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ANDREY V. STROMONOGOV (*National University of Science and Technology MISiS*)
VALERY I. SUPRUN (*National University of Science and Technology MISiS*)
YURY G. AGAFONOV (*National University of Science and Technology MISiS*)

TECHNOLOGY OF BLOCK STONE EXTRACTION IN COMPLEX STRUCTURED CARBONATE DEPOSITS

It was established that the carbonate deposits of the Russian Platform are mainly semi-consolidated massifs with a complex spatial network of vertical endogenetic cracks and bedding, expressed in the separation of limestone formations into strata and intermediate layers. Together, they determine the block structure of the massif. The geological structure is dominated by deposits (about 85 %) of small-stone blocks from 0.2 to 0.9 m^3 with a 15 %–25 % yield of commercial-grade stone blocks. All this, together with the increased demand for white stone for construction and restoration, places increased demands on the quality of blocks extracted and supplied for stone-working, which cannot be achieved using conventional blasting techniques of pre-excavation of natural massifs.

A promising blast-free technology for extracting limestone and dolomite blocks that improves their quality is proposed. The technology, based on the use of a hydraulic excavator and a bar cutting machine, takes into account, as much as possible, the natural features of the complex structured carbonate deposits. It was established that for the majority of carbonate deposits of the Russian Platform limestone block mining is possible as a "by-product extraction" with the integrated use of all the productive strata of carbonates. To implement this technology, it is proposed to create a mine site for block stone production integrated into the working area of a large quarry which mines the carbonate massif, with the formation of several goods flows of different types of raw materials. Technological solutions for screening block excavation sites from the working area of the main quarry were considered.

Keywords: limestone blocks, jointing, bedding, blast-free technology, hydraulic excavator, bar cutting machine.

Limestone and dolomites of the Russian Platform occupy an important position in the building materials sector. This is primarily due to the long history of their use in the construction of white-stone buildings that first appeared in Vladimir-Suzdal Russia at the turn of 15th-16th centuries [1–5].

Today, white rock is once more in demand. The ecological purity, historical value, and low cost of limestone and dolomite give them an unbeatable advantage compared to marble and granite from domestic and foreign deposits. The use of inexpensive local materials is increasingly attractive not only for the restoration of religious and historical buildings, but also for new construction [6–9].

Carbonate deposits of the Russian Platform are mainly semi-consolidated massifs with a complex spatial network of vertical endogenetic cracks, formed as a result deposit lithification, and bedding, expressed in the separation of limestone formations into layers (strata and intermediate layers). Together, they determine the size of geological structures or block structure of the massif. The geological structure of the Russian Platform deposits is dominated (about 85 %) by small-stone blocks from 0.2 to 0.9 m^3 with a 15–25 % yield of commercial-grade stone blocks.

Therefore, despite the fact that limestone is a widespread mineral within the Russian Platform, manifestations and areas of high-quality limestone blocks are rare natural anomalies. There are only two limestone deposits on the balance sheet which fall into the "Natural decorative stones" category — Korobcheevskoe in Moscow Region and Molokovskoe in Tver Region [10–15].

The main suppliers of white stone are quarries which use conventional blasting techniques for pre-excavation of natural massifs. As a result, stone processing companies receive so-called boulder blocks with explosive-induced micro-jointing and irregular geometric shape. The use of such materials reduces the product yield, worsens its quality characteristics, and, as a result, it makes the limestone stone-working process ineffective.

This confirms that the justification for advanced block stone extraction technologies, taking into account as much as possible the natural characteristics of the carbonate massifs, is a very relevant task.

To assess the possibility of creating specialized areas (quarries) for the extraction of block stone, the economic stripping ratio is proposed as a criterion [16]:

$$K_b = \frac{C_p - C_o}{C_s}, \qquad (1)$$

where C_p is the cost of mineral deposit production by underground mining;

C_o is the cost of mineral deposit production by open-pit mining;

C_s is the total cost for 1 m^3 (1 t) of stripping work.

The calculation scheme K_{pr} for exposed reserves of productive strata and the presence of residual quarry workings is given in Fig. 1.

It should be mentioned that we are not talking about placing a border zone between open-cut and underground mining, it is only about the ratio of the overburden formation to productive strata at which open-cut mining becomes efficient.

Considering K_b as an output function of (ρ) product from 1 m^3 of commercial-grade blocks and the output of blocks from massif (b), as well as the value of this type of basic material (M_ρ), after some simple mathematical transformations, we obtain the following:

$$K_b = \frac{\left(\dfrac{M_\rho}{f_\rho} - C_\rho\right)\cdot\rho - C_{m_\rho} - \dfrac{C_m}{b}\cdot f_g}{C_s \cdot f_g}. \qquad (2)$$

From here we can determine the ultimate capacity of formations of exposed carbonate rocks lying over the productive strata:

$$H_s^{pr} = \frac{\left(\dfrac{M_\rho}{f_\rho} - C_\rho\right)\cdot\rho - \dfrac{C_m}{b}\cdot f_g}{C_s \cdot f_g \cdot H_c}, \qquad (3)$$

where H_c is the productive layer thickness, m;

C_{m_ρ} is the cost of delivery of block stone to the stone-processing company, RUB/m^3;

C_ρ is the cost of cutting out 1 m^2 of product from the stone block, RUB/m^2;

ρ is the commercial-grade product output from the "raw" stone block, m^2/m^3;

M_ρ is the market price of the finished product made from natural stone, RUB/m^2;

b is the output of commercial-grade stone block from the massif;

C_s is the total development cost per 1 m^3 of overburden rock, RUB/m^3;

f_g is the quarry profitability ratio;

C_m is the cost of excavating 1 m^3 of stone block, RUB/m^3;

f_ρ is the profitability ratio of the stone processing company.

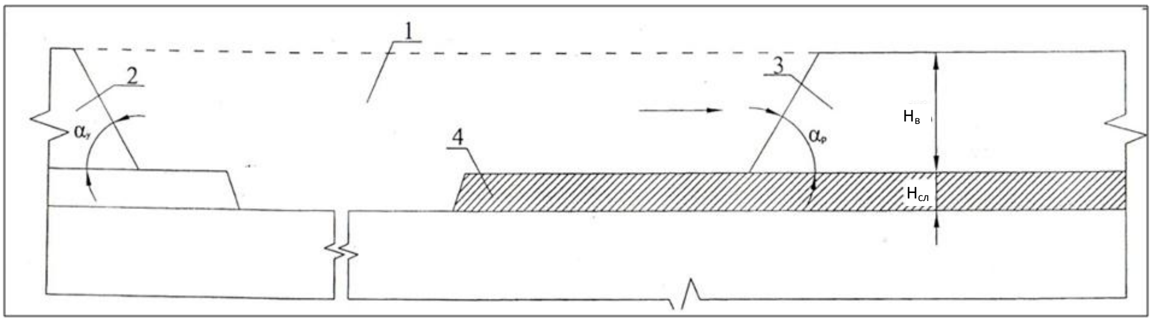

Fig. 1. The calculation scheme for the maximum stripping ratio under the conditions of exposed reserves of productive strata and the presence of residual quarry workings:
1 – exposed quarry space; *2* – fixed border; *3* – stripping bench; *4* – productive layer.

The main variables that affect H_s^{pr} are the output of slab (20-mm thick) from 1 m^3 (ρ), market price of the product (M_ρ), and output of blocks from the productive layer (b).

For limestones and dolomites of the Russian platform, the abovefigures vary in the following ranges: $\rho = 7...23$ m^2/m^3; $M_\rho = 44\ldots57$ c.u./m^2; $b = 0{,}1...0{,}5$.

For these variables, the calculated value of H_s^{pr} ranges from 3 to 7 m. With the average capacity of the productive stratum of 0.5 ... 0.6 m, this corresponds to an economic stripping ratio of 6 to 14 m^3/m^3.

If the actual thickness of stripped rocks lying over the productive stratum is higher than the value calculated by the formula (3), stone block can be extracted using by-product excavation where the overlying formation of carbonate is not taken out to the dump, but processed to obtain various kinds of commercial-grade products (gravel, slaked lime, crushed stone, cement raw materials, calcareous stone for the glass industry, lime flour, etc.). The profitability of processing host rock can be low. At this point it is important to decrease the C_s value.

Once again, it should be noted that the obtained H_s^{pr} values that determine the area and the possibility of creating independent sites (quarries) are valid when exposed geologic outcrops are present which were formed by quarries previously excavated in order to develop this deposit. When it comes to new deposits, initial capital costs associated with their development are an additional factor that seriously complicates the creation of an independent quarry in the deposit.

Analysis of the geological occurrence mode of productive limestone strata in the deposits of the Russian Platform allows us to conclude that limestone block development is only possible for the vast majority of carbonate deposits using "by-product excavation" with the integrated use of all productive strata of carbonates.

This allows us to formulate the following basic principles of block stone excavation for most carbonate deposits of the Russian Platform:

– the block stone mining site should be integrated into the working area of a large quarry, concentrated on carbonate massif, with multiple flows formed of different types of raw materials;

– nonconditioned blocks (up to 90% of extracted volume) from the block stone mining site should be efficiently processed into other types of products;

– rocks, enclosing productive strata of limestone, on a dedicated block limestone mining site should be developed using the blast-free method, and the site itself must be shielded from the main working area of the quarry.

When designing blast-free technology for block limestone and dolomite extraction, the natural characteristics of carbonate deposits of the Russian Platform (developed system of endogenetic cracks, stratification, small capacity of the productive strata) should be used as much as possible.

By virtue of these characteristics, as well as due to the fact that the productive horizons are located on the middle and lower hypsometric levels of the developed natural massif, it is highly unadvisable to use diamond-cutting rope [17–19]. Moreover, the majority of quarries use dependent development systems with a fixed sequence of stripping, mining and development operations. In addition, long-term reserves available for stripping are limited.

The geological structure of carbonate deposits of the Russian Platform are best suited for block limestone development technology based on the use of hydraulic excavator and bar cutting machines [20–21].

A process flow diagram of open mining is shown in Fig. 2.

Stripped loose soil and rocks are mined by excavators from the main quarry.

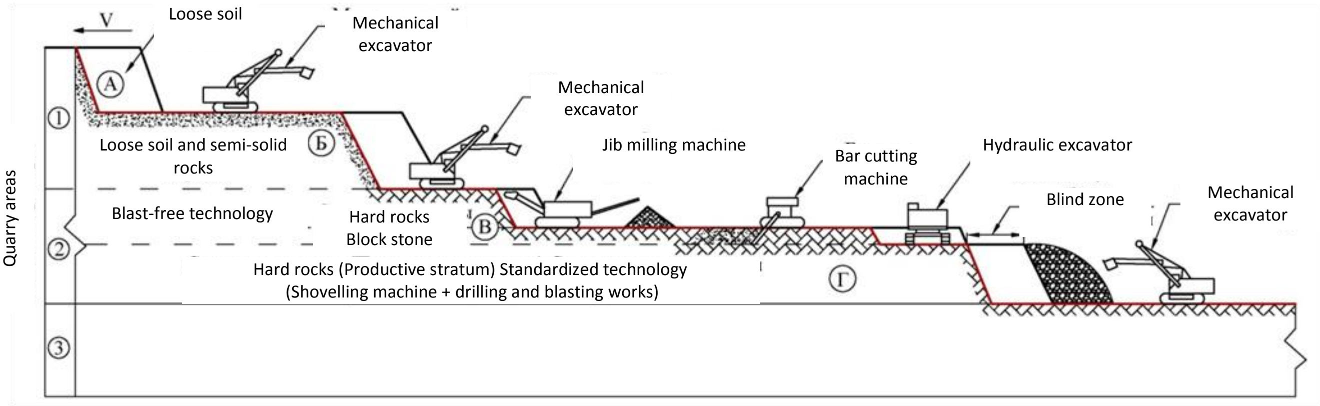

Fig. 2. Process flow diagram of the site allocated to process limestone into blockstone within complex mining of the deposit.

Fig. 3. Bar cuts in the Myachkovskiy stratum of limestone (Afanasyevsky quarry).

Because productive strata must be mined using the blast-free method to preserve their integrity, it is proposed to mine these strata using jib milling machines. Jib milling machines, but not auger milling machines [22] or machines with rotary wheels, should be used when mining rocks on the carbonate deposits of the Russian Platform due to the fact that jib machines do not require a large working face, which is very important for CSM-class machines.

Bar cutting machines produce vertical cuts to a depth equal to the thickness of the stratum. The bar cutter should function in such a way that the cuts are located perpendicular to the direction of the strike of endogenetic cracks (Fig. 3).

This allows rectangular blocks to be made and increases their output. The distance between the bar cuts should result in block sizes that can be loaded onto trucks. When the bar cutter works, cutting and chain vibration lead to weakening of the links between geological structures and emergence of anthropogenic weakness planes, which in turn leads to diminished effort of hydraulic excavator, thereby increasing its performance.

While using the bar cutters to prepare productive strata for extraction, it is important to perform geometrization of the massif, which makes it possible to determine the direction of the strike of the main endogenetic cracks that divide productive stratum into separate geological structures [23, 24]. The bar cutter makes it possible to cut taking into account the regularities of changes in the frequency and direction of endogenetic cracks, thereby increasing the chances that "actual" blocks will be formed.

Cracks in carbonate massifs of the Russian Platform form a complex three-dimensional network (Fig. 4). They arise due to a reduction in the volume of rocks during sediment diagenesis. These cracks are called endogenetic or primary cracks of desiccation or jointing. Such cracks are perpendicular to the stratification and form several systems that, together with stratification, predetermine the separation of the massif into geological structures with certain dimensions (Fig. 5).

Endogenetic cracks do not extend beyond the boundaries of the stratum. They are confined to the stratification planes.

The next process step is the breakdown of productive stratum into geological structures using three systems of weakening planes, namely, along the contours of stratification, vertical endogenetic cracks, and technological planes of weakening. The load should be applied to the individual geological structures from the bottom to the top (from the contact lying on the bottom lamination plane to the top of the productive stratum). Implementation of such an excavation process scheme is most suitable in terms of kinematic and power parameters for the use of front hydraulic shovels with breakout force of 17 … 27 t·H [25, 26].

The massif is excavated by the front hydraulic shovel by embedding the bucket into the cracks between the separate geological structures (along surface of the plane) and by breaking the blocks. The excavator operator sees the face, chooses the place of weakening, and breaks down the massif. For a backhoe excavator, the operator does not see the surface of the face and contact areas of rocks between different formations.

The experience of breaking carbonate rock into separate geological structures shows that the mechanical strength of the separate structures (it can even reach 110 ... 150 MPa) has little effect on the breaking efficiency. The most important factor here is the cohesion force between the extracted block and the underlying formation, and the separating forces occurring along the natural cracks that form contours of geological separate structures (blocks). The digging force for such rocks should be very high, since loading productivity is of secondary importance.

One of the major components of the proposed block stone extraction technology is shielding block extraction sites from the quarry's working area where mining operations are performed using standard blasting methods of massif pre-mining. Screening should be carried out along the three profile lines (Fig. 6). Two of them – AB and CD – cut off the block stone extraction site (F) from the working area at a height of the entire zone of rocks. The lower profile line AC AC cuts off section F from the group of lower quarry horizons.

An alternative option may be considered when, instead of shielding along $BACD$ contour, a section is temporarily preserved from mining operations (Fig. 6). However, this option may not be regarded as economically advantageous, because it assumes preservation of mining operations in the 40 ... 50 m band, which increases capital and operating expenditures incurred for creating a specialized area for block stone extraction. However, it can be used when the quarry has reserves that allow, without damaging the primary production, land stretching 250 ... 300 meters to be allocated on the flanks or in the center of the working area for blast-free block stone extraction.

The most effective method of shielding the block extraction sites in the carbonate massifs is presplitting. To be used as a means of shielding, the basic parameters of blasting works should be justified: well diameter, distance between the drill (blast) holes, alignment of the contour blast hole lines, charge design in the hole.

The application of the proposed technological solutions for mining limestone blocks in Afanasyevsky field in Moscow region and Maleevskoe field in Ryazan region showed their high efficiency.

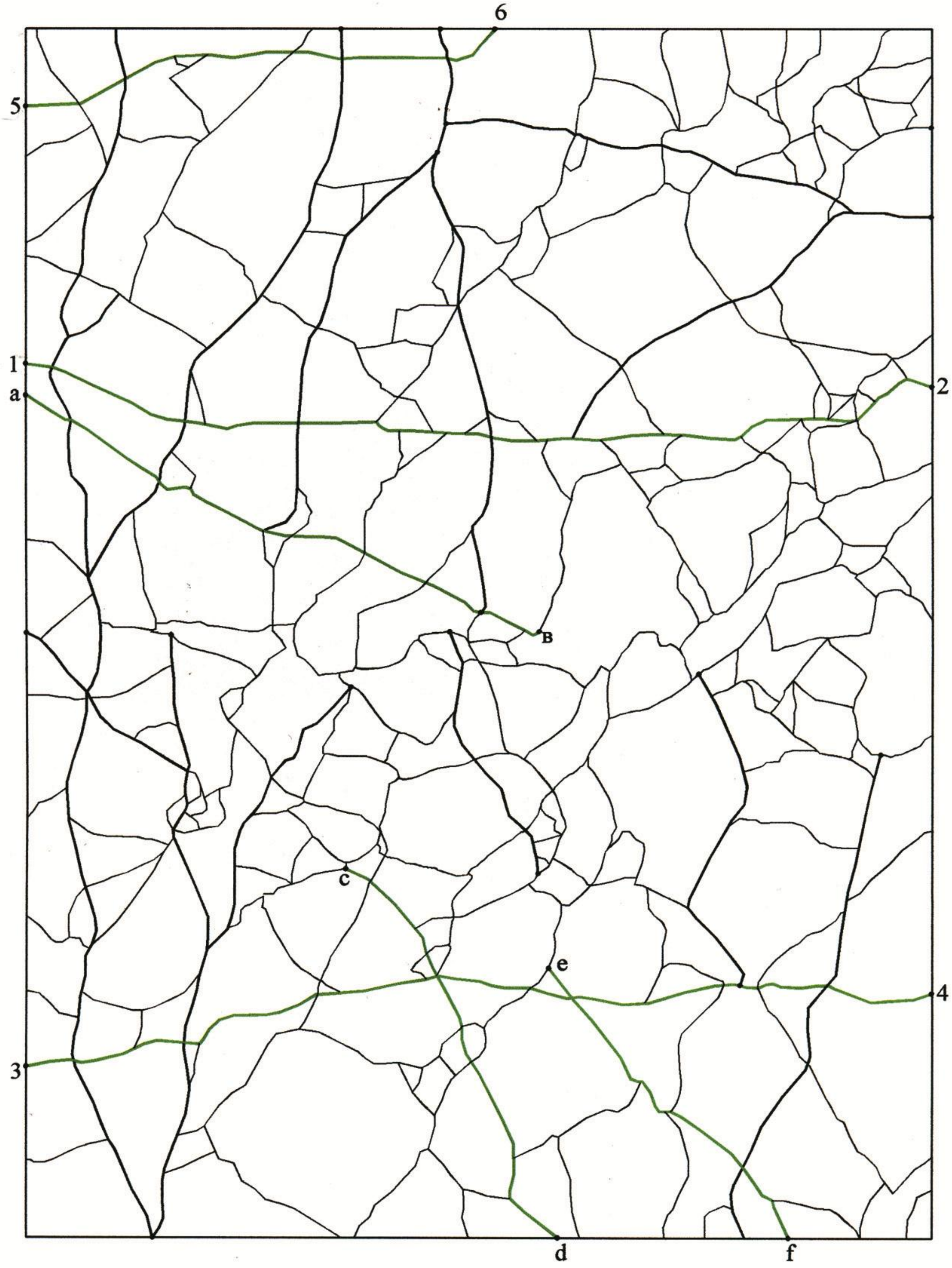

Fig. 4. Propagation pattern of endogenetic vertical cracks (cut along the top of carbonate massif stratum):
1–2, *3–4*, *5–6* – primary system cracks;
a–b, *c–d*, *e–f* – secondary system cracks.

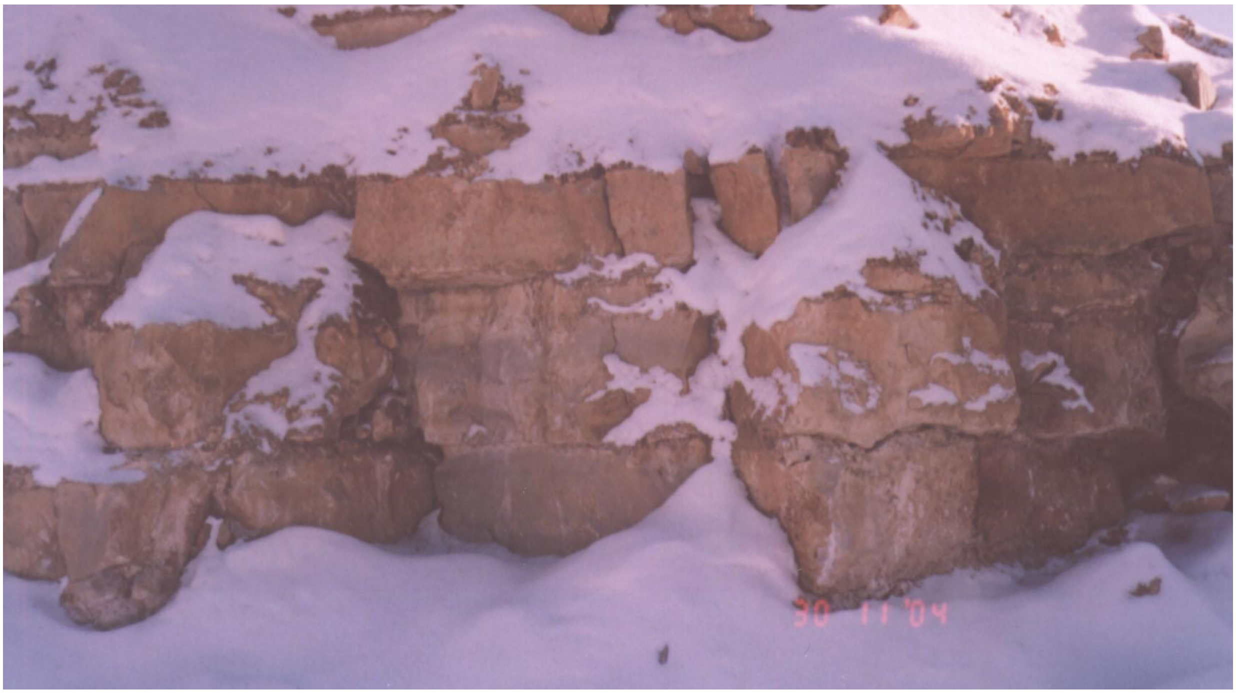

Fig. 5. Productive strata broken by vertical endogenetic cracks into separate geological structures.

