

**INNOVATIVE DEVELOPMENT  
OF RESOURCE-SAVING  
TECHNOLOGIES FOR MINING**

Multi-authored monograph

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The multi-authored monograph written by leading mining researchers suggests a complex approach to considering resource-saving technologies in mining by comprising all stages of the process from mining to recultivation of lands disturbed by mining operations.

The book targets mining scholars, practitioners, lecturers, postgraduates and mining students.

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## PREFACE



We are glad to present the multi-authored monograph “Innovative development of resource-saving technologies in mineral mining”.

The publication aims to generalize modern approaches to improving resource-saving technologies in mining. There are presented scientific developments by authors from different countries. Various directions of mining – from analysis of modern approaches to optimization of geological and economic estimates of mineral deposits to solving problems of land reclamation – are comprised.

The authors consider issues of resource-saving technologies applied to mining iron and uranium ores, hard and brown coal, amber as well as to developing oil and gas deposits. The monograph papers are united by a common aim of applying a complex approach to improving mining technologies and rational use of mineral resources by developing and introducing resource-saving technologies.

The scale of the problems mentioned here implies further consideration in order to calibrate a more precise picture of resource saving technologies application in mining that cannot be unanimously defined and described. Therefore, the authors are going to continue their research and present it in subsequent publications.

We hope that the multi-authored monograph will be of interest to scientists and lecturers of mining educational and research institutions, practitioners and students whose major is mining.

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# MODERN APPROACHES TO OPTIMIZATION OF THE ECONOMIC-GEOLOGICAL EVALUATION OF SOLID MINERAL FIELDS IN UKRAINE

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**Abstract.** The paper characterizes modern approaches to optimization of the economic-geological evaluation of solid mineral fields during the state expert appraisal of materials of the economic-geological evaluation of mineral deposits based on the Mineral Reserves and Resources Classification of the State Subsoil Fund of Ukraine, taking into account the world practice of mineral deposits evaluation, and proposes modern classification categories and subcategories.

*The purpose* of the economic-geological evaluation of mineral reserves and resources at all stages of geological exploration works is determination of economic feasibility of their industrial development.

*The object* of the economic-geological evaluation is mineral reserves and resources of a subsoil plot provided or supposed for rendering in use for the purpose of geological study or economically effective minerals production and use.

For a detailed characteristic of approaches to the economic-geological evaluation of mineral reserves and resources of solid mineral fields the authors have chosen their representative types: iron ores, gold and amber covering a range of processes of mineral deposits formation from magmatic and metasomatic to sedimentary ones.

**Introduction.** The main requirement for geological exploration works and industrial development of mineral deposits comes down to

ensuring their maximum efficiency and achievement of maximum results at the minimum time, labour and capital costs. On the subsoil owner's part the main requirement consists in the fullest and rational use of the available mineral and raw-material base.

Evaluation as determination of quantity and quality of minerals and components is carried out at all stages of geological exploration works.

Prognostic and prospective resources are evaluated at the stages preceding the prospecting, and mineral reserves are evaluated by the results of prospecting stages. Such an evaluation can be called ***geological***.

In addition, explored reserves of mineral raw materials are subject to the feasibility study by which the technical (technological) and economic feasibility of their production, processing and selling of sales production is determined. Thus, the evaluation of mineral reserves of a subsoil plot acquires attributes of the complex ***economic-geological evaluation***.

*Geological* evaluation is based on the results of performed geological exploration works which are generalized in the process of evaluation and calculation of mineral reserves and resources.

In the course of preparation of source materials for reserves calculation a geological model of a field or a subsoil plot is created according to the available detailed study of geological and structural formation, occurrence modes, morphology and an internal structure of mineral deposits, their substance composition, regularities of spatial location and a ratio of natural and technological types of mineral raw materials, and also surrounding rocks within the natural boundaries of development of mineralization. While determining the commercial significance of the accessed productive deposit a set of requirements for quantity of reserves and quality of mineral raw materials, technological properties, mining and geological conditions for mineral extraction as well as economical and geographical conditions for exploitation of the field enabling its profitable use as a mineral and raw-material base of the future enterprise is considered.

The application of conditions for natural geological models of productive deposits predetermines their certain changes due to which a geological model of deposit is transformed in the process of delineation of estimation blocks of reserves into its geological

production model.

Economic evaluation of fields determines a prognostic economic effect of the use of explored reserves. It is based on the results of reserves calculation and contains a cumulative analysis of factors which determine economic profitability of field exploitation and efficiency of capital investments in the construction of an industrial complex by calculating a cash flow.

Geological and economic evaluations of fields are interrelated and almost inseparable. If reserves are calculated on the basis of conditions justified according to the economic evaluation of efficient use of explored reserves, the exploitation reserves and quality of a mineral in sales production are taken as a basis of calculations of this efficiency. Trade terms of sales production and taxation of the production activity of mining enterprises are determined by the appropriate legislation of Ukraine. Interrelation and correlation of the geological and economic evaluations of mineral deposits (plots) predetermine a complex character and certain complexity of the evaluation methods. Therefore, although a basis of economic-geological evaluation of mineral deposits is the results of performed geological exploration works and exploitation, it can be carried out at a high-quality level only by joint efforts of experts in geology, mining, technology, metallurgy as well as economy, ecology and law.

**Main body.** The main instrument of state management of the mineral and raw-material base of Ukraine, economic-geological evaluation and state accounting of mineral reserves according to a level of their commercial significance, a level of their geological and feasibility studies and preparedness for industrial development is the Mineral Reserves and Resources Classification of the State Subsoil Fund of Ukraine approved by the Resolution of the Cabinet of Ministers of Ukraine dated 05.05.1997 No. 432 [4].

Economic-geological evaluation of mineral reserves and resources is carried out by the results of performed geological exploration works on search, search evaluation and prospecting of the respective objects. Reserves and resources of main and accompanying minerals are approved on the basis of the state expert appraisal of reports submitted to the SCMR which contain materials on the geological study of mineral fields (plots), calculation of their



reserves and feasibility study of their commercial significance [1-6].

A determining part of the economic-geological evaluation of mineral deposits is justification of *conditions* for mineral raw materials as a set of limiting requirements for quality and quantity of mineral raw materials in subsoil, mining and geological modes of occurrence, mine technical and other conditions of development of productive deposits, requirements for completeness and complexity of fields try-out, requirements for environmental protection measures. The observance of conditions at the time of calculation and industrial field development provides the most complete and rational use of available mineral reserves and resources.

The Mineral Reserves and Resources Classification of the State Subsoil Fund (hereinafter referred to as the Classification) establishes the principles of calculation, economic-geological evaluation, state accounting, and reporting on the use of mineral reserves and resources unified for the State Subsoil Fund of Ukraine according to a level of their economic and commercial significance (E axis), a degree of feasibility study and preparedness of mineral deposits for their further use (F axis), and also a degree of geological study and confidence (G axis).

The current Classification contains groups of reserves and resources by the following list of indicators:

1. By a degree of geological study.
2. By a degree of feasibility study.
3. By commercial significance.
4. By production and use conditions.
5. By a degree of preparedness for industrial development.
6. By geological structure complexity.

A tendency to international integration in the sphere of global economic activity intensified in the last decades made a need for creation of a uniform global system of mineral reserves classification more pointed. It became a major driving force for elaboration of the International United Nations Framework Classification for Fossil Energy and Mineral Reserves and Resources (hereinafter referred to as UNFC-2009). UNFC-2009 was developed according to the international mandate provided by the UN Economic and Social Council, the Ad-Hoc Group of Experts of the Economic Commission for Europe.

UNFC-2009 is a universal system in which the quantities of minerals are classified on the basis of three fundamental criteria: a level of project socio-economic viability (E), a degree of preparedness of a project for industrial development/project feasibility (F) and a degree of confidence of geological study (G) with the use of a numerical coding system.

The UN Framework Classification contains information on:

- 1) a stage of geological evaluation;
- 2) a stage of evaluation of development feasibility;
- 3) a degree of economic efficiency.

The system of grouping (categorization) of mineral reserves/resources based on the results of stage-by-stage geological exploration works performed according to the standard branch practice in all countries with the developed mining industry enables to apply the United Nations Framework Classification to all types of minerals and mineral raw materials. Terms and definitions currently used in the existing classification systems can be easily associated with the relevant evaluation stages in the United Nations Framework Classification enabling to keep national terminology and to provide its comparability. Main categories of the United Nations Framework Classification can be divided into subcategories at a national level, if required, for the purpose of considering certain needs that adds necessary flexibility to this classification system.

The confidence of geological study is divided into four levels, including: for the known field (deposit) - a high confidence level (G1), a moderate confidence level (G2) and a low confidence level (G3), and for the potential field a level of geological study is determined by probability of its identification (G4). On the basis of these criteria it is convenient to create four categories reflecting the increasing level of geological probability of mineral reserves and resources.

Feasibility study and preparedness for mineral exploration includes four degrees ranged by decrease in justification, including: feasibility of extraction is confirmed with mining operation (F1), feasibility of extraction is subject to further evaluation (F2), limited technical data for feasibility of extraction (F3), no development project (F4).

Ukraine is one of the first countries of the world that took the

International United Nations Framework Classification for Fossil Energy and Mineral Reserves and Resources as a basis for its mineral reserves and resources classification of the State Subsoil Fund. Since 2007, transition to the use of the United Nations Framework Classification adapted to the conditions in Ukraine is envisaged.

The draft Mineral Reserves and Resources Classification of the State Subsoil Fund of Ukraine corresponding to UNFC-2009 determines groups of mineral reserves and resources by the following classification indicators:

- by a degree of geological study of mineral reserves (explored and preliminary explored) [4];
- by a degree of geological study and confidence of mineral resources (prospective and prognostic);
- by a degree of feasibility study the mineral reserves and resources are divided depending on detailed economic-geological evaluation and feasibility studies of subsoil plots.

According to the Classification there are detailed, preliminary, and initial economic-geological evaluations:

- **detailed economic-geological evaluation (EGE-1)** is determination of a level of economic efficiency of production activity of the mining enterprise which is created or reconstructed, and expediency of investment in works on its design and construction. EGE-1 is carried out on the basis of explored and preliminary explored mineral reserves and includes a feasibility study (FS) of permanent conditions for their calculation. Detailed technical and economical calculations, reliability of EGE-1 financial indicators shall be sufficient for making an investment decision without additional investigations. The materials of the detailed economic-geological evaluation of a mineral deposit evaluated positively by the SCMR are a main document that justifies the expediency of financing of works on try-out of construction projects of the mining objects;

- **preliminary economic-geological evaluation (EGE-2)** is justification of feasibility of industrial development of a mineral field (deposit) and investment in geological exploration works on its prospecting and preparation for exploitation. EGE-2 is carried out on

the basis of preliminary explored and explored mineral reserves, and executed as a feasibility study report (FSR) on the expediency of further exploration, including research and industrial field (deposit) development. At the same time the efficiency of field development is evaluated at a level of end mining sales production; feasibility indicators are determined by calculations or taken by analogy;

- **initial economic-geological evaluation (EGE-3)** is justification of the expediency of investment in prospecting works on the plots promising as regards discovery of mineral fields. EGE-3 is carried out on the basis of preliminary explored reserves and quantitative evaluation of mineral resources and is submitted in the form of feasibility considerations (FC) on their possible commercial significance. Evaluation of the possible industrial development of foreseen mineral deposits is justified by aggregative technical and economical calculations on the basis of a proved analogy to the known commercial fields or a technical assignment given by a customer of geological exploration works.

Mineral reserves and resources are divided into categories and subcategories **by a level of economic, social and commercial significance** (Table 1).

A category of balance reserves includes quantities of minerals for which at the moment of carrying out economic-geological evaluation according to technical and economical calculations and/or materials of financial reporting it has been proved that a profitability ratio of production of the mining enterprise (estimated and/or actual) is sufficient for economically effective extraction of minerals from a subsoil plot and selling of end mining sales products. Code designation (hereinafter referred to as the code) of a category of balance reserves according to UNFC-2009 is **E1**. According to the terms of production and sale it is possible to determine subcategories - recoverable and subsidized balance reserves among the balance reserves by such criteria:

- for recoverable reserves (**E1.1 code**) - profitability of production activity of a mining enterprise (mine) determined by the SCMR exceeds a refunding rate of the National Bank of Ukraine subject to the rational use of technical facilities and technologies and the observance of requirements for subsoil and environmental

protection;

- for subsidized reserves (**E1.2 code**) - efficiency of extraction and use of minerals by a mining enterprise (mine) under design determined by the SCMR is possible only subject to granting to a subsoil user tax benefits, privileges, subsidies or other kinds of support at the expense of state or local budgets. Subsidized reserves of mineral deposits are accounted in the State Mineral Reserves Balance separately with the specification of certain subsoil users.

A category of conditionally balance reserves (**E2 code**) includes quantities of minerals, the efficiency of production and use of which at the moment of evaluation cannot be expressly determined at sufficient geological knowledge through limiting parameters, and also reserves that meet the requirements of conditions for balance reserves, but cannot be extracted and sold for different reasons at the moment of evaluation.

A category of off-balance mineral reserves (**E2 code**) includes reserves for which at the moment of carrying out economic-geological evaluation according to technical and economical calculations and/or materials of financial reporting it has been proved that a profitability ratio of production (mineral raw materials) of a mining enterprise (estimated and/or actual) has a level insufficient for economically effective mineral extraction at the moment of evaluation, but in the foreseeable future these reserves can become balance ones and, therefore, are considered as potentially balance.

A category of mineral reserves and resources with undetermined commercial significance (**E3 code**) includes reserves for which the economic-geological evaluation has not been carried out or only initial economic-geological evaluation, which does not enable to determine their commercial significance, has been carried out. Mineral reserves and resources with undetermined commercial significance according to UNFC-2009 include such subcategories:

a) reserves and resources that will be extracted, but won't be used or sold (**E3.1 code**) which include the quantities of minerals used for internal needs of a mining enterprise during production and preparation of raw materials for sale, and also normative industrial and operational costs;

b) reserves and resources whose economic viability of extraction and sale cannot be yet determined due to insufficient prospecting

information (**E3.2 code**);

c) reserves and resources which on the basis of realistic assumptions of current and future market conditions are considered as unsuitable for economic extraction and sale at present and in the foreseeable future (**E3.3 code**).

Mineral reserves and resources are divided into such categories and subcategories **by a degree of feasibility study and preparedness for industrial development**:

**The first category (F1 code)** includes mineral reserves on the basis of which the detailed economic-geological evaluation of efficiency of their industrial development (EGE-1) is carried out, and its materials, including the feasibility study of permanent conditions for mineral raw materials are approved by the SCMR. Justification of mineral extraction at implementation of the optimum development project or during the mining operations is approved.

The first category of reserves preparedness for industrial development according to UNFC-2009 is divided into such subcategories.

A subcategory of reserves ***on production***, the extraction of which is carried out during the economic-geological evaluation (**F1.1 code**). A main feature of the reserves of this subcategory is actual economic industrial realization (sale) of end products of a mining enterprise.

A subcategory of reserves as regards which the industrial development project has been approved, the capital funds have been invested in its implementation, the issues on licenses and sales of products have been settled (**F1.2 code**).

A subcategory of reserves ***approved for industrial development***, the detailed economic-geological evaluation of which (with the research and industrial development for minerals of nation-wide significance) is finished, the materials on feasibility study (FS) of conditions and quantity of reserves have been approved by the SCMR (**F1.3 code**).

**The second category** of project feasibility (**F2 code**) includes mineral reserves on the basis of which the preliminary economic-geological evaluation of their commercial significance has been tried out, and the materials of the feasibility study report (FSR) on expediency of further geological exploration works and quantity of reserves have been approved by the SCMR. ***A degree of justification***

*of the reserves extraction by a certain optimum development project requires further detailed evaluation.*

The second category of reserves as regards their preparedness for industrial development includes such subcategories.

A subcategory of reserves ***proved for development (F2.1 code)*** includes reserves whose positive results of the preliminary economic-geological evaluation point to the expediency of continuation of geological exploration works with the purpose of detailed justification of industrial development in the foreseeable future.

A subcategory of project feasibility for which the performance of geological exploration works on preparation for industrial development ***is on hold or significantly delayed (F2.2 code)*** on the basis of conclusions of the economic-geological evaluation.

A subcategory of project feasibility for which the ***current plans to develop or to acquire additional data have not been determined*** on the basis of the evaluation ***due to unprofitability of development or other limited potentials (F2.3 code)***.

**The third category** of project feasibility (**F3 code**) includes mineral reserves and resources on the basis of which only preliminary or expert economic-geological evaluation (EGE-3) is carried out, and their possible commercial significance and economic development cannot be significantly determined due to insufficient data. At the same time the available **positive** results of geological exploration works justify the expediency of further geological exploration works with the purpose of acquisition of additional data in order to evaluate the expediency of extraction.

**The fourth category** of project feasibility (**F4 code**) includes mineral reserves and resources which during preliminary, initial or detailed economic-geological evaluation are reasonably defined as such that cannot be rationally extracted by modern mining methods or mining operations and remain in subsoil (in situ) on the place of occurrence.

Mineral reserves are divided into two categories: explored and preliminary explored **by a degree of geological study and confidence.**

**Explored reserves (G1 code)** are the volumes of minerals on the place of occurrence whose quantity, quality, technological properties,

mining and geological, hydrogeological and other modes of occurrence are studied with a high level of confidence and completeness. Key parameters of explored reserves which provide design decisions as regards extraction and processing of mineral raw materials and environmental protection are determined by data of direct measurements or investigations conducted within the deposits by a dense spacing pattern combined with limited extrapolation justified by research data. Explored mineral reserves are determined by data of finished prospecting, and the research and industrial development of deposits is carried out for minerals of nation-wide significance. Explored reserves are a basis for planning of further industrial field (deposit) development.

**Preliminary explored reserves (G2 code)** are the volumes of minerals whose quantity, quality, technological properties, mining and geological, hydrogeological and other modes of occurrence are studied with the completeness sufficient for significant techno-economic determination of commercial significance of the field (deposit) on the whole. Key parameters of preliminary explored mineral reserves that influence a choice of mineral raw materials production and processing methods are mainly estimated on the basis of extrapolation of data of direct measurements or investigations conducted within the field (deposit) by a wide or a non-uniform spacing pattern. Extrapolation is justified by analogy to the explored field (deposit), and also data of geological, geophysical, geochemical and other subsoil study. The quantity of preliminary explored reserves can be determined with a moderate level of confidence. Preliminary explored reserves are a basis for justification of further prospecting and detailed economic-geological evaluation of mineral fields (deposits).

Mineral resources are divided into two categories: prospective and prognostic **by a degree of geological study and confidence.**

Prospective resources (**G3 code**) are the volumes of minerals quantitatively estimated by the results of geological, geophysical, geochemical and another study of subsoil plots within the productive areas with accessed mineral deposits of a known geological and industrial type. Prospective resources include quantities of minerals in the fields which can be evaluated only with a low level of confidence. Prospective resources consider also a possible discovery



of new mineral fields (deposits) of a determined geological and industrial type the existence of which is justified by positive evaluation of mineral manifestations, geophysical, geochemical and other anomalies the nature and prospectivity of which are proved. Quantitative and qualitative indices of minerals in the fields (deposits) are estimated on the basis of data of certain samples and data of interpretation of geological, geophysical, geochemical and other studies or statistical analogy. Prospective resources are a basis for economic-geological evaluation of the expediency of further searches or geological exploration works.

**Table 1**

**MINERAL RESERVES AND RESOURCES CLASSIFICATION**

Categories by a level of socio-economic viability ( <b>E axis</b> )	Categories by a field project status and feasibility ( <b>F axis</b> )		Categories by a level of geological study and confidence ( <b>G axis</b> )	Class code of Ukraine	UNFC -2009 class
1	2	3	4	5	6
1. Balance reserves (1.) E1 E1.1; E1.2	Reserves on production ( <b>FI.1</b> ) approved for development ( <b>FI.2</b> ) or for industrial development ( <b>FI.3</b> )	EGE-1 (.1.) <b>F1;</b>	Explored reserves (..1) <b>G1</b>	111 proved	Commercial reserves
			Preliminary explored reserves (..2) <b>G2</b>	122 probable	
2. Potentially balance and off-balance reserves (2.) E2	Development of reserves is pending, on hold or unprofitable	EGE-1 (.1.) <b>F1; F1.3</b>	Explored reserves (..1) <b>G1</b>	211	Probable commercial reserves
		EGE-2 (.2.) <b>F2 (F2.1; F2.2)</b>		221	
			Preliminary explored reserves (..2); <b>G2</b>	222	

**Table 1 continuation**

1	2	3	4	5	6
3. Commercial significance of reserves and resources is undetermined (3..) E3; E3.1; E3.2;E3.3	Development of reserves and resources unclarified	EGE-3 (.3.) <b>F3</b>	Explored reserves (..1); <b>G1</b>	331	Non-commercial reserves
			Preliminary explored reserves (..2); <b>G2</b>	332	
			Prospective resources (..3); <b>G3</b>	333	Resources of geological exploration works
			Prognostic resources (..4); <b>G4</b>	334	
	Residual (additional) unrecoverable resources in subsoil	<b>F4</b>	Explored reserves (..1); <b>G1</b>	341	Residual (additional) reserves and resources
			Preliminary explored reserves (..2); <b>G2</b>	342	
			Prospective resources (..3); <b>G3</b>	343	
			Prognostic resources (..4); <b>G4</b>	344	

Prognostic resources (**G4 code**) are the volumes of minerals which consider a potentially possible formation of fields of certain geological and industrial types based on the positive stratigraphical,

paleogeographical, lithologic, tectonic, mineragenic and other prerequisites established within the prospective areas where commercial fields have not been accessed yet. Quantitative assessment of prognostic resources is carried out on the basis of estimated parameters by analogy to the productive areas where there are accessed mineral deposits of the same geological and industrial type. Prognostic mineral resources are a basis for justification of regional and prognostic geological works.

By the geological structure complexity the mineral deposits or their plots suggested for development by certain mining enterprises (mines) are divided into four groups:

1) fields (plots) with a **simple** geological structure and undisturbed or weakly dislocated bedding, continuous quantitative and qualitative parameters of deposits with main minerals, uniform distribution of main useful and harmful components (a coefficient of variation of the most variable parameter is less than 40%);

2) fields (plots) with a **complex** geological structure and discontinuous quantitative or qualitative parameters of deposits with main minerals, non-uniform distribution of main useful or harmful components (a coefficient of variation of the most variable parameter ranges from 40 to 100%);

3) fields (plots) with a **very complex** geological structure and variable quantitative or qualitative parameters of main mineral deposits, very non-uniform distribution of main useful or harmful components (a coefficient of variation of the most variable parameter ranges from 100 to 150%);

4) fields (plots) with a **extremely complex** geological structure and sharply variable quantitative or qualitative parameters of deposits with main minerals, extremely non-uniform distribution of main useful or harmful components (a coefficient of variation of the most variable parameter exceeds 150%).

To determine the complexity of a geological structure of the mineral field (plot) the indices of parameter variability of the biggest deposits with main minerals containing not less than 70% of reserves are used.

By a degree of preparedness for use the mineral reserves and resources are divided into:

- prepared for geological exploration works with the purpose

of geological study, including research and industrial development of the mineral fields of nation-wide significance for the purpose of detailed economic-geological evaluation;

- prepared for industrial development for the purpose of mineral extraction.

Accessed mineral fields (deposits) are prepared for prospecting works and research and industrial development if a degree of their geological and feasibility study provides a possibility of determination of all minerals and components, estimated quantity of deposits and their geological structure, technological properties of minerals as well as mining and geological conditions of their bedding, mining and technical, ecological and other conditions of mineral raw material production and processing, and **profitable** sale of sales production with the details sufficient for correct evaluation of their commercial significance.

Explored mineral fields (deposits) are prepared for industrial development if:

- balance reserves of main, jointly deposited and associated minerals and useful components available in them with commercial significance are approved by the SCMR, prospective resources have been evaluated; the quantities of recoverable mineral reserves and resources within the field (plot) are established according to a degree of their geological study considered during the design of construction (reconstruction) of a mining enterprise (mine) for determination of the limiting depth of development, a method of opening and a scheme of mineral deposit development and manufacture of sales production of a mining enterprise, development of a production facilities layout, approach roads, waste disposal sites etc.;

- quantities of balance explored and preliminary explored mineral reserves used for the design of construction (reconstruction) of a mining enterprise have been determined, a possibility of their development without harm to the mineral deposits remaining in subsoil has been justified;

- hazardous ecological factors which affect or can affect the environment during the prospecting works and field development, mineral raw material processing, industrial waste disposal have been defined and estimated; a rational complex of environmental

protection measures has been developed; background parameters of environmental conditions have been determined;

- profitability of production activity of a mining enterprise being designed has been justified by technical and economical calculations, the efficiency of capital investments in the field (plot) development approved by a subsoil user has been provided.

### **Economic-Geological Evaluation of Iron Ore Fields**

Within Ukraine 95% of iron ore reserves and 100% of their production fall on the ferruginous quartzite fields and Early Proterozoic rich ore fields.

By a mass fraction of iron there are naturally rich and poor (to be dressed) ores.

Rich ores the iron content of which makes from 46 to 70% are mainly magnetite, hematite and martite by mineral composition. In their turn they are divided into blast-furnace, open-hearth and sinter ones.

Poor iron ores requiring concentration are divided into free-milling and hard-milling ores depending on their mineral composition and textural-structural features. Free-milling ores include mainly magnetite ferruginous quartzite and oolitic brown ores of the Kerch Basin.

The dressing method is determined according to mineral composition of ores, their textural-structural features, and also a character of non-metallic minerals and physicommechanical ore properties. Magnetite ores are dressed by a magnetic method.

The use of dry and wet magnetic separation for magnetite ore dressing provides obtaining of prime concentrates even at rather low content of iron in ore. If there are commercial volumes of hematite together with magnetite in ores the magnetic flotation (for finely impregnated ores) or gravity and magnetic (for largely impregnated ores) dressing methods can be used. The magnetite quartzite dressing schemes include: crushing, grinding and magnetic dressing.

Blast-furnace or open-hearth ores, concentrates, sinter cakes, rolls and cakes are sales production of mining and processing enterprises of the iron ore industry. After processing of ferruginous quartzites and skarn-magnetite ores the concentrates with the iron content up to

67% are obtained; prime concentrates from apatite-magnetite ores contain not less than 62-64% of iron; concentrates with the iron content of not less than 69.5% and silicon dioxide of not more than 2.5% are produced for electrometallurgical processing. Requirements for product quality are regulated by the respective standards and specifications.

In some cases iron ores contain associated useful components which get into cast iron and steel or slag.

Such useful impurities as nickel, cobalt, manganese being alloyed components enable to obtain special steels with set properties by passing from cast iron into steel.

Vanadium is extracted from slag of metallurgical processing of titanium magnetite concentrates; phosphorus-containing slag should be used as a fertilizer.

Iron ore fields or their plots under try-out or to be tried out by certain subsoil users are divided into three groups by the geological structural complexity considering variability of ore body forms, their internal structure, distribution of useful and harmful components and mining and geological modes of occurrence and other factors which influence a choice of the prospecting and development system:

- the first group - fields (plots) with a simple geological structure;
- the second group - fields (plots) with a complex geological structure;
- the third group - fields (plots) with a very complex geological structure.

Geological exploration works directed to search and prospecting of iron ore fields are performed according to the procedure established by the Regulations on the Procedure for Geological Exploration Works in Stages and Phases.

In all iron ore fields discovered in the course of search works in order to determine their commercial significance the prospecting and evaluation works are conducted and in case of positive results the exploration works are conducted with the purpose of preparation of the discovered ore reserves for industrial development.

Geological study of iron ore deposits provides determination of their substance composition, quantity, quality and technological

properties, a geological structure, hydrogeological, mining and geological and other occurrence modes of ore deposits for the purpose of justification of design decisions as regards the iron ore production method and system as well as the iron ore complex processing scheme.

Iron ore reserves are divided into two groups by a level of geological knowledge: *explored and preliminary explored*.

*Explored reserves (G1 code)* are the volumes of iron ores whose quantity, quality, technological properties, mining and geological, hydrogeological and other occurrence modes are studied with the completeness sufficient for working off of construction projects of mining objects and ore processing objects. Key parameters of explored reserves which provide design decisions as regards ore production and processing and environmental protection are determined by data of direct measurements or investigations conducted within the deposits by a rather dense spacing pattern in combination with the limited extrapolation justified by data of geological, geophysical and other studies.

By the study of morphological characteristics, an internal structure and ore deposit occurrence modes the explored iron ore reserves are divided into categories A, B, C<sub>1</sub> of exploration maturity.

Category A reserves are located on the plots of detailed elaboration and primary try-out of fields with a simple geological structure within which a form, quantity and occurrence modes of ore deposits have been established, a ratio and spatial location of the plots formed by ores of different natural types and industrial (technological) grades have been clarified, ore-free and substandard plots within ore deposits have been marked out and delineated, positions and displacement amplitudes of fractures have been established. A boundary of category A reserves is formed along the prospecting workings.

Category B reserves are located within the plots of detailed elaboration and primary development of fields with a complex geological structure and in the fields of the first group of complexity, in blocks for which the quantity, main features of a form, an internal structure and occurrence modes of ore deposits, spatial location of ore-free and substandard plots in the middle of ore deposits have been determined, positions and displacement amplitudes of big

opening dislocations and zones of development of low-amplitude opening dislocations have been established. In the category B blocks the main regularities of spatial distribution of different types and industrial ore grades within the boundaries of ore deposits should be determined. A boundary of category B reserves is drawn according to the prospecting workings with a limited zone of extrapolation.

Category  $C_1$  reserves are located on the plots of detailed elaboration and primary work-off of fields with a very complex and an extremely complex structure of field development and in the fields of the first and second groups of geological structure complexity, in the blocks within which the quantity and characteristic forms of ore deposits, key features of their occurrence modes and internal structure, possible discontinuity of ore bodies have been clarified, a position of main tectonic disturbances, and also zones and areas of their intensive development have been characterized. Spatial location and a ratio of different natural types and industrial ore grades, placement of ore-free plots composed of substandard ores have been generally established. The boundary of category  $C_1$  reserves is determined on the basis of prospecting workings and extrapolation by geological and geophysical data.

***Preliminary explored reserves (G2 code)*** are the volumes of iron ores whose quantity, quality, technological properties, mining and geological, hydrogeological and other occurrence modes are studied with the completeness sufficient for probable determination of possible commercial significance of the field. Key parameters of preliminary explored reserves that influence a choice of ore production and processing methods are mainly estimated on the basis of extrapolation of direct measurements or investigations conducted within the field by a wide or non-uniform spacing pattern. Extrapolation is justified by analogy with the explored field (deposit), and also according to geological, geophysical, geochemical and another subsoil study.

Preliminary explored reserves by a degree of study of morphological characteristics, an internal structure and occurrence modes of ore deposits correspond to category  $C_2$  reserves. Category  $C_2$  includes ore reserves on the subsoil plots within which a form, quantity, an internal structure and occurrence modes of ore deposits are estimated by geological, geophysical, geochemical data and



confirmed by a wide spacing pattern of sections of the prospecting workings. The boundary of the reserves of category C<sub>2</sub> is formed on the basis of well-grounded data extrapolation of prospecting workings. In the newly discovered fields where there are no reserves of high categories, category C<sub>2</sub> includes deposit plots in which the ore quantity and quality are measured in several points located not on the same line crosswise relative to the spread of ore bodies.

Iron ore resources are divided into two groups **by a degree of geological study and confidence: *prospective (G3 code) and prognostic (G4 code)***.

Prospective resources (**G3 code**) are divided into P<sub>1</sub> and P<sub>2</sub> categories by a degree of data confidence on occurrence modes and morphological characteristics of ore deposits.

Category P<sub>1</sub> includes prospective resources on the plots of extension of ore-bearing areas adjacent to the boundaries of explored and preliminary explored reserves of the discovered iron ore fields, and in the prospective subsoil plots where the existence of ore deposits, their form, quantity, an internal structure and occurrence modes are estimated by geological, geophysical, geochemical data and confirmed with the single sections of prospecting workings.

Category P<sub>2</sub> prospective resources consider a possibility of opening of new iron ore fields in the known ore region, ore field, the existence of which is foreseen on the basis of positive evaluation of ore manifestations of the ore-containing rocks, geophysical and geochemical and other anomalies, the nature and prospects of which have been justified on the discovered ore analog objects. Occurrence modes and morphological characteristics of mineralization are estimated by analogy.

***Prognostic resources (G4 code)*** are the volumes of iron ores considering a potentially possible formation of fields of certain geological and industrial types grounded on the positive stratigraphical, lithologic, mineragenic, paleogeographical and other prerequisites established within the prospective areas where commercial fields have not found yet.

Quantitative evaluation of prognostic resources is carried out on the basis of foreseen (prognostic) parameters by analogy with the productive areas where there are accessed iron ore fields of the same geological and industrial type.

Iron ore reserves and resources are divided into three groups by a degree of feasibility study:

*the first group* (F1 code) - explored reserves on the basis of which the detailed economic-geological evaluation (EGE-1) of the efficiency of their industrial development has been carried out. Its materials, including the feasibility study of permanent conditions for mineral raw materials, have been approved by the SCMR;

*the second group* (F2 code) - reserves on the basis of which the preliminary economic-geological evaluation of their commercial significance (EGE-2) has been carried out, and materials of the feasibility study report on the expediency of further deposit exploration, including justification of temporary conditions for mineral raw materials have been approved by the SCMR or a customer (investor) of geological exploration works;

*the third group* (F3 code) - reserves and resources on the basis of which the prefeasibility economic-geological evaluation of possible commercial significance of a prospective subsoil plot (EGE-3) has been carried out, and materials of feasibility considerations of the expediency of further prospecting and evaluation works, parameters of the previous conditions for mineral raw materials have been approved by a customer (investor) of geological exploration works.

Conditions for calculation of iron ore reserves shall provide a full complex and economically rational extraction of iron ore reserves and available associated minerals and components from subsoil on the basis of the use of up-to-date industrial mineral raw material production and processing technologies in view of observance of the requirements for subsoil and environmental protection. The feasibility study of conditions is carried out for all iron ore fields forwarded to state expert appraisal, direct calculations with the involvement of actual techno-economic indices of development of analog fields.

Iron ore reserves are divided into such groups by commercial significance:

*balance* (E1 code (E1.1, E1.2)) - (economic) reserves which at the moment of evaluation according to technical and economical calculations can be extracted and used on the economically efficient basis with the up-to-date equipment and iron ore production and processing technology that provide the observance of requirements

for a rational complex use of raw material resources and environmental protection;

*potentially balance* (E2 code) - (economically limited) reserves whose efficiency, production and use at the moment of detailed economic-geological evaluation cannot be expressly determined and also reserves which meet the requirements for balance reserves, but for different reasons cannot be used at the moment of evaluation;

*off-balance* (E2 code) - (potentially economic) reserves whose production and use at the moment of evaluation are economically inexpedient, but in the future they can acquire commercial significance;

*with undetermined commercial significance* (E3 code) - reserves and resources on the basis of which only initial economic-geological evaluation with the use of assumed technological and economic output data is carried out.

Iron ore reserves and resources which belong to certain groups by a level of their commercial significance, a degree of feasibility and geological study are divided into classes identified by means of the international three-digit code. In this code units correspond to the groups of reserves (resources) by a degree of geological study; tens correspond to the groups of reserves (resources) by a degree of feasibility study, hundreds correspond to the groups of reserves by a level of commercial significance.

In the explored field a topographical basis is formed on a scale meeting its quantity, features of a geological structure and a relief. Topographic maps and plans in the iron ore fields are made up on a scale of 1:1,000-1:10,000.

For the area of the field a geological map and a map of mineral resources are made up on a scale of 1:25,000-1:50,000 with the corresponding sections. The maps include location of ore control structures and ore-bearing rock complexes, mineral fields and manifestations of the area, and also plots, within which the mineral prospective and prognostic resources have been evaluated.

Mineral reserves on the plots of detailed elaboration of fields of the 1<sup>st</sup> and 2<sup>nd</sup> groups of complexity are mainly explored by categories A and B (respectively) and in fields of the 3<sup>d</sup> group - by category C<sub>1</sub>.

Reserves are calculated within the established industrial

boundaries of ore deposits drawn on the basis of test data and geological documentation of mining workings (wells).

Delineation comes down to making boundaries of commercial ore reserves which separate them from off-balance ores and from ore-containing rocks.

Explored and preliminary explored iron ore reserves are calculated by a level of geological knowledge.

**Explored (G1 code)** reserves are calculated on the plots provided for primary development and project justification of the first stage of a mining enterprise. Within the boundaries of explored reserves it is possible to distinguish categories A, B, C<sub>1</sub> reserves in the appropriate ratios by a level of exploration maturity and appropriate confidence.

Category A reserves are indicated in the blocks of detailed elaboration of explored fields with a simple geological structure (the first group) or according to mining prospecting and mining development works in the fields with a complex geological structure. In order to define the reserves as category A it is necessary to clarify in full morphology and an internal structure of ore bodies suitable for selective extraction, spatial location of the plots formed by ores provided by conditions of technological types and commercial grades, to delineate ore-free and substandard plots within the ore deposits, and also to study comprehensively mining and geological occurrence modes which determine a technology of mining works. Detailed study of an internal structure of ore deposits shall provide univariant data linkage between prospecting sections. A boundary of category A reserves passes through prospecting workings without extrapolation towards the reserves of lower categories and pinching-out of deposits.

Category B reserves are indicated in the blocks of detailed elaboration of the fields with a complex geological structure (the second group) or plots of primary development with a simple geological structure according to the data of advance mining prospecting. In the blocks of category B reserves the main features of occurrence modes, forms and a structure of ore bodies suitable for selective extraction, main regularities of spatial location of technological types and industrial ore grades provided by conditions, main regularities of location of ore-free plots and substandard ores within the ore deposits are established. Detailed geological study

shall make essential changes in the elements of extent and inclination of ore bodies, internal structure of ore deposits, character of relationship between ore bodies and ore-containing rocks impossible at further exploitation. The boundary of category B reserves in the fields with a complex geological structure is drawn under prospecting workings, in the fields with a simple geological structure, at stable depth and ore quality - with the use of limited extrapolation.

Category  $C_1$  reserves are indicated on the plots of detailed elaboration and primary development of fields of the third and fourth groups of geological structure complexity, and also in the fields with a simpler geological structure. Category  $C_1$  includes reserves explored with detailing that provides general clarification of occurrence modes, forms and a structure of iron ore deposits, their technological types and industrial grades, and also natural factors that determine carrying out mining prospecting works.

**Preliminary explored (G2 code)** reserves are indicated on the plots of fields which are not provided for primary development and are an object of future expansion of extractive operations. By a degree of study of morphological ore deposit features the preliminary explored reserves correspond to exploration maturity category  $C_2$ . In the iron ore fields under prospecting the category  $C_2$  preliminary explored reserves are located on the plots of extrapolation adjacent to a boundary of explored reserves by extent and inclination based on a wide spacing pattern of ore sections which confirm extrapolation; in the independent ore deposits - proceeding from the aggregate of ore sections established in outcrops, mining workings and wells taking into account data of geophysical, geochemical studies and geological structures.

In the process of calculation of the reserves with extremely interrupted, nested mineralization within the boundaries of productive (mineralized or ore-bearing) zones the plots characterized by low coefficients of ore-bearing capacity (0.3-0.1 and less) are usually defined as category  $C_2$ .

By a degree of preparedness for industrial development the indicated iron ore fields are divided into:

- prepared for prospecting works with the purpose of detailed economic-geological reserves evaluation;

- prepared for industrial development for the purpose of ore extraction.

In the iron ore fields under development the mining prospecting is carried out according to item 26 of the Reserves Classification under the methodological guidance which is developed as a standard of the mining enterprise.

Mining prospecting which advances the front of breakage works specifies geometrization of ore bodies with a high coefficient of ore-bearing capacity, determines boundaries of breakage blocks, and specifies evaluation of quantity and quality of ore reserves. Data of mining prospecting is used for design of mining development and heading works and annual planning of production rates.

Associated mining prospecting is carried out directly in the course of heading works in the operational block, and at the same time the boundaries of mineralization and ore quality are specified.

According to associated prospecting the boundaries of breakage reserves and workings are projected under the accepted development system, the volumes of reserves ready for production are calculated, loss and depletion at production are determined.

By the results of mining prospecting of the iron ore fields (plots) under development, the reserves are transferred to the appropriate categories of confidence; additionally indicated and recovered reserves are calculated and accounted.

Conditionally balance and off-balance reserves involved in development are transferred to balance ones. Iron ore reserves accessed, prepared and ready for production, and also available in the safety pillars of permanent mine workings and mining development headings are calculated and accounted separately according to their commercial significance, a level of geological and feasibility study.

### **Economic-Geological Evaluation of Gold Deposits**

Gold is a chemical element, a metal from the group of noble metals. Gold is a resistant inert metal. About 10% of total gold is used for production of industrial products.

Ores include gold mainly in its native form. It is mostly contained in quartz and sulfides (arsenopyrite, pyrite, chalcopyrite, faded ores, galenite and other minerals); it can be often in a scattered fine-

dispersed state.

Under conditions of formation the gold fields are divided into endogenous, exogenous, metamorphogene (metamorphic and metamorphosed).

By a mineral structure endogenous gold ores form several main ore formations as follows.

*Gold-quartz and gold-quartz-sulfide formation.* Gold in ores is generally free in quartz, partially - in sulfides and is characterized by non-uniform distribution. Depending on the composition of sulfides there are different mineral types in these formations. The deposits are presented by veins, vein zones and stockworks formed under conditions of average depths in sedimentary, volcanic, intrusive and less often in metamorphic rocks.

*Gold-sulfide formation.* Gold is closely connected with sulfides. Pyrite, chalcopyrite, arsenopyrite, pyrrhotite, sphalerite and galenite in variable quantities play a major role in the mineral composition of ores. Deposits belong to the zones with impregnated gold-bearing sulfides in sedimentary and sedimentary-effusive rocks. They are quite often bent for carbonaceous and graphite shale.

*Gold-carbonate-sulfide formation* covers fields, ores that make deposits, veins presented by nested or impregnated mineralization in carbonaceous rocks and metasomatites formed on them.

*Gold-silicate (skarn) formation.* Ores represented by skarn deposits with the imposed sulfide and gold ore mineralization belong to the contact areas of granitoid massifs of the Paleozoic Era, rarer of the Mesozoic Era.

*Gold-silver (gold-adular-quartz) formation.* Ores of this formation are localized, as a rule, in the near-surface conditions and connected with surface volcanism. Gold and silver ores are represented by veins, mineralized and vein zones, stockworks. According to the quantity of sulfides in ores, endogenous fields are divided into poor and sulphidic (within 2%), low-sulfidation (within 5%), moderate sulfidic (within 5-20%) and essentially sulfidic (more than 20%) types.

Besides the above ore formations forming namely gold fields, gold is an important useful component of many endogenous complex fields - mainly copper-porphyritic, copper-pyrite, pyrite-polymetallic, polymetallic copper-nickel, etc.

By morphological features, occurrence modes and an internal structure of ore deposits, a character of gold distribution in subsoil, the endogenous fields are divided into such main types: stockworks, mineralized and vein zones, veins, massive and impregnated ore deposits, tubular and irregular-shaped deposits and nests.

*Stockworks* are formed by a large number of differently oriented, unevenly formed and distributed thin quartz veins and thin vein lines, and also impregnated sulfide mineralization. As a rule, stockworks have quite considerable area and depth. Fields of this type are localized in metamorphosed sand and shale (carbonaceous) rocks, rarer in the igneous intermediate rocks and granitoids or subvolcanic acid rocks. Big, but extremely uneven in depth veins with a complex form belong to the fault zones within the stockworks. Plots with commercial ores in the stockwork fields have no clear geological boundaries and are indicated according to test data.

*Mineralized and vein zones* are plots of tectonically dislocated and hydrothermally changed terrigenous-sedimentary and volcanogenic-sedimentary rocks or an aggregate of closely located subparallel quartz veins, vein lines, flattened lenses localized in crystalline rocks, effusive and subvolcanic formations with the moderate-acid composition, and also in terrigenous-sedimentary rocks. Linearly-oblong forms, considerable depths (from 5-10 to 50 m and more) and the absence of clear geological boundaries of ore deposits are characteristic of them; as a rule, their boundaries are determined by test data. Vein-impregnated ores belong to gold-silver, gold-quartz-sulfide and gold-quartz formations.

*A vein type* of fields can be presented by one vein big in length, several veins disconnected between them, or a system of rather short veins. In all cases each vein is an independent ore body. The biggest part of vein fields with gold-quartz formation is deposited between the sand and shale flyschoid strata; the length of their ore bodies varies from tens to the first hundreds of meters - several kilometers.

Vein fields belonging to intrusive massifs are usually presented by veins with considerable length both by a course (within 1 km and more), and inclination. Ore bodies have gold-quartz or gold-quartz-sulfide composition.

Vein fields developed among young effusives and subvolcanic formations mainly with acid and moderate-acid composition belong



to gold-silver formation and are related to a near-surface type. The length of ore bodies reaches hundreds of meters.

Vein fields are mainly complex by ore composition: gold-copper, gold-antimony, gold-polymetallic.

Deposits (lens-like, vein-like, sheet and complex) can be formed by gold-bearing pyrite-chalcopyrite, pyrite-pyrrhotite, sphalerite-galenite, barite, magnetite massive and impregnated ores; besides, deposits can be presented by secondary quartzite, quartz-mica, quartz-manganic, and other rocks with impregnated or vein-impregnated mineralization.

*Tabular and irregular deposits and nests* of skarn fields have limited distribution.

Ore *dikes* are an independent morphological type of gold fields. Their mineralization relates either to a system of quartz or quartz-sulfide vein lines presented by cross cracks, or to thin quartz veins and vein lines coinciding with longitudinal fracturing of dikes. Gold is mainly concentrated directly in quartz veins and vein lines at its low content in rocks of the dikes.

Gold-enriched gossans of sulfide fields, crusts of weathering of mineralized zones and also gold-bearing placers belong to *exogenous fields*.

*Gossans* are a top oxidized part of sulfide deposits (pyritic sulphur, copper-pyrite and polymetallic) in which gold as a chemically resistant mineral is accumulated together with iron hydroxides, lead carbonates, secondary silver minerals. The biggest contents of gold relate to lower horizons of gossans formed by barite, quartz and pyrite granular formations.

*Crusts of weathering of mineralized zones* have considerable sizes. They develop on the areas of outcrop of gold-bearing mineralized zones the primary ores of which are poor in gold. Ore bodies in crusts of weathering have considerable areas and extend to the depths of 300-400 m. Fields are localized in terrigenous or volcanogenic-sedimentary rocks. In crusts of weathering ores are fully disintegrated, gold is mainly in a free state. Its content can be 1.5 - 2 times higher than in primary ores.

By natural conditions of formation the crusts of weathering are divided into residual and redeposited. Residual crusts are formed in aluminum silicate rocks. The fields in this type of crusts are

characterized by preservation of morphology and gold-bearing capacity of primary ores in the zone of hypergenesis. Redeposited crusts are formed in carbonate rocks or at the contact between carbonate and aluminum silicate rocks. Morphology and gold-bearing capacity of primary ores in the redeposited crusts experience essential changes.

*Metamorphogene fields* include gold-bearing conglomerates and sandstones of the Witwatersrand field in the Republic of South Africa - one of the biggest gold ore objects in the world.

*Technogenic fields* include dumps of off-balance ores extracted and stored as a result of development of gold fields, gold-containing waste (tails, sludge) formed in the process of ore dressing or processing of gold-bearing concentrates (drosses, cakes, ashes) of complex fields of ferrous, nonferrous, noble and other metals.

By the content of gold ores are divided into:

- low-grade (poor) ores with the gold content of 2-3 g/t and less;
- crude ores with the gold content from 3-6 to 10 g/t;
- rich ores with the gold content of 10-30 g/t and more.

Gold fields are divided into four groups by the quantity of evaluated and preliminary evaluated reserves:

- small fields with reserves up to 10 tons;
- average fields with reserves of 10-50 tons;
- big fields with reserves of 50-100 tons;
- unique fields with reserves of over 100 tons.

According to the Mineral Reserves and Resources Classification of the State Subsoil Fund the gold ore fields or their plots which are being tried out or are to be tried out by certain subsoil users correspond to three groups by geological structure complexity:

the second group - fields (plots) with a complex geological structure;

the third group - fields (plots) with a very complex geological structure;

the fourth group - fields (plots) with the extremely complex geological structure.

**Gold reserves are divided into groups: *explored (G1 code) and preliminary explored (G2 code) by a degree of geological knowledge*** Explored gold ore reserves are divided into exploration maturity categories B and C<sub>1</sub> according to a degree of study of

morphological characteristics, an internal structure and occurrence modes of ore deposits. Preliminary explored reserves correspond to category  $C_2$  reserves by a degree of study of morphological characteristics, an internal structure, occurrence modes and tectonic features of ore deposits.

**Gold ore resources are divided into two groups: *prospective (G3 code)* and *prognostic (G4 code)* by a degree of geological study and confidence.** Perspective resources are divided into categories  $P_1$  and  $P_2$  by a degree of data confidence on occurrence modes and morphological characteristics of ore deposits. Prognostic resources belong to category  $P_3$  by a degree of data confidence on occurrence modes and morphological characteristics of ore deposits.

**Gold ore reserves are divided into four groups by commercial significance (*balance (E1 code (E1.1, E1.2)), conditionally balance (E2 code), off-balance (E2 code), commercial significance of which is undetermined (E3 code)*).**

For the most effective study of indicated gold ore fields it is necessary to observe the established order of investigations according to the Regulations on Stages of Exploration Works which provides the observance of staging in the complex fields and in the fields of new types where there is a high risk to obtain unforeseen results. Topographic maps and plans in the gold fields are made up on a scale of 1:1,000-1:5,000. For the area of the field a geological map and a map of mineral resources are made up on a scale of 1:25,000-1:50,000 with the corresponding sections.

Technological properties of ores of gold deposits differ in their complexity. Such signs are of the most importance for determination of a process technology of gold-containing mineral raw materials:

- a characteristic of gold contained in ore (fineness, a form of occurrence, a nature of association with metallic and non-metallic minerals, a state of the particle surface);
- complexity of ores i.e. the content in ore of other useful components having commercial significance together with gold;
- a level of ore oxidation i.e. oxidized and sulfide minerals ratio (%);
- existence of components in ore which significantly complicate their process technology.

The main technological dressing schemes for mineral raw

materials of gold fields are in most cases a combination of dressing processes and pyro- and hydrometallurgy which include ore sorting, crushing, grinding, sludge removal, gravity flotation (collective or selective) dressing, amalgamation, cyanidation (by a filtration or sorption technology) or pyrometallurgical processing (burn, melting) of ores and concentrates. A final process is gold affinage.

New technological processes include: radiometric sorting, foam separation, heap leaching, bioleaching, chloride sublimation etc. and also geotechnological methods of gold mining (mine and well leaching systems).

The methods based on cyanide leaching of gold have been widely used in practice of gold mining companies.

Category B reserves at prospecting are calculated only in fields of the 2<sup>nd</sup> group. They include reserves located on the plots of detailed elaboration or within other parts of ore bodies, whose degree of exploration maturity meets the requirements of Classification for this category.

The boundary of category B reserves is drawn along the mining workings or wells without extrapolation, and the main geological characteristics of ore bodies and ore quality are determined by a sufficient volume of representative data.

Category C<sub>1</sub> includes reserves on the plots of fields within which a network of mining workings and wells accepted for this category is kept, and the reliability of information obtained is confirmed in the new fields by the results of study of the plots of detailed elaboration, and in the fields under development – by data of exploitation.

Category C<sub>2</sub> reserves are calculated by ore bodies by extrapolation under the course and inclination from a boundary of explored reserves of higher categories on the basis of geophysical works, geological and structural formations, study of regularities of change in depth of ore bodies and contents of gold in them and single ore sections that confirm this extrapolation; by independent ore bodies - proceeding from the aggregate of ore sections positioned in outcrops, mining workings and wells taking into account data of geophysical, geochemical studies and geological structures, and in case of impossibility of geometrization of ore bodies - statistically within the generalized boundary.

**By a degree of preparedness for industrial development the**

indicated gold fields are divided into:

- **prepared for prospecting works** with the purpose of detailed economic-geological evaluation of gold reserves including research and industrial development;
- **prepared for industrial development** for the purpose of gold mining.

Gold ore fields are considered as prepared for prospecting if their commercial significance and expediency of a prospecting stage of works have been determined, the general scales of the field have been indicated, and the most prospective plots for justification of the order of prospecting and further try-out have been outlined.

In the fields of the second group of geological structure complexity the explored gold ore reserves are used for design of construction (reconstruction) of mining and processing enterprises, in the fields of the third and fourth groups of geological structure complexity the explored and preliminary explored reserves are used. At the same time the quantity of explored reserves shall provide operation of a mining enterprise or its first stage for the period sufficient for return of funds invested in construction.

Mining prospecting is carried out in the fields under development according to the methodological guidance developed and approved as a standard of the mining enterprise.

By the results of the works performed on mining prospecting of gold fields (plots) under development, a transfer of reserves to the appropriate exploration maturity categories, calculation and accounting of additionally indicated and recovered reserves are carried out. Conditionally balance and off-balance reserves involved in development are transferred to balance ones. Gold ore reserves accessed, prepared and ready for production, and also that available in the safety pillars of permanent mine workings and mining development headings are calculated and accounted separately according to their commercial significance, a level of geological and feasibility study.

### **Economic-Geological Evaluation of Amber Fields**

In the territory of Ukraine amber manifestations and fields are presented by placers connected with the sediments of the Prypiat and

Dnipro Basins of the Baltic-Dnipro Amber Province and Carpathian Province. The commercial amber fields of the Prypiat Basin are connected with the sedimentary deposits of coastal marine lagoon-deltaic facies. Amber ores of the fields in Ukraine are developed by means of the open-cut excavation and transport method. Ore dressing is carried out by buddling on the metal sieves (griddles). Sand and clay are washed away with water and amber pieces are selected and directed to sorting and further use. A main area of amber application is jewelry industry.

The commercial fields of precious commercial fossil resins (amber) belong to three geological and industrial types of fields (biogenic-sedimentary, modern alluvial and of sea beaches, buried marine lagoon-deltaic and lacustrine-glacial).

Amber fields or plots provided for development by certain mining enterprises belong to the 2<sup>nd</sup>, 3<sup>d</sup> and 4<sup>th</sup> groups of the Classification by the geological structure complexity and features of distribution.

The 2<sup>nd</sup> group includes fields with a complex geological structure with the irregular quantitative or qualitative parameters of deposits and uneven amber distribution within productive deposits (a variation coefficient of deposit parameters provided by conditions - 40-100%). The fields of this group are characterized by the existence of an amber-containing layer of deposits uniform in length and width with its rather constant depth, but within the boundary of the layer there are relatively enriched and depleted plots. Amber ore deposits subject to a selective try-out in the fields of the second group cannot be delineated during the prospecting works, and their reserves cannot be determined within a geometrical boundary. A possibility of plots delineation with obligatory indices of ore quality shall be proved in the block of detailed elaboration or on the analog field exploited. Some fields in the coastal marine sediments with large amber reserves belong to this group.

The 3d group includes fields with a very complex geological structure with variable quantitative or qualitative parameters of deposits and very uneven amber distribution by the depth of a placer and the area of their development (a variation coefficient of deposit parameters provided by conditions - 100-150%). Within the deposit boundaries there are certain plots considerably enriched with amber or depleted. Reliable linkage of deposits in the fields of this group,

determination of their integrity, reliability of testing and delineation are possible only on the basis of data of mining operations in combination with drilling ones. Deposit localization and location are controlled by natural factors (lens-like clay bodies, clay rolls or very thin layers, coalified wood, heap of trees, a congestion of trunks and branches of trees, boulders, pebble and gravel) which can be traced beyond their boundaries and can be used for forecast and geological extrapolation of mineralization on the plots adjacent to productive mining workings. This group includes big channel fields, average and small deposits of buried marine lagoon-deltaic and lacustrine-glacial fields.

The 4<sup>th</sup> group includes fields with an extremely complex geological structure and sharply variable quantitative or qualitative parameters of deposits and extremely uneven nested amber distribution by the depth of deposits and the area of their development (a variation coefficient of deposit parameters provided by conditions - over 150%). Within the industrial boundaries of the field there are plots with the noncommercial content of amber. Deposit boundaries are established only according to test data. Deposits are controlled by natural factors which practically do not go beyond their boundaries. A possibility of forecast and extrapolation of a placer is very complicated.

By the quantity of balance amber reserves the fields are distributed into: big reserves of more than 100,000 kg; average reserves of 50,000-100,000 kg; small reserves up to 50,000 kg.

**Amber reserves in subsoil are divided into explored and preliminary explored by a degree of geological knowledge.** Explored amber reserves are a basis for design and development of the field, its plots or groups of blocks within a single open-pit field. A group of explored reserves includes amber reserves of categories B and C<sub>1</sub>. Preliminary explored reserves are a basis for justification of further prospecting or research and industrial development of a field, a plot or its blocks. A group of preliminary explored includes amber reserves of category C<sub>2</sub>.

**Indicated amber reserves are divided into such groups by commercial significance:**

- *balance (E1 code (E1.1, E1.2))* - reserves which at the moment of evaluation, according to technical and economical

calculations can be extracted and used on the effectively economical basis by means of up-to-date technical equipment and amber production and process technologies which provide the observance of requirements of the rational complex use of explored amber ores and environmental protection;

- ***conditionally balance (E2 code)*** – reserves the efficiency of production and use of which at the moment of evaluation cannot be finally determined and also reserves that meet the requirements for balance reserves, but for different reasons cannot be used at the moment of evaluation;

- ***off-balance (E2 code)*** - reserves the production and use of which at the moment of evaluation are economically inexpedient, but in the future they can become an object of industrial development.

As regards the explored amber field it is necessary to have a topographic basis the scale of which would correspond to sizes and geological features of the deposit.

Topographic maps for these fields are made on scales of 1:2,000-1:5,000. For small fields or certain plots of big and average fields a scale is enlarged to 1:1,000.

Amber placers are studied by means of wells with a big diameter, holes, trenches (ditches), investigation open pits, and also in natural outcrops.

Amber ore reserves are calculated in thousands of cubic meters; amber reserves are calculated in kilograms and rounded off to the first digit after the decimal point. Amber content in samples is determined in grams per cubic meter and rounded off to the first digit after the decimal point.

Both total and recoverable amber reserves if available on the place of bedding in subsoil shall be calculated and accounted. Loss and depletion during production and processing of amber ores determined according to the optimum field development system based on the comparative feasibility analysis and industrial (semi-industrial) technological study are considered at calculation of recoverable amber reserves.

According to the Mineral Reserves and Resources Classification of the State Subsoil Fund, amber reserves are calculated by the results of geological exploration works and exploitation of fields. Reserves belonging to different groups and classes under their



industrial significance, a degree of feasibility and geological study are separately calculated. Total and recoverable amber reserves are determined in raw amber and distributed by rating according to sales production specifications.

Reserves are calculated within the established industrial boundaries of amber deposits determined by test data and geological documentation of mining workings (holes, trenches, open pits).

Delineation of deposits comes down to making boundaries of commercial ore reserves which separate them from off-balance ores and amber-bearing deposits.

According to the geological structure complexity and a degree of study of amber fields during prospecting and exploitation works the explored ore and amber reserves are calculated by categories B and C<sub>1</sub>, preliminary explored - by category C<sub>2</sub>. Prospective resources are evaluated by category P<sub>1</sub>. Prospective resources of category P<sub>2</sub> and prognostic resources of category P<sub>3</sub> are determined during the regional geological study of the territory of Ukraine.

By the results of prospecting category B reserves should be indicated in the blocks of detailed elaboration of fields of the 2<sup>nd</sup> group of geological structure complexity within the boundaries of a uniform network of prospecting workings.

Category C<sub>1</sub> reserves shall be indicated on the plots of primary development of amber fields of the 2<sup>nd</sup> group, and also within the plots of detailed elaboration of fields of the 3<sup>d</sup> and 4<sup>th</sup> group of geological structure complexity.

Category C<sub>1</sub> should include blocks of amber placers explored in details sufficient for general clarification of the quantity of amber ores, amber quality and quantity, a form and an internal structure of deposits, mining and geological and other conditions of their bedding. In the fields of the 4<sup>th</sup> group of complexity category C<sub>1</sub> includes blocks adjacent to the open pits of research and industrial development.

Category C<sub>2</sub> reserves should be indicated in amber fields of all groups of geological structure complexity within the plots provided for prospective development of the mining enterprise.

The quantity, form, internal structure and occurrence modes of ore deposits are evaluated by analogy with the explored plots of amber placers confirmed with the available prospecting sections.

**By a degree of preparedness for industrial development** the indicated amber fields are divided into:

- prepared for prospecting works with the purpose of detailed economic-geological evaluation of reserves;
- prepared for industrial development for the purpose of mineral extraction.

In the fields of the 2<sup>nd</sup> group of geological structure complexity the explored amber reserves are used for design of construction (reconstruction) of mining and processing enterprises, in the fields of 3<sup>d</sup> and 4<sup>th</sup> groups of geological structure complexity the explored and preliminary explored amber reserves are used. At this the quantity of explored reserves shall provide operation of the mining enterprise or its first stage for the period sufficient for return of funds invested in construction. In case of deviation of the established ratio of balance reserves of different categories by more than 20% the SCMR of Ukraine makes a decision on preparedness of the field for industrial development. In the amber fields under development for recalculation of reserves their ratio by categories is justified by the subsoil user's technical assignment.

The SCMR of Ukraine determines a possibility of full or partial use of category C<sub>2</sub> amber reserves for design.

In the fields prepared for industrial development the substance composition and technological properties of amber ores shall be studied in details that provides basic data for design of a technological scheme for amber extraction; hydrogeological, engineering-geological, mining and geological and other natural and technogenic conditions should be studied in details that provides data acquisition for making up a field (plot) development project.

In the developed fields it is necessary to carry out mining prospecting.

Mining prospecting, which passes ahead of development of extractive operations (by 2-5 years), specifies boundaries of ore deposits, their internal structure, occurrence modes and development conditions, amber content in ores determined by geological exploration according to data of additional mining workings taking place prior to the extractive operations.

Mining prospecting that goes with extractive operations

determines boundaries of breakage blocks, quantity and quality of prepared reserves, a volume of operational reserves, loss and decrease in the average content of amber during production (depletion).

By the results of mining prospecting of amber fields (plots) under development the reserves are transferred to the appropriate exploration maturity categories, additionally indicated reserves are calculated and accounted. Conditionally balance and off-balance reserves involved in try-out are transferred to the balance ones. Amber ore reserves accessed, prepared and ready for production, and also available in the safety pillars are calculated and accounted separately according to their commercial significance, a degree of geological and feasibility study.

### **Conclusions**

According to the legislation of Ukraine on subsoil, mining and economic relations the economic-geological evaluation of mineral reserves and resources of subsoil plots determining their quantity, quality and cost takes a key place in relations as regards creation, use and protection of the mineral resources base of the state.

The Mineral Reserves and Resources Classification of the State Subsoil Fund is a key normative and legal document which regulates economic-geological evaluation of mineral deposits, establishes basic conditions and approaches to their observance. Detailed methods on the use of the Classification for mineral deposits by their types, and also evaluation indices characteristic of them are determined by the SCMR relevant guidance documents and the integrated Regulation on the Procedure for Development and Substantiation of Conditions on Mineral Raw Materials for Calculation of Solid Mineral Reserves.

The economic-geological evaluation of a mineral deposit or a productive subsoil plot is a wide range of investigations, multivariant engineering calculations and structures owing to which conditions, optimum boundaries of productive deposits, quantity and quality of balance and off-balance mineral reserves, a technological scheme and a level of performance indicators of future exploitation, commercial significance and cost of a field or a subsoil plots under

evaluation are determined.

The economic-geological evaluation is a research process associated with the study of a range of possible competitive variants of delineation and industrial development of mineral deposits of the field (plot) and determination among them the most effective one by optimality criteria.

Implementation of the economic-geological evaluation of mineral deposits makes conditions for the fullest use of their reserves at a level of present-day achievements of scientific and technical progress in the sphere of mineral raw material production and processing on a rational economic basis, as well as:

- determination of expediency of creation of new or expansion of operating mining enterprises (mines) and attraction of investments in the branch;
- prospective planning of needs of economic development of the country and their provision with mineral reserves;
- determination of directions of prospecting and regional works;
- keeping the State Balance of Mineral Reserves;
- determination of payment and tax obligations for the use of natural resources.

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## **THE STUDY OF THE STRESS-STRAIN STATE OF THE MASSIF IN MINING URANIUM AT “VOSTGOK” DEPOSITS**

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Being one of the world’s largest uranium producers (about 2% of the world’s production), the state enterprise “Vostochnyi Mining and Processing Works” (“VostGOK”) provides about 40% of Ukraine’s nuclear power stations with uranium raw materials.

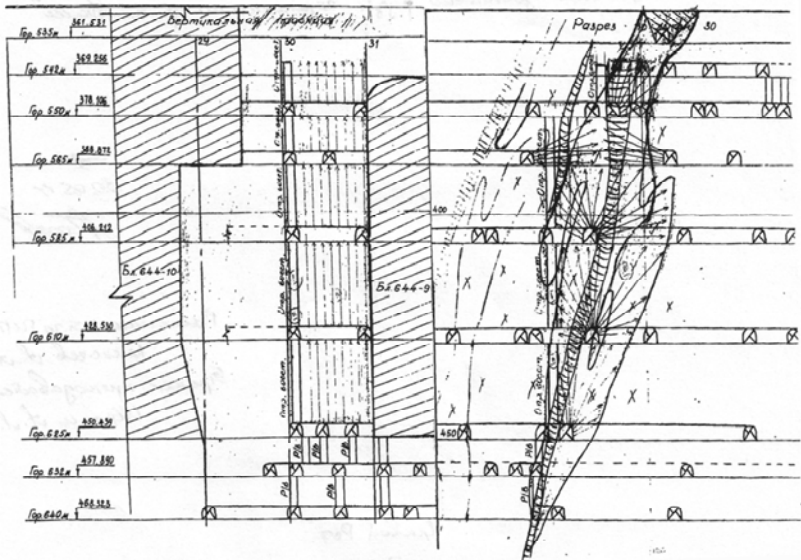
Considering the conditions of uranium deposits exploitation (location in densely populated areas, protected sites etc.), to protect the environment from possible emissions of radioactive elements room mining is applied with subsequent backfilling of the dead area with consolidating mixtures.

This technology is economically reasonable at deposits with the increased uranium content. To exclude a number of labour-consuming and environmentally dangerous operations from the production process, lean uranium-containing ores are reasonable to be mined applying underground block leaching. This enables reaching maximum values of mineral extraction and avoiding considerable material expenditures on backfilling mixture preparation and backfilling dead rooms, as they are almost completely backfilled with the muck pile, and on utilization of waste after the mined ore primary processing (barren rocks and off-balance ores) on the daylight surface.

Further cut of costs can also be achieved through mining deposits by vertical double blocks. Ore body 10 of the Michurinskoye deposit is supposed to be mined in blocks 10-2 and 10-3 at the 325-184 m level at the Ingulskaya mine (Fig.1).

The process is as follows. Another room is located under the

temporary pillar-crown below the dead room backfilled with the muck pile. Under this pillar in the block located further down the dip a compensatory room is placed to which reserves of this block are broken and the temporary crown is brought down. The solution for leaching uranium ores is fed from the existing workings over the room of the upper block. At this, volumes of mining are cut and pipes are again used for feeding the working solution to blocks.



**Fig.1.** The vertical plane of blocks 10-2 and 10-3 (ore body 10 of the Michurinskoye deposit)

The levels of stress in main structural units and in the enclosing rock massif, the condition of the crown (the degree of its disturbance caused by workings and deep holes) are different from those occurring when the traditional technology is applied. Besides, the crown is affected by reagents for underground block leaching. Due to all that, factors impacting the crowns' stability and mine safety on the whole require urgent investigations.

Determining permissible dimensions of main structural units of room mining systems [ , main regulatory documents 1, 2], however, do not consider the impact of the ore body dip and are not intended for determining the safe thickness of a crown. The technology of

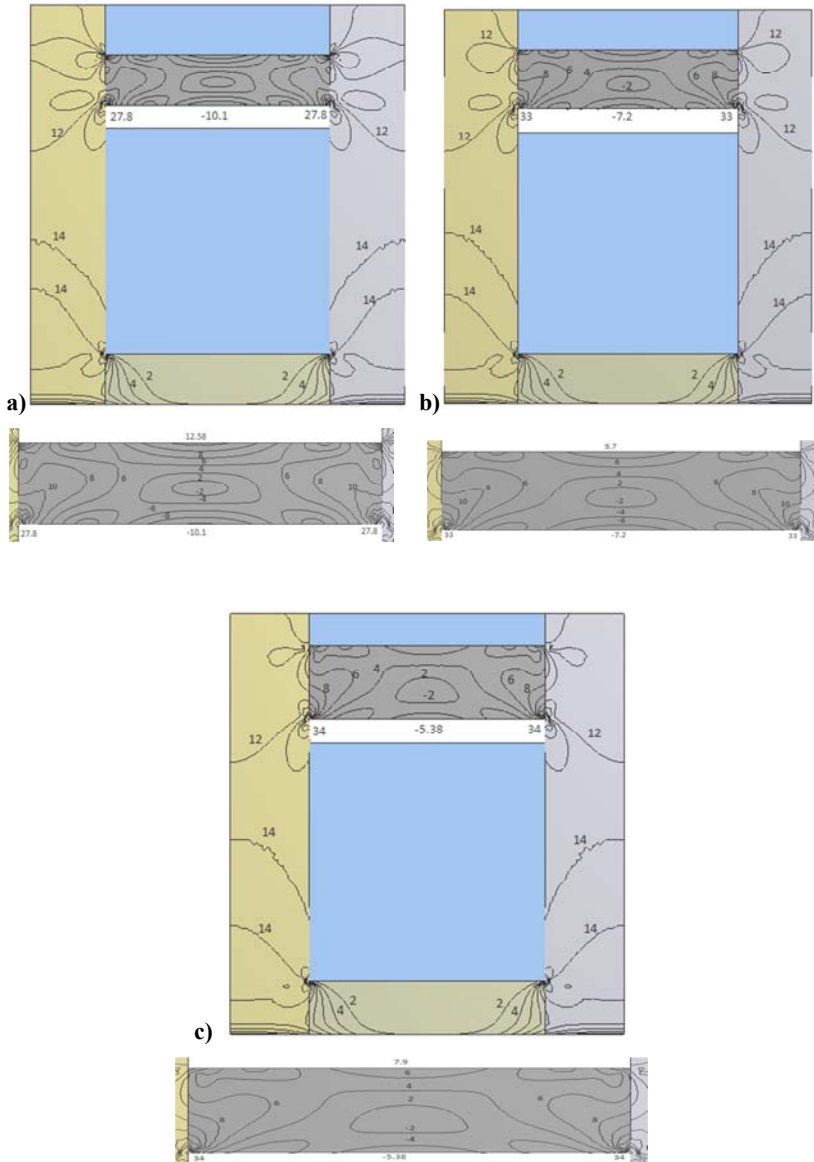
underground block leaching of uranium ores in vertical double blocks is a new one to be applied at VostGOK underground mines and requires scientific support.

To study the stress-strain state and stability of crowns depending on the ore body dip angle and the conditions of the above mentioned blocks, mathematical modelling applying the finite-element technique was applied. The range of boundary conditions of the impacting factors included values characteristic of all the underground mines of “VostGOK”. Uranium ore hardness varied from 9-11 to 14-16 on the Protodyakonov scale, that of the enclosing rock - 13-15, the ore body dip made from 60° to 90° (in increments of 10°). The stress-strain state was registered for crowns of 10 to 14 m thick. For calculating the stress field characteristics *Ansys 18* was used.

The results of modelling the stress-strain state of 10 m thick crowns in ores of various hardness with the ore body dip angles of 90°, 70° and 60° are presented in Fig 2, 3 and 4.

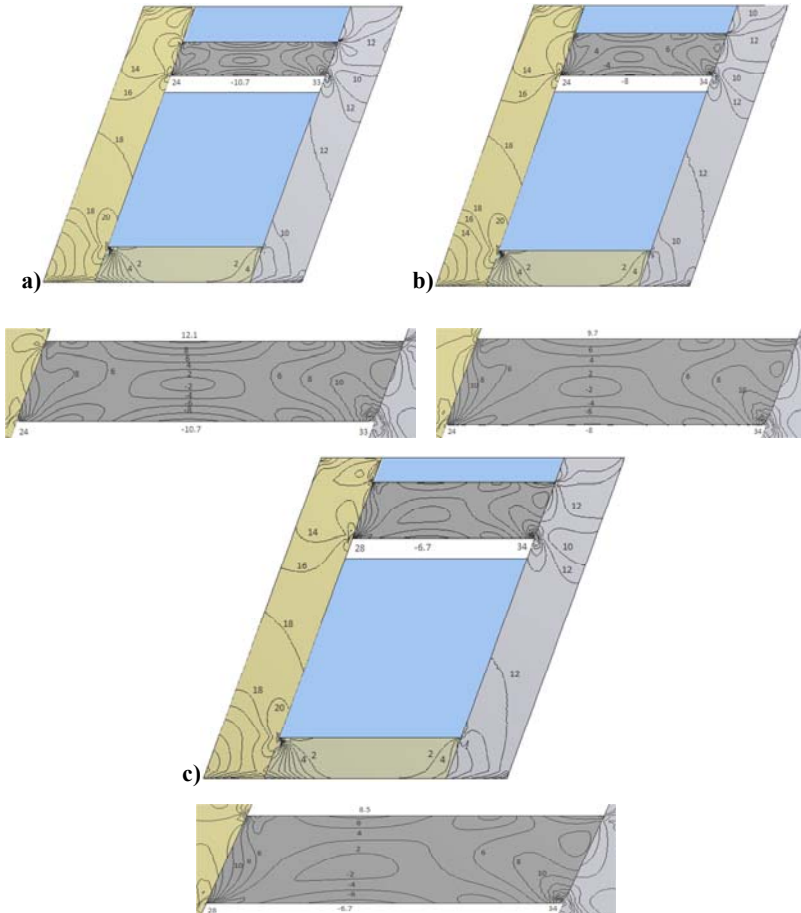
As is seen, the tension stress zone in the lower central part of the crown is the most dangerous. This corresponds to the classical concepts of stress field development in the so called “stress relief arch” that occurs when the massif is undermined by the lower block room. As ore hardness reduces, absolute values of stress in the crown decrease slightly (by 0.1...0.5 MPa, i.e. from 1...2 to 6...7%). This can be explained by the fact that less hard ores are less liable to accumulate stress as they get relieved through deforming towards a free surface (i.e. the room) and, on the opposite, harder ores tend to accumulate stress due to smaller deformations. However, stability of less hard ore crowns decreases due to reduction of their ultimate strength.

For instance, with the ore body dip angle  $\alpha = 90^\circ$  and hardness of 14-16 points, the value of tension stress in the lower part of the crown reaches 10.1 MPa (Fig. 2,a). However, as ultimate tension stress of such ores is about 11 MPa, these stresses will not cause failures. When the crown is made of ores of 10-11 points (Fig. 2,d), the tension stress level makes 9.9 MPa. With the ultimate strength of the ores of 7.7 MPa this will cause rock falls of about 100...150 m<sup>3</sup> (according to “Instructions...” [2] used at “VostGOK” mines, rock falls of over 250...300 m<sup>3</sup> are considered critical).



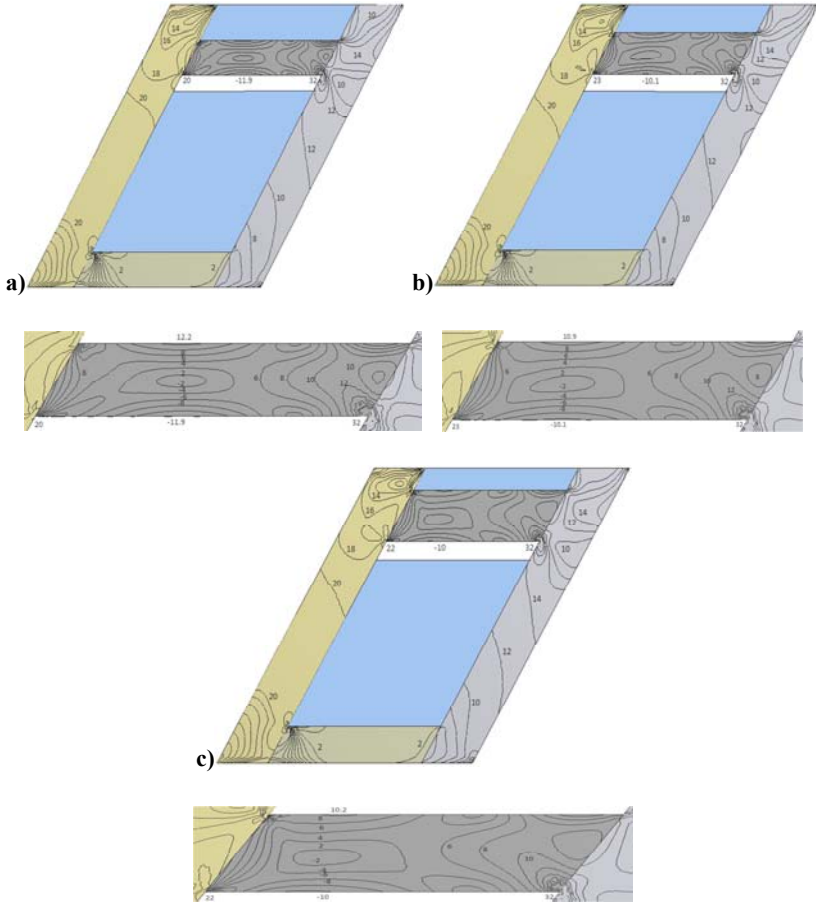
**Fig. 2.** Stress field development in 10 m, 12 m, 14 m crowns with the ore body dip of 90°, MPa





**Fig. 3.** Stress field development in 10 m, 12 m, 14 m crowns with the ore body dip of 70°, MPa

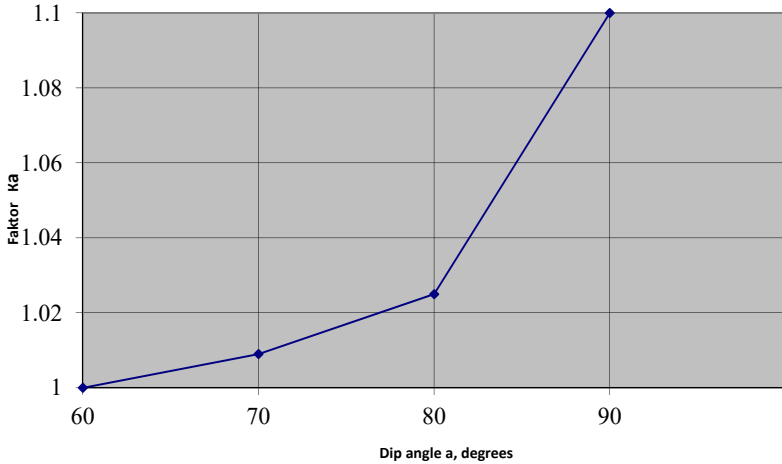
With the dip angle  $\alpha = 70^\circ$  and ores of 14-16 points crown failures do not practically occur (Fig. 3,a), with  $\alpha = 60^\circ$  small rock falls (3-5 m<sup>3</sup>) may occur even in crowns of ores of the same hardness (Fig. 4,a). In crowns of ores of 10-11 points with  $\alpha = 70^\circ$  (Fig. 3,b) and  $\alpha = 60^\circ$  (Fig. 4,b) the volume of rock falls will make from 150 to 200...220 m<sup>3</sup>, sometimes to 400...450 m<sup>3</sup> respectively. These values testify to the critical condition of the crown at angles about  $\alpha = 70^\circ$ , at about  $\alpha = 60^\circ$  the crown will fail.



**Fig.4.** Stress field development in 10 m, 12 m, 14 m crowns with the ore body dip of 60°, MPa

Thus, the obtained results testify to the considerable impact of the ore body dip angle on the stress-strain state of crowns and their stability and enable us to suggest application of the correction factor  $K_{\alpha}$ , whose numerical values are given in Fig. 5. So, when determining the minimum permissible thickness of the crown in certain conditions, its value obtained without this factor should be corrected through multiplying it by the corresponding value  $K_{\alpha}$ .

Changes in the existing stress fields, increase of absolute values of current stresses caused by technological workings result in decrease of the crown stability.



**Fig. 5.**  $K_a$  values depending on the ore body dip angle  $\alpha$

Due to this, when determining safe dimensions of exposures and pillars, they should be corrected considering the accepted criteria. In the first case, the crown thickness is determined according to conditions of the room mining order in compliance with the instructions developed by NIGRI (Research Ore Mining Institute) [1]. In the second case, the correction factor is applied.

We suggest correcting thickness of the crown with workings using the expression

$$h_{cr}^n = h_{cr} \cdot K_{dist}, \text{ m} \quad (1)$$

where  $h_{cr}$  is thickness of the monolith crown, m;  $K_{dist}$  is the factor considering disturbance of the crown resulted from mining, unit fraction.

As the disturbance degree of the crown depends on the number of workings in it, their geometrical dimensions and thickness of the crown itself, we suggest determining the numerical value of  $K_{dist}$  as the product of separate universal factors. Each of these factors differentially takes into account the impact of a particular working on

the crown stress-strain state and, consequently, on its stability, as follows

$$K_{dist} = K_1 \cdot K_2 \cdot \dots \cdot K_n, \text{ unit fraction,} \quad (2)$$

where  $n$  is the number of workings in the crown.

Numerical values of these factors calculated individually for each working can tentatively be determined as follows

$$K_i = \sqrt{1 + \left( h_i^w / h_{cr} \right)}, \text{ unit fraction.} \quad (3)$$

where  $h_i^w$  is the  $i$ -th working height (width), m.

For instance, according to the calculations, the minimum permissible thickness of the crown not disturbed by workings is  $h_{cr} = 10$  m. In case of workings of 2.5, 3.0, 3.5 and 4.0 m, the correction factors for each of them determined by (3) will equal 1.12, 1.14, 1.16 and 1.18 respectively. Thus, the crown thickness should be increased to 11.2, 11.4, 11.6 and 11.8 m respectively.

If there are 2 workings of 3.0 m and 3.5 m in the crown, the correction factor will make  $K_{dist} = 1.14 \cdot 1.16 = 1.32$ . Correspondingly, the disturbed crown thickness should be increased to 13.2 m.

If there are 3 workings of 2.5 m, 3.0 m and 3.5 m in the crown, the correction factor will make  $K_{dist} = 1.12 \cdot 1.14 \cdot 1.16 = 1.48$ . Under such conditions the crown thickness should be half as much as that of the monolith crown and make 14.8 m.

So, the crown thickness should be corrected considering decrease of its stability caused by workings. This will help avoid its complete or partial failure.

As shrinkage stoping with sulphuric acid treatment is one of the main components of underground block leaching of uranium ores, the crown separating the rooms will also be exposed to the sulphuric acid.

The research conducted enables the authors to assume that the longstanding (from 3-4 to 6 months) exposure to the sulphuric acid may negatively impact strength properties of the ore massif of the crown. This assumption is substantiated by data on the physical and mechanical properties of rocks of the Michurinskoye deposit, particularly albitites and migmatites which are the most

representative rocks in uranium ore occurrence zones. Thus, the average compressive resistance of rocks in their natural humidity conditions and when water-saturated makes 164.4 MPa and 127.5 MPa for albitites and 153.1 and 112.4 MPa for migmatites respectively. That is, if compared with the natural state, water saturation of rocks reduces their compressive resistance by 22...27%.

The analysis of the data on the physical and mechanical properties of rocks of the Michurinskoye deposit, particularly albitites and migmatites which are the most representative rocks in uranium ore occurrence zones, enables the authors to conclude that the longstanding (from 3-4 to 6 months) exposure to the sulphuric acid may negatively impact strength properties of the ore massif of the crown. The average compressive resistance of rocks in their natural humidity conditions and when water-saturated makes 164.4 MPa and 127.5 MPa for albitites and 153.1 and 112.4 MPa for migmatites respectively. That is, if compared with the natural state, water saturation of rocks reduces their compressive resistance by 22...27%.

The impact of the sulphuric acid solution on the crown stability was confirmed by the following investigation. Forty ore cubes with 50 mm sides were divided into two groups. The first group of 10 cubes was used to determine the uniaxial compressive resistance in the natural conditions, the remaining cubes were used for determining the degree of the sulphuric acid solution impact on the samples' strength.

To provide conditions of the crown contacting the acid solution, in the laboratory environment only one face of an ore sample contacted the acid solution. The other faces of the cubes were covered with two coatings of paraffin. These cubes were placed in a vessel with the sulphuric acid solution which is used for spraying the shrunk muck pile in underground mines of "VostGOK". Tests of uniaxial compressive resistance were carried out 2.5, 4 and 6 months after dipping to determine the impact of the exposing time on the uranium ore strength. These periods correspond to the minimum and maximum time of the reagent impact in real conditions.

The laboratory hydraulic press is able to produce pressure up to 50 t. In relation to the cubes' surface  $S = 25 \text{ cm}^2$  the corresponding

pressure makes about  $2000 \text{ kg/cm}^2$ , or 200 MPa. The press is coupled with a computer that sets the loading rate for the samples and forms the loading diagram for each of the samples with the automatic recording of the current load, maximum pressure at the moment of their destruction and calculates ultimate strength of each sample depending on its sizes. During the tests the minimum loading rate of 1 kN/s was set according to corresponding standards (from 1 to 5 kN/s).

The samples of the first group demonstrated the average value of the uniaxial compressive resistance of about 130 MPa. According to the instructions [3] this value corresponds to the rock hardness ratio of 11 points. For the samples exposed to the sulphuric acid solution during 2.5, 4 and 6 months, average strength values made 82...84.5, 79.5...80.5 and about 78 MPa respectively, i.e. their ultimate strength decrease (in relation to the samples of the first group) made 35...37%, 38...39% and about 40%.

Thus, the tests conducted confirmed the authors' assumption about the considerable impact of the acid solution on the uranium ore strength and, consequently, the stability of exposures and pillars. The determined dependencies should be considered in defining the safe crown thickness when applying the technology of underground block leaching of uranium ores.

So, the research conducted enabled determining the degree of impact of major factors (ore body dip, crown integrity loss caused by technological workings, impacts of reagent used when applying underground block leaching of uranium ores) on the crown stability. These factors should be taken into account when determining safe dimensions of exposures and pillars using corresponding correction factors. As a result, in concrete conditions it is necessary to correct parameters of structural units of blocks, particularly the crown thickness, considering the value of its stability changes caused by the above factors. This correction enables avoiding the crown failure and provides safety of works. The determined dependencies can then be corrected considering practical experience of "VostGOK" underground mines.

So, the research conducted has resulted in the following:

- the degree of impact of major factors (ore body dip, crown integrity loss caused by technological workings, impacts of reagent

used when applying underground block leaching of uranium ores) on the crown stability has been determined;

- the factors should be taken into account when determining safe dimensions of exposures and pillars using corresponding correction factors.

- in concrete conditions it is necessary to correct parameters of structural units of blocks, particularly the crown thickness, considering the value of its stability changes caused by the above factors. This correction enables avoiding the crown failure and provides safety of works.

The determined dependencies can then be corrected considering practical experience of “VostGOK” underground mines.

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# IMPROVING THE METHOD OF OPEN-PIT LIGNITE DEPOSITS DEVELOPMENT IN UKRAINE

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**Abstract.** *Subject of the research* is open-pit mining concerning a full-field industrial development of a number of lignite deposits; feasibility study has been performed for them as for the efficient environmental friendly processing of coal and associated minerals.

*Objective of the research* is to develop both methodology and conceptual approaches to high-productive, economically viable, and environmental friendly methods for open-pit lignite mining in the context of suprasalt depressive basins.

*Task of the research* is to analyze the current state of lignite mining in Ukraine; to characterize a new genetic Ukrainian coal type from geological and industrial viewpoint; to substantiate parameters of lignite open-pit mining on the basis of Novo-Dmytrivka mining and industrial district; to expand the capacities of lignite mining on the basis of Novo-Dmytrivka, Bantysh, Stepkivka, and Bereka deposits; to substantiate the integrated use of diverse rock masses in the context of the national economy; to use lignite in terms of its power; to produce montan wax; to apply sodium humite in the context of agriculture; to use overburden rocks for the construction of bordering dams of powerful water storages; and to develop recommendations concerning the design of Novo-Dmytrivka mining and industrial system with the integrated development of lignite and associated minerals.

*Methods of the research* are: analytical estimation of resources of lignite deposits; geological and engineering-technical analysis; and integrated and feasibility studies of indices of mining and opening operations. Optimization of the process solutions relies upon the analysis of changes in rock mass coefficient use within the open-pit area in the context of complete



land reclamation of the disturbed land and the development of new productive land instead of the littered territories. The updated research method is to determine the basic technological parameters of equipment taking into consideration significant water inflow in terms of working areas as well as the inflows effect on the output of the lignite open pit depending upon changes in the depth of mine workings.

The carried out research helped study more thoroughly the geological and engineering-technical features of lignite deposits in Ukraine. Their geological structures, coal-bearing capacity and the coal grades, total reserves, and their commercial significance have been determined. Parameters of benches and working sites have been substantiated. The parameters make it possible to decrease the current volume of overburden rock mining and to transfer their maximum values to the final stage of the open pit operation. Rational systems of mining and transportation equipment for the development of the open-pit field in terms of criteria of capacity, efficiency, and power consumption have been substantiated involving different traffic flows of rock mass movement in open pits and at the surface. There were issued recommendations to design the development of Novo-Dmytrivka lignite deposit. Relying upon the analyzed deposits of north-west Donbas, it is expedient to develop the unified coal-mining complex for the processing of lignite and associated minerals to be used by plants of building materials and structures as well as chemical and metallurgical plants as the basic raw material. There has been substantiated a possibility of commercial development of a number of lignite deposits in Ukraine to develop mining and preproduction complex with coal output at the level of 9-10 mln t/y and 23-24 mln t/y of coaly mass as well as their processing by thermal power station which capacity is 1800-2400 MW; a plant to produce 15 thousand tons of montan wax a year; briquetting factory which capacity is 2 mln t/y; and a concrete product plant to manufacture building structures with a capacity of 1 mln of m<sup>2</sup>/y.

**Introduction.** Industrial development of new coal deposits is one of the key problems of strategic advance of fuel and energy complex of Ukraine. If coal share is 67%, and oil and gas shares are 18% and 15% among the world reserves of raw hydrocarbons, in Ukraine their volumes are distributed differently: coal is 954%, oil is 2%, and gas is 2.6%. In this context, only coal, owing to its reserves which have already been prospected, is able to decrease energy dependence of Ukraine upon foreign raw material in future. According to expert estimations, as for the heat power industry and raw processing, the role of coal will become more important both in Ukraine and in the

whole world.

Bereka, Stepkivka, Biliaievka and Bantysh diapir structures are coal-bearing potentially. However, only the first stage of prospecting activities has been implemented within the deposits. The obtained data make it possible to evaluate the deposits in general to be insufficient for their complete commercial economic evaluation.

Up to now, Novo-Dmytrivka structure has already been studied in more detail; as for its scale and conditions of formation of a complex of various minerals and coal reserves it is among the unique ones. It is possible to build open pit with annual 9-10 mln t output of high-grade lignite on the basis of Novo-Dmytrivka deposit. In this context, opening coefficient will not be more than 4 m<sup>3</sup>/t; it is 3 times less to compare with that achieved by open pits of *Oleksandriia* SHC.

There are no national practices concerning the development of such deposits. Thus, objective of the paper is to develop both a method and conceptual approaches to high-productive, efficient, and environmentally friendly techniques of open-pit lignite mining under the conditions of suprasalt depressive basins.

In this connection, scientific and practical tasks of the research are to analyze the current state of lignite mining in Ukraine; to substantiate parameters of open-pit lignite mining on the basis of Novo-Dmytrivka mining and industrial district; and to substantiate the integrated use of diverse rock mass for the national economy needs.

### **Zonation of Ukrainian deposits**

Lignite is a natural type of solid sedimentary mineral as a transition form from peat to the rock varieties. Its deposits occur in the interior of the earth in the form of plate- and lens-shaped bodies with thickness being small to compare with its area. Lignite mining had obtained its wide significance since 1929 when *Ukrburvuhillia* Group was established. Dnipro lignite basin became a development centre. The basin is located within a central part of Ukraine; it stretches in the form of a wide belt from northwest to southeast passing through Zhytomyr Region, Kyiv Region, Cherkasy Region, Kirovohrad Region, Dnipropetrovsk Region, Zaporizhzhia Region and partially through Mykolaiv and Odesa Regions. Total extent of

the basin is 650 km, and width is 70-175 km. Its area is almost 100.000 square kilometers.

Ten geological and industrial districts are singled out at the territory of the basin (Fig. 1).

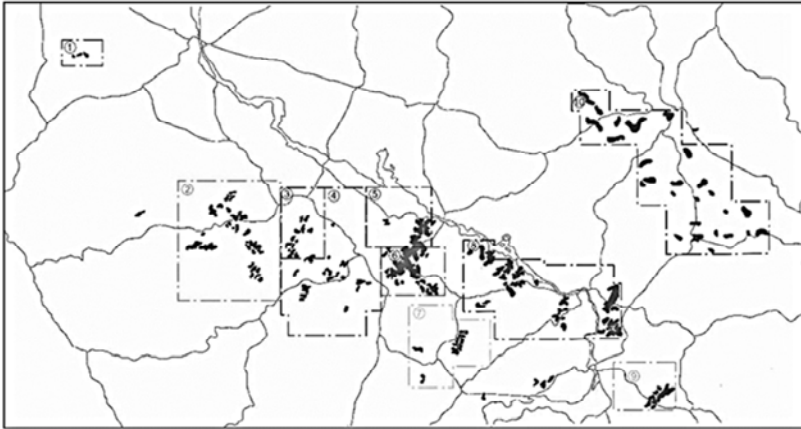


Fig. 1. Zoning of lignite districts in Ukraine

They stretch northwest to southeast as following [1]:

1. Korostyshiv – Korostyshiv deposit.
2. Zvenyhorodka – Yurkivka, Mokra Kalyhirka, Zelenkivka, Zalizniachkove, Oleksandrivka deposits and smaller ones.
3. Zlatopillia – Novo-Myrhorod, Zlatopillia, Zhurovka, Serdiukivka, Paleoloh (Kvytkovka and Mykhailivka deposits) formations.
4. Kirovohrad – Malo-Vyskivka, Zeleno-Haiska, Novo-Mykhailivka, Mamaika, Veselivka, Adzhamka, Balashivka and Zelenivka deposits.
5. Novo-Heorhiivka – Zolotarivka, Revivka, Taburyshche and Myronivka deposits.
6. Oleksandriia – Bandurovka, Morozivka (*Verbolozivska, Baidakivska, Semenivska #4, Semenivsko-Holovkivska* sites), Moshoryno-Svitlopil, Nova-Praha, Mykhailivka, Mariianivka and Balakhivka deposits.
7. Kryvyi Rih – Khrystoforivka, Hurievsk, Vesele-Ternivka,

Pychuhine and Kolomiitseve deposits.

