MINING SCIENCE and TECHNOLOGY

№1

MISIS

NATIONAL UNIVERSITY OF SCIENCE AND TECHNOLOGY

RUSSIA, MOSCOW 2016

CONTENTS

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PREPARATION OF PYRITE-ARSENOPYRITE CONCENTRATE FOR GOLD LEACHING THROUGH THE USE OF POLYREAGENT COMPLEXES

The article focuses on the causes of refractoriness of sulfide ore stock with superdispersed gold inclusions. In order to increase gold recovery, a hypothesis about effective (peroxide) preparation of refractory raw materials in the process of leaching has been described. The author focuses on theoretical justification of the physical-chemical model and the two-stage mechanism of oxidation (photoelectrochemical and bacterial) of material using microorganisms Acidithiobacillus ferrooxidans and Acidithiobacillus thiooxidans. A scheme of two-staged (physicochemical and bacterial) oxidation was developed. It is based on photoelectrochemical effects for the leaching of gold from refractory materials. It is shown that photoelectrochemical pre-oxidation makes it possible to increase development of the mineral matrix with its further bacterial oxidation. It also helps to increase the efficiency of invisible gold extraction from sulfide ores. Laboratory tests have been performed on the combined oxidation scheme on electroactivation carbon-in-leach of invisible gold from the Kokpatas deposit's sulfide ores. The authors present the results of the laboratory tests of the activation methods for impacting upon the refractory ores before metal leaching, which prove the efficiency of such non-traditional methods for developing mineral matrix. Gold extraction using the combined technology of sulfide and arsenopyrite oxidation increases by 18 %.

Keywords:refractory ores, pyrite, arsenopyrite, invisible gold, physical and chemical oxidation and bacterial oxidation, polyreagent schemes, reactive oxygen, electrochemical and photochemical processes.

The widespread natural occurrence of gold ores with invisible valuable components conditions the need for their large-scale involvement in ore processing. One of the ways to enhance the efficiency of refractory ore gold leaching is to intensify the process of refractory sulfide matrix destruction. In this regard, a new scientific approach to the selection of effective recovery methods for refractory gold minerals is required [1–10].

The research subject is poor bulk pyrite-arsenopyrite float concentrate with a gold content of 25 g/t, obtained from refractory sulfide ore from Kokpatas deposit ($\beta^{Au} = 2.9$ g/t, $S_{tot} = 5.8$ %, $S_{sul} = 5.6$ %, $S_{oxid} = 0.13$ %, $Fe_{tot} = 7.4$ %, $Fe_{sul} = 4.2$ %, $As_{tot} = 0.52$ %, $C_{tot} = 1.2$ %, $C_{tot} = 0.44$ %).

The research objective is to substantiate and develop combined methods for the processing of refractory gold-bearing raw materials through targeted photoelectrochemical effects using strong oxidizers.

The research task is to conduct experimental investigations of the influence of two-stage oxidation on mineral matrix development in the course of refractory mineral processing.

Research techniques: multifactorial experiment planning techniques, granulometric, mineralogical, spectral, chemical (including phase), X-ray phase, optical and electron microscopic, microscopic, bacterioscopic, atomic absorption, assay, X-ray diffraction analysis and other analysis techniques, plus aggregate laboratory tests on minerals. Experimental data were subjected to mathematical treatment.

Intensification of the refractory mineral oxidation process and enhancement of the efficiency of subsequent gold recovery in the course of sorption cyanidation is achieved through two-stage oxidation: physicochemical (through targeted photoelectrochemical effects) and bio-oxidation [11, 12].

The essence of the process is that prior to bacterial oxidation, sulfide materials are subjected to a preliminary oxidizing (peroxide) preparation by treating the mineral matter with reagents, containing reactive oxygen, which were obtained through a photochemical and electrochemical synthesis from primary gases, chemical compounds and water.

As is known, the Eh difference between the environment and electrical potential of sulfide minerals during oxidation shall lie within 100 to 200 mV; if they are equal, no oxidation will take place, while a lower difference value testifies to low oxidation efficiency, so

$$\mathrm{Eh}_{\substack{\text{react. medium}\\ \text{(oxidizer gas}\\ \text{mixture)}}} - \mathrm{Eh}_{\text{sulph. matrix}} = 100 \text{ to } 200\ \text{mV}. \quad (1)$$

The predominant propagation of Acidithiobacillus ferrooxidans and Acidithiobacillus thiooxidans is limited by a rather narrow interval of Eh (400 to 750 mV). Bacteria do not develop in the reducing environment and die after long-term exposure to such conditions. Inthis case, consideration should be given to the fact that at Eh 400 mV leaching of iron take place, and at Eh 750 mV that of sulfur. The required Eh values:

$$\mathrm{Eh}^{\text{opt}}_{\substack{\text{oxidizer gas}\\ \text{mixture}}} = \mathrm{Eh}_{\text{sulph. matrix}} + 200\ \text{mV} =$$

$= 1400 + 200 = 1600\ \text{mV}$ – for peroxide preparation, where $\mathrm{Eh}_{\text{sulph.matrix}} = 1400\ \text{mV}$ is sulfide matrix Eh.

After gas phase, UV irradiation is complete, decomposition of reactive oxygen and, primarily, O_3 takes place. Upon ozone introduction into the pulp, its greatest part degrades into O_2 and O^*. Reactive atomic oxygen, in its turn, transforms into singlet oxygen with a redox potential being 3-4 times lower than that of ozone. Other reactive forms of oxygen are more stable. As the ozone concentration experiences a 3 to 4-fold decrease, Eh decreases in direct proportion.

The two-stage oxidation duration (T_{treat}) is composed of the time of photoelectrochemical oxidation (t_1), bio-oxidation (t_2), and the reaction period of polyreagent complexes of reactive oxygen (chlorine-containing compounds) with the pulp mineral phase (Δt). The duration of photoelectrochemical oxidation depends on the constituent phases. The first is the phase of photoelectrochemical effects determined by the dissolution rate of reactive oxygen in the acid solution for obtaining the required reactive oxygen content: $t_1^{\mathrm{I}} = \dfrac{C^{\text{final}}_{O_2^*} - C^0_{O_2^*}}{v_1} = \dfrac{15-10}{2} = 2.5\ \text{h}$, where the initial reactive oxygen concentration $C^0_{O_2^*}$ = 10 mg/l, the final reactive oxygen concentration $C^{\text{final}}_{O_2^*}$ = 15 mg/l, the oxygen dissolution rate in the solution v_1 = 2 mg/(l·h).

The second phase of photoelectrochemical oxidation is determined by the reaction rate of hydrogen (low-soluble) and reactive oxygen with the production (synthesis) of reactive ion-radical complexes and the rate of their interaction with the mineral matrix:

$$t_1^{\mathrm{II}} = \frac{C^{\text{final}}_{H_2O_2} - C^0_{H_2O_2}}{v_2} = \frac{20-2}{1.8} = 10\ \text{h},$$

where the initial concentration $C^0_{H_2O_2}$ = 2 mg/l; the final concentration $C^{\text{final}}_{H_2O_2}$ = 20 mg/l, H_2O_2 formation rate during hydrogen and ozone interaction v_2 = 1.8 mg/l·h, Δt = 2 h.

Thus, the duration of photoelectrochemical oxidation is 14.5 h (the parameter values were assumed according to the experimental data obtained on the Yellow Jacket deposit ores [17]).

The duration of the bio-oxidation stage depends on the duration of four phases:

phase 1 being a lag phase, i.e. the initial development phase characterized by the bacteria adaptation to the ambient environment:

$$\frac{t_2^{\text{bio}}}{t_2^{\text{physchem+bio}}} = \frac{40\ \text{h}}{10\ \text{h}};$$

phase 2 being an exponential phase comprising an acceleration phase, an exponential growth phase and a growth deceleration phase:

$$\frac{t_2^{\text{bio}}}{t_2^{\text{physchem+bio}}} = \frac{50\ \text{h}}{20\ \text{h}};$$

phase 3 being a stationary phase:

$$\frac{t_2^{\text{bio}}}{t_2^{\text{physchem+bio}}} = \frac{15\ \text{h}}{15\ \text{h}};$$

phase 4 being the bacteria dying stage $\frac{t_2^{\text{bio}}}{t_2^{\text{physchem+bio}}} = \frac{20\ \text{h}}{20\ \text{h}}$, where t_2^{bio} is the duration of the bio-oxidation phases for the classic (tank) option, h; $t_2^{\text{physchem+bio}}$ is the duration of the bio-oxidation phases with preliminary two-stage oxidation, h.

$$\frac{\sum t_2^{\text{bio}}}{\sum t_2^{\text{physchem+bio}}} = \frac{125\ \text{h}}{65\ \text{h}} = 1.93 \qquad (2)$$

Consequently, the duration of bio-oxidation features a 2-fold decrease on average if preliminary photoelectrochemical oxidation is applied before leaching.

The experimental verification (aggregate laboratory tests) of the two-stage oxidation theory was conducted using a process scheme simulated on the pulp of leach plant GMZ-3 (ref. Fig. 1). The priority of the proposed process is confirmed by the following patents: 2350665, 2361937 [13, 14].

For experiments, use was made of an integrated sample containing 20 individual samples, weighing 10 kg each. The granulometric characterization of gold in the flotation pyrite-arsenopyrite concentrate showed that the major part of the valuable component (99.9%) falls under the 0.074-mm grain-size category, i.e. there is practically no gold in the ore (except for the South 1 site with visible gold).

The sulfide-arsenide (pyrite-arsenopyrite) concentrate is prepared using a combination of peroxide-hydroxide complexes produced by ozonized air bubbling of the electrolytic cell interelectrode space in sulfuric environment followed by additional oxidation using heterotrophic bacteria [15]. The pyrite-arsenopyrite ratio in pyrite-arsenopyrite ores is subject to continuous changes, sothe bio-oxidation efficiency is reduced. Combined oxidation is proposed to eliminate this deficiency.

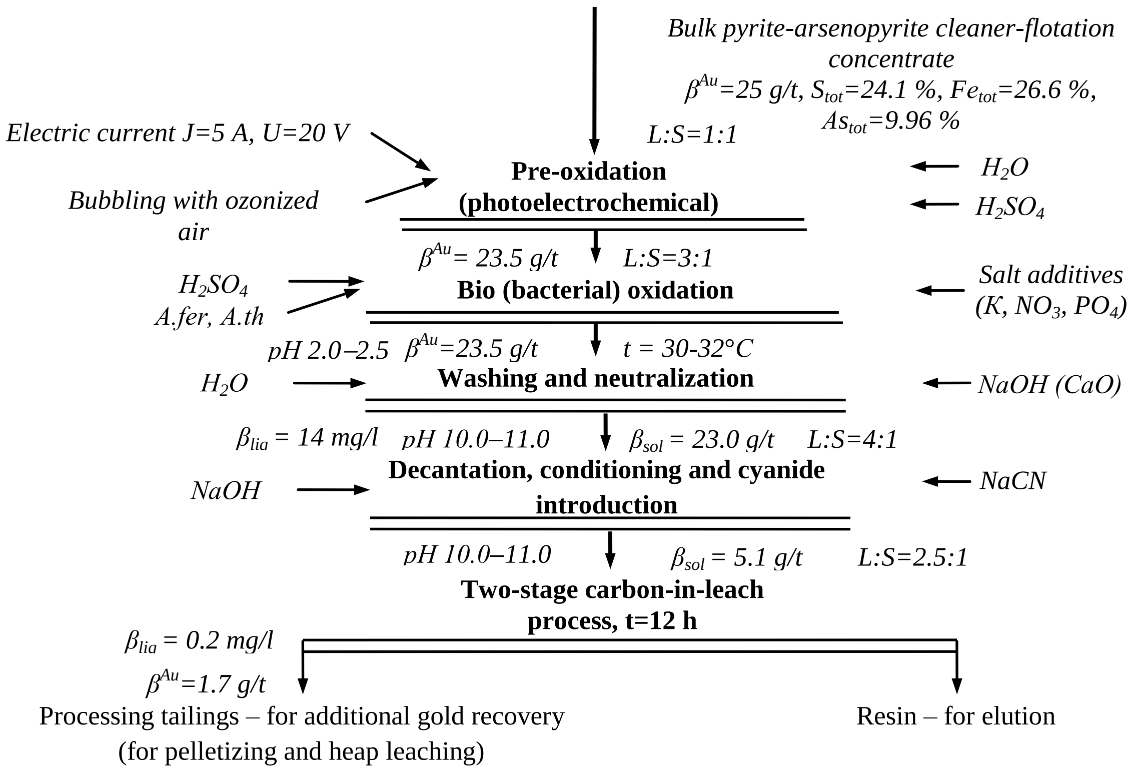


Fig. 1. Process scheme of activation leaching of poor pyrite-arsenopyrite gold-bearing concentrate through combined oxidation techniques

The laboratory setup for the processresearch of ores included the following equipment: a 30-liter laboratory agitator with tubular electrodes used as air tubes; a quartz flask with a fluoroplastic (chemically inert) plug having inlet and outlet glass tubes passed through it; a compressor and an electrical cabinet with a rectifier and a regulated unit; DRT-230 UV lamp mounted above the flask at a distance of about 15 cm (for direct pulp irradiation, it is secured by means of two stands); a photoelectrochemical reactor; a bacterial oxidation column; a bacteria propagator; a vessel for bacteria decontamination.

The proposed process included the following sequence of operations: preparation of a water-based leaching solution; electrolysis of the bubbled solution (≈ 1.5 h, U = 20 V), H_2SO_4 introduction; UV irradiation (DRT-230 UV lamp, 5 min with ongoing electrolysis); mineral matter treatment with a polyreagent solution, oxidation; introduction of salt additives, supply of an additional portion of sulfuric acid, introduction of Acidithiobacillus ferrooxidans and Acidithiobacillus thiooxidans; bacterial oxidation with controlled arsenate ion concentration; washing and neutralization with the addition of NaOH or CaO; conditioning up to pH 10–11, introduction of NaCN (up to 1 g/l) and precyanidation; two-stage sorption leaching.

Initially, two-stage oxidation of the pyrite-arsenopyrite float concentrate was performed using a sulfuric acid solution (3%) treated in an electrolyzer and bubbled for 1 hour with air supplied through the quartz flask arranged within the DRT-230 UV lamp radiation zone. The reactor capacity is 30 L. The obtained polyreagent solution representing a combination of a reactive oxidizer and a complexing substance was used to treat the mineral matter [16]. Exposure to the polyreagent complex caused intensive physical and chemical processes in the liquid and solid phases, which resulted in the primary oxidation of the sulfide matrix, transformation of its surface layers, mainly in the area of active centers, into a sulfate form and partially into a sulfide form, and creation of further favorable conditions for bacterial oxidation. The matrix of the invisible goal containing minerals demonstrated at least a 60-65% oxidation.

After subjecting ore to the photoelectric activation treatment, the second oxidation stage, bio-oxidation, was performed. Bacteria were introduced on a slurry carrier, formed by grinding. In this case, development of the bacteria on the slurry carrier is relatively fast and at the same time allows for concentrating the bacteria before introducing them into the basic matter for leaching, which results in the fast development of bacteria growth centers in the mineral matter.

Precyanidation was performed toobtain the maximum gold concentration into the solid phase solution. In the course of sorption leaching, full dissolution of gold takes place as well as its activated carbon sorption. The optimal physicochemical and process parameters (L:S = 5:1, duration of leaching (36 h) and precyanidation (5 h), gold content in liquid and solid phases versus the preliminary cyanidation cycle time) were determined experimentally. The precyanidation time was reduced by 1 h for the pulp air bubbling with a view to improve the recovery during sorption leaching. The experimental research results are presented in Fig. 2 to 7 and in Table 1. Arsenic contained in the arsenopyrite is an inhibiting factor for the bacterial leaching process, so the culture was adapted to the extremely high arsenic content. In order to reduce the dissolved arsenic concentration and preserve the residual reactive oxygen that would intensify subsequentbacterial oxidation, the solid content in the activated pulp was brought to L:S = 5:1. The pulp solid phase experienced a significant increase in the elemental sulfur (over 3 times) and sulfates (almost by an order).

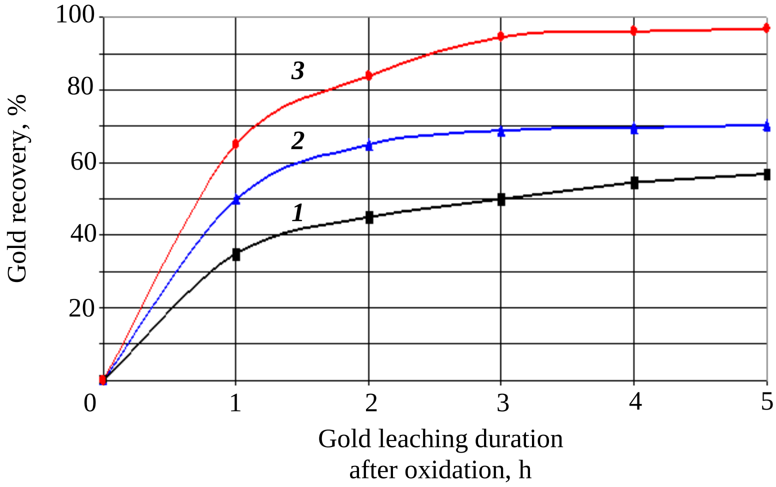


Fig. 2. Gold recovery vs pulp density during cyanidation:
1 – L:S = 1.5:1; *2* – L:S = 3:1; *3* – L:S = 5:1

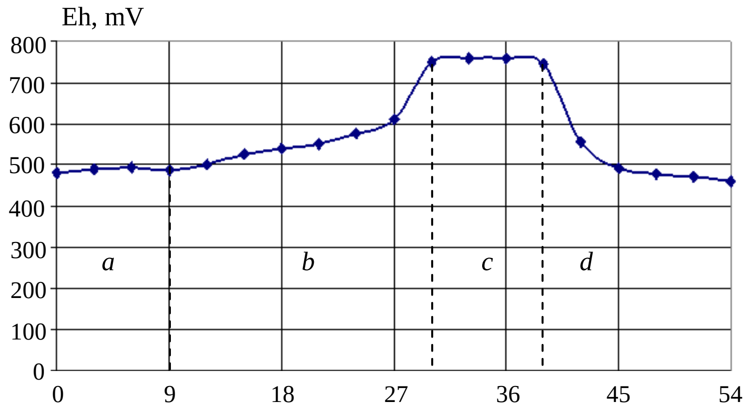


Fig. 3. Eh vs iron leaching duration:
a – lag phase;
b – exponential phase;
c – stationary phase;
d – dying phase

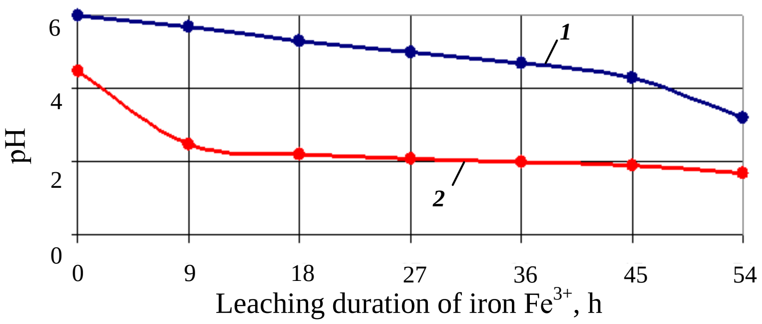


Fig.4. Acid production vs leaching duration:
1 – bio-oxidation (reference option)
2 – physicochemical and bio-oxidation (experimental option)

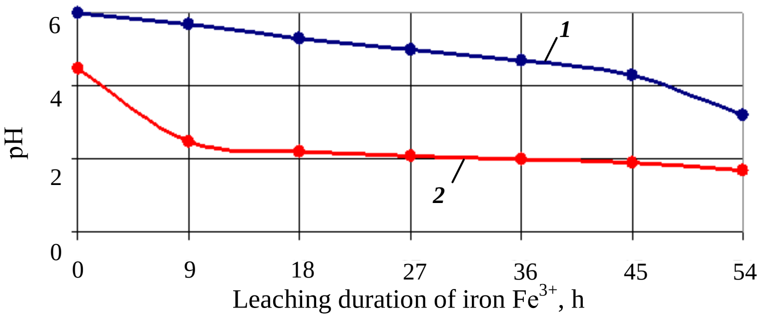


Fig. 4. Acid production vs leaching duration:
1 – bio-oxidation (reference option)
2 – physicochemical and bio-oxidation (experimental option)

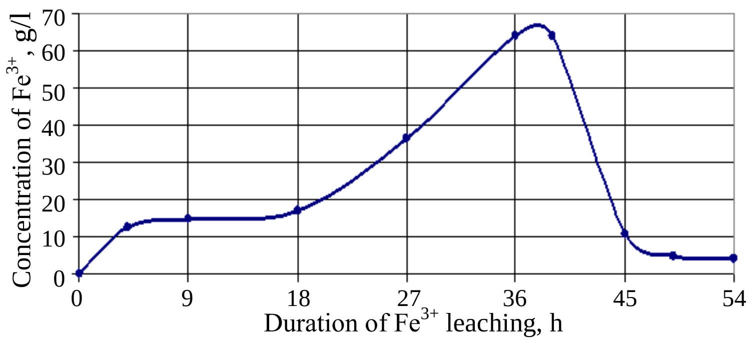


Fig. 5. Fe^{3+} concentration in solution vs leaching duration

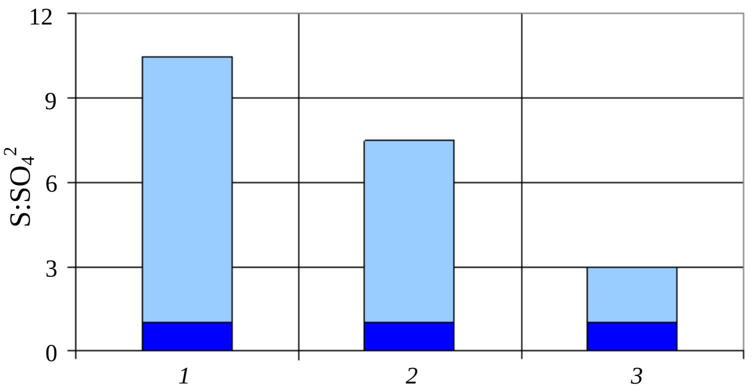


Fig. 6. Ratio of unoxidized (light) and oxidized (dark) sulfur species ($S:SO_4^{2-}$):
1 – in base ore (9.5:1);
2 – after physicochemical oxidation (6.5:1);
3 – after physicochemical and bio-oxidation (2.0:1)

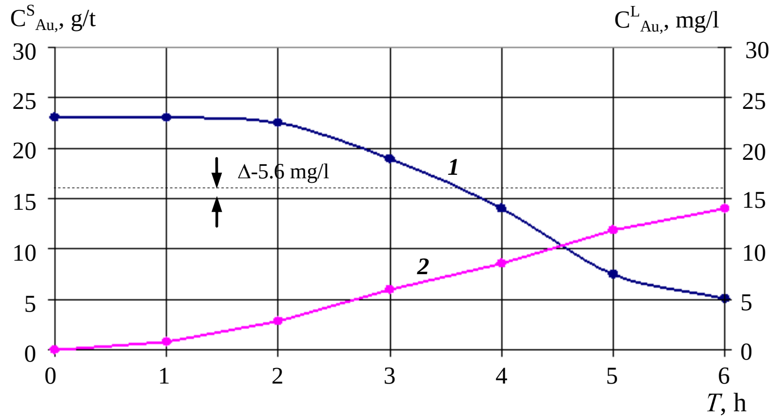


Fig. 7. Gold content in liquid and solid phases vs sorption cyanidation time:
1 – gold content in solid phase, g/t;
2 – gold content in liquid phase, mg/l.

Table 1

Sulfide concentrate leaching behavior

Time of leaching, h	Eh, mV	Period of leaching	Concentration in solution, g/l		Number of bacteria
			Fe^{2+}	Fe^{3+}	cells/ml
0	490	Initial	5.1–5.6	12.4–18.7	4×10^6
3	490				
6	493				
9	487				
12	500				
15	525	Active	0	33.2–63.9	10×10^7
18	540				
21	550				
24	577				
27	610				
30	750				
33	758				
36	760				
39	745				
42	555	Decline	0	20.7–26.3	4×10^7
45	492				
48	477	Final	0	5.5–5.1	3×10^6
51	470				
54	460				

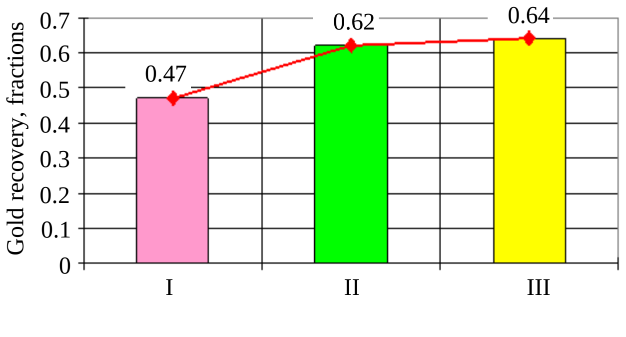
Durations of the lag phase (12 h) and exponential phase during which the microbial population growth and activity are described by the exponential law (30 h) are essential for the bacterial oxidation process. The total leaching duration was determined by the duration of the said two phases while the actual time, by the exponential phase duration.

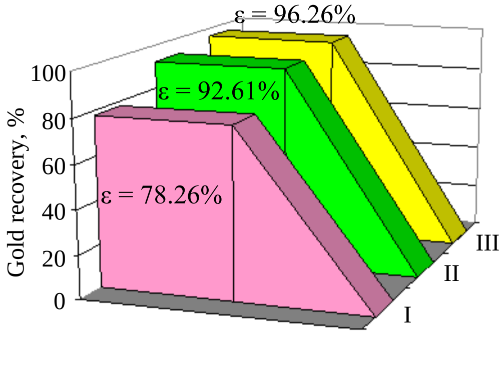
The bio-oxidation (BO) process was accompanied by a decrease in pH and an increase in the redox potential of the product solution. In order to ensure active activation of iron (II) – Fe^{2+}, the pH value was maintained at an optimal level of 2.0 to 2.5. In the course of 24-hour arsenopyrite and pyrite leaching, the solution's pH varied from 4.5 to 2.1 and reached a value of 1.7 in 72 hours. In so doing, active bio-oxidation of arsenopyrite and pyrite took place along with reactive iron hydrolysis with sulfuric acid generation. The Eh value was characteristic of the oxidation-reduction reaction progressing in the pulp. The sulfide-sulfosalt module $S:SO_4^{2-}$ experienced a 4.25-fold decrease, t_{opt} = 28-35°C. The research showed that the cyanidation time reduced significantly (down to 1 h) mainly due topulp saturation with reactive oxygen before cyanidation, which provided for the formation, in addition to cyanides, of hydroxide-peroxide complexes in the pulp liquid phase ensuring accelerated gold dissolution.