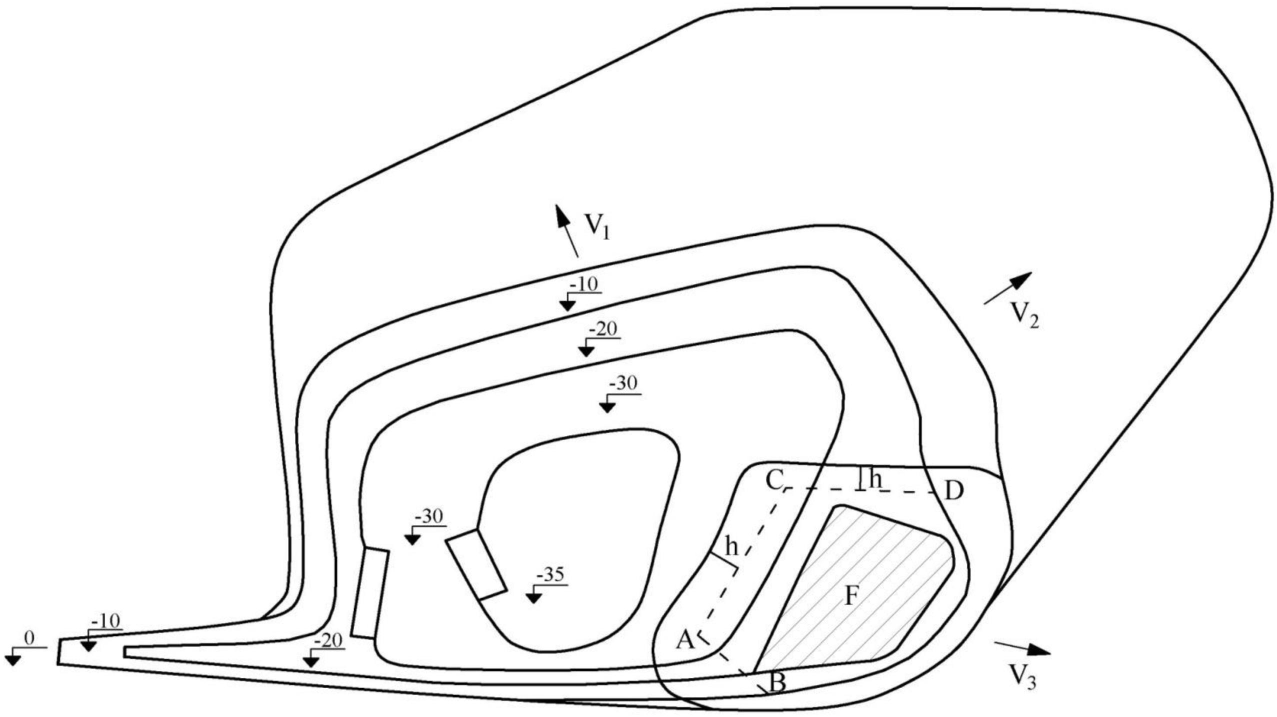

Fig. 6. Shielding pattern for the block stone extraction site from the working area of quarry.

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Title: **Technology of block stone extraction in complex structured carbonate deposits**

Author 1
Full name: **Andrey V. Stromonogov**
Company: **The National University of Science and Technology MISiS**
Work Position: **Leading Engineer of Design-Expert Center**
Contacts: **labstone@mail.ru**

Author 2
Full name: **Valery I. Suprun**
Company: **The National University of Science and Technology MISiS**
Work Position: **Head of Design-Expert Center**
Scientific Degree: **Doctor of Engineering Science**
Contacts: **labstone@mail.ru**

Author 3
Full name: **Yury G. Agafonov**
Company: **The National University of Science and Technology MISiS**
Work Position: **Deputy Head of the Design-Expert Center**
Scientific Degree: **Candidate of Engineering Science**
Contacts: **granat-agafonov@mail.ru**

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Abstract:

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SERGEY I. PROTASOV (*Innovation Firm KUZBASS-NIIOGR*)
DANIIL A. POKLONOV (*Gorbachev Kuzbass State Technical University*)

EXPERIMENTAL RESEARCH AND FEASIBILITY STUDY FOR THE USE OF GD-300 HYDROMONITORS IN KUZBASS MINES

The paper presents the experimental results for the new GD-300 hydromonitor used to determine the analytical dependence between the generalized hydromonitor resistance factor and nozzle diameters, and the algorithm for determining hydromonitor nozzle diameters subject to the operating conditions at the pump station. New hydromonitors improve hydraulic mining performance and efficiency.

Keywords: Kuzbass mines, hydraulic mining, hydromonitor testing.

INTRODUCTION

According to analysis, the amount of overburden rocks mined by hydraulic methods (hydraulic mining) in Kuzbass coal quarries has decreased in recent years [6, 7, 8]. One of the main reasons for this appears to be the fact that rocks become more complex for hydraulic mining equipment and require high-pressure water jets to break them [9]. New field blocks to be commissioned in Kuzbass in the coming years, including Sartakinsky-2 Block, are overlaid by thick Quaternary deposits, which makes it difficult to remove them using heavy-duty excavation equipment. Therefore, hydraulic mining technology needs to be developed further. Local experience shows that in such cases hydraulic mining offers substantial economic benefits [10–15].

The analysis [8] shows that GMD-250M hydromonitors with 100 and 110 mm nozzles are the most common type currently used in Kuzbass mines. These hydromonitors are able to supply up to 2,000 m^3 of water per hour for rock excavation. However, one such hydromonitor cannot cope with the regular seasonal scope of overburden removal operations. This is why two or more hydromonitors operating in parallel or several separate hydromonitor pump units are used for rock excavation. This results in an increased number of operating personnel and, therefore, impacts on the cost of mining operations.

Tests

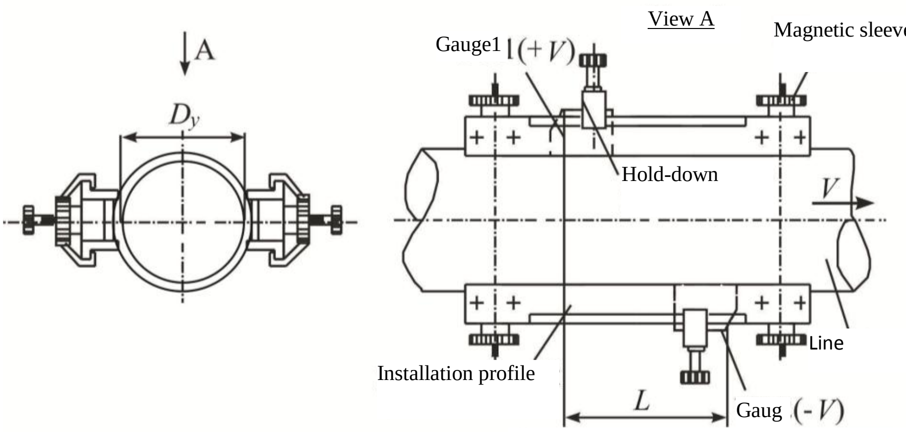
Coal Company Kuzbassrazrezugol OJSC ordered the development and manufacture of GD-300 hydromonitors (Zavod Gidromash OJSC, Novokuznetsk) and T-521 hydromonitors (Yurginsky mashzavod LLC, Yurga). Test programs were developed and test procedures approved by the Federal Environmental, Industrial and Nuclear Supervision Service (Rostechnadzor) for two GD-300 hydromonitors on site at Kedrovsky mine (Fig. 1). At the same time, the T-521 hydromonitor was tested in Taldinsky mine.

Fig. 1. GD-300 hydromonitor made by Zavod Gidromash OJSC at Kedrovsky mine.

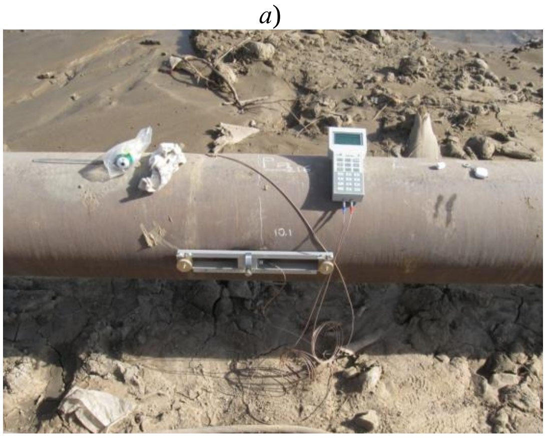
In both models, flow channel areas and nozzle diameters are larger than in the GMD-250M. These models use 125 ... 175 mm nozzles to deliver a flow rate up to 5,000 m^3/h depending on the head, i.e. more than double the flow rate of GMD-250M hydromonitors. The test results provide compelling evidence that these hydromonitors are much more efficient for rock mining than GMD-250M hydromonitors, which leads to substantial improvements in the technical and economic performance of hydromonitor dredging equipment. Since the hydromonitor meets all regulatory requirements, Rostechnadzor authorized its application for open-pit mining.

To demonstrate that these hydromonitors may be used to enhance the efficiency of overburden removal operations using hydraulic mining methods, various studies have been conducted to identify hydromonitor performance characteristics that allow for the optimization of hydromonitor dredging design parameters to reduce overburden removal costs.

The main hydromonitor operation parameters are known to be inlet head (or head upstream of hydromonitor nozzle) and flow rate, which determine the performance conditions for the hydromonitor and are interdependent. When the head at the hydromonitor inlet changes, the flow rate also changes. Each hydromonitor can have an infinite number of performance conditions depending on the head at its inlet. A combination of all possible hydromonitor performance conditions, i.e. numerical values of hydromonitor heads and flow rates in a specific operating environment (with a certain pump station which creates water head at hydromonitor inlet), shows a specific or actual performance of this particular hydromonitor. It should also be borne in mind that there is an optimum head for each nozzle at which the specific flow rate is minimal.



a)



b)

Fig. 2. Installation diagram for the primary instrument, *a*,
and the general view of the flow meter installed on the inlet line, *b*.

The ultimate goal of design calculations for hydromonitor pump units is to identify the actual performance of pumps and hydromonitors. To do so, a head-capacity curve should be available for the hydromonitor in addition to head-capacity curves for pumps and pipelines; the curve is a square parabola that passes through the origin of coordinates [1] and is described by the following dependence

$$H_{hm} = R_{hm}Q^2, \tag{1}$$

where H_{hm} is the total specific energy consumption for water movement in a hydromonitor, which is called its resistance, m; R_{hm} is the generalized hydromonitor resistance factor, s^2/m^5; Q is the flow rate through a hydromonitor, m^3/s.

The experimental research into head-capacity curves of a hydromonitor on site in Kedrovsky Coal Mine, branch of Coal Company Kuzbassrazrezugol OJSC, provided numerical values of generalized resistance factors for GD-300 hydromonitors for different nozzle diameters. Nozzles 118, 125, 140 and 160 mm in diameter were tested. In this case, the generalized resistance factors R_{hm} were 518, 399, 276.5 and 168.5, respectively. Fig. 2 shows the installation diagram for the primary instrument, *a*, and the general view of the flow meter installed on the inlet line, *b*.

The generalized resistance factor R_{hm} describes hydromonitor resistance H_{hm}, m, which is the total specific energy consumption by water moving in hydromonitor channels and is equal to total head losses in the hydromonitor h_{hm}, m, head losses in the nozzle h_n, m, and dynamic head H_d, m, at nozzle outlet, i.e.

$$H_{hm} = h_{hm} + h_n + H_d. \tag{2}$$

Again,

$$h_{hm} = k_h Q^2; \tag{3}$$

$$h_{ps} = \xi_{ps}\left(\frac{V_{ps}^2}{2g}\right); \tag{4}$$

$$H_d = \left(\frac{V_{ps}^2}{2g}\right), \tag{5}$$

where k_h is the head loss coefficient for the hydromonitor; Q is the flow rate through the hydromonitor, m^3/s; ξ_{ps} is the hydraulic resistance coefficient for the nozzle; V_{ps} is the nozzle jet velocity, m/s.

With regard to (3), (4) and (5), dependence (2) changes to

$$H_{hm} = k_h Q^2 + \frac{(\xi_{ps}+1)V_{ps}^2}{2g}.$$

Let us express V_{ps} in equation (6) in terms of flow rate Q, m^3/s, and nozzle diameter d_{ps}, m,

$$V_{ps} = \frac{4Q}{\pi d_{ps}^2}. \tag{7}$$

Taking into account (7), we get

$$H_{hm} = \left(k_h + \frac{0,0827(\xi_{ps}+1)}{d_{ps}^4}\right)Q^2.$$

So, according to dependence (1)

$$R_{hm} = k_h + \frac{0,0827(\xi_{ps}+1)}{d_{ps}^4}.$$

Taking into account the fact that head losses in the hydromonitor described by coefficient k_h are negligible compared to dynamic water head at nozzle outlet, dependence (9) may be reduced to

$$R_{hm} = k \cdot d_{ps}^{-4}, \tag{10}$$

where k is the empirical coefficient.

By processing the experimental data, we found the analytical dependence between the generalized resistance factor for GD-300 hydromonitor and the nozzle diameters [2] (Fig. 3) and the numerical value of the empirical coefficient *k*, which is equal to 0.10, i.e.

$$R_{hm} = 0.10 d_{ps}^{-4}. \tag{11}$$

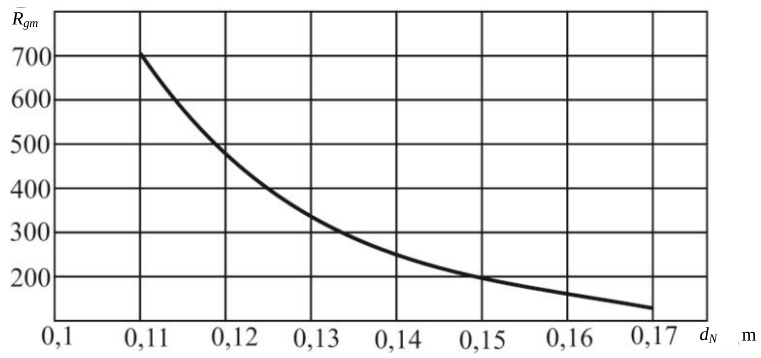


Fig. 3. The analytical dependence between changes in the generalized resistance factor for GD-300 hydromonitor and the nozzle diameters.

The relative error for calculations based on the semi-empirical dependence is 3.82%; the root-mean-square deviation is 10.31%, and the coefficient of variation is only 3.0%, which is quite fair for this type of calculation [3].

After the nozzles are selected, actual performance of the hydromonitor pump equipment is determined and analyzed [4]. If such performance fails to meet certain requirements, the selection should be adjusted by using bigger or smaller nozzle diameters.

If it is impossible to achieve the desired hydromonitor performance by adjusting nozzle diameters, the structure of pump stations should be changed, i.e. other types of pumps should be used, or the number of pumps connected in series or in parallel should be changed [5].

Fig. 4 shows the block flow diagram for selecting hydromonitor nozzle diameters subject to operating conditions at a pump station.

Assessment of Feasibility of New Hydromonitors

Hydraulic mining methods are used for overburden removal in Krasnobrodsky mine in Novosergeyevskoye Field (Razgen Block) located in the northwestern part of the Prokopyevsk-Kiselevsk geological and economic area in Kuzbass. The Krasnobrodskaya formation consists of mild clays and gray sand loams with greenish, bluish, and brownish shades; these are plastic and sometimes oversanded, with ocher-colored spots of ferrum oxides, both horizontally and cross-laminated. The rocks in the formation, especially at the bottom, have a lot of sand and gravel. The total volume of loose deposits within the hydraulic mining block is 78 million m^3. Taking into account the underbreak and the existing piles, the total volume of loose deposits to be removed by hydraulic mining methods is 68 million m^3 (87 %). In terms of development complexity, loose Quaternary deposits are classified as Group V. The overburden thickness in the hydraulic mining block varies from 5 to 25 m. The annual thickness of the hydraulic mining block is 2,000 thousand m^3 of overburden.

The calculations showed that two GMD-250M hydromonitors with 110 mm nozzles should be replaced with one GD-300 hydromonitor with a 160 mm nozzle, taking into account the operating conditions for hydromonitor dredging equipment in Krasnobrodsky mine.

To determine costs benefits, water supply costs were calculated for one hydraulic transport system. For Krasnobrodsky mine, these cost benefits are doubled because the project provides for the operation of two hydraulic transport systems. Based on the above calculations, costs were compared for two options: the operation of two GMD-250M hydromonitors (in service) with 110 mm nozzles, and the operation of one GD-300 hydromonitor (in service) with a 160 mm nozzle. The calculation results are given in Table 1.

After hydromonitor nozzle diameters were selected subject to pump station performance, the value of the actual water head upstream of the

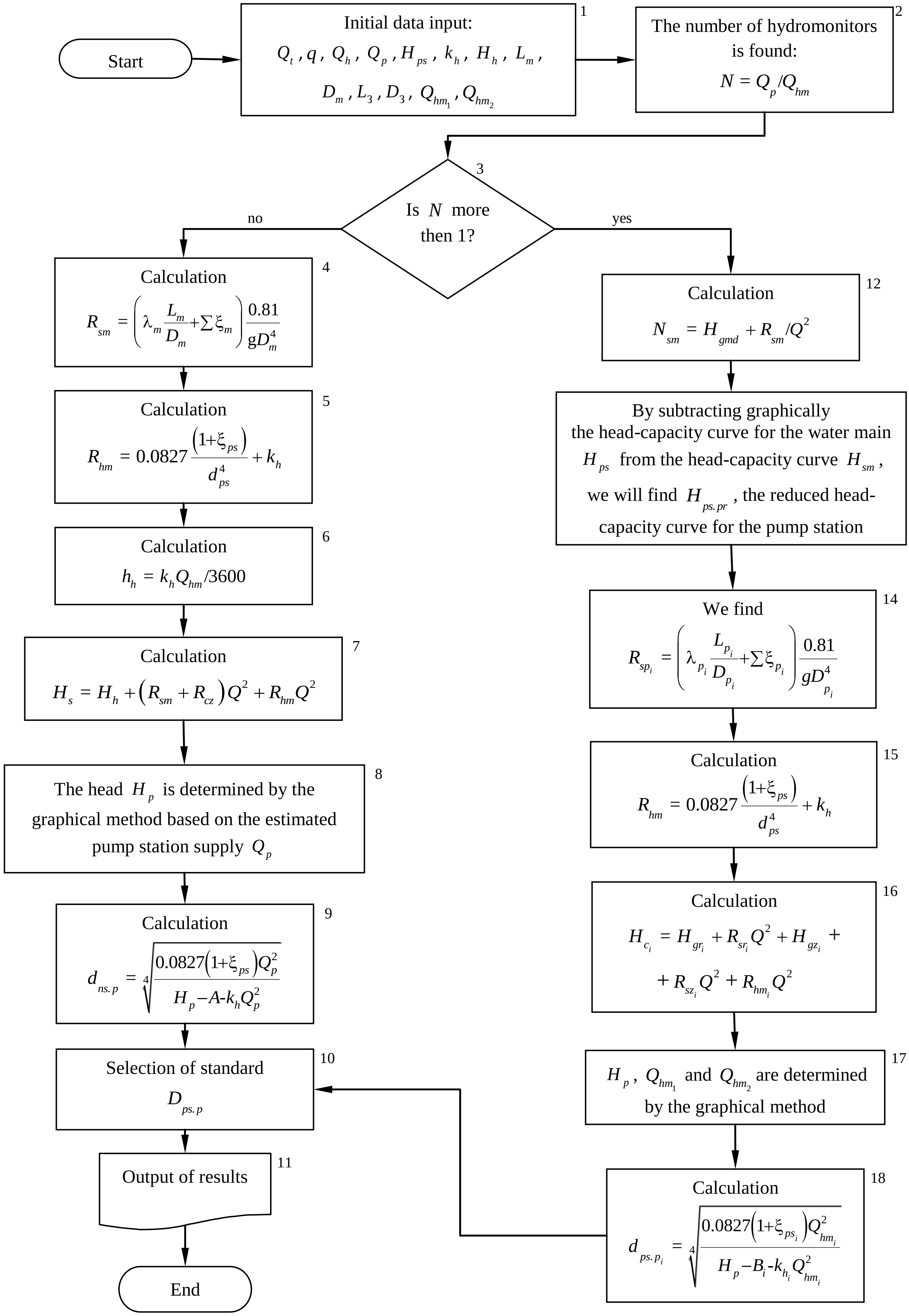


Fig. 4. Block flow diagram for selecting hydromonitor nozzle diameters subject to operating conditions at a pump station

Table 1

Calculation of Cost Benefits

Hydromonitor dredging equipment costs	Current option with two GMD-250M		Proposed option with one GD-300	
	Investments, thousand RUB	Annual operating expenses, thousand RUB	Investments, thousand RUB	Annual operating expenses, thousand RUB
For one GR-4000/71 hydraulic transport system	5,695.23	2,085.855	3,944.14	1,040.81
For Krasnobrodsky mine with two hydraulic transport systems	11,390.46	4,171.71	7,888.28	2,081.62
Cost benefits for Krasnobrodsky mine	–	–	3,502.18	2,090.09

hydromonitor was found: $H_{gmd} = 191.7$ m. Taking into account the head loss coefficient for the hydromonitor ($k_h = 26.6$ for GD-300 hydromonitor), the actual flow rate through the hydromonitor $Q_{gmd} = 3.780$ m^3/h, and the head upstream of each hydromonitor $H_{gmd} = 191.7$ m, we can now find the actual head losses in the hydromonitor $H_h = k_h \left(Q_{gmd}/3600\right)^2 = 2.6\times \times\left(3780/3600\right)^2 = 29.33$ m and the head for breaking $H_h = H_{gmd} - h_h = 191.7 - 29.33 \approx 162$ m.

The research results allow us to validate key mining design parameters when GD-300 hydromonitor is used, as well as block parameters, their number, and the front advance rate of mining operations per season, taking into account the seasonal capacity of 2,070 thousand m^3/year.

Design parameters for hydromonitor dredging equipment with a GD-300 hydromonitor recommended for Krasnobrodsky mine are given in Table 2.

CONCLUSIONS

The research results show that head-capacity curves found by experiment for GD-300 hydromonitors and the known pattern of changes in key parameters of hydromonitor pump units allow for the optimization of hydromonitor dredging designs, which substantially reduces hydraulic stripping costs and improves coal mining economics.

Table 2

Design parameters for hydromonitor dredging equipment with a GD-300 hydromonitor for Krasnobrodsky mine

Design parameter	Symbol and value
Actual head provided by the hydromonitor, mm w.g.	$H_d = 159.7$
Hourly capacity of the hydraulic transport system for hydraulic fluid by solids, m^3/h	$Q_p = 540$
Seasonal capacity of a hydraulic system (block), thousand m^3/year	2,070
Open pit bench height, m	$h = 20$
Hydromonitor/face approach coefficient	$\varepsilon = 0.4$
Safe distance from hydromonitor to face, m	$l_{min} = 8$
Hydromonitor advance increment, m	S = 21
Length of the working section of a hydromonitor jet, m	$L_p = 37$
Hydromonitor pull width, m	$A = 46$
Block width, m	$l = 230$
Length of a block covered by one sump position, m	$L = 460$
Sump depth, m	$h_з = 6$
Sump width and length (min), m	$\alpha \approx b = 5$
Total front length covered by one hydraulic transport system per season, m	$L_f = 900$
Front advance rate in the hydraulic block, m per season	$v_f = 230$

Recommendations for operating equipment provided by the authors ensure a match between the water supply and hydraulic transport system parameters for hydromonitor dredging equipment in Krasnobrodsky mine. The calculation of cost benefits demonstrated that the replacement of GMD-250M hydromonitors by a GD-300 hydromonitor to ensure operation of the hydraulic transport system in Krasnobrodsky mine reduces investments by RUB 3,502.18 thousand and annual operating expenses by RUB 2,090.09 thousand.

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Title:	**Experimental research and feasibility study for the use of GD-300 hydromonitors in Kuzbass mines**
Author 1	Full name: **Sergey I. Protasov** Company: **Innovation Firm KUZBASS-NIIOGR** Work Position: **Professor** Scientific Degree: **Candidate of Engineering Science** Contacts: **protasov@kuzbass-niiogr.ru**
Author 2	Full name: **Daniil A. Poklonov** Company: **TF Gorbachev Kuzbass State Technical University** Scientific Degree: **PhD student** Contacts: **psi.rmpio@kuzstu.ru**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-2-14-23**
Abstract:	The results of experimental studies of the new jetting DG-300, on the basis of which set of generalized analytical dependency jetting resistance ratio of the diameter of nozzles used and developed an algorithm for determining the diameter of the jetting nozzles, taking into account the mode of operation of the pumping station. Application of new jetting improves performance and efficiency jetting.
Keywords:	cuts of Kuzbass, hydromechanization, jetting test
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MAI NGOC LUAN (*Curtin University of Technology, Australia*)
BUI XUAN NAM (*Hanoi University of Mining and Geology, Vietnam*)

APPLICATION OF GEOSTATISTICAL TOOLS TO ASSESS GEOLOGICAL UNCERTAINTY FOR SINQUYEN COPPER MINE, VIETNAM

Geostatistics-based estimators, i.e. ordinary kriging and simple kriging, are state-of-the-art estimation techniques widely used in the mining industry. However, the estimation result of kriging techniques is not able to cope with the well-known uncertainty of geological attributes in mineralization deposits. Moreover, the smoothing effect of kriging algorithms has led to over/under estimation in many circumstances. To overcome these drawbacks, the deterministic estimation result of kriging is usually followed by stochastic results provided by stochastic simulation. In this paper, the application of ordinary kriging and stochastic simulation are performed to build the resource model together with the uncertainty assessment of the Sinquyen copper mine.

