive dust emissions. The efficient dust suppression methods were analyzed by assessment of efficiency of each one and as well required upkeep maintenance for protective coverings (Hadjiev and Tzekova, 1979). The goal of the investigation is the application of polyvinyl acetate water dispersible latex reagent solely and in combination with vegetation for short-term and long-term erosion control at surface layers of the tailings ponds and waste fly-ash dumps.

METHODS AND MATERIALS

The studies have been conducted on the soil and industrial wastes. Table 1 presents description of the samples.

Sample 1 was taken from the surface layer of the non-operational part of a tailings pond. This part is out of use for the reason that the thermoelectric power station was refurbished of from coal to natural gas firing. The projected filling up capacity of the tailings pond has not been achieved. It is considered that the deposited wastes will be recycled in the near future. The total surface area of the tailings pond is 80 hectares.

Table 1. Description of the studied samples.

No	Name, origin and source of collected sample
1	Slimes generated by wet dust collectors of blast furnace, basic oxygen and electric arc furnace; cinder and fly-ash discarded from thermoelectric power station and other production lines at a steelworks, disposed of on a tailings pond
2	Cinder and fly-ash left behind lignite coal burning at a thermoelectric power station
3	Sample collected from the water slope of the tailings dam of a copper mineral processor
4	Soil used in re-vegetation of the air slopes of the tailings pond of a copper mineral processor

Sample 2 was taken from a surface layer of the non-operational section of a waste fly-ash dump. The surface area of this part is 100 hectares.

Sample 3 was taken from the water slope of a dam of a filled up tailings pond. The lagoon is operated as a return water dam of working tailings pond.

Sample 4 was taken from the topsoil used in re-vegetation of the air slopes of a tailings pond.

The trials have been carried out with a binding agent, sold under trademark Terra-Control SC 823, and a mixture of grass seeds. Terra-Control SC 823 is synthetic resin dispersion based on polyvinyl acetate formulation in water. The product is normally diluted with water in most ratios. It is compostable, medium term biodegradable and non-phytotoxic agent. Non-toxic by-products are produced in its biodegradation. The technical data of Terra-Control SC 823 are density (20°C) 1.1 g/cm^3; solid content 57.5÷1.5%; pH=4÷6; viscosity (20°C) 25000±3000 Pa.S; and processing temperature above 5°C.

Terra-Control SC 823 has been developed and is manufactured by Cognis Corporation in USA.

The polyvinyl acetate dispersion was diluted to 10% with water and sprayed at 15g/m^2 dosage rate onto the wastes by means of a helicopter type Kamov 26. The conditions of the trial were as follow: fly altitude above the treated ground surface 2m; helicopter's velocity during the procedure 40 to 60 km/h; width of the treated surface strip 25 m; duration of operation 6-7 min; reagent tank volume 600 l; optimal solution consumption 20 l/s; flights per one hour 9; spray nozzle diameter 4 mm.

The mix of grass seeds, cultivated and produced by company DSV in Germany, consists (in %) of species Festuca rubra rubra 25, Festuca ovina 20, Lolium perenne 15, Festuca rubra trich. 15, Poa pratensis 20, Agrostis capillaries 5.

Both the binding agent and the mix of grass seeds were supplied by Gea International in Bulgaria.

The investigations have been performed taking into account the previous published methodical considerations (Hadjiev and Hadjiev, 1996). The vegetation trials have been performed after the well-known technique (Voznuk et al., 1985).

RESULTS

Table 2 presents the results of sieve and sedimentation analysis performed on the studied samples.

Table2. Granulometric analysis of the studied samples.

No in order	Fractional assay, %			
	Grain size, mm			
	> 1.0	1.0 - 0.4	0.4 - 0.01	< 0.01
1	0.00	16.07	59.36	24.57
2	0.00	14.00	85.32	0.68
3	1.67	17.37	80.48	0.48
4	11.35	15.65	45.70	27.30

The vegetation experiments have been carried out at a fixed dosage rate of grass seeds and dosage rate of a binding agent from varying 15 g/m^2 to 100 g/m^2. Table 3 presents data concerning the highest germination of the seeds and period to achieve maximum value, applying smallest amount of a binding agent.

Table 3. Period for achievement of maximum germination, applying 15 g/m^2 Terra Control SC 823.

No in order	Maximum germination, %	Period, days
1	61.11	30
2	72.22	35
3	52.83	48
4	92.45	28

The germination of the sown seeds into the sample 3 was raised by capping with soil and addition of water-based fertilizer.

Based on the assessment of local climatic factors and results obtained from laboratory scale investigations two projects were prepared. The company Gea International Ltd bid two tenders for dust suppression control on two waste sites offering these projects and undertook contracts for both of them. Both contracts were completed successfully.

Project 1 for long-term dust suppression of the wastes deposited into the non-operational part of the tailings pond of Kremikovtzi ShH Company

The slime generated from wet dust cleaning installations of blast, basic oxygen and electric arc furnaces, coke plant and other production lines are dumped into the tailings pond of the Kremikovtzi ShH Company. Lignite coal was combusted at the company's power plant until 1988 year, and waste fly ash and cinder were dumped into the part of the pond, which is non-in use now.

For the completion of the project, 14.170 tons binding agent Terra-Control SC 823 and 10 tons grass seeds were spent. The application dosage rates of the binding agent and seeds were respectively 17.71 g/m^2 and 12.5 g/m^2. It has been guaranteed long-term protection of the covering layer. For comparison, according to the second placed offer in the tender, it was proposed 40 tons water dispersible acrylic-based formulation binding agent to be used at dosage rate 50 g/m^2. It has been guaranteed providing up to 12 months of holding power and suppression dust.

Since July 1996 year, the project of Gea International Ltd is being put into effect. From that point to the present time (2003 year), it is not observed any dust drifting over the area around the tailings pond of Kremikovtzi ShH Company. The vegetated spot of the tailings site is not allowed for livestock grazing. There are examinations to establish commercial production of grass seeds from that site.

Project 2 for treatment of the dry section of the waste fly-ash pile of coal-firing power station Mariza Iztok 2

Table 4 presents some figures contained in the offers of four bidding companies in a tender for dust suppression treatment of the waste fly-ash pile of coal-firing power station Mariza Iztok 2. Number one is the offer of Gea International Ltd.

Table 4. Data offers of participating companies in the tender

No in order	Total sum (reagent and treatment), USD	reagent dosage rate, g/m^2
1	113 927	30.16
2	230 000	-
3	260 000	96.00
4	400 000	100.00

The project was implemented as the waste fly-ash pile surface was treated with 30.16 tons polyvinyl acetate dispersion Terra Control SC 823 to prevent dust drifting. The product was applied via helicopter at a rate of 30 g/m^2 and a 1:10 dilution rate. The dust was completely suppressed for the period to the next following put into operation. The requirements of the client were fulfilled completely. They included a usage of non-hazardous chemicals, creation of a short-term protective permeable covering film.

CONCLUSION

The performed investigations provide appropriate solutions for prevention of dust emissions triggered by wind erosion on tailings ponds and waste fly-ash piles in the following cases:

- It is necessary to apply temporary protective covering films on surface sections non-in use on waste disposal site.
- Protective covering layer has to establish long-term dust suppression action and there is soil available for disposal
- There is large amount of particles below 0.01 mm in the surface layer on the wasteland, and vegetative species are tolerant to the containing compounds.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.189-192

WASTE FROM HEAT ELECTRIC POWER-STATIONS WITH SOLID FUEL – ECOLOGICAL PROBLEM AND RAW MATERIAL

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ABSTRACT
In the first part of the paper an evaluation of the ecological danger from direct throwing out the cinder and fly ash (CFA) from heat electric power- station (HEPS) is presented. The possibilities of decreasing and eliminating the ill effects of the now practiced way of putting the CFA in depots are offered.
The second part concerns the possibilities for utilization of the CFA, based on the foreign and our experience. As a result of authors' research it is proved that the waste – CFA, may replace successfully the sand in underbasic concrete of temporary roads in the open mines.
A composite material and technology for cinder waste pools' covering are made on the CFA base – a radical step against dust emission and leaching heavy metals from them.

EVALUATION OF THE ECOLOGICAL DANGER FROM DIRECT THROWING OUT THE CFA FROM THE HEAT ELECTRIC POWER-STATIONS (HEPSs) OF "MARITSA IZTOK" MINES AND POSSIBILITIES FOR ITS PREVENTING.

Introduction to the problem

The production of electrical energy from lignite coal of joint-stock company (JSC) "Maritsa iztok" mines, concerning to the "dirty and risked" production, is a reason for considerable ecological problems and most of them are connected with the CFA from the HEPSs. The problem concerns the quantity as well as the qualities of the cinder, fly ash and gypsum.

It becomes clear from "Energoproekt" data that the quantity of the waste is 6-7 million tones a year. The problem "quantity" finds palliative decision in mixing the waste with the overburden cover and their putting in depots on external dumps.

The "Minproekt" research [1] shows that the mixing of the waste – cinder and fly ash (C + FA) with clays of the overburden cover approves the physical and mechanical properties of the mixture as the fly ash has stabilizing effect. This technology is accepted and is being realized.

Here the question of waste's qualities (C + FA) will be concerned. As a result of research of Direction "Scientific investigations", Section "Environment preservation" for defining the physical, mechanical and chemical indicators of (C + FA) [2] a conclusion is given: "the founded harmful components (according to Bulgarian State Standard BSS 5795-81) are: the unburned particles in the cinder are 9.17% average, in the fly ash - 5.49% average and water soluble salts in the fly ash are 2.47% average. This research is not finished yet, the data for contents of heavy metals and their water soluble connections in the C + FA, whose dust emission and leaching from the waste dumps will threaten soils, waters and air near them, is not given.

In the report for evaluation of effect on the environment (EEE) for HEPS "Maritsa iztok" 2 [3] it is pointed out that " the waste pools are a potential pollutant of the environment during the exploitation as well as after stopping their using before conservation and recultivation". Further in [4], (table 5, p. 24) the contents of heavy metals in the fly ash from the waste pools and after electrical filters are presented. The presence of harmful water soluble salts, the data for heavy metals in the cinder and evaluation of the ecological danger from putting in depots the C + FA in waste pools and after dehydration in the external dump "Staroselets" are not given here.

The wide-ranged research [8] shows that the presented disseminated elements (Zn, Cu, Ga, As, Se, Mo, Cd, Sb, W or Pb) in the coal increase their contents on the external surface of the particles with decreasing their diametre and it is essential for leaching processes.

Here the main conclusion is:

Putting the cinder from the HEPSs of coal in depots and undoubtfully its removal, as a rule, is connected with harmful substances' releasing. The quantity of released harmful substances depends mainly on the form (way) of fly ash's removal. Thus, for example, if the fly ash is stored in one ordinary depot and if there is a contact with the underground waters, this leads to releasing harmful substances, which may bring to intolerable pollution of the environment"[8].

Wind and atmosphere waters contribute to spreading harms of CFA when the present technology of their putting in depots is used.

Our investigation, done in authorized laboratories, shows the following contents of heavy metals and $SO4^-$ (table №1) in CFA from HEPS –2.

Table № 1

Serial №	Lab. №	Pb mg/kg	Cu mg/kg	Zn mg/kg	Cd mg/kg	Ni Mg/kg	Co mg/kg	Bi mg/kg	Ag mg/kg	As mg/kg	Sb mg/kg	Mo mg/kg	Hg mg/kg	SO4- mg/kg
1	13639	14	180	60	<1	80	18	<20	<1	36	<10	39	<0.05	8765.0
2	13640	30	159	73	<1	87	26	<20	<1	86	<10	38	<0.05	16337.5

Continuous leaching of heavy metals by atmosphere waters brings to their gathering in soils and waters near depots for CFA.

Possibilities for decreasing or eliminating the ecological danger from direct throwing out the CFA.

According to the present level of the ecological science and technology the radical decisions against spread of harms, contained in the waste C + FA are two:

The first one is its putting in depots to be done in ones, which are preliminarily provided by bedded and lateral seal stopping and at the end – by covering stopping.

The second one is immobilization (inertia) at first and putting the harms from CFA in depots after that using the present technology. This decision technologically is easier for execution and demands fewer funds.

Investigations [8,9,10] are done and their results show that at mixing in advance and certain proportions of C + FA, gypsum and water from the sulphur purifying, a solid body is formed and it is called stabilizat. Salts and disseminated elements (heavy metals), included in the stabilizat, leach only in very limited scale (quantities). Contained salts' concentrations are under limited values for the responsible class of depots" [8], i.e. under top allowable concentration (TAC).

The "stabilizat", got by us, was on the base of C+FA+G+lime+water. The samples from it showed satisfactory compressive strength (0.3÷1.0 MPa), good water soluble resistance and dust emission. Leaching the heavy metals from the stabilizat has to be researched additionally as well as its formation on a base of hydraulic binders and additions.

UTILIZATION OF THE CINDER, FLY ASH AND GYPSUM – WASTE PRODUCTS FROM HEPS OF THE JOINT-STOCK COMPANY "MARITSA IZTOK" MINES

Introduction to the problem

Waste from the production of electrical energy from brown and lignite coal in form of cinder, fly ash and gypsum (C + FA + G) are a problem for all countries, engaged with such activities. As we mentioned above, big areas are necessary for its putting in depots. They are ecological danger for soils, waters and air because of their dust emission and leaching of harms, which contain heavy metals and their combinations as soluble salts and because of high content of sulphates.

Since the 80s of the last century the waste – C + FA + G has mainly treated as a raw material in order to be saved areas for its putting in depots and kept deficit materials such as sand, stone, clays, etc. Because of the heavy metals and other harms presence in the C + FA, a method for immobilization these harms by means of suitable C + FA, gypsum and rarefied water mixing was developed.

The fields of real using the C + FA as a raw material – a component in construction materials and replacing some of them, for example the sand and stone, are systemized clearly in [8] (see table№2). Here we must add VIII.6 – the use of gypsum for creating stabilizat and immobilization the harms in the waste – C + FA before its putting in depots.

It became clear above that our practice for treating the C + FA does not correspond to that of the developed countries. Putting the C + FA in depots together with the overburden clays is a danger for the uncontrolled pollution of air, soils and waters, near and under the waste dumps with heavy metals and other harms through wind. The infiltration of atmosphere waters in the overburdens' body causes leaching of harms and from there they have their way as a polluted drainage waters to the near situated soils and underground waters. On the other hand, raw materials such as C + FA are being fully and irrevocably buried.

The data analysis of table 3 for the evaluation of effect on the environment of HEPS-2 [4] shows the presence of significant quantities of heavy metals in the cinder, thrown out to the waste pool. The following is written there: "When there are strong winds (with speed more than 5 m/s), the raising of such dust in the air is dangerous because of the presence of lead combinations, which reach the level of 25 mg/kg in the material of the waste pool.

Utilization of the cinder, fly ash and gypsum – waste products from the HEPSs

The eighth fields, shown in table №2 for using the cinder, fly ash and gypsum (C + FA + G) are absolutely realizable in our country. VIII.4 is being adopted in semi-industrial scale in recultivation of post-mining areas. VIII.6 and II.1 will be realizable soon. The formation of stabilizat through mixing the C + FA + G and the water from the sulphur purifying before their putting in depots together with the overburden eliminates one considerable potential danger and accumulates part of the gypsum, useless for other purposes.

One of conclusions in the report of evaluation of effect on the environment of HEPS-2 [4] is that "it is necessary to be foreseen and planned concrete investigations, connected with the possibility of waste calcium solutions from the sulphur purifying installation of the smoke gases from block 7 of HEPS-2. The possible positive results will help a traditional problem for putting the waste from the sulphur purifying installation in depots to be solved.

The received by-product from the sulphur purifying is gypsum with quantity of 300÷350 thousand tones a year.

TABLE №2.
FIELDS OF USING THE CINDERS FROM THE HEPS

Fields of using	Purpose of using
I.Production of cement and concrete	1.Addition to concrete for its production (sign for cinder testing). 2.Raw material and initial supply of energy during cement production. 3.Raw material for artificial cinder agglomerate which substitutes the additions to concrete – stone and felt. 4.Raw material for binder solutions.
II.Small blocks for construction	1.Raw material for small blocks used in construction (for example, light concrete small blocks and bricks in mixture with limestone and sand).
III.Highway and road construction	1.Filler in bituminous covering and bearing courses 2.A component part in hydraulic connected bearing courses (HGT). 3.Hydraulic binder cinder for reinforcement of soils as bearing, cold-preventing course in the upper and down structure. 4.Raw material for artificial cinder agglomerates (granules) which substitute stone and felt.
IV.Building and exploitation of depots	1.Material for stabilization or reinforcement of problem slimes (for example, $CaSO_3/CaSO_4$ – slimes from purifying (clarifying) and excavating). 2.Material for compaction of depots' bottoms (bottom isolation). 3.Material for covering and recultivation of depots (for example, depots for solid waste of life and mining halds). 4.Material for embankments and roads on depots. 5.Material for putting out mining fires in the halds. 6.Material for binding the dust in the mining halds.
V.Landscape forming	1.A component part of the filling in landscape forming (for example, in filling stone pits and building sound isolating shafts (embankments).
VI.Underground construction	1.Raw material for solution, respectively hardener for filling. 2.Using material for embankments.
VII.Raw materials providing	1.Raw material for metals extraction (for example, Al, Fe).
VIII.Others	1.Using cenospheres for special construction materials. 2.Filler for covering isolated bituminous layers. 3.Isolating material in metallurgic industry. 4.Way for improving soils' qualities. 5.Raw material for special glass (porous foamy glass). 6.Receiving a stabilizat during putting C + FA in depots in the waste dumps with immobilization of heavy metals and other harms.

In conclusion to the report for evaluation of effect on the environment for the project "Completion of block №8 and building sulphur purifying installation" [5] three possibilities for using the gypsum in sulphur purifying are considered:

- Production of gypsum with commercial qualities for using by final consumers;
- Putting in depots in the waste pool;
- Using gypsum for filling waste pits.

In that case a spreading the spectrum for using the waste gypsum according to the practice of developed countries, namely **production of light blocks** for construction on C, FA, G base plus certain additions is offered. It will take considerable part of the waste and save putting valuable raw materials (clay and sand) in present production of blocks and bricks. The necessary investigations for realizing that production are:

- Making and check the execution of prescriptions for production of construction elements;
- Defining the physical, mechanical and chemical properties of these elements, namely: volume weight, strength indicators, water absorption, resistance to cold, resistance to heat, thermal isolation qualities and other requirements to them according to our standards;
- Making a production technology;
- Evaluation free on production workshop.

The fields III.3. and IV.6. are very perspective too (see table №2).

The underbasic bearing courses in temporary roads of open mines is building now from gravel stone and constant ones – from concrete. In these two cases CFA may substitute the sand. The investigations carried out by the University of Mining and Geology and the University of Architecture, Civil Engineering and Geodesy, Sofia, show that CFA-concrete's compressive strength and elasticity modulus meet the requirements for underbasic bearing course. Its thickness is $15\div20$ cm and it can bear loading of machines running on mines' roads.

0.8 m^3 CFA with a value $8\div10$ times lower than that of the sand are used into 1 m^2 CFA-concrete at an average instead of sand.

The problem of waste pools covering near the HEPSs against dust emission of CFA, thrown out in them, is **unsolved**. The efforts in this direction had as a result creating a composite material on CFA base plus binders which together with our suitable technology [6] can temporarily seal the waste pools' surface and solve simply and cheaply the problem with dust, spreading from their surface in dry periods of the year. The strength and water resistance of the sealing course prevent dust emission and strong infiltration of surface waters in the waste pools' body. These qualities correspond to the technology of scraping up the dried sections.

CONCLUSIONS

Big fly ash and sulphur contents of East-Maritsa lignite coal determines considerable qualities of waste + CFAG during their burning-up. The situation is similar for the other HEPSs in Bulgaria. The lack of sufficient capacity of the waste pools sets the acute problem of areas, taken by them, and protection of the environment.

The Republic of Bulgaria "produces" 3% of the waste world volume [7].

The actuality of investigations for solving the ecological problems connected with the waste from the HEPS as well as their utilization in Bulgaria is undoubted.

Widening the fields of applying the CFAG by means of creating execution of prescriptions and technologies for certain cases is obligation of our scientific specialists on the base of foreign experience and own investigations.

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Recommended for publication by Department of Underground construction, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.193-195

ASSESSMENT OF QUALITY OBJECTIVES OF CAVED BENCH MINING METHODS

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ABSTRACT
The research has as main goal to assess the effects of the dynamic of basic qualitative parameters of mining products exploited, in the context of alteration of quantitative weight for the mining methods employed, consequently to the increasing scale operation with the use of caved bench mining method MEBS.

GENERAL CONSIDERATION

In order to approach, even synthetically, we consider useful to remind that the survival of an enterprise highly depends of his client's existence; to be able to maintain a client, it is required constantly to meet his more and more complex, evolutionary and exacting needs. Consequently, the industrial unit should constantly be in progress, being supported on the contribution of his personnel members from the quality department.

The quality management process consists in applying all the statistic principles and techniques in all the stages: design, production, operation and maintenance, in order to economically satisfy the demands.

To elaborate the quality characteristics for a product it will be employed the concept of optimization, meaning satisfaction both of client's and supplier's needs, with minimal cost prices.

MEBS – represents a process – "a systematic serie of actions directed towards reaching the coal exploitation from the deposit". The term "process" includes both the human factor and the equipment's and the environment, he accomplishes also the following criteria:

- *he is systematic:* the activities within the process are interconnected in a unique concept;
- *he detainees capability*: the final result of quality plans is a process which is able to reach the objectives concerning quality in given existing operating conditions;
- *is legitimate*: the process is developed through authorized paths; he will be approved by those to whom the assigned responsibilities were commissioned.

The quality objectives are issued from number of client's requirements and from the variable characteristics of mined out coal output and selected mine technologies.

Most of the quality objectives were established at bottom and mean levels of the hierarchy. Objectives are assesses very often on technological basis, expressed mainly by the ash content / qualitative parameter to be monitored by quality responsible and by workers.

The relationship between quality and selling a product are for the date, not enough understood, but the research work carried out are emphasising a direct connection between quality and benefits. The most efficient valorisation way for the mined out coal outputs, as far as the maximal cash is concerned, is not yet approached in his whole complexity.

To prioritise quality, it must be taken into account whit maximal weight when estimating the managing skills and performances whit now are having as first goals. Overall physical mined out output and working efficiency in physical units.

ESTIMATING THE CAPABILITY OF THE CAVED BENCH MINING METHOD

The studied the mining method belongs to a group of new tehnologies applied in the collieries from Valea Jiului coal basin, so it is a process containing some characteristics over taken from former processes / for whom operational knowledge exist and certain characteristics not completed by practical experience. In the sometime, the mining method be considered as a critical process, presenting some specific occupational safety and environmental problems, but also the risk of loosing important amounts of money as a consequence of the lower process capability.

The caved bench mining methods are designed to provide an average output of 1500 t exploited coal in every working face.

The processes excessively variable are not able to complete their objectives regarding the quality.

While the method has as main goal the reaching objectives of efficiency and reaching the proposed outputs per working face, we are foreseeing that miners will deliberately ignore the quality checking in the tehnological processes.

The capacity of the process to active quality products has two aspects

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.233-236

COMPOSITE MATERIAL FOR ISOLATION OF DEPOTS FOR INDUSTRIAL WASTE AND WASTE OF LIFE

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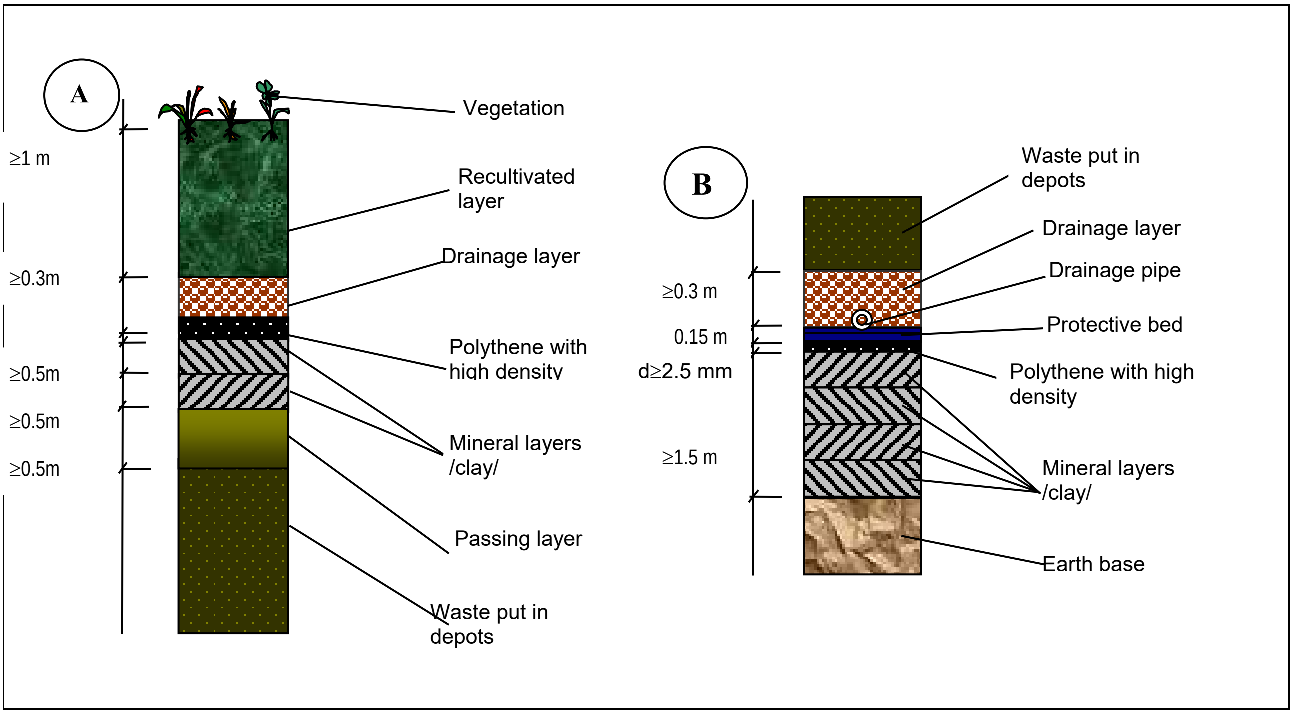
ABSTRACT
A new composite material on the base of industrial waste plus additions from inert materials is made. Many laboratory tests are carried out – for defining the convective and diffusion spreading of heavy metals and other harms; the chemical stability to suffosion and the basic mechanical indicators. The material's suitability as an element in the seal depots' stoppings is proved. The possibility of using the new product in building industrial waste depots and depots for waste of life is motivated. Ecological application of waste, polluting the environment, is found out.

INTRODUCTION

Basic way for decreasing the dangerous effect of waste is their preserving in suitable waste dumps (WD). The seal stoppings of contemporary WD are intended to decrease the waste influence [1]. That's why it is necessary the WS to be treated as very important engineering equipment.

The building of depots for solid waste of life and industrial waste is necessary to be combined with application of reliable bottom and covering seal stoppings which provide minimum movement of harms towards air environment and underground waters and soils. This is underlied in the European conception from 1986 [2] in which the idea of partitioning barriers, is developed in order the waste, put in depots, to be controlled and a monitoring to be realized.



A – covering seal stopping; **B** – bottom seal stopping

Figure 1. Vertical cuts of seal stoppings used for waste (industrial and of life) putting in depots

The contemporary technical barriers in depots have to be conformed to European standards for firming and seal stoppings. The latter are built from natural and artificial materials. The contemporary construction of bottom and covering seal stoppings contains layers with well-compacted clays with filtration coefficient ($K_f \leq 1.10^{-9} \div 1.10^{-11}$ m/s), and thickness of 1.5 m and 0.5 m respectively. In the first case clays are with predominant montmorillonite composition, in the second – with caolinite composition, which have the necessary stability to chemical suffosion. The construction of stoppings is shown on fig. 1 [3]. In order to be provided better isolation, high density polythene (HDPE) and polypropylen (PP) in the form of clothes [4] are put.

The type of bottom and covering stoppings must be conformed to the depot class. The depots are classified in three groups according to the quantity of organic hydrocarbons, which are emitted from them and infiltrated by the underground soils and waters [5].

Considerable quantities of high quality clays for mineral layers in the stoppings (1 $m^3 \div 3$ m^3 for $1m^2$ from the area of the depot) and big costs, connected with them, are the reason for searching and creating new materials.

According to German standards [6], using the alternative seal materials is allowed under the condition that they correspond to the criterion for classical ones and have proved qualities, for example, the asphalt concrete, which is a waterproof material. It is suitable for isolation and is applied for many years. As an example it may be given the isolation of more than 20 depots [7] with isolation area of 203 600 m^2 in Switzerland. The structure of asphalt concrete is given on fig. 2 [8].

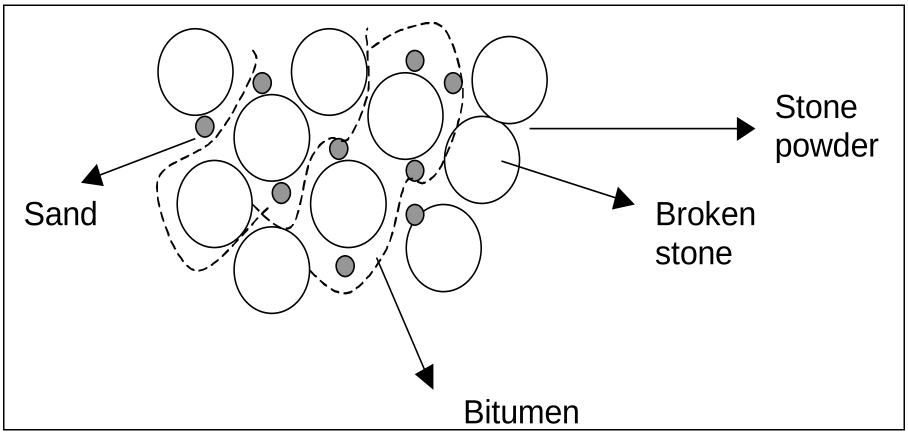


Figure 2. Characterizing scheme of the asphalt concrete

The bitumen and stone powder are put as filler in order to fill the gaps between particles of inert materials. The aim is a dense mass to be formed. It has to be said that the asphalt concrete is good for using under the condition that after compacting the pores is under 3% and the binder is more than 5 weight per cents towards the mass of material. It is necessary the filtration coefficient to be conformed to the European standards and requirements.

Material for seal stoppings

The new seal material, called tar concrete, is formed on the base of technological waste from crude oil processing through cracking plus fillers. The waste (tar) has the role of a binder. Its composition is defined through Markuson method (AASHTOT 59/Test methods for emulsified asphalts) and is shown in table №1.

Table 1. Waste composition

Components	Quantity, weight per cents
Water contents	33 % as emulsified water
Contents of dry substance	
A_7 (Asphaltens)	14.06 %
Paraffins	36.70%
Resins	49.24 %

Fillers in the new material, which are used for seal layers of the depots, are natural materials. They are shown in table №2 [9].

Table 2. Seal material composition

Composition	Weight per cents, %
Broken stone fraction 5/10	$20 \div 30$
Broken stone fraction 0/5	$50 \div 60$
Bentonite	$2 \div 5$
Stone powder (Si_2O)	$10 \div 20$
Waste	$8 \div 11$

The ratio between components of crude oil waste, which compose the oil phase of emulsion, are close to those of the distilled bitumen, used in road construction. The high presence of tars is the reason for high stability of the new material to oxydation and photooxydation ageing.

The new material has a filtration coefficient $K_f = 1.10^{-9} \div 1.10^{-11}$ m/s [10], which is proved for 12 000 minutes. Samples, compacted with static pressure till forming a seal layer with volume density $\rho = 1.85 \div 2.10$ g/cm^3, show a filtration coefficient $K_f = 5.10^{-10} \div 5.10^{-11}$ m/s (fig. 3).

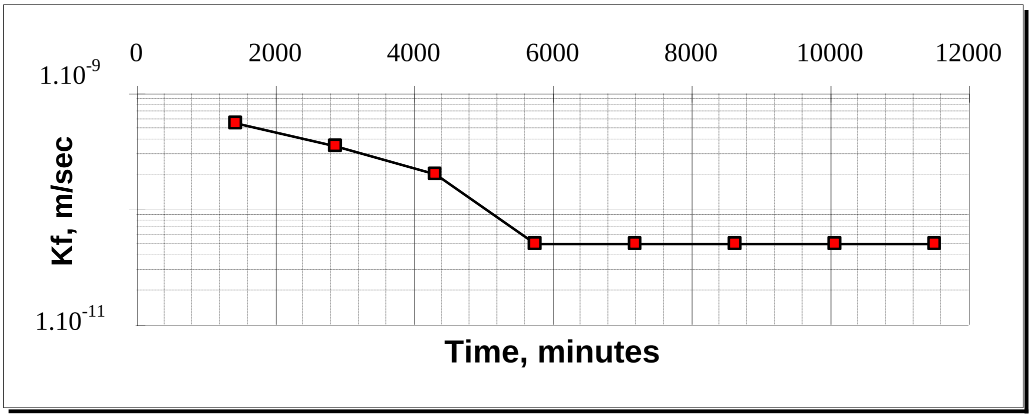


Figure 3. Filtration coefficient of the new material

These values correspond to the requirements, given in "Technische Verordnung über Abfälle".