The aggregate laboratory tests employed a process scheme involving two-stage sorption with a reduced reagent treatment developed by A.G. Sekisov, Doctor of Engineering Sciences (Institute of Mining, Siberian Branch of the Russian Academy of Sciences), jointly with JSC Geokhim [9, 17]. Unlike the traditional carbon-in-pulp process when the sorbing agent is advanced against the pulp motion, i.e. along the dissolved gold content gradient, the proposed two-stage sorption process involves the introduction ofthe initial sorbing agent following the flow movement of the pulp (the sorbent has been subjected to special preparation and participates actively in the local cyanidation process) followed by introduction of the sorbent primary part against the resin movement.

Three options (Fig. 8) were tested experimentally in the course of the research. The efficiency of the proposed process was evaluated by comparing the gold content in tailings using experimental (II-III) and reference (I) schemes. According to the experimental scheme (III), gold concentration in tailings amounted to 1.7 g/t (ref. Fig. 1), while in the reference option, to 5 g/t. Consequently, the valuable component content in the tailings decreased 2.8 times. The incremental gold recovery amounted to 0.17 units (from 0.47 to 0.64) for resin and 18 % (from 78.26 to 96.26 %) for solid phase.

The proposed process is promising for the national gold industry. Thus, Transbaikalia comes 5th place among the gold-bearing regions of Russia, where gold has always been and still remains the primary metal. Over 1000 deposits and occurrences of vein and placer gold were discovered and studied to various extents in this region.The potential gold reserves are concentrated not just in the deposits that have been developed for many decades, but also in those operated presently. In the Transbaikal gold ore province, such mineral stock site is represented by the Darasun ore field where the proposed technology can be applied successfully based on the ore material composition being close to that of the Kokpatas ore field. Within the ore field boundaries, the Darasun gold ore deposit is distinguished along with lesser ones – Teremkinskoye, Talatui and a number of ore occurrences.





I – bio-oxidation with one-stage sorption as per GMZ-3 regulations;
II – physicochemical and bio-oxidation with one-stage sorption;
III – physicochemical and bio-oxidation with two-stage sorption;
ε – gold recovery, %

Fig. 8. Comparative experimental results with a reference line:
a – resin gold recovery; *b*– solid-phase gold recovery

The distinguishing feature of the Darasun field ores (Darasun, Teremkinskoye, Talatui gold ore deposits) is the high content (15 to 60%) of sulfides (pyrite, arsenopyrite, chalcopyrite, galenite, sphalerite, pyrrhotine, etc.) and sulfosalts.

Further development of the Transbaikal raw material base is planned through the introduction of new competitive high-performance ore processing technologies requiring no high capital or operational costs and providing a quick return on investments as compared to the traditional valuable component recovery techniques.

Thus, use in manufacturing processes of highly reactive non-toxic oxygen-hydrogen ion-radical compounds, obtained through targeted photoelectrochimical effects, provides for a significant increase in the invisible gold recovery from the poor bulk pyrite-arsenopyrite float concentrate and involvement into processing of mineral stock sites with potentially large gold reserves that serve as sources for the national gold reserves.

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“Gornye nauki i tehnologii”/ “Mining science and technology”, 2016, No. 1, pp. 3–12	
Title:	**Preparation of pyrite-arsenopyrite concentrate for gold leaching through the use of polyreagent complexes**
Author 1	Name&Surname: **Lidiya. V. Shumilova** Company: **Transbaikal State University** Work Position: **Professor** Contacts: **shumilovalv@mail.ru**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-3-11**
Abstract:	The article focuses on the causes of refractoriness of sulfide ore stock with superdispersed gold inclusions. In order to increase gold recovery, a hypothesis about effective (peroxide) preparation of refractory raw materials in the process of leaching has been described. The author focuses on theoretical justification of the physical-chemical model and the two-stage mechanism of oxidation (photoelectrochemical and bacterial) of material using microorganisms Acidithiobacillus ferrooxidans and Acidithiobacillus thiooxidans. A scheme of two-staged (physicochemical and bacterial) oxidation was developed. It is based on photoelectrochemical effects for the leaching of gold from refractory materials. It is shown that photoelectrochemical pre-oxidation makes it possible to increase development of the mineral matrix with its further bacterial oxidation. It also helps to increase the efficiency of invisible gold extraction from sulfide ores. Laboratory tests have been performed on the combined oxidation scheme on electroactivation carbon-in-leach of invisible gold from the Kokpatas deposit’s sulfide ores. The authors present the results of the laboratory tests of the activation methods for impacting upon the refractory ores before metal leaching, which prove the efficiency of such non-traditional methods for developing mineral matrix. Gold extraction using the combined technology of sulfide and arsenopyrite oxidation increases by 18 %.
Keywords:	refractory ores, pyrite, arsenopyrite, invisible gold, physical and chemical oxidation and bacterial oxidation, polyreagent schemes, reactive oxygen, electrochemical and photochemical processes.
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MONITORING THE CONDITION OF GEAR UNITS FOR VARIABLE-FREQUENCY DRIVES OF MINE BELT CONVEYORS

This study provides the results of analysis of mine belt conveyor downtime and considers the relevance of monitoring their technical condition. The monitoring results of VFD conveyor gear units regarding lube oil and vibration parameters are presented.

Keywords: belt conveyor, variable frequency drive, technical diagnostics, vibration diagnostics, analysis of lubricants.

A great number of belt conveyors is currently in operation. In the near future, it is expected that their number and capacity will increase further, along with a greater transport distance for mined rock. The specific power and equipment for belt conveyors is growing and the variable frequency drive has been widely introduced. Operation of the entire plant is dependent on the conveyor's serviceability.

The growing volumes of coal mining using comprehensively mechanized long walls along with the increasing production safety require the development of reliable transport systems. High efficiency and fail-safe operation with a concurrent decrease in energy consumption is the primary challenge facing the manufacturers of mine belt conveyor lines. Another equally important task is to reduce expenses for maintenance and repair. In order to ensure fail-safe operation of the mine belt conveyor during the maximum possible time, the causes of various component failures should be determined [1−3].

Analysis of downtimes caused by the belt conveyor gear unit failure indicates from 7.4 to 18.2% of the total number of failures and amounts to 12% on average. It should be noted that the belt splice failures prevail among the total number of failures. Failures are rather frequent but the average time required for their rectification is 1.5 to 2 h. Downtime caused by gear unit failure are seldom but the average reconditioning time is from 24 to 48 h.

Therefore, identification of the actual technical condition of the mine belt conveyor gear units is a crucial task.

The study provides monitoring data of the lube oil and vibration parameters of the 3LL1600 mine belt conveyor with length L = 850 m, engineering capacity Q = 3500 t/h, belt speed v = 0 to 4 m/s.

The conveyor drives employ Moventas Santasalo gear units:

D3RST82XO, double-reduction right-angle type;

reduction ratio i = 20.6128;

rated mechanical power at service factor FS=1 $P_{\text{m rated}}$ = 995 kW;

rated thermal power at service factor FS=1 and ambient temperature t_{amb} = 20ºC $P_{\text{t rated}}$ = 779 kW;

permissible oil temperature t_{oil} = 90ºC;

installed motor power P = 500 kW;

high speed shaft rotation speed n = 1500 rpm (25 Hz).

The mining frequency converter station, type ChPSSh-1250/6-0.69-2-UHL5, is designed for step less electrical speed and torque control of the single or multi-motor belt conveyor drive and feeding the supply voltage to all station and conveyor auxiliaries in underground mining workings that fall under the gas (methane) and carbon dust hazard category as per the requirements of safety regulations [4].

The conveyors were commissioned on June 10, 2014.

Figure 1 presents the driving drum belt run-around scheme and arrangement of gear units designated conventionally as P1-P3.

Vibration parameters as a function of the belt conveyor loading and speed were measured during the two months after commissioning (Figures 2 to 4). The loading and speed data

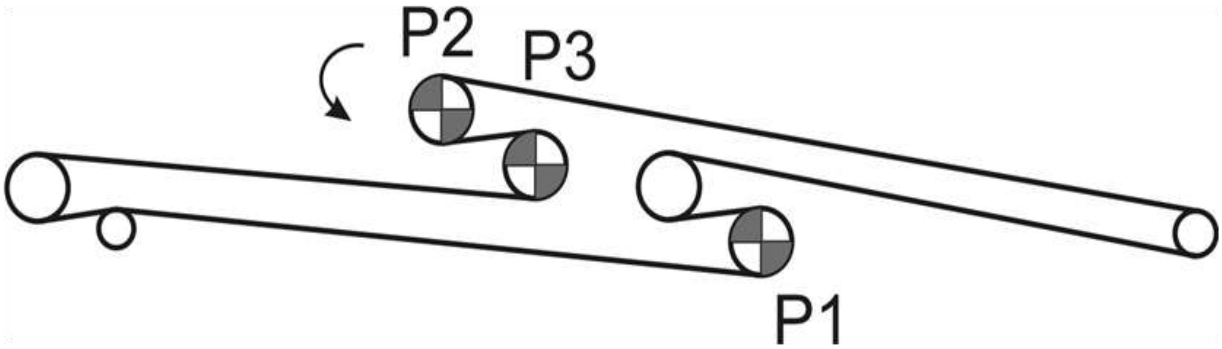


Fig. 1. Driving drum belt run-around scheme and arrangement of gear units

were recorded against the readings of the ChPSSh frequency converter station.

Having analyzed the vibration parameters, we can conclude that the gear unit P1 has the heaviest load while gear unit P3 is the least loaded, which is in line with the classical theory of calculating traction force on the belt conveyor drums [5].

A feature of the variable frequency drives is the dependence of vibration levels on the drive motor rpm [12]. At the run-in stage, the minimum vibration levels are observed for loading levels of 25 to 30%.

Then the lube oil and vibration parameters were analyzed depending on the belt conveyor operating time as per the requirements of regulatory documents [6–11].

The kinematic viscosity data are presented in Table 1 and Figure 5. The results of mechanical impurity accumulation for gear units P1 are presented in Table 2 while primary impurities for the remaining gear units are shown in Figures 6, 7.

The open-cup flash point data are presented in Table 3.

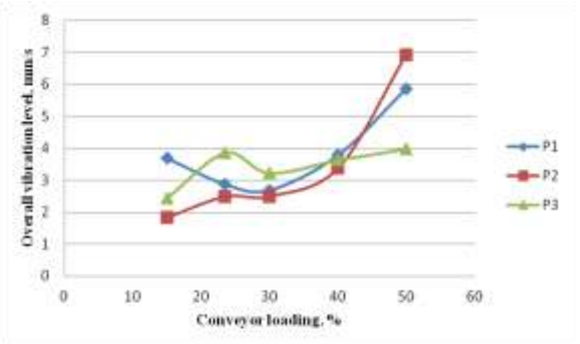


Fig. 2. Overall vibration level in 2 to 200 Hz range vs conveyor loading

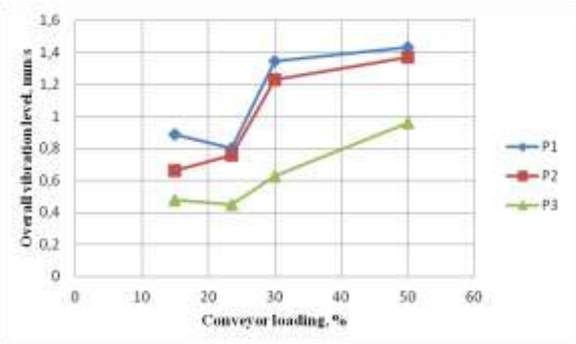


Fig. 3. Overall vibration level in 100 to 2000 Hz range vs conveyor loading

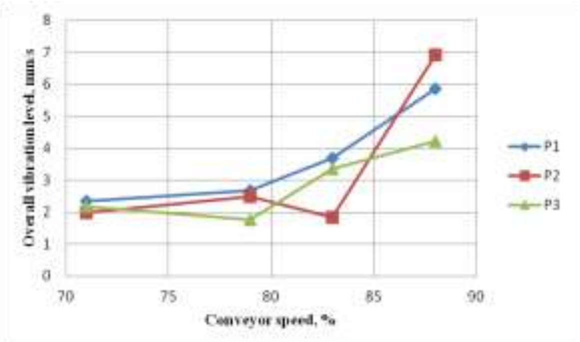


Fig. 4. Overall vibration level in 100 to 2000 Hz range vs conveyor speed

Table 1

Kinematic viscosity (KV) as per GOST 6258−85

Gear unit	Temperature, °C	Standard value	Sample of 29.08.2014	Sample of 19.02.2015	Sample of 12.03.2015		Sample of 06.07.2015	
		KV, mm^2/s	KV, mm^2/s	KV, mm^2/s		KV, mm^2/s	FV	KV, mm^2/s
P1	40	320.00	297.70	307.00		325.20	45.40	345.00
	100	24.10	25.33	24.17		24.50	3.50	25.33
P2	40	320.00	335.50	345.50		341.40	46.30	351.20
	100	24.10	24.67	24.50		25.83	3.58	26.00
P3	40	320.00	304.80	308.60		335.50	43.30	329.70
	100	24.10	23.33	24.67		24.67	3.58	26.00

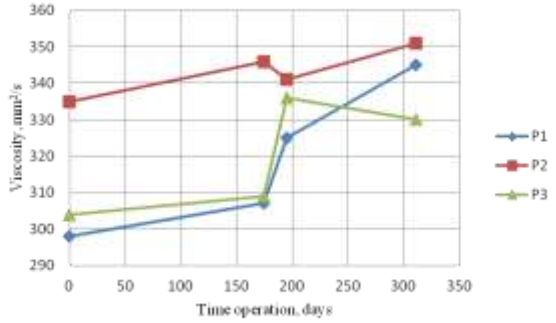


Fig. 5. Oil kinematic viscosity vs operating time

Table 2

Gear unit P1 wear products, g/t

Elements	Permissible limits	Sample of 29.08.2014	Sample of 19.02.2015	Sample of 12.03.2015	Sample of 06.07.2015
Fe	126–200	40.340	160.120	204.800	171.520
Si	21–30	37.640	33.880	44.550	39.830
Cu	100–150	5.526	0.480	8.960	1.650
Al	4–7	1.518	1.540	2.380	2.000
Cr	2–5	0.683	1.300	1.070	1.20150
Pb	–	2.547	3.450	4.230	2.480
Sn	–	5.781	9.340	7.900	6.980

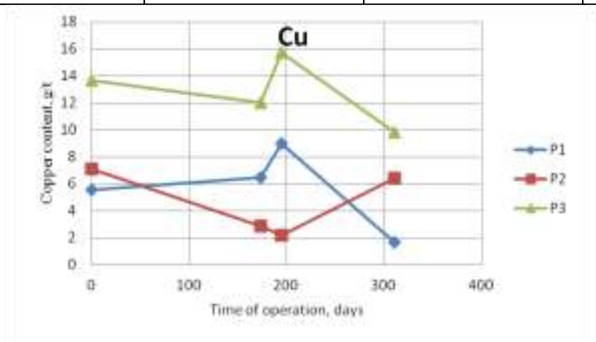


Fig. 6. Iron content vs operating time

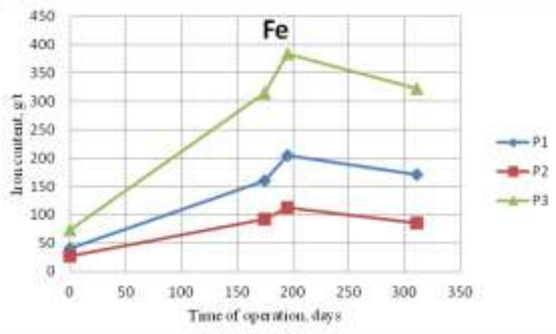


Fig. 7. Copper content vs operating time

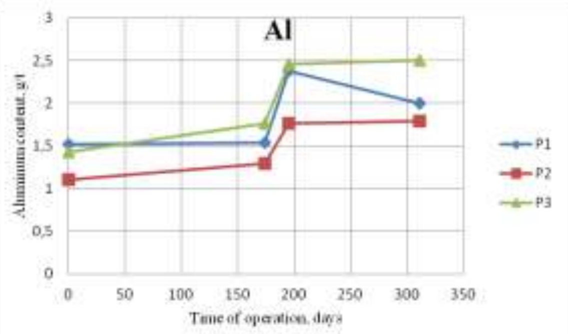
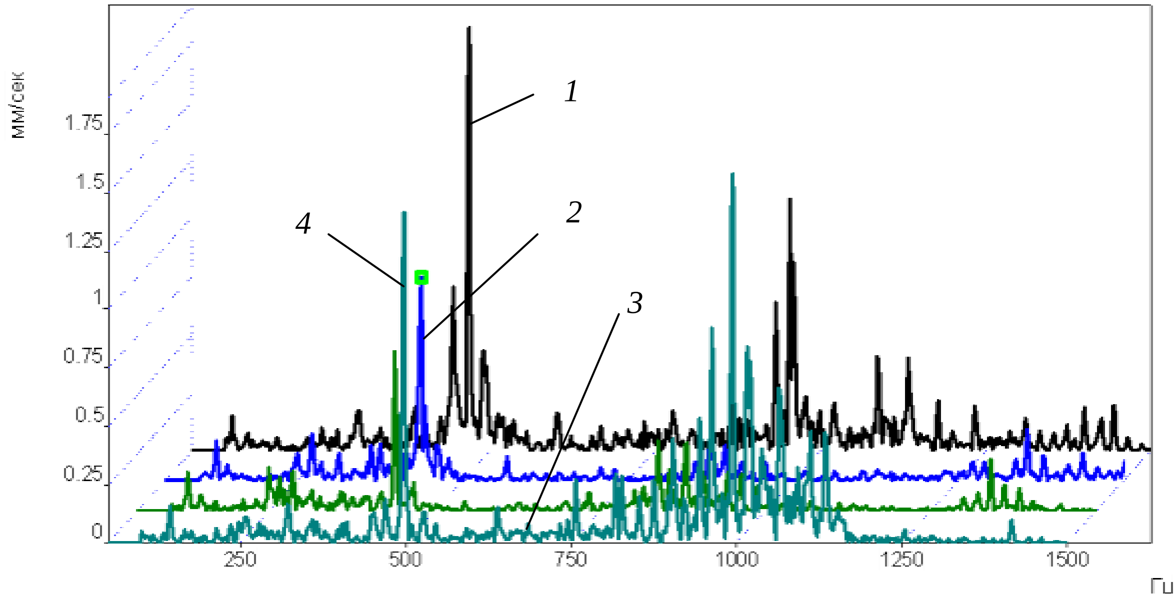


Fig. 8. Aluminum content vs operating time

Table 3

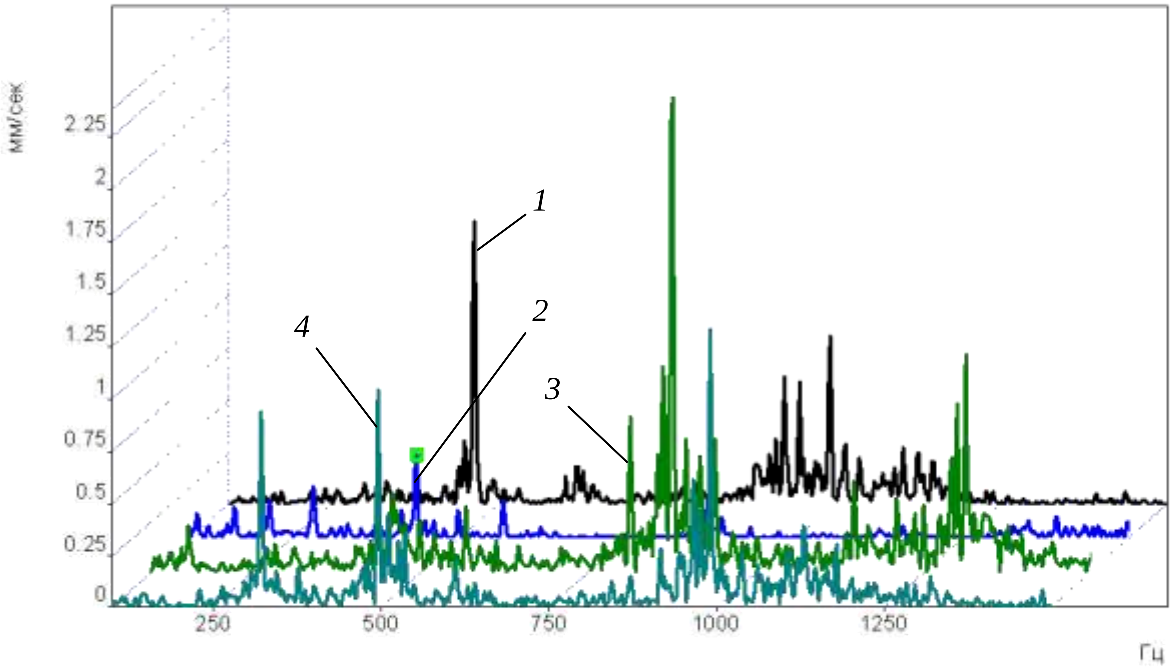
Open-cup flash point, °C (GOST 26378.4−84)

Gear unit	Standard value (DIN ISO 2592)	Sample of 29.08.2014	Sample of 19.02.2015	Sample of 12.03.2015	Sample of 06.07.2015
P1	255	226	222	238	236
P2	255	248	228	235	215
P3	255	234	225	230	235



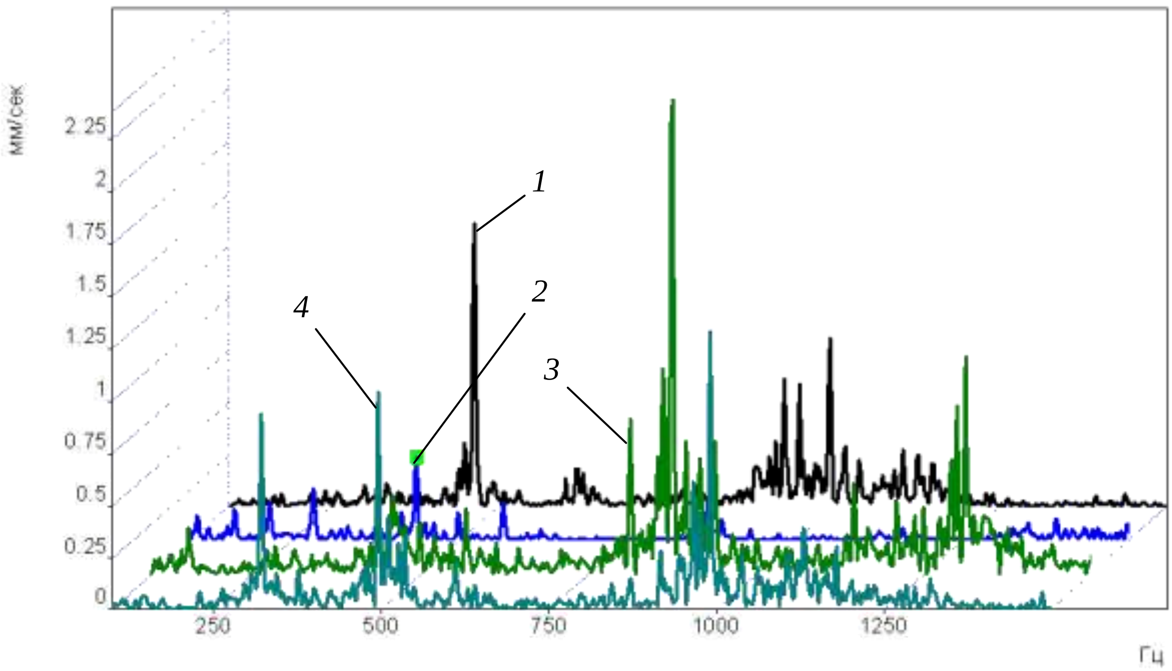
[vertical axis: "mm/sec"] [horizontal axis: "Hz"]

Fig. 9. Comparison of vibration velocity spectra in range from 100 to 1500 Hz for gear unit P1: *1* – 28.08.2014, *2* – 19.02.2015, *3* – 12.03.2015, *4* – 06.07.2015



[vertical axis: "mm/sec"] [horizontal axis: "Hz"]

Fig. 10. Comparison of vibration velocity spectra in range from 100 to 1500 Hz for gear unit P2: *1* – 28.08.2014, *2* – 19.02.2015, *3* – 12.03.2015, *4* – 06.07.2015



[vertical axis: "mm/sec"] [horizontal axis: "Hz"]

Fig. 11. Comparison of vibration velocity spectra in range from 100 to 1500 Hz for gear unit P3: *1* – 28.08.2014, *2* – 19.02.2015, *3* – 12.03.2015, *4* – 06.07.2015

Analysis of the behavior of oil parameters shows that viscosity rises in the course of operation owing to the evaporation of low-boiling fractions. In this case, the higher the initial viscosity, the less the mechanical impurity accumulation and the less the wear intensity of bearings and gear teeth in the gear unit (ref. Figures 5 to 8): *1* – 28.08.2014, *2* – 19.02.2015, *3* – 12.03.2015, *4* – 06.07.2015.

Figures 9 to 11 show the spectra of the vibration velocity RMS values in the range typical of gear unit tooth mesh frequencies. Measurements were performed according to [4] for a secondary shaft point in the axial direction being most characteristic of the vibration at the tooth mesh frequencies of the first gear pair.

Having analyzed the spectra (ref. Figures 8 to 10), we arrived at the conclusion that the running-in processes in the gear unit were completed, the highest vibration levels are observed in gear unit P1 with the lowest in gear unit P3.

The gear unit manufacturer, Moventas Santasalo, recommends the first oil replacement after 800 to 1000 hours of operation and subsequently after 10,000 hours of operation or annually. The first oil replacement was not actually performed, only replenishment after 5000 h. The present oil condition in gear units P1 and P2 is satisfactory while in gear unit P3 the concentrations of mechanical iron (Fe) impurities have been exceeded.