8. Dnipropetrovsk – Hannivka, Fastiv, Novo-Aleksandrivka, Verkhniodniprovsk, Sokolove, Saksahanske, Adalymivka, Pavlo-Hryhorivka, Shyroke, Barvinok and Kamiianske deposits, Karnaukhivka, Pavlivka, Synelnykove ra Pervozvanivka deposits.

9. Horikhivka – Horikhivka deposit.

10. Northwest Donbas – Novo-Dmytrivka, Sula-Udai, Bantysh, Stepkivka and ra Bereka deposits.

Almost 200 lignite deposits and occurrences are within the basin; however, not all of them have practical value and sufficient reserves to be mined. The Governmental balance of Ukraine has counted 27 deposits; 11 of them are suitable for open-pit mining. According to A+B+C1 categories, their reserves are 2409.3 mln tons; 215.7 mln tons belong to C2 category; and 393.1 mln tons are non-commercial reserves. Balance reserves, suitable for open-pit mining (according to A+B+C1 categories), are 510.9 mln tons; non-commercial reserves are 393.4 mln tons. They are mainly deposited in Kirovohrad Region (240.57 mln tons) and Dnipropetrovsk Region (106.62 mln tons); other their shares are in Cherkasy Region (187.4 mln tons), Zaporizhzhia Region (11.43 mln tons) and Vinnytsia Region (5.24 mln tons).

In terms of Dnipro basin, coal-bearing formation consists of one to three contiguous coal seams being, in actual fact, a common lignite layer divided by rock interlayers which thickness is 0.5 to 6 m and more. A bottom seam with up to 25 m thickness (when average value is 2-4 m) is the basic one. Upper seam is characterized by variable thickness (0.1 m to 3 m). Maximum thickness of all the seams is 29 m (Verkhniodniprovsk deposit). Configuration of the deposits is rather complicated; it duplicates borders of palaeovalleys within which they occur. The deposits are almost horizontal. Their depth from the surface varies from 10-30 m in the central part to 100-150 m within watersheds. Area of certain deposits is 50-60 km<sup>2</sup>.

The common feature of the basin coal is that, according to the degree of metamorphism, it belongs to lignite with the age varying for certain deposits and sites. As a rule, carbonaceous clays and sands occur in the upper part of the seams. Clay lenses, secondary kaolins, and sandstones are available among the sands. Fine sands and carbonaceous clays occurring more rarely are the floor of the

coal deposits; within the areas where crystalline basement rises, primary kaolins are the floor. In the context of coal basins, kaolins and systems of sedimentary formations capping them occur on the rocks of the crystalline basement. As for the sedimentary formations, sandy varieties are groundwater reservoirs (aquifers), and clayed as well as carbonaceous poorly permeable layers are aquifuges. Within the basin, water-bearing levels of the Quaternary System, a system of Kyiv, Kharkiv, Poltava, and Buchak suites as well as underground water of fissured zones of crystalline floor rocks are singled out.

In the context of Verkhniodniprovsk deposit, reserves for open-pit mining are 159.2 mln tons. Mining and geological conditions are favourable: thickness of a coal seam is 10.6 m; industrial opening coefficient is 6.6 m<sup>3</sup>/t; moisture is 51 %; ash content is 18.7 %; bitumen content is 8.3 %; and the heat of coal combustion is 2290 kkal/kg. There is a possibility to build open pit with annual capacity of 4.0 – 4.7 mln tons. However, the deposit is within a nature conservancy zone of the Dnieper River. Its allotment for mining in Dnipropetrovsk region being overloaded with mining enterprises is rather a problematic idea.

In the context of Synelnykovo lignite deposit, where total reserves are 350 mln tons, *Petrovska* site has been prospected for open-pit mining. Reserves of the site are 70 mln tons and opening coefficient is 9.1 m<sup>3</sup>/t. The lignite contains 58% of moisture, 4.8% of sulphur, 20.8% of ash and 7.4% of bitumen; heat of the lignite combustion is 1810 kkal/kg. Mining and geological conditions are complicated. Other deposits of Verkhniodniprovsk lignite district have been explored preliminary; they are subject to underground mining.

Novo-Dmytrivka lignite deposit is in Barvenkovo District of Kharkiv Region. The deposit is confined to a deep basin above salt rod; it has trough-like occurrence form. Three lignite seams, which thickness varies from 2.0 m to 60 m, are of commercial interest. Balance reserves are 390 mln tons. Ash content of the lignite varies from 13.5% to 40%; moisture varies from 48.5% to 56 %; sulphur content varies from 1.5% to 3.8 %; heat of the lignite combustion varies from 1435 to 2930 kkal/kg. Depth of the lignite seams at the output to the surface is 50 to 60 m and in the central part of a trough is 300-400 m. Industrial opening coefficient is 4 m<sup>3</sup>/t. The lignite is

suitable for briquetting, production of montan wax, direct burning, as well as chemical and technological processing [2-5].

Sula-Udai deposit in Poltava Region consists of four sites: *Voronky* site, *Melikhivka* site, *Senchanka* site, and *Dubrovka* one. Total reserves of the sites are 504.5 mln tons. *Melikhivka* site was prospected preliminary; other sites are under prospecting. Lignite deposits of *Melikhivka* site is represented by two layers – upper layer and lower one. According to the preliminary data, thickness of the upper layer is 2.7 m; thickness of the lower layer is 3.8 m. Occurring depth of productive layers varies from 16 m to 112 m. Working moisture is 58.8% to 60.2%; sulphur content is 1.46% to 1.7%; heat of the fuel combustion is 2080 kkal/kg. A site, allocated previously for open-pit mining, has average opening coefficient at the level of 9.5 m<sup>3</sup>/t; reserves are almost 100 mln tons. The lignite grade is understudied.

### **Analysis of practices concerning the development of lignite deposits**

In the mid 1940s, economic research made it possible to find an opportunity for the construction of electrical and chemical integrated works on the basis of Ukrainian lignite. The integrated works was a part of thermal power plant with 222 thousand kW capacity, semi-coking plant with the output of 2 mln tons, and by-product processing facilities. According to data by [6], semi-coking would make it possible to generate the following from a kilogram of the lignite: 66 % of semicoke, 8.95 % of resin, 0.46 % of benzol, and 100 m<sup>3</sup>/t of gas. On condition that 4.4 mln tons of lignite would be processed annually, there was proved the possibility to produce up to 2 mln tons of semicoke; it was planned to use the amount in such a way: 1.5 mln tons would be consumed by thermal power plant; 0.5 mln tons would be used for by-product. Moreover, following amount of chemicals was also involved: 21.5 thousand tons of benzol; 75 thousand tons of crude carbolic acid; 73 thousand tons of lubricants for impregnation of sleepers; 95 thousand tons of lignite tar pitch; 9 thousand tons of paraffin; and up to 32 thousand tons of sulphur.

Scientific and research as well as design and prospecting activities persisted during post-war period. They were focused on the

substantiation of the expediency of open-pit lignite mining. Thus, in his early publications O.S. Fidelev [7-11] listed innovative for that time methods to determine possible depth of fully-mechanized open pits with the substantiation of expedient parameters of their development. There were considered conditions for possible use of German transport-and-dumping bridges in complex with multi-bucket excavators to mine opening rocks. There were analyzed parameters of selective mining with the use of lignite and enclosing interlayers of barren rock as the associated minerals. There were identified parameters of open-pit field opening as well as expedient use of continuous equipment depending upon rock hardness. There were formulated basic requirements for preliminary drainage of watered lignite deposits and use of internal dump within the worked-out areas. There was proposed a new design for a stable dump with the stepped cross section with the formation of pre-dump with the use of sandy opening rocks. There were substantiated recommendations concerning the arrangement of dump support and excavator support to arrange the transport-and-dumping bridges on the dump, lignite and within the intermediate area [12].

Fundamental work by M.H. Novozhylov titled as “Open-pit mining” was published in 1950. The book has substantiated a tendency for the development of a complex of scientific and research activities not only for the national practices of open-pit mineral mining but also for the whole world community [13]. Along with the substantiation of engineering solutions concerning the provision of open-pit sides and protection against landslide phenomena, the book gives information on the methods of deposit dewatering, and consideration of the mineral reserves for the operation period of the enterprise. Moreover, it considers thoroughly the problems of mechanization of opening and mining activities with the use of various excavator facilities and hydraulic transport means; corrects the classification of opening schemes and systems of mineral mining; calculates and plans extraction volumes in terms of overburden and mineral in the context of different possibilities.

The methods of opening and further development of lignite deposits depend significantly on geological conditions of their occurrence. For instance, coal industries in Germany and the USA are characterized by the uniformity of deposits within the countries;

alternatively, in the context of Russia, Kazakhstan, and Ukraine occurrence conditions of deposits are rather various. It should be noted that the USA applies open-pit mining mainly for horizontal or flat seams which average thickness is 1.6 m, and maximum value is 4 to 6 m; Germany applies open-pit mining for horizontal seams which thickness is 10 to 30 m and more, occurring under overburden with 20 to 80 m thickness. Alternatively, at the territory of former USSR no less than ten types of coal deposits can be singled out. The deposits differ in their seam thickness, mode of occurrence (from horizontal to steep), hardness of coal and enclosing rocks, dilution of productive layers, water intake, climatic conditions etc. The above has become the foundations to apply various methods to open deposits; various extraction and transportation equipment as well as various mining parameters contrasting open-pit mining method applied in Germany and the USA.

Substantial volume of mining operations in the context of the national enterprises results from the following which took place while opening the deposits:

- flows of coal and rock are divided into zones which either unite several benches or separate them according to flanks;
- mobile machinery moves along the separated opening mine workings of freightless direction or freight-hauling; and
- transportation flows are organized according to flexible schedule and interchange points are transferred closer to excavator.

It should be noted that Ukrainian lignite industry was developed in a post-war period on the basis of specific deliveries of heavy mining equipment from Germany. Its task was to provide fuel (i.e. lignite briquettes) for rural areas mainly. Production of the briquettes involved a significant consumption of heat and electric energy. Hence, thermal power plants (TPPs) became a part of coal processing complexes. Neighbouring miner towns and settlements consumed a certain share of the produced heat and electric energy. Worsening of mining and geological conditions in mines and open pits, depreciation and physical wear of equipment, elimination of subsidies for the briquettes purchase by people and decreased level of their paying capacity as well as gasification of villages resulted in sharp drop of mining within the basin.

Currently, lignite is not mined in Ukraine; the product is



processed while using imported raw material only. Mining operations in Vatutino, Novomyrhorod, and Korostyshiv districts have been ceased due to low technical-and-economic indices. It is not expedient now to build new coal-mining enterprises on the account of the limited reserves, relatively high opening coefficients, and remoteness from processing plants in the context of the districts [14].

According to the data by the Ministry of Coal Industry, only *Kostyantynivsky* open pit with *Protopopivska* site and *Morozivski* open pit operated in 2005. They mined 313.4 thousand tons of lignite; 311 thousand tons were mined in 2006; 211 thousand tons were mined in 2007; and 40.7 thousand tons were mined in 2008. Then, the mining operations were suspended due to tear and wear of extraction facilities and underfinancing of the holding company.

It follows from the performed geological and industrial review that depending upon the occurrence mode of lignite deposits, their grade and amount of the commercial reserves it is expedient to restart inactive mining operations in *Kostyantynivsky* and *Morozivski* open pits. However, their commercial reserves and operational performance are not sufficient to meet energy requirements of the region. Thus, Novo-Dmytrivka deposit should be noted among the new top-priority lignite-mining objects to be built since it can be used as the basis for the development of a powerful fuel and energy complex. It should also be noted that several analogous basin-like lignite deposits (i.e. Bantysh deposit, Stepivka deposit, Bereka deposit, Biliaevka deposits and others) have been prospected in the neighbourhood of Novo-Dmytrivka deposit. They are ten salt-dome lignite deposits with gigantic reserves above salt rods. The lignite is characterized by high output of resin (up to 18.5%) and bitumen (up to 10 – 15 %) [15].

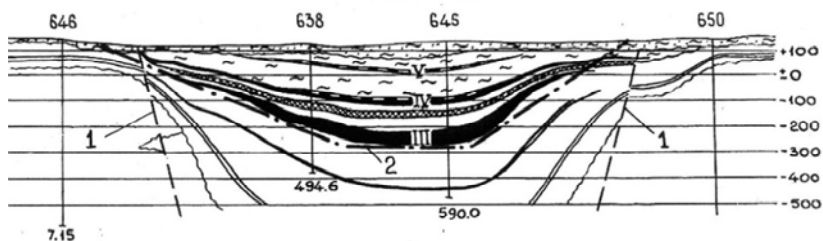
Relying upon the world practices and previous studies, the State Scientific and Research Design Institute of Coal Industry, *Ukrenergoprom* Institute, and LAUBAG Company [16] have proved it is expedient to use lignite for heat and electric energy generation while building fuel and energy complexes in the place of its extraction (on the basis of Novo-Dmytrivka deposit). Production of synthetic liquid fuel and lignite-based gas is both possible and useful owing to the number of the available engineering solutions [17-26].

Moreover, the deposit may be used as the basis to produce montan wax, humic fertilizers, sorbents, coal-alkali reagents, and building materials [27-34].

German Company LAUBAG made a forecast concerning the restart of lignite industry in Ukraine; according to the forecast, the use of reserves, having been out of operation lately, is interesting from the commercial viewpoint. First of all, it is planned to restart mining operations in the *Kostyantynivsky* and *Morozivski* open pits which have already been opened and equipped; their reserves are 48.6 and 20.1 mln tons respectively; their projected annual outputs are 2.3 and 1.5 mln tons respectively. In future, the additional development of Verkhniodniprovsk deposit is planned with 146.4 mln tons of reserves, Novo-Dmytrivka deposit with 390 mln tons of reserves, and Sula-Udai deposit with 130 mln tons of reserves and annual productivity of 4.7; 8 – 10; and 2 – 4.6 mln tons respectively [16].

### **Geological and industrial description of Novo-Dmytrivka deposit**

Currently, Novo-Dmytrivka suprasalt depressive basin (Fig. 2) has been investigated thoroughly and estimated industrially; genetic type of lignite deposits, being new for Ukraine, is connected with it. According to the conditions of the mineral complex origination and the lignite reserves, the deposit is among the unique ones. Its industrial coal-bearing properties depend on Oligocene deposits (Bereka series) and Miocene deposits (Poltava series). Structurally, they form syndepositional trough where amplitude along the floor of the basic coal lens is 330 to 360 m. In terms of three productive levels, commercial reserves of lignite are 394 mln tons, including 50.8% of A category, 21.8% of B category, and 27.4% of C1 category. Moreover, beyond the boundaries of original walls of the depression, non-commercial reserves have been estimated at a level of 52.5 mln tons (south-east part of the deposit). Hence, lignite reserves within the basic productive levels (i.e., 3<sup>rd</sup>-4<sup>th</sup>) are 446.5 mln tons; and total geological reserves are more than 600 mln tons. Basic coal-bearing capacity belongs to a central part of the depression where coal lenses are of maximum thickness (74 and 37 m) and coal-bearing coefficient is 34% [35-38].



**Fig. 2.** Geological section of Novo-Dmytrivka deposit:  
 1 – a boundary of a salt rod; 2 – boundaries of open-pit field;  
 III, IV, V – commercially important coal levels

Coal levels one and two are of minor thickness (to compare with other levels) in terms of their deeper occurrence depth; thus, they have not any industrial importance and cannot be considered as non-commercial reserves.

Level three of the deposit is of simple structure (i.e. it has no rock interlayers); it occupies 7.8 square kilometers within 2.0 m thickness forming genetically a syndepositional basin. Its thickness decreases gradually towards boundary walls; at the angle of 8 to 12° it thins away completely. The lignite grade is rather high: ash content is 5.6 to 15.8%; sulphur content is 2.0 to 3.2%; and combustion heat is 6700 to 6900 kkal/kg. Wax content is 56 to 62%; output of humic acids is 47.0 to 65.5 g/m<sup>3</sup>; and technological grade B is 1.2. The commercial lignite reserves are 296.1 mln tons; ash content of 290.7 mln tons of them is less than 20%.

Level four is characterized by complicated structure (2-3 members) and maximum thickness of 37 m. Technological lignite grade is B – 1; ash content is 8.1 to 29.2%; sulphur content is 2.6 to 4.1%; and combustion heat is 6350 kkal/kg. Bitumen output is up to 13%; output of humic acids is 47.7 to 76.0 g/m<sup>3</sup>. Commercial reserves are 98 mln tons.

Level five consists of two coal members with 2.5 and 5.4 m; total value is 8.2 m. It is characterized by high ash content (15 to 45%) and non-commercial reserves (58.3 mln tons). Modern techniques make it possible to apply such coal as low-grade fuel. At the same time, two above-mentioned levels may become a fuel source for

thermal power stations, a material to produce coal briquettes, and a raw material for chemical and technological processing to generate petroleum products, montan wax, and humic acids. Total lignite reserves of the three productive levels are more than 452 mln tons.

Carbonaceous clays, diatomites, native sulphur, fireproof and ceramic clays, glass sands and building sands as well as lead-zinc ores and mercury ores within original walls of the depressive basin are among the associated minerals.

Reserves of the carbonaceous clays with 35 to 45% of organic material are 480 mln cubic meters or almost 1 billion tons; sulphur reserves (if concentration is more than 7% to be minimum commercial content) are more than billion tons, and diatomite reserves are up to 160 mln cubic meters. Since 2001, Donetsk State Regional Geological Enterprise (town of Bakhmut) has been engaged in prospecting and evaluating activities within the walls of Novo-Dmytrivka depression to survey zink-lead ores.

Thus, Novo-Dmytrivka deposit should be considered as a complex one taking into account the availability of mineral being important for Ukraine. The minerals are 85 to 90% of the productive thickness overburden.

The current state of Ukrainian power industry and permanent deficit of energy carriers need immediate development of the deposit as a raw material base for active thermal power station. The detailed analysis of both grade and technological properties of the lignite demonstrates their complete applicability to be used as power fuel [2]. The performed calculations mean that Novo-Dmytrivka deposit can be used as the basis to build thermal power station with 2400 MW capacity and provide it with fuel for the period of 60 to 70 years.

### **Substantiating the prospects of Novo-Dmytrivka open-pit building**

As of 2018, Ukraine has extremely limited possibilities of coal mining to be independent in the progress of its heavy industry. It is particularly important for lignite. Oleksandriia lignite district has no long-term perspectives to support its productivity. Other promising deposits are not developed, and some deposits are under prospecting and detailed exploration. At the same time, Novo-Dmytrivka deposit

has been explored thoroughly; its reserves are considerable. Thus, the deposit may be taken as an example to restart the operations of coal industry in the near future [39].

Novo-Dmytrivka lignite deposit differs from Oleksandriia deposits not only in significant reserves of various minerals but also in the complicated occurrence conditions due to considerable depth and water intakes which may affect the economic expediency of raw material extraction [40]. Hence, its development should involve pre-estimation concerning the economic feasibility to build mining enterprise. In such a case, it makes sense to analyze the available mining systems used during many years to extract manganese ores and lignite in Ukraine along with the considered mining techniques involving water pressure [41]. Minimum value of the reduced costs to mine basic mineral (i.e. a ton of the raw material) is applied as the efficiency criterion of the considered systems. Thus, substantiation of economic expediency to develop Novo-Dmytrivka deposit with further construction is quite topical scientific and practical issue.

Basing upon the practice of opening activities, open-pit field can be opened with the help of the two high-productive rotary and conveyor systems: system 1 consists of a rotary excavator CPC-2000 which theoretical efficiency is 4900 m<sup>3</sup>/h, conveyor transport, interbench loader and a stacker of ARs-B 5000.60 type. The system operates starting from the initial construction stage moving gradually to lower working levels of the open pit; system 2 is as follows: opening operations are planned to be performed with the use of more powerful rotary excavator SRs-6300 which theoretical efficiency is 14000 m<sup>3</sup>/h and where rock is loaded on a belt conveyor with the interbench loader and stacker [42].

It is wise to perform mining by means of mechanical shovel, where capacity of a bucket is not less than 12 cubic meters, completed with dumping trucks. Such a composition corresponds to the increased strength of lignite within the rock mass; moreover, the method has been widely piloted in open pits having similar mining and geological conditions.

Preliminary estimation of capital investment to purchase and mount the mining equipment is USD 680.7 mln (Table 1). Operational costs for the lignite extraction were determined relying upon the recommendations of German “Master plan of lignite

industry development in Ukraine”. According to the technique, the estimated extraction costs differ from classical prime cost by the level of depreciation charges and involve the discounted equipment cost.

Operational costs are determined on the basis of the applied equipment. Workforce requirements depend upon the mounted technical facilities of the open pit and its operational mode (i.e., one-shift control and maintenance of the equipment, and three-shift production process).

A value of other categories of staff relations has been calculated according to Ukrainian standards. Workforce requirements to perform auxiliary and intermediate operations were estimated using the practices of Eastern European coal enterprises. For the selected types of costs, following prerequisites of their estimation have been calculated:

- depletion period is 36 years; and
- lignite mining is expected after 6 years of the point when equipment started to be mounted and auxiliary works concerning the open pit construction and lignite seam opening began.

**Table 1**

**The expected investment for the purchase of mining equipment to be used while Novo-Dmytrivka open pit operating**

<b>Capital equipment</b>	<b>Quant ity, units</b>	<b>Cost, USD thousands</b>	<b>Cost includ- ing VAT, USD thousands</b>
Excavator with no less than 12 m <sup>3</sup> bucket	1	3 754	4 505
Rotary excavator SRs-2000	1	36 993	44 392
Rotary excavator SRs-6300	2	177 878	213 454
Stacker of ARs-B 5000.60 type	1	16 614	19 937
Stacker of Spreader ARs- (K) 8800.195 type	3	144 078	172 894
Reloader of BRs (K) 1800.65 type	1	4 543	5 452
Reloader of BRs 1600.47/72 type	2	23 364	28 037
Face conveyor	2	23 665	28 398
Frontal conveyor	1	14 732	17 678
Main conveyor	2	29 446	35 335
Stacking conveyor	2	36 265	43 518
Dumping trucks	5	4 358	5 230
Unaccounted mining equipment (10 %)		51 569	61 883
<b>Total</b>		<b>567 260</b>	<b>680 712</b>

Maintenance cost depends upon the annual investment total:

- infrastructural devices – 2 %;
- chain and rotary excavators – 3 %;
- belt stackers– 3-3.5 %;
- dragline excavators, face shovels – 4 %;
- auto dumpers – 6 %;
- auxiliary facilities – 3 %;
- electric energy – USD80/MW excluding VAT; forecast of retail tariff for electric energy consumption for industry is prescribed at the level of class one;
  - average expenditures connected with a salary of a worker are USD 850/month;
  - payroll tax is 22%;
  - annual administrative cost and total production cost are USD 3 mln; and
  - land tax, rent payments, and other expenses connected with unaccounted cost.

Extra operational expenses of a mining enterprise have been calculated for a period of the projected capacity attainment (9 mln tons a year); they are USD 4.71/t.

The data, concerning the capital investment and operational cost, show that despite significant expenses connected with the purchase of mining equipment, operational cost to extract a ton of lignite are expected to be USD 4.7 when the open pit has attained its projected capacity. If combustion heat of lignite is 2527 kkal/kg, then extraction cost of a ton of a fuel equivalent is USD 13.1 excluding VAT.

However, such statistical data as CAPEX and OPEX prevent from judging about the operational efficiency on one or another deposit on the whole. It is usual to make the evaluations basing upon the cash flow forecast.

Hence, to have the opportunity to substantiate the expedient development of certain open-pit sites, it was decided to adopt a method turned out to be reliable while developing the calculation of lignite industry and power industry applied in Germany and represented in the “Master Plan of Lignite Industry Development in Ukraine” formed by German lignite and energy groups of companies. According to the Plan, alternatives of the development of

the projects are estimated by means of general internationally recognized methods of the dynamic business calculations. In this context, annual expenses and revenues are determined either for the whole period of a deposit operation or for the period of extraction planning.

Expenses and revenues have been mutually compared in terms of financial and mathematical averaged expenses and then determined with the help of the indices:

- actual cost in terms of different calculation types of interest rates;
- financial and mathematical averaged costs (FMACs); and
- a level of internal interest rate.

Prerequisite to apply methods of dynamic business calculations is as follows: each investment cost and current production cost is recorded on operating years of the whole period being the analysis in the form of a payment line. Neither depreciation deductions nor interests are involved. Possible interest rate on the loan or the invested capital value is involved in shares while cash flows forecasting. Unreported income is processed similarly. Annual excess of the income above expenses as well as excess of the income above expenses for the whole operational period of the deposit is also involved in the calculation process. Thus, operation of the enterprise should provide positive net cash flow (NCF). In this context, following ratio should be met:

$$NCF \geq 0 \quad (1)$$

As a rule, the payments are different at certain dates and certain years. To make them comparable, they should be estimated for a certain date of operation with the help of imputed interest rate. As a result, it is guaranteed that the payments which should be made later are insignificant to compare with those to be made earlier. The applied imputed interest rate depends upon the interest rate for long- and medium-term credit and the accepted amount of the charged interest. If the forecasts for future potential interest and expected investment risk are dangerous, then higher interest rates are usually involved. As a rule, starting point of the whole operational period or starting point of a production process is considered. Total payments of the date are indicated as net assets value.



Implementation of the projects is rational from the viewpoint of production and economy if in terms of the stated imputed interest rate (discount interest rate) difference between the net assets value and investment is at least equal to zero; positive difference is much better. As a rule, in the context of firmly stated imputed interest rate, such an interest rate is fixed when net assets value tends to zero. The interest rate is called internal one.

Application of the methods means that along with production costs, the expected prices for the end product are also known; moreover, income for the whole project period is earned. It often happens that prices for commercial output are unknown, unapproved, and unstudied. If so, then the averaged financial and mathematical expenses are set; the expenses may be considered as specific forms of capital cost representation. The calculated cost of the enterprise corresponds to the financial and mathematical value of a product unit which should be identified for the whole operation period. In other words, net assets value of the income excess above expenses will tend to zero. In this context, following ratio should be met:

$$NPV = 0 = -(CAPEX_i + CAPEX_{(i+1)} + CAPEX_{(i+n)}) + NCF_{(i+n+1)}/(1 + IRR) + NCF_{(i+n+2)}/(1 + IRR)^2 + NCF_{(i+n+3)}/(1 + IRR)^3 + NCF_{(i+n+4)}/(1 + IRR)^4 + NCF_{(i+n+5)}/(1 + IRR)^5 + (NCF_{(i+n+6)}/IRR)/((1 + IRR)^5) \quad (2)$$

If so, income excess above expenses is set basing upon the expenses and expected earnings from the end product sales; besides, its value is set as that one at the level of FMAC.

To consider and evaluate a new deposit in comparison with other ones, it is required to determine cost values as well as financial and mathematical expenses for the enterprises separately. As a result of the FMAC calculations, following values concerning the operation of an enterprise on the basis of Novo-Dmytrivka open pit have been obtained (Table 2).

**Table 2**

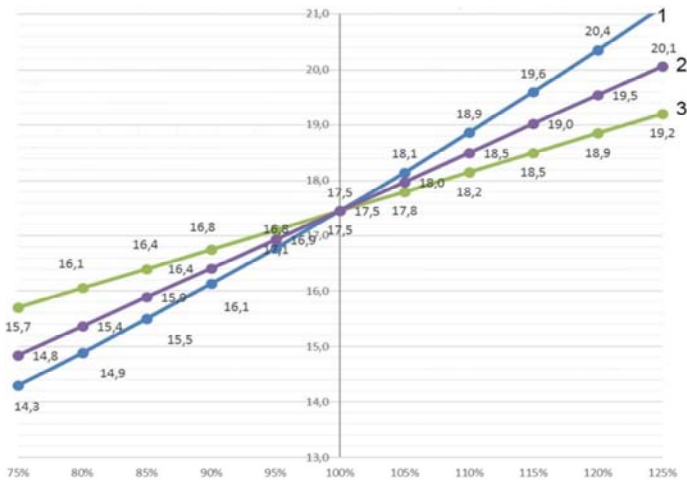
**Financial estimation of Novo-Dmytrivka open pit operations**

Index	Measurement unit	Cost connected with extraction of a ton of raw coal	Cost connected with generation of a ton of fuel equivalent
Capital cost	USD/ton	10.5	29.0
Operational cost	USD/ton	7.0	19.4
Total cost	USD/ton	17.5	48.3

Data from Table 2 indicate that in the context of implementation of the 10% interest of the rate of return on invested capital, the price for raw lignite extraction should not be less than USD 17.5 per ton excluding VAT or USD 48.3 per ton in the context of fuel equivalent.

The calculated price for fuel is quite comparable with the price for black coal extraction. Since the calculations involved certain share of assumptions, to determine stability of the FMAC-based price it is required to analyze its sensitivity according to the parameters: change in interest rate on capital is  $\pm 25\%$ ; change in operating cost is  $\pm 25\%$ ; and change in capital investment is  $\pm 25\%$ . Fig. 3 represents modeling of the fuel prices.

Data in Fig. 3 explain that the project to develop Novo-Dmytrivka deposit is mostly sensible to capital cost; the least sensibility is in production cost. It means that the improvement of financial stability of Novo-Dmytrivka-based mining enterprise should involve: the required capital raising with the lowest percent interest rate and downward capital investment optimizing. Reduction of operational cost is only possible at the expense of the electricity generation (construction of own technological thermal station) and income increase resulting from marketing of products.



**Fig. 3.** Analysis of sensitivity of fuel price FMAC, USD/t: 1 is interest rate change; 2 is change in the amount to be invested; 3 is change in production operational cost.

Reserves of Novo-Dmytrivka deposits involve carbonaceous clays referring to the Lower Pliocene (above coal deposit 5) and to Poltava Miocene formation (between coal deposits 4 and 5); the clays occupy the area of more than 6 square kilometers.

Average thickness of the carbonaceous clays is 75 meters and their previous reserves are more than 450 mln cubic meters; in terms of specific weight being 1.35 t/ m<sup>3</sup> they are more than 600 mln tons. Carbonaceous clays have never been studied before as energy raw material. On the analogy of Dnipro lignite basin, their ash content and combustion heat with a view to combustible mass and as-received fuel are 4500-5600 and 700-900 kkal/kg respectively. Typical ash content varies within 55-65%.

The research helped understand that the carbonaceous clays overlap both *Verkhni* and *Skladny* coal levels. Taking into consideration their high ash content and low combustion heat their use as a certain associated raw material is not possible. However, in a mixture with rough lignite, the rock mass (rough coal + carbonaceous clay) is energy raw material suitable for its further use. For a comparison: lower combustion heat of rock mass with a value of 1825.5 kkal/kg corresponds to so-called power generating coal used to be burnt at lignite thermal stations 1 and 3 in the town of Oleksandriia. Co-extraction of the rough lignite and carbonaceous clays makes it possible to increase the mineral amount by more than 20 mln tons a year with high economic indices (Table 3).

**Table 3**

**The data concerning calculations of lignite and carbonaceous clays extraction according to FMAC, USD/ton**

Index	Rough lignite extraction, mln tons a year		Carbonaceous clay extraction, mln tons a year	
	value	t.r.f.	value	t.r.f.
Capital cost	10.5	29.0	4.7	18.0
Production cost	7.0	19.4	3.2	12.1
Total	17.5	48.3	7.9	30.1

The data from Table 3 explain that co-extraction of the rough lignite and carbonaceous clay makes it possible to reduce the calculated price for fuel by 60 %. Analysis of the indices sensitivity

according to FMAC calculations has been carried out similarly as for the rough lignite extraction. Previous technical and economical modeling of Novo-Dmytrivka deposit mining demonstrates the feasibility of lignite complex use taking into consideration the extraction of carbonaceous clays. Despite the necessity in significant capital investment, integral financial and economic indices of the mining enterprise are high.

**Conclusions.** All known Ukrainian lignite deposits, occurred above salt rods, are complex ones; that is they involve other minerals in addition to lignite. Their list varies depending upon deposits: from diatomite and fireproof clays to building and glass sands. Aside from its basic use, lignite may be applied for briquettes and production of montan wax being valuable raw material for many industries.

Taking into account the form of Novo-Dmytrivka deposit and its borders, it is expedient to consider pivoting mining method with the opportunity of internal dump formation making it possible to use parameters of mining and loading equipment more effectively. The use of productive rotary and conveyor scheme to extract minerals regulates timing of mineral mining since processes of rotary and conveyor system production and maintenance may achieve three years.

To reduce introduction period of a mining enterprise and to accelerate its attainment of projected capacity it is wise to consider the use of cycle facilities as a part of hydraulic excavators and open-pit dumping trucks.

Innovative techniques and equipment were developed for selective mining of lignite and associated minerals, their transportation and transient dumping. Period of their reuse and supply to consumers help provide such related industries as building industry, metallurgical industry, chemical industry and agriculture with valuable mineral raw material. Its extraction from technogenic deposits will be performed with the use of the equipment systems applied while lignite deposits mining. Recommendations, being a result of the performed research, have been published. The recommendations concern the introduction of the obtained results while designing lignite open pits.

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# **MODERN GEOTECHNICAL METHODS OF MANAGEMENT OF THE PROCESS OF AMBER EXTRACTION**

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## **Abstract**

The current state of development of mining in the northern regions of Ukraine is characterized by the presence of a significant number of industrially significant deposits of amber. Large volumes of amber were found in the Volyn, Zhytomyr and Rivne regions of Ukraine. The most prominent is the Klesiv deposit, which is in the Sarny district of the Rivne region. On the territory of Ukraine, amber is found in Volyn, in the Pripyat river basin, in the vicinity of the city of Kyiv and in the Carpathian region. Many deposits are in complex mining and geological conditions and are not involved in the development due to the impossibility of their exploitation by traditional methods. The exploitation of such deposits by traditional methods is inefficient and costly.

Effectively to develop amber deposits allows the hydraulic and hydromechanical method of extraction, which is carried out with the help of hydraulic energy used to destroy amber-containing rocks and delivery of amber to the surface of the day.

The efficiency of the operation of the hydraulic equipment can be improved by improving the existing technological equipment based on the application of the system of automatic control of the process of hydro thinning. Known mining methods do not allow to fully extract amber due to the imperfection of technology for the complete removal of amber from amber-containing deposits.



The complexity and abundance of possible technological processes in the extraction of amphibious deposits of exploitation indicate that for the study and extraction of amber, there is a need for additional research. Improving the efficiency of hydromechanical complexes by reducing energy costs as the product range increases and product quality improves requires a scientific substantiation of the technological parameters of the extracted equipment.

The existing methods and means of amber extraction and the new method of amber extraction from an amber-containing sandy deposit have been analyzed, in which the high productivity of the extraction equipment is achieved and the negative impact on the environment is reduced.

## **Introduction**

In Ukraine, intensive development of amber deposits is under way. However, the complex and diverse mining and geological conditions of deposits, the imperfection of technological methods of extraction, the lack of special equipment lead to large losses of amber, especially of small fractions, as well as to disruption of the ecological situation in the mining areas. The search and implementation of new development methods will significantly expand the capabilities of traditional technologies.

The work deals with the conditions of amber extraction in the deposits of the Rivne-Volyn region, in which a large number of industrially significant sandy and sandy-clay deposits of amber are concentrated. In this region, only selected sections of the field are developed that are available for simple hydraulic well production using a hydro pump. This method is inefficient: it requires a large amount of water, amber is extracted without subsequent reclamation of the destroyed layer, with large losses in the dump of fine amber particles.

The developed hydromechanical production method for these conditions is promising, since it is carried out without destruction of the rock bed during amber mining and does not require subsequent reclamation of the surface in the area. At the same time, the high water consumption, the sources of which are limited in these areas, is a deterrent to its widespread use in sandy and sandy-argillaceous deposits. At the same time, the use of this method has shown that the effectiveness of amber extraction depends significantly on the rate of

its ascent from the well, and this requires a large flow of water. However, at present there are no studies to rationally limit the use of water without reducing production efficiency. One of the important factors is the density of the slurry, which significantly affects the rate of ascent of amber particles and the efficiency of the extraction process, but the nature of this effect has not been sufficiently investigated.

The complex effect of several mechanisms at amber extraction, for example, the creation of conditions for the ascent of amber particles of various masses with additional air flow (flotation or bubbling) and vibrational action in the working zone, deserves the attention of researchers. The effectiveness of this mining technology can be increased by selecting the dominant process factors and their rational parameters.

The aim of the study is to determine the density of the medium necessary for the ascent of amber of different sizes, as well as to determine the influence of vibration and bubbling factors on the process of ascent of amber.

## **1. Characteristics of deposits of amber and its properties**

Amber in various countries has its own distinguishing features that set it apart in different parts of the world [1]. Canadian researchers conducted the analysis of amber from 11 amber-containing fields of Canada by different chemical methods, which showed that they represented three different classes. Performed studies also showed how the trade of amber changed depending on the discovery of places of amber containing deposits. The studies indicate the features of the amber-containing fields in Canada and focus on the development of the region at increasing the extraction of amber. Physical–chemical properties of amber allow selecting a method of extraction, however, the studies have a scope that is limited by the region of amber extraction. Molecular component has little impact on production, and explores the uniqueness of a valuable sedimentary.

The same can be told about the research, during which scientists were interested in the presence of the particles and insects in amber. The impact of exploration changes the infrastructure of a region and

landscapes of areas. One needs to consider in what way extracting the amber must be carried out for the minimal technogenic and ecological impact on the environment.

Many researchers are interested in the physical and chemical properties of amber.

The extraction of amber is developed in those countries where its industrial production is in place [2].

In Rivnensky-Volynsky region of Ukraine significant fields of amber have been found. Extraction work is now underway in Klesivsky field (Sarny District) and in the areas of Volodymyrets'ky (village of Berezhnytsia) and Dubrovits'ky (village of Vilneh). The total reserves are estimated at 100 000 t, which mostly lie in the sand and sandy clay soils on the depth to 15 m and are sufficient for research and implementation of new technologies.

The discovery of industrial amber placers in Ukraine began from developing Klesivsky deposit in 1980, which is active until now. Further searching and search and estimates within the Klesivsky amber-containing area that have been carried out over the last 30 years, enlarged information about its geological structure and conditions of the formation of initial amber placers.

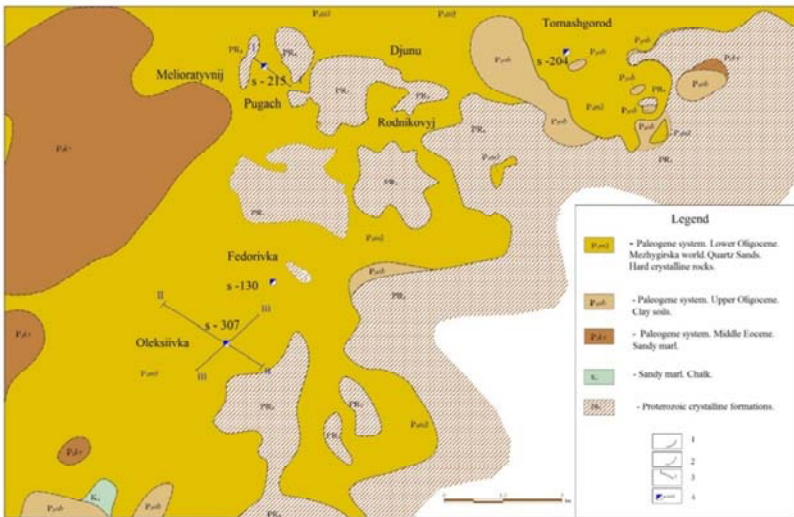
Productive (Mezhygir) deposits of the south-eastern part of the Klesivsky district (areas of Pugach, Rodnikova, Duny and Fedorivka) fill the lowering between the outputs on the pre-berrics surface of the Proterozoic crystalline formations and their weathering meales (Fig. 1). Productive fields of the Klesivsky deposit are deposited as a strip of the width from 200 to 700 m, which can be traced from the Southeast to the Northwest between the outputs of small pieces of the foundation on the pre-Mezhygir surface. The length of the stripe enriched with amber is over 2 km. The outputs of the crystalline rocks among the Mezhygir sediments are insignificant in size and in the Mezhygir time manifested themselves as numerous abrasion islands with the square about 50 %.

Extraction of amber from sandy fields are mainly carried out by two methods: mechanical and hydraulic.

Modern technologies and machinery installations can be applied for the production at industrial scale, and where production is not developed, it is necessary to use mobile extraction tools [4].

By the analysis of the scientific literature, one can conclude that

the researchers are more interested in the precious value and uniqueness of amber than in the technology. Research into improvement of technologies and technological equipment was tackled by individual scientists [2]. Further research should focus on the implementation of the newest technologies and extraction tools, optimization and intensification of the processes of extraction from amber-containing fields by using hydromechanical method for maximal extraction of amber from a deposit.



**Fig. 1.** Schematic geological map of the pre-berries surface of the southern part of the Klesivsky amber-containing district

Therefore, relevant scientific task at the amber exploration is reducing negative consequences of the application of extracting equipment on natural environment and substantiation of rational parameters of mining equipment to maximum extraction of amber from amber-containing fields.

## 2. Geotechnological methods for extracting amber

Extraction of amber in Ukraine is carried out by mechanical or hydraulic methods, which have a number of shortcomings. In

particular, these are large operational and economic costs, removal of mineral soil to the surface, altering the structure of the soils, formation of cavities and a negative environmental impact on the environment [4, 5].

### **2.1. Hydromechanical method of amber extraction**

At the National University of Water and Environmental Engineering (NUWEE, Rivne, Ukraine) hydromechanical method was proposed for the production of amber, based on saturation of the array with water, excitation mechanically (vibration excitation) and segregation of amber to the surface of the field by Archimedean force. The action and effect on the array by vibrating methods need analysis and study of the processes occurring in the amber-containing sandy soils.

When using this method, vibrating machines act on the soil environment from the top, or are put in the middle of the vibrated mass of soil. Ground mass is a three-phase dispersion system that consists of the solid (the skeleton), the liquid phase (water), which fills the pores, and gaseous (certain volume of air, water steam). The air is in the compressed state, so the bubbles hardly move relative to the skeleton. As for the liquid phase, there are two parts of water: water associated with molecular skeleton and free water, which can move around by the force of gravity and pressure gradient.

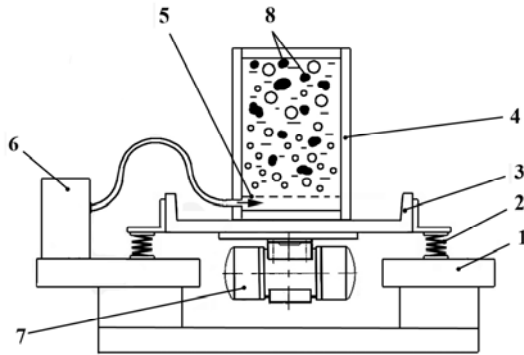
As the amber at Klesivsky field lies in sandy soils, then the sandy soil passes into a state of liquefaction at the saturation of water and mechanical action on the environment. Experimental studies of liquefaction of sandy soils were tackled by many researchers. A research of influence of vibration equipment to the soil environment was carried out by [6-8].

### **2.2. Installation for studying the main technological parameters of amber-containing sands**

For experimental studies of the influence of density environments and vibration excitation with hydro extraction method amber was created laboratory stand shown in Fig. 2.

The stand consists of the base 1, on which the body 3 is mounted

on the elastic supports 2 with a cylindrical tank 4 filled with water, a perforated surface 5 is installed at a distance from it, which forms a cavity with the bottom and walls of the tank, compressor 6 with a capacity of 20 l / min. The electric vibration exciter 7 of the brand EV63-4U3, 1420 rpm, an engine power of 0.18 kW, a maximum disturbing force of 1755 H, to create circular vibrational oscillations was attached to the body of the stand, experiments were conducted with the disturbing force of 800 H [9].



**Fig. 2.** Scheme of the laboratory stand

The principle of the stand is as follows: in the tank 4 filled with water, pieces of amber 8 of different mass were immersed, and salt or "blue clay" was added to increase the density of the liquid until the amber ascent ascertained, this process is also carried out with air supply (bubbling) and the action of circular vibrational oscillations separately from each other and together.

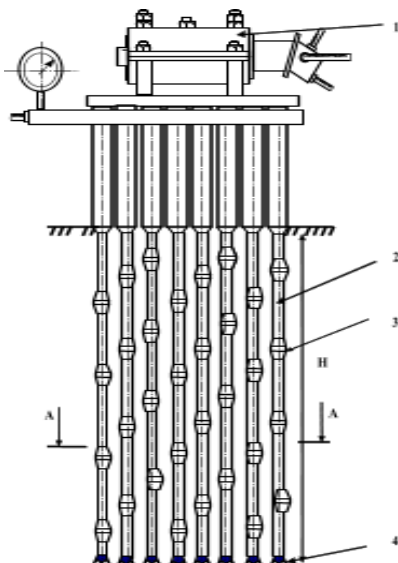
### **2.3. Industrial plant for amber extraction from amber-containing sands**

The National University of Water and Environmental Engineering has developed the method of hydraulic vibration intensifier engineering design which is based on the studies performed with the aim to determine the parameters and the design of vibrating tools intensifying the process of amber extraction.

After studying hydraulic vibration intensifiers [10], it has been found out that in order to determine the main intensifier parameters the following soil characteristics are determined on the basis of geological studies in the first stage: soil type, soil moisture, soil porosity, pore volume reduction factor, skeleton density.

The greatest rate of ascent ( $v = 0,1 \dots 0,15$  m/s) is provided by the medium density, which being based on experimental research has amounted to  $\rho_v = 1600 \dots 1850$  kg/m<sup>3</sup> [11].

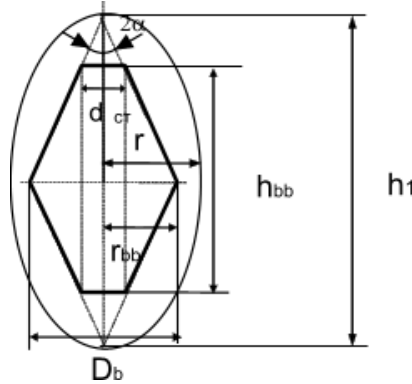
Focusing on the maximum depth of amber depositing, the length of its working bodies ( Fig. 3) is marked, the distance between the rods of the intensifier is accepted at a rate of overlapping of working elements zone in the horizontal and vertical planes. The zone of the working elements depends on physical and mechanical soil properties.



**Fig. 3.** General scheme of intensifiers

(1 - vibration exciter ; 2 - guiding rods ; 3 – vibrating irradiators ; 4 - cone tips).

Working element is designed in the form of biconic vibrating irradiator and the radius size that is defined by the formula depending on the height of the ellipsoid with the conicity of a biconic vibrating irradiator that amounts to  $15 \dots 21^\circ$  ( Fig. 4).



**Fig. 4.** Determination of geometric parameters of vibrating irradiator

The radius of the vibrating irradiator is determined as:

$$r_{bb} = \frac{h_1}{2} \operatorname{tg} \alpha \quad (1)$$

where:

$h_1$ - ellipsoid height of unstable layer of soil;

$\alpha$  - acuity angle of vibrating irradiator .

The volume of vibrating irradiator ( $V_{bb}$ ) in the ellipsoid volume ( $V_e$ ) corresponds to the pore decrease volume ( $\Delta\eta$ )

$$V_e = \frac{V_{bb}}{\Delta\eta} \quad (2)$$

The volume of biconic vibrating irradiator is determined by the expression

$$V_{bb} = \frac{2}{3} \pi \cdot r_{bb}^2 h \quad (3)$$

where:

$r_{bb}$  - vibrating irradiator radius

Ellipsoid radius ( $r$ )

$$r = \sqrt{\frac{3V_{bb}}{4\pi h_1 \Delta\eta}} \quad (4)$$

The number of vibrating irradiators ( $N_e$ ) at a packing density of  $\bar{P} = 0,74048$  (for maximum overlapping of vibrating irradiators



operation zones the packing density is  $\bar{P} = 0,74048$ ) in a particular volume which depends on the working depth and the ellipsoid size which is created around the biconic vibrating irradiator and determined by the equation:

$$N_e = \frac{ABH}{4\sqrt{2}r^2h_1}, \quad (5)$$

where:

A - length, B - width, H - height of the intensifier working area.

The volume occupied by ellipsoids in the bulk solid

$$W_e = \frac{\pi ABH}{3\sqrt{2}}. \quad (6)$$

The determination of vibrating irradiators quantity and the volume occupied by ellipsoids in the volume of bulk soil is performed with the help of a computer program.

The coordinates of rods shifting in the volume of bulk soil are determined as:

By width  $A'D' = r$ .  $A'D'$ ,  $O'D'$ ,  $M'S'$  – the distance between adjacent rows (Fig. 5, 6). By length  $O'D' = r \frac{\sqrt{3}}{3}$ . By depth

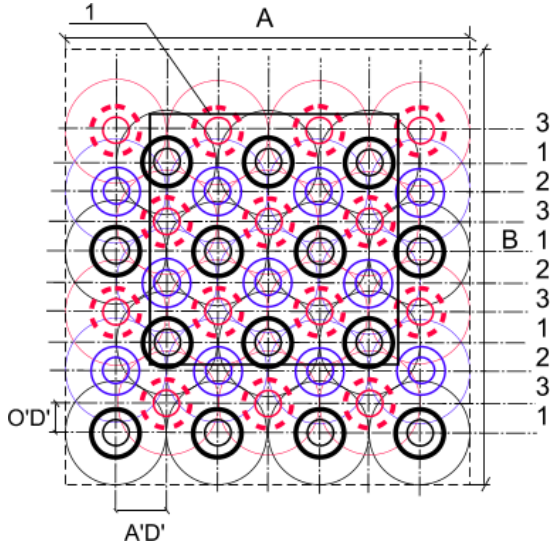
$$M'S' = 3h_1 \frac{\sqrt{2}}{\sqrt{3}}.$$

To determine the water expenditure, when the process is provided with a supply of water to the array without air quantity of the water supplied to the array of soil is determined by the amount of pore filling and water flow is calculated by the as follows:

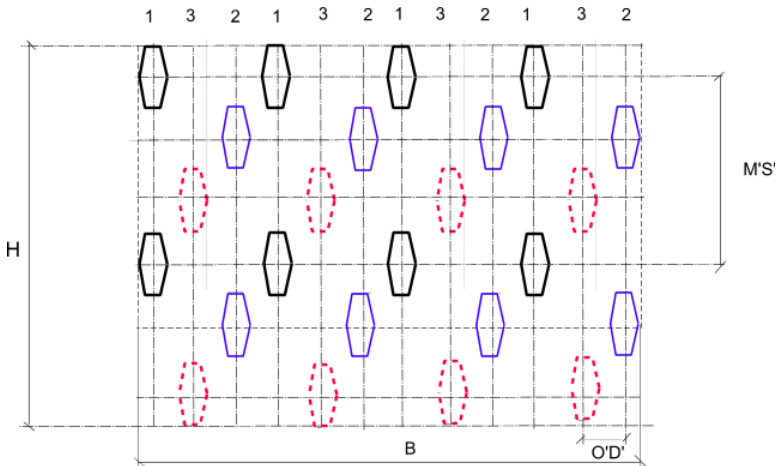
$$Q_{ps} = \frac{ABg}{1+e} \left( \frac{(\rho_{ck}(1+W) - \rho_p(1+e))}{\rho_p - \rho_g} \right), \quad (7)$$

where:

$\rho_p$  - medium density; e - the coefficient of soil porosity;  $\rho_g$  - working fluid density;  $\rho_{ck}$  - soil skeleton density; W - environmental humidity; v - installation deepening velocity.



**Fig. 5.** Location of vibrating irradiator along the rows in the horizontal coordinates (1, 2, 3 - consecutive rows of rods in a vertical way)



**Fig. 6.** Location of vibrating irradiators along the rows in the vertical coordinates (1, 2, 3 – rows of rods)

To determine the expenditure of air supplied to the array of soil saturated with water (at the rate of pore filling), the following relationship intensifying the process of amber surfacing and creating conditions for lifting it to the surface is considered.

Air flow along the supply is determined by the expression :

$$Q_{nos} = \frac{n_1 AB \mathcal{G} (\rho_s - \rho_p) + \frac{AB \mathcal{G}}{1+e} (\rho_{ck} (1+W) - \rho_p (1+e))}{\rho_p}, \quad (8)$$

where:

$n_1$  – medium porosity in the natural state.

Power ( $N_{np}$ ), which is necessary to maintain the workflow is determined as follows:

$$N_{np} = N_{\epsilon i \bar{\sigma}} + N_{pe} + N_{nos}, \quad (9)$$

where:

$N_{\epsilon i \bar{\sigma}}$  – the power, which is directly spent on the soil massif excitation;  $N_{pe}$  – the power required to supply the working fluid to the massif;  $N_{nos}$  – the power required to supply air to the massif.

Power ( $N_{\epsilon i \bar{\sigma}}$ ), which is directly spent on the excitation of the soil massif is determined by the expression:

$$\begin{aligned} N_{\epsilon i \bar{\sigma}} &= P A_{\kappa} \varpi \sin \delta = \left( m + \int_0^H \rho_p AB d H \right) A_{\kappa} \varpi^2 A_{\kappa} \varpi \sin \delta = \\ &= \left( m + \int_0^H \rho_p AB d H \right) A_{\kappa}^2 \varpi^3 \sin \delta, \end{aligned} \quad (10)$$

where:

$P$  – the maximum exciting force;

$$P = \frac{M_{\kappa} \omega^2}{g} = \frac{G A_{\kappa} \varpi^2}{g} = \frac{m g A_{\kappa} \varpi^2}{g} = m A_{\kappa} \varpi^2,$$

$M_{\kappa}$  – angular momentum;

$$M_{\kappa} = G A_{\kappa} = m g A_{\kappa};$$

$\omega$  – frequency vibration;  $g$  – acceleration due to gravity,  $g = 9,81 \text{ m/s}^2$ ;

$m$  – mass of equipment.

In determining the capacity of the vibrator motor drive it is necessary to consider the mass of soil massif in its depth, whereas the maximum exciting force is determined by the equation:

$$P = MA_{\kappa} \omega^2 = \left( m + \int_0^H \rho_p AB dH \right) A_{\kappa} \omega^2, \quad (11)$$

where:  $M$  - mass of equipment and Soil massif, ranging  $M = \left( m + \int_0^H \rho_p AB dH \right)$ ;  $\rho_p$  - medium density;  $A_{\kappa}$  - fluctuations amplitude.

Resulting from the experimental research the fluctuations amplitude is determined by the equation:

$$A_{\kappa} = \frac{\Delta h}{(0.28 \dots 0.32)h}; \quad (12)$$

where  $\Delta h$  - soil layer increase;  $h$  - soil layer height in its natural state.

$A, B$  - width and length of the soil massif that varies;  $\delta$  - phase angle,  $\delta = 20^\circ$ ;  $H$  - depth of vibrating boiling soil layer.

The power of vibrator motor drive is determined as follows:

$$N_{\text{vib}} = A_{\kappa}^2 \omega^3 \sin \delta \left( m + \rho_p AB \int_0^H dH \right); \quad (13)$$

The power of vibrator motor drive necessary for the stimulation of soil massif depends on the exciting force, amplitude and frequency of vibration, medium porosity, installation and soil massif mass as well as on the depth of the installation working zone.

The power ( $N_{p\phi}$ ), required to supply working fluid into the massif is determined by the equation:

$$N_{p\phi} = p_{p\phi} Q_{p\phi}; \quad (14)$$

where:  $p_{p\phi}$  - fluid pressure, MPa.

Power ( $N_{no\phi}$ ), required to supply the air into the massif is determined by the equation:

$$N_{no\phi} = p_{no\phi} Q_{no\phi}; \quad (15)$$

where:  $p_{no\phi}$  - air pressure, MPa.

The frequency of vibration is determined by the vibration

recorder . The optimum values of medium density ( $\rho_p$ ) are determined due to experimental studies.

The efficiency of hydraulic vibrating intensifiers is expressed as:

$$\Pi_{\epsilon.\text{itm.}} = 3600 \cdot \frac{nW_e K_\epsilon K_{nep}}{t_y}. \quad (16)$$

where:  $W_e$  - soil massif volume treated by the intensifiers;  $t_y$  - device operation time;  $n$  - number of cycles of immersion and surfacing from the soil massif;  $K_\epsilon$  - the coefficient calculating the use of working time;  $K_{nep}$  - the coefficient calculating the time to reshuffle intensifiers .

To determine the amounts of soil massif processing, the efficiency of hydraulic vibrating intensifiers is considered.

### **Results of experimental research into extraction of amber out of sandy fields**

Main experimental studies were performed in the laboratories of the University, Rivne geological expedition of the enterprise "Ukrpivnitchgeologiya" and the Klesivsky amber-containing sandy field, which is the most characteristic of this amber-containing region. The research found that the structure of sandy amber-containing environment is destroyed during liquefaction. Sand particles in the vibration zone, are separated from the total array and driven to vibrational motion at their equilibrium position and move by certain trajectory relative to the vibration projectile. In this case, there is an intense movement of gas and water that takes along sand particles and amber and throws them to the surface. Since the surface of amber considerably exceeds the square of the particles, then under the influence of Archimedean force the pieces of amber are pushed out to the surface.

At vibration action on a sandy array of soil, the following stages of conversion are observed:

- vibration liquefaction (maximum preparation for intensive mixing);
- vibration boiling (separation of particles and mixing in the array);
- the gradual compacting of an array of sandy soil from the

periphery to the source of vibration.

The point of transition from a state of vibration liquefaction to vibration linearization of dry friction is characterized by maximal sealing of the material and maximal resistance of the soil layer.

The effect of vibration liquefaction of the layer is analogous to the phenomenon of vibration linearization of dry friction, i.e. in the presence of vibration, transfer of relative motion to a particle in the environment needs less permanent effort than in its absence. When reducing the effective coefficient of friction in the vibration liquefied state, the particles slide relative to each other without tearing away. The layer is compacted while flowing.

It is known from the research [2] that sand displacement in the vibration boiling layer is not subject to the law of motion of the particles in the airless space. In addition to gravity, the environment produces significant influence on the trajectory of the movement of a layer of sand. Liquefaction forms during throwing, while during falling, the pressure increases in the medium. Greater pressure drop is executed on the lower layers of sandy soil than on the top, so the air is displaced from the bottom and the condensation occurs between the particles.

Thus, the vibration boiling layer of sandy soil acts as a pump that is pumping gas-liquid mixture to the surface, capturing the particles and transporting them to the top. In this case the rate of rise of a particle from the bottom to the top depends on the intensity of vibration excitation of the array, liquefaction of the medium, saturation by the air bubbles and the viscosity of the medium.

Pressure drop depends on the frequency and amplitude of the fluctuations, the height of the layer, particle size and moisture content of the sandy soil, and also on the coefficient of friction of particles one by one. The intensity of the pumping action of the vibration boiling layer is characterized by three parameters: pressure over and liquefaction under the vibration boiling layer, the difference of pressure in the layer.

Pumping effect is ensured during vibration in the following sequences: 24 – 26 Hz; 48 – 52 Hz; 96 – 104 Hz.

Hence, the formation of vibration boiling soil layer is affected by the following parameters:

- 1) amplitude of oscillations;

- 2) frequency of fluctuations;
- 3) forcing power;
- 4) water pressure;
- 5) air pressure;
- 6) geometric location of fluctuations exciter.

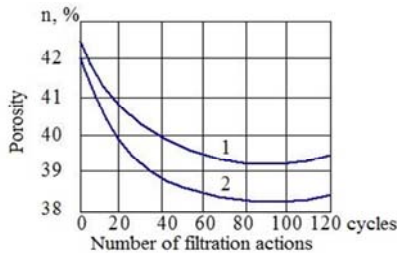
To a large extent these parameters are determined experimentally.

A significant factor that affects the loosening of the sand mass, is the porosity ( $n_1$ ) of the environment, which is determined from the expression:

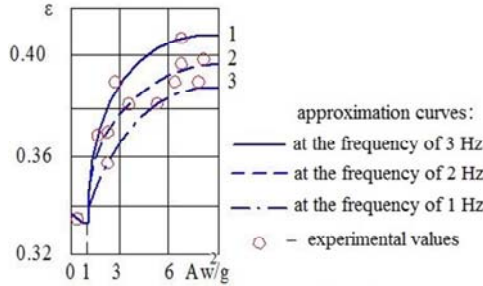
$$n_1 = \frac{(V_w - V_c)}{V_w}, \quad (17)$$

where  $V_w$  is the full volume of the soil layer;  $V_c$  is the volume of solid particles.

By the results of theoretical and experimental research into behaviour of sands under mechanical action on the change of aggregate state in the environment, minor changes in the array porosity were observed, namely within  $\Delta\eta = 1 - 1.5 \%$  [6] (Fig. 7). The liquefaction of environment is estimated by the porosity of vibration boiling layer that depends on the acceleration of vibration. It was found that with insufficient liquefaction the compression of a soil layer occurs, which leads to significant reduction of the amber recovery to the surface. Research into the changes of porosity ( $\epsilon$ ) of vibration boiling soil layer of the quartz sand under the acceleration of vibration on a laboratory installation allowed obtaining data, based on which a graphical dependency that is presented in Fig. 8 was built.



**Fig. 7.** Dependence of porosity of sandy soil under the influence of an fluctuations exciter: 1 – during the intensity of vibration excitation action for 1 sec.; 2 – during the intensity of vibration excitation action for 0.5 sec



**Fig. 8.** Dependence of porosity ( $\epsilon$ ) of vibration boiling soil layer of the quartz sand on the acceleration of vibrations  $\left(\frac{A\omega^2}{g}\right)$  at different frequencies (1 – 3 Hz; 2 – 2 Hz; 3 – 6 Hz)

For extraction of amber with the use of hydromechanical method at the National University of Water and Environmental Engineering a vibrohydraulic intensifier was designed, which includes a vibrator and guide rods with vibration emitters. The parameters of a vibrohydraulic intensifier were received by the conducted research.

Experimentally was determined that the amplitude of the oscillations ( $A_\kappa$ ) of a vibration boiling layer has the link between initial porosity of the medium and incremental rise of the the latter at vibration boiling and changes to the value, which is taken into account by the experimental factor  $K$ , determined on the basis of processing experiments data by known hydromechanics dependence:

$$\frac{\Delta h}{h_{\text{noy}}} = KA_\kappa \quad (18)$$

where  $K$  is the experimental coefficient;  $\Delta h$  is the increase of vibration boiling soil layer from the initial download;  $h_{\text{noy}}$  is the initial height of the installation load.

From the research [11], by experiments in laboratory and under the field conditions, it was proved that with increasing distance from the center of the vibration, the amplitude of fluctuations fades while the forces of traction and friction between the particles of amber-containing sand decrease and gradually stop. The more the environment is liquefied, the smaller is the radius of action of



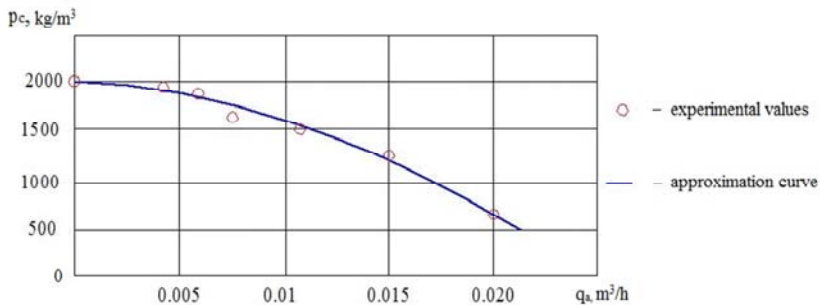
vibration projectile. When working with clay and certain loams, the efficiency of vibrators is not expedient.

To receive the data of the effective area of action of the working body of a vibrohydraulic intensifier, additional research was carried out in the laboratory to create a vibration boiling soil layer both in closed installations and in open systems.

This allowed vibrohydraulic intensifier to create a solid suspension environment and remove amber to the surface of the field by segregation in the amber-containing sandy layer.

Experimentally during full-scale research of a vibrohydraulic intensifier, the dependence of the speed of emersion of amber on the vibration parameters and the supply of gas-liquid mixture into the ground array was determined [11].

The maximum liquefaction of colloid mass (density of environment ( $\rho_c$ )) was observed in the sandy soil during the air supply to the boiling soil layer  $q_n = 0.02 \text{ m}^3/\text{h}$ . The density of the colloid mass ( $\rho_c$ ) was  $0.5\text{--}0.6 \text{ kg/m}^3$  at the frequency of vibrations of 30 Hz (Fig. 9).



**Fig. 9.** Dependence of the density of the medium on the air supply ( $q_n$ ) at the frequency of vibrations of 30 Hz

The minimal medium density ( $\rho_{max}$ ) is  $1800 \text{ kg/m}^3$  with the air supply ( $q_n$ )  $0.004 \text{ m}^3/\text{hour}$  was reached at the frequency of vibration of 30 – 35 Hz.

Air supply within  $0.01\text{--}0.12 \text{ m}^3/\text{hour}$  per  $1 \text{ m}^3$  of sand soil liquefies amber-containing sandy environment and intensifies the process of amber emersion. However, the increased consumption of air leads to a decrease in the speed of emersion. Dependence of the

change in density of the medium with the air supply to a boiling soil layer and approximation curve by the results of experimental research are presented in Fig. 10. In this case the volume of extracted amber is 90–95 % of the total reserves in the field.

Supply of gas-liquid mixture allows intensifying the process of amber recovery to maximal values, but during formation of air trunks, the boiling process passes into vibration liquefaction and eventually stops. Maximal speed of amber recovery to the surface is observed by changing the supply of gas-liquid mixture to sandy array within 0 to 0.020 m<sup>3</sup>/hour.

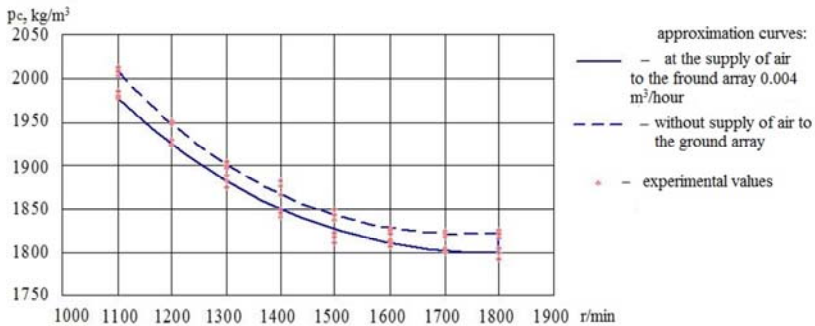


Fig. 10. Dependence of the density of the medium on vibration frequency ( $\omega$ )

Segregation in the amber layer and the speed of its recovery are decisive in the working modes of intensifiers.

### 3. Recommendations for the production of amber using a hydromechanical method using a vibration-hydraulic classifier

The data of the conducted experiments on determining the concentration of salt C (gram per liter, g / l) necessary for the ascent of amber, for various conditions, are presented in Tab. 1.

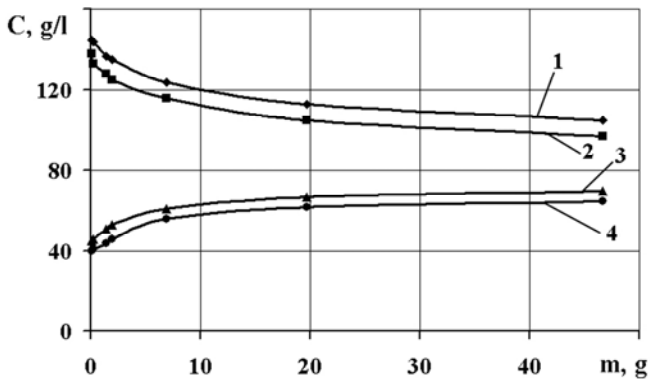
From Table. 1 it is established that the use of bubbling has a significant effect on improving the ascent of amber, with a lower salt concentration in water. To determine the nature of the dependences obtained, the graphs, shown in Fig. 11, of the required concentration

of salt C for the ascent of amber from the mass of pieces of amber  $m$  when air is supplied (bubbling) and the effect of circular vibrational oscillations separately from each other and together.

**Table 1**

**The concentration of salt C (g / l), necessary for the ascent of amber**

№	Type of additional effect	m, amber mass, g						
		0,1	0,2	1,4	1,9	6,9	19,7	46,7
1	without vibration and bubbling	145	144	137	135	124	113	105
2	with vibration	138	133	128	125	116	105	97
3	with bubbling	45	46	51	53	61	67	70
4	with vibration and bubbling	40	41	44	46	56	62	65



1 - without vibration and bubbling; 2 - with vibration;  
3 - with bubbling; 4 - with vibration and bubbling.

**Fig. 11.** - Dependence of the necessary concentration of salt C for the ascent of amber on the mass of pieces of amber  $m$ .

Since salt is a relatively expensive way to increase the density of the solution, and the use of salt in industrial conditions can lead to increased salinity of the environment, salt substitutes have been searched for in the area where amber mining is expected. The

material found is "blue clay", the deposits of which are in close proximity to the places of extraction and processing of amber. As a result of studies of this "blue clay" it was found that it contains 25% of clay, and the remaining 75% is sand. Since the influence on the increase in the density of the solution is provided by clay, the data on the change in the concentration of relatively pure clay, rather than the sandy-argillaceous mixture, will be given further. The data of the conducted experiments on determining the concentration of clay  $C_g$  necessary for the ascent of amber, for various conditions, are presented in Table. 2.

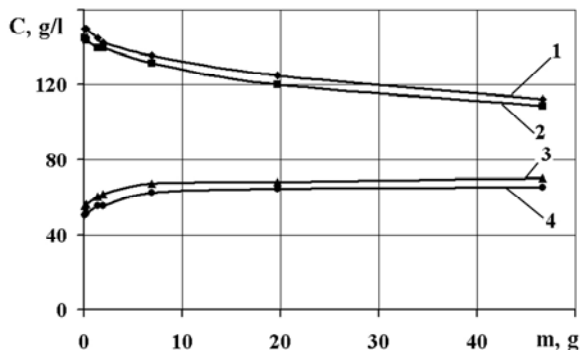
**Table 2**

Concentration of pure clay  $C_g$  (g / l), necessary for the ascent of amber

№	Type of additional effect	m, amber mass, g						
		0,1	0,2	1,4	1,9	6,9	19,7	46,7
1	without vibration and bubbling	150	150	145	142,9	135,6	125	112,2
2	with vibration	145	144	140	140	131,7	120	108,7
3	with bubbling	55	56	60	61,3	67	68	70
4	with vibration and bubbling	50	51	55	55	62	64	65

As a result of the analysis of the obtained data it was established that the replacement of salt with pure clay does not significantly affect the concentration of the solution necessary for the ascent of amber, only clays need 10-15% more. To determine the nature of the dependences obtained, plots of the necessary concentration of pure clay  $C_g$  were constructed for the ascent of amber from the mass of pieces of amber  $m$  with air supply (bubbling) and the action of circular vibrational oscillations separately from each other and together shown in Fig. 12

Taking into account the positive influence of vibration during the implementation of technology of amber extraction in sand deposits, a special design of the vibratory classifier [12] has been developed, which makes it possible to extract the amber even of small size classes with the least technological losses.



1 - without vibration and bubbling; 2 - with vibration;  
3 - with bubbling; 4 - with vibration and bubbling.

**Fig. 12.** - Dependence of the necessary concentration of pure clay  $C_g$  for the ascent of amber on the mass of pieces of amber  $m$ .

Subsequent improvement of this design allowed to create a complex action classifier in which the intensification of the process of ascent of amber particles due to air bubbles is provided.

## Conclusions

With the hydromechanical method of extraction and extraction of amber from sandy and sandy-argillaceous rock mass, the use of a suspension medium in the form of a liquid of increased density, vibration and air in the form of bubbles makes it possible to increase the rate of ascent of pieces of amber of various sizes, which contributes to its maximum extraction.

Application of hydromechanical method of extraction of amber with the help of the studied vibrohydraulic intensifier allows extracting 90 – 95 % of the amber from a deposit, in this case the medium density and vibration influence the liquefaction of amber-containing sand, namely, the optimal density ( $\rho_c$ ) is 1600–1850 kg/m<sup>3</sup>, the frequency of vibrations of 30–35 Hz, amplitude  $A = 1.07 - 2.5$  mm;

Maximal speed of amber recovery to the surface is achieved depending on the changes in the consumption of water and air by a

vibrohydraulic intensifier for specific amber-containing fields and the optimal values were obtained during performed laboratory and field experimental research.

When using table salt to increase the density of the suspension medium, its concentration is in the range of 105 - 150 g / l, and when using clay for this purpose, its concentration increases by another 10%.

The use of vibration allows you to reduce the concentration of the suspension by 10%, bubbling - in 1,5-3 times, and also contributes to the segregation of pieces of amber when they are separated from sand and clay. During the experiments it was proved the possibility of replacing the expensive and polluting salt with the "blue clay" with the increase in the concentration of pure clay in comparison with the salt by 5 - 10%.

In general, as a result of the studies carried out, it has been shown that the complex effect on the rock mass can be effectively used in the form of a liquid medium of increased density, vibration and air flows in the form of bubbles. At the same time, a closed cycle is performed on the liquid phase of the impact, which allows us to justify the new technology of amber extraction.

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## RESOURCE-SAVING TECHNOLOGY OF URANIUM ORE MINING

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**Abstract.** Uraniferous granites are characterized by a complex structure, observed both at the micro level and at the macro level. This anisotropy of physical and mechanical properties of rock-forming minerals with a spatial (preferred) orientation, and containing a large number of structural defects in the form of intragranular microcracks, as well as the presence of macrocracks a well-developed system with varying degrees of gaping. Vertical microcracks two mutually perpendicular systems form in the rock, a so-called fault-crack grid. One system of cracks is characterized by openness, another system is formed by densely-closed cracks. Granites are strained rocks, due to intracavitary stresses, which reach values of up to 100 MPa. However, in uraniferous granites due to metasomatism, intergranular stresses are virtually absent due to relaxation. Such structural features of the uraniferous granites determine the nature of their destruction by the explosion and eventually, the selection methods of their effective fracture energy of the explosion. Structural features uraniferous granites and their effect on the nature of their destruction by explosion caused new approach to resource-saving technology of uranium ores underground method.

**The purpose of the work** is to present new ways of complex structure rocks destruction by explosion during underground mining of mineral deposits.

**Methods of research.** When creating a resource-saving technology, modern methods of studying rock structure and its influence on the nature of explosion destruction were widely used. In particular, for the study of microstructure, state-of-art equipment and methods of microstructural analysis were used to establish the spatial orientation of the structure the elements of granites. When processing the measurement data, rigorous methods of mathematical statistics were used. Simulation of the destructive



effect of the explosion was carried out on samples made of uraniferous granite. The influence of the stress-strain state of the massif on its destruction by explosion was estimated using simulation methods. The data obtained were then confirmed by modeling the dynamic loads on the destruction of optically active transparent materials.

The results of research have been used to create new ways of rock destruction by explosion of new methods and explosive rock penetrations various mining. New methods of rocks destruction by explosion are based on the account of structural features of the medium being destroyed. These methods were patented and were laid in the foundation for the creation of resource-saving technology for uranium ore mining.

Mine workings of various technological purposes in strong strenuous rocks on uranium mines are carrying out using the energy of the explosion. The process of rock destruction by explosion proceeds on a small area of the production cross section at one free surface. Such conditions exist, as a rule, with the operation of the first charges forming the hollow cavity, which then plays the role of a free additional surface. Therefore, the improvement of ways to break strong hard-to-break rocks reduces to the development of new cuts (types of cut) and the order of its formation, taking into account the Rock Massif Strained-Deformed State (SDS). This way provides an increase in the borehole utilization ratio (BUR) and a decrease in the cost of mining.

### **Mining geological structure of Ukraine uranium deposits ores (on the example of the Vatutinsky deposit)**

The field of the “Smolinskaya” mine (State Enterprise “Eastern Mining and Concentrating Mill”, further in the text – SE “EM & CM”) is located within the ore field of the Vatutinsky deposit of the Central Uranium Area, in 80 km west of Kropivnicki (Kirovograd region). The boundaries of the deposit fit into the central part of the Ukrainian Shield, on the western wing of the Kirovograd antiklinorium and are confining to the Zvenigorod-Antonovo fault zone [1]. The structure of the deposit is determining by the intersection of a thinly layered rock monoclinaly falling to the west towards the Main Western fault of the East Kurnikovskaya zone (Fig.1). The area of the deposit is composed of Lower Proterozoic rocks – gneisses, migmatites, granites, pegmatoidal granites, gneiss-like microgranites, and alkaline me-tasomatosis rocks that develop over all these rocks.



**Fig. 1.** Geological plan of the Vatutinsky uranium ore deposit [1]:  
 1 – loose sediments; 2 – gneiss biotite; 3 – migmatites shadow; 4 – granites medium-grained; 5 – microgranites; 6 – albitites; 7 – syenites; 8 – ore bodies ;  
 9 – tectonic disturbances; 10 – rock contacts; 11 – metasomatic contacts

Gneisses are represented by biotite, garnet, graphite, amphibole-biotite and some other varieties with a foliation, banding or massive texture. Shadow migrants pre-dominant from migmatites, banded migmatites are less common. The granites are predominantly alaskite-like, uniformly grained, slightly porphyreous. Pegmatoid granites are characterized by coarse-grained pegmatoid structure and consistency of composition. Gneissous microgranites are found in the form of dykes and are a fine-grained rock of gray and dark gray color. Alkaline metasomatites in the deposit (albitites and albite-microcline syenite-like rocks) are accompanied by a halo of secondary changes in the enclosing rocks. The main rock-forming minerals of syenite-like rocks: microcline (30-80%), albite (20-60%) and dark-colored (0-30%). These rocks are characterized by high porosity and almost constant presence of pyrite in them. Albitites are composed mainly of albite (60-95%), dark-colored minerals (0-40%) and quartz (0-35%). According to the nature of dark-colored minerals associations, albitites are divided into alkaline-amphibole-aegirite and epidote-chloritic. The bulk of the ore-bearing albitites of

the deposit is located in the lying side of the Main Western fault, which is characterized by a northwest strike (310-320°) and a southwest inclination at an angle of 70-80°. The total length of the strip exceeds 3 km, its width reaches 0.6 km and the depth is more than 1.2 km. The largest bodies of albitites are connected with the Main West and East dip faults. The uranium mineralization in the deposit is spatially associated with albitites. In syenite-like rocks, in a number of cases, only a slight increase in the uranium content has been established. The ore bodies of complex shape and phacoidal systems are formed by mating and pass into one another or successive echelon lenses. A characteristic feature of them – a few dissenting orientation of the ore bodies in relation to their ore zones. With the general NW-trending mineralized deposits of corresponding strike major ore-controlling faults, some small ore bodies usually have a NS-strike, according to a stretch of the country rocks. Uranium minerals are represented by oxides (uraninite, nasturan, pitchblende, uranium hydroxide), silicates (nenadkevit-like mineral, coffinite, uranophane,  $\beta$ -uranotil, boltvudite), uranium titanates (brannerite, davidite), etc.

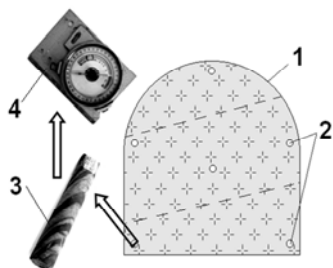
The ores structures are inherited from the original rocks. The most common in the deposit medium-grained ores, fine-grained and coarse-grained ores are of subordinate importance. Textures of ores also inherit the textures of the original rocks. There are massive ores formed on granites, banded and spotted, formed by migmatites, schistose – on gneisses. By the nature of the distribution of uranium minerals in the ores, disseminated, small-nest and fine-grained ores are distinguished. In general, the ore in the deposit is finely impregnated.

### **Forecast estimate the massif properties**

To adjust the rational parameters of drilling and blasting operations during the mine workings of various technological purposes, it is necessary to take into account the structure of the destroyed massif, the physical and mechanical properties of rocks and the influence of the rock massif strained-deformed state on the destruction rocks nature.

With this goal in mind, we patented a method for predicting the

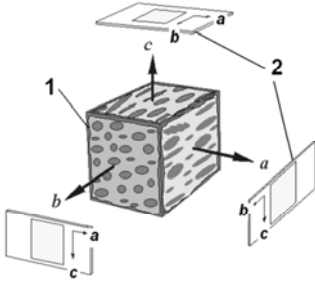
stress-strain state of a rock massif in the mine face (patent of Ukraine No.100054 [2]). The method includes boreholes drilling, rock samples (cores) selection, determination of physic mechanical characteristics and rock deformations. In accordance with the proposed method, first on the ore block, in the mine face **1** in the characteristic zones of its section (along the perimeter and in its center), at the sites of the bundle of fan wells, geological exploration holes **2** are drilled to a depth equal to half its length ( $0.5\ell_h$ ). Then it becoming removed from the drill hole and connects with Core Barrel (for example, EZY-MARK™) for coring **3**. The selected core is oriented in the drifting operation direction with a mining compass **4** (Fig.2).



**Fig. 2.** Cross section of the mine in the ore block with drilled geological exploration holes:  
 1 – mining; 2 – exploration holes;  
 3 – core rock; 4 – mining compass

According to the nature of the etching pattern, the direction of Sander's structural axes is determined [3] **a**, **b** and **c**, where **a** is the axis that lies in the flattening plane of mineral grains aggregates; the axis **b** – coincides with the linearity of elongated mineral grains aggregates, oriented normally to the flattening plane. All three axes are orthogonal to one another.

From the prepared sections of the rock, oriented petrographical micro sections are made: one parallel to the **ac**-plane and perpendicular to the **b**-axis, the other parallel to the **bc**-plane and perpendicular to the **a**-axis and the third parallel to the **ab** plane and perpendicular to the **c**-axis (Fig.3).



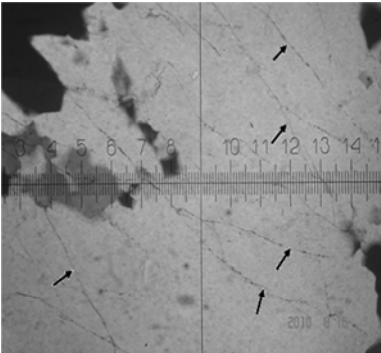
**Fig. 3.** Oriented ore lump (1) and microsections (2), made of core: *a, b, c* – Sander's structural axes

Using the ISA-integrator mounted on the object table of a polarization microscope, the morphological features of the intensity of microcracks  $I_{j1}$ ,  $I_{j2}$ ,  $I_{j3}$  are determined on petrographical sections (Fig. 4) in mineral grains, and with the help of the Fedorov Universal table FS-5 of the isolated microcracks spatial position.

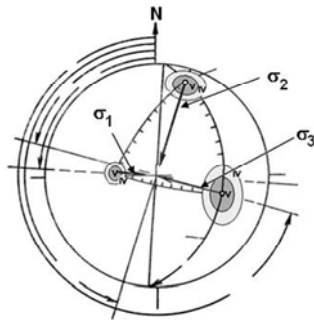
For example,  $I_{j1}$  is 30 joint/cm;  $I_{j2}$  – 19 joint/cm;  $I_{j3}$  – 5 joint/cm. Then, according to the relation

$I_{j1}:I_{j2}:I_{j3} = \sigma_1:\sigma_2:\sigma_3 = 30:19:5$ , determine the main compressive stresses ratio  $\sigma_1: \sigma_2: \sigma_3$  as 1:0,63:0,17, which assesses the Rock Massif Strained-Deformed State in the development opening face.

Using the equidistant Kavraisky grid construct a structural diagram (Fig. 5). The diagram shows the poles of the main microcracks systems (the microcracks resulting system vectors) and the main compressive stresses vectors ( $\sigma_1$ ,  $\sigma_2$  and  $\sigma_3$ ) with coordinates  $\sigma_1$  ( $\rho_1 = 72^\circ$ ,  $\varphi_1 = 90^\circ$ );  $\sigma_2$  ( $\rho_2 = 9^\circ$ ,  $\varphi_2 = 195^\circ$ );  $\sigma_3$  ( $\rho_3 = 15^\circ$ ,  $\varphi_3 = 285^\circ$ ).



**Fig. 4.** Uraniferous granite with microcracks in quartz grains (dark arrows, petrographical microsections, photomicrography)



**Fig. 5.** Structural diagram of main systems poles of microcracks and the main compressive stresses vectors ( $\sigma_1$ ,  $\sigma_2$  and  $\sigma_3$ )

The use of the proposed method in the mining industry will make it possible to scientifically substantiate the location of the preparatory mining work during the blasting of stressed rocks, to adjust the parameters of drilling and blasting operations (the location of cutters, felling and contouring blast-hole charges, their length and quantity, unit costs for explosives) in the breaking.

As a result, the blasting efficiency and the borehole utilization ratio, and the productivity of the handling equipment increase.

### **New ways of forming a hollow cavity**

Because of the studies carried out to establish the patterns of stresses distribution in the mining cut face the recommendations have been developed to improve the efficiency of breaking hard rocks and patented the method of forming a hollow cavity in strong strenuous rocks [4].

The mechanism of rocks destruction in the formation of a hollow cavity in strong overburden rocks is based on taking into account the redistribution of static stress fields during the process of preparing it for blasting with explosives charges to the compensation cavity in the boreholes of a three-tier walking cut.

Formation of the enlarged diameter compensation cavity in the center of the cutting is carried out with a plasma torch by expanding the forward boreholes.

When a quartz-containing rock is heated by a plasma torch to 450-

573 °C, a polymorphic transition of low-temperature  $\alpha$ -quartz to its high-temperature modification ( $\beta$ -quartz) occurs, as is known [5]. This process, known as  $\alpha \leftrightarrow \beta$  transition accompanied by an abrupt increase in the volume of quartz at 0.86 %. As a result, the rock in the heating zone by the plasma torch is easily destroyed (the rock is peeling off) and the separated rock fragments are carried out by the plasma torch gas jet from the central compensation borehole.

In addition, the linear expansion coefficient of the rock increases by 20 % when it is heated perpendicularly to the layering and by 22 % – in parallel to the layering. When drilling in the central part of the set of boreholes is 30 % greater than the cut length (the second variant of the compensation cavity formation), the reference pressure

zone moves beyond the limits of the compensation cavity and the cutter charges operate under more favorable conditions, namely, in the lower stresses zone. According to the sequence of operations set forth in the new forming a hollow cavity method we developed a blast pattern for drilling a horizontal mine with a cross section of 9 m<sup>2</sup>. Schemes of the holes location in the cross section of the cuts construction are shown in Fig.6 and Fig.7.

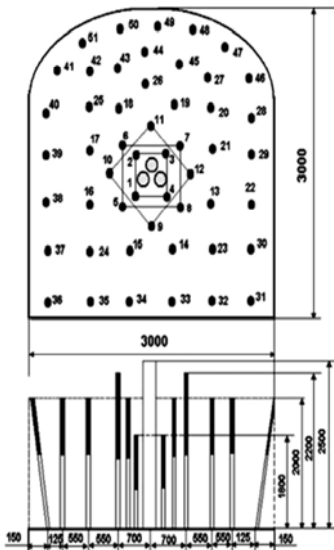


Fig. 6. Proposed blast pattern for horizontal mining

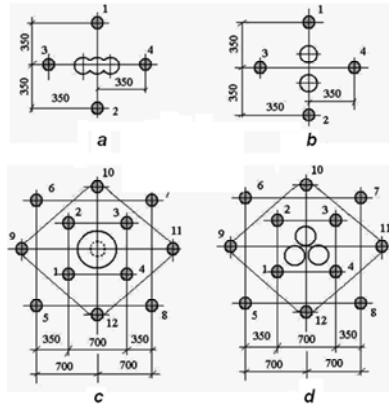


Fig. 7. Cuts design schemes with an additional compensation cavity:  
a - slit cut with uncharged compensation holes; b – straight prismatic cut with two vertical compensation holes; c – straight prismatic three-tier walking cut with holes extended plasma torch; d – straight prismatic three-storey walking cut with a bundle of three compensation holes

The peculiarity of the proposed three-tiered the cut is that the holes across the cross section generation drilled starting from the drilling of the cylinder circumference or at the vertices of a polyhedral prism. Loading the blastholes carried patronizes explosives, e.g., emulsion type explosive “Anemiks”. The holes of the *I*-th and *II*-nd storey are being charged, and in the holes of the *III*-rd storey, charges of cumulative action are formed, placing in the bottom part of the holes spherical cavities with 0.5 borehole diameter. The charged holes establish intermediate initiators

detonator cap of instant, short-delay and delayed action. Charges are stemming, and commute to groups. Charges detonate with deceleration compensation cavity using non-electric initiation system type NONEL, starting from the *III*-rd storey charges.

The efficiency of rock breaking into the compensation cavity during the realization of the method of formation of the hollow cavity is achieved by initiating charges with delay between stories, starting with charges of cumulative action, located in the zone of action of a stable volumetric thermal field and redistribution of static tensions in the center of cut.

The resource-saving technology of uranium ore mining developed by us, after its adaptation to specific geological conditions, can also be successfully applied for the underground mining of iron ore and polymetallic minerals.

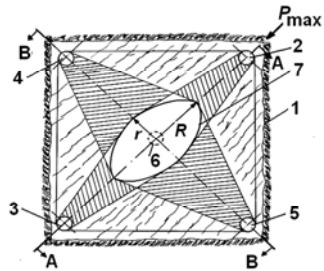
### **Effective ways of roadway development**

In the technological process of uranium ore mining intensification, the timely putting into operation mining blocks plays a special role. This process is caused by a large amount of various technological purposes roadway development, such as: opening slots, horizons of scarpering and rock mass movement, interstorey and sub-floor mining, cul-de-sacs, rise headings and blind pits etc. For effective raise driving, we patented the methods of their formation (patents of Ukraine No.79129, 88825 [6, 7]). According to the developed ways of roadway, we will consider the variants of these cuts formation.

*The peculiarity of the raise driving method (1st variant, patent No.79129)* is that the increase in the efficiency of the blasting of the boreholes explosives charge and the strong viscous rocks destruction quality is ensured by blasting explosives with a delay to the compensation cavity. First, holes are initiated that drilled in a plane, which coincides, with the natural cracks in the massif and the direction of the forces prevailing rock pressure. It should be noted that two peripheral holes are drilling in a plane coinciding with the direction of natural cracks in the massif and the prevailing forces of rock pressure, and two other peripheral holes – along a line perpendicular to the plane of the prevailing forces possible direction

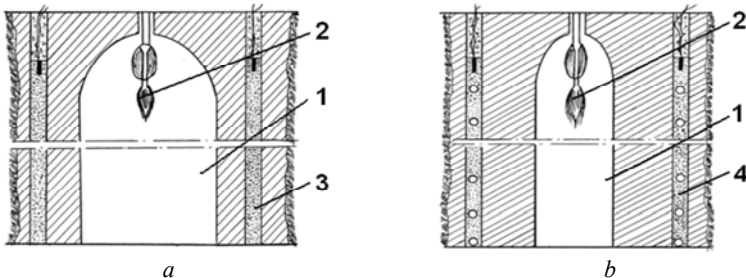


of the rock pressure in the compressive stress zone. Then, the central well is expanded with a plasma torch, giving it the shape of an elliptical cylinder. In this case, the larger radius  $R$  of the compensation cavity elliptical section is oriented in the direction of the prevailing forces plane of the rock pressure  $P_{max}$  in the zone of the tensile stresses action. In peripheral boreholes in the compressive stresses action area, cumulative explosive charges of a variable cross section are formed with uniform placement of spherical cavities on the charge column, and entire charges form in the wells in the zone of tensile stresses action. Blasting is carried out in one-step with a deceleration to the entire height of the workings being carried out, beginning with the boreholes in the zone of tensile stresses action, followed by the blasting of the boreholes in the compressive stress zone. The scheme of the raise driving is shown in Fig.8 and Fig.9, *a, b*.



**Fig. 8.** Scheme of drive working: 1 – massif; 2,3,4, 5, 6 – boreholes; 7 – compensating cavity

In our proposed method of mining technical result is achieved by improving the form of work explosion in the peripheral holes. Increasing the efficiency and quality of strong viscous rocks destruction is attained due to the redistribution of the explosion energy in an unequal component stress field.

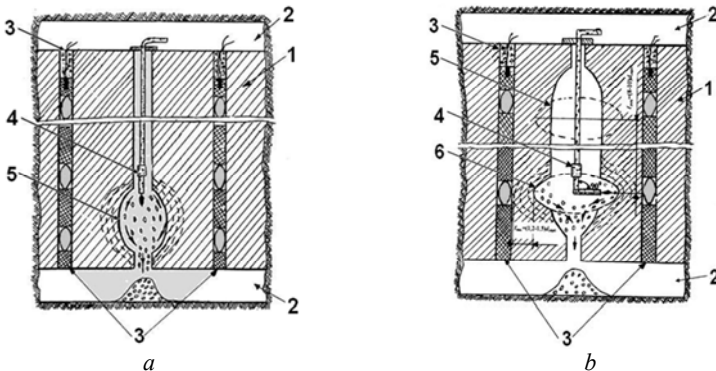


**Fig. 9.** Scheme the of mine workings excavation in sections AA (*a*) and BB (*b*):  
*a* – section along AA; *b* – section along BB; 1 – compensation cavity;  
 2 – plasma torch; 3 – constant-cross sectional charge;  
 4 – charge of variable cross section

***The raise driving method on the 2nd variant (Pat. No.88825).***

Forming of raise driving begins with the drilling of the central hole, and around it four peripheral holes are drilled at the tops of the design contour of the mine. The central borehole is expanded by a plasma torch to form a compensation cavity with a diameter  $d_{\text{comp.cav}} = (5-6)d_{\text{hole}}$  at the entire height of the raise driving. Then, the entire height of the compensating cavity starting from its end, the plasma torch rotation around its axis to form annular recesses diameter  $d_{\text{an.rec}} = (7-8)d_{\text{hole}}$  at a distance from each other equal  $l_{\text{rational}} = (8-10)d_{\text{hole}}$ .

In the peripheral boreholes, dispersed charges are formed by placing on a column ampoules with air of length  $l_{\text{a.g.}} = (5-6)d_{\text{hole}}$  and subsequent injection of explosive by portions of a pneumatic charger. After that, the charges commute to groups and are blown up in one-step with a delay. The efficiency and technical result is achieved by improving the operation of charges explosion distributed air gaps in the peripheral blastholes that were drilled in the zone of the thermal field stable action. Moreover, the quality of strong rocks destruction between the compensation cavity and blastholes in a different gradient stress field is increased, which is created in the massif by initiating charges with delay-action-blasting between groups of blastholes. Schemes of the raise driving are shown in Fig.10, *a*, *b*, and the design of the charge in Fig.11.