Tests for defining the diffusion coefficient (K_d) are carried out [10]. The values of K_d are in these limits – $4.10^{-11} \div 4.8.10^{-11}$ m/s. The results are stabilized and constant after the 50th day from the test beginning.

The chemical stability (Fig. №3) of the alternative material to different chemical solutions / Thulol (C_7H_8) with purity of 99,5 volume per cents and acid indicator pH - 5,5; NaOH and HCl with pH indicator of 8,0 and 2,0 / was tested in laboratory through the drop method [1]. Chemical reaction of the material with HCl and NaOH was not proved and it kept its stability for a long time. During tests of 720 minutes with thulol in concentrated form, only surface reaction is indicated. A trace, deep 2÷3 mm and with diametre of 10 mm, is seen. It shows that although the solution aggressiveness the binder is steady in the material and keeps its stability. Putting the inert fillers on silicon base (SiO_2) is obligatory. They determine the good chemical stability of the material while the carbonate fillers would react with acids and CO_2 will be formed. Components in binder's composition (asphaltens, oils and tars) are extremely stable to acids and alkalis. In the waste water from depots of waste of life, presence of thulol may be expected in concentration not higher than ppm (that is thulol, soluble in water in quantity of 515 mg/l at 20 °C). That's why it is possible to say that the new material is stable to the influence of organic solutions [8].

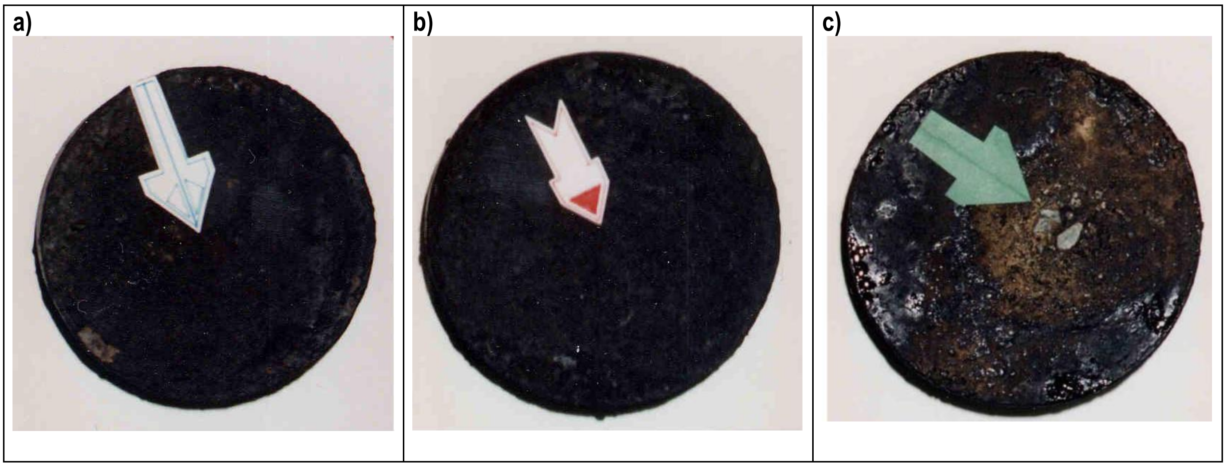


Figure 4. Chemical reaction of the material with: (a) HCl (b) NaOH and (c) Thulol

The composite material bears longitudinal deformation and can undertake considerable vertical bearings up to 0.15 MPa, which are typical for lower seal stoppings.

The technology of preparing and building waterproof and diffusion stable layers from the new material is extremely popular [9]. The homogenized mixture is put still hot on the preliminary leveled and compacted clay or earth base. The spreading is done by machines or by hand and the layer's width is not more than 15cm. The compaction is done through packing until the suitable volume density is reached. The seal stopping is covered with earth mass or clay layer when it cools down and gets the atmosphere temperature. The composite material stops water and humidity penetration from the underground water in the waste, put in depots, and vice versa, the diffusion process in two directions is practically equal to zero.

CONCLUSIONS

Using the waste products for economic aims is a real alternative of soils' recovery. With the help of waste, received from the deep processing of crude oil, a seal material, which is stable to chemical aggression of acids, alkalis and thulol solutions, is prepared. The calculated filtration coefficient of $5.10^{-9} \div 5.10^{-11}$ m/s shows that the material is suitable to be used as a seal base in building depots for industrial waste and waste of life. The high strength is another priority. It may be increased through putting geotextile as a structural material because of suitable relations between geotextile fibres and tars from the waste material. The new product is able to form steady waterproof and diffusion resistant covering stoppings. Constructions, which include a layer of tar concrete, are several times cheaper than classical because of considerable reducing the expensive clays; the expensive HDPE - cloth is eliminated; the building time is decreased; the classical road building machines are used.

Gratitude: *The authors heartily thank to the Austrian Institute for East and South-East Europe – Office Sofia for the given financial support.*

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Recommended for publication by Department of Underground construction, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.237-239

ECOLOGICAL RED GLASS

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ABSTRACT
The interest in glasses of the copper ruby type is re-actualized due to the present general tendency to avoid the fabrication and the use of products with a content of toxic or potential pollutant substances. Copper gives a red color as a result of some subtle redox processes doubled by thermal treatments. Although a great volume of practical experience was accumulated many aspects of ruby formation mechanism remained still insufficient elucidated and the rigorous control of numerous parameters influencing the color quality is still difficult or even impossible. In the present work some results are communicated, obtained in the frame of a study following both the more deeply and correct understanding of copper ruby formation processes and the accumulation of data about the role and the influence of technological parameters.

INTRODUCTION

Although in glasses can be obtained a great variety of colors and nuances in the case of red color the situation is special, the possibilities being more restrained. The electronic transitions of the usual coloring ions need, as a rule, energies corresponding to the middle of the visible spectrum towards red or to UV, the transmitted trough glass light having yellow – green – blue colors.

The transmission of yellow – red – IR radiations is possible by means of different mechanisms, preponderantly through light diffusion on colloidal aggregates nano or micro crystalline. In this aim most frequently are used cadmium sulphide, cadmium selenide and sulfo-selenide. On the basis of a similar mechanism one may obtain beautiful red colors by means of colloidal aggregates of Au, Ag or Cu [*Balta P.,* 1984; *Balta P. et al.* 2001].

In the last years glass industry was affected by the tendency, manifested throughout the world, to avoid the use of potential dangerous or pollutant substances. Such a new problem is contoured related to glasses with cadmium content, toxic for human body and pollutant for the environment.

The copper ruby seems to be a possible alternative. Because the implicated subtle redox processes and thermal treatments the obtainment of reproducible colors and nuances at an industrial scale is difficult. In this some results contributing to a more deeply understanding of the redox processes mechanisms and of the peculiarities of technological parameters are presented.

COPPER RUBY

The strong dispersed during melting metal atoms does not color glass. It is needed a developing, striking, thermal treatment to obtain the colloidal aggregates. The red ruby color is obtained when the colloidal aggregates reach dimensions of the order of 50 nm. The coloring mechanism is based preponderantly on the diffusion of light on colloidal particles.

The light diffusion intensity I_D may be evaluated by means of the Rayleigh's equation:

$$I_D = \frac{\pi . V^2}{\lambda^4 . r^2} . \varepsilon_r{}^2 (\Delta \varepsilon_r - 1) \sin^2 \alpha \qquad (1)$$

where V is the particles volume, λ the wavelength of the incident light, ε_r the dielectric relative permittivity, r the distance from the particle to the point light intensity measurement and α the angle between the diffused light fascicle and the incident one. It may be remarked the fact that, beside of some parameters related to the measuring conditions, diffusion is strong influenced by the wavelength of the incident light.

Diffusion is more intense when the wavelength is shorter, i.e. in UV – blue – green domain. Thus, through glass pass preponderantly the wavelengths corresponding to domain of yellow – red. The particle concentration has a strong influence too but at their increase, or at the increase of particle dimensions, glass become opalescent or even opaque [*Balta P.,* 1984; *Balta P. et al.* 2001].

THE WORKING MANNER

The copper ruby study followed particularity the redox processes both on simple glasses used as a kind of models and on the industrial type glasses. The model glass was chosen in the system Na_2O-B_2O_3 (10wt%Na_2O). Was used an organic reducing agent (saccharose) but also metallic Sn. Copper was introduced as CuO,Cu_2O or even Cu. Melting was made in an electric furnace, in ceramic or platinum crucibles, at 1000^{0}C for borate glasses or 1450^{0}C for silicate ones, in normal atmosphere. The samples having shapes of discs with a diameter of 25 mm and a thickness of 1-2mm were obtained by pressing molten glass in a metallic form. They were annealed and thermal treated in different conditions. Absorption spectra were recorded by means of a two-beam spectrophotometer Shimadzu UV-160A, in the domain 400-1100 nm, sometime beginning from 200 nm, in comparison with the same glass but without Cu.

COPPER RUBY IN BORATE GLASSES

In a first series of glasses copper was introduced as Cu_2O with the aim to check the opinion of some research workers according to that the colloidal aggregates are formed by cuprous oxide and not by elementary Cu. Even in the absence of the reducing agent the red color must appear [*Capatina C.*, 2001]. After melting glasses were colorless and remained so even after long heath treatments. The Cu^+ presence was evidenced the charge transfer absorption in far UV specific to this ion and by means of chemical analysis.

The conclusion was that, at least in this glass, cuprous oxide did not form colloidal coloring aggregates. Introducing an organic reducing in glasses containing Cu_2O or CuO, red colored glasses were obtained. In Fig. 1 are presented the electronic spectra of glasses initially containing CuO. The spectra interpretation seems to be quite simple. After a quart of hour of melting the peak at 590 nm has the maximum amplitude indicating the presence of a great quantity of colloidal aggregates but it can be observed also a small maximum at 800 nm corresponding to some not yet reduced CuO.

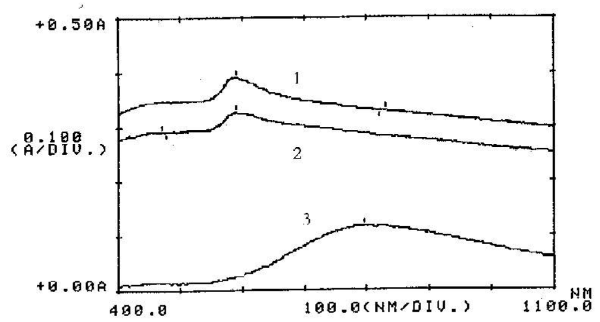

Figure 1. Electronic spectra of borate ruby glasses. Copper was introduced as a CuO and was used an organic reducer. Melting duration: 1 – 15 min., 2 – 30 min., 3 – 60 min.

After 30 minutes the peak at 590 nm is practically unchanged but that of Cu^{2+} totally disappeared. After 60 minutes the peak at 590 nm is not more visible but, in exchange, that at 800 nm is very pronounced.

The high absorption level showed by ruby glasses spectra over all visible is due to the presence of carbon resulted by organic compound decomposition. Following the carbon disappearance the necessary oxygen quantity was calculated resulting that the main oxygen source is the surrounding atmosphere. An apparent difussion coefficient of oxygen in borate melt at 1000 ^{0}C of the order of 10^{-4} – 10^{-5} cm^2s^{-1} was estimated [*Capatina C.,* 2000]. The very high value for this kind of glasses can be explained by the convection contribution, determined by gas bubbles elimination during the organic reducer burning.

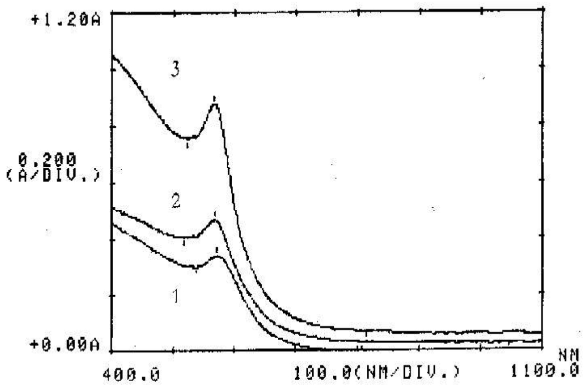

Figure 2. Electronic spectra of borate ruby glasses with copper from a Cu-Sn alloy. The heath treatement duration was: 1 – 120 min., 2 – 210 min., 3 – 240 min.

An usual soda-lime-silica glass was experimented using as a source of Cu and Sn the same alloy. The melting was made at 1450 ^{0}C in normal atmosphere. The striking treatment was made at temperatures and durations. Some of the recorded spectra are presented in figure 3.

Inspired from the Professor's Cristea and his collaborators works, in a series of borate glasses copper was introduced in the form of a Cu-Sn alloy [*Cristea V. et al.*, 1975]. The thermal were performed at 500 ^{0}C. The color appeared only after about 25 minutes. In figure 2 spectra obtained after longer treatments are presented.

At least three differences may be remarked comparing with ruby glasses obtained with organic reducer [*Ram A., et al.,* 1974].

1. the absorption due to carbon resulted by reducer burning is missing
2. a large window is present from about 568 nm up to IR
3. the peak amplitude increases apparently exponentially with time.

It may observe a strong absorption at about 566 nm due to copper colloidal aggregates, a sharp absorption limit a uniform transmission towards IR. The longest melting duration results in the apparition of a very small peak at about 800 nm specific for Cu^{2+} singaling the total oxidation of Sn. In this way a saturation is evidenced related to the end of Sn oxidation process and the beginning of the copper oxidation up to Cu^{2+}. The presence of this copper ion alters the ruby quality and represents the upper time for the good quality ruby glass elaboration process.

The striking temperature and time have a synergetic action the 566 nm peak amplitude depending linearly on temperature but logarithmically on time. These are the main technologic parameters determining, together with copper concentration, the nuance and the intensity of copper ruby glass color.

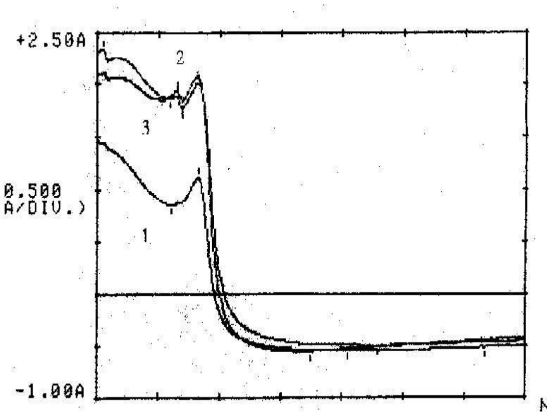

Figure 3. The electronic spectra of the industrial type ruby glasses with alloy, maintained at the melting temperature: 1 – 90 min., 2 – 150 min., 3 – 210 min
Copper ruby in industrial type glasses

Another important achievement consists in the fact that the glass composition has a low influence on the copper colloidal aggregates absorption peak position. Passing from silicate glass to borate and phosphate glasses the peak position modification is very small, inside of some 23 nm [*Weyl W.A.,* 1967].

CONCLUSIONS

Due to the actual tendency to avoid the use of cadmium in red glasses were explored the redox mechanism and the technology of cooper ruby, which seems to be a reasonable alternative.

Using a sodium – borate model glass were melted and studies containing Cu_2O, CuO or Cu^0, using an organics reducer and also Sn from a Cu-Sn alloy. The sodium – borate glass with Cu_2O content did not exhibit a red color. The advantage of using Cu-Sn instead of other copper sources and of the organic reducer was evidenced.

The copper obtainment in an industrial type glass was studied achieving some new information concerning a saturation phenomenon related to Sn quantity, the synergetic influence of striking temperature and time and the low influence of glass composition upon the colloidal aggregates absorption peak position.

The accumulated results contribute to the more deeply understanding of redox processes mechanism at copper ruby obtainment and the influence of some technological parameters. Now it is possible to imagine experiments at an industrial scale to establish the conditions for copper ruby glass obtainment in large quantities.

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THE INFLUENCE OF ADDITION ELEMENTS ON SINTERED IRON COMPACTS MICROHARDNESS

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ABSTRACT
New antifriction Materials based on iron powder, with several addition elements were developed using PM technologies. The samples were obtained by uniaxially cold compaction using pressure forces of 300 MPa and sintering in dry hydrogen atmosphere at three different temperatures. In this study the effect of copper, in and lead addition elements on Vickers hardness and microhardness as a function of process parameters was investigated. Vickers hardness was evaluated at an indentation load of 50 N and Vickers microhardness was evaluated at an indentation load of 0.1 N.
Keywords: powder metallurgy, sintered iron compacts, microhardness, addition elements

INTRODUCTION

Technical and economical advantages of the P/M technology lead to performing a wide type of new materials and products with special characteristic required by modern techniques [1].

In order to meet the requirements of future possible applications, it is important to improve the existing and develop new methods of enhancing the applied properties. This may be achieved by efficient alloying, using efficient combinations of alloying elements.

Many sintered parts reach sufficiently high strength properties, for example, similar to cast iron, already at a porosity of 20-15%. Controlled non-homogeneity of the structure by powder metallurgy processes maker it possible to obtain special properties of materials, which cannot be obtained by conventional technologies. It is essential to know the actual loading conditions of the part and modify alloying and the treatment conditions of the material on the basis of these conditions [2].

The metallic powder sintered parts present the remarkable physical, chemical and mechanical characteristics, which are determined by their composition and the phase structure as well as the shape and mass distribution of the grains [3].
The study also attempts to optimize the addition elements and sintering condition (temperature, maintaining time) in order to obtain adequate values of hardness and to investigate the quality of the sintered materials microstructure.

EXPERIMENTAL PROCEDURE

As experimental materials, iron powder by Ductil S.A. Buzău DWP 200 electrolytic copper powder, brass, tin and lead powder were used.

The powder mixtures were cold compacted in a die with single action of the upper punch at a pressure of 300 MPa obtaining 10 mm cylindrical sample. The compacted samples were placed in a tubular furnace having uniform heating zone and sintered at 600^{0}C, 650^{0}C and 600^{0}C for 20, 25 and 45 minutes.

The sintering atmosphere was dry hydrogen with a flow rate of 11/min. The samples were cooled in furnace by switching off the powder and maintaining the same flow rate of the hydrogen gas. The reference densities for selected composition were calculated by the rule of mixtures.

The sintered samples were metallographic prepared by polishing in order to investigate the porosity and by reactive etching in order to evaluate microhardness of the individual grain and structural constituents. Reference densities for each alloy composition were calculated by the rule of mixtures and the total porosity of the sintered specimens was evaluated from the difference between the reference density and calculated density.

The characteristics of the elemental powders and the composition of the mixtures are presented in Table 1 and the experimental conditions are presented in Table 2.

Table. 1.The influence of addition elements on the sintered iron.
Sample.

POWDER COMPOSITION							
	IRON %	Si %	Mu %	P %	S %	O_2 LOSS	O_2
DWP 200	0,020	0,050	0,200	0,020	0,015	0,200	0,220
CHARACTERISTICS OF ELEMENTAL POWDER							
Particle size (µm)		>160	160-100	100-63		<63	
DWP 200		15%	20-40%	20-40%		20-45%	

APARENT DENSITY 9/cm³	2,5-2,7
FLOW RATE (S/50 g)	33

From this tipe by iron powder obtained three materials FC-40 (iron-lase, C-0,4%), FC-80 (Iron-base, C-0,80%) and Fe 50 U_3 (iron base, C-0,5%, Cu-3%).

CONCLUSION

Hardness express the resistance of material to deformation of the surface caused by the effect of a geometrically defined body.

The value obtained by these methods ar referred to as macrohardness, the value increasing by addition elements and time by sintering process.

Table 2 Experimental conditions.

Sintering Conditions	Sintering temperature [⁰C]	600-650⁰
	Holding time [min]	20,35, 40
	Atmosphere	Dry hydrogen
Sintered density [g/cm³]		6,95

Tests of hardness of powder materials in determining the hardness of the material of a whole, including pores, were carried out using the Vickers method.

RESULTS AND DISCUSSIONS

Hardness expresses the resistance of the material to the deformation of the surface caused by the effect of a geometrically defined body. Although, the value of the porous materials hardness is always lower then the compact materials hardness, for a general qualitative characterization of the sintered alloys macrohardness Vickers method has been used. This method is based on the Vickers method with a very low load.

The influence of the additional elements contents from the hardness (the selected compositions containing cooper powder and brass powder) is in Figures 1.

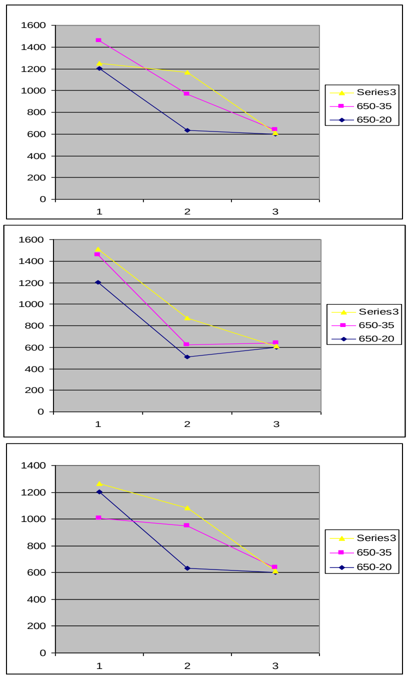

Figure 1. The influence of the additional elements and sintering parameters on the hardness.

The influence of hardness of additional elements from sintering parameters is descrived in Figures 2.

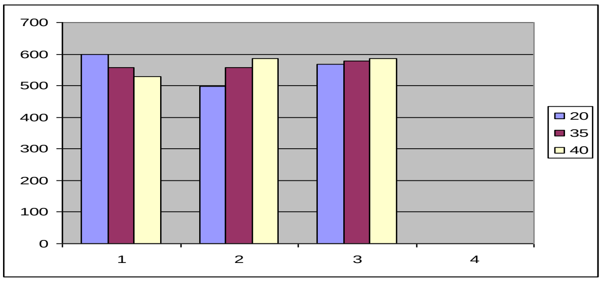

Figure 2. The influence of the additional elements and sintering parameters on the hardness of the alloys containing brass powder.

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RUBICI b: SINTERED MACHINE ELEMENTS ELLIS NORWOOD.

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.243-246

RADIATIVE HEAT TRANSFER EQUATION IN SYSTEMS OF GREY - DIFFUSE SURFACES SEPARATED BY NON-PARTICIPATING MEDIA

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ABSTRACT
The paper proposes a matrix model for solving the radiative heat transfer in enclosures based on mutual angular radiation coefficients and radiative properties of the surfaces that make up the enclosure.

INTRODUCTION

Radiative heat transfer represents the most important part of the global heat exchange in thermal installations working at temperatures, such as steam generators, boilers, furnaces and so on. The two components of the radiative heat transfer – (1) between two or more surfaces and (2) between gas and surfaces – on the one hand, and on the other hand the complex geometry of most heat exchange systems, make the simulation of the radiative heat transfer a complex process, more suited for the numerical approach. Besides, a serious drawback is the lack of a standard terminology in the field of radiative heat transfer. The nomenclature used in this paper is based on that of the illumination engineering community due to the close relationship between physical terms and standardized terms used in the field of illumination engineering.

The basic concepts of radiative heat transfer such as direction and solid angle, radiant intensity, radiosity, emittance, absorbance, reflectance and so on, are considered known and aren't discussed in this paper. The basic geometrical parameters are represented in figure 1. A summary of the basic terms used henceforth is given below:
- spectral radiation intensity due to the own emission of the surface:

$$I_{\lambda,e} = \frac{\delta \dot{Q}}{\cos\varphi\, dS_1 d\Omega\, d\lambda}$$

Index *e* means self-emitted.
- spectral power density:

$$E_\lambda(\lambda) = \int_0^{2\pi} d\psi \int_0^{\pi/2} I_{\lambda,e}(\lambda,\varphi,\psi)\cos\varphi \sin\varphi d\varphi$$

- spectral radiation intensity due to incident radiation:

$$I_{\lambda,i} = \frac{\delta \dot{Q}}{\cos\varphi\, dS_1 d\Omega d\lambda}$$

Index i means incident to the surface considered.
- spectral irradiation:

$$G_\lambda(\lambda) = \int_0^{2\pi} d\psi \int_0^{\pi/2} I_{\lambda,i}(\lambda,\varphi,\psi)\cos\varphi \sin\varphi\, d\varphi$$

spectral radiosity:

$$J_\lambda(\lambda) = \int_0^{2\pi} d\psi \int_0^{\pi/2} I_{\lambda,e+r}(\lambda,\varphi,\psi)\cos\varphi \sin\varphi\, d\varphi$$

Radiosity takes into account both self-emitted and reflected radiation

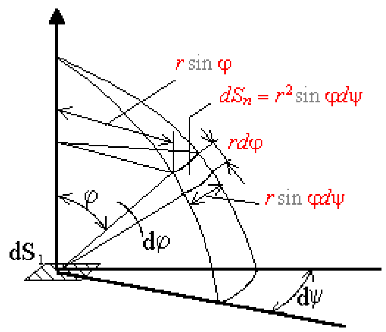

Figure 1

RADIATIVE HEAT TRANSFER IN A SYSTEM OF SURFACES

The system of surfaces considered consists of n flat diffuse-grey surfaces which make up an enclosure.
The net radiative heat flux of a surface i is given by the difference between the radiated heat flux due to emission and

reflection and the incident radiative heat flux between two random patches (figure 2):

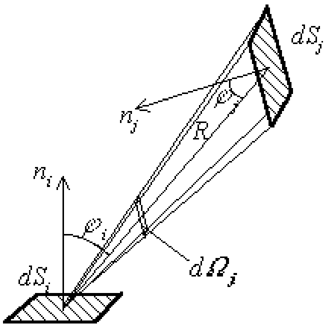

Figure 2

$$\dot{Q}_i = S_i(J_i - G_i) \tag{1}$$

in which the radiosity of surface *i* is given by: $J_i = E_i + R_i G_i$ with R_i – reflectance of the surface *i*; for an opaque surface, $R_i = 1 - A_i$, and (1) turns into:

$$Q_i = S_i(E_i - A_i G_i) \tag{2}$$

The radiative heat flux given by (2) has a non-zero value unless the considered surface gets an equivalent heat flux either through convection or conduction. If heat exchange is accomplished exclusively through radiation and if steady state is reached, then $\dot{Q}_i = 0$.

For diffuse-grey surfaces the emittance equals the absorbtance according to Kirchhoff's law: $\varepsilon_i = A_i$. In this case the radiosity is given by:

$$J_i = \varepsilon_i E_{b,i} + (1 - A_i)G_i \tag{3}$$

with $E_{0,i}$ the power density of the black body having the same temperature as the surface considered.
The above equation changes (1) to:

$$\dot{Q}_i = S_i\left(J_i - \frac{J_i - \varepsilon_i E_{b,i}}{1 - \varepsilon_i}\right) \tag{4}$$

or:

$$\dot{Q}_i = \frac{E_{b,i} - J_i}{\dfrac{1 - \varepsilon_i}{\varepsilon_i S_i}} \tag{5}$$

Equation (5) describes the radiative heat exchange from a surface. It suggests an analogy with Ohm's law leading to the definition of the radiative resistance of the surface *i*:

$$R_{r,i} = \frac{1 - \varepsilon_i}{\varepsilon_i S_i} \tag{6}$$

If the surface *i* belongs to an enclosure then the radiosity J_i depends on the heat transfer among surface *i* and surfaces *j* (*j*=1...*n*). The radiative heat flux radiated by surface *i* and received by the surface *j* is given by:

$$Q_{i \to j} = \varphi_{ij} J_i S_i \tag{7}$$

and the radiative heat flux radiated by surface *j* and received by the surface *i* is given by:

$$Q_{j \to i} = \varphi_{ji} J_j S_j \tag{8}$$

In which φ_{ij} and φ_{ji} are the mutual medium angular radiation coefficients.

Considering a system consisting of two surfaces *i* and *j*, the mutual medium angular radiation coefficient φ_{ij} is defined as the ratio between the radiative heat flux emerging from S_i and intercepted by S_j and the overall radiative heat flux emerging from S_i:

$$\varphi_{ij} = \frac{\dot{Q}_{i \to j}}{J_i S_i} \tag{9}$$

Isolating two elementary patches dS_i and dS_j and applying Lambert's law, the elementary radiative heat flux emerging from dS_i in direction φ_i in the elementary solid angle $d\Omega_{ji}$ is given by:

$$\delta \dot{Q}_{i \to j} = I_i \cos\varphi_i dS_i d\Omega_{ji} \tag{10}$$

Taking into account the definition of the solid angle, the above expression turns into:

$$\delta \dot{Q}_{i \to j} = I_i \frac{\cos\varphi_i \cos\varphi_j}{R^2} dS_i dS_j \tag{11}$$

For diffuse surfaces, $I_i = \dfrac{J_i}{\pi}$ and equation (11) changes to the following form:

$$\delta \dot{Q}_{i \to j} = J_i \frac{\cos\varphi_i \cos\varphi_j}{\pi R^2} dS_i dS_j \tag{12}$$

As a result of integration, equation (12) turns into:

$$Q_{i \to j} = J_i \int_{S_i}\int_{S_j} \frac{\cos\varphi_i \cos\varphi_j}{R^2} dS_i dS_j \tag{13}$$

The net radiative heat flux between surfaces I and j is given by:

$$\dot{Q}_{ij} = \dot{Q}_{i \to j} - \dot{Q}_{j \to i} \tag{14}$$

Taking into account the well-known reciprocity property of the medium angular radiation coefficients (14) becomes:

$$Q_{ij} = \varphi_{ij} S_i \left(J_i - J_j \right) \qquad (15)$$

or:

$$\dot{Q}_{ij} = \frac{J_i - J_j}{R_{r,ij}} \qquad (16)$$

In which the geometric radiative resistance $R_{r,ij}$ is given by:

$$R_{r,ij} = \frac{1}{\varphi_{ij} S_i} = \frac{1}{S_{ij}} \qquad (17)$$

The geometric radiative resistance $R_{r,ij}$ depends solely on the geometry of the radiant system, unlike $R_{r,i}$, which depends also on the radiative properties of the material.
The total incident heat flux on the surface i is given by:

$$G_i S_i = \sum_{j=1}^{n} \varphi_{ji} S_j J_j = \sum_{j=1}^{n} \varphi_{ij} S_i J_j \qquad (18)$$

from which emerges the irradiation of surface *i*:

$$G_i = \sum_{j=1}^{n} \varphi_{ij} J_j \qquad (19)$$

Combining (1) and (19):

$$\dot{Q} = S_i \left(J_i - \sum_{j=1}^{n} \varphi_{ij} J_j \right) \qquad (20)$$

or, taking into account the enclosure property of the medium angular radiation coefficients $J_i S_i = \sum_{j=1}^{n} \varphi_{ij} J_i S_i$, equation (20) yields consecutively:

$$Q_i = \sum_{j=1}^{n} S_i \varphi_{ij} J_i - S_i \sum_{j=1}^{n} \varphi_{ij} J_j \qquad (21)$$

$$\dot{Q}_i = \sum_{j=1}^{n} S_i \varphi_{ij} \left(J_i - J_j \right) = \sum_{j=1}^{n} \dot{Q}_{ij} \qquad (22)$$

Equation (5) and (22) lead to:

$$\frac{E_{b,i} - J_i}{\dfrac{1-\varepsilon_i}{\varepsilon_i S_i}} = \sum_{j=1}^{n} \frac{J_i - J_j}{\dfrac{1}{\varphi_{ij} S_i}} \qquad (23)$$

In a system of grey-diffuse surfaces (23) is written as:

$$\frac{E_i - J_i}{R_i} = \sum_{j=1}^{n} \frac{J_i - J_j}{R_{ij}} \qquad (24)$$

or:

$$\frac{E_i}{R_i} - \frac{1}{R_i} J_i = J_i \sum_{j=1}^{n} \frac{1}{R_{ij}} - \sum_{j=1}^{n} \frac{1}{R_{ij}} J_j \qquad (25)$$

which, upon rearranging, becomes:

$$\frac{E_i}{R_i} + \sum_{j=1}^{n} \frac{1}{R_{ij}} J_j = \frac{1}{R_i} + \sum_{j=1}^{n} \frac{1}{R_{ij}} J_i \qquad (26)$$

The radiosity is given by:

$$J_i = \frac{\dfrac{1}{R_i}}{\dfrac{1}{R_i} + \sum_{j=1}^{n} \dfrac{1}{R_{ij}}} E_i + \frac{1}{\dfrac{1}{R_i} + \sum_{j=1}^{n} \dfrac{1}{R_{ij}}} \sum_{j=1}^{n} \frac{1}{R_{ij}} J_j \qquad (27)$$

or:

$$\begin{bmatrix} J_1 \\ J_2 \\ .. \\ .. \\ J_n \end{bmatrix} = \begin{bmatrix} \dfrac{\rho_1}{\Phi_1} E_1 \\ \dfrac{\rho_2}{\Phi_2} E_2 \\ .. \\ .. \\ \dfrac{\rho_n}{\Phi_n} E_n \end{bmatrix} + \begin{bmatrix} \Phi_1 \rho_{11} & \Phi_1 \rho_{12} & ... & \Phi_1 \rho_{1n} \\ \Phi_2 \rho_{21} & \Phi_2 \rho_{22} & ... & \Phi_2 \rho_{2n} \\ ... & ... & ... & ... \\ \Phi_n \rho_{n1} & \Phi_n \rho_{n2} & ... & \Phi_n \rho_{nn} \end{bmatrix} \bullet$$

$$\bullet \begin{bmatrix} J_1 \\ J_2 \\ .. \\ .. \\ J_n \end{bmatrix} \qquad (28)$$

in which: $\Phi_i = \frac{1}{R_i} + \sum_{j=1}^{n} \frac{1}{R_{ij}}$, $\rho_i = \frac{1}{R_i}$ and $\rho_{ij} = \frac{1}{R_{ij}}$.