Keywords: Geostatistics, ordinary kriging, sequential Gaussian simulation, ore resource estimation

INTRODUCTION

In the modern mining industry, all mining projects are developed based under a block model in a three-dimensional space (hereafter called block model). The objective of block modeling is to intuitively represent the real but unknown geological conditions of the mineralization area. In the geological block model, each of the unit blocks contains geological attributes i.e., grades of different metals and minerals comprising the deposit, and their specific gravity, tonnage, and lithology. Once the geological block model is built, economic parameters such as mining and processing cost and commodity price will be applied to convert that geological block model into an economic block model, which forms the platform for the mine planning process. The source of information for the estimation process mainly comes from a limited number of exploratory boreholes, which is obviously not available at all blocks. The process that interpolates values of unsampled blocks from nearby boreholes is called ore resource estimation. Ore resource estimation is a complicated task in terms of algorithms, the large amount of data and computation cost.

To date, many estimation methods have been developed to tackle this challenge. One of the first approaches is to assign the value of the nearest sample point to an unknown point, called the nearest neighbor. The nearest neighbor methodology is illustrated in Fig. 1.

Fig. 1. Illustration of nearest neighbor method

Inverse distance is a more advanced interpolation technique, which is capable of considering the impact of distance between different sample points to the estimating point. The methodology of this approach based on the assumption that those points have the same distance to the unsampled point will have the same similarity. The similarity will decrease gradually as the distance increases. With this assumption, the weights of different sample points to the estimated point will be different, and only depend on the distance.

Due to the explicit interpolating mechanism, nearest neighbor and inverse distance are classified as deterministic methods. These methods are quite simple and easy to implement in terms of algorithm and computational intensity. The result is acceptable when the variation of dataset is not complicated or when the requirement of estimation quality is not too high, e.g. estimating attribute grades of waste blocks.

Danie Krige (1951), a mining engineer working in a gold mine in South Africa, suggested a new statistical estimation method where the gold content of blocks not only depends on the attribute inside but also the surrounding spatial information. This new idea was subsequently developed as a brand new scientific subject called geostatistics. The family of average estimation algorithms is named kriging, after the pioneering work of Danie Krige. In this approach, the spatial features of the survey area are quantified into a calculating equation, e.g., anisotropy, correlation between different variables, or spatial continuity. Kriging algorithms release estimate results with the smallest estimation variance constraint to an unbiased condition. Among the kriging algorithms, regular kriging is considered the most popular due to its simple application.

Generally speaking, the estimation methods, include kriging algorithms, aim to obtain a single “best” estimation result and they are not capable of accessing the associated uncertainty of the estimation. Dimitrakopoulos et al. (2002) proposed an application of stochastic simulation in resource estimation. Differing from kriging methods, the result of stochastic simulation is a series of probable possibilities (or realizations) that a mineral deposit may have; therefore, this technique can access the uncertainty associated with the resource estimation.

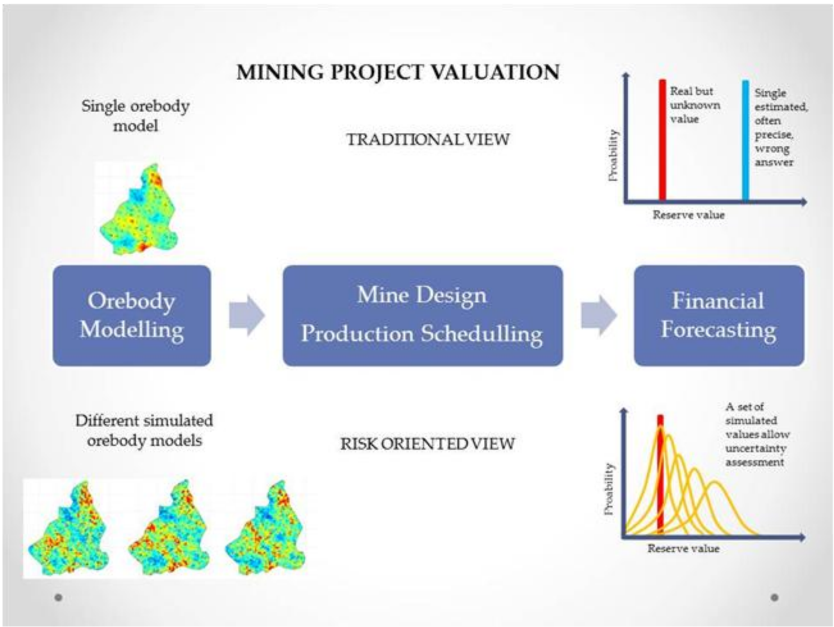

Fig. 2. Illustration of the application of estimation and simulation

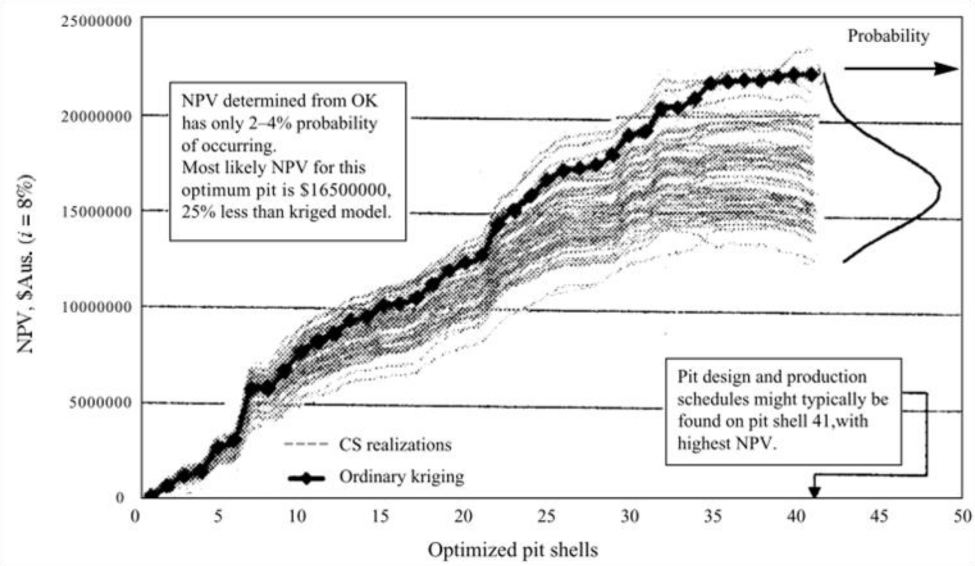

Fig. 3. Illustration of NPV distribution of a mining project

When it comes to mine planning, instead of using a single ore body model to release a unique project NPV as a traditional method, applying stochastic simulation will result in different schemes, from the most optimistic to the most pessimistic a mining project may experience. This new approach, so-called stochastic mine planning, has opened a new dimension for the mining industry. The differences of traditional and stochastic mine planning are illustrated in Fig. 2 and 3.

In this paper, the methodology of ordinary kriging (*OK*) and sequential Gaussian simulation (SGS) are reviewed in Section 2. Section 3 is devoted to a large-scale case study with results analysis and conclusions drawn in Section 4.

METHODOLOGY REVIEW

Ordinary Kriging

The general methodology of kriging interpolators is reviewed from the work of Goovaerts (1997).

Let's define study area *A* with a dataset including n sample points:

$$\{Z(u_\alpha), \alpha = 1, 2, ..., n\},$$

where Z is the continuous attribute that is being surveyed, such as metal content, mineral density, porosity ..., u_α is the location of the corresponding n sample point.

The problem is estimating the value of $Z^*(u)$ at any unsampled point belonging to *A*.

The estimating value at unsampled point u by kriging algorithm is defined as:

$$Z^*(u) - m(u) = \sum_{\alpha=1}^{n(u)} \lambda_\alpha(u)\left[Z(u_\alpha) - m(u_\alpha)\right], \quad (1)$$

where:

$Z^*(u)$ – Estimating value of unsampled point;

$Z(u_\alpha)$ – Value of sample point α;

$\lambda_\alpha(u)$ – Weight of sample point α to estimating unsampled point;

$m(u)$ – Local arithmetic mean;

$m(u_\alpha)$ – Arithmetic mean of study area.

Estimation error is defined as:

$$\sigma_E^2(u) = Var\left\{Z^*(u) - Z(u)\right\}.$$

All kriging methods aim to minimize estimation error $\sigma_E^2(u)$ by setting the unbiased condition:

$E\left\{Z^*(u) - Z(u)\right\} = 0$: Expected value of estimated value and real value (average estimated value) is 0.

According to the equation (1), these

parameters $Z(u_\alpha)$, $m(u_\alpha)$ are constant, so the difference between kriging methods basically concerns the issue of how to define local arithmetic m(u) and weight $\lambda_\alpha(u)$ of sample points.

With ordinary kriging, this method is named “ordinary” because it is based on the assumption that local mean $m(u)$ is constant and cannot be determined, and so can be removed from the equation. This makes the calculation very convenient to apply. This method even has the nickname “BLUE”: Best Linear Unbiased Estimator.

$$Z^*(u) = \sum_{\alpha=1}^{n(u)} \lambda_\alpha^{OK}(u) Z(u_\alpha) + \left[1 - \sum_{\alpha=1}^{n(u)} \lambda_\alpha^{OK}(u)\right] m(u),$$

where

$Z^*(u)$ − Estimating value of unsampled point;

$Z(u_\alpha)$ − Value of sample point α;

$\lambda_\alpha^{OK}(u)$ − Weight of sample point α by *OK* method;

$m(u)$ − Local arithmetic mean.

Local mean $m(u)$ is removed from the equation by the assumption that the sum of weights is 1:

$$\sum_{\alpha=1}^{n(u)} \lambda_\alpha^{OK}(u) = 1.$$

Therefore:

$$Z^*(u) = \sum_{\alpha=1}^{n(u)} \lambda_\alpha^{OK}(u) Z(u_\alpha).$$

Sequential Gaussian simulation (SGS)

SGS is one of the simplest and most common geostatistical simulations which relies on the normal (Gaussian) distribution of input data to have a zero mean and unit variance (Deutsch & Journel, 1998). This feature provides an extremely analytical simplicity for simulation process. Normally, the distribution of most raw datasets on earth is not normal; therefore, a normal transformation needs to be implemented before simulation.

SGS is a sequential simulation approach, which means that the simulation procedure of a node is conditional on all neighboring data inside the search radius, including both original data and previously simulated nodes. A random path will be set to ensure that all nodes are simulated randomly and sequentially.

Steps of SGS:

1) Normal transformation of original data;

2) Draw a random path to visit every node;

3) Withdraw a random value to a node from a cumulative distribution function generated from data within the search radius, assign this value to that node as the simulated value;

4) Repeat step 3 until all nodes, except original nodes, have been simulated;

5) Back transform the simulated values into their original form;

6) Check the result: reproduce and check the histogram and variogram of realizations to target features.

Repeat step 2 to step 6 to generate other realizations.

CASE STUDY

In the case study, the *OK* and SGS were applied to the Sinquyen copper deposit in Vietnam. The block model consists of 802,944 blocks, of which 10,613 are ore blocks with block size of 25×25×25 m.

Geostatistical analysis showed that the deposit has the best spatial continuity direction from North East to South West with the anisotropic ratio of 1.823. The geostatistical analysis results are presented in Fig. 4, 5, 6, and 7.

This variogram model was applied to implement *OK* and 10 realizations of SGS on the Sinquyen copper mine. The grade-tonnage curves are presented in Fig. 8.

It is clear from the grade-tonnage curves that *OK* has underestimated the grade of copper. For instance, at cut-off 1 % Cu, the metal tonnage estimated by *OK* is 47 % while SGS ranges from 68 to 72 %. The average copper grade

Azimuth: α = 45°

Range: a = 41,53

Sill: C = 0,50

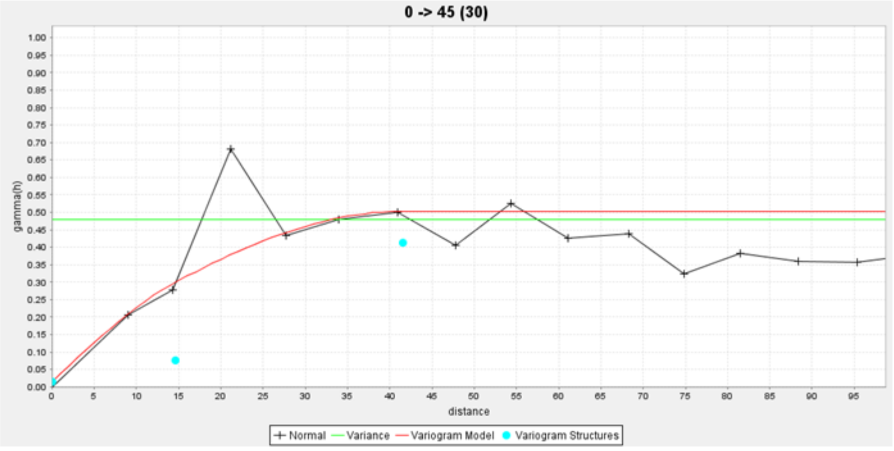

Fig. 4. Variogram model has the best spatial continuity

Azimuth: α = 135°

Range: a = 22,78

Sill: C = 0,50

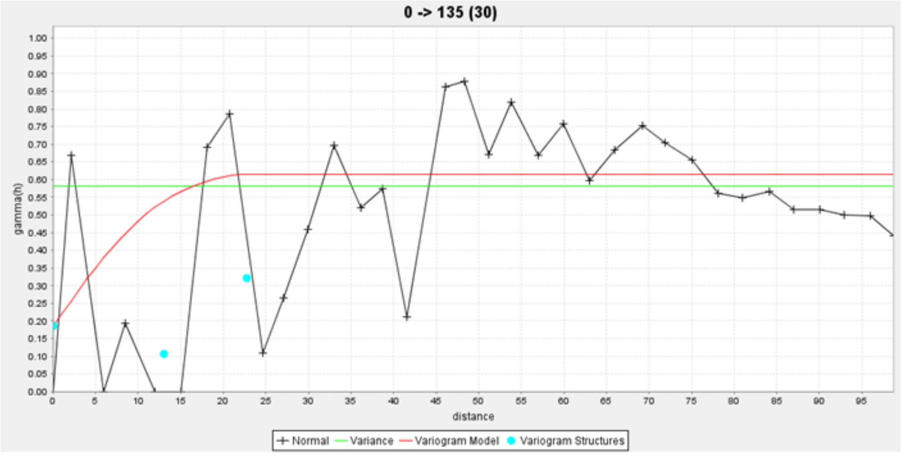

Fig. 5. Variogram model has the least spatial continuity

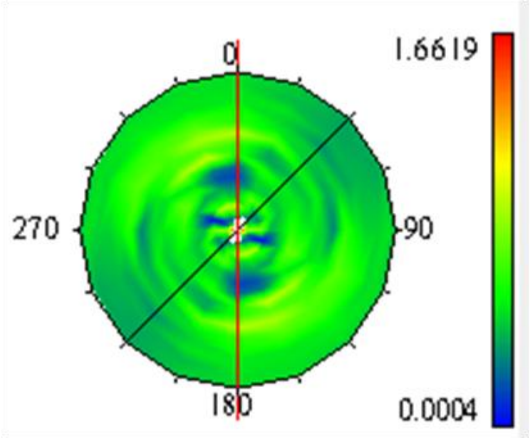

Fig. 6. Variogram map

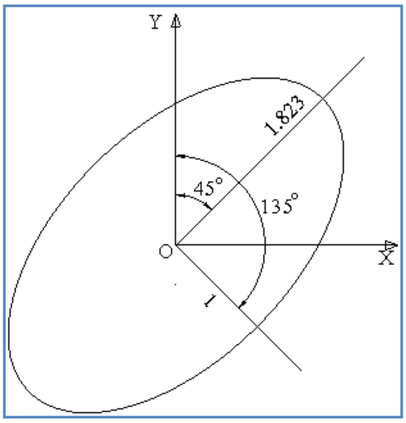

Fig. 7. Anisotropic ellipse

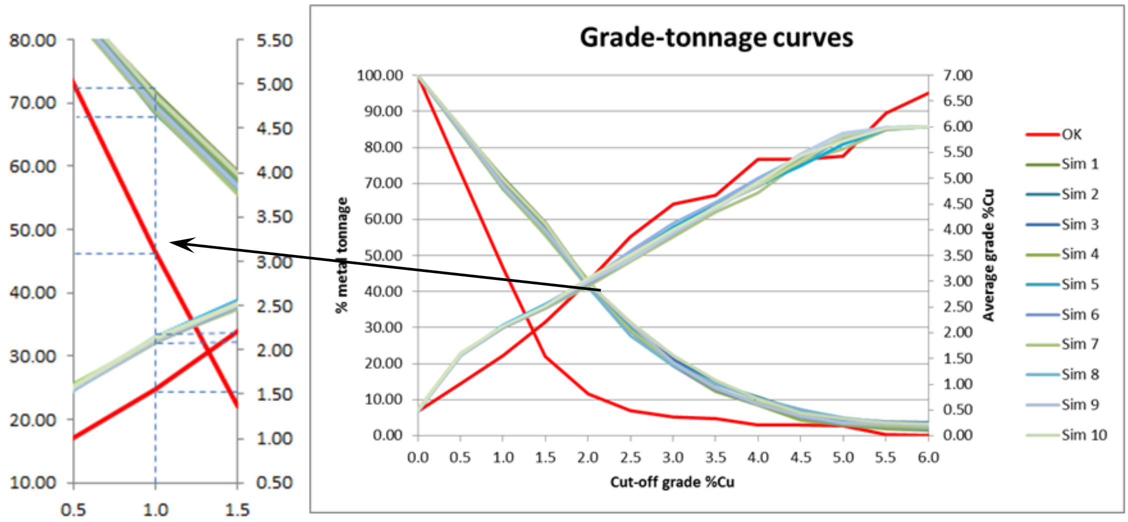

Fig. 8. Grade-tonnage curves of *OK* and SGS.
(*Follow increasing trend of cut-off: Downward lines: tonnage curves, upward lines: grade curves*)

estimated by *OK* with cut-off 1 % is 1.56 % and the result of SGS is from 2.1 to 2.2 %.

Another important conclusion from the stochastic simulation result is that the variability of simulation-based copper metal tonnage is around 4 %.

CONCLUSION

In this paper, ordinary kriging and sequential Gaussian simulation have been reviewed and applied at the Sinquyen copper deposit. The result shows that there is a high possibility that *OK* has underestimated the grade of copper. Moreover, the variability of copper metal in the deposit is 4%. This information is very helpful when performing the mine planning for this project.

In future, it would be interesting to incorporate the geological uncertainty generated by SGS into the mine planning process, so we can evaluate its impact on the project's NPV.

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Author 1	Name&Surname: **MAI Ngoc Luan** Company: **Curtin University of Technology, Australia**
Author 2	Name&Surname: **BUI Xuan Nam** Company: **Hanoi University of Mining and Geology, Vietnam** Work Position: **Vice Rector** Scientific Degree: **Dr.-Ing.** Contacts: **buixuannam@humg.edu.vn**
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Abstract: Geostatistics-based estimators, i.e. ordinary kriging and simple kriging, are state-of-the-art estimation techniques widely used in the mining industry. However, the estimation result of kriging techniques is not able to cope with the well-known uncertainty of geological attributes in mineralization deposits. Moreover, the smoothing effect of kriging algorithms has led to over/under estimation in many circumstances. To overcome these drawbacks, the deterministic estimation result of kriging is usually followed by stochastic results provided by stochastic simulation. In this paper, the application of ordinary kriging and stochastic simulation are performed to build the resource model together with the uncertainty assessment of the Sinquyen copper mine.

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VALERY V. MOROZOV (*National University of Science and Technology MISiS*)
VALERY M. SHEK (*National University of Science and Technology MISiS)*
YURY P. MOROZOV *(**Ural State Mining University)*
LODOY DELGERBAT (*Joint Mongolian-Russian Enterprise "Erdenet Mining Corporation"*)

IMPROVING INTELLIGENT CONTROL OF ENRICHMENT PROCESSES BASED ON ORE SIZING VISIOMETRIC ANALYSIS

For the flotation process, the use of open control loops used as ore quality input parameters is effective. The operational analysis shall be performed to determine the ore quality including its material and mineral composition and structure. The main operational analysis is measurements of X-ray and visible spectrum areas.

New techniques and devices have been developed to improve accuracy of ore composition measurements. Specific lighting systems that alternate micro-focused and flat light flows were used to analyze the conveyed ore. These systems ensure high measurement accuracy of the conveyed ore. A special flat-bed plant for fine ore visiometric analysis have been developed a part of the existing sampling and analysis system. The flat-bed plant ensures higher accuracy of ore mineralogical composition measurements.

Modern ore visiometric analysis systems help create an effective automated control approach for enrichment processes based on advanced monitoring of ore sizing. The systems implemented at the Erdenet enrichment plant (Mongolia) contributed to higher levels of copper and molybdenum extraction.

Keywords: ore enrichment, reduction in size, flotation, automated control, algorithm, optimization, ore size determination, visiometric and X-ray fluorescent analysis

INTRODUCTION

The analysis of ore and/or enrichment products in the X-ray and visible sub-spectrum [1, 2] is a new approach based on ore size radiometric data, which can improve the control efficiency of enrichment processes. A key advantage of such techniques is the ability to take operational measurements of ore composition parameters and enrichment products, including mass fraction of specific minerals.

Previously, Russian and foreign experts determined the size of processed ore based on measurements of source ore material composition and its enrichment flotation products with X-ray fluorescent analyzers [3, 4]. However, an adequate determination of ore size is only possible with the operational measurement of mineral composition parameters such as the ore oxidation level and ratio of basic mineral forms [5, 6].

Ore size operation analysis can be implemented based on the continuous measurement of mineral composition directly in the process flow or based on the analysis of samples specifically taken from this ore sample flow.

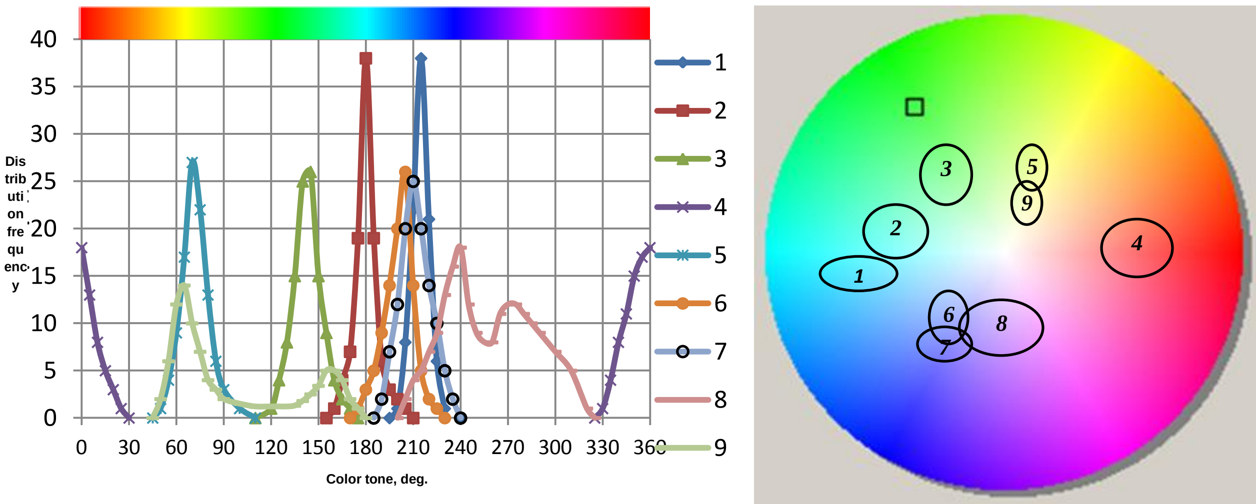
ORE SIZE VISIOMETRIC ANALYSIS METHOD

Reliable separate determination of ore minerals is possible with modern color pattern identification formats. The RGB format, which presents any color as a combination of red, green and blue, is an ideal format for primary processing of the color image. Other parameters, the most important of which are hue, saturation and value (HSV), are determined based on these three main parameters [7].

First, video images of all field minerals are entered in the system's database. Main ore copper minerals as well as primary (granodiorite and granosyenite) and accessory (quartz and sericite) rock forming minerals are presented in Figure 1.