Thus, the results of the conveyor monitoring since the commissioning date enable us to select the proper condition criteria for the gear unit components as a function of its loading and speed and elaborate oil replacement recommendations depending on the gear unit location in the conveyor process scheme.

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8. GOST 26378.4-84 Nefteproducty otrabotannye. Metod opredeleniya temperatury v spyshki v otkrytom tigle.
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“Gornye nauki i tehnologii”/ “Mining science and technology”, 2016, No. 1, pp. 14–19	
Title:	**Monitoring the condition of gear units for variable-frequency drives of mine belt conveyors**
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DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-13-18**
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REACTIVE POWER COMPENSATION IN DISTRIBUTION SYSTEMS OF MINING COMPANIES

Major mining sector power consumers use both active and reactive power to create network flows. Reactive power compensation and the effective placement of compensating devices is a top priority in view of energy efficiency policies. Power compensation is a priority issue when optimizing voltage modes and power consumption in order to reduce active losses and improve energy quality. An effective solution which could help reduce power network losses is the installation of reactive power compensating devices (RPC). Analysis of optimum power calculation methods and determination of the location of compensating devices takes into account the specific operating conditions of mining companies, namely, the necessity to separate power networks into surface and underground networks. The target function has been represented based on annual reduced cost data and a system of limitations for the target function has been determined. The target function analysis shows that the function is separable, and the task could be solved with discrete programming methods.

Keywords: mining companies, reactive power, compensation, target function, limits, cost function, minimization, discrete programming, equivalent characteristics.

Mining companies, in particular mining and processing integrated works, feature large installed capacity, a complex and divergent power network structure, specific operation modes as well as specific modes for interacting with power systems. In this respect, RPC and determining effective locations for compensating devices is a top priority for mining companies in view of energy efficiency policies. Power compensation is a priority issue when optimizing voltage modes and power consumption in order to reduce active losses and improve energy quality. An effective solution which could help reduce power network losses is the installation of RCP devices [1]. Effective power consumption becomes even more important when switching to energy efficiency technology and taking into account ever-growing power prices.

Analysis of existing optimum power calculation methods and the location of compensating devices (CD) shows that the discrete programming method is the most effective solution for mining companies to optimize reactive load compensation (RLC). This solution helps create an RCP model accounting for the specifics of mining companies' power networks (separated into surface and underground networks) and obtain simple and effective problem solution algorithms [2, 4].

The nominal power of compensating devices Q_{ki} (for power networks up to 1 kW and more than 1 kW) and reactive power generated by synchronous motors (SM) can be determined with a target function based on minimum reduced cost:

$$C=\sum_{1}^{N} C_{ki}\left(Q_{ki}\right)+C_E \qquad (1)$$

where $C_{ki}\left(Q_{ki}\right)$ is capital cost for CD installation, capacity is Q_{ki}; N is SM and CD number; C_E is the function of the cost of reactive power compensation.

Cost function C_E can be presented as follows

$$C_E=c_0\Delta P_P=\frac{c_0}{U^2}\left\|\bar{Q}-\bar{Q}_k\right\|R\left\|\bar{Q}-Q_k\right\|,$$

where c_0 is the unit cost of power and energy loss reduced to the active power cost; ΔP_P is active power loss; U – system voltage; $\bar{Q}$ – vector maximum designed reactive loads of network nodes; $\bar{Q}_k$ – vector of CD power; R – matrix of node active network (6−10 kW) resistance with respect to the main step-down substation (MSS).

The target function (1) is determined with specific limitations:

$$\sum_{1}^{N} Q_{ki} = Q_k,\ Q_{k\min i} \le Q_{ki} \le Q_{k\max i}\ \ i = 1, 2, \ldots, N,$$

$$Q_{ki} \in \{Q_{k\text{н}}\}\ \ i = 1, 2, \ldots, n,$$

where Q_k is the necessary total power of synchronous compensating devices in the installed power system; $\{Q_{k\text{н}}\}$ is the variety of standard CD power values; n is the CD number.

The function of the cost of compensating devices $C_{ki}(Q_{ki})$, due to specific operation principles of underground and surface networks, shall be provided separately for networks up to 1 kV and over 1 kV:

- load bus i up to 1 kV:

$$C_{ki}(Q_{ki}) = e\left[C_{LVB}(Q_{ki}) + C_{TSi}(Q_{ki}) + c_0 \rho_l Q_{ki}\right];$$

- load bus i up to 6–10 kV:

$$C_{ki}(Q_{ki}) = eC_{HVB}(Q_{ki}) + c_0 \rho_h Q_{ki};$$

where e is the annual deduction from capital costs, including standard coefficient; $C_{LVB}(Q_{ki})$ is cost for low-voltage battery; $C_{HVB}(Q_{ki})$ is cost for high-voltage battery; $C_{TSi}(Q_{ki})$ is the reduction of cost for transformer sub-station (TS) *after* installation of CD up to 1 kV Q_{ki}; ρ_l, ρ_h – specific loss of active power in CD up and over 1 kV.

Cost of CD up to and over 1 kV can be determined as follows:

$$C_{CBi}(Q_{ki}) = A_i K'_{0i} + \sum_{j=1}^{A_i} B_{ij} K_{0i} + k_{\text{un}i} Q_{ki},$$

where C_{CBi} is cost for condenser battery; A_i, B_{ij} is the number of TS and the number of CD in every TS; K'_{0i}, K_{0i} is the cost of switching and control equipment for one TS and one CD respectively; $k_{\text{un}i}$ is CD unit cost.

Power is supplied to mining company process sites through one- and two-transformer circuits, depending on consumer reliability category. Therefore, installation of CD on the low-voltage side of distribution transformers may result in a reduction of TS power (but this will not result in a reduction in TS quantity) by standard power stage. Then

$$C_{TSi}(Q_{ki}) = \begin{cases} C_{TS}(S'_{\text{nom}i}) - C_{TS}(S_{\text{nom}i}) \\ \qquad \text{when} \quad Q_{kmi} \le Q_{ki} \le Q_{kni}, \\ 0 \qquad \text{when} \quad Q_{ki} \le Q_{kmi} \text{ or } Q_{ki} > Q_{kmi}, \end{cases}$$

where $S'_{\text{nom}i}$ is TS nominal power ($S'_{\text{nom}i}$) which is one stage below the installed power of TS i ($S_{\text{nom}i}$); $C_{TS}(S_{\text{nom}i})$, $C_{TS}(S'_{\text{nom}i})$ – TS cost, capacity is $S_{\text{nom}i}$ and $S'_{\text{nom}i}$.

Maximum and minimum CD power (the range available for a decrease of CD installed power) can be determined by the following formula:

$$Q_{k\min i} = Q_i - \sqrt{\left(S'_{\text{nom}i}\right)^2 - P_i^2};$$

$$Q_{k\max i} = Q_i + \sqrt{\left(S'_{\text{nom}i}\right)^2 - P_i^2}.$$

Cost function $C_{ki}(Q_{ki})$ for same type CDs can be determined by the following formula

$$C_{ki}(Q_{ki}) = eK_p m + \left[\frac{M_{1i}}{Q_{\text{nom}i}} Q_{ki} + \frac{M_{2i}}{mQ_{\text{nom}i}^2} Q_{ki}^2\right] c_0,$$

where K_p is an SM exciting regulator; $m = N - n$ is the number of SM in a group; M_{1i}, M_{2i} are constant values determined by the motor nominal parameters [3].

The active power loss can be presented as follows

$$\Delta P_{\text{M}} = \Delta P_0 + \sum_{1}^{N-1}\left(a_i Q_{ki}^2 + b_i Q_{ki}\right) + \\ + \frac{2}{U^2} \sum_{i=1}^{N-1} Q_{ki} \sum_{j=i+1}^{N} a_{kj} b_{ij}, \qquad (2)$$

where $\Delta P_0 = \frac{1}{U^2} \bar{Q} R Q$; a_i and b_i are coefficients related to CD installation and mounting: $a_i = \frac{1}{U^2} R_i$; $b_i = -\frac{2}{U^2} \bar{Q} R_i$ at $i = 1, 2, \ldots, N$;

$$\sum_{1}^{N} R_i = \begin{vmatrix} R_{1i} \\ R_{2i} \\ \vdots \\ R_{Ni} \end{vmatrix} - i \text{ column of matrix } N.$$

If quadratic function i (in brackets) is referred to the CD cost in i load bus $C_{ki}(Q_{ki})$, then the cost function C can be presented by the following formula

$$C = c_0 \Delta P_0 + \sum_{i=1}^{N} C_i(Q_{ki}) + \frac{2c_0}{U^2} \sum_{i=1}^{N-1} Q_{ki} \sum_{j=i+1}^{N} Q_{kj} R_{ij}, \quad (3)$$

where $C_i(Q_{ki}) = C_{ki}(Q_{ki}) + c_0 a_i Q_{ki}^2 + c_0 b_i Q_{ki}$ is the cost function of load bus i or the cost of installation in bus i of CD with power capacity Q_{ki}.

Analysis of target function (5) shows if $R_{ij} = 0$ for all $j = 1, 2, ..., N-1$ and $j = i+1, i+2, ..., N$, the function is separable, i.e.

$$C = c_0 \Delta P_0 + \sum_{i=1}^{N} C_i(Q_{ki}) \text{ when } \sum_{i=1}^{N} Q_{ki} = Q_k, \quad (4)$$

and the problem can be easily solved with discrete programming (DP) methods [5, 6].

However, only radial networks meet the $R_{ij} = 0$ conditions, and this limits the use of DP algorithms for an arbitrary open-loop network.

The non-separability of the RLC target function can be overcome by multiple refinement of network power consumption mode during calculation and determination of additional power losses from two varying loads at common circuit branches. However, such multiple refinement of power modes can be avoided by reducing the target function (7) to a deterministic format.

Let us accept that the bus numeration starts from the end of the main line and take into account the bus resistance matrix properties, then we can get the following final sum of the target function (6)

$$\frac{2c_0}{U^2} \sum_{i=1}^{N-1} Q_{ki} \sum_{j=i+1}^{N} Q_{kj} R_{ij} = \frac{2c_0}{U^2} \Big[Q_{k1} Q_{k2} R_2 +$$

$$+ (Q_{k1} + Q_{k2}) Q_{k3} R_3 + \ldots + \left(\sum_{i=1}^{N-1} Q_{ki} \right) Q_{kN} R_N \Big],$$

where $R_2 = R_{12}$, $R_3 = R_{13} = R_{23}$, …, $R_N = R_{1N} = R_{2N} = \ldots = R_{N-1,N}$.

Let's denote the total power of CD in i load bus by $Q_k^i = \sum_{j=1}^{i} Q_{ki}$, then the expression (6) will take on the following form

$$C = \sum_{i=1}^{N} \left[C_i(Q_{k1}) + \frac{2c_0 R_i}{U^2} Q_k^{i-1} Q_{ki} \right]. \quad (5)$$

To simplify the calculation process, the expression (5), taking into account that $Q_{ki} = Q_k^i - Q_k^{i-1}$, can be presented as follows

$$C = \sum_{i=1}^{N} \Big[C_i\left(Q_k^i - Q_k^{i-1}\right) +$$

$$+ \frac{2c_0}{U^2} R_1 Q_k^{i-1} \left(Q_k^i - Q_k^{i-1}\right) \Big] \quad (10)$$

with limitations:

$$Q_k^0 = 0, \quad Q_k^N = Q_k,$$

$$Q_{k\min}^{i-1} \le Q_k^{i-1} \le Q_{k\max}^{i-1},$$

$$Q_{k\min}^{i} \le Q_k^{i} \le Q_{k\max}^{i}.$$

The calculation procedure is reduced to the determination of the absolute minimum value for C^* for the variables Q_k^i, Q_k^j, Q_k^r with fixed set variable $Q_k = \left(Q_k^i, Q_k^j, Q_k^r\right)$ on which function C depends.

By selecting Q_k^{N-1} as the total power $N-1$ CD and making this value fixed, we can set it to minimum for other variables Q_k^1, Q_k^2, Q_k^{N-2}.

Once all calculation procedures are completed, the function minimum can be presented in the following form

$$C^* = \min \Big\{ C_N\left(Q_k^N - Q_k^{N-1}\right) +$$

$$+ \frac{2c_0}{U^2} R_N Q_k^{N-1} \left(Q_k^N - Q_k^{N-1}\right) + C_{N-1}^E\left(Q_k^{N-1}\right) \Big\},$$

where Q_k^{N-1} may take on discrete values from $Q_{k\min}^{N-1}$ to $Q_{k\max}^{N-1}$, and

$$C_{N-1}^E\left(Q_k^{N-1}\right) = \sum_{i=1}^{N-1} \Big[C_i\left(Q_k^i - Q_k^{i-1}\right) +$$

$$+ \frac{2c_0}{U^2} R_i Q_k^{i-1} \left(Q_k^i - Q_k^{i-1}\right) \Big]$$

This function can be equivalent depending on Q_k^E:

$$C_{N-1}^E\left(Q_k^E\right)=\min\left\{C_{N-1}\left(Q_k^E-Q_k^{N-2}\right)+\right.$$
$$+\frac{2c_0}{U^2}R_{N-1}Q_k^{N-2}\left(Q_k^E-Q_k^{N-2}\right)+$$
$$\left.+\sum_{i-1}^{N-2}\left[C_i\left(Q_k^i-Q_k^{i-1}\right)+\frac{2c_0}{U^2}R_iQ_k^{i-1}\left(Q_k^i-Q_k^{i-1}\right)\right]\right\}$$

A similar approach is used to obtain the function $C_{N-2}^E\left(Q_k^{N-2}\right)$, etc. This process continues until the following expression is obtained at the final stage

$$C_1^E\left(Q_k^E\right)=C_1\left(Q_k^1\right),$$

where $Q_{k\min}^1 \le Q_k^E = Q_k^1 \le Q_{k\max}^1$; $C_1\left(Q_k^1\right)$ is the cost function for load bus 1.

Minimization is carried out for every fixed value Q_k^E from the range $\left[Q_{k\min}^i;Q_{k\max}^i\right]$ by resetting power values Q_k^{i-1} that meet the requirement $Q_{k\min}^{i-1} \le Q_k^{i-1} \le Q_{k\max}^{i-1}$. The variables Q_k^{i-1} take on discrete values with the pre-set quantization step *h*.

The equivalent cost characteristic $C_1^E\left(Q_k^E\right)$ determines the minimum cost for the main power line with optimum installation of reactive power sources in load buses *i* with total power of Q_k^E.

Development of equivalent characteristic $C_i^E\left(Q_k^E\right)$ means the replacement of load buses *i*, each with reactive power source (RPS) Q_k^{i-1}, by equivalent load bus with RPS power of $Q_k^Э = Q_k^i = \sum_{j=1}^{i} Q_{kj}$, the cost function of which is determined by the equivalent characteristic $C_i^E\left(Q_k^E\right)$.

The calculation shall start from the main line end ($i=1,2,3,...,N$) and continue until all main line load buses are reduced into one load bus and optimum cost $C_N^E\left(Q_k\right)=C^*$ is determined.

Conclusion. The discrete programming approach for determining the effective location of compensation devices in distributing networks of mining companies has been proposed. This method helps determine the optimum solution by applying specific limitations and rules on the target function.

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“Gornye nauki i tehnologii”/ “Mining science and technology”, 2016, No. 1, pp. 20-23	
Title:	**Reactive power compensation in distribution systems of mining companies**

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DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-20-23**

Abstract:	Major mining sector power consumers use both active and reactive power to create network flows. Reactive power compensation and the effective placement of compensating devices is a top priority in view of energy efficiency policies. Power compensation is a priority issue when optimizing voltage modes and power consumption in order to reduce active losses and improve energy quality. An effective solution which could help reduce

	power network losses is the installation of reactive power compensating devices (RPC). Analysis of optimum power calculation methods and determination of the location of compensating devices takes into account the specific operating conditions of mining companies, namely, the necessity to separate power networks into surface and underground networks. The target function has been represented based on annual reduced cost data and a system of limitations for the target function has been determined. The target function analysis shows that the function is separable, and the task could be solved with discrete programming methods.
Keywords:	mining companies, reactive power, compensation, target function, limits, cost function, minimization, discrete programming, equivalent characteristics
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THE RELATION BETWEEN TOTAL MOISTURE CONTENT INDEXES IN FROZEN ROCKS AND SOIL

Moisture in frozen rocks and soil is generally represented as two components – ice and unfrozen water. To quantify the moisture content in these rocks, two indexes called total and general moisture content were earlier proposed. The paper shows that such names do not properly reflect these indexes. In fact, it is the total mass moisture and water content. For a more complete and more accurate assessment of the impact of moisture on the physical properties of frozen rocks and soil, the paper proposes using total volumetric and absolute moisture and water content. The relation between total moisture indexes in frozen rocks and soil considered in the paper was identified.

Keywords: frozen rock, total moisture, total water content, total absolute moisture, total absolute water content, the relation between them.

To assess the content of all kinds of moisture (ice, frozen water) in frozen rocks and soil (hereafter frozen ground) two indexes, *total and general moisture* signified in the absolute majority of literature sources as $W_{t.}$ and $W_{g.}$ [1–6], were proposed. Total moisture was defined as the relation of moisture mass in frozen ground $M_m=M_{ice}+M_{nonfr.w}$, where M_{ice}, $M_{nonfr.w}$ is the mass, kg, ice and non-frozen water to mass of dry rocks and soil $M_{dry\ ground}$, i.e. to the mass of mineral matrix $M_{\min}$, and general moisture as relation of M_m to the mass of frozen ground M_{gr}.

However, it is necessary to note that the definitions proposed for $W_{t.}$ and $W_{g.}$ do not exactly correctly depict the essence of these indexes.

In the classic approach to determining moisture content in substances [7, 8], substance moisture means the moisture content in them, related to the wet condition of substances. If moisture content in a substance is related to its dry condition, this is already moisture content. Considering the fact that $W_{t.}$ is determined by the dry condition of rock, $W_{t.}$ will actually be *total mass water content* with a total dimension of "kg of moisture/kg of dry ground". It is better written as $w_{\Sigma,\text{M}}$ (in technical literature water content indexes are usually indicated by lower case w with corresponding indexes. Here the lower index Σ is a conventional designation of sum, and m of mass). General moisture W_g is total mass moisture. It is better written as $W_{\Sigma,\text{m}}$ (in technical literature moisture indexes are usually indicated by the capital letter W with corresponding indexes). Therefore, we have

$$W_{\Sigma,m} = \frac{M_m}{M_{fr.gr}} = \frac{M_{ice}+M_{nonfr.w}}{M_{fr.gr}};$$
$$w_{\Sigma,m} = \frac{M_m}{M_{dry.gr}} = \frac{M_{ice}+M_{nonfr.w}}{M_{dry.gr}}. \quad (1)$$

Dimension $\left[W_{\Sigma,\text{M}}, w_{\Sigma,\text{M}}\right] = \text{fraction}$.

If we multiply the first parts of the fractions in (1) by 100, $W_{\Sigma,m}$ and $w_{\Sigma,m}$ is measured in percent by weight (% mass), irrespective of the dimensions stated for these indexes, *unit fractions* or *mass percent*, their total dimensions will be:

$\left[W_{\Sigma,m}\right] =$ kg of moisture/kg of frozen grounds;

$\left[w_{\Sigma,m}\right] =$ kg of moisture/kg of dry grounds.

So, according to the dimensional theory

$\left[M_{fr.gr}\right] =$ kg of frozen ground =
= kg of dry ground + kg of moisture
and this is irrespective of whether the ground is loosen or frozen.

However, as noted in [9], the use of only mass characteristics of water content does not allow for a full analysis of moisture impact on the corresponding physical properties of frozen ground. It would be more convenient and clear to use volumetric data of moisture content determined in certain cases like $W_{\Sigma,m}$ and $w_{\Sigma,m}$:

$$W_{\Sigma,\,vol}=\frac{V_m}{V_{fr.gr}}=\frac{V_m}{K_v V_{dry\ gr}}=\frac{V_{ice}+V_{nonfr.w}}{K_V V_{dry\ gr}};$$
$$w_{\Sigma,\,vol}=\frac{V_m}{K_f V_{dry\ gr}}=\frac{V_{ice}+V_{nonfr.w}}{K_f V_{dry\ gr}}, \qquad (2)$$

where $V_{fr.gr}$, $V_{dry\ gr}$ is volume, m^3, frozen (loosen and unripped) ground and dry unripped ground; V_{ice}, $V_{nonfr.w}$ is the volume of ice and non-frozen water in frozen ground, m^3; $K_f \geq 1.0$ is the fragmentation index of dry ground, unit fraction; K_V is the ratio of ground change as a result of fragmentation (destruction) and volumetric swell due to watering and following freezing of water, unit fraction.

The feature of the changing ground condition as a result of their watering and following the formation of ice is that, during watering ground, the water is located in pores, cracks and cavities changing the volume of ground directly due to an increase in their volume and indirectly impacting their swelling and rippability. The same happens when freezing water. It should also be taken into account that in wet ground especially in cohesive and loosen ground, the fragmentation index K_f and swelling index K_{sw} depend on their moisture content. This dependence for a number of rocks can sometimes be rather complex. For example, by watering fine-grained ground such as quartz sand, a decrease in K, as well as its increase can occur. This is established by the fact that at certain ranges moisture can facilitate the agglomeration of fine hard particles into larger conglomerates, and the other way round, at other ranges. K_f and K_{sw} change through even more complex dependences while water freezes, especially in cohesive ground where the ice formation process ends at temperatures of much lower than 0°C.

Therefore, a change in volume of dry ground only happens due to their fragmentation (destruction), while K_f at plus and minus temperatures is actually the same. Wet grounds in unripped (undistracted) condition, in that case $K_f = 1.0$, change their volume in the process of watering and following freezing of water. So, in these grounds $K_V = K_{sw}$. In friable grounds the process of swelling due to watering and following freezing of water is accompanied by the fragmentation process. So, their K_V becomes an integral index indicated in general as $K_V = K_f K_{sw}$. In practice, when indicating the changing volume of friable grounds as a result of changing their fragmentation, watering and freezing of water, it is very difficult to distinguish the impact of K_f and K_{sw} on K_V. So, in these grounds K_V is experimentally determined, also taking into account the impact of K_f and K_{sw}.

If in (2) you multiply the right parts of fractions by 100, $W_{\Sigma,V}$ and $w_{\Sigma,V}$ will be measured in percent by volume (%V). However, you should not forget that irrespective of the dimensions of these indexes, unit fractions or volume percent, their full dimensions for unripped or friable frozen grounds will be:

$\left[W_{\Sigma,v}\right]=$ m^3 moisture/m^3 frozen grounds = = m^3 moisture/m^3 dimension of frozen grounds;

$\left[w_{\Sigma,v}\right]=$ m^3 moisture/m^3 dry grounds = = m^3 moisture/m^3 dimension of dry grounds.

As well as mass and volume indexes of moisture, in many cases it is convenient to use indexes of mass concentration of moisture that can be assigned to frozen and dry ground. In the first case it will be total moisture $W_{\Sigma,\,\mathrm{abs}}$ of frozen ground, and in the second case it will be total absolute water content $w_{\Sigma,\,\mathrm{abs}}$ defined as:

$$W_{\Sigma,\,\mathrm{abs}}=\frac{M_m}{M_{fr.gr}}=\frac{M_{ice}+M_{nonfr.w}}{M_{fr.gr}}=$$
$$=\frac{M_{ice}+M_{nonfr.w}}{K_V V_{dry\ gr}};$$
$$w_{\Sigma,\,\mathrm{abs}}=\frac{M_m}{K_f V_{dry\ gr}}=\frac{M_{ice}+M_{nonfr.w}}{K_f V_{dry\ gr}}.$$

All indexes of moisture content in frozen grounds $W_{\Sigma,m}$, $W_{\Sigma,vol}$, $w_{\Sigma,m}$, $w_{\Sigma,vol}$, $W_{\Sigma,\text{abs}}$ and $w_{\Sigma,\text{abs}}$ are banded by pairs through parameters of ground condition and density properties of ground and moisture components. This makes it possible to determine other indexes, knowing some of them, and in some cases assess the condition parameters of frozen ground and density properties. However, even for one pair of indexes, the types of relation can differ. This is determined by the approach to determining them. This will be shown below for a number of cases. It is not difficult to find these relations, however, it is important to clearly understand the essence of the indexes used. Relations between $W_{\Sigma,m}$ and $w_{\Sigma,m}$ have the same appearance as those between total moisture W_m and total moisture content w_m in wet ground:

$$W_{\Sigma,m} = \frac{M_m}{M_{fr.\,gr}} = \frac{M_m}{M_{dry\;gr} + M_m} = \frac{M_m / M_{dry\;gr}}{\dfrac{M_{dry\;gr}}{M_{dry\;gr}} + \dfrac{M_m}{M_{dry\;gr}}} = \frac{w_{\Sigma,m}}{1 + w_{\Sigma,m}}.$$

The relation between $W_{\Sigma,m}$ and $W_{\Sigma,vol}$ has a more complicated appearance:

$$\frac{W_{\Sigma,m}}{W_{\Sigma,vol}} = \frac{M_m}{M_{fr.gr}} : \frac{V_m}{V_{fr.gr}} = \frac{M_m}{V_m} \cdot \frac{V_{fr.gr}}{M_{fr.gr}} = \rho_m \frac{K_f V_{dry\,gr}}{M_{dry\,gr} + M_m} = \rho_m \frac{K_V}{\dfrac{M_{dry\,gr}}{V_{dry\,gr}} + \dfrac{M_m}{V_{dry\,gr}}} =$$

$$= \rho_m \frac{K_V}{\rho_{vol} + \dfrac{\rho_{vol} M_m}{\rho_{vol} V_{dry\,gr}}} = \rho_m \frac{K_V}{\rho_{vol} + \dfrac{\rho_{vol} M_m}{M_{dry\,gr}}} = \rho_m \frac{K_V}{\rho_{vol} + \rho_{vol} w_{\Sigma,m}}. \quad (3)$$

This relation may be also obtained in another way. For example:

$$\frac{W_{\Sigma,\text{м}}}{W_{\Sigma,\text{об}}} = \rho_m \frac{K_V V_{dry\,gr}}{M_{dry\,gr} + M_m} = \rho_m \frac{K_V}{\dfrac{M_{dry\,gr}}{V_{dry\,gr}} + \dfrac{K_f V_m}{K_f V_{dry\,gr}}} = \rho_m \frac{K_V}{\rho_{vol} + K_f w_{\Sigma,\text{abs}}}. \quad (4)$$

where ρ_m is moisture density in frozen ground, kg/m^3; ρ_{vol} is volumetric density, kg/m^3, of ground defined in accordance with the recommendations in the paper [10], $\rho_{vol} = M_{dry\,gr}/V_{dry\,gr} = \rho\left(1 - P_{v.d}\right)$; ρ – density, kg/m^3, of ground, $\rho = M_{dry\,gr}/V_{min.m} = M_{min.m}/V_{min.m}$, $V_{min.m}$ – volume of mineral matrix of ground, m^3; $P_{v.d} = V_{cav}/V_{dry\,gr}$ – index of total porosity, unit fraction; V_{cav} – total volume of all cavities in dry undisturbed (unripped) ground, m^3.