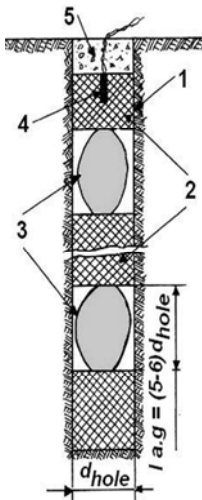
**Fig. 10.** Technological scheme for the preparation and operation of the raise driving: *a*, *b* – uprising mining technological schemes; 1 – massif; 2 – drilling room; 3 – peripheral blastholes; 4 – plasma torch; 5 – compensation cavity; 6 – annular recesses

**The stope in the deep-mined output of uranium ores.** For the uranium ore extraction in Ukraine, a chamber system is used with chambers preparation the by sub drifts and orts (Fig.10) and laying of the worked out space with a hardening mixture [8]. Ore breaking the cells is carried out by methods common power control explosion providing the most efficient blasting, namely: vertical layers on the compensatory space, in the clamped medium, beams of adjacent holes in the vertical compensation gap with a uniform distribution of explosives in holes slugger array of reduced diameter and the rows of short-delay between initiation of detonators.

The effectiveness of the ore breakage in this case depends on the factors influencing the degree of its crushing and reduction. The dimensions of the outlets (draw holes) and vehicles limit the output of ore from the chamber. Since uranium ore deposits in Ukraine are located, as a rule, close to industrial facilities and civil buildings, the underground mining of such deposits should be carried out taking

into account the seismic effect of the explosion on protected objects. The negative impact of mass explosions on residential and social development in the mining of ore deposits (The Central uranium ore deposit, Kropivnicki) complicates the process of operating units.

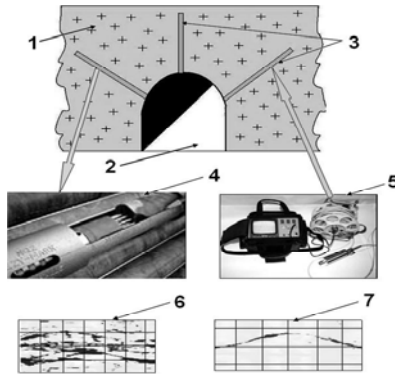
Anisotropy of rocks (texture, fracturing, tectonic disturbances and rock contact) significantly affects the propagation of seismic waves in massifs. At the same time the amplitude, frequency spectrum and attenuation of seismic oscillations generated by underground explosions significantly change. The more intense the vibrations occur in places where the massif to the surface or in areas of the small thickness of unconsolidated sediments. The amplitude of the seismic waves propagating in the cross stretch and stretch, often differ in 1.4-2.0 times, and in some cases – by 4 times or more [9]. These features of the



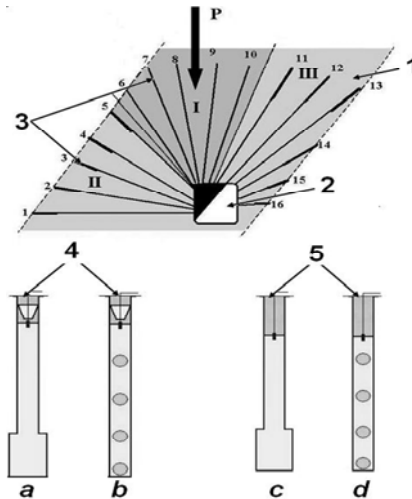
**Fig.11.** Design of well borehole charge:  
 1 – blasthole,  
 2 – explosive charge;  
 3 – ampoule with air;  
 4 – primed blasting cartridge; 5 – stemming

rock massif should be taken into account when designing parameters for drilling and blasting operations near protected objects. An effective method of reducing the seismic action of a underground explosion on surface objects is to shield seismic waves by forming a shielding zone above the located below stope and filling the used chambers with a hardening mixture. The weakening of the intensity seismic waves from an explosion thus occurs due to their intensive attenuation of the shielding zone.

To reduce dynamic (seismic), explosion impact mass for blasting operations in the block with the subsequent formation of the shielding contact zone patented method for breaking rock (Pat. No.107147 [10]). The method consists in the fact that in the ore block, prepared for the breaking, the drill drift locations laying kit fan wells staggered the entire height of floors or sub stage drilled exploration holes and conduct selection-oriented cores. In the laboratory, slices are formed, oriented petrographical sections are made, then the main systems of microcracks are identified using optical microscopy methods and their spatial position is established – the angle of incidence and its azimuth. Intensity, direction and depth of the development of joint systems in the massif are studying using a digital video camera, placed in the exploration well drilling in the mine roadway. Interpretation of the video camera recorded information allows us to determine the direction of the prevailing action of rock pressure forces. Then in the zone of these forces action, cut holes are drilling. In the hanging and recumbent sides, boreholes are drilled, and in the ends of the boreholes the plasma torch forms cylindrical cavities with a diameter  $d_{c,c}=(2-3) D_{hole}$  and a height  $h_{c,c}=(5-10)D_{hole}$ . In the cut-in holes, explosive variable cross section charges of the cumulative action are formed with uniform spherical cavities uniformly alternating in the charge column with a diameter  $d_{sp,c} = 0,8D_{hole}$ . In the hanging and lying sides form continuous charges of explosives. Explosive charge and detonate group commute with delay, since cut-off (within the area of the pressure forces prevailing), then are forming explosive charges in a lying and hanging laterally on checkerboard pattern. Schemes for the implementation of the method for breaking rock are shown in Fig.12 and 13.



**Fig. 12.** Scheme of preparation for conducting blasting operations in a steeply ore-run: 1 – massif; 2 – preparatory mining ; 3 – prospecting boreholes; 4 – core barrel EZY-MARK™; 5 – digital video camera; 6 – character of the cracks distribution in the massif under the influence of rock pressure prevailing forces; 7 – the cracks distribution character in the massif in the zone of reduced influence of rock pressure



**Fig. 13.** Technology of blasting operations in the ore-run block: 1 – massif; 2 – preparatory mining; 3 – blast boreholes in the block; 4 – design explosive charges in boreholes ascending; *a* – charge of variable section with boiler extension at the borehole end and the gate mouth stemming, *b* – charge variable section that making cumulative effect of spherical inserts and sealing the mouth of the gate); 5 – design explosive charge downholes (*c* – charge of variable cross-section with a boiler extension at the end of the well and sealing of the mouth with a stemming, *d* – charge of a variable cross section of a cumulative action with spherical inserts and sealing of the mouth with an incision); I – action prevailing forces of rock pressure zone ; II – zone lying rocks; III – hanging rock zone

Applying the developed method and structures charges enhances the efficiency of explosive charges in fan-shaped holes with ore breaking in the block. In addition, the rock mass crushing quality increases, the specific costs for explosive materials and explosives are reduced, and the seismic effect of a mass explosion is reduced. As a result, the explosion efficiency, the crushing productivity and sorting complex and vehicles are increasing.

## **1. Modern technology of mine drifting**

Geological structure of the deposit, as it is known, determines the choice of systems mining and technology blasting of the rock mass. For mining uranium deposits, we proposed camera systems field development sublevel drifts orts and alternate excavation of rock blocks and filling of mined-out space hardening mixture.

To effectively prepare the deposit for mining its reserves by underground method, a large number of preparatory workings of various values are usually formed, in particular, drilling drifts, orts and raise driving.

The implementation of the new method of formation cuttings cavity with rational parameters [3] was carrying out on Smolinskaya mine. In particular, a new technological blast pattern was proposed. This took place on a blocks mining №№ 105-3 (horizon + 100 m), 554-11 (horizon 490 m), 554-5 (horizon + 490m), 374-36 (horizon + 370m) of the (SE “EM & CM”) during the pilot explosions to break strong rock in the preparatory faces of the drilling workings. The experiments were carried out using several versions of the design of the jaws: with a basic blast pattern, a straight cylindrical (Fig.14, tabl.1), and with the new blast pattern (see Fig. 7) – a straight prismatic three-storey walking cut with a compensation hole of increased diameter at its center.

We also considered slit cuts with uncharged compensation boreholes in its center and a straight prismatic cut with two compensated holes drilled vertically in the center of this cut (see Figure 7). Provision was also made for the contour of the cavity to form cuttings cumulative effect charges in blastholes extended length of the three-storey.

Experimental cuts were formed by drilling in the central part of

the face of compensation blastholes of increased diameter. The horizontal and vertical boreholes, as well as the central borehole, were expanded with plasma torch to a diameter of 250-350 mm.

Volumetric thermal field in the rock massif created using a plasma torch. Around the compensation holes, then cut boreholes were drilled.

The new blast pattern differs from the base blast pattern in that a hollow cavity is formed using a straight three-storey prismatic walking cut with longer-length blastholes. In the third tier boreholes formed explosive charges to the cumulative effect of placing at the end of the spherical cavity.

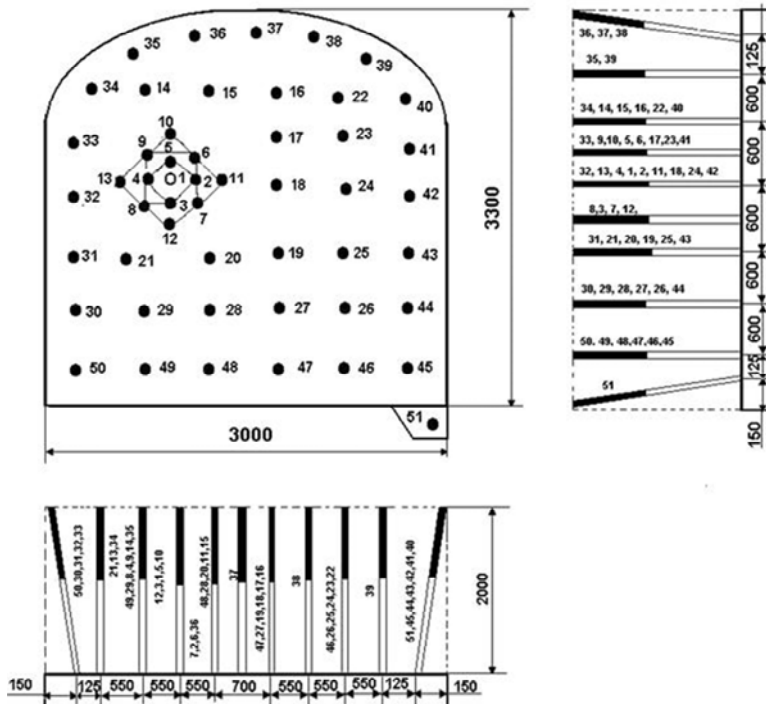


Fig. 14. Scheme drilling location in the working face at the base blast pattern

**Table 1****The sequence of detonation**

Boreholes number	Delay number	Deceleration time, ms
	1	not charging
2	0	0,0
3	1	0,025
4	2	0,050
5	3	0,075
6-7	4	0,100
8	5	0,150
9,10	6	0,250
11-15	7	0,500
16-29	8	0,750
22-26	9	1,000
40-44	10	1,5
35-39	11	2,0
45-51	12	4,0

Sealing the mouth of holes produced stemming from the hardening mixture. The distance between adjacent drill holes is increased to 0.55 m instead of 0.50 m as compared with the base blast pattern, while the remainder of the breaking rock by making the sectional area is carried out by entire charges, sealing the mouth of the drilling sandy-clay tamping.

The average length of the boreholes to be blasting on the base pattern was 1.8 m with simultaneously blasting 56 blastholes and the total mass of the explosive ammonite No.6GV – 61.2 kg, and according to the proposed blast pattern – 2.2 m, the number of blastholes – 51, the total mass of explosives is 56.2 kg.

Charges are initiated by electric detonators of instantaneous and short-delayed action with eight stages of delaying. Cut borehole charges are blasting by instantaneous and short-delayed electric detonators with a deceleration between 15-20 ms steps using a non-electric initiation system such as NONEL. Results of the experimental blast evaluated quality crushing rock (average diameter of the piece) and the borehole utilization ratio, which was 0.97.

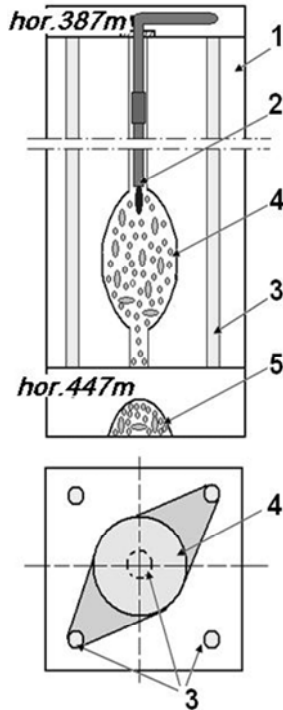


## 2. Modern technology of rising mining has been testing in the Kryvbas mines

The tests were carried out in a massif of strong magnetite quartzite ores with a coefficient of strength  $f=13-16$ . In particular, at the mine Guards from the horizon 447 m on the horizon 387 m, an uprising mining operation was built with a cross section of  $2.5 \times 2.5$  m at the entire height of the floor, the technological scheme of which is shown in Fig.15. For this purpose, a set of blast holes with a diameter of 105 mm was drilled at the tops of the square in the design contour of the expected development at the tops of the square to the full height, with access to the drilling horizon of the underlying or overlying floor (sub-floor). Compensation cavity was formed by expanding the central well with a plasma torch with a diameter of 105 mm to 500 mm.

The compensation cavity was given an elliptical cross-section with the direction of the ellipse major axis toward the holes forming the hollow cavity and the prevailing forces of the rock pressure.

The expansion of the central hole was carried from the bottom upwards, and the fracture products were moved by gravity to the underlying horizon. The wells were charged with granulated explosives such as gramonite 79/21 or emulsion explosives (EE) of the Anemiks type using a pneumatic charging machine. After charging the holes, intermediate initiators were installed, and the hole head was stemming with a special locking device. Then the explosion network was switched and blasted with a



**Fig. 15.** Technological scheme for creating an end-to-end uprising between the drilling horizons of the ore-run:  
 1 – uprising mining; 2 – plasma torch; 3 – blastholes; 4 – compensation cavity; 5 – broken rock

slowdown in one-step, starting from the boreholes, to the full height of the rising mining. The charges were detonated using a non-electric initiation system such as NONEL.

Thus, the effectiveness of the uprising mining formation was achieved by improving the quality of the massif destruction within the boundaries of the mining cross section to its full height. This decreases the explosives specific consumption and the explosives volume.

In accordance with the method **developed by us for breaking rocks** [10], the technology of breaking the ore in the chambers of the first stage begins with the formation of a cutting slot in the center of the block or the most powerful part of it. The cutting slot is formed by blasting parallel downholes in the sub-floors towards from the bottom up. Boreholes are drilling on 3-4 pieces in an each row. The line of least resistance is 1.2-2.0 m. Further from the drilling drift (ort), according to the developed method, geological prospecting boreholes are used to identify the fracturing features. The cores are extracted from the prospecting boreholes and are orienting their by means of a mining compass. In laboratory conditions, thin sections are made of cores, along which the density of cracks and their direction are determined. In geological prospecting boreholes, in addition, a rock surface is studying using a TV-camera to determine fracture parameters. The video information is output to the TV-camera monitor and a scan of the recorded image of the well surface is carried out, onto which a grid with a cell size equal to 1 cm is applied and the fracture coefficient of the rock mass is calculated. The coefficient of fracturing is  $K = N_{fr}/S$ , where,  $N_{fr}$  – number of cracks,  $S$  – the surface cracks area in the places of their concentration. According to the obtained data on the nature of the cracks distribution in the ore block, this is prepared for excavation. In addition, the direction and intensity of the stress field distribution in the massif are evaluated, and then the location and direction of the blast holes drilling in the face of the mine are corrected.

After the above operations, doing drilling of 3-4 rows of ascending or descending boreholes with a diameter of 65 or 85 mm on 13-16 pieces in a fan. The line of least resistance varies from 1.02 to 1.8 m, and the distance between the ends of the boreholes in the row is from 1.8 to 2.5 m. The charges burden-to-spacing-ratio

depends on the fracture and is in the range of 1.0-1.5. In drilled wells, charges of various designs are forming. In the zone with the predominant direction of the rock pressure forces, in order to improve the crushing quality and redistribution of the stress field (zone I), cut-in wells are placing and charges of variable cross-section with cumulative action are forming in them. These charges are spherical cavities uniformly spaced along the column with a diameter  $d_{sp} = 0.8D_{hole}$ . In the recumbent (zone II) and hanging (zone III), the sides in the fan wells form entire charges with boiler expansion at the end, which allows to qualitatively destroy these zones and reduce the out-of-gauge output. In this case, the boiler extensions in the boreholes of diameter  $d_b = (2-3)D_{hole}$  and height  $h_b = (5-10)D_{hole}$  are formed with the help of a plasma torch. Sealing the borehole mouths in the ascending fan do with the shutter, and downstream – by stemming. Boreholes are charging with explosives (for example, emulsion). Then, intermediate initiators with instant and short-delayed electric detonators or devices for non-electric initiation of the NONEL type are installing in the blasting holes.

Explosive charges commute on groups. Commutation begins with charges in the boreholes of the I zone to form a hollow cavity, and then connect the charges in the boreholes of the II and III zone in the chess-wave initiation scheme with decelerations between groups of charges in the fan-holes. In the first place, cut-in well charges are initiating in the zone of prevailing forces of rock pressure. Then, in a checkerboard pattern, charges are blowing up in the wells of the fan located in the lying side of the ore-run. Charges in the hanging side of a steeply falling ore-run are blowing up in the last place.

The method developed by us for breaking rocks has allowed ore output from 1 m of the borehole – 6-12 tons (ore density is  $2.65 \text{ t/m}^3$ ). The output of oversized fractions is 5-7% and the specific consumption of explosives is  $3.5-4.5 \text{ kg/m}^3$ .

### **The technology of filling the worked out space of the mining block (chamber) with hardening mixtures**

Mining of ore-runs at uranium ore deposits at Smolinskaya mine is carrying out by chamber systems. The worked out space is then filling with hardening mixtures [11-13]. This achieves reliable

control rock pressure, reducing dilution ore mining industry waste disposal, saving of fertile land, as well as the integrity of residential and industrial buildings. Preparation of the hardening mixture is carried out on a stationary complex specially built for this purpose. The productivity of this complex is 400-450 thousand m<sup>3</sup> of hardening mixture per year. Based on the results of the decision of optimization models for selecting and justifying the rational composition of the hardening mixture to fill the worked out space, we propose the following component ratio (in percentages by weight), namely: 32.6 %, dolomite dust 22.5 % water, 13.8 %, alumokalic alum, 0.6 % – lignosulfonates, 28.8 % – natural anhydride (or blast furnace slag), tails –1.7 %. The minimum cost of the selected mixture is 19.54 UAH/t, which are 0.45 UAH/t less than the cost of components of the base hardening filling mixture used in the Smolinskaya mine. At the same time, the consumption of materials for the preparation of 1 m<sup>3</sup> of the proposed filling mixture is: dolomite dust – 0.8 tons, alumokalic alum 0.3 tons, lignosulfonates 0.012 tons, natural anhydride (blast furnace slag) 0.8 tons, processing waste of the mining industry (tails of uranium ores and coal enrichment - mud) – 0.032 tons, water – 350 liters.

The use of technology for mining uranium ores with the filling of the worked out space with hardening mixtures ensures high safety of technological processes of extraction and processing of minerals. Moreover, ecological and man-caused safety is achieving in the territories with developed infrastructure, especially in the areas near the underground mines.

## **Conclusions**

The recommendations on the intensification of mine workings penetration of various technological purposes and maintenance of stoped excavation in ore blocks using the energy of explosion by borehole charges of explosive cumulative effect are developed. This is the method for forming a hollow cavity (Patent No.88827, Ukraine), a method of penetrating an uprising mine roadway (No.79129 and No.88825, Ukraine) and a method for breaking up the hard rock of the ore-run (No.107147, Ukraine).

Approbation of rational parameters of new ways of mine

workings penetration, such as the formation of a hollow cavity during the drilling of horizontal workings on the drilling horizon, the rising mining between the floors of the mining blocks, the breakage of ore in the block, was carried out in new blast patterns at the Smolinskaya mine. At the same time, experimental explosions were made in the preparatory faces of drilling workings and in operating blocks for breaking off uranium ore. Because of explosions were carried out in experimental preparations drilling cut and excavation in the operational blocks of breaking of uranium ore.

By increasing the borehole utilization ratio to 15 % (0.95-0.97), the specific costs for drilling operations, explosive materials decreased by 10 %. It increased the uniformity of crushing rocks, decreased ore dilution in the management of sewage treatment works, improved technical and economic performance of the mining enterprises.

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## **FORMATION OF EFFICIENT ENERGY FLOWS AT THE CONTOUR BLASTING OF BOREHOLE CHARGES**

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**Abstract.** When blasting by presplitting method is performed in hard rock masses, problems related to the choice of rational parameters for blasting by presplitting method and rationale for applying the most effective method of its implementation arise. This determines the relevance of studies aimed at developing methods for controlling the process of rock destruction, when performing blasting by presplitting method. Proceeding from this, the subject of these studies are technological parameters of blasting by presplitting method that ensure formation of effective energy flows of the borehole charge blasting to obtain design bank slopes. To solve research tasks, a comprehensive research methodology has been applied that consists of the analysis and generalization of previous scientific studies on the implementation of blasting by presplitting method, theoretical studies on achieving effective conditions for interaction of energy flows of pre-split borehole charges, as well as experimental tests of developed designs of pre-split charges in industrial conditions. The purpose of these studies is to develop a method and determine parameters for the formation of effective energy flows in blasting by presplitting method of borehole charges. Based on results of the studies, the dependence of the average linear mass of the explosive charge on the distance between boreholes that ensures the development of a directed network of fractures between pre-split charges upon their blasting has been obtained. It is established that each type of the design of the pre-split borehole charge that is characterized by the averaged linear mass corresponds to the optimal distance between pre-split boreholes. It is shown that the most optimal distance between the pre-split borehole charges ensures the development of a fracture network for various rocks varies from 1.5 m for quartz-mica slate to 4.2 m for weathered ferruginous quartz. In this case, the explosive charge should be 100 mm in diameter and

should be placed in the polyethylene sleeve along the borehole axis (the linear mass of the charge is 7.46 kg/m). 4. A design for a network borehole charge, which implies that an explosive is placed uniformly in a polymer shell along its length and held in a vertical position along the borehole axis. The proposed technology for the formation of pre-split charges of explosives has been successfully tested in industrial conditions.

**Introduction.** One of the negative consequences in carrying out mass explosions in open pits is the back-break. This leads to a decrease in stability of bank slopes, reduction in angles of bank slopes in comparison with the project and requires additional costs for artificial strengthening of banks or cutback in order to restore transport berms on the ultimate pit and considerably complicates subsequent drilling and blasting operations [1]. To eliminate these drawbacks, presplitting blasting method that due to the creation of a shielding plane makes it possible to obtain a relatively flat and stable surface of the bank and reduce the amount of destruction behind the final boundary is used [2, 3].

At present, a significant progress has been achieved in the field of application of contour blasting in open pit mining. However, a number of key issues related to the choice of rational parameters of contour blasting, predictive estimate and choice of methods that allow increasing the efficiency of the formation of the angles of bank slopes has not been solved. One of the most significant drawbacks is that the requirements for stability of pit banks is not taken into account when choosing parameters of drilling and blasting operations. The influence of methods of blasting works on stability of bank slopes is only ascertained after the performance of these works. This determines the relevance of studies aimed at the development of methods for controlling the process of rock destruction in contour blasting and determination of rational parameters of contour blasting that allow obtaining stable steep bank slopes.

**Characteristics of contour blasting methods** In the practice of blasting operations, three methods of contour blasting have become the most widespread methods: smooth-wall blasting, blasting of contour charges simultaneously with the main industrial holes and presplitting. The method of smooth-wall blasting involves the



blasting of contouring charges after the ripping of rock in the border zone. It ensures the breaking of the most broken part of the rock mass and obtaining of a relatively smooth surface of separation along the line of pre-split charges of explosives. The method fully meets requirements of high precision of contouring of the profiled development and found application in mine construction.

The use of boreholes with a large diameter (150–250 mm) in the crushing of rocks in the open pit leads to a sharp increase in the size of the fracture zone, especially in the upper part of the bank. In this case, the smooth-wall blasting does not usually provide the deformation zone required in terms of stability limitation. Positive results can be achieved only in large-block rock masses composed of hard ( $f = 15\text{--}20$ ) viscous rocks at bank slope works along intrinsic fractures falling at large angles towards the open pit.

The second method involves the use of contour contiguous boreholes with small charges of explosives that are blasted simultaneously with the main sets of boreholes. This approach is simplistic, but it does not fully ensure the security of the rock mass beyond the boundaries of the industrial block to be blasted.

The method of presplitting provides for the blasting of boreholes drilled along the contour of the block prior to blasting of the main boreholes or the approach of blasting operations in this section. In this case, even before the explosion of the main fragmenting charges, a screening fracture is created along the design contour. The presence of such fracture makes it possible to reduce stresses by 2-4 times at the front of the compression wave that is formed by the explosion of the main charges, and, thereby, reduce the width of the deformation zone [4].

The screening fracture along the design contour is created by simultaneous blasting of a set of contiguous contour borehole charges. Charges are located in the boreholes both uniformly along the entire length and uniformly. In terms of design, charges from strings of explosive cartridges, sleeve charges of sheathed slurry explosive, as well as charges with air or inert gaps. The presence of an air gap in the borehole facilitates the blast blow-off. As a result of this, even when blasting explosives are used, the effect of a blasting on the rock mass is close to the effect of cratering explosives.

When neighboring charges interact, one or more fractures that are

expanded under the influence of pressure of detonation products are formed between them. The number of fractures that are formed between holes depends on the fracture pattern of the rock mass, rock properties, velocity of the explosion energy supply to the mouth of the fracture. In practice, a fracture network usually develops along the pre-splitting line. The opening of the fracture between the contour boreholes occurs due to the compaction of cavities in the adjacent part of the rock mass. At present, several methods of approach to the determination of the parameters of contour charges are known.

Despite various approaches to the implementation of the aforementioned methods, upon blasting the contoured borehole charges, a fracture or a layer of broken rock mass that shields the shock waves, i. e. prevents their passing into the aquifer rock mass, is formed in all of them. Therefore, there is no fundamental difference between the mentioned methods. In this regard, the quality of contour blasting and influence of various factors on it can be considered jointly for all these methods.

**Analysis of scientific studies.** Conditions for the formation of a fracture during contour blasting have been considered by V. A. Borovikov. In particular, dependences that establish conditions for the fracture formation are proposed as a consequence of the development of the zone of plastic deformations during the interaction of two contiguous contour charges.

The authors of [5] offer to use the dependence obtained on the assumption of the development of the continuous main fracture along the line of charges for the calculation of contour blasting parameters. The calculation is based on the transition from compressive stresses to tensile stresses that should determine the tensile strength of the rock.

B. N. Kutuzov proposes the calculation of parameters of contour charges on the assumption of simultaneity of blasting along the entire surface of the contour that is only permissible for very small distances between the charges.

However, the above methods for the determination of contour blasting parameters offer general dependences that only establish the ratio of the main parameters of contour charges (borehole diameter, distance between boreholes, charge mass) taking into account certain

characteristics of rocks. The choice of rational parameters of contour blasting for specific mining and geological conditions remains completely unrevealed. In addition, taking into account the strength of fractured hard rocks based only on the strength factor and neglect of the influence of the fracture pattern of the rock mass lead to a sharp decrease in accuracy of calculations. In particular, the calculation of parameters of contour borehole charges according to the proposed methods gives a difference in the linear mass of the explosive charge by more than three times. Therefore, at present, real parameters of contour blasting are based on generalized actual industrial data.

In particular, reference parameters of contour borehole charges obtained as a result of the generalization of actual indices are given in work [6]. It is recommended to take the borehole diameter within the range from 100 mm to 150 mm. At the same time, it is noted that positive results of contour blasting can be achieved even with a larger borehole diameter. Approximate values of contour charges parameters, when using boreholes with a diameter of 100 mm to 150 mm are given in Table 1.

**Table 1**

**Reference parameters of contour charges**

Rock-hardness ratio, $f$	Explosive ratio, $\text{kg/m}^3$	Mass of charge per 1 meter of the borehole (kg) depending on distance (m) between contour boreholes		
		1,0	1,5	2,0
16-20	0,8-1,0	1,5	2,2	3,0
10-12	0,5-0,6	1,0	1,5	2,0
6-8	0,3-0,4	0,8	1,2	1,5

In work [7], it is recommended to select parameters of contour charges in accordance with Table 2.

The discrepancy between parameters recommended for the use in [6] and [7] is due to the difference in criteria for evaluating results of contour blasting. It is evidence that the choice of contour blasting parameters in accordance with the table data is rather approximate.

Table 2

## Parameters of contour charges

Distance between boreholes, m	Diameter of the borehole, mm	Linear mass of the charge, kg/m, at hardness ratio $f$		
		12-16	6-12	4-6
0,8-1,0	80-100	0,7	0,5	0,4
1,0-1,2	80-100	0,8	0,6	0,5
1,2-1,5	100	1,0	0,8	0,7
1,5-2,0	100-150	1,5	1,2	1,0
2,0-2,5	100-200	1,8	1,5	1,2

It should be noted that quality of contour blasting performance depends, first of all, on the distance between boreholes in the pre-split, density of explosive charging in the borehole, borehole diameter and charge design. To obtain necessary results of the explosion, it is necessary to consider the combined effect of these factors. In general, the effectiveness of contour blasting depends on the optimal distance between the boreholes in the pre-split that should ensure such interaction of energy flows of the explosion, at which the rock mass is only destroyed in the required rock sheet of a certain width.

In most cases, the justification of the method of contour blasting in open pits consists in choosing a rational design of borehole charges with a fixed distance between them [8] or in the optimal arrangement of contour boreholes of different diameters with a reduced content of the explosive in borehole charges [9]. The properties of the rock mass to be destroyed are generally accounted indirectly, based on production experience of the mining enterprise.

The authors [6] have also carried out an analysis of methods for determining parameters of contour blasting for the formation of permanent open-pit sides in rock excavation. It is established that they do not allow to reasonably justify parameters of drilling and blasting operations for specific mining and geological conditions, but they only allow to roughly estimate the relationship between the main characteristics of contour blasting.

The most complete studies were carried out in [10]. The author establishes a criterion for evaluation of the contour blasting effectiveness from the perspective of the condition for the formation of a fracture network between charges during their explosion.

However, the work does not consider the effect of structural features of borehole charges and their parameters on the optimal distance between the boreholes.

Analysis of the above research studies has shown that the most effective way of limiting the deformation zone along the design surface of separation is presplitting. The maximum reduction in the width of the fractured zone along the design surface of separation within the border zone can be achieved by:

- the use of inclined boreholes;
- short-delayed blasting;
- the use of special blasting schemes;
- the reduction of the mass of explosives to be blasted simultaneously by increasing the number of deceleration groups or reducing the size of the block to be blasted;
- the reduction of the borehole diameter;
- application of axial air gaps in the borehole;
- the use of decked charges.

The analysis of contour blasting methods and calculation of contour blasting parameters shows that, for effective destruction of rock masses, it is necessary to create conditions, in which directional blasting energy flows will be formed in the rock mass due to the total shock wave impulse.

#### **Peculiarities of the screening fracture formation process.**

According to the theory of blasting, when a detonation wave passes through an explosive, gaseous products of detonation are formed, the pressure of which is reached during the initial period of several hundred thousand atmospheres.

Rapid expansion of detonation products leads to a sudden change in the pressure and density in the environment (rock mass), i.e. a shock wave is generated. At the optimal parameters of contour charges, pressure at the front of the shock wave at the borehole walls should decrease substantially to let it be quickly transformed into the stress wave. This is confirmed by the absence of the deformation zone on the remained traces of contour boreholes in the rock mass.

In the zone of action of stress waves, the destruction of the mass occurs due to tensile stresses. Dynamic compression stresses caused by the contour charge blasting act for only tens of microseconds and only result in a certain weakening of the mass due to the increase in

the length of fractures that were in the rock.

Further development of fractures is determined by quasistatic stresses caused by pressure of gaseous products of detonation on the borehole walls. Depending on the height of the bank, the duration of the explosion gasses pressure phase is 8–20 ms.

Total stresses from adjacent charges determine the direction of maximum destruction of the mass along the line of charges. The bigger the distance from the plane where charges are located, the lower total stresses are. At some distance, the value of tensile stresses becomes equal to the dynamic strength of the rock. This condition determines the greatest deviation of the boundary of zone where blocks are fragmented from the plane where charges are located for any of its points.

The actual mass is usually cut by several fracture systems and, in addition, has a number of randomly oriented fractures that greatly complicates the mechanism of forming the screening layer from the crushed rock. With large-block structure of the mass, when the sizes of structural blocks exceed or are close to the accepted value of the distance between contour charges, natural fractures have little effect on the width of the fractured zone along the line of charges. In this case, the width of this zone along the line of charges is comparatively relevant, and the surface of the bank slope obtained is the most smooth. An exception is the case when the plane where pre-split charges are located is parallel to the main system of natural fractures (or crosses it at a small angle). In this case, opening of fractures that already exist in the mass occur and is accompanied by an insignificant formation of new fractures.

With a small-block structure of the mass and no fractures parallel to the plane where charges are located, the width of the zone of intensive fracturing is reduced. This is due to the partial unloading of the mass due to the possibility of some displacements along fractures of various systems and the uneven distribution of loads on adjacent structural blocks along the line of charges. Significantly smaller sizes of structural blocks in comparison with the distance between charges determine the presence of loosening surfaces (interblock fractures) in the immediate vicinity of the plane of charges. To destroy such mass, there is no need for such high stresses that are required for the destruction of a large block mass. Therefore, the fractured zone in

the small-block mass, along the plane of pre-split charges, is framed by the fractured zone along interblock fractures. This is the reason for the large unevenness of the slope surface in comparison with the large block mass.

Observations established that these irregularities usually amount to one quarter of the distance between charges, and, in the most unfavorable cases, they can reach half the distance between charges. However, the roughness of the slope surface does not characterize the stability of the bank. In open-pits, there is no need to achieve an ideally smooth surface of the slope and apply a certain degree of unevenness of the solid mass to evaluate the correctness of the selected parameters of blasting by presplitting method.

When performing blasting by presplitting method on the banks consisting of small block rocks, in the case of placement of pre-split charges in a plane intersecting a system of pronounced fractures at a small angle, the formation of a screening fracture occurs due to the opening of the fracture of this system and partially fractured joints along the line of charges. In this case, the slope surface has a stepped appearance. Traces of boreholes are poorly preserved.

Considering the high rate of fracture growth in the rock (from 200 to 1800 m/s) and a relatively small distance between pre-split charges (up to 3.0 m), it is possible to estimate the time necessary to form a continuous fracture along the line of charges. Even supposing that fractures develop only from boreholes, the time necessary for the growth of fractures and formation of the continuous fracture will amount to milliseconds that is almost an order of magnitude shorter than the time for supporting the detonation products of the explosive in the blasting cavity. Such significant difference makes it possible to consider that the rock fracturing along the line of boreholes occurs under the influence of tensile stresses caused by quasistatic pressure of detonation products equal to the average pressure in the borehole at the initial moment of loading.

As the opening of the gap progresses, the pressure decreases and becomes balanced by the mass reaction at some point. At that moment, the fracture opening reaches its highest value. The decrease in the pressure of detonation products leads to a partial collapse of the fracture due to the elastic restoration of the shape of the deformed joints and partial opening of interblock fractures. The complete

collapse of the screening fracture is prevented by the presence in it of a solid aggregate with fractured rock. Regardless of the fractured zone width and fracture opening, the cavity is usually filled by 25–35 %. The structure of the screening layer is a set of plate-like blocks of the rock, fractures between which are partially filled with clamped rock blocks of smaller fractions. All the blocks were subject to some displacement during the efflux of detonation products.

The upper part of the ground is significantly broken by the previous blasting of the above bank.

Within the broken zone, the opening of the screening fracture essentially depends on the degree of the ground destruction; it decreases with depth, and the width of the fractured zone along the line of charges depends on the degree of ground destruction slightly less.

Therefore, within the slightly ground zone, where the bank is still potentially stable, in calculating the width of the degradation zone along the line of charges can be considered independent of the depth, and, in calculating the fracture opening, it is necessary to take into account the degree of condition of solid mass ground at a given depth. Within the framework of an unbroken massif, the width of the fractured zone and fracture opening can be assumed as a constant one in calculations. It should be borne in mind that a local change in the blocky structure, orientation and opening of fractures can lead to a local change in the width of the fractured zone by up to 1.5–2.0 times.

The upper part of the slope in open pits is usually severely broken and is potentially unstable. The thickness of this zone depends on the mass structure, strength characteristics of the rock and contacts of interblock fractures, as well as on the adopted technology for blasting operations. The principal difference between the action of the presplit charge blasting within the severely broken zone and action in an unbroken mass does not allow to apply the description of the mechanism of the screening fracture formation to this zone. Therefore, when calculating the parameters of pre-split charges and evaluating blasting results, it is not necessary to take this zone into account.

#### **Theoretical studies of the interaction of blasting energy flows.**

Let's consider a blasting of the presplit charge of the explosive in the

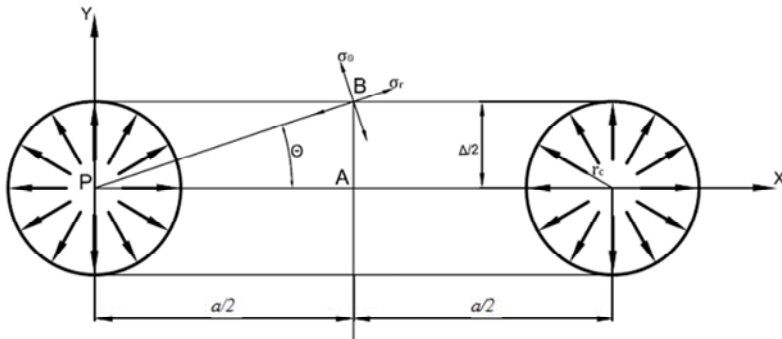


rock mass. According to [11], as a result of the blasting in the mass, a stress state is formed around the cylindrical cavity. In considering the two-dimensional problem, it can be described by radial component  $\sigma_r$  and azimuthal component  $\sigma_\theta$  of stresses in the polar coordinate system. The transition to a system of rectangular coordinates is carried out by the relations:

$$\begin{aligned}\sigma_x &= \sigma_r \cos^2 \theta + \sigma_\theta \sin^2 \theta; \\ \sigma_y &= \sigma_r \sin^2 \theta + \sigma_\theta \cos^2 \theta; \\ \tau_{xy} &= (\sigma_r - \sigma_\theta) \sin \theta \cos \theta.\end{aligned}\quad (1)$$

Since, in the case of the simultaneous blasting by presplit charges, it is necessary to create the minimum permissible stress state of the rock mass in the middle on the line where boreholes are located, when choosing the parameters of drilling and blasting operations, it is necessary to proceed from the condition that the process of rock destruction within the distance between charges with a given width  $\Delta$  should be ensured precisely at this place.

With the simultaneous explosion of adjacent cylindrical charges of explosives, the displacement of the rock mass at points lying along the AB line (Figure 1) in the direction radial from the charge is impossible. Therefore, to estimate the state of stress at point B, we consider  $\sigma_\theta^B=0$ . Tangential stresses  $\tau_{xy}^B$  at point B caused by blasting of adjacent borehole charges are equal in magnitude, but they are reverse in direction, and their resultant value will be  $\tau_{xy}^B=0$ .



**Fig. 1.** Scheme for the determination of blasting by presplitting method

As a result of summing radial stresses at point B, equations (1) will be as follows:

$$\begin{aligned}\sigma_x^B &= 2\sigma_r \cos^2 \theta; \\ \sigma_y^B &= 2\sigma_r \sin^2 \theta; \\ \tau_{xy}^B &= 0,\end{aligned}\tag{2}$$

where  $\sin\theta$  i  $\cos\theta$  are parameters that are determined by rations:

$$\cos^2 \theta = \frac{a^2}{a^2 + \Delta^2}; \quad \sin^2 \theta = \frac{\Delta^2}{a^2 + \Delta^2}.\tag{3}$$

When the rock mass is destroyed in the bank contour, the tensile stresses that are defined at point B by the dependence:

$$\sigma_p^B = \sigma_y^B - \nu\sigma_x^B,\tag{4}$$

where  $\nu$  is the Poisson's ratio.

Radial stresses resulted from the blasting of a single cylindrical charge are calculated by formula [10]:

$$\sigma_r = P \cdot f_p(r) \cdot f_s(r),\tag{5}$$

where  $P$  – pressure of detonation products on the borehole walls, Pa;  $f_p(r) = (r_c/r)^{0.5}$  – function of the geometric divergence of cylindrical waves and distance;  $f_s(r) = \exp(-\alpha r/r_c)$  – absorption function that takes into account the loss of stress waves;  $a$  – distance between boreholes, m;  $r_c$  – borehole radius, m;  $r$  – current distance to the charge, m;  $\alpha$  – absorption coefficient.

Absorption coefficient  $\alpha$  is determined based on experimental studies or by the empirical dependence:

$$\alpha = -0,155 \cdot 10^{-8} \rho c_l + 0,073,\tag{6}$$

where  $\rho$  – specific weight of rock, kg/m<sup>3</sup>;  $c_l$  – velocity of compressional-wave propagation in the rock mass, m/s.

Let's substitute in formula (5) value  $\sigma_y^B$  and  $\sigma_x^B$  by taking into account (4) and value  $\sigma_r$  c (6) at

$$r = \sqrt{\left(\frac{a}{2}\right)^2 + \left(\frac{\Delta}{2}\right)^2} \quad (7)$$

and we obtain equation [12]

$$\sigma_p^B = 2P\sqrt{d_c} \frac{va^2 - \Delta^2}{\sqrt[4]{(a^2 + \Delta^2)^5}} \exp\left[-\alpha \frac{\sqrt{a^2 + \Delta^2}}{d_c}\right]. \quad (8)$$

Due to the fact that the density of the explosive charge in the pre-split boreholes is low, the pressure of detonation products on the borehole walls is calculated as follows:

$$P = \frac{Q\omega(\gamma-1)\eta}{V_c - \alpha_k}, \quad (9)$$

where  $Q$  – mass of the explosive charge in the borehole, kg;  $\omega$  – specific energy of explosives, J/kg;  $V_c$  – borehole volume, m<sup>3</sup>;  $\alpha_k$  – covolume of blasting gases, i.e. volume of gas molecules per 1 dm<sup>3</sup>, dm<sup>3</sup>;  $\gamma$  – isentropic exponent:  $\gamma = 1.45$ ;  $\eta$  – energy loss factor.

Since the pressure in the pre-split borehole does not usually exceed 200 MPa, value  $\alpha_k$  can be neglected. Taking into account formulas

$$Q = q \cdot l \quad (10)$$

$$V_c = \frac{\pi d_c^2}{4} l, \quad (11)$$

where  $q$  – linear mass of the borehole charge, kg/m;  $l$  – charge length, m;  $d_c$  – borehole diameter, m, expression (9) will get the following form:

$$P = \frac{4q\omega(\gamma-1)\eta}{\pi d_c^2}. \quad (12)$$

Since the main destructive stress in blasting by presplitting method is the tensile stress that, in the ideal case, should create a separation fracture, in this case, the condition for the destruction of the rock mass are:

$$\sigma_p^B \geq \sigma_{kp}^t, \quad (13)$$

where  $\sigma_{kp}^t$  – ultimate tensile strength of rock, Pa.

In accordance with the basic principles of mechanics of fracturing the continuous medium, the critical stress value at two-dimensional state of stress for separation fractures is determined based on the expression:

$$\sigma_p = \frac{K_c}{\sqrt{\pi l}}, \quad (14)$$

where  $K_c$  — critical factor of stress intensity;  $l$  — half length of the fracture.

In connection with the fact that the blasting of two charges of explosives cause effect on the rock mass between pre-split boreholes, the value of half length of the fracture can be taken as the half distance between pre-split charges, i.e.  $l = a/2$ . Taking this into account, (14) takes the following form

$$\sigma_p = \frac{K_c}{\sqrt{\frac{\pi a}{2}}}. \quad (15)$$

Thus, using formulas (8), (12) and (15), we determine the necessary linear mass of the charge that ensures the condition for the fracture development between pre-split boreholes:

$$q = \frac{\sqrt{2\pi}}{4} \frac{K_c}{\omega\eta\sqrt{a}} \sqrt{d_c^3} \frac{4\sqrt{(a^2 + \Delta^2)^5}}{\mu a^2 - \Delta^2} \exp\left[\alpha \frac{\sqrt{a^2 + \Delta^2}}{d_c}\right]. \quad (16)$$

The analysis of Eq. (16) shows that there is a definite dependence of the linear mass of the pre-split charge of the explosive  $q$  on the distance between pre-split charges  $a$  at established properties of the rock mass.

Calculations have been made for rocks that are represented within the open pit of the Poltava Mining and Processing Combined Works to establish a graphical relationship between  $q$  and  $a$  under the following parameters of the pre-split borehole charge: borehole diameter –  $d_c = 0.25$  m; specific energy of gramonite blasting 79/21 –

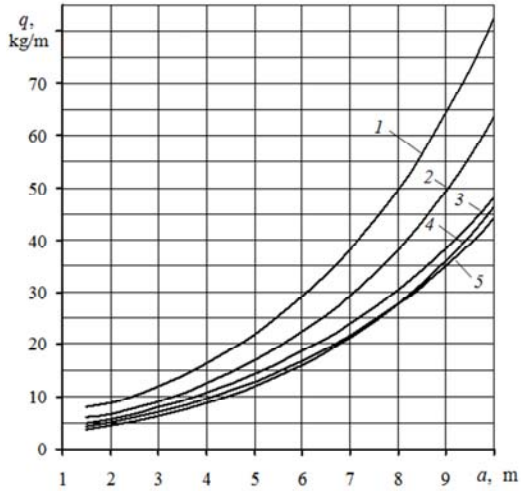
$\omega=4285$  kJ/kg; energy loss factor –  $\eta=0.7$ .

Indicators characterizing physical and mechanical properties of rocks are given in [13]. Values of the absorption coefficient  $\alpha$  were obtained from the calculation formula (6). Figures 2 and 3 show graphical dependences of the change in the linear mass of charge  $q$  on the distance between the pre-split boreholes  $a$  during the destruction of rocks of different hardness that are located within the open pit [14].

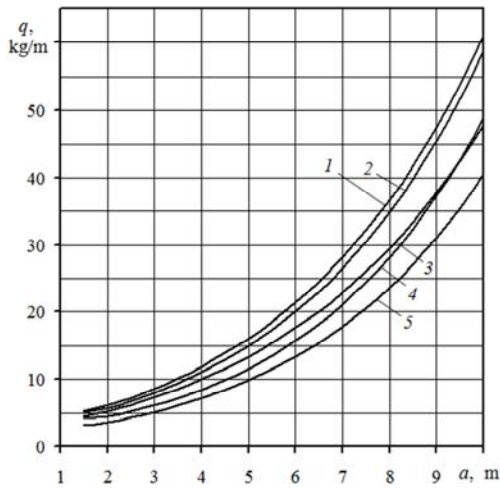
As a result of the analysis of dependencies, it is established that the optimal value of the distance between the boreholes corresponds to each type of pre-split borehole charge design that is characterized by the average linear mass of the explosive. This distance should ensure the development of a fracture network between pre-split charges of explosive. In particular, when blasting by presplitting method in the quartz-biotite shales (Fig. 2, dependence 1) with gramonite 79/21 at a distance of 3.0 m from each other, the linear mass of the charge should be 12 kg. If the work is carried out in the quartz-mica shales, the linear mass of the charge decreases to 9 kg (Fig. 2, dependence 2). Thus, to create a directed network of fractures with a distance between the boreholes of 3 m, the greatest amount of explosive per 1 m of charge required for the destruction of quartz-biotite shales is 12 kg, the smallest amount for weathered ferruginous quartz is 5 kg (Figure 3, dependence 5).

In general, it should be noted that with the increase in the distance between pre-split charges, the linear mass increases in the parabolic dependence. The nature of the change in established dependencies for different types of rocks is not the same, since some of them intersect between themselves (see Fig. 2, 3). This is explained by the combined influence of physico-mechanical properties of the rock mass on parameters of pre-split borehole charges.

Dependencies at Fig. 2 and 3 will also be analyzed from the point of view of placing the explosive charge in the borehole. If the borehole charge is placed in a polyethylene sleeve with a diameter of 100 mm, its linear mass will be 7.46 kg/m. The most optimal distance between such pre-split charges for various rocks varies from 1.5 m for quartz-mica shale (Figure 2) to 4.2 m for weathered ferruginous quartz (Figure 3).

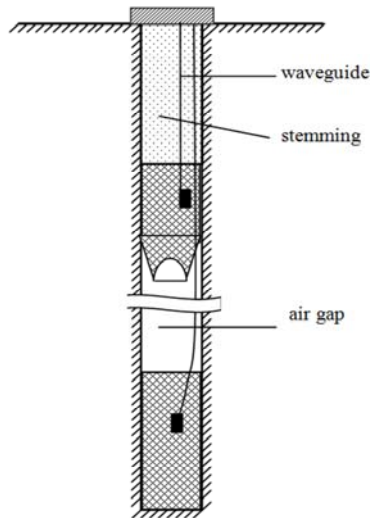


**Fig. 2.** Dependence of the linear mass of the charge  $q$  on the distance between the pre-split boreholes  $a$  during the destruction of rocks: 1 – quartz-biotite slate; 2 – quartz-mica slate; 3 – magnetite quartzite; 4 – buck quartz; 5 – cumingtonite-magnetite quartz



**Fig. 3.** Dependence of the linear mass of the charge  $q$  on the distance between the pre-split boreholes  $a$  during the destruction of rocks: 1 – plagiogranite, migmatite; 2 – weathered shale; 3 – amphibolites; 4 – granitoids; 5 – weathered ferruginous quartz

In the open pit of the Poltava Mining and Processing Combined Works, the formation of pre-split borehole charges is carried out with the use of a cumulative locking device, with the help of which one air or water-filled gap is formed (Figure 4) [15]. The gap is created by suspending the top of the charge or stemming. Thus, the value of this gap is regulated by the linear mass of the explosive of the pre-split borehole charge.

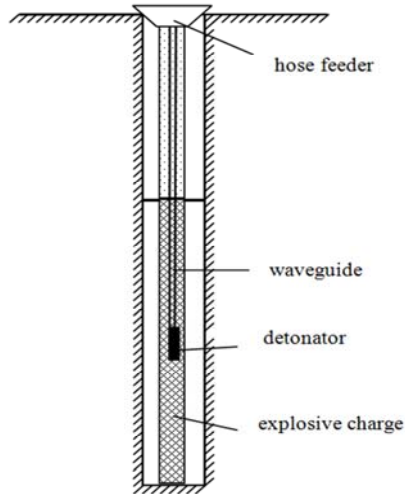


**Fig. 4.** Design of the borehole charge with the use of the cumulative locking device

However, such design of the pre-split charge explosive has some drawbacks. In particular, the technology of charging boreholes is complicated, the risk of stopping detonation in the borehole is increased due to the presence of discontinuities in the charge and it is necessary to place an intermediate detonator in each part of the charge. This, accordingly, increases the cost of blasting means and increases the laboriousness of the borehole charging process.

To eliminate these drawbacks, it is proposed to form a continuous borehole charge with a reduced diameter. The task of forming such pre-split charge was solved by placing it in a polymeric shell and holding the charge column in an upright position. For this purpose, the technology of loading industrial explosives into the sleeve feed

device is used [16]. Fig. 5 shows the design of the charge with the use of the sleeve feed device that form borehole charges with a diameter from 127 mm to 200 mm [17, 18].



**Fig. 5.** Design of the borehole charge with the use of the sleeve feed device

The above technology has been successfully tested in industrial conditions. In pre-splitting boreholes with a diameter of 250 mm, the charge with a diameter of 127 mm is placed in the polymer sleeve. Under such conditions, the consumption of bulk explosives per 1 m of the borehole charge was 13 kg and it was 19 kg emulsion explosives. According to Fig. 2, for the magnetite quartzites, the distance between the pre-split boreholes is 4.5 m. This design ensured the uniformity of the explosive distribution along the length, location of the charge along the borehole axis, technology of the charge formation in the borehole with any level of water cut [13].

**Conclusions.** Based on the results of the conducted studies:

1. The dependence of the average linear mass of the explosive charge on the distance between boreholes that ensures the development of a directed network of fractures between pre-split charges upon their blasting has been obtained.

2. It has been established that each type of the pre-split borehole



charge design that is characterized by the averaged linear mass corresponds to the optimal distance between pre-split boreholes. In particular, to create a network of directed fractures with a spacing between boreholes of 3 m, the greatest amount of explosive per 1 m of charge is 12 kg with the destruction of quartz-biotite shales, and the smallest amount is 5 kg with the destruction of weathered ferruginous quartz. In general, it should be noted that with the increase in the distance between pre-split charges, the linear mass increases in the parabolic dependence.

3. It is shown that the most optimal distance between the pre-split charges ensures the development of a fracture network for various rocks varies from 1.5 m for quartz-mica slate to 4.2 m for weathered ferruginous quartz. At this, the explosive charge is placed in a polyethylene sleeve with a diameter of 100 mm along the borehole axis (the linear mass of the charge is 7.46 kg/m).

4. A design for a network borehole charge, which implies that an explosive is placed uniformly in a polymer shell along its length and held in a vertical position along the borehole axis. The mentioned technology for the formation of pre-split borehole charges has been successfully tested in industrial conditions.

5. A promising direction of further scientific developments are theoretical and experimental studies of the effect of the natural fracture pattern of the rock mass on parameters of blasting by presplitting method.

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# GEOMECHANICAL MONITORING FOR UNDERGROUND MINING MINERAL DEPOSITS

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**Abstract.** The subject of research is geomechanical monitoring for underground mining of deposits. The results obtained by analysis and generalization methods, theoretical studies using methods of mathematical physics and rock mechanics, experimental studies based on methods of visual observations of electrometric, vibro-acoustic and electromagnetic measurements are used in the work. The purpose of the research is to substantiate and develop of geomechanical monitoring methodology for controlling the properties and diagnostics underground geotechnical systems state for various purposes.

The scientific bases and methodology of geomechanical monitoring of mines, as multilevel and multicomponent structures of control, forecasting and diagnostics are stated. Scientific principles of the development of diagnostic models for complex geotechnical systems and processes are substantiated, taking into account the interaction of diagnostic systems that ensure the effectiveness of monitoring. The peculiarities of using shock-wave, ultrasonic, electrometric and electro-magnetic methods in geomechanical monitoring are described. Mining technical principles of using elements of underground geomechanical monitoring in coal, ore and non-metallic mines have been developed to improve the safety and efficiency of their operation.

Combined solution of the tasks set up ensured the implementation of the developed concept, the core principal of which is in the separation of the geotechnical system for the control of individual objects and processes, the further synthesis of geomechanical monitoring of the production cycle of the entire underground system, and the development of measures to improve the efficiency and safety of mining. The implementation of this

concept ensured the solution of an important scientific and technical problem of improving the efficiency of mines and ores by creating and implementing highly effective technologies for assessing, forecasting and monitoring their condition.

### **Actuality and principles of monitoring organization**

In the modern world, energy and industrial potential remain the main components of the national security of any civilized state. That is why the mining industry plays a huge role in the development strategy of Ukraine, as the basis of the country's energy complex, the metallurgical and chemical industries. However, a constant increase in the depth of development of deposits and deterioration of mining and geological conditions of production leads to a decrease in the economic performance of most mines and an increase in the level of injuries in the industry. Efficiency, stability and safety of functioning of mining enterprises are determined by information on the state of the production environment and the level of control of most production processes. This level can be ensured by implementing a holistic system for integrated monitoring of mines and ores, for example, according to the scheme shown in Fig. 1 [1].

In the above block diagram, the main part is the control and prediction of geomechanical processes determined by the properties of the rock massif, its interaction with workings, supports and security structures, the initial state of the massif and its behavior during field development, dynamic and gas dynamic occurrences of rock pressure, etc. These equations were aggravated by the intensification and concentration of production, which required the development and attraction of new progressive methods of continuous monitoring of the environment and their transformation into geomechanical monitoring system.

According to the common terminology, geomechanical monitoring is a complex system for observation the state of a rock massif, estimating and forecasting technogenic changes in the lithospheric environment in order to identify negative consequences and develop recommendations for their elimination in the development of mineral deposits.

The main task of geomechanical monitoring is to promptly and

continuously determine, evaluate or forecast:

- physical and mechanical properties of rocks and massifs;
- the stress-strain state (SSS) of the rock massif with the identification of the parameters of inelastic deformation zones, reference and geostatic pressures;

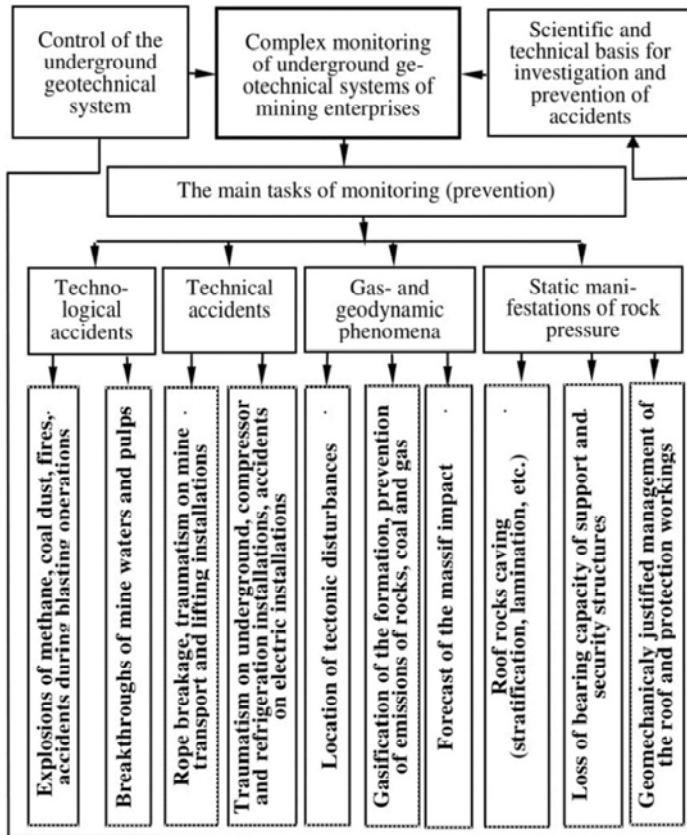
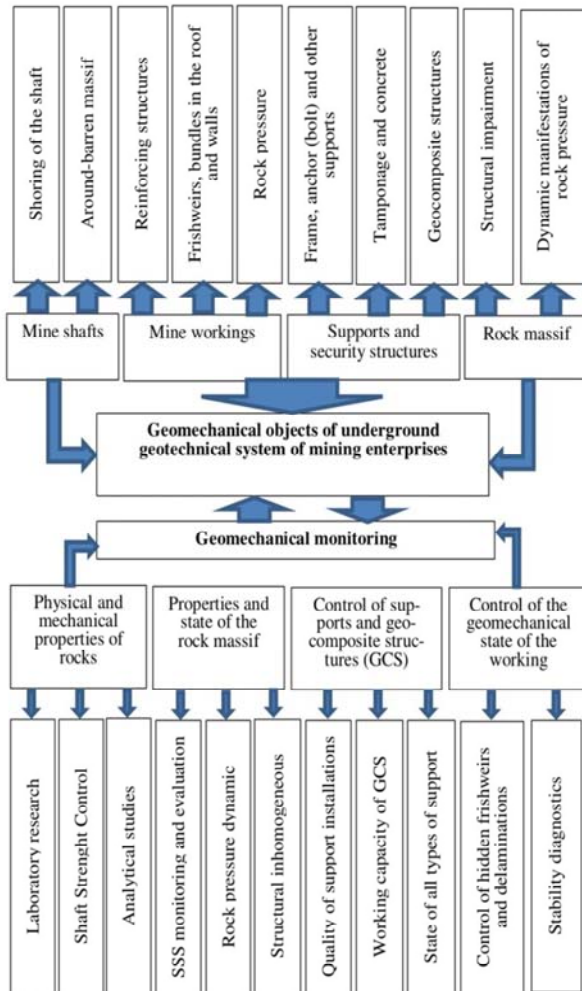


Fig. 1. Components of complex monitoring

- vector-force characteristics of the system "support-massif";
- state of supports and security structures;
- the possibility of anomalous manifestations of rock pressure, including rock collapse, dynamic and gas dynamic phenomena





**Fig. 3.** Structural diagram of geomechanical monitoring

Concerning the choice of methods and means of control, we should note that in all cases, instrumental methods of control and diagnostics should be preceded by visual studies, which results in the decision to involve one or (to increase the reliability) of the full complex of methods of mining geophysics.

The main prerequisite for geophysical control in mine workings is insufficient volume and rather low reliability of data that were previously obtained during visual inspection, as well as during exploration and control drilling. When choosing the methods of control, the following factors are taken into account: the nature of the tasks that can be solved with the help of certain geophysical control methods; a priori results on the informative degree of the method; conditions and possibilities for performing measurements; results of preliminary visual examination; availability of appropriate technical means; labor and material costs; duration of the entire work cycle. When performing diagnostics of the state of fixed and loose sections of mine workings, the following geophysical methods are recommended: ultrasonic, shock wave (vibro-acoustic), electro-metric and electromagnetic.

*The ultrasonic method* (US) is used to express the physico-mechanical properties of rocks in laboratory conditions, including using mobile laboratories, as well as assessing the stress-strain state of rocks and the degree of their fracturing in the massif.

*Shockwave method* is used to assess the state of concrete, reinforced concrete (including tubing), anchor, spattered concrete and multi-layered supports, as well as control of hidden hocks and delaminations in the roof and walls of the excavations.

*The electrometric method* (EM) is used to evaluate the stress-strain state, fracturing and moisture content of the rock massif.

*The electromagnetic method* (EMM), based on recording the intensity of pulsed electromagnetic radiation of rocks (IEMER), is used to quickly identify large sections of increased cracking in the massif and to assess the development of deformational processes in the "support-massif" system that occur under the influence of rock pressure, and also various technological and mining-geological factors, accompanied by triboelectric effects.

The application of each of these methods is preceded by the definition of its informativeness in relation to specific conditions and diagnostic tasks. This process involves the statistical processing of an array of data obtained at known abnormal and undisturbed sections. A specific type of equipment is selected based on the tasks assigned and the availability of serial or single production carried out in specialized organizations, as well as from domestic or imported



instruments used for monitoring and diagnosis in other fields of technology. The main criteria for choosing the equipment are as follows: compliance with the required technical characteristics, portability, availability of an autonomous power source, mine execution, immunity against electromagnetic interference of industrial frequency. When choosing modern technical means with the built-in software, it is necessary to be guided by such criteria: availability of an adapter for communication with personal computers; minimum requirements for hardware capabilities and software for processing data copied from the device's memory; the ability to test the device in the process of performing the work and automatically block the work in the event of faults, prompt viewing of the stored data.

### **Some features and order elements application of geomechanical monitoring**

The definition of physic-mechanical characteristics of rocks is regulated, as a rule, by regulatory documents. The periodicity of the control of the physical and mechanical characteristics of rocks, regardless of the availability of geological reports, is established from the following considerations: in the preliminary workings, the sampling interval should not exceed 50 m, with thickening in zones of tectonic disturbances up to 10 m; in the purification chambers, sampling is performed in each block with mandatory sampling of the ore body, as well as the rocks of the hanging and recumbent sides; The number of samples at each point of the bore should be at least 5.

To obtain a complete picture of the properties of the massif, it is necessary to perform a complex of physic-mechanical tests of rocks that involves the determination of the following characteristics by methods of direct measurements, non-destructive testing and, for some characteristics, calculation methods: density, specific gravity, porosity, shape and time of destruction in water, humidity, compressive strength in dry and water saturated states, tensile strength in parallel and perpendicular to stratification; modulus of elasticity, Poisson's ratio; angle of internal friction, coefficient of adhesion, abrasiveness, resistance to cutting. The determination is desirable to be performed by direct measurement methods in the

process of physical and mechanical tests on samples of a cubic shape with an edge of 40 mm and plates thickness of 20 mm using certified press equipment. To exclude the statistical error, the quantity of samples must be at least 6 cubes (cylinders) and 6 plates for each test. The loading process must be at a constant rate, maintained until the sample is destroyed.

In calculations at the transition from the sample to the massif, it is necessary to take into account the scale factor of strength [2].

Basic characteristics for the determination of elastic (dynamic) parameters – the propagation velocity of the longitudinal  $V_p$  and transverse  $V_s$  waves in the material, the measurement technique of which is regulated. Having determined the values  $v_d$  of the velocities, calculate the dynamic Poisson's ratio  $\nu_d$ , the dynamic shear moduli  $G_d$  and Young  $E_d$ , the dynamic coefficient of volume compression  $K_d$ , using the expressions [3]:

$$\nu_d = \frac{1 - 2(V_s/V_p)^2}{2 - 2(V_s/V_p)^2}; G_d = \gamma V_s^2;$$

$$E_d = 2G_d(1 + \nu); K_d = \nu_d \left( V_p^2 - \frac{4}{3} V_s^2 \right).$$

It should be taken into account that the values US of the elastic properties of rocks, obtained by mechanical methods and the ultrasonic method (by virtue of the difference in the physical principles of measurement) may not coincide. For rocks, as for all solids, the value of the dynamic modulus of elasticity is usually greater than the static one and the difference between them can fluctuate within a wide range. The discrepancy between the same rock characteristics is more than 2 % due to the presence of small cracks and pores in them, which, under considerable loads, are the main cause of the nonlinear dependence between strains and stresses. The denser the rock, the less the difference between the moduli of elasticity. Under the influence of static stress the rock is compacted. Therefore, for rocks that are in a stressed state, the difference between static and dynamic modules is less, that is, the value of the rock modulus in the array is closer to the dynamic one.

As a rule, geomechanical monitoring should be preceded by a complex of visual-optical observations of the state of the medium.

*The order of visual control.* The main objectives of visual inspection include: determining the feasibility of using specific geophysical control methods on the survey site; determining the boundaries of the controlled area; refinement of geophysical control parameters, as well as selection of the most suitable technical means for its implementation; elucidation of the need for additional technical measures to ensure the possibility of geophysical control; determination of the need to comply with special requirements for safety; obtaining initial data for the development of a work plan for the control of the surveyed site.

The simplest form of visual inspection is an examination with record keeping in the event of violations. A more thorough visual inspection involves photographing anomalies (if environmental conditions permit), as well as determining the linear dimensions of the anomalies and their position using technical means.

*Downhole Visual Control (DVC)* is separated into a different category, because it is based on the use of specialized technical means and solves problems that are unavailable when using other variants of visual observations. With reference to the problems of the state of assessing the support and the near-boundary zone of the rock massif, the use of visual downhole monitoring allows: to determine the type and structure of rocks surrounding the mine workings; identify the presence, determine the position and size of the segregation in the massif; identify the presence and determine the depth of voids behind the monolithic support of the mine; identify defects in concrete and reinforced concrete supports, cast strips; localize the location of water supply to production. To increase the informativeness of the studies, during the operation of the DVC complex, it is possible to switch the type of illumination in the visible and infrared light ranges, as well as in the mode of registering the protruding parts of the investigated surface. The set of equipment DVC include: a tablet (laptop); camcorder; a side observation nozzle for a video camera with light sources; a pre-pack with power, information and control cables. To conduct in-depth processing of information, calculations and forecasts, special programs are used, for example *Tiny Sheet 3*, whose calculation files are formed in advance and can be adjusted directly at work, based on their types and purposes.

The study is conducted both in specially drilled wells and in existing degassing, measuring and unloading wells.

The strength of geocomposite structures elements and rocks in the massif is determined by the shock pulse method according. For mass express determinations of the strength parameters of materials, one should use regression equations, each of which corresponds to certain lithotypes of the rock and reflects the relationship between the uniaxial compression strength limit and the shock pulse duration described by the parabolic mode of coupling. The regression equations for each type of rock and the type of geocomposite materials are determined experimentally.

The number of determinations on the plot should be at least 10, the distance between the measurement points is 15 cm, and from the edge of the structure 20 cm.

The measurements are carried out at a positive temperature (in exceptional cases, the structures can be inspected at a temperature of -10 °C, provided that the structure was previously at least one week with a positive temperature and relative humidity of not more than 75 %). When interpreting the results of measurements, calibration dependence is used between the strength of the material sample and the measured values of the informative parameter, constructed from the data of a series of tests of at least 15 sample cubes.

*Features of the methodology:*

- On the measurement plot there should be no cracks, segregations, as well as sources of groundwater filtration;
- when controlling the massif, impacts are applied across the layered, perpendicular to the surface of the outcrop, and when controlling hardening mixtures with filler in the form of rock crushed stone - approximately in the same ratio, both for the filler and for the binder component, followed by averaging the entire set of results.

Monitoring and evaluation of the SSS of the rock massif in the framework of geotechnical monitoring is carried out by indirect measurement methods: ultrasonic, seismic (SA), IEMER, which are characterized by high operational efficiency and low labor input.

*The ultrasound method* is based on measuring the acoustic parameters of the rock massifs and recalculating the obtained values into strains according to the calibration dependences obtained in the laboratory conditions. When planning the research, it should be take

into account that under uniaxial compression, significant changes (up to 30-40 %) in the propagation velocity of waves are observed in a direction parallel to loading, but in directions perpendicular to compression, there are practically no changes. More noticeable is the effect of pressure on the change in the magnitude of the damping coefficient. With increasing pressure, up to the onset of destruction, the attenuation coefficient decreases (by about 80-180 %).

The recommended frequency of elastic vibrations in the application of this method is in practice limited to a range from 25 to 200 kHz. Physical limitations of the method include fracturing and natural variability of the massif properties (heterogeneity).

Ultrasound is performed according to a borehole sounding scheme in mutually perpendicular directions. The values of normal stresses are determined by means of calibration graphs, and the quantitative evaluation of the components of the main stresses in the array is carried out according to the formulas:

$$\sigma_{xx} = 3K \left( \frac{V_{p_x}^2}{AV_{p_x}^o{}^2} - 1 \right); \quad \sigma_{yy} = 3K \left( \frac{V_{S_{yy}}^2}{AV_{S_{yy}}^o{}^2} - 1 \right); \quad \sigma_{zz} = 3K \left( \frac{V_{S_{zz}}^2}{AV_{S_{zz}}^o{}^2} - 1 \right),$$

where  $\sigma_{ii}$  - the components of the principal stresses;  $A$  - the acoustic fracture coefficient; the superscript "o" denotes the parameter for the sample.

*The seismo-acoustic (SA) method* is based on the established dependence of the spectrum parameters of the acoustic signal initiated in the massif by various sources, on the strained-deformed state of the massif. The source of SA pulses is the cracks appearing in the massif as a result of its deformation, under the influence of redistribution or changes in stresses.

Practically two methods are used:

- acoustic sounding, when the geophone is located on the surface of rock outcrops, and pulsed excitation of the massif is carried out by applying a series of impacts at a distance of 1-3 m from the geophone in 2-4 places. As an informative, for the conclusions about the state of the massif, the middle part of the spectrum of the acoustic signal is used;

- registration of the massif noise, when the source of sound is the process of crack formation in the massif, while the receivers are

located on a specific network, which allows not only to state the development of the process, but also to determine the coordinates of the source.

Seismo-acoustic information should be processed daily, and the results of processing are recorded in an electronic journal on which time series of SA emission activity with the necessary time resolution are constructed, reflecting the intensity of cracking processes at various sites.

It should be taken into account that SA emission is a non-stationary process, which is related to the cycle of work in the mine, periodic landings of the bottomhole formation, as well as seasonal variations in the amount of fluids. It is also important to have a random component due to objective and subjective errors. The reasons for objective errors include setting the equipment, the presence of electrical and acoustic noise, and subjective errors depend on the individual properties of the operator, his psychophysiological state, attention, etc.

*The method of IEMER* is based on pulsed electromagnetic radiation of rocks, which has a mechanic-electrical nature and is caused by deformation processes in the massif. In physical essence, the phenomenon is close to SA, but it differs by the electromagnetic carrier of information and other frequency and dynamic ranges, which makes it possible to apply it even when registering micro-fractures in the rock mass that result from changes in the stressed state of the massif.

The parameter to be recorded is the intensity of the magnetic component of the electromagnetic field averaged over the time interval, which is the cumulative result of global electromagnetic phenomena in the geospace and terrestrial atmosphere, technogenic processes, local radiation of the array in brittle fracture zones, and tribo-electric phenomena. At each point it is recommended to perform at least 6 measurements and averaging the result.

The technique assumes profiling along the workings with the location of the receiving antenna parallel to the direction of the profile. In the case of asymmetry of the conditions, measurements are made on two parallel profiles symmetrically disposed with respect to the production axis. The step along the profile is chosen in the range from 4 to 10 m. This option is used to map the zones of

increased stresses associated with the movement of the cleaning work front, as well as with the localization of zones of geological disturbances in the array. To assess the asymmetry of loads in the development, measurements are made on a series of parallel profiles, positioning them along the output contour, and the control profile in its central part.

Control of structural disturbance of the massif (cracking, stratification, etc.), weakening the strength of rocks, increasing the deformation of the massif, which leads to loss of stability of the massif, collapse of the roof, incursions, formation of fishweirs and segregations, increasing pressure on the support of excavations, etc. The problem is solved by the methods of ultrasound, EM and induced polarization (IP).

*Method of ultrasound.* It is based on the possibility of determining the Poisson's ratio, which is associated with the fracturing of rocks. For fracturing evaluation, the spectra of the received elastic waves are analyzed, which makes it possible to determine the frequency parameters of the medium. The range of frequency studies is limited by the acoustic properties of the rocks which study. With increasing frequency of ultrasonic oscillations, the accuracy of measurements increases, but the acoustic transparency of the material decreases. With increasing frequency, the wavelength becomes commensurate with the size of small inclusions and cracks. As a result, scattering and diffraction of elastic waves occurs, which can be detected by analyzing their spectral composition. At low frequencies, only large structural disturbances are detected.

*EM monitoring* of fracturing and heterogeneity of the massif is based on measuring the electrical resistance of a rock mass plot, which depends on the degree of fracturing of rocks and their heterogeneity. The method makes it possible to obtain information about the state of the massif at the outline, by profiling through pre-drilled holes (hole sounding), and without using them in the case of free access to the rock massif.

The electrical resistance of the minerals from which the rock is composed is significantly higher than the resistance of the saturating waters, so their total resistance is determined primarily by moisture, as well as porosity, fracturing or looseness of the massif. In this case, the moisture of the rocks and their fracturing most often change the

resistance in opposite directions: moisture reduces resistance, and fractures it increases.

The fractured porosity  $K_1$  of moisture-saturated rocks is determined by the formula:

$$K_1 = A_a \frac{\rho_3(\rho_1 - \rho_2)}{\rho_2(\rho_1 - \rho_3)},$$

where  $\rho_1$  - resistance of rocks without cracks;  $\rho_2$  - resistance of fractured rock;  $\rho_3$  - water resistance;  $A_a$  is a coefficient that takes into account the anisotropy (for isotropic rocks  $A_a = 1.5$ ).

Methodically recommended is a four-electrode symmetric measurement circuit, consisting of a power line, AB length, through which a low-frequency alternating current is passed, and a receiving line of length MN. To ensure the necessary uniformity of the electric field in the controlled area, the following relation should be satisfied:  $AB \geq 3 MN$ . The measurement step is chosen to be equal to the separation value of the receiving electrodes. The magnitude of the seeming specific resistivity for a given circuit is determined from the formula:

$$\rho_s = \frac{k_g \cdot \pi \cdot AM(AM + MN)}{MN} \cdot \frac{\Delta U}{I},$$

where  $k_g$  - the coefficient due to the geometry of the mass section;  $\Delta U$  - voltage drop in the receiving line;  $I$  - current in the supply line.

The method makes it possible to obtain information about the state of the contour part of the massif both by profiling and by hole sounding. The diameter of the hole for sounding is chosen in the range from 42 mm to 56 mm, and the length is not less than 3 m. The difference electrode on the probe should be of the order of 0.5 m. At the outcrops of the rock massif, the feeder line length is 1.5-12 m. To reduce the influence of metal structures, the profile is laid at the largest distance from them. Nearby electrical installations must be de-energized during the measurement period.

*The IP method.* The essence of the method is as follows: a section of the rock massif using feed electrodes is charged with current pulses of a rectangular shape and after breaking the supply circuit, a decrease in the induced polarization of the rocks is investigated, which is of a volumetric nature and is related to the electret and ferroelectric formation mechanisms. As an informative characteristic



of the structural disturbance of the medium with gas (radon, thoron, and carbon dioxide) and liquid filler, the relaxation parameter of the currents of the double electric layer, determined by the polarization capacitance and the specific electrical resistance of the medium, is used.

It should be remembered that with a significant moisture content of the rock massif and the presence of minerals with large electronic conductivity, the effect of IP is absent. As a rule, the intensity of fracturing is estimated through the value of the polarization capacitance, taking into account that the higher the fracture of the rock massif, the longer the decay of the IP.

Methodically, a system of observation points is broken on the measuring profile, the distance between which is calculated depending on the required detail of the studies. In the studied section of the rock massif, there is a feeding line AB of minimum size. In the center of the feeder line, three receiving electrodes are located symmetrically within the middle third, for example, at a distance  $(AB / 3 - 0.5 \text{ m})$  from the center. These electrodes are connected to two receiving lines (the central electrode is common for both lines). A current is passed through the supply line and the electrical component in the supply line and the signals from the MN output of the first and second receiving lines are alternately measured. Sequentially the sizes of a feeding line increase. At small depths of research, the length of the receiving line is kept, and measurements are made at each point by the probing method with two additional orthogonal receiving lines M1N1 and M2N2 with two feed line orientations that differ by some angle. The length of the supply line should be 1.5-2 times the depth of the maximum sounding of the array. Receiving lines are used in two sizes, which increase the resolution of the method necessary for the study of low-power objects. The measurements are carried out separately for each direction by installations with a size corresponding to the objects under study. When the structure of the rock massif is complicated by stratification or schistose, a study of the fracture should be carried out taking into account the anisotropy.

Interpretation of observations results using the IP method is based on a joint analysis of maps of geological material on the distribution of electrophysical properties in the rock massif and the results of

full-scale determinations of the parameters of the IP transient processes, applied to the approximation of bodies of regular geometric shape. When determining the position of structural disturbance or heterogeneity as diagnostic features, the parameters of the transient processes of the IP and the nature of their changes in the plot of the massif are considered for different sizes of the probing installations and orientation of the measuring lines relative to the elements of occurrence of the expected structural disturbance of the rock massif.

*The control of hidden fishweirs, segregations and hollowness* is performed by the method of shock wave diagnostics (vibro-acoustics VA), an alternative to which in the system of geomechanical monitoring is practically absent. The VA method is based on shock excitation, recording and analysis of the amplitude-frequency and temporal parameters of natural oscillations of the section of the roof (walls) of the development. The method makes it possible to detect hidden and visible segregations in the boundary plot of the massif and stratification in the roof at a depth of up to 3 m, and also perform diagnostics and forecasting of the roof state by monitoring the dynamics of the change in information parameters over time.

The basic characteristics of the method are the frequency and amplitude of the elastic vibrations of rock massif elements. The nature of the response of the controlled area of the massif to shock excitation is determined by resonant phenomena and damping of elastic vibrations in the medium. The main informative parameters of the method are the maximum amplitude of the oscillations, the duration of the oscillation process, and the spectral composition of the oscillations. The depth of the method is determined by the sensitivity of the receiver, the amplitude of the impact and the distance between the points of reception and excitation. The control base must be no less than the required depth of control.

When controlling segregations and fishweirs, the parameters characterizing the impact of the striker and the medium are, on the one hand, the amplitude and frequency of the roof vibrations, and on the other hand the acceleration of the impactor and the time of its collision. Inhomogeneities and disturbances in the near-well massif lead to an increase in the amplitude and duration of the oscillations, as well as the appearance of sharply pronounced resonances in the

spectrum. Criterion values, as a rule, are specified for specific mining and geological conditions.

Usually the geophone is installed so that the longitudinal axis is perpendicular to the surface of the object (with a deviation of not more than 200). Measurement should be made at least 3 seconds after the final installation of the geophone in a stationary state. The impact is applied at a distance of 0.4-0.6 m from the installation point of the receiver. If possible, the parallelism of the axis of the geophone and the direction of the impact should be observed. Measurements at one point produce 3-5 times. The measuring point is successively moved along the workpiece in steps of 1-3 m.

With a large deviation of the geophone from the perpendicular (more than 200), it is necessary to pay attention to the fact that at the time of impact does not occur its slip from the installation site. In this case, to obtain reliable results, the geophone tip should be installed in the unevenness (recesses) of the roof, and the number of measurements at one point should be increased to 4-7.

To reveal hidden segregations, a survey of the site in mutually perpendicular directions with a change in the base from 1 to 10 m is necessary.

The diagnostics of the state of the system "support - massif" and unclosed sections of the trunks is carried out by VA- and EMM methods. VA method is used to assess the state of concrete, reinforced concrete (including tubing), spattered concrete and multilayer fastenings of trunks, horizontal workings and their interfaces, and also to determine the influence of the perimeter barrel on the support. The EMM method is used to assess the stress-strain state, fracturing and humidity of the near-barrel massif, including in the "barrel-mining" coupling zones.

The informativeness of each of these methods with relatively to specific conditions and problems of diagnostics is determined a priori. This process includes statistical processing the array of data obtained on unconditionally anomalous and undisturbed areas (objects of research).

*Vibro-acoustic method* is the main and mandatory at all stages of the shaft survey.

When diagnosing the state of the support-massif system, one should take into account that defects in the support material reduce

its strength, which is manifested in a decrease in the amplitude, duration of the oscillations, and shift of the maximum of the spectral density toward low frequencies.

Reduction of the mechanical connection between contiguous elements, for example, concrete ring ruts, concrete and reinforced concrete blocks or tubing, causes a decrease in the amplitude of elastic oscillation, which is registered on the other side of the contact boundary of the elements relative to the site of the disturbance.

Increasing the pressure of the rock massif on the support proportionally reduces the amplitude of its natural oscillations in the frequency range of 0.06-0.6 kHz.

Performing measurements in the shaft is preceded by azimuthal breakdown of the circular profile (for vertical shafts) and from the vertical axis (for inclined shafts) to sectors in 600 with a shaft diameter of up to 5.5 m and 450 with a shaft diameter of more than 5.5 m. The following circular profiles are broken parallel to the given step, which is equal to the height of the tier (for vertical shafts), or 5 m (for inclined shafts). If a detailed survey of the anomalous section of the shaft is required, the latter is divided by a network of circular profiles with a sector of 300 and a step of 1-2 m.

To reveal hidden defects in the structure or loose mechanical connection of the support with the rock massif, the control base is chosen equal to 0.8-1.0 m. The measurements are performed approximately at the center of the sector, successively shifting the control base to one sector in a horizontal plane, then moving vertically to one step measurement. When assessing the depth of a crack on the surface of a support or determining the quality of a mechanical connection between individual structural elements, the control base is chosen to be symmetric with respect to a crack or visible interface of the elements. When examining plots with a regular spatial structure of timbering, for example, tubing, measurements are performed on each of the elements. The control step in this case is determined by the period of the structure.

When assessing the quality of VA concrete, the circular profiles are chosen so that they are intersected to with areas of corrosion that are observed visually, peeling and other anomalies. To build a detailed picture of the disturbance of near-surface layers of concrete, the control base and step can be reduced by 2.0-2.5 times.

When assessing the pressure of the circumshaft massif on the support, the measurement step is chosen equal to 1 m. Control of the strained-deformed state change of the near-shaft massif is conducted in dynamics. First, the value of the amplitude of the oscillation of the fastening is determined according to the selected profile scheme. The resulting picture is a conditionally initial reference of observations. Further, as the barrel is being used, the measurements are repeated at intervals of 2-3 times a year, during the off-season, depending on the geological conditions and the intensity of deformations in the controlled section of the shaft. According to the obtained data, circular diagrams of the pressure variation on the support in the selected sections of the trunk are constructed.

Evaluation of the quality of the sprayed concrete timbering is carried out by the method of longitudinal profiling on mutually perpendicular profiles with 1 m steps along and orthogonal to the axis of the shaft.

Criteria assessment of the state of the system "annular concrete support - massif" is determined by the thickness of the support, the properties of its material, and the type of equipment used.

*EM diagnostics* of loose sections of the shaft are performed in the following sequence:

- the sections in the middle part of the lithological homogeneous massif are selected and the altitude marks are determined;
- marking of drilling points in the direction of the sides of the horizon or the section axes (for inclined shafts);
- bore holes are drilled, followed by blowing compressed air into them (without rinsing).

In order to increase the reliability of the measurement, two steps are taken: when the probe is moved to the coal-face of the hole, and then when it is reversed. After the completion of the measurement cycle, the value of the measured parameter is directly compared to the forward and reverse run for each of the profile points. With a difference in the values, which exceeds the measurement error, they are repeated.

The EM method can be used: in the bottom-coal-face non-fastened part of the shaft to assess the asymmetry of the properties of the rock massif along the contour of the shaft; in excavations, which are adjacent to the shaft, to assess the zone of its influence; on the

sites of the absence of continuous attachment of inclined shafts to assess the stress-strain state of the near-shaft massif. In this case, the length of the supply line AB is 1.5-6 m. The optimum line size is in the range of 6-12 m.

*The EMM method* of shaft diagnostics is based on the registration of electromagnetic emission, which arises as a result of the formation of new surfaces during microdefections in the rock massif and is manifested in the form of the IEMER.

The method is used to isolate the areas of increased stresses in the massif by profiling along the axis of the shaft and to assess the asymmetry of stresses in the circumshaft massif. When profiling stretches of vertical shafts, measurements are made from the stand. Depending on the size of the monitored area and the degree of heterogeneity of the massif, the profile pitch should be from 5 to 20 m or be a multiple of the distance between tiers. In the presence of a directional antenna, radiation registration should be performed in three mutually perpendicular directions: along the axes "north-south", "east-west" and "top-bottom". In this case, the indication in the direction of "top-bottom" serves for estimating background noise not related to deformation processes in the massif.

When performing measurements on the bottom-fishweir non-secured massif, the following sequence of works is recommended: determination of the direction of zero azimuth; breakdown from the specified direction of the circular profile with an angular pitch of 30, 45 or 600; perform measurements on each of the angular directions at a distance of approximately 1 m from the surface of the contour with the orientation of the receiving antenna perpendicular to the radius of generation.

Areas of increased stress in the system "support - rock massif" are determined by the local increase in the level of electromagnetic radiation, provided there is no electromagnetic interference of anthropogenic nature. Results of studies in the borehole area are represented in the polar coordinate system in the form of a graph of the parameter change depending on the azimuth. According to the asymmetry of the radiation pattern, the nature of stress distribution is evaluated.

## Conclusions

Geomechanical monitoring of underground geotechnical systems should consist of three main directions:

1) the operational determination of the physical and mechanical properties of rocks on samples of an arbitrary shape, including directly on the site and in a massif of rocks;

2) control of the properties and condition of the near zone of the rock massif (up to 10 m from the outline), namely: the presence and size of fishweirs, segregations, cracks; parameters of anomalous stress zones, inelastic deformations and reference pressure; vector-force characteristics of the interaction in the "support-massif" system; condition of arch, anchor, reinforced concrete and concrete timberlines and a fixed massif; quality of plugging, etc.;

3) control of the far zone of the rock massif (10-200 m from the workings contour), in particular, determination of the presence and parameters of karsts, tectonic disturbances, man-made voids, structural inhomogeneities, forecast of gas dynamic phenomena and rock impacts.

Based on fundamental and applied scientific research, scientific foundations, methodology and means of control of the geomechanical state of the underground geotechnical system of mines and mines have been developed. The development is systematized to the level of monitoring and implemented at the mining enterprises of Ukraine.

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# THEORY AND PRACTICE OF CONTROLLING GEOMECHANICAL PROCESSES IN THE ROCKS AROUND WORKINGS WITH ROOF BOLTING SYSTEMS

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**Abstract.** In recent years roof bolting is getting widespread use the coal and mining industry. This increases stability of mine workings and allows increasing the rate of driving preparatory works while saving funds for their maintenance.

Depending on the structure of the roof rocks of the shape of the working section there are distinguished five main conditions of its use:

- semi-stabled layered rocks of the direct or false roof are suspended by anchors to the main roof;
- various rock layers being fastened by anchors, form a composite beam protecting the roof from collapse (the case of "stitching");
- at each anchor the pieces of fractured rock are pressed against each other forming a strong block. In this case along the perimeter of the working there is built a ring of such blocks with the help of an anchor that supports each other, as in a stone arch;
- in the fractured homogeneous mining rock of high thickness the roof bolting prevents the working from intruding individual pieces when natural cracks are opened under the impact of rock pressure;
- when driving in the highly fractured layered massif with the contour that closely coincides with the natural equilibrium, the roof bolting prevents the working from corrosion and rash of rocks.

**The subject of the study** is controlling the state of the massif around a preparatory working, its geomechanical state.

**The methodology of the study** consists in assessing mining and geological factors affecting the development of coal seams, mining and



technical factors affecting the use of the roof bolting and parameters of the deformation processes in mine workings.

**The purpose of the study** consists in improving the working state by using a combined anchor-frame support or roof bolting to support mine workings.

## **Introduction**

**The scientific research topicality.** At present at the mines of the Karaganda coal basin (the Republic of Kazakhstan) up to 30% of mine workings are supported with a metal support that is practically enclosing and serves to retain the exfoliating mass of rocks. However, even with careful backfilling, the frame support used in the development workings cannot always prevent the deflection of the roof rocks.

**The objective of the study.** There have been established regularities of changing the stress-strain state of coal in rock massifs depending on the main mining-geological and mining factors that will allow, under specific operating conditions, establishing optimal anchoring parameters to improve stability of preparatory mine workings.

### **Research tasks consist in the following:**

- 1) to analyze the factors influencing the geomechanical state of the workings at the mines of the Karaganda coal basin;
- 2) to assess mining and technological conditions for using roof bolting of mine workings at coal mines;
- 3) to carry out studies of changing and distributing stresses in the roof, soil and sides of the working;
- 4) to identify regularities of changing the stress-strain state of the coal of enclosing rock massifs depending on mining and geological factors that will allow, in specific operating conditions, establishing rational parameters for securing side rocks to improve stability of preparatory mine workings;
- 5) to determine the impact of mining-geological and mining factors on forming zones of inelastic deformations in enclosing rocks and to establish rational parameters for using the roof bolting in preparatory mine workings.

## **1. Using roof bolting at the mines of the Karaganda coal basin**

Selecting the type of the roof bolting depends on a number of geological, mining and technical-economic factors. These factors include: the structure and physical-and-mechanical characteristics of the rocks, the degree of disturbance and water content of the deposits, the depth of development, the spacing between the seams, the configuration, the purpose and lifetime of the mine, its operating conditions, consumption, cost, scarcity and strength properties of the roof bolting materials, and manufacturability of the support.

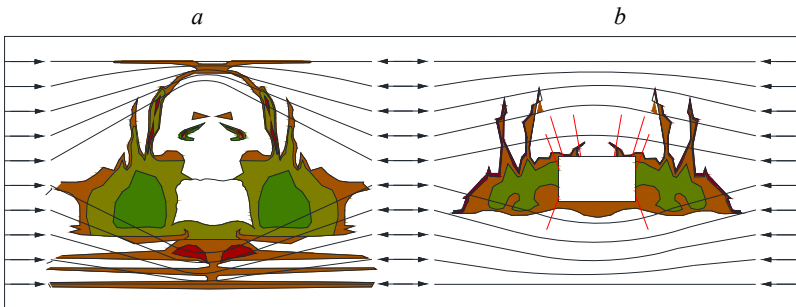
Roof bolting presents a system of anchor rods (metal, reinforced concrete, wooden or polymeric materials) inserted into perforated drilled holes (boreholes) and fixed in various ways in the rock strata. To the anchors there are suspended support plates, caps with puffs or metal nets. Anchors, working in stretching, keep the anchored rocks from stratification, displacement and destruction. In rocks with the layered structure, the layers of the unstable direct roof are either attached (stitched) by anchors to the stable main roof, or individual rock layers are anchored to one monolithic slab that is capable of absorbing the load from overlying rocks. In rocks with the unbroken structure, anchors fixed outside the arch of natural collapse, resist the tensile forces in the rocks of the arch.

The constant increase of the development depth is accompanied by the complication of mining conditions and increasing the total length of the mine workings network. At the development depths existing in the Karaganda basin, present day supports cannot achieve maintenance-free workings.

The rocks of the roof in the workings are of low strength, and when exposed more than one meter, they collapse. At this the rocks are prone to soaking and heaving. In the tectonic respect, the layers under development are complex. Wide introduction of technological schemes for the non-pillar mining of seams led to high costs of supporting the workings, the necessity of skin-to-skin workings driving with the worked-out space.

The transition to the roof bolting provides an entirely different geomechanical state of workings, since there is no stratification of rocks in the roof and the reference pressure on the soil is produced

due to local stresses formed by the frame support which causes the soil rocks to heave in the working (see Figure 1).



**Fig. 1.** Deformation pictures with the arch supporting (a) and the roof bolting (b) of the working

Particularly acute is the issue of increasing stability of in-seam winning workings supported in the conditions of weak enclosing rocks in the zone of intensive impact of stoping. One of the rational ways to improve the state of the workings and to save material resources is the use of a combined anchor-frame support or the roof bolting to support mine workings. Selecting the type of support is greatly affected by geomechanical processes near the mine workings. In this connection the development of technology for making excavations based on the determination of the stress-strain state (SSS) of the massif and its impact on the parameters of supporting and subsequent maintenance is an important scientific and technical task of mining.

When driving workings and developing coal seams, due to the disturbance in the balance of rocks and redistribution of natural stresses, in mines there arises mining pressure and various geological phenomena occur in the deformations, destruction, movement and displacement of their various masses. The development of rock pressure that arises as a result of the interaction of coal-bearing rocks with mine workings is greatly influenced by geological and mining-technical factors. The main geological factors include the geological structure, lithological types of rocks, their occurrence conditions, crack and fracture disruption, physical and mechanical properties, gas content and others which determine the complexity of the

engineering geological conditions of the mining fields. Among the mining-technical factors there should be noted the size, the shape and the location of the capital, preparatory and cleaning workings, the depth of their laying and the way of penetration, the development systems and the types of supports used.

Below there is considered the roof bolting use at the mines of the Karaganda coal field to secure the rods in preparatory, capital and auxiliary workings in order to ensure the mining operations safety.