Subtracting from both left and right member the term representing the radiosities matrix, equation (28) turns into:

$$\begin{bmatrix}0\\0\\..\\..\\0\end{bmatrix}=\begin{bmatrix}\frac{\rho_1}{\Phi_1}E_1\\\frac{\rho_2}{\Phi_2}E_2\\..\\..\\\frac{\rho_n}{\Phi_n}E_n\end{bmatrix}+\left(\begin{bmatrix}\Phi_1\rho_{11} & \Phi_1\rho_{12} & ... & \Phi_1\rho_{1n}\\\Phi_2\rho_{21} & \Phi_2\rho_{22} & ... & \Phi_2\rho_{2n}\\... & ... & ... & ...\\\Phi_n\rho_{n1} & \Phi_n\rho_{n2} & ... & \Phi_n\rho_{nn}\end{bmatrix}-I\right)\bullet$$

$$\bullet\begin{bmatrix}J_1\\J_2\\..\\..\\J_n\end{bmatrix} \tag{29}$$

in which I is the identity matrix:

$$I=\begin{bmatrix}1 & 0 & ... & 0\\0 & 1 & ... & 0\\0 & 0 & ... & 0\\0 & 0 & ... & 1\end{bmatrix}$$

Finally, the radiosities matrix is the solution of the following system:

$$\begin{bmatrix}1-\Phi_1\rho_{11} & -\Phi_1\rho_{12} & ... & -\Phi_1\rho_{1n}\\-\Phi_2\rho_{21} & 1-\Phi_2\rho_{22} & ... & -\Phi_2\rho_{2n}\\... & ... & ... & ...\\-\Phi_n\rho_{n1} & -\Phi_n\rho_{n2} & ... & 1-\Phi_n\rho_{nn}\end{bmatrix}\begin{bmatrix}J_1\\J_2\\...\\J_n\end{bmatrix}=\begin{bmatrix}\frac{\rho_1}{\Phi_1}E_1\\\frac{\rho_2}{\Phi_2}E_2\\...\\\frac{\rho_n}{\Phi_n}E_n\end{bmatrix} \tag{30}$$

CONCLUSIONS

Equation (30) can be solved by mean of numerical methods such Gauss-Siedel iteration. Gauss-Siedel iteration has the advantage of being absolutely convergent for diagonally dominant systems such as the one of interest here. A matrix *M* is diagonally dominant if for all *i* $\sum_{j=1, j\neq i}|M_{ij}|<|M_{ii}|$. This is the case of the matrix in equation (30) because medium angular radiation coefficient from a flat surface to itself is 0.

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.247-250

IMPACT ON REGIONAL AQUIFERS OF COAL EXPLOITATION IN OLTENIA REGION

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ABSTRACT
Mining works connected with coal extraction in Oltenia have an important effect on regional aquifers, such regional dacian artesian aquifer.
In the natural regime period the main flow direction in the artesian aquifer, in the zone of the huge quarry from the Oltenia (Rosia de Jiu), was north-south, with a hydraulic gradient of about 1%. As a consequence of a strong drainage in the artesian aquifer a great depression zone appeared (a drawdown about 90 m).
The variations of groundwater level under the conditions of a relatively constant recharge change the relations between regional aquifers and also the chemical characteristics of groundwater.
In many others regions (like Motru, Prunişor, Mihăiţa etc.), the water drainage change the natural regime of aquifers. The effects of are changes of chemical characteristics of groundwater and also on the stability of the slopes.
The paper presents some results of quantitative evaluation of mining works effects on aquifers in Oltenia region. Results are obtained with stochastic and numerical models for local and regional aquifers.

Key words: mining works, coal extraction, water drainage, stochastic model (kriging, conditional simulation), numerical model (finite difference), hydrodynamic and hydrochemical changes, monitoring network

INTRODUCTION

Mining works associated with coal exploitation in Oltenia have many kind of impact on aquifers systems that are important resources for water supply of towns, agriculture and industry:

- disappearance of shallow aquifers in the area of quarries (by excavation of aquifers);
- dewatering of shalow aquifers (above of the lignite bed number five, the deeper exploited in Oltenia's quarries) in the area of quarries
- dewatering of deep aquifers (confined artesian aquifers) in the area of quarries and their neighbourhood (below the coal bed number five).

Environmental impact assessement is realised on the base of a comparative analisys of the parameters of aquifers before the beginning of coal exploitation and after that..

In the areas of coal querries from Oltenia, the most affected aquifer by the dewatering system is the artesian aquifer situated below the coal bed number five.

The artesian confined aquifer system situated below the coal bed number five have the next main characteristics:

- the most developed pliocene aquifer in the Oltenia region, with a very good continuity;
- the superior pontian and dacian deposits of artesian aquifer are represented by:
  - o fine, medium and coarse grained sands at the bottom of the aquifer
  - o medium sand, fine sand, silt and clayey sands at the top of the aquifer
- a complex structure in the west side of Oltenia (between Danube River and Jiu River: ten or twelve permeable layers of sand separated by impermeable or semipermeable layers of clay) that became a simple one with no more than one or two layer at the east of the Jiu River.

The artesian-confined aquifer connects all effects of coal exploitation on all shallow aquifer systems that are in hydraulic communication with this regional aquifer. This aquifer can be the main physical support for a *numerical model*, used as a principal tool for development of environmental politics in Oltenia region, a region with an accelerated mining activity.

Specific aims of environmental assessement of lignite exploitation on aquifer systems from mining Oltenia region are:

- assessement of hydrodynamic regimen of aquifer systems;
- monitoring and prognosis of hydrodynamic regimen in the quarries for lignite exploitation;
- monitoring and prognosis of hydrochimic regimen in the quarries for lignite exploitation;
- identification of appropriate tools for removal of the negative effects of lignite exploitation on environment (for long life of "clean" aquifers).

ENVIRONMENTAL ASSESSEMENT METHOD

Environmental assessement of impact of lignite exploitation on aquifer systems from mining Oltenia region is realised on the basis of: selection, validation and storage of data, mathematical modelisation of aquifer system and assessement of mining impact on aquifer by numerical simulation.

The main criteria of *data selection* was the objectives of the project and database include:

- natural hydrodynamic regimen of the artesian aquifer system,;
- hydrodynamic regimen of the aquifer for the entire period of exploitation;
- chemical caracteristics of groundwater

Validation of data was realized by processing of experimental data and new hydrodynamic tests in some area like Motru (***Fig.3***) and Rosia de Jiu. All information are separated in two categories: hydrogeological parameters of aquifers and technical characteristics of exploitation process. National Company of Lignite Oltena (NCLO) is the beneficiary of the database realized.

Mathematical model have two components:

- the first component is for *stochastic analysis* of time series and spatial distribution of hydrogeological parameters;
- the second one for numerical simulation of flow in the artesian aquifer

Stochastic analysis of time evolution of piezometric head was used for identification of correelations between flow of dewatering system, dynamic resources an recharge of aquifer. Because of irregular measurement program, the stochastic analysis was necessary for identification of general trend of piezometric head. The spectral analysis and Markov chains was used to find the stuctures of time series: periodical character, time-correlations etc.

For a complex analysis of spatial distribution of hydrogeological parameters of numerical model was used kriging method [4]. The fictiv point method was used for improuvement of monitoring network of aquifer systems.

Numerical simulation of artesian aquifer is realized on the basis of a finite difference model. We use the numerical model for simulation of dewatering system impact on aquifers for different scenario. Conditional simulation was used to evaluate the degree of uncertainty of the results of mathematical simulation.

DATABASE

Geological investigation of lignite deposit was realized in Oltenia by a huge number of wells (***Fig.1***). Hydrogeological exploration is not so detailed like the geological one. The main characteristics of monitoring network for aquifer systems:

- the monitoring network is developed especially in the area of quarries;

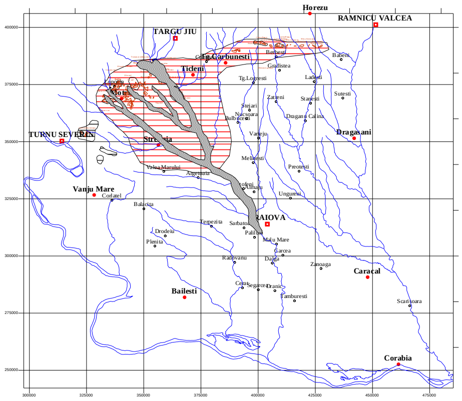

Figure 2 .Zona de impact a exploatării cărbunelui asupra acviferelor

- too few observation points between areas with mining works;
- an irregular program for measurements in the observation points
- not enough number of chemical analysis for groundwater

The achievement of a monitoring network for a realistic evaluation of impact enforces a *new strategy.* The main objectives of this new strategy for monitoring of aquifer systems are:

- a new permanent monitoring network for piezometric head of aquifers with a uniform spatial distribution on entire Oltenia region (not only in the area of mining works!);
- evaluation of an *optimal frequency* of measurement for piezometric head and water sampling for chemical analysis;
- monitoring of dewatering system (geometry of the drainage net, flow of drainage net etc.)

This objectives of this new monitoring network hint at protection of aquifers against the dewatering effects in the area of quarries for lignite exploitation.

The permanent network will use the existing water wells supply from mining areas and other piezometers situated outside the mining areas, till the boundary of dewatering influence.

At the moment, in the mining areas NCLO have 60 water wells supply. The spatial distribution of this wells is not adequate for a strict delimitation of influence area of dewatering system for quarries and mining works.

For the estimation of the impact on the artesian aquifer was used data measured between 1970 and 2001, in different mining areas (Motru, Prunisor, Rosia de Jiu etc.)

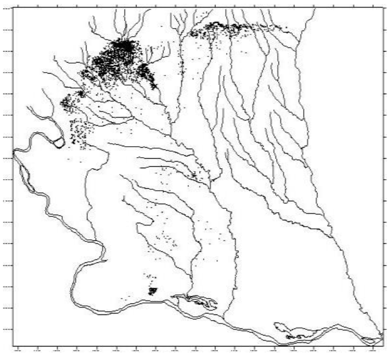

Figure 1. Foraje geologice de explorare pentru zăcămintele de lignit din Oltenia

REZULTS

There are different effects of lignite exploitation on aquifer systems from Oltenia region (***Fig.2***):

- *total or partial disappearance of aquifers be cause of:*
  - the excavation works;
  - the execution of drainage trench;
  - induced leakage
  - drainage and dewatering in the areas of the quarries
- *local and regional reduction of groundwater resources settled by:*
  - drawdown of piezometric surface in the areas of quarries;
  - change of hydraulic gradient;
  - decreasing or dissapearance of elastic store of confined aquifers;
  - increasing of deficit of water balance for regional aquifers
  - pollution of groundwater in some areas because of lekage and dewatering system
  - deterioration of water resources because of intensive exploitation
- the emptiness of groundwater resources by:
  - complete destruction of permanent water resources of aquifer;
  - intensive leakage of aquifers and exploitation for water supply; lock
- *decreasing of shallow aquifer recharge by*:
  - closure of pore, especially from shallow aquifer, with dust of lignite
- *destruction of domestic supply well by* :
  - total drainage of groundwater resource
  - disappearance of shallow aquifers

In this first stage, the assessment of coal exploitation impact on aquifers from Oltenia region consider only the drawdowns induced by dewatering systems of quarries and tapping of groundwater from mining areas.

For the analysis of piezometric head variation, from existing documentation, was selected 197 drainage wells from dewatering system of Rosia's quarry. In the analysed period, October 1979-December 2000, was realised a number of 8600 measurements of piezometric head.

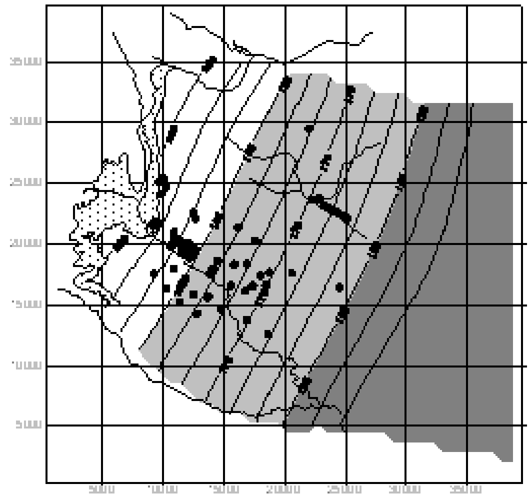

Figure 3. Natural regimen of flow in the aquifer situated below the lignite bed number one (area E.M.Motru)

The evolution of piezometric head in artesian aquifer (below the five lignite bed) in the five drainage lines of Rosia quarry indicate a variation of drawdown of piezometric head between 50 m and 75 m, with a stationary character at the beginning of studied period.

Natural hydraulic gradient of the flow in the area of Rosia quarry has not any influence on stationary character of piezometric head evolution for a long period of time.

For the area of Rosia quarry, on the basis of the hydrogeological parameters of artesian aquifer, was realized a hydrodynamic finite difference model for the simulation of the dewatering system effect. The drawdowns induced by the dewatering system for the entire period of dewatering have values between 80 and 95 meters.

Another hydrodynamic model was elaborated for the aquifer situated below the lignite bed number one, in the area of mining works Motru [1]. In this area the tapping of groundwater for the Motru town have an important impact on the aquifer. Natural regimen of groundwater flow (***Fig.3***) is influenced on some areas around the water wells supply (water catchings: Matasari, V.Manastirii-Lupoaia, CET.Motru, Lupoaia; ***Fig.4***). The drawdown induced by the tappings of water has values between 15 and 20 meters.

In a second stage, these models and others, elaborated for different areas like Prunişor (Palcu,M., 1985), Albeni (Scrădeanu D., 1990), was joining in a single model. This new hydrodynamic model will be used for the prognosis of the effect of lignite exploitation in the future and for the recovery of the piezometric surface after the closer of lignite exploitation in Oltenia region (Palcu, M., 2003).

Quality of groundwater was affected by lignite exploitation. In some areas was identified high concentration of ammonia, organic matter, nitrite, and nitrate (M.Palcu, 2002). Because of poor quality of chemical data, now is impossible to elaborate a representative hydrochemical model for the artesian aquifer system.

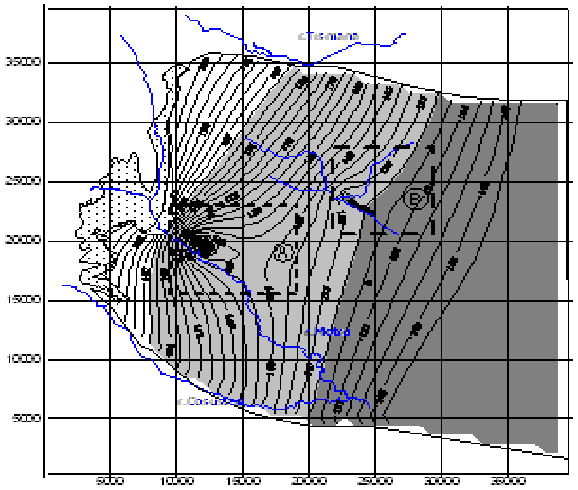

Figure 4. Regimen of flow in the aquifer situated below the lignite bed number one influenced by tappings or water (area E.M.Motru)

Pollution sources are represented by *agriculture activities* in the plane area and overflow meadows and *mining works* in the areas of quarries. Due to the incomplete burning of coal, the redox reactions and the others activities connected with lignite exploitation are introduced in the aquifers: carbon gaz and ammonia, oils from deposits and another contaminants from the waste dump.

CONCLUSIONS

Preliminary analysis of results measurements and simulation with hydrodynamic models make possible the following conclusions:

- impact of lignite exploitation on regional aquifer systems is important in the neighbourhood of coal quarries (drawdown of 95 m in the area of Rosia de Jiu quarry, 20 m in the areas of tapping of groundwater of Motru town);
- dynamic resources of regional aquifers from Oltenia region are huge so the time recovery of piezometric surface is very short, after the end of dewatering activities (only in the case of artesian aquifer and not for the destroied shallow aquifers)
- natural quality of groundwater is affected because of the new hydrodynamic relations between aquifers, induced by dewatering systems of quarries and mining works.

For the big proportion of mining works and dewatering systems connected with the coal exploitation in the Oltenia region, the impact on artesian aquifer is not so important be cause of his huge dynamic resource. Is possible a spectacular recovery of this aquifer in a short time after the end of exploitation of deep lignite beds in the west part of Rosia de Jiu quarry.

Although at regional scale, the recovery of artesian aquifer is quite rapidly, the unpleasant effects of coal exploitation at local scale are important for water supply of villages, landslides in the area of roads and railways. This damages can be resolved with low costs if will be realised a monitoring network for environmental parameters. We already have been proposed a monitoring network on the basis of our hydrodynamic model, with the following main objectives:

- the degree of fragmentation of shallow and confined aquifers, the main cause of reduction of groundwater resources and pollution of groundwater.
- the quality of groundwater in the water wells and the tapping of groundwater, affected by the changes of hydrodynamic relations between aquifers;
- the piezometric head of the regional and local aquifers, the main signal of dewatering systems influence on aquifer;
- groundwater level in the sterile dump situated inside and outside the quarries, an important cause of the landslides
- Subsidence of the land induced by dewatering and underground mining works, also an important cause of the landslide.

The data obtained from the monitoring network are used to make actual the hydrodynamic and hydrochemical models of aquifer systems for the prognosis of impact of coal exploitation on environment in Oltenia region.

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- the ability to reach the objectives concerning the outputs quality the so called targeting (which is for the MEBS s case, to provide the standardised ash content).
- The intrinsic capacity to constantly repeat the results, known as the capability of the process.

The effect of excessive variations of ash content, as qualitative target parameter at working face level will be the probably expressed in a graphical manner, using a diagram, as it follows (figure1).

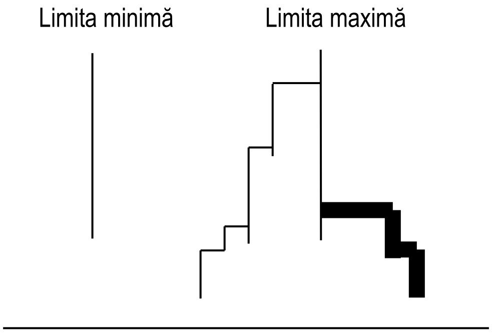

Figure 1

The mining process is characterised by the tendency allowable limit of ash content, in order to achieve the quantitative parameters.

A consequence according to the diagram consists in frequent exceeding of the allowable ash content with major effects regarding these products valorisation.

Regardless from the characteristics token over from methods usually employed in Valea Jiului coal basin mining practices, MEBS obviously dispose of new characteristics, for whom there is not yet experience and knowledge, for example coal outputs quality.

The measurement of coal quantity exploited from cased bench mining method, together with his quality expressed in ash contend are representing actual issues not completely solved out yet. Determination as accurate as possible the coal output and quality in comparison whit the reserve consumption from the considered deposit and seams ash content is having a real economical significance and importance given by the need to monitor technical and economical efficiencies of MEBS and also by requirement to measure the work and quantify the level of rewords.

Uncertainties on accurate knowledge of industrial product's ash content for every working place, affects negatively the try to rigorously analyse MEBS's efficiency, whit respect to recovery and dilution degrees.

A particular shape of process capability evaluation is the numerical simulation based on mathematical models.

For a first case study we employed the data specific for seam 13, block II, Lupeni colliery.

The main purpose of the case study consists in assessing a correlation between the qualitative parameters of mined out coal in high output working faces and castings volume expressed as marketable output. It is to be mentioned that an output of 1500 t day, as it is designed for the caved bench mining method represents for the great majority of collieries within our basin a weight exceeding 50 %, so the effects on coals quality delivered for coal processing plants or thermal powerplants will be major. The mathematical model and the results are presented in the following section

RESEARCH REGARDING THE POSSIBILITY TO EVALUATE MARKETABLE OUTPUT AS A FUNCTION OF THE QUALITY OF MATERIALS RESULTED THROUGH OUTPUT'S EVACUATION FROM THE CAVED BENCH.

The main parameters taken into account in this research were

a - ash content of material in the deposit, comprised between 6 - 38,5 %;

b - ash content of debris in the deposits roof comprised between 83 - 88 %;

v_a - the weight of coal from the seam in the entire coal output, comprised between 0 - 100 %;

v_b - weight of debris in the coal mass comprised;

w_c - moisture content of mined out coal comprised between 6 - 10 %.

As qualitative and quantitative index the marketable output is employed output resulted from selling the rough coal as sorted PM, lei Gcal

Description of the mathematical model proposed:

Using the following notations: Y = P.M., X_1 =a, X_2=b, X_3=v_a, X_4=v_b, X_5=w_c, the dependence between the six considered parameters can be described employing an equation having the shape:

$$Y = F(x_1, x_2, x_3, x_4, x_5) \qquad (1)$$

Taylor polynoms, can be used for the approached issues:

The linear polynom whose shape is:

$$T_l = a_0 + a_1x_1 + a_2x_2 + a_3x_3 + a_4x_4 + a_5x_5 \qquad (2)$$

Analysing the shapes of two degree Taylor polynoms we can observe that they can be turned in linear polynoms which are depending upon a higher number of variables namely:

a) For the quasicanonical polynom, by introducing the following variable $x_6 = x_1^2$; $x_7 = x_2^2$; $x_8 = x_3^2$; $x_9 = x_4^2$; $x_{10} = x_5^2$, T_c will take the shape:

$$T_c = a_0 + a_1x_1 + a_2x_2 + a_3x_3 + a_4x_4 + a_5x_5 + a_6x_6 + a_7x_7 + a_8x_8 + a_9x_9 + a_{10}x_{10} \qquad (3,a)$$

where: $a_6 = b_1$; $a_7 = b_2$; $a_8 = b_3$; $a_9 = b_4$; $a_{10} = b_5$;

b) For the general polynom, by doing these notations: $x_6 = x_1^2$; $x_7 = x_2^2$; $x_8 = x_3^2$; $x_9 = x_4^2$; $x_{10} = x_5^2$; $x_{11} = x_1x_2$; $x_{12} = x_1x_3$; $x_{13} = x_1x_4$; $x_{14} = x_1x_5$; $x_{15} = x_2x_3$; $x_{16} = x_2x_4$; $x_{17} = x_2x_5$; $x_{18} = x_3x_4$; $x_{19} = x_3x_5$; $x_{20} = x_4x_5$, T_g will admit the expression:

$$T_g = a_0 + a_1x_1 + a_2x_2 + a_3x_3 + a_4x_4 + a_5x_5 + a_6x_6 + a_7x_7 + a_8x_8 + a_9x_9 + a_{10}x_{10} + a_{11}x_{11} + a_{12}x_{12} + a_{13}x_{13} + a_{14}x_{14} + a_{15}x_{15} + a_{16}x_{16} + a_{17}x_{17} + a_{18}x_{18} + a_{19}x_{19} + a_{20}x_{20} \quad (3,b)$$

Based on the above presented equation (1) will be completely determined precisely or approximately / if the Taylor polynoms coefficients T_l, T_c, T_g, having the shapes (2a), (3,a) and (3,b) will be determined. To obtain their values in real conditions N groups of seven values having the shape { y_k, x_{1k}, x_{2k}, x_{3k}, x_{4k}, x_{5k} } ; $k = 1 \leftrightarrow N$; with which, using the least square roots method some linear systems of q equations with q unknowns are obtained. For these systems:

-q=6 for the linear polynom
-q=11 for the quasicanonical polynom
-q=21 for the general polynim
and their solutions can be obtained by computer

Mathematical model's validity testing: The proposed mathematical models was achieved based on data gathered from technical reports available at collieries.

Table 1 synthesize the main values of the above mentioned parameters, employed in validation of mathematical models. For each case, the computation of Taylor polynoms coefficiency was made by computer.

Table 1

X_1	X_2	X_3	X_4	X_5	Y
6	83	100	0	6	398
8	84	95	5	7	367
10	85	90	10	8	335
12	86	85	15	9	303
14	87	80	20	10	282
16	88	75	25	6	253
18	83	70	30	7	233
20	84	65	35	8	191
22	85	60	40	9	173
24	86	55	45	10	155
26	87	50	50	6	147
28	88	45	55	7	110
30	83	40	60	8	104
32	85	30	70	10	62,0
34	87	20	80	7	30,2
35	88	15	85	8	19,1
36	83	10	90	9	15,1
37	84	5	95	10	5,2
38	85	0	100	6	2,6

The solution adopted in solving these disadvantages consisting in conceiving an original software, allowing an accurate computation of coefficients and to obtain correct solution of the equation's.

We note that the model can be applied when other parameters characteristics to the mining process are used, of course with appropriate alterations in the proposed mathematical model.

CONCLUSIONS

- The quality of mined out output should became a priority objective in high rank manager's responsibilities, in order to obtain maximal incomes from coal valorization;
- MEBS represents a new mining method, at least when considering the qualitative influences which should be known as a effects on cost prices and incomes;
- Applying MEBS changes the weight of coal mined out by different mining methods, with 50% including major consequences on coal's quality. In working faces exploiting coal by caved bench method an upper control limit and an upper tolerance limit had to be fixed and permanently monitored and acting so that they will be not exceeded.
- The proposed mathematical models established a correlation between the qualitative and quantitative parameters and their influence on the possible incomes. The model can be also employed when other characteristic parameters of the mining process are introduced, with corresponding alteration of the mathematical model;
- It is advisable and feasible to make expeditious interventions for quality correction at high output level working faces, if portable measuring devices and instruments will be available to rapid assessment of coals ash content;
    - Correlation of rapid analysis data with proposed mathematical model considerably enables the obtaining of the economical effects of mined output's quality;
    - The research carried out can be spread to other mining methods currently applied in the practice of collieries within the Valea Jiului coal basin.

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50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.197-198

RATIONAL APPROACH FOR VALUATION OF EXPLOSIVES EFFICIENCY

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ABSTRACT
The article treats the problem related to the choice of suitable explosive agents and the valuation of its efficiency in given conditions. It is established that for the right choice it is requested to take into account the energetic characteristics of the explosives that affect considerably the expenses for realization of drilling and blasting works.

The real assessment of the economical efficiency for the used explosives can be completed on the bases of a complex valuation of the expenses for the general processes during mineral recourses extraction – drilling and blasting, loading, transporting, mechanical crushing etc. The full influence of the expenses for these processes regarding the choice of one or other explosive while making objective economical analyzes cannot always be absolutely reliable. That is why during optimization of the parameters of drilling and blasting works often analyzes only of the expenses for drilling and blasting are made. When calculating the efficiency of the explosive usually the relative consumption of explosives, the charge diameter, the volume of the exploded space, the volume of the single charge, the quantity of explosive in a single blasthole, the crushing degree and other factors related to the quality of the realized drilling and blasting works.

At the choice of one or other explosive the enterprise is usually interested in the price. Moreover, the commercial valuation of the energetic parameters, determining the use of the explosive usually is not taken into account. Together with the price of the explosives at first place it should be considered their effective performance. In fact during a choice of the explosive it should be rated the technical efficiency of the explosive trough the blasting characteristics, determining the charge capability to perform efficient work in demolishing, crushing and displacement of the mineral resource.

The general parameters of the explosive agent, that affect mostly the energy consumption of the blast destruction and the potential expenses that should be considered during valuation are:

- Absolute weight energy E_T;
- Absolute volume energy E_O;
- Relative weight energy e_T;
- Relative volume energy e_O;
- Detonation pressure P_D;
- Pressure in the charge P_C;
- Detonation velocity V;
- Power coefficient $K_{M.}$

The absolute weight energy E_T (kJ/kg) is the explosion heat per unite of explosive masse. It is determined trough theoretical way of calculation.

The relative weight energy e_T characterizes the performance of the explosive agent, $e_T=E_T/E_{TE}$, where E_{TE} is the weight energy of the reference explosive.

The absolute volume energy E_O (kJ/l) characterizes the energy volume concentration of the explosive in the charge and for unite of volume is calculated by the formula $E_O = E_T.\rho_z$, where.ρ_z is the density of the charge in the blasthole.

A number of studies show that the volume concentration of the explosive energy (E_O) is one of the most important parameters, determining the action efficiency of the explosion at rock destruction.

It can be used for valuation of blasthole charges the linear density of the explosive energy $P_O = P.\ E_T$ (kJ/m), where P (kg/m) is the capacity (the linear density) of the explosive in the blasthole.

The relative volume energy e_O characterizes the volume concentration of the energy of the chosen explosive regarding the relative volume energy of the reference explosive and is equal to $e_O = E_O/E_{oe.}$

The detonation pressure P_D characterizes the shattering effect of the explosion. It influences the crushing intensiveness of the zone nearest the charge.

The pressure in the blasthole P_C is created by the blast gas products after the end of the detonation impulse and the full finishing of the chemical reaction.

The blast gases pressure depend on the medium detonation pressure, the density of the charge and can approximately be calculated by the formula $P_c = 0{,}5P_d\ \rho_z^{2{,}5}$. It is seen from the formula that important influence for the rise of the Pc is due to the density of load, depending mostly on the adopted technology of explosive agent loading. At mechanical load of coarse disperse explosive agents Pc increases, while at hand loading of cartridge explosive agents it decreases.

The velocity of detonation V (m/s) characterizes the velocity of release of the heat energy in the explosive agent. Whit the rise of the detonation velocity the energy of the shock wave increases, that contributes rise of the crushing range. The detonation velocity is affected by a number of factors and varies in wide limits. The general factors are the charge diameter, the loading density, and the particle size of the explosive agent, the initiating method.

The power coefficient K_M= Eo.V/EoeVoe reflects the complex influence of the quantity released during explosion heat energy and its velocity of release by unite volume of the charge of explosive agent regarding the reference explosive agent.

Between two similar in volume charges of explosive agent, detonating with equal velocities, the explosive agent having higher Eo will be most powerful, as for the same period of time a bigger quantity of energy will be released. Between two similar in volume charges of explosive agent and equal Eo, the explosive agent having higher velocity of detonation will be most powerful, as its energy is released for a shortest period of time.

The parameters of the used in our country explosive agents for underground and open airs conditions are given in table. As reference explosive agents are used in underground conditions explosive Amonit – 6, and for open air-conditions the explosive agent Naftonit – 0.

On the ground of the analyze the following conclusions can be made:

1. The valuation of the efficiency of a given explosive agent should be made by taking into consideration the concrete conditions and all expenses for the realization of the drilling and blasting.

2. While choosing an explosive agent all technical and experimental indicators of the explosive agents should be considered.

Explosive agent	Density of loading, kg\dm³	Detonation velocity, m\s	Absolute weight energy KJ/kg	Relative weight energy	Power coefficient
1. Explosive agent for underground conditions					
Amonit – 6	650-750	3600	4187	**1,0**	**1,0**
Geleks - hand loading - mec. loading	 700-800 800-900	 5000 5500	4600	1,10	 1,63 2,01
Elacit -710	1220	5300	2822	0,67	1,61
Elacit -720	1220	4800	3009	0,72	1,56
Metanit	650-750	3000	3050	0,73	0,60
2. Explosive agents for open air conditions					
Naftonit - 0 - hand loading - mec. loading	 850-950 1100	 2700 3200	3852	**1,0**	 **1,0** 1,37
Naftonit - A4	850-950	2900	4280	1,11	1,33
GDA - 70/30	980	3200	5485	1,42	1,94
GDA - 79/21 - hand loading - mec. loading	 1050 1200	 3300 3650	4187	1,09	 1,64 2,07
Elacit - 3400	1250	3668	4500	1,07	2,33

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Recommended for publication by Department of Opencast mining and blasting technologies, Faculty of Mining Technology

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.199-202

ORGANIZING EFFECTIVE AND ECOLOGICALLY FRIENDLY PRIMARY BLASTING OPERATIONS FOR EXTRACTION OF INTERT MATERIALS

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ABSTRACT
The technology for Effective and Cleaner Primarily Blasting for Extraction of Inert Materials at the Skakacitsa Rock Quarry, part of the Bulgarian Rail Road Company, was developed, tested and implemented as part of the work performed under project sponsored by Ecolinks, initiative of the US Agency for International Development.
The up till now used technology for primary blasting at Skakavitsa quarry utilized detonating cord connected to doubled blasting circuit on the surface of the blasting field and two clinches detonating cord attached to the blasting cap in each borehole. The delay was achieved through time-delay device for the detonating cord. This technology required relatively large spending on explosives, poor fragmentation of the rocky mass, considerable rock scattering, large air blast and seismic effect.
The newly developed technology for millisecond demolition of the rocky mass incorporating charge initiation from the bottom, introduced effective and ecologically friendly blasting operations suitable for the quarry's conditions. The technology's main objectives were as follows:
1. Reducing the average cost of explosives for extraction of 1m^3 compact rock mass by 25-30%;
2. Decreasing, by approximately 30%, the harmful gas-dust emissions in the environment released during primary blasting;
3. Decreasing, by over 50% the seismic effects of blasting on the environment, buildings, and equipment at Skakavitsa quarry;
4. Decreasing, by approximately 30%, the production of oversize rock that requires secondary blasting, as well as achieving more uniform fragmentation of the rock mass produced;
5. Sharp reduction of flying rock pieces in the blasting field and the related air blast by approximately 50%;
6. Reducing, by over 20%, the cost of extraction of inert materials and improving the production efficiency of Skakavitsa quarry.
The implementation of the new technology improved considerably the rock production process at the Skakativsa quarry.