Moisture in frozen ground is a two-component substance consisting of unfrozen water and ice. Taking into account calculation methods of many-component substances density, we get

$$\rho_m = \rho_{ice} \frac{I_{\Sigma,v,ice}}{I_{\Sigma,v,ice} + W_{v,nonfr.w}} + \rho_w \frac{W_{v,nonfr.w}}{I_{\Sigma,v,ice} + W_{v,nonfr.w}} = \frac{1}{\dfrac{I_{\Sigma,m,ice}}{\left(I_{\Sigma,m,ice} + W_{m,nonfr.w}\right)\rho_{ice}} + \dfrac{W_{m,nonfr.w}}{\left(I_{\Sigma,m,ice} + W_{m\,nonfr.w}\right)\rho_w}}, \quad (5)$$

where $I_{\Sigma,m,ice} = I_{ice}/V_{fr.gr}$, $W_{v,nonfr.w} = V_{nonfr.w}/V_{fr.gr}$ is the total volumetric ice content of frozen ground, unit fraction, and its volumetric moisture due to unfrozen water, unit fraction; $I_{\Sigma,v,ice} = M_{ice}/M_{fr.gr}$, $W_{m,nonfr.w} = M_{nonfr.w}/M_{fr.gr}$ is the total mass ice content of frozen ground, unit fraction; and its moisture due to unfrozen water, unit fraction; ρ_{ice}, ρ_w – ice density, $\rho_{ice} \approx 917$ kg/m^3, and unfrozen water density, $\rho_w \approx 1000$ kg/m^3.

Taking into account values ρ_{ice}, ρ_w, let us present (5) as follows:

$$\rho_m = 917 \frac{I_{\Sigma,v,ice}}{I_{\Sigma,v,ice} + W_{v,nonfr.w}} +$$

$$+1000 \frac{W_{v,nonfr.w}}{I_{\Sigma,v,ice} + W_{v,nonfr.w}} \approx$$

$$\approx \frac{917 I_{\Sigma,v,ice} + 1000 W_{v,nonfr.w}}{I_{\Sigma,v,ice} + 1000 W_{v,nonfr.w}} \approx$$

$$\approx \frac{1}{\dfrac{0.0011 I_{\Sigma,m,ice}}{I_{\Sigma,m,ice} + W_{m,nonfr.w}} + \dfrac{0.001 W_{m,nonfr.w}}{I_{\Sigma,m,ice} + W_{m,nonfr.w}}}.$$

Let us check the obtained value expressions (3) and (4) using the dimension theory. Let's consider two cases. The first one is when dry ground is unripped, i.e. $K_f = 1.0$ and the second is when it is loosen, i.e. $K_f > 1.0$. So, in the first case we take into account that $K_V = \dfrac{\text{m}^3 \text{ fr. gr}}{\text{m}^3 \text{ dry gr}}$.

The dimensions of the left part of value expressions (3) and (4) are

$$\left[\frac{W_{\Sigma,m}}{W_{\Sigma,v}}\right] = \left[\frac{M_m V_{fr.gr}}{V_m M_{fr.gr}}\right] =$$

$$= \frac{\text{kg moisture} \cdot \text{m frozen ground}}{\text{m}^3 \text{ moisture} \cdot \text{kg frozen ground}},$$

and of the right part of (3)

$$\left[\rho_m \frac{K_V}{\rho_{об} + \rho_{об} w_{\Sigma,м}}\right] = \frac{\text{kg moisture}}{\text{m}^3 \text{ moisture}} \times \frac{\text{m}^3 \text{ fr. gr.}}{\text{m}^3 \text{ dry gr.}} \times$$

$$\times \frac{1}{\left(\dfrac{\text{kg dry gr.}}{\text{m}^3 \text{ dry gr.}} + \dfrac{\text{kg dry gr.}}{\text{m}^3 \text{ dry gr.}} \cdot \dfrac{\text{kg moisture}}{\text{kg dry gr.}}\right)} =$$

$$= \frac{\text{kg moisture} \cdot \text{m}^3 \text{ fr. gr.}}{\text{m}^3 \text{ moisture } \cdot \text{m}^3 \text{ dry gr.} \left(\dfrac{\text{kg dry gr.} + \text{kg moisture}}{\text{m}^3 \text{ dry gr.}}\right)} =$$

$$= \frac{\text{kg moisture} \cdot \text{m}^3 \text{ fr. gr.}}{\text{m}^3 \text{ moisture} \cdot \text{kg}^3 \text{ fr. gr.}}.$$

Comparing the dimensions of both parts of value expression (3), we can see that they are equal. Exactly the same conclusion is obtained by consideration of value expression (4).

Let us consider the second case for value expression (4). Dimensions of K_V and K_f and the left part of value expression (4) are:

$$[K_V] = \frac{\text{m}^3 \text{ loosen fr. gr.}}{\text{m}^3 \text{ dry gr.}};$$

$$\left[\frac{W_{\Sigma,m}}{W_{\Sigma,vol}}\right] = \left[\frac{M_m V_{\text{loosen fr. gr}}}{V_m \text{M}_{\text{fr. gr}}}\right] =$$

$$= \frac{\text{kg moisture} \cdot \text{m}^3 \text{loosen fr. gr}}{\text{m}^3 \text{moisture} \cdot \text{kg fr. gr}},$$

where $V_{\text{loosen fr. gr}}$ is the volume of loosen frozen ground, m^3.

When analyzing the left part dimensions of value expressions (3) and (4), it is considered that in the second case the same total mass moisture of loosen and unripped frozen grounds mass of loosen frozen ground equals to a mass of unripped frozen ground, since cavities between separate sections of frozen ground as a result of its fragmentation at $W_{\Sigma,m} = \text{const}$ mass of the ground is not changed. However, one should also take into account that $V_{\text{loosen fr. gr}} \neq V_{\text{fr. gr}}$.

For the right part of (4) we have:

$$[\rho_{вл} K_V] = \frac{\text{kg moisture}}{\text{m}^3 \text{ moisture}} \cdot \frac{\text{m}^3 \text{ loosen fr. gr}}{\text{m}^3 \text{ dry gr}};$$

$$\left[\rho_{vol} + K_f w_{\Sigma,м}\right] = \frac{\text{kg dry gr.}}{\text{m}^3 \text{ dry gr.}} + \frac{\text{m}^3 \cancel{\text{ dry loosen}} \text{ gr.}}{\text{m}^3 \text{ dry gr.}} \times$$

$$\times\frac{\text{kg moisture}}{\cancel{\text{m}^3\ \text{dry loosen}}\ \text{gr.}}=\frac{\text{kg dry gr.}}{\text{m}^3\ \text{dry gr.}}+\frac{\text{kg moisture}}{\text{m}^3\ \text{dry gr.}}=$$

$$=\frac{\text{kg fr. gr.}}{\text{m}^3\ \text{dry gr.}}.$$

Finally:

$$\left[\rho_m\frac{K_V}{\rho_{vol}+K_f w_{\Sigma,\,m}}\right]=\rho_m K_V:\left[\rho_{vol}+K_f w_{\Sigma,\,m}\right]=$$

$$=\frac{\text{kg moisture}\cdot\text{m}^3\ \text{loosen fr. gr.}}{\text{m}^3\ \text{moisture}\cdot\text{m}^3\ \text{dry gr.}}:\frac{\text{kg fr. gr.}}{\text{m}^3\ \text{dry gr.}}=$$

$$=\frac{\text{kg moisture}\cdot\text{m}^3\ \text{loosen fr. gr.}}{\text{m}^3\ \text{moisture}\cdot\text{kg fr. gr.}}.$$

Therefore, the dimensions of the left and right parts of (4) are the same. Exactly the same result is obtained by consideration of value expression (3) for loosen frozen grounds.

The relations between $W_{\Sigma,m}$ and $w_{\Sigma,vol}$ have the same appearance as the relations of (3) and (4) between $W_{\Sigma,m}$ and $W_{\Sigma,vol}$:

$$\frac{W_{\Sigma,\,m}}{w_{\Sigma,\,vol}}=\frac{M_m}{M_{fr.\,gr}}:\frac{V_m}{K_f V_{dry\ gr}}=$$

$$=\frac{M_m}{V_m}\cdot\frac{K_f V_{dry\ gr}}{M_{fr.\,gr}}=\rho_m\frac{K_f V_{dry\ gr}}{M_{dry\ gr}+M_m}$$

$$=\rho_m\frac{K_f}{\dfrac{M_{dry\ gr}}{V_{dry\ gr}}+\dfrac{M_m}{V_{dry\ gr}}}=\rho_m\frac{\text{K}_\text{p}}{\rho_{vol}+\dfrac{\rho_{vol}M_m}{\rho_{vol}V_{dry\ gr}}}=$$

$$=\rho_m\frac{K_f}{\rho_{vol}+\dfrac{\rho_{vol}M_m}{M_{dry\ gr}}}=\rho_m\frac{K_f}{\rho_{vol}+\rho_{vol}w_{\Sigma,\,m}};$$

$$\frac{W_{\Sigma,m}}{w_{\Sigma,vol}}=\rho_m\frac{K_f}{\dfrac{\text{M}_{dry\,gr}}{V_{dry\,gr}}+\dfrac{\text{M}_m}{V_{dry\,gr}}}=$$

$$=\rho_m\frac{K_f}{\rho_{vol}+\dfrac{K_f\text{M}_{\text{вл}}}{K_f V_{dry\,gr}}}=\rho_m\frac{K_f}{\rho_{vol}+K_f w_{\Sigma,\text{abs}}}.$$

Let's find the relation between $W_{\Sigma,m}$ and $W_{\Sigma,\text{abs}}$:

$$\frac{W_{\Sigma,m}}{W_{\Sigma,\text{abs}}}=\frac{M_m}{M_{fr.\,gr}}:\frac{M_m}{V_{fr.\,gr}}=\frac{V_{fr.\,gr}}{M_{fr.\,gr}}=\frac{K_V V_{dry\,gr}}{\text{M}_{dry\,gr}+M_m}=$$

$$=\frac{K_V}{\dfrac{M_{dry\,gr}}{V_{dry\,gr}}+\dfrac{M_m}{V_{dry\,gr}}}=\frac{K_V}{\rho_{vol}+\dfrac{\rho_{vol}M_m}{\rho_{vol}V_{dry\,gr}}}=$$

$$=\frac{K_V}{\rho_{vol}+\dfrac{\rho_{vol}M_m}{M_{dry\,gr}}}=\frac{K_V}{\rho_{vol}\left(1+w_{\Sigma,\text{м}}\right)}. \qquad (6)$$

In unripped frozen grounds ($K_f=1.0$) the relation (6) has the following appearance:

$$\frac{W_{\Sigma,m}}{W_{\Sigma,\text{abs}}}=\frac{K_{sw}}{\rho_{vol}\left(1+w_{\Sigma,m}\right)}.$$

If we use volumetric density of frozen ground $\rho_{vol,fr.\,gr}$ and pour density of frozen ground $\rho_{pour,fr.\,gr}$, the relation between $W_{\Sigma,m}$ and $W_{\Sigma,\text{abs}}$ can be presented as follows:

in unripped frozen ground

$$\frac{W_{\Sigma,m}}{W_{\Sigma,\text{abs}}}=\frac{V_{fr.\,gr}}{M_{fr.\,gr}}=\frac{1}{\rho_{vol,\,fr.\,gr}},$$

in loosen ground

$$\frac{W_{\Sigma,m}}{W_{\Sigma,vol}}=\frac{V_{fr.\,gr}}{M_{fr.\,gr}}=\frac{1}{\rho_{pour,\,fr.\,gr}}.$$

One of the types of relation between $W_{\Sigma,m}$ and $W_{\Sigma,\text{abs}}$ can be obtained as follows:

$$\frac{W_{\Sigma,m}}{w_{\Sigma,\text{abs}}}=\frac{M_m}{\text{M}_{fr.\,gr}}:\frac{M_m}{K_f V_{dry\,gr}}=\frac{K_f V_{dry\,gr}}{\text{M}_{fr.\,gr}}=$$

$$=\frac{K_f V_{dry\,gr}}{\text{M}_{dry\,gr}+M_m}=\frac{K_f}{\dfrac{\text{M}_{dry\,gr}}{V_{dry\,gr}}+\dfrac{M_m}{V_{dry\,gr}}}=$$

$$=\frac{K_f}{\rho_{vol}+\dfrac{\rho_{vol}M_m}{\rho_{vol}V_{dry\,gr}}}=\frac{K_f}{\rho_{vol}+\dfrac{\rho_{vol}M_m}{\text{M}_{dry\,gr}}}=$$

$$= \frac{K_f}{\rho_{vol}\left(1 + w_{\Sigma,m}\right)}. \quad (7)$$

Using another approach to search for a relation between $W_{\Sigma,m}$ and $W_{\Sigma,\mathrm{abs}}$, we get:

$$\frac{W_{\Sigma,m}}{w_{\Sigma,\mathrm{abs}}} = \frac{M_m K_f V_{dry\ fr.}}{M_{fr.\ gr} M_m} = \frac{K_f V_{dry\ fr.}}{M_{fr.\ gr}} =$$

$$= \frac{K_f V_{dry\ fr.}}{M_{dry\ fr.} + M_m} = \frac{K_f}{\dfrac{M_{dry\ fr.}}{V_{dry\ fr.}} + \dfrac{M_m}{V_{dry\ fr.}}} =$$

$$= \frac{K_f}{\rho_{vol} + \dfrac{K_f M_m}{K_f V_{dry\ fr.}}} = \frac{K_f}{\rho_{vol} + K_f w_{\Sigma,\text{аб}}};\rightarrow$$

$$\rightarrow \frac{W_{\Sigma,m}}{w_{\Sigma,\mathrm{abs}}} = \frac{K_f}{\rho_{vol} + K_f w_{\Sigma,\text{аб}}};\rightarrow$$

$$\rightarrow W_{\Sigma,m} = \frac{w_{\Sigma,\mathrm{abs}} K_f}{\rho_{vol} + K_f w_{\Sigma,\mathrm{abs}}}. \quad (8)$$

In unripped frozen grounds ($K_f = 1.0$) both relations (7) and (8) between $W_{\Sigma,m}$ and $W_{\Sigma,\mathrm{abs}}$ have the following appearance:

$$\frac{W_{\Sigma,m}}{w_{\Sigma,\mathrm{abs}}} = \frac{1}{\rho_{\text{об}}\left(1 + w_{\Sigma,m}\right)};$$

and

$$W_{\Sigma,m} = \frac{w_{\Sigma,\mathrm{abs}}}{\rho_{vol} + w_{\Sigma,\mathrm{abs}}}.$$

By analogy, let us find consistent relations between other total indexes of moisture content in frozen ground that were not considered above $W_{\Sigma,vol}$ and $W_{\Sigma,m}$, $W_{\Sigma,vol}$ and $w_{\Sigma,vol}$, $W_{\Sigma,vol}$ and $W_{\Sigma,\mathrm{abs}}$, $W_{\Sigma,vol}$ and $w_{\Sigma,\mathrm{abs}}$, $W_{\Sigma,\mathrm{abs}}$ and $w_{\Sigma,vol}$, $W_{\Sigma,\mathrm{abs}}$ and $w_{\Sigma,\mathrm{abs}}$, $w_{\Sigma,m}$ and $w_{\Sigma,vol}$, $w_{\Sigma,m}$ and $w_{\Sigma,\mathrm{abs}}$, $w_{\Sigma,vol}$ and $w_{\Sigma,\mathrm{abs}}$:

$$\frac{W_{\Sigma,vol}}{w_{\Sigma,m}} = \frac{V_m}{V_{fr.\ gr}} : \frac{M_m}{\mathrm{M}_{dry\ gr}} = \frac{V_m \mathrm{M}_{dry\ gr}}{V_{fr.\ gr}\mathrm{M}_{\text{вл}}} =$$

$$= \frac{V_m}{M_m} \times \frac{\mathrm{M}_{dry\ gr}}{\mathrm{K_v} V_{dry\ gr}} = \frac{1}{\rho_m} \times \frac{\rho_{vol}}{\mathrm{K}_V} = \frac{\rho_{vol}}{\mathrm{K}_V \rho_m},$$

$$\frac{W_{\Sigma,vol}}{w_{\Sigma,vol}} = \frac{V_m}{V_{fr.\ gr}} : \frac{V_m}{K_f V_{dry\ gr}} =$$

$$= \frac{V_m}{K_V V_{dry\ gr}} : \frac{V_m}{K_f V_{dry\ gr}} = \frac{K_f}{\mathrm{K}_V}, \quad (9)$$

if frozen ground is not loosen, i.e. $K_f = 1.0$, then in this case (9) transforms into (10):

$$\frac{W_{\Sigma,vol}}{w_{\Sigma,vol}} = \frac{1}{K_{sw}}; \quad (10)$$

$$\frac{W_{\Sigma,vol}}{W_{\Sigma,\mathrm{abs}}} = \frac{V_m}{V_{fr.\ gr}} : \frac{\mathrm{M}_m}{V_{fr.\ gr}} = \frac{V_m}{\mathrm{M}_m} = \frac{1}{\rho_m};$$

$$\frac{W_{\Sigma,vol}}{w_{\Sigma,\mathrm{abs}}} = \frac{V_m}{V_{fr.\ gr}} : \frac{M_m}{K_f V_{dry\ gr}} =$$

$$= \frac{V_m}{M_m} \times \frac{K_f V_{dry\ gr}}{K_V V_{dry\ gr}} = \frac{K_f}{K_V \rho_m};$$

$$\frac{W_{\Sigma,\mathrm{abs}}}{w_{\Sigma,vol}} = \frac{M_m}{V_{fr.\ gr}} : \frac{V_m}{K_f V_{dry\ gr}} =$$

$$= \frac{M_m}{V_m} \times \frac{K_f V_{dry\ gr}}{K_V V_{dry\ gr}} = \rho_m \frac{K_f}{K_V};$$

$$\frac{W_{\Sigma,\mathrm{abs}}}{w_{\Sigma,\mathrm{abs}}} = \frac{M_m}{K_V V_{\text{с.пор}}} : \frac{M_m}{K_f V_{\text{с.пор}}} = \frac{K_f}{\mathrm{K}_V};$$

$$\frac{w_{\Sigma,m}}{w_{\Sigma,vol}} = \frac{M_m}{M_{dry\ gr}} : \frac{V_m}{K_f V_{\text{с.пор}}} =$$

$$= \frac{M_m}{V_m} \times \frac{K_f V_{dry\ gr}}{M_{dry\ gr}} = \rho_m \frac{K_f}{\rho_{\text{об}}};$$

$$\frac{w_{\Sigma,m}}{w_{\Sigma,\mathrm{abs}}} = \frac{M_m}{M_{dry\ gr}} : \frac{M_m}{K_f V_{dry\ gr}} = \frac{K_f V_{dry\ gr}}{M_{dry\ gr}} = \frac{K_f}{\rho_{vol}};$$

$$\frac{w_{\Sigma,vol}}{w_{\Sigma,\mathrm{abs}}} = \frac{V_m}{K_f V_{dry\ gr}} : \frac{M_m}{K_f V_{dry\ gr}} = \frac{V_m}{M_m} = \frac{1}{\rho_m}.$$

The performed research allowed us to determine for the first time the type of relation between indexes of ice and unfrozen water content in frozen ground and the total moisture indexes in these grounds. It has been shown that, in a number of cases, the relation between the same indexes can differ, which is caused by their determination method. Using the obtained indexes in practice makes it possible to reduce the number of laboratory studies for determining the relations between other corresponding indexes of physical properties of frozen ground and total moisture content indexes and certain components in this ground.

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“Gornye nauki i tehnologii”/ “Mining science and technology”, 2016, No. 1, pp. 25-31	
Title:	**The relation between total moisture content indexes in frozen rocks and soil**
Author 1	Name&Surname: **Gennady A. Yanchenko** Company: **National University of Science and Technology NUST MISiS** Work Position: **Professor**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-25-32**
Abstract:	Moisture in frozen rocks and soil is generally represented as two components – ice and unfrozen water. To quantify the moisture content in these rocks, two indexes called total and general moisture content were earlier proposed. The paper shows that such names do not properly reflect these indexes. In fact, it is the total mass moisture and water content. For a more complete and more accurate assessment of the impact of moisture on the physical properties of frozen rocks and soil, the paper proposes using total volumetric and absolute moisture and water content. The relation between total moisture indexes in frozen rocks and soil considered in the paper was identified.
Keywords:	frozen rock, total moisture, total water content, total absolute moisture, total absolute water content, the relation between them.
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ABOUT THE HALF-WAVE LENGTH OF THE BOTTOM-HOLE CORE-DRILL COMPOSED OF STRUCTURAL ELEMENTS OF DIFFERENT STIFFNESS

In the formation of the well under the influence of axial load and rotation speed, the core drilling shell is curved, while its lower bottom-hole part consists of elements of different stiffness: core retrieving barrel and drilling pipe. To determine the half-wave length in this part of the process tools, the energy method was used, according to which, all external forces on the said length changes into potential energy of the curved drill string. The solution of the corresponding equation helped to obtain a formula for determining the length of the half-wave of the string bottom-hole area. The obtained relationship is different from the well-known formula of B.I. Vozdvizhensky and M.G. Vasilyev in that it additionally takes into consideration the ratio of the core retrieving barrel and half-wave, ratio of moments of inertia of the core retrieving barrel and drilling pipes, and the mass ratio of core barrel and drilling pipes per unit length. To reduce the well deviation, it is recommended to install rib stiffeners in the "half-wave crests" of the bottom-hole part of the drilling string.

Keywords: deviation, drilling line, rib stiffeners, half wave length, half-wave crests.

It is common knowledge that during well drilling the tool string under the influence of axial load and torsion torque becomes unstable and assumes a wave shape. As a consequence, additional conditions are created promoting deviation of the well and its departure from the design direction.

Any half-wave length in the compressed part of the tool string is calculated according to G.M. Sarkisov's formula [1, 2]:

$$l = \frac{9500}{n}\sqrt{\pm z + \sqrt{0.25z^2 + \frac{1.1jn^2}{100q}}}, \qquad (1)$$

where l – is the half-wave length, cm

n – is the string rotation speed, rpm;

z – is the distance from the zero section, where the compression of the lower part of the tool string conditional on the bottom-hole reaction proceeds to tensile, cm;

j – is the axial moment of inertia of the tool string cross-section; cm^4;

q – is the weight of 1 cm of drill pipes that form the tool string, kg/cm.

Equation (1) is valid if the length of the tool string half-wave, first from the bottom-hole, consists of the same rigidity element (for example, the drill collars for rotary drilling).

When core drilling, the said half-wave consists in general of two structural elements: core retrieving barrel and drilling pipe (Fig. 1).

In *A*, *B*, *C* points and other points the drill string touches the wellbore walls. The direction of the well is greatly influenced by first contact point of the drill guide on the borehole wall (point *A* in Fig. 1). The closer point *A* is to the bottom-hole, the more the well route will deviate from the projected route (along arrow *B* in Fig. 1). To move point *A* away from the bottom-hole to a greater distance, the stiffness of the tool string needs to be increased, which will contribute to a smaller wellbore deviation.

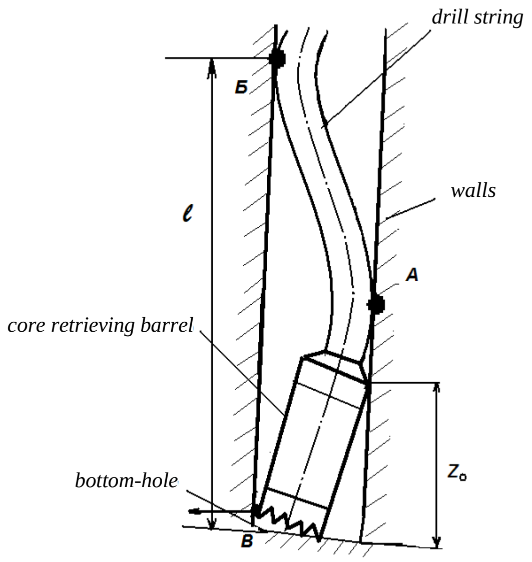

Fig. 1. Deviation of the bottom-hole part of the tool string under the influence of external loads.

A radical solution for increasing the stiffness of the tool string is to install centralizers at points *A* and *B* whose diameter will be close to the well diameter.

The diameters of drill pipes and core barrels, which the bottom-hole section of tool string consists of, are standardized. The following diameters are used in the coring process, mm: 42, 50, 63 (drill pipes), 46, 57, 73, 89, 108, 127 (core barrels). Therefore, the length of the half-wave of the tool string, located first from the bottom-hole, is generally equal to the sum of the lengths of two components with different diameters: core retrieving barrel and drilling pipe. The model of the bottom-hole part of the tool string represents a shaft with two stiffened areas (Fig. 2), where the bottom area shapes the core retrieving barrel, and the top one shapes the drill string.

The task of calculating the said length l (see Fig. 1) is solved using the energy method, according to which the whole work of external forces at the half-wave length (centrifugal and axial load) transforms into potential energy of the curved shaft (of the tool string) at the same length:

$V = A_C + A_1$ (V – potential energy of the curved shaft; A_C, A_1 – work of centrifugal and longitudinal (axial load) forces respectively).