## **2. Assessment of mining-and-technological conditions of using the roof bolting for mine workings in coal mines**

In order to ensure high rates of driving and maintenance-free supporting of development workings, especially in zones of increased rock pressure that cause a significant decrease of in-seam workings, additional work is underway to erect reinforcement pads and to strengthen the coal-bearing massif. These measures are labor-intensive (0.2 person-sh/m), require additional material costs (15-25 thousand tenges/m) and reduce the rate of reserves development by 30-40%.

One of the rational ways to improve the state of workings and to save material resources is using the roof bolting. The volume of implementing the roof bolting at the mines of the Karaganda coal basin is up to 25% in the pure form and 45% with a mixed support.

In connection with spreading of the roof bolting technology, the costs for supporting, the volume of material transportation, the safety of work, the efficiency of using the working sections, the end operations on mating lavas with adjoining workings were simplified.

When using the roof bolting, the cost saving is 250-350 US dollars per 1 m of the working compared to the metal arch support. The cost for driving a single meter of the working with the cross-sectional area of 14.4 m<sup>2</sup> make with the roof bolting 6, with the roof bolting in combination with a frame support 8-12, with a metal arch 60-80 thousand tenges. The dynamics of using the roof bolting technology at the mines of the CD ArcelorMittal Temirtau JSC is growing (see Table 1).

**Table 1****Basic technical-economic indicators at the mines of the Karaganda coal basin**

Indicators	Units of measurement	Operation years				
		2013	2014	2015	2016	2017
Driving mine workings supported with a support including	K	51.3	54	60.5	60	60
- arch	Km	35.4	34	29.8	26	25
- combined	Km	11.8	13.5	20	18	15
- roof bolting	Km	4.1	6.5	10.7	16	20
- workings volume for a lava	km/lava	4	4.5	5	5	5
Cost of driving mine workings						
- arch	tenges/m	80	60	65	100	100
- combined	tenges/m	60	60	65	65	80
- roof bolting	tenges/m	50	55	60	80	80
Driving specific volume	m/1000 t	4.2	4.9	5.4	5.4	5.3

Table 2 shows the factors caused by mining-and-geological features of developing seams in the countries with developed coal industry in comparison with the conditions of coal seams occurrence in the Karaganda basin. Geomechanical conditions for supporting winning workings in the Karaganda basin at great depths are complicated by the increased complexity due to the low strength of the rock beds enclosing the coal seams, especially soils that, even at low concentrations of rock pressure, are prone to intense heaving. In the coal basin there prevail medium-resistant roofs (50% of all the seams), there are 25% of semi-stable), 20% of unstable, and only 5% of stable seams.

**Table 2**

**Factors caused mining-and-geological features of developing seams**

Mining-and-geological parameters	Coal mining country				
	Germany	Great Britain	Australia	USA	Kazakhstan, Karaganda basin
Depth of mining operations, m	1000-1200	600-800	260-280	250-360	550-820
Geostatic rock pressure, MPa	20-25	10-15	5,0-6,5	7-9	10-15
Maximum horizontal rock pressure, MPa	20-25	20-22,5	10-13	15-18	10-15
Mined thickness of coal seams, m	2,0	2,5	3,1	2,2	1.0-8.5/ average 2.28
Seam inclination angle, degrees	mainly 5-10, no more than 15	no more than 5	no more than 5	no more than 5	7-25
Characteristics of the roof rocks, uniaxial compression strength, MPa	from thin argillites to sandstones, 35-80	from argillites to sandstones, 35-70	from argillites to sandstones, partly coal, 5-80	from argillites to sandstones, and limestone, 10-80	from argillites to sandstones, 10 - 80
Characteristics of the soil rocks, uniaxial compression strength, MPa	thin argillites, coal interlayers, 40-45	argillites partly crossed by roots, 40-45	argillites partly crossed by roots, 30-40	argillites, partly sandstones, 30-40	argillites, 20-30

Table 3 presents the factors and parameters associated with the technology of driving mine workings and stoping in coal-mining countries. A comparative assessment of mining-technical parameters indicates that the following trends are developing: the straight-through and recirculation scheme of preparation and ventilation of mining units is used with the advantage of the latter; rigid and pliable pillars are left (at the AMT it is planned to study the feasibility of this trend); the cross section shape: arched in capital, auxiliary and winning workings, rectangular with the prevalence of the latter; the working dimensions are growing, especially in width up to 5.5-6.5 m.

**Table 3**

**Factors and parameters associated with the technology of driving mine workings and stoping**

Technological parameters	Germany		Great Britain	Australia	USA	Karaganda basin
	Scheme of preparing and ventilating mining units	straight through	reverse through	straight through	straight through	straight through
Leaving pillars, m	-		rigid, 80-120	pliable, 10-20	pliable, 10-30	acc. to special project
Working cross section shape	arch	rectangular	rectangular	rectangular	rectangular	arch, rectangular
Working dimensions, m:						
height	4.1-4.8	2.5-4.1	2.5-5.1	2.1-4.2	2.1-3.5	3.5-3.7
width	6.1-7.5	5.1-5.8	4.5-6.1	5.1-6.2	4.7-6.1	5.5-6.5

Table 4 presents the technological factors of using the roof bolting in coal-mining countries. In the Karaganda coal mines of

Central Kazakhstan (the Karaganda coal basin) mine workings have both arched and rectangular cross-sections.) The evaluation of the technological factors in the operation of the roof bolting systems shows that the effective and reliable supporting of workings in the roof of weak fractured rocks, at large depths and in various zones of stowing impact can be provided with the use of steel-polymer anchors.

**Table 4**

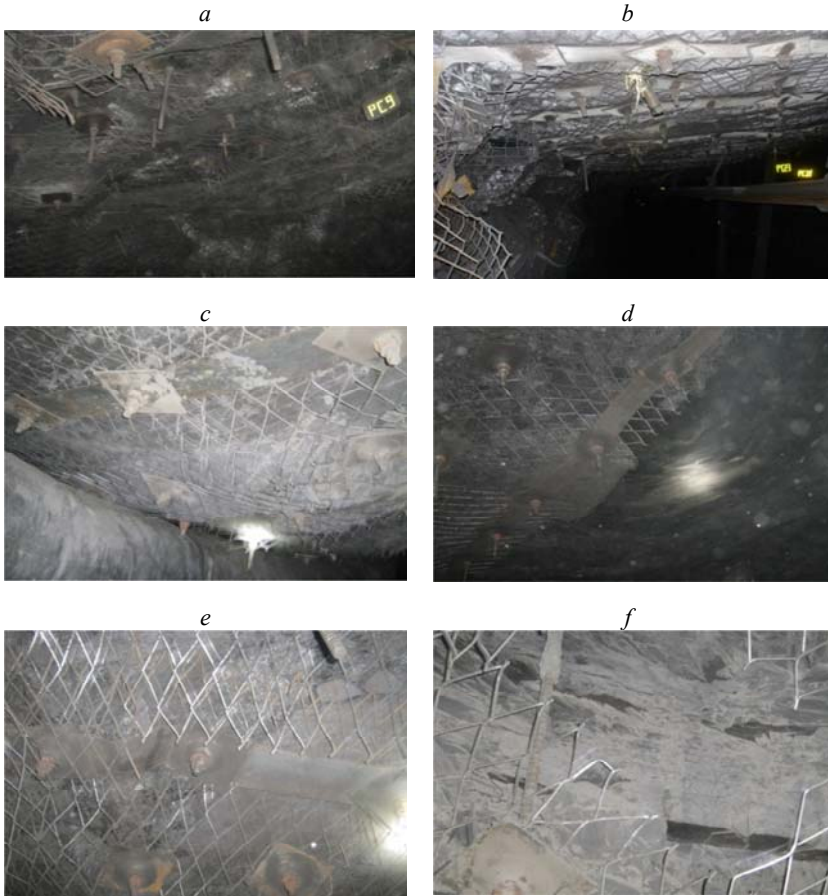
**Technological factors of using the roof bolting in coal mining countries**

Technological factors	Germany		Great Britain	Australia	USA	Karaganda basin
Anchor length on the coal, m	2.5-3.05		2.2	1.9-2.1	1.6-1.9	2.4
Anchor length on the rock, m	2.1-2.4		2.1-2.4	1.5-2.4	2.1-2.4	2.4 (2.9)
Design bearing capacity, kN (depending on the material)	360-540		310	220-320	150-220	250
Anchors mounting density, anchor/m <sup>2</sup> :						
roof	1-2	1.4-2.2	1.1-3.0	0.5-0.7	0.4-0.7	1.0-1.5
sides	0.6-1.9	0.5-1.2	0.3-0.9	0.11-0.23	0.09-0.15	0.6-0.7

Steel-polymer anchors are fixed along the entire length of the hole with fast-hardening resins, with the bearing capacity of 250-300 kN and the length of 2-3 m.

The main types of rock deformation in the development workings are collapse, precipitation, extrusion, cupping, squeezing coal, coal emissions and punching of soil rocks. The set of unfavorable factors (the depth of the location of the mine, the performance of mining in the zone of increased rock pressure, the increase in the concentration of works) affecting the state of the contour massif leads to deterioration of the conditions for supporting a working and practical absence of the possibility of mining operations (see Figure 2).





*a*-unstable roof condition;

*b*-panorama of deformations of production angles;

*c* - a dome in the roof of the mine;

*d*-sagging roof workings;

*e* - is the deformation of the anchor rod;

*f*-cracks in the roof of the mine and dome

**Fig. 2.** Manifestations of rock pressure when using anchorage of mine workings

Development workings of the main purpose have the rectangular cross-section, the width of 5 m and the height of 3.0-3.3 m and the arch shape (height and width 4.7x3.3 m). With the combined support (conveyor drifts and headways) there are mounted 12-14 steel-polymer roofing in the workings supported by the roof bolting with steps of 0.5 between rows (through one under the strip) and 0.65 m in rows with 4-6 fiberglass side anchors, under strips (channel 10, strip 150x5 mm) with a wire mesh of the MM type. Roof anchors of the AMB types with the length of 2.4 m, and lateral type AM of 1.6 m set at an angle of 35-40° to the bedding along the 1x1 m grid. For the full filling of the hole there are used 4 ampoules of the AMK-M type. In the arch shape of the workings there is used a combined support consisting of a metal arch support made of SVP 27 arches with the step of 0.5 m and an anchor support through 0.5 m in the number of 9 roofing and 2 lateral (1.8 m long) anchors or without them and with the metal mesh rundown.

The strength of lateral and roofing rocks varies from 20 to 40 MPa. The rate of driving is, when using the roof bolting: in pure form of field workings 100-140 m/month, in-seam 130-200 m/month; on coal in non-dangerous faces 90-120 m/month and is outburst-dangerous faces 120-180 m/month in mixed faces. The rate of driving when using a combined support is 60-100 m/month, in-seam 140-200 m/month. For split furnaces (dimensions 6.5 m wide, 2.5 m high), the driving rate is 50-80 m/month. The service life is 3-4 years.

The analysis and generalization of the state of mining preparatory works and survey of mines in the basin showed that at the stage of driving in about 25-30% of them there occur dangerous deformations and loss of stability of rock outcrops including 40% of them outside the zone of stopping impact and 60% in the zone of impact. The loss of stability of rock outcrops leads to reduction in the rate of driving by 40-45% and increasing the consumption of fixing materials. In addition, 35-40% of accidents in mining operations are due to the loss of stability of rock outcrops and the collapse of the roof rocks and the sides of workings.

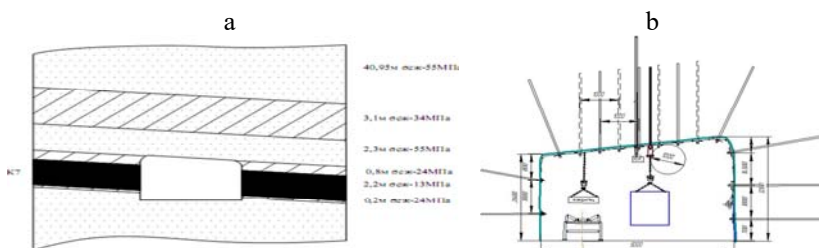
### 3. Assessment of the deformation processes parameters in mine workings with the roof bolting

The experience of exploitation of deep mines shows that one of the problems that needs to be solved is the problem of ensuring stability of mine workings. The analysis of the mine workings state of the Karaganda coal basin with significant depth of mining shows that more than 20% of their total length is in unsatisfactory condition.

In order to determine the optimum scope of using the roof bolting and to establish the regularities of deforming the near-boundary massif near the workings fixed by the roof bolting at the mines of the Coal Department, it was planned to install observation stations (including reference stations).

Studying the deformation processes parameters in operational workings is represented by the example of the results of field observations of the roof, sides and soil rock displacements of the conveyor intermediate drift 48κ<sub>7-3</sub> at the Kostenko mine.

The conveyor intermediate drift 48κ<sub>7-3</sub> (Figure 3, Table 5) is driven along κ<sub>7</sub> seam with the total thickness of 1.72 m, with the seam inclination angle 3-7°.



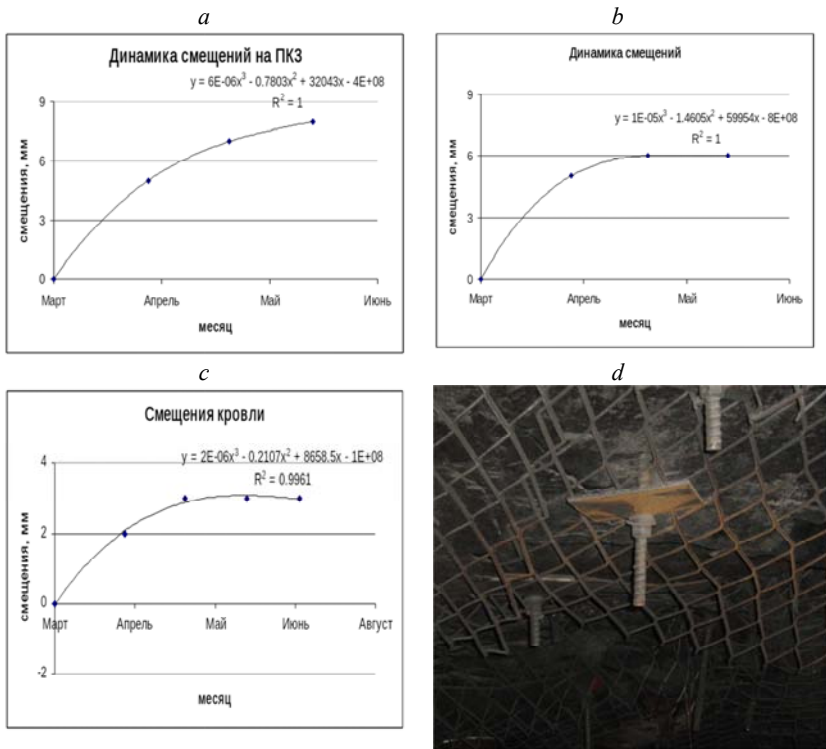
**Fig. 3.** Position of κ<sub>7</sub> seam in cross section (a) relative to the elements of occurrence and the passport of supporting conveyor intermediate drift 48κ<sub>7-3</sub> at the Kostenko mine in operation (b)

**Table 5**  
**Characteristics of the conveyor intermediate drift 48κ<sub>7-3</sub> at the Kostenko mine**

Working name	Length, m	Width, m	Height, m	Cross section area, m <sup>2</sup>	Support type	Anchors density, anch/m
Conveyor intermediate drift 48κ <sub>7-3</sub>	970	5.0	3.0; 3.5	16.2	Roof bolting	9

Within the first month there were observed active deformations with horizontal displacements from the left and right sides up to 5 mm (Figure 4, a, b). In the following months the intensity of displacements decreased. The obtained increased values of the displacement of the left flank can be explained by the fact that the working is driven along the coal bed which strength coefficient is 1.5 on the scale of prof. M.M. Protodyakonov.

Figure 4, c, d illustrates the dynamics of changing displacements of the roof rocks (c - instrumentally, d - visually).



**Fig. 4.** Dynamics of changing horizontal displacements of the contour massif rocks from the left (a) and right (b) sides of the working and the roof rocks (c, d)

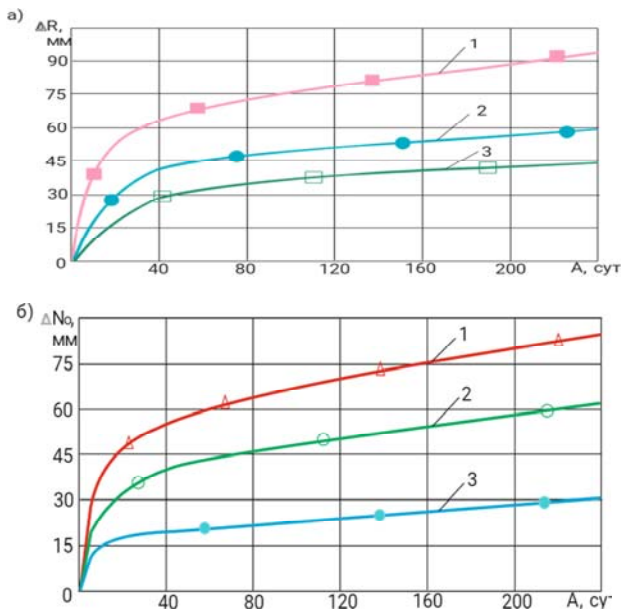
Active deformations of the roof rocks are observed within the first month. The total displacements within the entire observation period did not exceed 3.0 mm. Such displacements in the case of

weak rocks of the immediate roof are explained by the effective work of the roof bolting.

The presence of medium strength rocks in the immediate soil of the reservoir explains low activity of displacements.

Systematization of the results of measurements for winning workings depending on the main mining and technological factors, made it possible to establish the following regularities.

The performed measurements of the parameters of the deformation processes in the operational workings with various types of supporting made it possible to determine the nature of their stability including those in the zones of mining operations impact. At this, deformations of the outline contours with the combined roof bolting-frame support are 4-5 times smaller than with the pliable metal arch support (see Figure 5).

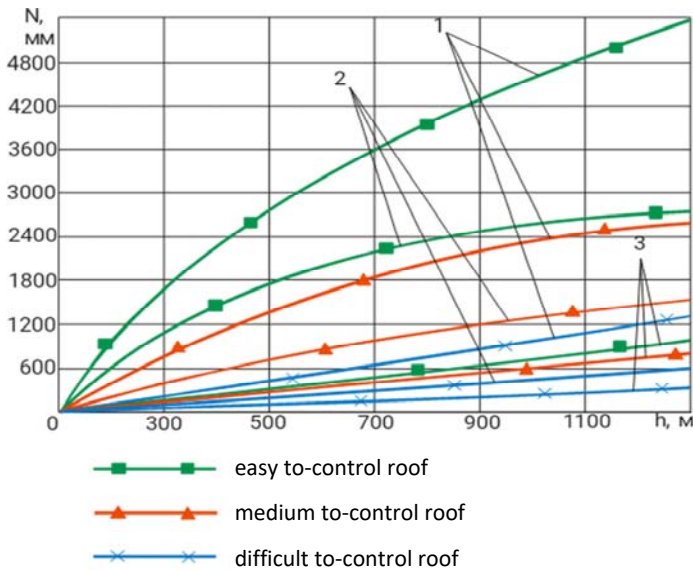


a) approaching the sides on the sections of the coal seam  $\Delta R$ ; b) lowering of the roof  $\Delta N_0$ ; with the support: 1 - frame; 2 - roof bolting; 3 - roof bolting-frame.

**Fig. 5.** Deformation of the rock massif around the preparatory working in the zone of stoping impact depending on the period of their supporting

There have been established the empirical dependencies of curve 2 (for the roof bolting) according to the presented plot (Figure 5):  
 a)  $\Delta R(T) = 15,165A^{0,294}$ , the correlation coefficient  $r = 0,854$ ;  
 b)  $\Delta N_t(T) = 1634A^{0,291}$ , the correlation coefficient  $r = 0,858$ .

Rock displacements also grow intensely in mining workings depending on the depth of the development, with varying controllability in the direction from difficult- to easy-to-control roofs (Figure 6).



1 – arch; 2 – roof bolting; 3 – combined

**Fig. 6.** Dependence of the rock deformations (N, mm) in mine workings on the depth of mining operations (h, m)

As a result of field observations there has been established the following:

- the total displacement of the roof when using the roof bolting does not exceed 30 mm;
- the adopted length of anchors in the roof (2.4 m) and installed in the sides (1.6-1.8 m) of the workings provide the needed stability;
- the total horizontal displacement of the rocks of the working sides does not exceed 100 mm which is permissible when using the

roof bolting;

- there have been obtained the values of the displacements in the zone of stoping impact (up to 2.5 m) which will require the advance installation of rope or cable anchors.

The established regularities of changing the stress-strain state of coal in rock massifs (displacements, stresses, fracture zones) depending on the main mining-geological and mining factors will allow, under specific operating conditions, establishing the optimum anchoring parameters to improve stability of preparatory mine workings.

Generalizing and processing the results of instrumental observations of rock convergence and deformation of the support elements by statistical methods make it possible to obtain an empirical database that can be considered as the basis for forming and correcting the analytical methods of justifying the parameters of workings supporting.

#### **4. Stress-strain state of the enclosing rocks around the mining working with the roof bolting depending on the seam inclination angle and the depth of anchoring the contour massif**

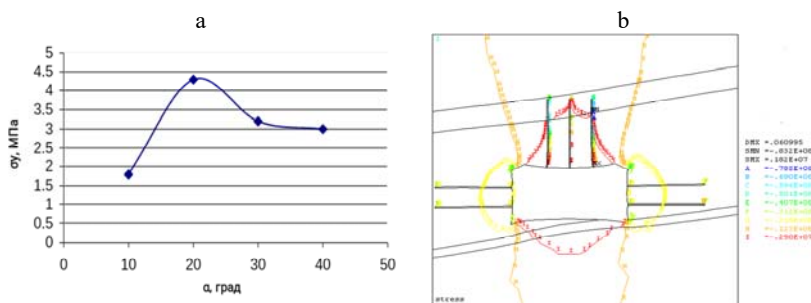
The weakest link in resolving the issues of increasing the efficiency of using the advanced technology of the roof bolting is insufficient knowledge of geomechanical processes near the mine workings [1, 2].

The conditions for supporting workings with various types of support in the zone of stoping impact have been studied using the example of the conveyor intermediate drift 48K<sub>7-3</sub> at the Kostenko mine of the Karaganda coal basin. The mined thickness of the K<sub>10</sub> layer on the west wing of the mine is 3.7...4.0 m. The immediate roof varies in the strike from 3 m to 7 m and is represented by argillites. The main roof is composed of weakly fractured sandstones with the thickness of 24-32 m. Figure 1 shows the state of the studied working in the zone subject to the stoping impact.

The working is characterized by the following changes in the condition of the support: deformation of the cap and its rush (Figure 7, a) along the lines of runs - 60%; deformation of composite columns (Figure 7, b) in the vertical plane - 1.5%; deviation of

friction racks from the vertical position - 70%.

To determine the conditional zones of inelastic deformations propagation there has been used the approach permitting to determine the stress-strain state at the considered point of the anthropogenic space, and then to establish the time to the failure and to estimate stability of the rock outcrop for the subsequent adoption of technological measures. For geomechanical interpretation of the modeling results, there has been considered the conveyor drift 64к<sub>10</sub>-3 with the cross section of 16.2 m<sup>2</sup> of the κ<sub>10</sub> seam of the Abayskaya mine at the depth of 630-640 m.



a – the nature of changing; b – the diagram (at  $\alpha=10^0$ )

**Fig. 7.** Distribution of maximum longitudinal stresses in the side rocks surrounding the working

In the presented studies the analytical modeling is performed using the numerical finite element method. The simulation has been carried out for the conditions of seam production of the seam conveyor working of the κ<sub>10</sub> seam of the Abayskaya mine of the ArcelorMittal Temirtau JSC with the depth of development of 400 m and the seam thickness of 3.8 m. There is considered the stress-strain state of the massif around the winning working. In the software complex ANSYS there has been built a model of the enclosing rocks massif that corresponds to the conditions of seam κ<sub>10</sub> bedding.

The values of maximum stresses in the side rocks surrounding the working for various cross section shapes of the workings has been studied. For example, the rectangular shape is shown in Figure 7.

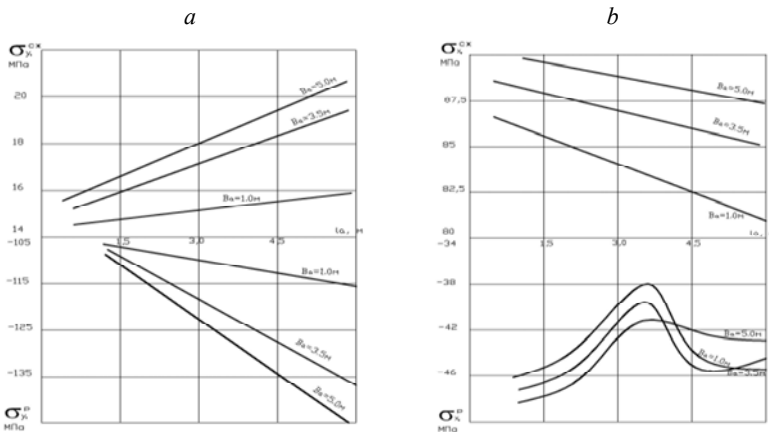
The carried out studies made it possible to draw a conclusion about the preference of using the rectangular section shape of the



winning workings with the roof bolting of the enclosing rocks for the development conditions of the  $\kappa_{10}$  seam of the Abayskaya mine of the CD ArcelorMittal Temirtau JSC.

There have also been carried out studies of the stress-strain state of the enclosing rocks depending on the thickness of the layer of easily collapsing rocks with different lengths of anchoring. The studies have been carried out with the example of producing the rectangular cross-sectional shape with the following parameters of the design scheme: the angle of the seam inclination is  $15^\circ$ , its thickness is 3.8 m; the development depth is 400 m; the working cross section is  $15.5 \text{ m}^2$ ; the diameter of the anchor is 0.022 m.

The nature of the change and distribution of stresses in the roof, soil and sides of the development is studied. With the layer of easily falling rocks from 1.0 to 6.0 m and the anchor length of 2.4 to 5.0 m, the following stress changes occur around the working. The maximum and minimum normal stresses with increasing the length of the anchor (from 1.5 to 6 m) and increasing the thickness of the layer of easily collapsing rocks (for example, built of argillite) from 1 to 6 m grow in the proportional linear relationship (Figure 8, a).



*a* - normal; *b*- longitudinal

**Fig. 8.** Dynamics of changing stresses in contour rocks of the preparatory working depending on the anchoring length and the rock layer thickness

## **5. Changing the strain state of the contour massif around the winning workings depending on the mining-and-technological factors impact**

In recent years there becomes widely used in coal and mining industries the roof bolting which increases stability of mine workings and makes it possible to increase the rate of driving development workings while saving funds for their supporting.

Manifestations of rock pressure have been studied with the establishment of the degree of the technological factors impact with the use of the finite element method [1, 2]. We consider the case of plane deformation with establishing stresses on the side walls, the roof and the soil of the working [3].

In the mining working with increasing controllability of the enclosing rocks, stresses in the massif grow linearly (Figure 9, a). Moreover, vertical stresses ( $\sigma_y$ ) grow insignificantly when the arch support and with the roof bolting and are close in magnitude.

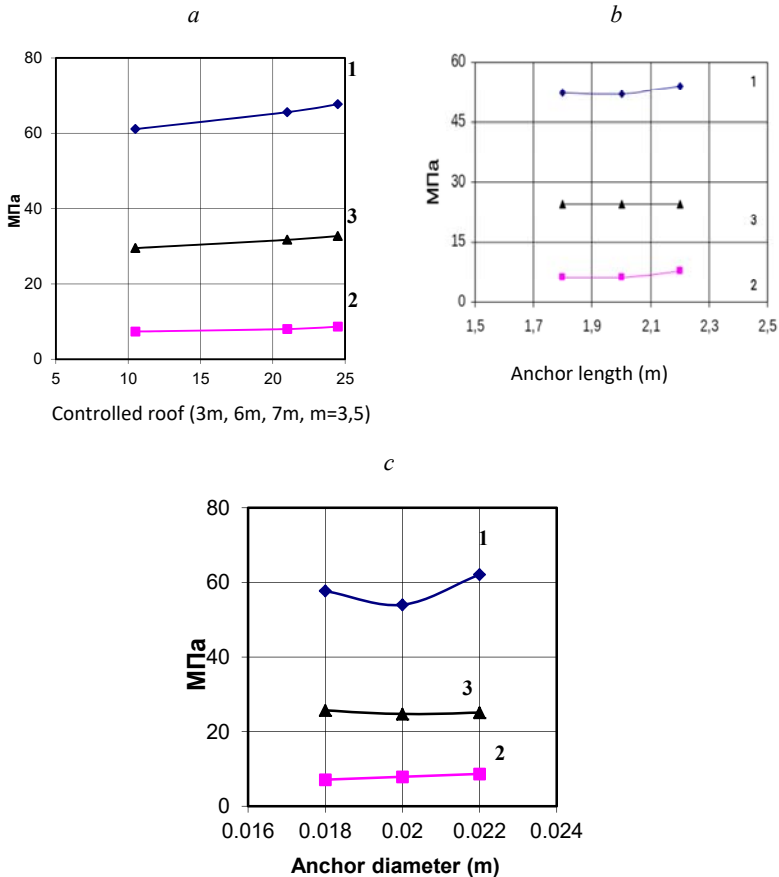
It has been established that with increasing the length of an anchor (from 1.8 to 2.4 m) and its diameter (0.02-0.0224 m) longitudinal stresses are more significant (55-60 Pa) with the tendency of increasing. The tangential stresses are practically unchanged (25 Pa) in the considered range, and the normal stresses increase insignificantly in linear dependence (from 5 to 10 Pa) – see Figure 9, b and c.

The carried out studies made it possible to establish the degree of the technological development factors impact on the efficiency of using the metal arch support and the roof bolting of winning workings.

The revealed regularities of changing the stress-strain state of coal in rock massifs (displacements, stresses, fracture zones) depending on the main mining-geological and mining factors will allow, under specific operating conditions, establishing the parameters of supporting for increasing stability of development mine workings.

There have been also studied manifestations of rock pressure with establishing the degree of the technological factors impact using the finite element method. The technological scheme of stoping with recurrent ventilation has been modeled for the conditions of the  $\kappa_{10}$  seam of the Kostenko mine at the 200 m length of the lava its

passage using the roof bolting. Changing stresses of the rock massif have been studied depending on the angle of the anchors inclination in the roof at design parameters: the length of the anchor is 2.4 m; its diameter is 0.022 m; the working cross section is 17.5 m<sup>2</sup>; the depth of development is  $Y^H=8,7138$  MPa. Table 6 shows the calculated values of maximum stresses and the parameters of the maximum displacement module.



1 – stress on x; 2 – stress on y; 3 – tangential stress

**Fig. 9.** The effect of the roof rocks controllability on the values of stresses arising around the working contour supported by the metal-arch support (a) and the roof bolting (b) with changing its length (c) and the diameter (d) of the anchor rod

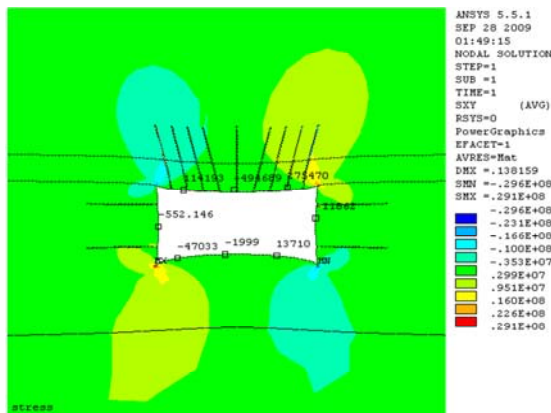
**Table 6**

**Maximum normal and tangential stress values**

$\alpha, \beta$ (deg.)	$\sigma_x$ (MPa)	$\sigma_y$ (MPa)	$\tau_{xy}$ (MPa)	$u_x$ (m)	$u_y$ (m)
30	93.6	19.7	54.1	0.0028	0.1382
45	93.4	28.2	46.5	0.0028	0.1382
60	93.4	26.1	30.6	0.0028	0.1382
75	93.4	9.05	29.1	0.0028	0.1382
90	93.4	5.73	29.1	0.0028	0.1382

From Table 6 it follows that with changing the angle of the anchors inclination, the normal stresses along the y axis and tangential stresses change. From the study carried out, the optimal angle of the anchors in the roof is the angle  $\alpha = \beta = 90^0$  (Figure 10).

The revealed regularities of changing the stress-strain state of the coal in the enclosing rock massifs depending on mining and geological factors, allow setting, in specific operating conditions, the rational parameters of supporting side rocks to improve stability of development mine workings.



**Fig. 10.** Distribution of stresses  $\tau_{xy}$  at  $\alpha=\beta=90^0$

## **6. Mining-geological and mining-technical factors impact on forming zones of inelastic deformations in enclosing rocks around workings**

To carry out the study, there have been selected the development conditions with assessing the impact of the following factors: the shape of the cross section of the development working; the type of support: metal arch, the roof bolting and combined; supporting conditions: in stability: unstable (fractured argillites); medium stability (argillites); resistant (siltstones); in controllability: easy-to-control; medium controllability; difficult-to-control; the development depth and the angle of inclination.

In the process of the studies there has been determined the stress-strain state around the mining: the roof, the soil and the sides; zones of fibrillation (fissuring); stresses (compression, tension and tangents).

There has been established the mining-geological and mining-technical factors impact on forming zones of inelastic deformations in enclosing rocks around workings.

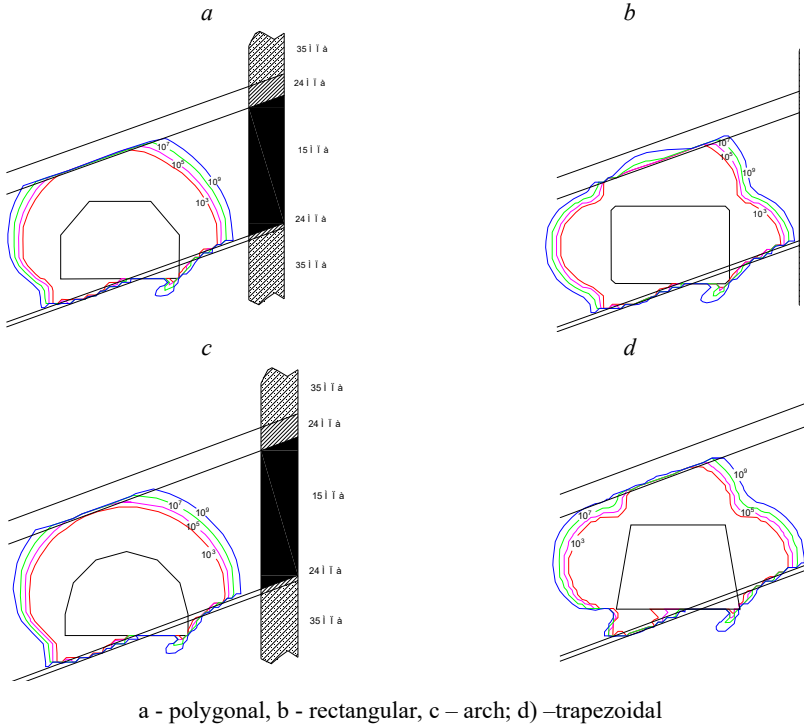
To determine the conditional zones of inelastic deformations, the program "Calculation of stress and durability" based on the theory of elasticity and kinetic strength of solids [4], has been used that allows determining the SSS at the considered point of the anthropogenic space, establishing the object durability and assessing stability of the rock outcrop. For geomechanical interpretation of the modeling results, there has been considered the conveyor drift 45 $\kappa_{10-3}$  with the cross section of 17.8 m<sup>2</sup> of the  $\kappa_{10}$  seam at the Kostenko mine at the depth of 630-640 m.

Figure 11 shows the isolines of the maximum tangential stresses arising in the side enclosing rocks with the arched, polygonal, trapezoidal and rectangular shapes of the working cross section.

The working shape has a significant effect on the pattern of the maximum tangential stresses  $\tau_{\kappa}$  distribution. The maximum concentrations ( $\tau_{\kappa} = 25$  MPa) for the arched and polygonal supports are located on the soil including the areas of contiguity to the side walls of the working; for rectangular shape in the same place and in the roof, and with the growth of  $\tau_{\kappa}$  to 35 MPa at the side walls of the working. Studies show that the more acceptable for the conditions

under consideration is the arched shape of the cross section of the working.

Changing the angle of inclination in the range from 0 to 30° does not lead to a significant changing of the loading pattern (for the example there is accepted the arch shape of the working support).



**Fig. 11.** Isolines of maximum tangential stresses arising in the enclosing rocks for various shapes of the working cross section

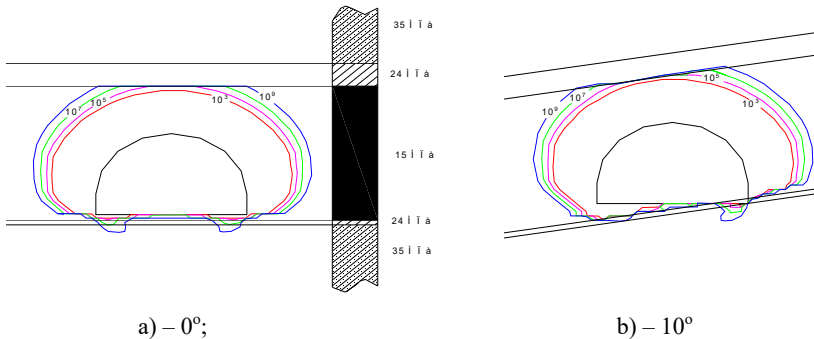
The zone of cracking of the enclosing rocks is maximal for horizontally lying layers and appears at the distance of 1.7...1.8 m and approaches the working section by 0.05 m with the growth of the seam inclination angle by 10° (Figure 12).

The assessment of the degree of the rock surface controllability impact on the rock massif deformability shows that for easy- and medium-to-control rocks in the roof, soil, and space of the side walls,

the fracture planes are formed at the distance of 2.5-2.9 m from the working contour, and with difficult-to-control rocks in the roof directly above the working.

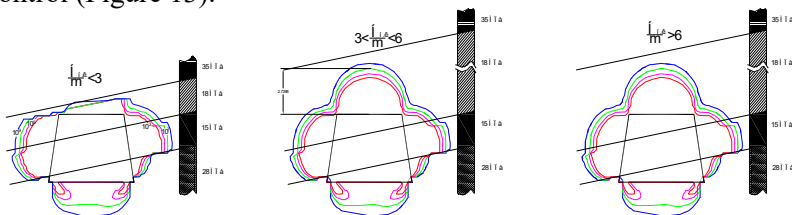
Depending on controllability of the enclosing rocks, the degree of developing the cracking intensity is considered in the trapezoidal shape of the working cross section. Fissuring is most widely spread, and on all sides of the working, at the distance of 2.4-2.5 m from the contour and even less (0.4-0.5 m) at the side racks in the soil near the working with easy- and medium-to-control roofing.

When the roof is difficult-to-control, the cracks in the roof are close to the workings and are at the distance from the working section that does not exceed 0.5 m.



**Fig. 12.** The coal seam inclination angle impact on the distribution of the maximum tangential stresses around the development working

Thus, characteristics of the enclosing rocks controllability are manifested to a greater extent by the proximity of newly formed cracks to the working contour only when the roof is difficult-to-control (Figure 13).



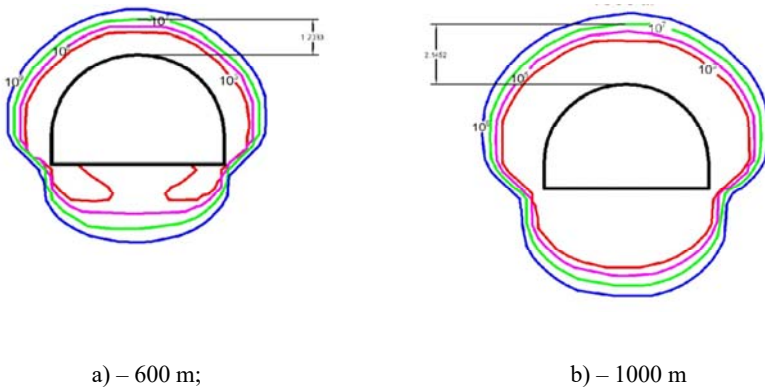
**Fig. 13.** The roof rock controllability impact on the rock massif deformability

Cracking in the roof and sides from the depth of the working location (on the example of the arched shape) of 600 and 700 m, differs only slightly and begins after 0.4 hours at the depth of 1.2-1.5 m in the contour rocks and within the study period of 1.5, 15 and 150 days spread in steps of 0.2 m deep into the massif. In the soil the tendency of crack propagation is similar to cracking in the roof: with the same parameters but at the initial distance from the working contour of 1.5 m [5, 6].

At the depth of 800 and 1000 m the process of crack initiation in the roof is shifted deep into the massif in comparison with the depth of 600...700 m by 0.5...0.6 m (i.e. up to 1.7-2.5 m), and in the soil by 1.5... 2.0 m and begins at the distance of 2.5-3.5 m. In general, the cracking in the roof and sides of the working depends on the depth in the directly proportional relationship, and in the soil it extends in the hyperbolic connection. With the growth of the development depth (from 600 to 1000 m), the cracking increases by the factor of 1.5.

The first diagrams of the conditional zones of inelastic deformations from the outline are located at the distance of 1.6-2.0 m.

Figure 14 shows the dynamics of the intensity of developing fracture zones in time depending on the depth of the working location in its roof, sides and soil.



**Fig. 14.** Dynamics of cracking in the working contour rocks with increasing the development depth



The carried out studies made it possible to determine the mining- and- geological and mining-technical factors impact on forming zones of inelastic deformations in enclosing rocks and to establish rational parameters for the roof bolting use in development mine workings.

### **Conclusions**

- there has been carried out the analysis of the factors influencing the geomechanical condition of workings at the mines of the Karaganda coal basin;

- there have been estimated mining and technological conditions of using the roof bolting of mine workings at coal mines;

- there have been established the regularities of the coal stress-strain state in rock massifs depending on the main mining-geological and mining-technical factors that will allow, under specific operating conditions, establishing the optimal anchoring parameters for increasing stability of development mine workings.

- there has been studied the nature of changing and distributing stresses in the roof, soil and sides of the working;

- there have been revealed the regularities of changing the coal stress-strain state in the surrounding rock massifs depending on the geological and mining factors that will allow, under specific operating conditions, establishing the rational parameters of supporting side rocks to improve stability of development mine workings;

- there has been determined the mining-geological and mining-technical factors impact on forming zones of inelastic deformations in the enclosing rocks around the workings. Stability of the contours of the development workings has been studied taking into account their stress-strain state depending on the geological and technological factors using the finite element method. There have been defined the boundaries of the region of inelastic deformations by the method of successive loading. There are considered the parameters of deformation of the side rocks of the mine workings depending on the angle of the seam inclination and the depth of anchoring.

At the shallow development depth (400 m), minimum stresses are inherent in the rectangular sectional shape of the working. For the

average development depth (600...700 m), the arched shape of the working cross section can be quite acceptable. Changing the angle of inclination in the range from 0 to 30° does not lead to a significant change in the loading pattern. Characteristics of the enclosing rocks controllability are manifested to a greater extent by the proximity of the newly formed cracks to the working contour only when the roof is difficult-to-control (5 times) and is located at the distance from the working section that does not exceed 0.5 m. With the growing development depth (from 600 to 1000 m) crack formation is increased by 1.5 times.

There have been carried out studies that allowed determining the mining-and-geological and mining-technical factors impact on forming zones of inelastic deformations in enclosing rocks and establishing the rational parameters for using the roof bolting in development mine workings.

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# DEVELOPMENT OF MINING TECHNOLOGY WITH MULTIPLE RELOCATION OF EXTERNAL AND INTERNAL DUMPS

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**Abstract.** Iron ore deposits with complex mining and geological conditions are being developed in Ukraine. The operating stripping ratio at most open pits is in the range of 0.6-0.8 m<sup>3</sup>/t, reaching 2.5-3 m<sup>3</sup>/t at a number of open pits. Further development of mineral deposits requires the alienation of large areas of agricultural land to locate the dump facilities (1.3-2.5 hectares per each 1 million m<sup>3</sup> of waste rock). One of the rational ways to solve the problem is to place dumps in the boundaries of mining allotment. The sites for placing the dump are selected taking into account the accepted strategy of the deposit development. Dumps may be located inside the pit or on the surface near the pit. These dumps will be temporary. The accumulated experience of using the temporary dumps in Ukraine shows that we do not always observe a responsible, balanced approach to determining the rational volume of temporary dumps, their location and the estimated costs of subsequent relocation. The use of a shovel-and-truck complex for relocation of temporary dumps increases the need of the pit for equipment and reduces its competitiveness.

The objective of the work is to develop a new method of forming and relocating the temporary dumps using a cyclic and continuous flow technology. The idea of the work consists in the result-oriented use of a gravitational segregation to form a temporary dump with parameters suitable for the subsequent relocation of waste rock by conveyors.

The developed technology allows for piling the temporary dump with one high lift. In this context, a process of natural gravitational segregation takes place during the movement of waste rock over the dump slope. Fine rock fractions accumulate in the upper part of the dump, large pieces slip to its base. Due to the gravitational separation of rock, the conditions are created to use the conveyors for relocating the temporary dump. Moreover, it is envisaged to use a complex of equipment consisting of a bulldozer and a conveyor loader to load the conveyor. A key element of this technology is a conveyor loader. An operating tool of the loading machine is a durable conveyor belt having a large width and a relatively low speed. The conveyor belt is protected by a casing covered with a rock cone during normal operation. The rock falls onto the conveyor belt through a feed

opening located in the upper part of the protective casing, which is able to operate under the gob. In order to improve the efficiency of rock waste separation in the dump body, it is proposed to use a screen on the dump. The work presents the results of theoretical and experimental studies of the developed technology for dump formation. The efficiency of the proposed technology is supported by engineering simulation and economic model.

## **Introduction**

The key feature of surface mining operations is the need for stripping and dumping operations related thereto. Complex mining and geological conditions of bedding the iron ore deposits in Ukraine cause a multiple excess of stripping over mining operations and a high need for alienation of agricultural land for the disposal of waste rock dumps. The residual capacity of most of the external dumps placed within the land allotments that have been allocated for the iron ore open pits under ongoing projects for their development does not allow disposing the waste rock volume to be removed from the open pits within their boundaries up to the end of development.

In order to locate 1 million m<sup>3</sup> of waste rock in an external dump on flat terrain, an area of 1.3-2.5 hectares is required. Therefore, the problem of reducing the need of iron ore pits for lands to place the dumps is of vital importance for Ukraine.

Many Ukrainian and foreign researchers devoted their works to the solution of this problem. One of the main trends that originated in the 80s of the last century and is actively developing to the present time is a theory of in-pit dumping on steeply dipping deposits [25-28]. However, in general, this theory with respect to the practice of surface development of iron ore deposits boils down to the use of solutions that are typical for development of a large horizontally located deposit. Mining conditions, under which the open pit continues its operation without increasing its depth, are created at a deep iron ore pit in the last period of its development, when the volumes of stripping operations are substantially reduced. During the normal pit operation, the benefits obtained from the disposal of waste rock on temporarily non-mining pit walls result in a loss in the subsequent development phase, since additional extraction and handling equipment is required to move the stored volumes. The author knows the design solutions that justify the need for in-pit

temporary dumps, but there are few examples of subsequent relocation of these dumps.

A peculiarity of mining technology used at the iron ore open pits of the CIS countries is the widespread use of conveyance by rail or by truck and rail. Rail conveyance causes the use of low dumps, which need for large areas to place the scheduled volumes of waste rock. Despite the unavoidable technological drawbacks (first of all, a small slope of opening), the design companies continue to use the rail transport in projects for mine development. The restriction caused by a small overcoming slope is eliminated in the feasibility study due to a nonexistent, prospective type of rail haulage being capable of climbing the slopes up to 60%. Another drawback of process flowcharts with rail haulage is the high demand for extraction and handling equipment. Shovels are necessary both for reloading stations and for dumps (for example, having the need for 1 shovel to excavate the rock mass in the face, the existing flowcharts for truck and rail conveyance require two more shovels: for a reloading station and for a dump). This drawback is also common with all flowcharts of cyclical and continuous method of waste rock conveyance applied at the Ukrainian iron ore open pits. Conveyors are used only uphill, a shovel reloads the waste rock onto the railcars at the surface stock and then it is hauled to the shovel dumps.

These process flowcharts are extremely expensive in relation to both natural and material resources, and to labor resources. Based on these process flowcharts it is impossible to implement a resource-saving and environmentally friendly technology for surface mining operations. Refusal of them (at least at the level of projects for mine development) will be a significant contribution to resource saving in the mining industry.

### **The concept of technology with multiple relocation of dumps**

The core aspects of proposed concept include the use of temporary internal and external dumps located and repeatedly relocated within the boundaries of mining allotment.

An efficient technology with multiple relocation of temporary external and internal dumps within the mining allotment should meet the criteria and requirements as follows:

1. implementation of this technology should not increase the need for basic extraction and handling equipment used in mining operations at the pit, that is, the number of shovels should approach the number of those required to excavate the scheduled volumes of rock mass at the pit. The temporary dumps should be relocated with minimum involvement of dump trucks.

In most cases, it is possible to predict 1-2 relocations of the dump. Moreover, the last relocation of dumps is carried out by casting over in the final phase of pit development. The proposed concept allows organizing an efficient separate storage for waste rock with different mineral composition that, in essence, will be a controlled process of formation of new technogenic deposits.

2. One of the prospective engineering solutions makes it possible to stop using the shovels for relocating the temporary dump in favor of bulldozers and conveyor loaders as the extraction and handling equipment [1]. When travelling for a short distance, the performance of bulldozers approaches the performance of shovels, and the conveyor loader operating at the gob loads the rock mass onto the transport facilities. When re-mining the proposed complex of equipment replaces a shovel and is characterized by low steel intensity, high mobility, and low energy consumption. This complex has a significant impact on technology of pit transport operation - a haulage level is located at the waste dump toe, there is no need for transport communications in height of the dump.

3. Implementation of the first two conditions is impossible without ensuring the required granulometric composition. A full changeover to a process with multiple relocation of temporary dumps requires the use of conveyor systems and in-pit crushing operations of all waste rock to be sent to the temporary internal and external dumps. This condition for waste rock accumulated in temporary dumps is not currently being implemented. It is known from the theory and practice of dumping operations that a phenomenon of spontaneous gravitational segregation is observed during the movement of rock mass over the dump slope. This phenomenon was experimentally investigated, its key regularities and factors influencing the efficiency of the separation process were established in the previous works [6]. Thus, the effect of repeating the granulometric composition, on the average, in height of the dump

layer, being very close to the granulometric composition of initial rock mass placed in the dump, was established. In order to intensify the segregation process, the ways of improving the quality of separation by screening before dumping were analyzed in works [6, 8]. The results of new experiments and theoretical studies aimed at improving the quality of rock mass fractionation during the dump formation will be presented in this work. Thus, the availability of rock mass with the granulometric composition that allows using the conveyor systems is the third condition required for implementing a resource-saving technology with multiple relocation of temporary dumps.

The above concept of resource-saving technology for surface mining operations may be fully implemented when developing the iron ore deposits with complex mining and geological conditions. Thus, this technology will allow us to involve a number of deposits of Pravoberezhny magnetic anomalies located at the Petrovsky and Artemovsky open pits of Central GOK to be developed. The proposed technology may be used in the course of technical reclamation associated with casting-over. Inclusion of the equipment for coarse magnetic separation in the process layout will enable us to identify the ores lost during development.

A prototype of the development system with multiple relocation of temporary internal and external dumps may be created and investigated in production within the framework of reconstruction of existing surface complexes for the cyclical and continuous method of waste rock haulage. At the Ukrainian iron ore open pits, all cyclical and continuous flow complexes deliver the waste rock to a surface store. Further, the waste rock is hauled by rail to the shovel dumps. In the current context, this flowchart for waste rock delivery is inefficient and resource-intensive; in the years ahead due to an ever-increasing demand for replacement of worn-out extraction and handling equipment, this development system will be economically inefficient.

Two sites may be identified in order to implement this technology in the process lay-out from stacking the waste rock at the surface store to the subsequent placement in the shovel dump:

- 1) replacement of the shovel at the surface store by a conveyor loader. Conveyor loaders are characterized by a capacity of up to 2-3

thou m<sup>3</sup>/h, that is, they may potentially replace a shovel at the reloading store. At the same time, the loading time of railcars will be reduced, that will result in the increase in the capacity of the reloading station. The use of a conveyor loader does not require significant reconstruction of the store yard. Since the rock mass has been crushed in the primary crusher, this eliminates any problem with the oversized pieces of rock mass.

2) replacement of a shovel on a dump by a conveyor loader. Just like the dump shovel, the conveyor loader should be located below the haulage level. The difference between the levels where a conveyor loader and a rail track are situated determines the capacities of a buffer store and dump rail tracks. A reloading station at the dump may include more than one conveyor loader that will reduce the time of unloading the railcars and will increase reliability of the reloading station operation. The complex of equipment should also include dump conveyors and conveyor dumping machines. The dump reloading station, arranged according to this principle, will be characterized by a long service life of dumping rail tracks in one position that will result in cost cutting and increase in operating reliability of rail transport.

### **The method of surface development of deposits by multiple relocation of dumps**

The developed method [9] may be used in surface development of steeply dipping deposits of a large extent.

The method [5] of surface development of steeply dipping deposits that involves multiple relocation of temporary dumps at the pit is known. A limitation of this method is an increase in the cost of relocation of the temporary dumps as the pit deepens and risk of a decrease in ore mining due to the influence of internal dumps on the size of mining zone.

A method closest in terms of technical essence and the result achieved to the method under consideration is the method [2,3] of surface development of steeply dipping deposits, which provides for dividing the pit in depth into push-backs. When moving from one push-back to another, the temporary dumps, piled during the previous push-back, move to a new location.



A limitation of this method is that the in-pit dump is piled in several lifts, which requires the temporary transport communications and discharge sites on the dump.

In addition, piling the internal dump with several lifts from the pit benches leads to joint placing in the dump body the waste rock, which is transportable by conveyors, and lump non-transportable rock. Therefore, when the secondary dump is piled, the waste rock from the primary dump may be hauled only by truck or by rail. The primary dump will be constructed by shovels from the top to the bottom in benches; however, it is necessary to build temporary transport communications for traffic between the reclaimed and piled dumps.

The use of conveyors to form the primary dumps will require pre-crushing the blasted waste rock that is associated with the need for additional capital expenditures.

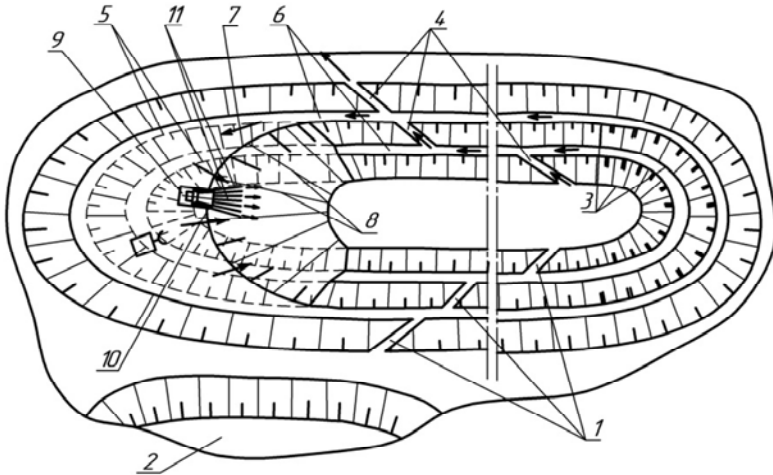
The objective of study is to improve the way for surface development of steeply dipping mineral deposits due to waste rock segregation when forming the primary internal dump, that will allow cutting the specific reduced costs for haulage and transshipment of waste rock from the primary internal dump to the secondary internal dump.

The task is solved due to the fact that when piling the primary internal dump in one lift in height determined by project, the slope surface of the internal dump is formed, where the process of gravitational segregation of rock pieces takes place. Subsequently, the secondary internal dump is formed by waste rock located in the upper and middle parts of the primary internal dump. Moreover, this waste rock is moved to the secondary dump by belt conveyors, which are loaded by a mobile conveyor loader operating at the gob. In this case, the rock located in the lower part of the primary internal dump is moved to the external dumps of the pit. Such division of the primary dump body in height leads to the fact that waste rock, a granulometric composition of which is suitable for transportation by belt conveyors, is used to form internal dumps.

In order to achieve an evident effect of segregation, a mobile discharge platform is equipped on the primary dump at the unloading site of waste rock. An inclined bar grizzly is installed on the platform, and the bars of grizzly are equipped with shock absorbers and arranged

fanwise angularly related to each other.

The development method is as follows. In the initial period, the mining operations are conducted according to a specific technology project. The waste rock is transported (Fig.1) through the system of ramps 1 and placed in the external dump 2 on the daylight surface of the pit. Minerals are mined on the benches of the working zone 3 and transported through a system of ramps 4.



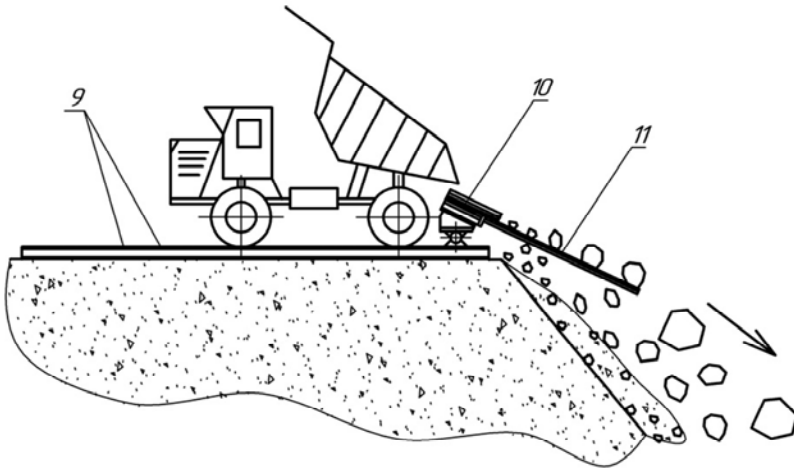
**Fig.1.** Development of the open pit with external dumping and formation of the primary dump.

The mining operations as per known technique are carried out until a stripped area sufficient to start placing the primary internal dump is formed.

After formation of required stripped area, the waste rock is hauled by truck from the working zone 3 through the system of ramps 4 and haulage berms 6 to the zone of non-mining benches 5. The primary dump 7 is piled on non-mining benches 5, the total height of which corresponds to its design height. During the initial period of piling the primary dump 7, backfilling of non-mining benches 5 with waste rock is made and a slope 8 of the internal dump, which will provide segregation (fractionation) of waste pieces, is formed.

In order to ensure a stable effect of waste rock fractionation (segregation) at the place of unloading on the primary dump (Fig. 2), a

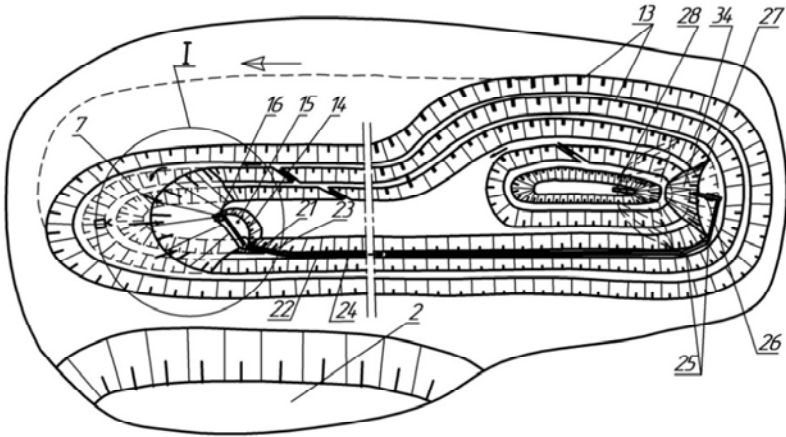
mobile discharge platform 9 is equipped and an inclined bar grizzly 10 is installed, where the bars 11 are sprung and arranged fanwise, angularly related to each other.



**Fig. 2.** A mobile discharge platform on the primary waste dump

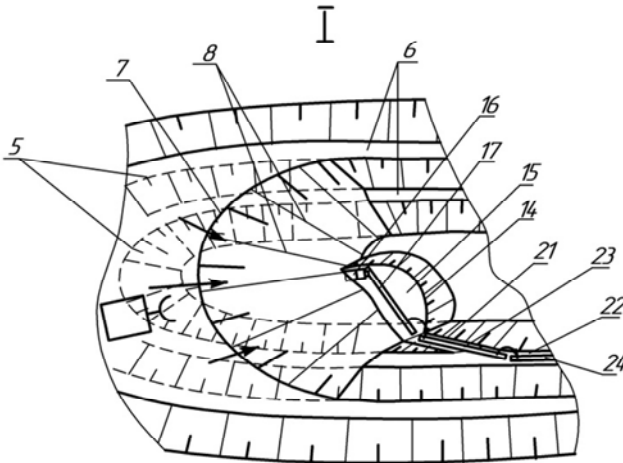
The trucks unload the waste rock at the discharge platform 9 onto the inclined grizzly 10. The unloaded rock is automatically divided into fractions in size - first, the smaller pieces, then the pieces of medium size and, lastly, the largest pieces fall on the sloped surface 8 of the dump. The large pieces of waste rock roll down the sloped surface 8 of the dump and form a lower layer of lumps of oversized fraction as per the condition of transportation by conveyor. The pieces of middle fraction are automatically stacked above this layer, and the smallest pieces of waste rock are stacked in the uppermost part of the primary internal dump 7. Thus, a stable mode of segregation (fractionation) of waste rock in the body of the primary dump is provoked. Moreover, fractionation will occur smoothly, and the height of the layer of each fraction will depend on the granulometric composition of waste rock and the angle of installation of bars 11 relative to each other.

After reaching a pit shell, the pit walls 13 are pushed back and the pit bottom is deepened (Fig. 3). The primary internal dump 7 is piled until the dump starts preventing the development of stripping and mining operations in the working area of the pit.



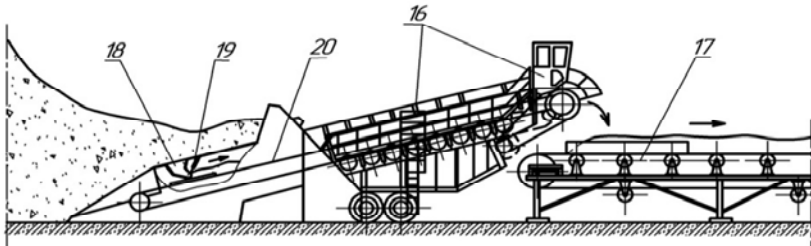
**Fig.3.** Development of the open pit when reaching the pit shell with deepening the pit bottom and simultaneously pushing back the pit walls

After reaching the maximum capacity of the primary dump 7, a bank 14 adjacent to the slope of the dump 7 is formed on the platform adjacent to its bottom edge and a platform 15 is planned thereon. The height of this platform 15 is assumed to be equal to the level of placing the lumps of oversized fraction in the dump body as per condition of transportation by conveyor (Fig. 4).



**Fig.4.** View I of fig.3

A conveyor loader 16 [1] is installed and a transfer belt conveyor 17 is mounted (Fig.5) on the platform 15. The conveyor loader 16 is equipped with a hopper 18 with a gate 19.



**Fig.5.** Loading the transfer conveyor by a mobile conveyor loader operating at the gob

An inclined belt conveyor 23 is installed in a steep half-trench 21, connecting the platform 15 with the haulage berm 22. A semi-stationary belt conveyor 24, a dump conveyor 25 and a spreader 26 are installed on the haulage berm 22 (Fig. 3 and Fig. 4). The spreader 26 piles the secondary waste dump 27.

Upon completion of this scope of work, bulldozers located on the upper platform of the formed primary dump 7, operate layer-by-layer from the top down and move the waste rock to the crest of the dump bench. Further, the waste rock under its own weight moves over the slope of the dump 8 and gradually forms a cone covering the receiving part of the conveyor loader 16 and fills the hopper 18 (Fig. 5). When developing each excavation layer, bulldozers move the waste rock from the periphery of the dump 7 to the point of intersection of its crest and the vertical plane going through the longitudinal axis of the conveyor 16. The gate 19 opens the hopper 18 to load the conveyor 20 of the loader, which reloads the waste rock to the transfer conveyor 17.

The conveyor 17 transfers the material onto the inclined conveyor 21 (Fig. 3), the semi-stationary belt conveyor 24 and the dump conveyors 25. The waste rock from the dump conveyors 25 is piled in the secondary dump 27 by a spreader 26. After transshipment of transportable waste rock located in the middle and upper parts of the primary dump 7, the large non-transportable pieces of waste rock located in the lower layer of the primary internal dump are moved to the external dumps 2 or crushed and transferred to the secondary internal dump 27.

When the pit shell reaches the left wing of the pit, the subsequent deepening is carried out. At the same time, the pit walls are pushed back and the direction of mining operations toward the right pit wing is changed. The waste rock of the secondary internal dump is also developed by bulldozers and conveyor loaders, and moved by the belt conveyor system to a new pit dump.

The use of proposed method will cut the costs for dumping operations and significantly reduce the area of alienated agricultural land that generally cut the costs for development of mineral deposit.

### **Analysis of flowcharts using bulldozers. Development of classification**

The variety of conditions for surface development of mineral deposits and existing types of mining equipment provides for the use of a large number of flowcharts in the world mining industry. However, the mining equipment is represented by a fairly narrow range of mining machines at the domestic iron ore pits, that predetermines a relatively small variety of flowcharts used.

The flowcharts that have not become common use include bulldozers. It is known that at appropriate haulage distances (up to 100 m) the capacity of a bulldozer may significantly exceed the capacity of a single bucket shovel. At the domestic iron ore pits, this equipment is mainly used to form the truck dumps and as an auxiliary equipment to plan the approaches to excavating faces and to clean the roads from spills and snow. More extensive experience with the use of bulldozers is accumulated when placering, as well as at the pits for extraction of limestone and raw materials for the chemical industry.

The most complete analysis of flowcharts using bulldozers is contained in the work [10]. It should be noted that the authors are considering the use of bulldozers when mining the rock mass at the face. The authors do not analyze the flowcharts for re-dumping operations using bulldozers. Also, the flowcharts based on the joint use of bulldozers and conveyor loaders are not considered in these works.

It is known that the technology of rock mining by bulldozers consists in successively removing chips of 0.3-0.5m thick on a

horizontal or inclined (up to 30°) surface. The average travel length where the space is filled in front of the dump is 8-16 m. When operating on an inclined surface, part of the bulldozer weight is used to increase the force for moving the prism of dragging. Analysis of using the bulldozers in the mining operations in the CIS countries allows distinguishing two main types of flowcharts:

1. a bulldozer moves the loosened waste rock into a hopper through which it is delivered to the conveyor or wheeled vehicle;

2. a bulldozer moves the loosened waste rock to a pile, from which a loader or shovel reclaims it to load into the continuous or cyclical transport.

These flowcharts have significant technological drawbacks. Conditionally dividing the process chain into two zones: a zone of bulldozer operations, a zone of the rock mass loading into vehicles, it may be seen that flowcharts of the first type are characterized by a rigid process interconnection of extraction and handling operations and rock mass haulage by main transport. A hopper of the loading equipment is a link in this system. If problems occur in one of zones, the whole process chain stops.

The reliability of flowcharts of the second type is much higher. In this case, the rock mass accumulates in a pile, from which it is then excavated and loaded into vehicles. In this case, if the vehicles are not available or the handling equipment is out of order, the bulldozers do not stop operation and the converse is also true. However, in contrast to the first flowchart, there is a need for additional extraction and handling equipment. It is possible to envisage the use of conveyors in both flowcharts.

The third class of flowcharts may be more perfect and should meet such requirements as independence of operation in adjacent zones of the process chain and the absence of expensive extraction and handling equipment (shovels or one-bucket loaders). A key element of this flowchart is a feeding hopper being capable of operating at the gob, the receiving part of which must be directly located in a pile of rock mass formed by a bulldozer. A flowchart with the use of conveyor loaders meets these requirements.

A conveyor loader is the installation with a belt conveyor, the receiving part of which is located at the gob of waste rock collapsed from the bench by bulldozers, and the unloading part provides the

supply of rock mass to the vehicles; the cyclic feed of waste rock onto the conveyor is converted into a continuous flow. The required strength of the conveyor loader is provided by a powerful welded shield, a conveyor belt of high resistance to abrasive rocks and a shockproof table in the loading zone. In order to reduce the loss of time for shifting the self-propelled conveyor loader, it is recommended to have two shields for each machine under operation at the same time. The conveyor loader is installed at the bench bottom. The rock, being pushed off the bench roof, rolls down the shield and falls in the receiving part of the conveyor. There are four phases in the operation. The machine is shifted and installed during the preparatory phase. The phase of high capacity continues until the rock feeding the conveyor falls off the bench under its own weight. The low-capacity phase takes place when the rock mass developed and moved by bulldozer is at the same level as the receiving part of the conveyor loader or below it. The cleanup phase is connected with the additional loading the rock left at the end of cycle near the conveyor loader. The conveyor loader is then shifted to another shield. Further, the rock mass from the conveyor loader is hauled by dump trucks, by rail or by conveyors.

The third flowchart combines the advantages of above flowcharts for mining operations with the use of bulldozers, being free of their drawbacks at the same time.

Analysis shows that the flowcharts based on using bulldozers may be promising in the development of deposits with complex mining and geological conditions, and in the development of old and temporary dumps. The use of these complexes allows the release of basic equipment for mining operations directly at the pit. However, at the iron ore pits, the flowcharts for surface development of deposits using bulldozer-conveyor complexes have not yet found their application.

The proposed classification is developed on the basis of generalization of domestic and foreign experience in the use of bulldozers for surface development. The classification of flowcharts using bulldozers is based on the features as follows:

1. by the method of mining operations and the type of transport equipment involved in the flowchart:

— cyclic technology;



- cyclic and continuous flow technology;
  2. by the method of preparing the rock mass for excavation:
    - without loosening and crushing the rock mass;
    - with mechanical loosening the rock mass;
    - with blast crushing the rock mass;
  3. by the conditions of bulldozer operation:
    - 3.1 by slope of the face area:
      - horizontal face area;
      - inclined face area;
    - 3.2 by the nature of bulldozer travel in the face area:
      - radial arrangement (the bulldozer moves all rock mass to a certain zone on the slope face);
      - parallel arrangement (the bulldozer places the rock mass in bulk, located parallel to the front slope of excavating block);
  4. by the method of loading and the kind of loading equipment involved in the flowchart:
    - loading the vehicles by extraction and handling machines of cyclic operation (single-bucket shovels and loaders) or continuous operation (compact rotary shovels);
    - hopper loading the vehicles (hopper with a gate, hopper with a belt (plate, vibrating) feeder, conveyor loader);
  5. by the availability of crushing and screening equipment in the flowchart:
    - without crushers and screens ;
    - with screening the rock mass before loading for haulage;
    - with crushing the rock mass before loading for haulage;
    - with screening and crushing the rock mass before loading for haulage.

### **Analytical and experimental studies of equipment and parameters of dumping**

Theoretical and experimental studies of gravitational segregation of rock mass showed that this phenomenon could be predicted in a fairly wide range of dumping equipment capacities - 0.5...33.0 m<sup>3</sup>/s [6]. It was found that the content of coarse and non-transportable by belt conveyors fraction in the lower part of the dump at a piling rate Q=20...20.5 m<sup>3</sup>/s did not exceed 50...53%. When piling the dump

with  $Q=1.8...1.9 \text{ m}^3/\text{s}$ , the content of non-transportable coarse rock is not more than 40%. The content of large pieces in the upper layers of the dump was 5% and 8% respectively. Thus, at natural spontaneous gravity segregation, a large piece of rock is present in all layers of the dump in its height. The obtained results are confirmed by the work [7], where it is found that in the process of spontaneous gravitational segregation, the large-piece material located in the lower part of the dump dominates over the fine fraction.

For the proposed technology [9], it is important when piling the primary dump to provide such a segregation mode, in which the possibility of the occurrence of a large piece in its upper and middle parts is minimized. To do this, it is proposed to pile the dump through a screen [8], which allows not only to separate the coarse rock, but also to assign to the pieces kinetic energy sufficient for relocation and placing in the base of the dump.

In the course of theoretical studies of the structural parameters of the screen, two geometric shapes of its throw-off surface were analyzed, having a radius and logarithmic forms. In each of these options, a rectilinear receiving plate is designed in the screen. The exit velocity of pieces from the throw-off part is found from I.V.Meshchersky's equation of the dynamics of a point of variable mass  $M$  [11]:

$$M \frac{dV}{dt} = M\bar{g} + N + F + \Phi \quad (1)$$

Projecting this equation to the tangent and normal to the surface of the screen, we obtain a system of equations

$$\begin{aligned} M \frac{dV}{dt} &= Mg \cos \delta - fN + \frac{dM}{dt}(u \cos \delta - V), \\ 0 &= N - Mg \sin \delta - \frac{dM}{dt}u \sin \delta. \end{aligned} \quad (2)$$

Solving equation (2), we determine the velocity of particles along the flat part of the screen

$$V_0 = 0.5 \cdot \left( u\mu + \sqrt{u^2\mu^2 + 2gl\mu} \right). \quad (3)$$

For a screen with a radial throw-off surface, the exit velocity of the pieces is equal to:

$$V_H^2 = V_0^2 2e^{-2f(\varphi-\delta)} + A(\varphi) - Ae^{-2f(\varphi-\delta)}, \quad (4)$$

where

$$A(\varphi) = 2Rg \left[ \left( (1 - 2f^2) \sin\varphi + 3f \cos\varphi \right) / \left( 1 + 4f^2 \right) \right], \quad (5)$$

at the dropping point  $\varphi = \frac{\pi}{2} + \beta$ .

For a screen with a throw-off surface that is described by a logarithmic curve, the exit velocity of the pieces may be calculated by formula

$$V_H^2 = V_0^2 e^{-2f(\varphi-\varphi_0)} + \frac{2g\eta e^{-2f\varphi}}{1 + (2f + ctg\mu)^2} [A(\varphi) - A(\varphi_0)], \quad (6)$$

where

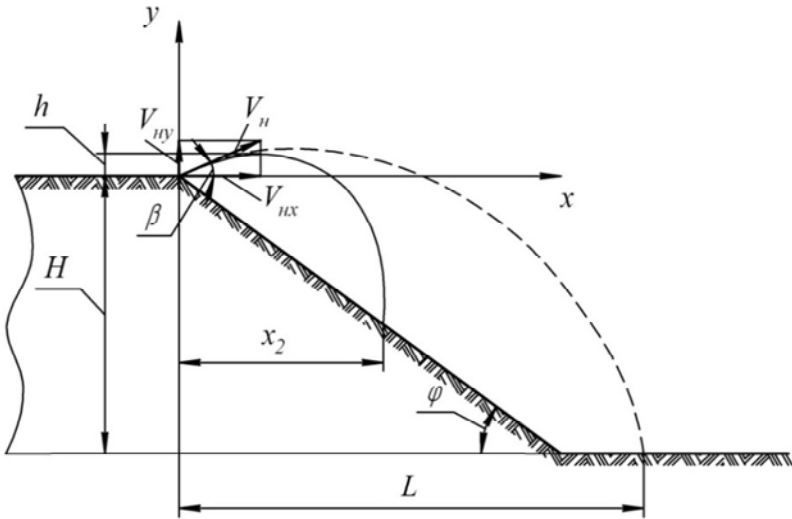
$$A(\varphi) = \left[ \left( 1 + 2fctg\mu + ctg^2\mu - 2f^2 \right) \sin\varphi + \right. \\ \left. + f \left( 3 + 2fctg\mu + ctg^2\mu \right) \cos\varphi \right] e^{(2f + ctg\mu)\varphi}, \quad (7)$$

at the dropping point  $\varphi = \pi - \mu + \beta$ .

The motion trajectory of rock pieces, when delivering the rock through a screen located near the dump crest was analyzed at the second stage of the theoretical studies (Fig.6).

Calculations were based on the assumption that air resistance had no effect on the motion trajectory of a rock piece. Assuming that the size of the screen is incommensurably small in comparison with the dump height, the beginning of the flight trajectory of rock pieces is combined with the dump crest. The surface of the dump slope is

represented by a plane with a slope angle  $\varphi$ , and the surface of the dump base is represented by a horizontal plane.



**Fig. 6.** Computational diagram of motion trajectory of a rock piece

When departing the throw-off part of the screen at an angle  $\beta$  to the horizon, a piece of rock will have an initial velocity  $V_n$  that may be decomposed into components: horizontal -  $V_{nx} = V_n \sin \beta$  and vertical -  $V_{ny} = V_n \cos \beta$ .

The equation of the motion trajectory of rock pieces [12] is

$$y = x \cdot \operatorname{tg} \beta - \frac{g \cdot x^2}{2V_n^2 \cdot \cos^2 \beta} \quad (8)$$

The equation of the dump slope surface as per the coordinate system is

$$y = (-\operatorname{tg} \varphi)x \quad (9)$$

Obviously, in order to determine the coordinate  $x_2$  of the place where the rock pieces fall on the surface of the dump slope, it is necessary (9) to substitute in (8)

$$(-tg\varphi) \cdot x = x \cdot tg\beta - \frac{g \cdot x^2}{2V_H^2 \cdot \cos^2\beta}. \quad (10)$$

Solving equation (10), we find the abscissa of intersection points of dependences (8) and (9):  $x=0$  is the origin of coordinates:

$$x_2 = \frac{2}{g} V_H^2 \cos^2\beta (tg\varphi + tg\beta) \quad (11)$$

The ordinate of the impact point is determined by the expression (9), substituting into it (11)

$$y = -tg\varphi \cdot \frac{2}{g} V_H^2 \cos^2\beta (tg\varphi + tg\beta) \quad (12)$$

If the initial velocity  $V_H$  is sufficiently large, then it is possible to determine the location of falling a rock piece on the dump toe (abscissa  $L$ ).

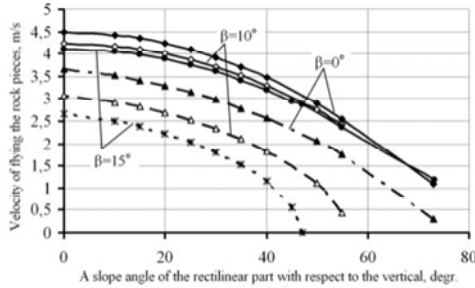
The total time of movement of each rock piece along the  $x$  axis before it falls onto the dump toe is defined as the total time  $t_l$  of the rock movement up to the level  $h$  of the motion trajectory (Fig.6) and the time  $t$  during which the rock will fall down on the dump toe, i.e.

$$T = \frac{1}{g} (V_{Hy} + \sqrt{2Hg + V_{Hy}^2}). \quad (13)$$

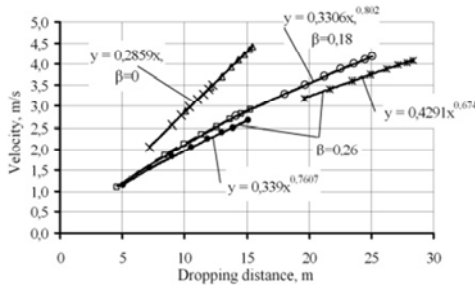
The distance between the dropping point and the point of falling a piece of rock onto the dump toe is

$$L = \frac{V_H \cdot \cos\beta}{g} \left( V_H \cdot \sin\beta + \sqrt{2Hg + V_H^2 \cdot \sin^2\beta} \right) \quad (14)$$

Effect of structural and technological parameters of the screen when piling the dump for various conditions was analyzed. It is established that an increase in the setting angle of the rectilinear part of the screen in relation to the vertical (the flattening of the screen) will lead to a decrease in the initial velocity of flying the rock pieces from the screen surface (Fig.7). The graphical dependences of dropping rate of rock pieces by the screen, the throw-off part of which is made in the form of a radial and logarithmic curve (Fig.8) are obtained as a result of joint solution and analysis of dependencies (4), (6) and (14).



**Fig. 7.** Dependence of flying velocity of a rock piece from the screen surface on the slope angle of its rectilinear part with respect to the vertical



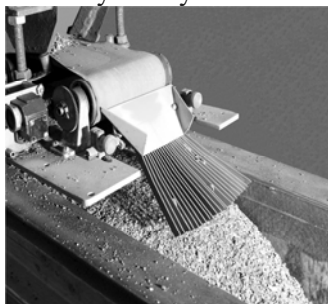
**Fig. 8.** Functional dependence of dropping rate and distance of falling the rock pieces

Using the constructed dependences for the analyzed conditions of dumping, it is possible to establish the yield rate of rock pieces, at which they fall on the dump toe at the lowest point of the dump slope. With a dump height of 60 m and a slope angle of  $36^\circ$ , the rock should cover a distance of 82.6 m. The required initial velocity of a rock piece at the dropping angle  $\beta = 10^\circ$  is 11.39 m/s, at  $\beta = 0$  23.6 m/s, and at  $\beta = 15$  8.407 m/s. However, the increase in the range of flying the oversized product due to the structural parameters of the screen is limited by dropping rates of 4.5...5 m/s (Fig.7). At the dropping rate of 4.0...4.5 m/s, the rock pieces will fall onto the middle part of the dump slope. Large rates of throwing off the oversized product may be obtained through the use of throwers of a special design, operating in tandem with screens.

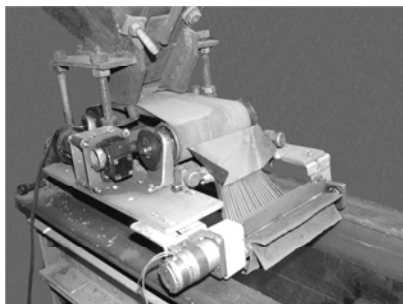
Experimental studies were carried out in the laboratory environment using a dump model (a testing bench) in a 1:100 scale

[6]. The bench was loaded by a conveyor feeder, which, along with the hopper, was moved along the guide rails. The rock was delivered by gravity to the feeding belt of 100 mm wide through a 26x15mm<sup>2</sup> rectangular slot of the hopper.

The studies were conducted in 2 stages. At the first stage, the dump model was piled through the screening model; at the second stage, a rotor-type thrower with elastic rubberized blades was used for dumping (Fig. 9 and fig.10). The experimental study technique is compiled using mathematical methods for planning extreme experiments based on the methods of statistics and known statements of similarity theory.



**Fig. 9.** Conveyor feeder with a screen



**Fig. 10.** A vane-type thrower installed on the screen model

In the process of study, the bench was conditionally divided into two parts – upper and lower ones, for each of which a mathematical model of the content of non-transportable material fraction was obtained as a function of productivity of waste dump formation  $Q$ - ( $x_1$  factor), and the layer level relative to the dump toe  $h$ - ( $x_2$  factor). Factors and their levels are justified by a number of preliminary experiments.

The obtained numerical values of regression coefficients were checked for significance by the Student's criterion, the adequacy of the models was estimated by the Fisher criterion and by means of an approximation coefficient.

In the laboratory, a model of the dump 60 m high was investigated (the height of the model was 600 mm). The regression equation for the percentage of non-transportable fraction, as a function of factors studied, is obtained in a coded form for the upper

part of the dump model, and has the form

$$y = 10,604 - 0,364x_1 - 12,973x_2 + 1,229x_1x_2 - 4,324x_1^2 + 5,589x_2^2, \quad (15)$$

for the lower part of the dump model

$$y = 25,721 - 1,855x_1 - 11,033x_2 - 0,919x_1x_2 - 2,049x_1^2 + 12,961x_2^2 \quad (16)$$

Analysis of the results shows that the regression coefficients  $b_1 = -0.364$  in (15) and  $b_2 = -0.918$  in (16) are statistically insignificant ( $t_1 = -1.83$ ,  $t_2 = -1.893$  at  $t_{table} = 2.101$ ). In view of the fact that in reality these factors influence the segregation phenomenon, we leave these coefficients in the models obtained.

It is found that the use of a screen when dumping allows us to completely exclude a large piece in the upper part of the dump. At the same time, the intensity of dumping affects the content of large non-transportable rock pieces in the layer under consideration to a lesser extent than the height of this layer ( $|b_1| \ll |b_2|$ ). Moreover, the greater the capacity of the dumping equipment and the higher the rock layer in the dump, the lower the content of large rock lumps in the dump layer. These results do not contradict the general idea of segregation and are accounted for the screening function of the screen.

The regression equation for the percentage of non-transportable coarse fraction as a function of parameters studied in its natural form for the upper part of the dump model under the laboratory conditions will take the form, %

$$y = 10,604 - 0,364 \frac{Q - 0,064}{0,036} - 12,973 \frac{h - 440}{140} + 1,229 \frac{Q - 0,064}{0,036} \cdot \frac{h - 440}{140} - 4,324 \cdot \left( \frac{Q - 0,064}{0,036} \right)^2 + 5,589 \cdot \left( \frac{h - 440}{140} \right)^2, \quad (17)$$

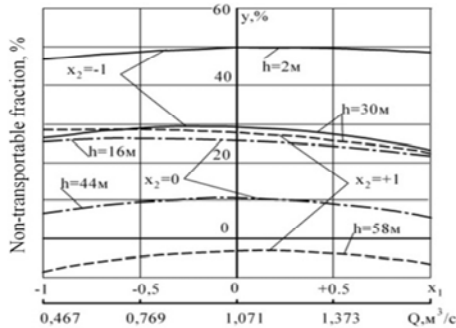
for the lower part of the dump model



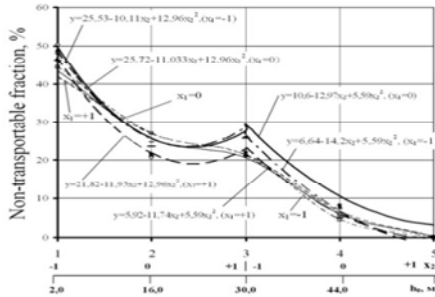
$$\begin{aligned}
y = & 25,721 - 1,855 \frac{Q - 0,064}{0,036} - 11,033 \frac{h - 160}{140} + \\
& + 0,919 \frac{Q - 0,064}{0,036} \cdot \frac{h - 160}{140} - 2,049 \cdot \left( \frac{Q - 0,064}{0,036} \right)^2 + \\
& + 12,961 \cdot \left( \frac{h - 160}{140} \right)^2.
\end{aligned} \tag{18}$$

Based on the results of studies of the content of non-transportable fraction in the parts of the dump (upper and lower parts), the combined graphical dependencies (Fig.11 and fig.12) of the percentage of non-transportable fraction, as a function of investigated factors, brought to real conditions, were constructed.

Analysis of the graphs shows that the use of a screen at dumping makes it possible to achieve a content of non-transportable large pieces in the lower part of the dump of at least 50% and completely eliminate the large fraction in its upper part (Fig.11, curve  $h=58$  m). It is important to emphasize that the negative calculated values of fractions were obtained by extrapolating the data using the least square method. In the middle part of the upper half of the dump, the content of large pieces does not exceed 10% (Fig.11, curve  $h=44$  m). In this case, in the middle layer of the entire dump (Fig.12, layer 3, the curves obtained from the actual measured values are plotted with thin lines), an increase in the percentage of coarse fraction up to 21-22% is observed under all conditions of piling.



**Fig.11.** Dependence of non-transportable fraction content in the dump layers, as a function of productivity of dumping (dump 60 m high)



**Fig.12.** Graphic dependence of non-transportable fraction content in the dump layers, as a function of the layer height (dump 60 m high. The abscissa value 0 corresponds to the dump base)

This negative fact when dumping through a screen may be explained. As theoretical studies show, the oversized product falls onto the dump surface, located close to its middle part in height. Then, the secondary segregation begins, on which the process of sliding the upper dump layers is superimposed with the result that the small fraction partially overlaps and retains the oversized product in the middle part of the dump.

In order to confirm this assumption, the experimental laboratory studies were carried out on the bench with dumping through a screen equipped with a vane-type thrower. It is established that the dumps, formed in this way, do not contain pieces of coarse rock in the upper layers. In the middle layers of the dump at a 30 m height, the content of large non-transportable pieces does not exceed 4...5%, and in the lower layers at the dump toe the content is more than 50...60%.

In the mining industry, there is experience in the use of throwers designed by Stephens-Adamson Manufacturing. This equipment may throw the stored rock at an angle of up to 50° to the horizon to a height of more than 12...13 m and a distance of up to 22 m along the horizon [13]. Also, the throwers of a different type may be used for this purpose [14].

### **Engineering simulation and economic model of relocation of a temporary dump**

Experience in the operation and design of iron ore pits of Krivoy Rog basin shows that in order to cut the operating and capital costs

for mining operations, it is often justified to place temporary dumps within the mining allotment [3,4]. At the first stage, i.e. the stage of building a temporary dump, an economic performance of mining operations is improving. At the second stage, the situation changes.

The second stage begins in 10-20 years after piling the temporary dump and is carried out under more severe economic conditions. Delay in transferring the temporary dumps leads to a decrease in the pit capacity in ore and may cause the revision of the pit shell.

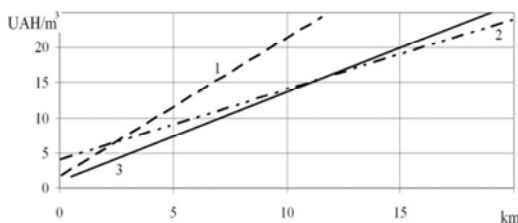
In order to compare the different technologies for piling the external temporary dumps, an economic model and mathematical simulation of three mining options were carried out:

1. development of a dump with the use of single-bucket shovels - rail transport - shovel dumping;
2. development of a dump by single-bucket shovels - truck - bulldozer dumping.
3. development of a dump by bulldozers - a conveyor loader - conveyors - conveyor dumping.

For each option, the haulage distance varied from 1 to 20 km.

For each of the above technologies, the development of three dumps was simulated, differing in the volume of stocked waste rock (6.6 million m<sup>3</sup>, 23.5 million m<sup>3</sup>, 42.7 million m<sup>3</sup>). The height of dumps are assumed to be equal to 45 m, the angle of the dump slope is 20°.

Analysis of the obtained dependences of the specific reduced costs for development of a temporary dump on the haulage distance (Fig.13) showed that the technology using conveyor loaders is most effective at the haulage distance of up to 10 km, at a greater distance the waste rock is more profitable to haul by rail.



**Fig. 13.** Dependence of specific reduced costs on the distance of relocating the temporary dump (1 - truck, 2 - railcars, 3 - conveyors).

Trucks are the most effective at the distance of relocating the dump up to 1 km, but at the haulage distance of up to 2.5-3 km is more beneficial than rail transport.

To achieve satisfactory technical and economic highlights of the process of relocating the temporary dumps is possible when using the developed technology based on the targeted application of gravitational segregation in the formation and subsequent development of temporary dumps [8,9]. This method of surface mining involves the conveyors to develop the upper and middle parts of the dump, containing mainly fine and medium rock pieces.

### **Conclusions**

The article analyzes the major technological challenges of deep open pits in Ukraine that reduce the level of their competitiveness. Sustainable development of mining companies is impossible without abandoning the use of obsolete process flowcharts and reducing the need for alienation of land to develop the waste rock dumps.

The criteria and requirements are formulated with which the resource-saving technology of mining operations with multiple relocation of dumps should be in full compliance.

A method of mining operations with multiple relocation of dumps using a cyclic and continuous flow technology is developed. The feature of the developed method is the result-oriented use of a gravitational segregation to form a temporary dump with parameters suitable for the subsequent relocation of waste rock by conveyors.

Classification of process flowcharts for mining operations using bulldozers is analyzed and performed.

The results of analytical and experimental studies of dumping technology with the use of screens in order to strengthen the influence of gravitational segregation on the internal structure of the dump are presented. The use of a simple screen for dumping allows us to completely exclude a large piece in the upper part of the dump, at least 50% of large pieces accumulate in the lower part of the dump, but the content of these pieces in the middle of the dump is large enough (about 20%). It is possible to significantly reduce the content of large rock fractions in the average in height of the layer using screens of a special design.

The efficiency of proposed technology is supported by engineering simulation and economic model. Further researches will be aimed at determining an optimum field of using the temporary dumps under the conditions of iron ore open pits.

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# INNOVATIVE TECHNOLOGIES FOR GENTLE DESTRUCTION OF ROCKS BY DYNAMIC METHODS

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**Abstract.** The problem of reducing the intensity of dynamic loads on destroyed objects (mountain ranges, oversized blocks, and building constructions) in the processes of gentle reflection of rocks and arrays by using low-energy non-electric initiation systems, domestic analogues of K-pipes and hydraulic hammer is considered. The dynamic of explosion with the use of shock-tube is reduced by 1.9 times compared with traditional means of mass delayed blasting. The use of explosive techniques of block separation from rock massive along with its high productivity leads to significant losses of raw materials due to disruption of the integrity of monoliths under the influence of dynamic loads. Circumstances require the equipping of these works with special domestic means of gentle blasting. The world experience in the use of the K-pipe technology shows that due to the low detonation rate of charges in the K-pipes, which does not exceed 2000 m/sec, the preconditions for a "soft" or gentle explosion are created. The tests of local analogues developed by K-pipes of local production indicate a decrease in the dynamic loads on the adjacent massif of stone from the explosion in comparison with smoke gunpowder by 1.4 times.

The use of dynamic non-explosive destructive methods virtually eliminates the seismic effect. In the case of shock damage, the level of external influence on the object or surrounding array can be controlled by the impact energy  $A_p$ , impact impulse  $I$ , which is characterized by the amplitude of the impact force  $F_{p,max}$ , by the duration of the pulse  $t_p$ , the shape of the pulse and the number of impacts sufficient to disintegrate the object.

The optimum volume of an oversized piece is established with its even placement for the DAEWOO DOOSAN DXB 90 hydraulic hammer based

on the DAEWOO DOOSAN SOLAR 255LC-V excavator. The productivity of the hydraulic hammer, depending on the volume of the oversized object of destruction, is described by a cubic polynomial. Exceeding the optimum volume of the oversized block or destroyed construction significantly reduces the productivity of the hydraulic hammer and increases energy costs. In this connection, the question arises of the combination of the advantages of two methods of destruction, that is, the joint use of a "soft explosion" through the use of K-tubes for the primary separation of the destroyed object into parts, acceptable for their subsequent grinding with a hydraulic hammer in the energy-efficient mode.

**Keywords:** K-pipes, hydraulic hammer, dynamics, rocks, block stone.

**Introduction.** The use of explosive equipment in mining and construction industry puts experts in the field of explosive activity a number of tasks - not only to achieve a qualitative crushing of the rock mass, the development of a sole with an allowable seismic effect, which, with the enlargement of the scale of the mass explosion and the approach of works to objects, becomes of increasing importance to provide for seismic resistance of buildings [1].