Software processing methods are used to create computer-aided images (reference standards) of the main minerals. The mineral spectrum characteristics in the wave radiation visible range are the source of data (database) during visiometric analysis of ore mineral composition (Figure 2).

Fig. 1. Photographs of ore and rock forming minerals of copper-molybdenum ores.



a *b*

Fig. 2. HSV color characteristics of minerals: *1* – azurite; *2* – turquoise; *3* – malachite; *4* – cuprite; *5* – chalcopyrite; *6* – chalcocite; *7* – bornite; *8* – covellite; *9* – pyrite.

As shown in Figure 2, *b*, the resolution of mineral spectrum characteristics is higher when using a two-parameter recognition system: color – HSV saturation. Fully using the HSV format features increases the reliability of mineral determination up to 0.95 even for such complicated systems as chalcopyrite – pyrite or chalcocite – covellite, bornite.

The program uses simple interval limits of RGB energy for a single image area (pixel). 3D areas of RGB energy intervals for surveyed color minerals, and general image element energy are formed on the basis of two-dimension probability areas (areal) that describe the presence of mineral products in the HSL system. Recognition takes place on 3-dimension probability areas of HSL system mineral products single pixel color pairs; the neuron networks technique and system learning approach is used.

Figure 3 presents a source image of the aggregate of chalcopyrite and bornite and covellite (*a*), and recognition data (*b*).

The recognition efficiency is high even for close spectrum minerals such as covellite and bornite. It should be noted that the aggregate size shown in the Figure is 2 mm.

The mass fraction of oxidized minerals, primary and secondary copper, pyrites, quartz, sericite, mica as well as other minerals whose ratio describing ore size is determined with spectrum mineralogical analysis.

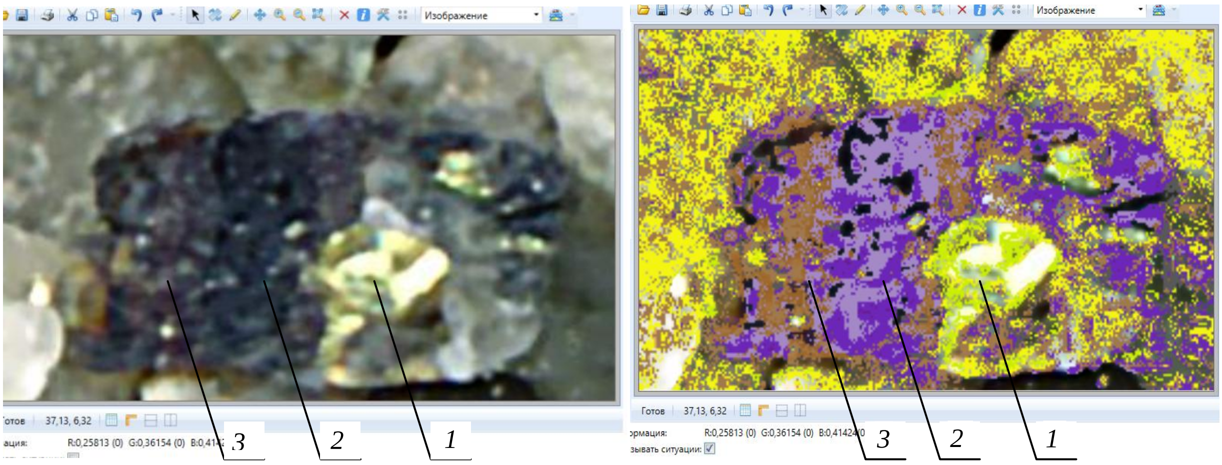


Fig.3, *a*. Image of the aggregate of sulphide copper minerals in copper-molybdenum ore and recognition data: *1* – chalcopyrite; *2* – bornite; *3* – covellite.

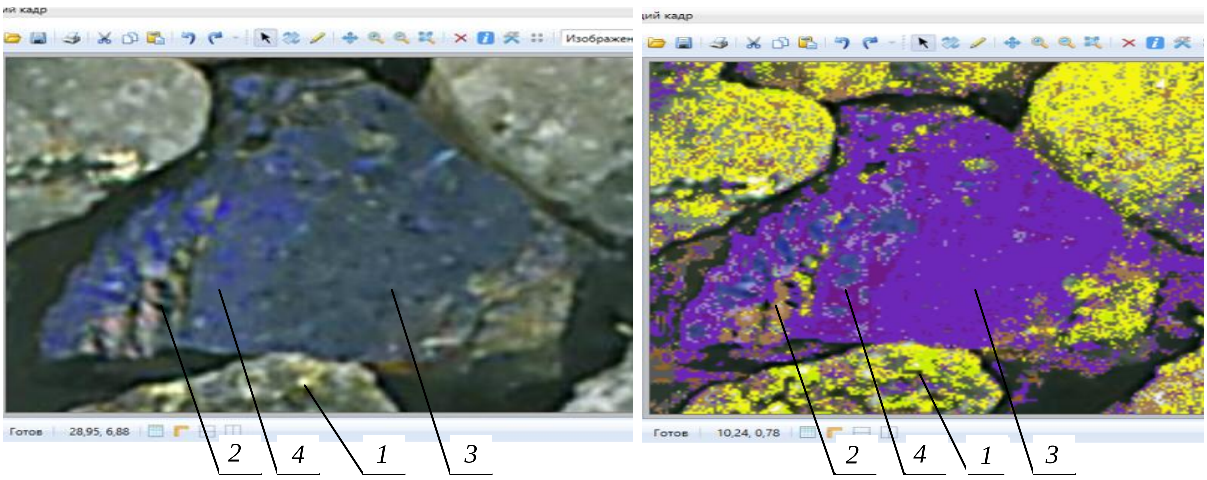


Fig. 3, *b*. Image of the aggregate of copper minerals in copper-molybdenum ore and recognition data: *1* – chalcopyrite; *2* – bornite; *3* – covellite; *4* – azurite.

This approach is most effective when we assess the ore oxidation level which can be calculated as a ratio of the integral intensity of spectrum characteristics of oxidized minerals to the general intensity of spectrum characteristics of all copper minerals. The ore oxidation level can be determined as a ratio of the mass fraction of copper in oxidized minerals to the general mass fraction of copper. Spectrum mineralogical analysis can also be used to determine the ratio between primary (chalcopyrite) and accessory (bornite, chalcocite, covellite) sulphide copper minerals. We can also determine the presence and measure the mass fraction of talc, mica, sericite and other minerals that have an impact on the flotation process.

The processable ore size is determined through its similarity to main process ore types [8]. The approach that we use is aimed at determining final ore size and composition − the fractions of main process ore types. X-ray fluorescent analyzer data (material composition, video-image analysis sensors data) are used as input data for analysis system.

Ore size is calculated using the multi-criteria accessory calculation method. The required solution is presented by standard ore types. The ore is presented as a combination of five ore types, while in-ore fraction of each ore type is determined. The system's mathematical pattern is responsible for calculating available ore size across six or more essential ore parameters (for example, copper, molybdenum and iron content in ore, mass fraction of oxidized copper minerals, accessory sulphide ore minerals in ore, primary copper minerals and sericite). Other parameters can also be used: for example, grain size composition, ore crushing, and grindability capacity [9].

The multi-criteria task solution is reduced to determining how much the reference point belongs to a specific set of points on a plane (2-dimension space) or on any other "plane". Such a calculation approach is used to determine the "similarity" of processed ore to any of five known ore types and establish the fractions of any of the five ore type in the ore available for processing [8]. Therefore, first of all we shall determine the distance from the point whose coordinates correspond to the ore available for processing, to any of the points whose coordinates correspond to the ore types established by process engineers as reference ore types (Figure 4).

Then, following valuation and assessment of the parameter significance, we shall use calculation equations to determine target mass fractions of standard ore types in the processable ore.

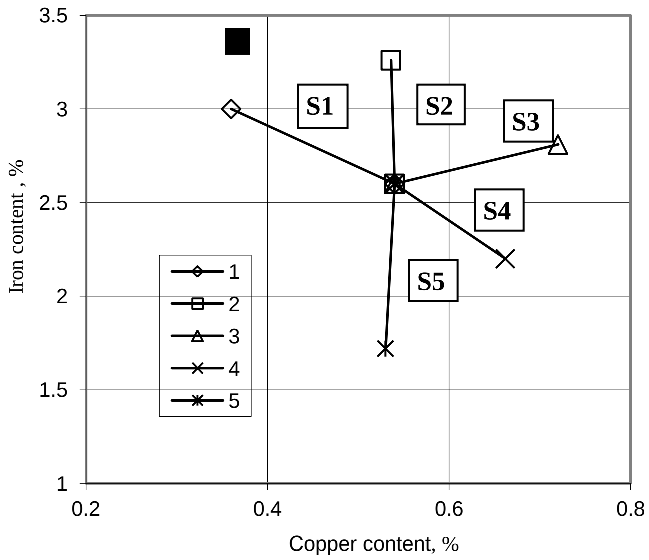


Fig. 4. Example of determining ore composition in 2-dimension space 1, 2, 3, 4, 5 – ore types; 6 (■) – current ore type; S1, S2, S3, S4, S5 – deviation of current ore parameters from types 1, 2, 3, 4, 5 ore.

The standardized deviation value (S_i) of the ore mix parameter (Z_n) from reference ore parameters (Z_{ni}) shall be calculated using the following formula:

$$S_i = \left(\left|Z_n - Z_{ni}\right|\right) / Z_{ni}, \text{ where } i = 1-5. \quad (1)$$

The standardized value for similarity of ore mix parameters to reference ore parameters shall be calculated using the following formula:

$$D_i = 1 / S_i, \text{ where } i = 1-5, \quad (2),$$

where S_i is the standardized deviation of ore mix parameters from reference ore parameters.

Calculation of the mass fraction of individual ore type (γ_i) in ore mix:

$$\gamma_i = kD_i / \sum\left(kD_i\right), \text{where } i = 1-5 \quad (3),$$

where k are significance coefficients of individual measured ore parameters.

The significance coefficients of individual measurement ore parameters are adaptive-adjustable parameters; the initial source for the adjustment of these parameters is actual ore size obtained from the samples taken and processed using classical techniques.

The size measurement operation is repeated at set time intervals. The final results of ore size and processed medium analysis are presented as time curves in Figure 5. These curves describe changes in processed ore composition.

Upgrading the optical analysis system for conveyed ore

Joint research by scientists from NUST MISiS and Erdenet Mining Corporation resulted in the development and testing of a new ore advanced diagnostic method based on optical analysis of ore composition.

The video-image analysis system (Figure 6) provides digital ore images. This was built with modern telemetric and software and hardware facilities. The system allows real-time data to be obtained on ore mineralogical composition and ore type. The system can also provide data on ore grain size distribution in the breakage operation and mineral impregnation.

The analyzer operates in two alternating modes. In operating mode 1: deflector *5* generates diffused light for uniform illumination of the sample. In mode 2: plane-parallel light flow is generated by deflector *6*. This promotes maximum image "contrast", which is necessary to measure ore sizing.

Conveyed ore undergoes continuous scanning. All data received from the system will be processed and averaged. Then, ore size and grading will be recognized subject to the algorithm described above. The obtained data are used for automated control of ore processing and floating processes.

Ore sample optical analyzer

The disadvantage of the conveyed ore analysis is low measurement reliability due to low definition of captured images. This is due to the insufficient homogeneity of the sample plane section and the impossibility of focusing on the entire ore surface.

Joint research by scientists from NUST MISiS and Erdenet Mining Corporation resulted in testing a new ore diagnostic method based on flat-bed analysis of mineral composition. The high accuracy of visiometric analysis is ensured by special devices embedded into the sampling system, and further ore sample analysis by the Quality Assurance Department. A sample analysis plant has been developed to support sample analysis using this method.

The plant includes a sample table made from transparent glass, a light flow source, optical system, and optical converter.

The measurement technique includes ore sample pre-processing, forming a measurement area in the form of a plane sample section, illumination and recording images of the sample plane section in the visible sub-spectrum. Sample images are illuminated and recorded from bottom to top, in 2D scanning mode.

By "placing" grains, significant reduction in sample roughness is possible. An almost plane-parallel, evenly illuminated surface without

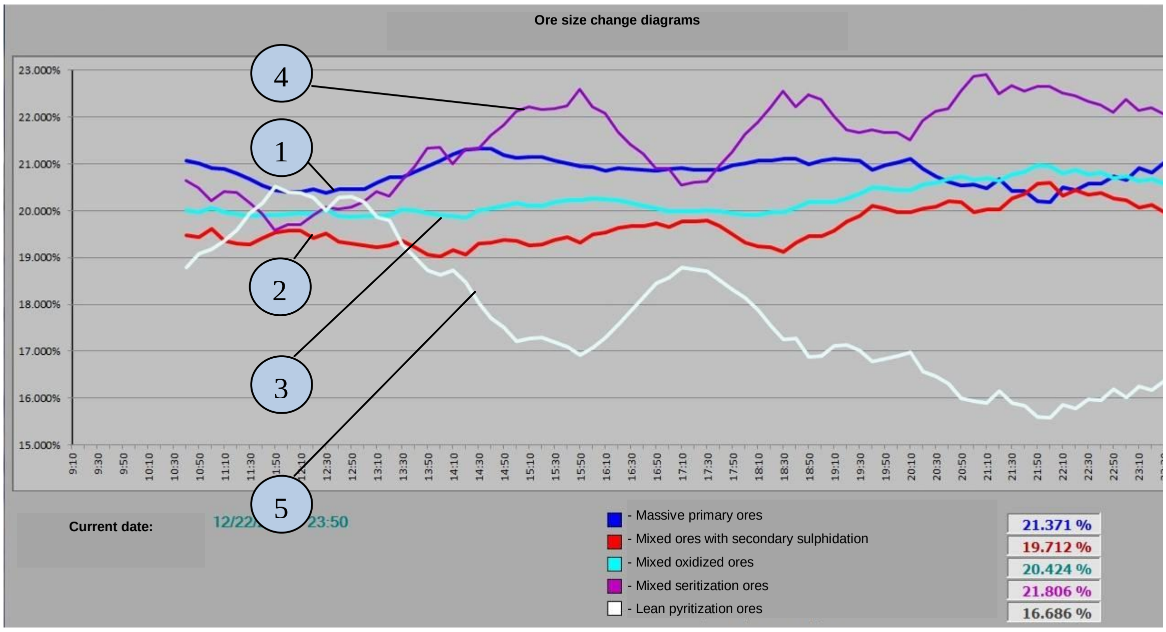


Fig. 5. Change in current ore composition: *1* – massive primary ores; *2* – mixed ores with secondary sulphidation; *3* – mixed oxidized ores; *4* – mixed seritization ores; *5* – lean pyritization ores.

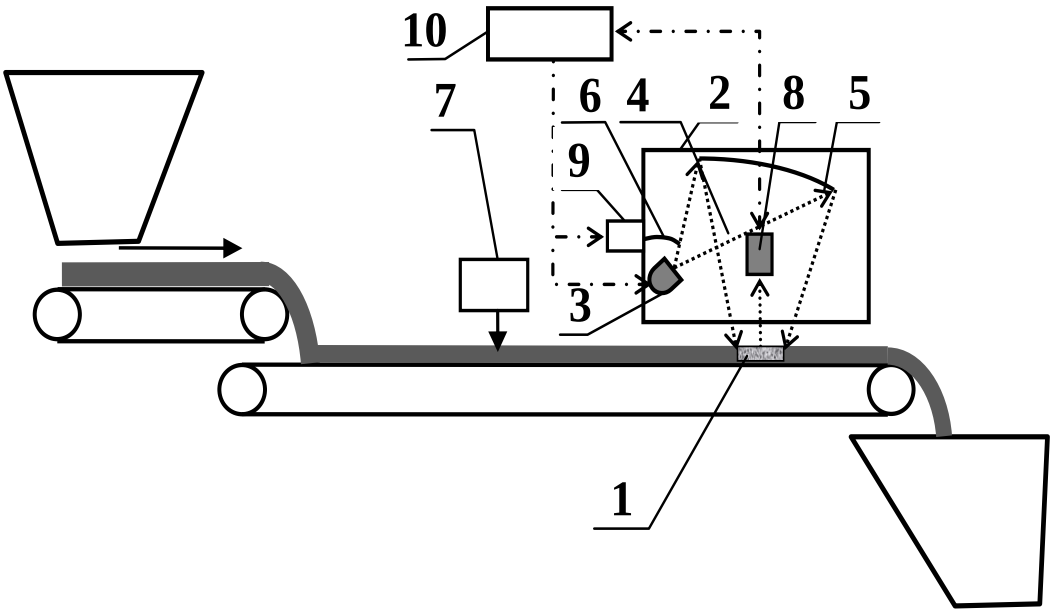


Fig. 6. Telemetric analyzer of ore size and quality. *1* – measurement area; *2* – body, *3* – light source, *4* – light flow; *5*, *6* – main and accessory deflectors; *7* – sprayers; *8* – video camera; *9* – deflector drive; *10* – microprocessor.

shaded sections will be formed. This ensures high image definition and reliability of mineral determination.

Control of enrichment processes based on ore sizing data

Initially, ore quality is inspected at ore development and transportation phases. This control process includes control for in-flow ore averaging, as well as separation of primary flow by mostly sulphide and mostly mixed ore flows. The proposed enrichment approach and automated enrichment control equipment are presented in Figure 8.

Phase 1 consists of sample taking and ore mineralogical analysis at the mining phase. "Ore oxidation" and "ore antecedence" are two criteria used to assess ore quality. Numerically, the "ore oxidation" criterion corresponds to that part of the ore which is in an oxidized mineral state (azurite, cuprite, malachite, chrysocolla, etc.). The "ore antecedence" will be calculated as an ore part in the form of chalcopyrite mineral (primary sulphide copper mineral). Depending on these parameters, the formed oxidized ore flows are sent for desalinization of primary and mixed ore available for enrichment.

To calculate optimum degradation and flotation parameters for various ore sizes, most typical ore samples were surveyed, and degradation and flotation process schedules were developed. Since it was impossible to obtain an ore sample of specific sizing, experts performed statistical processing of available data and developed "standard" ore enrichment models for massive primary ores (MPO), mixed secondary sulphide ores (MSSO), lean pyritization ores (LPO), mixed cetirizine ores (MSO), and mixed oxidized ores (MOO).

The model results presented as optimum degradation and flotation parameters for various ore size (Table 1) were recommended to be implemented in Erdenet enrichment plant's automated control systems.

Optimum degradation and flotation parameters were used in calculating setting functions SF for local automated systems of automated feedback regulation systems [10]. The process was subject to regulation after all input parameters, selected in accordance with ore sizing, were set.

The setting function value SF of any process parameter was calculated as a weighted optimum mean value of these parameters for every standard ore size (SF_i), taking into account the ore size contribution to ore mix subject to formula (4)

$$SF = \sum \gamma_i SF_i, \qquad (4),$$

where γ_i – is a relative mass fraction of ore type i available for ore mix processing.

The calculation data set includes the flow of reagents, ore processing, degradation sizing, etc.

While calculating reagent consumption, we may also take into account possible effects that can occur when processing various ore types. The ore cross-effect can be presented with correction coefficients L_i. Then, equation (4) will take the following form;

$$SF = \sum L_i \gamma_i SF_i. \qquad (5)$$

Pre-set functions were used as a basic level of local degradation and flotation automated control systems. Using the ore sizing determination procedure improves automated control reliability. Control stability improved by 5 % to 7 % after implementing the ore sizing determination algorithm. Maintaining optimum degradation and reagent consumption rate under current ore flotation provides a higher level of copper and molybdenum extraction to copper or molybdenum concentrate by 0.3 and 1.1 % respectively, as well as reagent consumption by 2–3 %.

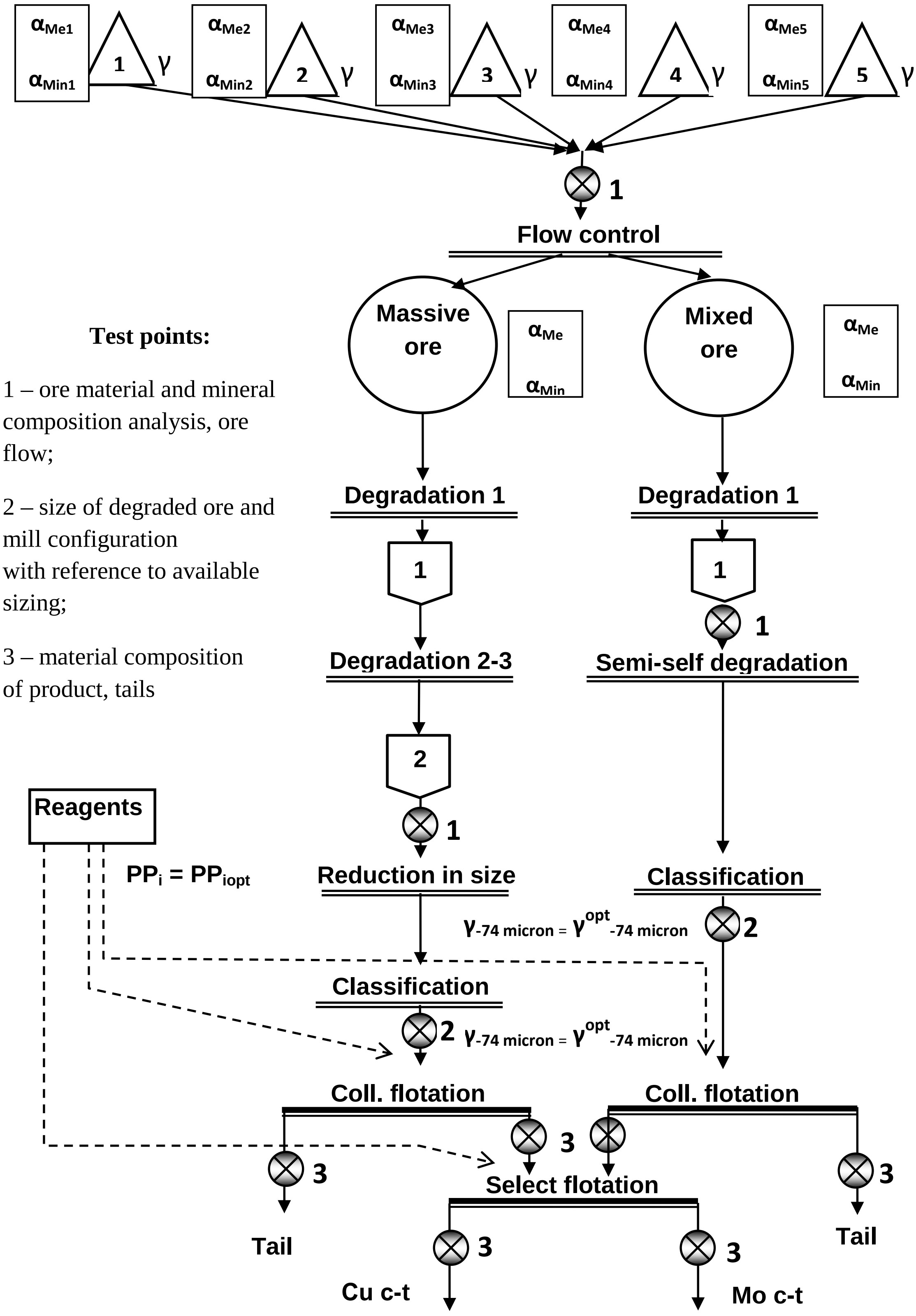


Fig. 8. Block Diagram - Formation of current ore flow as a mixture of specific process ore types and location of sampling points, IM – impregnation of valued components

Table 1

Optimum degradation process parameters and classification – target functions in control systems

Process parameters	MPO	MSSO	LPO	MOO	MSO
Degradation size, %, Class – 74 micron	67.50	64.50	67.00	66.00	66.00
Ore conveyed to mill, t/m^{3}h	1.65	1.74	1.71	1.75	1.75
Pulp density, %	43.50	41.00	41.50	40.00	40.00
Consumption of AeroMX-5140 collecting agent	10.00	12.00	13.00	17.50	10.00
Consumption of MIBK foaming agent	13.00	16.00	16.00	19.00	13.00
Lime consumption	1100.00	1300.00	1300	1300.00	1100.00

Note. Massive primary ores (MPO); mixed secondary sulphide ores (MSSO); lean pyritization ores (LPO); mixed cetirizine ores (MSO) and mixed oxidized ores (MOO).