First, we calculate the potential energy of the shaft:

$$V = V_1 + V_2 . \qquad (2)$$

Potential energy of the upper I (V_1) and lower II (V_2) shafts equals:

$$V_1 = \frac{EJ_1}{2}\int_{£}^{1}\left(y^{II}\right)^2 \mathrm{d}x;\; V_2 = \frac{EJ_2}{2}\int_{0}^{£}\left(y^{II}\right)^2 \mathrm{d}x, \qquad (3)$$

where E – is the elasticity modulus of the shaft (tool string), for steel E=2·10^5 MPa;

J_1, J_2 – is the moment of inertia of the drill pipe and the core barrel cross sections,

y – is the curve function along which the tool string bent, $y = e\sin\frac{\pi x}{l}$ (see fig. 2) (e – is the deflection of the core retrieving barrel, equal

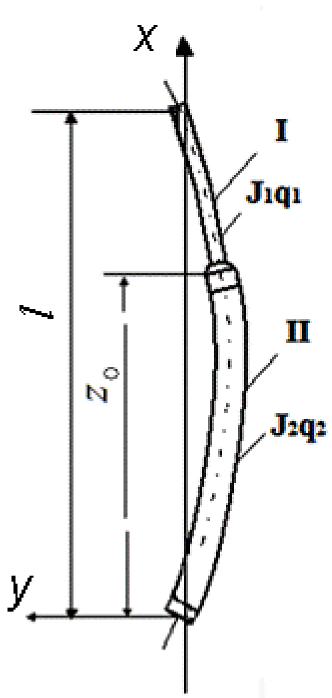

Fig. 2. Model of the bottom-hole part of the tool string: I – *drill string;* II – *core retrieving barrel.*

to the clearance between the well and core barrel, l – is the half-wave length of the tool string).

Having calculated the second derivative of the function (y) and applying it to the formula V_1 and V_2, denoting by $z_0 / l = \varphi$, we obtain

$$V_1 = \frac{EJ_1 e^2\pi^4}{4l}\left(1 - \varphi + \frac{\sin 2\pi\varphi}{2\pi}\right), \qquad (4)$$
$$V_2 = \frac{EJ_2 e^2\pi^4}{4l}\left(\varphi - \frac{\sin 2\pi\varphi}{2\pi}\right).$$

Applying (4) to (2), we will obtain the full value of the potential energy of the shaft:

$$V = \frac{EJ_1 e^2\pi^4}{4l^3}\left[1 + (K-1)\left(\varphi - \frac{\sin 2\pi 4}{2\pi}\right)\right], \qquad (5)$$

where $K = J_2 + J_1$.

We calculate the work of external transverse (centrifugal) forces, whose value is equal to

$$A_c = A_1 + A_2 \qquad (6)$$

The value of works of the centrifugal forces on upper A_1 and lower A_2 sections of the shaft is equal to:

$$A_1 = \frac{q_1\omega^2}{2g}\int_{£}^{1} y^2 \mathrm{d}x = \frac{q_1\omega^2}{2g}\int_{£}^{1} e^2 \sin^2\frac{\pi x}{l}\mathrm{d}x =$$

$$= \frac{q_1\omega^2 e^2 l}{4g}\left(1-\varphi+\frac{\sin 2\pi\varphi}{2\pi}\right), \quad (7)$$

$$A_2 = \frac{q_2\omega^2}{2g}\int_0^{£} y^2 dx = \frac{q_2\omega^2}{2g}\int_0^{£} e^2\sin^2\frac{\pi x}{l}dx = ,$$

$$= \frac{q_2\omega^2 e^2 l}{4g}\left(\varphi-\frac{\sin 2\pi\varphi}{2\pi}\right). \quad (8)$$

where q_1, q_2 is the mass per 1 m of drill pipes and core barrels respectively:

ω is the angular spin rate of the tool string;

g is the gravity acceleration equal to 9.81m/s^2.

The total work of the centrifugal forces accounting for (7) and (8) is equal to

$$A_C = A_1 + A_2 = \frac{q_1\omega^2 e^2 l}{4g}\times$$

$$\times\left[1+(m-1)\left(\varphi-\frac{\sin 2\pi\varphi}{2\pi}\right)\right], \quad (9)$$

where $m = q_2 / q_1$.

The work of longitudinal force (axial load C) at half-wave length is equal to:

$$A_l = \frac{C}{2}\int_0^1 \left(y^{II}\right)^2 dx = \frac{Ce^2\pi}{4l}, \quad (10)$$

The load on the bottom-hole C, exerted by the part of tool string mass, is equal to: $C = \alpha q_1 z$, (α – is the adjustment ratio to take into account the additional weight of drill pipe links; z – is the length of the compressed part of the tool string).

Since $V = A_C + A_l$, by substituting values of V, A_C, A_l, we obtain from (5), (9) and (10)

$$\frac{DEJ_1 e^2\pi^4}{4l^3} = \frac{Bq_1\omega^2 e^2 l}{4g} + \frac{Ce^2\pi}{4l}. \quad (11)$$

where $D = 1+(K-1)\left(\varphi-\frac{\sin 2\pi\varphi}{2\pi}\right)$;

$$B = 1+(m-1)\left(\varphi-\frac{\sin 2\pi\varphi}{2\pi}\right).$$

Consequently, to determine the half-wave length, we should solve the quadratic equation

$$Bq_1\omega^2 l^4 + Cg\pi^2 l^2 - DEJ_1\pi^4 g = 0. \quad (12)$$

Solving the equation (12), after algebraic transformations, we obtain

$$l = \frac{\gamma_0}{\psi_0 n}\sqrt{-z+\sqrt{z^2+(\psi_0 n)^2}}, \quad (13)$$

where $\gamma_0 = 6,28\sqrt{\frac{J_1}{\alpha q_1 D}}$;

$$\psi_0 = 0,0942\sqrt{\frac{J_0}{\alpha^2 q_1}}DB;$$

n is the toll string rotation speed, rpm.

We then analyze formula (13) depending on the value of ratio $\varphi = \frac{z_0}{l}$. If $\varphi = 0$, then $D = \mathrm{I}$; $B = \mathrm{I}$. Coefficients γ_0 and ψ_0 are respectively equal to $\gamma_0 = 6,28\sqrt{\frac{J_1}{\alpha q_1}}$; $\psi_0 = 0,0942\sqrt{\frac{J_1}{\alpha^2 q_1}}$. We get the first special case: formula of B.I. Vozdvizhensky and M.G. Vasilyev for determining the half-wave of the tool string, composed of drill pipes. If $\varphi = 1$, then $D = K = \frac{J_2}{J_1}$; $B = m = \frac{q_2}{q_1}$. Coefficients γ_0 and ψ_0 are equal to $\gamma_0 = 6,28\sqrt{\frac{J_2}{\alpha^2 q_1}}$; $\psi_0 = 0,0942\sqrt{\frac{J_2}{\alpha^2 q_1}}\cdot\sqrt{\frac{q_2}{q_1}}$. In this case, formula (13) will take the following form:

$$l = \sqrt{\frac{q_1}{q_2}}\cdot\frac{\gamma_0}{\varphi_0 n}\sqrt{-z+\sqrt{z^2+\frac{q_2}{q_1}(\varphi_0 n)^2}}. \quad (14)$$

Formula (14) expresses the second special case, where the length of the core retrieving barrel is equal to or greater than the length of the first half-wave from the bottom-hole, and the load to the bottom-hole is exerted by the weight of drill pipes.

Therefore, the length of the first half-wave from the bottom-hole in general, when the drill at the site consists of different diameter components, depends not only on the drilling conditions and rigidity of drill pipes, but also on the ratio of lengths of the drill pipe and core barrel constituting the half wave ($\varphi = \frac{z_0}{l}$); the ratio of momenta of inertia of the drill pipe and core barrel cross sections ($K = \frac{J_2}{J_1}$), as well as ratio of masses of drill pipe and core barrel per unit length ($m = \frac{q_2}{q_1}$).

Generally, bending the tool string will be influenced by the axial load *C*, centrifugal forces arising due to rotational speed and weight of the tool string, as well as the pressure of flushing fluid in the well.

Fig. 3, *a–g* shows the dependence of the length of the first half-wave of the tool string from the bottom-hole *l* on the axial load *C* exerted onto bottom-hole *n* at different values of rotational speed of the drill string. Solid lines represent half-wave lengths without accounting for flushing fluid, dashed lines account for this. As it follows from the graphs, flushing fluid, which fills the well, slightly increases the half-wave of the tool string, as if straightening the latter, but this effect is negligible (half-wave length increases by 3–5 %).

The conducted studies allow us to determine the location for installing the centralizers on the body of core retrieving barrel and the drill string. The first place is located at a distance equal to ½ half-wave, the second one at a distance equal to a half-wave length (Fig. 1). In addition, information on drilling mode parameters should be obtained.

As an example, Fig. 4 shows the tool string with coring consisting of tungsten carbide drill bits with a 76-mm diameter, a core barrel with 73-mm diameter and length of 3.5 m, and a drill string of 50-mm diameter. It is proposed to drill under axial load $C = 5$ kH and rotation speed n = 390 rpm. While calculating according to the formula (13), the length of the tool string's first half-wave from the bottom-hole was equal to $l_1 = 5.1$ m. Therefore, centralizers will be installed at: $5.1 : 2 = 2.55$ m and 5.1 m.

At the point where first crest of the half-wave l_{c1} is located, we place four carbide welds along the circumference of the core barrel. The weld length should be taken at 20 cm to compensate for the possible displacements of half-wave crest when the drilling mode is changed. At the point of the second crest of the half-wave l_{c2} = 5.1 m, we install rubber ring-protectors with longitudinal channels for passage of the drilling fluid. The length of the second centralizer protector is taken at 30–40 cm, based on the experience of exploration drilling, when such a measure helped to prevent wear of the drill pipes.

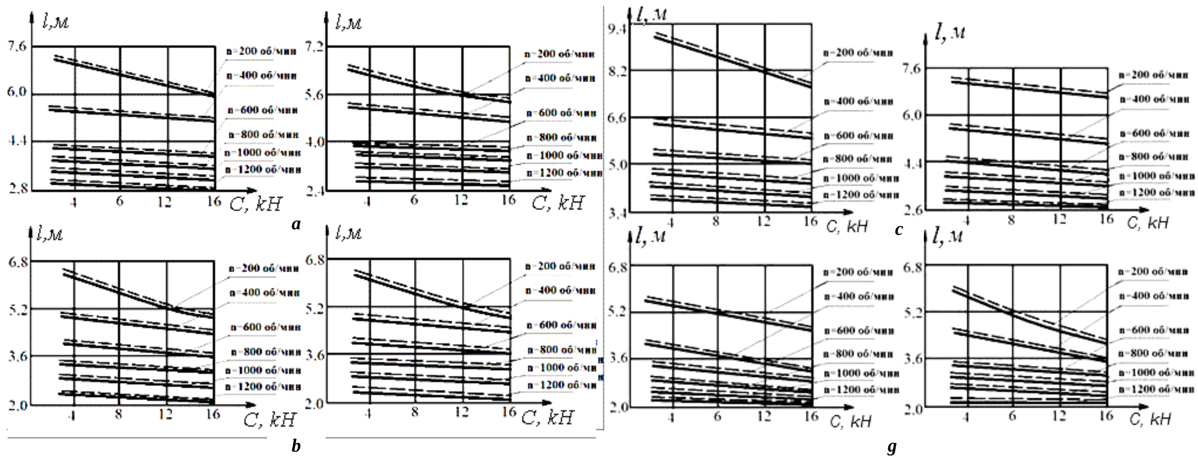

Fig. 3. Dependence of the half-wave length *l* on axial load *C* at different values of *n*.

Diameter of drill pipes: ***a***, ***c***, ***d*** – 50 mm; ***b*** – 42 mm; core barrels: ***a*** – 73 mm; ***b***, ***g*** – 57 mm; ***c*** – 89 mm; _____ – half-wave length without accounting for hydraulic forces; - - - – half-wave length accounting for hydraulic forces.

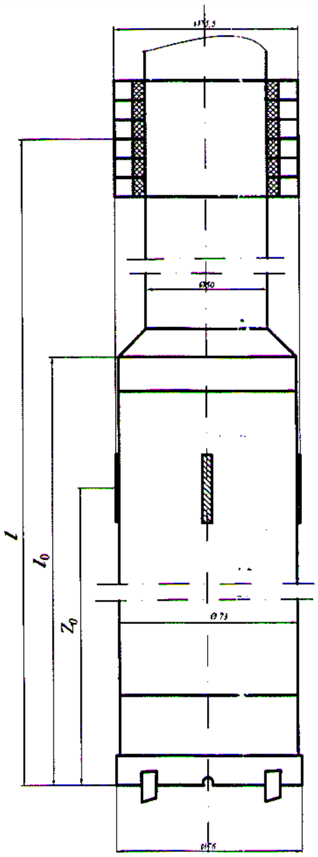

Fig. 4. Stiffened tool string.

The performed studies helped to formulate a number of conclusions.

The half-wave length *l* decreases with the increasing axial load C and decreasing drilling diameter (reduction in the rigidity of the core retrieving barrel).

For a tool string of a certain size, the half-wave length decreases with higher intensity as the number of revolutions of the drill column *n* increases than when the axial load *C* increases.

The resulting formula for determining the first half-wave length of the tool string from the bottom-hole, consisting of components of different stiffness, allows us to determine the location for installing centralizers to increase the rigidity of the bottom-hole part of the tool and contribute to smaller well deviation.

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“Gornye nauki i tehnologii”/ “Mining science and technology”, 2016, No. 1, pp. 33-37	
Title:	**About the half-wave length of the bottom-hole core-drill composed of structural elements of different stiffness**
Author 1	Name&Surname: **Fedorov B.V.** Company: **Kazakh National Research Technical University named after K.I. Satpaev** Work Position: **Professor** Contacts: **kuzinevgen@gmail.com**
Author 2	Name&Surname: **Korgasbekov D.R.** Company: **Kazakh National Research Technical University named after K.I. Satpaev** Work Position: **Lecturer** Contacts: **daha_korg@mail.ru**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-33-38**
Abstract:	In the formation of the well under the influence of axial load and rotation speed, the

	core drilling shell is curved, while its lower bottom-hole part consists of elements of different stiffness: core retrieving barrel and drilling pipe. To determine the half-wave length in this part of the process tools, the energy method was used, according to which, all external forces on the said length changes into potential energy of the curved drill string. The solution of the corresponding equation helped to obtain a formula for determining the length of the half-wave of the string bottom-hole area. The obtained relationship is different from the well-known formula of B.I. Vozdvizhensky and M.G. Vasilyev in that it additionally takes into consideration the ratio of the core retrieving barrel and half-wave, ratio of moments of inertia of the core retrieving barrel and drilling pipes, and the mass ratio of core barrel and drilling pipes per unit length. To reduce the well deviation, it is recommended to install rib stiffeners in the "half-wave crests" of the bottom-hole part of the drilling string.
Keywords:	deviation, drilling line, rib stiffeners, half wave length, half-wave crests.
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STATUS OF DEVELOPMENT ORIENTATIONS FOR MINING TITANIUM PLACERS IN VIETNAM

Vietnam has potential reserves of titanium placers, located mainly along the coast of the Middle spreading across provinces from Thanh Hoa to Ba Ria – Vung Tau. Mining titanium placers deposits has gained considerable profit for localities and the country as a whole, but it has also given rise to significant environmental issues.

The paper introduces the potential of titanium placers in Vietnam, current mining status and environmental issues faced, and then proposes a number of strategic orientations for sustainable development of the future titanium industry of Vietnam.

These orientations are completely feasible with regard to the titanium placer mines of Vietnam in order to consider and apply effectively for suitable conditions. They also represent a useful reference for similar titanium placer mines in the world.

Keywords: Vietnam, titanium placers, marine deposit, mining and processing technologies.

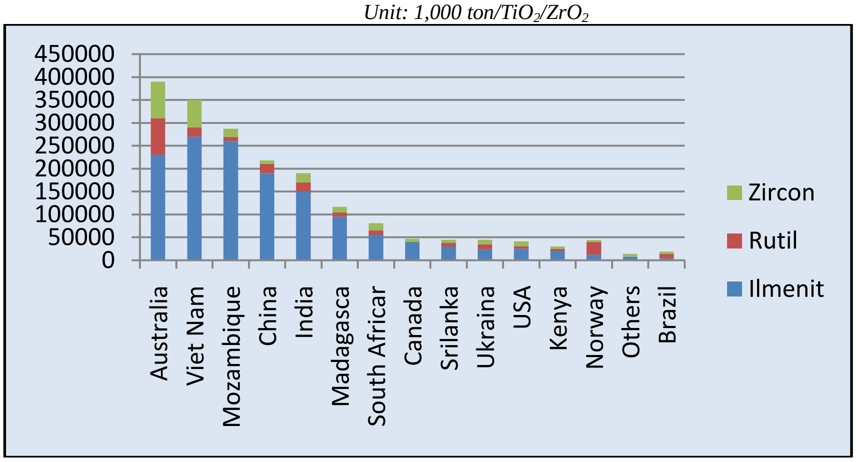
1. Potential of titanium placers in Vietnam

Sea-beach titanium placers are a mixture of heavy minerals such as ilmenite, rutile, zircon, monazite, sillimanite, garne, etc. The ratio of heavy minerals in ore is different from area to area because of their formations and economic values.

In Vietnam, titanium placers are discovered along the coast of the Middle, spreading across provinces from Thanh Hoa to Ba Ria – Vung Tau, concentrated at areas Ky Anh, Cam Xuyen (Ha Tinh province), Vinh Thai (Quang Tri province), Ke Sung, Vinh My (Thua Thien – Hue province), My Thanh – Phu My, De Gi – Phu Cat (Binh Dinh province), Ham Tan, Mui Ne (Binh Thuan province) – Figure 1. Total titanium resources and reserves of Vietnam (calculated until 2008) is about 100 million tons.

Since 2008, the Ministry of Natural Resources and Environment of Vietnam has been implementing an exploration project of titanium placers in the red sand formation in Ninh Thuan, Binh Thuan and North of Ba Ria – Vung Tau over an area covering 1,460 km2. The research result over an area of 760 km2 out of the total of 1,460 km2 estimated the titanium resources and reserves at about 300 million tons. Therefore, over the total study area of 1,460 km2, it is estimated that there may be up to 500 million tons of potential titanium placers, increasing the total titanium – zircon resources of Vietnam to over 600 million tons.

Unit: 1,000 ton/TiO_2/ZrO_2



[correct the corresponding countries as follows: Vietnam, Madagascar, South Africa, Sri Lanka, Ukraine]
[on right-hand side, change "Rutil" to "Rutile" and "Ilmenit" to "Ilmenite"]

Fig. 1: Titanium mineral reserves of some countries of the world (*Source: TZMI, USGS*)

Source: map from Worldofmaps.net

Fig. 2: Provinces with titanium placers in Vietnam
(Thanh Hoa, Nghe An, Ha Tinh, Quang Binh, Quang Tri, Thua Thien – Hue, Quang Nam, Quang Ngai, Binh Dinh, Phu Yen, Khanh Hoa, Ninh Thuan, Binh Thuan, Ba Ria – Vung Tau)

Since 2008, the Ministry of Natural Resources and Environment of Vietnam has been implementing an exploration project of titanium placers in the red sand formation in Ninh Thuan, Binh Thuan and North of Ba Ria – Vung Tau over an area covering 1,460 km2. The research result over an area of 760 km2 out of the total of 1,460 km2 estimated the titanium resources and reserves at about 300 million tons. Therefore, over the total study area of 1,460 km2, it is estimated that there may be up to 500 million tons of potential titanium placers, increasing the total titanium – zircon resources of Vietnam to over 600 million tons.

Sea-beach titanium placers in the Middle of Vietnam are similar to other titanium placer deposits in the world. They are synthetic placers, deposited by minerals with a density of 4.3–5.2 and grain size of 0.074–0.25 mm. There are many useful minerals, but including 4 main ones: ilmenite, rutile, zircon, and monazite, where ilmenite occupies 53.09–91.66%, zircon 3.56–18.45%, rutile 0.69–2.65%. The concentrating ore quality of the titanium placers in the Middle of Vietnam meets all requirements for export.

According to the prospecting and exploration results on the titanium placers along the coast of Middle Vietnam, synthetic titanium placers are mainly located in the marine deposit and formed during the Late Pleistocene of the Quaternary period.

Deposits of titanium placers in the Middle of Vietnam explored before 2005 with the cut-off grade of ≥ 10 kg/m3 have shown: length of these deposits is from several meters to 20 km, width of 25–700 m, and thickness of 0.5–10 m.

There are two parts in the deposits of titanium placers in the Middle of Vietnam: non-ore component is about 80.2–99.7%, and ore component is 0.3–18.8%.

– Non-ore component is mainly quartz (83.1–96.9%), sand (2.3–15.9%), and other minerals such as turmaline and distene (0.2–3.2%).

– Ore component accounts for 0.3–5.2%, somewhere over 10%, where content of titanium and zircon is more than 98%; other minerals such as monazite, xenotime, granate are about 0.01–2%. Minerals containing titanium comprise:

+ Ilmenite ($FeTiO_3$): is the main ore mineral with content from several kg/m3 to several hundred kg/m3, accounting for 50–90% of total useful heavy minerals. This mineral is black or grey black with ametallic luster. Grain size of ilmenite is from 0.02–0.4 mm, mainly 0.1–0.25 mm. Ilmenite consists of the following main components: TiO_2 (48.50–53.00%), FeO (39.96–44.78%), MnO (3.40–8.60%), SiO_2 (0.32–2.68%).

+ Zircon ($ZrSiO_4$): is also a mineral found often in ore. It can be colorless, light blue, pinkish, and transparent. The content of zircon ranges from 0.3–10 kg/m^3. Grain size is from 0.02-0.2 mm. Zircon consists of the following main components: ZrO_2 (62.94–64.83%), HfO_2 (1.56–3.34%), SiO (32.15–33.58%).

According to Decision No. 104/2007/QD-TTg on Master Plans for investigation, exploration, processing and use of titanium ore in the 2007–2015 periods, orientation to 2025 of the Prime Minister, demand for products from titanium and zircon ores and estimated production of ilmenite concentration are shown in Tables 3 and 4.

Fig. 3: Marine deposit of Pleistocene (red sand) in Ninh Thuận

Fig. 4: Marine deposit of Halocene at Quang Xuong district in Thanh Hoa

Table 1

Chemical composition of ilmenite

Components (%)	Areas				
	Mo Duc – Quang Ngai	Phu My – Binh Dinh	Phu Cat – Binh Dinh	Quy Nhơn – Binh Dinh	Tuy Hoa – Phu Yen
TiO_2	52.00	53.00	50.79	48.84	50.14
FeO	44.22	42.18	44.12	45.63	38.96
MnO	2.87	2.21	2.95	4.03	8.60
SiO_2	0.31	2.17	1.37	0.88	1.95
Total	99.40	99.56	99.59	99.38	99.65

Source: General Department of Geology and Minerals of Vietnam

Table 2

Chemical composition of zircon

Components (%)	Areas				
	Mo Duc – Quang Ngai	Phu My- Binh Dinh	Phu Cat – Binh Dinh	Quy Nhơn – Binh Dinh	Tuy Hoa – Phu Yen
ZrO_2	64.48	64.31	63.05	62.94	63.97
HfO_2	2.45	2.07	3.10	3.34	2.17
SiO_2	32.59	32.91	32.97	32.58	33.33
Tổng	99.52	99.29	99.12	98.86	96.16

Source: General Department of Geology and Minerals ofVietnam

Table 3

Demand of products from titanium and zircon ores to 2025 (1,000 tons)

No.	Demand	2007	2010	2015	2020	2025
1	Pigment	12	16	26	42	74
2	Artificial Rutile or titanium slag	0	30	30	45	80
3	Ilmenite	28	37	70	110	170
4	Fine zircon powder	10	12	15	25	40

Table 4

Estimated production of ilmenite concentration to 2025 (1,000 tons)

Products	2007	2010	2015	2020	2025
Concentration ore	460	250	350	400	600
Concentration ore for deep processing	40	250	350	400	600
Concentration ore for export	420	0	0	0	0

2. Current mining of titanium placers in Vietnam and environmental issues

2.1. Mining and processing technologies

There are two mining and processing technologies applied for titanium placers in the Middle of Vietnam:

a. Mining with excavator (wheelloader) and truck combined with the fixed processing complex (Figure 5):

Ore is loaded by excavator or wheelloader to truck to transport to the fixed processing complex located in the center of the mine, then mixed with water to pump to the raw processing components. Concentration ore gained from the raw processing component is reprocessed at the concentration processing component to obtain the required concentration ore (approximately 85% of heavy minerals) – Figure 6.

The mining and processing scheme was applied quite commonly between 1995 and 2002 at the concentrated mines of large reserves in Ha Tinh, Binh Dinh, etc, but it is no longer popular. The scheme is high in output, but it involves considerable investment capital, it is not well suited for small and complicated mines, and encounters many difficulties in land reclamation, especially when the distance between the fixed processing complex and mining face is considerable. For a deposit with a grade less than 2% of heavy minerals, this scheme is ineffective.