In the course of time, new technologies and materials are emerging, but the need for a natural stone does not diminish due to its high wear-resistant, as well as unique decorative properties. Highly productive dynamic methods for separating blocks from an array traditionally include the use of smoke gun or thread of a detonating cord (DC). However, smoky gunpowder has a high sensitivity to friction and fire, it is characterized by high hygroscopicity, which in operation results in initiation failure, and more reliable detonating cord with a high detonation velocity (over 6000 m/sec) and appropriate briarance has too intense dynamic effect on an array of stone, which leads to disruption of its structure and significant technological losses of raw materials.

In international practice, for the "soft" separation of monoliths, the method of K-tubes containing specially developed charges of explosives with a low detonation velocity was used. Ukraine has developed its own K-pipes, which can be used to separate a block stone combined with a hydraulic hammer [2]. Unlike the well-known foreign analogues, which contain high-sensitivity and dangerous nitro ethers in the charge, domestic K-pipes consist of polymer tubes



up to 27 mm in diameter fitted with non-explosive components, which mixture becomes explosive at the site of work due to saturation with a liquid fuel component. Traditionally, such blasting tools are used to separate stone blocks from an array when producing valuable cladding and decorative material.

The detonation rate of the charges in the pipes does not exceed 2000 m/sec, which is a precondition for the so-called soft explosion. For their initiation it is expedient to use low-speed non-electric means, as well as Ukrainian production, which can significantly reduce the seismic effect of blasting operations.

In recent years, along with explosive crushing of rocks, a mechanical grinding tool is successfully used, e.g., hydraulic shock machines, which are an excavator, on which the hydraulic hammer is hung around.

The parameters of the area of the excavator with a hydraulic hammer from its point of standing are determined by geometric dimensions, kinematics of the jib and the angle of rotation of the platform relative to the chassis.

Despite the increase in the efficiency of mass explosions of rock and semi-rocky rocks, the volumes of rocks subject to secondary crushing remain significant in connection with the growth of volumes of mining. Thus, in granite quarries, the average outlet of 1.2 m or more (oversized barrels for crushers) can reach 20-30% [3].

To place oversized blocks it is necessary to occupy large areas of the quarry faces which complicate mining operations, especially when the quarries are deepened; the presence of oversized pieces leads to a deterioration in the quality of the preparation of rock mass, reducing the productivity of excavating equipment and increasing the cost of mining.

Oversized blocks are characterized by the variability of physical and mechanical properties (density, strength, brittleness, etc.), shapes, sizes, etc. that determines the complexity of the choice of technical means to destroy oversized blocks, on the one hand, and the low efficiency of their use on the other hand.

Increasing the effectiveness of the destruction of oversized blocks can be achieved with a certain combination of parameters of both external power and rocks, which corresponds to the characteristics of the oversized blocks. One of the main directions of intensification of

mining is the concentration of production by increasing the unit capacity of equipment. In open mining operations, increasing the parameters of drilling operations (diameter of blast holes, the distance between wells, etc.) leads to an increase in the volume of secondary crushing of rocks and the growth of dimensions of oversized blocks. This requires the creation of high-productivity technical means for the destruction of the oversized blocks.

Given the fact that a large part of the minerals is in conditions that are economically advantageous for open development, the problem of intensification of work on additional crushing of rocks becomes extremely important in terms of optimizing energy consumption, ecology and the economy as a whole.

### **1. Seismic load from K-pipes and low-energy non-electric means of initiation**

In industrial mass explosions with a limited charge weight, for example, when block stone was broken off, in Ukraine's quarries using low-speed non-electric initiation systems, it was found that their seismic effect was 1.9 times lower than in traditional slow-detonating means and local explosions performed according to same schemes for the reduced mass of explosive, but using main detonating cords (DC) and delayers. In connection with this, there is an opportunity to increase the scale of mass or local explosion, on the other - the possibility of improving the seismic evaluation techniques in the production of explosive works with the use of non-electric systems initiating charges of explosives [1].

As the calculations have shown, a non-electric system of the "Nonel" type is a safe scheme for initiating borehole charges in a mass explosion, even without taking into account other its advantages, including such technologically important as the use of the possibilities of lower initiation of the borehole charge. In fact, the use of a combined scheme with electric means of delaying between groups of charges in 15 ms at initiation of a DS is a violation of the requirements of the Uniform Rules of Safety (separate groups are those slowing between at least 20 ms). This should lead to an increase in the risk of a seismic mass explosion, as indicated in the calculations given in the Rules, as well as seismograms, which

indicate that seismic waves are not segregated from certain groups of charges.

However, when it comes to low-speed non-electrical initiation, this rule does not apply. Calculations show that the seismicity scheme of the "Nonel" type with a delay of 17 ms is not inferior to all the parameters of the scheme of initiation of the detonating cord with a period of 20 ms due to the slower passage of the shock wave in the shock-tube.

Explosive charges of domestic K-pipes have low sensitivity to mechanical influences (36-40% sensitivity to shock, 3650-3740 kg/cm<sup>2</sup> sensitivity to friction), which eliminates the danger of their use as a means of block stone separating.

The explosive mixture for K pipes is made of potassium perchlorate and diesel fuel (or nitromethane). They are filled into plastic tubes with inner diameter of up to 27 mm, based on the critical diameter of the detonation of this composition, and accordingly, such charges can be placed in the holes 36-42 mm in diameter, which are most used in Ukraine when the block stone is separated.

The explosive device is made in the form of a plastic tube, which has connecting elements representing plastic nozzles with brake flaps (plumage) for placing the tube spatially along the axis of the hole, and also connect the tubes between them, continuing the charge to the desired length, and initiated by a non-electrical initiation system.

The device is a polyethylene tube filled with a low-explosive explosive mixture of potassium perchlorate, manganese oxide (IV) and diesel fuel (nitromethane). The tube may have an external diameter of different sizes (from 12 mm to 25 mm) depending on the diameter of the hole. The length of the explosive tube is not very significant, since it can be increased due to the connecting element (coupling sleeve) to a length of several meters, which makes it possible to use it when the block of decorative stone is reflected in height (thickness) to 6 meters, or when the grinding of the oversized blocks with the same thickness. Typically tubular charges are produced for an explosive device of 40-50 cm in length. At the end of the tube is a connecting element (coupling sleeve), which has an internal diameter corresponding to the external diameter of the explosive tube (Fig. 1). It allows increasing the length of the charge

to the required length.

The formation of local hot spots when K-tubes are initiated is due to the adiabatic compression of gas or air bubbles that are included in the composition of explosive in the form of inclusions. Near the hot spots, during the impact, a chemical reaction of explosive transformation takes place. An important condition for the propagation of an explosive transformation is that the explosive gases that have been formed are nowhere to expand due to the comprehensive compression of the explosive mass, so they create a high pressure that compresses the adjacent explosive layer. Compression causes heating and rapid reaction [4].



**Fig. 1.** K-tube

The use of domestic K-pipes of quarry production increases the output of industrial block stone and reduces the seismic effect of the explosion, compared with smoked gunpowder, by 1.4 times [5].

## **2. Research of technological parameters of the hydraulic hammer**

The use of dynamic non-explosive destructive methods virtually eliminates the seismic effect. In case of shock damage, the level of external influence on an object or surrounding array can be controlled by the impact energy  $A_p$ , impact impulse  $I$ , which is characterized by the amplitude of the impact force  $F_{p,max}$ , pulse duration  $t_p$ , the shape of the pulse and the number of impacts sufficient to disintegrate the object.

The use of hydraulic hammer actually eliminates the seismic effect on the mountain range. It is recommended to work with a hydraulic hammer only on the front or behind the base machine. Using a hydraulic hammer from the sides of the base machine can lead to the rollover of the excavator, or sharp, devastating vibrations of the machine.

It is necessary to press the hydraulic hammer to an oversized piece of rock with the help of hydraulic cylinders of the working equipment so that the resultant clamping force is directed along the hammer axis, which reduces the radius of the excavator and makes  $0,8-0,7R_{h,max}$ . Analysis of working parameters of digging of mine excavators with backhoe allowed to determine the dependence of the radius of excavator drawing on the weight of the excavator (Fig. 2).

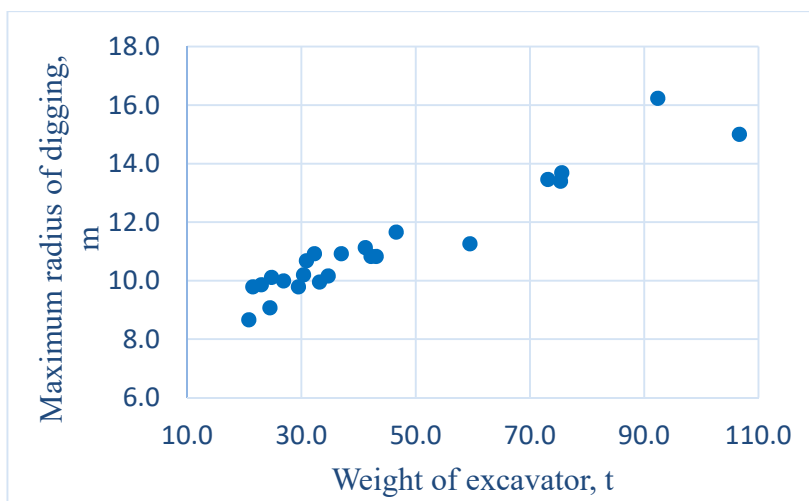


Fig. 2. Dependence of the radius of excavator digging on its mass

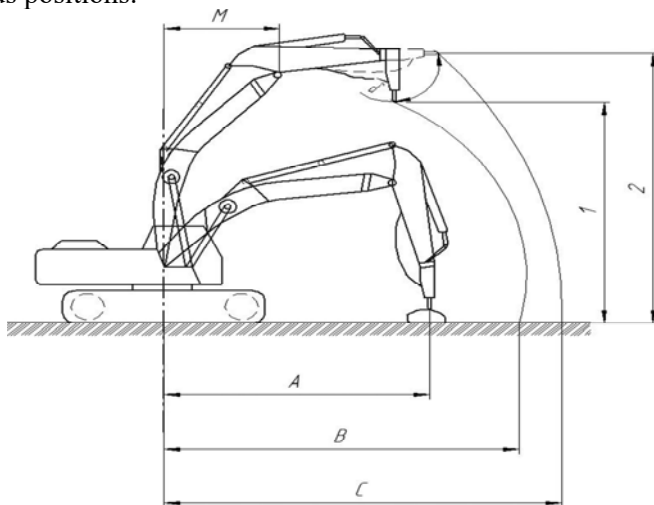
Excavators with a mass of 16-27 tons and a maximum radius excavator digging 9-11 meters have become the most widespread on gravel quarries.

A series of experiments were carried out on the Lesznikow granite mine, which is associated with the features of the destruction of oversized blocks with the use of the DAEWOO DOOSAN DXB 90 hydraulic hammer on the basis of the DAEWOO DOOSAN SOLAR255LC-V excavator. As a result of the experiment, it was found that the most efficient is the crushing of oversized blocks at a right angle ( $90^\circ$ ) to a hydraulic hammer, since with a different

position of the hydraulic hammer, it undergoes an oblique impact and its unfavorable operating conditions are observed - shock (single shock) or lateral bounce. The productivity of the destruction of oversized blocks depends on both the technical parameters of the hydraulic hammer and the characteristics of the excavator used as the base on which the hydraulic hammer is fixed, as well as on its proper use [6].

The passport of the DAEWOO DOOSAN SOLAR255LC-V excavator is shown in Fig. 3 that shows the main parameters, for which compliance (Table 1, 2), it is possible to get the maximum performance of this equipment.

When using a hydraulic hammer, first of all, it is necessary to determine the optimal range of operation of the hydraulic hammer, at which a blow with a maximum destructive energy, when crushing oversized blocks of the rock is possible. Also, the hydraulic hammer can be used for the selective extraction of blocks in gravel mines, so it is worth considering the actions of the hydraulic hammer in its various positions.



**Fig. 3.** Passport of DAEWOO DOOSAN SOLAR255LC-V excavator with DAEWOO DOOSAN DXB 90 hydraulic hammer

**Table 1****Specifications of DAEWOO DOOSAN DXB 90 hydraulic hammer**

Specifications	Value
Weight of the car, t	12~16
Operating mass, kg	940
Diameter of the tool, mm	107
Working pressure, bar	180
Flow, l/min	80 (min)
	140 (max)
Frequency of blows per minute	820
Energy of blow, J	2100

**Table 2****Specifications of DAEWOO DOOSAN excavator SOLAR 255LC-V**

Specifications	Value
Operating weight, t	24,6
Bucket capacity, m <sup>3</sup>	1,1
Engine power, (kW)/(rpm)	165/2000
Max. radius of digging, mm	10240
Max. digging height, mm	9500
Excavator's length, mm	10110
Excavator's width, mm	3200
Excavator's height, mm	3250

Parameters of the complex:

A - the maximum radius of destruction, in which the hydraulic hammer has the maximum clamping force, due to the weight of the handle;

B - the maximum radius of destruction, at which a direct blow of a hydraulic hammer is possible;

C - maximum radius of hydraulic hammer;

M - the minimum radius of the hydraulic hammer;

1 - the maximum height of the hydraulic hammer at its vertical position (8.1 m);

2 - maximum height of the hydraulic hammer at its horizontal position (9.5 m);

d - circular arc of hydraulic hammer's displacement.

Parameter A depends primarily on the length of the jib that for this equipment is no more than 5.9 m, the maximum radius of destruction, in which the direct blow of the hammer depends on the length of the handle and the length of the jib ( $B \leq 8.9$  m). The maximum radius of the hydraulic hammer C is 10.3 m for this equipment, depending on the length of the hydraulic hammer, the length of the handle and the jib. The minimum radius of the hydraulic hammer depends on the length of the jib ( $M \geq 3.1$  m). Knowing the parameters of the hydraulic hammer, we can investigate the influence of these parameters on the productivity of the destruction of oversized blocks, depending on the width and length of the collapse of oversized rocks.

When passing the hydraulic hammer along the width of the collapse, the conditional pieces make up one side of the collapse of the rocks, which ensures high productivity of excavation equipment when loading dump trucks, but the productivity of the hydraulic hammer decreases as the time for moving these pieces from the side which is closer to an array of rocks is lost.

When working hydraulic hammer large in length in bulk oversized blocks, conditioned pieces are formed on two sides that are perpendicular to the collapse of rocks. At the same time, in the hydraulic hammer, the maximum productivity is observed in one pass for large in length in bulk oversized blocks, but there are some difficulties at the entrance of dump trucks and the operation of excavation equipment. Also, during the development of the length of time, the time for moving the excavator to another stope is lost, since such stopes are more than in the development of large in bulk oversized blocks.

The technical productivity of the hydraulic hammer is determined



by its effective power, that is, the product of the impact energy and frequency of impact [7, 8]. The greater the strength of the material that needs to be destroyed with the help of a hydraulic hammer, the greater impact on productivity gives the amount of impact energy. The impact energy of the hydraulic hammer must be such that the destruction of the treated material under the points of its working tool took no more than 15-30 seconds. When destroying viscous materials such as, e.g., frozen ground, limestone and similar materials, the decisive influence on the productivity of the hydraulic hammer has the energy of blow, because for the formation of cracks in the material to be processed, the working tool should be immersed at a sufficiently large depth. The energy of hammering is the kinetic energy of the peen:

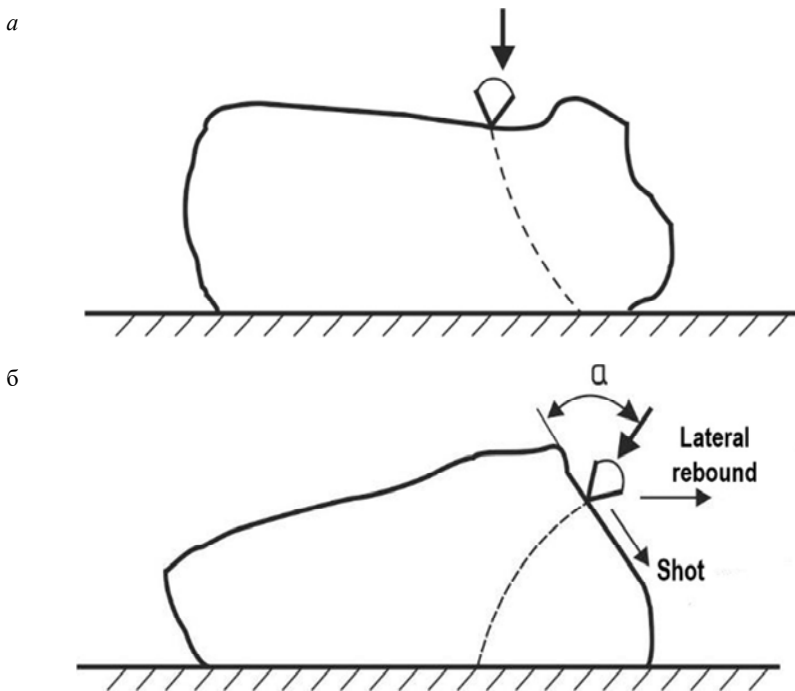
$$E = \frac{mv^2}{2}, \quad (1)$$

Where  $m$  is the weight of peen;  $v$  is the speed of peen at the moment of the collision with the tool.

One and the same amount of energy can be obtained at the expense of the speed of the peen or due to its weight. With equal energy of impact, the hydraulic hammer, which has more weight of peen, will be effective. Unfortunately, the weight of peen the manufacturers of hydraulic hammer do not specify in their specifications and catalogs. In many cases, the technical characteristics of the hydraulic hammer and the value of the impact energy are not given. This is probably due to the fact that this figure is practically impossible to measure under operating conditions, and the magnitude of this indicator is not stable, and depends on the productivity of the pump of the base machine, the angle of the axis of the hammer from the vertical, and the elastic properties of the material being processed.

With the destruction of oversized plates and lamellar forms, a high quality of crushing is achieved (Fig. 4, a).

In difficult configuration (angular shapes) of the oversized blocks during the process of destruction, there are adverse modes of operation of the hydraulic hammer - shock (single shock) or lateral bounce (Fig. 4, b).



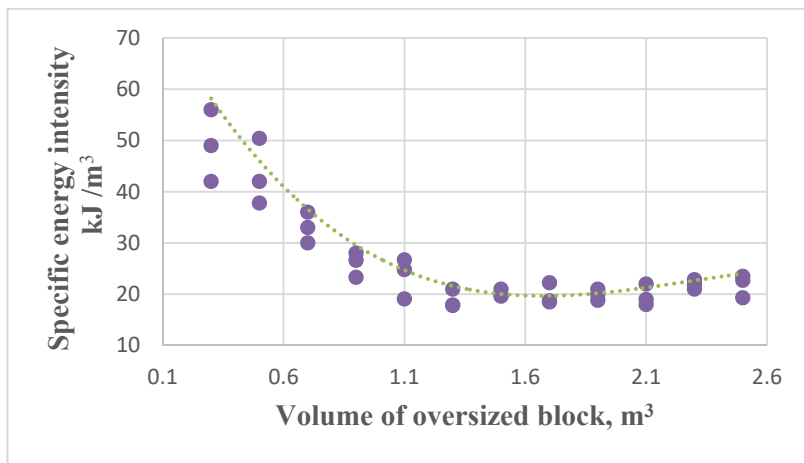
**Fig. 4.** Oversized blocks dimensions:  
 a - direct blow; b - oblique impact ( $\alpha < 90^\circ$ )

According to the results of the destruction of the oversized blocks, which were partially laid out into the invasions (the width of the working platform is 10 m, the length is 35 m), a schedule of the dependence of specific energy intensity on the volume of the oversized block was constructed (Fig. 5).

The graph shows a sharp decline in the specific energy intensity with an increase in volume from 0.8 to 1.3 m<sup>3</sup>.

Gradual increase of the specific energy intensity is from 1.6 to 2.5 m<sup>3</sup>. For these conditions, the optimum piece of oversized block is considered to be a block in the volume of 1.6 m<sup>3</sup>, which has the least specific energy intensity among other oversized large or smaller blocks. This may be explained by the fact that when expanding in bulk oversized blocks don't have a stable basis, larger in volume oversized blocks are more stable in bulk, the less are the opposite. In

the destruction of small oversized blocks the shocks occur more often when it is destroyed in bulk, and according to this energy intensity will be greater than the energy of destruction of large oversized blocks.



**Fig. 5.** Schedule of the dependence of specific energy intensity on the volume of the oversized block when placed in bulk

The specific energy intensity of the destruction of oversized blocks when placing them in bulk of Lesznikow field can be calculated by the formula:

$$E = -6,4V^3 + 43,8V^2 - 92,8V + 82,3 \text{ (kJ/m}^3\text{)}, \quad (2)$$

where  $V$  is volume of oversized block,  $\text{m}^3$ .

As a result of the destruction of the oversized blocks, which are laid out evenly (the width of the working platform is 15 m, length is 40 m), a schedule of the dependence of specific energy intensity on the volume of the oversized block was constructed (Fig. 6).

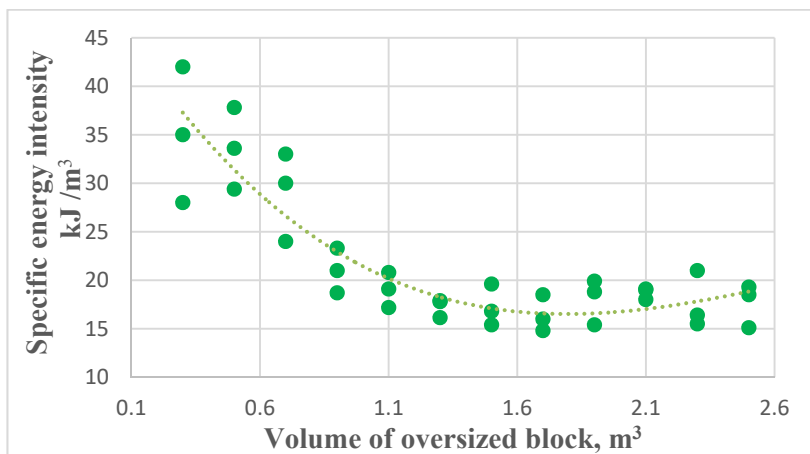
Specific energy intensity of oversized blocks that are laid out evenly can be calculated by the formula:

$$E = -2,1V^3 + 17,6V^2 - 42,5V + 48,5 \text{ (kJ/m}^3\text{)}, \quad (3)$$

where  $V$  is volume of oversized block,  $\text{m}^3$ .

From Fig. 5 it is evident that for these conditions, when placing

oversized blocks in bulk, as well as for evenly placement, the optimal piece in volume is  $1.6 \text{ m}^3$ .



**Fig. 6.** Schedule of the dependence of specific energy intensity on the volume of the oversized block when placed evenly

Also, the width of the gutter affects the productivity of the crushing of oversized blocks, since at a large width of the excavator drive; it takes time to take the working position. The optimum size of the wiring width is determined by the technical characteristics of the excavator.

When laid out oversized blocks in bulk hydraulic hammer in 10 hours destroyed 523 oversized blocks. Its average performance is 61 oversized blocks per hour. In the evenly laid out oversized blocks hydraulic hammer destroyed 610 oversized, therefore its average productivity is 52.3 oversized blocks per hour. In this case, the volume of oversized blocks ranged from  $0.3$  to  $2.5 \text{ m}^3$ .

The crushing of one oversized piece of rock is carried out [9, 10] in about 10-12 strokes of the hydraulic shock device. The hydraulic shock device can perform 300-500 beats per minute, which means that for the destruction of one oversized piece of rock, it is necessary to spend 2-3 seconds.

The productivity of a hydraulic shock device can be conventionally determined by the formula:

$$T_P = t_d + t_m + t_i, \text{ min}, \quad (4)$$

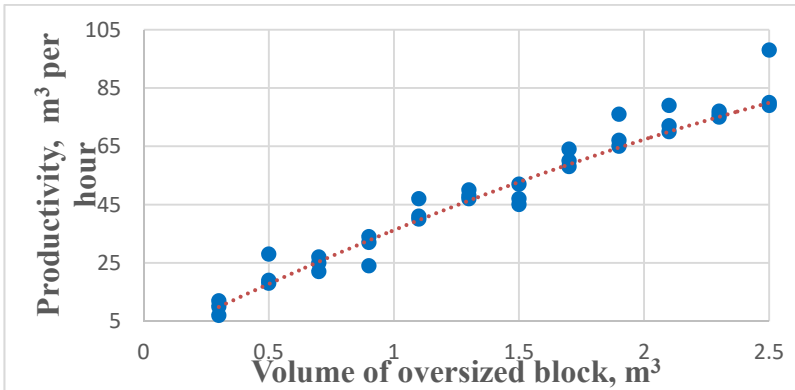
where  $t_d$  is the time spent on the destruction of one oversized block, depending on the distance between the oversize is 10-400 s;

$t_m$  is time spent on the movement of hydraulic shock device, c;

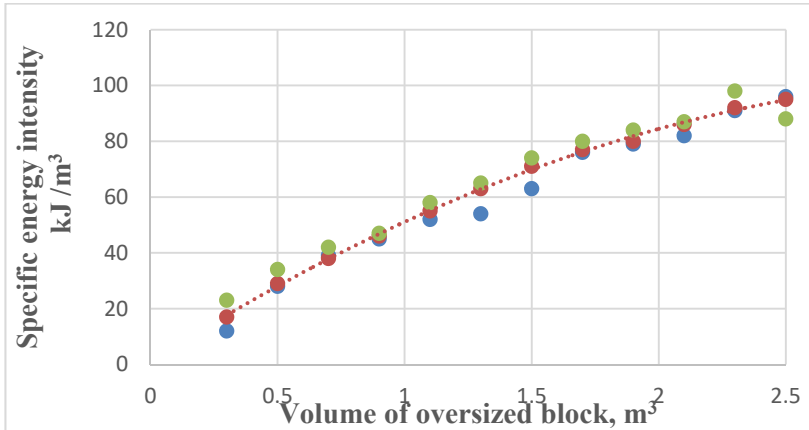
$t_i$  is time spent on installing a hydraulic shock device on the oversized block, depending on the qualification of the operator of the hydraulic shock device, should be from 10 to 60 seconds.

In the case of evenly laying out oversized blocks, the time for performing auxiliary operations will be less than when the oversized blocks are in bulk, as it eliminates the time to move the oversized material to effectively split it or isolate it from among other conditioned pieces of rock. Evenly layout of oversized blocks is characterized by larger workspace, so the time for moving an excavator will be greater than when crushing oversized blocks in bulk.

The productivity of the DAEWOO DOOSAN 90 hydraulic hammer was determined experimentally in part of layout of oversized blocks in bulk (Fig. 7) and evenly layout oversized blocks (Fig. 8), depending on the volume of the oversized blocks.



**Fig. 7.** Dependence of the productivity of the DAEWOO DOOSAN 90 hydraulic hammer, depending on the volume of the oversized blocks when placed in bulk



**Fig. 8.** Dependence of the productivity of the DAEWOO DOOSAN hydraulic hammer, depending on the volume of the oversized block when placed in evenly

Dependence of productivity on volume can be described by the following formula:

$$T = -3,9V^2 + 42,7V + 2,6 \text{ (m}^3\text{/h)}, \quad (5)$$

where  $V$  is volume of oversized block, m<sup>3</sup>.

Dependence of productivity on volume can be described by the following formula:

$$T = -8,4V^2 + 58,7V + 0,83 \text{ (m}^3\text{/h)}, \quad (6)$$

where  $V$  is volume of oversized block, m<sup>3</sup>.

### 3. Relationship of the destruction time with the volume of the destroyed oversized object

The productivity of the destruction of oversize is determined by the time for which the oversized object be completely destroyed.

Experimental investigations were carried out at the Lesznikow mine of granites using a hydraulic hammer. Measurement of the geometric parameters of the oversized blocks and time of their destruction is performed. In total, 250 measurements were made. With the destruction of the oversized tiled form on the conditional pieces of destruction occurred in several stages. Oversized blocks are

divided by the number of stages of destruction. On the charts of figures 9-11, which shows the destruction at a certain stage shows the volumes of formed pieces that are destroyed for a certain time depending on the group of oversized volumes.

Oversized volume was determined by measuring in three mutually perpendicular directions. When measuring, the volume of the oversized volume is set.

In the investigation of the destruction of oversized pieces of rock on the Lesznikow granite mine, it is found that their most widespread form is slab. The investigated pieces are conventionally divided by volume into 3 groups:

Group 1 - up to  $0.75 \text{ m}^3$ ;

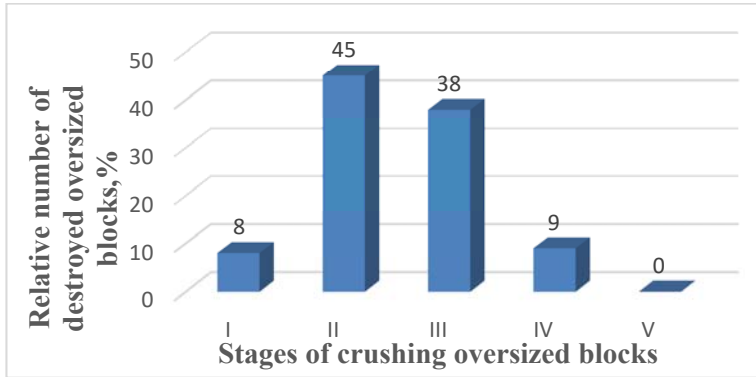
Group 2 - from  $0.75$  to  $1.5 \text{ m}^3$ ;

Group 3 - more than  $1.5 \text{ m}^3$ .

Depending on the stage of crushing, in which the oversized pieces of rock is completely destroyed on the conditional pieces, diagrams have been formed showing the relative number of the oversized objects being studied, which collapses at a certain stage, depending on the volume group (Figures 8 ... 10).

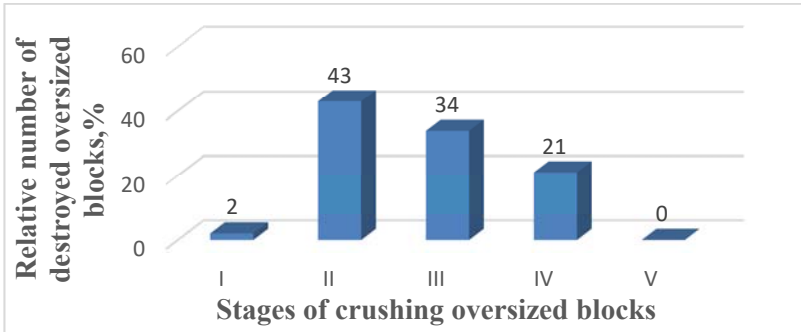
The destruction of the oversized objects on conditional pieces can take place in one or several stages. The number of stages of the destruction of the oversized objects depends on many factors, the main of which are the size and shape of the oversized object, the step of reflecting the oversized object. The stage of destruction of the oversized blocks is the process of breaking the oversized block into the conditional pieces, where the volume of the oversized block at the first stage is an oversized block with the initial volume formed from the array at primary destruction of the rock; the conditional volume is the volume of conditioned pieces formed during the crushing of the primary oversized volume. Oversized blocks that are the subject to the second stage of crushing, are those that were formed at the first stage of crushing. Accordingly, conditioned pieces are the result of 2 stage of crushing. Further characteristics of crushing stages are determined in the same way.

As can be seen from Fig. 9, the oversized blocks from 1st group in volume most often collapses in 2 stage, which is 45% of all destroyed oversized blocks, and also in 3 stage.



**Fig. 9.** Percentage of destroyed oversized blocks of group 1 in volume, depending on the final stage of its complete crushing

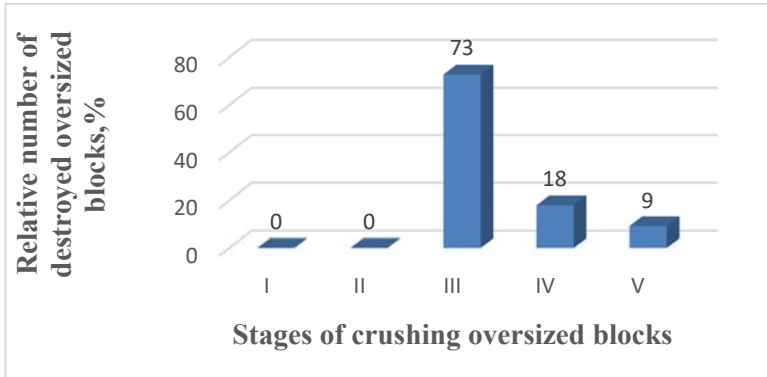
As can be seen from Fig. 10, the oversized blocks from 2nd group in volume are often destroyed also in the second stage, which is about 43% of all destroyed oversized, and in the third and fourth stages.



**Fig. 10.** Diagram of the relative amount of the destroyed oversized blocks of the 2nd group in volume, depending on the final stage of its complete crushing

Oversized blocks from the 3d group in volume (Fig. 11) at all do not collapse in 1-2 stages, and most often collapse in the third stage, which is about 73% of all processed oversized blocks. A significantly smaller amount of destruction occurs at the fourth stage, which is about 18% and in the fifth stage, about 9%.





**Fig. 11.** Diagram of relative amount of the destroyed oversized blocks of 3d group in volume depending on the final stage of its complete crushing

Occasionally, oversized blocks often collapse in two, three or four stages. Effective destruction in the second stage requires time consuming up to 9 seconds, while the volume of formed pieces is 0.75 m<sup>3</sup>; in the third stage is up to 14 seconds, while the volume of the formed pieces is 1.25 m<sup>3</sup>.

Naturally, with the increase in the initial volume of a piece that needs to be disintegrated, the number of the final stage of destruction increases, and hence the need for energy costs. Thus, the third group of oversized fractions, which only indicates the lower boundary of the volume (not less than 1.5 m<sup>3</sup>), which is mainly destroyed only at stage 3, does not collapse at the first and second stages at all. Of this group, 27% of stones, which in volume considerably exceed the specified limit, require 4th and 5th stages of crushing. As for the 9% of the oversized stone, which collapses only at the 5th stage, it is clear that it concerns fractions of considerable volume (in practice, about 10m<sup>3</sup>).

The reason for the origin of such fractions in the volume of rocks destruction should be considered the features of the mutual influence of the systems of detonation and shock waves and stress waves on massive explosions with the use of modern mixed explosives, especially sensitive to changes in their density. Recently, scientists and industrialists pay attention to the phenomena of attenuation or incomplete detonation of charges of emulsion substances that can

noticeably lose detonation properties due to lowering the sensitivity of charges [11]. Desensitization of emulsion explosive is associated with the following phenomena:

When initiating a wellbore charge, an intermediate detonator generates a detonation process in the mass of the emulsion explosive and a shock wave in a destroyed medium that moves in a solid rock mass at a rate greater than the detonation rate of the charge of the emulsion explosive. In this case, the front of the shock waves will pass into an undetonated part of the charge, consolidating the explosive mixture due to the feasibility of a gas sensitizer present in the amount of 1 ... 15%;

- while mass explosion with the use of the technique of short slowness, the stress waves from detonating charge reaches the charge of the next adjacent scheme of the group for 0.5 ... 1 milliseconds, while this charge must detonate only in tenths of a millisecond, that is, the stress wave passes into the undetonated charge, changing its properties.

According to theoretical investigations in both of these situations, the action of stress waves leads to an increase in the density of the emulsion explosive to values exceeding its boundary (about 1.2 t/m<sup>3</sup>), reaching the area of unstable detonation.

Under these conditions, a situation is possible when the explosive substance decomposes in deflagration mode, significantly reducing the energy parameters of the process, which leads to poor-quality crushing of the array and the growth of the output of the oversized blocks and increasing its parameters. In fact, instead of crushing, the array collapses into natural isolations, for which it is inappropriate to use a hydraulic hammer as the primary mean of destruction. At the stage of primary crushing of such fractions, the most reasonable is the explosive method, namely the method based on the use of domestic K-pipes, which is characterized by high safety of work, ecology and economic competitiveness.

#### Conclusion

The level of seismic loading on the array and surrounding objects during the time of mass explosions with the use of low-speed detonating waveguide is reduced by 1.9 times compared with traditional means of delayed detachment.

The use of domestic K pipes of quarry production reduces the

seismic effect of the explosion, compared with smoked gunpowder, in 1.4 times.

The seismic effect of mechanical crushing of rocks is practically absent. The level of external influence is determined by the elastic-plastic state of the oversized object and the nature of its destruction. With a fragile fault, reflection occurs with minimal energy absorption. With brittle-plastic and viscos-plastic types of destruction, energy costs increase significantly. Further research in this direction has great potential for the mining industry.

The optimum pieces of the oversized objects are placed in evenly for a hydraulic hammer based on the DAEWOO DOOSAN SOLAR 255LC-V excavator. The oversized block power transfer coefficient, the volume of the detachable piece and other parameters depend on the dynamic resistance of the rock to the depth of the tool.

The dependence of the productivity of the hydraulic hammer on the volume of the oversized object is described by the polynomial of the third degree. Shock influence on the destruction of oversized blocks is transferred to the base and is partially absorbed by an array of rock.

Oversized blocks are often destroyed in two, three or four stages. For the second stage, the effective destruction takes up to 9 seconds, while the volume of the formed pieces is 0.75 m<sup>3</sup>; for the third one - up to 14 seconds at the volume of formed pieces of 1.25 m<sup>3</sup>.

Instead of the 4th or 5th energy-consuming stages of mechanical destruction of oversized objects, a combined scheme based on their primary destruction using the "gentle" method using the analogue of K-pipes of domestic production with low-energy charges on the basis of non-explosive components and subsequent crushing with a hydraulic hammer is recommended.

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# TECHNOLOGIES OF “ON-LINE” QUALITY CONTROL OF ORES AND THEIR PROCESSING PRODUCTS AT KAZAKHMYN CORPORATION LLP

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## **Abstract**

**Objective.** Developing an effective on-line systems for controlling and managing the quality of copper-containing polymetallic ores and their processing products at the enterprises of the Kazakhmys Corporation LLP.

**Methodology.** The basic method of the studies is X-ray fluorescent (RFM) that is one of the most common methods of nuclear geophysics in mining. There have been used present day nuclear-geophysical technologies (NGPT), as well as the equipment for "on-line" testing and analyzing ores, products of ore processing.

**Results.** There has been developed and adapted for mining and geological conditions of underground mines, concentrating mills and copper smelteries, the NGPM complex for testing and analyzing ores that allows controlling and managing their quality at different stages of exploration, extraction and processing. The basis of the complex is energy dispersive X-ray fluorescent spectrometers of the Kazakh production: portable RPP-12, laboratory RLP-21T, RLP-21T (LA) and stationary conveyors of RLP-21T type. The complex provides an effective on-line control of the ores quality when testing breakage faces, the bulk of the mined ore, the ore on the belt of the concentrating mill conveyor, mine geological and factory technological samples, as well as certain types of finished products of the copper smeltery.

**Scientific novelty.** There have been developed the methodology of the RFM and the limits of its practical use at the stages of mining, concentration and metallurgical operations. It has been proved that the RFM provides an effective management of the ores quality for all industrial components, including light and associated elements. For the first time it shown that

geological sampling at mines and controlling the ore quality at the concentrating mills of the Kazakhmys Corporation LLP can be completely transferred to the NGPM "on-line" ore testing. RFM is providing rapid analysis of anodic copper samples with high accuracy. There have been shown favorable prospects of determining the content of gold in the raw materials coming to the metallurgical plants and in the samples of anodic copper.

**Practical relevance.** NGPT allow organizing the effective “on-line” control of the ore quality and there processing products in the chain: mine – concentrating mill – copper smeltery. There are recommended to implementing the methodology RFM testing and new specimens of nuclear-geophysical equipment.

## Introduction

The Kazakhmys Corporation LLC is the largest copper producer in Kazakhstan. The company includes 10 underground and open pit mines, five concentrating mills, two copper smelteries, and two coal mines.

The world experience of extracting and processing nonferrous metal ores clearly shows: in the market economy it is impossible to organize an efficient operation of mining and processing facilities without using on-line systems for quality control of extracted and processed multicomponent polymetallic raw materials. This circumstance brings necessity the development and implementation of on-line systems for quality control of ores and their products at the mining and processing enterprises of the Kazakhmys Corporation LLP.

Solving the problems of organizing an effective on-line control of the ore quality at the stages of mining and concentration processing complicates the multicomponent, characterized by a great range of contents of all industrial and interfering components composition of the ore of polymetallic deposits developed by the Kazakhmys Corporation LLC, such as: the copper-lead-zinc deposit Zhezkazgan (the main and associated ore components: Cu, Pb, Zn, Ag, Re, Cd, S, Os); the copper-lead-zinc deposit Zhaman-Aibat (Cu, Pb, Zn, Ag, Re, S); the gold-copper-porphyrus deposit Nurkazgan (Cu, Au, Ag, Mo, Se, S), the pyrite-copper-lead-zinc sites of the birth of Kusmurin (Cu, Zn, Pb, Au, Ag, Cd Se, Te, S) and Akbastau (Cu, Zn, Pb, Au,

Ag, Cd, Se, S, Te), the gold-pyrite-copper-lead-zinc deposit Abyz (Pb, Zn, Cu, Au, Ag, S, Se, Te, Cd, In, Hg), the Sayak group of copper-skarn deposits (Cu, Mo, Fe, Au, Ag, Bi, Te, Se, Re), the copper-porphyrific deposit Shatyrkol (Cu, Mo, Au, Ag, Te, Se, U) and others.

Zhezkazgan concentrating mills No. 1 (ZhOF-1) and No. 2 (ZhOF-2) of the Processing and Production Complex of the Kazakhmys Corporation LLP branch, the Zhezkazgantsvetmet LLC are processing ore from the mines and quarries of the Zhezkazgan mine site (Zheskazgan copper sandstone deposit), as well as from the Zhomart mine (the deposit of the median sandstone Zhaman-Aibat, 160 km to the southeast of Zhezkazgan, the transportation distance is 169.6 km).

Due to the large number of suppliers of ore at ZhOF-1, 2, the task of objective distribution of copper in the concentrate between the mines and quarries is transformed into the rank of the most important production tasks. This distribution to the latest time was carried out according to the input quality control of the ores organized by the quality control department (QCD).

Quality control of the ore entering ZhOF-1, 2 is performed according to the following scheme: ore that has passed the large crusher of type 900/160 (ZhOF-1) and KKD 1500/1800 (ZhOF-2), is tested at intervals of 3-4 minutes using semi-automatic samplers; the selected samples at the end of the shift are delivered to the sample preparation department of the QCD at ZhOF-2; the samples ready for analysis are sent to the express laboratory to the RLP-21T spectrometer. The speed of this scheme, despite the fact that the last link in the whole chain of testing is the express analyzer but not the chemical analysis, is from 5 to 12 hours. The current scheme cannot implement operational input "on-line" quality control of ores.

### **The work objective**

Developing effective on-line systems for controlling and managing the quality of copper-containing polymetallic ores and products of their processing at the stages of mining, dressing and metallurgical operations of the Kazakhmys Corporation LLP enterprises based on the most up-to-date equipment, methodological

and mathematical support. At this, "on-line" control should be carried out not only for the main (profile) industrial component, but also for all balance components, including the associated ones (silver in the first place).

### **Methodology of studies**

Determining the content of elements in the ore is the task of mine testing which has always been the most "narrow" link in the quality management systems of mined ores and metals of any configuration. With introducing (in 1978) nuclear-geophysical technologies for testing ores in natural occurrence, in transport tanks, in bulk of the mined rock mass in the practice of geological servicing of mining operations at the Zhezkazgan mines, the situation radically changed for the better. It was from that moment that the task of the "on-line" quality control of extracted and processed multicomponent polymetallic raw materials obtained a perspective practical solution.

The world experience shows that of the whole variety of present day nuclear-geophysical technologies of ore testing (NGPT), the task of "on-line" control is most closely matched by the X-ray fluorescent method (RFM). Today there is no need to prove that there is no alternative in the mining industry in terms of productivity, efficiency and representativeness. The questions arise only regarding the accuracy of sampling and analyzing; the number of basic and associated ore elements determined with the help of the RFM; limits of detecting elements; NGPT adaptation to specificity of the mining technologies used.

Though the RFM is a shallow (film) method of testing, theoretical studies of scientists (Bakhtiarov A.V., Bolshakov A.Yu., Krapivsky E.I., Leman E.P., Losev N.F., Meyer V.A., Ochkur A.P., Plotnikov R.I., Pshenichny G.A., Yakubovich A.L., and others) confirmed by numerous experiments at mining enterprises of nonferrous metallurgy in Kazakhstan, it was convincingly proved that by selecting the optimal testing network and its frequency, working only with "fresh" but not dusty faces, boreholes and ore piles there can be reliably achieved certainty and representativeness of the data of X-ray testing (RFT) quite sufficient to meet challenges in the operational management of the quality of extracted ore as both for



the entire list of commercial ore components and for the accuracy, validity and sensitivity of the “on-line” definition of individual elements [1].

Up-to-date software allows accurate determining of both the response function of the detector and the spectral composition of the exciting radiation. The secondary emission spectrum is reconstructed by using the nonlinear least squares method, taking into account the dependence of the relative intensities of the characteristic lines of the elements on the material composition of the ore. The secondary instrumental spectra are cleared from double and triple overlays provides a high-precision extraction of the true intensities of analytical lines of elements from the secondary instrument spectra, and, regardless of the detector (proportional, semiconductor). The use of fundamental algorithms to take into account matrix effects, including scattered radiation, provides highly effective accounting for changes in geometric measurement conditions with variations in the real composition, density of ores and the "sensor-probe" gap in a wide range.

Today with the use of NGPM, there are being solved not local analytical problems, as it was before, NGPM really pretend to play the main role in the geological support system for mining operations in the scale of such a giant of non-ferrous metallurgy in Kazakhstan as the Kazakhmys Corporation LLP.

Indeed, the RFM ore testing in conditions of natural occurrence, broken rock mass, transport tanks and conveyors, the RFM analysis of worn downhole, drill, slurry, carload and core samples, the RFM logging of production and exploration wells is not only in the rank of the main methods of geological support of mining operations at the mines of the Kazakhmys Corporation LLP, but also in the rank of virtually the only tool for forming the information blocks of subsystems for monitoring production processes for ores and metals and direct operational management of the quality of mined ores and metals of the new generation.

The RFM task for in the faces at the mines of the Kazakhmys Corporation LLP has been solved since 1977. Within these years a series of portable energy-dispersive X-ray fluorescent (EDXRF) spectrometers RPS4-01 Gagara, RRK-103 "Search" and, finally, since 1998 RPP-12 have been consistently involved. In all

spectrometers proportional radiation detectors and ionizing radiation sources Cd-109 and Pu-238 were used. The number of simultaneously determined elements was brought to four: Cu, Pb, Zn, Fe [1-6].

In total at the Zhezkazgantsvetmet PA, every day 18 teams of the face RFA of the geophysical service descend into the mine. The annual volume of the RFA was brought (in 2010) to 302,530 r. m. of the face cross-sections. A high efficiency of the face RFA was graphically confirmed by the tables in the annual geological reports, in which the average copper content according to the face RFA data was always between the drainage data of the concentrating mills and the carload testing of the QCD, both for a year as a whole and for months, moreover, much closer to the data of the discharge than to the data of car testing of the QCD. The total number of RPP-12 spectrometers at the mining enterprises of the Kazakhmys Corporation LLP exceeds 30 units.

At this stage of the RFM use one of the main problems of the method was finally solved: the problem of complete separation in the secondary spectra of analytical lines of elements with neighboring atomic numbers (copper and zinc) using a proportional radiation detector RPP-12 spectrometer. The task of testing high (up to 7 m and higher) mining faces and ledges has been successfully solved.

In addition the face RFA, at the Zhezkazgantsvetmet PA since 1987 there has been solved the task of the RFA for downhole, drill, slurry, carload and core samples. All these years the RFA was carried out using laboratory EDXRF spectrometers RPS4-01 "Gagara", BRA-3, RAL-M1M, RLP-21 and RLP-21T.

The largest lists of detectable elements were in the spectrometers RLP-21 (34 elements) and RLP-21T (31 elements). The maximum number of spectrometers in the laboratory reached 7 units. The maximum annual number of samples was 243 515 in 2006. The entire volume of the QCD carload samples passed through the RFA laboratory.

At this stage of using the RFA, there were solved the following complicated methodological tasks:

- there was developed the RFA methodology that made it possible to implement the principle that the developers and suppliers of the RLP-21 and RLP-21T spectrometers "Aspap Geo" LLP were guided

by: the RFA objects are different, the types of ores are different, the industrial products of ore processing are different: the calibration is the same which was confirmed by the analyses of polymetallic ores from different deposits of Kazakhstan, industrial products of Zhezkazgan copper plant (dump, converter and anodic slag, granules, matte, material from boxes, turnover from the shop of crusts, recycled dust, concentrate with onvertor) with the calibration by RLP-21T on state standard samples (SSS) of ores of Kazakhstan deposits. Moreover, the convergence of the results of RFA and chemical analysis was within the limits of tolerances of ordinary chemical analysis;

- there was developed a package of specialized application programs, unique in complexity and capabilities, in which there was used a modification instead of the method of spectral parameters to take into account the matrix effect, the method of spectral coefficients: corrections were introduced only for the elements determined by the RLP-21T, as well as certain undefined elements whose contents can be calculated through the correlation coefficients with the determined associated elements (for example, sulfur through iron at copper-pyrite deposits);

- there were achieved very low limits for detecting the content of elements (silver and cadmium in the first place): Ag = 0.74 ppm (SSS -3029; C (Ag) = 2.1 ppm); Cd = 1.05 ppm; C (Cd) = 5.0 ppm); Zn 0.0058% (SSS - 2887; C (Zn) = 0.011%), Pb 0.0084% (SSS - 2887; C (Pb) = 0.037%) when the measurements were exposed within 120 to 150 seconds;

- there was developed a highly effective identifier of the analytical lines of elements excluding the appearance of "false" contents of elements, such as Eu = 0.366%, Lu = 0.103%, Re = 288.9 ppm, Os = 11.3 ppm, as for the Epsilon-3 spectrometer -XLE PAN-analytical (Netherlands) for SSS - 3032. In RLP-21T in the application package there was a tool that allowed describing with the accuracy of 95-98% the spectrum of the L-series of lead, for example, in which there were present 20 lines, and removing these lines from further processing. Therefore, there were no "false" contents;

- it became possible to provide RFA for polymetals and light (Al, S, Si, P) elements in one measurement without using vacuum or inert

gas and without special sample preparation for the analysis;

- it became possible to ensure carrying out the RFA for rhenium with accuracy of ordinary chemical analysis (category 3), starting from 1.35 ppm (in one of the spectrometers RLP-21T there was a special option "RFA for Re").

The production geophysical work at the Zhezkazgantsvetmet PA was provided by the developed in 1988 geophysical service of up to 70 employees. All methodological and instrumental developments began to be introduced at the mining and processing enterprises of the other branches of the Kazakhmys Corporation LLP:

- RFA of ores on portable spectrometers RPP-12 was launched at the Abyz, Akbastau, Kusmurin and Nurkazgan underground Karagandatsvetmet pits, at the Sayak, Kounrad and Shatyrol of the Balkhashtsvetmet PA;

- RFA of ore samples on spectrometers RLP-21T is using at the Nurkazgan-underground, Abyz, Akbastau, Sayak and Shatyrol mines;

- RFA ores and processing products samples on spectrometers RLP-21T is used in the RFA laboratories at the Balkhash, Karagailinsky and Nurkazgan concentrating mills.

The task of organizing the "on-line" control at ZhOF-1, 2 is complicated by such factors as a large size (class 250 mm) pieces of ore which is to be tested; the need to determine the content of silver in ores in the range of contents from 2-3 to 50 ppm.

The world experience in organizing the on-line control at concentrating mills evidenced by using an ore-control station (OCS). The OCS market is very diverse: X-ray fluorescent OCS: STARK PC (Kras-Rados, Krasnoyarsk); RKS-KM (Tekhnoros LLP, Krasnoyarsk); Online Conveyor XRF Analyzer Con X-03 (Baltic Scientific Instruments Ltd, Latvia); RKS ARP-1C (Technoanalitpribor LLC, Moscow); RCC-1M (OJSC NIITFA, Moscow); RKS-A (INTEGRA GROUP, Moscow); RKS on the gamma-neutron activation analysis (PGNAA): CB Omni (Thermo-Fisher Scientific, Australia) [2].

The abovementioned factors that significantly complicate practical implementation of the analytical task, sharply reduced the list of OCS that can be recommended for using in Zhezkazgan. At this it is necessary to clearly understand: the overwhelming majority

of OCS represented in the market, when analyzing multicomponent polymetallic ores of non-ferrous metals are effective only with the ore size of no more than 40÷50 mm. The task of determining silver generally shortens this list to several units.

Having analyzed the possibilities of adaptation of the OCS available in the market to the realities of the ZhOF-1, 2, we chosen OCS RLP-3-02 (the Geotech LLC, S-Pb) that has a number of advantages over other OCS, namely: the presence of two X-ray tubes which allows significant increasing the area of the "spot of sight" of the ore surface on the conveyor belt ("spots" of this size are no longer present in any OCS) and increasing the reliability of sampling; the ability to really determine the content of silver in ores in the range of 2-3 ÷ 50 ppm. We took into account that OCS RLP-3-02 was implemented at the JSC "Mining Company" AIR "(Vostok s., Primorsky Krai, Russia), where it sorted ores in dump trucks "BelAZ" for copper and tungsten; at the CJSC "Silver Magadan", where it was used for testing ores for silver on the conveyor belt. In addition, RLP-3-02 works very steadily with large-lump (200÷400 mm) ore and is very reliable and unpretentious in operation.

The main problem that is to be solved at ZhOF-1, 2 is compensation of the effect of the variable profile of ore on the conveyor belt and the presence of large pieces of ore on the belt. For many OCS this problem is solved by using a special ultra-sound sensor. In the RCS RLP-3-02 for these purposes there is used an incoherent component of the scattered radiation (more precisely: some part of the spectrum of incoherent radiation, a kind of "know-how").

OCS RLP-3-02 works in the mode of scanning the ore surface on the belt of the conveyor. Continuous scanning time can reaches 30 seconds, then there is a short pause of up to 5 seconds. The removal of the results of the "on-line" testing (Cu, Pb, Zn, Ag) is made by entering the start and the end time of loading the car with the ore in the "Start" and "End" windows of the main dialog box of the OCS ARM.

OCS RLP-3-02 was put into operation on the conveyor No. 1 of ZhOF-2 in the large crushing plant in 2013, then it was transferred to the conveyor line No. 1A of the large crushing plant at ZhOF-1 in connection with the need for the on-line controlling the the ore

quality supplied by the Zhomart PA, that within some time was not included in the Zhezkazgantsvetmet PA. With introducing the OCS, the point of the QCD testing for the input ore at ZhOF-1 was abolished.

And now let us focus on the latest methodological and equipment developments, as well as research cycles that will form the subject of this work.

1. Despite the excellent work of the RPP-12 spectrometer, the ore RFA did not keep pace with the increasing needs of mining. In particular, one of the most important objects of quality control of ores was silver produced in passing. The requirements to the safety of carrying out the RFA faces were substantially tightened. The RPP-12 spectrometer (developed in 1998) requires modernization: the renewal of the element base, increasing the number of detectable elements, and increasing safety of the RFA process.

To solve the very difficult problem of "face RFA for silver" we selected three EDXRF portable spectrometers developed in: ElvaX GEO (Ukraine), Spectroskan GEO (Russia) and RPP-12T (Kazakhstan). The basic selection criteria were as follows: a) the minimum weight; b) the ability to test faces up to 7 m high; c) high reliability of the results of testing ores for silver at the average level of its contents of 15 ppm; d) the speed and quality of servicing.

According to the weight, the spectrometers made up the series: RPP-12T in the mine version 1.16 kg, ElvaX GEO 2.0 kg, Spectroskan GEO 2,4 kg. Only RPP-12T has the means of delivery of the sensor to the height of up to 7 m. For testing poor silver-bearing ores. In the speed and quality of service RPP-12T is out of competition (the experience of joint work with Aspap Geo LLP since 1996). Was chosen the RPP-12T spectrometer. That spectrometer has: a silicon drift detector (SDD) with the area of 25 mm<sup>2</sup> with thermal cooling and energy resolution of 140 eV; a device for fixing rods; an X-ray tube 50 kV, 4 W; a wireless (Bluetooth) scheme for transmitting signals from the sensor to the instrument; a Samsung smartphone with a shockproof case; 34 elements: Cu, Zn, Pb, Ag, Cd, As, Se, Ba, Fe, Mo, Mn, Ti, V, Cr, Co, K, Ca, Ni, Ga, Br, Rb, Sr, Zr, Y, In, Pd, Nb, Sn, Sb, Te, Bi, W, Th, U; the surface area of the sampling object is 4-5 cm<sup>2</sup>; the exposure measurement at one point is from 5 seconds; the detection limits for most ore elements range

from  $n \cdot 10^{-4}\%$  to  $n \cdot 10^{-3}\%$ .

Besides that it has one more important advantage: the device of registering and processing is a smartphone of the latest generation with the Android operating system and an impact-resistant casing. This innovation provides high speed, software flexibility, the possibility of voice control of the device, as well as additional possibilities for documenting the results of sampling (photographing the face, determining the coordinates, operative data transfer in the presence of the network). Competitors use portable pocket personal computers that are much more cumbersome and less productive.

In Figure 1 the RPP-12T spectrometer is shown in three versions: for testing faces (left), for testing ore in the bulk, the broken rock mass, the core of exploratory wells and cutting wells in the quarries (in the center), for analyzing powder samples using a special round nozzle in which there is inserted a cuvette with a sample (right). Figure 2 (on the left) shows the probe part of RPP-12T (a version for face RFA).



**Fig. 1.** Versions of the RPP-12T spectrometer

2. To solve the problem of upgrading the RPP-12 spectrometer, at the Aspaz Geo LLP there was developed and tested the RPP-12T the spectrometer and put into operation by the Geophysical Department of the Zhezkazgan region of the Quality Control Department: a 25 mm<sup>2</sup> SDD detector with thermal cooling and energy resolution 140 eV; 1-2 closed sources of ionizing radiation Pu-238; a device for fixing rods; a wireless (bluetooth) scheme for transmitting signals from the sensor to the instrument; a CAT smartphone with a shock-

resistant case; 12 elements (Cu, Zn, Pb, Fe, Ba, K, Ca, Ti, Mn, Ni, As, Sr) and more. The RPP-12T spectrometer is able to test faces with the height of 7 meters or more. The general view of the RPP-12RI spectrometer and the procedure for performing the face RFA is shown in Figure 2.



**Fig. 2.** RPP-12 and RPP-12RI spectrometers

3. The experience of industrial operation of the OCS RLP-3-02 has shown that this OCS is not most likely capable of fully ensuring the solution of the defined complicated tasks and, most of all, in the part of the confident work at low levels of silver and cadmium. Therefore, it was decided to attract the Aspap Geo LLP (Alma-Ata, Kazakhstan) to cooperation. In very short terms there was developed the OCS and delivered, and from October 2016 to January 2017 three OCS RLP-21T were put into operation at CM-1 ZhOF-1 and CM-2 ZhOF-2. The appearance of the OCS RLP-21T and the location of its installation are shown in Figures 3 and 4.



**Fig. 3.** OCS RLP-21T on conveyor No. 1 and 2 of CM-2 ZhOF-2





OCS RLP-21T (right), RLP-3-02 (left)

OCS RLP-21T

**Fig. 4.** OCS RLP-21T on conveyor No. 1A and OCS RLP-3-02.  
On conveyor No. 1 of CM-1 ZhOF-1

Due to the fact that all OCS were installed on conveyors before electromagnets, special attention was paid to protecting the OCS from impacts of metal that is in the ore.

The main structural elements of the OCS RLP-21T are: an X-ray tube VF-50J/W/S; a high-voltage power supply uX50P50 / XCC; a semiconductor detector XR-100SDD-X-Ray Detector; an ultrasonic distance sensor MaxBotix MB7067; a thermoelectric module Laird Technologies AA-150-24-44-00-XX [8].

The information of all parameters of the OCS is issued both in the form of a short and detailed reports. The important information is given by the report for a shift for the conveyor: how long the conveyor was idle (with ore and without ore), how long it worked (with ore and without ore). The software provides for keeping an electronic "Register of events on the OCS", in which a lot of parameters are recorded, including all cases of metal strikes with ore, by OCS or by its protection.

The innovative approach was also manifested in the fact that the secondary X-ray spectra were measured every second. This ensures the process of continuous testing ore on the conveyor belt, which in other OCS is not available. The contents of silver and cadmium are given out for 40 spectra, copper, zinc, lead and iron for 20 spectra. The average content of the elements in the train composition is generally determined by entering into the dialog box in which the data from the ZhOF-1 and ZhOF-2 server (the Libra database) and the input tools of the OCS operators, the time of the beginning and the end of unloading the train with ore corrected for the time of ore

run through the conveyors. After that a line is automatically generated in the OCS report document. The results of the "on-line" quality control of ores are now available to any user of the corporate network. Since the conveyor belt contains zinc, the time intervals when the conveyor without ore is standing or moving (there is a belt movement sensor) are excluded from the processing.

4. Laboratory spectrometers RLP-21T put in operation no later than 2012 also need to be replaced with more up-to-date modifications for solving new, more complex tasks. In particular, it is necessary to ensure the following: a) all RLP-21T spectrometers of RFA laboratories are to determine the content of silver and light elements; b) the list of elements determined in the RFA process is to include gold (for the RFA laboratory of the Balkhash copper smeltery that receives gold-containing ore).

The first task is solved by simple acquiring the latest modification of the RLP-21T spectrometer that provides the RFA with 38-42 elements.

To solve the second problem, it was necessary to introduce gold into the list of identified elements and to carry out a series of studies on real samples of gold-containing raw materials in order to clarify the lower bound of the confident RFA for gold. The methodical investigation of the task and a series of studies were carried out within the framework of the MISAC project: Modernizing the instrumental system of analytical control at the Balkhash copper smelting plant of the Kazakhmys Smelting LLP. In the course of the studies there were obtained positive results. They are detailed below in the section "The research results".

5. The studies were conducted to determine the feasibility of using nuclear-geophysical technologies to assess the quality of anodic copper in the Zhezkazgan copper smelting plant of LLP "Kazakhmys Smelting", which produced positive results.

### **The research results**

1. *Face RFA for silver.* Before making the final decision, the spectrometer RPP-12T was comprehensively tested at the mining and processing enterprises of the Zhezkazgantsvetmet PA and the Zhezkazgangeology JSC (Figure 5). Let us discuss the test results.



**Fig. 5.** Testing the RPP-12T spectrometer

Table 1 shows the results of testing core samples of one of the wells. The well was selected in such a way that the entire range of silver content in the ores of the Zhezkazgan deposit was presented, and the average silver content at the ore crossing was close to the average silver content in the commodity ore. There is shown a good convergence of the silver content, as well as copper and zinc, according to the RPP-12T and chemical analysis.

**Table 1**

**Results of testing the RPP-12T spectrometer RPP-12T  
on the core of exploratory wells**

Nos	Interval, m			N pct	Cu, %		Zn, %		Ag, ppm	
	from	to	L		chemmm/x/a	RFA	chem	RFA	chem	RFAΦO
1	131.3	132.4	1.1	20	0.93	0.848	0,003	0,0033	50,7	44,7
2	135.4	136.4	1.0	14	0.50	0.443	0,004	0,0033	20,8	17,4
3	136.4	137.5	1.1	16	0.46	0.432	0,004	0,0036	21,3	18,0
4	137.5	138.5	1.0	14	0.43	0.419	0,004	0,0038	16,8	14,7
5	138.5	139.5	1.0	14	0.34	0.316	0,005	0,0038	17,8	19,5
6	139.5	140.5	1.0	16	0.22	0.231	0,005	0,0041	6,1	6,1
7	140.5	141.5	1.0	19	0.42	0.418	0,005	0,0042	22,6	24,8
8	141.5	142.5	1.0	18	0.37	0.579	0,005	0,0064	20,7	23,7
9	142.5	143.5	1.0	17	0.44	0.449	0,004	0,0037	24,6	24,1
10	143.5	144.5	1.0	16	0.20	0.204	0,004	0,0034	6,8	5,9
Average contents, %					0.431	0.434	0.0043	0.0040	20.82	19.89
Error σ, %						0.70		6.98		4.47

The emphasis in studying crushed carload samples of the QCD was made on poor ores with low (<15 ppm) silver content. For each sample there were made 40÷45 ten-second measurements in the measurement mode "Natural". The results of the studies are shown in Table 2. It is shown that the RPP-12T spectrometer confidently determines the low silver content.

**Table 2**

**Results of testing the RPP-12t spectrometer on carload ore samples**

Element	Method of analysis	Mines, quarries, portals							Average
		57	65	Amn	55	Z.Port	Kr-C	65-B	
Cu, %	PIII12T	0.59	0.58	0.97	0.79	0.57	0.90	0.58	<b>0.716</b>
	x/a	0.61	0.63	1.07	0.82	0.54	0.86	0.51	<b>0.720</b>
Ag, ppm	PIII12T	12.72	12.27	23.88	7.64	9.67	8.57	8.65	<b>11.91</b>
	x/a	11.9	11.7	24.2	6.7	9.4	7.8	8.9	<b>11.51</b>

There has been tested the RFA of powder ore samples option. For this purpose RPP-12T has been completed with a special nozzle. The nozzle is put on a cuvette with a sample (Figure 5), RPP-12T is inserted into the nozzle (Figure 1). Such a construction is very stable. The measurements were performed in the Powders mode with an exposure of 30 seconds. The cycle results are shown in Table 3. It is shown that RPP-12T provides high accuracy of RFA for silver at a wide spread (2.9÷176.6 ppm) of its contents. The RFA of powder ore samples option allows testing sludge of blastholes and fan boreholes in the mine, and this is an important element of the "on-line" quality control of ores and a regulator of the mining process.

Table 3 shows the results of testing the RPP-12T spectrometer on geological powder samples of ores. A good convergence of the average results of copper and silver according to RFA data and chemical analysis of these samples is shown.

**Table 3**

**Results of testing the RPP-12T spectrometer for powder samples**

Element	Method	Samples								Average
		1	2	3	4	5	6	7	8	
Cu %	RFA	0.68	0.88	1.43	0.84	3.65	1.33	7.13	3.95	<b>2.486</b>
	chem	0.62	0.91	1.34	0.82	3.61	1.27	7.17	4.02	<b>2.470</b>
Ag, ppm	RFA	2.43	6.12	10.21	15.11	21.98	32.35	94.6	174.4	<b>44.65</b>
	chem	2.9	5.5	8.5	13.1	21.4	32.8	98.3	176.6	<b>44.89</b>

2. *OCS RLP-21T on the ZhOF-1, 2 conveyors.* There has been designed, tested during the bench tests and widely used the optimal scheme for monitoring the OCS parameters, including measurements of the amplitude resolution of the spectrometric path along the  $ZnK\alpha$  line to the empty conveyor; energy calibration of the spectrometer on a control sample containing iron and cadmium; measurements of the reference powder ore sample in a large-sized counter that is suspended under the OCS.

The OCS ARM works with the SQL scales database at "Verkhnyaya" railway station. Remote users receive the information of the OCS RLP-21T through the OCS-Client application.

Let us discuss the results of a complex study performed both in the process of bench tests of the OCS and directly on heavy ZhOF-1 and ZhOF-2 conveyors.

In the OCS RLP-21T the distance from the OCS to the ore on the belt of the conveyor is determined in two ways: a) by an ultrasonic distance sensor (main); b) by the value of scattered radiation from the X-ray tube from the ore on the conveyor. The convergence of the results of the distance estimation by both methods is shown in Table 4. It has been proved by the studies that both methods give close results. This confirms high efficiency of the mathematical apparatus used in the OCS RLP-21T.

**Table 4**

**Convergence of the distances measured by the ultrasonic sensor (USS) and by scattered radiation of the X-ray tube (SR)**

Method	Sensor- sample distance, cm								
	USS	22	32	42	47	52	57	62	67
SR	22.0	32.7	41.0	45.6	52.2	58.8	34.2	68.1	71.4

Table 5 shows the results of studying (at the stage of bench tests) the effect of the height of the OCS RLP-21T suspension on the dynamics of changes in Cu, Zn, Pb, Ag, Cd, Fe contents in one of the powder reference samples.

Table 5

**Dynamics of changing the elements contents in the reference sample  
depending on the height of the OCS RLP-21T suspension**

H, cm	Cu, %	Zn, %	Pb, %	Ag, ppm	Cd, ppm	Fe, %
22	0.778	0.613	0.556	10.96	24.38	6.30
27	0.778	0.608	0.547	10.28	22.72	6.20
32	0.763	0.623	0.556	9.71	22.06	6.07
37	0.788	0.620	0.561	12.18	24.23	5.97
42	0.791	0.625	0.563	10.37	23.79	6.01
47	0.771	0.630	0.567	10.68	25.81	5.96
52	0.776	0.609	0.559	12.18	24.92	6.05
57	0.753	0.602	0.553	10.04	23.17	5.82
62	0.733	0.617	0.540	10.22	24.67	6.08
67	0.784	0.633	0.548	11.08	23.07	6.16
72	0.785	0.617	0.545	10.86	23.02	6.17
Average	0.773	0.618	0.554	10.78	23.80	6.07

The sample was poured into an iron baking tray of a large size, the height of the sample poured into the tray was kept equal throughout its area. The tray with the sample could move vertically in 5 cm increments. It has been proved by the studies that the operation of the OCS RLP-21T depends little on the height of the sensor-sample surface gap. It is this circumstance that is considered the main stumbling block for most OCS used to organize the "on-line" monitoring of the material composition of large lumpy ore. Based on the results of the study, the working height of the OCS RLP-21T suspension was selected equal to 72 cm (up to the lowest point of the deflection of the conveyor belt that was not loaded with ore).

Table 6 compares the average monthly and mean annual (for 2017) copper and silver contents for the OCS RLP-21T on conveyors No. 1 and No. 2 and the discharge of the ZhOF-2 GK-2 classifier. In the same table there is shown the average annual content of copper and silver for conveyor No. 1A and for the discharge of the ZhOF-1 GK-1 classifier. As a result it has been proved on the extensive factual material that: a) the convergence of the average monthly contents of copper and silver on a large-lump ore of the 300 mm class is good; b) the representativeness of the initial ore sampling

data is higher than under the traditional QCD system; c) the "on-line" control of the initial ore is fully realized. All this made it possible to liquidate the points of the QCD testing at the cone crusher of the KKD 1500/1800 type at the ZhOF-2 KD. The point of input ore sampling at the cone crusher KKD 800/160 of the ZhOF-1 KD-1 was closed earlier.

**Table 6**

**Average monthly contents of copper and silver according to the data of the OCS RLP-21T and discharge of the ZhOF-2 GK-2 classifier**

Month of 2017	Cu, %			Ag, ppm		
	QCD	OCS	$\sigma$ , %	QCD	OCS	$\sigma$ , %
January	0.701	0.720	2.71	11.64	11.94	2.58
February	0.772	0.741	4.02	12.09	12.38	2.40
March	0.737	0.743	0.81	11.31	11.82	4.51
April	0.751	0.781	3.99	13.35	13.45	0.75
May	0.738	0.723	2.03	12.47	12.30	1.36
June	0.811	0.842	3.82	12.04	12.25	1.74
July	0.779	0.835	7.19	10.33	11.69	3.48
August	0.739	0.803	8.60	9.74	10.61	8.98
September	0.786	0.767	2.33	13.25	13.40	1.18
October	0.767	0.749	2.35	12.147	12.416	2.21
November	0.788	0.790	0.25	12.468	13.958	11.25
December	0.953	0.940	5.12	17.61	17.189	3.49
The average.ZhOF-2	<b>0.757</b>	<b>0.773</b>	<b>0.77</b>	<b>12.387</b>	<b>12.20</b>	<b>3.20</b>
The average.ZHOF-1	<b>1.037</b>	<b>1.022</b>	<b>1.45</b>	<b>14.408</b>	<b>14.10</b>	<b>2.16</b>

3. *RFA for gold samples of gold ore.* Using the latest modification of RLP-21T (laboratory of RFA LLP "Two Kay", Alma-Ata), two cycles of studies were performed.

Cycle No.1: RFA was carried out for several samples of gold ore coming from the Balkhash copper smeltery from one of the suppliers. No special preparation of samples before the RFA was carried out. Figure 6 shows: a fragment of the instrumental spectrum; the table of the RFA results for one sample in different scales; the "pure" lines  $L\alpha$ ,  $L\beta$ ,  $L\gamma$  of the L-series of gold selected using principles of mathematics. The gold content is defined as  $57.6 \pm 8.9$  ppm with the measurement exposure of 200 s. It has been proved that the positive potential for solving the problem of determining the gold content by the RFA method is at least at its high enough levels.

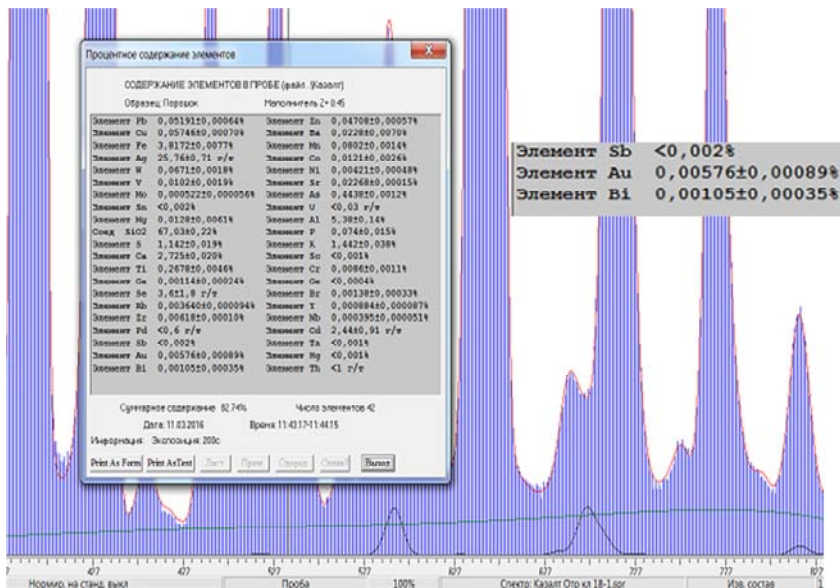


Fig. 6. Table of the RLP-21T RFA results for a gold ore sample

Cycle No.2: there were selected two non-ore samples in which there was no gold at all, and 10 and 20 ppm gold were added to these samples (Shevelev G.A.), respectively. In the RLP-21T there were optimized the conditions for exciting L-series gold lines. In particular, the tellurium target ( $TeK\alpha = 25.3$  keV line) was replaced by a silver target ( $AgK\alpha = 22.2$  keV line), that is, the exciting radiation was close to the lines of the L-series of gold. Tables of the contents of the elements based on the samples RFA results are shown in Figure 7. From these tables it follows: the situation with the RFA for gold has radically improved:  $10.17 \pm 0.60$  g / t and  $22.82 \pm 0.66$  g/t. The possibility of determining gold by the direct X-ray diffraction method (without preliminary chemical decomposition of the sample and concentrating gold with a solid organic extraction agent TVEKS) is proved at lower levels of its contents in ores, which brings organization of the on-line control of imported gold-containing raw materials entering the Balkhash copper smelter to the rank of the solved production tasks.



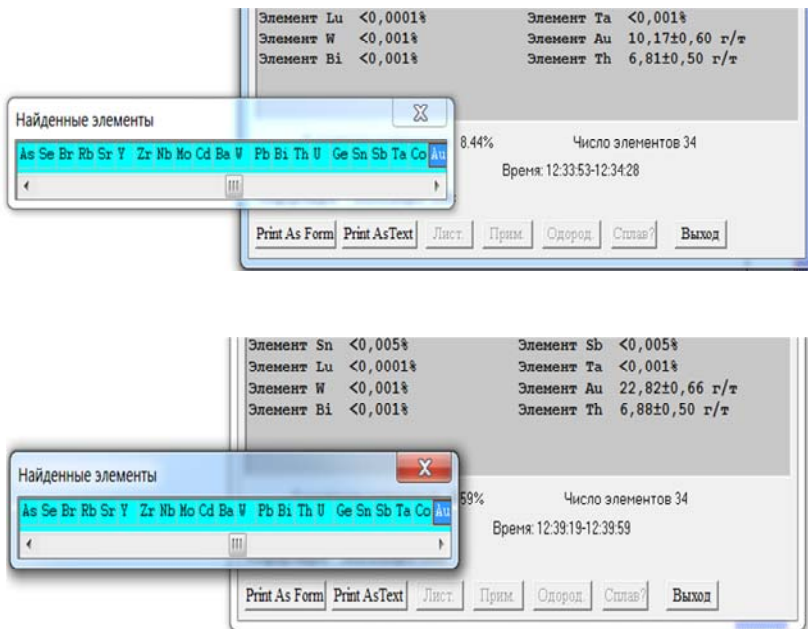


Fig. 7. Tables of the samples RFA results with gold addition

4. *RFA of anodic copper samples.* The studies were carried out on an average daily sample of anodic copper of a cylindrical shape. An EDXRF spectrometer for the local analysis of the RLP-21T (LA) samples was used for the 200 seconds exposure. The RLP-21T (LA) spectrometer, the sample and the results of the RFA of the anodic copper sample are given in Figure 8. There were identified 14 elements in total. According to the standard method, the copper content in the samples of anodic copper was determined by calculation: 100% is the sum of the impurity content.

The convergence of the RFA results and the chemical analysis of the anodic copper sample is shown in Table 7. For a number of elements chemical analyzes are taken from the monthly sample, since in the average daily samples these elements are not determined. This niche can be filled with the RFA on the RLP-21T (LA) spectrometer.

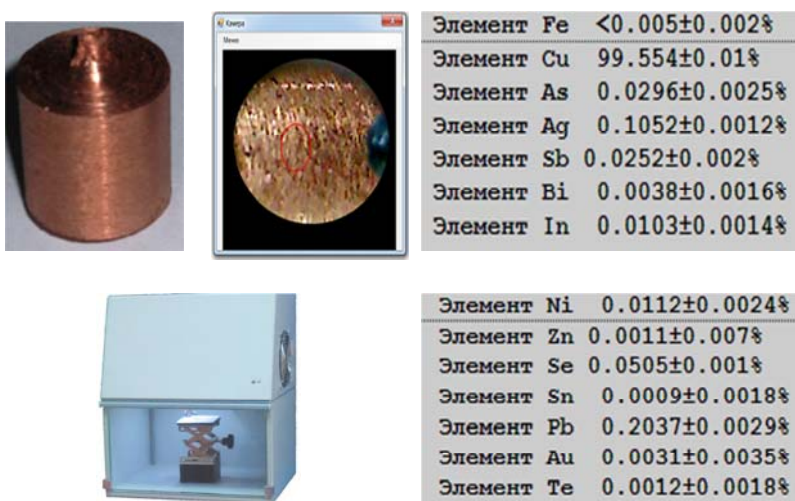


Fig. 8. RFA of the anodic copper sample: a sample, a measurement spot, the RLP-21T, the table of elements contents

Table 7  
Convergence of the results of the chemical analysis and the RFA of the anodic copper sample

Element contents, %	Chemical analysis		RFA	
	Mass fraction C, %	Accuracy factor $\pm \Delta_C$ , %	Mass fraction C, %	Accuracy factor $\pm \Delta_C$ , %
Copper	99.43	0.14	99.554	0.0100
Silver	0.1039	0.0042	0.1052	0.0012
Lead	0.17	0.04	0.2037	0.0029
Arsenic	0.019*	0.004	0.0296	0.0025
Antimony	0.019*	0.004	0.0252	0.002
Selenium	0.057*	0.020	0.0505	0.001
Iron	0.0032*		<0.005	0.002
Bismuth	0.0029*		0.0038	0.0016
Nickel	0.010*		0.0112	0.0024
Tin	<0.0010*		0.0009	0.0018
Zinc	Was not identified		0.0011	0.007
Indium	Was not identified		0.0103	0.0014
Gold	0.00244*		0.0031	0.0035
Tellurium	0.0039*		0.0012	0.0018

## Conclusions

As a result of the long-term scientific, methodological and mathematical studies based on the latest equipment base, there have been developed nuclear-geophysical technologies for the "on-line" quality control of ores, covering mining (in fact, in full), concentrating (partially) and metallurgical (fragmentary) processes of the flagship of non-ferrous metallurgy of Kazakhstan, Kazakhmys Corporation LLP. The technologies ensure determining the contents of all the main and associated ore components, thereby forming all the conditions for controlling the quality of ores by the contents of the main and associated ore components. The detection limits of the associated ore components correspond to these tasks. The possibilities of the "on-line" monitoring of elemental and gross composition of ores and concentrates on the RLP-21T spectrometers exceed many times the possibilities of chemical analysis.

For the developed nuclear-geophysical technologies of the "on-line" quality control of ores and products of their processing there has been selected an optimal analytical complex of the equipment of Kazakhstan manufacture, the portable spectrometers RPP-12, RPP-12T, RPP-12PI, the laboratory spectrometers RLP-21T. These spectrometers are based on the innovative ideology: deposits are different, technological grades of ores are different, enrichment products are different, but calibration of the spectrometers is the same. The task of the "on-line" quality control of ores and products of their processing for silver is completely solved at the stages of mining and concentration processing [3].

At the stage of the enrichment process there has been introduced the on-line technology for controlling the quality of incoming ores for six (Cu, Ag, Pb, Zn, Cd, Fe) elements on all four "heavy" conveyors of ZhOF-1 and ZhOF-2 by means of RLP-21T. This made it possible to abolish the ore sampling points by the QCD on cone crushers of both factories. The contracts have been signed for the delivery of the OCS RLP-21T to the Balkhash and Karagaylinsky concentrating mills, as well as to the conveyor of the Nurkazgan mine-underground (at the mine molybdenum will be determined instead of zinc). The OCS will be put in production in the second and third quarters of this year.

There have been obtained the encouraging results on the "on-line" quality control of gold-containing ore for gold and copper concentrates entering the Balkhash copper smeltery using the direct RFA method (without acid dissolution of samples and sorption of gold on TVEX) using the laboratory EDXRF spectrometer RLP-21T.

Encouraging results on the "on-line" quality control of anodic copper at the Zhezkazgan copper smeltery in the shift and daily samples by X-ray diffraction using the laboratory EDXRF spectrometer RLP-21T (LA). Gold is among the elements of the "on-line" control.

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# THE TECHNOLOGY OF NORMALIZATION OF THE MICROCLIMATE IN THE MINE WORKINGS OF DEEP MINES

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**The aim of the study** is the solution of urgent scientific and technical problems of increasing the efficiency of normalization of the microclimate in the mine workings of deep mines through the development of new scientifically based methods of effective air-cooling using groundwater.

The subject of research regularities of changes of thermodynamic parameters of mine air when it is cooling, irrigation and cleaning in the underground chambers.

**Research methods.** Scientific analysis and synthesis of previously executed theoretical and experimental studies on issues of microclimate normalization during underground mining of ore deposits; theoretical researches and mathematical modeling of heat transfer processes in mines.

**Results.** On the basis of conducted industrial researches it was established that in general thermodynamic processes in ventilation workings are polytropic. The thermal decomposition in polytropic processes at the expense of convective heat exchange in ventilation workings reaches 15 kJ / kg, which leads to an increase in air temperature to 26-30 ° C and deterioration of working conditions. Mathematical modeling is the quantitative and qualitative parameters of the process of normalization of the microclimate in mines, developed technology of irrigation cooling and cleaning the air in underground chambers of mines.

**Scientific novelty.** Set up the pattern to change the temperature of mine air in deep ore mines, which is formed polytropic processes in mine workings and corresponds to the temperature of the rocks plus 1-3 ° C due to the receipt of heat from operating equipment, blasting, process air compression, oxidation, evaporation and hydration tabs.

**Practical significance.** Developed and implemented in deep ore mines cooler and clean mine air with the use of groundwater. The method involves accumulation of groundwater in the upper horizon of the mine, they are cooled to a temperature of 10-11 °C, for termoisolation supply duct into the

chamber irrigation, cleaning and cooling of mine air at 8-10 °C and feeds it into the area of mining works to normalize the microclimate in the mine workings and hydration stone gobbing.

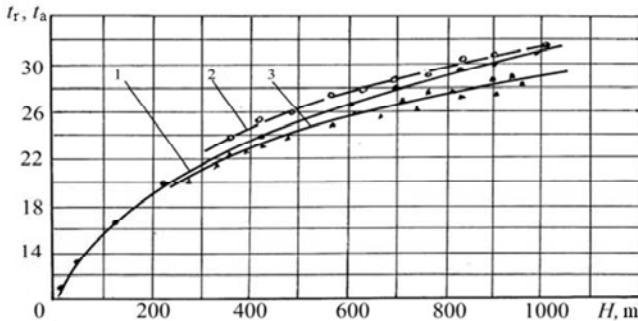
**Key words:** microclimate, mine working, chamber, temperature, underground water, cooling, air purification, thermal mode.

**The urgency of the problem** lies in the fact that exploitation of ores at great depths is accompanied by deterioration of working conditions. At depths of 1500-1700 m, the temperature mine air in mine workings exceeds the permitted value of 26 °C. Adverse weather conditions in mines lead to overheating of the body running, a malfunction of the respiratory system and reduce immunity to diseases. The problem of normalization of the microclimate in the mine workings of deep mines is dedicated to the research of many scientific-research, design and educational institutions. Among which the works of the National Academy of Sciences of Ukraine, State Makeevka Scientific Research Institute, Donetsk Coal Institute, National Mining University, Polyakov Institute of Geotechnical Mechanics of the National Academy of Sciences of Ukraine, State Research Institute of Labor Safety in the Mining Industry, Kryvy Rih National University.

The solution to this problem will allow to improve working conditions in mines, to reduce the risk of overheating of the body, the violation of the functional respiratory system and occupational diseases working in the region.

**Presentation of the basic material.** The results of industrial researches of thermal conditions in mining operations of ore mines are given. The studies used a complex method, which involved: conducting airborne temperature measurements in mines of Kryvy Rih and Zaporizhia iron ore plant; measurement of the temperature of rocks and mine waters in boreholes, wells, working face, the drainage grooves, and mine waters; inspection of ventilation systems of the mines and the work of the main fan. Measurements of air temperature were carried out in the exhaust chambers, in the process of insertion, and in the adjacent mine workings. The measurements used proven standard technique with the use of attorneys and deep electrical mercury thermometers, anemometers APR-2, aspiration psychrometer MV-4M, microbarometer MB-63, microprocessor-

based meters barometric pressure type MBTS-5. In fig. 1 shows graphs of change of temperature of rocks  $t_r$  and air  $t_a$  in working face mines, depending on the depth of the development of  $H$ .



**Fig. 1.** Graphs of the average temperatures of rocks  $t_r$  and air  $t_a$  jobs in working from the depths of development  $H$ : 1 – the results of thermometry of exploration wells; 2 – the results of measurements  $t_r$  in the holes; 3 – the results of  $t_a$  measurements in the workplace, in working face and into the mine workings.

These data indicate that the average temperature of the neutral species layer at a depth of 20-30 m is about 10-11 °C. At a depth of 800-1000 m the temperature of the rocks is 28-30 °C, and the temperature at these depths reaches 29-31 °C.

Temperature of mine air in mine workings depends on the temperature of rocks; work of machines and mechanisms; processes, compression and oxidation air in the workings, evaporation of moisture; heat of hydration stone gobbing in the chambers; blasting and is expressed as follows.

$$t_a = t_r + \Delta t, \quad (1)$$

where  $\Delta t = (1-3)^\circ\text{C}$  – temperature rise due to operation of machinery and other factors.

Factors that determine the condition of a microclimate in mines ore: the efficiency of their ventilation, velocity of air, its temperature and humidity in the workplace. The main method of normalization of the microclimate in mines, where the temperature does not exceed 26 °C is to increase its speed for effective heat factor, which is determined by the formula

$$V_i = k \cdot 0,02(t_a - 17)^2, \quad (2)$$

where  $k$  – coefficient taking into account working:  $k = 1$  for treatment working face,  $k = 0,3$  for a blin-drift, development and main workings;  $t_a$  is the temperature of the mine air, °C.

The evaluation of these parameters was made by comparing the required value with the actual data. Measurements indicate that the security of air and its velocity is 60-80 %, and the temperature there is around 26-31 °C and relative humidity 85-95 %. In such a climate to improve thermal environment in the workplace is only possible with effective regulation of thermodynamic processes in the ventilation network.

In the shaft depending on the season there is a heating or cooling air, increasing its relative humidity, and pressure. In horizontal workings is heat exchange with the wet walls, the increase of humidity, change in air speed, static pressure varies slightly. In mining, which removes exhaust air, there is a decrease in static pressure, which leads to condensation of moisture. Such change of parameters of the air determines the reality of polytropic processes and the ratio polytrope in shafts  $n=(1,0-1,4)$ , and corresponds to the transition from isothermal to adiabatic and is accompanied by a change in air temperature, and horizontal workings figure polytrope close to  $n= 0,0$  and isobaric process corresponds to the stationary distribution of the temperature. Heat in polytropic processes in the workings reach 15 kJ/kg, which leads to a temperature increase in the area of mining works to 29-31 °C and heavy working conditions in working face.

Below are the results of experimental research thermophysical properties of rocks and gobbing materials. The study was performed by the "regular mode" and "stationary source of heat," an improved method for "instant source of heat", by Professor Kondratyev G. M. Using differential equation the coefficients of thermal conductivity  $\lambda$ , W/m·K, thermal diffusivity  $a$ , m<sup>2</sup>/s specific heat, J/kg·K. we studied the rock samples (shale, quartzite hematite-martite ore hematite-martite) and stone gobbing from materials of different composition. The test results indicate that the quartzite gematit-martite have a coefficient of thermal conductivity  $\lambda = 4,95-5,23$  W/m·K and thermal diffusivity  $a \cdot 10^{-7} = 6,4-8,0$  m<sup>2</sup>/s, which allowed to recommend this rock as a filler in the amount of 30-40 % in the manufacture of the stone gobbing instead of sand, which has a



coefficient of thermal conductivity  $\lambda = 0,81 \text{ W/m}\cdot\text{K}$ .

Thermophysical data of samples of stowing with the addition of 30 % of the rock quartzite hematite-martite instead of sand, the works of Professor V. L.Sakhnovsky, are significantly different. Thus, the thermal conductivity  $\lambda$  increased by an average of 1,7 times, the diffusivity and increased on average 2,2 times, resulting in a rapid transfer of heat into the environment and reducing the temperature of hardening gob.

A pattern to change the temperature of mine air in deep mines has been investigated by means of mathematical modeling of heat transfer processes in mines, which produce the change of microclimate parameters. This approach allows through the use of rapid identification of parameters of mathematical models of heat transfer processes to carry out forecasting of the thermal regime of the mine and its regulation by calculating and implementing the appropriate control action. To build mathematical models of heat transfer processes is composed of the General scheme of ventilation of mine workings (Fig. 2).

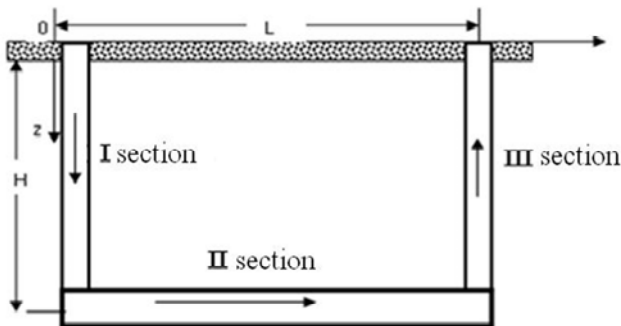


Fig. 2. The scheme of movement of air in mine workings

According to the scheme, the air moves down along the vertical shaft at a depth of  $H$  (I section) and then moves the horizontal workings of length  $L$  (II section), then rises up a vertical trunk (III section). On the section I allocated to two areas: first, the temperature change takes place without condensation and the second condensation is taken into account. For the two core sections  $z = z_1$  and  $z = z_2$  the rate of change in time of heat will be

$$\Delta W_c = \int_{z_1}^{z_2} c_1 \cdot \gamma_1 \cdot f_1 \cdot \frac{\partial t_1}{\partial \tau} dz, \quad (3)$$

where specific heat of air, j/kg °K;

$\gamma_1 = \gamma(z)$  – air density, kg/m<sup>3</sup>;  $t_1 = t(z, \tau)$  – air temperature, °C;  $\tau$  – time, s;  $f_1$  – is the cross sectional area of the shaft, m<sup>2</sup>.

In turn, the formula (3) consists of the following parts:

$$k_1 \cdot U_1(t_1 - t_m) + c_1 \cdot \gamma_1 \cdot f_1 \cdot \frac{\partial t_1}{\partial \tau} + \frac{\partial}{\partial z}(c_1 \cdot \gamma_1 \cdot f_1 \cdot w_1 \cdot t_1) = f_1 \cdot W_1, \quad (4)$$

where  $k_1$  – transfer coefficient in the shaft, W/(m<sup>2</sup>·°C);  $t_m$  – temperature rock mass around the shaft at a given depth, °C;  $W_1 = W(z, \tau)$  is the density of heat sources in W/m<sup>3</sup>.

In equation (4) the first component corresponds to the amount of heat per unit time which enters through the lateral surface of the shaft due to heat exchange with the external environment. The second component, referred to the unit of time the amount of heat coming from sources in the section that is considered. The third component is related to the unit of time the amount of heat that enters through the cross section of the bore due to the movement of air.

Condensation of water vapor leads to the allocation of heat, the density of which sources can be determined by the formula:

$$W_c = r \cdot \gamma_1 \cdot d'(t_1) \cdot \left( \frac{\partial t_1}{\partial \tau} + w_1 \frac{\partial t_1}{\partial z} \right), \quad (5)$$

where  $d'(t_1) = \frac{d}{dt_1}(d(t_1))$  – derivative of moisture content, temperature, kg/kg °C. To obtain the mathematical model of heat exchange process in the second region of the shaft is necessary in equation (4) to take into account the density of the sources (apparently, heat dissipation when condensing water vapor), is determined by the formula (5). After the developed mathematical model the got results of calculations of temperature of t, and also relative humidity  $\varphi$  and to pressure P, executed as a result of measurings for shaft «Exploited» resulted on fig. 3.



saturation with moisture of air, the cooling process is direct and occurs to the point A (first stage cooling). Then there is a decrease in temperature to full saturation (point P). Increasing the efficiency of cooling in irrigation chambers is possible by the use of additional means, for example, due to the condenser and water-air mixture.

