2nd INTERNATIONAL SCIENTIFIC AND TECHNICAL INTERNET CONFERENCE "INNOVATIVE DEVELOPMENT OF RESOURCE-SAVING TECHNOLOGIES OF MINERAL MINING AND PROCESSING"

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WELCOME SPEECH FOR THE PARTICIPANTS OF THE 2ND INTERNATIONAL SCIENTIFIC AND TECHNICAL INTERNET CONFERENCE "INNOVATIVE DEVELOPMENT OF RESOURCE-SAVING TECHNOLOGIES OF MINERAL MINING AND PROCESSING"

Dear colleagues,

On behalf of the Academy of Mining Sciences of Ukraine, let me greet you at the Internet conference and thank you for your contribution. I am confident that application of Internet resources will be very helpful for your further research activity.

Thanks to the assistance of the Academy of Mining Sciences of Ukraine, the conference has entered the international scene for the second time becoming a good annual event. In my opinion, it is a great advance for the mining science as the conference facilitates opinion exchange among young and experienced scientists worldwide and enhances links of science and practice.

It is a unique forum of mining researchers generating new ideas and solutions through scientific research, the results of which will be implemented in real-life industrial conditions.

The conference is also an important event for researchers of allied sciences. There is no doubt that it will contribute much to their intellectual interaction and cooperation creating additional incentives for their further efficient integrated work.

Contribution of well-known scientists makes the conference authoritative and valuable, while participation of young researchers is equally important as they represent the future of mining as a science. Integration of various levels and organization forms in education and science is a new trend spreading worldwide. In recent years, there have been numerous discussions concerning creation of forms comprising scientific and educational potentials. It is essential to realize that training highly qualified specialists is not the aim of a separate educational institution only, but that of the whole educational system. It concerns a system unique in its scale and dynamic in character resulting from mutual efforts of many Eastern European countries.

Our conference deals with a variety of urgent issues of the mining science, in particular:

- formation of the innovative concept of the mining strategy;
- innovations and investments in mining as a science and industry;
- integration of industry-specific and regional research centers;
- theory and practice of transformation of the global raw material market and international cooperation of its producers and suppliers;
- issues of reforming mining education and university research;
- social and economic conditions and development of mining regions;
- topical trends in developing geology, mining machine building, ecology and mine surveying;
- elaboration of methods of raw material prospecting and mining (coal, oil, natural gas and uranium);
- improvement of tools of economic and mathematical simulation of mining production;
- application of up-to-date information technologies to scientific and practical activity in mining;
- elaboration of methods of scientific forecasting, planning and designing of mining operations;
- industrial management, managerial and financial accounting, analysis and auditing;
- control over mineral quality at mining enterprises.

I think that the conference is aimed at exchanging efficient world practices and knowledge in the field of scientific support of mining, economics and management. I hope that the results obtained will be valuable for all conference participants, especially Ukrainian ones, and suggested recommendations will be implemented.

I am sure that all the conference participants – talented researchers – will achieve great results and contribute much to development of the mining science and implementation of innovative ideas in applied developments. New discoveries and breakthroughs are awaiting you.

I wish all the participants rewarding work, good discussion and interaction, new interesting projects, ideas and solutions and every success in search for the most efficient designs.

May good success attend you in your research!

Yours sincerely, Doctor of Engineering, Professor,
President of the Academy
of Mining Sciences of Ukraine Yuriy Vilkul

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APPLICATION OF PROGRESSIVE COAL PRODUCTION TECHNOLOGIES IN THE LVIV-VOLYNSK COAL BASIN

In connection with the situation that prevailed in Eastern regions of Ukraine, where the vast amount of coal reserves of the country, the issue of developing advanced methods of thin and ultrathin layers of Lviv-Volyn Coal Basin is certainly important.

Lviv-Volyn Coal basin is located in northwestern Ukraine in the upper reaches of the Western Bug is the south-eastern part of the Lu-blin Basin in Poland. Its area - 1,400 square kilometers (nearly pulled meridionally 190 km with an average width of approximately 60 km). In the coal-bearing formations in the basin developed 199 coal seams and layers, including 99 seams, of which 30 - labor power (0,6 m) and 17 - industrial value

For component composition of gases in coal-bearing strata dis-tin-guish three zones: methane-nitrogen to a depth of 400 m, nitro-gen-methane to a depth of 400-500 m and methane - less than 500 m. The methane gas zone dominated by methane and its homologues whose content ranges from 50.2 to 98.9%. In addition to methane, the gas mixture consists of nitrogen (0,21-46,5%), carbon dioxide (5.4%), oxygen (0,23-4,02%) and hydrogen (0,01-3,21 %), accompanied by a slight capable of migration component of helium (0,08-0,6%). Heavy hydrocarbons are especially significant with ethane content (0,076-0,46%), butane (0,396-0,834%), pentane (0,195-0,849%).

Given the conditions of occurrence and characteristics of coal seams of the Lviv-Volyn Coal Basin, with modern logistics mining basin, significant reserves of coal are off-balance sheet, that is not economically feasible to develop in today's time.

Therefore, finding promising methods of coal development is very important in terms of future energy independence of Western Ukraine.

By analyzing national and world scientific literature, there are two main perspective methods.

One of them is the introduction and development of developing coal seams using plow systems.

Compared to the harvester, plow systems main ad-vantages are more simple and secure their maintenance and operation, higher quality coal produced by using the principle of cleavage.

The second method is the underground gasification of coal (UGC) - obtaining energy from coal at the place of its occurrence, namely, its transfer to high-calorie gases, which is more calorific in value than coal.

The disadvantages of UGC include the lack of scrutiny and the considerable cost of building a full-fledged production, and the advantages are the ability, in the long term, to reduce the domestic use of expensive natural gas and the ability to process off-balance coal reserves.

However, all this requires further scientific research with national or regional support, as the experience of home-grown researchers is not sufficient for the wider application of UGC on an industrial scale.

Thus, the introduction of new technologies at the mines of the Lviv-Volyn Basin will help to increase productivity, production volumes, improve conditions and safety.

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BRIEF STUDY OF THE VOLUMETRIC DEFORMATIONS AND DILATANCY OF SOILS AND ROCKS AS A PRECURSORS OF FAILURE

Keywords: rock mechanics, laboratory tests of rocks, uniaxial and triaxial compression tests of rocks, volumetric strain, dilatancy, damage of rocks, failure of rocks

Brace et al. (1966, also citing the research of Bridgman (1949), Borg et al. (1960) and Handin et al. (1963)) write about crystalline rocks and stress causing cracking, after crossing the threshold of elastic behaviour and about the dilatancy phenomenon.

For granular materials slippage along irregular aggregate surfaces, even at low stress levels, is accompanied by increasing in volume (compare with the vertical cracks - friction phenomenon described by Jones (1952), Tocher (1957) and Matsushima (1960)).

Dilatancy is a phenomenon preceding the brittle failure of rock material (Kwaśniewski, 2007). It may forerun, in a macro scale, the phenomena of rock bursts, mining tremors and earthquakes. That is why the dilatancy effect has been- and still is being studied in Poland and in the world (e.g. Kwaśniewski, 1975-2014; Mogi, 1971-2006; Takahashi, 1994-still; Cieślik, 2018, still).

In 1964, Brace proposed ranges of behaviour of rocks undergoing a brittle failure: I and II - elastic behaviour, III - loosening of grain boundaries, IV - propagation and formation of new cracks.

In 1975, Kwaśniewski proposed several thresholds and boundaries of linear and non-linear behaviour on the stress-strain characteristics for rocks under uniaxial and triaxial compression tests. He also proposed the classification of $\square_D$ dilatation thresholds: low, moderate and high, and as well the intervals of volumetric deformations (1997). Together with Mogi (2000) they described the process of volumetric deformation of rock as a brittle material (Fig. 1).

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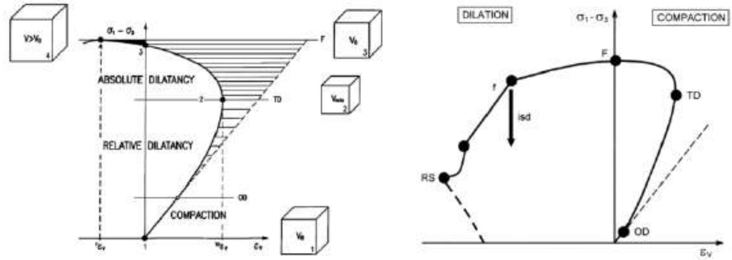

Fig. 1. Relative dilatancy threshold *OD*, absolute dilatancy *TD*, *F* - failure, *f* - damage in the post-critical stadium, *isd* - immediate stress decrease, *RS* - residual stress (Kwaśniewski, 1997; Kwaśniewski and Mogi, 2000)

Attempts are still being made to build a theoretical model that would describe both volume increase in the post-critical and compaction in the pre-critical parts of stress-strain characteristics. Such models have already been proposed, incl. by Vermeer and de Borst (1984) and Yuan and Harrison (2004).

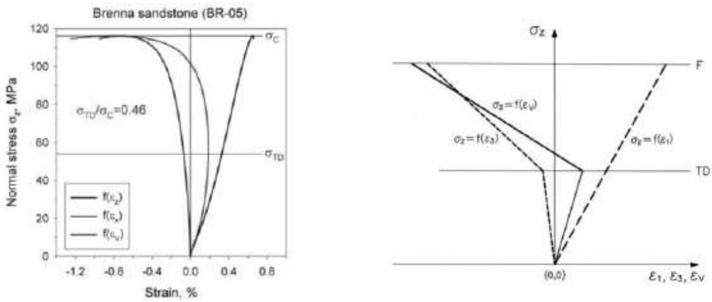

Fig. 2. Stress-strain characteristics of a medium-grained *Brenna* sandstone and simplified, piecewise linear characteristics representing behaviour of rock in the pre-failure domain (Kwaśniewski and Rodríguez-Oitabén, 2012)

Kwaśniewski· and Rodríguez-Oitabén (2012) proposed a model with simplified, piecewise linear stress-strain characteristics (Fig. 2).

· Professor Marek Kwaśniewski (1947-2014), the head of the Rock Mechanics Laboratory (1975-2014; Silesian University of Technology), President of the Polish National Group of the International Society of Rock Mechanics. For many years, he has conducted researches with Kiyoo Mogi and Manabu Takahashi (AIST, Geological Survey of Japan), including on volumetric deformation and rock dilatancy. His last monograph on volumetric strain and dilatancy of rocks that contained many materials most likely unpublished elsewhere ever then has never been published because of his death that precluded it to be finished and be made publicly known.

The approach allowed to obtain dilatancy of the characteristics and better fit the straight lines to the curves (Tab. 1).

Tab. 1

Uniaxial compression test results for Brenna sandstone, ${}^{p}\psi$ - dilatancy angle for plastic strains (Kwaśniewski and Rodríguez-Oitabén, 2012)

Parameter	σ_C MPa	σ_{TD}/σ_C, -	${}^{F}\varepsilon_{VD}$ %	ν -	ψ°	${}^{p}\psi$°
Mean value	109.5	0.50	1.38	0.20	32.9	65.0

Tab. 2

Values of the ψ dilatancy angle for different geomaterials (Vermeer and de Borst, 1984)

Normally consolidated clay	Loose sand	Concrete	Dense sand	Marble
0°	<10°	13°	15°	12°÷20°

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DEVELOPMENT OF MINE DEGASSING FOR THE USE OF CAPTAINABLE METHANE AND REDUCTION OF METHANE SAFETY

The development of methods for coalmine methane recovering, suitable for use in the national economy, should occur in two main directions:

- improvement of existing and the creation of new ways of degassing coal mines, significantly reducing the volume of methane emissions in mine workings and thereby ensuring the safety of work in methane copious mines, and at the same time increasing the productivity of sinking and mining machines and mechanisms by gas emission factor;

- the creation of highly effective methods for the degassing of coal seams that are not related to mining operations - the degassing of coal seams using vertically directional wells drilled from the surface using artificial methods to increase the natural gas permeability of coal seams (hydraulic fracturing, special processing methods, etc.) [1], and without the use of these methods.

Obtaining natural gas suitable for utilization during the development of the first direction is a subordinate (indirect) effect of degassing, the main purpose of which is to ensure the safety of works on methane release factor. The use of these methods, schemes and options for degassing is especially important in the complex degassing of mining sites with exceptionally high methane copiousness (30-50 m3/min). First, the indicated methods of degassing include various options for the suction of methane-air mixtures from the worked-out area of the existing sites, when methane is supplied to the site from reservoirs that are located in the close proximity of the developed seam and are associated with mining excavations with operational cracks.

The second direction in the development of methods for the degassing of methane producing coal deposits is, first of all, a very promising method for producing coal mine methane (gas with a high, stable methane content) for a wide range of methods of disposal in the national economy. At the same time, the application of the shaftless degassing methods discussed above contributes to a decrease in the natural content of methane in coal and definitely also helps to increase the safety of gas mines in the future.

Vertically directed wells can be used to produce gas containing 75-97% methane under the following conditions:

-for degassing of unloaded coal seams without and using special methods to increase the gas permeability of the seams;

-for degassing of undermined coal producing sections in conjunction with underground mining.

The experience of seams degassing by drillings from the surface showed [1] that at a well depth of 600-900 m, the caption gas contains at least 70% methane (when choosing the optimal degassing parameters, the methane content is 95-97%), and the well delivery, depending on the volume of degassed coal and the value of natural methane content of coal, is in the range from 1000 to 16000 m^3 /day of methane. The amount of methane captured by this method may reach 96-135 billion m^3 / year in the coming years [1].

The directions for improving the method of degassing undermined coal producing sections by directed vertically-deviated (horizontal) wells are as follows.

In order to prepare the gas producing sections containing a measure or measures of gently sloping remote seams in advance:

The development of measure of methane producing coal seams begins with a subsurface seam of measure with preliminary drilling from the surface of directed vertically-deviated degassing wells (VDDW) crossing the undermined (overlying) seams in the area of their future part-time work. In some cases, with relatively weak lateral rocks, VDDW should be drilled with the intersection of the developed seam in front of its mining face. The degassing well is cased for the entire length, except for the lower 25-35 m, which are cased with perforated pipes or not cased (with solid rocks).

Vertical-deviated directional wells are drilled to undermined seams (rock layers) and subsequently from one blast-hole along the developed massif (coal seams or parting rocks), the inclined-horizontal part of two to four degassing wells is drilled, thereby increasing the contact area of gas producing seams within the floor. This version of such method of degassing, in addition to increasing the rate of natural gas suitable for utilization, allows more deeply degassing the overlying seams of the measure, including more reliably preventing the possibility of sudden gas emissions into the mine workings.

In the descending order of mining methane-producing coal seams of the measure, VDDW should be drilled in such a way that the inclined (horizontal) part of the well passes through the produced seam in the active overworking zone or by bedding between the developed and degassed seams.

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REDUCTION OF ORE LOSSES AND DILUTION IN MINING CONTIGUOUS ORE BODIES

The problem of improving quality and quantity indices of mining Kryvyi Rih iron ores has been an urgent problem, especially in conditions of escalating competitive struggle of countries exporting iron ore materials to the European market. China, Australia, Brazil, India and Russia are major competitors of Ukraine in this regard. There are several reasons for that. First of all, it is the mining depth of over 1500 m, great rock pressure and relatively low iron content in the mined ore. Ukraine produces sintering ore of 58-60% Fe, while its competitors - 60-63% Fe. Therefore, search for new mining technologies considering geological conditions of Kryvyi Rih iron ore basin is of primary importance as it is aimed at improving quality indices of the ore mined.

Kryvyi Rih iron ore basin comprises more than 300 column and plate-forming deposits with the depth reaching 3000 m. The iron content varies from 58% to 64%.

The structure of occurrence of ore bodies is formed by contiguous ore bodies and rock interlayers.

Considering the necessity of mining contiguous ore bodies, the authors suggest the technology of steep deposit mining of contiguous ore bodies, which is characterized by maximum application of mining and geological characteristics of an ore deposit and separate technological factors to gaining ultimate ore recovery.

The technology implies disintegration of an ore body into columns with the height equaling in vertical thickness of ore bodies and the rock interlayer. The ore massif is drilled within a column considering cutting of the rock interlayer without destroying it and cutting a rock pillar in the hanging wall of the vertical column. Some of the boreholes located under upper and lower rock interlayers are left uncharged to control movement of rock interlayers in ore drawing.

To avoid early dilution, the ore is drawn from craters away from the rock. This time table of ore drawing allows irregular movement of the lower rock interlayer and its dumping onto the given angle exceeding that of the natural slope of broken ore. The upper rock interlayer in ore drawing is based on the upper ore pillar and becomes horizontal. The volume of the ore drawn determines the moment when the upper rock interlayer becomes horizontal. After that, the previously drilled boreholes are charged and the upper ore pillar is destroyed, this providing regular horizontal sinking of the interlayer where ore and rocks interact avoiding dilution caused by overlying rocks. After the lower rock interlayer takes a certain position and reaches drawing craters, the lower ore pillar is destroyed and the ore is drawn to obtain substandard ore. The described algorithm enable the maximum ore recovery with insignificant dilution.

Conducted laboratory experiments of the technology confirm possible control over upper and lower rock interlayers in ore drawing. Comparative calculations of the suggested technology and the one without upper rock overlapping and leaving the lower pillar reveal reduction of ore losses by 3.8 times, dilution by 5.8% and increase of ore recovery ratio by 9%.

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CHOICE OF AN INDEPENDENT ROCK BOLT SUPPORT FOR CROSS-CUT FACE

An independent rock bolt support has been known in Polish mining since 1917, but it began to develop after World War II. The rock bolt support is an alternative to the arch yielding support, which dominatess in Polish hard coal mining. Using this type of support to secure roadways compared to an arch yielding support is primarily: a larger useful surface of the excavation, the possibility of full mechanization of the bolting process, improvement of working conditions and measurable economic benefits.The selection of the rock bolt support in underground hard coal mining is made by the Head of Mining Department on the basis of a project developed by an expert, including in particular: recognition of geological and mining conditions, in situ test locations, range of rock recognition and results of rock mass tests, selection of installing bolt, arrangement of stratification indicators, organization of supervision and control. Requirements for the used of rock bolt support in underground hard coal mining are: Roof rocks have a weighted average uniaxial compressive strength (Cs), tested for a 3 thick rock package, not less than: 15*a* - for layers of slab structure and measured fracture of roof rocks (RQD) of not less than 20%; 10*a* - for layers of massive structure and measured fracturing of roof rocks (RQD) of not less than 40%. The rock mass is dry or not soaking up and the coefficient of slakeability (r) is not less than 0.8. It is used to support roadway and room excavations with a cross-sectional area not exceeding $30m^2$ and the excavation width not exceeding 7. In addition, the bolt rods should be made of steel with a tensile strength Rr≥500 MPa. Bending strength Rg≥300 MPa, while shear strength Rt≥250Pa. Load capacity for resin, adhesive and reinforced concrete rock boltss should be at least 120 kN. The selection of bolt support for preparatory and exploitation excavations is a very complicated issue due to the multitude of geological, mining, technological and organizational factors. In the case of using an independent bolt support in a cross-cut face, the vertical extent of the rock destruction zone in the roof, the length and spacing of the support should be determined by means of analytical, numerical methods and on the basis of in situ tests. In addition, it is necessary to determine the load that the bolt support is able to take over under static and dynamic loads. In this case, the best simulation of mine conditions are laboratory tests that can be used to model the bolting parameters such as: choice of the bolt hole diameter to the rod diameter, length of embedment and fixing mechanism: resin, expansion or friction.

In the article the execution technology of an independent rock bolt support for cross-cut face was presented. In order to drive the rise gallery (upcut) of face, the JOY 12CM30 mining and bolting roadheader will be used. It is a mining machine with a driving head for one cut across the entire width. In addition, it includes bolters fitted at the working platforms, which allow for simultaneous mining and bolting of the ceiling and side walls. At the beginning, the mining machine will be extended to the roof and floor through the frame mechanism. This will allow it to stabilize so that it does not move during mining. Then the driving of the excavation begins. The costruction of the roadheader allows the organ to move up and down over the entire width of the excavation. Then the falling rock will be loaded with the help of a grab loader located on the roadheader table. Once cut, the mining machine can drill 1 meter of the excavation. During mining operations are also performed activities related to rock mass bolting because in this method the most important thing is to immediately pass to the bolting of roof and side walls after cutting the next web. This prevents stratification of the rock layers above the excavation. One work cycle of the mining machine ends after the bolts have been built into the rock mass. Then the roadheader is forwarded, stabilized and the excavation re-driving. Bolt holes are drilled in accordance with the bolt metrics drawn up for the rise gallery (upcut) of faceof for the coal seam 505/1. Resin charges are placed in hole, introducing them pneumatically to the bottom of the bolt hole. Then the bolt's translational-rotation movement mixes the adhesive components. After installing the bolt in the hole, the M24 nut is screwed with a torque of not less than 250 kN and to give the bolt a pre-tension of not less than 30 kN. In the roof of raise gallery (upcut) a SP-270 plate girder and a Pollux mesh made of basalt fiber enriched with admixtures ensuring greater strength will be built. The plate girder is made of 3mm sheet metal, 6m long and 320 m wide. Adding a mesh and a roof-bar will ensure the safety of the crew working there, against falling rock fragments and will improve rock mass stability. According to the bolting metrics, in the roof 5 bolts in a row, with a length 2.5 and 21.7 diameter will be installed. There will be a distance of 0.8 between the bolts and spacing between the rows of bolts will be 1.2. The two outer bolts will be built at an angle of 20° towards the side walls. The bolts used have a high tensile strength of 770 MPa. In addition, on both sides, two side walls bolts KWSZ-1 type will be installed. It is a bolt made of polyester resin with a diameter of 18 mm.

Basalt mesh will be applied to the lagging of the side walls. The use of an independent rock bolt support for a cross-cut face can contribute to a reduction in the excavation time, reduction of operating costs and to an increase in driving progress, of course, if geological conditions allow its use. In addition, the use of bolt support can help to compete with other global coal producers.

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ABOUT RECEPTIONS OF INTENSIFICATION OF PROCESSES ORE GRINDING

Today in the world there is a steady growth of science and technology, education and culture, an informational high-tech society is forming, where intellectual capital plays a huge role.

The disintegration of hard ores is the result of processes occurring at various hierarchical levels - from the decay of interatomic bonds to the formation of discontinuities the continuity. Therefore, processes and schemes for the disintegration of minerals should be built so that with minimal losses to extract useful minerals in subsequent redistribution.

Preparation for the selective separation of minerals should begin at the stage of explosive breaking of mineral raw materials. It is necessary to use such explosive blasting systems that lay in the ore pieces an extensive network of embryonic cracks and provide softening of mineral aggregates along the planes of mineral aggregation. It is necessary to use such explosive blasting systems that lay in the ore pieces a branched network of embryonic cracks and provide softening of mineral aggregates along the planes of intergrowth of minerals This allows to save energy in the processes of ball grinding and increase the productivity of mills on concentrating mills.

The use of plasma heat generators in the mining industry during the breaking of hard ores allows the thermal method to create charging cavities of a volume larger than the volume of the explosive in it placed. The effect of the explosion is enhanced by the fact that the explosive gases transfer the energy of the explosion to a much larger surface surrounding the charge [1].

The IGTM NAS of Ukraine has developed a combined technology for drilling blocks of strong ores in underground mines.

When creating chamber the energy that is radiated into the rock mass changes. This means that it is possible to significantly change the energy radiated into the mass of ore, compared with cylindrical discharges placed in wells drilled by traditional mechanical machines. The authors [2] substantiated a theoretical estimate of the change in the specific energy of an explosion emitted into the rock mass when creating an expansion chamber in cylindrical wells previously drilled mechanically.

Destruction upon impact of pieces of ore against an obstruction occurs along the planes of intergrowth of minerals with the formation of a wide network of micro cracks. The specific energy consumption for ore breaking is 2-5 kWh / t and for grinding by standard methods - from 20 to 40 kWh/t. For fine grinding of minerals (smaller than 40 microns) already requires more than 100-150 kWh/t. Therefore, it is necessary to grind the ore thoroughly during blasting before loading it into crushers and mills.

The study of the process of charge formation and the method of impact on the mass of ore during the explosive breaking of magnetite quartzite in the Ordzhonikidze mine showed that the main element of the complex work is the preparation of the charging chamber. The quality of crushing of strong ores beaten by explosive charges in wells with a diameter of 243 mm, despite a lower specific consumption of explosives on breaking, is higher than in bundles of wells with a diameter of 105 mm. The average specific consumption of explosives for secondary crushing is 2-2.5 times less.

For the purpose of the experiment, in block 7 of the axis between the horizons 387 and 447 m of the Ordzhonikidze mine, magnetite quartzite were breaking with a hardness of *f*=15-18 according to Protodyakonov with advance of two stope [3]. The first - from 6 wells of variable cross-section expanded with a plasma installation to a diameter of 350-400 mm, and the second - from 12 wells with a diameter of 243 mm (drilled by the BSh-200 C machine). The drill pattern of the arrangement of charges is 7.5 m between stope and 6 m between pair of holes of diameter 243. The granulometric composition of the beaten ore turned out to be much better than with borehole blasting with a diameter of 105 mm, as well as vertical concentrated charges. This made it possible to increase the intensity of the release of blasted mass in draw hole and the productivity of delivery vehicles to the receiving level.

Industrial experiments have revealed effects such as the effect loss of strength, the effect of relaxation of rock strength after preliminary thermal loading. These effects not justified theoretically and today in practice have not found their worthy application, although they can be used in energy saving technology.

It is generally accepted that explosive destruction of ore mass is influenced by such characteristics as strength (compression, tension, shear), compressibility, porosity, viscosity, graininess, jointing, water cut, but the metasomatic state of rocks is not always taken into account. Explosives cannot be calculated taking into account all of the above characteristics, since for most of them quantitative values are unknown. Therefore, the power intensive of technological processes in the extraction of hard ores and their enrichment will remain at the same level for a long time.

The creation of chamber by the plasma method makes it possible to strength loss the mineral composition and increase the system of cracks. This not only increases the efficiency of the ore mass explosion, but also increases the role of mineral discovery in enrichment [4].

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USE OF THERMAL AND MAGNETIC DEVICES FOR PREVENTION OF ASPHALTENE, RESIN, AND WAX DEPOSITS ON OIL EQUIPMENT SURFACES

Extraction of liquid hydrocarbons is often complicated by the presence of asphaltene or resin and wax deposits in them, which causes formation of deposits inside the tubing string. Over time, these deposits - hydrocarbons methane from $C16H34$ to $C64H130$, silica gel resins, asphalt-resinous compounds, hydrates, and others - worsen, and, in some cases, make it impossible to extract liquid products, since they gradually obturate the passage section of the tubing string.

Insufficient presence of complexes for the prevention of ARWD on oil and gas equipment market, and the tendency to implement combined technological complexes, which are made on the basis of magneto-electrical devices (thermal-magnetic dewaxing units), and also the design of a multi-level complex for deposits prevention should allow placement of thermal-magnetic dewaxing complexes in the zone with a higher level of such deposits.

Crude oil is a complex chemical composition of components which, depending on the structure and the external environment, may be in different states of aggregation. Temperature reduction causes a change in the physical state of the components, leading to the formation of paraffin crystallization centers and their growth [1].

The intensity of the ARWD formation depends on the predominance of one or more factors that can vary in time and depth, so the number and nature of the deposits are not constant.

the most promising means of ARWD removal among the existing ones are magnetic devices. They do not change the chemical composition of the formation fluid, are not harmful to the environment, in most cases are installed at an arbitrary interval of tubing string pulling unit and are efficient.

To use magnetic devices, it is necessary to ensure the following conditions: gas factor (20-300 m^3/m^3), the presence of the micro-impurities in the form of iron ferromagnetic particles in the wells, the content of asphaltene and resins not less than the content of wax in oil, etc.

Laboratory studies have shown that the rate of wax formation affects allocation process and behavior of gas bubbles in the flow of mixture. It is known that gas bubbles can float suspended wax particles. When bubbles contact the tubes surface, wax particles come in contact with the wall and deposit on it.

On the following, the process of wax deposition increases because of its hydrophobicity. On the wall of the tube, is formed a layer of wax crystals and bubbles in the gas. The smaller gas-saturated layer, the greater density it has. Therefore, denser sediments are formed at the bottom of the lifting pipes where gas bubbles are small and have greater strength adhesion to wax crystals and tube walls [1-2].

The intensity of ARWD formation largely depends on the rate of fluid flow. At low flow rates, the formation of ARWD is quite slow. With speed increase (at transition to turbulent flow regime) deposits intensity initially increases. Further increase of liquid-gas mixture speed (LGM) leads to decrease of ARDW intensity, as the high rate of fluid flow allows wax to keep the crystals in suspension state and take them out of the well. Furthermore, the flow tears a part of deposits from the walls of the pipes, which explains the decrease of deposits in the range of 0-50 m from the wellhead. At high speeds, the flow of the mixture cools slower than at lower ones. Likewise, at low speeds, the formation of ARPD slows.

According to the results of industrial enterprises research, the ARWD appear at intervals of 500-900 m (in some cases, from 1200 m), which is explained by positive conditions: the temperature of the fluid decreases to 17-20°C, degassing within this interval, decrease in pressure, etc. The depth of the deposits layer on the inner walls of the tubing can reach 30 mm and more – until the closure of the passage section of the tubing string. At this interval, there is a need for the use of equipment to prevent this situation or deposition [1-3].

Application of a magnetic device allowed increase in the average repair time of wells, complicated by the formation of emulsions and ARWD, on average by 2 times. The introduction of a magnetic device in wells, complicated by the formation of ARWD, has allowed doubling the overhaul period during the chemical treatment of wells.

The use of thermal-magnetic dewaxing units is aimed at increasing the overhaul period of wells due to the action of the directed magnetic field and thermal energy. The mechanism of thermal-magnetic dewaxing device action is directed to the change in the viscosity of the liquid passing through the device.

The use of thermal-magnetic dewaxing devices can be effective at the operation of wells with deep-well, centrifugal and diaphragm pumps, as well as on oil pipelines.

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INTENSIFICATION OF UNDERGROUND URANIUM LEACHING USING HYDROGEN PEROXIDE

For uranium mining, the method of underground-borehole leaching is used. This method is conducted by leaching solution (sulfuric acid or carbonate solution) is fed into the poor ore layer, which, seeping through the ore layer, reveals uranium minerals. The resulting uranium solution rises to the surface through a pumping well using a submersible pump.

There are several schemes of location of injection and pumping wells. In the United States, hexagonal and square schemes are used, when the pumping well is located in the center, and the injection wells are at the vertices of a regular hexagon or square. In the CIS countries, in addition to the above, a rectangular scheme of wells is also used, when the rows of pumping and injection wells alternate.

Initially, when the unit is started, water is pumped into the injection well, which displaces groundwater, then acidification is carried out. Acidification of the mining block consists in the supply of acidic solutions with the maximum amount of acid and oxidants to create an appropriate geochemical environment in the ore body (pH=2.5-3). Acidification of the block lasts 20-60 days. The acidification is fed 2.4 kg of H_2SO_4 (sulfuric acid) per 1 ton of ore mass or L:S=0.18-0.25 m^3/t. The end of the acidification is determined by the achievement of the industrial concentration of uranium in the production solution (more than 30 mg/l). Active leaching is carried out after acidification of the block in the same hydrodynamic mode, but with a gradual decrease in the concentration of acid in the leaching solutions. As additional reagents, oxidizers are used: nitric acid, pyrolusite, air oxygen and others. The choice of uranium ore opening method is determined by the mineralogical composition of the ore, and both uranium minerals and minerals of the host rocks must be taken into account. Acid leaching is more often used, it is applicable to almost all ores except for ores with a high content of carbonates, the presence of which in the ore sharply increases the consumption of acid.

The fact is that the growing demand for uranium is limited by the number of reserves suitable for profitable mining, it is necessary to rationally and fully utilize existing proven reserves and resources.

To do this, new techniques and technologies should be used, as well as additional reagents, the costs of which will be lower from the traditional method of PSV. In this regard, the search for mechanisms of expansion and full use of the resource base of uranium mining enterprises of Kazakhstan is very relevant.

This article proposes the use of hydrogen peroxide as an additional reagent. Hydrogen peroxide, H_2O_2 - the simplest representative of peroxides. Colorless liquid with a" metallic " taste in appearance similar to water, infinitely soluble in water, alcohol and ether. Concentrated aqueous solutions are explosive.

The advantages of hydrogen peroxide are that it is one of the most powerful oxidizers, stronger than chlorine or potassium permanganate. Hydrogen peroxide is also produced by sunlight on water, which is a natural way to clean the environment. Therefore, hydrogen peroxide has no problems with its decomposition products, which are present in other oxidants. Hydrogen peroxide mixes with water in any concentration and therefore the problem of safe handling is easily solved by diluting it.[1] Due to its strong oxidizing properties, hydrogen peroxide has found wide application in everyday life and in industry. It is used in analytical chemistry, as a foaming agent in the production of porous materials, in the production of disinfectants and bleaching agents. And also in oil production to increase oil recovery from wells.[2]

Hydrogen peroxide is often used in metallurgical processes for ore leaching, concentrate preparation, or wastewater treatment. The production of gold and uranium are important examples of the application of hydrogen peroxide in the mining industry. Depending on the composition of the ore, as well as on the leaching conditions, the use of hydrogen peroxide leads to savings of the leaching solution, acids, as well as improves the efficiency of the process as a whole.

Depending on the leaching conditions and the concentration of the solid phase, the addition of hydrogen peroxide leads to an increase in the total oxygen concentration in the system, as well as promotes deeper leaching by direct oxidation.[3]

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INSTALLATION TECHNOLOGY OF ROCK BOLT SUPPORT IN POLISH ORE MINING

In underground copper ore mines „Rudna", „Polkowice-Sieroszowice" and „Lubin" belonging to KGHM Polska Miedź SA, as well as one zinc and lead mine „Olkusz-Pomorzany" belonging to ZGH Bolesław, rock bolt support is a basic way of securing roof stability and is treated as a kind of technological treatment strengthening the rock mass above or around excavations. In order to use bolt support in Polish ore mining, it must meet the requirements of standards and regulations. In particular: roof rocks have a weighted average strength: uniaxial compression (Cs), tested for a package of rocks with a thickness equal to the width of the designed excavation - not less than 15 MPa and tensile strength (Ts) - not less than 2 MPa. In addition, the rocks have in the zone intended for bolting in mining plants extracting copper ore have an average rock quality designation of not less than 20mm and do not show a natural tendency to break away. However, breccia nests found in mining plants extracting zinc and lead ores do not tend to fall off. The technical design of the bolt support, includes in particular: roof classes determined on the basis of geomechanical tests carried out by an expert, geomechanical tests of rock properties and supplementary tests, selection of the rock support, support of the side walls, protection of the excavation roof during the liquidation phase, method of installing the rock bolt, organization of supervision and control. One of the most important parameters of the bolt support is its load capacity, which for resin, adhesive and reinforced concrete bolts should be at least 100 kN for copper ore mines, and 90 kN for zinc and lead mines. However, the load capacity for expansion and wedge anchors should be at least 100 kN for copper ore mines and 90 kN for zinc and lead mines. In the case of friction bolts, the minimum load capacity is at least 80 kN.

The mechanism of cooperation of the bolt support with the rock mass is based on the principle of: beam building, suspending a weak layer to stronger layers, transferring tensile stress through the bolts, and limiting the displacement of rock mass layers to the excavation. Currently, there are basically three main types of bolting technology: installation of point, full column, fast and slow resin, mechanical and cable-binder bolts. Rock bolts up to 2.6 m long are considered as short. Above this length, long bolts are distinguished. In the „Olkusz-Pomorzany" mine, only 1.6 m long ribbed bolts are used. The bolts are installed at a spacing of 1×1 m. Bolts with 22 mm and 24 mm rod diameter (rod with a wound rod) are embedde along the entire length using resin cartridges. The 22 mm and 24 mm diameter bolt is installed in 32 mm and 35 mm diameter holes. For KGHM mines in the LGOM area, the range of rock bolt diversity is much larger. Short bolts are installed on 1.2 m, 1.6 m, 1.8 m, 2.2 m and 2.6 m lengths using resin cartridges or expansion head with single or double spreader. For resin bolts, 22 mm diameter rods are used, which are installed in 28 mm diameter holes. For example, for bolt with a length of 1.8 m, four resin cartridges with a diameter of 24 mm and a length of 450 mm are used, the first one being fast setting: 30 seconds and the other thres being 2 minutes slow setting. In the case of mechanical (expanison) bolts, smooth 18.3 mm diameter rods are used, which on one side have a metric thread and on the roof side have either a metric thread or swelling. The expansion heads have a diameter of 25.4 mm, 28.6 mm and 36 mm. For expansion boltss, the difference between the head diameter and the bolt hole diameter should not be greater than 2mm. It should be mentioned that in the 90s of the last century in the

„Rudna” mine there were attempts to use reinforcement bolts that were installed with cement binder. The cement binder bolting assembly consisted of a TP8 hydraulic pump with a capacity of 15 liters/minute and pressure of 210 bar and a cement mixer TM120 type with a capacity of 200 kg. The cement required for installing about 19 bolts with 1.8 m long and with a diameter of 20 mm was obtained from 50 kg of cement. The average number of installed bolts was 40-50 bolts/shift. In the tests Portland cement, class 350 was used. The water-cement ratio was 3:7 (by volume). The rods were installed in 28 mm diameter holes. At present, the Polish ore mines do not use a short bolt support installed on cement binder in mining excavations due to the setting time, and thus a longer period for the bolt to obtain the required load capacity of 100 kN. Certain possibilities exist when using a special binder based on quick-setting cements, which will undoubtedly increase support costs. The compromise solution can be obtained by using mixed embedment, by using one resin cartridge introduced into the bottom of the hole, which will reinforce the rod, while the remaining part will be filled with cement binder. Long bolting at KGHM Polska Miedź S.A. is intended to strengthen the existing excavation support, in particular crossing with increased area, excavations made in disturbed rocks of reduced strength and entering or located in the zones of exploitation impact, as well as prone to expanding. Long bolting can be divided into two technologies, namely. Mechanical bolts, which consist of smooth rods, usually 1.7 m long, screwed together using sleeves. The most commonly used bolts are 3.5m and 5.2 m. This technology results in an immediate bolting effect. The second technology is based on the use of cement binder. The most commonly used cable length are 5 m, 6m and 7 m lengths. Cable-binder with a diameter of 15.5 mm and 18 mm are installed in holes with a diameter of 51 mm. In installing technology it is distinguished: first, the hole is filled with cement binder from the bottom of the hole and then a cable is introduced or first a cable armed with a venting hose is introduced into the hole and then the cement binder is injected to the bottom of the hole. The self-propelled bolting car for the mechanical installation of cable is equipped with a drilling turret and a platform with an aggregate for forcing the binder into the holes. To vent the hole, a plastic tube with a diameter of 6mm or 8mm and a length at least equal to the length of the cable is used. The binder is a mixture of water and Portland cement, minimum class 25. The volume ratio of the components is: 3 volumes of water and 7 volumes of cement. The weight ratio of ingredients i.e. w/c=0.3. The binder should be characterized by strength parameters guaranteeing the load bearing capacity of the cable min. 150 kN after a period of no more than 24 hours. The minimum consumption of binder is 1.5 liters per linear meter of hole, and the average consumption of cement per 1 hole 7 m is about 26 kg. Cementation is carried out using a compressed air powered device consisting of a model 5290T pump and 2585T mixer. The device is intended mainly for water-cement binders for which low flows but relatively high pressures are used. Device 5290/2585T enables the preparation and pressing of a water-cement mixture with a weight ratio of not less than 0.28. For a binder with a weight ratio of 0.3, at a supply air pressure of 0.7 MPa, the device capacity is 12 liters/minute and pressure max. 2.5 MPa. The time for pressing the binder into the 7 m hole is about 4 minutes.