The technology for Effective and Cleaner Primarily Blasting for Extraction of Inert Materials at the Skakacitsa Rock Quarry, part of the Bulgarian Rail Road Company, was developed, tested and implemented as part of the work performed under project sponsored by Ecolinks, initiative of the US Agency for International Development.

The objectives of the newly developed technical process and technology were:
1. Execution of full millisecond demolition of the rocky mass with the most appropriate delay interval.
2. Substantial reduction of the seismic effect of blasting within the allowable limits of 10-15 mm/s total velocity.
3. Sharp reduction of the negative effects of flying rocky pieces on machines, technology and people as well as reducing the harmful gas-dust emissions in the environment.
4. Achieving better fragmentation of the rocky mass, resulting of better use of the blast energy.
5. Reducing the cost for explosives for demolition of 1m^3 compact rock mass.

The ultimate goal of the new process was to realize maximum gains of the widely practiced systems of millisecond demolition of the rocky mass, as instead of imported are used new, produced in Bulgaria blasting materials.

ANALYSIS OF THE EXISTING AND UP TILL NOW USED BLASTING TECHNOLOGY

In recent years at the Skakavitsa quarry were developed and implemented various technological solutions as some of the more important are:
1. The drilling layout was changed from 2.5x2.5m - 2.5x3m to an average 3.5x 3.5m configurations of the charges as the cost for drilling was reduced considerably.
2. Developed and implemented was a new construction of the primer that replaced the previously used initiating charge of 2.2 kg "Amonit 6" (Lazarit) with 400g TNT cast boosters. As a result improved was the initiation of the main charge of explosive.
3. Eliminated were the detonating cord and No 8 detonator, previously used for the initial detonation, considerably improving work safety and effectiveness.
4. Discontinued were the usage of the expensive and not that reliable delay detonators type "Shuffler" import from Austria. Instead, introduced were Bulgarian millisecond electric detonators. The charges are detonated with a delay interval of 25 milliseconds between rows of blastholes.

In spite of the considerable improvements, the current technology has the following shortcomings:
1. The strong airblast and large sound effect, formed mainly by the blasting circuit of detonating cord laid on the ground and the wrongful initiation of the charges;

2. Considerable amount of flying rock pieces in the blasting field, which could damage the nearby equipment and poses a threat for workers in the field;
3. Extensive seismic effect on the environment as a result of the relatively large number of blastholes, detonating at the same degree at the current blasting scheme;
4. Frequent refuses on the main line of detonating cord, as well as misfired charges in the blastholes due to the inferior quality of the detonating cord. This caused frequent suspension of the work process, required multiple detonations of the same field, and doubling of the blasting circuit;
5. Ineffective utilization of the potential energy of the explosive due to the incorrect construction of the charges;
6. Relatively dense blast design, leading to increased cost of drilling.

These shortcomings are mostly caused by the following:

1) Incorrect, in light of the newest studies, construction and initiation of the charges.

Figure 1 illustrates the up till now used construction of the charges at the Skakavitsa Quarry. The charges occupy 2/3 of the lower section of the blasthole. The remaining 1/3 is filled with stemming consisting of the removed material at drilling.

The charge is initiated by 400g TNT cast booster connected to one or two coils of detonating cord that pass through the entire length of the blasthole (explosive and stemming). The detonating cord connects to the main line on the ground, which detonates with speed of over 5500m/s, as it reaches the bottom of the blasthole and initiates the booster. The main charge of explosive is then being initiated by the booster in an upward fashion.

Although this method is theoretically sound, when carried out in practice a combined initiation occurs, leading to few negative phenomena.

The passage of the detonating cord through the stemming with a speed of detonation of over 5500m/s spreads out the stemming prior to the detonation of the charge. Consequently, the stemming does not absorb the toxic gases released during the blast.

Even more importantly, the charge is not kept intact until initiated by the TNT booster. The detonation of the cord creates air blast affecting the charge in perpendicular direction, on its short side. The charge decomposes, becomes tighter, and partly detonates. Therefore prior to the actual detonation, the blasting agent deforms causing total disturbance of the blast. According to tests conducted by Swedish investigators this leads 25-30% lost of the potential power of the charge which means that there is 25-30% unnecessary expense for explosive. The results of these tests and analyses had led to development of more effective blast systems that take complete advantage of the power of millisecond demolition of the rocky mass without preliminary partial detonation of the charged explosive.

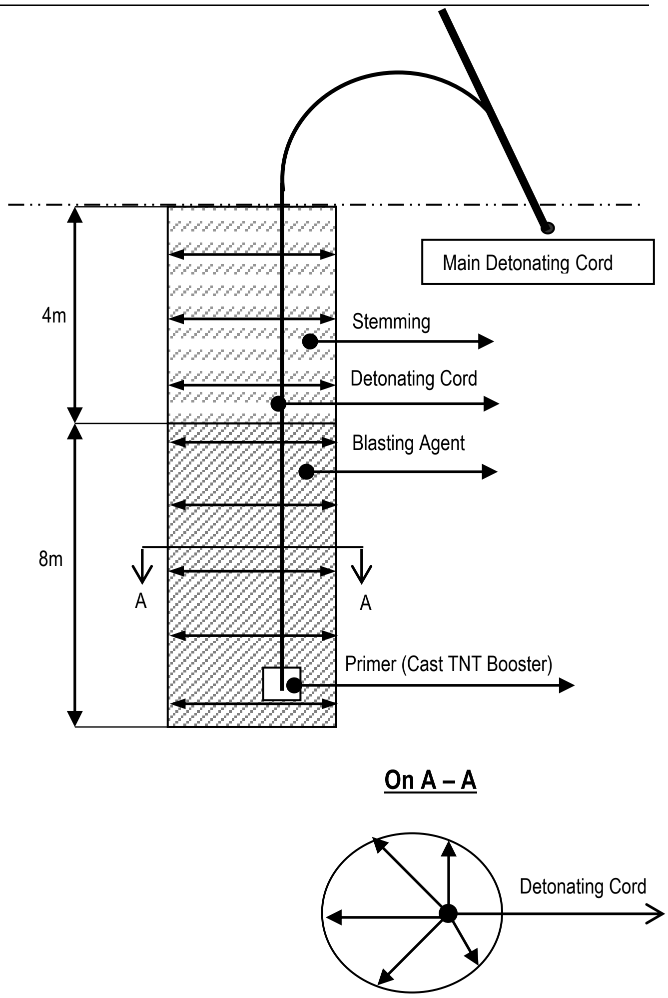

Figure 1: Charge construction and initiation with detonating cord.

2) Inefficient utilization of the millisecond delay to accomplish specific effects in blasting.

The up till now used millisecond delay design incorporates serial detonation. The main line of detonating cords are run along every row of blastholes as additional cords connect the charge with the main line. Each detonating cord on the ground is connected with one or two electric detonators of first to forth degree on the side of the blast field. When the delay interval between several rows of blastholes is the same, one detonator may connect two rows simultaneously.

This blast design is considerable improvement from the one previously used, however, it as well has number of weaknesses.

Firstly, one interval of millisecond delay is used to detonate large number of charges –from 12 to 20. This leads to relatively large seismic effect and poor fragmentation of the rocky mass.

Secondly, the large amount of detonating cord that detonates on the surface of the blast field causes substantial airblast, sound wave and flyrock.

Thirdly, the blasting circuit of detonating cord is not doubled although the inferior quality of the detonating cords leads to frequent refuses and the need of multiple detonations on the same rows of blastholes.

Based on the conducted investigation and analysis of the technological experience, abroad and at home, was developed new blast technology and process suitable for the conditions of the Skakavitsa quarry.

NEW BLAST TECHNOLOGY FOR SKAKAVITSA QUARRY

Based on the conducted preliminary research established were the following restricting conditions for the development of the new technology:

1. The number of concurrently detonated blastholes with first delay interval should be in the range of 2 to 6, as the optimal number is 3 to 5. This will help create better interaction between the charges, achieve better fragmentation of the rocky mass, will minimize the seismic effect and optimize the use of the blast energy.
2. The number of the blastholes at a single field should not exceed 60, as in particular circumstances they may reach but not exceed 80. This could be achieved by increasing the size of the blasting circuit and introducing ten degrees of delay; threby satisfying the requirement that 3 to 5 blastholes are detenoated at the same delay interval.
3. The resistance of the entire blasting circuit should not exceed 350 Ώ using electric detonators with increased safety class P, safety current 0.45 A, steady current 2A and safety impulse 18-20 A^2 ms.
4. In order to minimize the seismic effect the overall amount of explosive in a delay interval should not exceed 600-800 kg.

New charge construction and initiation method

A key element of the new technology for primary blasting is the new construction of the charge and the method of its initiation.

The leading goal of the new process is to keep the charge of blasting agent intact until the booster initiates it. The same can be said about the stemming above it. This goal could not be achieved at the up till now used charge construction (fig.1) that utilized detonating cord for initiation of the TNT booster.

At non-electrical systems this issue is solved by using special non-electric cord, which:

- Takes initiation by blasting cap detonator № 8;
- Transmits detonation to blasting cap placed at its base;
- Has a detonation speed of 2000 m/s, but due to its construction it does not breaks down and does not create blast effects on the ground.

The non-electric cord connects the delay detonator to the loaded in the blasthole charge. The primer is constructed by portion of the non-electric cord and the TNT booster. The initiation of the entire charge of explosive is carried out from down up, as the column of blasting agents and stemming remain intact.

This system is very reliable, however the cost of the cord with length of 12-18m ending with the detonator is on average 2.5 – 3.0USD, which is rather high for Skakavitsa quarry. To achieve the exact same effect we have developed charge construction and initiation with Bulgarian materials.

Instead of detonating cord, as used in the current charge construction or non-electric cord for use with non-electrical circuit, the new construction utilizes electric detonators with 12m copper wire, ending with delay detonator. The new construction is given in Figure 2.

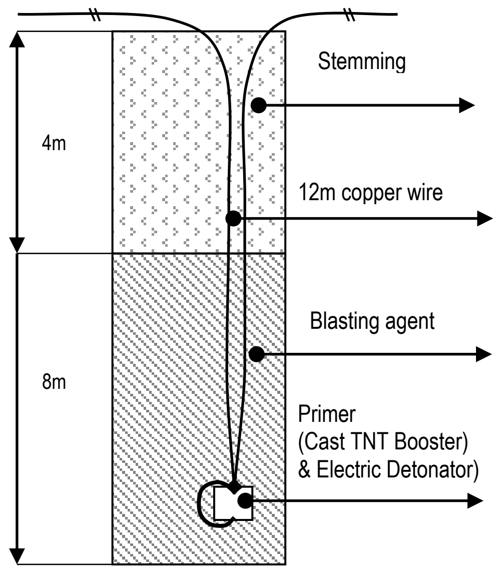

Figure 2: New charge construction for millisecond blasting circuit

The similarities with the non-electric system are:

- The charge of the blasting agent remains intact until detonation;
- The stemming remains intact until the blast and completely satisfy its functions;
- The charges are initiated from below with the selected most appropriate millisecond delay;
- The reliability of the blast has been increased;
- Better utilization of the power of the blasting agent with 25-30%.

The differences with the non-electrical system are:

- At the non-electrical system the delay detonator for the cast TNT booster is initiated by the detonation of the non-electric cord, whereas at the electric system initiation is carried out by electric detonator, class P with intrinsic safety;
- At the non-electric system the blasting system is composed of additional connection boxes (connectors) with pieces of non-electric cord and blasting cap detonator while the new blasting circuit is composed from the wiring of the electric detonators.

Preliminary investigations, performed during the development phase of the new charge construction, suggested more optimal connection pattern between the delay detonator with both the non-electrical and electrical blasting circuit.

Tests with both systems demonstrated reliability of the entire blast design as all expected results were attained.

New design of the blasting field

The main objective of the new blast design is to fully utilize the effects of the millisecond blasting which is the foundation for the two proposed technologies.

In accordance with the established restrictions, the blasting field has been designed with 60 to 80 blastholes, as the same delay interval detonates 3 to 5 blastholes and no more than 300-600 kg blasting agent.

The non-electrical system is based on the same basic restrictions. The delay boxes on the blasting field could have 3 to 4 connectors.

The key parameters of the new blast design are:

- The total number of blastholes on the blasting field is between 60 and 80.
- The maximum blastholes detonated with one delay interval are 3 to 5;
- The electrical detonator is of 10 degrees at a minimum and has delay interval of 25ms.

Figure 3 represents an example of millisecond blasting circuit that was proposed for experimentation and implementation at Skakavitsa quarry. The particular blast design depends on the number of the blastholes within and between rows.

At present work at Skakavitsa quarry is being done using blasting machines type L-200, having resistance 250 Ω, providing impulse over 18 A^2ms, and safety current 2 A. This blasting machine satisfies the requirements for 60 to 80 electric detonators in series.

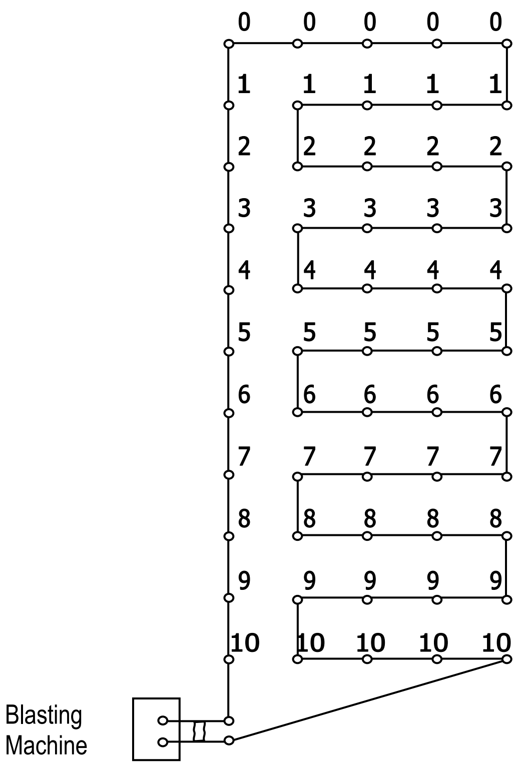

Figure 3: Blasting circuit with millisecond detonation for primary blasting operations.

When more detonators are being detonated in series, as well as from consideration for increased work safety we recommend blasting machine type L-300. Table 1 lists the main parameters of the blasting machines types L-200 and L-300.

Table 1: Technical parameters of blasting machines types L-200 and L-300

Parameters		*L-200*	*L-300*
1.	Mass, kg	1,6	1,6
2.	Tension, V	900±50	1200±50
3.	Capacity, mF	16	16
4.	Energy, Y	6,48	11,52
5.	Maximum resistance in series of Electric Detonators, class P, Ω	250	370
6.	Maximum amount Electric Detonators	80	100
	with 12m copper wire:	80	148
▪	in series	120	220
▪	in two parallel series	>18	>18
7.	Safety impulse at electricity 2A, A^2ms		

As a result of the development and implementation of the new technology for primary blasting operations at the Skakavitsa quarry utilizing millisecond demolition of the rocky mass can be summarized as follows:

1. Reduced was the average cost of explosives for extraction of 1m^3 compact rock mass by 25-30%;
2. Decreased were by approximately 30%, the harmful gas-dust emissions in the environment released during primary blasting;
3. Decreased were, by over 50% the seismic effects of blasting on the environment, buildings, and equipment at Skakavitsa quarry;
4. Decreased were by approximately 30%, the production of oversize rock that requires secondary blasting, as well as achieving more uniform fragmentation of the rock mass produced;
5. Reduced were the flying rock pieces in the blasting field and the related airblast by approximately 50%;
6. Reduced were by over 20%, the cost of extraction of inert materials and improving the production efficiency of Skakavitsa quarry.

The newly developed technology is applicable and it currently being used in other quarries for extraction of inert materials.

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Recommended for publication by Department of Opencast mining and blasting technologies, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.203-205

STUDY ON TOXIC GASES RELEASED FROM POWDERED AND EMOULSION TYPE EXPLOSIVES USED IN UNDERGRWOUND BLASTING

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ABSTRACT

The article describes the tests performed on toxic gas emissions of detonated powdered and emulsion explosives for work in underground conditions, not hazardous for explosions of gas and dust.

The study looked at and compared the two available methods for assessment of toxic gas emission – the traditional and presently used method utilizing 40-120 L chamber and a new method utilizing 142 m^3 chamber.

The following conclusions could be drawn from the obtained results:

1. The two tested ammonium nitrate type explosives makes "Amonit 6" and "Lazarit" release substantial amount toxic gases Carbon Oxide (CO) and Nitrogen Oxides (NO_x), 40-60 L/kg, exceeding the regulatory allowed 193 and 215 L/kg respecitvely.
2. The two ammonium nitrate type explosives emit considerable amounts nitric gases, which are extremely poisonous environmental pollutants. These gases are formed even though the tested explosives are balanced, i.e. have an oxygen balance close to zero. Both explosives, however, far exceed the allowed 100 L/kg toxic gas emissions.
3. The emulsion explosive "Elacit 710" emits relatively less toxic gases, probably because it contains 12-13 % water in its composition.
4. The conducted tests determined that the substantial toxic gas emission by the balanced explosives is caused by peculiar, newly discovered factor "L" in the chemical reaction. This factor affects all of the tested explosives almost uniformly. Factor "L" is currently being studied.

The most commonly used in Bulgaria industrial explosives designed for both surface and underground, not dangerous for gas and dust blasting operations are the powdered ammonium nitrate type explosives "Amonit 6" and "Lazarit", and the emulsion explosive "Elazit 710".

The Bulgarian regulatory documents (BDS, ON) stipulate that the allowed toxic gases released at detonation of a kilogram of the above named explosives are in the range of 40-60 L conditional carbon oxide (CO).

Explosives emitting more than 100 L toxic gases recalculated for conditional CO are not permitted for work in underground conditions by the Bulgarian Regulations for Labor Safety in Blasting Operations (article 215). Balanced explosives are those with zero oxygen balance.

The existing method for assessment of emitted toxic gases from powdered explosives is shown on Figure 1. In a small compression chamber (bomb), having volume from 20 to 120 liters is placed a small thick-walled mortar with 30-40mm groove. The sample of the tested explosive is placed in the groove. For best results the density of the sample (ρ) should be about 1 g/cm^3 as the tested explosive is between 20 and 40 g. Initiation is conducted with electric detonator of zero degree. In order to prevent damage to the chamber, it is sealed with vacuum pump, as the remaining pressure is carefully noted. Following the blast, the gas is measured using one of the below described techniques.

In the first technique, the gas sample is taken with vacuum pipette, which is refilled several times in order to reduce the concentration of the toxic gases.

The second, the atmospheric air is let into the chamber as the pressure is being equalized. Next, from the rarefied gas in the chamber is taken sample using vacuum pipette or other specialized apparatus (e.g. Dreger) for direct determination of the toxic gases discharged in the chamber.

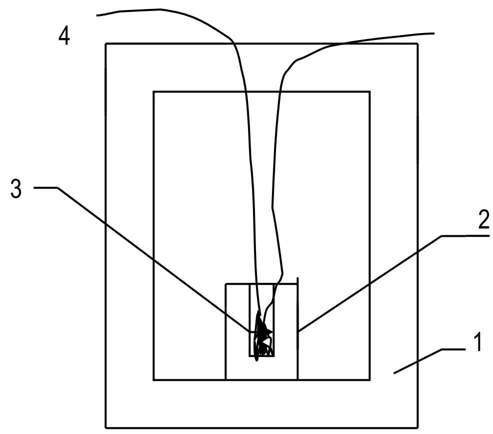

Figure 1. Small Chamber (bomb) 40- 120 L
1-chamber; 2-mortar; 3-explosive; 4- electric detonator

The quantity of gases emitted from a kilogram explosive is determined by the formula:

$$L_{(CO.Nox)} = \frac{(x)(1000)(V)}{(100)(g)}, \ L/kg \qquad (1)$$

where:
x is the percentage amount of CO and (NO_x) in the chamber
g is the weight of the sample explosive, g
V is the volume of the chamber, L.

From the so determined gas quantity, the gases resulting from the electric detonator being used are set apart and the remainder is transformed to conditional CO using the following formula:

$$\text{Conditional CO} = LCO + 6.5\ L\ (NO_x)\ ,L/kg \qquad (2)$$

The examined techniques, however, provide inaccurate results due to the following problems:
1. The quantity of the tested explosive is extremely small and the method does not give realistic idea of the true toxic gases emissions at detonation;
2. The test is performed in vacuum, however that is not suitable for this type of assessment;
3. The testing conditions does not correspond to the actual conditions in which industrial blasting are conducted.

To compensate for the shortcomings of the discussed techniques and in order to obtain a realistic idea of the mechanism of toxic gas emission in underground conditions we have developed a new method utilizing large compression chamber, with measurements 5x5x5 m and interior volume 142000 L or 142m^3. This method does not involve vacuum and the testing reflects real world conditions.

Shown in Figure 2 is the new compression chamber.

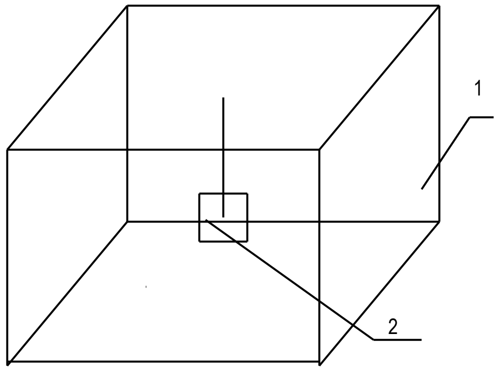

Figure 2 Compression Chamber (142m^3)
1- chamber; 2- charge explosive

The newly developed method is based on the following conditions:
1. The quantity of the tested explosive is in the range of 500 to 1000g
2. Pole blasting tests are performed on:
   - Open charge in original manufacturer package
   - Charge placed in thick-walled steel pipe

Table 1. Results from toxic gas study by the new methodology

No	Type Explosive	Test No.	Quantity of Explosive, g	Results		
				CO L/kg	NO_x L/kg	Cond.COL/kg
1.	Amonit6 32mm					
1.1	Open charge	3	700-1000	33,01	25,14	196
1.2	In steel pipe	1	600	46,39	21,06	183
	Average	4	700	36,36	24,118	193
2.	Lazarit 65mm					
2.1	Open charge	3	500	21,86	29,694	215
3.	Elazit 710					
3.1	Open charge	2	740	15,35	6,654	59
3.2	In steel pipe	2	740	45,06	4,965	78
	Average	4	740	30,20	5,827	69

3. Modern equipment with capability to simultaneously measure the total concentration of total toxic gases is being used, eliminating the need for various apparatus with different accuracy requirements for the different gases. For the purposes of this study we have used the licensed contemporary equipment of the Ministry of Environment and Water.

New round of tests were performed on the powdered ammonium nitrate types “Amonit 6”, “Lazarit”, and the emulsion explosive “Elazit 710” using the just described new methodology. The obtained reults are given in Table 1.

The toxic gases study by the new methodology leads to the following basic conclusions:

1. Found was that the two tested ammonium nitrate type explosives makes “Amonit 6” and “Lazarit” release substantial amount toxic gases Carbon Oxide (CO) and Nitrogen Oxides (NO_x), 40-60 L/kg, exceeding the regulatory allowed 193 and 215 L/kg respecitvely.
2. Both “Amonit 6” and “Lazarit” release large amount nitric gases, extremely poisonous heavy ecological pollutants. These gases are formed even though both explosives have oxygen balance of around zero, which means they are balanced. Furthermore, both tested explosives considerably exceed the regulatory standards for toxic gas emissions of 100 L/kg.
3. The emulsion explosive make “Elazit 710” releases relatively less toxic gases, probably because there are 12-13 % water in its composition.

4. There was not significant difference for tests conducted on explosives in original manufacturing package or placed in thick-walled steel pipe. Either way the above mentioned conclusions are valid.

The values presented in Table 1 have been corrected for toxic gases released during blasting by the electric detonator, which tested with the same apparatus, are on average 0.14 L/kg conditional CO.

Based on the conducted tests determined was that the substantial toxic gas emission by the balanced explosives is caused by peculiar, newly discovered factor "L" in the chemical reaction. The factor effect is almost equal for the all three tested explosives. The study of this "L" factor is an ongoing process.

The conducted study suggests that there is significant difference in the toxic gas values obtained by using the traditional and the newly developed methods. Extensive testing and investigation of the new "L" factor is also essential in the development and assessment of new makes explosives, permitted for industrial blasting operations. Serious investigation of these issues will facilitate resolution of many current problems caused by the negative affects of industrial blasting on work health and safety and the environment.

The results obtained from this study should be considered and incorporated in the Bulgarian regulations, in the training of specialists in the area of blasting techniques and technology, and in the assessment and measurements for reducing harmful emissions in the environment.

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Recommended for publication by Department of Opencast mining and blasting technologies, Faculty of Mining Technology

DETONATION PROCESSES AND THE EFFECTS THEREOF ON ECOLOGY DURING TOTAL LIQUIDATION OF URANIUM EXTRACTION CONSEQUENCES IN THE REGION OF ELESHITSA VILLAGE

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ABSTRACT
The present issue discusses some of the basic moments of liquidating the consequences of uranium extraction in the region of Eleshnitsa village. On the basis of the ecological risk specific aspects of the detonation processes are given for the technical liquidation of buildings and equipment as well as the effect on the ecology of the region. The financial aspect of the problem is mentioned in general.

CONDITION OF THE PROCESSES OF LIQUIDATING THE CONSEQUENCES OF URANIUM EXTRACTION IN THE REGION OF ELESHNITSA VILLAGE

Since the beginning of the 1960's the region of Eleshnitsa village has been subjected to an active extraction of uranium ore and its processing. The region covers the mountain slopes to the west of the village. The ore extraction was mainly subterranean but later it had been carried out in open mines as well.

To close the extraction/processing cycle an ore dressing plant "Zvezda" was built and set into operation at that time together with a toxic waste storage. Mainly ores from that region and from Barutin were processed in the plant. The final process phase was a semi-industrial product – ammonia uranylcarbonate (yellow cake).

The activities, connected with the extraction and processing of uranium ores in the region have, as a whole, caused a disbalance in the environment components, namely terrain structure violence, soil surface pollution due to ore transportation and deposition of extracted rocks in piles, underground water pollution, etc. But the most serious and problem causing pollution comes out of the plant operation.

The technological waste, deposited in the toxic waste storage is about 10 to 12 million cubic meters with the height of filling the storage reaching 70 meters and the length – 1 200 meters.

The intensive ecological disbalance is now covering the hollow with the Dindirishki, Oreovsko, Valcho and several other gulches. Together with the elimination of the ore extraction activities, started in 1992, some of the problems and mainly the ones related to the terrain, have been solved. The technical liquidation of the mines and the technical processing of the rock piles have shown no discredit up to now.

Under the PHARE Program the sealing and the treatment of the toxic waste storage have been started in 2002 and that will solve a further problem related to the ecological aspect of the liquidation process.

Anyhow, a problem remains the process of liquidating the plant site. The buildings and equipment on an area of about 60 000 square meters have been partially liquidated and partially repaired within the period of 1992 through 1996, but, practically, that has not created conditions for normal operation on the site and for avoiding the risks related to health and ecology.

The construction waste from the activities within that period is about 400 cubic meters in volume; it has been obtained conventionally (without applying detonation) and has been thrown on the slope by the toxic waste storage.

The seriousness of the problem of total and correct completion of the liquidation process comes also out of the fact that a part of the plant site will be used under exploitation conditions. The restored part of the regeneration process line for ion-exchange resin, intended to provide for the processing of enriched resins from all over the country, as well as the administrative building will require safe working conditions both during this process and after its completion.

The buildings and equipment with a total volume of about 6 500 to 7 000 cubic meters which remain for technical liquidation are situated between the administrative building and the regeneration line; this requires a careful and precise choice of the destruction technology to avoid the secondary dust pollution of the terrain of the already repaired plant site, on one side, and the remaining equipment, on the other side.

TECHNICAL LIQUIDATION OF THE BUILDINGS AND EQUIPMENT ON THE ZVEZDA PLANT SITE

When making a choice about the approach and methods of technical liquidation of any object with similarly situated buildings and equipment it is necessary to account for the following main requirements:
- guaranteed elimination of the health/ecology risk during the liquidation process;
- maximum preservation of the ecological balance of the various components of the environment during and after process completion.

The application of the manual (mechanical) method or the detonation one depends on the degree of the risk for the two methods.

Risk, arising in the destruction works from a dust cloud – carrier of a radioactive pollution, its orientation, size and intensity within the separate zones.

Risk, arising from loading-unloading works – size of the pieces of the destructed material, possibility for a secondary local dust pollution both during loading-unloading and transportation.

Risk, arising during depositing the material from the liquidation.

The analysis of the risk factors, as mentioned above, for the liquidation activities, carried out up to now for similar objects, shows a definite advantage in applying the detonation processes.

The assessment has been made by applying a rank system for risk assessment for defining the macro factors, giving each one a certain weight.

Taking into consideration the separate weights of the macro factor groups with an absolute priority given to the radioactive pollution resulting from dust pollution, the applying of the detonation method is recommendable for this object.

From ecological point of view, the effect to be reached in applying this method, consists mainly in minimizing the secondary pollution of the plant site and the necessity of secondary repair. The expected outlines of dust pollution remain within the frames of the work sites. When strictly conforming with the detonation technology (which is controlled and supervised during and at the time of detonation by Special Control Authorities)q the expected difference in dust pollution before and after the detonation is within the range of not more than 10% for the zone with d radius of 20 meters from the foundations of the destroyed object.

One advantage of the health-ecology aspect is the minimizing of the stay time of the operation personnel on the site, absolute elimination of any condition for accidents with the workers, elimination of pollution contours, even small in size, as a result from construction wastes falling from certain heights.

As the financing of the liquidation activities is provided by the State Budget the last but not the least reason for applying this method are the lower costs.

Big volumes of materials for destruction – about 7 000 cubic meters with prevailing contents of massive reinforced concrete (foundations, reservoirs, etc.) will require much more financing if manually destroyed in comparison with the detonation method, which represents a modern and dynamic way of terrain cleaning.

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Recommended for publication by Department of
Opencast mining and blasting technologies, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.209-213

POSSIBILITIES OF OBTAINING CONSTRUCTION MATERIALS USING THE SOLID WASTES FROM MINING INDUSTRY

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ABSTRACT
The coal mining processing industry is the central axis of the area's economical life in the Jiu Valley. Actually all these made the area to be faced with the sharpest problems regarding the environment pollution which have a direct impact on the health status of the population as well as on the area's fauna and flora, despite the fact that nature had endowed the Petroşani area with the most beautiful relief in the country.In this region there are wastes spoil dumps, decantation ponds of tailings from coal flotation and fly ashes deposits from the thermoelectric power station. They are occupying large amounts of lands, although they represent secondary sources of valuable raw materials, actually used on a small scale.The aim of our research was to establish the possibilities of using these secondary resources to obtaining construction materials and the first results are encouraging. We used tailings from decantation ponds, fly ashes and slag resulted from the coal burning process.
In order to obtain very good bricks with qualitative parameters acceptable by the beneficiary, we took into account the porosity, the compression resistance and the specific weight of the final products.We made very detail research to the obtaining mode of the mechanic froth so necessary for a good porosity of the bricks. Here we used a new device which is the object of an innovation certificate.

INTRODUCTION

The paper the research to the laboratory scale made up to the Mineral Processing Department in order to try an integral capitalization of the fly ashes from the thermo-electric power station.

In the construction materials industry exists a new trend to use so called "cheap raw materials" to the producing thermo and phonic isolation products obtained with or without froth adding. The chemical-mineralogical similitude of some industrial wastes which are partial used, could be a favorable premise to replace some usual materials in the same financial efficiency conditions. The industrial wastes quantities are higher every year, their depositing having a negative economical and environmental impact.

In the research approaching we took into account the achievement of new ceramics type BCFA with volumetric weight reduced and with better thermo-phonic isolation parameters, using fly ashes.

The fly ashes adding change the physical-mechanics and chemical properties of concretes in the next ways:

- Diminishes the hydration heat;
- Reduces the contractions and the deformation under the lasting charge;
- Increases the resistances given by waters and chemical agents;
- Lower costs by the partial replacing of the cement with fly ashes.

THE FLY ASHES CHEMICAL COMPOSITION

The inorganic substances from coal, by burning process are transformed in ash. The ash contains some elements so called "major elements" in higher quantities (above 1%). The decreased order of these elements is: Si, Al, Fe, Ca, Mg, S, Na, K, Ti, P. The major elements distribution determines the oxide composition of ashes. Moreover macroelements in coals there are also microelements (between 1% and 1 ppm content). In table no.1 are presented the macro and the microelements contents from the Paroseni thermo-electric power station deposit:

Table 1. The major and microelements contents of fly ashes deposit

Macroelements, %		Microelements, %	
SiO_2	47.12	Cu	0.039-0.055
Fe_2O_3	8.68	Pb	0.057-0.030
Al_2O_3	20.08	Zn	0.55-0.70
TiO_2	0.055	As	0.04
CaO	6.30	V	0.006-0.01
MgO	2.50	Mn	0.12
SO_3	0.22	Ni	0.005
Na_2O	0.98	Co	0.007
K_2O	1.87	S	0.22

The prevalent components are: SiO_2, Al_2O_3 and Fe_2O_3 in a weight of 70%, an indicator of the possibility to form of vitreous phases and of silicates, with favorable effects on the hydraulic capacity of these ashes.