The efficiency of the cooling chamber irrigation was investigated by mathematical modeling. So, the heat flow is given by the air drops of water will:

$$Q_a = mc_1(T_1 - T_{1f}) \quad (6)$$

and the amount of heat that is water droplets from the air is determined by the formula:

$$Q_w = Mc_2[T_2(t) - T_{2f}] \quad (7)$$

where M and m – the mass expense of air and water respectively through the nozzle, kg/s;  $T_1$ ,  $T_{1f}$  – initial and final temperature, °C;  $T_2(t)$  is the temperature drop at time (t), is determined by the formula:

$$T_2(t) = T_1(1 - e^{-\alpha}) + T_{2f}e^{-\alpha}, \quad (8)$$

where,  $T_{2f}$  – final temperature of water, °C;  $C_1$ ,  $C_2$  – heat capacity air and water, J/kg·K.

On the basis of the law of conservation of energy final temperature air will be:

$$T_{1f} = T_1 - \frac{Mc_2}{m_1c_1}(1 - e^{-\alpha}) + (T_{2f} - T_2) \quad (9)$$

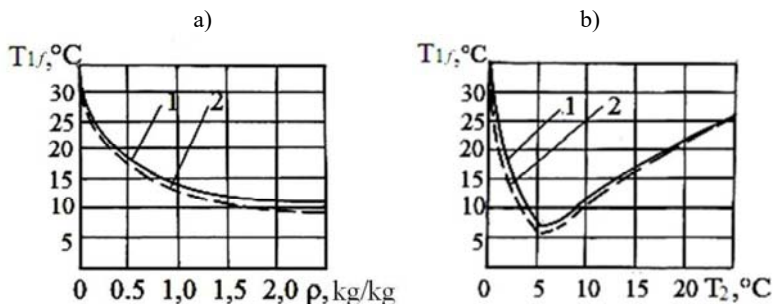
where  $\alpha$  is the relative time setting of the heating water drops ; a

$\alpha = \frac{t}{\tau} = \frac{A}{c_2M}$ ; A - is the heat transfer coefficient of the air and  $\lambda$

$A = \frac{Nu\lambda}{2R}$ , where Nu – Nusselt number and thermal conductivity of

water; R- is the radius of the droplet.

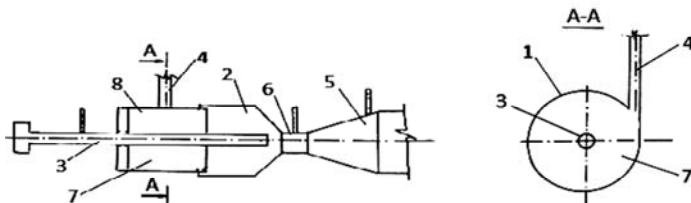
To improve the efficiency of cooling at irrigation with water should increase the water consumption, or reduce its temperature. On the basis of calculations by formula (9) plotted temperature  $T_{1f}$  to the water temperature  $T_2$  that is used for irrigation (fig. 5 a) and the extent of irrigation  $\rho = M/m$ , kg/kg (fig. 5 b).



**Fig. 5 a, b.** Plots of the finite temperature  $T_{1/2}$ ; temperature of water,  $T_2$  (a) and the extent of irrigation  $\rho$  (b): 1 – incomplete (intermediate) saturation of air by water; 2 – at full saturation of air with water

As evidenced by the results of the calculations (fig. 5 a, b), intense drop in temperature occurs when irrigation water with a temperature of  $T_2 \leq 5^\circ\text{C}$  (fig. 5 a) and the ratio of the irrigation  $\rho \leq 1,2 \text{ kg/kg}$  (fig. 5 b). A further increase of the coefficient of irrigation  $\rho$  has little effect on the decrease in air temperature.

Irrigation cooling of the air using water-air of the mixture was carried out using cooling ejector "Dispersed", whose scheme is shown in fig. 6.

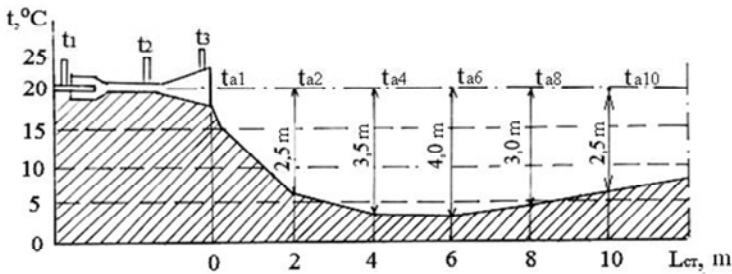


**Fig. 6.** Schematic diagram of the cooling ejector "Dispersed":  
1 – cylindrical body; 2 – confuser; 3 – nozzle; 4 – water nipple;  
5 – diffuser-mixer; 6 – cylindrical coupler; 7 – cavity of the housing

The formation of a water-air mixture is as follows. Compressed air with pressure of 0,7 MPa, which is supplied from the nozzle 3 is compressed to a pressure of 0,5 MPa, causes a vacuum in confusing of the housing 2. Due to this water pipe 4 and the cavity of the housing 7, the water enters the cylindrical coupler 6 where it is ejection and spraying with the formation of water-air mixture in diffuser-mixer 5. Air cooling occurs by adiabatic expansion of

compressed air during its exit from the nozzle 3 and the diffuser 5. Water-air mixture temperature below ambient air temperature by 3-4 times. Torch water-air mixture is expanded at the exit of the diffuser and eject the surrounding air and cool super by convective heat transfer and evaporation of water droplets.

Based on the analysis of studies of the effectiveness of the cooling air water-air mixture formed by cooling the ejector "Dispersed" the resulting dynamics of its temperature (Fig.7).



**Fig. 7.** The dynamics of the temperature along the length of the torch water-air mixture:  $t_1, t_2, t_3$  – temperature inside the "Dispersed";  $t_{a1}, t_{a2}$  – average temperatures along the length of the torch water-air mixture.

As can be seen, in the diffuser, a decrease in temperature from 20 °C to 17 °C, and at a distance of 4-6 m from it the temperature drops to 5°C. Is the temperature observed at a distance of 8-10 m from the diffuser, and it further increased to 8 °C and at a distance of 25 m remains close to ambient temperature. Water-air the effectiveness of cooling was determined by mathematical modeling. It is possible to find such parameters of the device, which provide optimum cooling of mine air. Structure of the mathematical model that describes the excess temperature at an arbitrary point of the cooled compact jet emanating from a circular hole of the diffuser has the form:

$$\Delta T(r, x) = \Delta T_n \cdot a \cdot \left( \frac{r_0}{x} \right)^b \cdot e^{c \left( \frac{r}{x} \right)^2}, (2 \leq x \leq 12) \quad (10)$$

where  $\Delta T(r, x) = T(r, x) - T_0$  ;  $\Delta T_n = T_n - T_0$  ;  $T(r, x)$  – the absolute temperature at the point of the jet  $(r, x)$ ,  $K$ ;  $t_0$  is the absolute ambient temperature;  $T_n$  is the average absolute temperature of the jet

at the exit of the diffuser,  $K$ ;  $r_0$  – the radius of the outlet of the diffuser,  $m$ ;  $a, b, c$  – numeric parameters.

To obtain the numerical values of the parameters  $a, b$ , use the results of the experiments. In the calculations was taken  $t_0 = 298 K$ ,  $T_n = 293 K$ ,  $r_0 = 0,3 m$ .

The result of the calculations was obtained the following values  $a = 8,365$ ;  $b = 2,104$ ;  $c = -2,197$ . The coefficient of multiple correlation was equal to  $R = 0,955$ , indicating a close correlation.

Taking into account the obtained values, the formula (10) takes the form:

$$r = x \sqrt{0,455 \ln \left( 4294,1 \frac{\Delta T_n}{\Delta T} \left( \frac{r_0}{x} \right)^{2,104} \right)} \quad (11)$$

The mathematical modeling of heat transfer processes water-air cooling of mine atmosphere allows organizing the computational experiment that gives the opportunity to explore the peculiarities and the efficiency of the process.

The calculation according to the formula shows the practical convergence of the test data and the results of the calculations. It allows investigating the temperature field of the jet. First of all, you can write the equation of the isotherms, i.e. lines of equal temperature jet

The calculations indicate that the cooling range of the jet reaches  $x_{\max} = 20 m$  and the greatest thickness is  $2 \cdot r_{\max} = 17 m$ .

As a result of the computational experiment heat transfer processes water-air cooling of mine atmosphere obtained the necessary parameters of the jet.

Experimental tests cooling ejector "Dispersed" occurred at the site of the mine to them. Lenin Public Joint Stock Company (PJSC) "KRIVOJ ROG'S IRON-ORE COMBINE" (Fig. 8).

The test procedure was provided to determine the dependence of the technical parameters of the installation of the air pressure in the line  $P_a, MPa$ , and also on geometrical parameters such as the diameter of the outlet of the diffuser  $d_d$  and the width of the annular hole for the release of the water-air mixture  $l_a$ . The number water-air of the mixture was determined as follows. First measured the speed  $V_m$  at the exit of the diffuser anemometer APR-2, then by the

obtained speed and the cross-section area of the diffuser  $S_d$  found a number water-air mixture  $Q_m$ . Thus, we measured the pressure in the supply piping with a pressure gauge. The length of the jet of the water-air mixture was determined by measuring the length  $L_j$ . The diameter of the droplets was determined by collecting them on a glass, greased with vaseline, with subsequent determination of their diameter under a microscope.



**Fig. 8.** Test cooling ejector "Dispersed" on the site of the mine to them. Lenin, PJSC "KRIVOJ ROG'S IRON-ORE COMBINE".

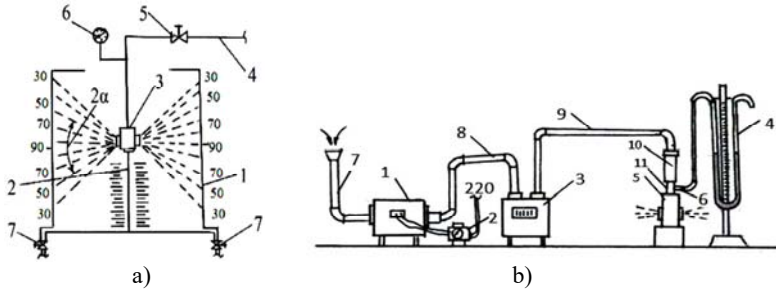
All were tested 4 types of units that differ in the length and the diameter of the mixing chambers. The test results show that with increasing air pressure increases the length of the jet of installation. So, if you increase the pressure  $P_a$  from 0,1 to 0,8 MPa length of the jet,  $L_{rj}$  increases from 6,5 to 36 m.

Analysis of test results shows that with the air pressure in the line  $P_a = 0,8 \text{ MPa}$  length of the jet is  $L_r = 34-36 \text{ m}$ .

During the test, the average droplet diameter  $d_{dr}$  ranged from 25 to 140 microns. The diameters of such droplets, mist, promote efficient evaporation and cooling air flow.

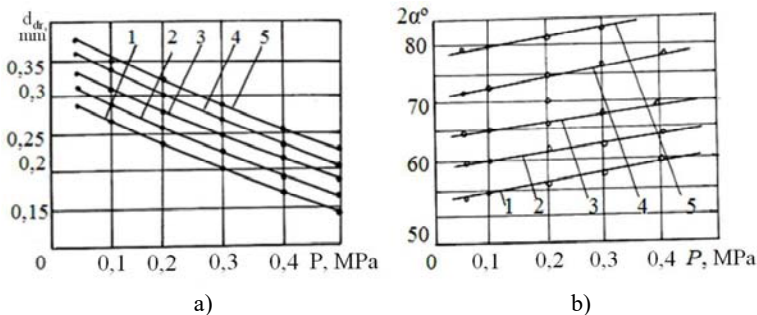
Research bidirectional nozzle and ejector "Dispersed" carried out with the help of stands (Fig. 9). During the test was determined: the opening angle of the torch nozzle  $2\alpha$ , dispersion of water droplets  $d_{dr}$ , the local resistance coefficient  $\xi$  depending on the pressures of water and air in the lines ( $P_w$  and  $P_a$ ) and the diameter of the injector nozzle  $d_i$ . During testing, the injector nozzle diameters were taken from 1 to 5 mm, and pressure of air and water before the nozzle was changed in the range from 1 to 5 kgf/cm<sup>2</sup>.





**Fig. 9.** Schemes of experimental setups for the study of the technical and aerodynamic parameters of a bidirectional tangential nozzle: a) scheme of stand to determine the technical parameters of the injectors: 1 – capacity for water collection; 2 – dividing wall; 3 – dual-sided nozzle; 4 – water pipeline; 5 – valve; 6 – manometer; 7 – drain cranes; b) scheme of stand to determine the hydraulic parameters of the nozzles: 1 – traction exciter; 2 – voltage regulator; 3 – gas meter; 4 – manometer; 5 – nozzle; 6 – a tube static pressure; 7, 8, 9 – flexible connection hoses; 10, 11 – water nipples.

The results of experimental studies of nozzle parameters are shown in Fig. 10.



**Fig. 10.** Based on the average dispersion of droplets and spray angle of water pressure before the nozzle: a) the dependence of the dispersion drops from water pressure: 1, 2, 3, 4, 5 – nozzle diameter respectively 1.5; 2; 3; 4; 5 mm; b) dependence of the angle of spray droplets from the water pressure in the supply piping before the injector with a nozzle diameter of respectively 1.5; 2; 3; 4; 5 mm

As you can see, the diameter of drops decreases with increasing pressure of water in front of the nozzle. Thus, with a diameter of a nozzle of 1.5 mm, the diameter of the droplets decreases from 0,3 to 0,15 mm, and at the diameter of the nozzle 5 mm - decreases from 0,4 to 0,25 mm.

The spray nozzle, on the contrary, increases the pressure of water in front of the nozzle from 1 to 5 kgf/cm<sup>2</sup> with a nozzle diameter of 1,5 mm, increasing from 55 to 60°, and with a diameter of the nozzle of 5 mm, the torch of sawing increases from 80 to 90°.

The results of experimental tests on the stand (Fig. 9a) showed that for reduce humidity of air after cooling it in the chamber irrigation (after the first stage of cooling) is apply cameras of irrigation, it is advisable to apply a tangential nozzle bilateral water spray with a nozzle diameter of  $d_n = 3 \text{ mm}$ .

Determinations of aerodynamic parameters of the nozzle were carried out on the stand (Fig. 9b). The test procedure provided for the determination of local resistance coefficient for both single and double nozzles. For each experiment, during  $t_e$  with fixed initial  $N_{in}$  ( $m^3$ ) and destination  $N_f$  ( $m^3$ ) of gas meter readings. Air flow that passed through the injector was calculated by the formula  $Q_{inj} = (N_f - N_{in})/t_e$ , where  $t_e$  is the time of the experiment, (s). In the inlet nipple with a diameter of 5/8" (with metric diameter  $d = 1,59 \cdot 10^{-2} \text{ m}$ ) and a section  $S = 1,99 \cdot 10^{-4} \text{ m}^2$  first, determine the air velocity by the formula  $V = Q/S$ , and then calculated the Reynolds number  $Re = d \cdot V / 15 \cdot 10^{-6} = 10,60 \cdot V$ .

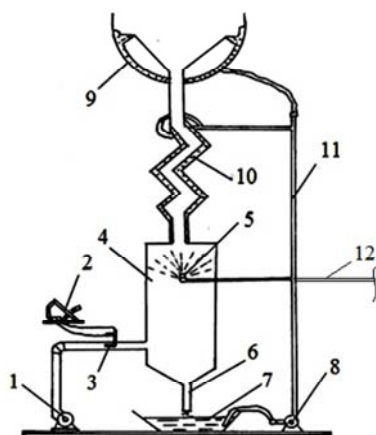
For the experimental conditions ( $t = 26 \text{ }^\circ\text{C}$  and  $P_{atm.} = 754 \text{ mm Hg}$ ) density of air amounted  $\rho = 1,17 \text{ kg/m}^3$ . On the basis of the difference between the heights of the liquid column in the water  $h$  U-shaped manometer calculated static pressure in the inlet nipple  $P_{st.} = \rho gh$  and the local resistance coefficient  $\xi = 2P_{st.}/(\rho V)^2$ . Studies were carried out: bilateral nozzle exit openings  $d \approx 3,5 \pm 0,1 \text{ mm}$  with a swirler in the form of cones (type 1); two-sided nozzle with  $d \approx 3,5 \pm 0,1 \text{ mm}$  without the swirler (type 2); nozzle unilateral action  $d \approx 3,5 \pm 0,1 \text{ mm}$  with guide cone (type 3).

Analysis of the results indicates that for each type of nozzle (single-sided, double-sided with the swirler, double-sided without the swirler) there is a correlation between the local resistance coefficient  $\xi$  and the number of Re. This connection is established by using the regression equation of the form  $\xi = aRe + b$ , where  $a$  and  $b$  are coefficients, determined by solving the system of equations obtained by the method of least squares.

$$\begin{cases} a \sum_{i=1}^{10} \xi_i^2 + b \sum_{i=1}^{10} \xi_i = \sum_{i=1}^{10} \xi_i \text{Re}_i \\ a \sum_{i=1}^{10} \xi_i + 10b = \sum_{i=1}^{10} \text{Re}_i \end{cases}$$

The test results of three types of tangential nozzles indicate that the smallest coefficient of the local resistance, on average,  $\xi_1=237$ , has the nozzle bidirectional action with an orifice diameter of  $d \approx 3,5 \pm 0,1$  mm with a swirler in the torsion chamber (type 1).

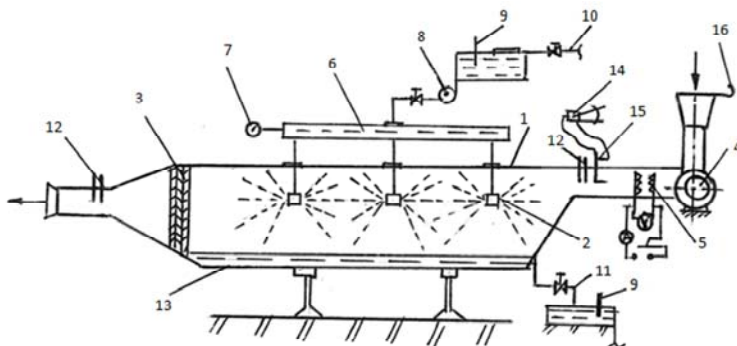
To reduce humidity of air after cooling it in the chamber irrigation (after the first stage of cooling) is applied to the contact condenser, the study which was carried out on models made of hollow corrugated elements, which circulate water (Fig. 11)



**Fig. 11.** Diagram of model for investigating the efficiency of the contact condenser: 1 – fan; 2 – micro-manometer MMN; 3 – air tube; 4 – the chamber of irrigation; 5 – nozzle; 6 – outlet nipple; 7 – under-pan; 8 – pump; 9 – device for the formation of film water; 10 – elements of the condenser; 11 – pipeline; 12 – main pipeline.

The results of studies on the efficiency of the contact capacitor show that the greatest decrease in temperature  $\Delta t$  is observed at air velocity up to  $2$  m/s and the distance between the elements  $b=15-25$  mm. Under these conditions, in the contact condenser the temperature drops by  $\Delta t = 2,5$  to  $3,0$  °C, and the amount of moisture in the air decreases  $t_o$  the level of  $d_m = 0,45-0,22$  kg/kg.

The cooling efficiency in the chamber irrigation was studied using the laboratory setup shown in Fig. 12.

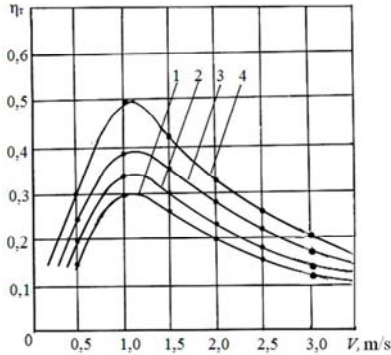


**Fig. 12.** Principal scheme of a laboratory installation for studying the effectiveness of irrigation cooling air: 1 – irrigation chamber; 2 – nozzles; 3 – condenser; 4 – fan; 5 – electric heater; 6 – collector; 7 – manometer; 8 – pump; 9 – mercury thermometer; 10 – inlet pipe; 11 – outlet pipe; 12 – dry and wet thermometers; 13 – under-pan; 14 – micro-manometer MMN; 15 – pneumometric tube; 16 – latch

In the irrigation chamber 1, there are three rows of nozzles 2 unilateral and bilateral actions. At the outlet of the chamber is the condenser 3. The air supplied by the fan 4 and heated by electric heater 5. Water supply for nozzles 2 was carried out through the collector 6 with the inlet pipe 10, and the temperature and humidity of the air was measured by thermometers and psychrometers 12, the temperature of the water coming from the collector 6 and is drained from the sump 13 into the container and measured using mercury thermometers 9. Airflows, and therefore the speed of his in the chamber irrigation 1 were varied with the help of the latch 16.

The experimental data were calculated the degree and the irrigation density according to the formulas respectively,  $\rho = G_w/G_a$ ,  $kg/kg$  and  $\mu = G_w/F_c$ ,  $kg/m^2$ , where  $G_w$ ,  $G_a$  – consumption of water and air, respectively,  $kg$ ;  $F_c = 0,3 \times 0,35 m^2$  – section plane of the chamber irrigation. Temperature and relative humidity of the air entering the chamber was varied within  $t_a = 25-35 \text{ }^\circ C$  and relative humidity  $\varphi$  in the range of 62-65 %. The temperature of the water entering the chamber irrigation was maintained within  $t_w = 4-7 \text{ }^\circ C$  by the use of ice, and the degree of irrigation  $\rho = 0,1-1,24 kg/kg$ . Water pressure before the nozzle was 0,3–0,4 MPa, and the pressure drop of the laboratory setup  $\Delta P$  was changed in the range of 13,7-117,6 Pa.

For fig. 13 shows the results of the experiments with the value of the cooling efficiency of air in the chamber irrigation obtained using the laboratory setup (fig. 12).



**Fig. 13.** The dependence of the magnitude of thermal coefficient  $\eta_T$  from the air speed in the chamber: 1 – one row of the unilateral nozzles; 2 – one a number of bilateral nozzles; 3 – two rows of unilateral nozzles; 4 – two rows bilateral nozzles

Processing of the results of laboratory tests using the methods of mathematical statistics dependence for determining thermal efficiency ratio of  $\eta_T$  cooling air in the chamber irrigation

$$\eta_T = \frac{kV^{-0.5}\rho^{0.9}\Delta t_a m_a c_a}{\Delta t_w m_w c_w}, \quad (12)$$

where  $k$  – coefficient characterizing the design of the chamber irrigation, the value of which is determined depending on the number of rows of nozzles, diameter of nozzles and heat losses through the walls;  $V$  – velocity of air in the chamber,  $m/s$ ;  $\rho$  is the degree of irrigation,  $kg/kg$ ;  $\Delta t_a$  – temperature difference between the air inlet ( $t_{in}$ ) and the output ( $t_{out}$ ) from the chamber,  $^{\circ}C$ ;  $\Delta t_w$  – temperature difference of water entering the injector ( $t_{win}$ ) and after irrigation ( $t_{wout}$ ),  $^{\circ}C$ ;  $m_a$  – the mass of air,  $kg$ ;  $m_w$  – mass of water,  $kg$ ;  $c_a$ ,  $c_w$  – heat capacity air and water,  $J/kg \cdot K$ .

For fig. 14 shows a technological scheme of the cooling of mine air in the chamber irrigation using groundwater that accumulated in the main reservoir 11 formed in the mine working 10 on the upper horizon of the mine. In the main reservoir 11 is pumped mine water from the auxiliary reservoirs 13. In the main reservoir 11 mine water cooled to 1-13  $^{\circ}C$  after mechanical treatment in a sand filter 12



about 18 million m<sup>3</sup> mine water in the calorimetry Results indicate that the specific heat capacity mine water (of 3.81 kJ/kgK) below the drinking water (3,92 kJ/kgK), but it is sufficient for the irrigation cooling of mine air.

When using water-air mixture using a cooling nozzle for the normalization of the microclimate in the ore mines developed the installation "the Oasis", which consists of trunk water and air pipeline, from two to four rows of nozzles, which are mounted hydraulic injector bidirectional action.

For air cooling is used, both horizontal and vertical chambers irrigation length  $L_{\text{cham}} = 10d_{\text{cham}}$ , where  $d_{\text{cham}}$  is the equivalent diameter of the chamber, which is determined by the formula

$$d_{\text{cham}} = 2 \sqrt{\frac{Q_a}{\pi V_{\text{dr}}}}, \quad (13)$$

where  $Q_a$  – the amount of air that enters the chamber irrigation, m<sup>3</sup>/s;  $V_{\text{dr}}$  – speed of free fall of drops of water, m/s.

Installation testing "Oasis" was carried out in 2 stages. The first phase of testing was carried out with cooling air, the spray water which is all collected from nozzles. The second stage is the cooling of the formed unit "Dispersed" water-air mixture coming from the nozzles. Application water-air mixture is recommended for cooling of mine air, whose temperature is above 30°C. The results of industrial tests show that the cooling of mine air by irrigation of mine water, a decrease in its temperature by 5-6°C, and using water-air mixture cooling temperature decreases to 9-11°C.

In order to control the process of cooling the air in the irrigation chamber, a program for a personal computer has been developed that allows you to calculate the final temperature of the cooling of the mine air in the irrigation chambers and the required amount of air entering the mining zone.

The program implies the application of formula (9) and constants: the initial specific heat of air and water, respectively,  $C_a = 1$  kJ / kg·K;  $C_w = 3,81$  kJ/kg·K; coefficient of thermal conductivity of water  $\lambda_w = 0,6$  W/m·K; coefficient of heat transfer per unit time  $\alpha = A/C_w m_w$ , where parameter  $A = Nu\lambda/2R$ ;  $m_w$  - mass flow of water, kg/s;  $Nu$  – Nusselt number;  $R$  - radius of the drop, mm.

The values of the variables are taken within: mass air flow  $m_a = 0,4-0,8$  kg/s; Reynolds number  $1 < Re < 10^4$ ; Prandtl number  $1 < Pr < 400$ ; radius of drops  $R = 0,1-3,0$  mm; mass flow of water  $m_w = 0,1-0,2$  kg/s; air temperature  $T_a = 25-35^\circ\text{C}$ ; the temperature of the water going to the nozzles,  $T_w = 11-13^\circ\text{C}$ . After entering all the parameters the program automatically determines the temperature of the air depending on its massive costs. Application of this program makes it possible, by adjusting the parameters of water or water-air mixture to maintain the temperature in the irrigation chamber at the level of  $20-22^\circ\text{C}$ . After entering the specified parameters, the program automatically builds the diagram.

The block diagram of the air conditioning program is implemented as follows:

1. Determine the amount of air  $Q_g$ , its temperature  $t_g$ , and humidity  $\varphi$ , in the mining area.

2. Identify the normative parameters of air in the mining area: the amount of air  $Q_n$ , which would ensure the removal of excess heat, the normative temperature  $t_n$  and humidity  $\varphi_n$ .

3. Match the output (actual) data of mine air in the area of mining operations with its normative values in the workplace.

4. Determine the required amount of air for the mining area, which should not be less than the normative value  $Q_g \geq Q_n$ .

5. Determine the required air temperature for the mining area, which should not be greater than the normative value  $t_g \leq t_n$ .

6. Determine the required air humidity, which should not be greater than the normative value  $\varphi_g \leq \varphi_n$ .

7. Create the air of the required condition.

8. Provide cooled and drained air to the mining area and carry it out to reduce the temperature and humidity of the air in the mining area to the standard value:  $Q_n, t_n, \varphi_n$ .

The experience of using mine water for irrigation of mine air proves that the main accumulating reservoirs is expediently located on the upper horizon in an array of rocks adjacent to a neutral layer of land and have a temperature of  $11-13^\circ\text{C}$ . The vertical or horizontal irrigation chamber is located closer to the mining area to prevent the heating of cooled air.

Taking into account heat exchange processes in mining operations, the amount of air necessary for the normalization of



microclimate in deep mines is determined by the effective rate of air movement adopted by the thermal factor:

$$Q_a = (\Sigma Q_{ma} + \Sigma Q_{sh} + \Sigma Q_{ch} + Q_{ca})K_m, \quad (14)$$

where  $Q_{ma}$ ,  $Q_{sh}$ ,  $Q_{ch}$ , respectively, the largest, calculated by the thermal factor of the air flow for ventilation of mining activity, shafts and chambers working equipment,  $m^3/s$ ;  $Q_{ca}$  - consumption of compressed air,  $m^3/s$ ;  $K_{gr} = 1,5-1,7$ - general factor of air reserve.

The general depression of the ventilation network of the mine is determined by the formula:

$$h_n = R_n Q_a^2, Pa \quad (15)$$

where  $R_n$  - is the value of the general aerodynamic resistance of the mine network,  $N \cdot s^2/m^8$ .

The microclimate of deep mines has the value of natural draft, which is determined by the formula:

$$h_{nd} = H \cdot g(\rho_{av1} - \rho_{av2}), Pa \quad (16)$$

where  $g$  - is the acceleration of free fall,  $m/s^2$ ,  $\rho_{av1}$ ,  $\rho_{av2}$  - the average density of air pillars in downcast shaft joint ventilation shaft,  $kg/m^3$ ;  $H$  - depth of location of the main working horizon,  $m$ .

The analysis of technical and economic indicators of the method of microclimate normalization with the use of refrigeration machines and the use of the recommended cooling technology of mine air with the help of mine waters allows obtaining the following results.

Calculation of the cost of cooling the mine air supplied to the mining activity, in the amount of  $55 m^3/s$ , by the method of irrigation cooling in comparison with the cooling of air by refrigerating machines of the type air conditioner mobile shaft CMS-300 evidence of its obvious efficiency. Thus, the material costs of purchasing equipment for cooling a given air quantity ( $55 m^3/s$ ) by refrigeration machines type CMS-300 in the amount of 7 pieces make up 2.8 million \$ C, which significantly exceeds the total amount of material costs for the implementation of the proposed method, which is estimated: the axial fan type AFE - 1000 \$; equipment for the installation of the "Oasis" installation - 63500 \$. The total material cost of acquisition, taking into account the additional costs, is 167 000 \$, which is 17 times less than the cost of cooling the mine air with the help of refrigerating machines.

The energy consumption when cooling the air by refrigeration machines is (75 kW/h multiplied by 7 machines = 525 kW / h) by 30 \$/h, and the cooling of the miner air in the proposed method involves compressed air consumption in the amount of 0,1-0,3 m<sup>3</sup>/s, cost 0,02 \$/m<sup>3</sup>, thus the cost of energy consumption is 6,1-17,7 \$/h. On this basis, the energy costs for the implementation of the recommended method are 1,7 times smaller than the refrigeration machines.

Irrigation chambers in mines workings in close proximity to the mining activity, and they are performed as a horizontal or vertical through-working. The air velocity in the irrigation chambers for its efficient cooling should be maintained within the range of  $V = 1-2,5$  m/s. At the temperature of the mine air in the mining activity up to 30° C it is expedient to use a system of irrigation cooling in the chambers, and at air temperature more than 30°C - use irrigation cooling using a water-air mixture, formed using the ejector "Dispersed". The cost of compressed air per one ejector is 0,1-0,3 m<sup>3</sup>/s, and the pressure in the compressed air line is 0,4-0,6 MPa. Number of row of nozzles in the irrigation chamber is accepted  $\geq 2$ . The uses of an accumulation reservoir allows the accumulation of mine water in the amount of 1200-1500 m<sup>3</sup> with a temperature of 11-13°C, and then use them to cool the mine air and irrigate the technological processes. The rest of the mine water is pumped off at night at a reduced tariff for electricity costs. This allows to improve the working conditions and reduce the annual cost of pumping out mine water within the limits of 45000-92000 \$ per mine.

Given the large amount of mine water that is pumped out from underground water bodies, the developed method of microclimate normalization also has significant economic expediency.

## Summary

1. The results show that when the velocity of air in the chamber irrigation within the  $V = 1-2,5$  m/s and the degree of irrigation  $\rho = 0,7$  to 1,2 kg/kg, cooling efficiency, specific heat ratio, is  $\eta = 0,31$  and 0,52, and the value of reducing air temperature is in the range  $\Delta t = 5-20$  °C. Increasing the cooling efficiency, this is expressed by the thermal coefficient, achieved through the use of underground cooled water.

2. Industrial tests of the irrigation cooling of mine air with the use of underground waters in conditions of mines of (PJSC) "KRIVOJ ROG'S IRON-ORE COMBINE" showed the possibility of reducing air temperature by an average of 8 – 10 ° C and humidity of 60 – 70 %.

3. The accumulation of underground water waste in the mine workings adjacent to the neutral layer of the earth, allows cooling it to 10 – 11 ° C with subsequent use for conditioning mine air.

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# **DESIGNING RECLAMATION OF POST-MINING AREAS. THE IMPORTANCE OF GEOPARKS IN GEOTOURISM AND ENVIRONMENTAL EDUCATION**

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## **Abstract**

The analysis of conditions for the process of reclamation and revitalization of areas after the completed opencast mining of rock raw materials in the light of strategic principles for the geopark creation and functioning was described. The model area for the analysis is the region of high concentration of extractive and processing industry, called Białe Zagłębie, located in the south-western part of the Świętokrzyski region (Poland). Due to exceptional geological and cultural values as well as well documented network of geosites, the significant part of this area is subject to protection as the Chęciny-Kielce Landscape Park. This area along with the eastern part of Białe Zagłębie is under the initiative of Geopark Geoland Świętokrzyski applying to the UNESCO Global Network of Geoparks.

Based on the analysis of natural, socioeconomic conditions and their connections with the sustainable development strategy, the optimal directions of reclamation and revitalization for the existing and future post-mining areas located at the described area were developed. Also, a model for planning the reclamation of mining areas functioning in areas prospective for the creation of geoparks, with the leading strategy for using natural (geological) and cultural resources for the sustainable development was proposed. The analyzed problem is of much importance in shaping ecological culture of the local community and tourists visiting these exceptionally attractive terrains in large numbers.

## 1. Introduction

Reclamation and use in order to give new functions to areas transformed by the mining activity constitute one of the key problems of industrial areas where extractive and processing industry played or still plays a leading role. The model example of such an area constitute the so-called Białe Zagłębie, the industrial district located in the south-western part of the Świętokrzyskie Mountains (Poland).

A good availability and significant resources of deposits of rock minerals are a reason why at present there is a number of large exploitation regions with open opencast mines of rock minerals and cement and lime industry plants within the area of Białe Zagłębie (its eastern part in particular). The choice of the optimal direction of reclamation and use of these areas after the completion of activity constitutes one of the key problems in the context of sustainable development strategy of the Chęciny-Kielce area. The basic principles of development strategy of this area specified in documents at the voivodeship and local level assume the optimal use of natural and cultural resources for the development of tourism. Post-mining areas combining elements of industrial and natural (geological) heritage constitute in this case potential centers of sustainable forms of tourism based on the above-mentioned resources. One of such forms is geotourism which in the holistic view constitutes a form of sustainable tourism being on the border of natural and cultural tourism, based on geological and cultural heritage of a given area [1]. Geopark as the area of protection and sustainable use of geological heritage for the stimulation of socioeconomic development of the region, recommended by UNESCO and supported by the European Economic Development Programme, is the most optimal solution for the realization of the above mentioned principles.

In this context, planning reclamation and revitalization of post-mining areas within the existing or designed geoparks requires merging this process with the strategy for their creation and functioning, assuming the optimal use of geological and industrial heritage for geotourism and geological education. This chapter constitutes an attempt to analyze the conditions essential for the

process of planning the reclamation and revitalization of future areas within the designed geopark. The article takes also the issue of incorporating the local community into the decision-making process by utilizing mechanism of functioning and managing the geopark into account.

## **2. Goal, range and methodology**

The basic goal is an attempt to specify conditions essential for the process of planning reclamation and revitalization of mining areas after the completed exploitation, in the light of functioning of Geopark Geoland Świętokrzyski applying to the UNESCO Global Geopark Network [6]. Authors made also an attempt to specify optimal forms of reclamation and revitalization of future post-mining areas, in the context of the sustainable development, taken into account in the development strategies at the local (commune) and voivodeship level. The proposal of the model for planning of reclamation and revitalization of mining areas with taking the mentioned principles into account was presented. MIDAS and INFOGEO SKARB databases, made available in services of the Polish Geological Institute - the Polish Research Institute and the results of conducted field studies including the terrain analysis of the current state of reclamation and evaluation of the land transformation scale caused by the mining activity in a given area were the basic source of information on the current state of exploitation and the assumed directions of reclamation of the most important exploitation regions of the Chęciny-Kielce area. The analysis of factors essential in the process of planning reclamation and revitalization was conducted by the method of source material analysis and design method.

## **3. Aims of reclamation and revitalization of degraded areas**

Reclamation of the area includes a number of activities such as improvement of physical and chemical properties of lands, regulation of water relations, redevelopment of soil, reinforcement of slopes and reconstruction or construction of necessary roads. The

aim of reclamation is reducing the nuisance of wastelands and creating a new facility, enabling to conduct a specific activity. Among directions of reclamation there are various types of reclamation directions: watery, forest, meadow, agricultural, recreational and special direction. The selection of the reclamation direction is influenced by external and internal factors. External factors include natural conditions, including climatic conditions of the surrounding area, legal and technical requirements, and above all expectations of residents [3]. Prior to the adoption of the decision about the direction of reclamation, consultations with the local community concerning the planned investments and possible dangers should be conducted. Taking the expectations of the local residents into account can add value to the restored areas. Social consultations relieve conflicts which often accompany industrial investments. The community often becomes involved in the reclamation process, which enables to create emotional bond with a given facility and gives the feeling of responsibility for the ongoing reclamation process.

Internal factors are the facility location and geometry, properties of works creating the surface layer of the facility, hydrographic conditions. Taking many external and internal factors into account during planning the reclamation works allows for better adjustment of the reclamation direction to the local conditions.

Revitalization is the process of economic and social transformations, increasing the quality of natural environment and life of residents accompanying reclamation. It facilitates economic recovery and rebuilding social bonds in the degraded urban areas. It combines technical actions with economic recovery programmes and undertakings to resolve social problems occurring in these areas, e.g. unemployment, crime, demographic imbalance [7]. Revitalization is reaction to crisis which involves a given area and manifests itself in many fields at once. It must be realized with cooperation of local partners: public sector (commune), local entrepreneurs, cleaning sector. Revitalization is comprehensive plan of multidirectional, mutually reinforcing actions, aiming at causing qualitative change in the whole indicated area, including changes in the negative image of the area.

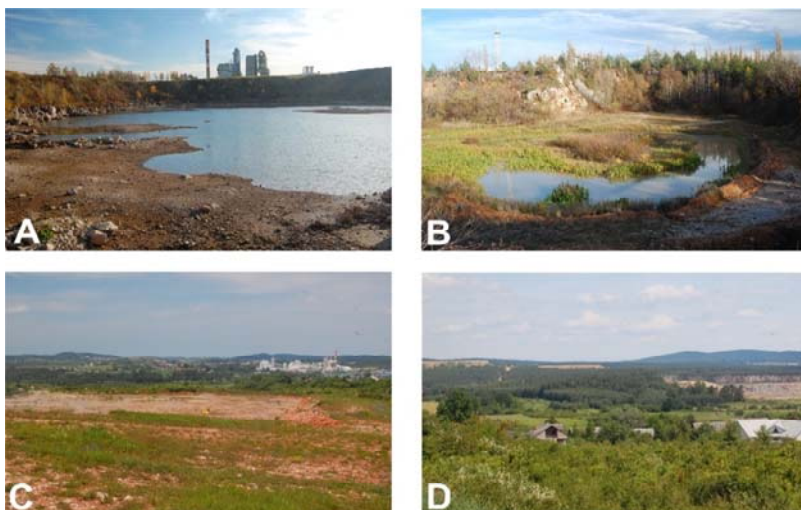
#### **4. The specificity of the research area in the light of mining activity**

The analyzed area is the surroundings of the Chęciny-Kielce Landscape Park constituting the active from the industrial perspective part of Białe Zagłębie (Poland). This area located in the south-western part of the Świętokrzyskie Mountains has constituted the important center of extraction of non-ferrous metal ores and decorative local lime varieties, the so-called Chęciny marble, for many centuries. Historic mining was characterized by the limited influence on environment and it introduced a new value to it. This value became the integral part of the local cultural and natural heritage in time. Residues from the extraction of copper and lead ores and exploitation of Chęciny marbles became one of the leading motives for the creation of the first geological park in Poland - the Chęciny-Kielce Landscape Park [3, 5]. The current protected surface of the area is 20 505 ha, and its protection zone (buffer zone) - 11 123.80 ha. The 5-kilometer-wide zone from the eastern border of the Park buffer zone constitutes the main subject of interest of authors due to the significant concentration of the extractive industry, particularly large exploitation regions comprising the characteristic elements in a form of: mining excavations, waste heaps and industry infrastructure facilities.

The most critical area in terms of the influence of opencast extraction on environment is located on the eastern side of the Chęciny-Kielce Landscape Park buffer zone. The region of Sitkówka-Nowiny is characterized by the most intense opencast exploitation per 1 km<sup>2</sup> of the land area in the voivodeship and one of the most intense exploitation in the country.

The character of the effect of opencast mining in the researched area is connected primarily to the visible transformations in the cultural landscape (Fig. 1) and changes in natural environment (soil and vegetation degradation, lowering the water table and hydrographical changes)]. Large exploitation regions are accompanied by smaller mining excavations and external waste heaps and industry infrastructure facilities, comprising together the mining basin landscape which in particular in the years 1970-1990 became the synonym of Białe Zagłębie.





**Fig. 1.** Opencast mining in the landscape of Białe Zagłębie:

A – excavation area "Zgórsko"; B – post-mining area, partly reclaimed in the exploitation region of "Zgórsko"; C – the view to the exploitation region "Kowala", in the background, the landscape of the eastern part of the Chęciny-Kielce Landscape Park; D – the view to the exploitation region "Trzuskawica"

[picture by M. Poros]

The specificity of opencast mining effect on environment, including the landscape of the researched area, manifests spatial diversification. It consists in concentration of large exploitation regions, characterized by high landscape impact in the eastern and north-western parts of the area. The remaining part of the largely protected area known as the Chęciny-Kielce Landscape Park is characterized by the presence of small and average post-mining facilities constituting the remnants after the historic mining of decorative lime varieties, the so-called "Chęciny marbles". Despite a similar genesis (opencast mining), the two mentioned categories are connected to a different strategy for the approach to reclamation and revitalization of the area after the completion of exploitation. In the case of post-mining areas after the historic exploitation of decorative varieties of carbonate raw materials self-acting reclamation (renaturalization) led to the creation of environmentally valuable areas, characterized by exceptional biodiversity, largely developed in

the natural direction as reserves and natural monuments as well as documentation sites. The example of such areas are "marble quarries" at Zelejowa Mt., Stokówka Mt. and Ołowianka Mt.

Large exploitation regions which are characterized by environmental transformations of large areas by opencast mining require considerably longer and more complex process of reclamation and revitalization. Exploitation regions located to the east from Sitkówka-Nowiny, in the region of Trzuskawica and Kowala (Fig. 1) are characterized by considerable diversification not only in terms of scale and technical parameters of mining excavations, but also due to their natural, scientific and didactic values.

The eastern part of Białe Zagłębie strongly transformed by the mining activity is simultaneously one of the most prospective areas of south-eastern Poland in the light of the geopark creation and geotourism development. Combining the area with high natural and landscape values protected within the Chęciny-Kielce Landscape Park, with typically industrial area of the eastern part of Białe Zagłębie and the municipal Geopark Kielce provides a unique possibility to use post-mining areas within one consistent strategy for the geopark development. The main principle of this strategy is utilization of geological and industrial heritage for the sustainable socioeconomic development of the region and involving the local community in this process. Conditions resulting from principles of geopark functioning for the process of planning reclamation and revitalization of future post-mining areas will be analyzed in the further part.

## **5. Conditions for the process of planning reclamation and revitalization of post-mining areas in the light of the strategy for the geopark creation and functioning**

In the light of UNESCO guidelines, constituting a specific collection of principles for the creation of geoparks, the use of geological heritage for the sustainable development of the region and its community constitutes the theme of the strategy for the geopark creation and functioning [6]. The area indicated as prospective for the geopark creation must distinguish itself in terms of geodiversity,

have a network of geosites and defined boundaries as well as the surface area sufficing for the independent socioeconomic development of the sustainable nature [1].

The process of planning reclamation and revitalization of future post-mining areas functioning within the designed or existing geopark, can be linked to the above mentioned principles in the following range:

- natural aspect related to the protection of geological and industrial heritage with particular value for the region
- scientific and didactic aspect related to the appropriate reclamation and use of these fragments of the post-mining area which demonstrate characteristics of geosites essential for scientific research and environmental education
- socioeconomic aspect and environmental education: involving the local community in the process of planning reclamation and revitalization and their participation in benefits resulting from the functioning of geotourist facilities and/or attractive investment areas in the reclaimed post-mining areas or in their close vicinity

### **Natural conditions and scientific and educational aspect**

The protection of geological heritage and functioning of geosites' network constitute one of the basic principles of the geopark creation and functioning [1, 2]. Opencast exploitation of rock raw materials causing degradation of the land surface is simultaneously the important factor affecting the specificity of the geopark, in terms of creation of new geosites significant for scientific research and environmental education (including geological one). Such a principle accepts diverse forms of reclamation and revitalization of the post-mining area, in the light of its later legal protection (as one of forms of nature conservation) and/or use in scientific research and environmental education.

The possibility to use post-mining areas in scientific and educational direction can be seen in many variants which can be connected with each other:

- post-mining areas, reclaimed in the forest and/or watery direction related to the natural aspect, as protected areas in which appropriate revitalization operations give new functions, enabling their use in scientific research and ecological education
- post-mining areas as ecological education centers related to the unique possibility to observe interaction between abiotic and biotic environment in conditions of natural succession functioning from the cessation of exploitation (renaturalization)
- post-mining areas, reclaimed and revitalized as geotourist facilities; the process of reclamation and revitalization strictly depends on the results of geosites' valorization conditioning possibilities of their separate use
- post-mining areas transformed as a result of reclamation and revitalization actions into parks/rock gardens where scientific research and actions related to ecological education can be conducted.

Natural conditions affecting the process of planning reclamation and revitalization are related to multidisciplinary valorization of the post-mining area as regards the indication and protection of environmentally valuable fragments. In the case of the protected area, which is also supposed to serve as a geotourist facility, one of the optimal methods introduced at the stage of planning reclamation of the post-mining area can be the method of geosites' valorization. It is standard method applied for evaluating the geotourist potential of geological sites among others in the designed or existing areas of geoparks. Assuming the appropriate modification as regards adjustment to the specificity of the active mining area, the method of analysis of geosites can provide essential information on the optimal direction of reclamation and use of selected fragments of the post-mining area.

In practice, this may mean introducing the modification of the plan for the opencast mine reclamation, taking the leading criteria into account: technical and economic ones as regards indicating the selected fragments for protection and/or making geotourism more available. The use of the geosites' valorization method in optimizing the process of reclamation at the planning stage has far-reaching effects. In the authors' opinion, the initial valorization of geosites

located in the future post-mining area, conducted at the planning reclamation stage, increases the chances of the appropriate use of sites which are the most valuable in terms of scientific and didactic aspect. Additionally, it can positively affect the reclamation process optimization as regards the appropriate preparation of the post-mining area for later revitalization actions.

### **Socioeconomic conditions and environmental education**

The strategy for the creation and functioning of geoparks takes the significant involvement of the local community in the decision-making process into account. This community is to be the main beneficiary of benefits resulting from the sustainable tourism development (primarily from geotourism) in the geopark area. Both factors (social and economic) combined with each other within the sustainable development strategy should be translated to the process of planning reclamation and revitalization of post-mining areas located in areas prospective for the creation of geoparks. The problem of the society's participation in the process of reclamation and revitalization and of social reception of these processes' effects was already the subject matter of many elaborations [4].

It follows from the research conducted by the above-mentioned authors that social consultations should be indispensable element of planning reclamation and revitalization as operations of far-reaching socioeconomic effects.

In the case of the designed or existing geoparks, the described problem is particularly of significance in terms of environmental education of the local community and criterion of this community's active participation in the geopark creation and functioning. Examples from European geopark indicate measurable benefits resulting from the long-term educational projects aiming at the local community's activation [5]. This process is related to building the regional identity and identification of residents with the geopark initiative and actions planned within its area. In addition, there is also the economic activation in terms of creation and support of the local business initiatives utilizing and promoting geological and cultural (including industrial) heritage of the geopark.

In the case of the area of Białe Zagłębie, significance of the above described factors for planning reclamation and revitalization is reflected in the example of the local, non-formal "Geopark Białe Zagłębie", which along with the Chęciny-Kielce Landscape Park and the neighbouring areas, was incorporated into the formalized Geopark Geoland Świętokrzyski functioning as the Association of Municipalities. The initiative of geopark resulting from the cooperation of residents with local authorities includes the area of communes: Kielce, Chęciny, Morawica, Sitkówka-Nowiny and Piekoszów and aims at promotion and geotourist use of the local geological and industrial heritage related to exploitation of rock raw materials and bullion. Actions of the local community primarily aimed at areas related to the historic rock and bullion mining. The subject of interest are also the active mining areas (among others large exploitation regions of Trzuskawica and Kowala). Incorporating the idea of using the post-mining areas for sustainable tourism, geological and environmental education into the strategy for the commune development starts to bring results in a form of greater opportunities to cooperate with extractive and processing concerns. Reclamation actions planned at the stage of mining plants' functioning can be subjected, after appropriate consultations, to slight modifications taking solutions proposed by the community and local authorities in the context of the strategy for the geopark functioning into account. Implementing such a model of action requires incorporating representatives of the managing body and programme council, comprising representatives of the local community, local authorities, business environment, non-governmental (including proecological) organizations and research and development units into the decision-making process related to planning reclamation and revitalization.

Despite making the process of reclamation and revitalization more complicated, such a scheme of action can be beneficial for both sides. Extractive concerns can realize an important element of their own politics of sustainable development assuming pro-environmental and pro-social actions for the local community, compensating the negative effect of mining on the environment and improving the company image in social reception in this way. Local authorities and residents by participating actively in the process of reclamation and

revitalization realize the local strategy for geopark development, which is to bring benefits related to the increased tourist traffic in the commune, and at the same time preserve the natural and cultural heritage for future generations – in accordance with the sustainable development principles. In this context, opencast mining of rock raw materials constitutes the specific "initial capital" which subjected to appropriate processes (of reclamation and revitalization) coupled with the strategy for the geopark functioning, brings tangible economic and social results.

## **6. The model of planning reclamation and revitalization of post-mining areas in the light of the strategy for the geopark functioning**

The process of planning reclamation and revitalization is a complex issue due to a large variety in criteria and legal and formal conditions. So far, results presented in elaborations by various authors indicate the need for an multidisciplinary approach to the problem. It assumes taking numerous criteria into account in the process of planning and involving specialists from various fields in their verification as well as social consultations [4, 8].

The standard procedure of reclamation applied in most cases results directly from the existing provisions of Community and domestic law [2]. The requirement of restoring practical and natural values to soil degraded by the mining activity specified in the relevant laws and ordinances (primarily in the Act of 3 February 1995 on the protection of arable and forest land). Along with technical and technological (quazi-constant properties and maintenance of the facility after the reclamation) factors, it decides about the selection by the majority of the forest or watery direction of reclamation as the most optimal in the existing conditions.

Data from MIDAS database concerning the basic directions of reclamation adopted for the mining areas in Białe Zagłębie confirms this regularity. In the case legal and economic factor, there is also additional problem of imprecise distinction of terms in legislation: reclamation and use as well as frequent conflation of the latter with revitalization by local government units. There often appear incoherence and difficulty in coordinating actions between the

company responsible for the liquidation of a coal mine and reclamation of the area and the recipient which often is the local government unit.

The described conditions are still relevant also in the case of the area which is to serve as the geopark, but additionally there is a significant criterion of conformity with the strategy of functioning of such an area based on the sustainable socioeconomic development which is based on the local geological and cultural (including industrial) heritage. In this context, mining area along with all its components is seen as the potential natural, landscape and cultural (in a sense of industrial heritage) value. Criteria resulting from the strategy for the geopark creation and functioning do not have to be contrary to the basic formal and legal, technical and economic factors conditioning the process of reclamation and, as a result, affecting the revitalization of the post-mining area.

The optimal solution taking both interests of a mining entrepreneur and criteria resulting from the geopark functioning into account is necessary. The above principles constitute the basis of the presented by the authors theoretical model of planning the process of reclamation and revitalization of the future post-mining area.

The principle that at the stage of planning reclamation by the mining entrepreneur the most important factors determining the choice of the reclamation direction are technical and technological (quazi-constant properties of the mining facility) and economic (the cost of reclamation endeavor) criteria was adopted for the needs of the developed model. Any modification of the reclamation plan resulting from the introduction of criteria related to the strategy of the geopark functioning must take these principles into account in such a way that their realization is possible from the technological point of view and does not generate excessive costs for the entrepreneur.

The most important functions which the revitalized post-mining area can have are related to preserving and using geological and industrial heritage in sustainable tourism and geological education. The proposed model assumes the introduction of two factors at the stage of planning reclamation: the modified method of geosites' analysis with a view to evaluate the geotourist potential of the future post-mining areas and incorporate entities responsible for the



geopark creation and functioning into the decision-making process.

The modified method of geosites' analysis is based on methods of geotourist potential evaluation, developed for geosites in Poland and in the world [3]. This method was to date used mainly for geosites outside the active mining areas, within areas of the designed or existing geoparks. Its use for sites within the active mining facilities requires modification, taking the specificity of the active mining area (among others, the criterion of location within the deposit being exploited, related to the possibility to preserve a given fragment of rock mass) into account. The aim of using this method at the stage of planning reclamation is to evaluate the applicability of specific fragments of the mining area for geotourism and geological education in a comprehensive way. This evaluation is indispensable for the appropriate planning of reclamation actions and then revitalization ones. The method in the version modified by the authors takes the following key criteria into account: location criterion, scientific value, didactic value.

The location criterion is of crucial importance for further actions related to, among others, the valorization of a given area fragment in terms of scientific and didactic value. It is strictly related to technical, technological and economic criteria which in the case of the active mining area determine the possibility to preserve a given site, also in the case when it is characterized by a significant scientific and didactic value. The possibilities to shut down are in such cases very limited; therefore, a 3-degree scale of evaluation is taken into account within this criterion, which takes location within the mining area and factors related to this location into account. A detailed description of this criterion is presented in Table 1.

In the case of the criterion taking the scientific value into account, the evaluation is based primarily on the initial valorization of a given fragment of the mining area as regards the occurrence of rocks, minerals, fossils and geological phenomena, being of significance at the local, regional and super-regional scale. Depending on the significance, each of the element receives a point evaluation: 1 pt – local significance, 2 pts – regional significance, up to 3 pts – super-regional significance. In the case of lack of a given element, the value 0 is attributed.

**Table 1****The location criterion within the modified method of geosites' analysis**

Characteristics of location factors	Point value (pts)	Evaluation and interpretation
The site located within the part of deposit currently being extracted / planned to be extracted which is characterized by beneficial parameters of the raw material, or within the part of the mining excavation for which the reclamation direction other than watery one is difficult to realize due to technical and/or economic factors	1	The lack of possibility to preserve a site and plan reclamation as regards the use for the needs of geotourism and/or geological education; the necessity to take the alternative actions related to the site's scientific documentation (including the evaluation of scientific and didactic value) and the protection of geological specimens valuable in terms of scientific and/or educational aspect into account
Sites located within the deposit part planned to be shut down where actions related to the maintenance of the remaining exploited deposit part (among others, construction of technological roads) are realized	2	There is a possibility to preserve the site and plan reclamation in terms of its use for the needs of geotourism and/or geological education, taking the minimization of costs related to the modification of the current technological actions by the entrepreneur into account; reclamation actions preceded by the comprehensive evaluation of the site's scientific and didactic value
The site located in the shut down part of the mining excavation due to the unfavourable geological situation and/or parameters of the raw material	3	There is a possibility to preserve sites and plan and gradually realize reclamation actions in terms of their use for the needs of geotourism and/or geological education; reclamation actions preceded by the comprehensive evaluation of the site's scientific and didactic value

In the case of presence of at least one element, being of super-regional significance, the site receives additionally 6 points due to the essential scientific value. The sum of points constitutes the evaluation of the site's scientific value. The evaluation of this criterion, depending on the number of points, is presented in Table 2.

**Table 2**

**The scientific and didactic criterion within the modified analysis of geosites**

Criterion	Evaluation scale	Point value	Interpretation
Scientific	High	> 9	High scientific value; preferred reclamation and use direction: natural (inanimate nature reserve, documentation site or natural and landscape complex)
	Average	5 – 9	Average scientific value
	Low	1 – 4	Low scientific value
Didactic	High	> 9	High didactic value
	Average	4 - 6	Average didactic value
	Low	1 - 3	Low didactic value

The criterion related to the didactic value concerns the evaluation of the degree of applicability for educational purposes. The following elements essential from the perspective of the use in geological education at various levels of teaching are subject to the evaluation:

- illustrativeness of geological phenomena occurring at the site (low – 1 pt; average – 2 pts; high – 3 pts)
- availability of minerals, rocks or fossils in the number and form allowing for their acquisition for educational purposes (low – 1 pt; average – 2 pts; high – 3 pts)
- additional characteristics of the site essential in terms of their use for educational purposes (each characteristics – 1 pt; whereby the maximum admissible evaluation of this element is 3 pts).

The total number of points determines the site's general didactic value, whereby in the case of the general average and maximum (3-point) evaluation of one of the elements it must be assumed that the possibility to use it in geological education should be taken into account in the context of its future use.

The use of the modified method of geosites' analysis is strictly connected with incorporating entities responsible for the geopark creation and/or functioning into the decision-making process. Developing the spatial management of the area with exceptional geological heritage, which is constituted by the geopark [6], requires the participation of multidisciplinary research team responsible for

the preparation of comprehensive substantial documentation already at the planning stage. If the modified method of geosites' analysis is applied for mining areas, the need to incorporate the mentioned team in the process of the separate criteria analysis is assumed, with special attention paid to the scientific and didactic value evaluation.

These actions can be helpful in optimizing the decision-making process within developing and modifying the plan of the future post-mining area reclamation. The entity responsible for conducting the reclamation can simultaneously eliminate the endeavor costs and prepare the functioning mining area in a better way in terms of the new functions adjusted to the geopark's strategy.

Examples of post-mining area use located in areas of existing and designed European and domestic geoparks have particular significance in the context of this article's subject matter. Among Polish solutions, there is an interesting example of reclamation and revitalization actions within "Bełchatów" Coal Mine. Apart from the set of planned and successively conducted reclamation actions in various parts of the mine, depending on technical and technological conditions, a number of concepts related to the future post-mining area use was developed so far. One of them is a proposal prepared by the research team from AGH Kraków concerning the transformation of the post-mining region of the mine into the sports, recreation and culture center of super-regional significance [2]. The geotourist and educational aspect was taken into account in the revitalization concept, introducing the element of use of the mine part with the existing mining infrastructure for the needs of a museum presenting the mining heritage. In the context of scientific and didactic attractiveness of sites within the part of the mine envisaged for reclamation in the watery or forest direction, the town of Bełchatów in cooperation with the management of PGE KWB "Bełchatów" and research team PIG-PIB realized the interactive exhibition entitled "Giants of Power" (pl. "Giganty Mocy"). The comprehensive scientific documentation was prepared and valuable geological specimens were secured for the needs of the exhibition. The described example is of great significance in the concept of the natural and cultural geopark "Szczercowskie Diamenty" creation.

In the case of European geoparks, interesting examples of post-mining areas' use for the needs of geotourism and geological

education come from, among others, Great Britain, Germany and Portugal. Due to minutely developed methodology of mining areas' reclamation as regards their future functioning as geotourist sites, the British model of geosites' preserving contributed substantially to the discussed issue. Examples of its use constitute among others active mines of rock raw materials: Barrington Chalk Pit and Broadway Quarry, located in counties of Cambridgeshire and Worcestershire [4]. In both cases opencast exploitation of rock raw materials revealed geological profiles significant from the super-regional (Barrington Chalk Pit) and local (Broadway Quarry) perspective. In agreement with authorities of mining facilities, selected fragments of mining areas were incorporated into the special system of protection of geosites, taking securing the geological heritage facilities, scientifically and didactically valuable at the national (SSSI - Site of Special Scientific Interest) and regional and local (RIGS - Regionally Important Geological and Geomorphological Sites) level into account. An example of the geopark in which a number of actions concerning reclamation and revitalization of post-mining areas for the needs of geotourism and geological education was realized is English Riviera Geopark. Currently, there are 8 geosites related to post-mining areas which are available for tourist traffic at the geopark.

The use of active and closed mining areas in geotourism and geological education is known also from two Portugal geoparks incorporated into the European and global network: Naturtejo and Arouca [5]. One of examples is a quartzitic sandstones and slates mine in Canelas where a part of a mining facility was shut down and it was turned into an interpretation center and site for the needs of geological education.

Planning reclamation and revitalization as regards the mentioned functions referring to the strategy of the geopark functioning, does not exclude other possibilities related to the multifunctional revitalization of the post-mining area or its immediate vicinity. Numerous examples from Poland and Europe indicate the beneficial effect of reclamation in the natural, tourist and didactic direction on the investment attractiveness of the immediate vicinity of post-mining areas [2]. This phenomenon concerns both investments strictly connected to the function of the area (investments from the

tourism sector in the broadest sense), and developer investments being the natural reaction of the market to the creation of the area attractive for settling down. Both phenomena under the rigours related to the geopark functioning principles can affect its socioeconomic development in a beneficial way.

## **7. Summary**

The nuisance of mining activity for the population inhabiting areas subjected to the direct or indirect impact of the mining industry is the important social and environmental problem. The opencast exploitation of rock raw materials is most often seen as devastation of nature and landscape whose effects must be eliminated using the available legal instruments and technological solutions. Revitalization includes the comprehensive actions conducted by the commune. Cooperation with residents, cooperation based on the agreement of administrative bodies with social entities is indispensable in these actions.

Coupling the process of reclamation and revitalization with the sustainable socioeconomic development strategy of the geopark enables to look at the mining activity through the prism of potential benefits and can positively affect the change in its perception by the local community. This results directly from the principles of the creation of the geopark within which the local population is co-creator and at the same time beneficiary of benefits resulting from its functioning [5]. Participating in the geopark functioning is closely linked to the participation in all decision-making processes related to revitalization of the land within it. The key importance of the process of reclamation and revitalization of post-mining areas for the geopark tourist attractiveness justifies the incorporation of the local community into planning of these actions. At the same time, it is the important aspect of environmental education, assuming activation of the population inhabiting the geopark area.

The process of reclamation aimed at the realization of the criteria essential from the perspective of the geopark development causes that post-mining areas additionally subjected to appropriate revitalization operations can constitute important geotourist facilities affecting the local socioeconomic development. The examined

example of the non-formal Geopark of Białe Zagłębie, promoting the local geological and industrial heritage related to the rock raw materials' exploitation within the area of the industrial basin, constitutes the exceptional testing ground, enabling to examine conditions of the process of reclamation and revitalization during the geopark creation and functioning. A great emphasis was placed on the perception of mining lands as new natural and cultural values of the designed or existing geoparks in the presented model of planning the reclamation and revitalization process developed by the example of the above mentioned area. The problem of using the mechanisms related to the creation of geoparks in planning reclamation and revitalization of post-mining areas can be, in the authors' opinion, an interesting research topic, promising the possibility of the practical use in areas with significant concentration of opencast mining.

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# **INVESTIGATION OF THE MUTUAL INFLUENCE OF A NUMBER OF INDUSTRIAL AND NATURAL FACTORS THAT HAVE A NEGATIVE IMPACT ON THE PROCESS OF THE UNDERGROUND COAL MINING**

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**Abstract.** Each type of industrial production of a rather high degree of complexity, for example, the coal industry, metallurgy, coke chemistry, possessing a number of negative factors, represent complex volumetric ergatic systems requiring management of the industry or even management of such a separate enterprise, non-standard approaches to solving problems, at least, minimizing occupational injuries. However, in the case when natural factors are added to industrial factors (the composition and movement of soils and rocks, abundant water inflows, both seasonal and annual), the task is beyond the reasonable use of even a sufficiently powerful computer, or requires a different formulations and other system approaches.

The analysis of the production process first of all draws attention to the following negative factors: traumatism associated with transport; injury during the maintenance of stationary mechanisms; electric shock; explosion; collapse; gassing; fire; GDP (gas-dynamic phenomena); other.

So, to start with the analysis, you can take all of the above factors, and then add or remove from the calculations and analysis some of them, using the methods of classical high-level statistics.

The methodology developed for the assessment of underground coal mining from the point of view of its safety allows give a deep and comprehensive assessment of mining production, outlining ways to reduce the accident rate and then shifting the minimization of the cost of safety in the industry or a separate enterprise.

The numerical estimation of the probability of occurrence of severe man-made accidents in coal mines can serve as a basis for further introduction of heuristic approaches in the industry, i.e., assessment of



production from the position of "man-machine-environment" and risk assessment for specific enterprises of the coal industry.

## **Introduction**

Each type of industrial production of a rather high degree of complexity, for example, the coal industry, metallurgy, coke chemistry, possessing a number of negative factors, represent complex volumetric ergatic systems requiring management of the industry or even management of such a separate enterprise, non-standard approaches to solving problems, at least, minimizing occupational injuries. However, in the case when natural factors are added to industrial factors (the composition and movement of soils and rocks, abundant water inflows, both seasonal and annual), the task is beyond the reasonable use of even a sufficiently powerful computer, or requires a different formulations and other system approaches.

Therefore, proceeding from the considerations that the coal industry is sufficiently conservative (new common technologies have not been introduced in recent decades), as factors influencing both the process of underground coal mining and its safety, on the basis of already available experience [1, 2], it should be consider the following phenomena: 1) Traumatism associated with transport; 2) Injury during servicing of stationary mechanisms; 3) Electric shock; 4) Explosion; 5) Collapse; 6) Poisoning with gas; 7) Fire; 8) GDP (gas-dynamic phenomena); 9) Other.

So, to begin with, all of the above factors can be taken into account in the analysis, and later on, or on the contrary, some of them can be removed or calculated from the calculations and analysis, using the methods of classical high-level statistics.

### **1. Methods for obtaining data on the work of the coal industry**

It is known that expert evaluation is a procedure for obtaining an assessment of a problem on the basis of the opinion of the specialists (experts) with a view to a subsequent decision [3].

There are the following traditional stages of expert evaluation:

- Setting of the research purpose;

- selection of the research form, definition of the project budget;
- preparation of information materials;
- selection of experts;
- performing of expert analysis;
- analysis of results (processing of expert estimates).

There are two groups of expert estimates:

- individual assessments, which are based on the use of the opinions of individual experts, independent of each other;
- collective assessments, which are based on the use of collective opinion of experts;

The stages of expert evaluation include:

1. Statement of the purpose of the study.
2. Choice of the research form, definition of the project budget.
3. Preparation of information materials, questionnaires forms, procedure moderator.

4. Selection of experts.

5. Performing of expert analysis;

6. Analysis of the results (expert evaluation processing).

7. Preparation of a report with the results of expert evaluation.

Expert estimation presupposes the creation of some kind of intelligence possessing great abilities in comparison with the capabilities of an individual. In this case, one of the main tasks is to search for weak associations and assumptions based on the experience of an individual specialist. The expert approach allows solving problems that cannot be solved by the usual analytical method, including choosing the best solution among the available ones and forecasting the development of the process, including using statistical methods, of a sufficiently high level.

Before the beginning of the expert study, it is necessary to clearly define its purpose (problem) and formulate an appropriate question (or questions) for experts. It is recommended that you follow the following rules:

- a clear definition of the conditions, time, external and internal constraints of the problem. The possibility of answering a question with the accuracy available to human experience;

- it is better to formulate the question as a qualitative statement than as an estimate of the number. For numerical estimates, it is not recommended to specify more than five gradations;

- experts evaluate possible options, and we should not expect them to build a complete plan of action, a detailed description of possible solutions.

Existing types of expert assessments can be classified on the basis of:

- in the form of participation of experts: full-time, correspondence. Full-time method allows you to focus the attention of experts on the problem being solved, which increases the quality of the result, however, the correspondence method can be cheaper and faster;

- by the number of iterations (repetitions of the procedure for increasing accuracy) - one-step and iterative;

- on the tasks being solved: generating solutions and evaluating options;

- according to the type of answer: ideological, ranking, evaluating the object in a relative or absolute (numerical) scale;

- on the way of processing opinions of experts: direct and analytical;

- by the number of experts involved: unrestricted, limited. Usually 5 to 12 experts are used.

Experts should consider the problem presented before a judgment is made. To conduct this procedure, it is necessary (or only expedient) to prepare information materials with a description of the problem, available statistical data, reference materials, questionnaires, inventory. It is necessary to avoid the following errors: mention the developers of the materials highlight one or another variant of the solution, express the attitude of management towards the expected results. Data should be versatile and neutral.

It is necessary to develop questionnaires for experts in advance. Depending on the method, they can be with open and closed questions; the answer can be given in the form of a judgment, a paired comparison, ranked series, in points, in percentage or in the form of an absolute score.

The choice of experts is carried out in such a way that they have full experience in the areas corresponding to the tasks being solved. When selecting experts, one should take into account the moment of personal interest, which can become a significant obstacle to obtaining an objective judgment. For this purpose, for example, the

methods of Shar are used, when one expert, the most respected specialist, recommends a number of others and further along the chain until the necessary collective is selected. The procedure differs depending on the method used. General recommendations:

- to discourage the pressure of authorities (the expert is often afraid to contradict the opinion of the majority or the most respected specialist);

- to establish and comply with the regulations. Increasing the time to make a decision beyond the optimum does not improve the accuracy of the answer.

The main stages of expert evaluation processing:

- determination of the competence of experts;
- definition of the generalized estimation;
- construction of a generalized ranking of objects in the case of several evaluated objects or alternatives);

- definition of dependencies between rankings;

- an assessment of the consistency of expert opinions. In the absence of significant coherence of experts, it is necessary to identify the reasons for the inconsistency (the existence of groups) and to recognize the lack of an agreed opinion (vague results);

- evaluation of the study error;

- building a model of the properties of the object (objects) based on the answers of experts (for analytical or statistical expertise);

- preparation of the report (indicating the purpose of the study, the composition of the experts, the evaluation received and the analysis of the results).

Thus, the method of expert evaluation, as applied to the mine evaluation object, was implemented as follows. Directors, Technical Directors and Deputy Directors on Safety Issues of Operating coal mines and associations were invited as the experts. The involvement of specialists of such a high level ensued from the need to provide perception (and information) of the picture of the functioning of the entire production, rather than any of its individual parts (transport, ventilation, etc.).

Specialists were asked to fill in the following questionnaire (Table 1.) When completing the questionnaire with several specialists from one enterprise or association, each filled out its part regardless of the opinion of colleagues.

**Table 1**

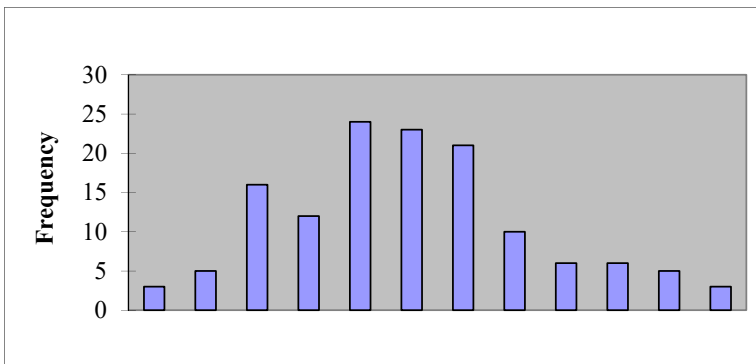
**Questionnaire for collecting data on accidents at mines**

№	Association	Mine	Responsible person Name, Position		Telephone fax e-mail	
			Accidents			
	Expert №		1	2	3	4
1	Transport traumatism					
2	Mechanical					
3	traumatism					
4	Electric shock					
5	Explosion					
6	Collapse					
7	Gassing					
8	Fire					
9	GDP					
	Other					

$\Sigma = 100\%$  Coordinator: tel .; fax; e-mail.

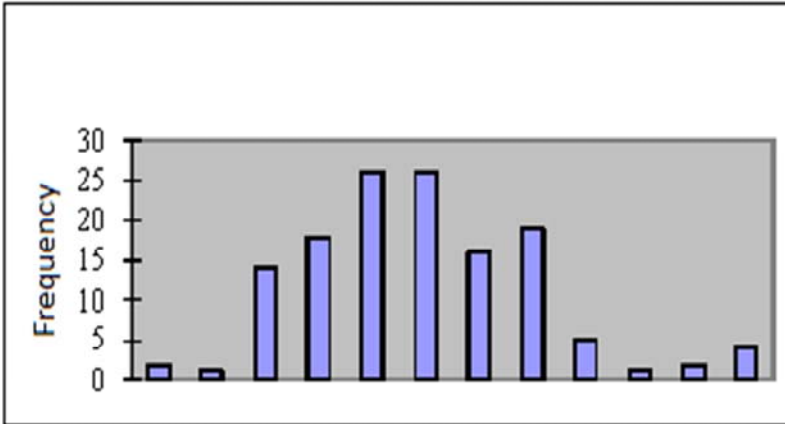
The hypotheses on the distribution laws were tested with a reliability of 0.95 (see Figures 1-7)

The calculation found that the data on factor 1 (transport) are distributed according to the normal distribution law with parameters: average  $a = 24,72$  and variance  $\sigma^2 = 162,03$  (Fig. 1).



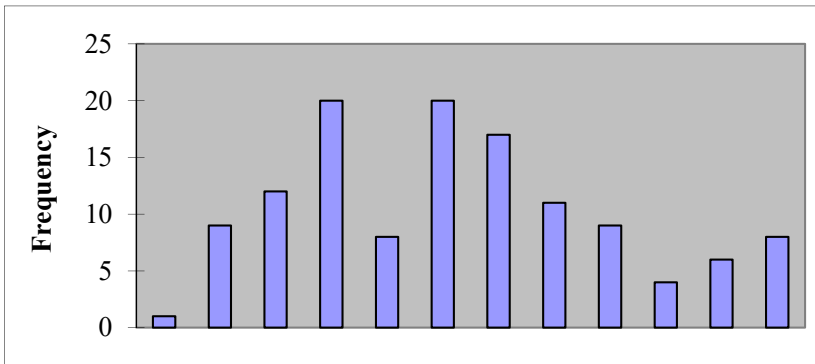
**Fig. 1.** Histogram of the distribution of data by factor 1 (transport)

Data on factor 2 (mechanical) are distributed according to the normal law with parameters: average  $a = 19,82$  and variance  $\sigma^2 = 108,94$  (Fig. 2).

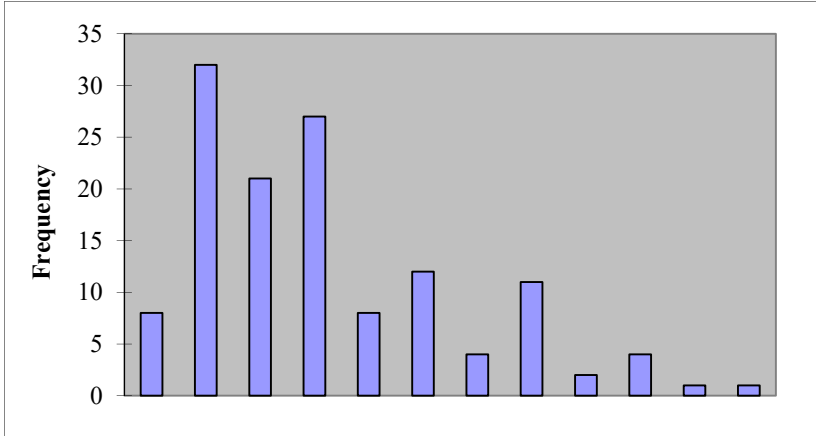


**Fig. 2.** The histogram of data distribution by factor 2 (mechanical)

The law of distribution according to factor 3 (Electric shock) can be established only for smoothed data - by averaging them by factor 10. The averaged data are distributed according to the normal law with parameters: average  $a = 6,23$  and variance  $\sigma^2 = 7,24$  (Fig. 3).

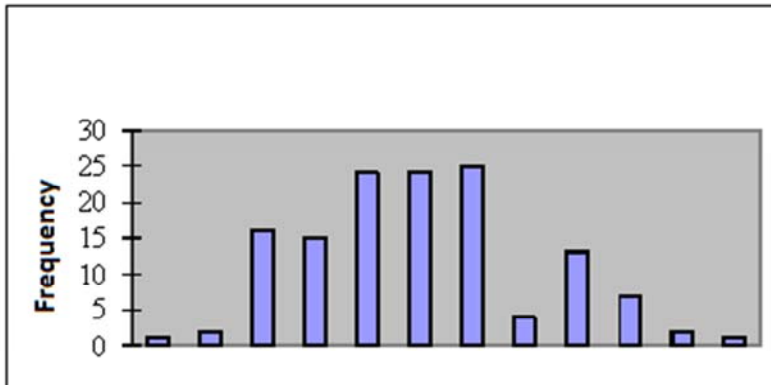


**Fig. 3.** Histogram of the distribution of data by factor 3 (electric shock)



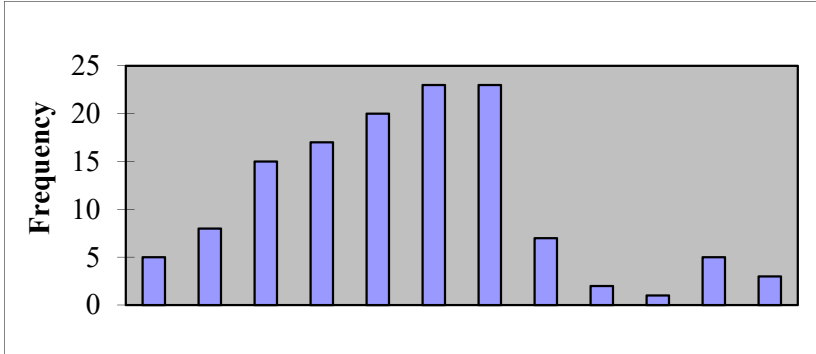
**Fig. 4.** Histogram of the distribution of data by factor 4 (explosion)

Data on factor 5 (collapse) are distributed according to the normal law with the following parameters: средним  $a = 31,56$  and variance  $\sigma^2 = 192,54$  (Fig. 5).



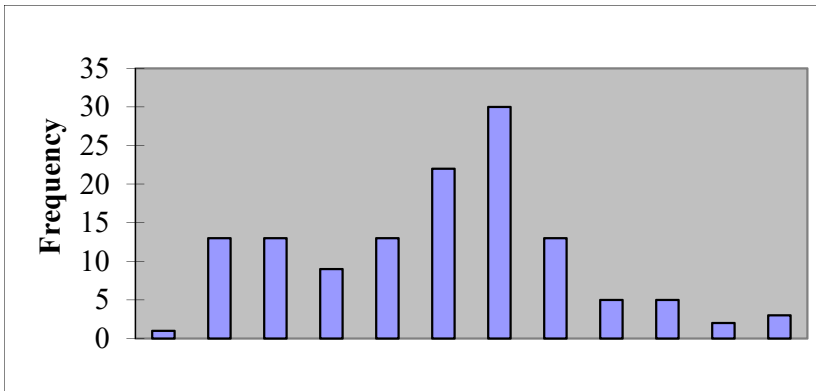
**Fig. 5.** Histogram of the distribution of data by factor 5 (collapse)

The law of distribution according to factor 6 (gassing) can be established only for smoothed data - by averaging them by factor 6. The averaged data are distributed according to the normal distribution law with parameters:  $a = 3,91$  and variance  $\sigma^2 = 5,46$  (Fig. 6).



**Fig. 6.** Histogram of the distribution of data by factor 6 (gassing)

The law of distribution according to factor 7 (fire) can be established only for smoothed data - by averaging them by factor 6. The averaged data are distributed according to the normal law with parameters:  $a = 5,92$  and variance  $\sigma^2 = 9,95$  (Fig. 7).



**Fig. 7.** Histogram of the distribution of data by factor 7 (fire)

The data for factor 8 (GDP) contain only two different values of 0 and 9.

The data for factor 9 (other) contain only ten different values. The value 0 occurs 125 times, and the remaining 9 values once, therefore, in the latest two cases, the distribution laws cannot be determined.



## 2. Primary assessment of indicators to be further analyzed

Mathematical analysis and statistical processing were performed on the basis of the initial data. The data corresponded to the work of 19 associations and individual mines, which were affected by 134 independent respondents.

Initially, it should probably be noted that all calculations, in this section, and other, were also performed with a reliability of 0.95 (including confidence intervals for the average).

The generalized statistical data for all nine factors considered are given in Table 2.

**Table 2**

**Statistical characteristics used in the calculation of data**

Factor №	Average value of confidence interval	Lower value of confidence interval	Lower value of confidence interval	Width of confidence interval	Note
1	24,72	22,53	26,91	4,33	
2	19,82	16,02	21,61	3,59	
3	6,09	5,13	7,20	2,07	
4	5,01	3,53	6,91	3,38	
5	31,56	29,17	33,94	4,77	
6	3,98	3,18	4,85	1,67	
7	5,94	4,75	7,39	2,64	
8	0,14	-	-	-	To build confidence interval is impossible, because 0 occurs 132 times of 134
9	2,49	-	-	-	To build confidence interval is impossible, because 0 occurs 126 times of 134

Graphically, the data can be presented in Fig. 8, in the form of diagrams of the experimental data determined on the basis of [4].

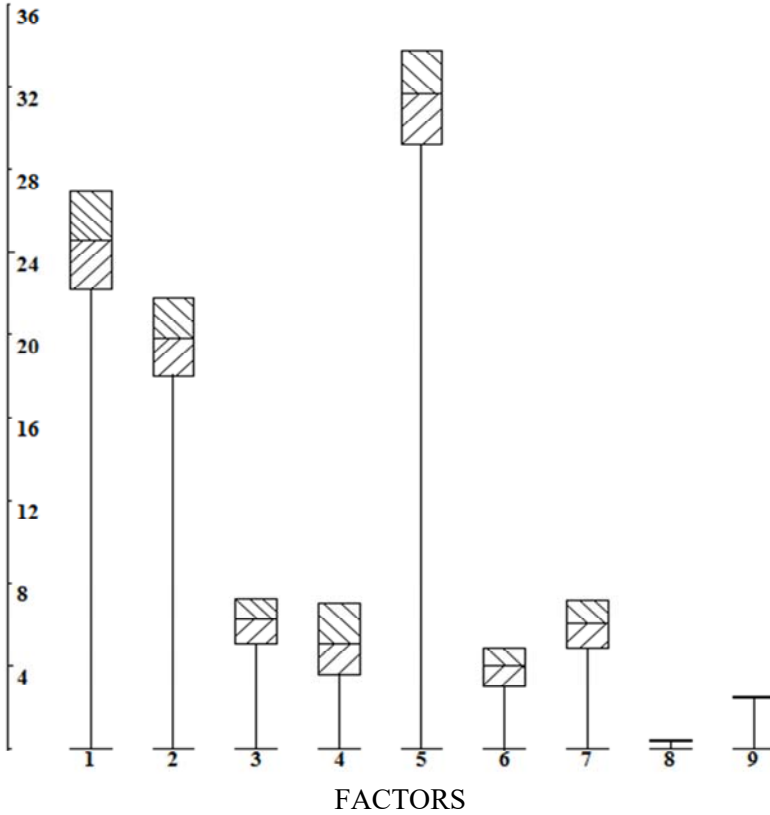


Fig. 8. Diagrams of experimental data on [4]

### 3. Construction of equations, based on the statistical properties of the source material.

Based on the obtained data, the following relationships were initially constructed:

$$1=f(4,6,7) \quad (1)$$

$$2=f(4) \quad (2)$$

$$3=f(5,6,7,9) \quad (3)$$

$$3=f(7) \quad (4)$$

$$4=f(5,6,7) \quad (5)$$

$$4=f(6,7) \quad (6)$$

$$5=f(7) \quad (7)$$

$$6=f(5) \quad (8)$$

$$7=f(1,3,4,5,6) \quad (9)$$

$$7=f(3,4,6) \quad (10)$$

Regression curves obtained by calculation [6] are given below:

$$x_1=-0,636x_4+27,91 \quad (11)$$

$$x_2=-0,455x_4+22,10 \quad (12)$$

$$x_3=-0,12x_5+0,05x_6+0,21x_7-0,12x_9+8,85 \quad (13)$$

$$x_3=0,36x_7+3,96 \quad (14)$$

$$x_4=-0,06x_5+0,68x_6+0,43x_7+1,61 \quad (15)$$

$$x_4=0,72x_6+0,46x_7-0,54 \quad (16)$$

$$x_5=-0,7x_7+35,7 \quad (17)$$

$$x_6=-0,09x_5+6,76 \quad (18)$$

$$x_7=0,05x_1+0,31x_3+0,34x_4-0,03x_5+0,08x_6+4,40 \quad (19)$$

$$x_7=0,35x_3+0,38x_4+0,11x_6+1,15 \quad (20)$$

Regression curves constructed according to formulas (11) - (20), taking into account the data contained in Table. 2 are shown in Fig. 9. The same figure shows the confidence intervals for the output factors, which make it possible to give a complete numerical estimate of the phenomenon under consideration in terms of its variations under given conditions. Confidence interval limits, determined at 95% confidence interval for all output factors, are shown in Table 3 and in Fig. 9.

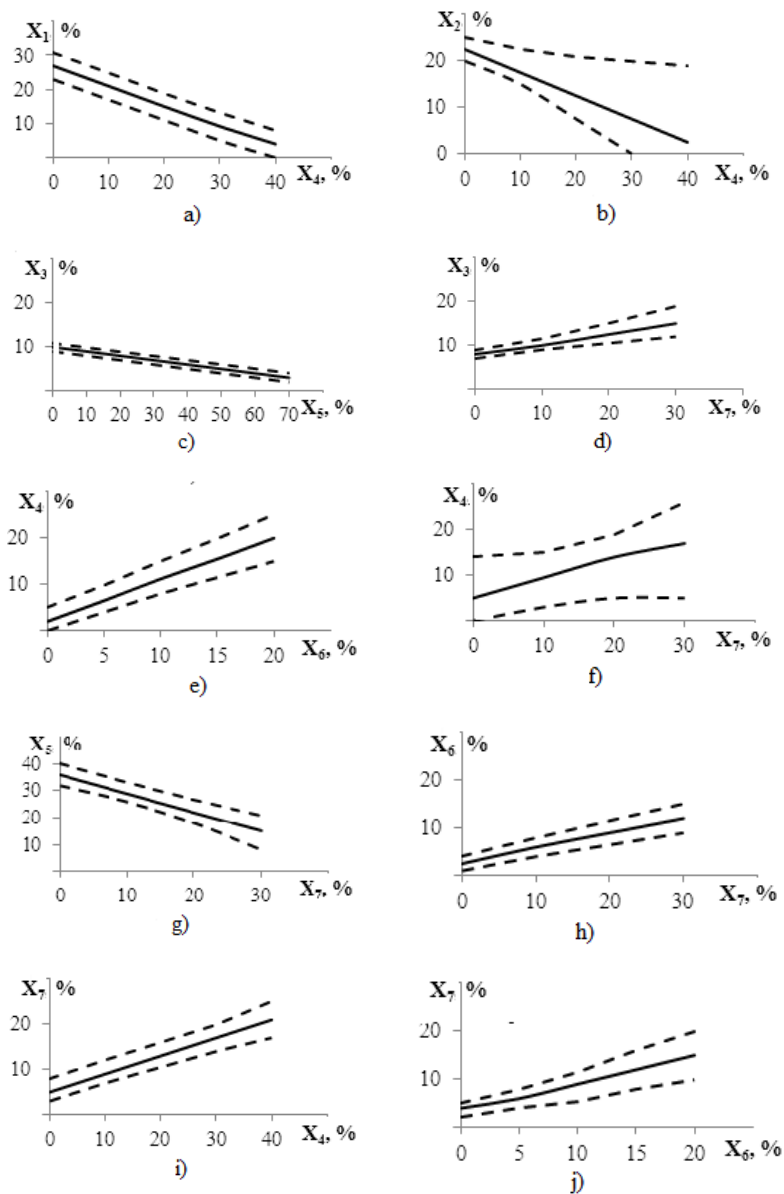
#### 4. Analysis of mining production, based on the characteristics that determine its factors

This analysis can be successfully performed by comparing the data contained in Tables 2 and 3 and the regression dependences presented in Fig. 9. Thus, analyzing the first Fig. 9 a, it can be concluded that transport accidents (factor №. 1) prevail in the case of a non-gas mine, while in gas mines, accidents associated with the explosion (factor №. 4) become decisive. A similar conclusion can be drawn with regard to injuries to personnel when servicing machines and mechanisms (Fig. 9b).

**Table 3**

**Statistical criteria**

Input factor №	Coefficient of correlation R	Confidence interval for input factors														Free member		
		1		3		4		5		6		7		9		min	max	
		min	max	min	max	min	max	min	max	min	max	min	max	min	max			
1	-0,35	-	-	-	-	-0,93	-0,35	-	-	-	-	-	-	-	-	-	25,33	30,43
2	-0,31	-	-	-	-	-0,70	-0,21	-	-	-	-	-	-	-	-	-	20,00	24,20
3	0,56	-	-	-	-	-	-	-0,18	-0,06	-0,15	0,25	0,07	0,35	-0,19	-0,05	-	6,25	11,44
3	0,43	-	-	-	-	-	-	-	-	-	-	0,22	0,49	-	-	-	2,85	5,06
4	0,69	-	-	-	-	-	-	-0,13	-0,01	0,44	0,92	0,26	0,60	-	-	-	-1,26	4,48
4	0,68	-	-	-	-	-	-	-	-	0,48	0,95	0,29	0,62	-	-	-	-1,91	0,82
5	-0,3	-	-	-	-	-	-	-	-	-	-	-	-1,08	-0,31	-	-	32,47	38,92
6	-0,29	-	-	-	-	-	-	-0,14	-0,04	-	-	-	-	-	-	-	5,06	8,46
7	0,65	-0,13	0,02	0,13	0,49	0,19	0,49	-0,10	0,04	-0,16	0,32	-	-	-	-	-	-0,23	9,02
7	0,64	-	-	0,18	0,51	0,24	0,52	-	-	-0,13	0,35	-	-	-	-	-	0,16	2,83



**Fig. 9.** Regression dependencies of the safety level of the coal mining process

**Table 4**

Confidence Areas for Regression

Output factor	Upper limit / lower limit
X <sub>1</sub>	$\bar{X}_1 = -0,636X_4 + 27,91 + 28,45\sqrt{0,0002(X_4 - 5,01)^2 + 0,007}$ $\underline{X}_1 = -0,636X_4 + 27,91 - 28,45\sqrt{0,0002(X_4 - 5,01)^2 + 0,007}$
X <sub>2</sub>	$\bar{X}_2 = -0,455X_4 + 22,10 + 23,71\sqrt{0,0002(X_4 - 5,01)^2 + 0,007}$ $\underline{X}_2 = -0,455X_4 + 22,10 - 23,71\sqrt{0,0002(X_4 - 5,01)^2 + 0,007}$
X <sub>3</sub>	$\bar{X}_3 = -0,135X_5 + 11,37$ $\underline{X}_3 = -0,135X_5 + 9,31$
X <sub>3</sub>	$\bar{X}_3 = 0,36X_7 + 3,96 + 10,81\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$ $\underline{X}_3 = 0,36X_7 + 3,96 - 10,81\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$
X <sub>4</sub>	$\bar{X}_4 = 0,995X_6 + 1,06 + 13,62\sqrt{0,0004(X_6 - 3,98)^2 + 0,007}$ $\underline{X}_4 = 0,995X_6 + 1,06 - 13,62\sqrt{0,0004(X_6 - 3,98)^2 + 0,007}$
X <sub>4</sub>	$\bar{X}_4 = 0,67X_7 + 1,02 + 13,89\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$ $\underline{X}_4 = 0,67X_7 + 1,02 - 13,89\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$
X <sub>5</sub>	$\bar{X}_5 = -0,7X_7 + 35,70 + 31,63\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$ $\underline{X}_5 = 0,7X_7 + 35,70 - 31,63\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$
X <sub>6</sub>	$\bar{X}_6 = 0,3X_7 + 2,18 + 8,99\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$ $\underline{X}_6 = 0,3X_7 + 2,18 - 8,99\sqrt{0,0002(X_7 - 5,94)^2 + 0,007}$
X <sub>7</sub>	$\bar{X}_7 = 0,47X_4 + 3,55 + 11,67\sqrt{0,0002(X_4 - 5,01)^2 + 0,007}$ $\underline{X}_7 = 0,47X_4 + 3,55 - 11,67\sqrt{0,0002(X_4 - 5,01)^2 + 0,007}$
X <sub>7</sub>	$\bar{X}_7 = 0,61X_6 + 3,51 + 12,77\sqrt{0,0004(X_6 - 3,98)^2 + 0,007}$ $\underline{X}_7 = 0,61X_6 + 3,51 - 12,77\sqrt{0,0004(X_6 - 3,98)^2 + 0,007}$

It is established that a serious accident rate is a consequence of man-made accidents due to an explosion, which gives an increase traumatismto 29.2%.

Interestingly, the association of accidents is monitored, or, more precisely, personnel injuries are inflicted on electric shock (factor 3) depending on the collapse (factor 5). Although this dependence is not so pronounced (Fig. 9c), the role of electric traumatism in mines dangerous for gas dynamic phenomena in comparison with mines of other categories decreases from 10,15% to 0,72%. Here and below, when determining the maximum and minimum values of the considered quantities, the upper and lower envelopes of the confidence interval respectively are taken into account.

It is obvious that the probability of electric shock is directly related to the fire (Fig. 9d). This is explained by the fact that, if necessary, to eliminate the negative phenomenon (fire) that has arisen, haste arises in the implementation of a number of unplanned emergency situations, which leads to an increase in the electric accident. In this case, the accident rate can increase from 2,43% to 18,46%, i.e.in 8.66 times. The dependence between the explosion (factor 4) and the respective poisoning by gas (factor 6) is similar in nature, Fig. 9 e.

The physical essence of the phenomenon is the following. When a gas is poisoned, that is, in the presence of a gaseous medium, the probability of an explosion increases sharply. Practically, judging by the accumulated experience, it can reach 22,2%. A similar, albeit non-linear, relationship exists between the explosion (factor 4) and the fire (factor 7) - Fig. 9 f. Thus, to determine the level of safety of mining operations, it can be considered for the first time, nonparametric statistics methods have been applied.

With a fire probability of 28,52%, the probability of an explosion, rounded at a 95% significance level, can reach 32,21% of all other made-man accidents at the mine.

Considering the dependence of the collapse of excavations (factor 5) as a function of fire (factor 7), it can initially be shown that burning coal, supports and other combustible materials can have a twofold impact on the nature of collapse of stationary and other excavations [5]. In this case, the likelihood of a fire, and, consequently, the collapse of excavations increased (Figure 9 g). So,

if the probability of a fire is up to 28,33%, the probability of collapse is reduced from 39,82% to 8,00%.