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APPLICATION OF MULTI-CRITERIA DECISION-MAKING IN SELECTION OF AN UNDERGROUND MINING METHOD

In the paper we show the procedure for solving problems that belong to the field of underground mineral deposits exploitation when valuing and selecting an optimal mining method and when applying multi-criterion optimization. Classical methods of multi-criterion optimization and fuzzy optimization have been applied. Fuzzy logic in the process of decision making is especially favorable for the application in cases when we do not have at hand sufficient number of information about the system that is being studied, also when we appreciate experience and knowledge of experts. In accordance with the defined goal, multi-criterion decision making has been applied in order to solve the problem of rating of certain mining methods as regards the set of analyzed criteria, as well as choice of optimal solutions. In the paper we have analyzed mining methods that represent alternative solutions. On the other hand we have defined and analyzed different criteria that influence the choice of optimal decision. There has been done mutual rating of analyzed criteria, as well as rating of alternative

solutions as regards the given criteria. By applying the mathematical model the final decision was made as regards the optimal mining method. In the paper we have shown the applied methodology.
Key words: optimization, mining, fuzzy logic, expert judgement

1. Introduction

Taking into consideration the complexity of the problems that are encountered and solved when mining, it is clear that projects are unavoidably of the multidimensional, and frequently of the controversial nature. Multidimensional nature of mining projects may be made exceptionally simple, and in a certain way it may be distorted if it is described and presented by way of a single criterion.

Theory of management suggests that the consequences of the managing actions always and explicitly aggregate into one criterion function. In case that criteria functions are of similar values relation of indifference is valid. But if there is even a slightest difference then the relation of preference is valid. Such an approach has been explained by the tendency to make comparisons simpler. However, practice refutes this idea.

Abandoning unconditional acceptance of such an approach brought about development of multi-criterial optimization, which in this paper represents the basis for the development of the model for decision support.

2. General terms on multi-criterion optimization (MCO)

Every model of multi-criterion optimization contains the following elements:

- more criteria (target function, criterion function) for decision making;
- more alternatives (solutions) for selection;
- selection process of one final solution.

There are 2 kinds of MCO problems if viewed from the point of description of the considered reality by means of a mathematical model multi-target decision making (MTD) or target programming as the subgroup of MCO, and multi-attributive decision making (MAD) or multi-criterion analysis.

It is customary that MTD problems are called well structured, and MAD problems poorly structured.

3. Fahp method application

The basic idea represented by the Fuzzy Analytical Hierarchy Process method is that it is applied in group decision making, however its application is possible in individual decision making as well. The basic steps to be taken when applying this method (irrespective of the coefficient criteria defining or alternative) are:

- collecting data from experts;
- application of FAHP for each expert separately;
- aggregation of received vector priorities.

Before the beginning of the method application the basic criteria on the basis of which the rating of risk is made are to be defined.

4. Conclusion

The aim of this paper is to analyze the sensitivity in decision-making for the selection of a mining method when using the FAHP model. The results indicate that the proposed FAHP decision-making model could be used for the selection of a mining method, as the uncertainty levels of the factor do not influence the final decision. It has been observed that the rank of the highest priority alternative never alters with either changes in the fuzzification factor or the decision-making attitude. It can be inferred from the results that the ranking of the most suitable alternative remains the same irrespective of the fuzzification factors and decision-making attitudes.

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TYPES OF GAS-DYNAMIC AND DYNAMIC PHENOMENA IN COAL MINES

Gas-dynamic and dynamic phenomena occurring in mines can be classified by energy sources, seam properties and external features.

Gas-dynamic phenomena include sudden emissions of coal and gas, sudden extrusion and precipitation (collapse) of coal, accompanied by increased gas evolution, emissions of rock and gas.

These phenomena occur in the mining faces of the capital, preparatory and treatment openings and represent a rapidly occurring destruction of the bottom-hole part of gas producing coal or rock mass under the influence of mining pressure, gas pressure and the dead weight of coal, accompanied by the rejection or collapse of coal (rocks) in mining and increased gas production, exceeding the usual one.

Coal and gas emissions occur on seams characterized by different angles of incidence, coal quality, the number of coal packs containing lateral rocks, etc. Emissions occur during blasting, drilling, mechanical and manual breaking of coal and rock, when controlled roofing by smooth lowering, full collapse, partial or full laying.

Types of gas-dynamic phenomena.

The sudden release of coal and gas is a complex gas-dynamic phenomenon, proceeding in several stages:

- brittle fracture of the stressed bottom-hole part of the seam caused by production processes;
- self-developing destruction of the coal mass with the formation of a characteristic cavity;
- removal of destroyed coal in the flow of expanding gas;
- constantly fading gas evolution from the exhaust cavity and the ejected coal.

The main signs of a sudden release of coal and gas are:

- refuse from the coalface to a distance exceeding the length of the possible placement of coal at an angle of original dip;
- increased compared with conventional gas generation in the mine;
- formation in the coal mass of a characteristic cavity;
- presence of fine coal dust on the slope of the discarded coal, on the lining, and sometimes in the entire rejected mass.

The consequence of sudden emissions of coal and gas is lining damage of the workings by discarded coal; damage and waste of equipment, lining breaking; test-blowing of a ventilation stream, etc.

Coal and gas emissions during blasting are characterized by the same signs as sudden emissions; however, they are not classified as sudden emissions, as they are caused by blasting operations that are carried out in a safe operation mode.

Sudden extrusion of coal with increased gas emission is a fast-flowing extraction of the bottom-hole part of the coal mass, caused by a change in its stress state, occurs without the release of coal by gas, and is accompanied by increased gas emission.

The sudden collapse (precipitation) of coal with increased gas emission is the destruction of the overhanging part of the coal mass due to the dead weight of coal, which occurs as a result of untimely or poor-quality fastening.

Rock and gas emissions during blasting occur in sandstones ahead of the mining face beyond the direct impact of IP and are characterized by rapidly developing destruction of the massif with the refuse of rock and gas emission.

Dynamic phenomena may include rock blows, shooting, sudden elevation of the soil, etc.

Mine bumps are a brittle destruction of the rock pillar or the edge of the seam, being in a stressed state [1]. Mine bumps usually occur on the seams with relatively high mechanical strength. Destruction occurs under the influence of an external impulse when coal in the seam is in an extremely stressed state. The refuse of coal during destruction occurs, as a rule, along a rectilinear trajectory, rejected coal is located at an angle of original dip.

Roof smashes occur during the development of coal seams with strong host rocks. Smashes occur due to the discharge of elastic energy accumulated in the roof rocks during bending. Observations show that bending under the influence of reference pressure begins from a distance of 5 - 10 m ahead of the mining face, and the total rock compression during bending is 1.37 mm / m. In cases where the roof rocks do not withstand bending stress, their destruction sets in, accompanied by smashes.

Coal bursting occurs due to the discharge of the elastic energy of rocks and coal in the process of stimulating the seam. It manifests itself in bouncing pieces of coal from the mining face with great speed, which can cause injury to workers.

Sudden elevations of the soil usually occur behind the preparatory or treatment mining face, as well as during the passage of mineshafts. The elevations are the result of the release of the elastic energy of strong rocks [1].

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TO THE QUESTION ABOUT NECESSITY OF THE IMPLEMENTATION OF THE MICROBIOLOGICAL ENHANCED OIL RECOVERY INTO THE OIL FIELDS IN THE CONDITIONS OF THE REPUBLIC OF KAZAKHSTAN

Due to the fact, that majority of Kazakhstan's liquid hydrocarbon deposits has been in operation for the long-time and some of them have already been mothballed due to low productivity, and some others are on the verge of the low economic production efficiency, there is an obvious necessity to increase the introduction of tertiary recovery's methods for developing of the oil reservoirs.

For example, in accordance with available data, the current oil recovery factor at the Tengiz field is 0.042 units. As is known, this deposit is confined to the southeastern part of the Caspian Basin, connected with a carbonate massif formed on the ancient Dodevonian base and is a high-amplitude isometric anticline trap with an area of 414 km^2. Three productive objects stand out in the section: I - Uppernevisei-Bashkir; II - Lowervisean-Tournaisian; III - Devonian. Water-oil contact is conditionally accepted at around 5450 m.

The average water cut of the production at the Kalamkas field for the same period was 83%, and the oil recovery factor was 19 %. This field, discovered in 1976, is confined to a slightly disturbed brachyanticline fold of latitudinal strike, within which gas content of six formations in the Neocomian, two in the Apt and seven gas-oil and oil horizons in the upper and middle Jurassic is proved.

The situation is similar at Uzen field: water cut - 80%, oil recovery factor - 26.5%. The field, discovered in 1961, is confined to the slightly disturbed brachanticline fold of the northwestern strike. In the Cretaceous complex, twelve gas-bearing horizons were identified in the Jurassic - thirteen oil-bearing and oil-gas-bearing. The total height of the productive floor is 1500 m.

The Karazhanbas gas-oil field, discovered in 1974, is confined to the brachianticline fold of the latitudinal strike. The oil potential of the Neocomian (five oil deposits) and the Bati tier of the Middle Jurassic (two oil horizons) is proved, the water cut is 77.7 % [1].

In the aforementioned fields and a number of others due to the increase of the complexity of the oil recovery in-depth and the conditions of the mode of occurrence, would be applicable such tertiary recovery methods as:

1. Thermal. A decrease of the viscosity of the hydrocarbon deposits is achieved by injecting water vapour, hot water into the oil-bearing stratum, and setting fire to the oil-bearing horizon [2].

2. Miscible displacement. The same result is obtained when carbon dioxide, air, natural gas, nitrogen, flue gases, etc. are injected into the formation.

3. Chemical. The displacement of residual oil is carried out by means of water flooding with solutions of surfactants, polymers, alkalis, acids, chemicals, microbiological effects.

4. Hydrodynamic. Are able to intensify during flooding the current oil production, increase the degree of oil recovery, as well as reduce the volume of water pumped through the reservoirs and reduce the current water cut of the produced fluid.

5. Physical. In this case, the formation is amenable to wave or electromagnetic effects, and horizontal wells and hydraulic fracturing are also used.

6. Combined. As a rule, for increasing oil recovery hydrodynamic and thermal methods, hydrodynamic and physico-chemical methods, heat along with physicochemical methods, etc. are used in combination.

Relating to the researches, field trials conducted in the different countries of the world (Azerbaidjan Republic [3], Denmark [4], PR China [5], Russian Federation [6,7], USA. [5], etc.), the most promising in terms of oil recovery efficiency and the absence of harmful environmental impacts is a microbiological impact, especially considering the positive results of industrial applications and a number of observations, including the laboratory tests for the formation fluids in the Uzen field in Kazakhstan [3].

Its advantages:

Low processing costs;

Environmental safety;

High technological efficiency.

In reservoir conditions, the products of bacterial biosynthesis can increase the mobility of oil by reducing its viscosity and reducing interfacial tension at the interface, which improves the separation of oil from the formation rock [7]. Also, organic acids and carbon dioxide secreted by microorganisms contribute to a long-term change in the filtration-capacitive properties (porosity, permeability) of the oil re-servoir.

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INFLUENCE OF LITHOLOGICAL ROCK DIFFERENCES ON INDICATIONS OF ELECTRIC CAPACITIVE INTROSCOPE

Certain experience has been accumulated in the creation and use of methods and equipment for monitoring geomechanical processes. The most perfect method for determining the state of a rock mass is electro-capacitive one [1]. However, random factors affect the reading of the introscope and the coefficient of disturbance [2] can be used only as a qualitative characteristic of the rock mass at the place of testing.

Through laboratory studies, the dependence of the instrument readings on the main influencing factors was previously determined [3].

However, to date, the relationship between its readings and the fracture volumetric ratio of the rock mass, which is understood as the ratio of the volume of voids formed by cracks to the volume of the rock, has remained undetermined. The purpose of the study was to determine quantitative indicators that allow us to calculate the fracture volumetric ratio depending on the readings of an electric capacitive introscope. Such a dependence is necessary for the subsequent calculation of the displacements of the rock contour of the mine [3], the prediction of the aftereffects of rock pressure on it, and the selection of rational support parameters [4].

This goal is achieved through mathematical modeling by the finite element method. The methodology was developed for modeling the electric field; also an application program was developed by means of which a series of calculations was performed on models simulating a rock mass with different lithological character; then simulation results were processed and the dependence of the instrument readings on the fracture volumetric ratio was obtained.

In the simulation, an axisymmetric volumetric problem was solved. The axis of the probe was aligned with the axis of the model. The number of annular finite elements (having triangular cross section) in the model was 5416, the number of nodes was 2905. The potential difference at the electrodes of the probe was 1 Volt. Around the probe electrodes, a 2 mm thick polyethylene layer was simulated on the model, which corresponds to the wall thickness of the plastic pipe in which the probe's sensitive element is enclosed [5]. The outer diameter of the probe is 20 mm, the length is 150 mm. So the diameter of the model is 200 mm and its length is 600 mm. The ratio of the diameter of the hole to the outer diameter of the probe of the introscope was taken equal to 1.1. The relative dielectric constant of air was assumed to be approximately equal to 1.

The relative dielectric constant of polyethylene was taken equal to 2.35 [6]. During the calculations, side rocks represented by mudstone, sandstone and limestone were simulated. The dielectric constant of these rocks was specified as follows [7]: mudstone - 5.5, sandstone - 4.35, limestone – 8. A fractured medium was considered as a two-component heterogeneous system in which the cracks are filled with air. Wet fractured rocks at this stage were not the subject of research, since their dielectric constant substantially depends on filling not only the cracks, but also the pore space [8]. To calculate the dielectric constant of dry fractured rock, the mathematical model of Maxwell-Wagner was used [9]. When performing calculations, the fracture volumetric ratio varied in the range from 0 to 0.15. The frequency of generation of the device in air was taken equal to A_{air}=2385 kHz.

According to the results of the first calculation, with a fracture volumetric ratio of 0, the readings of the device in the undisturbed A_r rock mass were determined. According to the results of subsequent calculations, the readings of instrument A were determined and then the coefficient of disturbance P_d [2] was calculated.

The matching of empirical dependences showed that with a relative error not exceeding 5%, they can be approximated by a linear function: for sandstone k_f=0.17·P_d, for mudstone k_f=0.43·P_d and for limestone k_f = 0.91·P_d. The obtained dependences are true when the ratio of the diameters of the hole and the probe of the device is 1.1.

For other values of this ratio the readings of the device should be corrected in accordance with the results of previous laboratory studies [3].

The conducted studies are part of a concept being developed for monitoring geomechanical processes and their effects, which can be the basis for an automated system of geomechanical control in coal mines.

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NEW TYPE OF ANCHOR SUPPORT

At the present stage, the mining industry is introducing new innovative technologies that help accelerate the construction processes of conducting and securing mine workings, with the goal of maximizing the following technical and economic indicators: strength, stability, reliability, durability, reducing the cost of mining.

Anchor support in horizontal and inclined workings is one of the rational types of supports that reliably protects against collapse of rock formations.

The classification and design of anchor supports are very diverse, for example, Russian or American metal anchor bolts of various types and the cost of manufacturing spacer rods, with a rod diameter of 19 to 22 mm, polymer anchor supports developed at the A. A. Skochinsky IPD, etc.

A new type of anchor support, which is proposed by the method of jamming the lock is classified as spreader rods of increased ability and by design it consists of made rubber, polymer or rubber - plastic cases and a rod with a metal or polymer composite material (PCM), which is a modern material competing with traditional materials like metal and its alloys - by weight (10 times lighter), strength (higher than 3 times), durability, neutrality to acid-based media of PH (stable Chiv corrosion).

A feature of the new anchor support is the manufacture of cases with an external diameter of 2-4 mm, less than the diameter of the fixed borehole, and the inner one is 6-8 mm less than the diameter of the threaded rod.

The inner diameter of the case for a firmer grip of the rock castle from the front end at a distance of 10-15 cm is reduced when the rod is screwed here, it expands by the thickness of the case, and in this way provide a strong grip of the rock castle.

The installation technology is simple, a case is put on a drilled hole, then the rod is screwed up to the stop, manually or mechanically.

Conclusions:

The advantages of the new anchor support are:

- lining of increased bearing capacity, due to continuous consolidation in the rock;
- a wide area of application for both ground and underground structures;
- resistance to corrosion;
- the ability to dismantle the lining, for re-use in other mine workings;
- the use of a new anchor support in the vicinity of the mine workings as an elastic damping structure;
- simplicity of design and relatively cheap to manufacture.

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ONLINE CONTROL OF THE CONTENT OF COPPER AND SILVER-RA IN THE ORE BY THE WALLS OF THE MINES OF THE MINES OF THE MINING COMPANY "KAZAKHMYS CORPORATION"

LLP Kazakhmys - the flagship of the copper industry in Kazakhstan, is one of the largest silver producers in the world. The silver is an accompanying ore component in all developed copper-containing polymetallic deposits,. The development of deposits is conducted by mining enterprises of the Mining and Processing Complex (GOK). The largest structural subdivision of the GOK is Zhezkazgantsvet-met LLP, which is developing the deposits of copper sandstones Zhezkazgan and Zhaman-Aibat.

In connection with the constant growth of the role of silver in the corporation’s economy, the task of online monitoring of silver contents in operating faces, as well as in faces prepared for mining operations, at the mines of GOK acquires the rank of first priority.

Three circumstances complicate the solution of the problem: low (3-15 ppm) silver contents in ores; height (up to 8 m) of the working faces; the presence of a linear, reverse, concentric, and vertical zonality of the Zhezkazgan deposit of copper minerals, acting both in the context of the entire ore-bearing stratum and in terms of individual horizons and individual ore deposits (due to the presence of zoning, the possibility the use of correlation between the contents of copper and silver).

Since 1978, the mines of LLP Kazakhmys have been using online monitoring of copper contents using energy dispersive X-ray fluorescence (EDXRF) spectrometers. The research method is X-ray fluorescent. Now the RPX-12 EDXRF spectrometers (4 elements: Cu, Pb, Zn, Fe) and RPP-12RI (12 elements: Cu, Zn, Pb, Fe, Ba, K, Ca, Ti, Mn, Ni, As, Sr) of the production of “Aspap Geo” LLP (Alma - Ata) are being used. These spectrometers operate confidently at high faces, but do not determine the silver content.

In order to organize an effective control system for the associated silver mining, the wearable EDXRF spectrometer must confidently determine the silver content, starting from 1+ ppm. Given the circumstances listed above, for the EDXRF spectrometer is an extremely complex scientific, methodological and apparatus task.

Under its decision, on the terms of reference of the corporation, Aspap Geo LLP developed the RPP-12T portable EDXRF spectrometer. Under its decision on the technical task of the corporation, Aspap Geo LLP developed the RPP-12T wearable EDXRF spectrometer. The RPP-12T spectrometer has: two versions: mine (with a device for attaching rods) and core; three operating modes: nature, core, powder; high-speed silicon drift detector (SDD) with an area of 5 mm^2 (thermal cooling, energy resolution - 140 eV along the 5.9 keV line); small x-ray emitter of 50 kV, 10 W; analytical information collection area - up to 4 cm^2; 34 detectable elements (Cu, Zn, Pb, Ag, Cd, As, Ba, Fe, Mo, Mn, Ti, V, Cr, Co, K, Ca, Ni, Ga, Br, Rb, Sr, Zr, Y, Nb, Sn, Sb, Bi, Se, In, Pd, Te, W, Th, U); serial shockproof and waterproof (IP-68 certificate) smartphone Caterpillar CAT S41 with the Android operating system, instead of a portable handheld personal computer; continuous operation time without recharging the batteries - at least 10 hours; the weight of the sensor is not more than 1.5 kg [1-3].

The use of a high-speed SDD and small-sized X-ray emitter, optimization of the excitation conditions of the K-series silver analytical lines, application of the most modern high-speed electronics, powerful methodological and software allowed to increase the luminosity (input load over 100 kHz) of the excitation unit and the detection of the RPP - 12T spectrometer and the signal/background ratio, and thereby, significantly accelerate the sensitivity of the spectrometric path to silver contents, which together ensured the possibility of reliable operation of the RPP-12T spectrometer at a silver content in ores from 1+ ppm when the measurements are exposed at the observation point for 10 seconds.

We have carried out an extensive research program to determine the ability of the RPX-12T EDXRF spectrometer to solve the problem of online monitoring of silver contents, starting from 1+ ppm, which included:

1. Core research of exploratory wells at Zhezkazgan field. Measurement modes: continuous (the sensor moves along the meter core interval for 20 seconds) and intermittent (sampling step 10 and 5 cm, the exposure of measurements 10 and 5 seconds).

2. Research on reference ore ores.

3. Research on crushed (class -30 mm) carriage ore samples.

4. Research on geological samples of ores.

5. Research on state standard ore samples (GSO).

6. Studies to determine the feasibility of high testing (up to 8 m) faces.

The research results [1-3] convincingly proved: the RPP-12T spectrometer really provides effective online monitoring of silver contents in the core of exploratory wells, along the walls of mine workings, in broken rock mass, in powder samples of ores in the range of silver contents from 1+ ppm. At present, 8 RPP – 12T spectrometers are involved in silver RFO ores.

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OPERATIONAL AND ECONOMIC ANALYSIS AS ONE OF THE PRODUCTION MANAGEMENT FUNCTIONS ENTERPRISES OF THE OIL AND GAS COMPLEX

Further improvement of economic management of social production is an important condition for improving its efficiency. Improvement of the mechanism of economic management of the enterprise constantly requires fundamental changes in methods and forms of activity of restructuring personnel management psychology [1].

The modern manager of the enterprise of each branch of the country's economy should be guided in the scientific and technical bases of production, its improvement of labor productivity and quality of products, combine professional competence with ideological horizons and ability to work with people [2].

The need for improvement of all aspects of the management of the enterprises of the oil and gas complex is connected with the growth of its scale, accelerated scientific and technological progress, entry into the Ukrainian market of well-known oil and gas companies.

In recent years, economists have paid much attention to the issue of further accounting improvement. The accounting function is closely linked to the control, which does not act as a standalone function, but plays an ancillary role in all other management functions. The essence of control lies in the detection and timely elimination of errors in the planning, accounting and organization of production.

An important function of management is analysis. With its help the quantitative and estimation of all production changes at the enterprise is carried out. The analysis reveals the reasons for the change in indicators and the interdependence between the individual parameters of the object they are exploring. By means of the analysis it is possible to set trends in the development of certain processes in time and to determine the best, most effective variants of the solutions [3].

The analysis reveals unused reserves and opportunities that can improve the management of the enterprise. Therefore, the analysis serves as the basis for management decisions making [4].

Business executives need to learn about business and careful analysis of practical errors and their timely correction. This will allow systematic regulation of production, coordination of economic processes. At the enterprises special economic services are engaged in economic analysis. In the analysis, they determine the most important factors of the enterprise on which, in the first place, depends on the quality of work performed or the quality of products produced. This approach to the study of economic activity allows us to determine its most important points without compromising the management system with multifaceted and unnecessary information and significantly reduce the complexity of economic analysis.

In the practice of analytical work at oil and gas enterprises it is necessary to strive not to increase the number of analyzed indicators and analytical calculations, but to identify those indicators that need detailed analysis to make specific management decisions.

Further refinement of the analysis, as a function of management, must be achieved by its promptness. The object of operational and economic analysis is the economic activity of the enterprise, which is reflected in the system of indicators that characterize the results of work for each day. It is carried out with the purpose of timely identification of the reasons for deviations of planned tasks for making appropriate management decisions.

The purpose and objectives of operational and economic analysis are closely related to the achievements of the enterprise, both during the month and throughout the year.

In the process of economic analysis, the short-term changes that occur in economic indicators are studied every day and can be influenced by the management system at once [3].

At the present stage of economic development, it is not enough to know only the results of production activity after the end of the reporting period. Because significant changes can occur during this time, both positive and negative, and need to be addressed in a timely manner. Only operational and economic analysis provides managers with the necessary operational information for each day, indicates the reasons, shortcomings of individual production units, divisions, and crews, allows to compare the obtained indicators with the previous period and summarize the obtained results [5].

Information after the operational and economic analysis should be provided to the management of the enterprise in the form of tables and specifically designed measures aimed at the fastest possible elimination of the identified deficiencies.

A well-established system of operational and economic analysis will give the expected results, provided that it is coordinated organization at all levels of production management.

Therefore, a well-established operational and economic analysis at the enterprises of the oil and gas complex is an important means of identifying the reserves of economic growth of these enterprises, improving their efficiency, rational use of resources.

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ONLINE MONITORING OF THE CONTENT OF COPPER AND SILVER IN THE ORE SUPPLIED BY GOK LLP «CORPORATION KAZAKHMYS» TO ZHEZKAZGAN CONCENTRATING FACTORIES

The largest structural unit of the Mining and Processing Plant (MPP) of Kazakhmys Corporation LP is the Zhezkazgantsvetmet Production Association, the mining enterprises of which are developing deposits of copper sandstones Zhezkazgan and Zhaman-Aibat. In connection with the closure of underground testing sites for OTK in mines, the most important task for miners was the argument for upholding the contents of copper and silver in the ores shipped to Zhezkazgan processing plants ZhOF-1 and ZhOF-2. One of the ways to solve this problem is online monitoring of copper and silver contents on input conveyor belts №. 1, №. 1A (ZhCF - 1), №. 1T, №. 2T (ZhCF - 2). Suspended ore control stations (RCM) could serve as a tool for such monitoring.

The effective operation of CSMs in the commercial ore of Zhezkazgantsvetmet is greatly complicated by two factors: ore size (grade –300 mm) and low silver contents (average 15 ppm) in ores. World experience shows that under such initial conditions it is almost impossible to choose an effective method for studying the material composition of ores.

Nevertheless, the problem was successfully solved. In 2014, on the conveyor No. 1T ZHCF - 2, the energy dispersive X-ray fluorescence (EDXRF) RCM RLP-3-02 LLC Geotech (S - Pb, Russia) was put into operation [1]. RCM determined the contents of copper, lead, zinc and silver. At that time, the main demand was for the accuracy of monitoring copper contents. And for the accuracy and reliability of monitoring the contents of silver, high requirements were not imposed. The production operation of the RCM RLP - 3-02 confirmed its high efficiency.

With the increase in the specific gravity of silver in the corporation's economy, the requirements for monitoring its content in the commodity ore of Zhezkazgantsvettsmet were tightened. The following tasks were set: to organize online monitoring of the contents of copper and silver on all input conveyor belts ZhOF-1 and ZhOF-2; determine silver content from 1+ ppm. RCM RLP-3–02 could not work at such low silver contents. Therefore, to solve the problem, they attracted Aspap Geo LLP (Alma-Ata), the leading manufacturer of EDXRF spectrometers in Kazakhstan.

In October 2018, RCM RLP-21T was launched on conveyor No. 1T ZHCF 0 2. Testing elements: copper, pig-iron, zinc, silver, cadmium, iron. At the end of January 2018, the RLP-21T CS was launched on conveyors № 2T (ZhOF-2) and №. 1A (ZhOF-1). The device and test results of the RCM RLP - 21T are described in detail in

[2, 3]. Thus, on all four input conveyor belts ZHCF - 1 and ZHCF - 2, an effective online monitoring of the contents of copper and silver was organized.

In the created system of online monitoring of copper and silver contents, RCM RLP-3-02 turned out to be a “weak link” because it did not provide the required sensitivity of monitoring for silver. Therefore, at the end of August 2019, it was replaced by the RCM-RLP-21T [4].

This RCM in its capabilities exceeded the capabilities of the previously installed three RCM RLP-21T, since it took into account the experience of installation (2018) of the RCM RLP-21T on tape input conveyors of the Balkhash OF (BOF). BOF receives ores with a significantly lower silver content (for example, the Nurkazgan underground mine supplies ores with an average silver content of 2.7 ppm; even lower silver contents in the ores of the Kounrad mine).

All four RCM RLP-21T installed on input conveyor belts ZHCF - 1 and ZHCF - 2 have the same measurement technology. Single measurement - 1 sec. Measurements go one after another without pauses. The contents of copper, lead, zinc and iron are determined by the sum of 20 single measurements. The silver and cadmium content are determined by the sum of 40 single measurements. Reports on the results of the work of the CSW are available to all interested specialists of the corporation.

If you use the latest information, then for the 20 days of October 2019 the following results are obtained.

Enrich factories	Cooper, %			Silver, ppm		
	RCM	слив	Δ	RCM	слив	Δ
JOF –1	0,96	0,96	0	8,14	8,26	0,18
Including nickname .Zhomart	1,21	1,23	-0,02	6,13	5,95	0,18
JOF –2	0,71	0,76	-0,05	14,89	13,61	1,28

Findings. As a result of methodological studies, bench studies and hardware developments, the following results were achieved:

- for the first time in Kazakhstan’s non-ferrous metallurgy, the problem of reliable online monitoring of coarse-grained ores of the –300 mm class for copper, lead, zinc, iron, and, and most importantly, Ag and Cd at very low (1+ ppm) contents has been solved last in ores;

- the online monitoring technology provides the mining units of Zhezkazgantsvettsmet software with a reliable evidence base for arguably defending the interests of mines and quarries in the distribution of ZHOF – 1 and ZHOF – 2 discharge metal based on the results of work over the past calendar month.

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DEVELOPMENT AND OPERATION OF OIL AND GAS FIELDS IN THE CONDITIONS OF THE DNIEPRO-DONETSK CAVITY USING INNOVATIVE COILED TUBING TECHNOLOGIES

The global trend of natural reserves depletion of hydrocarbons requires the use of more sophisticated wells and drilling techniques, since "classical" drilling no longer sufficiently satisfies the needs of the productive reservoirs. In some cases, the best solution is not a new technology, but a new use of already known technologies [1].

In recent years, virtually all oil and gas companies operating in Ukraine have paid great attention to the quality of well construction and the opening of productive layers. New progressive technologies in oil and gas production are widely used for this purpose.

Today, it is difficult to imagine a situation in which work to increase the oil recovery of layers and repair wells, especially in the fields that are in the late stages of development, would be carried out without the use of coiled tubing technology.

Coiled tubing is widely used in technological, as well as repair and restoration works, carried out on gas, oil and gas condensate wells [1].

Coiled tubing technologies are widely used: for well drilling, overhaul of wells, waterproofing works, etc. Therefore, the issues related to the use of coiled tubing technologies in the development and operation of oil and gas fields in Ukraine are relevant and worth considering. One such technology in drilling, which provides the opening of productive layers, is the use of coiled tubing.

The tubing drilling method based on the use of flexible tubes is widespread when drilling new wells as well as new old wells. High technical and economic efficiency is achieved when drilling inclined and horizontal side wells of existing wells.

Particularly effective is the use of coiled tubing at the late final stages of development in the operational wells of the inactive wellbore when cutting the side trunks. In addition to drilling, wells are also widely used in well overhaul.

Currently, oil and gas industries located in the Dnieper-Donetsk region are increasingly gaining popularity with wells.

Waterproofing technologies are also being used with the use of coiled tubing installations in oil, gas and gas condensate wells to improve the efficiency of wells.

Coiled tubing machinery is mainly designed for overhaul of wells and is widely used in operations related to injection into the well of technological agents - various liquids, gas or foam and does not require the lowering of the wells equipment. The following operations include: the invocation of the inflow by aerated liquids; flushing wells to remove chips; acid treatment of the bottom zone of the formation.

Operations with the use of coiled tubing technologies include the following: the challenge of the tide by lowering the level in the well, gas lift operation of wells, removal of fluid from gas wells, operation of wells through flexible pipes, removal of plugs of different densities, acidic treatments of ripples reservoirs of different types, perforation of wells, installation of gravel filters, destruction of solid deposits (cement, milling, etc.), cutting of tubing (tubing) and casing, side well drilling, well drilling (for depression and balanced), cement works, wellbore drilling, slope repair, horizontal and well drilling, geophysical studies [2].

Complex operations performed by coiled tubing installations include operations for well exploration, logging, visual inspection of the state of the well. In fact, the coiled tubing installations significantly increase the efficiency of the work on the wells preparation for repair and insulation work, for the purge of the face of water, the testing of columns for leakage by the method of reducing the level of fluid in the well, foam acid treatment, purification from salt deposits and gas-hydrate nitrogen, blowers and crimping of pipelines.

One of the promising innovative technologies that can be applied to coiled pipe installations is the introduction of metal-polymer coiled tubing (MPCT), which combined the advantages of flexible pipes and geophysical cable. Of course, MPCTs cannot completely displace metal flexible pipes. However, there are a number of wells in which the use of these pipes will also be effective from both a technological and economic point of view. MPCTs have less weight, minimal hydraulic pressure losses during flushing, which allows them to be used in deeper wells of the Dnieper-Donetsk basin.

In addition, MPCT can be used instead of geophysical cable in a number of downhole operations.

The advantages of MPCT over the geophysical cable are the more rigid MPCT design, which allows the pipe to penetrate inclined and horizontal well intervals, as well as the presence of an internal cross-section for the possibility of supplying fluid to the wellbore. The advantages of MPCT over the coiled tubing are the more flexible MPCT design.

It is known that the metal coiled tubing has a limited resource for possible bending due to the cracking of the pipe wall, and in MPCT this resource is higher, based on the physical properties of the components used. In addition, MPCT is not susceptible to corrosion and aggressive effects of hydrogen sulfide [3].

This helps to reduce the cost of drilling.

If, under traditional technologies, the technical capabilities of machines were determined by the modes of operation, then bending can provide conditions for rational exploitation of the deposit, optimal modes of disclosure, development, operation and overhaul.

Similar tasks were solved and partly solved in drilling and cattle using traditional constructions of columns, but in full volume they can be solved only with the use of coiled tubing [1].

Today, innovative coiled tubing technologies have great advantages over traditional drilling, overhaul and well development technologies.

Therefore, the use of coiled tubing, compared to traditional technologies, on average, reduces the time to clean tubing, wash sand plugs, acid treatment of the reservoir, which, in general, reduces the time for repair work in the well.

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THE TECHNOLOGY OF MINING THIN VEIN ORE BODIES

Due to the fact that with intensive development of deposits with favorable mining and geological conditions over the past 20-25 years, more and more deposits remain with complex mining and geological conditions. That is, the content of the useful component decreases, the power of the ore body decreases. The development of such deposits is carried out by various development systems, the use of which is accompanied by an increase in the cost of extraction and contamination of ore, which in some cases reaches up to 500-600%. Thin deposits work out with the shrinkage system or stull-set system. There are a number of studies to improve the technology for the development of thin ore bodies [1,2,3], however, since the applied development system and the proposed technology do not give the desired results, we offer the technology of mining thin vein ore bodies.

The object of study is the Verkhne-Andasayskoye gold deposit in the Republic of Kazakhstan. The Verkhne-Andasayskoye field is located on the border of the Chuy Depression and the southwestern wing of the Chu-Balkhash anticlinorium. The main structure that determined the geological structure of the region and metallogeny is the Jalair-Naiman zone of deep faults, falling to the northeast at an angle of 70-80 degrees.

The Verkhne-Andasayskoye deposit belongs to the gold-sulfide-quartz ore form, the gold-sulfide-quartz (gold quartz) geological and industrial type. This deposit is small in reserves and rich in gold content (27.04 g/t). The power of the ore body ranges from 2 cm to a meter. Ore and rock are close to each other in physical and mechanical properties. By strength, ore belongs to the strong (III) category, the breed - to medium (IV). Water absorption is small (1.4-2.3), humidity is low 0.42-0.64%. Ore and rock are non-radioactive, not prone to spontaneous combustion. The Verkhne - Andasayskoye field - is mined using the underground method, characterized by the presence of thin rake veins; a large fortress (14-18 according to M.M. Protodyakonov) and rock stability.

The project, for the development of the deposit, provides for a mining system with ore shrinkage with breaking-out out in bulk with scraper haulage.

To reduce contamination of ore, the consumption of explosives and reduce the complexity of work, proposed advanced technology for mining thin ore bodies.

The essence of our proposed technology for the development thin rake vein is that the ore deposit prepared by the floor method is divided into mine section. Preparing the block for an actual mining includes: carrying out floor ventilation and haul roadway, as well as two shrink drift and flank rising workings. After conducting uprising workings between the drifts, which limit the block along stretch, in the center of the latter, mill hole is mined in the usual way to the entire height of the block, leaving the drift pillar.

Each block, in height, is divided into working floor with diagonal drilling workings. The diagonal workings are tilted towards mill hole with an inclination angle of 45 °. This angle is necessary for the unimpeded release of ore into the mill hole, so that the inclination of its wall has an angle of repose for the free movement of the beaten ore towards the mill hole, eliminating the possibility of clogging. The distance between the diagonal workings coincides with the length of the oncoming holestaking into account the breakdown distance. The ore is beaten by oncoming holes up to 2 m long, drilled from diagonal drillings. An additional ore launch is also carried out, which connects the scraper and haul drifts. All this can reduce contamination of ore during breaking, thereby improving its quality and ensuring the completeness of excavation.