Ore is broken by hydraulic monitor (if necessary) and pumped for feeding to the raft-mounted mobile processing complex in the mine in order to separate into 3 kinds of products (tailings, intermediate ore, and concentration ore). The intermediate ore gained from the raw processing component is pumped to the

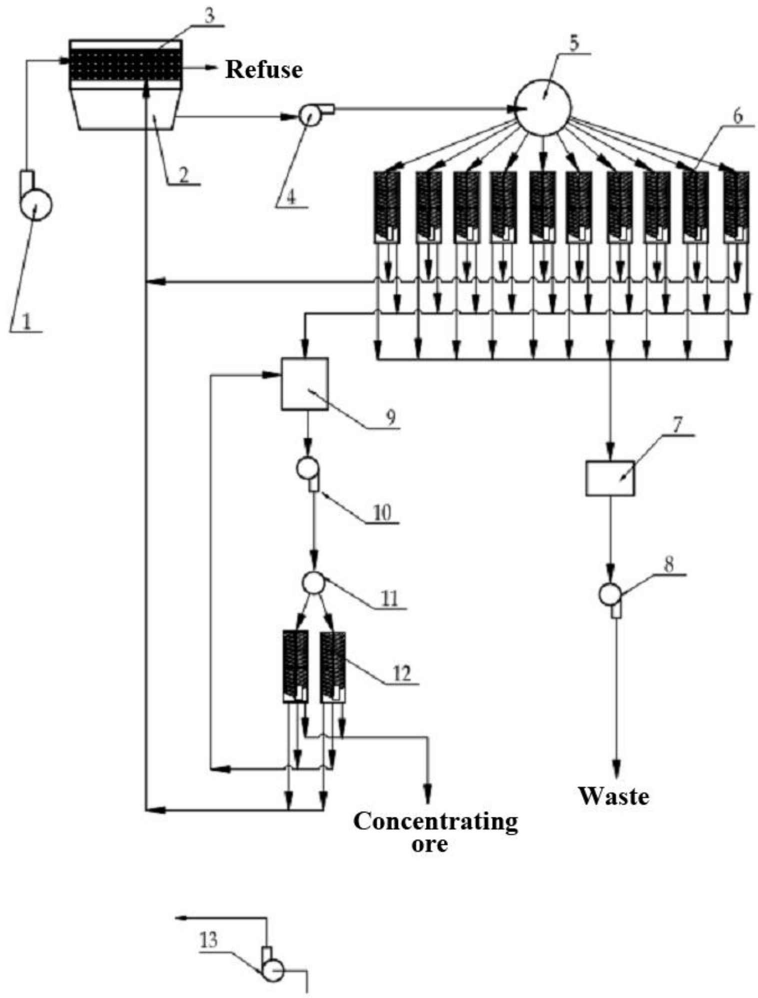


Fig. 5: Flow chart of the process scheme with fixed processing complex
(1 – Feed pump of run-off mine ores, 2 – Tank for collecting run-off mine ores, 3 – Screen for separating refuse, 4 – Pump supplying raw processing component, 5 – Tank for dividing into flows, 6 – Raw processing component, 7 – Tank for pumping waste sand, 8 – Pump for removing waste sand, 9 – Tank for pumping to the concentration processing component, 10 – Pump supplying for concentration processing component, 11 – Tank for dividing into flows, 12 – Concentration processing component, 13 – Water pump)

a

b

Fig. 6: Ore is extracted and transported by excavator and truck (a), and fed to the fixed processing complex by pump machine (b)

b. Mining with hydraulic monitor and sucker pump combined with the raft-mounted mobile processing complex (Fig. 7):

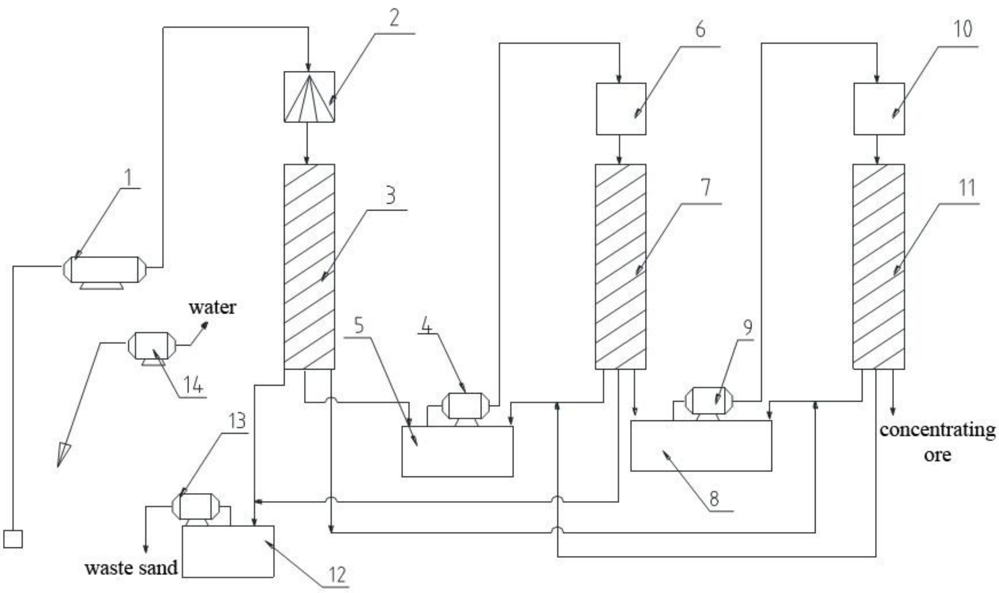


Fig. 7: Flow chart of the process scheme with mobile processing complex
(1 – Pump for sucking ore sand, 2- Tank for dividing into flows with structure of separating refuse, 3 – Raw processing component, 4 – Pump for intermediate processing, 5 – Tank for pumping to the intermediate processing component, 6 – Tank for dividing into flows, 7 – Intermediate processing component, 8 – Tank for pumping to the concentration processing component, 9 – Pump supplying for concentration processing component, 10 – Tank for dividing into flows, 11 – Concentration processing component, 12 – Tank for pumping waste sand, 13 – Pump for removing waste sand, 14 – Water pump)

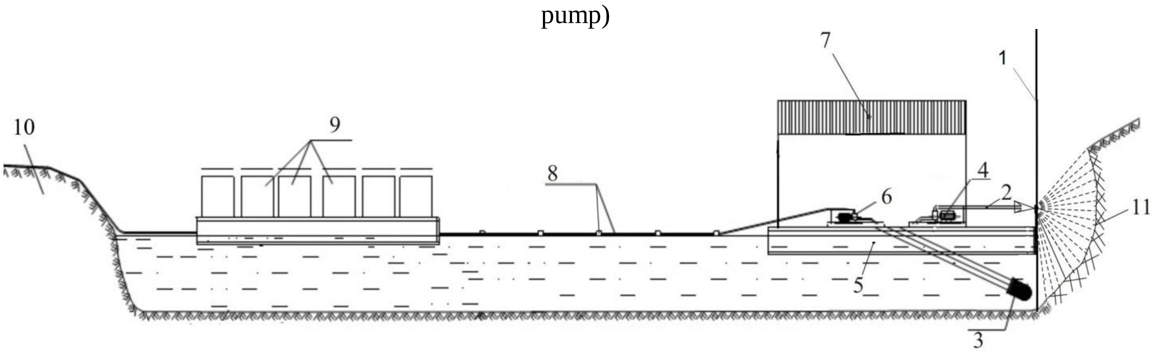


Fig. 8: Ore is exploited effectively with the mobile mining and processing system
(1 – anchor, 2 – nozzle, 3 – suction head, 4 – Pump for nozzle, 5 – raft, 6 – pump for mixture of titanium ore and sand, 7 – roof of the raft, 8 – pipeline to processing system, 9 – processing system, 10 – titanium ore)

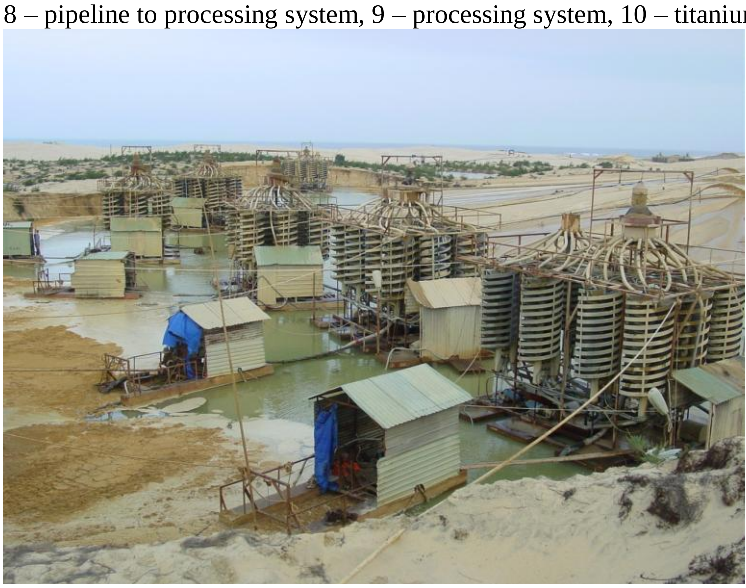


Fig. 9: Mining titanium placers with the mobile mining and processing system at Phu My district, Binh Dinh province, Vietnam

Fig. 10: Changing terrain of the area caused by mining and processing activities of titanium placers

intermediate processing component in order to obtain concentration ore. The concentration ore that meets the quality requirement (>85 % of heavy minerals) is pumped to the store. Waste sand is pumped to the waste dump at the mined-out areas (Figure 8).

The mining and processing scheme is flexible and suitable for deposits located under the underground water level. The process scheme is currently applied effectively for almost all titanium placers mines in the Middle of Vietnam.

2.2. Environmental issues caused by the mining and processing activities of titanium placers

Hazardous environmental impacts caused by the mining and processing activities of titanium placers can be listed as follows:

a. Changing terrain of the area: Over the time of the mining and processing, the terrain of the mine area has changed in altitude, slope, landscape, etc. These changes cause deformations of the coastline, depression of mine slope and beach, fly sand, etc.

b. Influence on the ecosystem of forest and land: Many protective forests have been cut down and overrun to create areas for ore exploitation. These have caused serious environmental pollution and soil depression. The key hazardous impacts of the mining and processing activities of titanium placers on the coastal soil environment are long and expansive land appropriation, a large volume of waste sand and areas of land reclamation that are limited in comparison with the mined-out regions.

c. Impact on the water environment: The rivers and lakes surrounding the mining areas are the sources of water supplying all mining and processing activities. Underground water in sand is used circularly. Mining and processing activities have lowered the local water level, and can cause salt incursion into water bearing aquifers.

Wastewater bearing sediment mud, heavy metals, lubricants, and so on is the reason for pollution of the running water for the people living in the areas surrounding the mine.

Mining and processing activities create a large volume of wastewater that is absorbed directly into the sand without treatment. This causes underground water pollution of the surrounding areas

d. Loss of mineral resources: Without the master plan for exploring and mining titanium placers, the area and mining sequence of licensed mines are not suitable. They lead to a loss of mineral resources and hamper the land reclamation process. Moreover, poorly developed mining and processing equipment is also a reason for mineral loss and low ore recovery.

e. Radioactive pollution: Mining, hauling, and processing activities of ilmenite can disseminate radioactive substances with the sources are radioactive minerals going along such as zircon, monazite, etc, containing thorium, uranium, etc. In particular, concentration ores usually have a radioactive field in excess of radioactive safety norms.

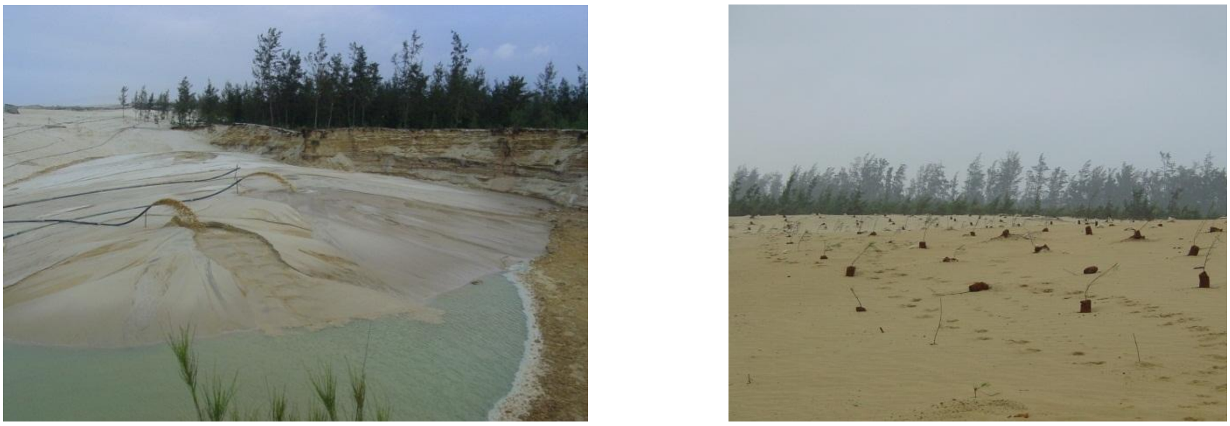


Fig. 11: Areas of land reclamation are limited compared with the mined-out areas

Fig. 12: Loss of titanium placers in mining and processing activities

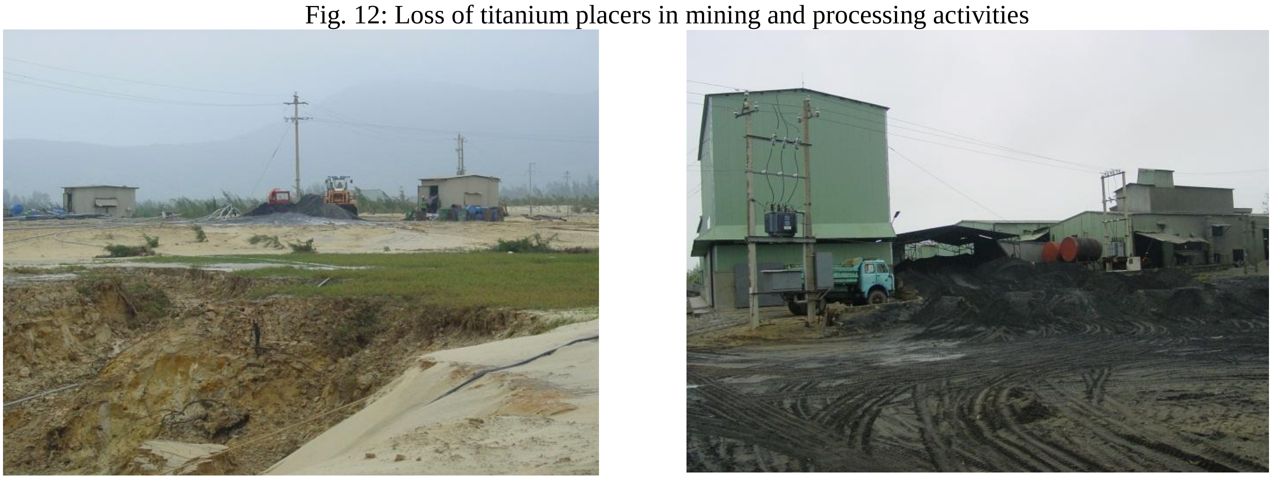


Fig. 13: Danger of radioactive pollution in mining, hauling, and processing of ilmenite

3. Directions for future development of the titanium industry in Vietnam

3.1. A master plan for exploration, mining and processing of titanium placers

Based on the demand for products from titanium placers for domestic consumption and export, and given the local potential of titanium placers, the government should have suitable master plans in place for the exploration, mining and processing of these kinds of minerals in provinces in the Middle of Vietnam.

To avoid the overlapping in the master plans of titanium ore exploitation, construction of an ecological resort, aquaculture, the construction of a coastal industrial park, and so on, which complicate the management process, the regulations of the Minerals Law should be observed, meaning that the areas containing titanium ore must be developed before conducting other works in these areas.

Additionally explore in particular the older titanium placer deposits in order to increase the reliability of ore quality and reserves, and to effectively justify investment and exploitation. At the same time, prospecting work should be performed for potential new deposits.

Areas for the mining and processing of titanium placers and the production there of should be planned in line with the development of industrial zones, export processing zones, meet all requirements of domestic consumption and export, and convenience for transportation, business, etc.

3.2. Improving the role of state management in licensing and ore consumption

a. Licensing management

Although the Provincial People's Committee is assigned to license artisan mining, in reality, its licensing power has not yet been executed properly. The "ask-and-give" mechanism still exists in the state management of mining licensing.

To avoid the above-mentioned problem, a new point, which is deemed a breakthrough of the Minerals Law of 2010, is a fundamental change in the mineral management mechanism, especially concerning regulations on finance, which are suitable for the market economy, such as regulations on payment for mining rights (Article 77); regulations on auctioning mining rights (Article 82), and regulations on licensing powers (Articles 77 and 78). These new regulations are transparent and in the public domain, helping abolish the "ask-and-give" mechanism, select technically and financially cabable investors, and avoid improper licensing as was the case in the past.

To guarantee the comprehensiveness between exploitation and processing, it is specifically stipulated in the Minerals Law of 2010 that mining licenses shall only be granted alongside commitments to build deep-processing plants to ensure added value for the mineral commodities.

b. Management and ore consumption

According to the Minerals Law, as amended in 2005, the export of raw minerals is limited by the state. In reality, however, this activity is still very much a work in progress. The deeper the processing of titanium ore, the higher the potential benefit. If the titanium ore is extracted and exported under the form of concentrates, its value is low and the revenue that the state can receive is not commensurate with the true value of the resources.

The Minerals Law of 2010 stipulated that it is forbidden to export crude ores. As regards the amount of titanium ore that had been extracted before the Minerals Law of 2010 came into effect, the Prime Minister released Directive No. 02/CT-TTg dated January 9, 2012, allowing for the export of in-stock ilmenite ore concentrates till June 30, 2012. This is an extremely important alternative that allows companies to release titanium ore concentrates from storage, ensuring are turn of capital to build deep-processing plants, registered in the provincial territories. Since July 1, 2012, the export of unprocessed mineral commodities has been forbidden, unless otherwise approved by the Prime Minister.

3.3. Investing in advanced mining and processing equipment and researching suitable process schemes

a. Investing in advanced mining and processing equipment

To increase productivity and ore recovery, and to decrease ore loss, investment into advanced mining and processing equipment is essential for titanium placer mines in Vietnam, especially for kinds of ores in the red sand formation.

Dredgers and bucket chain excavators with a pipeline system and processing complex are options that should be considered carefully. The experience of Germany and other countries are useful lessons for application at titanium placer deposits in the Middle of Vietnam.

b. Researching suitable process schemes

To decrease ore loss, increase the ratio of land reclamation and protect the environment, a suitable process scheme and mining sequence are very important for the titanium placer mines in Vietnam.

Suitable process schemes and mining sequences must facilitate all activities in mining, processing, waste dumping, and land reclamation,

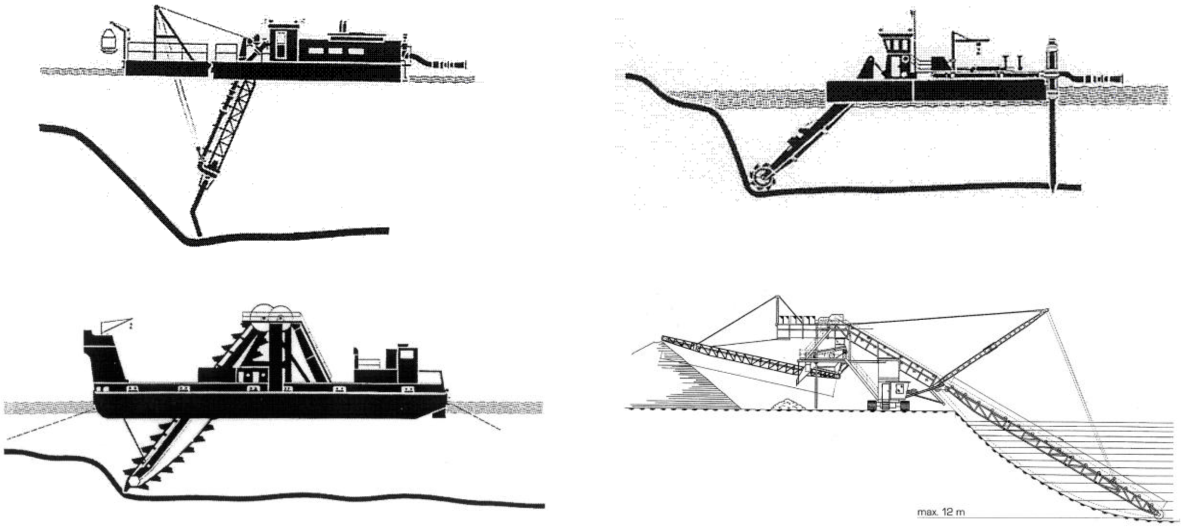


Fig. 14: Use of dredger and bucket chain excavator to extract and process placer ores

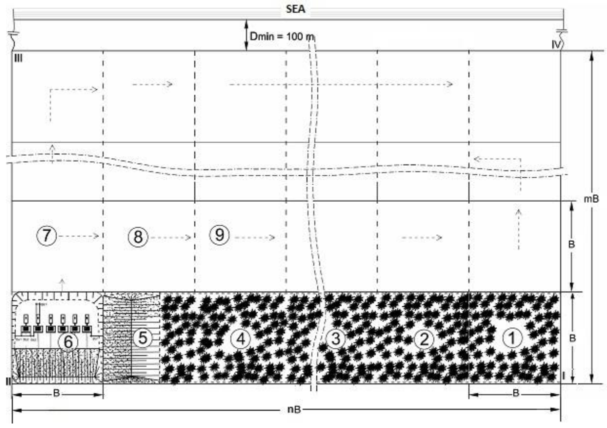


Fig. 15: Proposed process scheme with model mining sequence
(1), (2), (3), (4) – mine-out blocks have been reclaimed; (5) – block for waste dumping; (6) – mining block; (7), (8), (9) -next mining blocks

to be performed in sequence. In the process scheme, the area of the mine should be divided into blocks for mining periods, and developed in a specific direction.

In Figure 15, suitable process schemes and mining sequences are proposed for application at titanium placer mines in Vietnam. Here, blocks are square with B is the side of the square. D_{min} is the minimum distance between the mine limit and the sea. The mining direction and sequence are shown by the number and arrows in Figure 15.

The above sequences of mining, processing, waste dumping, and land reclamation represent a truly environmental-friendly process scheme for titanium placer mines in Vietnam.

4. Conclusions

Vietnam has potential reserves of titanium placers, located mainly along the coast of the Middle, spreading over provinces from Thanh Hoa to Ba Ria – Vung Tau. There are two mining and processing technologies applied for titanium placers in the Middle of Vietnam: mining with excavator (wheelloader) and truck combined with the fixed processing complex, and mining with hydraulic monitor and sucker pump combined with a raft-mounted mobile processing complex.

Mining titanium placer deposits has generated considerable profit for localities and the country as a whole, but it has also caused significant environmental issues, such as a changing terrain, impact on the ecosystem of forestry and land, effect to water environment, loss of mineral resources, and radioactive pollution.

For sustainable future development of the titanium industry in Vietnam, this paper proposes a number of directions:

- there should be a master plan for titanium placer exploration, mining and processing;
- the role of state management in licensing and ore consumption should be improved;
- investment should be directed to state-of-the-art mining and processing equipment and to research into suitable process schemes and mining sequences.

It is completely feasible to consider and effectively apply under the right conditions these directions for titanium placer mines in Vietnam. They are also useful reference material for similar titanium placer mines around the world.

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"Gornye nauki i tehnologii"/ "Mining science and technology", 2016, No. 1, pp. 39-49	
Title:	**Status of development orientations for mining titanium placers in Vietnam**
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DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-40-50**
Abstract:	Vietnam has potential reserves of titanium placers, located mainly along the coast of the Middle spreading across provinces from Thanh Hoa to Ba Ria – Vung Tau. Mining titanium placers deposits has gained considerable profit for localities and the country as a whole, but it has also given rise to significant environmental issues. The paper introduces the potential of titanium placers in Vietnam, current mining status and environmental issues faced, and then proposes a number of strategic orientations for sustainable development of the future titanium industry of Vietnam. These orientations are completely feasible with regard to the titanium placer mines of Vietnam in order to consider and apply effectively for suitable conditions. They also represent a useful reference for similar titanium placer mines in the world.
Keywords:	Vietnam, titanium placers, marine deposit, mining and processing technologies.
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ENGINEERING AND GEOLOGICAL SUPPORT FOR SLOPE STABILITY MONITORING AS A PART OF TRANSPORT INFRASTRUCTURE CONSTRUCTION PROJECTS

Exploitation of landslide-prone escarpments with low safety factor, or in a state of limiting equilibrium, can lead to a disaster when changing the seismic and hydro-geological situation. Therefore, recommendations must be developed for the deployment of a high density network of hydrogeomechanical monitoring information collection points for rapid notification when the critical water level is reached and for making management decisions to strengthen the slope.

Keywords: Landslide danger, slopes, slope processes, geological engineering surveys, hydrology, monitoring, sounding, mathematical modeling.

Engineering activities may result in the formation of complex natural and industrial systems which require interactive monitoring to ensure their sustainable development. A set of monitoring activities will make it possible to assess the rock mass space-time behavior and control its condition. This would ensure the industrial and environmental safety of construction projects delivered under complex engineering-geological, hydraulic geological and geomechanical conditions, and result in the development of new territories and mineral deposits.

Over the last decade the North Caucasus territory has experienced major new developments, mainly associated with the 2014 Winter Olympic Games. In particular, the combined (road and rail) Adler-Mountain Climatic Resort Alpika-Service highway was commissioned in Sochi to provide a reliable and regular transport link between the mountain cluster facilities and the main transport hubs. The road commissioned in November 2013 is approximately 50 km in length and has several junctions connecting to Federal Road M-27. In terms of the number of tunnels, this is a unique project for Russia. The entire construction site for this transport line features very complex engineering and geological conditions. The works were performed under young Alpine orogenesis conditions that give rise to the development of hazardous geological processes of both endogenous and exogenous nature. The northern section of the road is in a high seismic region (up 8–9 intensity), and this area also features minor surface slopes that results in the active development of slope processes (rock slides, landslides, mudflows, avalanches), as well as water erosion.

The slopes on the left bank of the Mzymta River and those on the section from Alpika-Service terminal to the nearest railway tunnel portal pose the highest soil slip hazard. The main conditions for the formation and development of hazardous slope processes are the specific nature and geological structure of the mountainous area, including the composition and mode of occurrence of geological material, structural and tectonic characteristics of rock mass, hydrodynamic mode of underground waters, neotectonic movements and other factors. The area's upper rock layer includes eluvial, diluvial and soil slip deposits comprising argillite, sandrock, porphyrite, and others bed rock land wastes. In this respect, these surface deposits include crushed stone, granitic subsoils with various joining materials (mild clay, sandy loam), as well as mild clay and clay [3].

To ensure safe operation of the road and on-time implementation of modern measures to minimize slope process hazards, in 2011□ 2013 the NUST MISiS Geology and Mine Surveying Department, together with Alkomp-Evropa LLC, developed a complex program for monitoring potential landslide slopes. The monitoring activities include collecting geodetic and geological information (directional survey, automated tacheometric survey, collection of piezometric data across the well network) with the aid of a set of interactive equipment. Most survey activities of potentially hazardous slide masses were performed in 2012–2013.

A reconnaissance survey of the territory of the Alpika-Service terminal construction site was performed at the initial phase. The main sections for field activities were determined based on the site route survey and geological structure data.