CONCLUSION

Joint research by scientists from NUST MISiS and Erdenet Mining Corporation resulted in further development of the flotation regulation approach based on advanced assessment of processed ore sizing. New techniques and devices for automated measurement of composition of conveyed and sampled ore have been tested. The control algorithm uses ore size data and available information on optimal degradation and flotation process parameters. Implementation of the proposed system will help to reduce losses of valued components and reagent consumption by 3–5 %.

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Ganbaatar Z., Djelgjerbat L., Duda A.M., Morozov V.V. Upravlenie obogashheniem medno-molibdenovyh rud na osnove kompleksnogo radiometricheskogo analiza rudy [*Management enrichment copper-molybdenum ores based on a comprehensive analysis of radiometric ore*] // Plaksinskie chtenija [Plaksin read] / Materialy mezhdunarodnoj nauch-prakt. konf. [Proceedings of the int. scientific-practical conference.] – Ekaterinburg, 2011. – pp. 118-121.

"Gornye nauki i tehnologii"/ "Mining science and technology", 2016, No. 2, pp. 29-38

Title:	**Improving the intelligent methods of management of processes of enrichment on the basis of visiometrics analysis of the grade of the ore**
Author 1	Full name: **Valery V. Morozov** Company: **The National University of Science and Technology MISiS** Work Position: **Professor** Scientific Degree: **Doctor of Engineering Science** Contacts: **dchmggu@mail.ru**
Author 2	Full name: **Valery M. Shek** Company: **The National University of Science and Technology MISiS** Work Position: **Professor** Scientific Degree: **Doctor of Engineering Science** Contacts: **shek@geotwo.ru**
Author 3	Full name: **Yury P. Morozov** Company: **Ural State Mining University** Work Position: **Professor** Scientific Degree: **Doctor of Engineering Science** Contacts: **tails2002@inbox.ru**
Author 4	Full name: **Delgerbat Lodoy** Company: **Erdenet Mining Corporation, Mongolia** Work Position: **Chief Specialist at ACS** Scientific Degree: **Doctor of Engineering Science** Contacts: **delgerbat@erdenetmc.mn**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-2-31-42**
Abstract:	For the flotation process using of open control loops, which are used as input parameter of the ore quality, is effective. To determine the quality of ore, on-line analysis of its elemental and mineral composition and structure is carried out. The basis of the operational analysis is the measurement of X-ray and visible light. To improve the accuracy of determining ore mineralogical composition, new techniques and facilities have been developed. At the conveyor ore analysis special lighting systems were tested, using alternating focused and flat light fluxes. Such systems provide highly accurate in-stream measurements of ore properties. A special flatbed facility for optical spectrum-based estimation of fine-crushed ore has been developed, being incorporated into the existing system of ore sampling and analysis. Application of the flatbed facility renders possible a high accuracy of determining ore mineralogical composition. Modern systems for optical spectrum-based estimation of ore grade provide a basis for effective automated control of ore beneficiation processes, based on the principle of advanced ore grade control. The systems implemented at the Erdenet processing plant

	(Mongolia) contributed to increasing copper and molybdenum recovery.
Keywords:	mineral processing, milling, flotation, automatic control, algorithm, optimization, estimation of ore grade, visible and RFA analysis
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LUDMILA S. KSENDZENKO (*Far Eastern Federal University)*
ALEXANDR S. LOSEV (*Institute of Applied Mathematics, Far Eastern Branch of Russian Academy of Sciences*)

OPTIMIZATION OF THE CALCULATION OF THE PERIODICITY PARAMETERS OF THE ROCK MASS ZONAL DESTRUCTION MODEL

An overview of new and most significant mining research using the non-Euclidean continuum model to describe the stress field distribution around a circular cross-section roadway. The accuracy of the calculation of created model parameters has been studied and compared with real experimental data. Specific periodicity parameters of the rock mass zonal destruction model around a deep circular roadway have been studied individually. The selection of this parameter is based on its prognostic significance for solving problems in mining deposit research. A comparative analysis has been made of the analytical and numerical determination of this parameter with physical data from two independent fields. The technique for combined use of the analytical and numerical approach, depending on the active task at surveyed field, has been described and justified.

Keywords: highly compressed mass; zonal destruction; parameters; non-Euclidean model; mass; field data

INTRODUCTION

Research into geological materials to resolve issues related to the mining industry is currently a priority field in many countries and involves experts from around the world. Recent efforts by A.M. Guzev and A.A. Paroshin helped them obtain significant results in developing new methods and approaches in this field. They were the first to propose the non-Euclidean continuum medium model to describe stress field distribution around a circular cross section roadway [1]. Instead of Saint-Venant's strain compatibility condition, the authors introduced the rock mass defectiveness function for plane deformation; this function complies with the biharmonic equation with preset limit conditions. The obtained result served as a basis for new surveys and provided new understanding of rock mass zone disintegration around deep underground mines [2–8].

In particular, the authors of the work [2] found a solution to end the problem of stress field distributed around a mine in the event of deformation and under non-hydrostatic loads. The task is presented as two integral components, while the solution refers to the expansion of the elastic stress field into the sum of the fields; the first is due to incompatible deformations of disturbed zones; the second is due to joint deformations of non-disturbed zones. In the first case, the elastic stresses can be determined with the non-Euclidean model; for the second one — using the standard elastic-mechanical model. As a result, using the Mohr-Coulomb stress criteria, the authors determined the number of disturbed areas, their location, that depend on physical and mechanical properties of the rock material as well as on non-Euclidean parameters.

In their work [3] the authors determined the quantity and size of disturbed and non-disturbed zones using the proposed non-Euclidean model of zone disintegration of geological material around a circular cross-section roadway. Zone quantity and size characteristics are shown with reference to natural, tangential and radial stresses, intermediate main stress coefficient as well as RMR (Rock Mass Rating) classification coefficient.

The work [4] presents the dynamic model for survey zone disintegration in enclosing mine rock mass around deep circular tunnels under hydrostatic stress condition. The non-Euclidean dynamic equation is obtained based on non-equilibrium thermodynamics; the solution of the equation is determined by direct and inverse Laplace transformations. It has been shown that the number of disturbed areas increases depending on the step-up of rock disturbance factor, relief rate, as well as subject to a decrease in monoaxial compression strength, GSI geological index, strength m_i, and Poisson coefficient ν.

The work [5] presents a new non-Euclidean model to survey impacts of deep penny-type mining cavities and natural axial stress on zone disintegration. The deformation energy density coefficient was used to determine the stress intensity coefficient for penny-type cavity peaks. The numerical calculation shows that both size and location of the rupture zone are sensitive to micro- and macromechanical parameters as well as non-disturbed mass stress.

The work [6] described a non-Euclidean model which was used to analyze the relation between zone disintegration and rock bumps. The formation mechanism for secondary micro cracks and the mechanism for their transfer to a non-stable state, expansion and secondary micro crack build-up and macro crack formation (resulting in rock bumps) were studied.

Despite the uniqueness of the research, the authors of the works [2–6] made the following conclusions:

- under high depth conditions, rock mass zone disintegration around underground mines is normal;
- zone disintegration structure features repeated the roadway contour and sequence of disturbed and relatively non-disturbed geological material;
- non-Euclidean mathematical models are the most suitable models for describing zonal disintegration effects.

However, there are some serious issues with the works presented here: there is no comparison with real experimental data, the algorithm used to determine the created mathematical models parameters is incomplete. In view of the above, the reliability of the built model parameters, as well as calculation accuracy and its influence on surveyed physical effects and rock phenomenon, remain uncertain.

Stress field distribution around a circular cross-section roadway

Let's consider field stress distribution around a circular cross-section roadway. This shall be analyzed as plane and stationary, non-compressible, with hydrostatic load for infinite distance [1]. This task shall be solved based on the obtained equilibrium equation

$$\frac{\partial \sigma_{rr}}{\partial r} + \frac{1}{r}(\sigma_{rr} - \sigma_{\phi\phi}) = 0, \tag{1}$$

biharmonic equation for defect function

$$\Delta^2 R - \gamma^2 R = 0, \tag{2}$$

and limiting conditions

$$R\big|_{r=r_0} = 0, \ \frac{\partial R}{\partial r}\bigg|_{r=r_0} = 0, \tag{3}$$

where σ_{rr} is normal radial stress; $\sigma_{\phi\phi}$ – normal tangential stress; $\sigma_{r\phi}$ – shearing stress; Δ – Laplace operator; γ – model periodicity parameter. In polar coordinates for biharmonic equation

$$\left(\frac{\partial^2}{\partial r^2} + \frac{1}{r}\frac{\partial}{\partial r}\right)^2 R = \gamma^2 R \tag{4}$$

the solution under the condition $r \to \infty$ for the distance between the roadway center to a mass point is presented as equality:

$$R(r) = aJ_0(\sqrt{\gamma} r) + bN_0(\sqrt{\gamma} r) + cK_0(\sqrt{\gamma} r), \tag{5}$$

where J_0, N_0, K_0 is the Bessel, Neumann, and Macdonald zero-order function.

The solution [7] of biharmonic equation (2), unlike [1], provides the following for limiting conditions:

$$\frac{\partial R}{\partial r}\bigg|_{r=r_0} = 0, \ \frac{\partial R}{\partial r}\bigg|_{r=r*} = 0, \tag{6}$$

corresponding to the following zonal rock disintegration, subject to the following approach: first limiting condition for the function R shall be considered as its extremity at the roadway boundary; while the second limiting condition shall be considered as the extremity in the middle of the first disturbed zone, etc.

However, here the reliability and quality of the analyzed model parameters remain uncertain. An important problem is the reliability of the determination of the periodicity parameter and its influence on the reliability of predictions, number, locations, and length of rock mass disintegration zones. While studying this problem, we developed two separate approaches to determine the parameter γ.

Method for determining the rock mass zone disintegration periodicity model parameter

Analytical approach. For the surveyed fields, the analytical dependence between the parameter γ and the distance from roadway boundary to the middle of the first disturbance zone (in roadway radius units) was obtained based on real experimental data

$$\gamma(r^*/r_0) = -10(r^*/r_0) + 23, \tag{7}$$

where r^* is the middle of 1st disintegration zone starting from the roadway boundary; r_0 is the roadway radius, m [7].

Numerical approach. This approach is based on the numerical selection of the parameter γ, with which the parameters *a* and *b* of the defect function take on maximum negative values at the roadway boundary, i.e. they will reach defect function at the roadway boundary extreme value and, therefore, the presence of a disintegration zone [8].

Each of the proposed approaches has its specific advantages and disadvantages; in particular, the analytical approach determines the parameter with an accuracy up to an γ integral number; however, it has not been fully tested on unknown fields, but it is very simple for calculations. The numerical approach is a common method to determine the parameter γ. This method helps calculate the target parameter with preset accuracy level. However, this is a time-consuming calculation method.

Using these two approaches for calculating the periodicity parameter of surveyed fields shows a variance in accuracy of up to 11.21 % in some cases (Table 1).

The calculations of relative accuracy for the surveyed fields (Tables 2, 3) show that an increase in the number of characters after the comma produces a positive impact on the calculation of the disintegration boundary in general.

For Norilsk mine (see Table 2) it can be concluded that the relative error of disintegration zone I boundary calculation is less, when $\gamma = 3$, e.g. the parameter, takes on an integral value. Analysis of the Dingji coal mine (China) data (Table 3) shows that an increase in digits after the dot in the parameter γ reduces the accuracy error for the calculation of all disintegration zone boundaries. With a more detailed analysis, it can be concluded that the increase in the number of digits for Zone I defines more exactly the accuracy error, since such an error reduces by more than 1 %, while the relative accuracy error for other disintegration zones reduces much more − by 2–4 %.

CONCLUSION

With reference to the above, we still cannot say that one periodicity calculation approach is better than the other. Therefore, both numerical and analytical approaches shall be used, depending on the specific task. In particular: for initial analysis of the roadway at a new field, the analytical approach is recommended to decide on further development of the mine. However, if a more detailed analysis is required to determine the number of zones, their radial length, zone occurrence depth, exact location of the last disintegration zone and the boundary of a single mine area, the numerical method shall be used to obtain more reliable data.

The combined use of numerical and analytical methods to determine the periodicity model of rock mass zone disintegration will result in a higher reproducibility of theoretical and field survey data. Such combined usage helps reduce the relative error during calculation of the surveyed model parameters, which would increase the accuracy and reliability of the results obtained with this model.

Table 1

Parameter γ for various fields

Geological rock location	r^* / r_0	γ	γ^*	Relative accuracy, %
Norilsk fields	2.0	3	3.336526	11.21
Donbass fields	1.0	13	13.050143	0.38
Primorsky Krai, Artyom Mine (settlement Shkotovo)	0.6	17	17.637309	3.75

Table 2

Norilsk field

Parameter value γ	Relative accuracy calculation of close and remote disintegration zone boundaries, %			
	Zone I		Zone II	
$\gamma = 3$	0.71	4.86	19.06	6.8
$\gamma = 3.2$	2.14	5.36	17.18	5.29
$\gamma = 3.29$	3.57	5.67	16.25	4.53

Table 3

Dingji coal mine (China)

Parameter value γ	Relative accuracy calculation of close and remote disintegration zone boundaries, %							
	Zone I		Zone II		Zone III		Zone IV	
$\gamma = 9$	0	2.69	10.60	10.80	11.60	3.00	9.51	7.04
$\gamma = 9.9$	0	1.81	8.09	8.17	8.00	0.13	5.45	3.44
$\gamma = 9.93$	0	1.61	8.00	8.07	7.88	0.22	5.33	3.33

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"Gornye nauki i tehnologii"/ "Mining science and technology", 2016, No. 2, pp. 40-44	
Title:	**Optimization of calculation of parameter of the frequency model zonal destruction of rocks**
Author 1	Full name: **Ludmila S. Ksendzenko** Company: **Far Eastern Federal University** Work Position: **Associate Professor of the Department of algebra, geometry, and analysis** Scientific Degree: **Candidate of Physico-Mathematical Sciences** Contacts: **ksendzenko@mail.ru**
Author 2	Full name: **Alexandr S. Losev** Company: **Institute of Applied Mathematics, Far-Eastern Branch of the Russian Academy of Sciences** Work Position: **Senior Researcher** Scientific Degree: **Candidate of Physico-Mathematical Sciences** Contacts: **A.S.Losev@yandex.ru**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-2-43-49**
Abstract:	An overview of the new, the most significant results in the field of mining research based on the use of non-Euclidean continuum model to describe the distribution of the stress field around the development of circular cross-section. The question of the accuracy of calculation of the parameters of the models constructed and compared with field experiments. Considered separately taken model parameter frequency zonal fracture rock mass around deep development of circular cross section. Selecting this option justified its prognostic significance for solving problems in the study of mining deposits. A comparative analysis of analytical and numerical determination of this parameter with the field data of two independent fields. The technique is described and sharing analytical and numerical approach, depending on the task on the test field.
Keywords:	highly compressed array; zonal destruction; options; non-Euclidean model; array; Full-scale data
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of surrounding rocks around a deep circular tunnel in a non-hydrostatic pressure state. Journal of Mining Science, 2011, no. 47(1), pp. 37–46. DOI:10.1134/S1062739147010059

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IVAN B. RATNIKOV (*Neftekom LLC*)
ROMAN S. SHULGA (*Siberian Research Institute of Petroleum Industry JSC*)
EVGENIY A. ROMANOV (*Siberian Research Institute of Petroleum Industry JSC*)
SERGEI N. BASTRIKOV (*Siberian Research Institute of Petroleum Industry JSC*)

ANALYSIS OF THE POROSITY PARAMETER USING ELEMENTS OF REGIONAL PETROPHYSICS (ELECTROFACIES) AND CLAYINESS TYPE

The most important calculation parameter, the open porosity ratio (K_p), is generally determined in petrophysical laboratories for the object (stratum) as a whole and without taking into account its regional heterogeneity. A range of the porosity value is taken as a basic criterion for selecting samples to construct the model $R_p = f\left(K_p\right)$ K_p. In this case it is assumed that: the higher ΔK_p is, the higher the validity of the relation $R_p = f\left(K_p\right)$ is. However, in practice such a concept often creates an ambiguity in determining K_p the porosity by electric resistivity of water saturated samples in different wells, even within one field. This phenomenon is caused by underestimating the features of the regional petrophysics object, the facies, which are reflected in the form of well logging curves, including SP, and the element of the general petrophysics object – the clayiness type. The breakdown (classification) of the formation by facies, determination and taking into consideration the type of clayiness, as well as distinguishing and analyzing the petrophysical types of rock based on rock porosity and permeability characteristics within the formation, make it possible to obtain satisfactory relations between electric resistivity and porosity of K_p the rock.

Keywords: open porosity, electric resistivity, facies, clayiness.

INTRODUCTION

The estimated and actual data for calculating the reserves and production of raw hydrocarbons depend, in addition to the quality of sampling and experimental set up, on the degree of approximation of the results of petrophysical studies to the real situation. Frequently, porosity in the comparison planes K_p with R_p and K_p with geophysical well logging data is not differentiated adequately. There are at least two reasons for this discrepancy.

Firstly, in the summary analysis of dependencies of K_p with petrophysical data and geophysical well logging data, regional factors that sometimes have a decisive impact on the petrophysical properties of rocks, are not taken into account by geophysicists (or petrophysicists). First of all, there are such features as: type of clayiness (structured, layered, dispersed), the specific surface of the pore channel, the value of its tortuosity, and shape of the packaging of rock particles, and the "maturity" degree of rocks. All these parameters are not included in the standard set of laboratory studies due to the complexity and lack of time to determine them. Nevertheless, the nature of the flow of fluid and electrical charge in mudded rocks is greatly determined by the combination of these symptoms. The magnitude and variation of these characteristics are defined by the conditions of the sedimentation. Various parts (elements) of the same layer (object) may belong laterally to different geological bodies which were formed in different paleogeographic conditions with their own dynamics of sedimentation, redox, and acid-base environments. A different lithogenesis situation provided the rock with properties that are characteristic only to these sedimentation conditions. Petrophysical characteristics, including electrical, are not an exception.

Secondly, the adequacy between the above parameters is violated for a trivial reason – poor sampling (core) within the stratum (object) section for laboratory research purposes. We can agree that a perfect selection of samples is not possible. However, petrophysicists should draw the attention of experts, using the results of laboratory studies, to the unsystematic sampling and examination of samples.

The objective of this work is an attempt to identify the nature of the variation only of $R_p = f\left(K_p\right)$ dependence that is defined for the homonymic layer on different wells within the same deposit.

General and regional petrophysics. The object and subject of general and regional petrophysics

Before proceeding to the analysis of K_p and R_p, we need to formalize the concepts related to ***the general*** and ***regional petrophysics*** and their objects and subjects. This is due to the fact that in petrophysical literature there is still no clear distinction between objects and subjects of both general and regional petrophysics, which is a consequence of the weak development of a systematic approach in terms of the theoretical framework of petrophysics.

Petrophysics is a geological cycle science. This means that it studies the geological environment (GE), which is determined as an object of petrophysics. The body of knowledge on the object is regarded as the subject of science. In the published materials the term "geological environment" is reduced to the following definition: the GE – is a mineral substance in solid, liquid and gaseous states, and the physical fields inherent to it. The GE is characterized by attributes such as structure, properties, and motion (chemical and physical processes). The GE includes the concept of "rock" as a phase system consisting of solid, liquid and gaseous phases.

It should be noted that geologists understand only the solid phase as the rock [14, 15, 18]. In order to be able to distinguish the concept of the rock in the geological and petrophysical aspects, it is necessary to further formalize this concept. The term "reservoir" is also not suitable for a petrophysical understanding of the rock – it increases, without any doubt, the scope of the concept, but at the same time limits significantly its content.

One of the important features of the GE is its variability over time and space [10]. Variability causes heterogeneity of the object, which is determined by the difference in its properties at different points. G.K. Bondarik [11] writes: "The heterogeneity is detected by comparing the elements of the set with respect to certain properties, by establishing the measures of similarity and relation between elements". For example, it is possible to select the GE heterogeneity based on its different parts (elements) belonging to a different formations (facies) and, as a consequence, to detect the heterogeneity of its physical properties.

Petrophysics, which has GE as its object, studies primarily the (petrophysical) rock, its material composition, structural and textural features, physical, chemical and mechanical properties, and changes in these properties over time (for example, in the process of deposit development).

Following the logic of the system approach, the science of "petrophysics" should be divided into at least two areas: ***general petrophysics*** and ***regional petrophysics*** (there is also general geology and regional geology, general hydrogeology and regional hydrogeology, general soil science and regional soil science, etc.). Each section shall have its own object and system of knowledge (theory, methodology, and application attributes).

From a formal point of view, the basis of ***general petrophysics*** should consist in teaching about the nature and the principles of the formation of the material composition, structural and textural characteristics and properties of (petrophysical) rocks during lithogenesis, and human engineering activities. The system of this science includes the theory and modeling of the formation of the composition, state, structural and textural distinctive features and properties of (petrophysical) rocks. Although the GE is an object of general petrophysics, it is regarded as *a system of attributes*, which has neither physical volume, nor fixed coordinates in geological space (for example, classification of rocks by permeability, density, Young's modulus).

Often, when analyzing this petrophysical characteristic, it is generally considered within the entire stratum without regard to heterogeneity of the stratum itself (belonging of different parts of strata to the different facies). The analysis results using such an approach are often contradictory and ambiguous.

The above difficulties can be overcome (or minimized) if the properties of the (petrophysical) rocks are analyzed using the knowledge system ***of regional petrophysics***. Following on from G.K. Bondarik (1981) who developed a general theoretical foundation for engineering geology, we believe that the basis of the subject of regional petrophysics should be a system of knowledge on the principles of spatial distribution of (petrophysical) rocks, on the principles of spatial variability of their composition, state and properties, and on the nature of the formation of spatial patterns. In contrast to general petrophysics, regional petrophysics should represent the rock not only in the attribute space, but also in geological space. It should consider the rock as a geological body occupying a fixed position in geological space. The object of regional petrophysics are *the geological bodies* selected by a real factor. The geological bodies, their structure, material composition and properties are the result of the interaction of physical fields of a specific dynamic system. Therefore, the properties of geological bodies formed in different paleogeographic conditions reflect the characteristic inherited features that are unique to these conditions.

Separating general petrophysics from the regional framework leads to a measuring technique for rock parameters that is only used in statistical calculations. Calculations are aimed mainly at obtaining a high degree of reliability of approximation of the compared parameters, therefore projected and actual results are often difficult to explain.

K_p and R_p were analyzed taking into account the features of the attribute and geological spaces. In the attribute space, in addition to porosity, the clayiness type is considered, as well as in integral form – a specific surface of the pore channels, their tortuosity, the form of packing of structural rock-forming elements and in the geological form – the facies. The division of the GE by material composition and formation conditions in the aspect of regional petrophysics presents certain difficulties in terms of methodology and requires considerable material and financial costs. Therefore, on the basis of practical tasks, the geological bodies were selected based on the facies, which, in turn, were diagnosed with the use of SP (spontaneous potentials) curves. The technique of determining the facies with SP data (hereinafter – electro-facies) for solving geological issues is given in detail in the well-known monograph from V.S. Muromtsev [19].

Research methods and results

$R_p = f\left(K_p\right)$ was analyzed for Yu2 stratum on the 215, 318, 417, 613 and 1610 wells of Vyngapurovsky deposit.

Sediments of Yu2 reservoir formation within well 215 are presented by light gray, sometimes with a brownish color tone, fine-grained silt and silty sandstone with clay and carbonate-clay cement. The structure is predominantly massive. Thin interlayers of carbonaceous-micaceous and clay material occur, concretions of siderite, carbonaceous plant detritus and pyrite inclusions are noted.