Our studies obtained the dependence of ore dilution on the distance between the diagonal drilling workings, the width of the working space and the height of the ore body, which will allow you to choose the parameters of the development system, taking into account the height of the ore body. With the right choice of development system parameters, the proposed technology will reduce the amount of ore dilution to 35%.

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THE RESEARCH OF THE INFLUENCE OF TECTONIC FRACTURE ON SECONDARY FRACTURE FORMATION IN THE COAL WHEN UNLOADING THE COALBED SATELLITE

At present, in all coal-mining regions of the world, the problem of methane from coal deposits is of particular importance. This problem, first of all, affects labor safety during the treatment works, methane production, as an alternative source of energy resources, as well as environmental issues related to methane emissions into the atmosphere of coal mining enterprises.

With increasing depth of mining, the geological and mining conditions sharpened badly. The emission of methane into the developed space increased significantly, the main sources of which are sandstones and satellite beds. There are many schemes for drilling degassing wells used to increase the efficiency of methane extraction. However, not all schemes take into account the peculiarities of geomechanical processes occurring in a coal-rock massif and the influence of tectonic disturbance on the formation of zones of accumulation of free methane in the places of unloading of undermined coalbed satellite formations.

In order to increase the degassing efficiency of the under-produced coal-rock massif, an analysis was made of the nature and mechanism of coal destruction under conditions simulating the stress-strain state of the disturbed coalbed satellite formation during treatment. To study the effect of natural fracturing on secondary fracture formation in coal during its destruction, an unequal true triaxial compression unit (TTCU) installation was used, which allows creating independent stresses in three mutually perpendicular directions that are similar in magnitude and direction to the rock massif [1]. The modeling of the zone of the limiting state with unloading was carried out at different loading levels, different orientations of the natural (tectonic) fracture, and different parameters of the type of stress state.

An analysis of the results of modeling the stress-strain state of the coalbed satellite formation under the condition of generalized compression with a transition to a generalized shear during its unloading showed that regardless of the orientation of natural cracks in the coal sample when the stress state type parameter changes from generalized compression to generalized shear (from - 1 up to 0) the destruction of coal occurs due to the growth of shear cracks located at an angle of 18 - 22 degrees. to the line of action of the maximum principal voltage [2].

At the same time, the results of the studies of the influence of the interaction of tectonic and technogenic fractures on gas production from the massif, in the conditions of the first northern lava of formation d_4 of block 3 of PJSC ShU Pokrovskoye, showed that the methane production rate from degassing wells exceeds 1.5 - 2 times of average indicators in places where the ratio of the tectonic to technogenic fracture angles is in the range from 0.5 to 1.25. With this ratio of catch in places of local discharge of coalbed satellite layers and intense tectonic fracture more favorable conditions are created to conducive to less energy for crack opening therein and accumulation of methane.

Thus, when the lava leaves the critical span of the primary roof landing, under conditions of volumetric uneven compression of the rock mass to a level exceeding the compressive strength of coal by 2.5 to 3 times, during its operational unloading, the deformation state of the coal seam is formed close to the generalized shift $\mu_\varepsilon = -0.5 \div 0$, which leads to the destruction of the carbonaceous massif by shear technogenic fracture with the formation of local zones of destruction of satellite layers before the initial landing and the accumulation of free methane [3].

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GEOLOGY AND MATERIAL COMPOSITION OF THE ORES OF THE ABYZ DEPOSIT

We have studied the Abyz deposit of the Kosmurunsky Upper Ordovician metallogenic complex. Volcanic-sedimentary deposits of the Devonian, interrupted by intrusive and subvolcanic formations of medium-basic composition, take part in the geological structure of the field. At the deposit, within the zone of metasomatites, more than 13 ore bodies are distinguished. The morphology of the ore bodies is rather complex, the forms are lenticular, lenticular-stratum. Both in the dip and in the strike, swelling and pinching, bends, apophyses are noted. Wedging out is mostly gradual, smooth. In some places, it is rather sharp and dull, which is confirmed by the results of drilling during network thickening and by the data of borehole geophysics.

From the surface and to a depth of 20-35 m, the main ore body is composed of oxidized ores, which below, through the compression of substandard ores, become sulfide. The thickness of the ore body varies widely, from 0.3 to 40-50 m, averaging 12.1 m in the East section and 12.2 m in the West. According to the main ore body, as a rule, in the contours of solid ores, 2-3 enriched zones with a high content of all components and traced by dip and strike are distinguished. Ore body 1 is localized in the lying side of the Main ore body and consists of 6 separate, spatially separated lenses in the Eastern section. In the western section, the ore body consists of two branches. Ore body 3 is located even lower along the section of ore bodies 1-2, has a length of 400 m in the East section, and 110 m in the West section. The thickness of the ore body ranges from 0.8-8.6 m, averaging 3.0 and 4.6 m in the eastern and western sections, respectively. Ore body 4 is present only in the Eastern section, has a length of 350m with interruptions, consists of 4 lenses. Ore body 5 is cut only in 2 profiles and has no industrial significance. Ore body 6 is the second ore body in the Western section. It is localized in the hanging side of the Main ore body, and if on the East site it is represented by 6 lenses disparate in space, then on the West - by two, but combined by a single mineralized zone, not contoured neither in the south, nor in the north, nor in depth. The thickness of the ore body ranges from 0.8-8.6 m, averaging 3.0 and 4.6 m in the eastern and western sections, respectively. Ore body 4 is present only in the Eastern section, has a length of 350m with interruptions, consists of 4 lenses. Ore body 5 is cut only in 2 profiles and has no industrial significance. Ore body 6 is the second ore body in the Western section. It is localized in the hanging side of the Main ore body, and if on the East site it is represented by 6 lenses disparate in space, then on the West - by two, but combined by a single mineralized zone, not contoured neither in the south, nor in the north, nor in depth. Within the profiles 17a, 19-20, the ore body reaches the day surface, where it is opened by ditches and pit. To the depth, the ore body is explored mainly by wells. Ore body 7 is located along the section above the Main and 6 ore bodies, but below the thickness of polymictic gravelites in the Eastern section. In the Eastern section, it is represented by four lenses of two branches, conventionally separated, in space. The total length of these lenses was 570 m with an average power of 1.9m. In the Western section, the ore body is cut only in 2 profiles and has a length of 100m with an average thickness of 7.3 m. In profile 19, the ore body emerges to the surface where it is traced by ditches, pits and at a 775 m horizon it is opened by cuts. Ore body 8 is located in the upper part of the section above ore bodies 6 and 7 and the stratum of polymictic gravelites and sandstones in the Eastern section, where it is represented by two branches (or 3 lenses) with a length of 470 m. Ore body 9 is cut above the section of ore body 8 and is found only in the Eastern section, where it is represented by four lenses with a total length of 210 m, with an average thickness of 3.5m. Ore bodies of the Eastern zone are mainly lenticular, curved in areas and are blind, occurring at a depth of 30-250 m. Ore bodies of the Western zone also have a lenticular shape complicated by flexure-like kinks, are completely blind and lie at a depth of 250-700 m. They are composed exclusively of hypogenic ores. Hypogenic ores in the deposit are represented by two types: solid (massive and banded) and disseminated (vein-disseminated), usually spatially combined. Solid ores are of primary value in the deposit.

The texture is banded. Disseminated ores account for approximately half of the total ore reserves in the deposit. The ore bodies of the eastern zone of the deposit are mainly blind, the apical parts of which are located at a depth of more than 30 m. Only in the northern part of the zone, between 13 and 20 prospecting lines, do ores reach the surface and are subject to oxidation processes. The depth of the oxidation zone ranges from 24.7 to 39 m. In the oxidation zone, the ores are dense quartz-limonite or slag-like dark brown aggregates, sometimes of a collomorphic texture and intersected by veins of malachite, azurite with small nests of cuprite. The main ore-forming minerals in the deposit are pyrite, sphalerite, chalcopyrite and galena, secondary ones are faded ore and rare ones are compounds of silver, bismuth and tellurium. Of non-metallic minerals, quartz, sericite, and chlorite are widespread. Further, the following minerals are noted in terms of prevalence in ore deposits: pyrrhotite, bornite, enargite, chalcopyrrhotite, cubanite, arsenopyrite, magnetite, hematite, molybdenite, aikinite, tellurium native, altaite, tellurous bismuthite, tetradimite, coloradoite, hessite, petritite, arthritis, petcitazite, petcitazite, petcitrite, arbite, petcitazite, petcitazite, petcitrite, and proustite-pyrargyrite, stephanite. Native gold and electrum are present in all types and varieties of ores.

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RESOURCE-SAVING TECHNOLOGY OF LOW-COAL SEAMS UNDERGROUND MINING

Mining and geological conditions of stope operation in Ukrainian mines needs solving complicated technological problems providing an opportunity of mineral mining [1-3]. Rock mass features, and mechanical characteristics of rocks, composing it, form the conditions under which deformation of a mine working boundary may achieve up to 90% of its initial linear dimensions.

Selective coal mining helps solve a number of problems arising in the process of extraction performed within the coal seams which thickness is less than a meter. However, considerable volume of rock, being cut, which had to be transported to the surface initially, is the basic major problem of the procedure. Nevertheless, the development of a method to backfill the mined area by means of the cut rock has helped solve the problem. Boundaries of the technology to protect mine workings as well as its efficiency determine volume of rock available for backfilling. To be more accurate, empty volume of the mined out area-volume of rock being cut ratio is meant.

For instance, in terms of *Samarskaia* mine [1, 2] the powered support was reinforced with the help of pit props providing immediate roof holding within the operating area of a shearer. Hence, design characteristics of the powered support in the context of backfilling technique implementation have been determined for two alternatives – rigid alternative, and flexible one.

The proposed design of the powered support section does not involve opportunity to resist caving of hanging roof rocks of the stope. Response of hydraulic cartridges with following console lowering takes place if pressure, bearing on the reversed console of the support. Local rigidity of the design is 20-30% lower to compare with other components of the structure. The fact raises a question concerning its efficiency under such mining and geological conditions when a chance of the immediate roof failure in terms of partial backfilling is more than 0.3. As a result, a design of the powered support section with the strengthened bearing structure of the reversed console has been proposed.

The identified discontinuities and areas of rock softening have been analyzed as for their interaction and integral effect on a stope support. As a consequence, 3-D model of the layered rock mass has been developed involving maximally each feature of strengthening characteristics and deformation characteristics of the rock and coal seam.

Standard form of vertical stresses within the stratified rock mass accords well with general ideas concerning rock pressure formation which can also be indirect confirmation of correctness of the parameters selected to describe geomechanical model for the computational experiment. The obtained curves of vertical stresses are separated into two areas of compressive stresses and tensile stresses by means of a parallel plane passing at the distance of 1.5 m from the stope face towards the mined-out area. Certain share of the analytical model, neighbouring the undisturbed rock mass, experiences compressive loads and the share, around the mined-out area experiences tensile loads. Ultimate compressive stresses are concentrated right behind the stope face.

Deformation processes of a stope boundary takes place due to displacement of roof and floor of a mine working. That depends upon significant length of a stope together with minor height of the mine working. Roof fault and floor heaving follow the rule of plane-parallel displacements oriented perpendicularly to a gravity force axis [4, 5]. In such a case, vertical stresses become the dominating conditions providing equilibrium of geomechanical system of a mine working. The stresses also exert forming influence on other components of stress-strain state of the rock mass within areas neighbouring the stope and the mine workings.

The analysis of geomechanical model of rock mass, and a mine working, driven in it and propped up by the powered support section has shown that, if block caving of roof takes place, dynamic changes in stress-strain state of rock mass depend upon height and location of the roof layers within which stress concentration arises with up to 80% values of stresses arising in front of the stope. In this context, geometry of the blocks varies as well as mechanical nature of rock layer failure.

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OIL RECOVERY ENHANCEMENT TECHNOLOGY IN KENKIYAK FIELD CONDITIONS

The Kenkiyak field was put into development in 1966.

In October 1956, structural exploratory drilling was launched at Kenkiyak Uplift. Oil of Jurassic and Cretaceous horizons is close to oil, specific gravity is 0.9127 g/cm^3, sulfur content is 0.59%, paraffin is 0.64%, oil viscosity at 20 ° C ranges from 50 to 900 MPa *s [1]. Within the Kenkiyak-nadsoleva field, there are 8 horizons in oil production development: in the Jurassic and Cretaceous sediments, 400 m deep and the Upper Permian oil deposits with depths of 1250 m. The Kenkiyak field belongs to the category of high-viscosity oil fields (in the deposits of the over-salt complex), very complex, oil (up to 60% yield), heavy and oxidized, deprived of fractions and gas factor. The approved oil recovery factor is 0.409.

The movement of the infiltration flow occurs in a southwest direction. Static levels depending on the terrain are set at depths of 15-36 m from the wellhead. The average water saturation is 22%. The thickness of the aquifer reaches 54 m. The formation water of the Middle Jurassic horizons is high pressure, the reservoir pressure reaches 3.97 MPa. The maximum productivity of wells is 541 m^3/day of water. To maintain reservoir pressure (RPM), the field has reservoir - fresh water from the Kok-Zhide field, used to produce steam. It is most advisable to use waste water for injection, which is formed after the separation of the oil-water emulsion collected from various sites. In the process of developing a field, two methods for increasing oil recovery, that is, heat and steam exposure and water injection, are mainly used [2-4]. The efficiency of oil displacement by water is very low due to the high ratio of oil and water viscosities. After two years of putting the field into operation, most of the producing wells began to be watered, with an increase in the increase in water cut, oil production dropped sharply.The main reason for the steam and thermal effects on the reservoir of production wells are, first, the low ratio of the number of injection and production wells (1:9); second, at low formation pressure, the method of steam impregnation is not adopted, but the method of oil displacement by steam with low parameters of steam injection is adopted. The accepted method of heat and steam exposure to an oil reservoir consists in pumping the calculated volume of coolant through injection wells, creating a heat rim and then moving it along the reservoir towards producing wells with unheated water. The increase in oil recovery from the reservoir when the coolant is injected into it occurs due to changes in the properties of oil and water in the reservoir as a result of temperature increase. With increasing temperature, the viscosity of the oil, its density and interfacial relations decrease, and the vapor pressure increases, which positively affects oil recovery.In the process of steam injection, the oil reservoir is heated primarily through the use of latent heat of vaporization. In this case, the steam entering the pore space condenses. The formation is subsequently heated by using the heat of the hot condensate, as a result of which the condensate is cooled to the initial temperature of the formation. When oil is replaced by steam, there is an improvement in the evaporation of hydrocarbons by reducing their partial pressure. The decrease in the partial level is associated with the presence of water vapor in the evaporation zone. Light components evaporate from the residual oil and are transferred to the front boundary of the vapor zone, where they condense again and dissolve in the oil shaft, forming a solvent rim, which provides an additional increase in oil recovery. Under the heat and steam action, three zones will form in the formation; all these zones undergo mutual influence. The increase in oil recovery from the reservoir during the injection of steam is achieved by reducing the viscosity of the oil, resulting in improved exposure coverage; due to the expansion of oil.

The viscosity of oil decreases significantly with increasing temperature, especially in the range of 30-80 °C. A relatively high rate of decrease in oil viscosity is observed with an initial increase in temperature. With increasing temperature, the viscosity of the oil decreases more intensively than the viscosity of water, which also positively affects the increase in oil recovery. In the process of steam injection, oil, depending on its properties, can expand, due to which additional energy appears to displace formation fluids. During the displacement of easily volatile oil by high-temperature steam, lighter fractions of the oil pass into the vapor phase, i.e., oil distillation is possible. In the colder zone of the formation, these fractions are concentrated, forming a solvent shaft or a mixing shaft in front of the vapor zone.

The effects of gas pressure conditions, changes in relative permeabilities and their mobility, etc. can contribute to an increase in oil recovery during steam and thermal exposure. As it moves through the reservoir, the steam heats the rock containing oil and displaces it towards production wells. Laboratory studies have found that capillary impregnation of core samples occurs at both low and high temperatures. At the same time, with increasing temperature, capillary impregnation of them occurs much faster. When deciding on the use of heat and steam, it must be borne in mind that the oil-saturated thickness of the reservoir must be at least 6 m.

The depth of the formation should not exceed 1000 m due to heat losses in the wellbore, which reach about 3% for every 100 m of well depth, and great technical difficulties in ensuring the strength of the string. If the total heat loss in the wellbore and in the formation exceeds 50% of the heat received at the wellhead, then the process of heat and steam exposure will be ineffective and uneconomical. The permeability of the formation should not be less than 0.1 μm^2.

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THE INFLUENCE OF THE ROLLER MODES ON THE LINKAGES BETWEEN PHYSICAL AND DEFORMATION CHARACTERISTICS OF COMPACTED OVERBURDEN SANDS

It's economically and environmentally advisable to use overburden and their mixtures formed during the extraction of minerals as a material for the construction of massive soil pillows [1]. Determining the mechanical characteristics of compacted rocks for the pillows in a volume sufficient for their design is associated with significant resource costs [2]. However, it has been found that correlation or even functional dependencies can be obtained between mechanical properties and physical condition of rocks under certain conditions [3].

The condition of the linkages between physical (humidity W, porosity coefficient e) and mechanical soil parameters (specific penetration resistance R, internal friction angle φ, specific cohesion c, deformation modulus E) is the accumulation of test data to determine these characteristics with respect to the plasticity and homogeneous genetically [3]. The determination of the coefficients of the correlation equations with each array of experimental data is usually performed by the least squares method, with the calculation of the correlation coefficients, the variations coefficients and measurement errors.

Determination of the correlation equation coefficients for each array of test data is usually performed by the method of least squares with the calculation of the correlation coefficients, variation coefficients and measurement errors. The imperfection of standards for the design and construction of pillows is the lack of requirements for determining the mechanical properties of compacted rocks. It is possible correlation or functional dependencies between their mechanical and physical properties. These dependencies for small-connecting overburden have hardly been investigated. The effect of rolling technology (parameters and mechanism mode; number of passes per track Δh; initial layer thickness h) on the mechanical properties of rocks has not been established.

Therefore, for the practice of designing sand pillows it is important to investigate the influence on the mechanical properties of compacted rocks of the parameters of the mechanisms (parameters and mode of the mechanism; the number of passes per track; the initial thickness of the layer). The situation at the site on the Vorskla Steel metallurgical plant was characteristic. It was planned to be built on the basis of modern nature and resource-saving technology of the Austrian company "Voest Alpine Industrieanlagenbau". For artificial mass, the overburden (sandy shallow and medium size) for the quarries of the Yeristovsky and Lavrykovsky iron quartzites deposits near the Horishni Plavni town of Poltava region was used.

Field studies of the influence the technological parameters of rolling of sand pillows layers (static or vibration work mode and roller mass, number of passes in one trace) on regularities of interrelation of physical and deformation properties of compacted overburden, in particular, for fine, homogeneous sand were carried out within such limits: 1) single drum self-propelled vibrating roller NAMM 3520: 12 passes in vibration mode; 8 passes in vibration mode; 6 -in vibration mode; 5 - in vibration mode; 4 - in static mode; 2) single drum self-propelled vibrating roller Vibromax VM132: 4 passes in vibration; 4 -in static mode; 3) single drum self-propelled vibrating roller ATLAS 1140: 8 passes in vibration mode; 4) trailed pneumatic roller DU-16: 4 passes; 5) combined rolling: DU-16 roller for 4 passes and NAMM 3520 rollers for 6 passes in vibration mode or Vibromax VM132 for 6 passes in vibration mode.

A statistically substantiated linkages between the physical and mechanical properties of rocks has been obtained, taking into account the influence of the parameters of the rolling, the volume of soil characteristics determinations (granulometric composition, w,ρ_d,E) and the technological parameters of rolling ($h,\Delta h$, static or vibration mode) for: shallow sand -humidity - 314; small, homogeneous sand with sandy loam -144; medium size, homogeneous sand - 61; the measurements number of roller passes with one track - 45; the measurements number of the thickness of the deposited layers - 45.

The possible influence the type and mode of rollers operation on the dependence patterns of the modulus of deformation of the soil on its specific volume of the skeleton $\lg E=f(1/\rho_d)$ for fine, homogeneous sand, for the humidity intervals is investigated by drawing on the graphs of the symbols of each of the above technological modes of rolling the sand pillows layers. Similar graphs were obtained for fine, homogeneous sand with impurity of sandy loam and medium uniform sand.

It can be seen from them that for each type of compacted overburden, regardless of the technological regime and the mass of the roller, the the deformation modulus increases linearly at the specific volume of the soil skeleton decreases.

The the coefficients of variation of virtually all the correlation equations beetwen physical and mechanical properties of the sealed, small-connecting overburden do not exceed v=0,20 and the correlation coefficients are greater than that r=0,85, which proves that the empirical expressions are correct.

Therefore, under whatever rolling technological parameters, a certain value of the density of the skeleton of the soil ρ_d would not be obtained, for a sand of a certain particle size distribution (and for the same pressure interval in the process of compression testing of specimens) a certain value of the modulus of deformation corresponds.

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DEVELOPMENT OF METHODS FOR CRUSHING AND GRAVIMETRIC SEPARATION OF THE POOR HEMATITE ORE

The main purpose of the presented research is to improve the technology of rock destruction by explosion. For this purpose, new methods for creating a differentiated stressed state of a poor hematite ore massif as well as for constructing borehole charges of explosives have been developed.

The methods of research are to analyze of geological data on the iron-ore deposits of Kryvbass, project documentation, production data of the mining enterprises and open pits, scientific publications and their subsequent synthesis and formulation of the developed provisions.

Explosive destruction of the rock massif with the use of vertical slopes of the quarry ledges, with the making of shielding slots, which are created by advanced explosion of special directional charges, as well as the reverse sequence of blasting of the borehole series, provides a reduction in the specific consumption of explosives by 15-17% and substantially increases the uniformity of rock crushing.

The originality is to determine the comprehensive approach in solution to the stated problem, and a number of the proprietary technical solutions.

Further development and manufacturing application of the research results will provide a significant reduction in the cost price of iron ore products by increasing the efficiency of drilling and blasting operations in the open pit.

The experiments of gravimetric separation were carried out following: the crushing of the ore coming from the spoil heap which has undergoes a coarse crushing until obtaining class (-40÷+0) mm, then after sifting we obtained the (-6÷+0) mm, class that we propose to direct towards the process of crushing.

After this mechanical preparation these classes underwent a gravimetric separation by pistonage in an a MOD-1M type apparatus (of Ukrainian republic).

These experiments showed that the pistonage does not give satisfying results. it was noted that a light fraction has weak index of enrichment.

Thus after its extraction class (-40÷+0) mm has undergoes an enrichment in heavy suspension according to the diagram calculated.

The separation of the classes was to realize in 2 steps:

In the first we used the mixture of magnetite and ferro-silice, or the density of suspension is higher than 2600 g/cm^3.

In the second we used the ferro-silice of 3600 g/cm^3 in density.

The enrichment by heavy suspension enabled us to eliminate the part of sterile which found mixed with ore and which can to be use as building materials.

The enriched product still requires additional treatments. In riding quality we used the ferro-silice granulated, which has a high friction resistance and corrosion.

During the separation of minerals by fractions of various densities we obtained the light fraction of densities<2700 g/cm^3, which is characterized only by the sterile with a small quantity of useful ore, (with microscopic inclusions), which cannot be extracted by physical methods of enrichment and are to be consider as losses without return.

The different results of gravimetric separation per fraction of density show us that we can divide our food product into:

A concentrate with a content of iron, equalize to 52.4 %, from the class of (-40÷+6 mm) a content of 38.8 % from food product.

A half product of 39.0 % in iron, with 41.2% degree of extraction.

A light product of 12.3% in iron, with 3.4%degree of extraction.

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MAIN DIRECTIONS OF IMPROVEMENT OF QUARRY DRILLING TECHNOLOGY

Machinery and technology for drilling in quarries is a complex system, the study of which is aimed at improving drilling equipment, tools and the drilling process.

Of the three most widely used main drilling methods in Ukraine's quarries (roller cone, rotational and rotational percussion), the cone method prevails, which accounts for up to 90% of the total drilling volume. In iron ore quarries, represented mainly by hard rocks, the cone method is, as a rule, the main one and accounts for 95-100% of the total drilling volume [1].

On the open mining operations of Ukraine, drilling rigs of various firms are used. In particular, cone drilling machines - SBSh-200 and SBSh-250 (Russia), Sandvik (Sweden), Atlas Copco (Sweden, USA), Terex SKS (USA, Great Britain), etc. The machines of these manufacturers have various modifications and differ in principle the construction of rotary-feeding mechanisms that determine their structural appearance and technological capabilities.

The productivity of the SBSh-200 and SBSh-250 machines is 3-5 times less than in drilling rigs of other companies. Higher productivity of drilling rigs of Western European and American firms is achieved primarily due to the quality of manufacturing, reliability, efficiency of maintenance and operation, the use of electronic control systems and diagnostics. However, they have a cost of 3.5-4 times more than the cost of SBSh machines. According to production experience and research by specialists, despite the higher productivity and reliability in operation of such machines compared to SBS machines, the specific cost of drilling wells will in many cases be higher (at least 4-5 times) [1]. That is, increasing the reliability and productivity of equipment is usually associated with significant costs and has rational economic boundaries.

It should be noted that a significant influence on the economic performance of cone drilling has the right choice and the rational operation mode of cone bits. In particular, during the life cycle of the machine, the costs of roller cone bits are usually several times higher than the cost of the equipment itself. This excess is especially significant (5-8 times) for the machines used for drilling hard rock [1]. In the cost of drilling rock with a coefficient of strength $f>12$, the costs of roller cones reach up to 65% of all the costs of operating a drilling rig.

Existing trends in the development of drilling equipment determine the main directions for improving the standard size of a number of drilling rigs based on the modular principle of their construction. Along with specialized machines, the expansion of a number of combined modifications is considered, which will allow the use of various drilling tools, in particular, all mechanical drilling methods and various methods of well cleaning - pneumatic and screw-pneumatic.

Another important aspect is the establishment and maintenance of rational modes of well drilling. This problem is most fully solved by optimal control of the drilling process, which is implemented by an automatic control system. In [2] it was established that the transition to automatic control of the drilling process compared to manual control increases the productivity of drilling rigs by 15-18%, reduces the cost of drilling by 10% and significantly increases the service life of the drilling tool. Such results are achieved with the right choice of cone bits in accordance with the strength of the rocks. Otherwise, financial losses from the introduction of an automatic control system are usually not compensated.

Currently, there has been a significant increase in the cost of drilling at quarries. In this regard, increasing the efficiency of equipment and technology for drilling wells in open cast mining has become necessary. To increase the efficiency of drilling operations, the following main areas can be distinguished:

- improving the design of drilling rigs in order to increase their reliability, the introduction of automatic control systems, the development of modular equipment based on basic models;

- improvement of existing and creation of new wear-resistant structures of drilling tools for typified mining, geological and technological conditions of quarries;

- improving the organization of management of drilling operations in quarries, the introduction of automated systems for supervisory control of machine tools;

- optimization of the technological process of drilling wells directly in the industrial conditions of quarries.

The first two areas are implemented in the design bureaus of manufacturing companies. The remaining directions are technically and organizationally implemented directly at the drilling sites. It should be noted that the core in the drilling system “rock - drilling tool - drilling rig” is the drilling process, since the main elements that determine the productivity and cost of drilling wells interact in it. Such elements include rock properties, the design and quality of the drill bit and the parameters of its force (axial load on the bit P, bit rotation speed ω) on the bottom, taking into account the degree of cleaning of the well from sludge.

Thus, when optimizing mineral development processes, the urgent scientific and technical task of substantiating and choosing rational drilling systems in an existing quarry is highlighted.

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TENDENCY OF QUALITY CHANGE OF GRAVEL AND SAND RESOURCES IN POLAND

Mineral aggregates are the basic group of minerals mined. In Poland, their share in the extraction of solid minerals over 30 years (1989-2018) increased from approx. 24 to 54%. Aggregate extraction is currently around 300 million Mg/year. The production of gravel-sand aggregates in Poland accounts for 60-65%. The remaining part consists of crushed-stone aggregates (25-30%) produced from compact and medium-compact rocks of igneous origin, sedimentary and metamorphic as well as recycled and artificial aggregates (about 10%). The gradual deterioration of the quality of the raw material base of deposits and, at the same time, the increase in the demand of the construction industry for the best quality thick aggregate fractions (5/8 mm, 8/11, 8/16, etc.) has a large impact on the increase a quantity in aggregates hard to market and non-marketable fractions. This applies especially to gravel-sand aggregates in resources where the proportion of fine fractions (below 2 mm) is steadily increasing, for which the demand for construction is limited and they are often treated as unusable material. As the production and consumption of these aggregates are practically unknown, an attempt was made to assess the tendency of changes in the quality (grain size) of gravel and sand resources on the national and three regional zones.

The basic data source was adopted, published annually by the Polish Geological Institute - National Research Institute in Warsaw, Balance of Resources of Mineral and Water Resources in Poland [1]. In the “Balance...”[1] three major subgroups of gravel and sand deposits can be differentiated the criterion for a division is a sand point (PP) denoting (the percentage of small fractions up to 2 mm in resources): gravel (SP<30%), sands with gravel (SP-30-75%), sands

(SP>75%). According to the analysis of change in resource volume in the period of 10 years (2007-2016) [1,2] despite exploitation, we have a relatively large growth of resources, because the total balance resources of gravel and sands increased by approx. 26.6%, and economic resources by 75% (fig. 1). The unfavourable factor is that mainly resources of sand are increasing (SP>75).

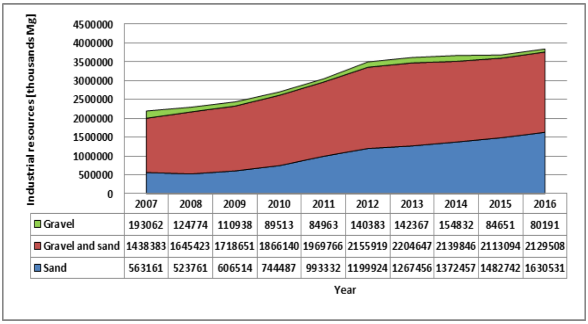

Fig. 1. Economic resources of gravel and sands in particular subgroups of deposits in 2007-2016 [3]

The change of structure of deposits affects the contribution change of particular subgroups of gravel and sands in total balance and economic resources. The contribution of sands in economic resources grows particularly fast, from 25.7% to 42.5% (in 2017 there was a further increase). The increase of documented and excavated resources of sand deposits caused an increase of sand points (percentage contribution of fractions up to 2 mm) in the deposits. Assuming average values of sand points (SP) in particular subgroups of sands and gravel, the approximate average SP value can be calculated in total resources documented and excavated in the given year (Fig.2). For Polish resources and in three regional zones these increases were as follows:

- balance resources: country - 4.5%, in zones: 2.0-6.6 %,
- economic resources: country - 10.4%, in zones: 4.9-16.4 %,
- exploited deposits: country - 4.8%, in zones: 5.5-8.7 %,

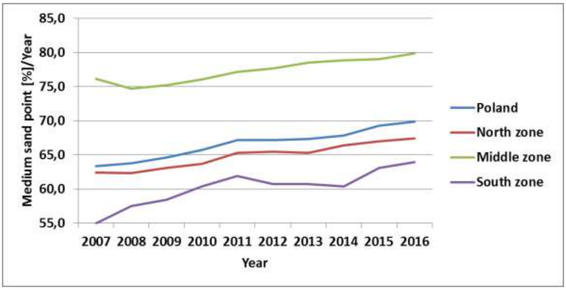

Fig. 2. Medium sand points for industrial gravel and sands in 2007–2016

The presented tendency of growth of fine, sand fractions of aggregates in the deposits of gravel and sands allows forecasting changes in SP in future years; for instance, for industrial resources the likely SP value in 2020 may increase up to 72.0% and in 2030 up to 80.0% [3]. The building industry requires most of all coarse aggregates (both gravel and crushed), whereas the demand for fine aggregates (sand) is fluctuating and to a large degree depends on the needs of the road construction industry. In recent years the demand was significantly lesser than the volume of sand production and mines had major problems with selling them (aggregates hard to market and non-marketable). Non-marketable and hard to market sand fractions of aggregates are often used in mines to re-cultivate post-excavation pits. The scarcity of natural aggregates in many countries and regions and limited resources of aggregates in Poland and increasing difficulties in obtaining a license for their extraction indicate the need of selective landfilling of sand aggregates instead of “drowning” them in post-excavation pits.

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OPEN PITS PRODUCTIVITY CONTROL ALONG WITH IRON ORE PRODUCTS DEMAND VARIATION

One of the most important conditions for any deposit efficient mining is justified determination of its operational capacity. At that the productivity of ore mining refers to strategical design solutions, which are very hard to be changed in case of necessity.

Operation in the contest of market economical relations puts mining plants in dependence on global mineral market situation, which is characterized by significant variability over the last ten years. Therefore when mines operate with a constant operational capacity, there are additional expenses related to storing of unsold final product in warehouses- in the period of demand decrease. Plants also lose the opportunity to increase the profit when demand is increased due to impossibility of production active intensification to cover the market demand. Due to that the objectives of local mining plants have been changed. Ensuring of competitiveness which depends on mining strategy became a priority.

Under these conditions the productivity of mine is to be adjusted to varying external conditions during its entire life time. It means to increase economical efficiency of mining is possible by implementation of flexible changes in mining volumes when demand is varied. The productivity is to be managed at the level of one open pit or group of them which belong to one Plant as well as at the level of Company Group with one owner in order to get the maximum profit. If the Plant owns several mines than the productivity of each one is to be defined based on the best performance of Plant.

Mineral demand increase brings to production volume increase (at that there is no change in stripping ratio). Demand decrease causes mining volumes decrease, equipment, buildings and facilities downtimes, manpower reduction, and equipment utilization time decrease. At the same time in order to decrease production cost the stripping ratio is decreased. However, the current methods for mining operations planning don't include any changes in mine productivity for ore extraction within long periods of deposit development. In addition there are no mechanisms for justified selection of open pit production capacity and operation mode, considering their interrelation along with iron ore demand variation. As a result, there is a delay in stripping operations, unscheduled temporary non-operational walls generation because of failure to follow the law of well-proportioned operations and mine development, and also, generation of temporary non-operational walls in operational zone which is unacceptable. Therefore, it is necessary to adjust mining operations to variable market conditions.

Due to that the productivity reallocation method, when iron ore products demand is varied, was developed for the group of open pits which are the part of one mining & benefication plant. By the example of Annovsk and Pervomaysk mines which belong to Severnyi GOK, there was ore mining productivity reallocation done without any changes in general strategy for final product production. In order to adjust ore extraction productivity to product demand increase or decrease, it is first required to define the mine maximum productivity based on mineral availability and also economic possibility, i.e. investments availability for plant capacity increase. In this case for each open pit there can be defined a scope of possible operation options which includes two ultra ones:

Mine operation with minimum stripping ratio and low ore extraction productivity.

Mine operation with maximum ore extraction productivity and high stripping ratio.

Within the possible options for ore extraction productivity and mining operation mode the best option of their combination is selected for long or entire period.

Iron ore concentrate demand can be decreased or increased. Therefore, apart from the strategy for ore extraction and concentrate production, defined by Design Institute, changes of net present value (NPV) based on concentrate production enforced changes for both cases were studied. At that, the issue of ore extraction productivity set level achievement was first considered, taking into account the current condition of mining operations in open pits and also operations mode change depending on ore extraction productivity change.

Ore mining productivity reallocation between Pervomaysk and Annovsk mines just by 1 m. in favor of Pervomaysk mine will allow increasing the profit of Severnyi GOK by 96 m.UAH. It is proved that the best reallocation for Severnyi GOK is the option when Pervomaysk mine operates with the maximum productivity and Annovsk mine ensures productivity of 9 m.t/y which is required to complete the strategy of Plant.

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DETERMINATION OF THE BOUNDARY CONTOURS OF THE OPEN-PIT MINE TAKING INTO ACCOUNT THE TRANSPORT PARAMETERS

Presently, the task of determination the boundary (end) contours of an open-pit mine is solved by comparison of the economic stripping ratio and incremental, the average, and / or current stripping ratio. This measure was introduced in the mid-twentieth century due to the complexity of the calculations associated with determining an economically viable depth of the surface mining by calculating the mining costs.

The introduction of these factors was aimed at simplifying the calculation of the limit depth of surface mining. It was assumed that the economic stripping ratio is constant. Various ratios were adopted for the development of soils and rocks as an exception. With this in mind, it should be emphasized that the economic stripping ratio value depends on the actual production cost of an extracted mineral, overburdened rocks and the acceptable production cost of an extracted mineral (for example, the cost of underground mining).

The academician Rzhevsky V.V. emphasized that the expenses for transportation of rock mass influence the increase in production cost of mining rocks with expansion of mining depth [1]. He noted that with the change of mining depth the error in calculations of deposit mining cost makes 3-15% and may not be taken into account. However, with increase of surface mining depth up to 500 m or more, the share of transportation expenses in the cost of rock mass production increases and currently makes 50-70% [2-4].

Therefore, the assumption that the economic stripping ratio is the constant is wrong and cannot be further used to establish the boundary contours of an open-pit mine. The attempts to apply correction factors to this ratio contradict the idea of application the stripping ratio – simplification of the procedure of determination the boundary contours of an open-pit mine.

Therefore, with the increase accessibility and simplification of operation of electronic computing tools during the determination of the boundary contours of an open-pit mine, it is more appropriate to comply with the conditions proposed by prof. Blyznyukov V.G. [5] that the actual mining costs of during the whole period of its operation should not exceed the eligible cost levels.

The value of eligible costs must be calculated taking into account the market conditions and the profit return ratio from the mining. The actual costs should take into account the cost of the surface mining operations in certain contours of an open-cast mine. Moreover, accounting for the above mentioned, it is recommended that the production cost of all processes, except transportation, be conditionally accepted as a constant value, and to take the cost of transportation rock mass increasing depending on the mining depth.