Dependence of the consequences of human exposure to a poisoning medium (factor 6) and an underground fire (factor 7) is shown in Fig. 9 h. This connection has a solid logical basis. Indeed, with the increase in the probability of fire (burning out, first of all, oxygen), the probability of personnel poisoning with carbonic or other gas increases. Thus, the increase in the probability of a fire up to 28,33% leads simultaneously to an increase in the probability of poisoning from 1,32% to 13,52%, i.e. in 10,2 times.

Naturally, the fire (factor 7) can be a consequence of the explosion (factor 4) - Fig. 9i. To calculate both the main regression curve and its confidence intervals, nonparametric statistics methods were used [6]. A similar conclusion can be drawn for the curves shown in Fig. 9j. Analyzing the curve shown in Fig. 9 and, it can be concluded that an increase in the probability of an explosion from 0 to 40% leads simultaneously to an increase in the probability of a fire from 3,51% to 27,50%, i.e., explosions are usually accompanied by fires, which looks quite really, proceeding from the available long-term experience of exploitation of coal mines of all categories in gas or dust.

At the same time, it is obvious that the fire, or rather the likelihood of its occurrence at the mine (factor 7), depends on the state of the gas environment, which in some cases is a source of poisoning of personnel of underground mining enterprises. Thus, with an increase in the probability of poisoning of mountain personnel (factor 6) from 0 to 20%, the probability of ignition (fire) in the mine environment increases from 1,75% to 20% (Fig. 9 j).

## **Conclusions**

The methodology developed for the assessment of underground coal mining from the point of view of its safety that was worked out and realized in the present work allows give a deep and comprehensive assessment of mining production outlining ways to reduce the accident rate and then shifting the minimization of costs to safety in the industry.

The numerical estimation of the probability of occurrence of



severe man-made accidents in coal mines can serve as a basis for further introduction of heuristic approaches in the industry, i.e., assessment of production from the position of "man-machine-environment" and risk assessment for specific enterprises of the coal industry.

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# UTILIZATION OF COAL WASTE IN SORBENTS OF WATER PURIFICATION FROM RADIONUCLIDES

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## Abstract

**Purpose.** The main purpose of the work is to increase the adsorption capacity of the carbon-containing sorbent.

**Methods.** The study of adsorption and filtration processes were studied with the aid of chemical and physical and chemical methods, construction of adsorption curves and their processing.

**Findings.** The technology of obtaining effective sorbents multi-purpose processing based on evil, studied changes in the coal structure, and the mechanism of purification and filtration of wastewater in the application of sorbents made in terms of thermal power.

**Originality.** It consists in the study of the surface of the sorbent, sorption mechanism according to the sorbent. It is shown that the structure of the sorbents have non-localized electrons, which are part of the graphite-like carbon crystal mesh.

**Practical implications.** The application of the carboniferous sorbent can improve sorption properties and, consequently, improve the degree of purification of waste water and increase the rate of filtration. Compared with known sorbents cleaning efficiency is increased by 20 – 25% and the filtration rate of 2 – 3.

**Keywords:** adsorption, carbon-bearing products, filtration, waste treatment

## 1. Introduction

In the production of electricity is widely used solid fuel, with the combustion of which generates a significant amount of ash and slag.

In Ukraine, up to 75% of all electricity is supplied by thermal power plants, the main fuel for which is coal, which leads to the release of ash and slag waste in power plants in huge quantities. Every 10 years (according to statistical data), the amount of ash and slag produced by thermal power plants doubles.

Valuable and widespread mineral raw materials for the production of building materials are burnt rocks and waste of coal enrichment.

Ashes are dust-like remnants formed by burning solid fuels, the sizes of which are from 5 to 100  $\mu\text{m}$ . The ash is classified according to the type of fuel to be burned, the way it is prepared and burned, the location of the deposit in the boiler unit, the method of disposal.

Burning of fuel is accompanied by burning of an empty rock, at which the clay substance is dehydrated and low-basic aluminates and calcium silicates are formed. Depending on the amount of liquid phase in the ash, the glass content, which usually has a gellenite-melilite composition, can vary widely.

Ash consists of organic and inorganic phases. The inorganic phase, in its turn, includes aggregated and non-aggregated glass particles, as well as crystalline components.

Coal ash may include the following components according to volume: 1-22% of unburned fuel particles, i.e., the organic phase, 25-85% of aggregated, 8-57% of non-aggregated vitreous, 0.2-20% of crystalline components.

Density and bulk weight of ashes varies depending on the content of unburned fuel particles and granulometric composition in them. The volumetric mass of the ash is also affected by the combustion temperature of the fuel, humidity and degree of compaction of the ash, the value of which is in the range from 1.75 to 2.4  $\text{kg}/\text{m}^3$ , and the bulk density in the friable state ranges from 600 to 1300  $\text{kg}/\text{m}^3$ . The specific surface area of the ashes of different TPPs varies and ranges from 1100 to 3000  $\text{cm}^3/\text{g}$  and more.

Fuel slags are vitreous grains of irregular shape up to 40 mm in size, this is the main type of waste in lump fuel combustion. In the case of pulverized combustion, the slag is 10-25% of the mass of the formed ash. Slags are formed either as a result of sintering of

individual particles on the grate at a temperature above 1000° C, or as a result of cooling the mineral part molten at a temperature of more than 1300° C.

## **2. Purpose**

The main purpose of the work is to increase the adsorption capacity of the carbon-containing sorbent. Isolation of aluminosilicate fraction from fly-ash, study of physical and mechanical properties of astringent materials obtained on the basis of TPP wastes.

## **3. Formulating the problem**

Taking into consideration the growth in the use of solid fuels, in particular a low-calorie with high ash content, the issue of effective use of ash from thermal power plants remains urgent. The output of ash and slag for 1 million KW of power plants is 500 thousand tons. This ash and slag are transported to special dumps. The removal of thermal waste from power plants to dumps uses an average of 0.3% of the total amount of electricity produced by TPPs, that is, considerable material resources are expended. In addition, the storage of waste requires the alienation of land that can be used in agriculture. For example, modern TPP requires for ash dumps from 1000 to 2000 hectares of area.

The level of use of ash and slag is about 15% (for comparison: in the USA - 20%, in France - 62%, in Germany - 80%), and they are used as a rule in the building materials industry: as additives to cement, concrete and mortar, in the road construction, in the manufacture of bricks, and also as a raw material for the production of  $\text{Al}_2\text{O}_3$ ,  $\text{Fe}_3\text{O}_4$ ,  $\text{TiO}_2$ ,  $\text{K}_2\text{O}$ ,  $\text{Na}_2\text{O}$ ,  $\text{P}_2\text{O}_5$ ,  $\text{U}_3\text{O}_8$ , etc. [1]. In some works, methods for the integrated use of ash from TPPs to produce conditional products, used as substitutes for natural raw materials in various industries [2].

Important studies on the immobilization of heavy metal wastes in Portland cement mixtures – ground fly-ash. For the study, aqueous solutions containing zinc, cadmium and mercury were used. It has been established that in the presence of ash the degree of immobilization of metals is significantly increased [3].

In this way, attempts are being made to systematize the ash of TPPs by certain indicators in order to predict the ways of the most rational use of both ash and its individual components obtained by separating the ash in various ways (mechanical, physicochemical, chemical, etc.).

The described technologies include in the technological process the use of plenty of water for washing of the prepared product, which contain insolubles. These coagulants are ineffective for cleaning of radio-active waters. In this connection there is a necessity of production of new coagulants for cleaning of mine waters.

One of asks of creation of new effective coagulants is a reception them on the basis of utilization of wastes of metallurgical, chemical, and also mining industry. We offered the coagulant, consisting of gidrargilit and tails of the oxidized quartzites which are the wastes the process. The technology of reception of the coagulator is based on the conducting of the joint mechanochemical activating of iron-ores and trihydrate of aluminum (gidrargilit). Afterwards will designate this mixture (I).

#### **4. Results of analysis**

Mechanochemical of reaction, mainly, depend on power effort of vehicle for grinding down. Changing the correlation of different types of affecting the ground material blow and abrasion, it is possible in a different degree to change the inner structure of particles. Mechanochemical activating (I) was conducted in a vertical oscillation mill basic advantage of which is the vibroshock affecting the destroyed material. Preliminary experiments on grinding down of different materials showed the high degree of mechanoactivation them as compared to the results of grinding in the vehicles of other types.

The number of experiments on grinding was conducted on a laboratory vertical mill with the use of the different technological modes the row (I). Components were mixed up and jointly ground in the correlations of  $\text{Fe}_2\text{O}_3:\text{Al}_2\text{O}_3 = 1:8, 1:1, 8:1$ . A grinding chamber was filled with grinding bodies such as balls from steel of different diameters. Material was skipped through a grinding chamber certain amount of times for the getting of general way of grinding 1, 2 and 3

m. Thus time of being it in the working organ of mill was regulated. Initial, the intermediate and eventual products of grinding were exposed to the analysis. At the same time power descriptions of the activated surface were measured by the method of pH-titration.

The degree of disactivating is determined by isotopic composition of radionuclides and their condition of in the solution. If radionuclides are adsorbed on dispersible admixtures or are in the colloid-dispersible state, steady decontamination of water is achieved at 97-99%. Thus, the degree of decontamination depends in this case on the degree of lighting up of water.

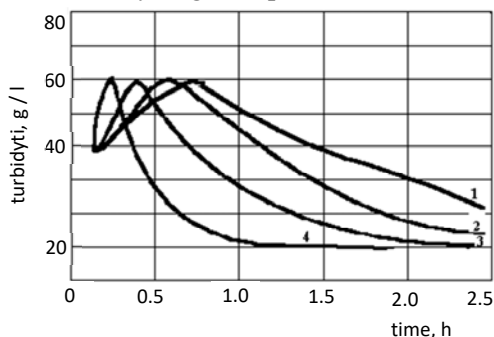
Tests of activated (I) were conducted in the process of cleaning of water on such parameter as turbidity of water and isotopic composition of radionuclides. The rate of turbidity depends on intensity of light diffusion and is proportional to the concentration of the self-weighted matters. Determination of the content of the self-weighted matters was conducted by weighing of dry sediment after coagulation of water.

As is known, process of lighting up of water, from formation of micelle and concluding their besieging, can divide of into a few stages. On the first stage, after introduction of coagulants into the cleared water, there is a hydrolysis of it with formation of micelle and their subsequent aggregate in bigger spherical particles sol (about 0.01-0.1  $\mu$ ). Opalescence appears. This period is called of the hidden coagulation. The second stage is formation of chain structures and tiniest flakes which aggregate in bigger ones. The third stage is connected with segmentation, i.e. settling under the action of gravity flakes of certain sizes. Often these stages do not follow in sequence, but recovered, complicating the process of lighting up.

On a Figure 1 curves characterizing a change turbidity of water in the process of coagulation at a standard coagulant and activated under various conditions (I) are presented. From experimental information it is evident, that the process of lighting take place in of presence activated (I) more quick and practically at once there is the stage of flocculation passing the stage of the hidden coagulation. Rational is the correlation of components 8:1

The substantial disadvantage of the use of salts of  $\text{Fe}^{2+}$  as a coagulant are corrosive activity of solutions, large expense of

chlorine and necessity of careful dosage of the applied reagents. Insignificant rejections in dosages result in substantial violation of the technological mode, to cause by the underoxidation of iron, and, as a result, to the incomplete process of hydrolysis. As a result of it there is appearance of  $\text{Fe}^{2+}$ , due to what water has unpleasant taste, the its coloured and turbidity increase. At application as a coagulant salts of iron it is necessary to give a preference to salts of  $\text{Fe}^{2+}$ .

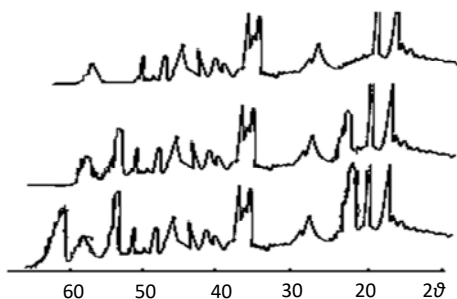


**Fig. 1.** Change turbidity of water in process of coagulating:  
 1 – industrial coagulant; 2 – activated 1:8; 3 – activated 1:1; 4 – activated 8:1

The process of oxidization intensively take place at pH 8. With this purpose before addition of green vitriol or simultaneously with it in more frequent than all dead lime lye is put in water. Oxidization of  $\text{Fe}^{2+}$  is more effective in  $\text{Fe}^{3+}$  passes at joint treatment of water a green vitriol and chlorine. The use of additional reagent results in limitation of application of green vitriol for lighting up and discolouring of water. However in the case of simultaneous lime-soda made soft of water he is an extraordinarily useful reagent.

As a result mechanical activation components takes place before oxidization of iron and its transition of  $\text{Fe}^{2+}$  in  $\text{Fe}^{3+}$  about what testify spectrums, presented on a Figure 2.

From a Figure 2 it is evident, that lances, proper formation of trivalent iron, appear at mechanical activation As a result of activating there is formation of hydrokco-combination of trivalent iron, type  $\alpha$ -  $\text{FeO}(\text{OH})$ , which is the product of hydrolysis of trivalent iron. Therefore the finished product of hydrolysis  $\beta$ -  $\text{FeO}(\text{OH})$  appears mush more quick, than at the hydrolysis of chlorides of iron (contain of iron coagulants).



**Fig. 2.** X-ray tables of activated components:  
1 – initial mixture; 2 – activated 1:1; 3 – activated 8:1

During the mechanochemical activation of ores in a vertical oscillation mill there is formation of clusters in the continuous mode that is obviously evident from radio structural spectrums.

Thus, in a coagulant, got at a vibroladening, iron additionally is not needed to oxi and it forms chains of few atoms, thus accelerates the process of coagulation.

The application of coagulants is very effective with enhance able basicity. They are polymeric hydrocomplex and require considerably less alkaline reserve. As a result mechanical processing (I) ions of iron of hydrolysis in a greater rate and are the centers of coagulation, and as a result of, it large and durable enough flakes. Appear durability of which is determined by the presence of hydrocomplexs aluminum. Exactly these complexes lug away the admixtures of radio-active elements [3].

At the application of the mixed coagulants, got by the method of oscillation ladening, there is stabilizing of pH environment. After activating (I) such important factor for coagulation, as pH environment was measured. It appeared to be equal 8. The application of this coagulant results in diminishing of salts of radio-active ruthenium.

During the mechanochemical activating there is an increase of total adsorption potential. It results in that besides molecular and electrostatic forces the structural factor of aggregate stability of the dispersible systems appears addition.

Because of presence on-the-surface of particles of active centers there are electrostatic forces of pushing away, which cause an



antihunt action. Hydrate shells a round particles promote stabilizing of the dispersible system. At the taking away particles on large as compared to their sizes distance of co-operation between them does not take a place. As a result of brownian motion of the positively charged particles they are drawn together and electrostatic forces pushing away appear, which are added up with molecular attractive powers. With diminishing of distance between particles the resulting action of these opposite forces results in predominating of pushing away. At further rapprochement of force of pushing away diminish and attractive powers begin to predominate. In order that coagulation takes place, particles must overcome forces of pushing away (so-called "power barrier"), that can happen in the case of sufficient large energy of motion of particles or decline of height of barrier. The higher this barrier and the less than energy of motion of particles, the less than probability of their clinging and slower the process of coagulation is absent at all.

To obtain the raw material for heat-insulating materials of fly-ash was subjected to the following operations: preliminary flotation was carried out to separate the inorganic and organic parts, and then the inorganic part was activated in two ways, using chemical and mechanochemical activation. The consumer properties of the initial materials were determined by the phase composition, physicochemical and physico-mechanical properties, as well as conformity with the technical specifications of the subsequent processes of redistribution of the obtained products.

Ash, carbon concentrate (underburn), ash concentrate and products of their treatment with reagents were subjected to optical testing. The morphology of the particles of the objects of investigation was studied with the help of a scanning electron microscope SEM - 100. To identify the composition of ash phases, determinations were made on the X-ray installation DRON-2.

The ash from coal combustion is formed due to all mineral impurities from the coal packs, rock layers of rocks of the roof and soil, getting into the commercial coal during the working out. The main impurities are clay minerals and quartz, carbonates, sulphides, iron oxides and other minerals are present. The phase composition of the ash is mainly determined by clay minerals, which are converted into other aluminosilicates (glass phases, possibly mullite,

crystalite) during high-temperature processing in the furnace of boilers. It should be taken into account that this is a large group of minerals with a variable composition in a wide range [4].

In natural conditions, a group of mixed-layer minerals is the most common. More often there is a combination of layers of hydromica, montmorillonites, chlorites and vermiculites. A common feature of all clay minerals is their aluminosilicate composition.

As a result of the electron-microscopic studies of the morphology of ash phases and its enrichment products, X-ray diffraction analysis revealed features of the microstructure of their surface.

Ashes consists mainly of two components: alumina-silicate particles of spherical shape and unburned coal particles (underburn). Silicate particles of irregular shape are few mainly fragments of spherical particles, occasionally they occur particles of slag and other minerals there. Aluminosilicate particles have a various degree of hollowness: from balls with a thin shell (microspheres), to practically dense (with relatively small inclusions of the rock inside). In all cases, their surface is smooth, sometimes characterized by the presence of small protrusions-nipples.

Through-pores are met on the surface, but as a rule, porosity in the form of small bubbles is common only within the shell of spheres. The size of spheres varies in wide limits: from shares of a micron, to hundredths of millimeters. Large spheres are single and are formed by smaller microspheres. Areas with a size of 5 to 20  $\mu\text{m}$  predominate.

Some spheres are located in the deepings of coal particles, formed during its uneven burning (gasification), therefore the carbon concentrate consists of a mechanical mixture and technogenic intergrowths of aluminosilicate spheres and carbon particles. There are dark opaque spheres, which along with the silicate part contain dendritic magnetite. In addition, during the X-ray phase analysis of samples of carbon concentrates of ash, quartz, magnetite, hematite peaks were determined, and the presence of pyrrhotite and maghemite was also observed.

The chemical composition of the fly ash of the Pridneprovskaya HRES is presented in Table. 1.

**Table 1****Chemical composition of fly ash**

Pro- duct	Content, % on dry substance										
	SiO <sub>2</sub>	Fe <sub>2</sub> O <sub>3</sub>	CaO	MgO	SO <sub>3</sub>	Al <sub>2</sub> O <sub>3</sub>	TiO <sub>2</sub>	P <sub>2</sub> O <sub>5</sub>	K <sub>2</sub> O	Na <sub>2</sub> O	C
Fly ash	49.7	12.3	2.1	1.3	0.4	12.8	0.5	0.9	3.1	0.6	12.0

As you can see from Table 1, after the separation of aluminosilicate spheres (AS) from fly ash, they can be used to produce various grades of concrete. The most common astringent substance used in construction is portland cement

Carbon sorbents can be obtained by high-temperature processing of coal and, in particular, when they are burned. The optimum content of carbon in sorbents can be achieved by controlling the combustion process of coal combustion or by separating the obtained products. The main parameters influencing the carbon content in sorbents obtained by burning coal are the dust dispersion and the air excess factor. When coal grinding is the carbon content in the fly-ash grows and the same with decreasing in the excess air factor.

The physical and physico-chemical properties of coal in ashes and fly-ashes are closely related to their molecular structure, which was formed during metamorphic transformations in the coal body [5]. A feature of this structure is that this picture of a complex structure is realized not only inside the samples, but also on their surface.

It is known that the distance between the carbon layers in the coals is 0.344 nm (in some cases for well-designed coals it reaches 0.399 nm), i.e. somewhat larger than the interlayer distance in graphite (0.335 nm). These figures allow us to talk about the weaker interaction between carbon layers in active coals compared to graphite. At present, there is a common opinion, that the interaction between the carbon layers in the coal crystallites is due to the Van der Waals forces interaction.

Thus, from the above discussion of the molecular structure of coals, that an element of the three-dimensional structure of the active coal is graphite-like crystallite, which for the nonactivated samples in ideal case has a cylindrical shape with a base-to-height ratio of 2:1 and a surface area is about 13 nm<sup>2</sup>. When looking at such a

crystallite, attention is drawn to the existence of two types of surfaces, one of which is formed by graphite-like carbon networks (the base of the cylinder) and the other by edge atoms located along the periphery of the carbon grids (forming the surface of the cylinder). This very fact alone causes the existence of at least two types of inhomogeneity of the adsorbing surface of active carbons.

The porous structure includes the characteristic of free space in the volume of carbon bodies, which is formed as a result of the action of oxidizing reagents on the carbon material at increased temperatures. The most extensive and systematic, theoretical and experimental research in this field belong to M.M. To Dubinin. Macro pores of active carbon have radii of curvature of the surface of 100-200 nm, their specific surface area is 0.5-2.0 m<sup>2</sup>/g, the volume is 0.2-0.5 cm<sup>3</sup>/g. For the mesopores, these values are respectively 2-100 nm, 20-70 m<sup>2</sup>/g and 0.1-0.8 cm<sup>3</sup>/g.

The smallest variety of pores of carbon adsorbents can be conveniently divided into micropores for which the characteristic dimensions in the direction normal to the movement of molecules during filling are less than 0.6-0.7 nm and supermicropores for which the indicated dimensions are in the range 0.6-1.6 nm.

The total volume of micropores is usually in the range 0.2-0.6 cm<sup>3</sup> / g. It's known for a long time that micropores of active coal are not random intervals between randomly arranged crystallites, but free volumes in its molecular structure that appear when the carbonaceous substance burns out. Micropores with a more regular structure are formed with partial or complete combustion of carbon layers of crystallites, at this mainly lesser perfect crystallites of small sizes burn out. As a result, slit-like micro-pores are formed in the range between the circular bases (walls) corresponding to the shape of carbon crystallites. The parameters of such an idealized model of micropores are the radius of the circular base and the half-width of the flat slit [7].

Surface chemical compounds of active carbons do not represent a new phase on coal, but are functional groups of one or another nature, connected with peripheral carbon atoms in graphite-like grids of coal crystallites. Despite of a large number of works devoted to the chemistry of coal surfaces, the question of the nature of surface chemical compounds of active coals can be considered one of the

most complex and least understood. In this regard, that most researchers connect both the acid-base and oxidation-reduction properties of active carbons with the existence of certain functional groups of different nature on them. Since the study of the chemical nature of the surface compounds of coal used an investigation of their sorption capacity with respect to acid and alkali, a representation of the basic and acidic oxides on the surface of the coal was introduced. Taking into consideration, that the amount of oxygen chemisorbed by coal always exceeds its content in basic or acidic groups, to explain this discrepancy, a notion of non-ionic surface oxygen compounds was introduced.

If the coal after activation comes into contact with oxygen or air at a low temperature, for example 0-100° C, surface compounds of the main character were formed on the carbon. Although the nature of the main oxides is still insufficiently understood, it has been established that the main oxides are products of chemisorption of oxygen on coal, which correspond to high values of the oxygen adsorption energy at the more active sections of the coal surface. The oxygen of these oxides, when in contact with water or aqueous solutions, penetrates into solution as hydroxyl ions, charging the surface of the coal positively.

At temperatures of 400-500° C, the interaction of oxygen with coal takes place in a different way. In this case, coal chemisorbs 13-15%, and according to some data up to 22-25% of oxygen, which is firmly bound by coal in the form of various surface oxygen-containing compounds. Approximately 1/5 part of the oxygen chemisorbed under these conditions is included in the various protogenic groups.

The aim of this work is to increase the adsorption capacity of the carbon-containing sorbent.

The features of the porous and molecular structure of active coal, its electrophysical properties, the chemical nature of the surface work forward the fact that the dissolved substances on the carbon adsorbent can be absorbed by completely different mechanisms, their sorption is due to the different nature of the forces participating in the absorption process.

So, dissolved substances are absorbed on the coal in the form of whole molecules due to the high specific surface area and the special

nature of the porosity – the manifestation of van der Waals interaction forces; according to the process of ion exchange, determined by the chemical nature of the surface of coal; by the formation of surface complexes of donor-acceptor type and the reduction of adsorbed substances, which is determined by the electron state of the coal.

To study the mechanism of interaction of dissolved substances with active carbon, we used the method of changing the gas atmosphere. This method is based on the ability of platinized coal to recharge during changing the gas atmosphere and, in this connection, change its adsorption behavior with regard to dissolved electrolytes. In an atmosphere of air (oxygen), platinized coal adsorbs anions from electrolyte solutions. On contrary, in hydrogen, coal is recharged and acquires the ability to absorb cations from solutions.

To clarify the mechanism of sorption of solutes by this method, adsorption experiments are carried out in parallel in an air atmosphere and in hydrogen. If adsorption is carried out by ion exchange, then, depending on the sign of their charge, they must be absorbed in the atmosphere of only one gas (the anions in the air atmosphere, cations – in hydrogen). The change in the gas atmosphere has practically no effect on the adsorption of a purely molecular type. In the case of specific sorption (donor-acceptor interaction), its value in one of the atmospheres should respectively decrease or increase.

This method was used in our work to understand the mechanism of sorption of a large number of organic and inorganic compounds by active carbon, depending on their chemical, electronic structure, dissociation ability, instability constant (for complex compounds), etc.

To obtain the sorbent, the coal raw material was subjected to a coarse vibrograde to a particle size of 10 to 500  $\mu\text{m}$ , with a coarse fraction of  $-90 \mu\text{m}$  10 to 70%, and burned in boilers of thermal power plants (TPP) at 1200 to 1500° C. As a result of this processing of coal raw material at the indicated temperature regime, the yield of the sorbent increases and the obtained product consist 45-60% of a carbonaceous fraction and 20-55% of spherical silicate particles.

The mineral component of the sorbent consists of particles of aluminosilicates of globular shape, the presence of which allows to

increase the filtration rate of the water to be purified through the sorbent bed, and also to increase the efficiency of their purification. The presence of particles of coal and aluminosilicates possessing different hydrophobic properties in the composition of the filter mixture increases the sorption capacity of the filter.

The sorbent becomes particularly effective in the purification of wastewater from viscous oil products having combination of these properties. When the sorbent is mixed with wastewater contaminated with, for example, fuel oil and viscous machine oils, oil agglomeration take place, i. e. the formation of agglomerates of granule type with a size of 1 to 5 mm, consisting of sorbent particles.

The choice of the granulometric composition of the feedstock for obtaining the sorbent according to the proposed method is determined by the coal brand. Coals grade AH and, partly, lean coals should be used with a content of the fraction  $-90\ \mu\text{m}$  10-40%, and coals with a lower degree of metamorphism, for example grades G and ZH, should be use of coarser grinding: the content of the fraction  $-90\ \mu\text{m}$  is 50 -70%.

The specific surface of the sorbent (ash) obtained by the known method from different grades of coal is 1.1-5.9 m/g. The specific surface of the sorbent in the proposed method exceeds this value as a result of an increasing of the content of the coal fraction, and the porosity of the coal particles as well. It is 20-30 m/g for different fractions of coal with the tendency of increasing for coals with a low degree of metamorphism (grades G and ZH).

Thus, the use of the proposed method for the preparation of a carbon-containing sorbent allows to improve its sorption properties and, as a result, to increase the degree of wastewater of purification and to increase the filtration rate. In comparison with known sorbents, the cleaning efficiency is increased by 20-25%, and the filtration rate 2-3 times. In addition, the method makes it possible to reduce the cost of the filter mixture, increase the yield of the sorbent, and expand the base of sorption materials.

The features of the molecular structure and electrophysical properties of active coals (AC) of the porous structure and the chemical nature of the surface determine the energy inhomogeneity of the coal surface, which affects chemical, chemisorption and adsorption interactions.

Due to the fact that active coals along the plane of carbon grids and in a direction perpendicular to the carbon layers of crystallites exhibit different properties, we can suppose that the bond strength of the sorbed molecules and ions on different sites of the surface of the carbon sorbents will differ substantially. The method of radioactive indicators was used to evaluate the strength of the bond of sorbed ions on the carbon isolated from the fly ash.

The method of heterogeneous isotopic exchange has become very widespread for solving many problems of the theory of chemical structure, reactivity and the mechanism of chemical reactions. Like any isotope exchange reaction, heterogeneous redistribution of isotopes is a reversible chemical reaction with kinetics and an equilibrium constant. In many cases, exchange equilibrium is achieved relatively quickly. However, in some cases, the exchange is slow, and its speed is influenced by various factors, including the energy inhomogeneity of the surface of the adsorbent. In connection with this, this method has been successfully used to study the nature and heterogeneity of the surface of the adsorbent.

The essence of this differential isotope method is as follows. A monolayer of labeled adsorbate is applied to the surface under study and then brought into contact with the same adsorbate, but no longer labeled. The state of adsorbed molecules is significantly different on a homogeneous and inhomogeneous surface. On a homogeneous surface, all molecules or ions, regardless of the order in which they were placed on it, are in the same energy state, that is, are connected with the surface by the same forces. Conversely, on an inhomogeneous surface, the molecules are in unequal states and differ in the strength of the bond with it. This is clearly seen in studies of the kinetics of the exchange of isotopes or in the desorption of isotope-labeled particles. There are three reasons for the heterogeneity of the surface: the presence of places with different adsorption energy; the presence of interaction between adsorbed particles; adsorbed particles change the properties of the surface, which before the sorption was uniform.

The study of exchange reactions between metal electrodes and ions of the same name in solution using radioactive isotopes is of great interest, since the specific character of the adsorption behavior of active carbon in electrolyte solutions lies in its ability to function



as a gas electrode, and the kinetic regularities of the exchange of ions adsorbed on the active carbon with the same-named solution ions can largely be identical with those of metal electrodes.

In literature, however, there are no papers in which the method of radioactive indicators would be used to study the regularities of isotopic exchange in the coal-solution system. The use of the isotopic exchange method in this system is of interest for solving a number of theoretical problems: the ascertaining of the effect of heterogeneity of the surface of carbon sorbents on their sorption properties, the reasons for the selectivity of individual modifications of active carbons, as well as in the practice of selecting moderate adsorbents for the separation of mixtures of ions of complex composition.

Studies have shown that the most stable modification of active coal oxidized coal can be successfully used to separate cations in multicomponent solutions. Despite the large number of works, devoted to the study of active coals, the question of the quantitative evaluation of the strength of the bond between absorbed ions and the surface of this sorbent can't be considered definitively clarified, although it is the subject of many studies.

The sorption selectivity of oxidized coal was estimated from the concentration constants of the exchange of cations of different nature and valence on it. Concentration exchange constants on the whole correctly characterize the selectivity of ion-exchange properties of coal in accordance to ions of different nature and allow one to foresee the features of its adsorption behavior in various systems.

To assess the strength of the bond of ions to the surface of coal, it is naturally necessary to involve research methods, allowing to clarify the inhomogeneity of the adsorption properties of the surface of carbon adsorbents.

One of the ways by which it is possible to estimate the relative strength of the bonding of sorbed ions to the surface of coal and its heterogeneity on various sites of the surface is to study the kinetics of exchange with the same ions, which are in solution.

As follows from the results of investigations of the equilibrium exchange of ions on smooth metal surfaces, the strengthening of the bond in the metal - solution system is expressed in the slowed-down kinetics of the exchanging ions. To obtain reliable data on the true exchange rates in the coal-solution system, in which the solid phase

is represented by a granular porous material, it was necessary to eliminate the diffusion limitations associated with both the grain size of the adsorbent and its porous structure. On the basis of the above considerations, such fractions of active coal were used in the work, so that, according to the equation of ion diffusion in solution, the completion time of the exchange in the absence of bond strengthening did not exceed the time required for sampling the system (1 min.). This condition was satisfied by powdered coals with a grain diameter of 0.25 - 0.50 mm, activated prior to firing  $\geq 50\%$ .

The reaction of exchange of cations adsorbed on oxidized carbon with the same ions, which are in solution were investigated. The exchange was carried out in the state of adsorption equilibrium of the coal - electrolyte system in order to exclude any effects associated with adsorption during the exchange process. The rate of redistribution of ions between the solution and the surface of the coal was connected with the change in radioactivity by a certain point in time.

From Table 2, in which data are presented on the kinetics of the exchange of  $\text{Na}^+$  and  $\text{Rb}^+$  cations on the oxidized carbon, it is clear that although a significant portion of the adsorbed cations is exchanged in less than 1 minute, in general, the exchange here takes place slowly and reaches equilibrium after 2-3 hours.

**Table 2**

**The exchange of  $\text{Na}^+$  and  $\text{Rb}^+$  cations on the oxidized carbon in 0.01N solutions of NaOH and RbOH**

Exchange time, min	Exchange in solution NaOH		Exchange in a solution of RbOH	
	Specific radioactivity of coal after exchange, imp/min·mg	Degree of exchange, %	Specific radioactivity coal after the exchange, imp/min·mg	Degree of exchange, %
1	740-7	29	—	—
3	820-8	32	324-6	69
5	995-10	39	320-6	68
15	1940-14	76	—	—
30	2040-15	80	337-6	72
60	2370-16	93	357-7	76
180	2550-16	100	—	—
240	—	—	399-8	85
350	—	—	425-9	91

A convincing illustration of the fact that the cause of the slowed exchange of ions on coal is not associated with the manifestation of factors of diffusive nature, are comparative data on the kinetics of isotopic exchange of cations on a porous adsorbent coal and nonporous oxidized (ductile) soot (Table 3).

**Table 3**

**Kinetics of isotopic exchange of  $Tl^+$  and  $Sr^{2+}$  cations on oxidized soot and carbon in nitrate solutions**

Exchange time, min.	Degree of exchange of cations $Tl^+$ , %		Degree of exchange of cations $Sr^{2+}$ , %	
	soot	coal	soot	coal
1	67	33	43	26
3	67	58	52	36
5	83	66	53	47
15	90	96	56	70
30	91	100	60	80
60	93	102	65	82
120	100	–	62	96
180	–	–	64	98
240	–	–	69	99
360	–	–	94	100
720	–	–	95	100
Total sorption of cations, mg-eq/g	0,014	0,110	0,008	0,107
Strong sorption of cations, mg-eq/g	0,010	0,097	0,003	0,077
The rate constant of isotopic exchange, $K_p \cdot 10^2, \text{min}^{-1}$	0,330	9,750	0,160	0,730

There are all reasons to believe that a slowed exchanged; part of sorbed cations on oxidized coal and soot is due to the action of the same factors and, above all, to the manifestation of specific interaction forces of sorbed ions and oxygen atoms of surface groups. It is noteworthy that the exchange of cations in soot is slower than in coal. This fact can apparently be explained by the fact that all the chemisorbed oxygen of the soot (canal soot it contains 3.75-4.25%) is part of the surface functional groups. The oxygen content in the oxidized coal does not exceed 14%. And only 20% of this amount are included in the surface functional groups.

The decrease in the bond strength of adsorbed cations in the

surface layer of coal is due to a decrease in the electron density on the oxygen atoms of the surface groups due to an increase in the relative content in the coals of electron-withdrawing atoms of nonionic chemisorbed oxygen.

In this way, strengthening of the  $-O-Me^{n+}$  bond in the surface layer of coal is determined by the peculiarities of the chemical nature of the surface of the carbon adsorbent, rather than by the porous structure of the coal on which the diffusion kinetics depends. The kinetic isotope method makes it possible to divide the total amount of sorption in a weak and strong one former of which is due to purely electrostatic interaction in the double layer of coal, while the latter is a manifestation of specific chemical forces during the interaction of cations with oxygen-containing surface compounds of coal.

A more accurate characteristic of the exchange is its rate constant, which for isotopic exchange on a homogeneous surface can be calculated by the formula:

$$K = \frac{c_p \cdot V}{(am + c_p \cdot V)} \cdot \frac{1}{t} \ln \frac{A_0 - A_\infty}{A_t - A_\infty} \quad (1)$$

$c_p$ ,  $V$ ,  $am$  – the number of exchanging ions that are in solution and adsorbed on coal, mg-eq;  $A_0$ ,  $A_t$ ,  $A_\infty$  – specific activity of the solution at the beginning of the experiment, at time  $t$  after reaching the equilibrium distribution of isotopes on coal and in solution, imp/min.

The expression for the exchange rate constant was obtained on the basis of the following considerations.

At the beginning of the experiment there is  $A_0$  activity in the solution. Its decreasing as a result of exchange of ions in a time:

$$dA = \frac{A_0 - A}{am} K_a dt - \frac{A}{cV} K_g dt \quad (2)$$

$A$  – the activity of the solution at a given time;  $m$  – the weight of coal;  $a$  – adsorption;  $c$  – concentration of solution;  $V$  – volume of solution;  $K_a$  – is the adsorption rate constant;  $K_g$  – is the rate constant of desorption.

The exchange takes place in the state of adsorption equilibrium, so  $K_a = K_g$ . Equation (2) takes the following form:

$$dA = K \left[ \frac{A_0 cV - A(cV + am)}{amcV} \right] dt \quad (3)$$

We change expression (3) to a form convenient for integration:

$$\frac{dA}{A_0cV - A(cV + am)} = \frac{K}{amcV} dt \quad (4)$$

We change the left-hand side to the form

$$\frac{dx}{x} = d \ln x \quad (5)$$

$$\frac{dA(am+cV)}{(am+cV)[A_0cV - A(cV + am)]} = \frac{K}{amcV} dt, \quad (6)$$

Since  $adx = d(ax) = d(ax + b)$ , if  $ab=0$ , then

$$-\frac{d[A(am+cV) - A_0cV]}{(am+cV)[A(am+cV) - A_0cV]} = \frac{K}{amcV} dt. \quad (7)$$

We have  $dA_0cV = 0$ , as  $A_0cV$  - is a constant term.

The activation energy for the isotopic exchange of cations of different nature is given in Table. 4. Its values were calculated from the temperature dependence of the rate of isotopic exchange of  $\text{Na}^+$ ,  $\text{Ca}^{2+}$ ,  $\text{Co}^{2+}$  cations sorbed on the oxidized coal. The activation energy of the exchange processes of cations  $\text{Na}^+$  approaches its value for diffusion processes and is 10.3 kJ/mol. An increase in the activation energy of this process for  $\text{Ca}^{2+}$  cations up to 16.7 kJ/mol and  $\text{Co}^{2+}$  up to 33.4 kJ/mol indicates an increasing influence of specific forces of interaction at the sorption of these cations, the activation energy is of the order of the activation energy of the chemical reaction.

**Table 4**

**Dependence of the values of the total and strong sorption and the rate constants of the exchange of  $\text{Na}^+$ ,  $\text{Ca}^{2+}$ ,  $\text{Co}^{2+}$  cations on the temperature**

T°C	Total sorption, mg-equiv/g	Strong sorption, mg-eq/g	$K \cdot 10^2$ , $\text{min}^{-1}$	E, kJ/mol
$\text{Na}^+$				
5	0,060	0,035	27,8	–
20	0,060	0,019	33,0	10,3
50	0,060	0,017	–	–
80	0,060	0,001	–	–
$\text{Ca}^{2+}$				
20	0,101	0,047	8,2	–
50	0,120	0,037	11,8	1,7
80	0,136	0,027	21,3	–
$\text{Co}^{2+}$				
5	0,148	0,114	2,0	33,4
20	0,148	0,103	3,8	–
50	0,156	0,065	10,3	–

This correctly reflects the huge differences in the sorption of these cations by oxidized coal. All mentioned above suggests that the selectivity of coal as a cation exchanger is indeed due to the strong binding of a part of the sorbed cations in the surface layer of coal, a qualitative characteristic of which can serve the exchange rate constant, determined in the state of equilibrium of the coal-solution system by means of an isotopic method.

To obtain data confirming the rightness of the above considerations we investigated the kinetics of the isotopic exchange of cations on synthetic cation-exchange resins with the KU-2 sulfonate cationate without selective action and having selectivity of relation of two or more charge ions of the carboxyl cationate (KB-4P) and for comparison on an oxidized coal.

We see from Figure 3 that for cations  $\text{Na}^+$  on cation exchangers KU-2 and KB-4P-2 the dependence is expressed by straight lines passing through the coordinates origin. This indicates that the isotopic exchange of cations on them obeys the same regularity within the entire value of the sorption.

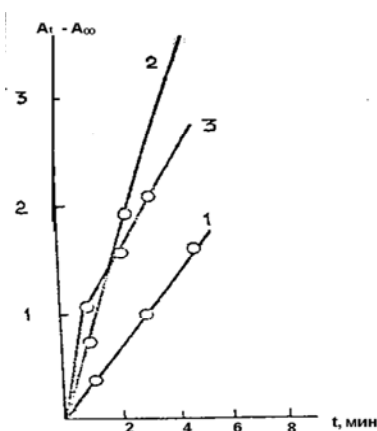
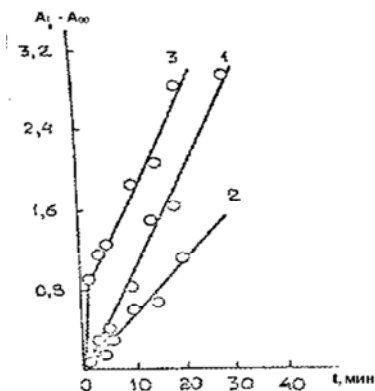


Fig. 3. Kinetics of isotopic exchange of  $\text{Na}^+$  cations in 0.001 N solution on cation exchangers: 1 - KU-2, 2 - KB-4P, 3 - oxidized coal

An analogous dependence is also observed in the case of the exchange of  $\text{Ca}^{2+}$  cations (Figure 4), whereas for both mentioned cations, the isotopic exchange on the oxidized coal is described by a polygonal curve consisting of two rectilinear sections.

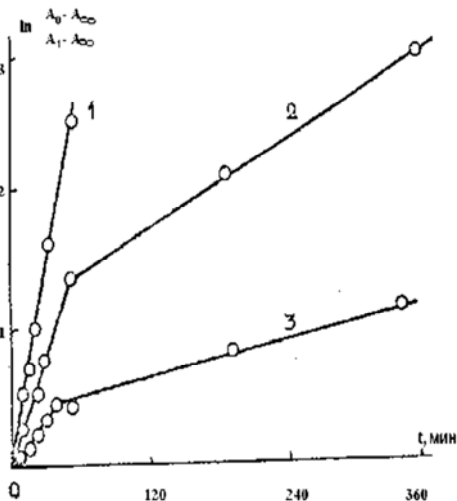


**Fig. 4.** Kinetics of isotopic exchange of  $\text{Ca}^{2+}$  cations in 0.001 N cation exchangers: 1 - KU-2, 2 - KB-4P, 3 - oxidized coal

As might be expected, the isotopic exchange of  $\text{Fe}^{3+}$  cations on the carboxyl cationate KB-4P-2 proceeds like an exchange on adsorbents with an inhomogeneous surface (Figure 5). The presence of two sections on the kinetic curve for these ions is probably associated with two types of interaction of  $\text{Fe}^{3+}$  ions with a cation, which can be due to both pure ion-exchange absorption and the formation of complexes with functional groups of the resin. In this case, the selective effect of the carboxyl cation exchanger with respect to cations of ferrous metals is determined only by the stronger electrostatic interaction of multiply charged ions with a polyanion, but also by the formation of intracomplex compounds with a coordination bond between the counterion and the carboxylic residue in resin.

It can be reasonably assumed that there must be a definite quantitative relationship between the values of the strong sorption of cations on the oxidized coal and the rate constants of the exchange of these ions, on the one hand, and the sorption selectivity of the oxidized coal with respect to ions of this species on the other hand.

For this purpose, the kinetics of exchange of a large number of ions of various nature and valence adsorbed on an oxidized carbon with the same ion solutions of the corresponding salts was studied.



**Fig. 5.** Kinetics of isotopic exchange of  $\text{Fe}^{3+}$  cations in 0.001 N solution on cation exchangers: 1 - KU-2, 2 - KB-4P, 3 - oxidized coal

In Table 4 shows the calculated values of the strong sorption of the rate constants for the exchange of cations of alkaline, alkaline-earth, some transition elements of the IV period, as well as metal cations of subgroup IIIB of the Periodic Table of Elements. Among the ions studied, the alkali metal cations are less strongly bound to carbon, the exchange rate constants of which have the highest values. Fact, that the exchange rate constants for one charged cations of the elements of the IA group decrease with increasing atomic number of element it is possible to tell about the effect of the polarizability of the ion on the strengthening of its bond with oxygen-containing compounds of the coal surface, which increases in the row from  $\text{Na}^+$  to  $\text{Cs}^+$ . Similar kinetic regularities of exchange are also observed in the case of alkaline earth metal cations, for which the values of strong sorption increase, the exchange rate constants decrease in a row from  $\text{Ca}^{2+}$  to  $\text{Ba}^{2+}$ .

So,  $K_p$  of  $\text{Fe}^{3+}$  ions is 300 times more than the corresponding value for  $\text{Na}^+$  ions. (Table 5).



Table 5

**Kinetic characteristics of the isotope exchange of cations on oxidized coal in the state of the adsorption equilibrium of the coal system – solution**

Cation	Electrolyte	pH	Total sorption, mm <sub>e</sub> / g	Strong sorption, mm <sub>e</sub> / g	Constant rate of exchange, K·10 <sup>-2</sup> , min. <sup>-1</sup>	The criterion of adsorption
Na <sup>+</sup>	NaCl	3,1	0,048	0,019	35,0	1,00
Rb <sup>+</sup>	RbCl	3,1	0,056	0,018	21,2	1,56
Cs <sup>+</sup>	CsCl	3,0	0,067	0,048	7,92	10,50
Ca <sup>2+</sup>	Ca(NO <sub>3</sub> ) <sub>2</sub>	2,8	0,101	0,047	8,16	10,60
Sr <sup>2+</sup>	Sr(NO <sub>3</sub> ) <sub>2</sub>	2,8	0,107	0,077	4,99	28,40
Co <sup>2+</sup>	Co(NO <sub>3</sub> ) <sub>2</sub>	2,8	0,112	0,91	3,42	49,00
Y <sup>3+</sup>	Y(NO <sub>3</sub> ) <sub>3</sub>	3,0	0,184	0,81	2,24	66,60
Sc <sup>3+</sup>	Sc(NO <sub>3</sub> ) <sub>3</sub>	2,7	0,330	0,198	2,36	109,00
Fe <sup>3+</sup>	Fe(NO <sub>3</sub> ) <sub>3</sub>	2,8	0,304	0,230	1,03	184,00

Since the ability of any ion to be selectively sorbed by oxidized coal from a mixture with other ions is determined as a whole by the strength of its binding in the surface layer of coal; a quantitative characteristic that must take into account both the value of strong sorption and the rate constant of exchange in order to take into account the influence of these two interrelated factors that determine the selectivity of the sorption of ions by coal, the concept of the adsorption criterion is introduced. The criterion for the adsorption of a given ion with respect to some other ion taken as a standard is

$$K_p = a_{np1} \cdot K_0 / a_{np0} \cdot K_1.$$

By the criterion of adsorption, the quantitative differences in the relative ability to sorption by coal are most clearly expressed.

## 5. Conclusions

As a result mechanochemical treatments there is a change of power descriptions of the ground up matter, that results in structural changes in a superficial layer (I). Appearny power barrier layers are cause intensification of process of coagulation.

The addition of clean hydracids of aluminum as a coagulant results in formation of flakes with the relative density of equal 1.001-1.003 kg/l. It is related to the fact that the volume of particulate matters in

unit of volume flakes is small and is measured as the tenths of percent. Thus, appearing loose and easy flakes can't fully clean the colored little turbid waters. The application of the mixed coagulants, got by the method of vibroladening results in formation of durable smaller crystals flakes not depending upon the cleared water. It can be explained that the activated mixed coagulant contains hydro complexes of aluminum and iron simultaneously.

The analysis of experimental data is shown the expedience of the application of these coagulants. On their basis compositions can be also offered and for ionochange resins, applied in industry for cleaning of sewer and industrial waters.

1. It has been revealed that during high-temperature processing of coals, multi-purpose sorbents can be obtained.

2. It is established, that the most effective sorbents of low cost can be produced in TPP conditions with special burning of coals of certain grades.

3. It has been found that the adsorption mechanism involves the donor-acceptor interaction, which results in the formation of strong structures immobilizing water-pollutant reagents

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# THE SITUATION FORECAST FOR THE EXTRACTION OF ORE RESERVES IN THE WEAKENED SECTIONS OF THE DEPOSIT

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**Abstract.** The paper considers the methodology of the situational forecast for mining operations in weakened sections of the field. The urgency of the forecast is due to extraction of ore in conditions of complex geotechnical situation. The methodology is based on three innovative methods of zone zonation of the day surface of the deposit by the degree of weakening and a complex geomonitoring system for the state of the earth's surface in problem areas based on modern surveying topographic and geodesic and aerospace technologies. The digital geoinformation model of the deposit (DGMD) and the geoinformation model of geomechanical risks (GMGR) have been created for the effectiveness of the use of the method. Based on the simulation results, continuous situational maps of geomechanical risks are constructed that reflect the development of deformation processes in the mountain range. To make forecasts in the model, the parameters of various mining options are loaded, the forecast situation maps of each variant of the plans are issued at the output. Practical approbation of the method was carried out in the conditions of the Annenskoye deposit. The technique has shown its effectiveness. As a supplement to it, on the recommendation of economists, for the convenience of calculating loss of balance reserves of mineral resources (BRMR) for zone zoning, a parameter characterizing the distribution of the FWIP for each element of the zone was introduced.

## Introduction

At the stage of completion of ore deposits depletion of ore reserves and a steady tendency to reduce the content of the useful component in crude ore toughen in the conditions of a market

economy requirements for quality and quantity of raw materials. In this regard, an increase in the extraction of mineral raw materials necessitates the extraction of remaining ore reserves in the mined rock mass, including those supporting the overlapping strata.

Geotechnical processes complicate mining, creating a threat to safe mining. This leads to loss of ore, causing the destruction of structural elements of mine workings and damage to objects on the surface. In this case, large geodynamic events (caving and man-made earthquakes) can cause death of people and lead to huge material costs.

The most important part of the assessment of operational risks is risk management, i.e. a set of measures to prevent a dangerous situation.

Therefore, in order to ensure measures to prevent a dangerous situation, safety and increase the efficiency of mining, first of all, it is necessary to know the places of possible catastrophic events in advance before geotechnical support and to choose the technology of extraction according to the current situation. This can be provided by preliminary diagnostics of the state of the rock massif containing the deposit of minerals and dividing the earth's surface into zones according to the degree of weakening. The results of such diagnostics make it possible to increase the validity of geotechnical solutions while further development of the remaining reserves of mineral resources and on the sites of possible geodynamic events of the weakened part that have already been identified as a result of diagnostics, monitoring should be organized with the aim of predicting them.

The identification of weakened zones on the surface of the deposits allows us to concentrate monitoring, increasing its intensity and accuracy.

Studies of deformation processes, their control and prediction determine in many cases the efficiency and safety of the development of deposits of solid minerals. A practical forecast can be implemented as a result of continuous tracking in space and time for deformation processes.

The system of geomechanical monitoring of deformations of the earth's surface should contain in its composition the following basic methods:

- methods of preliminary diagnostics of the rock massif;
- repeated high-precision leveling;
- satellite geodetic methods, first of all, methods of space-based radar interferometry;
- Other methods of instrumental observations at regional and local sites.

In this connection, it should be noted that the ground methods used for geomechanical monitoring of deformations of the earth's surface, such as re-geodetic leveling, the use of GPS measurements do not fully reflect the temporal detail and spatial scale of the changes that have taken place in the relief of the earth's surface.

At the present time, the methods and technologies of the SRI, which allow us to obtain area estimates of vertical and planned displacements of the earth's surface with accuracy to the first millimeters regardless of the conditions of illumination and cloudiness, are of particular practical value. Space based radar interferometry (SRI) is an effective means of direct mapping of earth surface movements and deformations of structures on large areas of the study areas.

Therefore, great prospects in terms of creating an effective system for geomechanical monitoring of deformations of the Earth's surface have the joint use of methods of space radar interferometry (SRI) and ground-based surveying-geodetic measurements. At the same time, surveying-geodetic methods (electronic tacheometric survey, laser scanning, GPS, high-precision ground leveling) can be used both independently and to verify the results of the application of space-based radar interferometry. Thus, each monitoring method is mutually complementary, expanding its capabilities.

Based on the zone zoning of the surface and the results of geomechanical monitoring, a method for situational forecasting of mining operations has been developed. An important part of the methodology are DGMD and GMGR - models. Based on the simulation results, continuous situational maps are compiled.

### **Identification of weakened zones on the surface of the deposit**

Experimental studies of forecasting issues have shown that the most important is the time aspect of the forecast. Determination of its

prognostic characteristics at all the identified stages of the development of extraordinary events right up to the moment of their implementation, based on kinetic representations ( $t \rightarrow t_0$ ), is possible only by continuous monitoring of the rock mass, including observation, assessment and prediction of the state of the mining massif. The identification of weakened zones on the surface of the deposits allows us to concentrate monitoring, increasing its intensity and accuracy.

Thus, prior to the implementation of geotechnical support, there are a number of limitations and specific problems that need to be foreseen and solved. This is primarily achieved by preliminary diagnosis of the rock massif and allows to provide a long-term forecast of the expected consequences of mining operations, determining for the future the strategy of finalizing the mine fields of the fields.

### **Zone division into districts of a surface by the method of multiplicity criterion $H/m$**

The known fact that the parameters of the geodynamic event and the time of its development depend mainly on the height of the worked area and the overlapping rocks from the boundary of the worked out space to the surface, taking into account the opening ratio. In this case, the collapse of the overlapping rocks and the failure funnel depend on the ratio of the volumes of the overlapping rocks and the worked out space.

Since the collapse of the overlapping rocks and the collapse funnel depend on the ratio of the volume of overlapping rocks and the worked out space, determining the ratio of the  $H$  - depth to  $m$  - the thickness of the worked-out space in the deposit space on the surface plane, we obtain the contour of possible funnel formation.

On the basis of a retrospective analysis of the collapses occurring at the Zhezkazgan deposit, the conditions for the formation of a dip are determined by the ratio  $H/m < 10$ . Method [1] is carried out as follows. For the entire area of the deposit, using existing sections based on geological and exploratory wells, as well as geomechanical documentation, all overlaps are applied to the plan, including arrays that are not subject to development.

The depth and thickness of each spent deposit are measured along the vertical axis. Then the values of the depths and powers are summarized, the results of which are carried to the plan. Using the set of available geological and exploratory wells over the whole area of the deposit or its individual sections, the values of total depths  $H$  and powers  $m$  are determined for each surface point. Based on the results of the total power on the plan, the extrapolation method constructs isolines with a certain interval.

The revealed volumetric anomalies of the total power are superimposed on the mining plan of the deposit. The most dangerous are the zones with the maximum values of the total production capacity of the deposit. Then, using the obtained values of the total power and depth, determine the ratio of the total depth  $H$  to the total power  $m$ . Based on the results, the contour multiplicity  $H/m$  is also constructed by the extrapolation method. The criterion for forecasting caving is  $H/m < 10$ .

By superimposing these plans with the isolines of the identified criteria for a mining plan, anomalous zones are identified that are potentially dangerous due to collapse. In this case, for hazardous areas are taken areas where the isolines of the values coincide according to the criteria  $m$  and  $H/m$ . These plots are the most dangerous in terms of collapse compared to the rest of the field, as evidenced by the practice of collapses in the field.

With further mining operations, the mining and geomechanical situation changes, and the values of  $m$  and  $H / m$  undergo changes. On such sites, for each cycle of clearing works, changes are made on the corresponding plans and again carry out a forecast of zones that are dangerous for collapse.

This method is particularly effective for such sections of the field where there is no possibility of using rock mass monitoring. The share of such sections of the field is currently more than 30%. At sites where monitoring is carried out, the hazardous areas identified by this method are subject to special control to determine the lifetime of their monitoring facilities, including monitoring deformations, seismic monitoring and monitoring of local areas.

The introduction of this method allowed, according to the established criteria, to perform preliminary diagnostics of the rock massif, to predict long before the realization of the place of possible

catastrophic phenomena, such as collapse with access to the earth's surface. The introduction of the method also contributes to the timely adoption of protective measures, the effective and safe management of mining operations in difficult geotechnical conditions.

As a result of using the present invention, the anomalous zones of the Zhezkazgan deposit, which are dangerous for collapse, have been identified by the multiplicity of work  $H/m < 10$ . These dangerous zones are presented in the form of a forecast map, according to which preliminary diagnosis of the mined rock mass is carried out.

Such diagnostics of the final state of the developed spaces allows to substantiate the possibility of excavating ore in mines, the ways and order of repayment of accumulated potentially dangerous voids, and the need to transfer surface structures.

Knowledge of such potentially dangerous zones on the caving in the zones allows us to identify the places of possible collapses at the stage of mining planning and take long-term preventive measures. With the further formation of voids in the final state of work-in-hand, the volume of voids increases with time (This effect was discovered in 1949 by Bridgman and is called the dilatation, arising from the growth of existing voids and cracks).

In the current situation, an indispensable condition for the formation of funnels on the surface is the initial destruction of the supports supporting the worked out space and further development of the process of deformation and destruction to the surface.

Practical instrumental observations of the displacement of the earth's surface and visual observations in the mine indicate the existence of a direct relationship between the increase in the parameters of the trough of displacement and the state of the void formed by the cleaning underground works. Therefore, it is important to take this link into account when performing a preliminary diagnosis of an underworked rock massif.

Analysis and generalization of the results of long-term observations of displacement of the earth's surface (>50 years) showed that the nature of its deformation is uneven, i.e. in the conditions of ore deposits, in spite of strong rocks, analogous to coal seams, there are spatio-temporal deformation anomalies in the zone of influence of work-in-hand.



## **Innovative methods of zone division into districts of the deposit surface**

The main disadvantage of the above method is that the obtained criterion  $H/m < 10$  is determined only by geometric parameters: the depth of the worked out space  $H$  and the thickness  $m$ . At the same time, it is known that the parameters of the geodynamic event mainly depend on the pressure on the work area on the side of the overlapping rocks extending from the upper boundary of the worked out space to the surface of the deposit in proportion to their weight, that is, the density distribution in this volume.

Therefore, on the parts of the array that are in different parts of it but at the same depth from the surface due to the difference in density, the pressure will be different and, consequently, the expected geodynamic events will differ substantially in all parameters. At the same time, the  $H / m$  criterion established in the method for these sites is the same. Thus, the accuracy of determining the boundaries of attenuated zones on the earth's surface depends on the degree of homogeneity of the array. With increasing homogeneity of the array, the accuracy of the method increases. In real conditions, the heterogeneity in density can be up to 30%.

In this regard, the established multiplicity criterion  $H/m < 10$  for an anisotropic mountain mass may impose unjustified restrictions on the extraction of significant amounts of minerals.

We have proposed a method for improving the criterion  $H / m$ , taking into account the anisotropy of the density of the array and allowing to increase the accuracy of control and the degree of reliability of forecasting deformation processes[2]. In the case of a discrete mass distribution (the density remains in a certain interval of altitude unchanged) for the pressure  $P$  on the region to be poisoned from the side of the overlapping rocks extending from the upper boundary of the worked out space to the surface:

$$p = k \sum_{i=1}^n h_i \rho_i g \quad (1)$$

there  $h_i$  – length of the interval with density  $\rho_i$ ,  $n$  – the number of such intervals,  $g$ - acceleration of gravity,  $k$  – coefficient of proportionality, which depends on the physical and mechanical properties of the array. The height of the worked out space  $H$  is equal

to the sum of the heights of the intervals:

$$H = \sum_{i=1}^n h_i \quad (2)$$

The average density  $\rho_{md}$  in height  $H$  is determined by the expression:

$$\rho_{md} = \frac{\sum_{i=1}^n \rho_i h_i}{H} \quad (3)$$

Taking into account (2) and (3)

$$P = k \rho_{md} H g. \quad (4)$$

Equal pressure on the boundaries of the worked-out space on the side of the overlapping rocks assumes the identity of the development of geodynamic events in these areas.

The height and average density of the overlapping rocks for the isobaric areas are related by the relation:

$$H_{cr} \rho_{md.cr} = H_1 \rho_{md.1} = H_2 \rho_{md.2} \dots = H_i \rho_{md.i}, \quad (5)$$

there  $H_{cr} \rho_{md.cr}$  – parameters of the area of the actual collapse (critical state) actually occurring at the implementation stage, the relevant criterion  $H/m < 10$ .

From (5) :

$$H_{cr} = H_1 \frac{\rho_{md.1}}{\rho_{md.cr}} = H_1 \frac{\rho_{md.2}}{\rho_{md.cr}} = \dots = H_i \frac{\rho_{md.i}}{\rho_{md.cr}} \quad (6)$$

The value  $H_{npi} = H_i \frac{\rho_{md.i}}{\rho_{md.cr}}$ , is the depth of the overlapping rocks from the boundary of the worked out space to the surface (for the critical state coincides with  $H_{cr}$ ), and along with the geometric parameters characterizes the density distribution in the array.

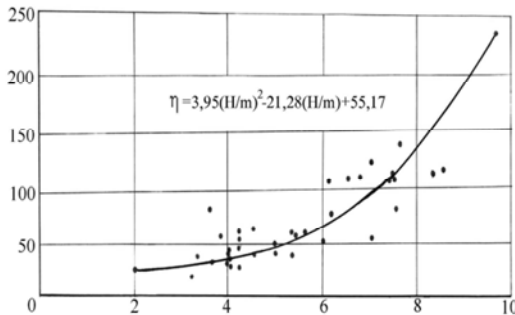
The use of the 1case of collapse in the Zhezkazgan field of the criterion  $H/m < 10$  based on a retrospective analysis, with the replacement of  $H$  by  $H_{pr}$ , makes it possible to significantly improve the accuracy of zonal zoning, especially in the boundary regions.

The correlation dependence describing the parameters of the depleted space and the displacement of the earth's surface is shown in Figure 1. This dependence can be used to approximate the maximum permissible values in those areas where there are no instrumental observations of surface deformations (profile lines of displacement).

Criterion  $H/m < 10$ , with the conditional assumption that supporting the developed space of the pillar have lost their bearing

capacity, allows the diagnostics of the final state of the mined rock mass.

Artificial extraction of supporting ends, which is possible by physical simulation of the final state of the worked out space, will allow to solve such long-term tasks of mining production as evaluating the consequences of field development on the surface with the purpose of taking preventive measures for their protection, identifying the possibility of finalizing remaining stocks, including in supporting ones, the choice of mining technology and their planning.



**Fig.1.** Graph of dependence of subsidence of the earth's surface from the multiplicity of work

The reliability of the regionalization is confirmed by the fact that all the realized cracks are in the zones identified by the specified criteria (Figure 2).

In another method proposed by us, such quantities as the height of the worked area and the overlapping rocks from the boundary of the worked out space to the surface (H), the power (m) are replaced by the vertical component ( $Z_c$ ) coordinate of the center of gravity of the column of the array extending vertically up to the surface of the deposit.

It should be noted that the position of the center of gravity is determined by the distribution of the mass in depth, and not just in depth, and therefore its change is most sensitive to processes occurring in the mountain massif.

The center of gravity characterizes the distribution of mass in the column, and its change in the process of mining operations

redistribution along the entire depth of the massif. In the process of such changes, due to internal and external factors, in conditions of exceeding the rate of energy accumulation over its distribution and dissipation in some areas, it leads to its excess, and hence to instability of the system. Based on these physical assumptions, the degree of change in the position of the center of gravity of the rock mass in the column can serve as local indicators, the formation of anomalous regions.

The proposed method is carried out as follows: on the field plan, draw a coordinate grid (x, y). For the beginning of the report along the Z axis, the lower horizon of the deposit is taken (the Z axis is directed vertically upwards). Using the coordinates and data of the set of available geological and exploratory wells over the whole area of the deposit or for its individual sections determine at each point of the surface the vertical component of the coordinate of the center of gravity of the column.



**Fig. 2.** Scheme of the forecast plan for the multiplicity of the Zhezkazgan deposit

From the beginning,  $Z_{c0}$  is determined by the center of gravity of the column in the originally untouched mountain massif, if possible, otherwise the intactness of the massif is achieved by calculation based on the known mass of extracted ore and the geometric dimensions of the mine workings. The next stage is the center of mass  $Z_c$  of the column as of the current time. Then for each point of the plan there is a difference  $\Delta Z = Z_{c0} - Z_c$  the results are taken out on the plan. On the plan, by extrapolation, connecting points with the same values of  $Z$ , construct isolines that divide the surface of the deposit into certain zones. The most dangerous are the zones with the maximum value of  $\Delta Z$ , which corresponds to the largest volumes of excavations and the depth of occurrence. The degree of danger of the zone depends on its width and the magnitude of the difference  $\Delta Z_c$  with neighboring zones (from the gradient  $\Delta Z_c$ ).

Considering the importance of monitoring the state of the day's surface of the deposit, we derive the initial coordinates of the vertical component of the center of gravity of the column on this surface. To do this, consider a vertical column, the bottom half of which positions itself the mountain massif in the initial state, and the second - virtual, represents a mirror image of the first half. All structural changes that occur in the array are displayed in the virtual half and determine the position of  $Z_c$ .

The change in position is characterized by a relative displacement of the center of gravity  $\varepsilon$  equal to:

$$\varepsilon = \frac{Z_{c0} - Z_c}{Z_{c0}} = 1 - \frac{Z_c}{Z_{c0}}, \quad (7)$$

there  $Z_{c0}$  and  $Z_c$  – vertical components of the coordinate of the center of gravity, respectively, in the initial and current state.

On the plan, using extrapolation method connecting points with the same values of  $\varepsilon$ , isolines are constructed, dividing the surface of the deposit into certain zones (by analogy with  $\Delta Z$ ).

The most dangerous are the zones with an increased value of  $\varepsilon$ , which correspond to the largest volumes of excavations and the depth of occurrence. The degree of danger of the zone depends on its area and the difference  $\varepsilon$  of adjacent zones (from the gradient  $\varepsilon$ ). The zonal surface of the deposit covered by the isolines can be characterized by an effective radius  $R_{eff}$  - the radius of a circle

whose area is equal to the area of the zone. Accordingly, the value of the gradient  $\varepsilon$  is determined by the ratio of the difference in the values of the  $\varepsilon$  neighboring zone to the difference in their effective radii:

$$\text{grad } \varepsilon = \frac{\varepsilon_i - \varepsilon_{i-1}}{\Delta R_{eff.}} \quad (8)$$

For the Annenskoye mine, which is part of the Zhezkazgan deposit, the relative displacement of the vertical component of the center of gravity of the column  $\varepsilon$  is given in Table 1.

**Table 1**

**Relative displacement of the vertical component of the center of gravity of the column  $\varepsilon$**

Zone	Safe	Dangerous	Highly dangerous
$\varepsilon, \%$	$\varepsilon < 7$	$7 < \varepsilon < 12$	$\varepsilon > 12$
The colour of zone on plane	green	yellow	red

The area of hazardous areas, determined by the value of  $\varepsilon$  due to the increase in the accuracy of the criterion characterizing the degree of danger of the zones and their boundaries is 10-15% less than the area of the corresponding zones, determined by the multiplicity of work  $H/m < 10$ , with strict compliance with the requirements for ensuring the safety of mining operations. This makes it possible to increase the extraction of minerals by increasing the area of the licensing zones.

Based on the experimental data (geological sampling materials, the results of observation of geodetic surveys, using remote sensing data of the Earth and interpretation of space images), taking into account the physico-mechanical properties of the rock mass, the correlation dependence of the maximum permissible value  $\eta$  was obtained, the causes and consequences of the occurred collisions, the correlation dependence of the maximum permissible settling value  $\eta$  on  $\varepsilon$  is obtained:

$$\eta = \alpha\varepsilon^2 + \beta\varepsilon + c \quad (9)$$

there  $\alpha, \beta, c$  - experimental constants determined for a given deposit.

Thus, the proposed forecasting method, provided that the criteria can be quickly determined, can provide a sufficiently high level of

prediction of geodynamic events. This is evidenced by a sufficiently high convergence of monitoring data with technological situations in actual conditions.

For zone districting of the ore deposit surface, a method based on measuring the acceleration of gravity  $g$ , for which data from the exploratory (initial) and geodetic (current time) of gravimetry was used, was proposed by the project executors in terms of the potential hazard to collapse.

Gravimetric surveying under real conditions is always discrete. In this connection, the problem arises of interpolating the value of the acceleration of free fall on intermediate points and estimating the accuracy of the solution, taking into account the measurement error. The determined parameters of the zone zoning are the chosen difference in the acceleration values of the free fall  $\Delta g$  between the initial (initial)  $g_0$  and the current value of  $g$ .

$$\Delta g = g_0 - g \quad (10)$$

and the relative value

$$\beta = \Delta g / g \quad (11)$$

From gravimetry it follows that  $g$  characterizes the mass distribution in a given state and  $\Delta g$  is its redistribution as a result of processes occurring inside the array during the transition from one state to another. The use of the difference value  $\Delta g$  avoids the determination of the absolute value of the acceleration of free fall, which greatly simplifies the process of zone zoning.

The proposed method is implemented as follows: on the field plan draw a coordinate grid ( $x, y$ ). Using coordinates, the values of  $g_0$  and  $g$  over the entire surface area of the deposit or its individual sections determine at each point the relative change in the acceleration due to gravity  $\beta$  (11). On the plan, by extrapolation, connecting points with the same value of  $\beta$ , construct isolines that divide the surface into certain zones. The degree of danger of the zone is determined by the value of  $\Delta g$ . The most dangerous are the zones with the maximum value of  $\Delta g$  ( $\beta$ ). By the meaning of  $\beta$ , the zones are conventionally divided into safe, dangerous and especially dangerous. The boundary values of  $\beta$  for each zone are determined by the criteria that are established for each deposit on the basis of a retrospective causal

analysis of the occurring geodynamic events, taking into account the structural features of the rock massif (geological structure, tectonic disturbance, fracturing of the applied development systems), physical and mechanical properties and strained-deformed state of the mountain massif.

### **Complex system of the geomechanical monitoring**

Taking into account that ground instrumental geodetic measurements make it possible to obtain exact values of displacements of the earth's surface only at points and along the profile, but it is not possible to build a reliable continuous map of deformations of the earth's surface on the entire area of the field from these data, since between the nodal points and profiles it will be necessary to perform the usual interpolation. Therefore, the work used a comprehensive approach to geomechanical monitoring, based on the use of both topographic and geodetic survey methods and methods of space-based radar interferometry.

Space radar interferometry is the most effective method of remote sensing, which provides high accuracy in determining the changes in altitudes and area displacements of the earth's surface due to the use of the phase component of the radio signal.

The application of methods and technologies of differential radar interferometry [4-5] makes it possible to monitor deformations of any territories (including those of significant area and extent) with high accuracy (up to the first millimeters) at a low cost of work.

The resulting displacement map, in addition to answering the question of the presence and magnitude of displacements on the earth's surface, carries information about natural and technogenic geodynamics and can be used to assess the ecological and geodynamic safety of field development, forecasting the risks of emergencies, etc.

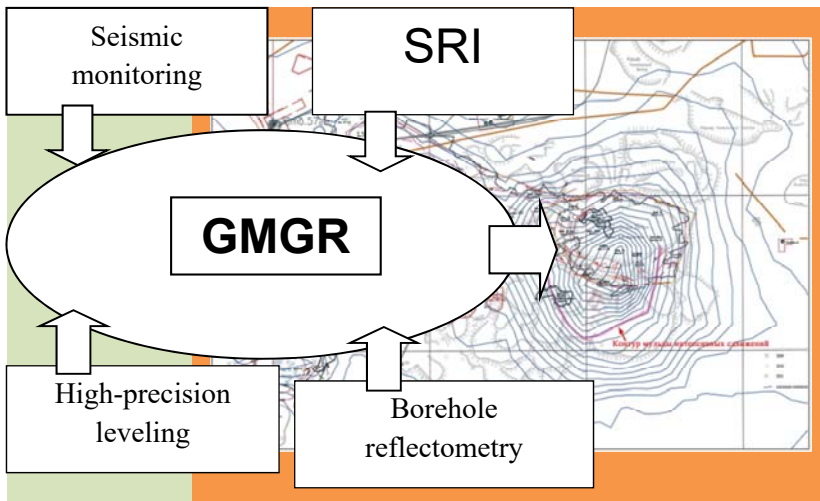
A comprehensive geoinformation model of geomechanical risks (GMGR), which performs a complex analysis of the results of space radar interferometry (SRI) and topographic and geodetic measurements, will be the basis for a comprehensive geomechanical monitoring of the development of the sink of the Anenskoe mine.



The geoinformation basis of the GMGR will be the topographic map of the industrial territory of the Anenskoe mine (scale no worse than 1: 2000), on which, in the form of thematic functional layers, the results of geomonitoring are superimposed:

- seismic monitoring;
- space radar interferometry;
- borehole reflectometry;
- high-precision leveling.

As a result of the subsequent geoinformation analysis of geomonitoring data, a resulting map of geomechanical risks will be constructed (Fig. 3).



**Fig. 3.** Enhanced Geoinformation model of geomechanical risks (GMGR)

The urgency of the development of forecasting methodology is due to the performance of mining operations in conditions of complex mining and technical conditions. On the example of the Annenskoye field, as a result of conducting mining operations for many years, extensive rock shifts, as well as subsidence and collapse of the earth's surface, have formed on the territory of the deposits. As a result of such processes, a rock formation shift formation was formed. Under the trough were significant balance reserves of ore,

which must be extracted as far as possible from the bowels, in compliance with the standards and safety requirements for mining. In the process of extraction, activation of such processes is possible, therefore, the creation of a forecasting technique in weakened zones when planning mining operations is relevant.

### **Situational forecast for underground mining in weakened zones**

The algorithm for implementing the forecasting methodology is as follows: at the initial stage, zonal zoning of the deposit surface is carried out according to the degree of weakening at the current time on the digital geoinformation model of the field (DGMD-model). Complex monitoring of the deposit surface is carried out. Based on the monitoring results, a situational map of the current situation is created. According to the data on the geoinformation model of geomechanical risks (GMGR - model), a continuous situational map of geomechanical risks is constructed, which reflects the development of deformation processes in the mountain massif. To make the forecast in the model of DGMD, GMGR, the parameters of various variants of the mining plan are loaded. At the output, predictive situation maps of each variant of the plan are issued. Practical approbation of the method was carried out in the conditions of the Annenskoye deposit. To test the efficiency of the methodology and the reliability of the forecast, a retrospective development of the situation with a depth of immersion of three years was simulated. The technique has shown its effectiveness. As a supplement to the methodology, it is planned to introduce a parameter characterizing the distribution of the BRMR for each element of the deposit surface, for the purpose of calculating the loss of balance reserves of mineral resources (BRMR), on the recommendation of the company's economists, for zone zoning.

The forecast for the development of information processes is an integral part of a set of measures to prevent dangerous situations, ensure security and improve the efficiency of mining. For forecasts, the project uses the created universal geoinformation system, an integral part of which are digital models. An example of using the capabilities of GIS technologies is the following algorithm for

calculating the expected surface movements and deformations resulting from mining operations.

In the process of mining and mining operations, the stressed state of rocks changes. The movement of rocks at a certain ratio of the dimensions of the worked out space and depth reaches the earth's surface, and it also undergoes deformation. The application of methods of technology of differential radar interferometry makes it possible to monitor the deformation of the surface displacement of deposits with high accuracy. The resulting offset map shows the value at the current time. After the subsequent geoinformation analysis of the monitoring data, a "duty card" of geomechanical risks is built. The process of development of shifts and deformations is modeled on the extended GMGR model. Calculation of shifts and deformations of the earth's surface for their prediction is carried out by modeling on the DGMD-model. The results of simulations on these models for verification are continuously compared. The forecast for DGMD is as follows. For the entire area of the deposit, using existing sections based on geological and exploratory wells, as well as geomechanical documentation, all overlaps are applied to the plan, including arrays that are not subject to development. On the plan, draw a grid of X, Y (the coordinate system and the scale used are calibrated from the GIS). The depth and thickness of each spent deposit are measured along the vertical axis. Then the values of the depths and powers are summarized, the results of which are carried to the plan. Using the set of available geological and exploratory wells over the entire area of the deposit or its individual sections, the values of total depths  $H_{pr}$  and powers  $m$  are determined for each surface point. The data is entered into the database management system (DBMS) of the model. Based on the results of the total power on the plan by extrapolation, the computer constructs isolines at a given interval. The revealed volumetric anomalies of the total power are superimposed on the mining plan of the deposit.

The most dangerous are the zones with the maximum values of the total production capacity of the deposit. Then, using the obtained values of the total power and depth, determine the ratio of the total depth  $H_{pr}$  to the total thickness  $m$ . Based on the results of the extrapolation method, the contours of the increment  $H_{pr} / m$  are constructed. The  $H_{pr}/m < 10$  criterion for the prediction of caving is

the value according to the imposition of the obtained plans with isolines according to the established criterion for the mining plan, anomalous zones potentially dangerous to collapse occur.

In this case, for hazardous areas are taken areas where the isolines of the values coincide according to the criteria  $m$  and  $H_{pr}/m$ . In the modeling process, various options for the direction of mining operations are lost, which leads to a virtual change in the mining and geomechanical situation,  $H_{pr}$ ,  $m$ . The received parameters are entered in the DBMS model. Based on the analysis of modeling results, a situational forecast is compiled.

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# **INNOVATIVE APPROACHES IN OIL AND GAS WELLS ENVIRONMENTAL SAFETY INVESTIGATION**

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## **Abstract**

There has been developed the life cycle of an oil well in the oil industry system. We used the systematic approach in order to investigate the ecological safety of oil wells, based on the analysis of each stage of life cycle of every well. The primary stages of well's life cycle have been defined, namely: technical and technological projects design; drilling rig installation; well drilling; trial; dismantlement of the rig; equipment installation; conversion; field operation; close-out; taking the rig out of service; maintenance service and repair works; complications and emergencies.

The ecological risks emergence has been theoretically analyzed at every stage of a well's life cycle. The risk factors that may occur on every stage have been defined: human, technological, qualitative and wear-out. The wells that had been taken out of service were examined. The uncertain performance map of an oil well has been designed where the unsealing spots in casing are marked. The field investigations of wells taken out of service have been performed and it has been defined that about 30% of the objects' equipment under investigation is unsatisfactory.

On the basis of conducted theoretical and field investigations a set of recommendations in order to increase the ecological safety of oil wells taken out of service has been formulated. The primary aspects of attention aimed at the ecological safety increase of the industry have been outlined. Namely, the accent is made on the necessity of wells monitoring after they have been taken out of service. The equipment that needs perfection at the stage of an oil rig installation has been defined and the technical solutions

of its modernizing have been given.

The given approach makes it possible to trace the connections of the combination of factors and the consequences of the impact on the environment as the whole set of elements. The implementation result of the given approach is the identification of real environmentally hazardous processes and preventing of their advance.

## **Introduction**

Oil and gas industry complex is one of the objects of high ecological hazard. It uses ignitable, highly flammable, dangerously explosive and toxic substances. According to 2008/50/EC Directive of the European parliament and council [1] it is necessary to decrease pollution to the levels minimizing the hazardous impact on human health and environment in general. One of the strategic directions is preventing hazardous substances emission from particular sources. At the same time the most effective measures of defining and implementation of air pollution reduction are taken at the local level.

At the time being all the regions of the world possess a great number of oil wells. These objects are quite dangerous for the environment as in the course of regular operating processes and also at states of emergency. A great amount of scientific papers in the sphere of ecological safety are about oil and gas industry complex problems. However, the analysis of the whole state found the problem of defining separate problematic fields.

The decrease of environmental pollution at oil and gas extraction works demands thorough investigation of every stage from the ecological safety viewpoint. It will make it possible to establish faults in operation of the equipment, technological processes and management decisions that need to be disposed of. An offer has been made to use systematic investigation approach that will be based on the analysis of each life cycle stage of the well. The above-mentioned approach will make it possible to trace the connections of the combination of factors and the results of the environmental impact as the whole set of elements. The result of implementation of this approach will be the definition of real ecologically hazardous processes and their delimitation.

The aim of the given work is implementation of innovative approach in environmental safety investigation of oil and gas wells in order to define environmentally hazardous units of equipment and technological processes.

In order to achieve the aim it is necessary to solve the following problems:

- to form conditions for systematic approach of achievement the goal set;
- to define the basic stages of life cycle of oil and gas well;
- to conduct theoretical analysis of the risks of emergence of ecologically hazardous effects at every stage of life cycle of oil and gas well;
- to perform practical investigation of particular stage of life cycle of the well;
- to offer technical, technological and management solutions in order to prevent the emergence of ecologically hazardous situations.

### **1. The definition of basic stages of life cycle of oil and gas extraction wells**

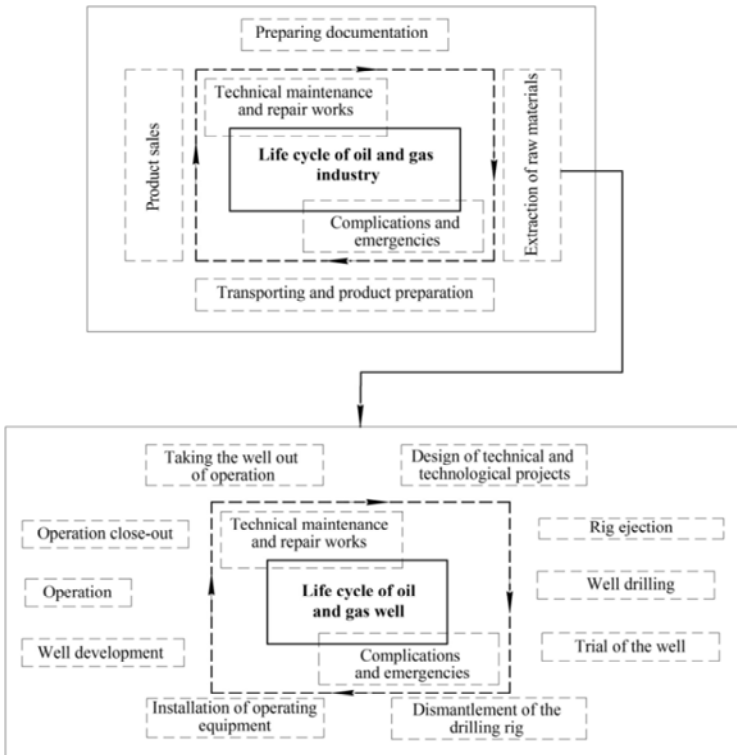
The solution of the tasks set is based on the complex implementation of environmental management system, the basis of which is described in international standards ISO series 14000 [2]. The international standards ISO series 9000 hold the ideas of systematic management approach, decision-making on the basis of facts and constant performance upgrade of the organization in general [3].

The ecological safety should serve as the basis of the systematic approach. The factual data about environmental pollution by the oil and gas extraction industry objects prove the necessity to aim the companies' strategy at ecological safe technologies during the whole life cycle.

Thorough investigation of oil and gas industry impact on the environment is based on stage by stage investigation of its life cycle. The life cycle of oil and gas wells as a whole is to be investigated by the principles of quality system. The basis of its system is "the quality loop" where one of the main criteria should be "ecological safety".

In order to investigate the cycle the regulations of ISO 14000 standards are applied, enabling to understand the notion of ecological management of the production process and the organization as a whole [2]. The standards mentioned demonstrate the evaluation methods and their possible interpretation in the management system of the life cycle in order to achieve better results of standard production and to improve the perfection process of every stage constantly.

In order to enable the systematic approach in solving the goal set it is important to create proper conditions. Taking into account the above-given materials the chart of life cycle of oil and gas well has been created where the well stands as a separate element of overall life cycle of oil and gas industry (fig. 1).



**Fig. 1.** The chart of life cycle of oil and gas well in the system of oil and gas extraction industry



The indicators of life cycle quality of the well in terms of ecological safety depend on the quality of all successive stages and their implementation means. Negative features of the quality of ecological safety can be altered by means of impact at every stage.

In the life cycle of the well as it is shown at fig.1 the following basic stages can be outlined: the design of technical and technological projects, drilling rig installation, the trial of the well, drilling rig dismantlement, completion installation, the exploration of the well, the operation of the well, operation close-out, taking the well out of operation.

## **2. The analysis of risks of ecologically hazardous effects at every stage of life cycle of oil and gas wells**

At the “*Design of technical and technological projects*” stage it is necessary to take into account the basics of ecologically safe resources exploration [4]: to obtain the business purpose on condition of minimum impact at environment; prevention and forecast of ecological hazard sources emergence; responsibility for breaking technological regulations and limits; minimizing environmental pollution during operation, connected with drilling, conversion and the performance of the well. At this stage the direction of the whole life cycle of the well is stated where the important role should be given to ecological safety. At this stage the calculation of risks at the stages of design still goes on in compliance with ISO 14040-14043, ISO/TS 14048-14049 standards. While calculating the risks they are divided into technological, labour protection and environmental protection. This division often neglects their real importance that is why it is important to consider the issue of risks as a complex [5]. It is important to pay special attention at the characteristics of the region as to the probability of natural hazards (floods, earthquakes, etc.), environmental stability of the region where the well is made, which is defined by the degree of reproductivity and the level or soil cultivation, the distribution populated locality within the radius of the well’s impact and the natural value of the territory. The well’s radius of impact should be calculated as for the defined process and for the emergency situation which will make it possible to estimate the probable environmental

risks.

*"Drilling rig installment"* stage is accompanied by intensive intrusion into the environment. At this stage the driveways to the nearest well are created the topsoil is taken out and the site is being got ready for the rig, the industrial and residential objects of the work area are erected. The distribution of industrial and residential objects during the rig installment should be conducted taking into account prevailing wind rose. The quality of installment works affects the quality of the process of drilling rig installment in general.

At the *"Drilling the well"* stage the following environmental risk factors are present, like the positioning of the well, management level and technological processes and substances used during oil and gas wells installment. Technological processes of this stage are accompanied by the loss of energy, high pressures and the presence by a big amount of chemical substances as of human origin (drilling mud, drilling waste water) and also of natural one (oil, hydrogen disulfide). While using power drives with internal combustion engines a huge amount of exit gas is produced.

*"Trial of the well"* stage is characterized by considerable emissions of oil into mud pits and gas which is being burned at the torch. This process has negative influence as on the ground layer and also on other environmental components.

*"The dismantlement of the drilling rig"* stage consists of the dismantlement and moving-out the drilling equipment and salvaging drilling wastes (drilling fluid, bore mud, drilling waste waters, chemical agents etc.) In a number of cases when the remediation is improperly conducted in places of presence of sludge pits and waste waters animals may die and territories like this may be impracticable for making use.

*"The equipment installation"* stage can be performed with not fully dismantled drilling rig or with the help of tower elevators (special equipment). Wells can be equipped with the following equipment: christmas tree, gas lift equipment, aligned pumps etc. In order to avoid the penetration of hazardous substances into the atmosphere the high air-tightness of drilling wellhead equipment is demanded.

*"The conversion of the well"* stage is ecologically hazardous at

the expense of emission into the atmosphere substantial amount of products from the rig. The emissions are performed by the take-offs at a certain distance from the well outside. The territory of the given surface is specially equipped by jack shafts. When the gas is present it is burned. If the mixture contains liquid or oil products the territory should have the earthen container for their collection. At the given stage the whole production from the well goes onto the open surface where it intensively spreads into environment.

*“The production activity”* stage is the longest in time compared to other production activities. When the oil wells are used the level of man-made burden is defined by the production activity of deposits and huge well stock, big length of internal industrial pipelines, tens of different technological equipment, etc.

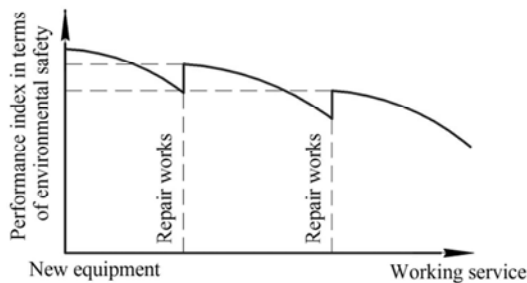
At *“The closure of production activity”* stage the environmental risk increases when inappropriate, outdated technological equipment is used, when physical-chemical properties of the fluids used lead to overload and reduce the efficiency of equipment productivity.

*“Taking the well out of production ”*stage. Currently a problem exists as to the wells that are being taken out of production. From its early days the oil industry has drilled a great deal of wells and up to present time the most of them are not identified and is left unattended which creates certain danger for the environment. Damaged well equipment resulting in the damage of sealing and leads to uncontrolled pollution of formation waters. After the investigations conducted by the scholars from Princeton University the abandoned oil and gas wells may serve as the source of considerable emissions of greenhouse gases, firstly methane into the atmosphere [6]. Only in the USA there are about 3 million of abandoned wells. Seismic activity may set these wells into activity and provoke the emergence of emergency emissions of oil and gas. As a result vast territories turn into zones of ecological disorder. The sealing of the well after its cold-stacking expires after 20-30 years. In the course of time the concrete drilling floors may ruin, the wellhead equipment starts corroding and as well as the drill string itself, resulting in the unsealing of the well. There exist investigation data and real facts of penetrating of hydrocarbons from underground layers which results in uncontrolled fluid or gas leaks into the environment when the well's construction is unsealed. The media

hold information about the oil leaks and damaging of close-out constructions at abandoned wells in different countries of the world. That is why the problem of wells' handling after their closing-out is quite acute for the time being.

"*Technical maintenance and repair works*" is conducted at all stages of life cycle of the well from the moment of installment the rig up to the closing-out of the production activity. Technical maintenance and repair works are accompanied by atmospheric pollution due to technological processes of these stages and using different aggressive substances for their conduction. For example during repair works at oil and gas wells in order to increase the productivity of strata different acids and different dangerous chemical substances.

According to the chart given at fig. 2 every next repair of the equipment lessens their quality coefficients in terms of environmental safety.



**Fig. 2.** The change of coefficients in terms of environmental safety of oil and gas extraction equipment starting from the moment of production activity beginning

"*Complications and emergency situations*" may occur at different stages of life cycle of the well. The main reasons of their emergence are technological processes violation, faults in equipment construction and also unpredictable situations during drilling. These processes are the source of pollution of all spheres of environment by hazardous substances of different classes of hazard, namely with the help of atmospheric air the pollutants are spread onto vast territories.

The operation of oil and gas complex very often violates the

equilibrium of natural geological systems – activates earthquake activity which may lead to calamities. Extracting hydrocarbon raw materials from the earth depths may provoke local earthquakes of considerable destruction power. There are some well-known historic facts – huge earthquakes close to oil and gas deposits in Rocky mountains (USA) and gas deposits in Uzbekistan and also local of 4 points of local intensity that took place in march 1986 at Khrestyshchen gas-condensate deposit in Ukraine [7]. When things like this take place there may be uncontrolled hydrocarbon flows and mud springs and oil and gas fountains. Whenever there is lack of attention to preventing and forecast the emergence of extreme conditions then the probability of accidents and complications increases.

The Table 1 contains risk factors that need to be taken into account and the conditions for their prevention at all the stages of an oil and gas well's life cycle should be created.

At all the stages of a well's life cycle there exist different factors that influence the rate of ecological hazard. A set of factors reflecting technical conditions of the equipment is worth mentioning (high quality, wearing out).

The equipment used at all the stages of life cycle of an oil and gas well is being under aggressive influence. The study of technical indices of the equipment is very important for evaluation of its reliability, especially if the terms of operation are higher than the standards. In cases like this the intensity of hardware faults increases and so does the level of environmental risk. The probability of failure-free operation of equipment is defined by the formula [8]:

$$P(t) = \Phi \left[ \frac{T_0 - t}{\sigma} \right], \quad (1)$$

where  $P(t)$  – is failure-free performance of the equipment under study;  $T_0$  – average time between failures;  $t$  – working life of the equipment under study;  $\Phi$  – Laplace's function;  $\sigma$  – mean square deviation of working life.

On the basis of probability of failure-free performance the following parameters are defined like the probability of failure  $Q(t)$  and the intensity of failure  $\lambda(t)$ .

Table 1

Risk factors at stages of life cycle of an oil and gas well

№	Life cycle stages of oil and gas well	Risk factors					
		Human (wrong decision-making)	Technological		Qualitative and (or) wearing-out (long term of operation)		
			Presence of aggressive substances	High pressures	Unsealing of wellhead equipment	Unsealing of drilling equipment	Faults in rockfall equipment
1	Design of technical and technological projects	+	+	+	+	+	+
2	Drilling rig installment	+					
3	Drilling of the well	+	+	+	+		+
4	The trial of the well	+	+	+	+	+	+
5	The dismantlement of the drilling rig	+	+				
6	Operating equipment installment	+					
7	Well development	+	+	+	+		
8	Operation	+	+	+	+	+	
9	Operation close-out	+	+		+	+	
10	Taking the well out of operation	+	+		+	+	
11	Technical maintenance and repair works	+	+	+			+
12	Complications and emergencies	+			+	+	+

The probability of failure of the unit under study – is the probability of the fact that under certain operation conditions during time  $t$  none of the faults appears in the given equipment.

$$P(t) = 1 - Q(t), \tag{2}$$

The intensity of faults – is the probability of reservoir faults after a certain period of time on condition that none of the faults have occurred before this moment. It is defined by the formula:

$$\lambda(t) = \frac{\Delta n(t)}{N_{u.o.} \Delta t}, \tag{3}$$

where  $\Delta n(t)$  – the number of objects that have failed during a certain period of time;  $N_{u.o.}$  – number of objects that are useful in the

interval of time  $\Delta t$ .

Most of the oil and gas extraction equipment especially at late operation stages exceeds the terms of their standard operation time that is why the risks of environmentally hazardous effects increase.

### **3. Environmental safety investigation of the wells that have been taken out of operation**

The investigation of the impact the wells that have been taken out of operation have on the environment is a very important stage for evaluation cause-and-effect relationship and defining the most environmentally hazardous situations. The closing stage of life cycle of wells is characterized by high risks of uncontrolled processes that may harm the environment. Environmental threats from oil and gas leaks during field development are theoretically less great than possible leaks after closing-up the operation. The Table 2 contains risk factors of uncontrolled fluid leaks.

**Table 2**

**Risk factors of uncontrolled emissions and fluid leaks**

<b>Risk factors</b>	
<b>Technical</b>	<b>Natural</b>
Failure of equipment reliability	Seismic activity rise
Cement bridge demolition	Flooding the wells during floods
Unsealing of wellhead equipment	Corrosion active surface waters
Unsealing of casing strings as well as the drillstring	Hydrocarbon penetration from deep strata

Besides during well's operation one can prevent pollutions and eliminate them by well-known nature protection measures and the wells that have been taken out of operation in most cases have no control.

The above-mentioned "favourable" conditions for defining leaks are accompanied by fountains, namely by open leaks of oil and gas onto the surface in the well zone [9].