In the lack of an international criterion of the ashes classification, it was proposed one, by Academic Press (USA). The classifying criterion are the report $\%SiO_2/\%Al_2O_3$ and the quantities of %CaO and $\%SO_3$. In according with this criterion, the ashes can be classified in four classes:

- aluminous – siliceous ashes with $\%SiO_2/\%Al_2O_3<2$ and %CaO < 15;
- siliceous – aluminous ashes with $\%SiO_2/\%Al_2O_3>2$ and %CaO < 15;
- sulphur – calcium ashes with %CaO>15 and $\%SO_3$ >3
- calcineous ashes with %CaO>15 and $\%SO_3$ < 3.

We made the trials to obtain bricks with siliceous-aluminous ashes and the choice is based on the following conclusions:

- the microelements contents are very small comparatively with a value minimal exploitable and therefore it is not viable the idea of the recovering these elements;
- comparing the microelements contents from the Jiu Valley coal and from the ashes resulted by burning, it is obviously a tendency of few chemical elements enrichment, e.g. : Ag, Au, Pb, Be, As, Mo, Ge etc.

THE FLY ASHES MINERALOGICAL COMPOSITION

The fly ashes pond from Paroseni (Jiu Valley) is a result of the superposition of two feeding sources, one inorganic and another of organic nature.

Above these two chemical and mineralogical basis structure there is a thermal structure with uncombustible elements concentration.

The inorganic sedimentary zone is a clay mass which includes a chemical-mineralogical complex formed by iron and titanium oxides and hydroxides, calcium and magnesium carbonates and sulphates, iron, lead, zinc and native elements e.g.: gold, silver, platinum elements, etc.

The organic sedimentary zone is represented by the chemical elements accumulated in the plants mass from coal forming process. This organic mass contains the basis compounds such as: carbon, hydrogen, oxygen, nitrogen, but also a lot of useful elements from soil. These elements are: gold, silver, zinc, beryllium, cadmium, tin, tellurium, germanium, manganese, cobalt and nickel and in addition there are a lot of elements without nutritive value e.g.: sodium, chlorine, radium, rubidium.

In the table no. 2 are presented a mineralogical analysis of the fly ashes from the Paroseni pond.

By the mineral substances included in the thermodynamic burning conditions followed by a cooling process, the fly ashes are constituted by two phases: a crystalline phase (12-14%) and a vitreous phase (66-88%).

By a correlation of thermal and roentgenographical analysis it was pointed out the main transformations of the mineral components of coals to their burning in the thermo-electric power station. The physical analysis shows that the fly ashes deposit from Paroseni looks as a compact powder, microporous spheres or compact glassy spheres; another characteristics are: the grinding fineness is about 68 % - 0.074 mm, low permeability and a high magnetic susceptibility.

Table 2. The mineralogical composition of the fly ashes from Paroseni pond

The mineral	Chemical formula	Content, %
Magnetite	Fe_3O_4	10
Hematite	Fe_2O_3	
Sphene	$CaTi(SiO_4)$	0.1
Pyrites	FeS_2	0.35
Calcite	$CaCO_3$	3.0
Dolomite	$Ca, Mg(CO_3)$	
Pb, Zn, Cu sulphates and carbonates	-	0.1-0.15
Metakaolinite	$Al_2O_3*2SiO_2$	
Kaolinite	$Al_4(Si_4O_{10})(OH)_8$	30-35
Chlorite	$(Mg,Fe)_5(AlSi_3O_{10})(OH)_8$	
Artificial silicates	-	35-40
Quartz	SiO_2	5-10

THE GEO-CHEMICAL PROPERTIES OF THE FLY ASHES

The geo-chemistry of the fly ashes from the Jiu Valley are grouped in: siderophile elements (Fe, Pt, Pd, Au), lithophile elements (Zn, Ag, Ga, In, Tl, Pb). The microelements from coals and which by the burning process pass in fly ashes proceed from two sources:

- elements from plants, e.g. : Fe, Zn, Au, Ag, Bi, Pt, Ge:
- elements accumulated in separable ash by deterioration processes, levigation, sedimentation and diagenesis e.g.: Si, Al, Fe, Mn, Ti, Zr.

In conclusion, starting from the physical-mechanical characteristics of the fly ashes we tried an integral and efficient capitalization in the construction materials domain.

EXPERIMENTS AND RESULTS

The physical-chemical processes which determine the hydraulic reinforcement capacity of the fly ashes is the fundamental problem in order to establish the optimal methods of the fly ashes using.

The hydraulic properties of the ashes are the result of the presence of two oxides categories: acidic oxides (SiO_2, Al_2O_3, Fe_2O_3) and basic oxides (CaO, BaO, Na_2CO_3, Na_2SiO_3).

The basic salts using in the system ash-water generates reactions with hydrated mineralogical compounds which assures the resistance structure development. The activation and the reinforcement process is encouraged also by the chemical, mineralogical composition and by the report between the vitrous and crystalline mass, etc.

The fly ashes, after a cooling process in fast thermodynamic conditions has a strong acid character and a microstructure with a great vitrous weight (45-88%). The main crystalline components from the Jiu Valley ashes deposit are: mullite, quartz, hematite, magnetite and feldspars.

In the hydraulic reinforcement process, some mineralogical components are inert (mullite, quartz, magnetite) and another are active, determining the selfreinforcing reactions. The report between the vitrous and crystalline phases determines the value of hydraulic activity.

Simultaneously with the cvasicrystalline structure forming, increases the number of structural faults, the dezassociation degree and the polarity. These aspects conduct to the increasing of the ashes hydraulic activity.

A higher potential energy is benefit on the hydraulic activity of the ashes and conducts to superior resistance in a short time. The ashes from the Jiu Valley have very good reinforcement characteristics which could be increased by using some chemical substances called activators. The activators enhance the starting of some chemical reactions and physical processes which determine the obtaining of resistance structure and durability similar with the hydraulic binding materials. The best activator is the calcium oxide. This is indicated for his activity, first of all because it creates an optimum basic medium in the ash-water-activator system; this medium is capable to release the chemical reactions which is the basis of the resistance structure. The kinetics of the physical-chemical process at calcium activating ashes depends of some influence parameters such as: the $SiO_2+Al_2O_3+Fe_2O_3$ content, the specific surface, crystalline structure, carbon content, the $Ca(OH)_2$ quantity, etc. It was established that for the siliceous-aluminous ashes with the parameter % SiO_2 + %Al_2O_3 + %Fe_2O_3 > 70, the activating is efficient.

Starting from these reasons, we made up a lot of trials to the laboratory scale in order to obtain different types of BCFA (cellular concrete without autoclavisation) bricks. The different prescriptions for bricks tried to replace the cement from bricks structure with industrial wastes from ores processing plants and from thermo-electric power station. The wastes type was: slag and ashes from the coal burning in thermo-electric power station bellow 3 mm and tailings from cupriferous flotation plant.

The qualitative characteristics of bricks were: the compression resistance, the porosity and the bulk density.

The bricks achievement with high porosity requires the utilization of some froth agents. These froth agents are very important for the stability, the mechanical resistance and the bubbles uniformity. We tried 5 froth agents type and we found out an efficient frothing agent, a Romanian product called "Spumar".

An important place in our research had the modality to produce the froth added to the mixture: sand-cement- lime-ash. For this operation we used a new device so called hydro-airator. Inside this device there is an injector-jet exhaustor closed in a rolling case having an interior profile especially designed to perform a depression for assuring the suction of liquid phase. The process evolution in centrifugal field causes the concentric stratification of the principal fluid (the air) which shifts on descendent spiral path favoring the appearance of depression cone and formation of a spindle liquid phase (solution frothing agent in water) emphasized by the tangential velocities values of the concentric sheets.

No matter what the input velocity is, when evacuating the apparatus it is multiplied by approximately 8 times, extremely advantaging both for the amount of solution aspirated and for assuring suitable dynamics of the froth jet. The amount of aspirated solution quantity is directly dependent on the air velocity and pressure (about 0.3 barr) which could be modified by vertical adjusting of the injector. To get greater values of the feeding current velocities is possible not only by decreasing the concentric zone but by changing the flow direction of fluid sheets in the places where the direction changing are located the Coanda effect occurs, its consequence making possible the suction of the solution. In the figure 1 we present the frothing device used in the laboratory trials.

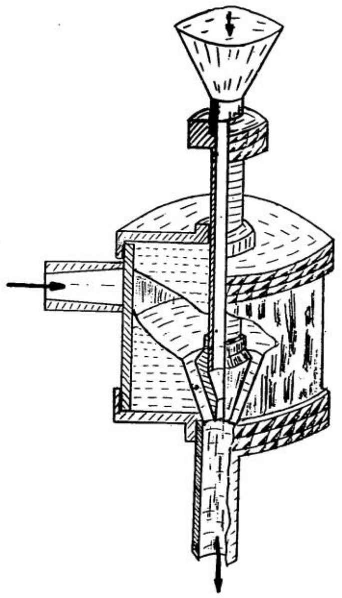

Figure 1. The hydro-airator device

The aim of our research was to obtain good results in economical efficient conditions, using different prescription for materials construction.

The replacing of the cement – an expensive product – with fly ashes in a proportion about 40%, was possible due to the vitrous phase and mineralogical components present in these ashes.

An increasing of the lime weight in the bricks composing had also a positive effect on the obtaining high values of the resistance structure. The fly ashes using has also a good effect in the porosity increasing and implicitly for the density reducing. These characteristics allow the utilization of the obtained bricks as a very good thermal-phonic construction material.

The component weights and the bricks characteristics are presented in table no.3.

The variation of the main characteristics of the bricks, with the ashes content are presented in the figures no.2,3 and 4.

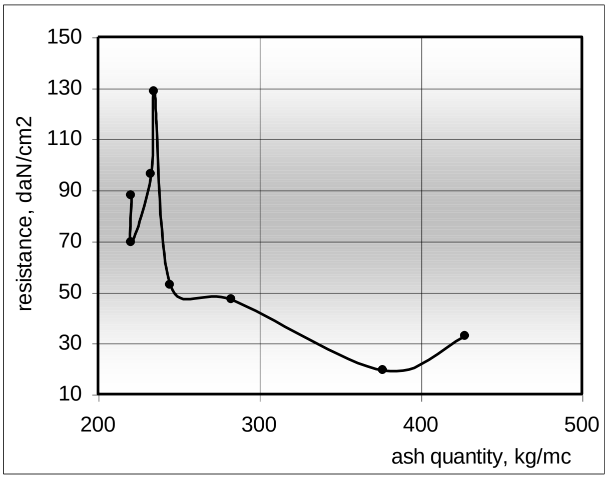

Figure 2. The variation of compression resistance of BCAF with ashes content

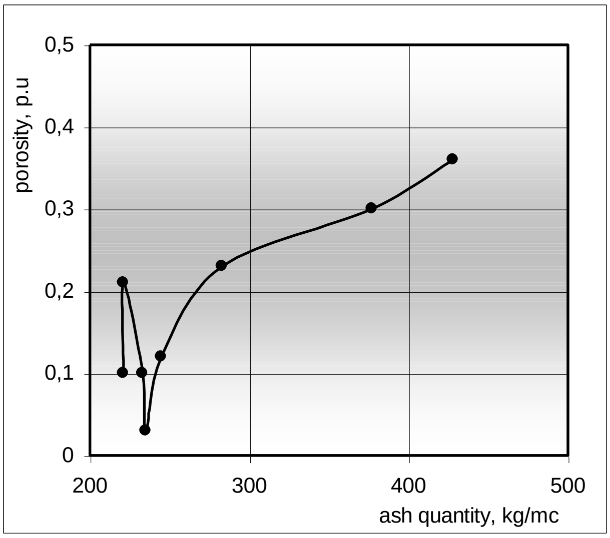

Figure 3. The variation of the porosity of BCAF with ashes content

The results of our research point out that the fly ashes can be successfully used in the construction material domain, because the characteristics of bricks type BCFA correspond to the legal prescriptions. For the greatest ash quantity used, we obtained bricks with a compression resistance above 15 daN/cm^2, which is the inferior limit. It's important to point out

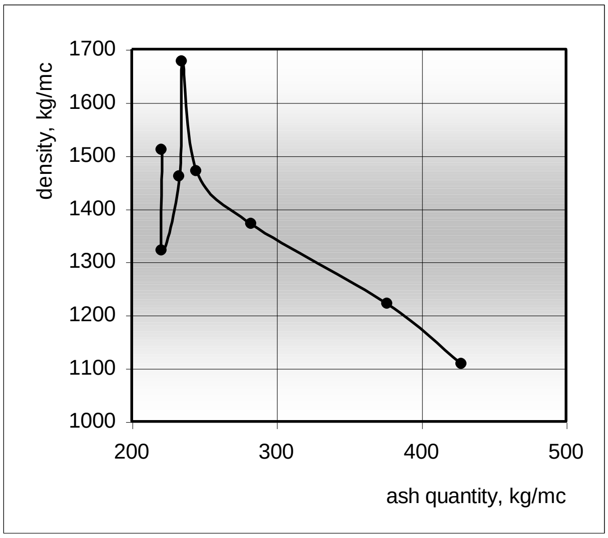

Figure 4. The variation of the density of BCAF with ashes content

also that in these conditions, the density is lower and this is another positive characteristic for a thermal-phonic material.

The results obtained to the laboratory scale conduct to the very important conclusion that this waste could be easily used in very efficient conditions in the construction materials domain, with a benefit impact on environmental pollution.

CONCLUSIONS

- the fly ashes deposits represent a viable secondary resources possible to be used with very good results in the construction materials industry with a positive effect on the ecological reconstruction of the Jiu Valley region;
- the specific characteristics of the fly ashes (especially the vitrous mass) have a benefit effect on the hydraulic reinforcement of the bricks type BCFA, which is a very good thermal-phonic construction material;
- the trials to the laboratory scale made up to our university have a novelty character by: the utilization of a new device for the obtaining the mechanical froth necessary to obtain a demanded porosity of the bricks, the increasing of the financial efficiency to the bricks processing by the replacing an expansive froth agent and of the cement weight with fly ashes;

Table 3. The characteristics of the BCFA bricks

Sample / Composition	1	2	3	4	5	6	7	8
Sand – 0.3 mm, kg/m^3	283	353	332	332	350	388	209	214
Cement Port.32.5	317	348	332	332	350	153	104	285
Ash , kg/m^3	283	235	221	221	233	245	377	428
Froth agent, l/m^3	0.8	1.2	1.0	2.0	3.0	1.0	10.0	7.9
Compression resistance, daN/cm^2	46.9	128.4	87.69	69.40	96.10	52.6	19.13	32.6
Porosity, p.u.	0.23	0.03	0.10	0.21	0.10	0.12	0.3	0.36
Density, kg/m^3	1371	1676	1510	1320	1460	1470	1220	1107

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Recommended for publication by Department of Engeneering Geoecology, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.215-220

ELEVTROCHEMICAL TREATMENT OF MINE WASTE WATERS CONTAINING HEAVY METAL IONS

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ABSTRACT
The paper shows an application of electrochemical method for purification of model solution and "Asarel-Medet"AD mine waste water, regarding heavy non-ferrous metal ions (Cu, Zn, Fe, Mn). The total activation energy of dissolution (48.5 kJmol) of iron electrode in sulphate solution during the electrocoagulation process was determined. The influence of electrocoagulation process conditions, such as, current density, pH of the solutions, duration of the treatment, additional aeration (50 l/h), and stirring of the solutions (1500 min $^{-1}$) was investigated. A comparison of the residual concentrations of the investigated elements in as-treated solutions, with and without addition of flocculent (Praestol 2531) was made. A technological scheme for purification of "Asarel-Medet" mine waste water is suggested, which meets the regulations for the analyzed elements, for waste water infused in water pools. (The Official Gazette, issue 97/2000, Ordinance Nr. 6). The advantage of the suggested technological scheme, in reference to one being currently in use in Asarel-Medet", is that, it eliminates the lime-wash feeding and the subsequent adjustment of pH up to 6, producing not only hydroxide and carbonate sediments, but also large amounts of redundant unconditioned sediments, as well as gypsum.

Key words: activation energy, iron electrode, electroflotocoagulation, heavy metals, mine waters

INTRODUCTION

Waste water (WW) amount grows steadily worldwide, especially the copper mine waters. According to K. Kovachev et al (1990) the amount of mine waste waters from copper mines in Bulgaria is 13.5 million m^3 per year, which is approximately 11% of the total amount of waste water resulting from non-ferrous metal yielding and processing. These waters contain insoluble coarse and fine-dispersion impurities, as well as dissolved substances that contaminate the environment. This requires a prompt development of waste water purification methods, aimed at attaining the emission regulations for WW from yielding and mineral processing of ores (The Official Gazette, issue Nr.97, Nov. 9, 2000).

Electrochemical purification of waste water (Iakovlev, 1987; Kulskii et al, 1987) is widely used. A number of autors take electrocoagulation and electroflotation not as separate process but like a combination of the two processes (Harchenko and Nasonova, 1985; Tabakov, 1986; Kovacheva and Parlapanski, 1996). It is implemented in specific devices called electrolytic cells, by electric current through waste water acting as electrolyte. Due to electrolysis, some changes occur, in round the electrode space, including changes in ionic composition (pH) and redox potential (Eh) of the water, coagulation of colloid-size impurities occurs, precipitation of coarse impurities is speeded up, and total hardness of the water decreases. Substantial part of ionic-molecular impurities gets removed with the sediment. Colloid-dispersion components in WW (the electrolyte) result from dissolution of metallic anode, usually iron (steel) for acidic medium. During electrochemical dissolution of iron electrode the following reaction could occur:

at anode:

$$Fe \xrightarrow{H_2O} Fe(OH)_{ads} \xrightarrow{H_2O,O_2} Fe(OH)_2 + Fe^{2+}.nH_2O \xrightarrow{H_2O,O_2} Fe(OH)_3$$

at catode:

$$2H_2O \leftrightarrow O_2 + 4H^+ + 4e^-$$

$$2H_2O + O_2 + 4e^- \leftrightarrow 4OH^-$$

Electrochemically obtained colloid iron hydroxides possess much higher sorption ability towards the impurities, than chemically obtained ones (Kulskii et al, 1987; Chanturiya, 1977).

Present study aims to investigate the opportunity for purification of model solutions and "Asarel-Medet" mine waters, from heavy metal ions (Cu, Fe, Zn, Mn).

EXPERIMENTAL PART

Determination of activation energy of anodic dissolution of iron electrode in a solution

Determination of activation energy of this process was reported elsewhere (Panayotova., Kovatcheva-Ninova et al., 2000). Polarization curves were taken by means of equipment shown on Fig.1. Polarization curves represent a dependence between current intensity (I, [A]) and the potential of working electrode in regard to calomel one(E, [V]). The curves were taken at four different temperatures: 20, 30, 40, and 50^0 C. Anode (low carbon steel 0.8 KPN, 0.25 cm^2 surface) dissolution was carried out in sulphate solution with sulphate ion concentration of 400 mg/L and pH 2.8, which is close to the real parameters of mine waste water. The curves are shown on Fig. 2, current values have been re-calculated as current density values (i, A/cm^2). Current density values *i* for each temperature *T* have been determined from the polarization curves plotted at constant potential value of 2 [V]. As-obtained values were used for plotting the dependence $\ln i/T^{-1}$, ^0K (Fig.3). This dependence shows the correlation between the velocity constant of iron electrode dissolution reaction and temperature. The equation, which expresses that dependence is Arhenius equation:

$$\ln i = \ln A - \frac{E_a}{RT} \quad (1)$$

$$i = A \exp\left(-\frac{E_a}{RT}\right) \quad (2)$$

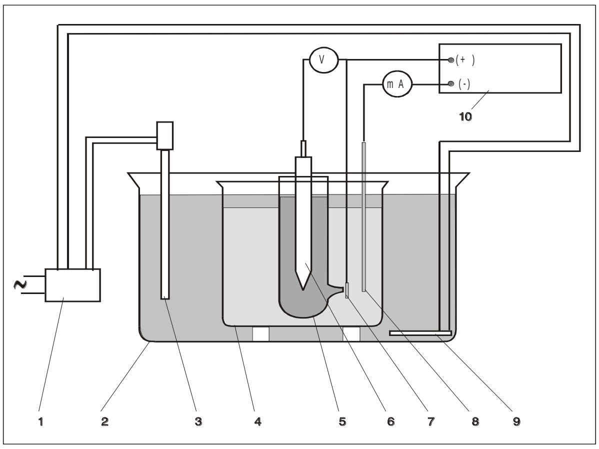

Figure 1. Draft of equipment for obtaining of polarization curves for anodic dissolution of a steel electrode.
1- Mercury relay; 2-tank; 3-contact thermometer ; 4- electrolytic cell; 5- test-tube with Lugin capillary; 6- reference electrode (calomel); 7-working electrode (steel); 8 - counter-electrode (Pt- electrode); 9-electrical heater; 10- stabilized rectifier.

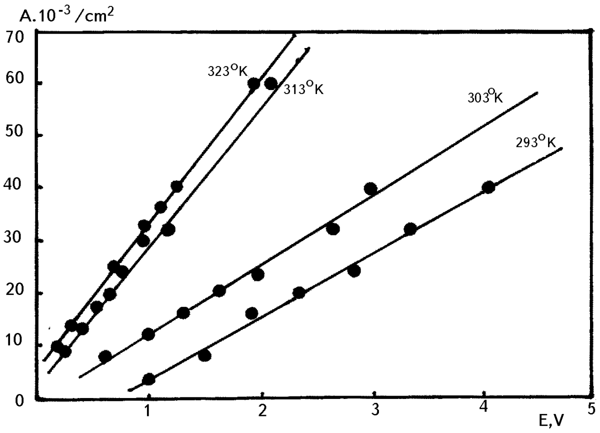

Figure 2. Polarization curve for anodic dissolution of a steel electrode in a sulphate model solution, at 20, 30, 40 and 50^{0}C.

The total activation energy was measured from the graphic dependency $\ln i/T^{-1}$, ^0K (Fig.3), taking into account that:

$$tg\alpha = {E_a}/{R}, \quad E_a = tg\alpha . R \quad (3)$$

where, E_a is the activation energy for anodic dissolution of steel; *R* is the universal gas constant 8,314 J/mol.gr; *tgα* is a tangent of the angle between the straight line and the ordinate.

The total activation energy was determined to be 41,6 kJ/mol.gr.

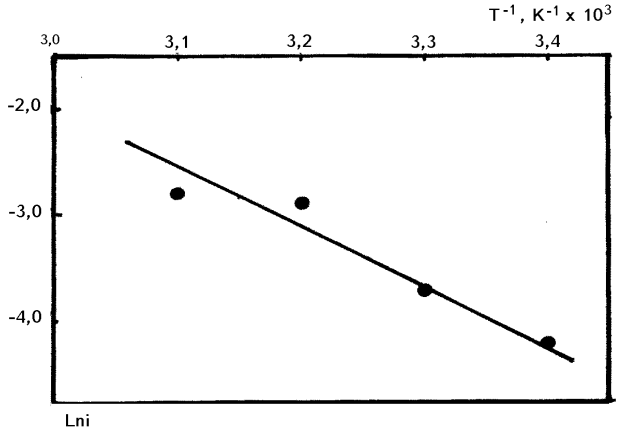

Figure 3. Dependence between velocity constant of the anodic dissolution reaction of a steel electrode and T^{-1}.

Electrochemical purification of waste water containing heavy metal ions.

Methods of investigation: For electroflotocoagulation treatment of the water, an equipment was used, consisting of: mechanical flotation machine (cell volume 500 cm^3, rotor revolution 1500 rpm, and aeration 50 l/h), voltmeter, ampermeter and selenium rectifier. Disc shape round electrodes were placed on a distance of 1 cm from the bottom of the cell. A dense anode having 50,24 cm^2 surface was used, while the cathode was perforated (3 mm hole diameter), it had 38,46 cm^2 surface, and was placed on a distance of 0,5 cm above the anode. A stainless steel was chosen as a material for the cathode, while for the anode a low-carbon steel of the type 0.8 KPN was used. It had the following composition: Mn 0.25-0.5%, C 0.05-0.11%, Si до 0.03%, Cr 0.1%, P 0.04%, S 0.04% и Ni 0.25%. The difference in electrode potentials was attained by using selenium rectifier.

A model solution of $CuSO_4.5H_2O$, $FeSO_4.7H_2O$ and $ZnSO_4.7H_2O$ was used for conducting the experiments. It had concentration of heavy metals close to that of a real mine water: Cu 14.91 mg/l, Fe 23.47 mg/l, Zn 16.39 mg/l and pH 3.0. Waste mine water from “Asarel”, (Table.1) was also used in the experiments. The concentration of the analyzed elements was determined by the method of inductively coupled plasma.

Waste water purification was conducted in different electroflotocoagulation conditions, such as: without additional stirring and aeration; with stirring at 1500 rpm and without aeration; with stirring, aeration of 50l/h and addition of flocculent in the treated solutions. “Praestrol” 2531 was used, which represents polyacrilamide based organic flocculent. It is a medium-anionic, with effective action at 6 – 13 pH range.

The residual concentration of the analyzed elements is a measure for the degree of purification of the waters.

Table 1. Chemical composition of mine water (pH 3.0)

Qualitative composition	Quantitative composition, mg/l	Qualitative composition	Quantitative composition Mg/l
Pb	0.06	Sn	0.05
Cu	10.00	Sb	0.05
Ni	0.026	Na	25.2
Fe	19.19	K	4.3
Mn	3.8	Mg	61.2
As	0.024	Ca	104
Cd	0.004	SO_4^{2-}	570
Co	0.054	NO_3^-	5.2
Mo	0.004	NO_2^-	0.02

Electroflotocoagulation of model solutions

A study on the influence of the current density and experimental conditions: $Fe(OH)_2$ was formed during the anodic dissolution of the iron electrode. In increasing pH values, $Fe(OH)_2$ vigorously absorbed oxygen (which is dissolved in the water, or artificially infused into the solution) and rapidly transformed in yellow-orange-red-brown colored $Fe(OH)_3$. Intermediate oxidation products were produced at pH 4-5, having blue-green to black coloration. These hydroxides contain simultaneously two- and three- valent iron. The hydroxides produced in this way, possess basic properties and they are practically insoluble in water.

Current density (i, A/cm^2) affected the amount of the iron hydroxides sediments (Fig.4). In increasing current densities values, their quantity increased, and solution pH from 3.0, reached 6.65-7.20 at i= 0.04-0.05 A/cm^2, by the end of the process (Fig.5). It can be seen from the graphs that experimental conditions (stirring, aeration), at a same current density, also affect the amount of the sediments and solution pH. The largest sediment amount and neutral pH was produced in electrocoagulation carried out with stirring and aeration of the solution.

In increasing the current density, residual concentrations of Cu, Zn, and Fe in model solutions decreased (Table 2). At the same current densities, residual concentrations dropped down if stirring and aeration was applied on the solution in the cell. The concentration limit regulations for Cu and Zn ions in waste water were reached at 0.04 A/cm^2, however the residual concentration of Fe ions was still higher. This can be attributed to the fact that micron size fine-dispersion sediments of $Fe(OH)_2$/$Fe(OH)_3$ were presented in the filtrated solutions after the termination of the process.

A study on the influence of electrochemical treatment duration and flocculent addition on the process: Based on the results obtained at the best electroflotocoagulation conditions (I=0.04 A/cm2, stirring and aeration of the solution), the following experiments were made at different duration of the treatment (Table 3). The residual concentrations of the analyzed elements were compared after electrofloto-coagulation, without and with, addition of “Praestol 2531” floculant.

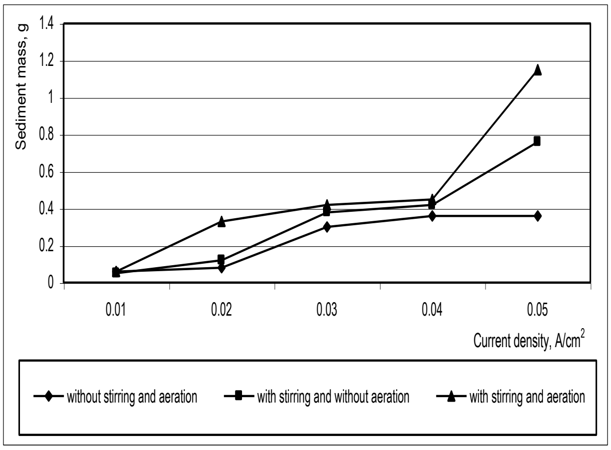

Figure 4. Influence of the current density on the formation of hydroxide sediments.

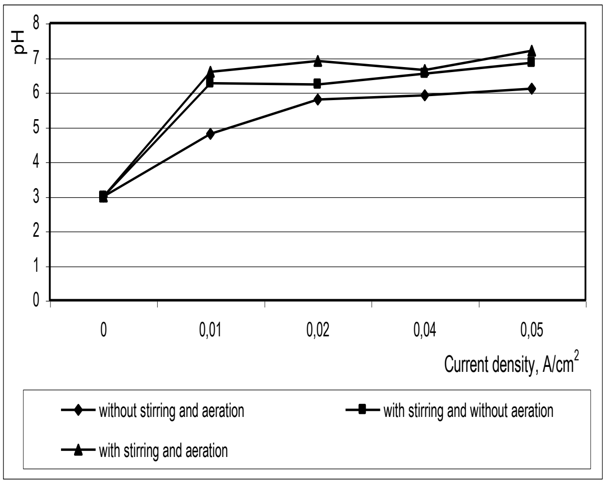

Figure 5. Influence of the current density on solution pH.

The residual concentration of the analyzed elements at specified current density decreased when treatment duration was increased. Electrochemical purification without floculant couldn’t achieve regulation limits for iron ions, even in larger duration of the treatment (25 min). Feeding with flocculent after the electrochemical process was terminated, induced a completion effect on the purification, which amplified in increasing the duration of the treatment, i.e. increasing the amount of produced hydroxide sediments. According to Table 5 data, the duration of the treatment drops to 15 min when flocculent was added to achieve regulation limits for surface water. The flock, which formed when the flocculent was added, aggregated in large clusters and was not destroyed from a turbulence.

Table 2. Influence of the current density and experimental conditions upon electrochemical purification of Cu, Zn and Fe solutions

Current density,	Residual concentration, mg/l			Extraction %		
A/cm²	Cu	Zn	Fe	Cu	Zn	Fe
0	14,91	16,39	23,47	-	-	-
Without aeration and stirring						
0,01	3,51	4,40	19,30	76,5	73,2	17,8
0,02	2,29	2,68	16,50	84,6	83,6	29,7
0,04	1,14	1,49	14,90	92,4	90,9	36,5
Without aeration, with stirring						
0,01	3,00	2,14	17,95	79,9	86,9	23,5
0,02	1,83	1,47	15,40	87,7	91,0	34,4
0,04	1,03	1,38	12,90	93,1	91,6	45,0
With aeration, with stirring						
0,01	1,14	1,44	12,6	92,4	91,2	46,3
0,02	0,87	1,27	11,12	94,2	92,3	52,6
0,04	0,45	1,15	9,72	97,0	93,0	58,6
Reg. limit conc	0,5	2,0	3,5	-	-	-

Таблица 3. Influence of the duration of the treatment and flocculent addition upon electrochemical purification of Cu, Zn and Fe solutions.

Duration of treatment	Residual concentration, mg/l					
Min	Without flocculent			With flocculent 150 g/t		
	Cu	Zn	Fe	Cu	Zn	Fe
0	14,91	16,39	23,47	14,91	16,39	23,47
5	9,76	12,60	18,6	3,50	7,80	12,50
10	3,22	5,15	14,7	1,69	3,75	8,35
15	0,45	1,15	9,72	0,10	0,80	3,21
20	0,37	0,76	7,90	0,10	0,34	2,28
25	0,15	0,20	6,63	0,08	0,17	0,61

Table 4. Electrochemical purification of waste water from “Asarel” mine.

Duration of treatment, min	Residual concentration, mg/l			Extraction, %		
	Cu	Mn	Fe	Cu	Mn	Fe
0	10,00	3,8	19,19	-	-	-
Without flocculent						
10	0,54	2,5	16,60	94,6	34,2	13,5
20	0,23	1,35	10,80	97,7	64,5	43,7
30	0,13	1,07	8,80	98,7	71,8	54,1
50 g/t flocculent						
10	0,41	1,75	8,90	95,9	53,9	53,6
20	0,19	1,5	5,01	98,1	60,5	73,9
30	0,07	1,0	2,38	99,3	73,7	87,6
100 g/t flocculent						
10	0,21	1,7	6,00	97,9	55,3	68,7
20	0,12	1,0	3,20	98,8	73,7	83,3
30	0,07	0,8	1,64	99,3	78,9	91,5
150 g/t flocculent						
10	0,13	1,5	5,2	98,7	60,5	72,9
20	0,10	0,8	1,10	99,0	73,2	94,3
30	0,06	0,7	0,50	99,4	81,6	97,4
Concentration limits	0,5	-	3,5	-	-	-

Ellectrolfotocoagulation of mine waste water from “Asarel” mine.