It is recommended to determine the position of the open-pit mine contours in space in condition of a maximal efficiency of mining. It is known that a combined scheme of transport is used in operation of steep-sloping deep-bedding deposits, which requires at least one board placed in the design position. Therefore, prof. Dryzhenko A.Yu. proposed to form a non-working board at the end of a deposit to make possible the creation of permanent transport communications, to achieve the lowest current volumes of overburden works and the possibility to make internal dumping under certain conditions [3].

However, the formation of a working board in the design position at the end of a deposit gives an incomplete determination of the open-pit mine contour position. Therefore, it is suggested to use the parameter b_x – the distance from the upper edge of the open-pit mine board in the bottom layer of deposit and in projected position to the point of intersection of the contour line of a mineral in the bottom layer of deposit and the surface.

For the same mining depth, the design of the board is assumed to be unchanged, regardless of its position in space. Therefore, the volume of rocks within the open-pit mine contours is equal. Therefore, the greater the mineral share in this volume, the more efficient is the deposit exploitation.

Based on the above, it can be argued that the position of the open-pit mine project contours in a plane at the same depth is determined by the parameter b_x, the value of which should be such that the mineral volume in the area near the open-pit mine contour is minimal. The area near an open-pit mine contour is a part of a mineral deposit that is not prepared for mining and to open which it is necessary to remove an overburden rock and which is located above the bottom level of mine [6].

After setting the optimal parameter b_x for the depths surface mining with an increment of 100 m, it is calculated the cost of a deposit development at these depths and compared with the acceptable one. Further calculations and comparisons are specified for depths with a step equal to the height of the bench (10, 12, 15 m). The mining depth at which the cost of surface mining is close to the acceptable is considered as the final.

Thus, in determining of position of the design contours of an open-pit mine it is relevant to set an optimizing parameter b_x, and to accept the mining depth accounting for the cost of extraction and transportation the entire volume of rocks and to compare it with the acceptable, which is determined by the market conditions and profit return ratio.

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TECHNOLOGY OF CARRYING OUT RECLAMATION AT THE TECHNICAL STAGE

Mineral mining is carried out in the areas with different land productivity. Mineral deposits suitable for an open-pit mining method are characterized by a wide variety of geological conditions, the development is carried out at various depths, mineral and overburden thicknesses, by various development systems: transport, non-transport, dump and various combinations thereof.

About 3-4% of overburden rocks of the current overburden is used for domestic purposes: for laying and filling rock voids, building roads and embankments, leveling the relief. But the use of overburden from already accumulated dumps is practically not carried out.

Due to the variety of mining and geological conditions, and, consequently, the applied development systems and their parameters, the disturbances of the earth surface by open pits are very diverse, both in character and in size.

Reclamation of the land disturbed by mining is carried out in all the developed countries of the world using modern equipment and technologies. The orientation of the reclamation work mechanization is selected specifically for each enterprise, depending on the technology of extraction of raw materials, the amount of work.

There are certain areas of reclamation: construction, agriculture, water management, fisheries, recreation, sanitary and hygienic.

Reclamation of dumps is usually carried out in two stages:

I – mining engineering, II - biological.

The main processes of mining engineering reclamation are the flattening of slopes, the layout of the upper surface of the dumps and the coating, if necessary, of the inclined and horizontal surfaces of the fertile soil layer.

The engineering phase includes preparation for subsequent targeted use in the national economy. It includes: planning, slope formation, transportation and application of soils and fertile rocks on the reclaimed land, construction of roads, hydraulic structures.

The engineering stage of reclamation is an integral part of the overall mining process. A number of works of the engineering phase, such as soil removal, formation of dumps of the necessary shape, construction of reservoirs and others are carried out in the process of mining with the main technological equipment.

The technology of reclamation works depends on the type of violations, the accepted orientation of reclamation and the equipment used.

The value of the advance removal of the fertile soil layer in relation to the upper overburden ledge, or the lower tier of the external dump should not exceed the annual advance of the front of mining (dump) operations.

The layout of the dump surface should be made in accordance with the accepted orientation of reclamation in two stages. The secondary planning is done after the final shrinkage of the blade, the period which is determined by the project but should not be shorter than two years.

The soil application to the surface of the dump is carried out only after the secondary planning.

Fire hazardous dumps are reclaimed only after work to prevent spontaneous combustion. The surface treatment of dumps with anti-pyrogenic substances toxic to plants should be injected to the depth of at least 3 m.

In the conditions of a soil layer absence and hilly terrain, the storage of rocks not suitable for biological reclamation and developing overburden dumps is affected not so much in the seizure of land but in deterioration of the environment. In such conditions reclamation should have an environmental landscape orientation (sanitary-hygienic orientation).

In order to fix dusty surfaces, in some cases there is used stabilization of surfaces by binders.

One of the largest coal deposits in Kazakhstan is the Ekibastuz coal basin. Total reserves are estimated at 10 billion tons. Based on the natural conditions of the region, we take the sanitary-hygienic orientation of reclamation. The purpose of the sanitary-hygienic orientation is to prevent the negative impact of disturbed areas on the environment.

So, on the dumps of the Ekibastuz basin open pits an effective binder compositionshould be considered the emulsion based on resin.

The composition is applied by an irrigation machine, and the resulting 2-3 cm thickcrust is elastic. The removal of a part from the fixed surface is not observed at the air flow rate up to 15 m/s; it is also recommended to use fixing reagents, which service life is about 2-3 years.

The area of the disturbed lands along the Bogatyropen pit amounts to about 1,500 ha. The overburden dumps of the open pit are composed of loams, quartz sand, sandstones, siltstone, and mudstones. The main factor limiting the possibility of biological reclamation isrock toxicity. The rock is characterized by the absence of humus, in most cases it is saline.

Therefore, the orientation of reclamation at the Bogatyr and Vostochny open pits is accepted as sanitary-hygienic.

As the information analysis on the state of reclamation work at the leading mining enterprises of the Republic of Kazakhstan shows, their situation requires improvement, both in terms of developing new scientific provisions, and in terms of practical implementation of the existing methods of reclamation work.

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FORMATION OF QUALITY MINERAL RESOURCES AND ORE OF MINING ENTERPRISES

The problem and its relation to the scientific and practical problems. Production of high quality iron ore products in the processing area to a large extent on the quality of the ore, which is consumed. The higher the quality of the ore, the smaller the value of irreversibly - a public work that is spent, and vice versa. When using low-quality ore, at equal cost of one and the same technique, technology and skills, ready to iron ore production, which is produced, is of lower quality, technical and economic indicators of production and consumption, iron ore than finished products, which is made of higher-quality ore. Therefore, one of the main issues the problem of increasing the quality of the finished products of iron ore is, the study of factors that shape the quality and the interaction between them. Assess the impact on the quality of the finished products of iron ore each factor alone is not always possible or quite hard, which necessitates the study of the influence of factors other than each factor separately.

Statement of the problem. Named the problem is quite relevant to the mining industry. Its successful solution is possible only on the basis of in-depth study and substantiation of functions and objects of geological surveying and security analysis of the factors that influence the formation and management complexity ore quality, choice of a rational set of mathematical methods for solving problems and building an efficient system of processing volumes of information using computer technology.

Presentation of the material and results. Typically ore mineral balance stocks are mineral complex aggregates which consist of a mixture of metallic and non-metallic minerals - silica, alumina, lime and magnesia, etc. In addition, ores contain varying amounts as useful (iron, manganese, nickel, vanadium, copper, titanium, etc.) and harmful (sulfur, phosphorus, zinc, lead and other) impurities. Useful impurities affect the quality of the metal, which is melted. They may be of value for their independent use. The nature of the action of harmful impurities diametrically opposite effect useful, so they must be removed completely or their contents should be brought to the permissible limits.

A characteristic feature is that the mining industry that raw ore, extracted even from one unit, characterized by variability in physical and chemical properties which are enhanced by the fact that the ore raw materials for processing, in particular to the processing plant comes from several units, horizons, mines, quarries and having a different enrichment. Variation of physicochemical properties of the ore requires operational variability or adjustment manufacturing process, otherwise it is connected to additional loss of ore, increases product yield (concentrate, agglomerate, pellets) lower quality.

Thus, the formation of end-product quality mining enterprises, next to a natural ore quality, the conditions of their occurrence, physical and mechanical properties and mineralogical components, provides a significant proportion of the impact of varieties of ores, which are applied to processing, and stability of their physical and chemical properties. The natural quality of the ore in the mining process takes variability depending on the destination, set the properties and requirements that are put forward by the consumer.

The main and defining moment in the formation of the quality of ore production is a process in which, joining together, the subject of the work, a tool of work and labor creates, in the end, the consumer properties of ore, which determine its quality.

The quality of the production process depends on the quality of the ore raw materials, equipment used and its settings, skills, quality management, which ensures the rational interaction between elements of the manufacturing process, stability, durability and efficiency of the production process.

The current practice of quality management ore is reduced mainly to the implementation of separate, unrelated events into a single system, with respect, the improvement of its quality. To create a quality management system of ore and targeted impact in promoting the quality of products is necessary to develop a program of action, the main provisions of which shall be: the goal to be achieved and activities to be developed and implemented; methodology to assess progress; events respect, ensuring stability and progress further progress. The quality management system, as well as any management system must include input and output, which are connected to each other backward and forward linkages.

The mechanism of action of the quality management system should be similar to the mechanism of action of a cybernetic system. Contradictions arising in the system through feedbacks, affecting entry system, restore the affected her balance.

Thus, when the consumer demands for quality ore variability required for the quality production, which in turn makes it necessary to influence the factors that determine the level, whereby, a new quality of manufactured products.

On the basis of the results of the consumption of the ore is again satisfied technological regulation of the finished product, which is manufactured.

This process is repeated until it reaches the desired level of consumer quality ore.

Thus, the basis and prerequisite for effective operation of the system of quality management of ore is an effective rapid feedback between the quality that is produced and consumed by which the possible provision of corrective influence on the course of the process for the manufacture of products of required quality.

A necessary condition for the normal functioning of the quality management system is the availability of timely and reliable information about the quality of ore that is mined and consumed, as between information management and quality of ore there is mutual dependence of communication - in the absence of sufficient information can not be an effective implementation of the process management, and in the absence of control loses meaning and information. In managing the quality of ore need to use both external and internal information.

Conclusions and direction of future research. The decision of the majority of these problems is possible only with the use of computer technology, which allows you to decide how organizational objectives and main tasks of process control.

In other words, the introduction of mining enterprises automated control systems, is, along with the improvement of engineering and technology in mines, quarries and mineral processing plants, the foundation for high-quality ore and production efficiency.

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NUMERICAL MODELLING OF THE INFLUENCE OF THE PLATE TAPERS ANGLE ON STRESS OCCURRING IN A CUTTING WEDGE OF DRILLING CUTTERS

Natural stones, such as marble, granite, limestone are traditional high-quality materials are widely used in construction and constitute an important source of economic wealth. The international natural stone quarry production sums up to 68.7 million tonnes per annum and is characterised as a low-tech, traditional sector with fragmented commercial activities and small size of companies [1].

Speaking about granite extraction, electrical energy, cooling water and steel are the major industrial requirements [2]. The resource efficiency of granite production chain is 0.31 [2]. Diesel fuel combustion for transport activities is the greatest contributor to GHGs emissions (68 %) compared to the purchased electricity and explosion process, with 31 % and 1 %, respectively [3]. Ukraine annually extracts about 104 million tonnes of granite; however the domestic mining industry still uses energy intensive techniques [4]. Implementation of the EC directives for building products and for energy efficiency in buildings impose the requirement for evaluation and reduction of the energy consumption in stone quarries. Therefore, despite minor share of drilling processes in energy consumption for granite extraction, it is actual to improve their energy and resource efficiency. Furthermore, in open cut operations the need to be competitive with world markets has place a heavy demand to be able to drill and blast considerable tonnages of ore in the quickest possible times [5].

There are a lot of factors of the drilling performance, for example, blast pattern [6], tools [7], pre-existing cracks [8] etc. In this article, the parameters of the drill cutters will be considered with the use of modelling.

The study of the processes of rock destruction by working tools (cutters) and the selection of their design parameters are complex tasks that are solved mainly experimentally. Modern methods of mathematical modelling eliminate the need for long and expensive experiments.

In this work, the influence of factors on stresses in the cutting part of drill cutters characterizing its strength was investigated. Using the developed numerical model, studies were carried out to establish the influence of plate size and sharpening angle on cutting strength parts to determine the maximum tensile stresses on the front edge of the incisor.

The studies were carried out with a chip thickness m=2, a back angle α=25 ° and with a variation in the point angle δ = 50- 95 ° with a step of 5 °. The values of the maximum contact height h, forces acting F_x, F_y on the cutting wedge of the cutter, and normal stresses $\sigma_{\max}$ arising in the reinforcing plate were calculated and summarized in Table 1.

Table 1

The results of the calculation

δ, deg	50	55	60	65	70	75	80	85	90	95
h, mm	0.80	0.84	0.88	0.92	0.97	1.03	1.09	1.17	1.26	1.38
F_x, N	290	337	385	414	476	575	679	788	995	1138
F_y, N	-70	-57	-32	9	42	101	180	286	464	657
σ_{max}, MPa	1117	1005	875	719	620	558	481	414	362	316

According to the results, the maximum normal stresses take place on the front face. At sharpening angles δ>90 °, the maximum normal stresses slightly decrease, but the cutting forces increase significantly. Consequently, it is impractical to use drill cutters with taper angles of more than 90 degrees, since they do not provide high work efficiency. It is possible to select the minimum allowable angle of sharpening of the cutting part of the drill bit for specified application conditions, characterized by the contact strength of the rocks of the field, under which its strength and high speed of drilling will be ensured.

Conclusions

1. Numerical studies of the stress-strain state of the cutting part of the drill bits make it possible to determine how the structural parameters of the drill bits have an influence on their strength and drilling efficiency. In this case, it is possible to determine the critical values of each parameter taking into account the strength of the cutter and drilling efficiency.

2. Modelling using the finite element method provides a fairly accurate solution to the problem of the stress-strain state of drill bits, which enables the design of new drill bits to select rational structural parameters of the elements of the cutting part under the condition of strength.

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INNOVATIVE DEVELOPMENT IN THE CREATION OF AN EARLY WARNING SYSTEM OF STRONG UNDERGROUND TREMORS FOR THE CITY OF DIAMONDS

General information on the area of works to the south-east of Almaty.

Data on the geological structure of the region, tectonic zoning, seismicity, availability of operating seismic stations, etc. will allow to place the points of the warning system in the most optimal places, taking into account the real seismotectonic conditions.

For the last 100 years the territory of Almaty and Almaty region has been affected by destructive earthquakes, such as Vernenskoye in 1887. (the epicenter was located on the territory of modern Almaty), Chilikskoye 1889. (the epicenter was located 150 km away from the city) and Keminskoye in 1911. (The epicenter was located 40 km away from the city).

The Institute of Seismology has identified seismogenerating zones in the Almaty region, timed to the epicenters of the strongest earthquakes, and it was established that the greatest seismic hazard is represented by the areas of Zailiysky and Kungei Alatau ridges, where the magnitude of earthquakes can reach the value of Mmax = 7.5-8.5.

The basis for the creation of an early warning system (SAO) of a strong earthquake is the long-term research carried out by the Institute of Seismology in the region. In particular, the results of research conducted by the Institute for the compilation of the Map of seismic zoning of the Almaty region [1] were used.

Thus, the region is characterized by a complex geological structure, due to the diversity of different age complexes of rocks, as well as the intensity of repeatedly manifested tectonic processes [3].

Seismically, the territory of the Almaty region and directly in the city of

Almaty. Almaty is one of the most seismically dangerous regions of Kazakhstan, so in accordance with the map of seismic zoning, the area of work is located in the zone of possible occurrence of earthquakes of 9 points on the scale of MSK-64.

Seismic generating zones (figures in circles): 1-Kemin, 2-Zailiysk, 3-Almaty. 92 - Zones of intensity of earthquakes on medium soils in MSK-64 scale, index 2 corresponds to the average frequency of occurrence of earthquakes 0.5 times in 500 years

Layout of early warning system (SAO) stations taking into account seismotectonic conditions

Taking into account the real seismotectonic and geographical conditions, the presence of active seismic and many other factors, the scheme of the Early Warning System (EWS) stations location was drawn up.

At a similar arrangement of points of supervision of system we will supervise seismogenerating areas - Kemin (partially), Zailiyskaya and Almaty representing real threat for region. The Kemin seismogenerating region is especially difficult to monitor due to its inaccessibility (high altitude) and the presence of the state border with the Kyrgyz Republic. The number of observation points included in the early warning system presented in this scheme - 39 points, with a distance between the stations 10-20 km.

A logical step is to use as a basis for an early warning system the network of seismic stations operating in the territory of Almaty region, where long-term monitoring observations are carried out and where there is a ready infrastructure (access roads, buildings, pedestals, electricity supply, communication channels, etc.).

It should be noted that the above scheme is the initial one and will be corrected during further reconnaissance works.

SRO station equipment.

The project's early warning system is based on the Geosig (Switzerland) GMSplus measuring system with integrated accelerometer AS-73.

This system is widely used in strong movement networks around the world, in particular, the seismic network of the NetQuakes project of the U.S. Geological Survey is fully based on these devices and currently has about 600 instruments. The GMSplus measuring system allows the use of all available data transmission, power supply and other possibilities. [2].

In 2018, Talgar (TLG), Tian Shan (TNS) and Medeo (MDO) observation points were installed and put into trial operation. In 2019, the equipment was installed at 3 more points of the system, including two points, Chilik (SHL) and Jarquent (DJR), where the equipment was launched. The work on installation and launch of the equipment at SAT is in progress.

Data transfer from the points is performed using GSM networks.

The system is configured to register earthquakes starting with M≥4.

The GeoDas-EEW software manages the operation of the stations and the system as a whole. The B-delta algorithm of the Japan Meteorological Agency (JMA) is used as the working algorithm of the early warning system. Figure 4 below shows a screenshot of the GeoDas-EEW program running in operation mode.

Green indicates normally operating stations, yellow indicates stations from which no data are available

Work on the establishment of a strong earthquake early warning system (SEA) has just begun. The stations installed and commissioned in 2018-19 are in pilot operation, and the system is being tested and configured in close cooperation with the Vulcan Research and Production Corporation (RF) and GeoSig (Switzerland).

The early warning system being developed at the Institute of Seismology has been integrated into the notification system of the KChS of the Ministry of Internal Affairs of the Republic of Kazakhstan. The received signal of early warning of an earthquake is automatically transmitted to the Emergency Situations Department in Almaty for further distribution.

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DEVELOPMENT OF RESOURCE-SAVING TECHNOLOGIES FOR DUSTYING OUT OF OPEN MINING IN THE CONDITIONS OF TAJIKISTAN

Mining in open pits causes various negative environmental impacts. Basically, such effects manifest itself in dust pollution, and the degree of pollution is directly proportional to the intensity of technological processes for the extraction and movement of rocks. So, during such technological processes as drilling, excavation, loading the rock mass from conveyor to conveyor, transporting rock mass by road, the dust emission rate is 1-10 g / s, and when crushing, averaging, loosening, blasting, and reloading from cars in the receiving hopper, this indicator is 10 g / s. The dustiness of the air in the excavator face during coal loading is 20 g / m3, and at the receiving hopper when loading it is 15-32g / m3. The volume of dust emitted by the explosion into the air reaches 17 g / m3. In iron ore quarries and coal mines, 70-80% of dust particles are less than 3 microns in size.

Specific indicators of the resource intensity of today's dust removal methods and tools vary up to 10 times. This suggests that there is a significant reserve for their improvement.

The intensive development of mining machinery and technology, especially in the last 20–25 years, has led to such significant dust generation that dust removal inefficiency can lead to irreversible environmental consequences. And also, standard technologies for dust control are largely obsolete, and the costs associated with dust removal are commensurate with the costs of mining and processing the rock mass. Thus, the problem of developing effective resource-saving technologies for dust removal processes of the mining industry, which is of great economic importance is becoming increasingly urgent.

In connection with the current situation in this area, studies of dust formation at various stages of field development, dust-forming ability of developed minerals, as well as studies on the development of methods and means of eliminating dust formation, it is necessary to give special attention, which could make a significant contribution to the theory and practice of solving these issues in open cast mining.

In order to minimize dust formation in open pits, it is necessary to create special modeling and design bases for dust removal technologies at various stages of open cast mining. Along with this, the application of stringent environmental requirements, the growth of costs associated with dust removal, scientists and engineers pose completely new tasks to create environmentally advanced development technologies and effective resource-saving dust removal technologies in open pit mining.

Methodically, the solution of these issues are based on the application of the theory of optimization of systemic and aero hydrodynamic approaches to the study of dust generation objects, equivalent circuits of specific dust removal devices, as well as the development of dust removal technologies are implemented, firstly, by the optimization of dust removal processes and secondly by their rational combination in the device.

To do this, need to solve the following tasks:

1. Substantiate the factors determining the choice of methods of dust control in mining processes.
2. Justify the optimality criterion for evaluating the recommended means of combating dust formation.
3. Assessment of the possibility of using water dust removal in conditions of varying temperatures on the basis of water-air jets, curtains, etc.
4. Consideration of the possibility of developing resource-saving technologies for the conditions of volley overloads of bulk material at varying temperatures.
5. Investigate the possibility of creating spray units for conditions of varying temperatures.
6. Investigate the possibility of creating aspiration systems based on the vortex effect.
7. To investigate the possibility of creating air and water-air curtains for dust removal systems with predetermined parameters.
8. To develop on the basis of the results obtained, resource-saving dust removal technologies in open mining works.

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INVESTIGATION OF PARAMETERS OF JET GROUTING DEVICE USING NUMERICAL MODELLING

Underground transport is a significant component of mass transportation of passengers, which is why it is built mostly in densely populated areas and central parts of large cities of Ukraine [1, 2]. Problems of efficient construction and maintenance of tunnels are relevant for Ukraine considering the scale of underground transport system and growing passenger traffic.

Soft soil improvement technologies by chemical reactions such as deep mixing and jet grouting are commonly used to improve the soft deposit to ensure safety during construction of tunnels [3, 4]. Typical applications of jet grouting are embankment foundations, water sealing bottom plug and excavation support, provisional tunnelling support and water cut-off [5]. In our investigation, the jet grouting is applied for maintenance of existing underground tunnels. Similar positive experience was described in [6].

The essence of the jet grouting method is the parallel destruction of rock and mixing of soil and stabilizing solution in mix-in-place format. It saves resources by performing two technological operations at the same time.

The drilling tool for the jet grouting is designed differently from the standard tool due to the presence of a channel for the supply of stabilizing fluid. Upon reaching the end of the screw, the cement slurry is fed into the inside of the monitor. The monitor is equipped with nozzles that convert a high-pressure energy into kinetic energy of the flow, which, in turn, destroys and mixes the soil. To cut the soil, the flow rate must be at least half the sound velocity [7]. It affects the diameter of the column [8-10], its composition and properties [11], etc.

Described above drilling tool for the jet grouting has sophisticated design. Therefore, it is hard to calculate mixture flow parameters using theoretical approach. Experimental or simulation data are usually used.

In this investigation, 3D-model of jet grouting device was developed to calculate the pressure that needs to be created in device to achieve mentioned above velocity of the grouting mixture. The material of jet grouting device (drilling tool and hydraulic monitor at the same time) was structural steel, diameter of the device – 100 mm. The inlet diameter of the hydraulic monitor's nozzle was 19 mm and the outlet diameter was 2 mm. Feed channel diameter of the device was 12 mm.

Further numerical simulations were aimed on calculating the pressure p and fluid velocity v inside the jet grouting monitor depending on position x. The results are shown in Table 1.

Table 1

The results of the calculation

x, mm	0	5	8	12	23	27	30	39
p, bar	-	166.7	169.3	172.8	174.7	174.9	174.9	175.0
v, m/s	132.4	50.2	37.9	24.4	5.9	2.4	2.4	0.8
x, mm	44	63	67	71	77	91	97	105
p, bar	175.0	175.0	175.0	175.0	175.0	175.0	175.0	175.0
v, m/s	0.3	0.1	0.1	0.1	0.1	0.7	0.1	1.9

The developed model and calculations show that to create a flow rate at the output close to half of the sound speed, namely 130 m/s, it is necessary to create a pressure of 175 bars. It is in line with other investigations [3, 5] and close to carried out theoretical calculations. Therefore, all the devices in hydraulic system should be designed for the high pressure.

Conclusions

1. Numerical modelling of jet grouting device simplify the task of calculation of its hydraulic parameters like resistance, velocity, pressure etc. In addition, it assists in strength calculations to ensure the safety and durability of the device.

2. In order to obtain the speed of grouting mixture that close to half of the sound speed, it is needed to feed the mixture with a pressure at list 175 bars.

3. Further investigations should consider the whole hydraulic system with a pump capable to create the flow with pressure and productivity to reach the speed of grouting mixture in the monitor's outlet nozzle required for proper soil cutting and column formation.

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ENERGY-EFFECTIVE METHOD OF INSTALLATION OF FOUNDATIONS UNDER OBJECTS OF MINE SURFACES

Technological complexes of mines and metallurgical enterprises in various combinations include: buildings of air heaters, fans, electrical substations, crushing and screening complex, compressor plants, repair and electromechanical workshops, etc.

Among them, we can conditionally distinguish:

- the main buildings and structures that are directly related to the technology of extraction and distribution of minerals;
- auxiliary - not directly involved in the technological scheme of the movement of minerals.

One of the possible ways to improve the foundations for mine surface objects is the use of thin-walled spatial reinforced concrete shells.

The currently used monolithic foundations for columns of industrial buildings, along with well-known advantages, have a great complexity of erection on a construction site and a high consumption of materials.

The use of monolithic thin-walled spatial reinforced concrete shells for objects of mine surface will reduce the volume of earthworks by 3...4 times, eliminate formwork almost completely compared to conventional tape, columnar and pile foundations, reduce concrete consumption by 2...2,5, metal about 2 times, labor costs and estimated cost of zero-cycle work by 40% [1, 2].

When the envelope are immersed, under them and around them, a compacted zone with increased strength characteristics is formed, due to which their bearing capacity is significantly increased.

When arranging foundations from monolithic thin-walled spatial reinforced concrete shells, all technological processes and operations related to dragging, installation at the dive site, guidance, orientation and immersion of the shells are performed by copra or by copra equipment [3].

The study of the process of immersion of monolithic thin-walled shells in the form of a truncated cone in the ground was carried out in the immediate vicinity of the buildings and structures being erected on the construction sites of the cities of the Dnieper and Volnogorsk.

To clarify the qualitative laws and physical essence of the process of immersion of shells in the form of a truncated cone, a special stand was used, consisting of the following main parts: anchor piles, reinforced concrete blocks, transverse beams, the shell under study, headband with a core, a bearing slab and a jack.

Immersion efforts were determined using a manometer according to the developed technique.

When the envelope were immersed in subsidence type II soils, the soil was soaked. For this purpose, wells with a diameter of 200 mm and a depth 2 times the height of the envelope with a pitch of 2 m. were drilled around the studied envelope in a radius of 2 m.

The distance between the axes of the experimental shell and anchor piles should be determined by the dependence L = 2D, where D is the diameter of the upper section of the shell. borehole are combined by drainage trenches and filled with sand or grave.

As a result of the experiments carried out in the conditions of various construction sites, studying the density of loesslike and clay subsidence soils at the base of the immersed envelope, it was found that when the area of the lower base is 0,2 m^2, the compacted zone extends to a depth of 0,8 m from lower cut of the envelope.

With an increase in the area of the lower base of the envelope to 1 m^2, the compaction depth reaches 2,1 m.

In its shape, the compacted zone in various soils around the dived envelope approaches an ellipsoid of revolution whose large axis coincides with the vertical axis of the envelope.

In this case, a substantial part of the densified zone is formed under the base of the envelope.

An analysis of the studies showed that when elements with a base area of at least 1 m^2 are immersed in loesslike loams to a depth of 1 m, they become denser and particles move down and to the side.

Herewith, a compacted zone is formed in density from 2,0 g/cm^3 in the densest zone to 1,6 g/cm^3 of density, which corresponds to the natural composition of the soil.

Further immersion of the envelope occurs with an increase in the transverse and vertical dimensions of the densified zone. Upon reaching a depth of 2 m, the dimensions of the densified zone stabilize, i.e. the process of its formation is being completed.

This depth is considered optimal and is determined by the expression: $h_д = 1{,}9 d_н$. The diameter of the compacted core is equal to: $d_я = 1{,}1 d_н$.

Since it is impossible to visually observe the process of deformation of soil layers and the formation of a compacted zone when immersing the envelope in the soil when conducting experiments in the conditions of the construction site, studies were conducted on a special bench in laboratory conditions on gypsum envelope models using photograms of the study process.

These data confirm the physical nature of the process, established in the course of research in natural conditions.

As a result of the research, a methodology has been formulated for determining the basic parameters of equipment for immersing envelopes.

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SECTION "MINE SURVEYING"

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THE CHOICE OF METHODS FOR OBSERVATION OF ROCKS AND EARTH SURFACE

Using modern instruments for measuring distances, angles and elevations, as well as GPS for coordinating points, methods for observing the movement of rocks and the earth's surface should be improved.

In the current instruction and methodological guidelines for monitoring deformations, the recommended methods for taking measurements and evaluating results are based on the use of linear measurements between benchmarks and determining their altitude from geometric leveling. The resulting line lengths between the benchmarks and their marks allow us to calculate the horizontal and vertical components of the deformations, which are compared with critical values, indicating the presence of a zone of a dangerous effect of underground mining.

It is known that the values characterizing deformations are: shrinkage (η), curvature (k), slope (i), tensile-compression (ε), calculated from changes in elevations and lengths of segments over a certain period.

Today, when a more effective way to determine the position of benchmarks in plan and height is to coordinate them using GPS, the urgent issue is the transition from the critical values η, k, i, ε adopted in the instruction to the critical values and directions of the vectors reference offsets.

At twenty-meter intervals between benchmarks, the critical value is the slope $i=4\cdot10^{-3}$, and the difference in shrinkage at adjacent points is 0.08 m. At a critical value of tension-compression $\varepsilon=2\cdot10^{-3}$, the distance difference between the benchmarks at the initial and current moment is 0.04 m, and at a critical value of curvature $k=2\cdot10^{-3}$, the difference in the slopes of adjacent lines is 0.004. The critical value of shrinkage on the benchmark is 15 mm, corresponds to the previous statements about the difference in precipitation of adjacent benchmarks.

An analysis of these data allows us to conclude that it is possible to determine the critical values of the displacements of the benchmarks in the plan and add them to the current instruction.

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CAPEX AND OPEX: THIS INDICATORS BE RESPONSIBLE FOR TAILINGS DAMS?

The mining process is very old, and records of this activity can be found in times like 1556, when Agricola launches in this period his book: De Re Metallica. Surface (or open pit) mining can be considered the percussion for so many others, since the scarcity of technologies and materials for extraction, gold and copper attracted attention at that time. In Brazil it was no different, alluvial gold caught the attention since the "discovery" of the country and attracted the attention and propitiated the development of cities like Catas Altas, Rio Piracicaba, Governor Valadares and many other cities in Minas Gerais, State that carries in the name the mineral activity.

There is in the State of Minas Gerais a region called Quadrilátero Ferrífero, which is located in the southeastern portion of Minas Gerais and occupies an area of approximately 7,000 km2. This region is named after iron ore deposits that occur in an area that has the cities of Itabira in the northeast, Mariana in the southeast, Congonhas in the southwest, and Itaúna in the northwest (GEOPARK, 2019).

Technology advances over time and its advancement is also the result of the discovery of new minerals, which have different uses and allow the creation of new technologies. In mining, technology has allowed poor ores, such as Itabirito, to be benefited and sold. An adversity in this system is the generation of materials without economic value: the tailings.

The process of extracting mineral resources from the crust basically generates a concentrate (ore), sterile and tailings. The last two should be arranged in a controlled and / or reused area, thus minimizing the impacts generated by the environment. The sterile, in most cases, is arranged in the form of piles, which when finished, are lined with grass. , legumes or bio-blankets, which assist in the stabilization of the disposed mass, reducing the chance of erosion, etc. Already the tailings are commonly disposed in dams, called tailings dams (De Abreu, I.F.; Discacciati, G.C.P .; *et al*, 2014).

Both the dams and the waste piles occupy large areas that, previously occupied by vegetation and diverse fauna, become a dedicated place for such storage and that is the result of anthropological action, with no possibility of later use. Two major tragedies linked to the issue of dam rupture were observed in Minas Gerais: the rupture of the Fundão dam (Mariana, 2015) and dam I (Brumadinho, 2018). The first breakup raised a big question: are these structures really safe? And then another, post-rupture: is there no other method to avoid dams?

It is unclear why miners do not use any other technique, but what it looks like is that it is tied to CAPEX and OPEX, which is not cheap (dam OPEX is cheaper) and makes us reflect.

The OPEX of a dam does not include energy consumption, filter materials, mechanical parts and the like, as dams are simplified by an area (trough) that receives the material and opex for magnetic concentration still shops the weight of processing route change - beyond the factors related to capex.

Regardless of the reason, it is known that the tailings filtration method is a solution and has been used successfully for a long time by iron miners (which is the main activity of the state), such as Vallourec in the Pau Branco Mine, which still adds use to this dry tailings, making it co-product. One challenge of this process is pulp density, which affects the productivity of the filter press.

In Brazil there is also the effort of companies, such as New Steel (acquired by the largest mining company in the country), in the search for the solution of this problem called dam, but which has added to the effort the choice of the dry route, as observed in your institutional site. The process, with magnetic concentration, excludes the existence of dams and is a breakthrough in facing the water crisis.

Based on this discussion, it is necessary to bring to society the real reason for choosing the current iron beneficiation routes, the challenges that mining companies have been facing and what is being done to overcome them. Only in this way can mining really be trusted again by civil society.

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GEOMETRIZATION AND PROGNOSTICATION OF GEOLOGICAL PARAMETERS OF IRON ORE DEPOSITS USING NEURAL NETWORKS

Ukraine is a country whose earth bowels have unique mineral deposits in terms of chemical composition; unfortunately, in terms of quantity, the deposits are limited, as are the amounts of useful components in them.

Resulting from the process of their exploration, the mineral deposits are regarded as natural systems with a predominant stochastic character. The model approach is basic because of the indefiniteness in the whole continuity "exploration – mining – preservation – liquidation". The purposes, tasks and interests of a variety of specialists form the links and the elements of a system which is financially and administratively managed by a concession owner. The principal requirement for these two systems to function in unison is to observe the safety regulations, to extract deposits fully and rationally, and to preserve earth bowels. In those two systems, definiteness becomes an element at a different stage.

As well as the methods of mathematical statistics applied mathematics used in mineral deposit investigation, geo-statistical and heuristic methods are also employed. The purposes, matter and tasks of Artificial Intelligence (methods and means which imitate the role of human intelligence) were outlined in the second half of the 20th century.

The main reasons for artificial intelligence to be used in mineral deposits investigation are: understanding and implementing of information technologies, deposit parameters modeling, evaluation of the models and using them in the designing and mining stage, earth bowel preservation, elaborating a great number of solutions and choosing the optimal solution in real time, and perceiving mineral deposits as multi-functional and multi-parametric systems.

In general, the issue of prognostication is reduced to obtaining evaluation of future values of data which are well-ordered in time and/or space and are based on already known data. Recently, along with the traditional methods (parametrical models) for regularity searching in the known data, are methods have been applied that use neural networks.

The proposed approach was used for prognostication of geological parameters which are the basic factors in the technical and economic evaluation of mineral deposits, as well as in planning the extraction operations in a mine.

Part of an iron deposit with a complex structure, extraction level was explored with a regular drill hole grid (100×100 m). Because the grade changeability is high, most of the grid density was increased up to 50×50 m. The results from mine-geometrical analysis of the basic parameter – grade of iron – show that the empirical distribution is quite different from the normal and has positive asymmetry. Investigating the character of grade changeability by the random functions theory gives reasons to affirm the presence of anisotropy in the variability of the basic parameter.

Reports can be found in the bibliography about using different types of neural networks in similar conditions. Uniform recommendations, however, cannot be found in them. Most of the investigations are often made on the artificially created mineral deposits.

Considering the quantity of initial information, the sequence of its providing and entry, and the price of unit of information, the following question is asked: "Can we train a neural network with data from the first stage of exploration so that it could prognosticate the basic parameter of the deposit - the grade of ore in the sample locations that complete the termination of the second and third stage - and what is the influence of the changing ratio of “training/testing” data on the prognostication error?”

If the second stage of exploration is partly realized, a second question arises: “With the same density of drilling grid, but with a parallel movement, do the type of neural network and its working capacity for the next stages of exploration change?”

A possible third question arises then: “Where is the threshold if information quantity resulting from the first and second stage of exploration, which provides a reliable choice of models for prognostication of geological parameters value at the third stage exploration?”

All the operations of searching for an answer to the second question are reflected in the construction of prognostication models – neural networks - through training based on the output information (located in a grid 200×200 m) and comparison of training quality.

Designing neural networks involves the following stages:

- determining a series of configurations (number of layers, number of nods in every layer, etc.);
- performing iterative experiments with every configuration;
- assessment, correction and completion of the training of selected configurations;
- final testing of the best neural network selected.

Statistical Neural Networks includes a tool for automatic creation, training and testing of a multitude of networks with different characteristics, and for selecting the best network according to chosen criteria. Experiments are made with various types of network containing different parameters (number of hidden layers, methods and phases of training, transfer functions, etc.). A number of training series with every network testing structure are performed. Applying the selection error appraisal criterion, the best network is chosen as a model. This method comprises the most advanced algorithms, such as regularization and analysis of accuracy. It can test hundreds of network combinations, concentrating on particularly promising network architectures. This method searches for optimal networks of different types (e.g., multilayer perceptions and radial basis functions) simultaneously. Another advantage of this method is the high performance speed, even if the answer is crude.

The information for iron grade includes the coordinates of the prospect hole (within the extraction level area) and the value of useful components. Thus, the neural network will have two input variables and one output. During using this method analysis, the data are divided into three groups – for training, for testing in the process of training, and for post-training testing.

So far, neural network application has not been employed extensively in estimations and prognoses concerning mineral deposits. Besides, the unique character of natural conditions, varying even within the same type of deposit, requires experimenting with different types of prognostication model.