Based on reconnaissance survey data and available geological information, several clay slate block sample sites (interstratified argillite) with not more than 2.5 m loose deposit thickness were determined. Then, within each selected site, the points were positioned in accordance with modern approaches for network development and testing, with the aid of mathematic models and complying with the main engineering and geological survey techniques: equal reliability, survey completeness, step-by-step approximation of data collection, minimum time and financial expenditure. In every selected site the general variability functions were determined based on the sum of surveyed deposit characteristics. This made it possible to design an engineering and geologic sampling network. The used method helped to minimize the number of sampling sites without any loss of data integrity [3, 4].

Cores were drilled directly from the mass following the cleaning (cutting down) of the upper waste layer and taking grab samples from the lumps and uncoverings in perennial and dry water stream beds (Figure 1).

To take the required amount of samples, 10 sample pits (up to 2 m in depth) were made; 65 samples were taken in total, including 42 cores with a diameter of 42 mm and minimum length of 85 mm; 17 sample pits for drilling samples with required geometrical shapes and sizes, as well as 6 samples of non-disturbed connected geological material (clays formed after slacking of waterlogged slate stone).

a *b*

Fig. 1. Core drilling: *a* – directly from rock mass; *b* – from lumps.

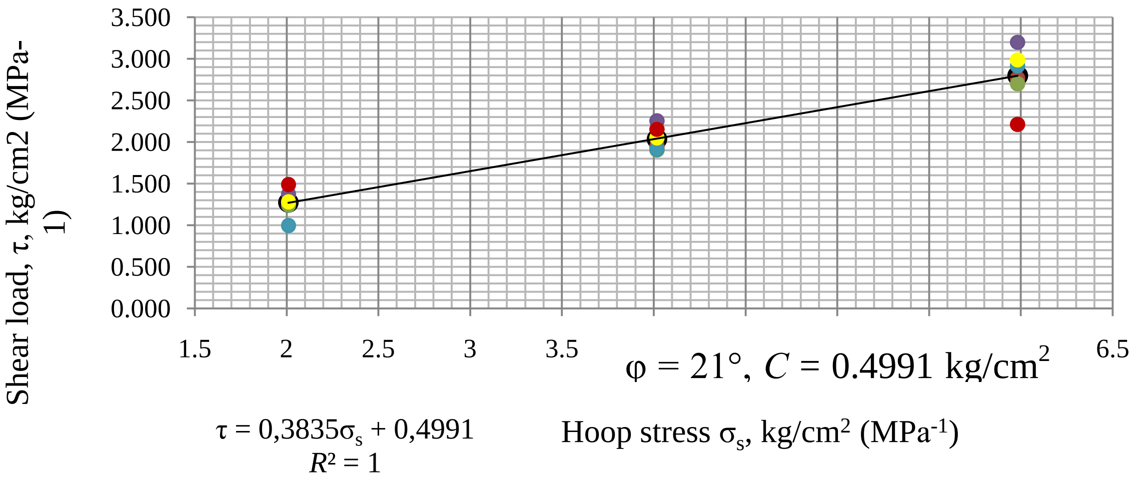
The samples were also taken from stream flow sites (18 sampling points in total in two sampling sites): 134 cores (diameter of 42 mm, length from 88 to 181 mm), 23 grab samples for further drilling-out in laboratory conditions. The uncovered clay slate and black earth area near the exit from the rail road section was subject to 100% sampling with average sampling step from 1–1.5 m.

The following was determined during the laboratory test phase: density (ρ), rock strength under uniaxial compression (σ_c), rock strength under uniaxial tension (σ_t), elastic modulus (E_{el}), deformation modulus (E_d), Poisson ratio (ν), angle of shear resistance (φ), specific cohesion (Figure 2).

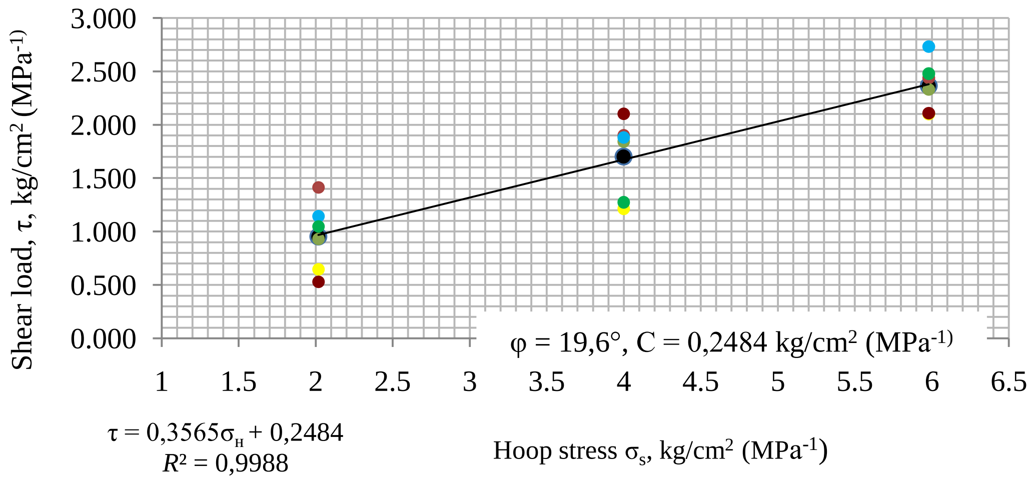
Shear testing of clay slate for lamination as well as testing of the land waste mild clay (with crushed stone inclusions) provides an almost identical angle of shear resistance and cohesion. This proves the presence of clay material between the slate stone layers. This leads to major weakening inside the mass and increases the soil slip hazard.

The obtained field and laboratory survey data made it possible to create a pattern and a model of slopes accounting for the physical and mechanical properties of build-up rocks and local hydrogeological conditions.

Stability factor (η) was calculated at monitoring sites 8 and 9 to assess potentially hazardous slopes; the calculations took into account the test profiles and various rock mass waterlogging levels. The work was performed using a software system developed at the Geologic Department of Moscow State Mining University.



a



b

Fig. 2. Dependencies $\tau(\sigma_s)$ for clay slates (*a*); mild clay (*b*).

The calculations are based on methods approved by the Russian State Mining and Industrial Inspectorate (Gostekhnadzor); these methods are founded on the "loosen medium" limit equilibrium approach which also covers limit equilibrium of frictions connected medium (i.e. geological material under consideration, Figure 3) [1].

Slope stability under various seismic, hydro-geological and climatic conditions was assessed in 2013. This work helped to develop criteria (in terms of numbers) to assess soil slide hazards. For the project area under consideration, the obtained dependencies of the stability factor on water levels in hydro-geological wells, accounting for the amount of precipitation, were used in the automated rock mass condition data collection and processing system (Figure 4).

From the works completed in 2012–2013, it can be concluded that the slopes near Alpika-Service rail road terminal feature a low stability ratio; some potentially sliding bodies are in the limit equilibrium state, which could result in emergency situations if seismic and/or hydro-geological conditions change. Therefore, recommendations were developed on deploying a high density hydrogeomechanical monitoring network for rapid notification when the critical water level is reached and for making management decisions to strengthen the slope.

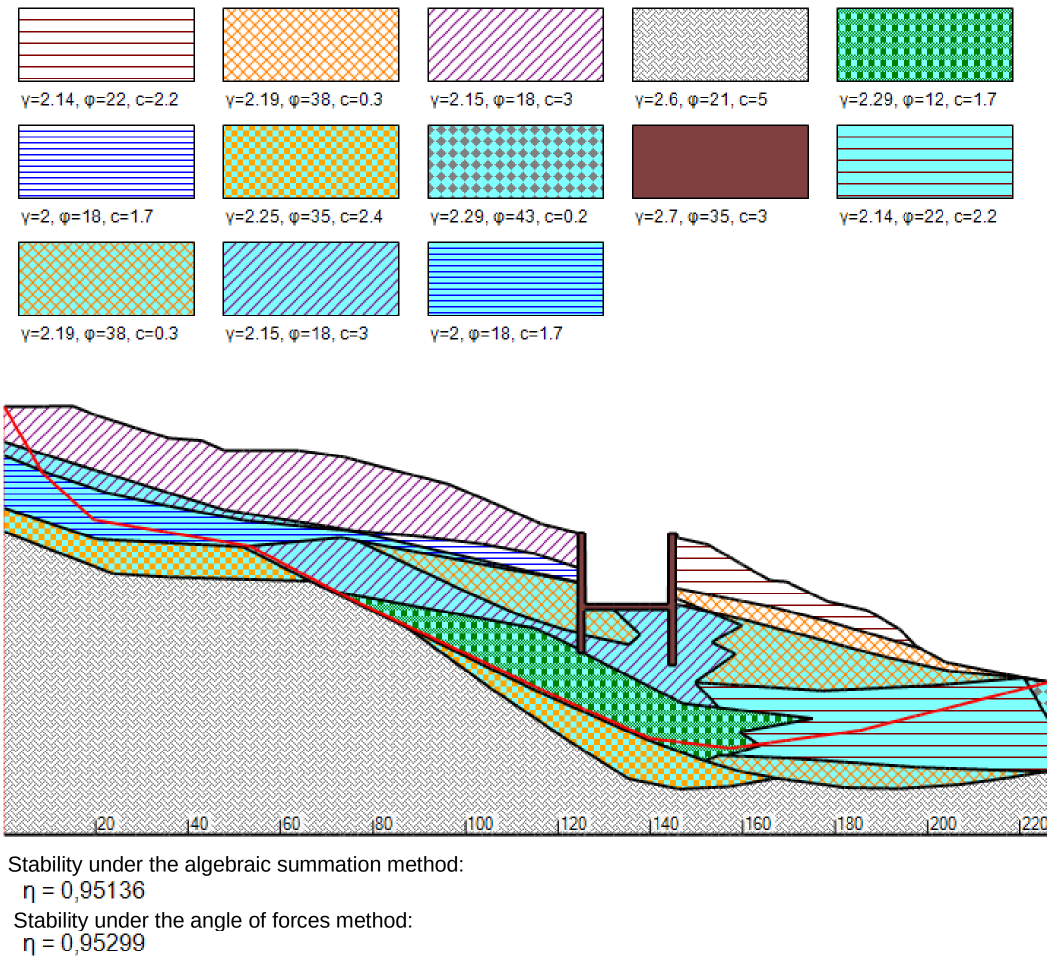


Fig. 3. Calculation of stability factor in Geodamp with preset water level in hydro-geological wells.

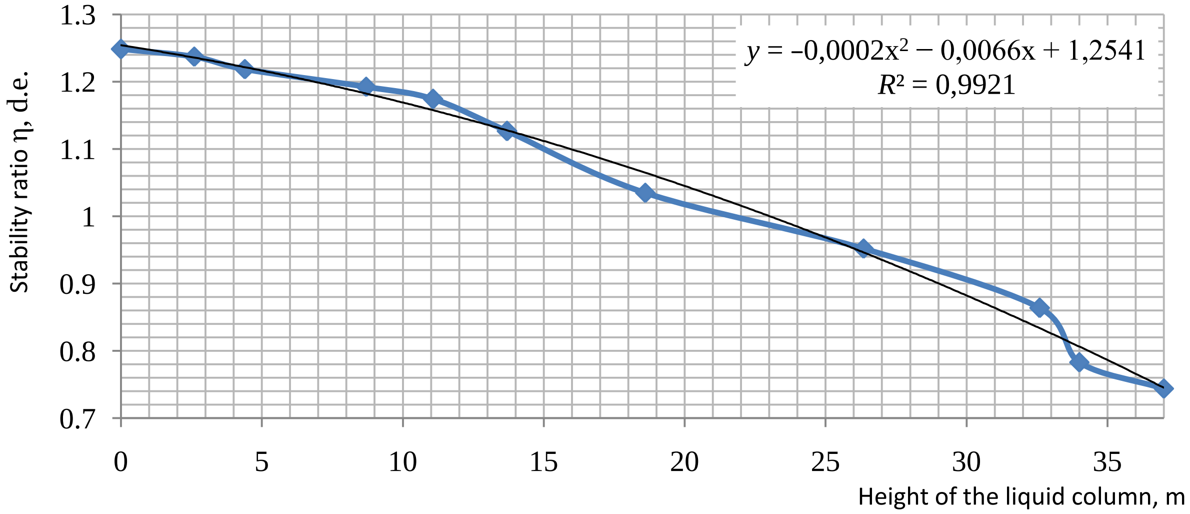


Fig. 4. Stability ratio dependence on the height of the liquid column in a hydro-geological well

A unique complex engineering and geological remote monitoring automated system was developed. The experience of deploying the system can be used by mining companies or building contractors when implementing construction projects on soil sliding areas. The developed system is capable of recording geological material condition changes in time and space; the sample collection networks are designed based on statistics methods and cluster analysis; this improves the reliability of geological material behavior assessment. The complex approach is achieved by calculating limit values using sets of parameters as the function of dependence of the system stability on the rock mass state vector.

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"Gornye nauki i tehnologii"/ "Mining science and technology", 2016, No. 1, pp. 50-55

Title:	**Engineering and geological support for slope stability monitoring as a part of transport infrastructure construction projects**
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DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-51-57**

Abstract:	Exploitation of landslide-prone escarpments with low safety factor, or in a state of limiting equilibrium, can lead to a disaster when changing the seismic and hydro-geological situation. Therefore, recommendations must be developed for the deployment of a high density network of hydrogeomechanical monitoring information collection points for rapid notification when the critical water level is reached and for making management decisions to strengthen the slope.
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LE HUNG TRINH (*Le Quy Don Technical University*)

APPLICATION OF REMOTE SENSING TECHNIQUE TO DETECT AND MAP IRON OXIDE, CLAY MINERALS, AND FERROUS MINERALS IN THAI NGUYEN PROVINCE (VIETNAM)

This article presents a study on the application of remote sensing techniques to detect clay minerals, iron oxide and ferrous minerals using LANDSAT 7 ETM+ multispectral images. We used band ratio method to determine areas that are rich and poor in mineral composite content. The results obtained in this study can be used to create a distribution map of clay mineral and iron oxide, and to facilitate mineral mining and exploration.

Keywords: remote sensing, iron oxide, clay mineral, ferrous mineral, band ratio.

I. INTRODUCTION

Mineral resources are one of the most important natural resources of every country. Minerals are a source material for many industries, such as energy production, building materials, metals, agriculture, and other branches of industry. The exploration of mineral composites is a complex and urgent problem in the research and monitoring of natural resources. Traditional methods based on field surveys only solve the problem on a small scale because of the high cost involved. Remote sensing technology with advantages such as wide area coverage and short revisit interval has been used effectively in the study of mining and exploration minerals [1-5].

Band ratio is a useful method for preprocessing satellite images, especially in areas where topographic effects are important. Band ratio is based on dividing the pixels in one band by the corresponding pixels in a second band. There are two reasons for this: one is that the differences between the spectral reflectance curves of surface types can be highlighted. The second is that illumination and, consequently, radiance may vary, while the ratio between illuminated and non-illuminated areas of the same surface type will be the same [2, 5].

Today, band ratio has been widely used in spectral index building (soil degradation index, leaf area index) for monitoring land cover and minerals, and for analyzing pollution. This article indicates a band ratio method for building spectral indices and mapping the distribution of iron oxide, clay minerals and ferrous minerals.

II. MATERIALS AND METHODS

2.1 Study area

Thai Nguyen district is located in the northeast path of Vietnam, in the Pacific mineral belt. The study area is located within the latitudes $20^0$20’N and $22^0$25’N and longitudes $105^0$25’E and $106^0$16’E, covering an area of approximately 3,562.82 km². The average temperatures in the hottest and the coldest months are 28.9 °C in June and 15.2°C in January. The lowest recorded is 13.7°C. The total number of sunshine hours in a year ranges between 1300 and 1750, which is equally distributed over the months in a year. The climate of Thai Nguyen has two distinct seasons: the rainy season from May to October and the dry season from October to May. The average rainfall per annum lies in the range of 2000 to 2500 mm; it rains most in August and least in January. Generally speaking, Thai Nguyen’s climate is favorable for developing agriculture and forestry. With its rich mineral resources and favorable climate, the province offers significant opportunities for industrial development for both domestic and foreign investors [8].

2.2 Methodology

Band ratio is a technique that has been used for many years in remote sensing to display spectral variations effectively. It is based on highlighting the spectral differences that are unique to the materials being mapped [1, 2, 5]. Identical surface materials can give different brightness values because of the topographic slope and aspect, shadows, or seasonal changes in sunlight illumination angle and intensity. These variances influence the viewer’s interpretations and may lead to misguided results. Therefore, the band ratio operation could be able to transform the data without reducing the effects of such environmental condition. In addition, ratio operation may also provide unique information that is not available in any single band which is very useful for disintegrating the surface materials. The band ratio images are known for enhancing spectral contrast among the bands considered in the ratio operation and have successfully been used in mapping alteration zones [1−4].

Fig. 1 shows the reflectance spectra for iron oxide (goethite and hematite). The vertical axis shows the percentage of incident sunlight that is reflected by the materials. The horizontal axis shows wavelengths of energy for the visible spectral region (0.4 to 0.7 µm) and the reflected portion (0.7 to 3.0 µm) of the infrared (IR) region. Reflected infrared energy consists largely of solar energy reflected from the earth at wavelengths longer than the sensitivity range of the eye. The spectral reflectance curve shows that the maximum reflectance of iron oxide occurs in the red band (band 3) and that reflectance is considerably lower in the blue band (band 1). The brightness signatures in the ratio image band3/band1 correlate with iron oxide [2, 5].

Fig. 2 shows the reflectance spectra of clay minerals and other hydrothermal alterations. The vertical axis shows the percentage of incident sunlight that is reflected by the materials. The horizontal axis shows the wavelengths of energy for the reflected portion (1.0 to 3.0 µm) of the infrared (IR) region. The spectral reflectance curve shows that the maximum reflectance of iron oxide occurs in band 5 and that reflectance is considerably lower in band 7 LANDSAT multispectral images. The brightness signatures in the ratio image band5/band7 correlate with clay minerals [2, 5].

The methodology is based on the mineral composite (clay mineral, ferrous mineral and iron oxide) and normalized difference vegetation indices NDVI, calculated according to the following equation:

$$NDVI = \frac{\rho_{RED} - \rho_{NIR}}{\rho_{RED} + \rho_{NIR}},$$

where ρ_{RED}, ρ_{NIR} - surface reflectance of red band and near infrared band respectively.

The role of NDVI is to mask dense plant areas. From the theoretical knowledge of the mineral’s spectral properties, it is well recognized that the LANDSAT ETM+ band ratios of 3/1, 5/7, 5/4 are analyzed for iron oxides, clay minerals and ferrous minerals respectively [1, 5, 6, 7].

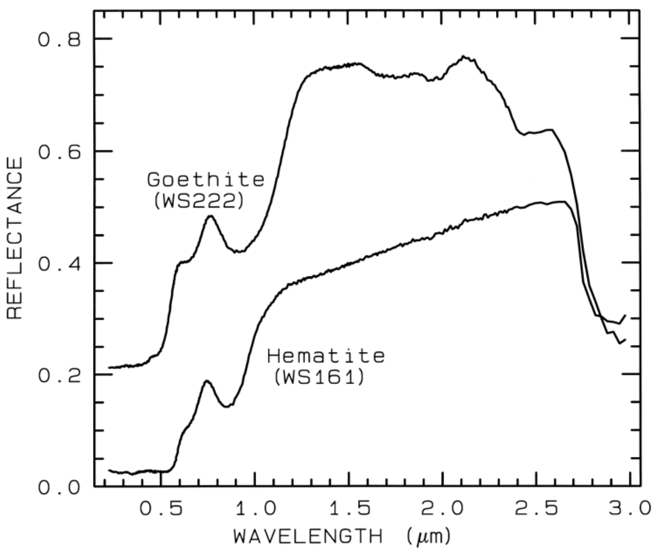


Fig. 1: Spectral features of iron oxide

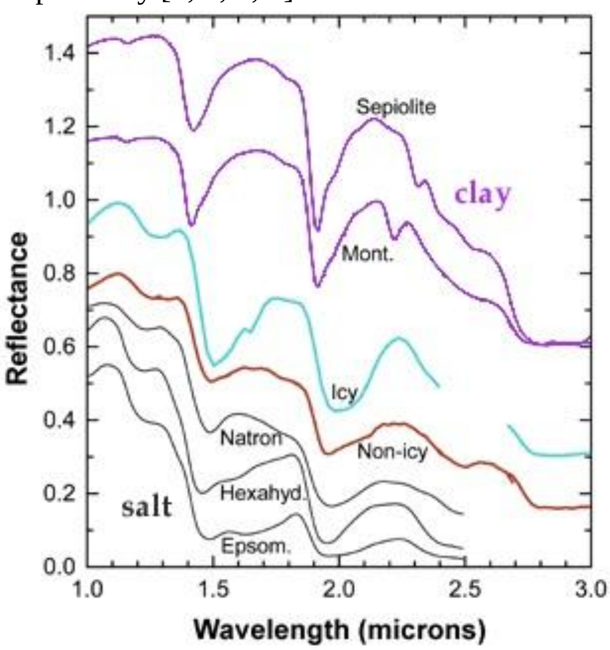


Fig. 2: Spectral characteristic of clay minerals

Table 1

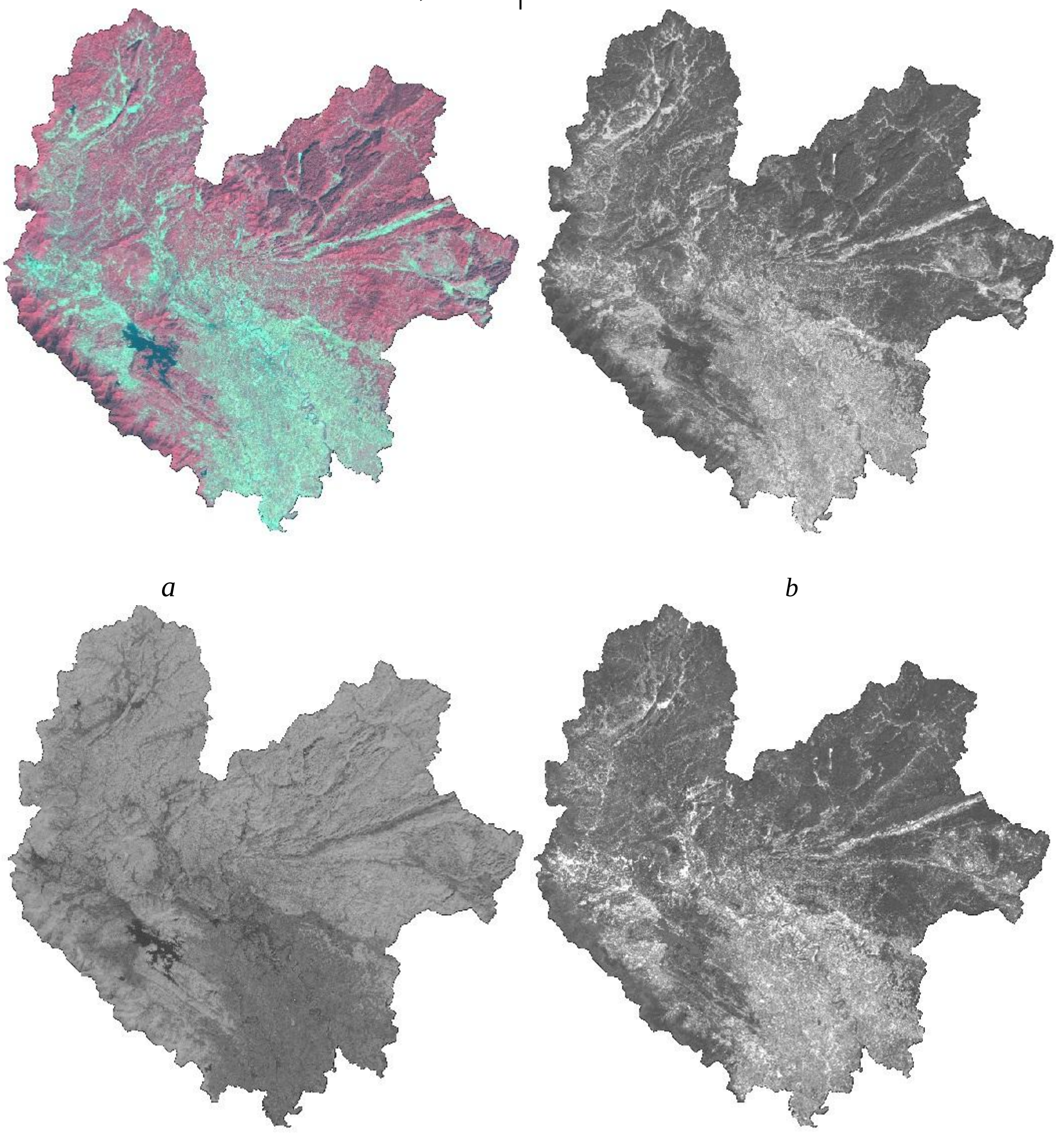
Algorithms of employed indices

No.	Indices	Algorithms
1	Clay minerals	Band5/band7
2	Iron oxide	Band3/band1
3	Ferrous minerals	Band5/band4

III. RESULTS AND DISCUSSIONS

To detect and map iron oxide, clay and ferrous minerals we used a LANDSAT ETM+ satellite image on November 8, 2007 (fig. 4). The Enhanced Thematic Mapper (ETM+) on board LANDSAT-7 is a multi-spectral radiometric sensor that records eight bands of data with varying spectral and spatial resolutions (30 m spatial resolution for red, green, blue, near infrared, and two bands of medium infrared; 60 m for thermal infrared; and a 15 m panchromatic band). With average spatial resolution, thermal infrared LANDSAT ETM+ image applications in the region of study (fig. 3).