Work of mine water purifying station (MWPS) in “Asarel –Medet” AD:, Water from the eastern waste-storage and drainage gallery of “Asarel” mine is being purified to characteristics meeting concentration limit regulations of 2nd category. The content of main cation water pollutants varies, on average, in the following ranges: Cu from 6.96 to 24.6 mg/l; Fe from 3.54 to 20.11 mg/l; Mn from 3.24 to 6.12 mg/l; Ca from 38.80 to 100.4 mg/l; Mg from 23.0 to 38.0 mg/l. In smaller amounts there are also found: Zn from 0.14 to 1.0 mg/l; Co from 0.09 to 0.04 mg/l; Cd from 0.04 to 0.11 mg/l, Ni from 0.03 to 0.09 mg/l and Cr from 0.04 to 0.1 mg/l. The technological process of mine water purification is based on neutralization with $Ca(OH)_2$, in which, water pH exceedes 10, and insoluble substances (metal hydroxides, hydroxicarbonates, and other colloid sediments) are formed. Anionic flocculent Magnaflock E24 is infused, to ensure rapid sedimentation, while for adjustment of water pH to 6.5-7.5, a H_2SO_4 is infused. Disadvantages of this technological scheme are: large amounts of $Ca(OH)_2$, infused during the neutralization, lead to higher Ca^{2+} amount in the water; H_2SO_4 adjustment of water pH results in formation of unconditioned sediments and $CaSO_4$ (gypsum) i.e. there is a formation of redundant sediments.

Study of electrochemical purification of mine waste water from "Asarel" mine: Based on the results from electrochemical purification of model solutions, experiments were made with mine waters from "Asarel" mine, under the following experimental conditions: current density 0,04 A/cm^2; 1500 rpm of flotation machine impeller; aeration of 50 l/h; duration of treatment 10, 20 and 30 min; 50, 100 and 150 g/t infusion flow of "Praestrol" in the treated water. Cu, Fe and Mn ions are the major water pollutants, while the rest of the heavy metal ions are within the regulations for drinking water and they are not traced. After water treatment a comparison was made between residual concentrations of the analyzed elements, without and with flocculent addition (Table 4). Data show that at the specified current density, it was achieved a purification of Cu ions up to the regulations for surface flowing water, while for Fe and Mn this was not achieved. The higher Fe content is probably due to the produced colloid-size iron hydroxides. In increasing the amount of hydroxide sediments, flocculent infusion in treated water produced a completion effect upon the cleaning, which amplified in increasing the flow. The regulations for surface flowing water were reached in 15 to 20 minute electrochemical treatment and 100 to 150 g/t flocculent infusion in treated water. Water pH at the end of the process were 6.2-6.5, while residual concentrations of analyzed elements, Cu, Fe and Mn, were 0.1, 1.1 and 0.7 mg/l correspondingly.

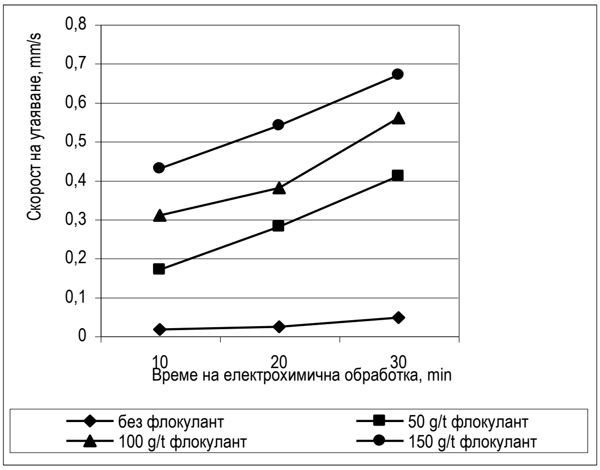

Figure 6. Velocity of precipitation of hydroxide sediments without and with sediments

The velocity of sedimentation of the sediments produced after water treatment, without and with flocculent infusion are shown on Fig.6. Sediments without flocculent infusion precipitated very slowly, with a velocity from 0-016 to 0-047 cm/s, depending on the amount of the produced sediments. The flock produced was sleazy and the sedimentation layer was large. Infusion of flocculent substantially increased the velocity of sedimentation, and at 150 g/t it was from 0.43 to o.67cm/s, the flock produced was much denser, and the sedimentation layer much smaller.

CONCLUSION

Total activation energy for electrochemical dissolution of iron anode in solution with 400 mg/l sulphate ion content and pH 3 was determined. Experimentally obtained value of 41.6 kJ/mol.gr looks quite feasible.

The results obtained in this study suggest the opportunity for application of technology, which includes electrochemical treatment of waste water from "Asarel" mine, with subsequent addition of "Praestol 2531" flocculent. The residual concentrations are below the emission regulations for waste water from yielding and mineral processing of ores.

Process parameters, such as current density, treatment duration, stirring, aeration and water pH, have an essential influence upon electroflotocoagulation purification of Cu, Fe and Mn ions. In this specific case, process parameters were: 0.04 A/cm^2 current density, 10-15 min duration of the treatment, 1500 rpm stirring, 50 l/h aeration, and 150 g/t flocculent infusion in the treated solution. Residual concentrations of Cu, Fe, and Mn, in these experimental conditions, were below the regulations for surface flowing water.

An advantage of the suggested technology in regard to currently existing one, is that it will eliminate the requirement of lime-wash infusion leading to a formation of large amounts of metal hydroxides, hydroxicarbonates and other colloid sediments. This causes a substantial increase of Ca ions concentration, and during the infusion of sulphuric acid for conditioning of water pH, additional large amounts of gypsum sediments are formed.

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Recommended for publication by Department of Mineral processing, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.221-223

APPLICATION OF A DIFFERENTIAL DOUBLE PULSE POLAROGRAPHY FOR ARSENIC DETERMINATION IN WASTE WATERS FROM COPPER PRODUCING PLANTS

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ABSTRACT
A method for direct As(III) determination in technological and waste waters from copper producing plants has been developed, using differential double pulse polarography. Its high resolution allows avoiding the preliminary samples' processing - masking and elimination from the matrix of the interfering components. In the same time, the two essential problems, related to the arsenic determination - the lowering of the detection limit and the distinction between its two forms have been solved.

INTRODUCTION

Arsenic, chemically active, toxic and carcinogenic element [Moore, et al., 1984], is one of the important environmental pollutants. It enters the air, the ground and the water mainly as a result of the activity of the ore-extracting and mineral-processing plants [Moore, et al., 1987], in form of As(III) and As(V). Because of the high toxicity of the arsenic and especially of As(III), its admissible concentration limit is low - 20 μg/L for first category waters, 50 μg/L for second category waters and 200 μg/L for third category waters [Bulgarian Official Journal. No 96, 1986].

Arsenic concentration in environmental samples is determined, before all, using atomic absorption spectrometry (AAS), inductively coupled plasma (ICP) and neutron activation analysis (NNA) [Stoytcheva, et al., 1998]. These instrumental methods are sensitive and allow the determination of arsenic in concentrations from the order of the admissible concentration limit. However, the results they provide express the total arsenic content, without distinction between its two forms As(III) and As(V), with different toxicity, biological activity and physiological action. It is known that the toxicity of As(III) compounds is more important that this of As(V) compounds, fact of primordial importance for the ecological expertise of the pollution with arsenic. An other disadvantage of the enumerated instrumental methods is the high price of the equipment and by consequent - of the analyses.

In order to determine the two forms of arsenic, As(III) and As(V), they have to be separated by preliminary chemical or physical - chemical processing of the samples, in general by extraction, by sorption or by chromatography [Tam, 1974; Kamada, 1976; Chan-Huan Chung, et al., 1984; Mok, et al., 1986; Slovák, et al., 1977; Mu-Ging Yu, et al., 1983; Howard, et al., 1989; Russeva, et al.,1989]. However, this approach renders complex the determination, prolongs the analysis and introduces errors in the results.

The voltammetric methods - before all differential pulse anodic stripping voltammetry (DPASV), and differential pulse polarography (DPP)) [Forsberg, et al., 1975; Davis, P., et al., 1978; Myers, et al., 1973; Henry, et al., 1979; Reed, et al., 1987] constitute an alternative of the mentioned methods of analysis of arsenic containing samples. These electrochemical methods allow the direct determination of the electroactive As(III) and its distinction from As(V) that has not an electrochemical activity. The interference of Pb(II), Tl(I), Tl(III), Sn(II) и Sn(IV), whose peaks overlap that of arsenic imposes their elimination by preliminary processing of the sample. For this purpose cation-exchange resins are used. They retain the mentioned cations, but not the arsenic, that is in anionic form (AsO_3^{3-}) [Reed, et al., 1987].

The polarographic methods of second order as radio frequency polarography (RFP) [Barker, 1958; Barker, et al., 1973], differential polarography with Faradaic rectification (DFRP) [Saur, 1979], differential double pulse polarography (DDPP) [Zlatev, 1984] or second harmonic AC polarography (SHACP) allow to solve in a simple manner the problem of the interference of the mentioned cations. The recorded polarogrammes for each component possess the form of a second derivative of polarographic wave with an insignificant half-width of the peak. When determining components with overlapping peaks, the first one is identified by the "positive" peak and the second one - by the "negative" peak remaining on the polarographic curve after their recovery. The disadvantages of the polarographic methods of second order consist in the complex and expensive equipment (RFP и DFRP), or in their insufficient sensitivity (SHACP), or in the problems generated by the applied high frequency.

Among the cited methods of second order solely the method DDPP combines high sensitivity and simple equipment, while its resolution is more important than that of others methods of second order [Zlatev, 1984]. For this reason DDPP has been applied for the direct determination of As(III) in waters coming from ore-extracting and mineral-processing plants.

EXPERIMENTAL

Apparatus and reagents

The determinations have been accomplished with a computerized polarograph Z52 (Zenith Laboratory, Bulgaria), conjugated with an electrolytic cell of conventional type with a working electrode - static mercury drop electrode (SMDE), a counter electrode from Pt, and a reference electrode - saturated calomel electrode.

The standard solutions of As(III) and Pb(II) with concentrations of 1000 mg/L have been purchased from Merck. The solutions with a lower concentration, necessaries to construct calibration curves or for the determination of the interference have been prepared directly in the electrochemical cell using automatic pipettes of small volume.

The solution was purged with Ar during 5 minutes in order to eliminate the dissolved oxygen.

HCl – 1 mol/L served as supporting electrolyte.
The experiences were carried out at ambient temperature.

RESULTS AND DISCUSSION

The differential double pulse polarographic method, as well as the other polarographic methods of second order, is based on the non-linearity of the voltammetric curve. As a result, when superposing pulses with different polarity on the potential of the working electrode, pulse faraidaic current cmponents with opposite polarity and with different value amplitude appear. The sum of each couple of pulses with opposite polarity for the corresponding value of the electrode potential, presented graphically as a function of the potential, possesses the form of second derivative of polarographic wave (figure 1).

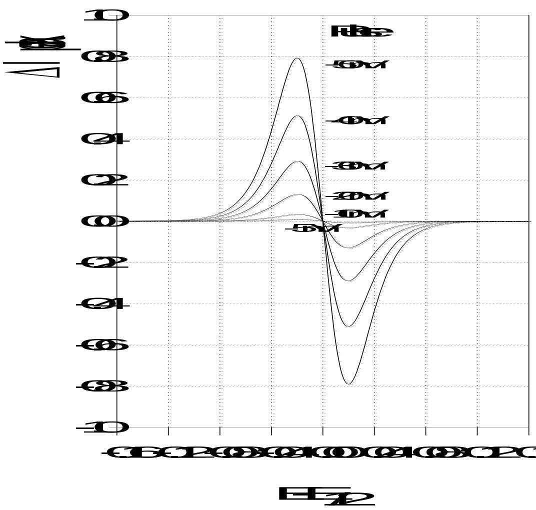

Figure 1. Theoretical DDPP polarographic curves as a function of the pulse amplitude.

The peak current is proportional to the concentration, and the potential of the peak corresponds to the nature of the determined component. The half-width of the peak for DDPP is only 60% from the half-width of the peak obtained by the most widespread conventional polarographic method differential pulse polarography (DPP) [Zlatev, 1984]. Namely that small peak's half-width allows the simultaneous determination of components with very close peak's potentials, as in the case of As(III) et Pb(II).

For overlapping peaks, As(III) concentration could be determined measuring the hight of the negative peak, and Pb(II) concentration - by the high of the positive peak, remaining on the polarographic curve after the recovery. For a comparison, in figure 2 is presented the polarographic curve (the upper curve), recorded using the conventional method differential pulse polarography (DPP).

As(III) produces two peaks in HCl and H_2SO_4 medium. The first one corresponds to the three electronic reduction of As(III) to As(0). The second peak, expressed better, corresponds to the posterior reduction to AsH_3. For concentrations higher than 300 μg/L, between the two peaks appears a third very narrow peak, that is a polarographic maximum. It could be eliminated by using suitable surfactants. In figure 2 the second peak of As(III) is presented - the first is recorded at more positive potentials.

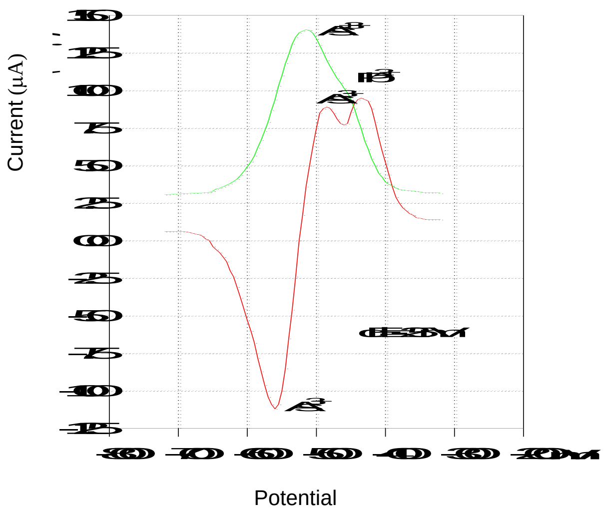

Figure 2. Polarographic curve of a solution, containing As(III) - 20 μg/L and Pb(II) -16 μg/

It has been established that the hight of the second reduction peak is a linear function of the concentration of As(III) in the analysed sample in the concentration range 10 μg/L – 60 mg/L. The experimental data have been used for constructing the calibration curve in the following experimental conditions:

- potential = 250 mV
- scan rate = 5 mV/s
- pulse amplitude = 50 mV.

The standard curve is presented in figure 3.

It has been established that the detection limit for As(III) determination by the method DDPP is 0.3 μg/L (confidence interval 99 %). This value coincides with the detection limit of the determination reached with the method differential pulse polarography (DPP) [Myers, 1973].

The relative standard deviation characterising the reproducibility was inferior to 3 %, in the upper limit of the linear concentration.

The error of As(III) determination in presence of Pb(II) for a concentration ratio of the components [As(III)]/[Pb(II)] varying from 100:1 to 1:10 has been determined. In the less favourable case - concentration ratio [As(III)]/[Pb(II)] = 1 :10, the relative standard deviation did not surpass 6.5 %.

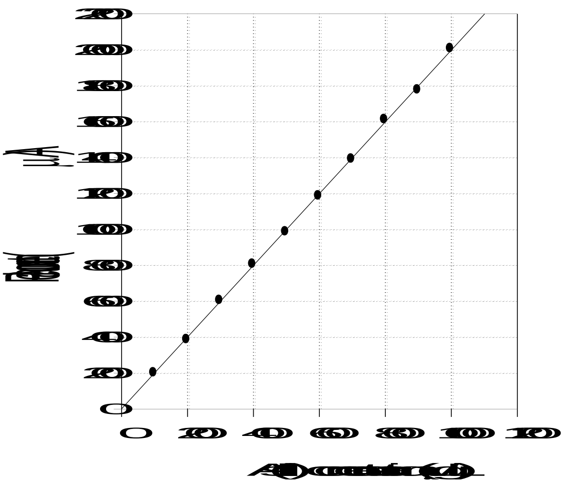

Figure 3. Standard curve – peak current as a function of As(III) concentration.

The developed method for As(III) determination on the basis of DDPP has been applied for the direct analysis of technological and waste waters from Pirdop copper producing plant. The results have been compared to those, obtained by others laboratories – the Central Research Laboratory at the University of Chemical Technology and Metallurgy, and the Laboratory of Sofia Regional Ecological Service (Table 1).

Table 1. As(III) content in industrial waters, mg/L

№	Method			Error, %
	DDPP	AAS	Spectrophotometry	
	1340	1360	-	1.47
	1018	1030	-	1.16
	990	1000	-	1.00
	810	830	-	2.41
	712	730	-	2.46
	1.42	-	1.46	2.07

The results presented in table 1 obtained by applying different methods in independent laboratories match very well.

CONCLUSION

A new polarographic method for arsenic determination is proposed. It combines the high sensitivity of the differential pulse polarography with the great resolution of the polarographic methods of second order and allows the direct determination of arsenic in low concentrations, as well as the distinction between As(III) and As(V).

The method found an application for As(III) determination in industrial waters.

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Recommended for publication by Department of Chemistry, Faculty of Mining Technology

50 Years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.225-227

CHARACTERIZATION OF CHEMICAL, MINERAL AND RHEOLOGICAL PROPERTIES OF OIL-POLLUTED DRILLING WASTES AND THEIR DETOXICATION*

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ABSTRACT
Chemical, mineralogical (XRD, FTIR, SEM) and rheological investigations of oil-polluted, spent drilling muds were carried out. The muds represent polydispersive, colloidal, mineral-organic systems with tixotropic properties. The analysed changes of microstructure of these slurries during successive stages of remediation involved their transition from highly dispersed, delaminated organic-mineral systems to the better ordered structures. The authors also present the results of catalytic and biochemical degradation of hydrocarbons contained in the spent drilling muds.

INTRODUCTION

Contamination of drilling waste as well as soil-ground environment in the course of prospecting and exploitation with some toxic organic compounds, especially oil-based hydrocarbons, is a serious problem in environmental researches and technologies.

Microbiological methods and methods based on extraction processes generally cannot guarantee high efficiency of removal of organic pollutions. The efficiency of these methods considerably drops down with the increment of quantity of fine-dispersive colloidal particles in the cleaned environment.

It is difficult to detoxicate drilling mud wastes with traditional methods, especially those which have low permeability, reductivity, very high content of colloidal dispersion particles and are hardly accessible to oxygen.

The subject of the researches was the drilling waste gathered in drilling pits in the South Carpathian Voivodeship, south of Poland. During their deposition, zones of these sediments were polluted with oil-based hydrocarbons. The objective of the research was to recognize the intensity of pollution of these wastes and to determine the efficiency of their detoxication with the use of a method worked out by the authors Fijał, *et al.* (2002).

RESEARCH METHODS

For all types of wastes gathered in drilling pits, full or index geochemical analyses were made, e.g. Zn, Cr, Pb, Fe, CO_2, phenols, H_2S (aliphatic and aromatic hydrocarbons including) for bottom sediments and muds from surface layers.

Dry residue left out after drying off water and volatiles from mud samples underwent mineralogical-phase analyses. XRD (employing a diffractometer by Philips), FTIR (Bio-Rad spectrometer, 165 type) and SEM were used.

Rheological analyses of the spent drilling muds were made with the use of a rotation viscometer Chan 35 API. When selecting the rheological model of muds, the Flow-Fluid Coef computer program was employed.

RESULTS

Mineral-phase analysis

Waste samples from drilling pits for phase analyses should have the form of dense muddy dispersions. They should undergo mild drying, in the process of which water and volatile organic matter, hydrocarbons including, will be removed. The analysed dusts were made of:

- rock cuttings representing the whole drilled profile in the form of Cretaceous and Tertiary clays and sands, Eocene spotted shales with shale intercalations, Cretaceous inoceramic material, shales and sandstone intercalated with shales, Badenian shales with sandstone horizons;
- spent bentonite, potassium-polymer and clay-free drilling muds based on bentinite and carbonacous mineras, as well as organic matter: starch- and cellulose-derivatives, synthetic polyacryloamide polymers, potassium salts and sodium base.

Mineralogical-phase analyses were mainly based on XRD.

X-ray analyses of mud wastes were made before and after their pre-oxidation, Figure 1.

* Support of Scientific Research Committee (KBN) is fully acknowledged, research no. 11.11.190.408 & 11.11.190.01

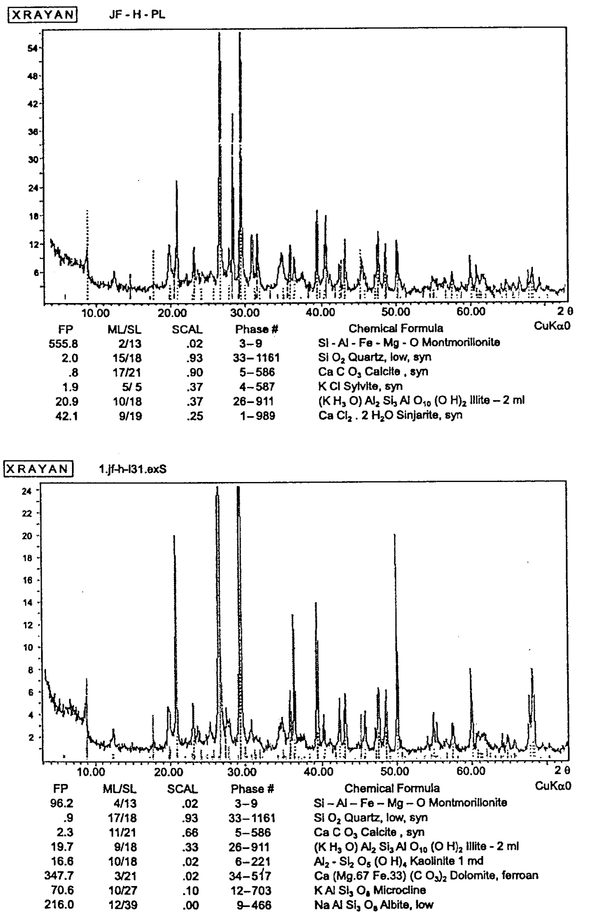

Figure 1. X-ray diffractograms of drilling waste sample before (a) and after (b) its pre-oxidation

The X-ray diffractogram of a raw waste (unoxidated) evidences a high level of delamination of minerals having a swelling structure (montmorillonite and mixed-package smectite-illite) based on a fuzzy diffraction band in a range of 4 to 10° 2°Θ. It corresponds to the presence of organic-mineral complexes, where small packages assume a random orientation. Oxidation of reduction substances cause that a diffraction package is well visible on the diffractogram with a maximum of about 12.9Å, which corresponds to the increasing orientation of packages of a surface-surface type. Therefore, we have a proof for a partial disconnection between organic matter present in drilling waste system and the surface of clayey minerals. This was attributed to oxidation of reduction organic matter, where aldehydes are oxidated to form carboxyl, and where the negative charge organic ions increase. This is also related with the increase of repulsive electrostatic forces between negative charge organic ions and clayey packages. Liberated from organic polymers, the packages may re-group, forming their own domain aggregates, resulting in the increased intensity of a peak of 12.9Å (smektite).

The increase of orientation of packages in the oxidated mud samples has also been confirmed by SEM analyses. Partial oxidation of organic matter encapsulated in muds was documented with the use of FTIR.

Chemical characteristic of wastes

Chemical analyses were carried out for waste samples. Index chemical analyses were made for heavy metals content (Zn, Cr, Pb), as well as phenols, H_2S, reduction substances and hydrocarbon content.

It follows from these analyses that content of reduced organic matter, formed in the course of oxygen-free fermentation in the sediments, is considerable (Table 1); this can additionally be evidenced by the presence of H_2S (0.024 to 0.098%). Some of the analysed samples had an increased Zn content (to 410 ppm). The analysed samples showed a considerable share of mineral matter making up the cuttings; H_2O content in the samples ranged from 32 to 43% wt.

Among the components environmentally hazardous, an increased reduction substance content was observed. It can be measured by 0.125 M of iodine solution, expressed in cm^3, which reacts with 1 kg of the analysed sample. The reductivity of individual parts of sediments varies. Reduction compounds are formed by transformation of organic matter of muds in the condition of a limited access to oxygen. The reduction matter content correlates with H_2S content (expressed by H_2S content). The presence of this form of sulphur should be associated with reduction of sulfate sulphur (SO_4^{2-}).

It follows from the analysis of hydrocarbons making up the samples from the drilling pits (Table 2) that substances are distributed non-homogeneously. Their total concentrations vastly change from about 600 to almost 15,000 mg/kg sediment.

Rheological analyses were made for spent drilling muds, as they dominate in the drilling wastes composition. To show the variability of rheological properties, muds of bentonite, potassium-polymer, clay-free and potassium-chlorine types were examined. Such muds are used in the area where oil and gas wells are drilled in the south of Poland.

Spent muds are multiphase, polydispersive systems. Mineral-organic associates cause that muds are disordered in structure, including the beginning of intercalation structures. The best rheological description can be made with the use of the Herschel-Bulkley and Casson models.

Detoxication and management of oil-polluted waste

Detoxication procedures were applied to the most polluted zones of sediments gathered in the drilling pits, about 300 m^3 of volume. These wastes were moved in the vicinity of a side of the container and stirred to make the hydrocarbon concentrations even, obtaining average concentration of 6268 mg/kg. This detoxication procedure was patented, Fijał *et al.* (2002).

This procedure lied in transformation of sediment into a multilayer detoxication structure, where catalysts were introduced to enable the process of gradual degradation of hydrocarbons. The initial analysis of these catalytic and biochemical reactions transforms hydrocarbons into partly polar substances, which in turn, link with doted activators to enhance migration of produced complexes to the surface of decontamination structure. Such a surface was a basis on which the processes continued at the increasing share of biodegradation processes.

Table 1. Content of selected chemical components in mud and sediments samples

No.	Zn ppm	Cr ppm	Pb ppm	Fe %	CO_2 %	Phenol Ppm	H_2S %	Red.subst. V/kg dry mass	Humidity %
1	302	25	31	2.5	2.9	0.12	0.024	915	43.07
2	160	27	23	2.45	3.15	0.07	<0.005	255	36.51
3	157	39	27	2.6	3.15	0.27	<0.005	155	34.85
4	410	39	42	3.3	3.2	0.08	0.067	1540	42.77
5	323	44	23	2.5	1.95	0.38	0.098	1920	36.12
6	88	27	19	1.8	1.55	0.14	0.078	1630	32.40

V/kg dry matter – volume of 0.125 iodine solution (cm^3) reacting with 1 kg of analysed sample

Table 2. Oil pollutions in samples from muds and sediments in Container I

No. of samples	Hydrocarbon content (mg/kg wet waste			Dry mass content % wt.
	Total aliphatic and aromatic hydrocarbons content	Sum of aliphatic hydrocarbons	Sum of aromatic hydrocarbons	
1	14,414.6	14,251.7	162.9	36.95
2	3028.9	3009.6	19.3	59.49
3	12,403.2	12,395.6	7.6	32.87
4	894.5	888.9	5.6	64.40
5	1107.7	1042.8	64.9	37.72
6	1928.5	1909.7	18.8	63.98
7	1960.3	1912.1	48.2	63.40
8	629.5	615.7	13.8	63.98
9	5895.2	5863.1	32.1	56.06
10	14,874.2	14,822.2	52.0	61.84

The efficiency of this method was documented by an analysis of changes of hydrocarbon concentration in the decontamination layer in the function of time of reaction. After 3 months, a radical drop of hydrocarbon concentration was observed (below 70 mg/kg).

CONCLUSIONS

The following conclusions can be drawn from the analyses:
1. It is possible to efficiently detoxicate oil-polluted drilling waste, in the course of which concentrations of these compounds can be safely managed.
2. Further research on remediation of areas subjected to detoxication is planned.

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Recommended for publication by Department of Engeneering Geoecology, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.229-231

TREATMENT OF WATERS FROM A COPPER MINE BY MEANS OF A PERMEABLE REACTIVE BARRIER

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ABSTRACT
Acid mine waters contaminated with heavy metals (Cu, Cd, Fe), arsenic and sulphates were treated by means of a permeable reactive barrier. The barrier was constructed in a site with a high rock permeability. It represented a ditch located perpendicularly to the direction of the water flow and filled with a mixture of biodegradable solid organic substrates (leaf and spent mushroom compost, cow manure and sawdust). The barrier was inhabited by a mixed microbial community consisting of sulphate-reducing bacteria and different metabolically interdependent microorganisms. An efficient removal of the pollutants was achieved within the 18 - month experimental period during which portions of the partially exhausted solid organic substrates were periodically replaced by fresh batches of such substrates. The microbial dissimilatory sulphate reduction and the sorption of pollutants by the organic matter in the barrier were the main processes involved in this removal.

Key words: microbial sulphate reduction; sulphate-reducing bacteria; heavy metals; arsenic.

INTRODUCTION

Acid drainage waters are a persistent environmental problem at many active and abandoned mine sites. This phenomenon is connected with the oxidation of pyrite and other suilphide minerals as a result of which acidic waters containing sulphuric acid, dissolved heavy metals and solid iron precipitates are released to the environmental. Toxic and radioactive elements such as arsenic and uranium are also solubilized from minerals containing these elements.

Several methods for treatment of acid mine waters exist, depending upon the volume of the effluents, the type and concentration of contaminants present. The most largely used method is connected with the chemical neutralization of the waters followed by the precipitation of metals. Such active treatment requires the installation of a plant with agitated reactors, precipitators, clarifiers and thickeners with high costs for reagents, operation, maintenance and disposal of the resulting metals laden sludge. The only alternative of such high-cost schemes are the passive treatment systems (Cambridge, 1995; Gusek, 1995; Groudev et al., 2000). These systems have been developed on the basis of naturally occurring biological and geochemical processes in order to improve the quality of the influent waters with minimal operation and maintenance costs. The main advantage of these systems over chemical neutralization is that large volumes of sludge are not generated, the contaminants being precipitated mainly as sulphides.

A wide range of passive treatment systems is available currently. This paper contains some data about the treatment of acid drainage waters from a copper mine by means of an alkalinity-producing and sulphate-reducing permeable reactive barrier.

EXPERIMENTAL

The copper ore deposit Elshiza, Central Bulgaria, for a long period of time was a site of intensive mining activities. As a result of this , a significant portion of the mine waters is contaminated with heavy metals (copper, cadmium, iron), arsenic and sulphates. The pH of these waters is acidic (usually in the range of 2.8 – 4.5) which facilitates the transportation of the dissolved contaminants.

The contaminated waters from the main underground mine in the deposit were discharged into a ravine and mixed with surface waters. The bottom of this ravine consisted of a layer of clay and different sediments and was located on weathered rocks with a high permeability down to a depth of about 3.5 m. The filtration coefficient of these rocks was in the range of about $3 \times 10^{-3} – 8 \times 10^{-3}$ m/s and there were numerous leaks in the rock matrix. As a result of this, a considerable portion of the waters infiltrated and saturated the weathered rocks. These weathered rocks were underlined by an intrusive rock with a low permeability. The filtration coefficient of this intrusive rock was about 6×10^{-8} m/s. In this way, a flow of contaminated groundwaters was formed below the bottom of the ravine. These waters were only partially separated from the surface waters in the ravine.

The permeable reactive barrier was constructed in the ravine, within the rock mass located below the bottom of the surface waters, in a site with a high rock permeability. It represented a ditch located perpendicularly to the direction of the water flow. The ditch was 5 m long, 2.5 m wide and 3.2 m deep and its bottom was located on the impermeable intrusive rock. The side walls of the ditch were covered by a corrosion-resistant cement layer to prevent the seepage of non-treated waters into the ditch. The ditch was filled with a mixture consisting of leaf and spent mushroom compost, cow manure and sawdust and was inhabited by a mixed microbial community containing sulphate-reducing bacteria and other metabolically interdependent microorganisms. (Table 1).

The ditch was covered by plastic sheets on which a 30 cm clay layer was formed. This cover efficiently isolated the surface waters in the ravine from the waters entering into the ditch through the highly permeable weathered rock.

Several piezometers were installed around and within the barrier for sampling and monitoring the process of water treatment.

Table 1 Microflora composition of the permeable barrier.

Microorganisms	Cells/ml
Sulphate-reducing bacteria	$10^5 – 10^8$
Cellulose – degrading microorganisms	$10^4 – 10^7$
Bacteria fermenting sugars with gas production	$10^5 – 10^7$
Denitrifying bacteria	$10^2 – 10^5$
Methane-producing bacteria	$10^1 – 10^3$
Anaerobic heterotrophic bacteria related to other physiological groups	$10^2 – 10^5$

Data about the waters treated in this study are shown in Table 2. The flow rate varied in the range of approximately 3 – 10 m^3/24 h. It must be noted, however, that the larger mass of the barrier influents and effluents passed through the cracks and leaks in the weathered rock and no through the basic rock matrix. The quality of the barrier influents and effluents was monitored at least once per week in the period May 1997 – October 1998. The parameters measured in situ included pH, Eh, dissolved oxygen, total dissolved solids and temperature. Elemental analyses were done by atomic absorption spectrophotometry and induced coupled plasma spectrophotometry in the laboratory.