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THE RELEVANT IMPORTANCE OF THE ISSUE OF ENHANCING THE EFFICIENCY OF ORIENTATION AND CONNECTION SURVEYS AND HEIGHT BENCH MARK TRANSFERS

Transferring height bench marks into a mine's underground workings and orientation-connection surveys are specific types of mine surveying that impact the accuracy of the surveying support for underground mining and it is therefore necessary to take into account the particular importance of their performance.

The increasing depth of mining makes rock bursts and other cataclysms more likely, so the issue of enhancing the accuracy of determining the height bench marks used to solve a wide range of problems in underground mining is becoming more relevant.

As is known, the orientation of underground mine survey networks completes two tasks: the transfer of the directional angle (network orientation) and that of the planned coordinates X and Y (network centring). The transfer of the height bench mark (the third coordinate) information is carried out independently of the results of the orientation-connection survey, since it involves the use of completely different instruments and equipment.

The aforementioned types of operations are usually performed by using known conventional methods. The main disadvantages of those methods are the considerable complexity of work, the necessity of introducing a large number of adjustments into measurements (for temperature; for comparing tapes, wires, measuring disks; for expansion due to the own weight and the suspended load, etc.) and, as a consequence, a high error rate of determining the height difference between the bench marks. Other disadvantages include limitations of using traditional methods due to the considerable depth of mine operations.

Quite frequently, a mineshaft has to be stopped from operating for a day or more, which results in significant financial losses and the impossibility to use the shaft as an emergency exit.

When making adjustments, there must be a strict compliance with the reference samples of the materials used for manufacturing parts of measuring instruments, which is almost impracticable. The fact of a slight mismatch between the differences in height when retransferring a bench mark data is therefore not a sign of the high accuracy of determining the difference in height between the bench marks on the surface and in the mine.

When orienting mine workings, problems sometimes arise with lowering plumb bobs into the shaft and with the bobs' oscillations, which significantly complicates measurements. Similarly, the probability of foreign objects accidentally falling into the shaft from the mine surface or from the mine's upper levels constitutes a heightened risk of work on a lower mine level, while the separate performance of the related horizontal and vertical surveys increases the complexity of work and the duration of the entire set of operations.

The use of gyroscopic orientation and laser scanning systems is almost indispensable at depths of underground workings of more than 500 m, since it is not dependent on installing plumb bobs in mineshafts and stopping the descent or ascent of people or loads. Specialized organizations usually perform this kind of work. They have the appropriate equipment such as gyro-theodolites, gyroscopic fixtures or lasers. Such operations are quite expensive and not every mining company can afford them. However, the qualification of surveying personnel of most mining companies makes it possible to perform the orientation independently, so the traditional orientation methods have not lost their importance, and are used quite often.

In order to enhance the efficiency of mine orientation and connection surveys as well as height ones, a comprehensive new method is required that would improve the accuracy of transferring spatial coordinates and directional angles and would be significantly more cost-effective for companies.

Theoretical studies in this area have shown that it is possible to carry out orientation and connection surveys simultaneously with the transfer of the height bench mark information, and this can be done by using digital photography and processing the obtained images with the help of digital photogrammetry methods.

In order to make a transition from theoretical research to its practical implementation, it is necessary to investigate the issues concerning the choice of the most advantageous geometry of the mine's vertical shaft entry figure and to substantiate a correct number of binding bench marks. That should be done considering the impact of underground mining conditions on the modern photo cameras and the minimization of error sources.

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PREDICTION OF MINERAL RESOURCE QUALITY INDICATORS IN BALANCE RESERVES

The problem and its relation to the scientific and practical problems. The problem of forecast evaluation of the mean values of geological features in the interior - one of the most important functions in the implementation of geological surveying and quality control of ores and ore processing. The use of methods based on the theory of random functions, gives satisfactory results for blocks of small size, low "lit" intelligence, which most often occur in existing enterprises.

Analysis of studies and publications. Forecasting issues of quality indicators ores in the bowels of the leading scientists engaged Margolin O.M., Ershov V.V., Ushakov I.M. and others. Prompted a lot of prediction methods, most of which are not effective from the point of view of the estimation error, since it does not take into account the nature of the variability in performance.

Statement of the problem. Objectives of the evaluation of quality indicators in the depths of ore can be successfully solved the equations and Kriging estimation of the variance if the model of the placed sign in the space does not contain a legitimate component of $C_T(X,Y,Z)$, and the autocorrelation function or private function is defined.

Presentation of the material and results. There are two variants of the method indicated-mentioned conditions. The first is related to the exact solution of the Kriging system of equations (discrete Kriging), the second - with the possible simplifications based on the physical meaning of (random Kriging).

Simplify equations may, in particular, in such situations, when the result of evaluation to conduct an obvious calculation. Suppose, for example, the linear dimensions of the block, which is estimated to have the same order and that the average distance between the points of testing. Then it is clear beforehand that the evaluation unit will make an important contribution, only those samples that are closest to the center of the assessed unit, ie weighting samples, withdrawn from the unit will be equal to zero. This conclusion is confirmed by the exact solution of the Kriging system of equations that take into account all the original data. The second option involves the assessment of relatively large blocks. In this case, it is expected that all the samples that are within a block which is judged to have approximately the same weight. Assays peripheral unit will also include in the evaluation with the same weights of different weights in the middle of the sample block.

Thus, the conclusion about the possibilities of simplifying the equations of Kriging, and hence the estimates of the mean values of the index should be based on a study of the variability in, and on the analysis of the geometric characteristics of the blocks, which are estimated parameters intelligence networks. Calculations show that for small blocks rather bring to evaluate three - six nearest samples. In this case, the regular testing of networks for coefficients can be made in advance for the most common configurations of the relative position of the block and samples that are involved in the assessment.

If you are using computer technology to find estimates of mean values of quality indicators in the unit in the event of deviations from the standard conditions are algorithmically easier to solve by setting the number of samples that are involved in the assessment, or the boundaries of the halo within which the samples are attracted to the evaluation. And in fact, and in another case, the order Kriging system of equations does not exceed six.

If the linear dimensions of the blocks exceed the average distance between samples several times, and the volume of units is much smaller than the volume of the ore body in which they are found, the assessment can be simplified. Evaluation parameters and block Kriging variance depend on the number of samples and to the characteristics of the autocorrelation or structural functions.

Thus, when evaluating large and small blocks of bulky Kriging equations that come out of the general theory, are greatly simplified. Calculations can be performed quickly with no regularity in the distribution characteristic component $C(X,Y,Z)$.

In the presence of a trend, all the above approaches are ineligible. In this case it is necessary to allocate the trend for example, using the method of least squares. The coefficients of the equation of the trend, caused by this method are not biased, but they do not minimize the variance, if the deviation from the trend of characteristic values auto correlated. In determining the deviation of the trend can be examined for autocorrelation. With its presentation of autocorrelation coefficients, which are calculated from the difference would be biased.

Geostatistical calculations for deviations from the trend, which is caused by the least squares method may not be effective, you must also take into account the autocorrelation of deviations in the selection equations of the trend. This approach to the assessment of performance in geostatistics is called universal Kriging. Registered This is not necessary if the number of testing points greater than 100. Otherwise, modify the approach to the assessment, which leads to Kriging equations.

The above methods of geostatistical evaluation of quality indicators (discrete, random and universal Kriging) is the most effective (in terms of accuracy) in a geological and statistical homogeneity of the blocks, which are estimated. However, most of these conditions deposits rarely performed, which leads to the necessity of dividing the block into homogeneous volume. Recently proposed modification Kriging - indicator Kriging that allows you to get effective geostatistical evaluation of the quality in the blocks, which are made up polytypic ores. The method is as follows. In order to justify inclusion in the system of geological surveying and quality control of ores and ore the most effective (in terms of accuracy) Mathematical methods for predicting the quality of a large amount of factual material on the exploited ore deposits, we compared the above modifications Kriging of the three most frequently used in practice traditional methods: weighted arithmetic; weighted back to the squared distance, taking into account the weighted anisotropy.

Conclusions and direction of future research. The comparability of the results on the accuracy of traditional and geostatistical methods in the evaluation of ore deposits in the depths contradicts the theory of optimal statistical estimation and can be explained only by the unfortunate choice of the variability in the model or some of its parameters. Detailed analysis of the variability in the specific conditions and evaluating the quality of the blocks must always be preceded by reasonable selection of the mathematical method of forecasting.

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RECOMMENDATIONS FOR PREVENTING PROFESSIONAL DISEASES FROM EXPOSURE TO NOISE AND VIBRATIONS IN MINING AND CONSTRUCTION ENTERPRISES

The employer is obliged to conduct effective monitoring of working conditions, the process of which is associated with such hazards as increased noise and vibration. For example, vibration control of machines and mechanisms, which is carried out constantly (once a year), selective control after each repair or when the mechanism changes (state sanitary standards SSS 3.3.6. 039-99 clause 9.6).

Preventive measures can prevent the appearance of diseases from exposure to noise and vibration.

In Ukraine, the minimum safety and health requirements when using personal protective equipment at the workplace are indicated in regulatory document No. 1804 of November 29, 2018. These minimum requirements are developed based on the EU Directives, namely: No. 89/656 and part 1 of Article 16 № 89/391. In addition, general and local production vibrations are regulated by state sanitary standards No. 39, Recommendations for the protection of workers from occupational hazards No. 156 and Recommendations on the selection, use, appearance and maintenance of hearing protection (EN 458: 2004, IDT) DSTU EN 458: 2005, № 187.

In accordance with paragraph 5 of the Minimum Requirements No. 1804, hearing protection in case of exceeding the noise level of more than 80 dB, it is recommended to use headphones, earbuds, and noise protection helmets - for such work:

- on metal presses;

- with pneumatic tools, perforators;
- during operation and maintenance of pumping equipment;
- with copra for clogging pallets;
- ground personnel at airports;
- in the woodworking industry.

For each employee, his / her card of personal protective equipment and a table of risks should be compiled to justify the selection and use of personal protective equipment following normative document No. 1804.

To protect the hearing organs of personnel from the negative effects of various noises, personal protective equipment must be used, which must comply with EN 352-1 and EN 352-2.

To protect the face and head mast be used anti-noise headphones complete with helmets and shields.

The analysis of statistics of noise and vibration occupational morbidity among workers in the mining and construction industries allowed us to develop the following measures for the prevention of diseases.

1. Each type of protection (headphones or earbuds) should be selected for specific working conditions depending on the prevailing frequency range of noise exposure at the workplace (Table 1).

Table 1

Technical characteristics of personal protective equipment

Frequency Hz	63	125	250	500	1000	2000	4000	8000
Headphone/Antinoise Performance								
Attenuation coefficient, dB	4,0/ 33,4	3,1/ 34,1	1,9/ 35,5	8,6/ 37,6	29,0/ 34,9	29,7/ 35,7	37,1/ 42,5	31,9/ 44,1
Deviations, dB	2,0/4,6	3,6/4	2,4/4,6	2,7/4	3,0/5	3,2/2	4,0/2,9	6,3/4,2
Protection margin, dB	9,0/ 28,8	9,5/ 29,4	9,5/ 30,9	15,9/ 33,5	26,0/ 29,9	26,5/ 32,9	33,1/ 39,6	25,6/ 39,9

2. Headphones are recommended for the following types of work (at a noise level of 120-125 dB): in stationary mine installations, tunneling pneumatic equipment, during drilling, and during compressor maintenance.

Reusable liners (made from flexible polymers) are recommended for noise protection at 105-110 dB. Disposable liners are recommended for workers in mining and construction enterprises when exposed to industrial noise with a level of 100-105 dB.

3. Miners and builders who work directly with sources of increased noise and vibration should comply with recommendations on the regime of work and rest (Table 2).

Table 2

Recommendations of work and rest for workers working when exposed to noise and vibration

№	Mode of operation	Measures to prevent the negative effects of noise and vibration
1	monthly	Uniform distribution of days off and days of work. The total monthly duration of work with negative factors is no more than 18 days.
2	weekly	The transition to different work shifts must be done every 5 days.
3	daily	Prohibition of excess work. Reduced night shifts. Rest between work shifts for at least two shifts.
4	shift rotation	Distribution of the technological operating time schedule with constant contact with noise and vibration - with an obligatory double additional break. The first break must be lasting 20 minutes - after the first two hours of work. The second break must be lasting 30 minutes - after two hours of afternoon work. For work with noise and vibration that is not constant during the shift, it is recommended that it must be regularly alternated work with and without negative noise and vibration factors.

4. Constant monitoring of the health of workers and regular preventive physiotherapeutic procedures for their recovery is necessary.

Compliance with the proposed recommendations will prevent possible occupational diseases from exposure to noise and vibration, extend the able-bodied age of skilled workers and improve the work culture.

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LOGIC AND LINGUISTIC MODELING ACCIDENTS AND INJIRIES

The most promising approach to taking into account all the essential factors in predicting anthropogenic risk is logical-linguistic modeling of incidents in human-machine systems. Its advantages are high flexibility and focus, the ability to take into account an arbitrarily large number of important factors, the speed of evaluating the nature of the system's response to the proposed change in individual properties of its elements or their assemblies. Logical-linguistic modeling for the study of human-machine systems must take into account the influence of the psychophysiological properties of personnel on their implementation of activity algorithms.

The most typical causal chain was a sequence of prerequisite events of the following form: *a* - human error and/or failure of process equipment and / or an external impact unfavorable to them; *b* - the appearance of a hazard in an unexpected place and/or at the wrong time; *c* - malfunction or lack of protective equipment and/or inaccurate actions of personnel or unauthorized persons in this situation.

The need to improve the professional suitability of personnel as the main component of the human-machine system and the main element of any organization involved in ensuring safety is due, first of all, to the high importance of the human factor, including in the formation of the conditions for the occurrence of accidents and injuries. Among the reasons for the high importance of the human factor in technological events, a special place is occupied by the mismatch between the requirements of modern technology and the psycho-physiological capabilities of human.

Studying of the influence of individual human properties on organizational characteristics explains the nature of the uncertainty introduced by them into the behavior of the human-machine system. This uncertainty comes from the peculiarities of each person's interpretation of the information available to him. This process is not statistically reproducible and cannot be described by exact quantitative methods. Therefore, it is more advisable to use approximate estimates based on the theory of fuzzy sets, which is based on the presentation of elements of human thinking not by numbers but by objects, when making the results of professional selection.

In models of accidents and disasters formed on the basis of the fault tree, with the possibility of an approximate estimate of the values of the probabilities of failures at the lower level elements, fuzzy modeling can be applied. In this case, using the random number sensor, you can set the possible values of these probabilities, and then, using the model of the fault tree, determine the corresponding probabilities of the main event. The random number sensor is used to form various combinations of the probability of failures at the lower level elements, and the event tree is used to evaluate the corresponding accident probabilities and the spread of these estimates. For all assessments of the quality of personnel actions, a triangular membership function of the form is used

$$\mu_i(C_i) = \begin{cases} \dfrac{C_i - a_i}{m_i - a_i}; C_i \in [a_i, m_i] \\ \dfrac{b_i - C_i}{b_i - m_i}; C_i \in [m_i, b_i], \end{cases} \qquad (1)$$

where

i - the level of quality of staff actions (i=1,2,3,4);

C_i - is the level of complexity of the situation;

a_i,b_i -lower and upper boundaries of distribution at i - m level of quality of personnel action;

m_i - the modal meaning of linguistic quality assessment;

$\mu_i C_i$ - some measure of the possibility of being in a situation with a i - level of quality of personnel actions.

Substituting the found estimates of the complexity of the situation into expression (1), we can obtain the corresponding values of the probabilities of transferring the failure to the next level of the system. The objects of imitation can be: the law of distribution of the quality of personnel actions; type of membership function $\mu_i(C)$ distribution measures the complexity of the situation.

Assuming that the complexity of the situation is determined on the [0;1] scale, we can distinguish 13 levels of complexity (0,0.09, 0.17, 0.25, 0.33, 0.41, 0.5, 0.58, 0.66, 0.75, 0.83, 0.92,1) on the quality scale, for example, located at equal intervals of values (poor, satisfactory, good and excellent staff actions). The random number sensor is used in assessing the probability of accidents in human-machine systems, forming with it the variants of the probability of human errors at different levels of control of the system, incorrect decisions to eliminate defects in the system, etc.

Using the sensor, various options for the values of the probability of failures at the considered level of the system are set. Given the known probabilities of the quality levels of the personnel's actions, a measure of the complexity of the situation is determined according to the formula of total probability.

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THE IMPROVEMENT OF WORKING CONDITIONS AT THE INDUSTRIAL WORKSHOPS OF MINING ENTERPRISES

Technological processes of rock processing are accompanied by intensive dust emission which increases allowable values of dust content in the air of working places. Available methods of dust suppression such as aspiration, ventilation, dust binding by moistening processed materials, capsulation of trans-loading units, etc. are unable to localize dust sources completely.

Emitted dust spreads all over industrial premises and settles on various surfaces (construction structures, pipelines, cable trunks, equipment elements, etc.) making layers of various thickness. Under the action of various factors (equipment operation and vibration, repair works, aeration flows, etc.), settled dust can repeatedly turn into aerosols. Due to this, dust content of the air increases that can provoke vocational diseases of processing shop employees.

At the same time, the dust accumulated on various surfaces complicates the maintenance and repair of technological equipment and communications, accelerates the wear of individual units and parts, reduces the time needed for overhaul maintenance, disables control and measuring equipment and weighing equipment. In a number of industries, the settled dust causes explosions or fires. For many enterprises, the dust settled on various surfaces is considered as a valuable resource in solving problems of energy saving and energy efficiency, due to utilization and recycling.

That is why, solution of dust collection problem is urgent for processing shops of mining enterprises, it allowing them to facilitate working conditions, reduce vocational disease rates and cut down equipment wear.

Technologies of rock processing are noted for a variety of sources of intensive dust emission and great areas of its precipitation to be cleaned. Centralized industrial dust collectors can be applied to different surfaces in processing shops of mining enterprises.

Yet, these machines are stationary and applicable to just one premise. The stationary pipeline system when in long service tends to be polluted and requires either cleaning or demounting. Dust precipitation reduces the amount of air exhausted through dust-cleaning nozzles and changes aerodynamic indices and efficiency of the vacuum system as a whole.

The length of pipelines of stationary vacuum systems is conditioned by large areas to be cleaned and a great amount of equipment there. This requires highly efficient traction activators consuming much power.

Thus, a long stationary system of pipelines is the most significant disadvantage of all modifications of centralized industrial dust collectors. It should also be noted that one machine is applicable to one premise only. For large-scale enterprises, there should be several single-types machines, each of them having a stationary system of pipelines. It makes control over pipe blockage more complicated and is associated with higher maintenance costs of such a long system within one enterprise.

Dust accumulation on surfaces of industrial structures and equipment does not occur uniformly. It enables an enterprise to keep its industrial premises clean by using a mobile dust collector that is transported to another object after cleaning the previous one. Basic equipment of a mobile dust collector is in the body of a van outside a premise to be cleaned.

The mobile industrial dust collector is serviced by three people, one of them is a driver and the rest are dust-collecting operators. The driver controls basic equipment, while operators clean a premise. Both operators also deal with high-up surfaces and hard-to-reach places. One of them cleans the surface using a step-ladder, while the other watches out for him, switches a flexible hose from one pipe union to another, gives required nozzles, etc.

A mobile industrial dust collector is able to move independently to a required object of cleaning within a given enterprise as a traction activator and dust catching devices are inside a mobile van, while detachable pipelines and nozzles are mounted from separate sections during cleaning. It reduces the total length of the vacuum system and helps avoid blockage of pipelines in the long run. There is also an opportunity of cleaning hard-to-reach surfaces. As the van is located outside the cleaned object, the air does not re-circulate in heavily polluted premises.

Mobile dust collectors are applicable to shops with any appliance saturation. The machines are able to collect great amounts of dust and transport it to unloading points without intermediate loading-unloading operations. Dust unloading is performed outside the industrial premise to avoid its secondary dust pollution.

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INCREASING THE EFFICIENCY OF DUSTING THE FIBER FILTER BY ELECTRIFYING THE FIBERS

The dedusting of aspirated air, which is removed from the sources of dust extraction during the processing of rock mass, in the aspiration-traction units of the mining and processing enterprises of the Kryvyi Rih region, as a rule, occurs by means of a two-stage purification system. The first stage of cleaning is preferably used cyclone apparatus, the second - scrubbers (Venturi pipes) or bag filters. A number of organizational and technical reasons (changing the aerodynamic parameters of dust flow, violation of schedules of unloading dust bins, etc.) causes the low efficiency of these systems, which leads to deterioration of sanitary and hygienic conditions of workers of industrial enterprises.

To increase the efficiency of industrial enterprises' aspiration systems, a dust chamber equipped with a modular fiber filter is proposed as the first stage of cleaning.

As the airflow passes through the fibers of the modular fiber filter, the kinetic energy of the dust particles is lost, which contributes to their intense coagulation with subsequent deposition. In addition to the purely mechanical coagulation of dust particles on the fibers, the latter create an electrostatic effect in the moving dust stream.

Fibers made from synthetic materials or particles themselves can receive and hold electrical charges. Thus there is an electrical interaction, which has a sufficiently large radius of action, which promotes the movement of particles to the fibers and touching the particles to their surface. Electric forces are widely used in practice to increase dust efficiency, and the charging of dust particles and fibers can occur spontaneously due to friction when hitting the particles against each other and when rubbing them against a solid surface (aspiration covers and pipelines, etc.). The polarity, the magnitude of the charges charged on the fibers, and the rate of their loss depend largely on their nature.

Electric forces contribute to the aggregation of dust particles and the formation of branched chain structures on the filter fibers. Particle aggregates are more easily captured by the filter, and the structures formed on its fibers serve as an auxiliary filter medium.

The study of the possibility of electrification of different types of fibers was carried out as follows. To determine the sign and measure the magnitude of the surface charge density, a PC2-3A device with a measurement range of 0 to 20 $\mu C/m^2$ was used. The sign of the electric charge was determined by the deviation of the arrow of the device from the middle of the scale (zero): right – «+», left – «-».

To obtain an electric charge on the fibers, they were rubbed on ebonite and fiber on fiber. The electric charge sign and the magnitude of its surface density were determined immediately after rubbing and again after 30-60-90 minutes to determine the amount of charge loss as a function of time.

Studies have shown that the highest positive triboelectric charge accumulates on the fibers of kapron and triacetate silk, and the smallest - on the fibers of nitron; the highest negative charge accumulates on the polyvinyl chloride fibers and the smallest charge on the wool fibers.

The results of determining the dependence of dust capture efficiency on the magnitude of surface triboelectric charge accumulated on the dust chamber model showed that as the triboelectric surface charge of fibers increases, the dust capture efficiency of the filter increases and this growth is subject to parabolic law. The largest surface charge on kapron fibers was 4.0-4.2 $\mu C/m^2$ on polyvinyl chloride – 2.0-2.3 $\mu C/m^2$.

Over time, charged filter fibers will lose their charge. Even in conditions of normal relative humidity of ambient air (40-50 %) in 80-90 min. after pointing the charge at the fibers under study, its density decreases by 45-60 %. At relative humidity of more than 65% the loss of electric charge from the fibers occurs within 20-30 s.

However, when exposed to constant factors that can lead to the formation of charges on the fibers (the presence of electrically charged dust or dust, which when interacting with the fiber forms an electric charge), they can be stored for a long time.

Therefore, the action of electrostatic forces increases the efficiency of dust collection of dust chambers with a modular fiber filter. It is therefore of interest to investigate the effect of these forces on the capture of industrial dust in relation to existing types of fibers. These studies will help to scientifically justify the choice of fiber type, as well as to find new ways to ensure high dust content and ease of regeneration of the fiber filter under conditions of normal and high humidity, which occurs in the production conditions. The use of a dust chamber with a modular fiber filter will ensure the stability and efficiency of aspiration systems, which will improve the working conditions of workers of industrial enterprises.

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WORKPLACE RISK ASSESSMENT IN MONGOLIA: A CASE STUDY OF A MINING INDUSRTY

Mongolia is a developing country and faces many challenges to meet customer requirements related to social compliance, health and safety, environment and well-being at the workplace. Workplace risk assessment is a new approach in Mongolian traditionally oriented occupational safety and health. This research aims to assess the level of risk attached to existing working strategies in the mining sector. For this purpose, a case study has been conducted at the Ukhaa Khudag coal mine, situated in the south of the country.

Numbers of accidents are very high, especially in the mining industry. During 2005-2013 years 225 has died, 100 disabled, and 401 injured and temporarily lost employability as a result of 629 work accidents. In 2013-14 at the Ukhaa Khudag mine there were 21 accidents, 15 of them are near misses, 3 first aid-injuries, 3 minor injuries, and 1 property damage.The reasons for these are identified and appropriate measures have been taken.

Risk Assessment are takenat Ukhaa Khudag Mine in the following five simple steps and the step 3 - Risk Assessment is explained in this article.

Step 1. Data collection

Step 2. Hazard identification

Step 3. Risk assessment

Step 4. Risk treatment

Step 5. Monitoring and review

A common method of risk assessment is consideration on what consequence could result from a hazard, and its likelihood to occur, analyzing with the hazard. A risk level is determined in this method. Risk level depends on both of consequence and likelihood.

Consequence definition. The severity of a consequence values numerically and tells us what impacts to personnel safety, environment, property, work performance and institutional reputation. (table 2, only personal safety is in scope)

Table 2

Definition of consequence

Consequence levels					
Scope	1	2	3	4	5
People	No Treatment Case	First Aid Case	Medical Treatment Case	Permanent disability	Loss of life

Likelihood definition. Likelihood is estimated by impacts to the following: Person or people, Project, and Company. (table 3, only impact to person is here)

Table 3

Definition of likelihood

Level	5	4	3	2	1
Occurrence	Very likely	Likely	Possible	Unlikely	Remote
Impact to person**	More than once a year	At least a time in 3 years	At least a time in 10 years	At least a time in 30 years	Once in 30 years
Likelihood	90-99%	65-89%	35-64%	10-34%	<10%

** Note: people including.

Risk matrix. Considering the category of likelihood against the category of consequence severity risk level is defined. Risk matrix is a simple mechanism to increase visibility of risks and assist management decision making.

Table 4

Risk levels

Likelihood	**Consequence**				
	1 Insignificant	2 Minor	3 Moderate	4 Major	5 Catastrophic
A – occurs frequently	H 11	H 16	E 20	E 23	E 25
B -recurrent	M 7	H 12	H 17	E 21	E 24
C– could occur	L 4	M 8	H 13	E 18	E 22
D– occurs rarely	L 2	L 5	M 9	H 14	E 19
E- almost never	L 1	L 3	M 6	H 10	H 15

Risk control. Risk level determines which type of risk management measures are needed and when to implement it. The higher the level of risk, it is necessary to confirm the higher the management and the urgent measures to control the risks.

Table 5

Risk control measures

Risk level	Risk control measure
Extra	Lower the risk to a low level very urgently Reduce residual risk to a reasonable level by the engineering method (administrative control is not possible)
High	Lower the risk to a low level urgently Reduce residual risk to a reasonable level by the method of hard and soft
Medium	Continue or adjust appropriate monitoring to reduce risk Reduce residual risk to a reasonable level by a method in risk control triangle (anyone is possible)
Low	Take corrective actions in the required part No more control for residual risk

To eliminate hazards and accidents at workplace, to constantly monitor them and prevent potential conflicts, risk assessment is required. A common way of risk assessment is a definition of risk severity level using the matrix. Combining Consequence table with Likelihood table will significantly improve risk assessments within any organization. The results of risk assessments were not used enough in risk management actions.

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EXPLOSION PROTECTION AND LABOR PROTECTION REQUIREMENTS FOR COGENERATION GAS COAL MINES

The current standard for mine degassing in Ukraine [industry standard of Ukraine] regulates the percentage of methane in the degassing mixtures. Methane concentration should be less than 3.5% or more 25%. A mixture of gases with a methane concentration in the range from 3.5 to 25% explosive and must be disposed of "candle". This restriction underestimates the potential of degassing gas mixtures, which leads to low energy efficiency expensive cogeneration stations. It is worth noting that methane concentration of 3.5% to 25% is typical for degassing systems of Ukraine.

An analysis of the operation of recycling facilities showed that the supply of hydrocarbon gas and air is carried out as usually by hand.

The aim of the work is the development of information support for explosion safety and labor protection during the utilization of gases from coal deposits at mining allotments and mines.

According to research [1], it was found that the explosion safety of hydrocarbon gases depends not only on the concentration of the main combustible gas - methane. The explosiveness of the mixture is affected by the mutual concentration of methane and oxygen in it. Explosiveness is determined in accordance with the concentration of oxygen in the gas mixture, which is inversely proportional to the concentration of methane in the range from 0 to 5% and directly proportional to the concentration of methane in the range from 20 to 25%. With an oxygen concentration of up to 7%, the mixture is explosion proof at any methane concentration. Based on these regularities, by the method of signature mathematics for solving boundary value problems of the information space, equations are obtained for calculating the control parameters of the relay controller.

Ensuring the explosion safety of mine methane is carried out by the regulator of the gas mixing regulatory point.

The heat load regulator ensures the operation of the boiler [2] in one of the following modes: on mine methane; on coal gas; on a mixture of coal gas and mine methane. The system simultaneously provides a pre-emptive signal to all controller inputs proportional to the first derivative of the disturbance of the boiler operation mode from the steam flow rate [2].

Coal mine methane flow regulator is designed to maintain a given coal mine methane flow rate when the boiler is operating on a mixture of coal mine methane and coal gases. To ensure the necessary range of regulation of the heat load of the boiler when the minimum consumption of coal gas is achieved, it is controlled by a self-tuning subsystem [3] according to labor protection and energy efficiency criteria, which can reduce the formation of toxic gases (*NO* and *CO*) in the furnace without decreasing the coefficient of performance (COP) boiler unit.

The rarefaction regulator is designed to maintain a given rarefaction in the furnace by changing the degree of opening of the exhaust fan guide apparatus according to the safety and labor protection criteria of the boiler unit according to the rarefaction parameter given in [3].

Conclusions

1. Distinctive advantages of the developed software for the information management system are:

- the presence of a gas mixing regulatory point (GMRP), in which, using the established explosion safety criterion, all extracted coal mine methane, including explosive concentrations, is utilized;

- criteria for labor protection and energy efficiency can reduce the formation of toxic gases (*NO* and *CO*) in the furnace without reducing the efficiency of the boiler;

- the issuance of control actions is formed that ensure trouble-free operation due to the extinction of the torch and the requirement for labor protection for gas contamination of the station premises with toxic gases.

2. On the basis of the presented scientific research works of the authors, work projects can be developed for the creation of pilot industrial models of accident-free gas cogeneration control stations at forges and mines.

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USE OF NATURAL BISCHOFITE WATER SOLUTION FOR WARNING OF AIR DUSTING

Current technology of mineral deposits mining operations with the purpose of low-grade ore enriching is needed enormous areas in order to arrange special places for washery refuse storages (mine refuses). Today “wet milling” of washery refuse storages is the most popular in our region, we say about sludge ponds [1].

Dry areas are created at the mine refuses outlet zones after the process of pulp aggradation on the maps. If wind speed is more than 3,0 м/s mine refuses after wind erosion are fulfilled into dust emission sources. The most quantity of mine refuses according to the fraction consistence is belonged to the erodible dangerous dust emissions. About 90% of these emissions are air borne particles with diameter less than 50 mkm [2]. This is very important to take into account that mine refuses areas are situated not far from the residential communities, 1-5 km usually. Analysis of mine refuses dust emissions shows that in such zones these dust emissions have a negative influence to the environmental as well as to the health of people, especially children. In these ecological dangerous zones level of the diseases of the children respiratory system is higher in 1,3 times [3].

For the decision of problem of moisture loss from the upside ball, it was decided to use water solution of chlorides, namely chlorides of magnesium - natural bischofite solution (MCl_2*6H_2O). This solution has got a 4th class of danger level. It does not burn. It has got nominal low corrosive ability. It is used in the interval from 55°C till - 35°C and this solution is produced in Ukraine.

Conducted laboratory and industrial researches are showed that after buttering of natural bischofite solution (NBS) to the wet upside area of the exist mine refuses storages, with the charges on the level 1,5-2,0 l/m^2 and concentration 100 % (density of the material not less then 1250 kg/m^3), this surface is well fastened. Due to the high NBS hygroscopicity this surface is remained moisture during the long period (no less than 75 days) taking into account the hottest days. Moistening of surface allows decreasing of air dust emissions fundamentally.

When the weather is stable dry and hot we have some moisture decreasing at the mine refuses upside ball during the day (there is a salt membrane on the upside surface and this membrane keeps air dust emission because of daily temperature surges - became known as “dew effect”, so the surface is moisten against). During this process air dust emissions are fundamentally decreased - Table 1.

Table 1

Results of production researches of keeping of upside mine refuses storage areas affectivity of Private Joint-Stock Company «PivnMCC»

Tempera-ture of air °C	Ddegree of air satura-tion, %	Wind speed, m/s	Moisture of mine refuses		Air pollution, мg/m^3	
			proc. NBS	final	proc. NBS	final
4,8	70	3,0-4,0	5,52	4,61	0,16	2,6
8	84	5,4-6,0	11,3	1,6	1,13	6,0
25	63	4,4-4,8	9,01	1,3	1	4,6
26	60	2,5-3,0	8,8	0,15	0,26	4,5

According to the researches, NBS, contrary to the other methods, can be used during the whole year. NBS use is not needed any special equipment, any prior operations. NBS is totally mechanized and can be used through different actual water distributing vehicles (water sprinkler tank, moisture monitor, etc). All these methods are increased flexibility of the use of the solution and are allowed easily to fasten the areas of different size and form.

But use of NBS with the concentration 100%, is not always economically appropriated. According to the results of the laboratory researches we approved that use of the solutions with low concentrations (60-80%) has made possible to save money for the fastening. We can reach efficiency of solution at the level during no less than 10 days. That is why we can decrease fastening expenses at the expenses of use of low concentration water solution in case of short term fastening of mine refuses storage areas (dam reconstruction, dredge piping laying).

This developed dust emission surfaces fastening technology of existing mine refuses storage areas passed industrial researches at such plants as PJSC “ArcelorMittal Kryvyi Rih” and PJSC “PivnMCC”. According to the results of these researches it was developed technology of industrial use of natural bischofite solution in local conditions. Also it was developed the most optimal schemes of solution application to the fastening surfaces.

Conclusions:

According to the survey results it is set that:

- Water natural bischofite solution is the optimal method of the dust surfaces fastening;

- A high NBS hygroscopicity allows to fasten the high humidity of the upside ball of the dust materials, decreasing dust emissions from the upside areas of the mine refuses storages;

- At optimal charges of NBS 1,5-2,0 kg/m^2 air dust emissions at outlet from the timbered area of mine refuses storages in 4,6-16 times less than at outlet from the ordnance datum taking into account the basic wind speeds;

- In order to block dust upsides during the short period we can use NBS with the concentration less than 100%.

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SECTION "MINERAL PROCESSING"

UDC [658.78:622'17]:622.012

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SUBSTANTIATION OF EXPEDIENCY OF INVOLVING STOCKPILED WASTES INTO PRODUCTION AT CENTRAL ORE-DRESSING AND PROCESSING ENTERPRISE, PJSC

Economical use of physical resources is of significant importance for improving the efficiency of public production since the national economy demand for raw materials and supplies is increasing steadily, while their production is becoming more and more expensive.

From extraction to obtaining finished products, raw materials undergo a large number of stages of processing. As a result of this process, only part of the production is used as intended, whereas the remaining products are stored in disposal areas and tailing storage facilities.

One of the enterprises of the Kryvyi Rih iron-ore basin, which mines and concentrates magnetitic quartzites, is PJSC Central Ore-Dressing and Processing Enterprise (PJSC "CODPE"). Approximately 30% of mineral raw materials extracted at this at production site is headed for metallurgical treatment in order to obtain cast iron, while considerable part (70%) is stored as wastes.

Previous research showed that the wastes of the enterprise contain valuable components; that is why it is expedient to consider them as sources of additional products.

Involving stockpiled wastes of the PJSC Central Ore-Dressing and Processing Enterprise into production will make the following possible: firstly, to reduce the territory of tailing dumps and decrease the negative impact of discharges into the environment; secondly, to additionally obtain both iron-ore concentrate and other marketable products to be used in various sectors of the national economy.

In this regard the following tasks are to be solved:

- to study material composition of dumping sites and mature tailings of the PJSC "CODPE" along with defining their mineral composition, sizing characteristics, physical properties (gravitational, magnetic, electrical ones);
- to conduct engineering studies on dumping sites and mature tailings of the PJSC "CODPE" to develop an integrated technology for obtaining sellable concentrate and additional products.

During elaborate study on each of the objects, it was established that when stockpiling the dumping sites different methods for dumping were used which influence considerably their further involvement into production of standard quality raw material:

- those which considered their further use (dumping site "Pivdennyi"), where sites of valuable components have been purposefully formed within the outlines of this technogenic deposit;
- those which did not consider their further use (dumping site No.6, South-West dumping site, the inner dumping site); mineral compo-

nents are distributed in a random way within the outlines of these arrays.

These will influence significantly the process of reworking dumping sites.

The analysis of the research studies showed that dumping site "Pivdennyi" is presented by two major locations of valuable components: oxidized banded iron formations and indurated talc.

From the site of oxidized banded iron formations, at this stage of research substandard hematite-martite ores can be involved for obtaining sintering ore from them. According to the recommended technology, from the source raw material with 43.8 % assay, it is possible to obtain sintering ore with 55.2 % assay with yield of a product of 36.9 % and product recovery of 46,5 %.

Obtaining hematite concentrate from stored oxidized quartzites requires more detailed research into peculiarities of physico-chemical and mechanical properties of raw material considering the influence of atmospheric phenomena on it.

Based on the conducted research, variations on process flow schematics on raw material processing using selective extraction of talc-magnesite raw materials are considered: difficult ones - carbonated serpentinites and moderate ones – talc-carbonate rocks.

It is shown that treatment schemes differ critically by degradation coarseness which makes 85% of the class -0.074 mm for carbonated serpentinites, for talc-carbonate rocks – 78% of the class -0.074 mm, at stagy I of degradation, and, respectively, 93.5 and 95.7% of the class -0.074 mm, at stage II.