Yu2 reservoir formation near the well 215 consists of light gray and brownish-gray fine-grained and moderately fine-grained silt and silty sandstone with clay and carbonate-clay cement. Interlayers and lenticules of carbonaceous-micaceous and clay material occur. The structure is massive, discontinuous and undulated, gently sloping undulated and lenticular. Alluviations of carbonaceous-micaceous material, siderite

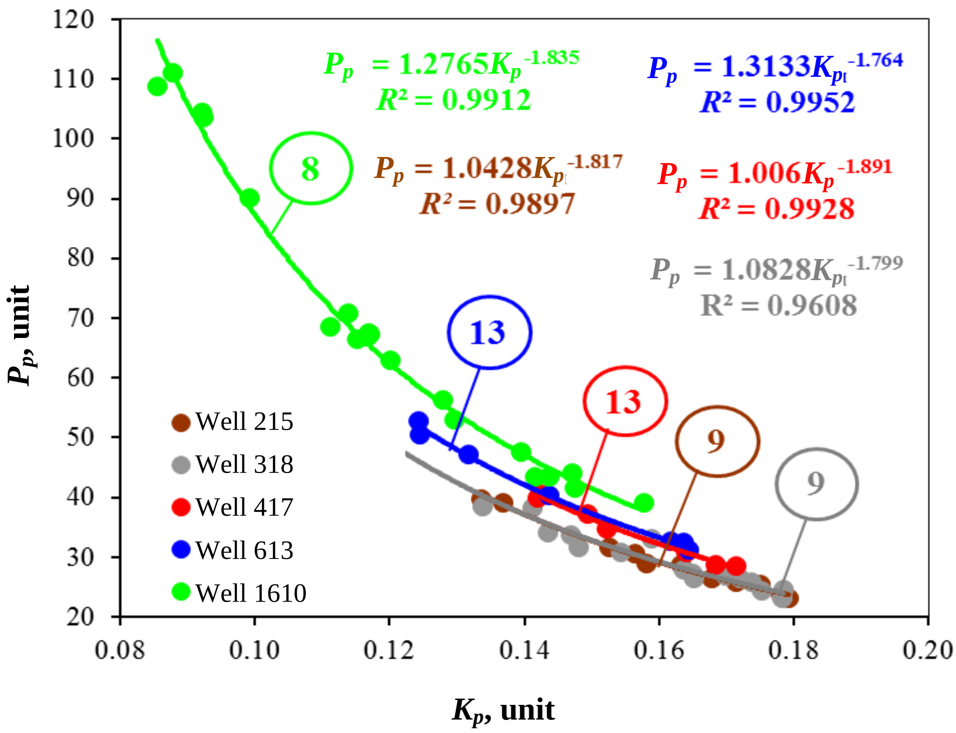


Fig. 1. Comparison of the porosity parameter (P_p) and open porosity (K_p) for the rocks of Yu2 layer of Vyngapurovsky deposit.

concretions and debris of coalified wood are noted.

The rocks of Yu2 reservoir formation exposed by well 417 represent a gray fine-grained silt and silty sandstone with uneven clay interlayers, sometimes changing into a fine- and coarse-grained sand siltstone with clay cement. The structure is mainly microlaminated.

Alluviations of carbonaceous-micaceous material and dark gray silty argillite interlayers are noted in thicknesses of 1 mm to 2 cm. Siderite is developed largely on alluviations. Traces of burrowing organisms are observed.

The collector of Yu2 layer in the well section 613 is composed of light gray fine-grained silt sandstone with clay and clay-carbonate cement, and light-gray fine-coarse-grained siltstone with a clay cement. Carbonaceous plant detritus and carbonaceous-micaceous material, thin slices and lenticules of clay matter are found in the sandstone, the passage ways of burrowing organisms made by clay matter are marked. Numerous slices and lenticules of clay material and carbonaceous plant detritus and aggregates of pyrite and siderite concretions occur in the siltstone.

Sediments of Yu2 reservoir formation exposed by well 1610 represent a light gray fine-grained silt sandstone with carbonate-argillaceous and argillaceous carbonate cements with occasional thin slices of micaceous-carbonaceous material, accentuating the parallel horizontal flazer bedding. Carbonaceous plant detritus is noted.

The composition and shape of Yu2 reservoir rocks imply that the sedimentation paleoenvironment in areas of dislocation of wells 215, 318, 417, 613 and 1610 are, to some extent, different from each other. At first glance, it could be assumed that the difference in the environment of GE formation predetermined the heterogeneity of the petrophysical properties of rocks.

Fig.1 presents a comparison of K_p and R_p by 5 wells (to make it easier to read, graphics data are not presented in a logarithmic scale). It can be seen that the points representing K_p and R_p do not belong to the single curve, but are divided into three groups: 1st group – wells 215 and 318; 2nd – wells 417 and 613; 3rd – well 1610.

We will try to understand why the relation between K_p and R_p is so different within the same reservoir in terms of laterality.

Reason No. 1 − external. SP data indicate that the shape of the curves within the studied formation differs significantly from well to well. Numerous domestic and foreign published materials show that the shape of the SP diagram is closely related to the sedimentation environment and the different geological parameters [2–9, 12, 13, 16, 17, 18, 20, 21, 23, 24]. After completing analysis of the SP waveforms according to Muromtsev [19] for this formation, we managed to identify 3 electrofacies (Figure 2): 8 shores (well 1610), 9 – coastal banks (wells 215 and 318), 13 headers of rip flows (wells 417 and 613).

It is also known that the clay type, and the quantity and composition of clay often have a decisive importance on the value of the electrical resistance of rocks. Unfortunately, to date, there has been no comprehensive study of the clayiness of oil deposit rocks.

Fig. 3 shows the result of the mass determination of the type of distribution of clayiness of Yu2 reservoir of wells J2 215, 318, 417, 613 and 1610. The algorithm for establishing the type of clay is very clear from the figure. In the layered distribution, the clay particles replace the sandstone grains and fill the pore space; for diffused dissemination the clay particles fill only the pore space and in the structural – clay minerals replace only the sandstone grains (Fig. 4).

With the same clayiness index (K_{cl}), the capacitive properties of rocks are defined as follows: in laminated clay volume $K_p = K_{cl}^{\max}\left(1-K_{cl}\right)$, in dispersed – $K_p = K_{cl}^{\max} - K_{cl}$, in structural – $K_p = K_{cl}^{\max}$ [22]. Fig. 3 also shows that dispersed clay predominates in wells 215, 318, 417, 613 and 1610. Clay minerals dispersed in the pore space can occur in the form of individual particles, coatings of sandstone grains (this promotes the appearance of fused clay crystalline coatings on the grains) and crystalline streaks or bridges of clay minerals within the rock pore space (Fig. 5). For all these types of clay distribution in the pore space of rocks, the specific and extremely strong influence of clays on the permeability of the formation is characteristic (Fig. 6). For example, with the same porosity, the clay streaks in the pore space are characteristic mainly for dense rocks (with less than 1 mD permeability). Clay coatings on the surface of the sandstone grains have rocks with permeability of 1 to 200 mD. In the presence at the pore space of discrete clay particles, permeability increases significantly and reaches 100 ... 1000 mD that is inherent in clean rocks. In other words, the permeability of the reservoir is highly dependent on the type of distribution of clay minerals in the pore space [22].

For analysis of resistance, porosity and permeability of the Yu2 formation J2, petrophysical types of rocks are selected (Fig. 6). For this purpose, the theoretical and empirical findings of Poiseuille, Darcy, Kozeny and Karman are used explaining the dependence of permeability on porosity:

$$K_{perm} = \frac{K_p^3}{(1-K_p)^2} \times \frac{1}{F_s \tau^2 S_{gv}^2}, \qquad (1)$$

where F_s is the coefficient depending on particle packing shape, τ^2 is pore tortuosity, $F_s\tau^2$ is the Kozeny constant, S_{gv}^2 is the surface area of particles in the rock unit volume minus pore space [1].

Transforming the equation (1), we obtain:

$$0,0314\sqrt{\frac{K_{perm}}{K_p}} = \frac{K_p}{1-K_p} \times \frac{1}{\sqrt{F_s}\tau \cdot S_{gv}} \qquad (2)$$

where 0.0314 constant is the conversion factor from square micrometer to mD.

In the formula (2) the expression $1/\left(\sqrt{F_s}\tau \cdot S_{gv}\right)$ is a hardly determinable parameter. Let us denote it as FZI (F_{zi}) – flow zone position indicator. Ratio $\left(K_{perm}/K_p\right)^{0,5} - RQI$ is the index

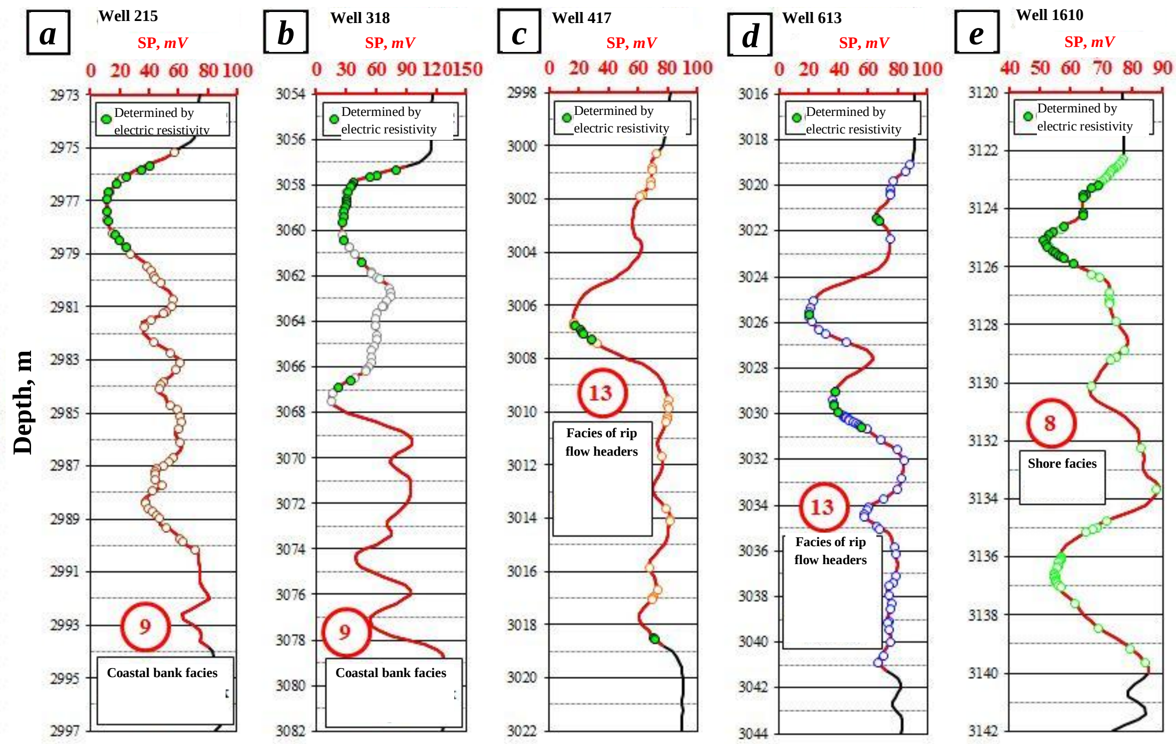


Fig. 2. SP data for Yu2 reservoir formation of wells 215 (*a*), 318(*b*), 417 (*c*), 613 (*d*) and 1610 (*e*) of Vyngapurovsky deposit.

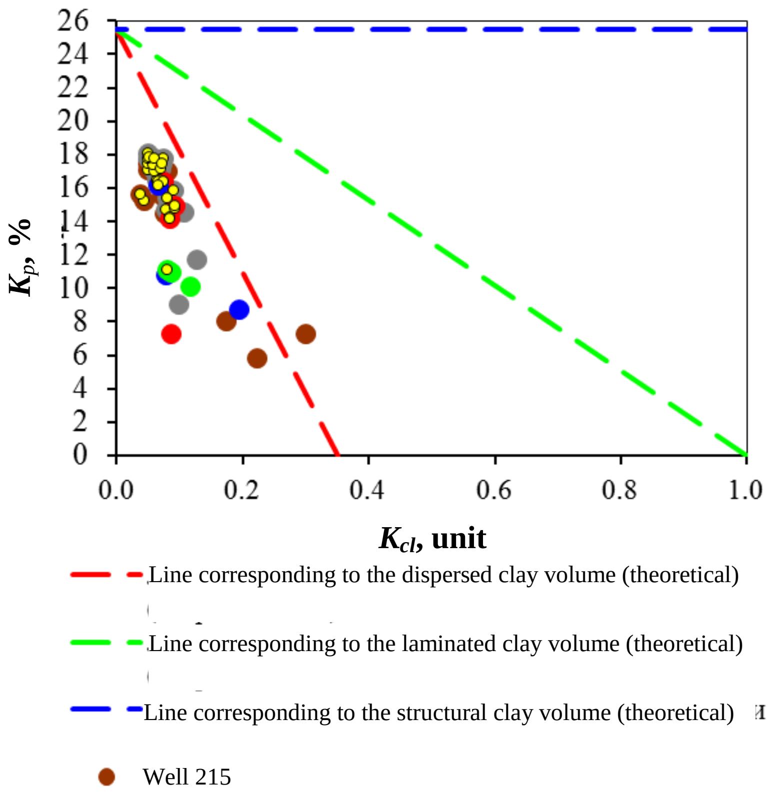


Fig. 3. Comparison of clay volume (K_{cl}) and open porosity (K_p)) for rocks of Yu2 reservoir formation of Vyngapurovsky deposit.

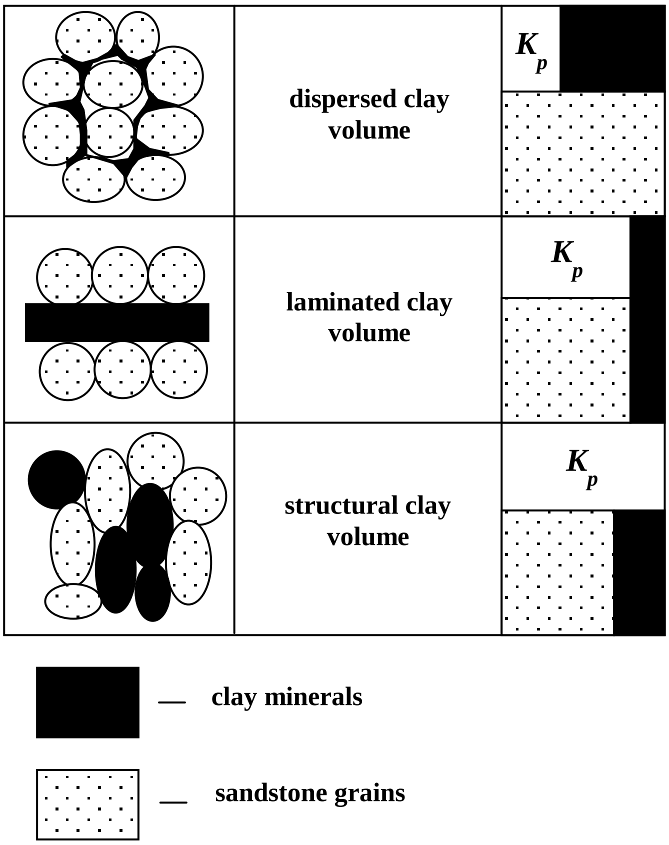


Fig. 4. Types of clay distribution in the clayey sandstone and their influence on porosity [3].

(indicator) of reservoir quality or I_{rq}.

Then, equation (2) will take on the following form;

$$I_{rq} = K_z F_{zi}, \tag{3}$$

where $K_z = K_p / (1 - K_p)$.

These conversions are required in order to evaluate, firstly, the reservoir quality, and, secondly, the values F_s, τ^2 and S^2_{gv}, on which the permeability and electrical resistivity of rocks largely depend [1].

Fig. 6 shows that in the rock samples of Yu2 reservoir formation where electrical resistivity is determined, dispersed clay covering grains mainly predominated (wells 215 and 318). The rocks of the same formation of wells 417 and 613 are distinctive, in that clay forming streaks predominate in the pore space. In the rocks of well 1610 there are two above types of clay distribution. With equal capacitive properties, rock with dispersed clay covering the grains transmits better electrical charges than rocks with dispersed type of clay in the pore space, forming streaks. This is confirmed by the graphs in Fig. 1. The exceptions are the rocks of well 1610, in which the high content of clay and carbonates exerts a decisive influence on the value of electric resistance of rocks (Fig. 7).

This inadequacy between electrical resistance, permeability and porosity is primarily due to unequal conditions of sedimentation in different parts of the Yu2 formation and subsequent lithogenetic transformations. The difference in sedimentation environment, in turn, has determined the regional heterogeneity of petrophysical properties within the reservoir under consideration. The consequence of this is a high variation in ratio R_p and K_p in comparison with different parts of the formation.

Reason No.2 – internal and external. An important point distorting the interpretation results of data R_p and K_p is often a lack of

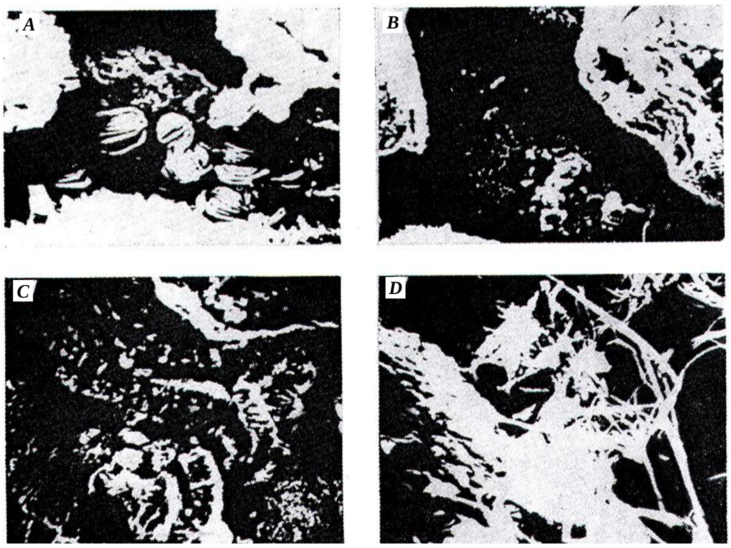


Fig. 5. Examples of distribution of clay minerals occurring in the pore space in the rocks in dispersed form according to V.H. Fertlu:

A – partial filling of the pore space by autologous kaolinite crystals; *B* – smectite film on the surface of the sandstone grains; *C* – authigenic chloride covers the surface of the grains and fills the pore space of rocks; *D* – filling the pore space of rocks by authigenic illite

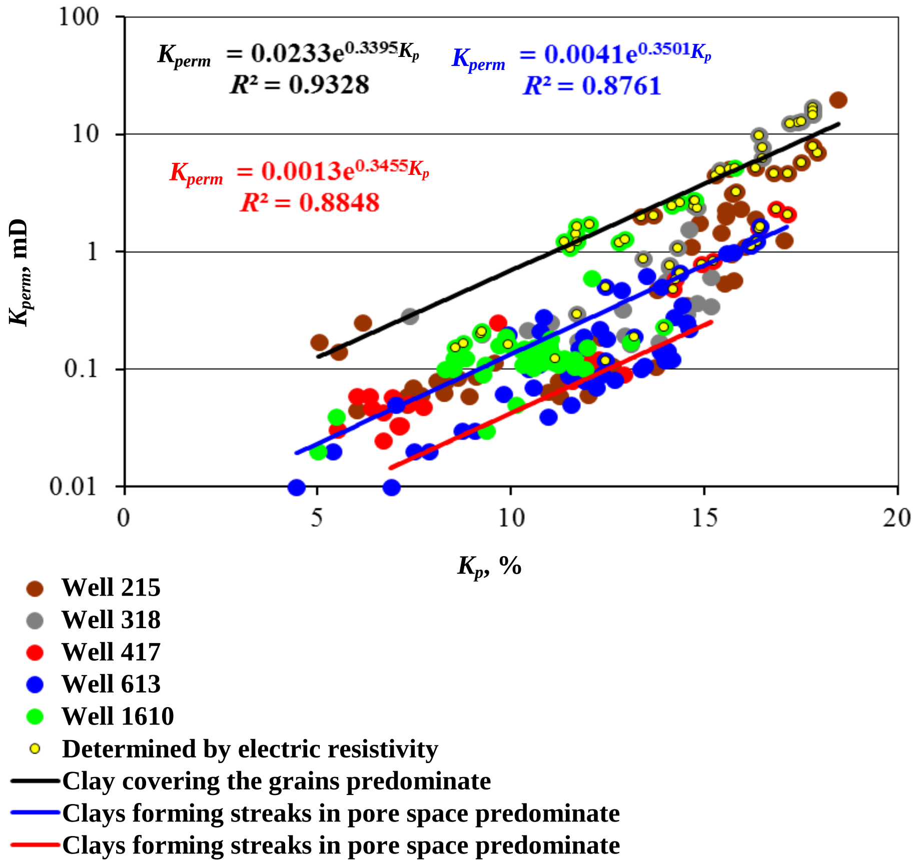


Fig. 6. Dependence between open porosity (K_p) and gas permeability (K_{perm}) for rocks of Yu2 formation of Vyngapurovsky deposit.

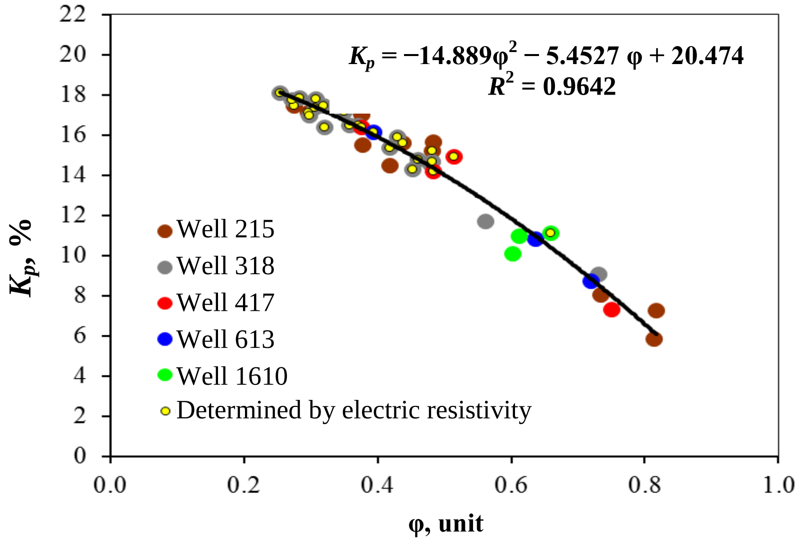
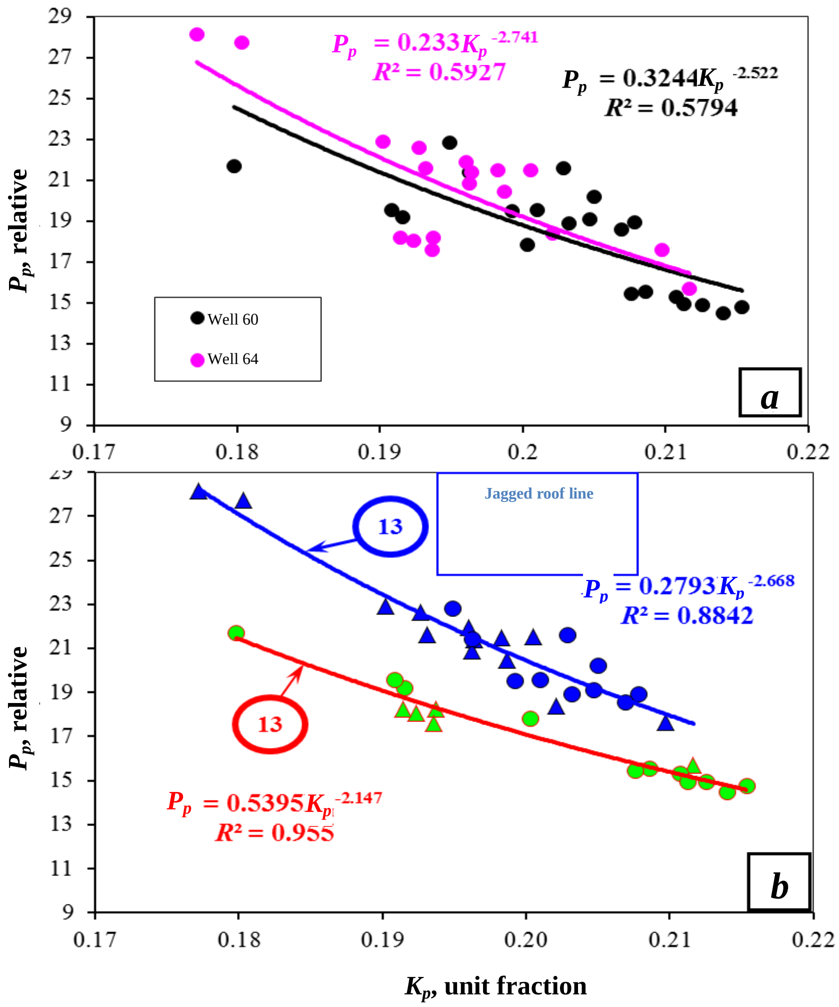


Fig. 7. Comparison of relative clay-carbonate content (φ) and open porosity (K_p) for rocks of Yu2 formation of Vyngapurovsky deposit.