Sulphate, nitrate and ammonium concentrations were measured photometrically. Organic substrate utilization was estimated by measuring the chemical oxygen demand (COD) of representative solid samples from the reactive barrier at the beginning and conclusion of the experiment. Cellulose was analyzed using the sulphuric acid digestion method (Updegraff, 1969). The method involves extraction with an acetic/nitric acid reagent to remove lignin, hemicellulose, and xylosans, followed by digestion with 67% sulphuric acid and final determination using the anthrone reagent.

Table 2. Data about the drainage waters before and after their treatment by the permeable barrier.

Parameters	Before treatment	After treatment	Permissible levels for waters used in agriculture and industry
pH	2.8 – 4.5	7.1 – 7.5	6 - 9
Dissolved O_2, mg/l	0.2 – 0.6	0.1 – 0.2	2
Solids, mg/l	23 – 77	14 – 37	100
Oxidativity (by $KMnO_4$), mg/l	5.1 – 12	75 – 710	40
SO_4^{2-}, mg/l	824 – 1540	244 – 424	400
Cu, mg/l	3.81 – 14	<0.05	0.5
Cd, mg/l	0.15 – 1.2	<0.01	0.02
Fe, mg/l	145 – 325	<1.0	5
As, mg/l	0.37 – 2.8	<0.05	0.2

In October 1997 and in May 1998 portions of the partially exhausted solid organic substrates were replaced by fresh batches of such substrates.

The isolation, identification and enumeration of microorganisms were carried out by methods described elsewhere (Karavaiko et al., 1988; Groudeva et al., 1993).

RESULTS AND CONCLUSIONS

It was found that an efficient removal of pollutants from the waters being treated was achieved in the barrier (Table 2). Even at dilution rate as high as about 0.5 h^{-1}, the concentrations of pollutants were decreased below the relevant permissible levels for waters intended for use in agriculture and industry. This was due to different biological, chemical and physico-chemical processes but the main role was played by the microbial dissimilatory sulphate reduction. This conclusion was made on the basis of the data about the generation of hydrogen sulphide, the significant decrease of the concentration of sulphate ions and of the levels of redox potentials (Eh) as well as about the increase of the number of sulphate-reducing bacteria, the level of the pH and the content of insoluble sulphides of copper, cadmium, iron and arsenic in the barrier. The above-mentioned heavy metals and arsenic were precipitated mainly as very fine particles of the relevant sulphides. However, portions of these contaminants were precipitated as hydroxides and carbonates or were removed as a result of their sorption by the organic matter in the barrier.

The microbial community in the barrier was able to survive and to act efficiently even at the lowest pH value of the waters treated during this study, i.e. at pH 2.8. The alkalinity produced by the solubilization of the carbonates contained in the spent mushroom compost (it was characterized by a positive net neutralization potential of about 250 kg $CaCO_3$/t) as well as by the hydrocarbonate ions formed during the sulphate reduction gradually increased the pH and stabilized it around the neutral point.

The microbial sulphate reduction was a function of the digestibility of the organic substrates in the barrier. Different saprophytic microorganisms degraded the biopolymeric organic compounds and provided the sulphate-reducing bacteria with suitable monomeric organic sources of carbon and energy. The organic substrates used in this study were slowly degradable and supported microbial growth over the long experimental period. The concentration of dissolved organic compounds was high during the first 3-4 months after the start of the experiment (values of the permanganate oxidativity in the range of about 600 – 700 mg/l were measured during this period). Then the permanganate oxidativity was decreased, initially to about 350 – 500 mg/l and later to about 80 – 150 mg/l due to the partial exhaustion of the available easily biodegradable solid organic substrates and to the low temperatures during the cold winter months (December 1997 – February 1998). In the spring of 1998 the permanganate oxidativity started to increase and was in the range of about 200 – 250 mg/l until the end of the experiment (in October 1998). It was found that the substrate utilization was about 35% during the entire incubation period.

The temperature was an essential factor affecting the rate of both substrate biodegradation and microbial sulphate reduction. The temperature inside the barrier varied in the range of about 5 – 32^{o}C during the different clymatic seasons. The temperature coefficient Q_{10} within this range varied from 1.5 to 2.1. The maximum sulphate reduction rate achieved during this study was 190 mg/l.h. The main factor limiting the rate of the sulphate reduction was the relatively low concentrations of the electron donor (i.e. of the dissolved organic carbon). Regardless of this, the effluents from the barrier were enriched in dissolved organic compounds. The concentration of ammonium ions in the effluents was also increased considerably due to the ammonification of the organic matter. The concentration of phosphate ions was also increased and this was connected with the solubilization of a portion of the phosphate present in the spent mushroom compost. The effluents still contained high concentrations of hydrogen sulphide.

Sampling of the groundwaters after their treatment in the reactive barrier revealed that a watercourse of about 12-15 m through the rock mass was sufficient to decrease the concentration of dissolved organic compounds, ammonia, phosphates and hydrogen sulphide below the relevant permissible levels.

It must be noted that the permeability of the rock serving as a back wall of the barrier steadily decreased due to different precipitation products (mainly fine particles of different mineral sulphides) present in the barrier effluents.

The data from this study revealed that the permeable sulphate-reducing and alkalinity-producing permeable barriers can be efficiently applied in commercial scale to treat acid drainage waters.

Acknowledgements A part of this study was funded by the National Science Fund (Research Contract № TH-803/98).

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Recommended for publication by Department of Engeneering Geoecology, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.233-236

COMPOSITE MATERIAL FOR ISOLATION OF DEPOTS FOR INDUSTRIAL WASTE AND WASTE OF LIFE

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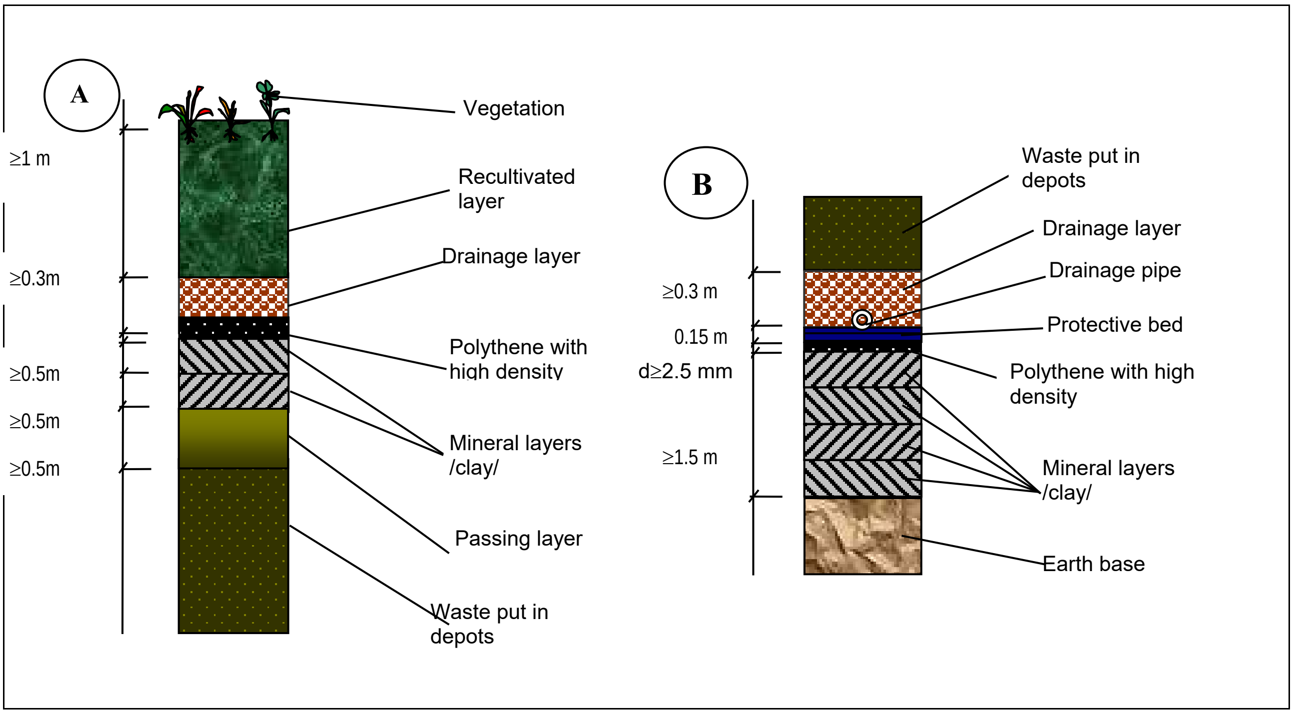
ABSTRACT

A new composite material on the base of industrial waste plus additions from inert materials is made. Many laboratory tests are carried out – for defining the convective and diffusion spreading of heavy metals and other harms; the chemical stability to suffosion and the basic mechanical indicators. The material's suitability as an element in the seal depots' stoppings is proved. The possibility of using the new product in building industrial waste depots and depots for waste of life is motivated. Ecological application of waste, polluting the environment, is found out.

INTRODUCTION

Basic way for decreasing the dangerous effect of waste is their preserving in suitable waste dumps (WD). The seal stoppings of contemporary WD are intended to decrease the waste influence [1]. That's why it is necessary the WS to be treated as very important engineering equipment.

The building of depots for solid waste of life and industrial waste is necessary to be combined with application of reliable bottom and covering seal stoppings which provide minimum movement of harms towards air environment and underground waters and soils. This is underlied in the European conception from 1986 [2] in which the idea of partitioning barriers, is developed in order the waste, put in depots, to be controlled and a monitoring to be realized.



A – covering seal stopping; **B** – bottom seal stopping

Figure 1. Vertical cuts of seal stoppings used for waste (industrial and of life) putting in depots

The contemporary technical barriers in depots have to be conformed to European standards for firming and seal stoppings. The latter are built from natural and artificial materials. The contemporary construction of bottom and covering seal stoppings contains layers with well-compacted clays with filtration coefficient ($K_f \leq 1.10^{-9} \div 1.10^{-11}$ m/s), and thickness of 1.5 m and 0.5 m respectively. In the first case clays are with predominant montmorillonite composition, in the second – with caolinite composition, which have the necessary stability to chemical suffosion. The construction of stoppings is shown on fig. 1 [3]. In order to be provided better isolation, high density polythene (HDPE) and polypropylen (PP) in the form of clothes [4] are put.

The type of bottom and covering stoppings must be conformed to the depot class. The depots are classified in three groups according to the quantity of organic hydrocarbons, which are emitted from them and infiltrated by the underground soils and waters [5].

Considerable quantities of high quality clays for mineral layers in the stoppings (1 $m^3 \div 3$ m^3 for 1m^2 from the area of the depot) and big costs, connected with them, are the reason for searching and creating new materials.

According to German standards [6], using the alternative seal materials is allowed under the condition that they correspond to the criterion for classical ones and have proved qualities, for example, the asphalt concrete, which is a waterproof material. It is suitable for isolation and is applied for many years. As an example it may be given the isolation of more than 20 depots [7] with isolation area of 203 600 m^2 in Switzerland. The structure of asphalt concrete is given on fig. 2 [8].

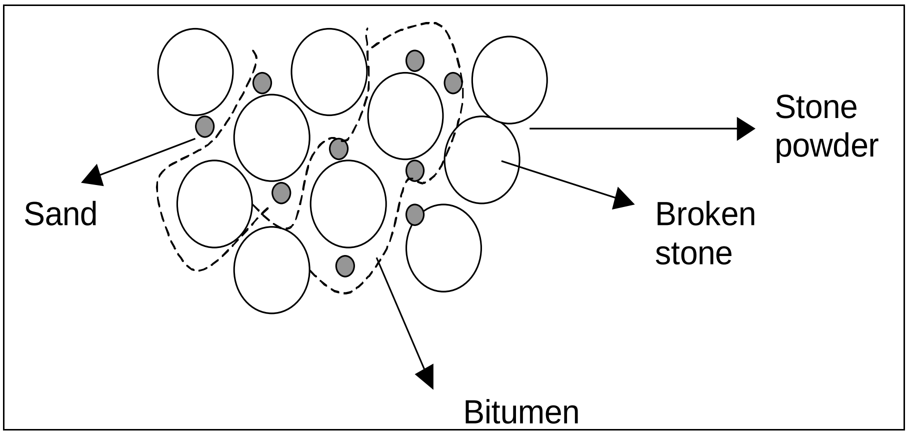


Figure 2. Characterizing scheme of the asphalt concrete

The bitumen and stone powder are put as filler in order to fill the gaps between particles of inert materials. The aim is a dense mass to be formed. It has to be said that the asphalt concrete is good for using under the condition that after compacting the pores is under 3% and the binder is more than 5 weight per cents towards the mass of material. It is necessary the filtration coefficient to be conformed to the European standards and requirements.

Material for seal stoppings

The new seal material, called tar concrete, is formed on the base of technological waste from crude oil processing through cracking plus fillers. The waste (tar) has the role of a binder. Its composition is defined through Markuson method (AASHTOT 59/Test methods for emulsified asphalts) and is shown in table №1.

Table 1. Waste composition

Components	Quantity, weight per cents
Water contents	33 % as emulsified water
Contents of dry substance	
A_7 (Asphaltens)	14.06 %
Paraffins	36.70%
Resins	49.24 %

Fillers in the new material, which are used for seal layers of the depots, are natural materials. They are shown in table №2 [9].

Table 2. Seal material composition

Composition	Weight per cents, %
Broken stone fraction 5/10	$20 \div 30$
Broken stone fraction 0/5	$50 \div 60$
Bentonite	$2 \div 5$
Stone powder (Si_2O)	$10 \div 20$
Waste	$8 \div 11$

The ratio between components of crude oil waste, which compose the oil phase of emulsion, are close to those of the distilled bitumen, used in road construction. The high presence of tars is the reason for high stability of the new material to oxydation and photooxydation ageing.

The new material has a filtration coefficient $K_f = 1.10^{-9} \div 1.10^{-11}$ m/s [10], which is proved for 12 000 minutes. Samples, compacted with static pressure till forming a seal layer with volume density $\rho = 1.85 \div 2.10$ g/cm^3, show a filtration coefficient $K_f = 5.10^{-10} \div 5.10^{-11}$ m/s (fig. 3).

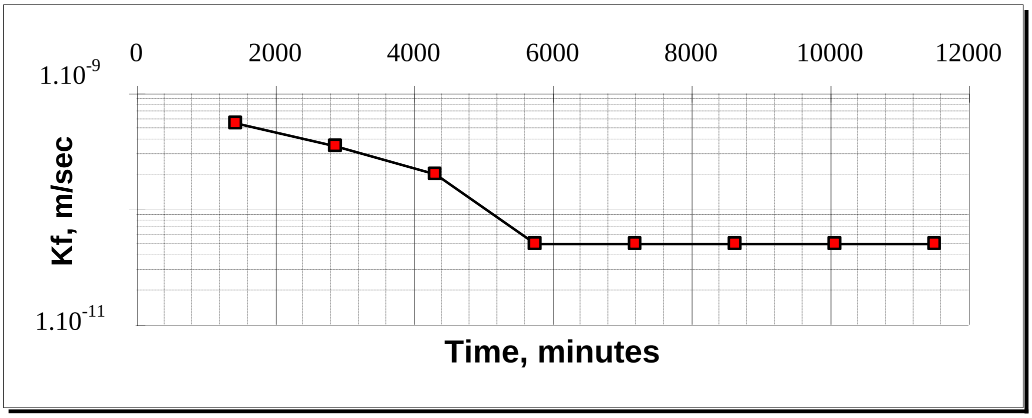


Figure 3. Filtration coefficient of the new material

These values correspond to the requirements, given in “Technische Verordnung über Abfälle”.

Tests for defining the diffusion coefficient (K_d) are carried out [10]. The values of K_d are in these limits – $4.10^{-11} \div 4.8.10^{-11}$ m/s. The results are stabilized and constant after the 50th day from the test beginning.

The chemical stability (Fig. №3) of the alternative material to different chemical solutions / Thulol (C_7H_8) with purity of 99,5 volume per cents and acid indicator pH - 5,5; NaOH and HCl with pH indicator of 8,0 and 2,0 / was tested in laboratory through the drop method [1]. Chemical reaction of the material with HCl and NaOH was not proved and it kept its stability for a long time. During tests of 720 minutes with thulol in concentrated form, only surface reaction is indicated. A trace, deep 2÷3 mm and with diametre of 10 mm, is seen. It shows that although the solution aggressiveness the binder is steady in the material and keeps its stability. Putting the inert fillers on silicon base (SiO_2) is obligatory. They determine the good chemical stability of the material while the carbonate fillers would react with acids and CO_2 will be formed. Components in binder’s composition (asphaltens, oils and tars) are extremely stable to acids and alkalis. In the waste water from depots of waste of life, presence of thulol may be expected in concentration not higher than ppm (that is thulol, soluble in water in quantity of 515 mg/l at 20 °C). That’s why it is possible to say that the new material is stable to the influence of organic solutions [8].

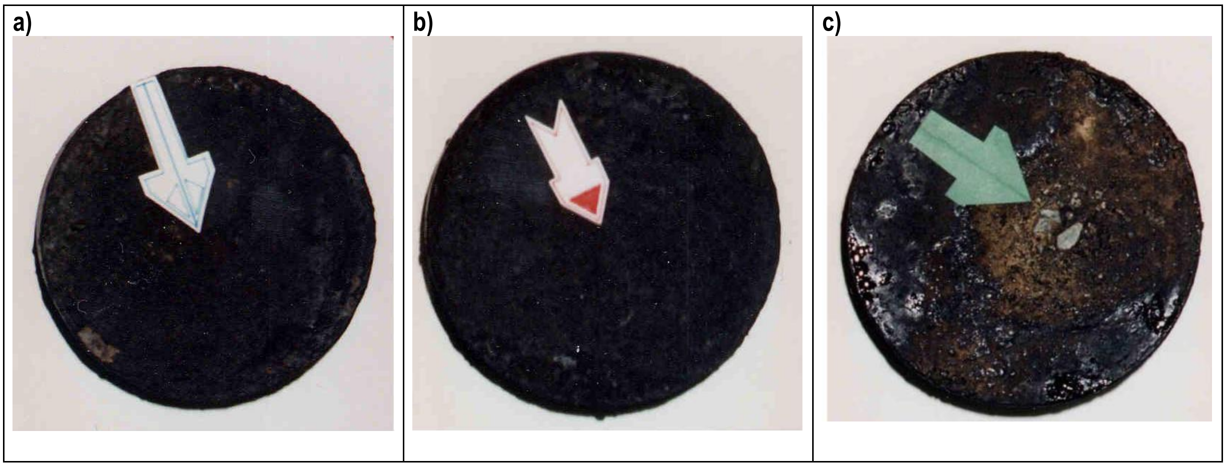


Figure 4. Chemical reaction of the material with: (a) HCl (b) NaOH and (c) Thulol

The composite material bears longitudinal deformation and can undertake considerable vertical bearings up to 0.15 MPa, which are typical for lower seal stoppings.

The technology of preparing and building waterproof and diffusion stable layers from the new material is extremely popular [9]. The homogenized mixture is put still hot on the preliminary leveled and compacted clay or earth base. The spreading is done by machines or by hand and the layer’s width is not more than 15cm. The compaction is done through packing until the suitable volume density is reached. The seal stopping is covered with earth mass or clay layer when it cools down and gets the atmosphere temperature. The composite material stops water and humidity penetration from the underground water in the waste, put in depots, and vice versa, the diffusion process in two directions is practically equal to zero.

CONCLUSIONS

Using the waste products for economic aims is a real alternative of soils’ recovery. With the help of waste, received from the deep processing of crude oil, a seal material, which is stable to chemical aggression of acids, alkalis and thulol solutions, is prepared. The calculated filtration coefficient of $5.10^{-9} \div 5.10^{-11}$ m/s shows that the material is suitable to be used as a seal base in building depots for industrial waste and waste of life. The high strength is another priority. It may be increased through putting geotextile as a structural material because of suitable relations between geotextile fibres and tars from the waste material. The new product is able to form steady waterproof and diffusion resistant covering stoppings. Constructions, which include a layer of tar concrete, are several times cheaper than classical because of considerable reducing the expensive clays; the expensive HDPE - cloth is eliminated; the building time is decreased; the classical road building machines are used.

Gratitude: *The authors heartily thank to the Austrian Institute for East and South-East Europe – Office Sofia for the given financial support.*

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Recommended for publication by Department of Underground construction, Faculty of Mining Technology

50 years University of Mining and Geology "St. Ivan Rilski"
Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.237-239

ECOLOGICAL RED GLASS

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ABSTRACT
The interest in glasses of the copper ruby type is re-actualized due to the present general tendency to avoid the fabrication and the use of products with a content of toxic or potential pollutant substances. Copper gives a red color as a result of some subtle redox processes doubled by thermal treatments. Although a great volume of practical experience was accumulated many aspects of ruby formation mechanism remained still insufficient elucidated and the rigorous control of numerous parameters influencing the color quality is still difficult or even impossible. In the present work some results are communicated, obtained in the frame of a study following both the more deeply and correct understanding of copper ruby formation processes and the accumulation of data about the role and the influence of technological parameters.

INTRODUCTION

Although in glasses can be obtained a great variety of colors and nuances in the case of red color the situation is special, the possibilities being more restrained. The electronic transitions of the usual coloring ions need, as a rule, energies corresponding to the middle of the visible spectrum towards red or to UV, the transmitted trough glass light having yellow – green – blue colors.

The transmission of yellow – red – IR radiations is possible by means of different mechanisms, preponderantly through light diffusion on colloidal aggregates nano or micro crystalline. In this aim most frequently are used cadmium sulphide, cadmium selenide and sulfo-selenide. On the basis of a similar mechanism one may obtain beautiful red colors by means of colloidal aggregates of Au, Ag or Cu [*Balta P.,* 1984; *Balta P. et al.* 2001].

In the last years glass industry was affected by the tendency, manifested throughout the world, to avoid the use of potential dangerous or pollutant substances. Such a new problem is contoured related to glasses with cadmium content, toxic for human body and pollutant for the environment.

The copper ruby seems to be a possible alternative. Because the implicated subtle redox processes and thermal treatments the obtainment of reproducible colors and nuances at an industrial scale is difficult. In this some results contributing to a more deeply understanding of the redox processes mechanisms and of the peculiarities of technological parameters are presented.

COPPER RUBY

The strong dispersed during melting metal atoms does not color glass. It is needed a developing, striking, thermal treatment to obtain the colloidal aggregates. The red ruby color is obtained when the colloidal aggregates reach dimensions of the order of 50 nm. The coloring mechanism is based preponderantly on the diffusion of light on colloidal particles.

The light diffusion intensity I_D may be evaluated by means of the Rayleigh's equation:

$$I_D = \frac{\pi . V^2}{\lambda^4 . r^2} . \varepsilon_r{}^2 (\Delta\varepsilon_r - 1)\sin^2\alpha \quad (1)$$

where V is the particles volume, λ the wavelength of the incident light, ε_r the dielectric relative permittivity, r the distance from the particle to the point light intensity measurement and α the angle between the diffused light fascicle and the incident one. It may be remarked the fact that, beside of some parameters related to the measuring conditions, diffusion is strong influenced by the wavelength of the incident light.

Diffusion is more intense when the wavelength is shorter, i.e. in UV – blue – green domain. Thus, through glass pass preponderantly the wavelengths corresponding to domain of yellow – red. The particle concentration has a strong influence too but at their increase, or at the increase of particle dimensions, glass become opalescent or even opaque [*Balta P.,* 1984; *Balta P. et al.* 2001].

THE WORKING MANNER

The copper ruby study followed particularity the redox processes both on simple glasses used as a kind of models and on the industrial type glasses. The model glass was chosen in the system Na_2O-B_2O_3 (10wt%Na_2O). Was used an organic reducing agent (saccharose) but also metallic Sn. Copper was introduced as CuO,Cu_2O or even Cu. Melting was made in an electric furnace, in ceramic or platinum crucibles, at 1000^0C for borate glasses or 1450^0C for silicate ones, in normal atmosphere. The samples having shapes of discs with a diameter of 25 mm and a thickness of 1-2mm were obtained by pressing molten glass in a metallic form. They were annealed and thermal treated in different conditions. Absorption spectra were recorded by means of a two-beam spectrophotometer Shimadzu UV-160A, in the domain 400-1100 nm, sometime beginning from 200 nm, in comparison with the same glass but without Cu.

COPPER RUBY IN BORATE GLASSES

In a first series of glasses copper was introduced as Cu_2O with the aim to check the opinion of some research workers according to that the colloidal aggregates are formed by cuprous oxide and not by elementary Cu. Even in the absence of the reducing agent the red color must appear [*Capatina C.*, 2001]. After melting glasses were colorless and remained so even after long heath treatments. The Cu^+ presence was evidenced the charge transfer absorption in far UV specific to this ion and by means of chemical analysis.

The conclusion was that, at least in this glass, cuprous oxide did not form colloidal coloring aggregates. Introducing an organic reducing in glasses containing Cu_2O or CuO, red colored glasses were obtained. In Fig. 1 are presented the electronic spectra of glasses initially containing CuO. The spectra interpretation seems to be quite simple. After a quart of hour of melting the peak at 590 nm has the maximum amplitude indicating the presence of a great quantity of colloidal aggregates but it can be observed also a small maximum at 800 nm corresponding to some not yet reduced CuO.

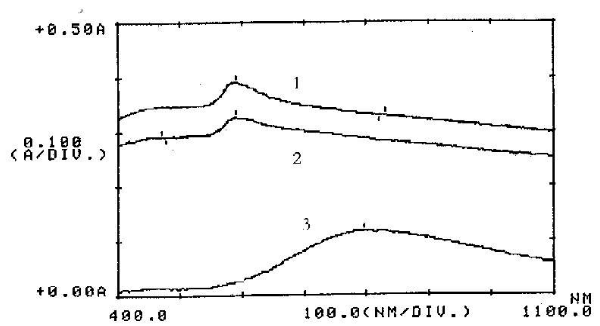

Figure 1. Electronic spectra of borate ruby glasses. Copper was introduced as a CuO and was used an organic reducer. Melting duration: 1 – 15 min., 2 – 30 min., 3 – 60 min.

After 30 minutes the peak at 590 nm is practically unchanged but that of Cu^{2+} totally disappeared. After 60 minutes the peak at 590 nm is not more visible but, in exchange, that at 800 nm is very pronounced.

The high absorption level showed by ruby glasses spectra over all visible is due to the presence of carbon resulted by organic compound decomposition. Following the carbon disappearance the necessary oxygen quantity was calculated resulting that the main oxygen source is the surrounding atmosphere. An apparent difussion coefficient of oxygen in borate melt at 1000 ^{0}C of the order of 10^{-4} – 10^{-5} cm^2s^{-1} was estimated [*Capatina C.,* 2000]. The very high value for this kind of glasses can be explained by the convection contribution, determined by gas bubbles elimination during the organic reducer burning.

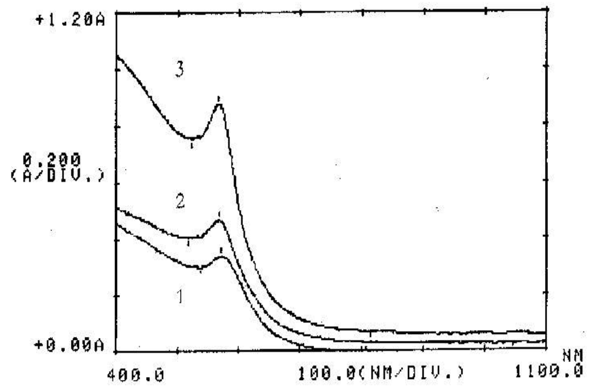

Figure 2. Electronic spectra of borate ruby glasses with copper from a Cu-Sn alloy. The heath treatement duration was: 1 – 120 min., 2 – 210 min., 3 – 240 min.

An usual soda-lime-silica glass was experimented using as a source of Cu and Sn the same alloy. The melting was made at 1450 ^{0}C in normal atmosphere. The striking treatment was made at temperatures and durations. Some of the recorded spectra are presented in figure 3.

Inspired from the Professor's Cristea and his collaborators works, in a series of borate glasses copper was introduced in the form of a Cu-Sn alloy [*Cristea V. et al.*, 1975]. The thermal were performed at 500 ^{0}C. The color appeared only after about 25 minutes. In figure 2 spectra obtained after longer treatments are presented.

At least three differences may be remarked comparing with ruby glasses obtained with organic reducer [*Ram A., et al.,* 1974].

1. the absorption due to carbon resulted by reducer burning is missing
2. a large window is present from about 568 nm up to IR
3. the peak amplitude increases apparently exponentially with time.

It may observe a strong absorption at about 566 nm due to copper colloidal aggregates, a sharp absorption limit a uniform transmission towards IR. The longest melting duration results in the apparition of a very small peak at about 800 nm specific for Cu^{2+} singaling the total oxidation of Sn. In this way a saturation is evidenced related to the end of Sn oxidation process and the beginning of the copper oxidation up to Cu^{2+}. The presence of this copper ion alters the ruby quality and represents the upper time for the good quality ruby glass elaboration process.

The striking temperature and time have a synergetic action the 566 nm peak amplitude depending linearly on temperature but logarithmically on time. These are the main technologic parameters determining, together with copper concentration, the nuance and the intensity of copper ruby glass color.

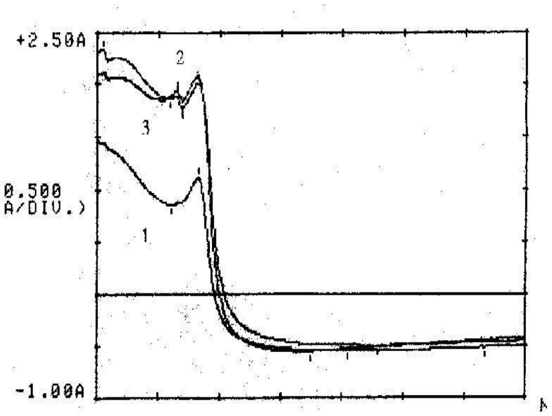

Figure 3. The electronic spectra of the industrial type ruby glasses with alloy, maintained at the melting temperature: 1 – 90 min., 2 – 150 min., 3 – 210 min
Copper ruby in industrial type glasses

Another important achievement consists in the fact that the glass composition has a low influence on the copper colloidal aggregates absorption peak position. Passing from silicate glass to borate and phosphate glasses the peak position modification is very small, inside of some 23 nm [*Weyl W.A.,* 1967].

CONCLUSIONS

Due to the actual tendency to avoid the use of cadmium in red glasses were explored the redox mechanism and the technology of cooper ruby, which seems to be a reasonable alternative.

Using a sodium – borate model glass were melted and studies containing Cu_2O, CuO or Cu^0, using an organics reducer and also Sn from a Cu-Sn alloy. The sodium – borate glass with Cu_2O content did not exhibit a red color. The advantage of using Cu-Sn instead of other copper sources and of the organic reducer was evidenced.

The copper obtainment in an industrial type glass was studied achieving some new information concerning a saturation phenomenon related to Sn quantity, the synergetic influence of striking temperature and time and the low influence of glass composition upon the colloidal aggregates absorption peak position.

The accumulated results contribute to the more deeply understanding of redox processes mechanism at copper ruby obtainment and the influence of some technological parameters. Now it is possible to imagine experiments at an industrial scale to establish the conditions for copper ruby glass obtainment in large quantities.

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.241-242

THE INFLUENCE OF ADDITION ELEMENTS ON SINTERED IRON COMPACTS MICROHARDNESS

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ABSTRACT
New antifriction Materials based on iron powder, with several addition elements were developed using PM technologies. The samples were obtained by uniaxially cold compaction using pressure forces of 300 MPa and sintering in dry hydrogen atmosphere at three different temperatures. In this study the effect of copper, in and lead addition elements on Vickers hardness and microhardness as a function of process parameters was investigated. Vickers hardness was evaluated at an indentation load of 50 N and Vickers microhardness was evaluated at an indentation load of 0.1 N.
Keywords: powder metallurgy, sintered iron compacts, microhardness, addition elements

INTRODUCTION

Technical and economical advantages of the P/M technology lead to performing a wide type of new materials and products with special characteristic required by modern techniques [1].

In order to meet the requirements of future possible applications, it is important to improve the existing and develop new methods of enhancing the applied properties. This may be achieved by efficient alloying, using efficient combinations of alloying elements.

Many sintered parts reach sufficiently high strength properties, for example, similar to cast iron, already at a porosity of 20-15%. Controlled non-homogeneity of the structure by powder metallurgy processes maker it possible to obtain special properties of materials, which cannot be obtained by conventional technologies. It is essential to know the actual loading conditions of the part and modify alloying and the treatment conditions of the material on the basis of these conditions [2].

The metallic powder sintered parts present the remarkable physical, chemical and mechanical characteristics, which are determined by their composition and the phase structure as well as the shape and mass distribution of the grains [3].
The study also attempts to optimize the addition elements and sintering condition (temperature, maintaining time) in order to obtain adequate values of hardness and to investigate the quality of the sintered materials microstructure.

EXPERIMENTAL PROCEDURE

As experimental materials, iron powder by Ductil S.A. Buzău DWP 200 electrolytic copper powder, brass, tin and lead powder were used.

The powder mixtures were cold compacted in a die with single action of the upper punch at a pressure of 300 MPa obtaining 10 mm cylindrical sample. The compacted samples were placed in a tubular furnace having uniform heating zone and sintered at 600^{0}C, 650^{0}C and 600^{0}C for 20, 25 and 45 minutes.