The tailing storage facility of the PJSC "CODPE" also features certain peculiarity of its formation, which distinguish it among tailing storage facilities of other ore-dressing and processing enterprises of Kryvorizhzhia.

This peculiarity involves simultaneous filling of containers and dropping of a pulp from pulp lines in one place (in the top of the bar), which has created favourable conditions in the tailings storage facility for gravitative differentiation of the solid phase of the pulp and creation of a dimensionally isolated, rather large iron fortified area, which is suitable for immediate selective mining. The dimensions of this area make 1.3 – 1.8 km across the width, over 2 km lengthwise and up to 3 – 20 m by capacity.

Wastes of the ore-dressing plant, which accumulate in the tailings storage facility, are considered as raw materials for recleaning. The research showed a possibility of using earlier warehoused tailings of wet magnetic separation of the "CODPE" PJSC as additional raw materials for obtaining saleable concentrate and mortar sand.

Considering the above-described we can make a conclusion we can make a conclusion that the material stockpiled at dumping sites and tailing storage facility of the PJSC "CODPE" forms "technogenic" deposits which can be involved into production to obtained finished products, whose uses are diverse and include: 1) metallurgy industry (magnetite and hematite concentrates); 2) construction industry (crushed aggregate, sand, limestone, clay, structural stone); 3) paint and coatings industry (mineral pigments); 4) agriculture (mineral fertilizers).

To involve stockpiled production wastes, apart from additional studies on finding optimal parameters of the technologies, the enterprise is to transfer them to the status of technogenic deposits and obtain a license for their development.

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METHOD FOR PROCESSING OF METALLURGICAL AND FLOTATION WASTE AND IMPOVERISHED ORES

The entire extraction of metals from impoverished ores through the classical methods of mineral processing meets many difficulties in the practice. The application of these methods to the treatment of metallurgical waste often does not lead to the desired results.

Pre-concentration of valuable metals facilitates their further extraction.

Recently more attention is paid to the possibility to improve the extraction of valuable components from waste and low-grade ores by applying an electrochemical impact [1-3].

Alehin and co-authors [1] developed a method for complex enrichment and deep recovery of metals from waste slurries or impoverished ores of nonferrous metals, in which a layer of the treated material is formed onto impermeable lining. Discharge drillings for submission of chemical reagents (0.15 M H_2SO_4 + 0.15 M $Fe_2(SO_4)_3$) were made in the layer. Carbon anodes and aluminum cathodes were inserted into the material layer. The contact of the material with the chemical reagents solution fed by the pressure drilling is realized by passing the solution through a layer to the collecting well. As a result, there were areas where the material did not come into contact with the reagents, which led to low efficiency. Moreover, the mobility of the geochemical barriers, which were formed and were able to concentrate the valuable components, made difficult the collection of the enriched material. In addition, $Fe(OH)_3$ was formed that hindered the following flotation of the nonferrous metals.

Maintenance of metals long enough in a mobile state and concentrating them in the near cathode or near anode space (depending on the type of metal and the electrolyte used) would ease significantly further metals' recovery.

Here a method is presented where by means of an electrochemical impact the metal components of slag, flotation waste and low-grade ores have been concentrated in the near-electrode spaces with the aim to facilitate their further recovery.

The treatment has been carried out in specially designed device [4] consisting of series of electrochemical cells where the ground material is mixed with suitable solutions and then subjected to DC voltage applied by an axially rectifier. Cathodes have been made of stainless steel, anodes have been made of graphite. Material enriched in different metals can be separated by means of special slots on the bottom of the device. Some representative results from the experiments carried out are summarized in Table 1.

Table 1

Results from studies on the electrochemical concentration of valuable metals

Material for treatment	Treatment conditions	Enriched material
Metallurgical slag: 0.73 % Zn, 0.30 % Pb, 1.60 % Cu, 24.6 % Fe	10 % H_2SO_4; solid to liquid 2:1; 0.4 mA/cm^2; room temperature; 168 h	Near to cathode area 9.75 % Cu, 3.2 % Zn
Metallurgical slag: 0.71 % Zn, 0.30 % Pb, 1.65 % Cu, 24.8 % Fe	10 % H_2SO_4; solid to liquid 2:1; 10 mA/cm^2; room temperature; 48 h	Near to cathode area 10.1 % Cu, 3.4 % Zn
Flotation waste: 0.23 % Zn, 0.16 % Pb	10 % HCl; solid to liquid 1:1; 10 mA/cm^2; room temperature; 120 h	Near to cathode area 4.9 % Zn, 3.8 % Pb
Low-grade lead-zinc sulfide ore: 1.61 % Zn, 1.16 % Pb	10% HCl; solid to liquid 1:1; 15 mA/cm^2; room temperature; 120 h	Near to cathode area 9.7 % Zn, 9.2 % Pb

When the treatment medium is weekly acidic (as is the case when 10 % acid is added to the studied material), the reactions proceeding on the electrodes can be generally presented with the following equations:

at insoluble anode: $2H_2O \rightarrow O_2 + 4H^+ + 4e^-$

at cathode: $2H^+ + 2e^- \rightarrow H_2$.

The reaction at anode contributes to keeping the acidic conditions, which prevents precipitation of metal hydroxides. The oxygen formed facilitates oxidation, thus the leaching of the desired metals from the solid material.

Comparison made with the same materials leaching under the same conditions, except of the DC impact, showed that the electric field facilitates leaching of those elements from the studied waste material, as ions. The field also causes the ions' electro-migration to cathode.

In conclusion it can be stated that by the proposed DC treatment, the valuable metals are concentrated in the near cathode area, i.e. in smaller mass of the treated material that can be easily separated from the other material. This in turn leads to a possibility for enhanced extraction of valuable metals at decreasing in the costs.

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CURRENT STATUS OF COAL ENRICHMENT IN POLAND

Introduction

Hard coal deposits in Poland occur in two basins: the Upper Silesian Coal Basin and the Lublin Coal. In recent years, hard coal mining in Poland has been on a downward trend. As shown in Table 1, coal extraction in the last 5 years (2014-2018) has decreased by more than 9 million Mg. This is particularly true for steam coal, for which the decrease amounted to over 8.8 million Mg.

Table 1

Production data for Polish hard coal industry [1]

Year	2014	2015	2016	2017	2018
Total production [mln Mg]	**72,5**	**72,2**	**70,4**	**65,5**	**63,4**
Steam coal [mln Mg]	60,2	59,2	57,2	53,0	51,3
Coking coal [mln Mg]	12,3	13,0	13,2	12,5	12,1

The trade balance of hard coal in Poland (Table 2) in 2015 and 2016 showed a slight surplus. In 2014, 2017 and 2018, however, it had a deficit which in 2017 exceeded 7 million Mg and in 2018 – 15.8 million Mg.

Table 2

Import and export of hard coal in Poland [1]

Year	2014	2015	2016	2017	2018
Import [mln Mg]	10,4	8,3	8,3	13,4	19,7
Export [mln Mg]	9,0	9,2	8,9	6,3	3,9
Trade balance [Mg]	**- 1,4**	**+ 0,9**	**+ 0,6**	**- 7,1**	**-15,8**

Washed production [2]

Figure 1 presents a simplified schematic diagram of a hard coal enrichment plant (steam and coke) in Poland.

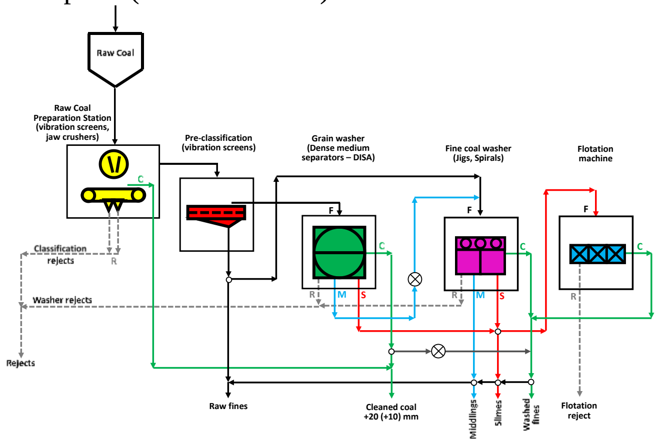

Figure 1. Schematic diagram of hard coal enrichment in Poland
(*F – Feed, C - Clean Coal, M- Middlings, R- Rejects, S – Slimes*)

There are 26 mechanical coal processing plants in Poland on production capacity 440.000 Mg/day, including 20 steam coal enrichment plants and 6 coking coal enrichment plants. The most common technologies for enrichment of coal of grain fraction above 20 mm are DISA heavy liquid separators. Grain grades up to 0.5 mm to 20 mm are enriched using Allmineral, OM or BATAC type fine coal jigs, heavy liquid cyclones and spiral separators. The finest grain grades below 0.5 mm are enriched in floatation machines.

Currently, about 90% of the steam coal mined in Poland is subject to processing. Out of the volume of steam coal processed, 12% is subject only to the process of preliminary classification and enrichment of grain fractions above 20 mm. The remaining part is additionally enriched in grain class above 0.5 mm. At the same time, only 11% of the processed steam coal undergoes a full range of enrichment. In the case of coking coal, the entire extracted volume undergoes a full range of enrichment.

Operating in Poland, the mechanical processing plants of steam and coking coal are characterized by a diversified level of technological development.

Barriers for the technological development of processing plants

The basic factors influencing the development of processing technologies of steam and coking coal include the capital barrier, legal status and location conditions.

The first of the above factors, i.e. the capital barrier, may be the factor hindering or even preventing the implementation of modern processing technologies. It is therefore necessary to draw up a multi-variant balance sheet taking into account, on the one hand, the market needs (this applies to the main customers of the commercial products offered) and, on the other hand, the costs related to research, design and purchase of machinery and equipment.

What also constitutes an important factor is the legal status, or rather frequent changes in legal regulations concerning coal mining, coal enrichment and environmental protection.

Location conditions are a further factor. A number of processing plants operating in Poland are located in multi-storey buildings adapted to the technological solutions applied. Therefore, their space and volume is limited. Changing the technology in such limited location conditions is very difficult and sometimes even impossible.

Summary

The present analysis of the condition of mechanical coal processing in Poland shows that over the last several years, many new technological and technical solutions have been implemented. This is visible mainly in the area of enrichment of the finest fractions of steam coal, and in particular in the area of mechanical dehydration. The technological solutions applied in processing plants, especially in relation to coking coal, are at a good level. However, it is still possible to increase the innovativeness and effectiveness of applied technological solutions and production efficiency in order to meet the expectations set by the market.

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SEMI-COKING OF SAPROPELITE COAL FROM THE MINES OF THE LVIV-VOLYN COAL BASIN

The problem of sapropelites accumulation in the coal mining and processing areas has arisen since the first years of exploitation of the Lviv-Volyn coal deposits. Significant reserves of sapropelite coal, which is unfit for industrial use, are located in the coal-mining region [1].

The extraction of sapropelites on the surface occurs simultaneously with the extraction of humus coal in cases of joint bedding.

Geological studies performed in the basin have shown that sapropelite coal reserves make up a significant proportion of all coal in the basin, up to approximately 0.1 bl.t. Individual sapropelitic stratums have a thickness up to 1m. In basic layers n_7, n_8 the sapropelite coal spreading is more than 75%.

In some areas, the continuous distribution area of sapropelites reaches 50-80 km^2 with average stratum thickness of 0.5-0.7m.

The purpose of this work is to extract tar, combustible gas and semi-coke from Lviv-Volyn coal basin (LVCB) high-ash sapropelites. The removed tar divided into five fractions according to the fuel scheme. Few samples of sapropelite coal selected from three mines of SC “Lvivvugillya”, crushed and fractionated.

For them, the technical characteristics were determined; they showed high ash content of all samples, which averaged 36.3%, with average humidity - 1.7% and volatile substance yield - 25.35% (from the analytical sample).

We used two apparatus for semi-coking: a small one with a loading capacity of 100 g and a large one with a camera volume of more than 1.5 dm^3.

Semi-coking carried out in the temperature range 540-550 °C. Using a small apparatus and a wet gasometer, we found that the gas yield of the samples averages 6.56 dm^3 per 100 g. The gas composition determined by gas chromatography. The calculated (in the ideal gas model according to ISO 6976:1995) higher and lower volumetric heat of gas combustion was 22.59 and 10.29 MJ/m^3 respectively.

After a series of experiments carried out on a large installation, we have collected a sufficient amount of primary tar for fractionation and investigation of individual fractions. The tar was a dark brown, moderately viscous liquid, with a high water content. We established that only part of the water could separate by the method of separation of unmixed layers. The residual water enters in light distillation fractions (gasoline and ligroin). The average water content in the tar was 43.7%, of which more than 90% easily separated on the separatory funnel. The relative density of the dehydrated tar samples is in the range of 0.975-0.996.

The fractionation of the dehydrated tar carried out according to the fuel scheme. On average, the gasoline fraction was (in wt. %) - 3.2, kerosene - 4.28, ligroin - 9.41, diesel - 43.95 and fuel oil - 39.16. For two tar fractions - kerosene and diesel, an IR-spectroscopy performed to determine the group composition. It showed that with increasing boiling point, the decarboxylation of the tar components occurs, and the fractionation product acquires an aliphatic-aromatic character with an admixture of a small amount of oxygen-containing compounds represented by phenols, ethers and alcohols.

According to the results of our research, we have made a material balance of sapropelite coal semi-cocking:

Raw material:
Sapropelite coal 1000 kg
Products:
Combustible gas 65.6 m^3;
Semi-coke 811.8 kg;

Tar 58.55 kg, including fractions

- gasoline 1.87 kg,
- ligroin 2.51 kg,
- kerosene 5.51 kg,
- diesel 25.73 kg,
- heavy fuel oil 22.93 kg;

Water (including pyrogenetic) 45.36 kg.

Gas and technological losses 84.29 kg.

The results obtained from the study of LVCB's sapropelites semi-coking indicate that it is a source of various energy and chemical raw materials: liquid, gaseous and solid.

The tar formed in moderate but sufficient quantities, and its fractionation shows the high content of diesel and heavy fuel oil fractions (82% of the tar total weight).

Studies of the group composition of kerosene and diesel fractions concluded that they are a mixture of aliphatic and aromatic hydrocarbons, which can be used in the chemical industry or as a raw material for the fuel production.

Checking the coke for its ability to burn showed that it is a combustible material. Its flammability is due to the high content of organic component - 49.7 %, at low humidity (1.05%) and significant but non-critical ash content (42.4%).

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LOADING DEVICE FOR BALL MILLS WITH DIFFERENT SIZED GRINDING BODIES

Ball miles got widespread in the first stages of ore grinding at enrichment factories, ball mills were obtained where they operate in closed cycles with a single-spiral classifier. They consume electric energy and materials in the form of balls and lining, the reason for this is the non-compliance with the optimal grinding environment due to the lack of an effective device for loading different sized balls. Practice has shown that mills loaded with balls of four to five sizes provide +10...15% higher productivity compared to technological units with a one-dimensional grinding environment.

Such tools have been developed since the middle of the last century. They were mainly intended for balls of the same size. They did not solve the problem comprehensively, could not supply the required number of balls in their rapid operation. Therefore, single copies of such devices were not distributed in processing plants. The low reliability of their work was the main disadvantage of these devices. Devices for various sizes of balls were not even offered. At the same time, devices for this purpose got widespread in mechanical engineering. [1, 2, 3]. Occasionally they appeared in the enrichment industry [4]. Despite this, the problem has not been solved in the enrichment industry.

The purpose of this work is to develop a device for loading ball mills with grinding bodies, four to five sizes, with their automatic feeding.

The energy approach to ball wear in mills has not yet become widespread. There are no ready-made solutions for the loading device. The analysis shows that it is fundamentally possible to use two approaches. The first is active devices of the type [4]. This device can be directly loaded with balls by the crane without their orientation, however, it can to jam grinding bodies, there is a need to turn on the vibrator, control the exit state of the balls, spend energy, etc.

The device will be more progressive, considered in [3]. In it, jamming of balls is practically excluded. It is more compact and can be made in the form of a single monoblock. The previous device requires the parallel use of four systems for each ball size. A multi-threaded loading device can be made in four to five sizes of balls.

More promising is the transition to more stable balls and linings. There are balls where output is 15 g per ton of ore. These are steel balls of the fifth hardness group, which are now produced in Ukraine. Under these conditions, when grinding 165 tons of ore, the actions of the balls is 16.5 kg, and the lining is 3.3 kg, which together is 19.8 kg. These indicators physically modelled a multi-threaded loading device. The simulation results are given in table 1.

We set certain technological parameters, we get that the loading device will have the main dimensions of 1.3×0.45×0.325 m. This will be a compact device, in which many multi- threaded for different-sized balls are located in width.

Each loading device set has the same mechanism for controlling the release of balls. It contains a spring-loaded latch that holds the ball in the gutter, and an electromagnet that changes the position of the valve. A device with many multi- threaded sets, assembled as a whole, ensures the operation of the ball mill during the day. It can be done with a width of 0.65 m. Then it can feed the technological unit with balls for two days. If you still increase the height to 0.9 m, providing the device with a size of 1.3×0.9×0.65 m, then it can feed the mill with balls for four days. That is, the capabilities of this loading device are wide, its reliability is extremely high.

Table 1

Physical simulation results of a lot of multi-threaded loading device

Diameter of loaded balls d_k, mm	50	65	75	90
The number of balls in the loading cycle, pcs.	7,0	4,1	3,1	2,13
The daily number of balls to load, pcs.	168	98,4	74,4	51,1
Rounded number of channels, pcs.	7	6	5	4
Rounded number of balls in the channel, pcs.	24	19	16	14
Accepted device length, m	1,3	1,3	1,3	1,3
Rounded channel height, m	0,45	0,45	0,45	0,45

This loading device must be filled with 418 pieces of different-sized balls. This is best done by an industrial robot of a loading and transport type. Such works were used one of the first in robotic technological modules in mechanical engineering, but their efficiency was low [5]. In this situation, the use of an industrial robot improves all of the indicators noted in [5]. In addition, in this process of filling a loading device with different-sized balls, all the advantages of an industrial robot are effectively used compared to a human - it realizes high productivity, does not get tired and can work for a day, does not require comfortable temperature conditions, housing, food, rest, holidays, days off, infrastructure, medical care, etc. [6]. At the workplace to the ball mill, filling devices filled with balls are transported by a crane, where they are installed at a precisely defined position with a fixed position. Before this, the loading device freed from the balls is removed and transported to the loading position. It is advisable to perform operation after the balls have been released from current loading device in last cycle.

After installing the filled loading device at the operating position near the ball mill, it is ready to perform technological operations of supplying different-sized balls. To do this, the device still needs to be connected to the power cable for the supply of executive electromagnets and the data cable for receiving pulses of passing balls into the technological unit, this is done by the ball mill operator.

Thus, this device can solve the problem of improving the energy efficiency of technological units and increase their performance in the finished product.

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FGX AIR-VIBRATING SEPARATORS FOR CLEANING STEAM COAL – 21ST CENTURY TECHNOLOGY

Introduction

At the break of the 20th and the 21st century, Chinese company Tangshan Shenzhou Manufacturing Ltd. constructed dry separators FGX type. Their construction combines a vertical jet air classifier and a vibration screen. In Poland, there is an experimental installation in the Katowice Branch of Institute of Mechanised Construction and Rock Mining) with the capacities 10 Mg/h and two industrial installations in Gdańsk and Toruń with the capacities 30 Mg/h [1]. FGX air-vibrating separators are devices for dry enrichment of coal. Because they work in the thickness range from 2.0–2.2 g/cm^3, they are commonly called - *devices for dry deshaling of excavated coal*.

Principle of operation

Excavated coal is transported by the feeder to a perforated and slatted separator working plate, which is made to vibrate by a vibrator. Under the plate, there are several air chambers fuelled by a centrifugal fan. The air passes through holes in the plate and creates air jets. Under the combined forces of vibration and air jets, the excavated material is lifted. Owing to the vibration and proper slating of the working plate, the material is rotating, which allows the grains to pass into separate containers and forces the grains to pass between separate slates (Figure 1).

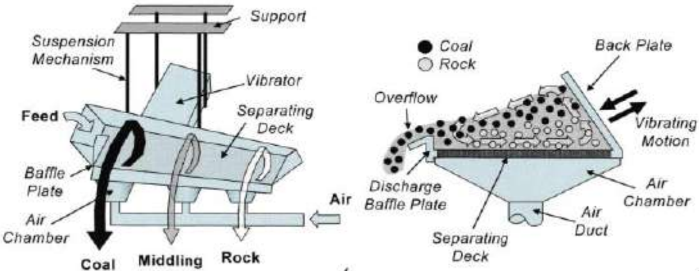

Figure 1. FGX Air-Vibrating Separator [2]

Depending on the thickness of grains, the materials are divided. Lighter material goes to the surface of the suspension layer, and fractions of grains with higher densities are in the lower parts. Fine material and the air is an autogenic medium. A suspension of air and solids is formed, sometimes referred to as fluidized bed. There is a condition for restricted movement within the bed, depending on their size and thickness. Fluidization effect is used which occurs as an effect of the density of fine grains forming the suspension and the bigger grains, which causes an improvement of separation of thick grained fractions.

Application & Economic efficiency

FGX air-vibrating separator can be used as:

- individual facilities for enrichment of excavated coal,
- “bypass” in the enrichment sections in coking coal processing facilities. They are used to reduce the quantity of gangue before the feeding of partially enriched raw materials to the water-based enrichment processes.

FGX separators, owing to the dust removal modules, allow for removal of the finest grains of the raw materials, which will not be introduced into the water system. Owing to this, coal sludge management is much less problematic. Dry dust can be a trading product and can be added to a concentrate. FGX dry separators are perfect for improving the quality of traded energy industry coal products for individual recipients. Dry separators can be used to adjust the raw material if it has varying levels of gangue. This happens in the case of mining several layers and a lack of selective feeding of the raw material to the processing facility [3].

Economic efficiency analysis of dry coal separation technology was a subject of many reports developed in the USA, South Africa and Poland. Comparative studies have shown that the investment outlays of dry separation technology account for 25% of investment outlays of wet enrichment methods (4 times lower), while operating costs of dry separation technology account for 32% of operating costs of wet enrichment methods (3 times lower).

Application of the dry separation process allows for [4,5]:

- lowering of investment and operational costs of coal processing facilities,
- effective deshaling, improvement of calorific value and a reduction ash content of the clean coal, removing pyrite sulphur and mercury and other ecotoxic elements,
- secondary enrichment of the trade product for high-quality ecological qualified fuels,
- effective use of separation rock products as substitutes for natural aggregates, recovery of coal substances from mining waste dumps.

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ITABIRITIC ORE WASTE REFINING THROUGH SCREENING

Mining is present in the evolutionary process of humanity, from the emergence of the first tools used by man to the development of cutting-edge technologies in the various spheres of modern society.

In recent years, with the austerity of non-renewable natural resources available and the advancement of technology, significant changes have occurred in business strategies. These transformations are consequences of factors that have been influencing the reduction of environmental and economic damages, making it possible to have a better perception of the opportunities of rational exploitation of available mineral resources.

According to the National Mining Agency (ANM, 2018), 90 % of the production of metallic substances in Brazil comes from the states of Minas Gerais and Pará, iron corresponds to 71.1 % of this production.

The processing of iron ore generates as products concentrate (iron), sterile and tailings. The latter need proper disposal in order to reduce the environmental impact caused by mining activity. However, when it comes to the disposal of tailings care should be even greater.

In the last 5 years, two events involving the breaking of tailings dams have killed hundreds of people in the state of Minas Gerais, thus raising the need for improvements in dam structures and construction methods, and the disuse of this disposal method itself. decommissioning existing dams that pose a risk of disruption.

Decommissioning is a costly activity for companies, but this cost can be minimized if the tailings are reprocessed for economically viable iron.

In the past the world market used high-grade iron ore, this factor, coupled with less efficient extraction and processing technologies, made the older tailings dams highly iron ore-rich, which makes their processing feasible.

The use of tailings is of interest not only to the private producing sector, but also to the public sector, given the interference of waste stocks, especially in environmental matters. The efficiency of the environmental management of the mining companies is related to the implementation of preventive actions for the mining dams, considering that, from an environmental and economic point of view, it is better to prevent the environmental damage than to look for alternatives for its correction, in which the Financial costs are higher in an attempt to recover degraded areas.

In order to create environmentally and economically viable solutions for decommissioning dams, this research aims to experimentally analyze the separation between Silica (SO_2) and Iron (Fe) from the tailings studied.

The research object presented was sampled at the Itabirusul dam in Itabira - Minas Gerais. The waste was characterized as inert, ie not considered a contaminant, according to NBR 13028 (ABNT, 2017), as it is basically composed of silica and iron ore, originating from itabiritic iron ore.

In the preparation of the samples, the material was taken to the sun to dry and then it was sieved in a 30 cm 0.6 cm rim sieve, with the purpose of extracting organic and inorganic impurities, thus making the material suitable for its characterization. Then the material was properly homogenized and quartered until it reached the ideal weight for 4 sieves, a key process in this study.

Following the above procedures, the sample was sieved for 15 minutes in a suspended sieve. The sequence of sieves used follows the order of 0.3; 0.15; 0.075; 0.053; 0.045 mm.

As a result, it was observed that in the particle size range 0.3mm and 0.15 mm the retained material corresponds to 58.3% of the initial mass, but composed almost entirely of silica only. In the 0.075mm range, the retained material corresponds to about 1% initial mass and average iron content. The retained material in the 0.053mm range also presented an average iron content. It was the materials, retained and passing in the range of 0.045mm, corresponding to about 10% of the sample, that presented a very high iron content. This pattern was repeated in the 4 screening trials.

Thus, it is observed that only with the screening of the tailings was it possible to obtain an efficient separation of silica and iron in the beginning and end bands of the sieve sequence. As for the intermediate bands, it is necessary to better distribute the particle size so that a better separation efficiency and observation of their effects is possible.

As a future objective, the authoring team of this paper will deepen the analysis of the contents in each particle size range of the sample through X-ray Diffraction (XRD), so that the efficiency and viability of separation screening can be concluded with more efficiency and assertiveness. silica-iron tailings.

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TECHNOLOGICAL PREMISE OF EFFICIENCY EMPLOYMENT OF RESOURCE-SAVING TECHNOLOGIES AT MINING AND PROCESSING FACTORIES

Humanity with the help of technological systems extracts and process natural resources and consuming them, satisfy their needs. But at the same time, due to imperfections in the "technology-consumption" relationship, various economic and environmental crises arise. To adjust the goal of technology development and reorientation from process innovations to the rational use of resources, a transition to the use of more resource-saving technologies should be made.

Analysis of a possible transition to the use of resource-saving technologies in mining and processing plants makes it possible to systematize the main directions of reorientation from process innovations to rational nature management:

- maximum metal recovery during ore beneficiation;
- the principle of comprehensive use of resources;
- improving the structure of consumption and saving resources;
- increase the metal content in the concentrate;
- minimization or complete elimination of resource losses;
- establish chains of re-enrichment of tailings;
- introduction of treatment facilities.

It should be noted that the complete extraction of metal in ore is the most urgent task in the mining industry. Production losses during enrichment, unlike other stages of metal processing, where waste can be recycled efficiently, can be irretrievable or ineffective.

Incomplete metal recovery during ore beneficiation and some situations leading to inefficiencies in the re-mining of unrecovered metals are a result of the imperfection of the beneficiation technology. Today, the highest share of metal losses is accounted for enrichment - on average, in the metal production process chain, losses are about 60%, of which 23–25% during mining, 24–26% during enrichment, 7–8% during blast furnace redistribution, 3– 5% for steelmaking and 2-4% for rolling production. Due to the level of existing technological excellence, minimizing this indicator is less possible. Since re-extraction of metal in non-ferrous metallurgy sectors requires more costs. And the under-extraction of metal itself leads to an increase in the cost of extracted metal - to produce one ton of metal it is necessary to mine from 1 to 5 thousand tons of overburden, in the process of beneficiation of ores containing non-ferrous metals, a dump in the amount of 30 to 100 tons is formed.

Ore mining and processing volumes have led to such a situation that the dumps of mining enterprises that are designed specifically for waste turned into deposits of mineral resources. According to experts, in the tailings of non-ferrous metallurgy there are thousands of tons of various types. In some places, the metal content in these wastes is high in comparison with the mined minerals.

A feature of natural resources is that they exist in the form of poly metals, i.e. includes many types of metals. Thus, the principle of orientation of enrichment processes on the extraction of one type of metal is an irrational way of nature management.

The use of resource-saving technologies in the enrichment of minerals contributes to the optimization of the existing system at the processing plants. The use of natural resources in all sectors of the economy has brought the issue of maximum metal extraction to the forefront. By maximizing metal recovery of increasing their concentration, relative reductions in ore production can be achieved, as well as lower levels of environmental pollution.

Thus, using resource-saving technologies in mineral processing, certain results can be achieved regarding improving the efficiency of enterprises, the rational use of mineral raw materials, reducing losses and waste and ultimately, reducing the environmental impact of mining enterprises.

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TECHNOLOGY OF COMPLEX PROCESSING OF PHOSPHOGYPSUM

The technology of complex processing of phosphogypsum includes the manufacture of building materials (slabs, blocks) for non-residential premises and the associated extraction of rare earth metals [1, 2, 3].

According to the data of Ukrainian scientists, it is not possible to obtain high-quality products of the 100 mark by the method of β-polygurat casting, then at the stage of formation it is envisaged to carry out semi-hard pressing on the knee-lever press. With this method of formation it is easy to switch to the production of other types of products, for example, gypsum tile, architectural products, etc.

The production of β-half hydrate is carried out by heat treatment in a gas stream at an initial temperature of 300°C and a final 100°C. At the same time it is possible to dehydrate small (up to 0.1mm) grains of gypsum of a second due to intensive mass and heat exchange with the environment.

Phosphogypsum is developed by an excavator on the tailing dump. If it is possible to resolve positively the issue with the city administration and PJSC «Rivneazot» about the placement of production in the immediate vicinity of the quarry, it is expedient to provide transportation by conveyor transport, the cost of which will not exceed 30% of the cost of traditional transport by road. At the same time, the efficiency of exploitation of excavating machinery will increase. In the development of phosphogypsum in this case, periodic action excavators may be replaced by cheaper multi-axle loaders of elevator type [4].

In this feasibility study, we will rely on a more unfavorable case that results in the use of motor vehicles. The phosphogypsum extracted from the deposit is loaded into a receiving hopper with a volume of $20m^3$ with a belt feeder with a width of 1000mm. Dosing of the flow of raw materials is carried out by the method of the embossing.

Crushing and partial dehydration are carried out in a MMT-1300 mine mill with a productivity of 16-20 t/h with a rotor diameter of 1300 mm and a power of 150 kW with a rotational speed of 960rpm. Since the source material in the form of phosphogypsum has good crushing it is assumed that the resistance would be about two months.

It should be emphasized that all machines, apparatuses and technological pipelines operate in an aggressive environment, so they must be made of acid-resistant steels.

Purification of air flow at the first stage is carried out in a battery with 8 cyclones in diameter of 800 mm, and then in a wet hydrocyclone with a diameter of 1500 mm. This purification allows you to allocate 99,8% of the particles, which can be considered satisfactory. The smoke exhaust fan must have the following characteristics: performance 40 thousand m^3/h, vacuum 450Pa. The mill is supplied by combustion products of combustion of fuel (gas) with a temperature of 800 °C under vacuum of 300-320 Pa.

After a series of cyclones, the phosphogypsum continues to be dehydrated in a bunker of $30m^3$ where its temperature drops to 100°C and below, and the sludge of the wet cyclone is collected in its lower part, from which it is transported by gravity. From the hopper, the gypsum is fed into the blade feeder into a blade single-acting mixer with a capacity of 16-20t/h, where partially sealed with wet sludge and technical water to a moisture content of 20-22% with a theoretical moisture content of 18,6%. Pressing of products is carried out by the knee-lever press with the productivity of 5-7 thousand pieces of conditional brick per hour, which corresponds to the productivity of the mill. Pressure of pressing up to 200 kg/sm^2. Ready-made products are packed on pallets by an automaton-inlayer and transported by a forklift truck to the warehouse of finished products, which solidify for 2 hours to the transport strength of 30 kg/cm^2. The loading of the motor transport of finished goods is carried out by an onloader with a carrying capacity of 1t. The possibility of using mobile power loaders with cable power is foreseen.

All technological equipment works under vacuum, therefore there is no dust formation.

To clean the outflow of air from the acid pairs, it is necessary to install a rectification column or scrubber after the smoke exhaust fan, in which harmful vapors will be absorbed into the water. It is supposed to separate about 1% of acids by weight from the weight of phosphogypsum with a content of 1,5%.

Power supply is supposed to be made from existing power lines of a chemical plant without the construction of a substation. The length of the cable line is up to 300 m. Water supply is also organized from existing networks. The need for technical water up to 15% by weight of phosphogypsum or 15000 m^3 per year. In the day will be consumed technical water 58,4 m^3. The amount of drinking water is calculated as 0,5 m^3 per worker. When calculating the number of employees 28 people need 14 m^3 per day. Hot water comes from existing plant networks. Heat supply is performed from existing networks in the form of a pair. It is possible to provide heat supply to workplaces and electric heaters. In this case, the power of electric heaters will not exceed 30kW. The profitability of production is 5,7 times higher than standard profitability.

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ANALYSIS OF TRADITIONAL METHODS FOR BENEFICIATION OF GOLD-BEARING REFRACTORY ORES

In the mineral resource complex of the deposit located on the territory of the Republic of Kazakhstan, there is a tendency to increase the imbalance between mining and the increase in off-balance reserves of gold-bearing ores, which ultimately poses a serious problem for the development of the country's economy. A large part of the reserves of rich, easily enrichable gold-containing mineral raw materials has been exhausted. The Kazakhstan gold mining industry is constantly involved in the processing of new gold ore deposits. In most cases, the ores of these deposits are classified as hard and especially hard. According to preliminary data, the share of these ores is more than 30% of all reserves, which means that due to the wider involvement of refractory gold and complex gold-bearing ores in the 21st century, a significant increase in gold production can be achieved. A significant part of the indigenous gold reserves is composed of refractory ores in which gold is in a finely disseminated state in sulfides and rock-forming minerals. The particle size is from tens to thousandths of a micron. Gold is mainly associated with pyrite and arsenopyrite; therefore, it does not dissolve during cyanidation with a standard grinding fineness of 80-95% of the class -0.074 mm and even with ultrafine grinding, without the use of preparatory technologies before cyanidation. Also, a preparatory stage is required, which will improve the indicators of further extraction to acceptable values. The preparation consists in opening the dispersed impregnation of gold, in various ways, to ensure access to it of a cyanide solution. Since the main mineral carriers of gold are sulfides and arsenopyrite, the preparation usually consists in the oxidation of a sulfide or arsenide matrix. Ores that contain active carbon also require oxidation or passivation. Primary gold ores are called refractory if the extraction of gold using traditional cyanidation technology does not exceed 80%. The persistence of gold is primarily associated with its dispersion in sulfide minerals and arsenopyrite in isomorphic or dispersed form, which impedes the access of the leach solution.

The second reason for persistence is the presence in the ore of significant amounts of a carbonaceous substance, which is a gold sorbent, or other natural sorbents that can sorb precious metals from cyanide solutions, which thereby increases the loss of gold and silver with the tailings of the technological process. These two factors can occur simultaneously, which complicates the processing technology. Such ores are commonly called double refractory ores. The problem of processing these ores is very relevant today. Ores and concentrates that contain organometallic, cluster, colloidal and other chemical and composite compounds that also complicate the technological extraction of useful components should also be classified as technologically refractory mineral raw materials. The prevalence of stubborn gold is high. The largest and most famous representatives in Kazakhstan are the Vasilkovsky deposit of persistent arsenic-sulfide gold ore.

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STUDY OF THE RELATIONSHIP OF THE DEGREE OF GOLD EXTRACT FROM GRANULOMETRIC COMPOSITION OF THE PROCESSED ORE

Ore preparation has a decisive influence on the final results of the processing of gold-bearing ores using all known methods of extraction. A special role is given to this process when using the heap leaching method. Incorrectly substantiated ore preparation parameters lead to irretrievable losses of gold extraction from the heap. In this regard, special attention is paid to the selection of parameters for the preparation of ore material before placing it in a pile.

The size of crushed ore is characterized by a grade of +3.35 mm, which determines the degree of opening of gold particles and, ultimately, the degree of its extraction from ore. From the size of the final material depends on the permeability of the heap, the filtration rate of solutions, the stability of individual tiers and the stack as a whole. The size of the final product has a high correlation with the content of small classes in the finished material, the degree of extraction of gold from which is higher than from large classes.

Table 1 presents the results of sieve analysis, gold content by size class and gold recovery for each size class of samples of the finished product entering the stack.

Table 1

Particle size distribution of the crushed material

Size class, mm	Class yield,%	Gold content, g / t	Gold recovery,%
+3,35	2,4	0,96	19
-3,35+1,7	28,5	1,04	27
-1,7+1,18	13,3	1,14	39
-1,18+0,60	18,5	1,15	49
-0,60+0,30	12,0	1,28	63
-0,30	25,3	1,72	81
Всего	100,0	1,27	50,4

The effect of these factors on As can be seen from Table 1, the gold content by size class is not uniform, and in small classes it is almost twice as large as in large ones. Extraction of gold by size class is also not uniform. In small classes, the degree of opening of gold is much higher, which certainly affects the extraction of gold.

The filtration rate of the solutions depends on the size of the finished material and the degree of agglomeration before laying in heaps. It determines the possible density of irrigation of the surface of the ore heap. At low filtration rates and high irrigation density, the ore pile is saturated with solutions and the threat of its destruction is created.

Leaching ore crushed to a fraction of less than 4 mm usually encounters the problem of filtering solutions in the heap, which leads to a decrease in the degree of gold recovery.

The content of clay particles, as well as a large amount of fine fraction formed during the crushing process, can create the problem of poor permeability of the heap of ore. This is due to the migration of small particles inside the ore laid in a pile until the formation of a poorly permeable layer that prevents the solution from passing to the ore.

directly below it. The presence of fines and clay particles is a decisive parameter in the operation of a multi-layered stack of ore.