In Fig. (*b*) – Well 64, clays covering grains and forming streaks in the pore space predominate
In Fig. (*b*) – Well 64, clays in the form of discrete particles predominate

In Fig. (*b*) – Well 60, clays covering grains and forming streaks in the pore space predominate
In Fig. (*b*) – Well 60, clays in the form of discrete particles predominate

Fig. 8. Comparison of the parameter of porosity (P_p) and open porosity (K_p): *a* – before and *b* − after analysis (Yaraynerskoe deposit, wells 64 and 60, formation BV6)

sampling system for the formation section for laboratory studies. Fig. 8 shows joint and separate comparison of R_p and K_p of BV6 formation of wells 60 and 64 of Yaraynerskoe deposit. Referring to Fig. 9, we can see that BV6 layer is represented by one geological body (GB) – facies of rip flow headers.

However, the samples selected for the study are not representative for well 60 or for well 64 – in the first well samples are taken generally from the area of maximum anomalies of SP and in the second from the average and minimum area. Areas in the minimum area of SP anomalies in both wells are characterized by different shapes, and that indicates the difference in the properties of rocks making up the formation. Although different parts of the reservoir were attributed to one facie, upon closer inspection it can be seen that in the final phase the sedimentation conditions of this facie often changed.

The upper part of the formation of deposits (serrate roof line) is characterized by a high content of clay and carbonate material (Fig. 10), which has a significant impact on the electrical conductivity of rocks. For formation BV 6 the definition of the type of distribution of clayiness of wells 60 and 64 was also determined (Fig. 11). It can be seen from Fig. 11 that dispersed clay predominates in BV6 formation. Fig. 12 shows that in the rock samples of upper part of BV6 formation (jagged roof line) where the electrical resistivity is determined, dispersed clay coating grains predominate, and in the middle part of the formation – clay in the form of discrete particles.

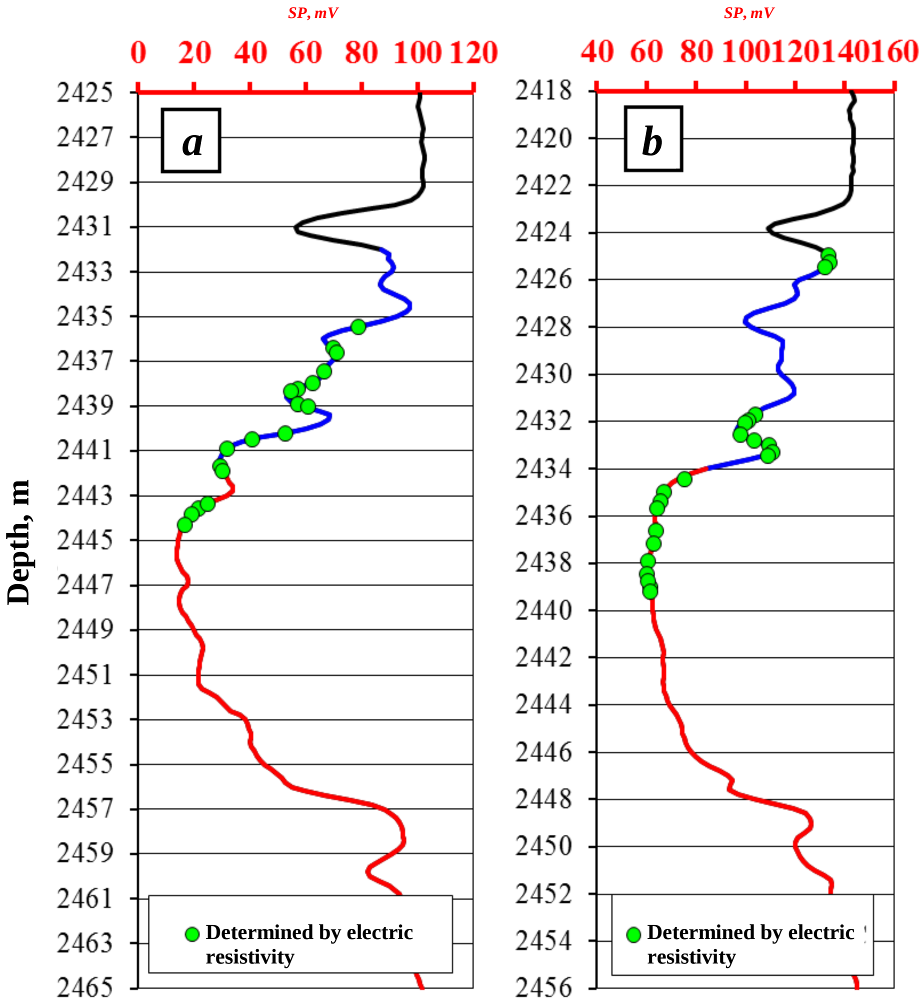


Fig. 9. SP data for rocks of BV6 formation of wells 64 (*a*) and 60 (*b*) of Yaraynerskoe deposit

With equal capacitive properties, rock with dispersed clay volume in the form of discrete particles has a better ability to pass electric charges compared with rock with dispersed clay volume covering the grains, as shown by the graphs in Fig. 8. In this case, we believe that the inadequacy of the relation between R_p and K_p has two causes:

1) absence of a representative set of samples (core);

2) neglecting the features (relationships) of formation forming conditions.

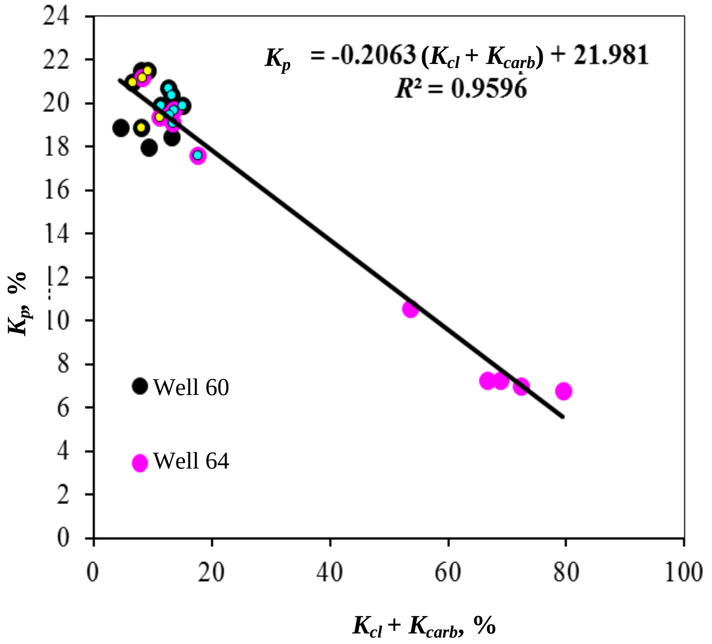


Fig. 10. A comparison of the total value of clay volume and carbonate content ($K_{\mathrm{cl}} + K_{\mathrm{carb}}$) with open porosity ($K_{\mathrm{p}}$) for rocks of BV6 formation of Yaraynerskoe deposit.

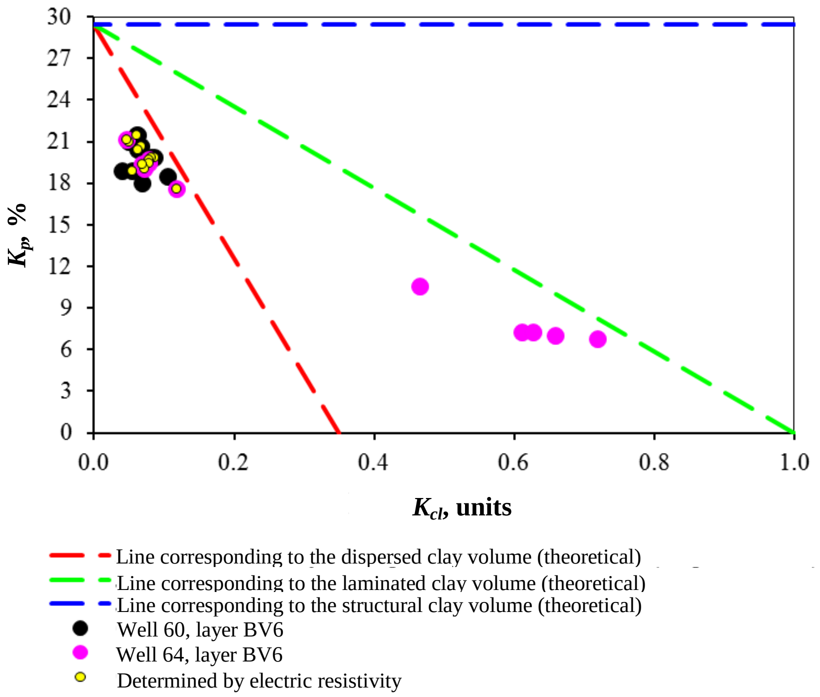


Fig. 11. Comparison of clay volume (K_{cl}) and open porosity for wells 60 and 64 of Yaraynerskoe deposit.

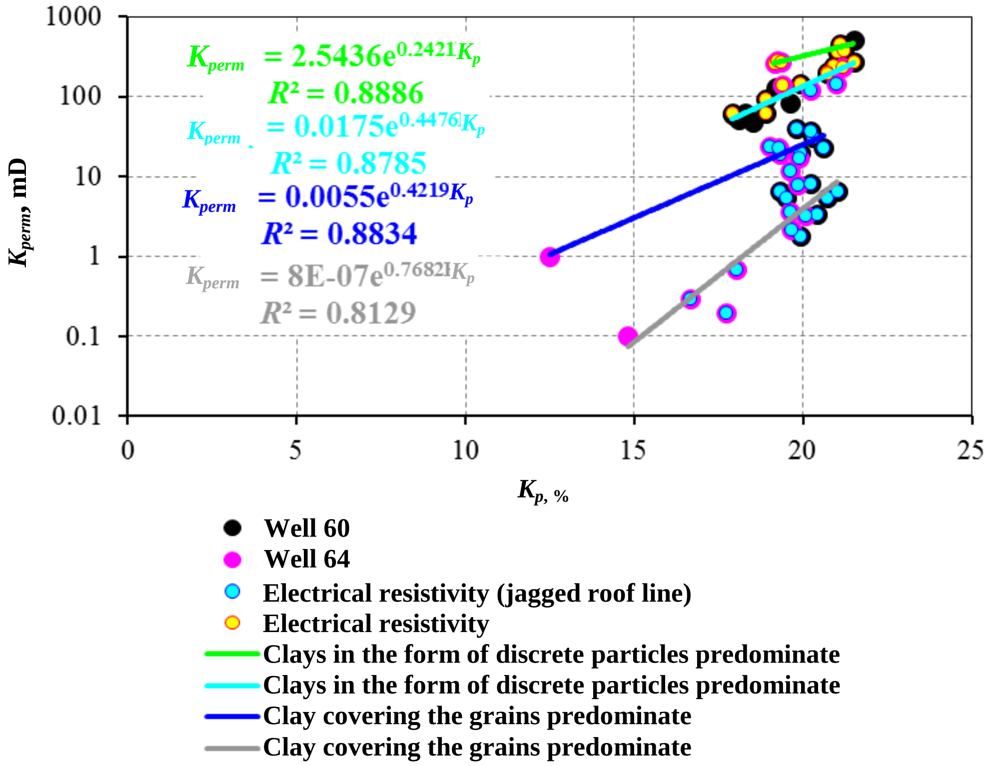


Fig. 12. Dependence between open porosity (K_p) and gas permeability (K_{perm}) of rocks of BV6 formation

CONCLUSION

On the basis of numerous data analyses of K_p, R_p and geophysical well logging we can conclude that it is inappropriate to process together petrophysical data and well logging data which were obtained in the GE representing different facies – geological bodies. It is clearly necessary to focus on where, on what basis and how to allocate the geological body (GB), i.e. to classify the GB – the geographical demarcation. It is reasonable to perform petrophysical modeling within the selected areas. In solving theoretical and practical problems of geophysics and petrophysics we cannot use only such information which only characterizes the object of general petrophysics. With a view to improve the reliability of the analysis results of petrophysical and geophysical data, it is necessary to make the best use of regional petrophysics attributes. It is necessary to develop a theoretical foundation for petrophysics, to form a systematic approach for solving various problems.

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Title:	**Analysis of the porosity parameter using elements of regional petrophysics (electrofacies) and clayiness type**
Author 1	Full name: **Ivan B. Ratnikov** Company: **Neftecom LLC** Work Position: **Deputy General Director for Science** Contacts: **IBRatnikov@mail.ru**
Author 2	Full name: **Roman S. Shulga** Company: **Siberian Research Institute of Petroleum Industry JSC** Work Position: **Head of the Department of Petrophysics** Contacts: **romshulga@mail.ru**

Author 3	Full name: **Evgeniy A. Romanov** Company: **Siberian Research Institute of Petroleum Industry JSC** Work Position: **Director of the Petrophysics Department** Scientific Degree: **Ph.D** Contacts: **romanovea@list.ru**
Author 4	Full name: **Sergei N. Bastrikov** Company: **Siberian Research Institute of Petroleum Industry JSC** Work Position: **General Director** Scientific Degree: **Ph.D.** Contacts: **bastrikov@sibniinp.ru**

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Abstract:	The most important calculation parameter, the open porosity ratio (K_p) is generally determined in petrophysical laboratories for the object (stratum) as a whole and without taking into account its regional heterogeneity. A range of the porosity value is taken as a basic criterion for selecting samples to construct the model $R_p = f(K_p)$. In this case it is assumed that the higher ΔK_p is, the higher the validity of the relation $R_p = f(K_p)$ is. However, in practice such a concept often creates an ambiguity in determining the porosity by electric resistivity of water saturated samples in different wells, even within one field. This phenomenon is caused by underestimating the features of the regional petrophysics object, the facies, which are reflected in the form of well logging curves, including SP, and the element of the general petrophysics object – the clayiness type. The breakdown (classification) of the formation by facies, determination and taking into consideration the type of clayiness, as well as distinguishing and analyzing the petrophysical types of rock based on rock porosity and permeability characteristics within the formation, make it possible to obtain satisfactory relations between electric resistivity and porosity of the rock.

Keywords:	open porosity, electric resistivity, facies, clayiness

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BORIS N. ABRAMOVICH (*Saint Petersburg Mining University*)
IRINA S. BABANOVA (*Saint Petersburg Mining University*)

ENERGY CONSUMPTION PREDICTION SYSTEM USING AN ARTIFICIAL NEURAL NETWORK

The article considers the possibility of increasing the energy efficiency of a mining enterprise by correctly choosing the price category (PC) and the electricity. The effectiveness of a forecasting model for energy consumption by a rational choice of PC is shown, and a system for predicting energy consumption using an artificial neural network is developed. The prediction error is 0.908% using the architecture network of a multilayer perceptron (MLP 24-18-1).

Keywords: energy management; artificial neural network; electricity tariff; price category; intelligent metering system; error prediction; architecture network; multilayer perceptron

The modern Russian electricity system is a set of interrelated consolidated electric power systems supplying electricity to the electrified part of the country and uniting generation and electric power transmission facilities. It should be noted that mineral resource sector enterprises are the most dynamically developing segment of the FPC (fuel-power complex) providing the required growth capacity for industrial production. On the one hand, being electricity market participants, they must comply with a number of requirements based on Russia's legislative framework and regulated electricity market, on the other hand, seek to minimize the costs related to the purchase of electricity by correctly choosing the PC and electricity tariff.

The crucial task for mining industry enterprises is to reduce production costs while increasing volumes and profit. Electricity is one of the most significant monthly costs for the enterprises. The final electricity price for the enterprise is formed from five indices (Fig. 1).

The electricity tariffs for the population and their equivalent consumer categories are fixed throughout the year and have three components (peak zone, intermediate zone and the rest of the day). For legal entities the situation is more complicated – they have to choose the tariff from six PCs for the calculations. The question arises: "Is it possible to raise the mining enterprise's energy efficiency by choosing the correct electricity tariff?" The decision is between: monetary economy of the long-term funds due to the rational choice of PC and the Automated Information and Measuring System of Commercial Energy Metering (AIMSCEM) or fixed costs when choosing the wrong tariff.

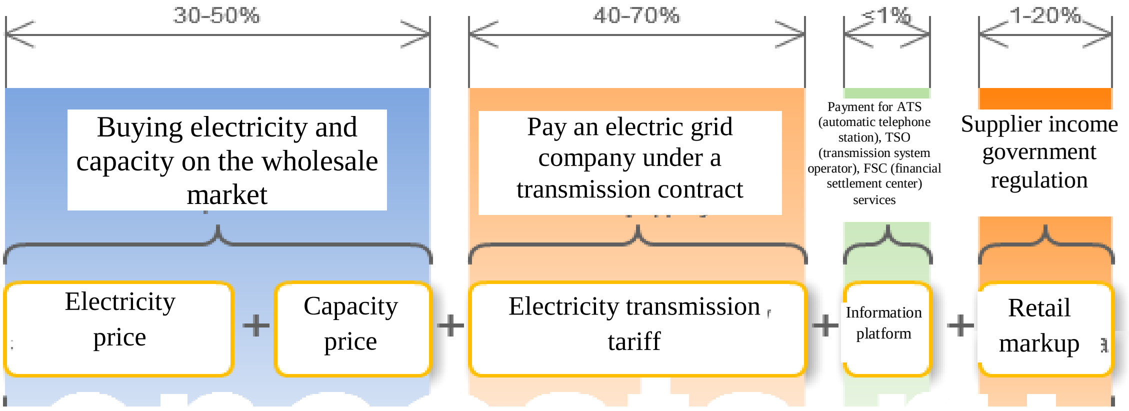

Fig. 1 Final electricity cost.

According to Russian legislation [1, 2], for legal entities, there are six free (non-regulated) prices PCs for electricity and capacity. For each PC, consumers are separated by the maximum capacity of power receivers, as well as the level of supply voltage.

The mining enterprise energy saving program is the result of an energy survey of production that makes it possible to develop a system of energy-saving measures and to assess commercial efficiency. Such a program includes low-cost, average cost, and high-cost measures. Energy saving requires the investment of certain financial assets and should be considered as one of the mining enterprise's investment activities.

The introduction of an AIMSCEM does not reduce the costs of energy resources. Experience shows that the installation of precision electronic electricity meters instead of induction ones may lead to an increase in electricity payments. However, a positive aspect of the AIMSCEM is the basis for the development and implementation of an energy saving measures system which can brings significant economic benefits.

In this regard, the following issues are relevant: energy consumption management for the mining enterprise; development of an electric power accounting monitoring system; development of algorithms of energy consumption functioning; increased reliability and accuracy for planned calculations by using artificial neural networks.

The choice of a rational electricity tariff with the development of an artificial neural network model made it possible to predict the power load [3], choose the optimal tariff for electricity payment, adjust both the electricity and capacity consumption (based on the predicted monthly chart of the enterprise), reduce the error prediction and electricity payment costs [4, 5].

The aim of the work was to increase the mining enterprise's energy efficiency by correctly choosing the electricity tariff with the subsequent development of an intellectual energy consumption prediction system.

Research objectives:

1) Identify the main factors influencing the choice of PC for energy and key performance indicators of the AIMSCEM system calculation.
2) Make a comparative analysis of the calculation of one-part and two-part tariffs for electricity payment.
3) Perform economic evaluation of project efficiency under different PCs.
4) Develop an intellectual power consumption prediction system using an artificial neural network for a rational electricity tariff.
5) Make a prediction for the day ahead and a prediction for monthly energy consumption for a mining enterprise using back propagation of the error algorithm [6].

The calculations were performed based on the example of a mining enterprise belonging to the subgroup III of consumers for power distribution, with power above 670 kW. The voltage level should be considered high taking into account the balance participation limits (BPL) and a 110/10 kV subtransmission substation.

The primary analysis of load profiles allowed us to make an assumption about choosing the best tariff based on finding a form factor. The enterprise can enter into an agreement with a supplier that guarantees from 3rd to 6th PCs taking into account the consumer subgroup III. The correct choice of PC and AIMSCEM system is not enough, it is necessary to create an intellectual electric account monitoring system using artificial neural networks (the ability to predict energy consumption with high precision).

To solve the task, the following were used: mathematical modeling and time series prediction methods, statistical and regression analysis, artificial neural networks theory, mathematical package STATISTICA, Matlab, expert analysis methods.

The main factors influencing the choice of PC for electricity and the development of the AIMSCEM system for a mining enterprise are: determining the power consumer subgroup by capacity (III, capacity from 670 kW to 10 MW; exclusion of I and II PCs from consideration); analysis of daily, weekly load and planned monthly schedule (calculation of basic indicators for the analysis of electric load schedules); price calculation 1 MVt/h by PC (the account of PC features and components for electricity: purchase on the wholesale electric power and capacity market (WEPCM); transmission services tariff; the guaranteeing supplier's retail markup; payment for the services of infrastructure organizations), a comparative analysis of the electricity cost different categories; comparison of one- (III and V PC) and two-part (IV and VI PC) tariffs.

The Key Performance Indicators of the AIMSCEM system calculation system are: energy savings from implementation for a year, capital investments, implementation period, operating costs, project performance indicators (for 5 years): net discounted net income (NPV); profitability index (PI); discounted payback period (DPB). The choice of PC and AIMSCEM system selection includes consideration of organizational, technological (system of transition to a different voltage, reserve reduction) and research (intellectual energy consumption prediction system based on neural networks, identification of risks for the AIMSCEM system) aspects of the issue.

Energy consumption prediction for a mining enterprise using artificial neural networks is shown in Fig. 2.

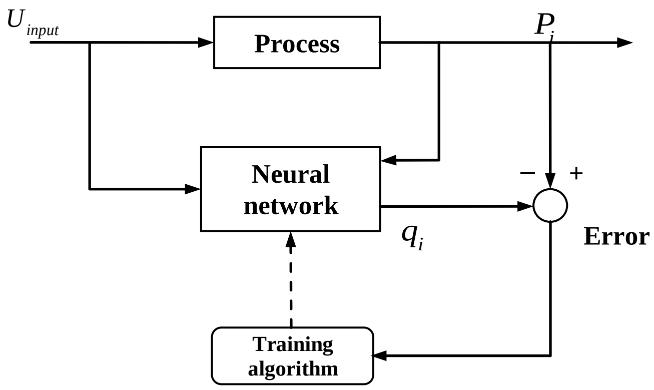

Fig. 2 Energy consumption prediction using ANN for a mining enterprise.

The task of energy consumption prediction can be regarded as constructing the function P depending on actual data on load, energy consumption history, temperature, lighting, humidity, precipitation, wind speed and direction:

$$P = F\big(t, \Delta t, n, k, P(t-\Delta t), ..., \\ ..., P(t), P(t-n\Delta t), ..., \\ ..., X_i(t), X_i(t-n\Delta t)\big). \quad (1)$$

The prediction P value should be in a δ given prediction confidential interval with a given probability belief p. The value k specifies the prediction type – short-term, etc. Where t is the current time, Δt is the time interval between the measurements; n is the number of intervals in the past, k is the number of intervals in the future, m is the number of the measured characteristics; X_i at $i = 1, ..., m$ is the measured characteristics included in the list of historical influencing factors (except for the consumed power itself these are the temperature T and humidity w, the length of the day, etc.). The depth in the past n and the list of independent variables or influencing factors X_i should also be determined.

The function depends on the independent variables taken as factors that influence the value of energy consumption for which we have reliable evidence: daily schedules of meteorological parameters, temperature, lighting, humidity, precipitation, wind speed and direction.