The sintering atmosphere was dry hydrogen with a flow rate of 11/min. The samples were cooled in furnace by switching off the powder and maintaining the same flow rate of the hydrogen gas. The reference densities for selected composition were calculated by the rule of mixtures.

The sintered samples were metallographic prepared by polishing in order to investigate the porosity and by reactive etching in order to evaluate microhardness of the individual grain and structural constituents. Reference densities for each alloy composition were calculated by the rule of mixtures and the total porosity of the sintered specimens was evaluated from the difference between the reference density and calculated density.

The characteristics of the elemental powders and the composition of the mixtures are presented in Table 1 and the experimental conditions are presented in Table 2.

Table. 1.The influence of addition elements on the sintered iron.
Sample.

POWDER COMPOSITION							
	IRON %	Si %	Mu %	P %	S %	O_2 LOSS	O_2
DWP 200	0,020	0,050	0,200	0,020	0,015	0,200	0,220
CHARACTERISTICS OF ELEMENTAL POWDER							
Particle size (μm)		>160	160-100	100-63		<63	
DWP 200		15%	20-40%	20-40%		20-45%	

APARENT DENSITY 9/cm³	2,5-2,7
FLOW RATE (S/50 g)	33

From this tipe by iron powder obtained three materials FC-40 (iron-lase, C-0,4%), FC-80 (Iron-base, C-0,80%) and Fe 50 U3 (iron base, C-0,5%, Cu-3%).

CONCLUSION

Hardness express the resistance of material to deformation of the surface caused by the effect of a geometrically defined body.

The value obtained by these methods ar referred to as macrohardness, the value increasing by addition elements and time by sintering process.

Table 2 Experimental conditions.

Sintering Conditions	Sintering temperature [⁰C]	600-650⁰
	Holding time [min]	20,35, 40
	Atmosphere	Dry hydrogen
Sintered density [g/cm³]		6,95

Tests of hardness of powder materials in determining the hardness of the material of a whole, including pores, were carried out using the Vickers method.

RESULTS AND DISCUSSIONS

Hardness expresses the resistance of the material to the deformation of the surface caused by the effect of a geometrically defined body. Although, the value of the porous materials hardness is always lower then the compact materials hardness, for a general qualitative characterization of the sintered alloys macrohardness Vickers method has been used. This method is based on the Vickers method with a very low load.

The influence of the additional elements contents from the hardness (the selected compositions containing cooper powder and brass powder) is in Figures 1.

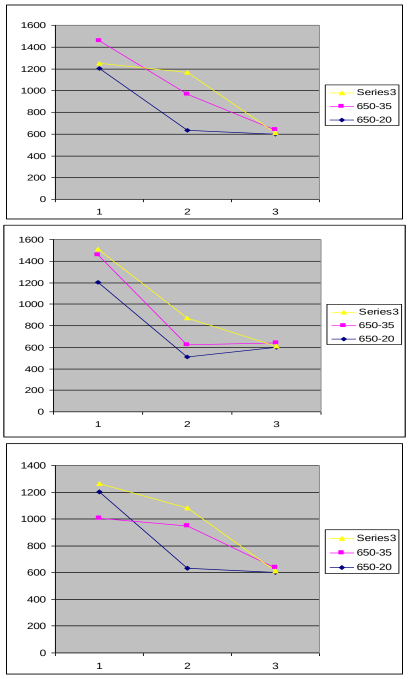

Figure 1. The influence of the additional elements and sintering parameters on the hardness.

The influence of hardness of additional elements from sintering parameters is descrived in Figures 2.

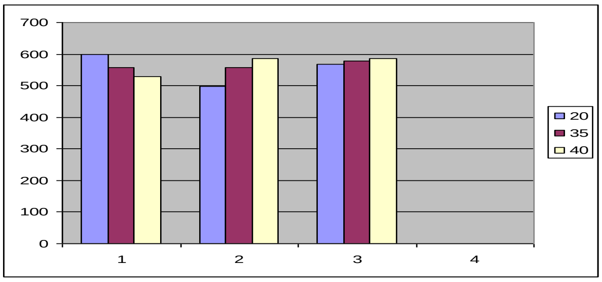

Figure 2. The influence of the additional elements and sintering parameters on the hardness of the alloys containing brass powder.

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RUBICI b: SINTERED MACHINE ELEMENTS ELLIS NORWOOD.

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Annual, vol. 46, part II, Mining and Mineral Processing, Sofia, 2003, pp.243-246

RADIATIVE HEAT TRANSFER EQUATION IN SYSTEMS OF GREY - DIFFUSE SURFACES SEPARATED BY NON-PARTICIPATING MEDIA

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ABSTRACT
The paper proposes a matrix model for solving the radiative heat transfer in enclosures based on mutual angular radiation coefficients and radiative properties of the surfaces that make up the enclosure.

INTRODUCTION

Radiative heat transfer represents the most important part of the global heat exchange in thermal installations working at temperatures, such as steam generators, boilers, furnaces and so on. The two components of the radiative heat transfer – (1) between two or more surfaces and (2) between gas and surfaces – on the one hand, and on the other hand the complex geometry of most heat exchange systems, make the simulation of the radiative heat transfer a complex process, more suited for the numerical approach. Besides, a serious drawback is the lack of a standard terminology in the field of radiative heat transfer. The nomenclature used in this paper is based on that of the illumination engineering community due to the close relationship between physical terms and standardized terms used in the field of illumination engineering.

The basic concepts of radiative heat transfer such as direction and solid angle, radiant intensity, radiosity, emittance, absorbance, reflectance and so on, are considered known and aren't discussed in this paper. The basic geometrical parameters are represented in figure 1. A summary of the basic terms used henceforth is given below:
- spectral radiation intensity due to the own emission of the surface:

$$I_{\lambda,e} = \frac{\delta \dot{Q}}{\cos\varphi \, dS_1 d\Omega \, d\lambda}$$

Index *e* means self-emitted.
- spectral power density:

$$E_\lambda(\lambda) = \int_0^{2\pi} d\psi \int_0^{\pi/2} I_{\lambda,e}(\lambda,\varphi,\psi)\cos\varphi \sin\varphi d\varphi$$

- spectral radiation intensity due to incident radiation:

$$I_{\lambda,i} = \frac{\delta \dot{Q}}{\cos\varphi \, dS_1 d\Omega d\lambda}$$

Index i means incident to the surface considered.
- spectral irradiation:

$$G_\lambda(\lambda) = \int_0^{2\pi} d\psi \int_0^{\pi/2} I_{\lambda,i}(\lambda,\varphi,\psi)\cos\varphi \sin\varphi \, d\varphi$$

spectral radiosity:

$$J_\lambda(\lambda) = \int_0^{2\pi} d\psi \int_0^{\pi/2} I_{\lambda,e+r}(\lambda,\varphi,\psi)\cos\varphi \sin\varphi \, d\varphi$$

Radiosity takes into account both self-emitted and reflected radiation

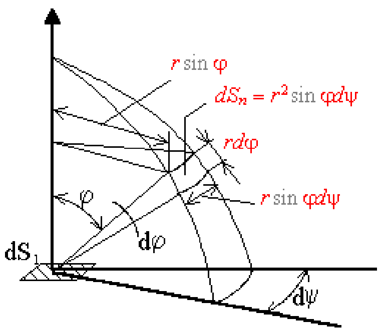

Figure 1

RADIATIVE HEAT TRANSFER IN A SYSTEM OF SURFACES

The system of surfaces considered consists of n flat diffuse-grey surfaces which make up an enclosure.
The net radiative heat flux of a surface i is given by the difference between the radiated heat flux due to emission and

reflection and the incident radiative heat flux between two random patches (figure 2):

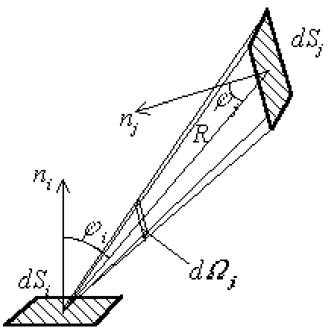

Figure 2

$$\dot{Q}_i = S_i(J_i - G_i) \tag{1}$$

in which the radiosity of surface *i* is given by: $J_i = E_i + R_i G_i$ with R_i – reflectance of the surface *i*; for an opaque surface, $R_i = 1 - A_i$, and (1) turns into:

$$Q_i = S_i(E_i - A_i G_i) \tag{2}$$

The radiative heat flux given by (2) has a non-zero value unless the considered surface gets an equivalent heat flux either through convection or conduction. If heat exchange is accomplished exclusively through radiation and if steady state is reached, then $\dot{Q}_i = 0$.

For diffuse-grey surfaces the emittance equals the absorbtance according to Kirchhoff's law: $\varepsilon_i = A_i$. In this case the radiosity is given by:

$$J_i = \varepsilon_i E_{b,i} + (1 - A_i) G_i \tag{3}$$

with $E_{0,i}$ the power density of the black body having the same temperature as the surface considered.
The above equation changes (1) to:

$$\dot{Q}_i = S_i\left(J_i - \frac{J_i - \varepsilon_i E_{b,i}}{1 - \varepsilon_i}\right) \tag{4}$$

or:

$$\dot{Q}_i = \frac{E_{b,i} - J_i}{\dfrac{1 - \varepsilon_i}{\varepsilon_i S_i}} \tag{5}$$

Equation (5) describes the radiative heat exchange from a surface. It suggests an analogy with Ohm's law leading to the definition of the radiative resistance of the surface *i*:

$$R_{r,i} = \frac{1 - \varepsilon_i}{\varepsilon_i S_i} \tag{6}$$

If the surface *i* belongs to an enclosure then the radiosity J_i depends on the heat transfer among surface *i* and surfaces *j* (*j*=1...*n*). The radiative heat flux radiated by surface *i* and received by the surface *j* is given by:

$$Q_{i \to j} = \varphi_{ij} J_i S_i \tag{7}$$

and the radiative heat flux radiated by surface *j* and received by the surface *i* is given by:

$$Q_{j \to i} = \varphi_{ji} J_j S_j \tag{8}$$

In which φ_{ij} and φ_{ji} are the mutual medium angular radiation coefficients.

Considering a system consisting of two surfaces *i* and *j*, the mutual medium angular radiation coefficient φ_{ij} is defined as the ratio between the radiative heat flux emerging from S_i and intercepted by S_j and the overall radiative heat flux emerging from S_i:

$$\varphi_{ij} = \frac{\dot{Q}_{i \to j}}{J_i S_i} \tag{9}$$

Isolating two elementary patches dS_i and dS_j and applying Lambert's law, the elementary radiative heat flux emerging from dS_i in direction φ_i in the elementary solid angle $d\Omega_{ji}$ is given by:

$$\delta \dot{Q}_{i \to j} = I_i \cos\varphi_i dS_i d\Omega_{ji} \tag{10}$$

Taking into account the definition of the solid angle, the above expression turns into:

$$\delta \dot{Q}_{i \to j} = I_i \frac{\cos\varphi_i \cos\varphi_j}{R^2} dS_i dS_j \tag{11}$$

For diffuse surfaces, $I_i = \dfrac{J_i}{\pi}$ and equation (11) changes to the following form:

$$\delta \dot{Q}_{i \to j} = J_i \frac{\cos\varphi_i \cos\varphi_j}{\pi R^2} dS_i dS_j \tag{12}$$

As a result of integration, equation (12) turns into:

$$Q_{i \to j} = J_i \int_{S_i}\int_{S_j} \frac{\cos\varphi_i \cos\varphi_j}{R^2} dS_i dS_j \tag{13}$$

The net radiative heat flux between surfaces I and j is given by:

$$\dot{Q}_{ij} = \dot{Q}_{i \to j} - \dot{Q}_{j \to i} \tag{14}$$

Taking into account the well-known reciprocity property of the medium angular radiation coefficients (14) becomes:

$$Q_{ij} = \varphi_{ij} S_i (J_i - J_j) \tag{15}$$

or:

$$\dot{Q}_{ij} = \frac{J_i - J_j}{R_{r,ij}} \tag{16}$$

In which the geometric radiative resistance $R_{r,ij}$ is given by:

$$R_{r,ij} = \frac{1}{\varphi_{ij} S_i} = \frac{1}{S_{ij}} \tag{17}$$

The geometric radiative resistance $R_{r,ij}$ depends solely on the geometry of the radiant system, unlike $R_{r,i}$, which depends also on the radiative properties of the material.
The total incident heat flux on the surface i is given by:

$$G_i S_i = \sum_{j=1}^{n} \varphi_{ji} S_j J_j = \sum_{j=1}^{n} \varphi_{ij} S_i J_j \tag{18}$$

from which emerges the irradiation of surface *i*:

$$G_i = \sum_{j=1}^{n} \varphi_{ij} J_j \tag{19}$$

Combining (1) and (19):

$$\dot{Q} = S_i \left(J_i - \sum_{j=1}^{n} \varphi_{ij} J_j \right) \tag{20}$$

or, taking into account the enclosure property of the medium angular radiation coefficients $J_i S_i = \sum_{j=1}^{n} \varphi_{ij} J_i S_i$, equation (20) yields consecutively:

$$Q_i = \sum_{j=1}^{n} S_i \varphi_{ij} J_i - S_i \sum_{j=1}^{n} \varphi_{ij} J_j \tag{21}$$

$$\dot{Q}_i = \sum_{j=1}^{n} S_i \varphi_{ij} (J_i - J_j) = \sum_{j=1}^{n} \dot{Q}_{ij} \tag{22}$$

Equation (5) and (22) lead to:

$$\frac{E_{b,i} - J_i}{\dfrac{1-\varepsilon_i}{\varepsilon_i S_i}} = \sum_{j=1}^{n} \frac{J_i - J_j}{\dfrac{1}{\varphi_{ij} S_i}} \tag{23}$$

In a system of grey-diffuse surfaces (23) is written as:

$$\frac{E_i - J_i}{R_i} = \sum_{j=1}^{n} \frac{J_i - J_j}{R_{ij}} \tag{24}$$

or:

$$\frac{E_i}{R_i} - \frac{1}{R_i} J_i = J_i \sum_{j=1}^{n} \frac{1}{R_{ij}} - \sum_{j=1}^{n} \frac{1}{R_{ij}} J_j \tag{25}$$

which, upon rearranging, becomes:

$$\frac{E_i}{R_i} + \sum_{j=1}^{n} \frac{1}{R_{ij}} J_j = \frac{1}{R_i} + \sum_{j=1}^{n} \frac{1}{R_{ij}} J_i \tag{26}$$

The radiosity is given by:

$$J_i = \frac{\dfrac{1}{R_i}}{\dfrac{1}{R_i} + \sum_{j=1}^{n} \dfrac{1}{R_{ij}}} E_i + \frac{1}{\dfrac{1}{R_i} + \sum_{j=1}^{n} \dfrac{1}{R_{ij}}} \sum_{j=1}^{n} \frac{1}{R_{ij}} J_j \tag{27}$$

or:

$$\begin{bmatrix} J_1 \\ J_2 \\ .. \\ .. \\ J_n \end{bmatrix} = \begin{bmatrix} \dfrac{\rho_1}{\Phi_1} E_1 \\ \dfrac{\rho_2}{\Phi_2} E_2 \\ .. \\ .. \\ \dfrac{\rho_n}{\Phi_n} E_n \end{bmatrix} + \begin{bmatrix} \Phi_1 \rho_{11} & \Phi_1 \rho_{12} & ... & \Phi_1 \rho_{1n} \\ \Phi_2 \rho_{21} & \Phi_2 \rho_{22} & ... & \Phi_2 \rho_{2n} \\ ... & ... & ... & ... \\ \Phi_n \rho_{n1} & \Phi_n \rho_{n2} & ... & \Phi_n \rho_{nn} \end{bmatrix} \bullet$$

$$\bullet \begin{bmatrix} J_1 \\ J_2 \\ .. \\ .. \\ J_n \end{bmatrix} \tag{28}$$

in which: $\Phi_i = \frac{1}{R_i} + \sum_{j=1}^{n} \frac{1}{R_{ij}}$, $\rho_i = \frac{1}{R_i}$ and $\rho_{ij} = \frac{1}{R_{ij}}$.

Subtracting from both left and right member the term representing the radiosities matrix, equation (28) turns into:

$$\begin{bmatrix} 0 \\ 0 \\ .. \\ .. \\ 0 \end{bmatrix} = \begin{bmatrix} \frac{\rho_1}{\Phi_1} E_1 \\ \frac{\rho_2}{\Phi_2} E_2 \\ .. \\ .. \\ \frac{\rho_n}{\Phi_n} E_n \end{bmatrix} + \left(\begin{bmatrix} \Phi_1 \rho_{11} & \Phi_1 \rho_{12} & ... & \Phi_1 \rho_{1n} \\ \Phi_2 \rho_{21} & \Phi_2 \rho_{22} & ... & \Phi_2 \rho_{2n} \\ ... & ... & ... & ... \\ \Phi_n \rho_{n1} & \Phi_n \rho_{n2} & ... & \Phi_n \rho_{nn} \end{bmatrix} - I \right) \bullet$$

$$\bullet \begin{bmatrix} J_1 \\ J_2 \\ .. \\ .. \\ J_n \end{bmatrix} \qquad (29)$$

in which I is the identity matrix:

$$I = \begin{bmatrix} 1 & 0 & ... & 0 \\ 0 & 1 & ... & 0 \\ 0 & 0 & ... & 0 \\ 0 & 0 & ... & 1 \end{bmatrix}$$

Finally, the radiosities matrix is the solution of the following system:

$$\begin{bmatrix} 1-\Phi_1\rho_{11} & -\Phi_1\rho_{12} & ... & -\Phi_1\rho_{1n} \\ -\Phi_2\rho_{21} & 1-\Phi_2\rho_{22} & ... & -\Phi_2\rho_{2n} \\ ... & ... & ... & ... \\ -\Phi_n\rho_{n1} & -\Phi_n\rho_{n2} & ... & 1-\Phi_n\rho_{nn} \end{bmatrix} \begin{bmatrix} J_1 \\ J_2 \\ ... \\ J_n \end{bmatrix} = \begin{bmatrix} \frac{\rho_1}{\Phi_1} E_1 \\ \frac{\rho_2}{\Phi_2} E_2 \\ ... \\ \frac{\rho_n}{\Phi_n} E_n \end{bmatrix} \qquad (30)$$

CONCLUSIONS

Equation (30) can be solved by mean of numerical methods such Gauss-Siedel iteration. Gauss-Siedel iteration has the advantage of being absolutely convergent for diagonally dominant systems such as the one of interest here. A matrix *M* is diagonally dominant if for all *i* $\sum_{j=1, j\neq i} |M_{ij}| < |M_{ii}|$. This is the case of the matrix in equation (30) because medium angular radiation coefficient from a flat surface to itself is 0.

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IMPACT ON REGIONAL AQUIFERS OF COAL EXPLOITATION IN OLTENIA REGION

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ABSTRACT

Mining works connected with coal extraction in Oltenia have an important effect on regional aquifers, such regional dacian artesian aquifer.
In the natural regime period the main flow direction in the artesian aquifer, in the zone of the huge quarry from the Oltenia (Rosia de Jiu), was north-south, with a hydraulic gradient of about 1%. As a consequence of a strong drainage in the artesian aquifer a great depression zone appeared (a drawdown about 90 m).
The variations of groundwater level under the conditions of a relatively constant recharge change the relations between regional aquifers and also the chemical characteristics of groundwater.
In many others regions (like Motru, Prunişor, Mihăiţa etc.), the water drainage change the natural regime of aquifers. The effects of are changes of chemical characteristics of groundwater and also on the stability of the slopes.
The paper presents some results of quantitative evaluation of mining works effects on aquifers in Oltenia region. Results are obtained with stochastic and numerical models for local and regional aquifers.

Key words: mining works, coal extraction, water drainage, stochastic model (kriging, conditional simulation), numerical model (finite difference), hydrodynamic and hydrochemical changes, monitoring network

INTRODUCTION

Mining works associated with coal exploitation in Oltenia have many kind of impact on aquifers systems that are important resources for water supply of towns, agriculture and industry:

- disappearance of shallow aquifers in the area of quarries (by excavation of aquifers);
- dewatering of shalow aquifers (above of the lignite bed number five, the deeper exploited in Oltenia's quarries) in the area of quarries
- dewatering of deep aquifers (confined artesian aquifers) in the area of quarries and their neighbourhood (below the coal bed number five).

Environmental impact assessement is realised on the base of a comparative analisys of the parameters of aquifers before the beginning of coal exploitation and after that..

In the areas of coal querries from Oltenia, the most affected aquifer by the dewatering system is the artesian aquifer situated below the coal bed number five.

The artesian confined aquifer system situated below the coal bed number five have the next main characteristics:

- the most developed pliocene aquifer in the Oltenia region, with a very good continuity;
- the superior pontian and dacian deposits of artesian aquifer are represented by:
  - o fine, medium and coarse grained sands at the bottom of the aquifer
  - o medium sand, fine sand, silt and clayey sands at the top of the aquifer
- a complex structure in the west side of Oltenia (between Danube River and Jiu River: ten or twelve permeable layers of sand separated by impermeable or semipermeable layers of clay) that became a simple one with no more than one or two layer at the east of the Jiu River.

The artesian-confined aquifer connects all effects of coal exploitation on all shallow aquifer systems that are in hydraulic communication with this regional aquifer. This aquifer can be the main physical support for a *numerical model*, used as a principal tool for development of environmental politics in Oltenia region, a region with an accelerated mining activity.

Specific aims of environmental assessement of lignite exploitation on aquifer systems from mining Oltenia region are:

- assessement of hydrodynamic regimen of aquifer systems;
- monitoring and prognosis of hydrodynamic regimen in the quarries for lignite exploitation;
- monitoring and prognosis of hydrochimic regimen in the quarries for lignite exploitation;
- identification of appropriate tools for removal of the negative effects of lignite exploitation on environment (for long life of "clean" aquifers).

ENVIRONMENTAL ASSESSEMENT METHOD

Environmental assessement of impact of lignite exploitation on aquifer systems from mining Oltenia region is realised on the basis of: selection, validation and storage of data, mathematical modelisation of aquifer system and assessement of mining impact on aquifer by numerical simulation.

The main criteria of *data selection* was the objectives of the project and database include:

- natural hydrodynamic regimen of the artesian aquifer system,;
- hydrodynamic regimen of the aquifer for the entire period of exploitation;
- chemical caracteristics of groundwater

Validation of data was realized by processing of experimental data and new hydrodynamic tests in some area like Motru (***Fig.3***) and Rosia de Jiu. All information are separated in two categories: hydrogeological parameters of aquifers and technical characteristics of exploitation process. National Company of Lignite Oltena (NCLO) is the beneficiary of the database realized.

Mathematical model have two components:

- the first component is for *stochastic analysis* of time series and spatial distribution of hydrogeological parameters;
- the second one for numerical simulation of flow in the artesian aquifer

Stochastic analysis of time evolution of piezometric head was used for identification of correelations between flow of dewatering system, dynamic resources an recharge of aquifer. Because of irregular measurement program, the stochastic analysis was necessary for identification of general trend of piezometric head. The spectral analysis and Markov chains was used to find the stuctures of time series: periodical character, time-correlations etc.

For a complex analysis of spatial distribution of hydrogeological parameters of numerical model was used kriging method [4]. The fictiv point method was used for improuvement of monitoring network of aquifer systems.

Numerical simulation of artesian aquifer is realized on the basis of a finite difference model. We use the numerical model for simulation of dewatering system impact on aquifers for different scenario. Conditional simulation was used to evaluate the degree of uncertainty of the results of mathematical simulation.

DATABASE

Geological investigation of lignite deposit was realized in Oltenia by a huge number of wells (***Fig.1***). Hydrogeological exploration is not so detailed like the geological one. The main characteristics of monitoring network for aquifer systems:

- the monitoring network is developed especially in the area of quarries;

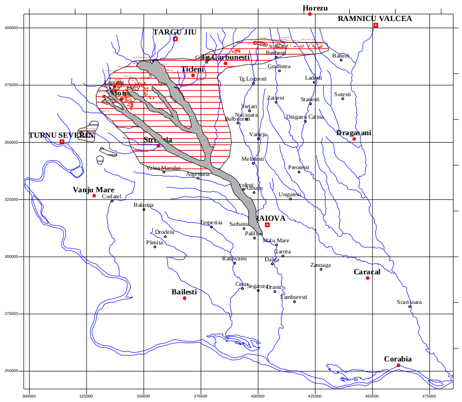

Figure 2 .Zona de impact a exploatării cărbunelui asupra acviferelor

- too few observation points between areas with mining works;
- an irregular program for measurements in the observation points
- not enough number of chemical analysis for groundwater

The achievement of a monitoring network for a realistic evaluation of impact enforces a *new strategy.* The main objectives of this new strategy for monitoring of aquifer systems are:

- a new permanent monitoring network for piezometric head of aquifers with a uniform spatial distribution on entire Oltenia region (not only in the area of mining works!);
- evaluation of an *optimal frequency* of measurement for piezometric head and water sampling for chemical analysis;
- monitoring of dewatering system (geometry of the drainage net, flow of drainage net etc.)

This objectives of this new monitoring network hint at protection of aquifers against the dewatering effects in the area of quarries for lignite exploitation.

The permanent network will use the existing water wells supply from mining areas and other piezometers situated outside the mining areas, till the boundary of dewatering influence.

At the moment, in the mining areas NCLO have 60 water wells supply. The spatial distribution of this wells is not adequate for a strict delimitation of influence area of dewatering system for quarries and mining works.

For the estimation of the impact on the artesian aquifer was used data measured between 1970 and 2001, in different mining areas (Motru, Prunisor, Rosia de Jiu etc.)

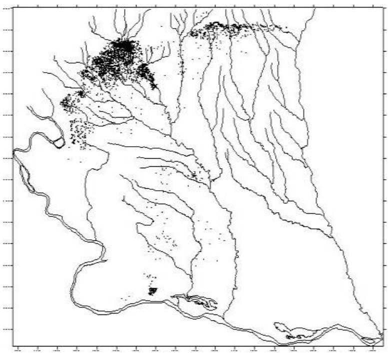

Figure 1. Foraje geologice de explorare pentru zăcămintele de lignit din Oltenia

REZULTS

There are different effects of lignite exploitation on aquifer systems from Oltenia region (***Fig.2***):

- *total or partial disappearance of aquifers be cause of:*
  - the excavation works;
  - the execution of drainage trench;
  - induced leakage
  - drainage and dewatering in the areas of the quarries
- *local and regional reduction of groundwater resources settled by:*
  - drawdown of piezometric surface in the areas of quarries;
  - change of hydraulic gradient;
  - decreasing or dissapearance of elastic store of confined aquifers;
  - increasing of deficit of water balance for regional aquifers
  - pollution of groundwater in some areas because of lekage and dewatering system
  - deterioration of water resources because of intensive exploitation
- the emptiness of groundwater resources by:
  - complete destruction of permanent water resources of aquifer;
  - intensive leakage of aquifers and exploitation for water supply; lock
- *decreasing of shallow aquifer recharge by*:
  - closure of pore, especially from shallow aquifer, with dust of lignite
- *destruction of domestic supply well by* :
  - total drainage of groundwater resource
  - disappearance of shallow aquifers

In this first stage, the assessment of coal exploitation impact on aquifers from Oltenia region consider only the drawdowns induced by dewatering systems of quarries and tapping of groundwater from mining areas.

For the analysis of piezometric head variation, from existing documentation, was selected 197 drainage wells from dewatering system of Rosia's quarry. In the analysed period, October 1979-December 2000, was realised a number of 8600 measurements of piezometric head.

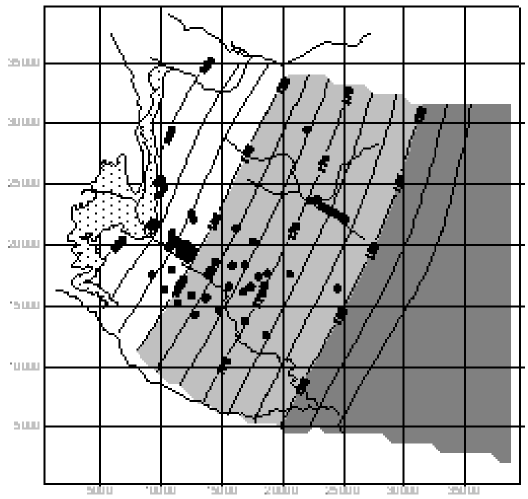

Figure 3. Natural regimen of flow in the aquifer situated below the lignite bed number one (area E.M.Motru)

The evolution of piezometric head in artesian aquifer (below the five lignite bed) in the five drainage lines of Rosia quarry indicate a variation of drawdown of piezometric head between 50 m and 75 m, with a stationary character at the beginning of studied period.

Natural hydraulic gradient of the flow in the area of Rosia quarry has not any influence on stationary character of piezometric head evolution for a long period of time.

For the area of Rosia quarry, on the basis of the hydrogeological parameters of artesian aquifer, was realized a hydrodynamic finite difference model for the simulation of the dewatering system effect. The drawdowns induced by the dewatering system for the entire period of dewatering have values between 80 and 95 meters.

Another hydrodynamic model was elaborated for the aquifer situated below the lignite bed number one, in the area of mining works Motru [1]. In this area the tapping of groundwater for the Motru town have an important impact on the aquifer. Natural regimen of groundwater flow (***Fig.3***) is influenced on some areas around the water wells supply (water catchings: Matasari, V.Manastirii-Lupoaia, CET.Motru, Lupoaia; ***Fig.4***). The drawdown induced by the tappings of water has values between 15 and 20 meters.

In a second stage, these models and others, elaborated for different areas like Prunişor (Palcu,M., 1985), Albeni (Scrădeanu D., 1990), was joining in a single model. This new hydrodynamic model will be used for the prognosis of the effect of lignite exploitation in the future and for the recovery of the piezometric surface after the closer of lignite exploitation in Oltenia region (Palcu, M., 2003).

Quality of groundwater was affected by lignite exploitation. In some areas was identified high concentration of ammonia, organic matter, nitrite, and nitrate (M.Palcu, 2002). Because of poor quality of chemical data, now is impossible to elaborate a representative hydrochemical model for the artesian aquifer system.

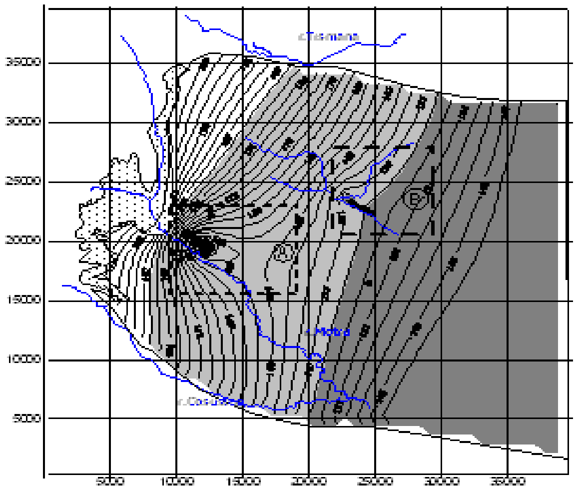

Figure 4. Regimen of flow in the aquifer situated below the lignite bed number one influenced by tappings or water (area E.M.Motru)

Pollution sources are represented by *agriculture activities* in the plane area and overflow meadows and *mining works* in the areas of quarries. Due to the incomplete burning of coal, the redox reactions and the others activities connected with lignite exploitation are introduced in the aquifers: carbon gaz and ammonia, oils from deposits and another contaminants from the waste dump.

CONCLUSIONS

Preliminary analysis of results measurements and simulation with hydrodynamic models make possible the following conclusions:

- impact of lignite exploitation on regional aquifer systems is important in the neighbourhood of coal quarries (drawdown of 95 m in the area of Rosia de Jiu quarry, 20 m in the areas of tapping of groundwater of Motru town);
- dynamic resources of regional aquifers from Oltenia region are huge so the time recovery of piezometric surface is very short, after the end of dewatering activities (only in the case of artesian aquifer and not for the destroied shallow aquifers)
- natural quality of groundwater is affected because of the new hydrodynamic relations between aquifers, induced by dewatering systems of quarries and mining works.

For the big proportion of mining works and dewatering systems connected with the coal exploitation in the Oltenia region, the impact on artesian aquifer is not so important be cause of his huge dynamic resource. Is possible a spectacular recovery of this aquifer in a short time after the end of exploitation of deep lignite beds in the west part of Rosia de Jiu quarry.

Although at regional scale, the recovery of artesian aquifer is quite rapidly, the unpleasant effects of coal exploitation at local scale are important for water supply of villages, landslides in the area of roads and railways. This damages can be resolved with low costs if will be realised a monitoring network for environmental parameters. We already have been proposed a monitoring network on the basis of our hydrodynamic model, with the following main objectives:

- the degree of fragmentation of shallow and confined aquifers, the main cause of reduction of groundwater resources and pollution of groundwater.
- the quality of groundwater in the water wells and the tapping of groundwater, affected by the changes of hydrodynamic relations between aquifers;
- the piezometric head of the regional and local aquifers, the main signal of dewatering systems influence on aquifer;
- groundwater level in the sterile dump situated inside and outside the quarries, an important cause of the landslides
- Subsidence of the land induced by dewatering and underground mining works, also an important cause of the landslide.

The data obtained from the monitoring network are used to make actual the hydrodynamic and hydrochemical models of aquifer systems for the prognosis of impact of coal exploitation on environment in Oltenia region.

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