Very often, as a result of this, there is a decrease in the extraction of gold, or at least it can lead to a lengthening of the leaching time to obtain maximum extraction of gold from the ore in the heap.

The most acceptable way to avoid such a situation is the agglomeration of the ore mass. In general, the agglomeration process consists of adding binding agents and a solution to form ore pellets. Typical binding agents are Portland cement and lime lime. Various high molecular weight A distinctive feature of the technology of heap leaching of gold from off-balance ore is that the ore of a very small fraction (95%), less than 3.35 mm in size, fits into a pile of ore. With such a crushing size, the problem of heap permeability comes to the fore, and the sintering process is crucial, as it directly affects the filtration properties of the ore massif.

In connection with the above, a considerable amount of research was carried out to determine the optimal agglomeration options, in the course of which the influence on the quality of such factors as the amount of cement, moisture during gold recovery has also been studied.

The pH of leach solutions is the most important indicator of the magnitude on which hydrolysis depends, and ultimately, solvent consumption. The pH is regulated by lime added to the ore after grinding and placed in a pile with it. This figure depends on the time of leaching

As a rule, in order to obtain quick information about the technological properties of ore, tests are carried out by cyanidation in rotating bottles with free access of air. The results of bottle research allow us to obtain data on the extraction of gold at different material size.

A sample of the ore was placed in a bottle and poured with a leaching solution with concentrations of sodium cyanide 1 g / l at the rate of 40% solids. A high concentration of cyanide aims to determine the final extraction of the metal in the solution. In heap leaching of metals from ores, the solution fed into the ore mass moves under conditions of partial filling of the pore space, covering the heap only with a thin film. Based on the research conducted, the following conclusions can be made:

Of the total amount of gold in the residues 32-42% of gold was extracted from the enrichment product in the process of washing in the form of relatively large particles (0.5 mm or less) metallic gold. The enrichment product consists of approximately 75% pyrite, 25% arsenopyrite and incidental iron impurities, containing 28–38% of gold not subjected to cyanidation.

Thus, the total extraction of gold after all the tests in the bottles and the mineralogical tests carried out indicates a strict dependence of the extraction of gold on the particle size.

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GRAVITY DRESSING OF REFRACTORY GOLD ORE

The enrichment process is a single system in which the individual elements are interconnected. High results can only be achieved with a systematic approach, which will take into account the interaction of system elements, that is, in this case, a full range of processes. Gravity enrichment is undoubtedly one of the most famous processes. It is to him that history owes to the fact that gold became the first metal with which mankind became familiar several millennia BC.

Nature herself took care of this, freeing gold from the minerals that enclose them in river beds and streams that flowed through the gold-bearing rocks, giving them such an attractiveness that our distant ancestors could not help but pay attention to. Mass production of gold from placers began with gravity enrichment methods, after which these methods actively “stepped” into the factory technology for processing ore from primary deposits.

At present, gravitational concentration of gold is widely used in gold recovery plants (ZIF) in all countries of the world, including those that account for the majority of the mining of this metal (China, Australia, the Russian Federation and Kazakhstan.

In 2005, an analysis was made of the performance of more than two hundred gold mining plants from most gold mining countries, including Kazakhstan and other republics of the countries.

By the nature of the processed raw materials, these factories are divided into 3 groups:

Group 1 includes enterprises engaged in the extraction of gold and, together with it, silver from relatively technologically simple quartz and quartz-sulfide ores containing precious metals mainly in a cyanide-soluble form.

Group 2 includes gold processing plants processing pyritic and arsenic-pyrite ores with finely disseminated gold in sulfides refractory for cyanidation, as well as ores containing a sorption-active carbonaceous substance.

Finally, group 3 consists of enterprises for processing complex ores containing, along with gold and silver, heavy non-ferrous metals (copper, lead, zinc, antimony), as well as uranium.

For research, two samples of a mixture of rich ores (5.65 g/t) and poor ores (1.2 g/t) were prepared at the Vasilkovsky deposit. A sample of ore mixture № 1 (50% rich ore and 50% poor ore was mixed) had a gold content of 5.5 g/t. Sample № 2 of the ore mixture in the ratio of 25% rich ore and 75% poor ore had a gold content of 2.3 g/t.

The results of gravity enrichment methods (on a screw separator, jigging machine, concentration table) are given in table 1.

Table 1

The Results of experiments on enrichment on a screw separator

Products	Exit		Gold Content, g/t	Mass of gold, g	Distribution %
	g	%			
1-concentrate	43.0	2.15	21.5	0.4622	13.39
2-concentrate	65.0	3.15	7,6	0.2470	7.15
Tails	1892.0	94.6	2.9	2.7434	79.46
Ore	2000	100	3.5	3.4526	100

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ENERGY SAVING IN MINING BY USING RATIONAL OPERATING REGIME OF ROLLER-BIT DRILLING MILLS

At the mining enterprises of Ukraine one of the most common methods of drilling mechanization is using a roller-bit drilling mills. As it is known that operation of roller-bit drilling mills is followed by the equipment increased vibration, which affects negatively its longevity, increases energy consumption and deterioration of working conditions of the operating personnel [1, 2].

During the research of vibration of the drilling rig and increasing energy costs amplitude of oscillations of the drill rods was simulated. Oscillation of the drill rods 16 m in length (Ø215×51.5 mm) was simulated in the Solidworks environment using the FFEPlus application.

Calculations showed that at a resonant frequency of the first mode (in round figures, 1.885 Hz on axes x and y) rotations of a drilling flight with number of turns n=113 min^{-1}, vibration amplitude a_1 had to reach 171 mm in the center of a drilling flight. However, due to restriction of vibrations resonant amplitude by bore hole walls, the rod of drilling flight starts scraping on the bore hole wall by the external surface in the vibratory-percussion mode, and not only on the first mode, but also on the second one with a frequency of 7.5 Hz and amplitude of 21.7 mm that also exceeds the available gap S=14.75 mm between a bore hole wall and rod. Such mode causes extreme dynamic loads on the rig equipment and as a result of increased energy consumption.

The evaluation actual of energy consumption when drilling is conducted on the RBDM-250MNA-32 rig (Ore Mining and Processing Industrial Complex “Ukr-mekhanobr”, Kryvyi Rih) in three modes: subresonance, at drilling flight resonance when bore hole gumming. The feeding pressure, drilling flight rotation number, values of voltage and rotator flow were fixed from the display in the operator cab. From the evaluation, it is seen that in the resonance mode, the energy consumption increased from 23.3 to 31.2 kW, i.e. by 34% in comparison with the subresonance mode. In order to reduce the vibration of the drilling rig, the bore hole was gummed (it is filled with a dense abrasive slurry). In this case, energy consumption increased from 31.2 to 35.5 kW, which is by 18% more in comparison with the resonance mode.

The investigation of drilling rate and energy consumption at various drilling regimes was conducted also in the conditions of Ingulets GOK on the RBDM-250A mill No 87 (drilling by two heavy rods of Ø215×51.5 mm in the rock with a strength of 16-18 according to M. M. Protodyakonov scale with drilling flight feeding pressure of 220 kN). Previously, considering wear and actual rods sizes, modeling of natural vibrations and amplitude of resonant vibrations was carried out, therefore the following numbers of rotations are determined: *a*) - subresonance mode n_{subr}=100 min^{-1}; *b* - resonance mode - loss of vibration resistance n_{res}=115 min^{-1}; *c* - superresonance - nominal detuning from the resonance mode n_{super}=130 min^{-1}.

At drilling rate determination, depending on the drilling modes, this parameter was registered after the second rod adding in order to avoid the errors because of the bottom hole top layer, which is partially destroyed after the previous blast. Drilling time along the length of the second bar of 8m was determined, and then, the drilling rate and specific energy consumption were calculated. Dependences of drilling speed and power consumption on operating mode of RBDM-250A mill are 38,4 kw in 100 min^{-1}, 46,3 kw in 115 min^{-1} and 43,6 kw in 130 min^{-1}. From these dependences, it is seen that the energy consumption increases in the resonance mode by 20.5%, and the drilling rate decreases by 4% in comparison with the usual drilling mode.

The energy consumption decreases by 7% in comparison with the resonance mode on a frequency of detuning by increase in rotations number. Thus, the drilling rate increases by 15.5%. Specific energy consumption and drilling rates of the rig are 2.4 kW·h/l·m. of bore hole for Subresonance mode, 3,0 kW·h/l·m of bore hole for Resonance mode and 2,4 kW·h/l·m. of bore hole for Superresonance mode. It is seen that specific energy depending on the drilling modes.

Specific energy consumption increases considerably (by 25%) in the resonant mode (from 2.4 to 3 kW·h/l·m of bore hole). Specific energy consumption in the superresonance drilling mode is reduced to the value of the superresonance mode. However, the superresonance drilling mode is more rational in comparison with subresonance one. In this mode, the drilling rate increases by 11% in comparison with subresonance mode.

To summarize one of the way to reduce energy costs for drilling blastholes is using rational operating regime of roller-bit drilling mills. The drilling modes by heavy rods of the flight length of L_f =16 m with number of turns n=125-130 min^{-1} at standard axial feeding pressure of drilling flight P_0=200-220 kN provide the minimum specific energy consumption of 2.4 kW·h/l·m of bore hole, maximum drilling rate v_d=18 m/h and the absence of vibration rigidity loss of drilling flight, extreme loads on the equipment and operator workplace.

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PROFITABILITY OF MAINTENANCE CONDITIONELLE

As a result of three years of studying (i.e. 720 days corresponding to 11520 hours of unceasing functioning) of installation (soutireuse) which has been subjected remedial maintenance, set of experimental data has been received. This data has allowed to estimate a phase of life of this installation which is a phase of ageing and-or deterioration in which the price of failures increases quickly depending on time.

As a result of the analysis of experimental data according to model WEIBULL value MTBF (Mean Time Between Failure) to equal 9748 hours (with η=11000 hours in the range of time of trouble-free functioning TBF) has been found. And the question which henceforth arises, consists in learning, whether there is the periodicity of regular preliminary maintenance giving cost on a unit of equipment below cost of remedial maintenance.

To answer this question cost C_1 for a unit of equipment has been calculated at remedial maintenance

$C_1=(p+P_1)$ / MTBF and an indicator "r": $r=P_1/p$,

where p - average cost of intervention of remedial maintenance; P_1 - the average cost of equipment downtime connected with remedial maintenance (at elimination of refusal of the equipment).

Results of calculations are presented in the below-mentioned table.

P	P_1	C_1	R
233909$	223751$	46,95$/h	0,96≈1,0

Using tables KELLY, we find economic periodicity θ and the corresponding cost C_2 of regular maintenance of a unit of equipment. Considering that $C_2=(p+P_2)/\theta$, we find from this P_2 - cost of impossibility of use (idle time) of the equipment at regular maintenance (we consider that cost of regular maintenance p is identical to cost of remedial maintenance). If r=1,0, then approximately

$$C_2/C_1=0{,}9,\text{ whence } C_2=0{,}9\times 46{,}95=42{,}26\$/\text{h}$$

$$\text{and } \theta=\eta\times x=0{,}85\times\eta=9350 \text{ hours}$$

We recommend to service of maintenance of the equipment of the enterprise to try to develop the policy based on conditional maintenance to achieve the object ‘a zero of refusals’.

For this purpose it is necessary to define profitability of conditional maintenance in relation to remedial maintenance or in relation to regular maintenance. Conditions of profitability the following (C_3 - cost of conditional service for a unit of equipment)

$$C_3\ C_1;\ C_3\leq C_2 \text{ with } C_3=(p+P_3)\ \tau,$$

where p - cost of conditional maintenance (identical to cost of remedial maintenance); τ - periodicity in conditional maintenance. τ=0,9×MTBF.

The maximum cost of impossibility of use (idle time) of equipment P_3, for an estimation of profit of conditional maintenance in relation to remedial maintenance or regular maintenance (the below-mentioned table) is defined.

Profitability in relation to remedial maintenance	Profitability in relation to regular maintenance
136796$	177985$

The conclusion: for the studied conditions, doing comparison between both expenses at the equipment downtimes, caused by conditional maintenance, we find that it is more profitable in relation to regular maintenance and in relation to remedial maintenance.

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UDC 620

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FUNDAMENTALS OF CREATING A MOBILE SYSTEM OF GEOTECHNICAL MONITORING

The problem of man-made contaminated areas as a result of the functioning of industrial and military complexes requires immediate intervention by scientific, industrial and educational organizations. Particular attention should be paid to areas where, in addition to environmental problems inherent in production processes, there are other types of hazards, such as those related to hostilities (mining, use of phosphorus mines, destruction of urban infrastructure, etc.). Therefore, the development of a reliable complex for collecting and processing information on potential and real hazards in such territories, both for biotopes and for humans, is an urgent scientific and technical task [1].

The main methods for detecting objects of different origin include: non-linear radar, sub-ground, ultrasonic, laser locators, thermal imaging, chemical analyzer, nuclear resonance-based instruments, etc. None of the above methods provide the required probability (0.97) required by the UN.

These methods and means of soil surface research in difficult conditions have disadvantages that affect the accuracy and speed of monitoring, which is unacceptable for the localization of dangerous devices with the subsequent determination of the degree of degradation of technogenically contaminated sites.

The purpose of the project is to develop an algorithm for the functioning and creation of an industrial-experimental sample of the information geomachronous monitoring complex of man-made contaminated territories.

The general drawback of the automated complexes developed is the focus on establishing common chemical and physical parameters without the possibility of locating dangerous techno genic devices.

The main purpose of the geoinformation complex is to study the state of the surface of the biotope of technogenically transformed territories, including those that have undergone military intervention, in order to identify and identify chemical and biological transformations of soil and objects that are hazardous to humans and the environment.

The developed broad (dual) geomachron complex is an air-to-ground information system, which contains the advantages of air systems, such as speed and informative collection of information, as well as terrestrial, to which the accuracy and high probability of the obtained data relate.

Structurally, the complex will consist of three main blocks: the air complex, the ground and the control system. The peculiarity of this complex is the application of a fundamentally new method of visually-automated soil condition research, which allows to identify and locate dangerous technogenic soil transformations with the necessary accuracy. The absence of an operator from the study area minimizes the probability of damage from dangerous man-made processes and devices. Most of the dangerous objects in the upper soil layers are made of polymers, which makes it impossible to use electromagnetic localization methods. The use of contact search methods allows you to pinpoint objects of any nature with high precision and accuracy.

The main advantages of the developed complexes among the existing ones are: low cost; guaranteed safety for operators; simplicity of construction; absence of serious damage when exposed to dangerous factors; simplicity of management; mobility; transportability.

To create this monitoring system, it is necessary to carry out thorough theoretical and experimental studies on the mechanics of contact interaction of indenters in different types of soils with different variants of laying dangerous objects, which will form the basis of the algorithm for recognition of these objects. The speed of the system and the operational data transfer will be provided by the air complex.

At this stage, the components of the geotechnical monitoring system of the sampled territories are tested on the basis of the experimental wheelbase (Fig. 1).

Establishing a geomachatron complex for territorial monitoring will help to reduce human capital losses, and the development can be beneficial for the military of Ukraine and the world. Low cost, mobility and transportability will allow the use of the complex by virtually all units of the armed forces. Particular interest in this complex has special forces units that perform latent engineering reconnaissance terrain, traffic route for the presence or absence of dangerous devices and other man-made hazards.

Fig. 1. Experimental wheel base of the geotechnical monitoring system of the exemplified territories

The main stages of work on creation of mechatronic monitoring complex of technogenic - dangerous territories will be development of: monitoring systems for dangerous devices; information management system; communication and command transmission systems; motion control systems; technical vision systems for motion control; systems of top-tethering and orientation; power supply systems.

The basis of reliable operation of mechatronic systems is the power supply system, which provides energy to the entire complex of installed equipment. The main energy consumers of this equipment are data acquisition and movement systems. When operating the data collection system, the complex is in a state of rest, which allows to reduce the overall power of the complex.

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FIBER CONCRETE IS AN EFFECTIVE MATERIAL FOR THE MANUFACTURE OF PUMP HOUSINGS

Almost at all enterprises of a mining and processing complex centrifugal pumps pumping aggressive liquids and the pulps containing a considerable quantity of the abrasive particles are used. To manufacture the housings of such pumps, taking into account the severe conditions of their operation, they use expensive alloy steels, which significantly increases the cost of the pump.

Monitoring, theoretical and experimental studies conducted by the authors have shown the prospects of using composite material - fiber concrete for the manufacture of pump housings.

Fiber concrete is a composite material obtained by adding fiber to the polymer concrete. The fiber is manifested in this case by micro-fittings, evenly reinforcing polymer concrete in all planes, increasing the strength, impact resistance, resistance to aggressive substances on the polymer concrete. In addition, the fiber reduces the formation of shrinkage cracks [1].

Polymer concrete, which is the basis of fiber concrete, is a composite material consisting of 80 ... 90% aggregate and filler, and the binding agent is a polymer (epoxy resin).

The efficiency of fiber concrete with hardening fibers depends on the nature of their arrangement and orientation in the product. Two types of reinforcement can be distinguished: with an obvious orientation of the fibrous filler in the direction of the acting forces and reinforcing with fibers (mostly of limited length) with an arbitrary orientation in the product.

Increase of tensile strength in the latter case is explained by the fact that the fibers at free orientation and sufficient uniformity of distribution in the material are able to perceive the efforts of almost any direction and thus will prevent the formation and development of cracks in the product. Fibers slow down the movement of microdefects in fiber concrete.

To obtain the product reinforced with steel fiber, it is necessary to observe the following technological operations: first, it is necessary to stir the dry aggregate with the required amount of fibers. After that, a binding agent is added to the mixture and mixing is performed until a homogeneous composition of the fiber concrete mixture is obtained [2].

The materials studied as aggregate for the preparation of fiber concrete were: granite crushed stone (black and white), butane crushed stone, quartz sand. As a filler were quartz flour, andesite flour, ED-20 resin filler, hardener-polyethylene polyamine PEPA. Fiber was used steel-wire, fiberglass and steel anchor type.

Humidity of fillers and aggregates of the polymer concrete should be no more than 0,5...1%. This is due to the fact that the strength and other properties of fiber concretes fall sharply with the use of wet aggregate: the thinnest layers of water on the particles of the aggregate worsen the curing of the polymer binder and reduce its adhesion to them. Therefore, the aggregates and fillers were dried in a drying cabinet at a temperature of 80-110 °C and must be cooled down to normal temperature before dosing.

The preparation of fiberconcrete mixture was made in the following order. At first, the binder was prepared, and then added it into the prepared mixture of aggregates. The binders were prepared within

30...60 seconds. The ready mixes were immediately loaded into a special container, where pre-mixed and processed modifying additives (with a small amount of binder) were already present. Mixing of aggregates with binding agent was performed within 1,5-2 minutes.

Phasic preparation of the mixture has number of advantages: the total duration of the mixing cycle and the consumption of resin are reduced. In this case, the binder is more homogeneous in composition and can be heated or cooled during preparation, which allows to regulate its viscosity and activity.

The carried out researches have shown the optimum addition of fibers in quantity of 3-5% by weight is optimal.

he results of experiments show the most rational composition of fiber concrete (by volume): granite crushed stone 50-52%, quartz sand 25-26%, quartz flour - 11%, steel fiber anchor type - 3-4%, epoxy resin - 10-11%, hardener - 2% This composition of fiber concrete has a density of 2200-2300 kg⁄м^3, and compressive strength of 230-240 MPa, bending strength of 80-90 MPa.

The final stage of researches was manufacturing of the housing “snail” of single-stage centrifugal pump from fiber concrete of the chosen composition. Special tooling was developed for casting the pump housing. Matrices were designed for casting two halves of the “snail”. After their casting they were glued together with epoxy glue thickened with andesite flour. The glued housing was covered with two layers of gelcoat.

The tests confirmed the technical feasibility and expediency of manufacturing centrifugal pump housings made of fiber concrete. They are much stronger than the housings made of metal, which makes it possible to reduce the thickness of the housing wall by 1.5 times, and therefore the weight of the housing itself.

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SPECIALIZED MACHINES AND EQUIPMENTS FOR THE COMPLEX MINING OF NON-METALLIC MINERALS

The current state of development of the Ukrainian economy is characterized by a shortage of construction sands, which leads to maintaining high prices for building materials with attendant consequences for the economy. At the same time, a significant number of industrially significant deposits remain not involved in development. A review and analysis of sand and gravel deposits located in Ukraine shows that the vast majority of them are watered [1, 2]. The economic feasibility of applying the method of sands developing by floating pump dredges during pressure hydraulic transport of rock to the place of its storage or processing has been proved in previous works [3, 4].

An analysis of the characteristics of the sands shows that the overwhelming majority of them in terms of the set of basic controlled physical and mechanical properties (particle size modulus, percentage of clay and dust particles, etc.) rarely corresponds to current regulatory documents. In this regard, there is a need for sand enrichment, which basically boils down to the removal of large inclusions from the rock mass, for example gravel fractions, lump clay, plant residues and shell impurities, as well as small dusty and clay (dispersed) impurities.

Thus, the development of an integrated approach to the development of watered deposits of non-metallic sand using rational approaches, both in mining and in substantiating the technology of mineral processing, is an urgent and important scientific and technical problem.

The simplest and most frequently used method of enriching construction sands from clay impurities during hydromechanical development is storage on alluvial maps. However, significant volumes of bulldozer and excavation works, a constantly changing hydrodynamic regime of the hydrotransport system, large losses of minerals due to dilution of clay particles accumulating in the alluvium map, do not allow recommending such technology for mass application.

Improving the quality of marketable products while lowering operating costs is possible through the use of special processing plants operating in combination with a pump dredger directly in the quarry [5, 6]. Of the variety currently available on the market for devices that implement the hydraulic method of processing non-metallic sands, enrichment complexes based on such plants are used in sand quarries: washing and drainage buckets from Shtichwe (Germany), installation for sand enrichment Fines Master Powerscreen (Great Britain), gravity sand washing MPG (Ukraine) [7, 8], direct-flow hydraulic classifier type GKD (Russia), screw classifier, etc.

The main criterion for adopting the technological scheme of mineral processing is energy intensity or cost, reduced to a unit of output. The most rational solution for each of the technological schemes is the one that provides the minimum specific energy consumption, and, as a consequence, the minimum cost of processing a unit of production.

During the design works by specialists of the design bureau Hydraulic technology of mining (Dnipro, Ukraine), a technical and economic comparison of some technological schemes for processing non-metallic sands using the above processing complexes was performed [9]. The calculations were carried out on the basis of real data received from mining enterprises, manufacturers and official representations that implemented enrichment complexes in Ukraine. As a result of such an analysis, it was established that it is rational to carry out the extraction and enrichment of construction sands with technological complexes consisting of a “dredger - slurry pipeline - classification installation”. As a result of the calculation of technical and economic indicators carried out during the design of technological schemes for the extraction and processing of construction sands, the economic efficiency of the use of enrichment complexes “gravitational sand washing” is substantiated. In some cases, to organize the process of enrichment of sand mined by floating pump dredger, it is enough to organize only the screening stage. In this case the rationality of the use of bar inclined grizzlies of the GNK type is justified.

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EFFECT OF CHEMICAL ELEMENTS ON THE PROPERTIES OF PIPE STEEL

The effect of titanium on the properties of steel is manifested depending on its state (in pure form or in the form of compounds). In hot-rolled steel, titanium with a total amount of 0,1% is oxidized in carbides (about 0,055%), nitrides (about 0.025%) and oxides (about 0,002%), as well as in partially solid solution (about 0,02%). After low temperature heating (negative hardening), the amount of titanium solid solution sharply decreases and usually does not exceed 0,003%

with the growth of titanium carbides. As a result, the impact toughness increases while reducing strength [1].

It should be noted that minor additives of titanium (0,01-0,03%) are effective in crushing the normal structure of the metal, as formed in liquid steel titanium nitrides serve as centers of crystallization.

The size of these particles of titanium nitride is about 0,02 microns. They serve as a barrier to grain growth and facilitate their fragmentation [2].

With small aluminum additives (up to ~0,1%), the critical brittleness temperature decreases, which is a result of the reduced content of dissolved nitrogen. After removal of nitrogen from the solution, the aluminum content practically does not have an effect on this indicator, but with its excessive amount of grain growth occurs, accompanied by an increase in the brittleness temperature.

Studies of the influence of various elements on the roperties of normalized low alloy steel type 16G2AF containing vanadium (0,10-0,15%), aluminum (0,02-0,04%) and nitrogen (0,013-0,024%), showed [2].

Increase in carbon content from 0.16 to 0.23% strengthens the properties, reduces the toughness at sub-zero temperatures and slightly affects the cold brittleness threshold (T50), which is explained by perlite share increase without changing the size of the ferrite grain.

Manganese with a content up to 1,7% moderately trengthens the steel, slightly increases the toughness and cold resistance, which is associated with the fragmentation of grain, so it is advisable to alloy steel of this type with manganese to 1,7%.

Silicon in the amount of 0,45 to 1,5% gradually increases the strength characteristics with a decrease in impact toughness and increased cold resistance, which is associated with some enlargement of the grain, a strong curvature of the crystal lattice and the elimination of this element.

Chromium up to 1,3% increases the strength, but decreases the yield strength, lowers toughness and increases the cold shortness threshold, which is associated with grain consolidation and increased perlite formation.

With an increase in the content of vanadium to 0,28%, there is a slight increase in strength properties with a decrease in impact toughness (when the content is more than 0,15%) and a slight increase of cold shortness threshold. Therefore, steel with carbonitride strengthening is advisable to microalloy vanadium to 0,15%.

Niobium in the amount up to 0,08% in steel with 0,09% V practically does not affect the mechanical properties, so the use of this element in normalized steel with carbonitride strengthening is impractical.

Nickel and copper in the amount of 0,5% each slightly increase the strength and impact toughness, but have virtually no effect on the cold shortness threshold, so it is possible to use these elements if necessary to further strengthen the steel with carbidonitride.

With increasing nitrogen content, the strength properties increase due to the fragmentation of grain. With a constant content of vanadium, the impact toughness decreases and increases the threshold of cold shortness. However, while increasing the content of vanadium, these indicators are improving, so to obtain optimal properties, it is necessary to ensure the complete binding of nitrogen with vanadium and aluminium.

Small molybdenum additives have little effect on the properties of such steel. Small titanium additives slightly reduce the yield stress due to the depletion of the metal with nitrogen, which goes into the formation of stable titanium nitride. Joint microalloying with nickel, copper and molybdenum is accompanied by a significant strengthening with the deterioration of cold shortness impacttoughness.

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A RESEARCH ON THE POSSIBILITY OF PLASMA-ASSISTED MACHINING OF MINING EQUIPMENT PARTS

Modern mining equipment parts are manufactured from steels and alloys with high mechanical characteristics, which leads to significant complications during their machining. The machining process is accompanied by heightened cutting resistance, high temperatures in the machining area, and a rapid loss of cutting ability of the tool. The reduction of the negative impact of those factors is possible due to the application of high-temperature plasma heating of the tolerance range cut-off material. Heating the work-piece with a plasma arc causes a number of phenomena: an increase in the ability of the metal being machined to plastic deformation; a decrease in its strength; the emergence of a system of structural transformations and thermal stresses in the surface layers of the work-piece; melting-off of the surface tolerance layer; changes in the friction parameters of the contact surfaces of the work-piece and in those of the tool; a change in the contact temperature; a change in the chemical composition of the surface layers [1]. The nature and the extent of the above-mentioned phenomena depend on the thermo-physical properties of the material being machined, the speed of the relative movement of the work-piece and that of the source of heat, as well as the thermal output of the plasma arc. By changing those parameters, it is possible to achieve an increase in the manifestation degree of the plasma effect aspect, whose dominance is considered appropriate for the work-piece material [2,3,4].

The method of plasma heating of the tolerance layer consists in melt-free plasma heating of the surface being cut at a given value of the electric current, softening of that layer, moving the latter to the machining area at the cutting speed and finally removing it with a cutter. Heating is done by an electric arc directed frontally to the surface being cut and oscillating relative to its mid-position with the frequency of an external alternating magnetic field across the cutting speed vector, with an amplitude equal to 0.8-0.9 of the width of the surface being cut. That being the case, the heating mode parameters should be set in a way that ensures softening of the structure of the work-piece surface layer down to a predetermined depth, which would make it possible to enhance the efficiency of the cutting process and to prolong the life of the tool.

The value of the cutting speed that equals the speed of movement of the plasma arc relative to the work-piece is set taking into account the value of the maximum heating temperature of the tolerance layer, without melting-off of the work-piece surface, to ensure structural transformations within a predetermined depth

$$V_p = \left(\frac{I \times U \times \eta}{\theta_{\max} \times b_n \times a_n \times \lambda} \right)^2 \times \frac{\omega \times b_n}{\pi},$$

where V_p - tool-cutting speed, m/min; I, U - current intensity and plasma arc voltage; η - efficiency factor of the plasma arc; $\theta_{\max}$ - maximum temperature plasma heating of the tolerance range cut-off material ($\theta_{\max} \leq \theta_{\text{melting}}$); b_s, a_s - width and length of the heating spot in the cutting zone, mm; λ - heat conductivity factor, W/cm·°C; ω - thermal diffusivity factor cm^2/s.

The above method is illustrated by the chart (Fig. 1) showing the location of the source of heat and the area being heated relative to the work-piece. The direct-action plasma generator 1 (PVR-401) was selected as the source of the plasma arc 3. The plasma generator is equipped with the magnetic deflection system 2 that consists of two cooled magnetic cores, whose ends are located near the nozzle of the plasma generator and arranged perpendicular to the cutting speed vector. The magnetic deflection system is connected to an adjustable AC power source. The work-piece is set on a lathe, on which the plasma generator is mounted. The plasma-forming gas is air. The decrease in the localization of heating by the plasma arc is achieved through superposing the external alternating magnetic field on the flow of the generated plasma. In order to create an external alternating magnetic field, the plasma generator is equipped with a special magnetic system. The plasma generator produces an electric arc that burns between the cathode of the former and the surface being cut. At the point of the contact, thermal output thus comes about through the reference straight line of contact between dimensions a_s and b_s.

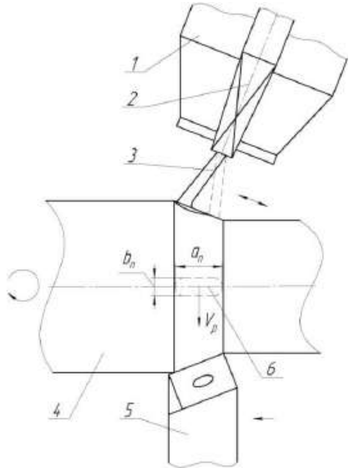

Fig. 1. Diagram of the plasma-assisted machining

As a result, an area of thermally softened metal emerges on the surface of the work-piece, which makes it possible to enhance the machining performance of the cutter while ensuring the stability of the parameters of plasma heating due to excluding melting-off of the work-piece surface, reducing the degree of the cutter overheating and, consequently, increasing the cutter's service life.

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IMPROVEMENT OF WELDED PIPE JOINTS FOR MINING EQUIPMENT

Mining enterprises use many steel pipes for pumping sludge from ore dressing waste. These pipes are often repaired by welding. This work requires a lot of time and high quality connections. In this regard, the research below have been performed.

The macrographic examinations enabled us to conclude that, the weld bead has a satisfactory penetration lacking defect. They confirm the visual examinations which reveals a regular cord and of beautiful aspect. It is noticed that the heat treatment does not change the macrographic structure except on the level of dimensioning of the grains. This obliges someone to see the size and the nature of the grains by micrographic examinations. The zone of connection is the seat of a thermal contribution in short and rather important conditioning the enlargement of the grains and the formation of an acicular structure, It is thought that the zone of overheating is the seat of structures which have the reduced plastic properties of the welding and weaken the structure slightly. The structure of our product after welding is primarily ferritic with some small islands or beaches of pearlite (percentage of weak carbon).

External master keys (4^{th} and 5^{th}) where the cooling speed is more important, give a needle structure characteristic which is a ferrite out of balance.

Internal master keys (having undergone treatments of reheating give a regular structure of ferrite, the coalescent ferrite needles end in structures closer to the state of balance or in mixed textures). The lower part of the joint thus corresponds to heated master keys several times than zones regenerated (master keys 1 and 2), have a ferritic structure with regular grains. The weld bead presents broad zones affected by the heat which can be treated as being ZAT1 and ZAT2. The ZAT1 close to the zone of connection, where the temperature is lower than 1100°C (zone of standardization), We can meet a fine structure which is due to the effect of a heating during short times and which did not reach the temperature of transformation AC3. In this zone metal acquired higher mechanical properties compared to those of the base metal which remained far from the action of heat. This phenomenon explains the increase in the hardness recorded in this zone. The ZAT2 having undergone a transformation leading to an incomplete recrystallization because the temperature remained constant between AC1 and AC3. At this temperature the most noticeable structure is the coarse ferrite grains accompanied by fine grains of ferrite and pearlite. In the zone close to the ZAT2, one observes a recrystallization which restores the form and initial dimensions of the grains deteriorated by the plastic deformation imposed by rolling. Beyond this limit the structure is identical to that of the base metal. This recrystallization results in a fall of the micro hardness. The analysis with the Charpy pendulum sheep highlights for product SG3 deposited by MIG welding may lead ductile character of the rupture at the ambient temperature and even at - 20 °C, and an unquestionable brittleness with the die of – 40°C. The analysis of facies reveals this brittleness within an overall ductile behavior because the level of the loads is close except for the temperature - 40 °C where a weak energy is observed.

It is Noticed that brittleness corresponds to ferrite and pearlite structure that can be found in various forms. The applied heat treatments affect little the structure of the base metal but the effect is spectacular on the joint, where it is noticed a considerable change inducing a total recrystallization which lead to a fall in microhardness on the level of the joint but lead to an increase in impact strength on the other hand. This Ci is justified by the return towards a ferritic structure because ferrite is fragile at low temperatures.

High tensile residual stresses, at or above the yield stress level, exist near the weld toe area, especially at the weld start/stop location. The magnitude of the residual stresses reduces quickly as the distance from the weld toe increases.

Post Weld Heat Treatment (PWHT) does not relax residual stresses completely from the socket-welded piping joints; the maximum tensile residual stress relaxes about 48%, where as the maximum compressive residual stress relaxes 50%. The reason that the residual stresses do not relax completely by PWHT is the different cooling rates at different locations of the welded joint, especially near the weld area. Different cooling rates regenerate residual stresses which are not much different from those originally introduced by welding.

The residual stress distribution does not change much when the slip-on gap in the socket weld joint is reduced to zero. Hence, the increase in fatigue life of socket welds with no slip-on gap is unrelated to residual stress. The improvement in fatigue life may come from the change in failure mode, which in turn, may be influenced by the change of the external load stress or strain distribution.

Fabrication of piping weld joint by *quarter circumferential welding* yields a more favorable residual stress distribution than the *full circumferential welding* pass. Less distortion and increase in fatigue life are observed in the earlier method of welding.

Moreover, in the quarter circumferential welding it does not matter much if the last welding pass is on the socket side or the pipe side. In case of the full circumferential welding, the location of the last pass does influence fatigue life, as demonstrated by Barsoum.

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TECHNOLOGICAL SCHEME OF A MULTISTAGE HYDROTRANSPORT FACILITY FOR TRANSPORTATION OF FOSSILS AND OTHER SOLID LOOSE MATERIALS BY PIPELINES AT FAR DISTANCES

The article reviews the technological scheme of a multistage hydrotransport facility for transportation of fossils and other solid loose materials by pipelines at far distances. To perform its function, it has a mechanism for obtaining hydraulic fluid which is installed inside the drum and consists of auger and quill shafts which are fixed to the axis between the intake pump wings. Inside the chamber for producing a

solid loose component there is a nozzle performed with an auger inside it, at the same time each stage (each station) is equipped by tanks located under the perforated drum for obtaining liquid and solid components which are respectively connected to the pump inlet pipe and the chamber for producing a solid component.

Multistage hydrotransport installation works as follows.

The high-pressure water pump from the inlet sump through the suction pipe draws in clean water and pumps it into the pipeline, at the end of which a nozzle is mounted for supplying a high-pressure jet to the chamber. There, a screw placed on a cylinder, which rotates around a pressure pipe on bearings, delivers solid bulk material from the feeder. From the latter, the hydraulic mixture enters the pipeline and into the pipe, where there is a complete loss of energy developed by the head unit.

The formation of slurry occurs in the mixing chamber, from which it enters the diffuser, in which the static pressure will be restored. A screw is fixed on this pipe, which between the cavities has nozzles for discharging the hydraulic mixture and rotates on bearings. The slurry enters a perforated drum rigidly connected to the housing.

Water from the body enters the tank, and from its upper part through the pipe to the high-pressure water pump located on the pumping unit. Wet solid material for further transportation is fed to a receiving hopper located on the first 1PU (first pumping unit). The rotation of the pipe on the control unit (pumping unit), as well as the screw on the control unit (head unit) occurs with gears and gears driven by electric wires. The space in which the gear train, gearbox and electric drive are located, against the accidental inlet of water or hydraulic mixtures protects the partition.

Thus, on the pumping unit, in the device for separating the slurry into solid and liquid components, the solid bulk material is separated from the liquid medium - water, and then the slurry is again formed with the same consistency that it had before the pumping unit into the pipeline.

The considered installation allows transporting hydraulic mixtures over long distances with the same - optimal consistency throughout the main line.

In addition, the possibility of the complete use of the energies developed by the HU and PU, as well as the use of high-pressure water pumps as energy sources, minimizes the number of stages of main hydrotransport systems.

An approximate preliminary calculation of the technical and economic effect achieved by the implementation of the considered technological scheme of a multi-stage hydrotransport installation for transportation of minerals and other solid bulk materials over a long distance through the pipeline has shown that, compared with the existing installations operating for the same purpose significant economic effect can be achieved.

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SIMULATION RESULTS OF THE HYDROLIAN ENERGY PRODUCTION SYSTEM (MADA)

Simulation of the hydrolian system with an asynchronous two (2) MW dual power generator shall be carried out under the following conditions :

The electricity grid in which the power produced by the hydrolian is injected is assumed to be three-phase balanced with infinite power;

The tidal velocity profile used is identical to that shown in Figure 1.

Simulation time is reduced to one day (24 hours).