According to the information when the well has finished its operation the hydrocarbon leaks can be visually noticeable. The approximate working life of casing strings comprises from 25 to 30 years. After this term expires the casing due to corrosion and wearing

out should be renovated or destructed (plugged out). According to the data about the speed of corrosion the expected working life terms are calculated starting from the investigation moment up to terminal corrosion wear out (in years) [10]

$$\Delta T = \frac{\delta_{val} - \delta_{a.c.}}{v_c}, \quad (4)$$

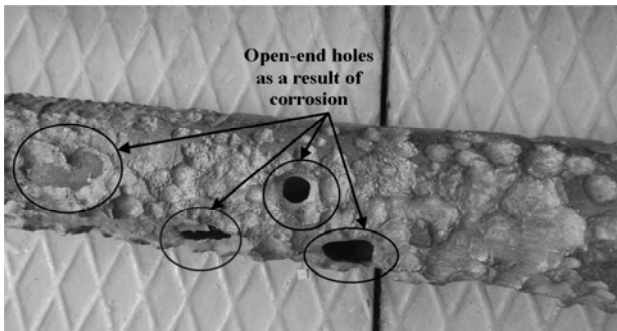
where  $\Delta T$  – is terminal corrosion wear out;  $\delta_{val}$ . – valid corrosion wear out;  $\delta_{a.c.}$  – actual (measured) corrosion wear out;  $v_c$  – average corrosion speed.

Knowing terminal corrosion wear out it is possible to calculate forecasted working life of the equipment from the moment of the beginning of operation up to the moment of terminal corrosion wear out.

$$T_F = T_{a.w.} + \Delta T, \quad (5)$$

where  $T_F$  - forecasted working life of the equipment (in years);  $T_{a.w.}$  – actual working life of the equipment for the moment of investigation.

Figure 3 shows pumping-compressor station pipes (PCP) that have been more than ten years in the well that contained corrosive fluids. These pipes can be taken out of the well and replaced in contrast to conductor, flow string and or protection string which are cemented in the well. In abandoned, taken out of operation the casing strings look like it is shown in Fig. 3.



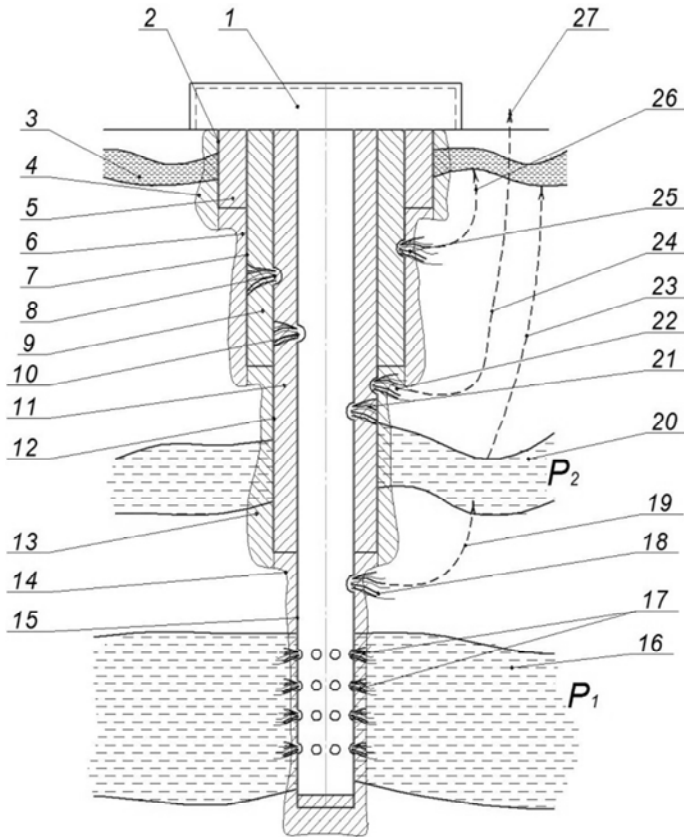
**Fig. 3.** Pumping-compressor strings destroyed by corrosion

Repair works consist of second cementing of the well but the casing string continues destroying. Because of this the earths depths



may be latently polluted and be driven to water-bearing stratum and under favourable conditions (permeable formations) penetrate into the atmosphere in form of gas shows near the wells – fountains. That is why these measures are considered to be environmentally hazardous.

Fig. 4 has the chart of the well that has been taken out of operation and probable ways of hydrocarbon penetration in different environments from different unsealed spots of casing strings.



**Fig. 4.** The chart of operating oil and gas well with possible variations of casing strings unsealing

1. Concrete plinth set when the well is being taken out of operation instead of drilling wellhead equipment.
2. Conductor – casing string for upper water-bearing layers isolation.
3. Water-bearing stratum.
4. Cemented conductor after its lowering.
5. Cemented section of the first technical casing string in the conductor.
6. Cemented section of the first technical casing string in the open hole.
7. First technical casing string.
8. The unsealing of the second technical casing string at the section cased by the first technical string.
9. Cemented section of the second technical casing string in the well cased by the second technical string,
10. The unsealing of the operating string at the section cased by the second technical string.
11. Cemented section of the operating string in the well cased by the second technical string.
12. Second technical casing string.
13. Cemented section of the second technical operation string in the open hole.
14. Cemented section of operating string in the open hole.
15. Operating string.
16. Pay oil and gas horizon with high stratum pressure  $P_1$ .
17. The punching of the operating string in the pay horizon zone
16. 18. The unsealing of operating string in the open hole.
19. Possible fluid penetration with high pressure into the stratum
- 20 with low pressure  $P_2$ .
20. Productive stratum with low pressure  $P_2$ .
21. The unsealing at the lower section of operating string cased by the second technical string.
22. The unsealing of the second technical string in the open hole of the well.
23. The relocation of the fluid from stratum 20 with low pressure  $P_2$  after it is penetrated by the stream 19 with high pressure in surface water strata 3.

24. The relocation of the fluid to the water stratum 3 and to the surface and to the atmosphere.

25. The unsealing of the first technical string.

26. The relocation of the fluid to the water-bearing horizon 3 and to the surface waters.

27. Uncontrolled discharge of the fluid to the surface (water bodies, atmosphere).

When the well is being conserved the marks are used for its identification. In some cases during next agricultural exploration of lands the wells were killed at 2 m deep. Recent wells that have been taken out of operation can be identified by informative marks but the old wells are difficult to find. One of the methods of their identification is georadar territory investigation aimed at spotting subsurface structural-material inhomogeneity of different origin caused by uneven moistening of sedimentation, different composition of rocks and texture of rocks (porosity, interstratification), inhomogeneity of deposits or materials, rock fracturing and deformity of the environment by inclusion of foreign objects [11].

***Field investigations of the wells taken out of operation.*** Field investigations were conducted on the territory of the deposits near the village of Solotvyno, Bohorodchany district, Ivano-Frankivsk region. 9 oil and gas wells that had been taken out of operation were investigated. The basic evaluation criteria of their probable environmental impact were: the location of the wells regarding the inhabited regions, rivers and other terrain features, the outlook of the equipment either oil or gas, the external manifestation of hydrocarbon leaks, the presence of data carriers.

During the investigation of the wells the methods of visual evaluation and the help of tools had been used (Table 3). In order to identify gas release of hydrocarbon the gas-analyzer “DOZOR-S-M” had been used. The gas-analyzer “DOZOR-S-M” is used to measure the concentration of components of gas air environment (fire gases  $C_nH_m$  and fumes of  $O_2$ ,  $SO_2$ ,  $Cl_2$ ,  $CO$ ,  $CO_2$ ): the price of one unit of the smallest digit (0.1% LFL (0.01% vol.) – fire gases and fumes; 0.1 mg/m<sup>3</sup> – sulfur dioxide; 0.1 mg/m<sup>3</sup> – chlorine; 0.1%vol. – carbon dioxide, oxygen); dimensional specifications 200x200x81mm; weight – 2.0 kg.

**Table 3**

**Methods of evaluation and latching data during wells investigation**

<b>№</b>	<b>Index</b>	<b>Evaluation and data noting method</b>
1	The positioning of the well	Visual, (GPS)-navigator, camera
2	Terrain peculiarities	Visual, cartographic, camera
3	Physical condition of the equipment	Visual, camera
4	Outer manifestations of hydrocarbons	Visual, gas-analyzer, camera
5	Gas condensate or oil	Visual, by the words of the locals, camera
6	Presence of data carriers	Visual, camera

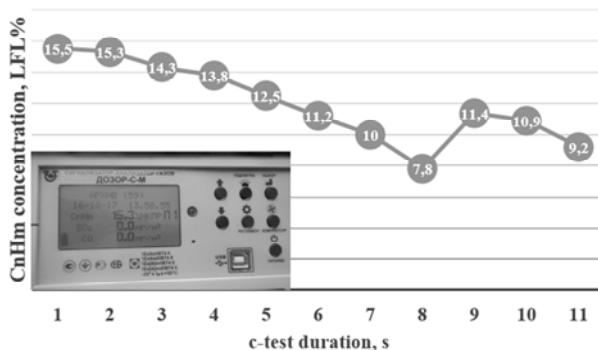
During the investigation the irrational, environmentally hazardous positioning of the objects had been noted. The attention had been drawn to the well near the river, which had been flooded several times as it was said by the local people. This well has Christmas trees dated by the year of 1965 which means that the well is significantly old. Among the objects under investigation some enclosed wells were found having concrete piers (fig. 5a) and unsealed ones (fig. 5b).

While investigating the condition of the wells the discharge of gaseous substances had been measured. At the Monastyrchany №8 well the concentration of methane had been noted (fig. 6). The measurements were conducted with the help of gas-analyzer DOZOR-S-M.



a – suspended well with mounted concrete pier;  
 b – unsealed well that has been taken out of operation

**Fig. 5.** Well site construction of the wells that have been taken out of operation



**Fig. 4.** The dynamics of concentration changes of CnHm at the Monastyrchany №8 well

At the end of investigation of the well it was established that:

- the surface was polluted by oil spills (Rakovets №1 and the well with the name unknown);
- the discharge of methane into the atmosphere (Monastyrchany №8);
- close presence of inhabited territories and the river (fig. 3b);
- unsatisfactory or poor (fig. 3b) data carriers;
- the equipment of the most objects under investigation has satisfactory physical configuration or relatively satisfactory, some units of the equipment are dated by 1965;
- 30% of the equipment with unsatisfactory condition has been separated after the investigation.

#### **4. Environmentally safe technologies and technical means in the life cycle process of oil and gas well**

The investigations conducted make it possible to define the acutest problematic issues at different stages of life cycle of oil and gas wells. The investigations of the wells that have been taken out of production show how important it is to maintain their constant control. Due to the decrease of oil extraction index some oil companies got interested in the redevelopment of the wells that use different reagents but we are not sure of whether these technologies are environmentally safe [12]. One of the methods of secondary use

of oil and gas wells is petrothermal power based on the thermal gradient rise with the depth. With this aim the wells should be quite deep – the earth temperature rises approximately by 25-50 degrees per each kilometer. The [13] states that heat-flow rate – is the amount of heat that is being taken out onto the surface per one unit of time and per one unit of per one area unit. It is measured in  $\text{mWt/m}^2$  and is defined by Fourier's law as the result of thermal gradient multiplication in a certain depth interval by heat-conducting of rocks of this interval. On the territory of Ukraine the heat-flow rate changes from 25– 30  $\text{mWt/m}^2$  up to 100 – 110  $\text{mWt/m}^2$ . The temperatures at 1 km deep change from 20 to 70°C and at 3 km – from 40 to 135°C. At the same time the depth of exhausted oil wells may constitute several kilometers and most of them have preserved pipeline infrastructure. An assessment of the heat-flow rate through the drillstring walls at the Karadag #189 has been conducted [13]. At 3765m stationary temperature of drilling fluid was 85 °C and the temperature of the drilling fluid after flushing was 70 °C. The total heat-flow size from the well-head up to the total amount of the drilling fluid in the drillstring at the amount of 40 m<sup>3</sup> (60000 kg) comprises 3.4 MWt.

The investigators offer to pump water into the earth by the “pipe in pipe” system. The water goes down be the broad pipe where it heats up to 130 °C. Then through the central pipe of fewer diameters the fluid flows to the surface and sets the steam turbine to motion or for other heating purposes [14].

Currently there are several petrothermal stations built in seismically quiet regions with absent temperature abnormalities in geological environment. The example of that kind are the petrothermal stations in Germany and France that have been functioning for 20 years [15] The main recommendations for reducing hazardous environmental effects of the wells that have been taken out of production are:

- constant control of all the wells that have been taken out of production;
- the development of all directions of rational use of the well data for constant control maintenance;
- the application of gas wells with low pressure that have been taken out of operation for local needs;

- the development of petrothermal engineering as environmentally clean and perspective direction of alternative energy;
- creating the database of the wells that have been taken out of production;
- constant technical and environmental control of the wells that have been taken out of operation should be placed upon the organizations that are going to use them.

The conducted investigations of the processes of oil and gas wells construction made it possible to identify the groups of equipment that need perfection. The main penetration source of hazardous substances into the atmosphere of this stage is pumping-circulating system of the drilling rig containing drilling fluid saturated by aggressive components [16, 17, 18]. In order to increase environmental safety of the given stage a set of technical solutions has been offered, namely: modernized vibrating sieve for rough drilling fluid purification, complex hydroclone for fine drilling fluid purification and the device for cleaning the outer surface of the pipes that are being taken out of the well.

The use of the above-mentioned equipment will make it possible to reduce the penetration level of hazardous substances into the environment, to prevent hazardous impact of aggressive substances on staff health and to increase fire safety of the drilling rig. The important part of environmental risk reduction is played by the high level of work process management, high qualification and responsibility of the staff, the use of modern highly effective equipment, compliance with strict technological process.

## **Conclusions**

The chart of well's life cycle in oil and gas industry has been developed in order to simplify the environmental safety investigations of oil and gas wells. In this chart the primary stages of a well's life cycle are the following: the design of technical and technological processes; drilling rig-up; drilling the well; trial; rigging-down; production activity; project close-out; taking the well out of operation; technical maintenance and repair works, complications and emergencies.

The risks of environmentally hazardous situations at every stage of life cycle of the well have been theoretically analyzed. The risk factors at every stage have been defined: human, the presence of aggressive environment, equipment under pressure, unsealing of wellhead and drilling equipment and faults in rockfall equipment.

The wells that have been taken out of operation were studied. The conventional chart of operating oil and gas well with the spots of unsealing in drilling strings has been created. Field investigations of the wells taken out of production have been performed and it was found that about 30% of the objects under study have the equipment in poor condition.

On the basis of theoretical and filed investigations a set of recommendations has been offered in order to increase environmental safety of oil and gas wells that have been taken out of operation. Primary aspects that deserve attention have been drawn in order to increase environmental safety of the industry. Besides it is very important to control the wells after they have been taken out of production. The equipment that needs modernizing at the stage of drilling rig installment was defined and some technical ways of its modernizing have been offered.

Further scientific investigations should be directed at perfection of every stage of life cycle of oil and gas well from environmental viewpoint. The given method of investigation is practical to use in the system of oil and gas extraction industry for description every component of the system.

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## **DETAILS LOADED SURFACES PLASMA STRENGTHENING**

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### **Abstract**

In the industry, large-scale and large-body gear transmissions are widespread. For example, ball mills, intended for grinding various ores, coal and other raw materials, are equipped with open gears. They also, like all open transmissions, work under shock-cyclic contact loads. Characteristic reasons for lowering the bearing capacity are determined not only by the hardening of the tooth in its base but also by fatigue cracking of the contact surface and wear of the profile of the tooth. The key issues in the design of gear transmissions are the selection of gear material and the method of strengthening it.

The work is devoted to the increase of wear resistance of large-modular gears due to the application of the innovative method of surface plasma-arc reinforcement working surfaces of teeth. The subject of the study is the processes of superficial plasma-arc reinforcement (SPAR) of large-modular gears. In the work the complex method of tests was used, including analytical analysis, scientific generalization of preliminary and experimental research. The basic theoretical results are obtained using approaches based on classical calculations of machine parts, the theory of mechanisms and machines, tribology and tribotechnics, the theory of wear and reliability of large-modular gears. The purpose of the research is to increase the wear resistance of large-modular gears through surface plasma-arc reinforcement.

The analysis of types of wear and damage of teeth of open pairs of ore-grinding mills has been carried out. Comparison of experimental wear rates with the estimated values characterizing the geometry and kinematics of the grip received a parameter that took into account the abrasive influence of the medium and the physical and mechanical properties of the materials of the teeth for these experimental conditions.

A critical analysis of approaches to solving the problem and increasing the reliability and durability of open, large-modular tooth gearings has been carried out. The influence of the physical and mechanical properties of the

gear wheels' material on the intensity of wear in the period of tire work and the established uniform wear of the contact surfaces of the teeth has been determined, as well as the substantiated possibilities of their productive surface plasma-arc reinforcement.

The method of determination and estimation of the residual resource of large-modular gear transmissions after SPAR is developed and the influence of technological parameters of this process on wear resistance and mechanisms of deterioration of the surface layer have been investigated. The process of SPAR has been experimentally investigated, an estimation of the obtained physical and mechanical properties and hardness of the processed material during plasma heating has been made. The increase of durability of the work of large-modular gears passing by the SPAR has been experimentally investigated and the calculations on durability have been performed for the wear parameter, and it has been established that the application of this heat treatment method increases the durability 2.28 times.

A complex of specialized equipment was developed for the implementation of the technological scheme of plasma-arc surface reinforcement of the large-modular gears' working surfaces without melting, using the original design of the magnetic-deflection system for the creation of an external magnetic field.

## **Introduction**

The problem of extending the operational life of the components is very relevant in the economic, environmental and resource-saving aspects, since their primary production and utilization are accompanied by consumption of raw materials and energy resources, as well as industrial pollution of the environment. In modern mining and concentrating production, redroot mills are widely used for grinding mountain mass. The large gear gears are one of the most loaded and responsible elements of the mill, which determines the reliability and durability of the equipment as a whole. Repair and replacement of such gear gears of mining machines requires the decommissioning of the main production equipment and the stop of the technological line leading to significant losses. The analysis of damaged gears shows that their premature failure is due mainly to the destruction processes in the surface layers of the teeth.

A promising direction of solution of this problem is hardening thermal treatment of loaded surfaces of parts with a concentrated energy flow [1]. Generated by high-speed heating and cooling of the

quench-type structure, they have high hardness, wear resistance, and fracture resistance.

The wide industrial application of most known methods of hardening treatment with a concentrated energy stream (laser, electron beam, cathode-ion, etc.) is constrained by the high cost and complexity of the equipment, insufficient reliability and performance, the need for vacuum, special premises with special requirements, the need for qualified service, high operating costs, etc. In these conditions, in order to extend the operational life of wear parts, a method of surface heat treatment by a plasma arc is considered rational in terms of universality, availability, environmental friendliness and economic efficiency [1,2,3]. Without changing the surface roughness parameters, such heat treatment is easy to integrate into the technological process of preparation and repair of parts, being a finishing operation, low-cost, sufficiently productive and allows to effectively increase their operational durability.

However, the properties of hardened layers after surface plasma arc quenching are insufficiently investigated, nor rational ways of controlling the specific power in the heating spot of the plasma arc have been proposed, as a consequence, it is not possible to use the large electric power necessary to increase the efficiency of heating. There are no scientifically substantiated principles for choosing modes and technologies for strengthening large geared gears, taking into account their load conditions. Therefore, increasing wear resistance of open gear gears of ore-mills, by surface plasma-arc quenching, has important practical value and is an actual scientific task.

The purpose of this work is to increase the wear resistance of large-modular gears by surface plasmo-arc hardening.

Formulated goal of work has determined the need to solve the following tasks:

- investigation and determination of the causes and nature of fractures of gearing gears of ore-grinding mills and substantiation of the possibility of increasing wear resistance of large gears by plasma-hard hardening of tooth working surfaces;
- development of a technique for studying the mechanisms of structural transformations in steels for surface plasma arc treatment

in order to select the optimal modes for hardening gears, to obtain the required structure, phase composition and hardness of the surface layer;

- investigation of the influence of technological parameters of the plasma quenching process on wear resistance and wear mechanisms of the surface layer and development of a technique for determining and estimating the life of large-size gears after plasma hardening;

- optimization of technological parameters of the surface plasma-arc hardening by mathematical modeling, with the purpose of revealing their influence on the relative service life of gears;

- experimental studies of the process of surface plasma-arc hardening, development and implementation of the thermophysical study methods;

- an experimental study of the gears durability, which have been quenched and calculations for durability in terms of wear parameters.

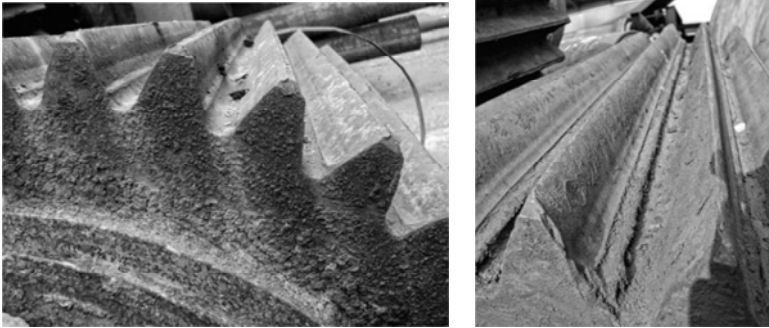
### **1. Problems of operation and analysis of technologies of surface hardening of coarse-grained gears**

One of the main conditions that technological equipment must meet is its trouble-free operation with the necessary reliability and durability in accordance with the technical operating conditions for a given period of time. In the mining industry, large-format open gears of external gear have found wide distribution. As a result of the study of the nomenclature of coarse-grained cogwheels used in mining enterprises, it was concluded that special attention should be given to the gears used for ball mills to extend the service life. The key issues in the design and manufacture of gears is the choice of the material and the way it is hardened.

Wear of the teeth is the main type of destruction of the teeth of open gears. Dustiness of the working area puts open gears in very unfavorable working conditions. Dust, containing a large number of solid abrasive particles, falls on the working surfaces of the teeth, mixed with grease and causes their intensive wear (Fig. 1).

Thus, the analysis of the damageability of gears makes it possible to assume that premature failure is due mainly to processes in the surface layers. Contact endurance of the surface layer is characterized by the ability of the material to resist the development

of dying on the working surface, perceiving variable contact loads, and is determined by structural and technological factors. Strengthening technologies, form the structure of the surface layer, create the basic level of hardening of the teeth of the wheels, determining their performance.



**Fig. 1.** Wear of the involute profile of the pinion shaft of the ball mill CBM 40x50

There are various methods of surface quenching, differing in the way of heating, cooling and temperature distribution in the treated layer, such as gas flame, induction or pulse. The main distinguishing feature of surface hardening methods by highly concentrated heating sources is the possibility of obtaining heating and cooling rates of materials that are several orders of magnitude higher than those typical for traditional methods of hardening (furnace hardening, quenching, high-temperature quenching, etc.), which facilitates the obtaining of hardened layers with previously unattainable level of operational properties.

A comparison of surface hardening methods, such as HFC, carburizing, nitriding, and ionic nitriding with respect to the hardness of the surface layer and the hardening depth, is given in Table 1.

However, here it is necessary to take into account the fact that surface hardening technologies have their own specific features and their effectiveness depends on the consideration of various factors that reflect both the protected part itself, its chemical composition, dimensions, geometric shape, initial surface condition, and the expected operating conditions for the whole complex of external factors. Therefore, the choice of hardening technology, in particular

gears, can not be universal in nature, but must be adapted to the specific wheel size in machines and aggregates. All this requires an individual approach both in the development of hardening technology and methods for assessing the damageability in accelerated tests.

**Table 1**

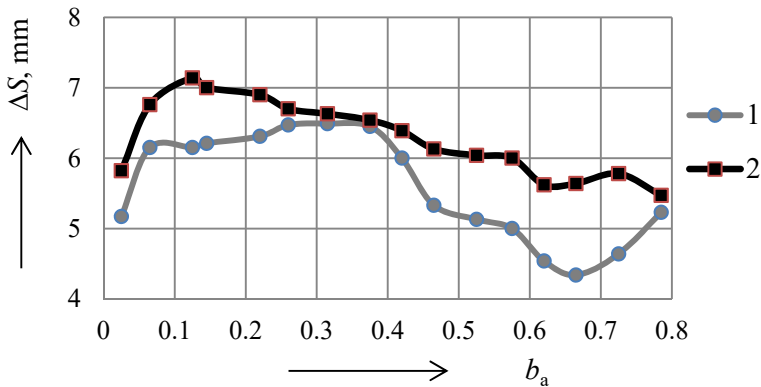
**Comparison of surface hardening methods for the hardness of the surface layer and the depth of hardening**

Type of surface hardening	Hardening of HFC	Cementation	Nitriding	Ionic nitration	Arc treatment	Laser hardening	Plasma hardening
Thickness of the surface layer h, mm	to 10	up to 2	up to 0,6	up to 0,35	up to 5	up to 1,5	up to 5
Hardness of the surface layer, HRC	56-60	58-63	56-66	56-70	35-50	55-65	55-65

The widespread industrial application of most known methods of strengthening treatment with concentrated energy flow is hampered by the high cost and complexity of the equipment, its inadequate reliability and productivity, the need for vacuum, special rooms with special requirements, the need for qualified maintenance, and high operating costs. In these conditions, in order to extend the operational life of wear parts, a method of surface heat treatment by a plasma arc is considered rational in terms of universality, availability, environmental friendliness and economic efficiency. Without changing the parameters of the surface roughness, such heat treatment is easily integrated into the technological process of preparation and repair of parts, being a finish operation, low-cost, sufficiently productive and effectively increasing their operational stability. As a result, it is proposed to apply plasma surface hardening (PPC) of the profile of the gear to improve the contact strength of the working surfaces of the teeth.

## 2. Investigation of a process of the open gears wearing

The experimental research of the teeth wearing of open gear gears of rotor mill mills has been carried out. As a result of the processing and analysis of experimental data averaged values (Fig. 2), which characterize various stages of wear, are obtained. It was established that the limit wear of the gear is limited to the period established and for the studied transmissions is  $3 \div 4$  mm from each tooth surface. The speed of work-wear wear on the gear is on average equal to  $V_{n1} \approx 5V_1$  ( $V_1$  is the speed during the wear period of the gear), and its duration is  $t_{\theta} \approx \Delta S_{10} / 5V_1$ .



**Fig. 2.** Wear of a profile of teeth ( $m = 20$ ) of mills CBM 4500x6000 in width

1 – the value of wear at a distance from the top of the tooth is 5 mm

2 – the value of wear at a distance from the top of the tooth 20 mm

For heavy-loaded elements of friction pairs, which include tooth gearings of open gears of mills, the wear rate, depending on the conditions of abrasive impact  $A$ , the physical and mechanical properties of materials  $M_{1(2)}$ , the geometric and kinematic parameters of conjugate  $K$  can be represented as:

$$U_{1(2)} = 6,8 \frac{AK}{M_{1(2)}} \quad (1)$$

Taking into account (1) and the results of experimental studies, the analysis and evaluation of the speed of wear of open gear gears



of ore-mills will be carried out according to the following formula:

$$U_{1(2)} = 60 N_{1(2)} K n_{1(2)} c_2 L_{1(2)} \quad (2)$$

where  $N_{1(2)}$  is the parameter characterizing the abrasive influence of the medium and the physical and mechanical properties of the materials of the teeth for the given experimental conditions (skid gear not protected from mechanical impurities, the material of the gears - structural steel, gear teeth with hardness  $HB_1 = 260-300$ , hardness crown teeth  $HB_2 = 180-200$ ; for mills working on roughly equal conditions, the average values of  $N_1 = 3,4 \cdot 10^{-10} \text{ mm}^{1/2}$ ,  $N_2 = 6,4 \cdot 10^{-10} \text{ mm}^{1/2}$ );

$n_{1(2)}$  – rotational speed, rpm;

$c_2$  – the number of engagements of crochet teeth in one turn;

$L$  is the coefficient taking into account the difference in the abrasive influence of the medium, the physical and mechanical properties of the material and the conditions of loading of the teeth for the calculated case:

$$L = \frac{\xi}{K_{q_a}^{2/3} K_R^{0.5} K_\sigma^{2.5} \mu_{\delta_{1(2)}}^2 \mu_{HB_{1(2)}}^{1.5} \mu_{HB_{2(1)}}$$

$$K_{q_a} = \frac{q_{a_N}}{q_{a_P}}; K_R = \frac{R_N}{R_P}; K_\sigma = \frac{\sigma_N}{\sigma_P}; \mu_\delta = \frac{\delta_P}{\delta_N}; \mu_{HB_{1(2)}} = \frac{HB_{P(1)2}}{HB_{N(1)2}};$$

(the quantities with indices  $N$  correspond to those for which the parameter  $N_{1(2)}$  is defined, and with the index  $P$  to the calculated case).

The rate of abrasive wear is affected by a value characterizing the physical and mechanical properties of the material of the gears  $M = \delta' HB_{1(2)}^{1.5} HB_{2(1)}$ . The value of  $\sigma$  is the composite value of the parameter characterizing the abrasive action of the medium. Nevertheless,  $\sigma$ , in addition, is a function of the hardness of the active surfaces of the teeth. Therefore, considering the influence of physical and mechanical properties of materials, we will take into account that the wear rate depends on the following quantities

$$U_{1(2)} \approx \frac{\sigma^{2,5}}{\delta' HB_{1(2)}^{1,5} HB_{2(1)}} \quad (3)$$

Increasing the hardness of the active surfaces of the teeth leads to

a decrease in  $\sigma$ . Analyzing expression (3), we note that increasing the hardness of the active surface of even one of the gears leads to a decrease in the strength of the abrasive particles and, consequently, the rate of wear.

Increasing the hardness of the active surfaces of the teeth of open gears, in addition, contributes to reducing the attendant types of wear, increasing their contact strength.

Thus, as a result of processing and analysis of the experimental data, averaged values characterizing the various stages of wear are obtained. Comparison of the experimental wear rates with the design values characterizing the geometry and kinematics of the gears, highlighted the parameter that took into account the abrasive action of the medium and the physical and mechanical properties of tooth materials for these experimental conditions. The latter allowed to substantiate the application of existing patterns of abrasive wear for engineering calculations for the wear of open gears of ore-grinding mills.

### **3. Technology and equipment for surface hardening of teeth with a scanning plasma arc**

The effect of plasma hardening is determined by an increase in the operational properties of the part due to a change in the physical and mechanical characteristics of the surface layer, due to the formation of a specific structure and phase composition of the metal, as well as the production of compressive residual stresses on the surface. In the transition zone, the heterogeneity of the structure increases in the following sequence: martensite-troostite, martensite and troostite grid, and transition to the ferrite grid at the boundary with the initial one.

Structural transformations as a whole correspond to those occurring in bulk hardening, however, high rates of heating and cooling cause a change in the relationships between structural components, a change in their morphology due to increased defectiveness of the crystal structure [1, 2].

The formation of stresses when cooling a surface subjected to plasma quenching can be represented as follows. After the termination of the action of the plasma arc, the inner layer of metal

located near the unheated source layer closes most rapidly, and the surface layer last. In the process of compression upon cooling, it acts on the inner layer, forming compressive stresses in it, while stretching is formed on the surface.

At the same time, in the surface layer of steels, the martensitic transformation occurs in the last place. Since martensite has a larger volume, in the near-surface layer, at this time, expansion takes place due to phase stresses and compressive stresses arise. The nature and extent of these phenomena depends on the thermophysical properties of the material being processed, the speed of relative movement of the workpiece and the heating source, and the thermal power of the plasma arc. By changing these parameters, it is possible to achieve an increase in the degree of manifestation of that side of the plasma effect on the workpiece material, the dominance of which is considered appropriate for this case.

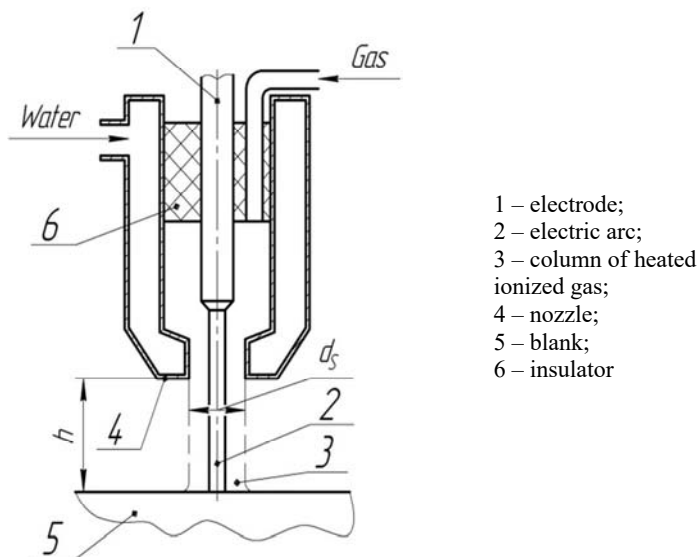
The structure of the reinforced layer, characterized by high hardness and high dispersion, has a determining effect on the change in the performance characteristics of hardened materials - wear resistance, mechanical properties (strength, ductility, crack resistance, endurance), heat and corrosion resistance.

In the beginning, it is necessary to formulate requirements, or criteria, which the source of heating must meet under the conditions of this processing method, namely hardening of coarse-grained gears [3]:

- 1 – the possibility of adjusting the heating width;
- 2 – no reflow of the surface layer;
- 3 – ensuring a uniform depth of structural transformations;
- 4 – Achievement of the necessary microstructure of the strengthened layer.

Plasma torches for plasma hardening should satisfy both the general requirements inherent in all plasmatrons (high power, stability of plasma flow parameters, significant energy efficiency, long duration of continuous operation, design reliability, ease of operation) and possess a number of specific properties: the protection of live parts and the ability adjustment of the plasma torch position. In addition, to ensure the effective heating of materials with different thermal characteristics in the design of the plasma torch, it is necessary to vary the heating parameters within fairly wide limits.

When choosing plasma generators, preference should be given to plasmatrons of direct action (Fig. 3). This is due to the fact that the operation of the plasmatron of indirect action is accompanied by the release of noise with a high overall level, which sharply worsens the sanitary and hygienic conditions of the operation of the thermist. The disadvantage of direct-acting plasmatrons is the high localization of heating by them of the surface of the workpiece.

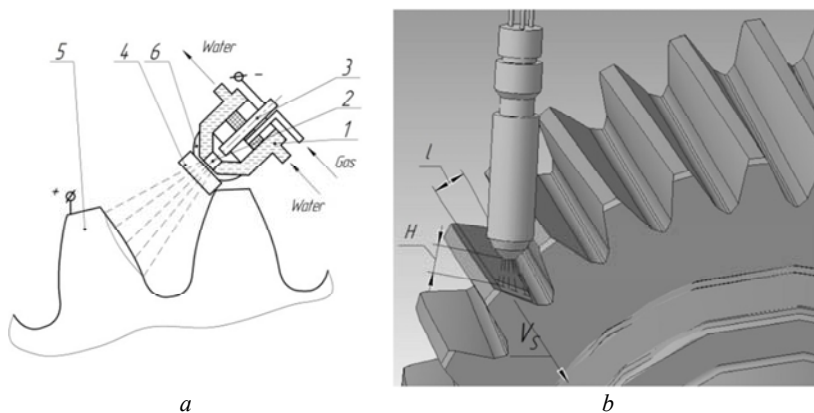


**Fig. 3.** Schematic diagram of a direct-acting plasma torch

To create an external magnetic field, a serially produced direct-acting plasmatron (PVR-402M) is supplied with a special magnetic system.

The idea of scanning a plasma arc over the surface of a product, the use of which reduces the specific heating power by an order of magnitude, increases the depth of hardening, reduces the probability of reflow and, ultimately, makes it possible to direct a high electric arc power to increase the productivity of the hardening process, was realized in a technological complex of equipment for the study of this process [4].

The method is implemented in this way (Fig. 4).

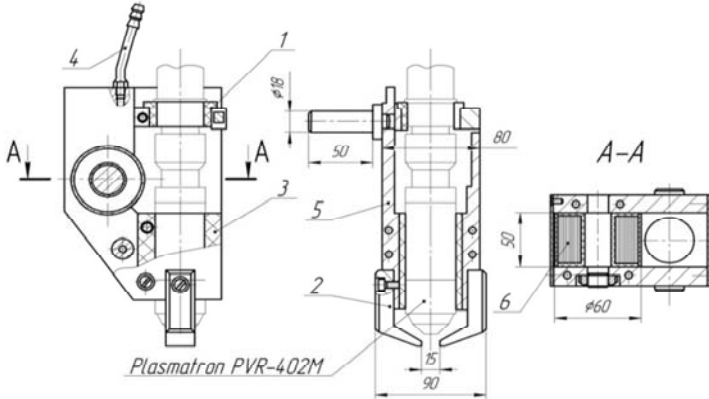


**Fig. 4.** The scheme of the device: *a* - the view in the processing plane; (1 - plasmatron, 2 - nozzle, 3 - cathode, 4 - magnet, 5 - gear tooth; 6 - electromagnetic coil); *b* - spatial view of the processing process

The toothed wheel or gear shaft is mounted on a machine, the plasma generator is a single-arc plasma torque converter. To create an external magnetic field and thereby reduce the location of heating, the plasma torch of direct action 1 equipped with a special magnetic system. The plasma torch generates an electric arc that burns between the cathode 3 and the lateral surface of the tooth of the wheel 5 and in the point of contact of the arc with the surface heats it. In addition, in the gap between the ends of the magnetic cores 4 the arc is influenced by the magnetic field, and since the electric arc is a conductor with current, then in the magnetic field on it acts Lorentz force, which deflects the arc in a plane parallel to the ends of the magnetic conductors 4 and perpendicular to the longitudinal line teeth. The alternating voltage on the coil 6 of the magnetic system provides fluctuations of the electric arc across the longitudinal line of teeth. As a result, when scanning the plasma arc with the frequency of the current in the network (50 Hz), the stain of the stain should be drawn along the working surface of the tooth 5.

For realization of this method a pilot-industrial complex was created on the basis of a lathe 1A660. Installation for the implementation of the method of SPAR contains: 1) plasma

installation APR-402; 2) Plasmatron PVR-402M with an electromagnetic scanner mounted on it (Fig. 5); 3) adjustable alternating current transformer to generate voltage in the magnetic system.



**Fig. 5.** General view of a plasma torch with a magnetic-deflection system:  
 1 – insulating sleeve; 2 – tip; 3 – placement; 4 – fitting; 5– Magnet's cheek; 6 – electric coil

The general view of a plasma torch with a magnetic-deflection system in the process of processing large-modular shaft-shaft ( $m = 22$ ) under production conditions is presented in Fig. 6



**Fig. 6.** The process of processing shaft-shaft by the method of PPP in production conditions

A choice of rational heating parameters at SPAR is made. The ratio of the size of the source and the value of the current strength of the arc determine the intensity value:

$$q_s = \frac{I \times U \times \eta}{\ell_s \times b_n}, \quad (4)$$

where  $U$  – voltage in the circuit of the plasma arc;  $\eta$  – efficiency of the heating source,  $\eta = 0,4$ ;  $\ell_s, b_n$  – dimensions of the heating source.

Having such a set of variable parameters, it is advisable to determine their rational ratio in accordance with the objectives of the SPAR. As a target function, it is possible to take some maximum temperature of surface layer heating, which provides the necessary structural transformations, which is determined from the expression:

$$\theta_{\max} = \frac{2q_s \sqrt{\omega \times \ell_s}}{\lambda \times \sqrt{\pi \times V_s}}, \quad (5)$$

where  $q$  – intensity of the source;  $\lambda, \omega$  – respectively, coefficients of heat and temperature conductivity of the material;  $V_s$  – the speed of the source movement.

Assuming that  $\theta_{\max}$  is equal to the melting point, determine the rate of displacement of the source  $V_s$ , at which the condition  $\theta_n \leq \theta_{\max}$  will be fulfilled:

$$V_o = \left( \frac{I \times U \times \eta}{\theta_{nn} \times b_n \times \ell_s \times \lambda} \right)^2 \times \frac{\omega \times b_n}{\pi}. \quad (6)$$

The SPAR process has a number of specific features, due to which there are limitations imposed on the parameters of the heating process. For the choice of the optimal values of the latter, the analysis of constraints is very significant.

#### **4. Evaluation of wear intensity of gears and structural transformations after SPAR**

Diagrams of temperature changes, obtained from the results of theoretical and experimental study of thermal fields from the action of a plane fast moving source, show that when processing steel 40X, the depth of the layer that underwent structural changes depends on the power of the heating source, the cooling rate, and the thermophysical properties of the material.

In Fig. 7 shows a sample of 40X steel, the surface of which has undergone plasma heating. On the end surface of macrosection, a layer is visible in the form of a uniform light band, differing in its etchability from the base metal.

In order to try to explain the phenomena occurring in the more deep-lying layers, it is necessary to return to the consideration of the initial structure of 40X steel, obtained as a result of previous technological operations of production of blanks.

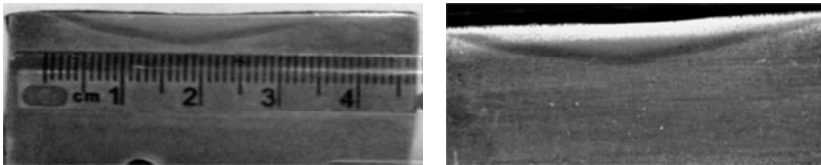


Fig. 7. Sample heated by a scanning plasma arc

Blanks heated to the temperature of  $A_{c3}$  and subjected to deformation, during subsequent cooling, a number of stages of decay of supercooled austenite proceed successively: diffusion pearlite, intermediate (bainitic), and diffusion-martensitic. As a result, a very complex metal structure is formed, consisting of upper and lower bainite, martensite and residual austenite.

Bainite is a two-phase mixture of ferrite and cementite crystals. Ferrite has low strength and high ductility. With a small number of cementite inclusions, plastic deformation develops relatively freely and the properties of the material are characterized by low hardness. If, as a result of heat treatment, cementite particles coalesce, some volumes of ferrite are released for the movement of dislocations, and the steel's ability to plastic deformation increases, i.e. plasticity increases. This fact is of great importance from the point of view of softening of the material, It is known that when a moving dislocation encounters insuperable inclusions, it passes through them, leaving the dislocation loops around the inclusions each time. The higher the accumulation of loops, the greater the hardening. As a result of high heating ( $\theta > 500^{\circ}\text{C}$ ), the metal layers located below the austenite structure obtained as a result of plasma heating undergo a high tempering. In this case, several processes occur. The main one is the decay of martensite, which consists in the release of carbon in the

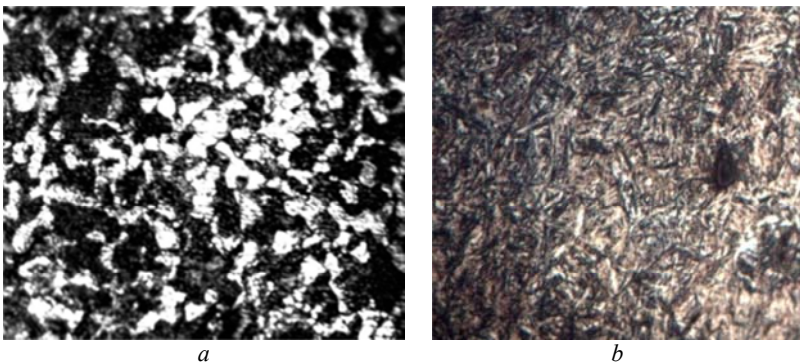


form of carbides, in addition, the residual austenite decomposes, carbide transformations and coagulation of carbides occur, the imperfections of the crystalline structure of S-solid structure and residual stresses decrease.

If, during slow heating, the removal of internal stresses and the coagulation of carbide particles occur in the range 300 ... 400 ° C, then the temperature of these transformations shifts upward along the temperature scale for high-speed heating, to which the plasma arc is heated with SPAR. At these temperatures (500 ... 700 ° C) there is a tendency for the hardness to fall, as well as for other strength parameters, while the plasticity indices ( $\Psi$ ,  $\delta$ ) increase.

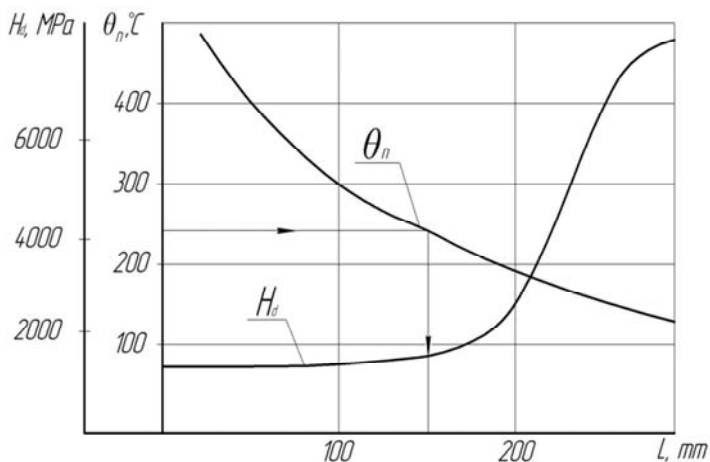
As the tempering temperature rises, cementite particles enlarge (Fig. 8), which, as indicated above, can lead to an increase in the plasticity of the material. With SPAR, metal layers lying at a depth of 7 ... 9 mm, although they do not heat above the  $Ac_3$ , but nevertheless, under certain heating conditions, temperatures of 600 ... 700 ° C can penetrate here. for example, at  $V_s = 3 \cdot 10^{-3}$  m / s, the size of the source  $l_s \times b = 0,05 \times 0,1$  and the current of the plasma arc  $I = 390$  A, the temperature of 600 °C, as calculated, can penetrate to the depth 9.8 mm.

Thus, the data presented show that the plasma heating of the stock material in the SPAR conditions can significantly change the state of its structure, which can lead to a significant change in the properties of the material in a favorable direction.



**Fig. 8.** Change in the structure of the material under the influence of plasma heating (increase 1000<sup>x</sup>): *a* – source structure; *b* – after SPAR

In order to study the properties of the alloyed steels (40X) in the cooling process, a measure of dynamic hardness was made, consisting in the introduction of an indenter in the investigated surface having a certain reserve of kinetic energy. As a result, graphs of temperature change of the surface and its dynamic hardness were obtained (Fig. 8). The analysis of the graphs showed that in the process of cooling there is a strengthening of the surface layers of the billet, heated above the temperature of  $A_{c3}$  and a sharp increase in hardness begins when cooled to a temperature of 200 ... 230 °C.



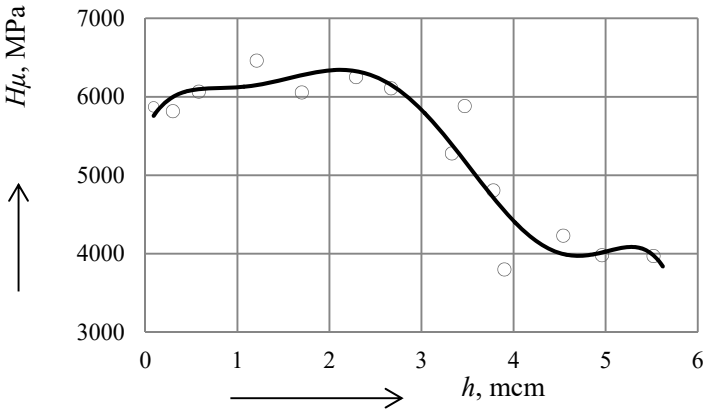
**Fig. 8.** Temperature and dynamic hardness change in the cooling process after plasma heating (steel 40X;  $I = 250$  A;  $V_s = 300$  mm/min;  $d_s = 6$  mm;  $Q_{pp} = 2.5$  m<sup>3</sup>/h)

Results of measurement of microhardness are presented on (Figure 9). From the presented graphs it is clear that the microhardness of the surface layers of the workpiece, processed in different thermal regimes, at a depth of 3 ... 5 mm significantly differs in size from the microhardness of the original structure of the material.

The presence of micro cracks in the course of research was not revealed. The analysis of the results showed the conformity of surface quality to the requirements proposed for the processed surfaces of parts.

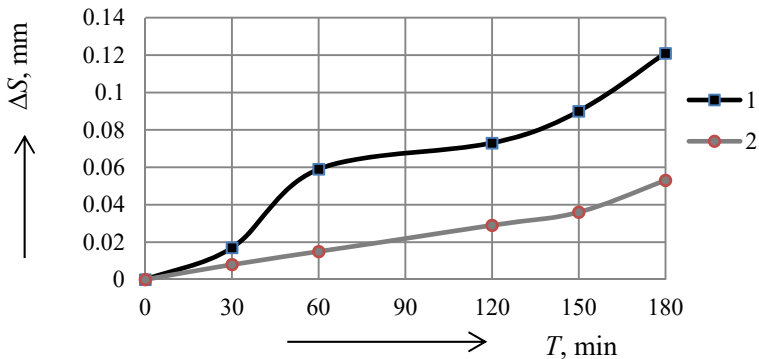
The tests (accelerated method) showed that SPAR of alloyed steels (40X) leads to an increase in its abrasive durability in 2.28

times, which can be explained by the structure of hardening, the formation of martensite and increased hardness.



**Fig. 9.** Change the microhardness of the surface after the SPAR

In fig. 10 shows wear curves of gear gears. Wear was measured by changing the thickness of the tooth as a difference in the thickness of the tooth in the dividing circle before the test and after a certain time.



**Fig. 10.** Intensity of abrasive wear of toothed wheels:  
 1 – wear of a non-tempered toothed wheel;  
 2 – wear of the toothed wheel of the subjected to the SPAR

The research has established that surface treatment with a plasma jet using a scanning magnetic system is an effective method of surface hardening of gear wheels and provides higher values of hardness and wear resistance.

### **Conclusions**

One of the most promising directions for the development of modern production is the introduction of intensive technologies for surface reinforcement of large-body gears using concentrated energy sources. Particular advantages over massive parts have a superficial plasma-arc reinforcement, due to high thermal power and the possibility of obtaining strong layers of considerable thickness (up to 5 mm). Increasing the durability of rotor mills' open large-modular gears, through surface plasma-arc reinforcements, has an important practical significance and is an urgent task.

The following advantages of plasma quenching can be noted in comparison with other methods of heat treatment:

1) when quenching concentrated energy sources, due to the specificity of the treatment (high heating and cooling rates), it is possible to obtain such a structure and properties of the surface layer, which are not achievable in the traditional methods of thermal treatment;

2) only the surface layer hardens and the core remains viscous, which provides increased resistance at the same time wear and fatigue;

3) the absence or minimal deformation of reinforcing parts, which makes it possible to increase the precision of their manufacturing, reduce the labor intensity of machining and the cost of manufacturing;

4) high performance;

5) when quenching without fading the surface, no subsequent machining is required, which makes it possible to use it as a finishing process of the process;

6) the presence of compressive stresses in the surface layer and

the presence of residual austenite increase the resistance to nucleation and crack propagation;

7) ease of maintenance, mobility, low cost and operating costs, small dimensions of equipment, the possibility of automation and rotation of the technological process.

The influence of technological parameters of this process on wear resistance and mechanisms of the surface layer wearing, namely current ( $I$ ), velocity ( $V$ ) and intensity ( $q$ ) of the heating source, were investigated in the work. It has been established that a significant reduction in wear will occur with the effect on the physical and mechanical properties of materials from which toothed transmissions are made, namely, increasing the hardness of the contact surfaces of the teeth through plasma strengthening. Under the conditions of SPAR, the efficiency of the plasma arc as a heating source depends essentially on the ratio of the thermal power; it is rational to consider such a ratio in which the temperature of the heated surface at a given size, its speed and the back of the heating spot reaches but does not exceed the melting temperature for this material.

The regularity of the formation of a tempered layer at the surface plasma-arched reinforcement of large-modular gear wheels has been experimentally established. It is shown that in the case of high-speed heating without melting by a plasma arc of direct action, which scans in an alternating magnetic field, the cooling rate of the surface layers reaches the rate of quenching, which provides structural transformations in the metal and forms a strengthened surface working layer (the strength of  $\sigma_v$  and HRC hardness increase). This ensures high performance of the surface working layer: wear resistance of gears increases by 2.28 times.

The method of heat treatment of large-wheel gear wheels is developed, which includes the heating of the lateral surface of the tooth without the melting of the plasma arc with a given current, with the displacement of the arc carried out translationally from one end of the gear to another, at a speed exceeding the rate of temperature propagation in this material.

A complex of specialized equipment was developed for the implementation of the technological scheme of plasma-arc surface reinforcement of the working surfaces of large-modular gears without melting, using the original design of the magnetic-deflection system for the creation of an external magnetic field.

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# **AN ANALYSIS OF SURVEYOR CONTROL OF LOSSES OF BALANCE-INDUSTRIAL SUPPLIES IS AT MASTERING OF BOWELS OF THE EARTH**

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## **Abstract**

Executed calculations of supplies taking into account the balanced manner on maintenance quality indexes minerals of supplies, show that the losses of balance-industrial supplies and impoverishments of content of quality indexes of minerals far more than they are certain on the accepted methodologies and that is why careful attitude toward the well balanced on maintenance quality indexes minerals of supplies in iron-ore mass, their timely maintenance and bringing in exploitation are major measures in relation to the guard of bowels of the earth and environment.

Offered methodology of control of the use of found-balance and maintenance of balanced on maintenance quality indexes minerals of supplies, that is attracted in exploitation, calculations conduct that after formulas, that take into account the volume of useful components attract that with breeds and balance on maintenance the quality indexes of minerals of supplies, so losses of balance-industrial supplies of that or other part of found-balance supplies of deposit. If not to conduct the separate account of all sources of entering iron-ore mass of useful components, then throw away opportunity objective comparison of job performances for the improvement of the use of bowels of the earth of areas of arrays of hard minerals that are in different mining-and-geological terms.

The considered methods of the surveyor providing of works are on determination of volumes of crop and setting of norms of quality indexes of balance industrial supplies of bowels of the earth, in that examine the losses of balance-industrial supplies and impoverishments of content of quality indexes of minerals as the determined (non-random) sizes. The conducted review of methods and technical upshots will allow to bring down losses of

balance industrial supplies and impoverishments) of content of quality indexes of minerals at a booty, ware housing and necessity of variegation of content of quality indexes of minerals in the stream of iron-ore mass. It is well-proven that for the correct choice of optimal (normative) the level of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass in every concrete case of the use of criterion of estimation of economic efficiency, that full enough takes into account the difference of variants of development for operating and capital charges. The criteria of economic evaluation at setting of norms of losses of balance-industrial supplies are differential mountain rent and income, that is counted on 1 τ balance supplies.

**Introduction.** Estimate rationality of process of mastering of mineral resources the indexes of plenitude of exception of them from the bowels of the earth and to the further processing. The complete losses of minerals consist on the average of losses: in the process of booty - 10-30 primary processing (enriching) to 20-40 metallurgical redistribution - 10-15. Especially severe losses at the primary processing of multicomponent mineral resources. Therefore, the number of «passing» components withdraws that from complex mineral raw material increases continuously. If in 1970 from the supplies of the colored and black metals withdrew 35 useful components, in 1990 their number attained 70, then in the beginning of the XXI of century - over 80. That is why a task of the complex mastering of bowels of the earth is *actual*.

Researches are based on materials of work of ore mining enterprises of Krivbass, that are in the central part of the Ukrainian shield that is the basic geostructural element of south-west of the east Europe platform. In the structure of the district, two structural floors participate: the crystalline foundation, made by metamorphism volcano upsetting and by granitoid formations of Pre-Cambrian and upsetting cover the cut of that is presented by the sedimentations of cainozoic. The structure of Kryvyi Rih belongs to one of the most interesting geological objects of Ukrainian of shield, that explains not only localization of bowels of the earth of unique supplies of iron-ore components but also original geological structure, history of geological development of region, that represents all basic stages of the formation.

*The aim of the work* is the development and introduction of methodology of determination of losses of balance-industrial



supplies and impoverishment of content of quality indexes of minerals taking into account the complex mastering of bowels of the earth.

For the achievement of the aim, such tasks are untied: The analysis of present methods of determination of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals; The improvement of existent methodologies of determination of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals taking into account the complex mastering of bowels of the earth; The establishment of norms of losses of balance-industrial supplies and impoverishment of content of quality indexes useful minerals.

*The idea of work* is analysis and determination of methods of calculation of optimal losses for development of economy of ore-mining enterprises and indexes of plenitude of the use of resources of bowels of the earth at present labor and material resources.

*The research object* is balance-industrial supplies of bowels of the earth.

*The subject of research* is losses of balance-industrial supplies and impoverishments of content of quality indexes of useful minerals.

**Analysis of present methods of determination of losses of balance-industrial supplies and impoverishment of content of quality indexes useful minerals.** Basic indexes of the use of supplies of bowels of the earth are losses of balance-industrial supplies and impoverishments of content of quality indexes of minerals in an array and in the stream of iron-ore mass [1]. In quality of indexes to them reciprocals – coefficient of exception of minerals from the bowels of the earth and coefficient of changeability of content of quality indexes of minerals are accepted at the booty of balance-industrial supplies. Methodology recommends also [2] the coefficient of obstruction of content of quality indexes of minerals. The first four indexes of the use of balance supplies are accepted officially by all ore-mining enterprises in accordance with [2] on determination, setting of norms and account, to the economic evaluation of losses of balance-industrial supplies of hard minerals at a booty, that is ratified [2]. Based on these, pointing corresponding branches and pool instructions were made: for the mines (quarries) of

ferrous metallurgy – one, for the mines (quarries) of standard – other, for enterprises of coal industry – third, to industry of building materials – fourth etc. These instructions are obligatory leading materials at planning, building and exploitation of all ore-mining enterprises. In this connection will bring only generals over on an account, estimation and setting of norms of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass with some working out in detail of the surveyor providing of works at working mine of hard minerals, in particular taking into account the complexity of the use of the bowels of the earth.

For correct determination of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass, the same as indexes of exception from the bowels of the earth and changeability of content of quality indexes of minerals of iron-ore mass, value has a choice of method of the surveyor providing of works, that most full answers that is why or to other type of minerals. The indexes of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals are needed for the decision of economic tasks in iron-ore mass must take into account not only content of quality indexes of minerals, lose that impoverishing breeds, but also where and on what stage of the survey or providing of project mountain works lose these minerals and impoverish.

Only classification of losses of balance-industrial supplies of hard minerals [3–5], built because of division on the technological processes of booty and places, where losses of balance-industrial supplies are. This classification is given for all methods of development and all hard minerals. Taking her for basis, will consider the types of the surveyor providing of works on determination of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in relation to the terms of development of iron-ore deposits.

Under the losses of balance-industrial supplies mean that part of balance supplies that do not withdraw, and under impoverishment of content of quality indexes of minerals is a decline of content of quality indexes useful to the component in digging in comparing to his content in the array of balance supplies. Subdivide the losses of

balance-industrial supplies into the losses of balance-industrial supplies in guard that does not withdraw even after liquidation of ore-mining enterprise and if barrier temporal in some period of time envisage their partial or complete exception, then minerals in that does not attribute to the losses of balance-industrial supplies, but set off to the balance supplies) the operating losses of balance-industrial supplies(quantitative and quality), that is related directly to the booty of balance-industrial supplies, as they largely differ in technological reasons and places of their formation.

Taking classification for basis [5–8,11–13], the basic operating losses of balance-industrial supplies in relation to the terms of development of iron-ore deposits will present as a table 1. In relation to that or another way of development or to the certain mining and geological conditions, a number of varieties of losses of balance-industrial supplies will be it is or diminished, or megascopic. Depending on the type of losses of balance-industrial supplies and impoverishment to content of quality indexes of minerals in an array and in the stream of iron-ore mass choose the corresponding methods of the surveyor providing of works on their determination, and the detailed selection of types of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in an array and in the stream of iron-ore mass allows in every separate case to choose more exact methods of the surveyor providing of their determination.

**Determination of volume of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass** matters for the decision ore-mining and economical and surveyor tasks. Foremost at a choice and comparison of methods and systems of development, determination of production capacity of mine (career), height of floor (to the ledge), estimation of balance-industrial supplies and establishment of standard on minerals, comparison of opening methods, determination of rational parameters of the systems of development and surveyor providing of technology of mountain works. Characterize the losses of balance-industrial supplies an amount and quality of part of minerals that abandon in the bowels of the earth, in comparing to liquidate balance-industrial supplies.

Table 1

**Basic operating losses of balance-industrial supplies and impoverishments of content of quality indexes of minerals are in relation to the terms of development of iron-ore deposits of Kryvyi Rih**

<b>Group, sub-group, kind</b>
<p><b>Losses of balance-industrial supplies in an array:</b> between areas (under blocks, interpanel, barrier, interchamber), in the sides of quarry; into the area, block, chamber, panel, post, quarry field of array of hard minerals; close the fire, flooded or heaped up areas, geological violations and mine-out space; in an array from incompleteness of exception: in a laying, hanging side, roof, sole, in parties of chambers; between the layers, in the attached areas; making that was brought down, fire, flooded areas heaped up; close inwardly contour breeds that remain, including; close array of establishment; as a result of complication of deposit, bed, ore body or area; on the bottom of quarry.</p>
<p><b>Losses of balance-industrial supplies of dissociated from an array:</b> in mine-out space: on the sole of open chambers, from incompleteness of producing and loading, on the ledges of quarry; on on the bottom of block and on a laying side at the systems with bringing down of minerals; on the bottom of chambers(to the sole), laying side from mixing at producing with the heaped up breeds; in a book-mark; in the places of bringing down, in obstructions, in the fire, flooded and attached areas: in obstructions from bringing down of roof of chambers and sides of quarries; in mine-out space from destruction of bottom of block; in the heaped up block and shop, in days a quarry; in the attached areas; in cleansing and entry-driving coalfaces: from mixing with breeds at general and at the separate exception of minerals and breeds; in the places of unloading and overload: in the underground making; on superficial folds, in bunkers; in the places of sorting and previous concentration; in the dumps of breeds and balanced minerals, in the assorted breeds; on transport ways of mountain enterprise : in the underground making, in the middle of quarry; on a surface.</p>
<p><b>Impoverishment of content of quality indexes of minerals</b> is on the stage of loosening the array: as a result of bringing in of breeds at the breakage: from the breeds of hanging and laying parties(to the roof, sole), on the contours of deposit, bed, ore body, layer; from the inwardly contour including there are the more set standards; from material(dry hydraulic book-mark that hardens and т. п.) of establishment, stale and heaped up breeds from the side of earlier exhaust blocks, chambers; from bringing in of breeds at separate breakage of minerals and in entry-driving coalfaces; as a result of abandonment of part of rich on maintenance quality indexes minerals : from the losses of balance-industrial supplies of more rich minerals(in ore layers in the bottom of blocks, sole, roof of chambers, at a laying side); from bringing down of breeds of hanging and laying parties and destruction of bottoms of blocks.</p>
<p><b>Impoverishment of content of quality indexes of minerals after loosening of array:</b> in cleansing coalfaces from adulteration of removed or breeds that was brought down: from bringing down of breeds of hanging and laying parties, on the contours of deposit, bed, ore body, at producing from chambers; from penetration in useful fossil, that apply lateral breeds at producing from blocks; from bringing down of breeds of roof in open chambers at loading; from the losses of balance-industrial supplies: more rich part of removed useful minerals(in particular, to the ore change) on the sole of chambers, ledges, on a laying side, on the bottom of block, inwardly-contour including, not articles of exception; from bringing down of material (book-mark that hardens) of establishment or heaped up breeds from the side of earlier exhaust blocks; from mixing with breeds at a separate exception; in the places of overload, warehousing, previous concentration and sorting.</p>

The coefficient of exception of balance-industrial supplies from the bowels of the earth characterizes an amount and quality of the obtained part of balance supplies. If balance-industrial supplies are lost on content of quality indexes of minerals does not differ from the balance-industrial supplies of block determine their volume directly in the process of the surveyor providing of realization of mountain works in a coalface, for example at the semilongwall of development of loss of balance-industrial supplies determine in parts units by a direct method after formulas

$$\Pi = \frac{\overline{\Pi}}{B} \quad \text{or} \quad \Pi = \frac{\sum_1^k \overline{\Pi}}{B}, \quad (1,2)$$

where is a  $\overline{\Pi}$  volume (mass) of the lost balance-industrial supplies;  $B$  is a volume(mass) of the liquidated balance-industrial supplies;  $k$  is a number of types of losses of balance-industrial supplies of minerals.

If content of quality indexes of minerals, lose that, differs from quality of balance-industrial supplies in middle on a block, then the losses of balance-industrial supplies it follows to determine after a formula

$$\Pi = \frac{\overline{\Pi}c_{\Pi}}{Bc} \quad \text{or} \quad \Pi = \frac{\sum_1^k \overline{\Pi}c_{\Pi}}{Bc}, \quad (3,4)$$

where  $c_n$  and  $c$  is content of quality indexes of useful component accordingly in the lost balance-industrial supplies and balance-industrial supplies of array of hard minerals.

At the surveyor providing of mountain works at the systems of development with bringing down of array of hard minerals of loss of balance-industrial supplies determine an indirect method after next formulas

$$\Pi = 1 - \frac{\Delta a}{Bc} \quad \text{or} \quad \Pi = 1 - \frac{\Delta a(a - b)}{B(c - b)}, \quad (5,6)$$

where  $\Delta$  is a booty of balance-industrial supplies of loosening iron-ore mass;  $a$  and  $b$  is content of quality indexes of metal accordingly

in the obtained iron-ore mass and impoverishing content quality indexes of useful component breeds.

Impoverishment of content of quality indexes useful minerals in iron-ore mass (changeability of content of quality indexes of minerals) characterize a decline in the process of booty of balance-industrial supplies of content in them quality indexes of useful components and increase of part of the finished mixing breeds in comparing to the same indexes in the balance supplies of array of hard minerals. In separate case at the surveyor providing of mountain works of development of iron-ore deposits of hard minerals, impoverishment of content of quality indexes of minerals in iron-ore mass determine attitude of mass of the finished mixing breeds toward the obtained iron-ore mass. Characterize the coefficient of changeability of content of quality indexes of the obtained balance-industrial supplies content of quality indexes in him (in comparing to the balance supplies of array of hard minerals) useful components or harmful admixtures, parts of the finished mixing breeds, by humidity, grade and other factors on that the degree of fitness of minerals depends for the further processing or use in a national economy. Impoverishment of content of quality indexes of minerals in iron-ore mass determine in parts of units a direct method after one of the over brought formulas.

$$P = \frac{B}{D}; \quad P = \frac{\sum B}{D}; \quad P = \frac{B}{A+B}, \quad (7,8,9)$$

where  $B$  is a volume (mass) of the minerals of breeds finished mixing on maintenance quality indexes; and  $A$  is a volume (mass) of the obtained balance-industrial supplies (part of the liquidated balance supplies).

$$P = 1 - \frac{a}{c}; \quad P = 1 - \frac{a-b}{c-b}; \quad P = \frac{c-a}{c-b}. \quad (10,11,12)$$

Both types the brought formulas over allow to estimate the use of balance-industrial supplies in two cases, when lose balance-industrial supplies with balance content of quality indexes of useful components, and at impoverishing on maintenance the quality indexes of obtained minerals breeds there are not useful components or there are useful components.

In two another cases, when lose balance-industrial supplies that on content of quality indexes differ from middle balance on maintenance the quality indexes of minerals and at impoverishing on maintenance the quality indexes of obtained minerals breeds there are not useful components or there are useful components. These formulas for the estimation of impoverishment of content of quality indexes of minerals in iron-ore mass not enough. For more clear surveyor providing of works on control of the use of found-balance and maintenance of balanced on maintenance quality indexes minerals of supplies, that will be in the near time attract in exploitation, calculations conduct after formulas, that take into account both mass of useful components attract that with breeds and balanced on maintenance the quality indexes of minerals of supplies (what attract sometimes in a booty technologically) and losses of balance-industrial supplies of that or other part of the found out supplies.

In third case, when lose the balance-industrial supplies of minerals, content of quality indexes of that differs from balance supplies on maintenance the quality indexes of minerals, but at impoverishing on maintenance quality indexes minerals breeds there are not useful components, determine the indexes of the use of balanced on maintenance quality indexes minerals of supplies after next formulas.

$$\Pi = \frac{Bc - \mathcal{D}a}{Bc_{\Pi}}; \quad P = \frac{B(c - c_{\Pi}) - \mathcal{D}(a - c_{\Pi})}{\mathcal{D}c}. \quad (13,14)$$

In most general, fourth case, when content of quality indexes of minerals in balance-industrial supplies, lose that, differs from middle content of quality indexes of minerals in the balance-industrial supplies of array of hard minerals and impoverishing on maintenance the quality indexes of minerals of breed contain useful components, formulas for determination of indexes of the use of balance-industrial supplies of bowels of the earth have such kind:

$$\Pi = \frac{B(c - b) - \mathcal{D}(a - b)}{B(c_{\Pi} - b)}; \quad P = \frac{B(c - c_{\Pi}) - \mathcal{D}(a - c_{\Pi})}{\mathcal{D}(c_{\Pi} - b)}. \quad (15,16)$$

Formulas are however above-mentioned suitable only for one component minerals without the account of possibilities of the

complex use of passing components of minerals. Today all less than one component minerals become and less than. In ferrous quartzite's except a basic component there is much copper, vanadium, zinc, lead and other useful components, part from them in composition wastes use as building material. Thus the cost of such macadam approximately equals prime prices of booty of iron-ore minerals. On some deposits, beds, ore bodies or areas of array of ferrous quartzite's content of quality indexes of titan, vanadium, cobalt, copper, zinc, sulphur, nickel, phosphorus, germanium and non-metallic minerals sometimes higher, what in the basic deposits of minerals of the coloured metals. The applied formulas are for determination of indexes «visible» losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass concordantly [5,7,8] both adulterations in iron-ore mass of useful components of containing breeds and their additions or reductions take into account in her due to abandonment in the losses of balance-industrial supplies of impoverished or enriched on maintenance the quality indexes of minerals of part. However the end-point allows exactly to take into account and divide the sources of bringing in iron-ore mass on maintenance the quality indexes of useful components and source of losses of balance-industrial supplies and on maintenance quality indexes minerals of supplies, as a result visibility of prosperity is created sometimes even in case of impermissible severe losses of balance-industrial supplies. At content of valuable component in breeds that apply, (often it arrives at 0,3–0,5 middle content of quality indexes of minerals) such visibility of prosperity is possible even at 30 % losses of balance-industrial supplies.

**For example.** In the balance-industrial supplies of ferrous quartzite's is to 32 % cities of quality indexes of iron, and at applying and containing on maintenance quality indexes minerals breeds is a 16 %. Volume of losses of balance-industrial supplies even 30 balanced supplies, but due to producing on maintenance the quality indexes of minerals of breeds in the volume of to 30 of 100 e exception on mountain mass (that quite possible), then on a formula (5) at content quality indexes of iron in the obtained iron-ore mass 27,2 %.

$$P = \Pi = 1 - \frac{Da}{Bc} = 1 - \frac{100 \cdot 27,2}{100 \cdot 32} = 0,15$$

what testifies to safe position, but 30 balanced supplies it is lost beyond retrieve, similarly as thrown away opportunity the use in the future presently balanced on



maintenance quality indexes minerals of supplies and breeds with content of quality indexes of iron 16 And the supplies of these breeds in a pool are enormous.

If not to conduct the separate account of all sources of entering iron-ore mass from the balance-industrial supplies of useful components, then lose another possibility of objective comparison of work for the improvement of the use of bowels of the earth of areas of arrays of hard minerals that are in the different mining(at presence of in the breeds of useful components and without them, at possibility of abandonment in the losses of balance-industrial supplies of poor on maintenance quality indexes minerals and without them) and geological conditions. In an order to take into account this important circumstance, some other factors (even partly), for example multicomponent of minerals and possibility of determination of losses of balance-industrial supplies at the surveyor providing of booty and complexity of the use of mineral raw material, it is expedient to replace an index – content of quality indexes of useful components (metals) by next indexes:

- minerals, that withdraw the value of content of quality indexes in the balance supplies of  $u_{\delta}$ ;
- in supplies, that lose  $u_n$ ;
- at impoverishing on maintenance quality indexes minerals breeds of  $u_p$ ;
- by a value on maintenance quality indexes in digging, that withdraw  $u_{\delta}$ .

Then for four losses of balance-industrial supplies and impoverishment considered earlier cases on maintenance the quality indexes of minerals of breed determine after next formulas:

a) when  $u_{\delta}=u_n$  i  $u_p=0$

$$\Pi = 1 - \frac{D u_{\delta}}{B u_{\delta}}; \quad P = 1 - \frac{u_{\delta}}{u_{\delta}}; \quad (17,18)$$

б) when  $u_{\delta}=u_n$  i  $u_p \neq 0$

$$\Pi = 1 - \frac{D(u_{\delta} - u_p)}{B(u_{\delta} - u_p)}; \quad P = 1 - \frac{u_{\delta} - u_p}{u_{\delta} - u_p}; \quad (19,20)$$

в) when  $u_{\delta} \neq u_n$  i  $u_p=0$

$$\Pi = 1 - \frac{D u_{\delta}}{B u_{\delta}}; \quad P = \frac{B(u_{\delta} - u_n) - D(u_{\delta} - u_n)}{B(u_{\delta} - u_p)}; \quad (21,22)$$

г) when  $u_{\sigma} \neq u_n$  i  $u_p \neq 0$

$$\Pi = \frac{B(u_B - u_n) - D(u_D - u_p)}{D(u_{\Pi} - u_p)}; P = \frac{B(u_B - u_n) - D(u_D - u_p)}{D(u_n - u_p)} \quad (23,24)$$

With these formulas study not only entering sources balance-industrial supplies of useful components (exception of rich part of balance-industrial supplies, addition useful components with impoverishing on maintenance the quality indexes of minerals breeds) but also entering sources iron-ore mass of harmful components.

**Indexes of the use of bowels of the earth are with taking into account of sibilance on maintenance quality indexes minerals of supplies.** Experience of development of iron-ore deposits testifies that in many cases she is carried out in a few stages. On the measure of working off the richest deposits, beds, ore bodies or areas of array of hard minerals in exploitation attract more poor on maintenance quality indexes minerals. In a number of cases with high efficiency already work off deposits, beds, ore bodies or areas of arrays of balance-industrial supplies that yet recently distinguished as sibilance. Maintenance and account presently of sibilance on maintenance quality indexes minerals of supplies matter Therefore, especially for creation of raw mineral-material base of country on the nearest years. If to take into account that content of quality indexes of the coloured and liquid metals in the obtained balance-industrial supplies annually goes down on 2,0-3,0 % of units, and content of quality indexes of iron is on 0,5-1,2 % then it is possible to assume, that breeds with content of quality indexes of the colored metals and iron according to a 0,25-0,32 % 0,52-0,65% city them quality indexes in balance-industrial supplies already in 5-10 % can be attracted in exploitation, because content of brack of quality indexes of useful components usually presents for the minerals of the coloured and liquid metals 0,42-0,53 % and for iron-ore - to a 0,63-0,84 % city them quality indexes in balance-industrial supplies. On many deposits the supplies of zabalance and poor on maintenance quality indexes minerals considerably exceed balance-industrial supplies both on a volume and on content of quality indexes useful minerals considerably exceed balance-industrial supplies both on a volume and on content of quality indexes of useful components.

On many deposits of hard minerals exploitation of poor and sibilance on maintenance quality indexes minerals of supplies is already conducted. In the Kryvyi Rih pool develop ferrous quartzite's in that on maintenance quality indexes iron presents 32-37 % id est 0,76-0,82 % and 0,61-0,73 % medium on maintenance the quality indexes of minerals in the balance-industrial supplies of rich on maintenance quality indexes minerals. Mark at the same time, that a guard and rational use of balanced on maintenance quality indexes minerals of supplies on enterprises carry out levels not on a due. Yes, from data [8,11–13], at development of rich on maintenance quality indexes minerals of supplies of Kryvyi Rih counterfeited and, thus, up to a point 2,6 milliards of  $\tau$  of ferrous quartzite's are lost for future development. On some ore mining enterprises the surveyor providing of mountain works is at development of balance-industrial supplies on mine enterprises carry out levels not on a due. To such attitude toward the balanced on maintenance quality indexes minerals of supplies the methods of their determination and account applied at this time promote in a great deal. In accordance with them the volumes of losses of balance-industrial supplies, impoverishment on maintenance the quality indexes of minerals and indexes of exception of balance-industrial supplies provide for to determine without an account on maintenance the quality indexes of minerals, lose that and volumes presently balanced on maintenance quality indexes minerals of supplies, that bring over to the booty. On the deposits of hard minerals where clear differentiations are between ore bodies and containing breeds that apply and does not contain the quality indexes of useful components, and also on deposits work off that the systems with a book-mark et al, fully sufficiently methods of the surveyor providing of works on an account, that provide for [5,14], however in most cases they do not allow to provide the complex and rational use balance-industrial and balanced on maintenance quality indexes minerals of supplies.

Zabalance on maintenance the quality indexes of minerals supplies enormous money is expended in secret service of that, unfortunately, while economically not estimated and, thus does not have a value. In the total in mining districts a giant loss the consequences of that it is while difficult to estimate is inflicted a future raw mineral-material base. Quite obviously, that the problem

of the complex mastering of bowels of the earth can not be decided on the basis of estimation and account of the use only of balance-industrial on maintenance quality indexes minerals of supplies, id est supplies of today, without a corresponding estimation and account presently of zabalance on maintenance quality indexes minerals of supplies, or supplies of future periods the especially nearest. Not to take into account them – so to conduct disorderly predatory exploitation of bowels of the earth, but especially their part - balanced on maintenance quality indexes minerals of supplies, that can as it is visible on the example of Kryvyi Rih considerably exceed balance supplies.

The decision of problem of the complex mastering of balance-industrial and balanced on maintenance quality indexes minerals of supplies requires substantial changeability of scientific bases of choice of basic parameters of mine (career), methods of opening of deposit, bed, ore body or areas of array of hard minerals and redemption of emptiness's, systems of development and first of all account, estimation, determination and setting of norms of losses of balance-industrial supplies at the surveyor providing of works at a booty. The same touches the applied methods of account and estimation of exception from the bowels of the earth of balance-industrial supplies and content of quality indexes of minerals, obtain that, including to both balance and balanced on maintenance the quality indexes of minerals supplies. If application of the systems, that answer more complete mastering of balance and balanced on maintenance quality indexes minerals of supplies and methods the redemption of emptiness's, related sometimes to some increase of charges (for the sake of the future considerable winning), then application of more exact methods of the surveyor providing of works on the account of losses of balance-industrial supplies, impoverishment on maintenance the quality indexes of minerals and indexes of exception of balance-industrial supplies provides a considerable economic effect without additional charges.

From position of the complex mastering of bowels of the earth, that apply at this time the methods of the surveyor providing of works on determination and account of indexes of losses of balance-industrial supplies, impoverishment on maintenance the quality indexes of minerals, rational exception from the bowels of the earth

of balance-industrial supplies and content of quality indexes of digging.

Accordingly [3] from the surveyor providing of works on the account of losses of balance-industrial supplies and impoverishment on maintenance the quality indexes of hard minerals, the «actual» losses of balance-industrial supplies and impoverishments determined only on maintenance the quality indexes of minerals, losses of balance-industrial supplies and impoverishments are on maintenance the quality indexes of minerals of balance supplies. The lack of these methods consists in that they do not take into account useful components in finished mixing to iron-ore mass of breeds, losses of balance-industrial supplies balanced after impoverishment on maintenance the quality indexes of minerals and corresponding to them indexes of exception from the bowels of the earth of balance-industrial supplies and changeability of content of quality indexes of minerals in the obtained balance-industrial supplies concordantly [3]. Use of indexes and formulas in [2,8] and branch instructions [5,7] take into account a multicomponent on maintenance the quality indexes of minerals of balance-industrial supplies on the basis of determination of their value, and also total content of useful components in the obtained balance-industrial supplies regardless of or there were they in balance-industrial supplies or introduced with impoverishing on maintenance the quality indexes of minerals breeds or balanced on maintenance the quality indexes of minerals supplies. At the same time, unlike the actual losses of balance-industrial supplies and impoverishment on maintenance the quality indexes of minerals, such indexes, as a coefficient of exception of balance-industrial supplies from the bowels of the earth, the coefficient of changeability on maintenance the quality indexes of minerals and «visible» losses of balance-industrial supplies and impoverishments on maintenance the quality indexes of minerals (concordantly [5]) quite not take into account neither a possible rejection on maintenance the quality indexes of minerals, lose that from middle on maintenance the quality indexes of minerals of balance-industrial supplies, nor part and impoverishing on maintenance quality indexes minerals of breeds (of content in them useful or harmful components).

None of formulas [8] and branch instructions [2,5] for determination of volumes of losses of balance-industrial supplies,

impoverishment on maintenance the quality indexes of minerals, coefficients of exception of balance-industrial supplies from the bowels of the earth and changeability of content of quality indexes of minerals does not take into account the possible bringing in exploitation with the impoverishing breeds of balanced on maintenance quality indexes minerals of supplies or them possible earning additionally, that eliminates normal exploitation of these balanced on maintenance quality indexes minerals of supplies in the future.

On ore-mining enterprises planning content of quality indexes of useful components in iron-ore mass is provided due to earning additionally presently substandard balanced on maintenance quality indexes minerals of supplies, that eliminates their normal exploitation in the future. Earning additionally of supplies of poor on maintenance quality indexes minerals is inflict a loss not only to the bowels of the earth, but potentially and to the environment, because instead of present it will be to master the new deposits of hard minerals and accordingly to distort the ecological balance in new districts. Therefore for more complete and complex use of bowels of the earth and improvement of conservancy it is necessary to take into account all subtleties of the use of balance-industrial and balanced on maintenance quality indexes minerals of supplies in close connection with conservancy.

That more exactly and clearly to control the use self of balance-industrial supplies, but not balance-industrial supplies together with balanced on maintenance the quality indexes of minerals of supplies and by the enriched useful components by breeds and simultaneously to provide an account and maintenance of balanced on maintenance quality indexes minerals of supplies, that can be in the near time also attracted in exploitation, it is necessary to use formulas [9,10], what is taken into account by receivables in iron-ore mass of useful components separately from balance-industrial supplies, containing breeds and balanced on maintenance quality indexes minerals of supplies, and also what quality indexes and how many useful components abandon in the lost iron-ore mass, but not in general in the bowels of the earth.

**Taking into account of amount and quality of attracted in the booty of balanced on maintenance quality indexes minerals of**

**supplies**, and also amount and content of quality indexes of attached here and broken, restored to a state, useless for further development in the future, balanced on maintenance quality indexes minerals of supplies. Balance of content of quality indexes of metals at the surveyor providing of works at development of balance-industrial supplies with the partial bringing in exploitation of him balanced on maintenance quality indexes minerals of supplies and adulteration it is hard to iron-ore mass of certain part of impoverishing on maintenance quality indexes minerals of breeds looks like

$$\Delta a = Bc + q_1 Bc_3 - \Pi Bc + Bb, \quad (25)$$

where  $\Delta$ ,  $B$ ,  $B$  is mass accordingly the liquidated balance-industrial supplies obtained, and finished mixing to loosen to iron-ore mass of impoverishing on maintenance quality indexes minerals of breeds,  $\tau$ ; and,  $c$ ,  $c_3$  and  $b$  is content of quality indexes of metal or other useful component accordingly in the obtained iron-ore mass from the array of balance-industrial supplies, content of quality indexes of minerals in balance-industrial supplies, at balanced on maintenance quality indexes minerals supplies and impoverishing on maintenance quality indexes minerals breeds,  $q_1$  is a fate of attracted in exploitation of balanced on maintenance quality indexes minerals of supplies (in relation to balance-industrial), part of units.

If mass of the lost balance-industrial supplies equals  $\Pi \times B$ , then mass of finished mixing to loosen to iron-ore mass of impoverishing on maintenance quality indexes minerals of breeds can be certain after expression:

$$B = \Delta + \Pi B - B - q_1 B. \quad (26)$$

Putting of this expression in a formula (25), determine the losses of balance - industrial supplies (in parts of units) after a formula:

$$\Pi = 1 + \frac{q_1(c_3 - b)}{c - b} - \frac{\Delta(a - b)}{\Delta(c - b)} \quad (27)$$

and impoverishment of content of quality indexes of minerals in iron-ore mass after a formula:

$$P = \frac{\Delta(c - a)}{\Delta(c - b)} - \frac{Bq_1(c - c_3)}{\Delta(c - b)}. \quad (28)$$

If balanced on maintenance the quality indexes of minerals supplies does not attract in exploitation, and only earn additionally and violate, id est when  $q_1=0$ , then formulas (27) and (28) look like:

$$\Pi = 1 - \frac{\mathcal{D}(a-b)}{B(c-b)}; \quad P = \frac{c-a}{c-b}. \quad (29,30)$$

Calculations in obedience to these the formulas show, if to examine the supplies o minerals taking into account the balanced on maintenance quality indexes minerals of supplies, then volumes of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass in reality far more than they are certain on the accepted methodologies.

**For example.** Will expect the volume of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass for the terms of the Kryvyi Rih pool, if  $c=56$   $c_3=35$ ;  $b=16$ ;  $q=0,1$ ;  $a=50$ ;  $\mathcal{D}$ =of 100 т;  $B$ =of 100 т. On the usually applied formulas (5) and (10) the volumes of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass will present

$$\Pi = 1 - \frac{\mathcal{D}a}{Bc} = 1 - \frac{100 \cdot 50}{100 \cdot 56} = 0,11$$

$$P = 1 - \frac{a}{c} = 1 - \frac{50}{56} = 0,11$$

In reality according to formulas (27) and (28)

$$\Pi = 1 + \frac{0,1(35 - 16)}{56 - 16} - \frac{100(50 - 16)}{100(56 - 16)} = 0,2 ;$$

$$P = \frac{56 - 50}{56 - 16} - \frac{0,1 \cdot 100(56 - 35)}{100(56 - 16)} = 0,1$$

Therefore careful attitude toward the balanced on maintenance quality indexes minerals of supplies, their timely maintenance and bringing in exploitation are major measures in relation to the guard of bowels of the earth and accordingly all environments. Like a previous conclusion balances of mass and values to the case of the surveyor providing of works on the account of the partial bringing in exploitation and earning additionally of balanced on maintenance quality indexes minerals of supplies in relation to the terms of multicomponent minerals can be presented by the next system.

$$\begin{cases} \mathcal{D}u_o = Bu_o + q_1 Bu_3 - \Pi Bu_o + Bu_p, \\ B = \mathcal{D} + \Pi B - B(1 + q_1) \end{cases}, \quad (31)$$



where  $u_{\delta}$  and  $u_{\delta}$  is a value of digging from the array of balance-industrial supplies and minerals in the array of balance-industrial supplies;  $u_3$  and  $u_p$  is a value of mineral sat the balanced on maintenance quality indexes minerals of supplies and impoverishing on maintenance quality indexes minerals of breeds.

Untiing the system of equalizations relatively  $\Pi$ , obsessed:

$$\Pi = 1 + \frac{q_1(u_3 - u_p)}{u_{\delta} - u_p} - \frac{D(u_{\delta} - u_p)}{B(u_{\delta} - u_p)}; \quad (32)$$

$$P = \frac{B}{D} = \frac{u_{\delta} - u_{\delta}}{u_{\delta} - u_p} - \frac{Bq_1(u_{\delta} - u_p)}{D(u_{\delta} - u_p)}. \quad (33)$$

Expressions of indexes of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass (27) and (28) characterize only the state of the use of sibilance on maintenance quality indexes minerals of supplies basic to the component. They are the useless for description uses that sometimes attract in exploitation or the counterfeited balanced on maintenance quality indexes minerals of supplies that also beds together with other minerals, breeds of opening and driving of making, feigns etc. For the decision of task of the complex use and mastering of deposit, bed, ore body or areas of array of hard minerals, creation had rejected and unrejected technologies of booty of balance-industrial supplies and processing of content of quality indexes of minerals in iron-ore mass it is necessary to create the corresponding methods of the surveyor providing of works on determination of losses of balance-industrial supplies and processing of content of quality indexes of minerals in iron-ore mass.

**Determination of volumes of losses of balance-industrial supplies and impoverishment is on maintenance the quality indexes of minerals taking into account the complex mastering of bowels of the earth.** At development of complex deposit, bed, ore body or areas of array of hard minerals withdraw one useful fossil, and other often loses for further development. The value of passing minerals sometimes considerably exceeds the value of basic useful fossils and, naturally, loss from the losses of balance-industrial supplies large enough. However, at determination of volume of

losses of balance-industrial supplies and loss from them this variety of losses of balance-industrial supplies, similarly as volumes of losses of balance-industrial supplies balanced on maintenance the quality indexes of minerals supplies that operate in present tense methods quite not take into account. In quality of losses of balance-industrial supplies will consider privation of possibility of the useful use of mine-out space. So, for example, at the systems with bringing down and with a complete book-mark this possibility is practically eliminated, at the systems with cleansing open-space and with his partial book-mark there is large possibility of the use of greater volumes of mine-out space.

The complex mastering of bowels of the earth requires application of corresponding methods of determination and estimation of losses of balance-industrial supplies of minerals, that take into account the features of complexity and plenitude of the use of all supplies (balance and balanced) and useful all of the tools, and also all other possibilities of receipt of that or other effect from the booty of balance-industrial supplies of concomitant minerals (use of making, mine-out space, different sort of wastes, cultivation of earth surface and fertile earth, productive water sources etc.). From positions of the complex mastering of bowels of the earth and other natural resources a loss consists of losses of balance-industrial supplies: useful components in the contours of balance-industrial supplies; balanced on maintenance quality indexes minerals of supplies, that violate at development of balance-industrial supplies of basic useful fossils; passing minerals; mine-out space suitable for the useful using (for at additional charges on his equipping) with modern amenities; tails, wastes suitable for the use today or in the future; to fertility of earth busy mountain taking and his economy and from the decline of fertility of the surrounding mountain taking of earth; water resources (part of these losses, that will be on lost useful fossils); animal kingdom.

At the surveyor providing of works on the account of potential value of passing minerals that bed together with a basic useful component, mine-out space of breeds and wastes of mountain and ore mining and processing production balance of values on a deposit, bed, ore body or areas of array of hard minerals is presented like previous. At terms, when lose all passing minerals and does not use

other possibilities (mine-out space, opening breeds, wastes of booty of balance-industrial supplies and enriching of content of quality indexes of minerals, is in iron-ore mass), losses of balance-industrial supplies in two simplest cases, when  $u_n = u_{\sigma}$  and  $u_p = 0$ , and also  $u_n = u_{\sigma}$  and  $u_p \neq 0$ , determine by formulas:

$$\Pi = 1 - \frac{\mathcal{D}_1 u_{\mathcal{D}1}}{B u_{\sigma 1} + \sum_{i=1}^n r_i B_i u_{\sigma i}}; \quad \Pi = 1 - \frac{\mathcal{D}_1 (u_{\mathcal{D}1} - u_{\mathcal{D}p1})}{B (u_{\sigma 1} + \sum_{i=1}^n r_i B_i u_{\sigma i} - u_{\sigma p1})}, \quad (34,35)$$

where  $\mathcal{D}_1$ ,  $B$  is mass of digging from the array of balance-industrial supplies and balance-industrial supplies in the bowels of the earth of basic useful fossil,  $\tau$ ;  $u_{\sigma 1}$  and  $u_{\sigma i}$  is a value of content of quality indexes of minerals in iron-ore mass of obtained useful fossils from the array of balance-industrial supplies and balance supplies of basic useful fossil, hrn./of  $\tau$ ;  $r_i$  is a coefficient that shows, in the how many times greater or less balance-industrial supplies of  $i$ -a of useful fossil from the array of balance supplies of basic useful fossil;  $B_i$  is balance-industrial supplies of  $i$ -a of useful fossil,  $\tau$ ;  $u_{\sigma i}$  is a value of content of quality indexes of minerals in iron-ore mass of  $i$ -a of useful fossil obtain that from the array of balance-industrial supplies to the hrn./of  $\tau$ ;  $u_{\sigma p1}$  is a value of impoverishing on maintenance quality indexes minerals of breeds, obtain that, on basic useful fossil, hrn./of  $\tau$ .

Like there can be the decided tasks on determination of losses of balance-industrial supplies and impoverishment to content of quality indexes of minerals in iron-ore mass for another cases, id est when  $u_n \neq u_{\sigma}$  and  $u_p = 0$ , and also when  $u_n \neq u_{\sigma}$  and  $u_p \neq 0$  [9,10].

**Thus**, basic indexes of the use of supplies of bowels of the earth are losses of balance-industrial supplies and impoverishments of content of quality indexes of minerals in iron-ore mass and in relation to that or another way of development or to the certain mining and geological conditions, a number of varieties of losses of balance-industrial supplies can be it is or diminished, or megascopic.

1. Depending on the type of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass choose corresponding to them methods of the surveyor providing of works on their determination. The detailed selection of

types of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in iron-ore mass allows in every separate case to choose the exact methods of their determination.

2. On the basis of the conducted analysis of present methods of determination of volumes of losses of balance-industrial supplies it is analyzed to content of quality indexes of minerals in relation to the terms of development of iron-ore deposits and it is set that the complex mastering of bowels of the earth, requires application of corresponding methods of determination and estimation of losses of balance-industrial supplies of minerals that take into account to the feature of complexity and plenitude of the use of all supplies balance and all minerals.

3. Methodology of control of the use of found-balance and maintenance of balanced on maintenance quality indexes minerals of supplies, that is attracted in exploitation, calculations it is necessary to conduct after formulas, that take into account both mass of useful components attract that with breeds and balanced on maintenance the quality indexes of minerals of supplies and losses of balance-industrial supplies of that or other part of found-balance supplies and if not to conduct the separate account of all sources of entering iron-ore mass from the balance-industrial supplies of useful components, then throw away opportunity objective comparison of work for the improvement of the use of bowels of the earth of areas of array of hard mineral, that be in different mining-and-geological conditions.

4. On the ore-mining enterprises, working off balance-industrial supplies execute with the groundless bringing in exploitation of part of balanced on maintenance quality indexes minerals of supplies, as a result violate their arrays and throw away opportunity the effective use in the future of basic part, or application of the system of development and the methods of redemption of mine-out space do not allow to return on old areas for the exception of balanced on maintenance quality indexes minerals of supplies.

5. Implementation calculations show, if to examine the supplies of minerals taking into account the balanced on maintenance quality indexes minerals of supplies, then the losses of balance-industrial supplies and impoverishments of content of quality indexes of

minerals far more than they are certain on the accepted methodologies in reality.

6. The complex mastering of bowels of the earth requires application of corresponding methods of the surveyor providing of works on determination and estimation of losses of balance-industrial supplies of minerals that take into account the features of complexity and plenitude of the use of all supplies and all minerals, and also all other possibilities of receipt of that or other effect from the booty of balance-industrial supplies of concomitant minerals.

7. Certainly, that for the correct choice of optimal (normative) level of losses of balance-industrial supplies and impoverishment of content of quality indexes of minerals in every concrete case use the criterion of estimation of economical efficiency, that takes into account the difference of variants of development on running and capital expenses. The criteria of economic evaluation at setting of norms of losses of balance-industrial supplies are differential mountain rent and income calculating on 1 t of balance supplies.

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