Based on the function (1), an energy consumption prediction algorithm was drawn up based on the operation of the multi-layer perceptron shown in [7], the use of artificial neural networks in the tasks of energy consumption prediction, as well as modeling and prediction in works [8, 9]; the study of short-term prediction of electrical load using the artificial neural networks [10–12].

Fig. 3 shows an algorithm for constructing a neural network for diurnal prediction. This algorithm is applicable for both weekly and monthly prediction.

As part of the research, back propagation of the error algorithm was used to predict the energy consumption of a mining enterprise; factors that influence power consumption were taken into account.

In blocks *1–3* the initial conditions for the neural network were started and installed (weighting values, number of training samples NP, ANN parameters, the given small value e which determines the prediction accuracy), the values of the electrical load $P_1, \dots, P_{24}$ were read, and then the conversion process was performed in the relative values Y_i that are within the range $0 \le Y_i \le 1$ where $1 \le i \le 24$).

In block 4, the calculation of the values of the signals on the inputs and outputs of the hidden layer j and output layer neurons k was made using the following formulas:

– j layer neurons inputs

$$net_j = \sum_{i=1}^{24} w_{ji} Y_i, \quad j = 1, 2 \dots, 5; \tag{2}$$

– j layer neurons inputs

$$Y_j = 1 / \left(1 + e^{-\left(net_j + \Theta_j\right)}\right); \tag{3}$$

– k layer neurons inputs

$$net_j = \sum_{i=1}^{5} w_{kj} Y_j \ , k = 1; \tag{4}$$

– k layer neurons inputs

$$\left(P_{prog}\right) Y_k = 1 / \left(1 + e^{-\left(net_k + \Theta_k\right)}\right). \tag{5}$$

Here w_{ji} and w_{kj} are the weight coefficients respectively between the neurons of j-rd and i-th layers and k-th and j-th layers; Θ – bias. To limit the search space during training, the objective error function found by the least square method [6] is minimized:

$$E_p = \frac{1}{2} \sum_{k=1}^{KN} \left(d_k - Y_k\right)^2, \tag{6}$$

where d_k is the desired value of the load at the output, Y_k is a calculated value, KN is the number of neurons in the output layer.

Then the gradient descents in the scales space are calculated w_{ji} and w_{kj}, and scales adjustment is made.

The following characteristics were obtained on the basis of gradient descents calculations: h – training speed ratio; a – moment determining the training acceleration. In this algorithm $h = 0.3$; $a = 0.61$; $e = 10^{-6}$ (chosen according to the prediction error of minimization criterion).

Block *6* determines whether all samples were used. If all, then the total error for all samples in block *7* is calculated, and the condition in block *8* is checked. If the condition is fulfilled, the training process ends, otherwise the process repeats.

To solve the energy consumption prediction task, the back propagation of error algorithm was used to minimize the root-mean-square deviation of the current output of multilayer perceptron and the desired output.

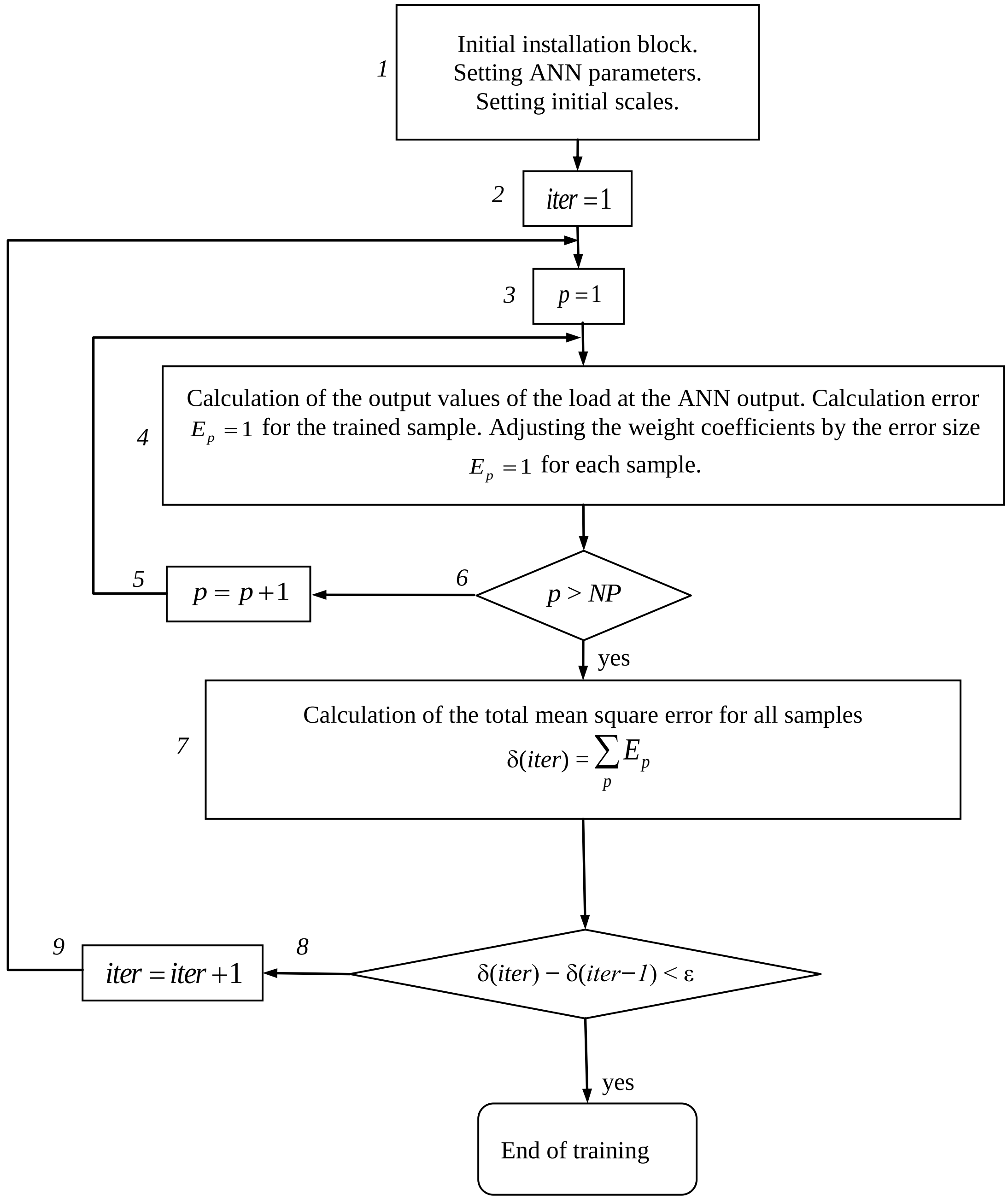

Fig. 3 Back propagation of error algorithm.

Prediction at the system output is compared with the actual load value, and, when the error exceeds a predetermined allowable level (for example, over 2 %), the system re-retrains using the new data.

Table 1 shows the main characteristics of the obtained predictive neural network for the prediction of energy consumption.

Training performance/Control performance/Test performance: network performance on the selections used for training. The value of the network performance depends on the type of output variable network.

Training error/Control error/Test error: network error on the selections used for training. This is a number that is actually optimized by the training algorithm. It is a root-mean-square value of errors under certain observations where the individual errors are calculated by the network error function which is a function of the observed and expected levels of the output neuron activation. The error in the test selection is interpreted as a model error prediction accuracy measure.

Training. This field contains a brief description of the training algorithms used for network optimization. It stores several codes, followed by the number of epochs performed by the algorithm, and an optional stop code which shows how the final network was selected. OR91b code shows that the backpropagation method was used, that the best found network was chosen (by the "best” that is the "minimum test error"), and that this network was found at the 91st epoch.

Prediction research of the best prediction models for an artificial neural network with activation function obtained, the hyperbolic tangent, MLP 24-18-1 architecture, the prediction error was 0.906 % without taking the factors into account.

In addition, the economic evaluation of the project effectiveness for various PCs taking into account the main factors of the final cost of electricity was determined: price (range of transformation voltage, power distribution categories, pricing zone, maximum capacity); schematics of the enterprise power supply from which the voltage level is dependent; the BPL affecting the formation of the weighted average price of the electricity; AIMSCEM system installation with the calculation of project payback cost.

The schedule of the enterprise site load for a typical day is shown in Fig. 4.

Comparative analysis of one-part and two-part tariffs under the per-unit system for a mining enterprise by price categories is shown in Figure 5. As a base price, the cost of 1 MWh for the third PC was taken.

Comparative analysis showed that the cost of 1 MWh is higher for the two-part tariffs (IV and VI PC) than for the one-part tariffs (III and V PC). Fig. 5 shows that for the one-part tariff for the 3rd PC, the payment cost for 1 MWh is minimal. Nevertheless, will such a tariff be optimal throughout the year subject to changes in energy consumption for this enterprise? Depending on the payment for the electricity chosen by the PC consumer, the consumption pattern and the ability to plan the final cost of energy consumed per month can vary by 50%. Consideration should be given to the specifics of the price categories:

– the third PC provides for the payment for transmission services under the one-part tariff; it does not require hourly planning of electricity consumption;

– the fourth PC provides for the payment for transmission services under the two-part tariff; it does not require hourly planning of electricity consumption;

– the fifth PC provides for the payment for transmission services under the one-part tariff (like the third one); it requires hourly planning of electricity consumption;

– the sixth PC provides for the payment for transmission services under the two-part tariff (like the fourth one); it requires hourly planning of electricity consumption;

Therefore, to choose the best PC, two questions must be answered: What type of boiler transmission tariff is more beneficial? Can a mining enterprise carry out detailed planning of hourly consumption?

Table 1

NN model result

Architecture	Training performance	Control performance	Test performance	Training error	Control error	Test error	Training/ Elements
MLP 24-18-1	0.169209	0.182035	0.141871	0.00906	0.0520651	0.0524561	OP91b

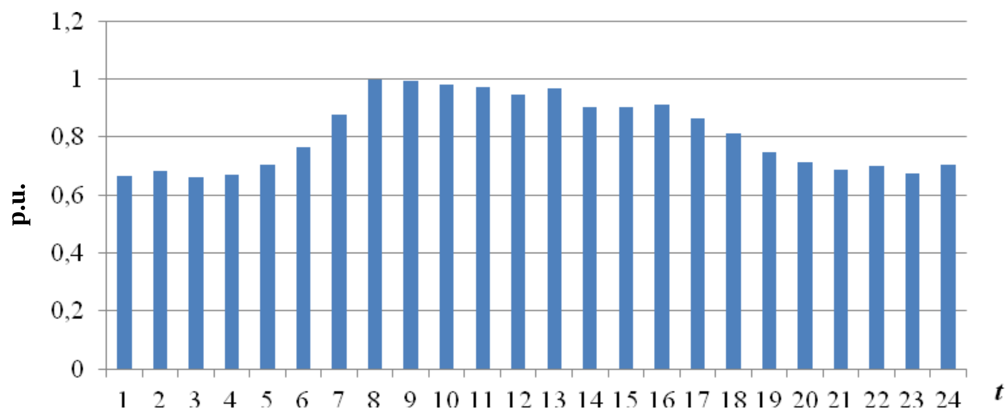

Fig. 4. Schedule of the mining enterprise site load on a typical day.

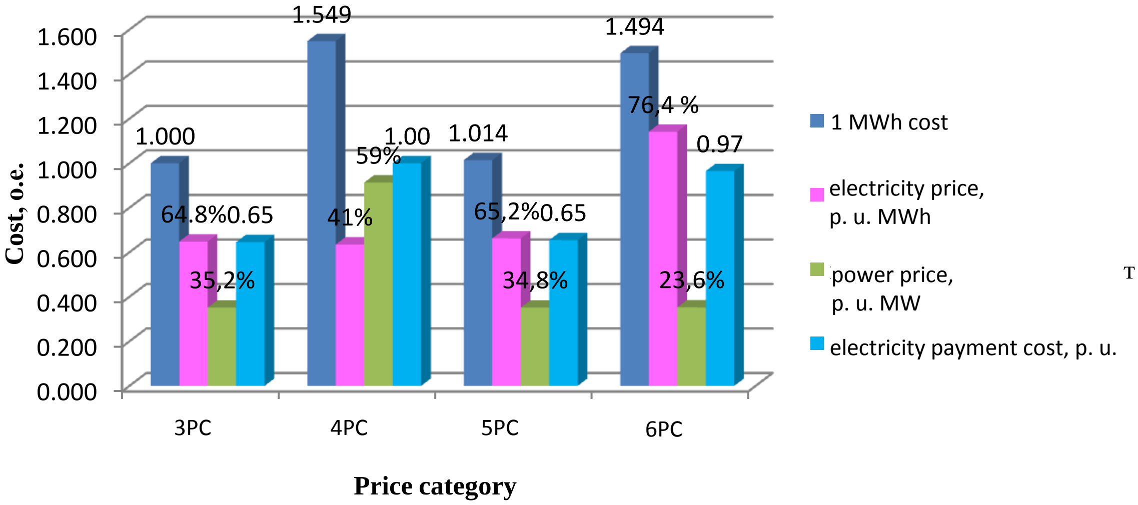

Fig. 5 Comparative analysis of the electricity cost in various price categories

The PC choice economic efficiency estimation taking into account the costs of AIMSCEM system is shown in Fig. 6.

The graph shows that the choice of prospective price category is effective for the enterprise for the 5th and 6th PCs, taking into account the calculation of the economic indicators of the project. Negative values NPV, P_i will make it possible to judge the ineffectiveness of the PC choice, the deviation of the 4th PC choice because the $DPB = n = 5$, project will not yield either profit or loss, therefore the project should be rejected. For the 6th PC the profitability index P_i and net present value (NPV) is higher than for the 4th and 5th PCs.

The development of an energy consumption prediction system using an artificial neural network was carried out according to the algorithm described above. The graphs shown in Fig. 7 were obtained.

The input variables to create the neural network were the hourly load values P_i $(i = 1,...,24)$ for the day preceding the predicted one (24 values) and for the day a week ago (24 values). In the input layer they get rated, that is, converted to the relative values. In the output, 24 values of the predicted value (a day ahead) are obtained. Results for diurnal prediction: the error of the best neural network was 0.908 %, multilayer perceptron type, MLP 24-18-1 architecture, hyperbolic tangent activation function.

The prediction error can be reduced by setting additional parameters on the network input: day status (weekday, weekend), average temperature value [13]. When taking into account these parameters, the relative error of the neural network was reduced to 0.799 %.

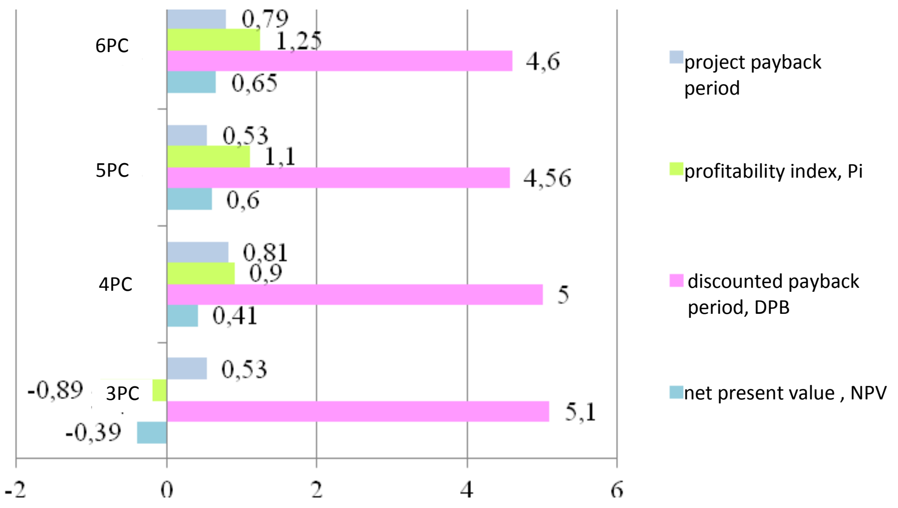

Fig. 6. Calculation of the project economic efficiency.

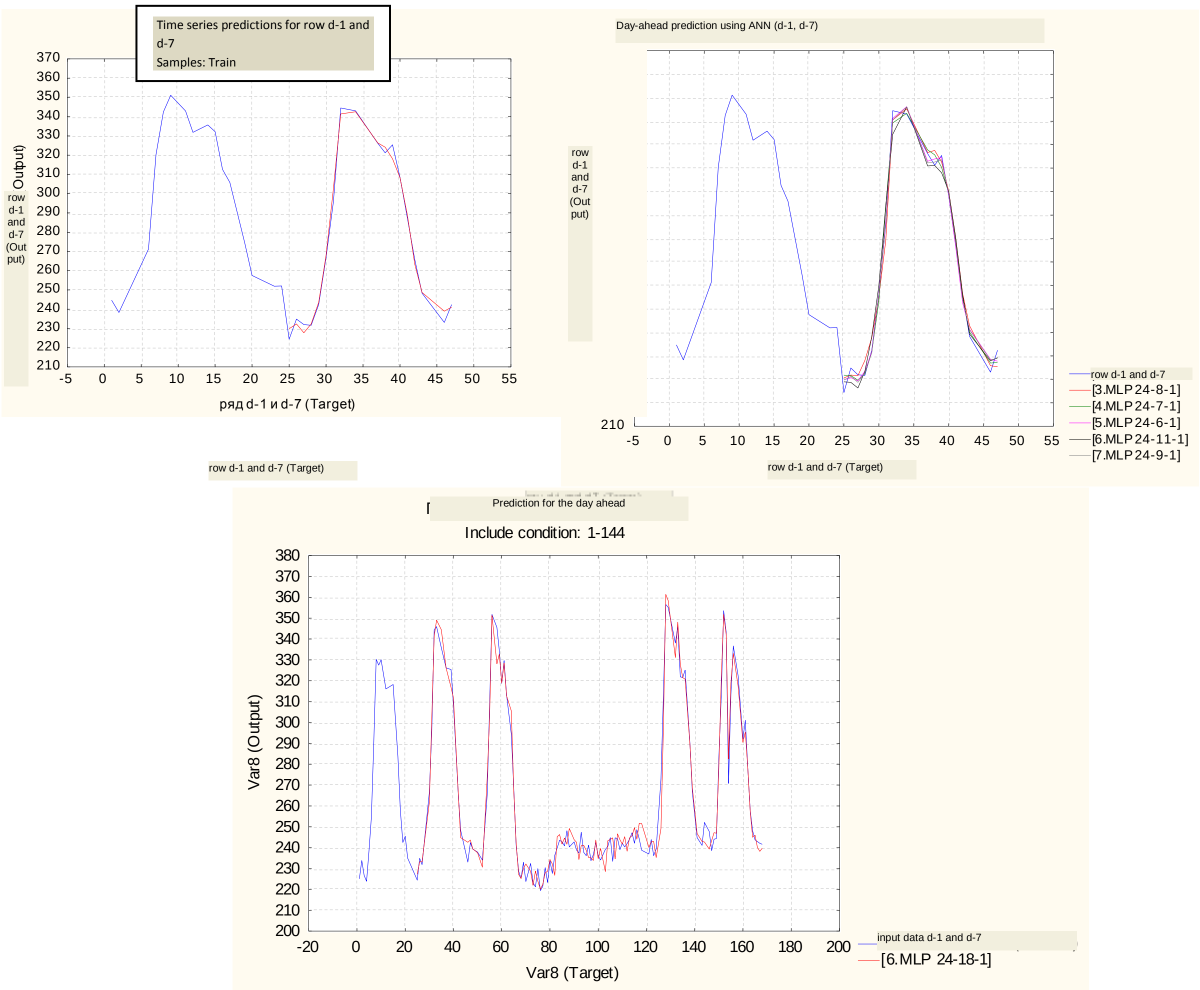

Fig. 7. Day-ahead prediction graphs using MLP 24-18-1 for mining enterprise.

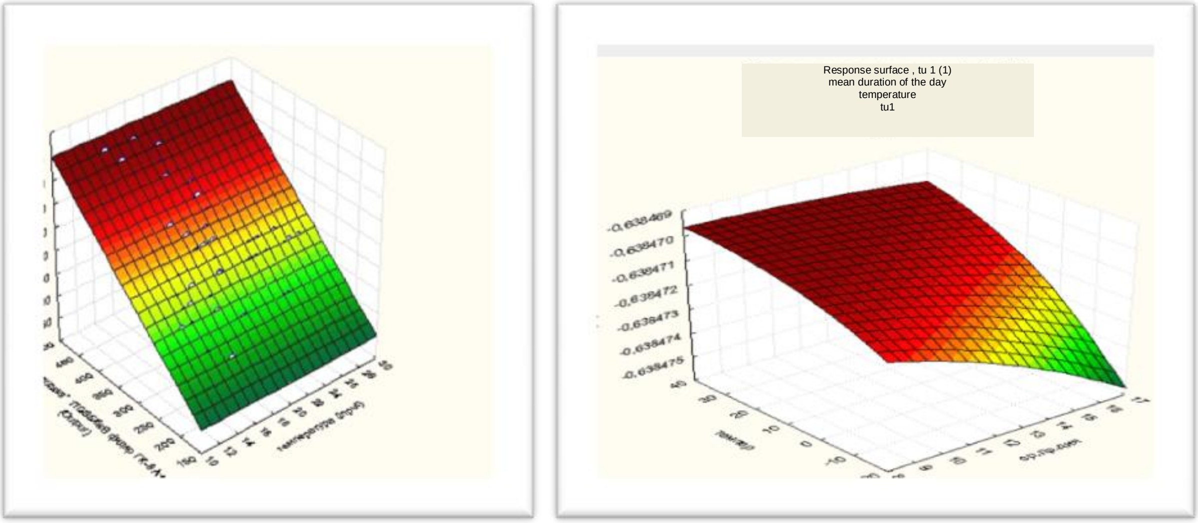

Fig. 8. Level of element activation for day status (weekday, weekend) and temperature.

The response surfaces of the dependent variables for energy consumption prediction were obtained. The element activation level is the weighted sum of its inputs with the addition of thresholds. Therefore, the activation level for the neural network is a simple linear function of inputs. Further, this activation is transformed by a sigmoid (*S*-shaped) curve.

CONCLUSION

It is possible to improve energy efficiency for a mining enterprise through the rational choice of PC and tariff for electricity costs with the creation of an intellectual electricity accounting monitoring system through the introduction of prediction functions based on the work of artificial neural networks. The cost of 1 MWh for two-part tariffs (fourth and sixth PCs) is higher than for one-part tariffs (third and fifth PCs). However, the greatest reduction in payment for electricity is observed in the application of tariffs (fifth and sixth PCs) where it is possible to predict the schedules of electrical load, thereby reducing the maximum power at peak hours (reduced power payment component).

The use of artificial neural networks makes it possible to predict the energy consumption of mining enterprises on the basis of the data for a weekly load, the data on energy consumption in the last month, air temperature, day status (weekday, public holiday, pre-holiday day). The development of the intellectual energy consumption system for a mining enterprise makes it possible to choose a rational PC, thereby reducing electricity costs.

Introducing such a system will allow to optimally plan and monitor the fulfillment of the enterprise load schedule; predict the production cycles and peak energy consumption values; the possibilities of redistribution of load and analyze the possibility of changes in the enterprise's operating schedule.

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Title:	**Energy consumption prediction system using an artificial neural network**
Author 1	Name&Surname: **Boris N. Abramovich** Company: **National Mineral Resources University** Work Position: **Professor of Department of electric power engineering and electromechanics** Scientific Degree: **Doctor of Engineering Science** Contacts: **babramov2bn@mail.ru**

Author 2	Name&Surname: **Irina S. Babanova** Company: **National Mineral Resources University** Work Position: **Department of electric power engineering and electromechanics** Scientific Degree: **PhD student** Contacts: **irina_babanova@mail.ru**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-2-66-77**
Abstract:	The article considers the possibility of increasing the efficiency of a mining enterprise by correctly choosing the price category and tariff for electricity. The efficiency of a forecasting model for energy consumption by a rational choice of price category is shown, and a system for predicting energy consumption using an artificial neural network is developed. The forecast error is 0.908% with the architecture of the network type MLP (MLP 24-18-1)
Keywords:	energy management; artificial neural network; electricity tariff; price category; intelligent metering system; error prediction; architecture network; multilayer perceptron
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