No loss of energy (power) in converters.

Figures 1 and 2 show respectively the 24-hour speed profile used for our study and the corresponding Pt mechanical power curve of the turbine.Mechanical power shall take into account the variation in tidal speed;This is quite normal, since this power is a cubic speed function.

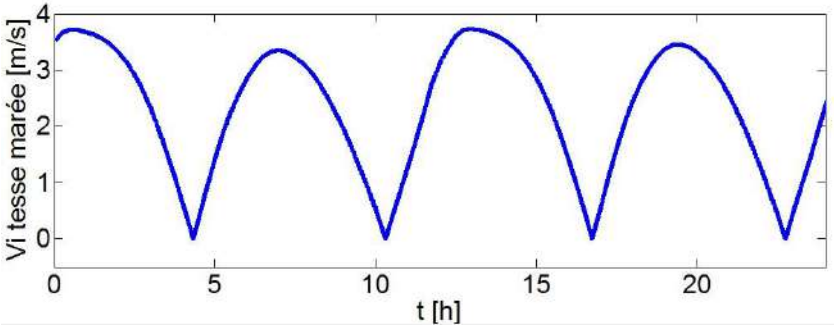

Figure 1. Tide Speed Profile

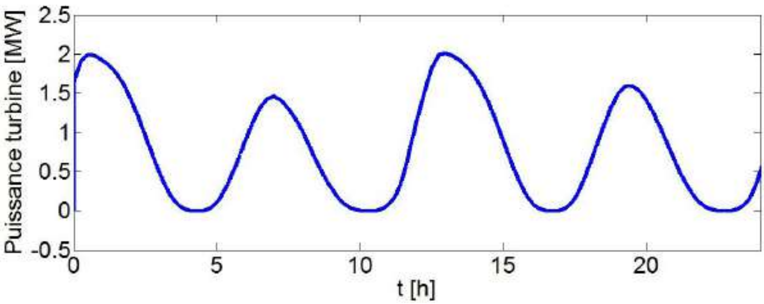

Figure 2. Mechanical power of the turbine

The result of the control of the Ω mechanical speed of the hydrolienne is shown in Figure 3. These lines show that the proposed control strategy is satisfactory, i.e. the controlled speed is the same as the reference estimated through the MPPT strategy.

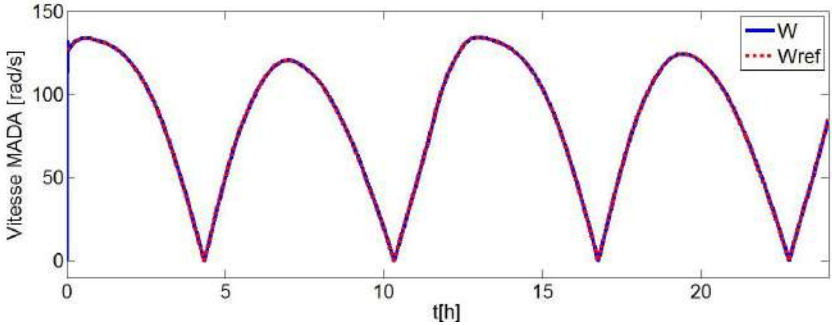

Figure 3. Generator speed control result where: W=Ω and Wref = Ωref

In order to take into account the intermittent production of hydrolian, the mechanical power of the turbine is taken as a reference to the active power supplied by the generator. The result of monitoring the active power produced in the MADA stator is shown in Figure 4.

The hot spare is set to: -400 kVAR between zero 0 and 6 hours;zero 0 kVAR between six 6 and 12 hours; then 400kVAR between 12 and 24 hours. It is positive for output, and negative for consumption. The reactive power check result is shown in Figure 5, where the reference power and the reference power are very consistent.

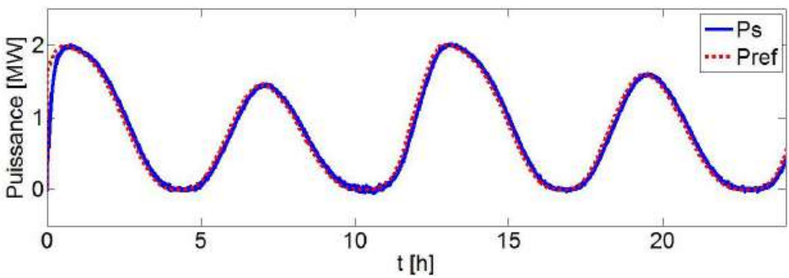

Figure 4. Control result of the active power of the generator

The result of monitoring the voltage of the continuous bus is shown in Figure 6, where the reference is fixed at 660V between zero(0) and 15 h and 580V respectively between 15 and 24 h.

The voltage of the continuous bus follows its instructions perfectly. Figure 7 shows the current in the continuous bus. As a result, the current in the bus smoothing capacitor will cancel as soon as the bus blends. The current straight at the entrance of the rectifier and the inverter are then identical.

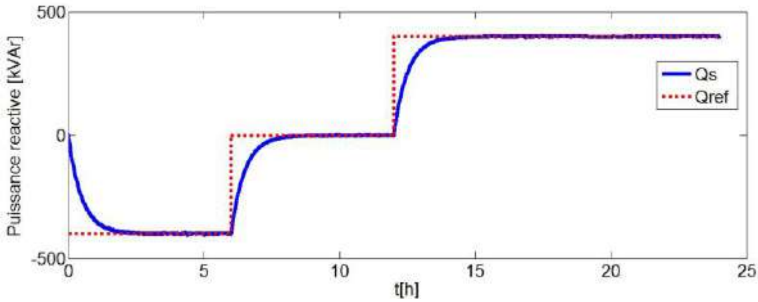

Figure 5. Reactive power control result of the generator

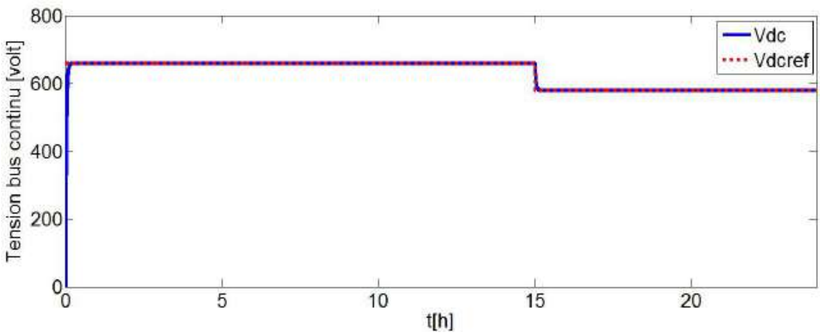

Figure 6. Continuous Bus Voltage Check Result

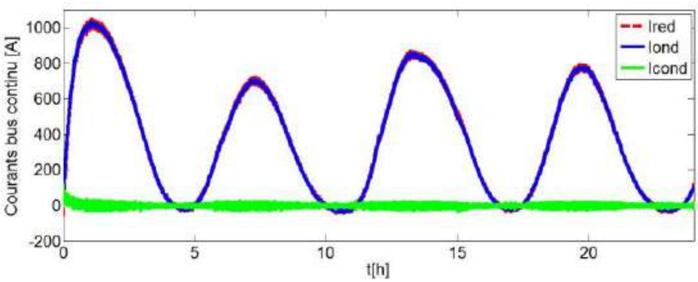

Figure 7. Continuous bus currents

Parameters used for system simulation will be given in the Annex.

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PROSPECTS OF GAS OIL PIPELINES RELIABILITY GROWTH BY PIPE STEELS IMPROVEMENT

Pipeline transportation is considered to be one of the most economical means of delivering liquid and gaseous products over long distances with minimal loss of product in the course of delivery to consumers. Modern pipelines are exceptionally long metal structures with a length exceeding thousands of kilometers.

It is known [1] that the destruction of gas pipelines is too dangerous. The area of environmental impact from the site of destruction ranges from several hundred meters to several kilometers. Particular danger during destruction is associated with the possibility of gas contamination of territories and settlements, the formation of an explosive mixture of gas and air, inflammation of transported products, their possible penetration into large bodies of water. It is known [1], for example, that only 1 tonne of spilled oil creates an 18 km^2 oil film on the surface of water bodies. In such cases, the complete restoration of the ecological balance requires the implementation of a whole complex of remediation works, which is associated with high material costs.

The working conditions of the pipe metal in a high-pressure gas pipelines are very specific, sharply different from the operating conditions of the metal in other metal structures, which is caused by the following factors [1-3].

Exploitation of metal pipes of the same pipeline due to its large length is carried out in dramatically different natural and climatic conditions - from sub-zero temperatures in the northern regions of the country to plus in the southern. The same conditions determine a wide range of types and mechanical characteristics of the soils in which the pipeline is laid; possibility of plastic deformation of the pipes when crossing various natural obstacles - water obstacles, swamps, mountains, lakes, etc [4-5].

In underground gas pipelines, the metal works at ground temperature. Fixing of the pipes with a diameter up to 1020 mm with soil is carried out on the area of several tens of meters long. As shown by the results of experimental measurements of elastic axial displacements and stresses taken during the cutting of an emergency underground gas pipeline with a diameter of 425 mm, fixing of the pipes with soil is carried out at a length of 25-50 m. Moving the ends of the pipes at the point of incision reached ≈ 29 mm, and longitudinal stresses a ≈ 200 MPa. In the pipelines with diameters of 1020 mm or more, fixing of the pipeline with soil is not always sufficient; the temperature regime and the magnitude of the longitudinal deformations are largely determined by the conditions of operation of the air cooling apparatus (ACA) and their number. In the case of ACA absence, the temperature of the pipeline may increase along its length, as the soil is no longer able to absorb the heat obtained by gas during compression. Therefore, securing absence of powerful gas pipelines in soils, preventing them from rising or bending in swampy and flooded places is a difficult task that cannot always be reliably solved, so the stability of the pipeline is not always secure.

Depending on climatic conditions, the metal of the pipes is operated in a wide temperature range - from 30 ... 40^{o} C in summer to -15 ... -20^{o} C in winter, and in the northern climatic zones in the areas of the above-ground routing the minimum operating temperature can be much lower. Construction and installation works on pipelines is in some cases are carried out only in winter at temperatures up to -40^{o} C.

During the amortization cycle (more than 30 years), the pipe metal works almost constantly in a two-axis stress state with different, depending on many factors, ratio of stresses in the circular and longitudinal directions. In addition, metal pipelines are subjected to low-cycle loads, which in some cases can cause stresses that reach a yield strength.

The influence of pipelines scheme stressed state on the plastic properties of the pipes metal is clearly traced by the change in a relative elongation. Thus, when on flat fivefold specimens with a uniaxial tensile elongation is 20-30%, then in the conditions of flat stress state in hydraulic tests before the destruction of full-sized pipes, the plastic elongation of the perimeter reaches only 3-7%, and in high-viscosity plastic pipes of controlled rolling steels - 8-12%.

In metal pipelines, as a rule, the inevitable presence of concentrators - burrs, scratches, oriented along the forming pipe.

Experimental studies have allowed to determine the change of circular deformations of the outer surface from the internal pressure in pipes with different ovalities (in section along the small axis of the oval). In particular, as the ovality decreases due to the increase in internal pressure, the value of deformations in the specified cross section increases the faster the greater the ovality in the initial state.

At a pressure of 1 MPa, local deformations can reach values corresponding to the yield stress, at a more or less low value of the average stresses in the pipe metal. The maximum value of local deformation is about 0.6% at a pressure of 5.5 MPa, then the increase in deformation ends, which corresponds to the moment of acceptance of the cylindrical shape pipe.

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STUDYING A PROCESS OF WET MATERIAL DRYING IN A VORTEX PLANT

Nowadays, vortex-type flow is finding ever-widening application in both national and foreign drying equipment. In terms of similar temperature modes, specific moisture content and capacity of vortex-type drying units is much higher than the ones of the drum or "boiling"-bed units. That is stipulated by the possibility to apply high-temperature heat carrier, developed surface of solid phase, and available high relative velocities between the latter and the drying agent reaching 50-150 m/s. In terms of "boiling" bed, they are only 0.2-0.5 m/s. The available vortex-type drying units differ in their structures, dimensions, and arrangement variety of certain components [1].

To carry out experimental research of the drying process involving vortex-type drying plant, a combustion chamber of aviation jet engine was used.

Technological process of the initial material drying is as follows: using the auger and chain disintegrator, dewatered concentrate is supplied into a distillation chamber where it is caught by a high-temperature flow of heat carrier and put into the lower part of the chamber.

Material is subject to intense drying within the formed vortex-type flow. While rising to the upper part of the apparatus, the flow comes into the precipitation chamber, equipped with the immobile guiding device, where main mass of solid particles is separated from the gaseous medium. Dry material is sent to a weighing device through the discharge sleeve; combustion gases are emitted into the atmosphere after their cleaning in wet scrubbers.

Air, required for fuel combustion, comes to the heating chamber through a spade-shaped vortex generator and hole in the fire tube. Another share of the air passing along the circular duct between the case and fire tube is mixed with the fuel combustion products within the chamber end section. Devices aimed at fuel parting are dismounted to enlarge the passage section of a combustion chamber for proper operation of combustion chamber on natural gas.

Having good aerodynamic properties, such combustion equipment provides high temperatures of a heat carrier in the center of gas flow while the temperature right in the near-wall area of the distillation chamber remains rather low (up to 600 °C) forming favourable conditions for wet material drying. Combustion chamber operates from the heat volume of the combustion space several times more than in other combustion spaces of similar purpose.

Analysis of cost-performance ratio of drying unit operation makes to possible to draw conclusion on the expediency of using vortex-type units to dry loose wet material. Comparing to “drying”-bed and drum drying units, the unit under consideration is much more economic along with less specific amount of metal.

According to the experimental studies, in terms of vortex flow, heat transfer coefficient is 40-70 times higher than in case of direct flow; moreover, mass transfer increases by 2.5 times and more.

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STUDYING THE LOAD JAM MODES WITHIN THE FRAMEWORK OF A FLAT MODEL OF THE ROTOR WITH AN AUTO-BALANCER

Passive auto-balancers, ball (roller), pendulum, etc., are used for balancing high-rotational rotors at operation. The same devices with a single or more loads can be used in vibration machines in order to excite vibrations [1].

The use of auto-balancers for different purposes is possible due to that a rotor machine with loads in the form of balls (rollers), pendulums can execute various steady modes of motion that correspond to:

- auto-balancing or synchronous rotation of loads together with the rotor (stationary movements);
- load jams (caused by the Sommerfeld effect) [2];
- parametric and other oscillations of loads.

Theoretically, the rotor machine auto-balancer system can execute various steady movements. Among all possible steady movements, over time, the system would execute only steady movements. Therefore, when constructing an analytical theory for such machines, one searches for all possible steady movements of the system and investigates their stability.

The most complete information on the origin, disappearance, conditions of existence and stability of different motion modes of the system is provided by the bifurcation theory. Typically, the speed of rotor rotation is accepted as a bifurcation parameter. In this case, at certain speeds of rotor rotation an auto-balancer can balance the rotor, and at certain – to excite vibrations. To build a bifurcation theory, it is necessary to find and investigate all possible steady movements of the system; this is, however, a complicated mathematical problem.

Today, most analytical results were obtained in the framework of a flat model of the rotor on isotropic elastic-viscous supports carrying an auto-balancer with identical loads. However, there remain the insufficiently studied modes of load jams caused by the Sommerfeld effect. Only those jam modes have been found and investigated at which loads are combined.

It is a relevant task to analytically find, within the specified model of the rotor and auto-balancer, all possible modes of load jams. This is important for building analytical theories both for passive auto-balancing and resonance vibration machines in which an auto-balancer is used as a vibration exciter. Our theoretical study shows that the load jam modes in the rotor auto-balancer system are the single-parametric families of steady movements. Each jam mode is characterized by a certain load configuration and the appropriate frequency of jamming. In the coordinate system that rotates synchronously with loads:

- the rotor displacement is constant;
- the parameter is the angle defining the direction of the rotor displacement vector;
– loads take certain fixed positions relative to the rotor displacement vector and these positions depend on rotation speed of the rotor.

The auto-balancer with n_b+2 of the same loads have n_b+1 different load configurations. The total number of different modes of load jams is:

- 2(n_b+1), if n_b is odd;
- 2n_b+1, if n_b is even.

The total number of different jamming frequencies is:

- 3(n_b+1)/2, if n_b is odd;
- 3n_b/2+1, if n_b is even.

The total number of different characteristic speeds is n_b+2. Characteristic speeds are the points of movement bifurcations, because their transitions give rise to the emergence or disappearance of single-parametric families of movements that correspond to a certain jam mode. At these points, the jam modes may acquire or lose stability.

It should be noted that the results were obtained for the cases of small forces of viscous resistance in the system or at low mass of loads in comparison with the system mass. However, this assumption is relevant for practice. In addition, the assumption was accepted at the stage of finding the expansion of characteristic speeds and jam velocities into series based on the small parameter power.

The solved problem can act as a model problem, particularly in order to estimate:

- efficiency of approximated methods for studying the dynamics and stability of movements of mechanical systems;
- the completeness of solving the problems on studying jam modes within other models of rotor machines with auto-balancers, the unbalanced vibration exciters, etc.

The results obtained make it possible both to reduce the regions of the existence of jam modes and to increase them. This could be used in the design of auto-balancers for balancing rotors or vibration exciters in the form of auto-balancers.

Among all theoretically possible jam modes, only stable movements would be executed in practice. Therefore, in the future it is planned to investigate the stability of the established jam modes and to conduct computational experiments. Note that the study can be carried out using the fixed-motion stability theory for nonlinear autonomous systems. At the same time, it is possible to analytically find the "exact" boundaries of movement stability regions in the parameter space.

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GEARBOX BODIES MADE OF POLYMER CONCRETE FOR MINING AND METALLURGICAL COMPLEX

The equipment of the mining and metallurgical complex operate in difficult conditions: high humidity, gas contamination, a sharp change in temperature, dustiness, contact with aggressive substances, and a number of others. This requires manufacture of equipment from expensive materials, increased thickness of the walls of the bodies or the use of protective covers and the complexity of their manufacture and operation. All these factors increase the cost of manufacturing and operating equipment. Therefore, an urgent problem is to search for new materials that could minimize the disadvantages noted above.

The analysis showed that among the new structural materials that can be used in mining engineering, polymer concrete is effective. It consists of: aggregate (large and small), filler, binder, surface-active substance (SAS) and hardener. The material is characterized by a number of positive characteristics: high strength, low weight, high impact resistance, elasticity and ductility, calm attitude to chemicals, lack of reaction to humidity and temperature, quick solidification of the mixture, dense and smooth surface of the product [1].

The authors conducted research on the use of polymer concrete for the manufacture of hull parts of mining equipment gearboxes.

The studies found that it is most advisable to use an aggregate with intermittent granular, consisting of three fractions: large, medium and small. In this case, the void in the large fraction is filled with granules of the middle fraction, and the void in the middle fraction is filled with small particles. In this case, a high density mixture is formed, which ensures high strength of the polymer concrete product.

Based on the results of theoretical and experimental studies, the rational composition of the polymer concrete mixture was determined, which, according to its strength and rheological characteristics, meets the requirements for the material for the manufacture of gear cases. This mixture is as follows: rubble -51%, quartz sand -25.5%, quartz flour-11%, ED resin - 10.5%, hardener-2% (mass ratios). Such a composition of polymer concrete after hardening makes it possible to obtain compressive strength σ_c= 170 ... 180 MPa, bending σ_b=80-85 MPa.

One of the main factors ensuring the constancy of the characteristics of polymer concrete is the observance of the technology of preparation of materials and the components mixing regime.

The rotation frequency of the working body of the mixer provides an efficient distribution of the components of the polymer concrete mixture with their uniform distribution throughout the whole volume. This allows you to get polymer concrete with guaranteed characteristics. Conducted experimental studies found that the rotation frequency of the working body of the mixer should be 600-700 rpm.

One of the most important factors affecting the quality of polymer concrete mix is the mixing time. Given the difference in the characteristics of the starting materials, it should be sufficient for uniform distribution of all components in the volume of the mixture. This ensures its homogeneity, which in turn enables its rational placement in forms without the formation of internal voids (shells), which reduce the strength of the hardened polymer concrete. The optimal time for mixing the components of the polymer concrete mixture, as shown by studies is a time of 3-4 minutes.

For the best mode of resin impregnation with EDN of 20 components of the polymer concrete mixture, a study was conducted of the effect of the temperature of the resin on the final result - the strength of the hardened polymer concrete. It was established that the heating of the resin to 50-60 ℃ is optimal, which gives a strength increase of 12-15%.

The conducted studies have confirmed the possibility of obtaining a new composite material - polymer concrete with high technical characteristics. Its density is 2200 ... 2300 kg / m^3 compared with the density of cast iron or steel 7500 ... 8000 kg / m^3. The study of the influence of these factors on the strength characteristics of polymer concrete made it possible to develop a technology for preparing the mixture and a technology for casting the upper and lower covers of the Ts2-250 gearbox from it. It has a number of advantages such as 2.6 times lighter than the same body of steel or cast iron, has a higher strength, does not require additional machining.

Studies have shown that the achieved strength and rheological characteristics of polymer concrete are guaranteed by the optimal structure of the mixture and the rational technology of mixture formation and molding of the product from polymer concrete. The advantages of the proposed technology are its simplicity, accessibility, the ability to obtain products of high degree of readiness without the need for mechanical processing, profitability.

At the same time, it should be noted that the composition of polymer concrete is not universal, which requires additional research when changing the characteristics of the components or the requirements for the product from polymer concrete.

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SUBSTANTIATION OF RATIONAL PROFILE OF ROLLS FOR MINING ROCKS DISINTEGRATION

The crushers of quasi-static action, such as jaw, cone and roll ones, are the main types of equipment for disintegration of mining rocks. The pieces are destructed during grab between two approaching working surfaces [1, 2].

The analysis of roll crushers operation has shown, that the teethed profile of rolls increases the grab angle of crusher, the size of pulled-in pieces and facilitates the pieces disintegration process compared to the case of smooth rolls. But, the main problem here is the high rate of teeth wear, which is the higher the sharper the teeth are [3].

So, choice of the rolls profile, balancing the process energy efficiency and the working surfaces wear, is an actual problem.

It is recommended to replace both the purely compressive loading, being characteristic for smooth rolls, and the splitting action of separate teeth with the following intermediate variant. The rolls profile (along the axis) has periodically alternating conic sites with equal taper angle and serial change of inclination direction, having rounded ledges and hollows [4]. The neighboring profiles form constant gap in the plane of rolls axes, guaranteeing minimal dispersion on the crushing product size.

The modelling of conditions for the destruction of cubical granite [5] of size from 5 to 20 mm, having less than 6 percent of lamellar pieces, was conducted. The cases of modelled pieces in the form of cuboid having rounded edges [6] getting in the gap between the ledge and the hollow of neighboring rolls have analyzed. The variants of pieces destruction by shear or bend loadings are considered. Here, the criterion of pieces destruction in a complex stressed state was used together with analytical interrelation between stresses, corresponding to the compressive, shear and tensile destruction [7]. It is set, that reaching of the ultimate strength of a piece was fixed at the calculated destruction stresses being 1.9 times less than in the case of the crusher with smooth rolls. At the same time, shear stresses were responsible for the destruction of not more than 2 percent of pieces by weight. The rest of pieces were destructed by bending stresses. So, the destruction caused by bending loading is most characteristic even for cubical granite. The efficient decrease of ultimate destruction stresses was much more sufficient for lamellar pieces included in material and made 6 times in average. It confirms sufficient energy conservation compared to the case of smooth rolls application.

The disintegrators with wave profile of rolls can be used for processing of lumpy mining rock mass with sufficient decrease of the energy consumption for crushing and enhancing of the cubical pieces yield. Also, the mode of pieces shape correction for material, having essential part of lamellar and needle-shaped pieces, can be realized.

The research results allow to make a reasonable choice of the rolls geometrical parameters while designing of disintegrator operational part.

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INCREASE IN MAXIMUM LIFTING DEPTH FOR A DOUBLE-DRUM MINE HOIST

The problem of increasing the rope capacity of existing cylindrical drum hoists is considered. To reduce capital costs while increasing the depth of the mine hoist, the use of a conical insert on the drum is proposed.

The maximum lifting depth H is determined by the rope capacity of the rope winder. If it is necessary to increase the lifting depth (within 100-180 m) without significant costs for the reconstruction of the mine hoist (replacing the lifting machine, drive, foundation, etc.), the conical inserts on the drum can be used. The increase in the drum capacity can be reached due to the increase of the winding radius on the conical inserts and reducing the cutting step on the profiled surface of drum [1].

As a basic option, we take a double-drum skip cylindrical mine hoist with the following parameters: drum diameter D=4 m; drum width B=2,3 m; rope winding pitch on the drum t=46 mm; the mass of the lifted mineral (coal) Q=8460 kg; skip with a sector shutter with a capacity of V_{sk}=9,4 m^3 and weight with a tow hitch Q_m=5430 kg; double-layered steel rope with a diameter of d = 42 mm, linear weight q=65,35 N/m, total tensile strength of the wire rope Q_z=1240 kN; maximum lifting speed $v_{\max}$=5,37 m/s; drive motor with a power of N=550 kW, with a rotation speed of n_{dr}=295 rpm, a multiplicity of starting torque λ=2,1; gear ratio of speed reducer $i_{\text{ред}}$=11,5.

Let us estimate the magnitude of the increase in the lifting depth H during the reconstruction of the basic hoist by using conical inserts on the drum.

The maximum lifting depth of a cylindrical drum without conical inserts is 553 m. The safety factor of the rope is 7,058.

The rope capacity of the drum when re-equipping it with the conical inserts increases by ΔH=177 m with a taper angle β=18°. In this case, it is not necessary to change the type of rope, since the safety factor m=6.75>6.5 corresponds to the norms of the Safety Rules.

For a modernized machine with conical inserts, verification showed the validity of using the engine of the original machine.

It would be possible to increase the maximum lifting depth by ΔH by increasing the diameter of the cylindrical drum, which is simpler constructively. But at the same time, it will be necessary to change the drive motor, since the existing one will not satisfy the overload capacity.

Thus, the use of a conical insert increases the rope capacity of the drum, while the existing engine satisfies in terms of power and overload capacity.

Conclusion. If necessary to reach the increase of maximum lifting depth within 100-180 m on existing hoist with cylindrical drums, it is most efficient to modernize its drum using conical inserts. The increase in rope capacity of the drum, in this case, is due to an increase in the radius of winding on the conical insert and the possibility of reducing the cutting step. At the same time, the power of the drive motor increases slightly and there is no need to change it.

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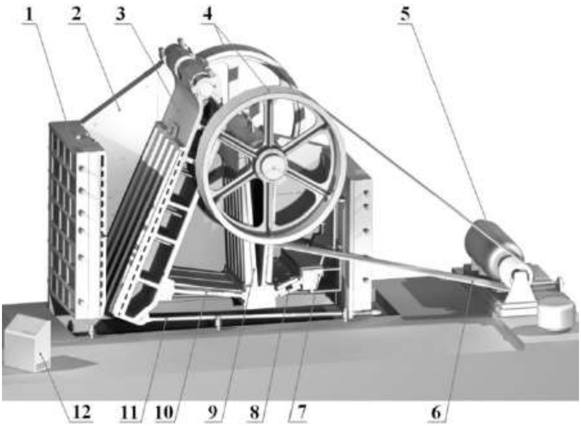
DEVELOPMENT OF THE ALGORITHM FOR OPTIMIZING THE BODY OF THE FIXED JAW

Designing mining machines provides for the solution of the main task of choosing the design parameters capable of ensuring the operability working capacity of these devices with regard to various requirements. The scientists of the Engineering and Design in Machinery Industry specialize in finite element analysis of thin-walled reinforced structures; their knowledge has already helped production workers to solve the number of practical problems. The FEM method allows not only to analyze the stress-strain state of structures, but also to solve the problems of parametrical and structural optimization. Let us demonstrate this by the example of a parametrical optimization of the jaw crusher with a simple movement of the jaw, the design of which is shown in picture. Here it is indicated: 1 - fixed jaw; 2 - bed; 3 - movable jaw; 4 - flywheel and drive pulley; 5 - motor; 6 - V-belt drive; 7 – adjusting device; 8 - rear toggle in the assembled state (safe one); 9 - connecting rod in the assembled state; 10 - front toggle; 11 - locking rod; 12 - control panel. In the design development, an important issue is the optimization of the mass of the following assemblies: the fixed jaw, the movable jaw, taking into account the geometric parameters that determine their shape and dimensions under the set strength conditions. Taking into consideration the fact that the designs of the movable and fixed jaws are similar, let us take the design of the fixed jaw body as an object of the analysis.

Let us develop an algorithm for the parametrical optimization of the design of the crusher fixed jaw. Let us find the values of the geometrical parameters of the jaw, at which the minimum value of the body mass is reached. The geometric shape of the body is determined taking into account the parameters controlled in the process of optimization, their values are accepted on the basis on design considerations, in particular, these are the height H and the length L of the body, width l_c and height δ_c of the ledges for connecting the jaw with the base. The parameter controlled during the optimization process can be the jaw thickness δ. In addition, the functional limitations associated with such strength condition on equivalent stress as $\sigma_{eq} \leq [\sigma]$ must be taken into account.



Picture. Appearance of the jaw crusher with the simple movement of the jaw

The geometric shape of the body is determined taking into account the parameters controlled in the process of optimization, their values are accepted on the basis on design considerations, in particular, these are the height H and the length L of the body, width l_c and height δ_c of the ledges for connecting the jaw with the base. The parameter controlled during the optimization process can be the jaw thickness δ. In addition, the functional limitations associated with such strength condition on equivalent stress as $\sigma_{eq} \leq [\sigma]$ must be taken into account.

The preliminary analysis of the results of solving the problem showed that the safety margin of the structure is too high and the body can be significantly lightened.

Let us set the task: for the crusher ShhDP 9×12 with the given values of H, L, l_c, δ_c and $\sigma_{eq} \leq [\sigma] = 165$ MPa, we will find the value of the parameter δ at which the body mass will be minimal.

The SOLIDWORKS Simulation application software package provides an option to optimize the design, taking into account the criterion of the minimum of its mass while maintaining the specified margin of safety. Let the permissible stresses be 165 MPa, and the safety margin is 1,5. We will find for this condition the minimum value of the parameter δ [1].

The analysis of the results of solving the optimization problem shows that the automatic search for the optimal value of the parameter δ gives the result that contradicts the physical meaning, i.e., if the case is thicker than it is provided by the test problem stipulated in the condition (δ=750 mm), then the stresses exceed the allowed ones. This is due to the peculiarity of the maximum stress search algorithm, which explores all parts of the structure. Thus, the need arises in determining the parameter δ to analyze the picture of the stress-strain state in the body of the crusher's jaw.

In accordance with the Hot Spot Stress (HSS) method, to eliminate the singularity in the analysis, stresses are determined at a distance of 0.5 t and 1.5 t from the place of their concentration, where t is the thickness of the rib. The nominal voltage at the sites of concentration is determined by the method of interpolation. So, applying the HSS technique, we will solve the problem of parametric optimization of the body of the fixed jaw. To do this, we will carry out a computational experiment, which consists in the sequential calculation of the stress-strain state of the body and its analysis when the parameter δ changes in the range of 750-350 mm with the pitch of 20 mm.

Using the method of approximation, we will find a mathematical model describing the dependence of the maximum equivalent stresses σ on the thickness of the body: $\delta(\sigma) = 2632.6 \cdot \sigma^{-0.375}$.

Conclusions. For the first time, the algorithm for parametric optimization of a structure has been proposed, consisting of a sequence of computational experiments with subsequent analysis of the obtained calculation results based on the HSS method.

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EXPERIENCE OF PNEUMO-PULSE CLEANING AUTOMATIC CONTROL SYSTEM DEVELOPMENT FOR VACUUM DEVICES

Operation of pneumatic equipment involves the use of atmospheric air, which has harmful impurities, as a technological environment. As a result, contaminants passing through gas turbine and compressor units cause abrasive wear of their mechanical parts. Typically, partial protection of the piston pairs and blades of the turbines is achieved by installing filters which efficiency depends from their periodic cleaning.

There are a few ways to restore filters. One of the promising automatic methods of cleaning is the impulse one, which allows to perform cleaning in the current mode. Its implementation will simplify the cleaning procedure, increase the time between mandatory maintenance of the system, and reduce the cost of it.

The modern trend is to equip industrial enterprises with various automation systems that are built on the basis of digital software. They include sensors for processing data, process visualizations, and automated system management simplifications. That is why the practical acquaintance with the popular samples of such equipment and their software is an important issue for both the trained professionals and the current employees.

Data collection and processing is a kind of bridge between the "real world" of physical quantities and a computer. Sensorial information (sound, light, temperature, etc.) is converted by sensors into real-time signals and can be processed by special hardware. By the reason that direct transmission of information from input sources to a computer is not possible, automatic systems use drivers or an OPC-server with specialized software to visualize the data, received on the monitor, while performing a technological task.

During the internship at the Department of Environmental and Energy Engineering at the Austrian Montana University (Leoben)[1], the author developed the Eurotherm 2500 device and iTools and Lookout software[2]. On this basis, practical work was done which modelled impulse method of filter cleaning.

Laboratory work includes two magnetic valves (normally closed), one pressure sensor, and an air source at 1 bar. Data processing is carried out by the Eurotherm 2500 module.

The principle of operation of the automatic system provides for a predetermined period of activation of the inlet magnetic valve. As a result, the pressure in the pneumatic system increases. Then, after a certain time, the inlet and outlet valves are inverted and the system pressure drops abruptly. The control of the process, that takes place in the installation, is carried out by the controller. In particular, there is the discharge and etching times of the pressure and the state of the valves (which of them are open / closed at a specific time). The progress of the system operation is saved in a file every two seconds with simultaneous real-time display of the results on the graphical monitor.

As a result of the practical work, it has been created a system which allows to manage process and store the results of slow processes, useful for some specific tasks. The disadvantage of the system is that the pressure regulation is affected by the magnetic valves not by the electric drive of the compressor, which is the source of pressure. In this way, the compressor pumps the pressure continuously, and therefore, consumes electricity continuously. The more successful solution for such a task could be to create a control system for the electric drive of the compressor, which will provide the same parameters with the use of the same hardware and software, but with less energy losses.

Conclusions

The necessity constantly clean the air filters of compressor and gas turbine installations has been proved.

The impulse method of filters current cleaning is proposed which one reduces costs and increases the time between their capital maintenance.

Experience in developing and creating of an automatic impulse filter cleaning system with real-time visualization of control results has been received.

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DEVELOPMENT OF COMPLIANCE FAILURE MODEL TECHNICAL OBJECT OF MINING EQUIPMENT

Sophisticated technical objects in modern society are extremely important. This is primarily about various special-purpose complexes, which include mining machines. Such objects belong to the class of restored objects of long repeated use. They are usually expensive and require significant costs for their operation. To ensure the required level of reliability during their operation, maintenance is usually carried out, the essence of which is the timely preventive replacement of elements in a precautionary state.

A characteristic feature of complex technical objects of special purpose is the presence in their composition of a large number (tens, hundreds of thousands) of diverse types components that have a different level of reliability, different patterns of their wear and aging.

This feature requires a more subtle approach to the organization and planning of maintenance during their operation.

The problem is that when developing such facilities, all issues related to maintainability and maintenance should be addressed already in the early stages of designing the facility.

The model under development is intended to obtain the probability functions of failure-free operation (or the distribution function of MTBF) for the object as a whole and all its structural elements from the available information about the failure-free indicators of component parts.

The model does not take into account the possibility of multiple failures, since within the framework of tasks for which this model is developed, the probability of multiple failures can be neglected.

From the above it is clear that the initial information for the model should be the probability functions of the failure-free operation products of zero rank (PZR). For all structural elements of higher levels, including the object as a whole, the functions must be calculated.

In practice, functions are rarely exactly known. In the best case, the first two points are known and there are certain assumptions about the class of distribution laws to which the function possibly belongs. As a rule, only the estimate of first moment is known (mathematical expectation of the time between failures). In the worst case, neither the distribution function nor its moments are known. Therefore, in practice, it is necessary to make an assumption about the form of the distribution law taking into account the type of this element and available information about the physical laws of failure occurrence for elements of this type. The average operating time to failure of the elements must be estimated from the information about the analogous elements.

The model under development is intended to solve the problems of assessing the reliability of aging objects, therefore, we need to use the laws of distribution operating time to failure, taking into account the degradation processes in the materials of different types elements. Currently, it has become generally accepted that gradual failures occur due to the fact that value of some determining parameter reaches the maximum permissible value.

To test and study the developed models and methods, test objects with different structures and reliability were used. The characteristics of test objects are selected in such a way as to cover all typical cases of possible real objects encountered in practice. Using the test objects in following sections, we demonstrate the features of application developed models and their capabilities.

Thus, we can draw the following conclusions:

1. The reliability model (RM) allows you to get estimates of the reliability indicators (RI) of individual structural elements and the object as a whole based on information on the RI of elements lower structural level. In RM, a hierarchical structural structure of an object is presented. The structural elements of a certain u-th structural level are a sequential (in the sense of reliability) connection elements of the (u + 1)-th level included in it. Individual structural elements can be a redundant group (parallel connection) of the same type elements. Thus, using RM, representation of the hierarchical structural structure is combined with an arbitrary series-parallel reliability structure of the object, which is an acceptable representation for most technical objects encountered in practice.

2. The DN-distribution is used as a model of failures for all elements and the object as a whole. DN-distribution is considered an adequate model of gradual failures both for electronic products and for various mechanical components and elements. An important advantage of the DN-distribution is also that its appearance is preserved during the transformations of the reliability structure of the system. It is this feature of the DN-distribution that made it possible to apply it to a system with a hierarchical structure.

3. The software implementation of RM is developed in the Delphi programming system. The hierarchical constructive structure of an object is programmatically represented using list data structures (TList lists are used). List elements are objects (instances of Delphi classes) that represent individual structural elements of a technical object. Such objects encapsulate all the necessary data related to individual structural elements, including the parameters of DN-distributions of the mean time between failures.

Information about the composition, structure and reliability of the elements objects is stored in database of the model built using tables of the InterBase database format.

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BOOK OF ABSTRACTS

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