Spatial distribution of clay mineral, iron oxide and ferrous mineral classes is determined and given in figure 4 (a-c). Nine-index classes ware interpreted into four categories named: very race, race, and medium, high – very high.



a *b* *c* *d*

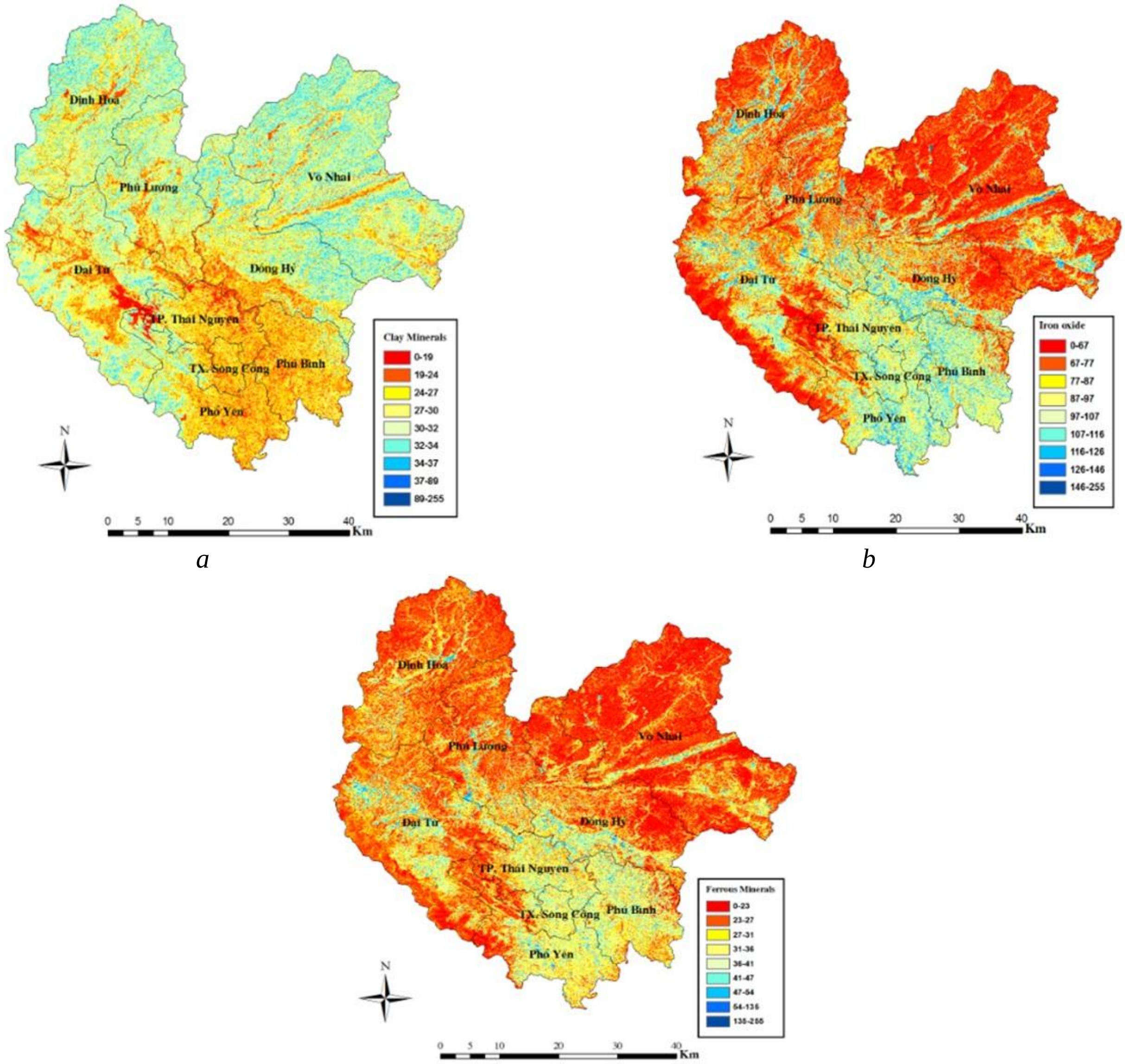
Fig. 3: LANDSAT ETM+ image 08 – 11 – 2007 of study area in color composite 432 (*a*), iron oxide index (*b*), clay mineral index (*c*) and ferrous mineral index (*d*)

Spatial distribution of clay mineral, iron oxide and ferrous mineral classes is determined and given in figure 4 (a-c). Nine-index classes ware interpreted into four categories named: very race, race, and medium, high – very high.

According to the spatial distribution of iron oxide, the main part of the study area (76.79%) is assessed in very race – race category, while the areas “medium” category covered a small portion (23.04%) of the total study area. The areas that contain iron oxide in “medium – high – very high” category covered minor portion (0.17%) of the study area (table 2).

Spatial distribution of clay minerals shows that more than half of the study area (62.14%) participates in the “race” category and this is ensued by “race – medium” (37.86%) and no area is detected for “high – very high” categories (Table 3).

Spatial distribution of ferrous minerals shows that the majority (72.13%) of the study area is evaluated in the “race” category. The area that contains ferrous minerals in the “medium” category covers 25.75% of the total study area and the area “medium – high” categories covers only a very small part (2.11%) of the study area and no area is detected for the “very high” category (Table 4).



a *b*

c

Fig. 4: Clay mineral (a), iron oxide (b) and ferrous mineral (c) index maps of Thai Nguyen district

Table 2

Class areas of iron oxide

Class	Index values	Cover area (km^2)	% of the total district area	Interpretation	
				Category	% of the total district area
1	0-67	781.2	22.10	Very race – race	22.10
2	67-77	971.85	27.49	Race	54.69
3	77-87	541.51	15.32		
4	87-97	420.04	11.88		
5	97-107	361.06	10.21	Medium	23,04
6	107-116	249.42	7.06		
7	116-126	149.23	4.22		
8	126-146	54.73	1.55		
9	146-255	6.14	0.17	Medium – high – very high	0.17
Total		3535	100		100

Table 3

Class areas of clay mineral

Class	Index values	Cover area (km^2)	% of the total district area	Interpretation	
				Category	% of the total district area
1	0 – 19	58.39	1.65	Race	62.14
2	19-24	511.07	14.46		
3	24-27	724.09	20.48		
4	27-30	903.14	25.55		
5	30-32	579.9	16.4	Race – medium	37.86
6	32-34	520	14.71		
7	34-37	220.02	6.22		
8	37-89	18.66	0.53		
9	89-255	0.01	0	Medium – high – very high	0
Total		3535	100		100

Table 4

Class areas of ferrous mineral

Class	Index values	Cover area (km^2)	% of the total district area	Interpretation	
				Category	% of the total district area
1	0-23	878.20	24.84	Race	72.13
2	23-27	1061.69	30.03		
3	27-31	610.15	17.26		
4	31-36	498.32	14.10	Medium	25.75
5	36-41	266.11	7.53		
6	41-47	146.18	4.13		
7	47-54	60.29	1.71	Medium – High	2.11
8	54-135	14.24	0.4		
9	135-255	0.01	0	Very high	0
Total		3535	100		100

IV. CONCLUSIONS

Spectral characteristic analysis of minerals shows that the LANDSAT 7 ETM+ multispectral image with 30 m spatial resolution can be used effectively for detecting and predicting the density distribution of iron oxide, clay and ferrous minerals. Consequently, the areas that are rich and poor in mineral composite content were determined with their coverage and geographical locations faster and more reliably.

The overall results indicated that a considerable proportion of the area of Thai Nguyen has moderate iron oxide, but poor ferrous and clay mineral content. The results obtained in this study can be used to create a distribution map of clay minerals, ferrous minerals, and iron oxide for the purposes of mineral mining and exploration.

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“Gornye nauki i tehnologii”/ “Mining science and technology”, 2016, No. 1, pp. 58-63

Title:	**Application of remote sensing technique to detect and map iron oxide, clay minerals, and ferrous minerals in Thai Nguyen Province (Vietnam)**
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DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-60-65**
Abstract:	This article presents a study on the application of remote sensing techniques to detect clay minerals, iron oxide and ferrous minerals using LANDSAT 7 ETM+ multispectral images. We used band ratio method to determine areas that are rich and poor in mineral composite content. The results obtained in this study can be used to create a distribution map of clay mineral and iron oxide, and to facilitate mineral mining and exploration.
Keywords:	remote sensing, iron oxide, clay mineral, ferrous mineral, band ratio.
References:	1. Hankan Mete Dogan. Mineral composite assessment of Kelkit River Basin in Turkey using remote sensing (2012), Journal Earth System Science 118, No. 6, pp. 701 – 710. 2. Md. Bodruddoza Mia, Yasuhiro Fujimitsu. Mapping hydrothermal altered mineral composite using LANDSAT 7 ETM+ image in and around Kuju volcano, Kyushu, Japan (2012), Journal Earth System Science 121, No. 4, pp. 1049 – 1057. 3. David M. Sherman. Electronic spectra of Fe^{3+} oxides and oxide hydroxides in the near IR to near UV (1995), American Mineralogist, Vol. 70, pp. 1262 – 1269. 4. Amro F. Alasta. Using remote sensing data to identify iron composite in central western Libya (2011), International Conference on Emerging Trends in Computer and Image Processing, Bangkok, pp. 56-61. 5. Trinh Le Hung (2013), Application of band ratio method to detect iron oxide, clay minerals and ferrous minerals, Mining Industry Journal, Vol. 4, pp. 19-24. 6. Trinh Le Hung (2013), A method for analyzing the principal components to detect clay

minerals and iron oxide distribution using images from LANDSAT satellite, Journal of Science, Ho Chi Minh National University of Education, Vol. 10, pp. 147-156.
7. Trinh Le Hung (2014), Application of remote sensing to detect iron composite and clay minerals based on the Crosta technique, Mining Industry Journal, Vol. 1, pp. 36-40.
8. www.thainguyen.gov.vn/.

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EVALUATION OF JET GROUTING EFFICIENCY AS A MEANS OF MINIMIZING THE HARMFUL IMPACT OF ESCALATOR TUNNEL

The article describes escalator tunnel construction using tunnel boring machines in Saint Petersburg. Rock mass jet grouting technology is used to reduce the harmful impact of underground construction on the earth surface in the mouth of the tunnel. Detailed analysis of the earth surface and rock strata subsidence data proved the effectiveness of the jet grouting technology. Subsidence of the surface and within the grouting area is almost zero.

The fact that the development of subsidence occurs mainly outside the zone of jet grouting soil is shown. A method for reducing subsidence by grouting additional unstable rocks in the massif, which will reduce the harmful impact of underground construction on the massif and the earth surface, is offered. To assess the efficiency of the proposed method, numerical modeling based on the finite element method was carried out. The modeling results showed that the proposed method reduces surface subsidence by 2-3 times.

Keywords: displacements and deformations, escalator tunnel, jet grouting, field survey data, modeling, finite element method.

Current projects related to the construction of underground escalator tunnels within the city limits face serious challenges due to engineering and geological conditions, the limited size of construction sites, and strict requirements for ensuring the safety of existing structures near the construction site. One of most effective solutions is to build escalator tunnels using power-driven tunnel boring machines (PTBM) with active face pressure support (Fig. 1).

The greatest impact on rock trough displacements on the Earth's surface is the physical and mechanical properties of rock mass and the displacement of the tunnel contour rock (convergence), determined by the boring process methods (weight pressure, level of grouting mortar outside construction zone). The construction of escalator tunnels using a PTBM in Saint Petersburg shows that strict adherence to process methods does not serve as a guarantee for preventing rock displacement across the tunnel contour. Currently, this technology is not capable of ensuring zero convergence, since rock subsidence in the arch crown outside the jet grouting zone is not less than 50–60 mm [1] in any conditions.

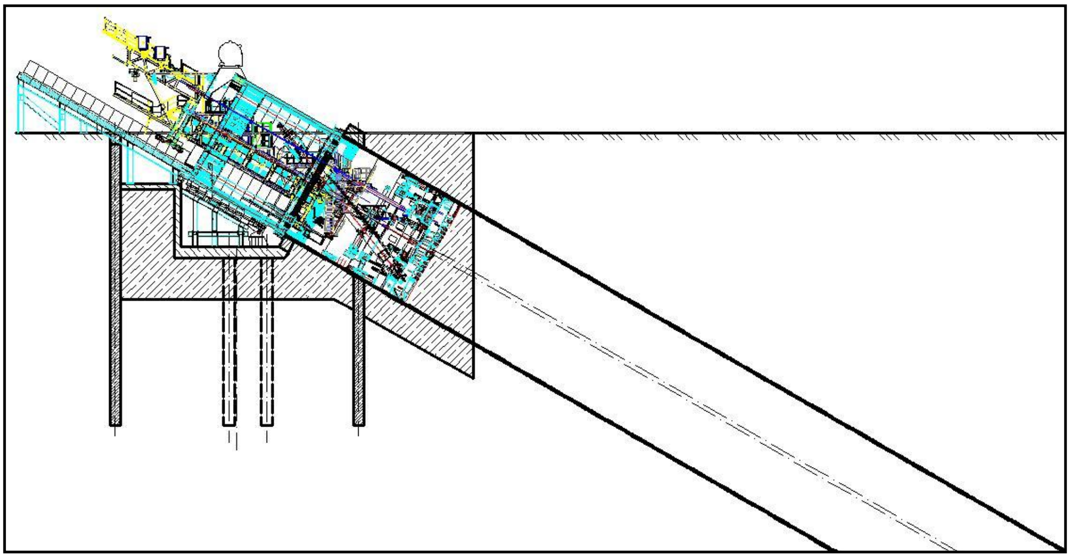


Fig. 1. Escalator tunnel boring with a PTBM

The main approach to reduce surface subsidence is to employ methods that help change the properties of geological material around the tunnel, for example, jet grouting around the tunnel mouth (Fig. 2). This strata consolidation technique is widely used due to its high process parameters and is described in various sources [2, 3]. As well as reducing surface subsidence level, this technology may result in major elevation of the grouting area (by 0.5–0.8 m), which places additional restrictions on this technology.

The field mining survey data obtained during the construction of Admiralteyskaya and Spasskaya underground stations prove the effectiveness of loose ground consolidation with jet grouting to reduce subsidence on the earth surface.

Field mining survey data related to soil subsidence levels, obtained using traditional geodetic methods (survey of subsidence level on wall check points, monitoring marks on building facades) and hole monitoring with extensometers were analyzed.

The method is based on the following approach: vertical holes with extensometers installed at various depths (spacing is approximately 10 m) are drilled along and perpendicular to the tunnel axis. These devices help obtain information about the displacement of the installation point with respect to the hole mouth. It should be noted that levelling hole heads is also very important to obtain information on point displacement in rock strata. The main distinctive feature of hole monitoring is the possibility to automatically collect continuous data which makes it possible to monitor occurrence and determine general mechanism for subsidence distribution in rock strata.

During construction of the escalator tunnel at the Admiralteyskaya underground station, hole E1 with extensometer was within the soil consolidation area (Fig. 3). The survey showed that point displacement in the consolidated area on the surface and in rock strata was practically zero. It can, therefore, be concluded that the jet grouting area fully prevented soil subsidence [4].

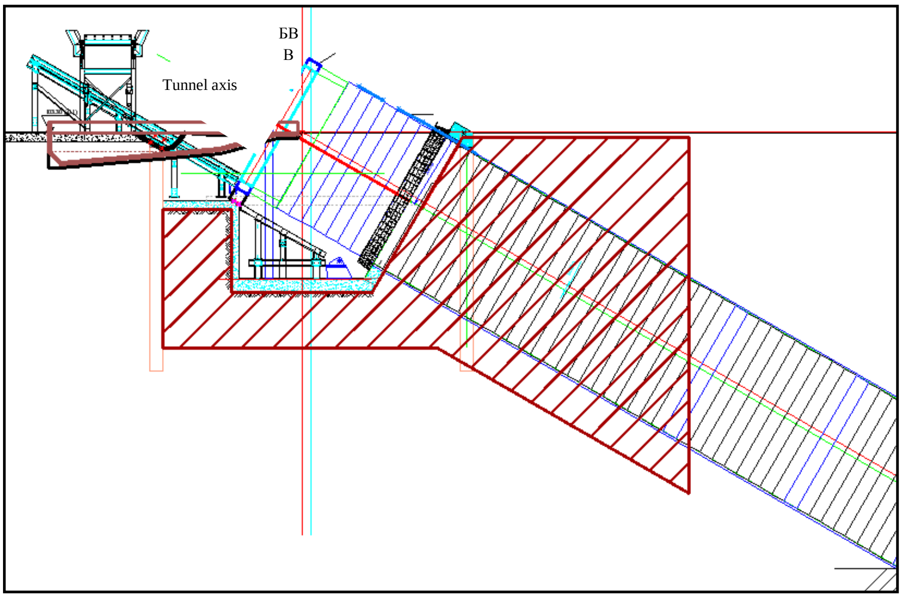


Fig. 2. Jet grouting of soils during construction of the escalator tunnel at the Spasskaya underground station – Cross-section along the tunnel axis

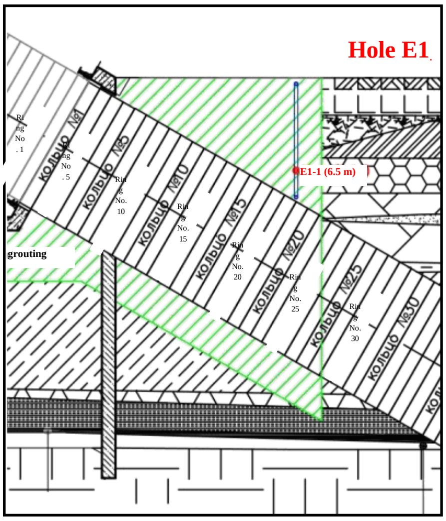


Fig. 3. Hole E1 with subsurface check mark (extensometer) at jet grouting section

Using hole check points to monitor soil subsidence shows that the most intensive geomechanical processes are being developed in the first part of the strata – along the tunnel length outside the jet grouting area. In this area, the subsidence level varies in depth, the maximum values in the tunnel arch are more than 100 mm, and the level reduces as it gets closer to the surface. During construction of the escalator tunnel at the Spasskaya underground station, the maximum earth surface subsidence (44 mm) was in the mouth of hole E1 located 7.4 m from the soil consolidation area.

The displacement and deformation mechanisms determined as a part of field survey activities allow us to provisionally separate the strata tunnel route into three sections subject to the subsidence intensity level (Fig. 4):

1. Minimum subsidence area – PTBM bores the construction site mouth jet grouting area. The subsidence level data obtained during mining survey activities lie within the limits of the measurement accuracy.

2. Large subsidence area starts where the boring machine exits from the grouted area to loose quaternary deposits and extends up to Proterozoic clay boundary. This area features major subsidence in the arch crown as well as large vertical on-surface displacements.

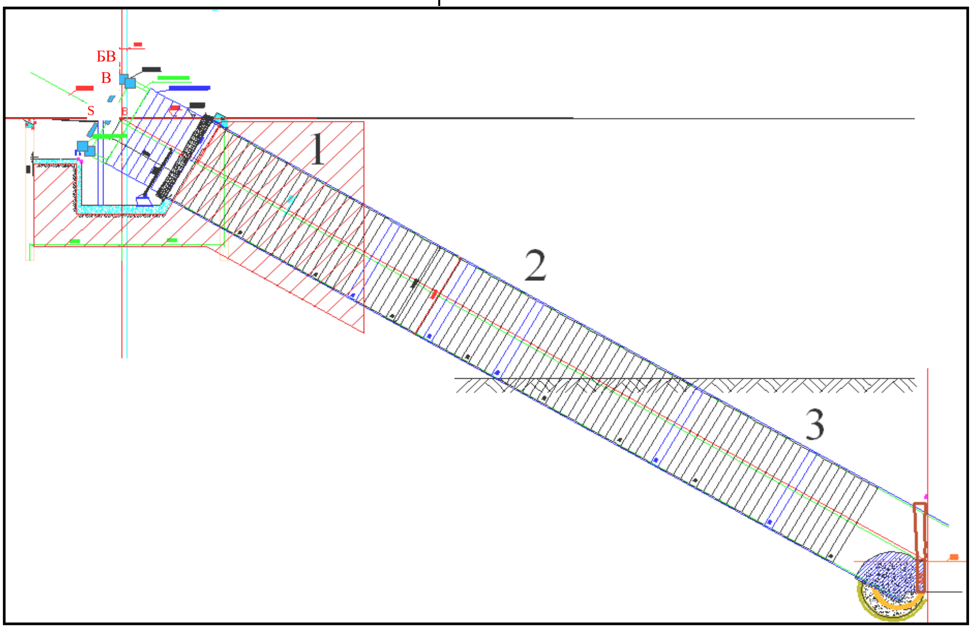


Fig. 4. Provisional sectioning of the massif during construction of the escalator tunnel at the Spasskaya underground station (in accordance with subsidence rate).

3. The subsidence decay area, where the PTBM enters solid clays with high deformation and strength characteristics (the total modulus of deformation exceeds the respective values for quaternary deposits by tens of times).

The subsidence decay area, where the PTBM enters solid clays with high deformation and strength characteristics (the total modulus of deformation exceeds the respective values for quaternary deposits by tens of times).

Despite relative deepening, section 2 has largest surface subsidence level. To reduce hazardous strata movement processes in this section, additional consolidation by jet grouting was proposed for the tunnel mouth (section 1) and in the strata depth up to the consolidated clays (section 2); it was also proposed to consolidate the soils without disturbing the sub-surface area (Fig. 5).

This approach would improve the mechanical properties of the massif near the tunnel and minimize the impact of jet grouting on the earth surface. "Protective" sub-surface mass makes it possible to reduce surface "swelling" which can occur when performing these works.

In order to analyze the effectiveness of the proposed method, 3D geomechanical modelling with Plaxis 3D software tool was proposed. This tool is an optimum solution for geo-engineering calculations; it is widely used around the world.

The created model (see Fig. 5) included the current mass consolidation method (section 1), mass section proposed for consolidation, tunnel lining constructed in a step-by-step manner. The total modulus of deformation for these sections is determined in accordance with test field data and taking into account actual physical and mechanical characteristics of the geologic material mass inside the tunnel. The total modulus of deformation values were checked on a previously created model for boring the Admiralteyskaya underground station escalator tunnel; on different models, these values reside in the interval of 200–500 MPa.

Analysis of the calculation results proves the effectiveness of additional soil consolidation of section 2. Fig. 6, *a* shows that major subsidence occurs within the section described above. As expected, the subsidence level is reduced in the additional consolidation area; the subsidence residual part is located in section 3. The calculation results show that the subsidence level can be reduced by 2-3 times compared to initial boring conditions (Fig.6, b).

In addition to the changed subsidence level we also should note the changes in distribution of vertical surface movements.

Here, the maximum subsidence levels areconcentrated near vertical projections of the

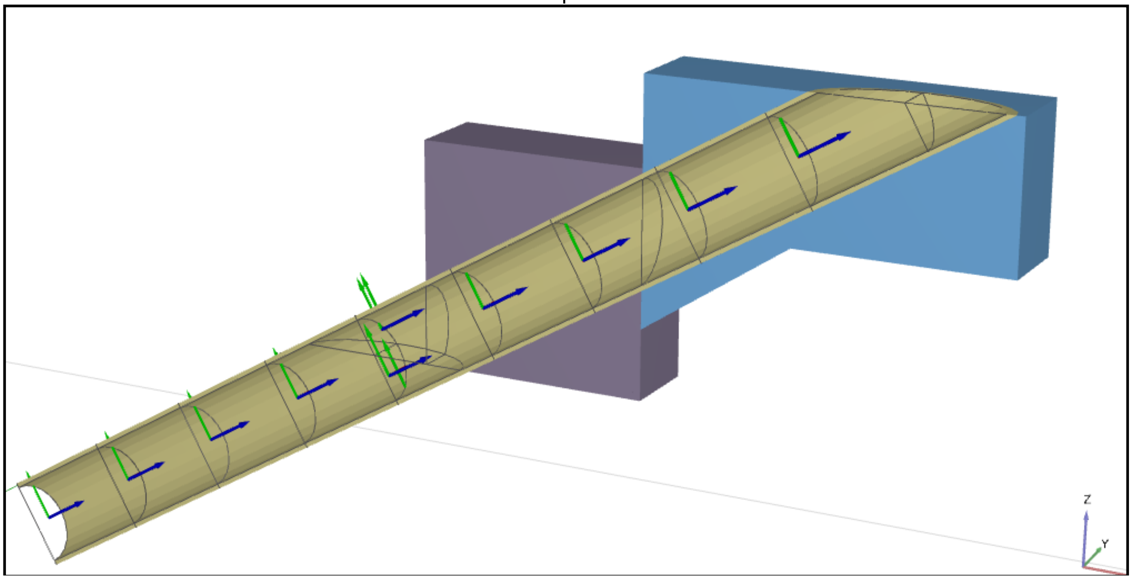
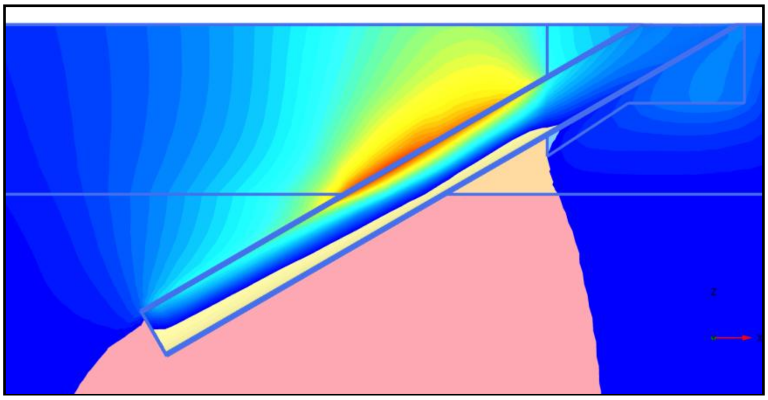


Fig. 5. Spasskaya underground station escalator tunnel model with consolidation of middle mass section.



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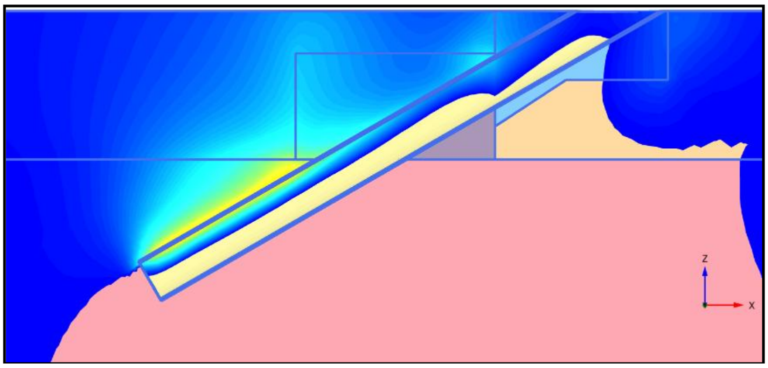
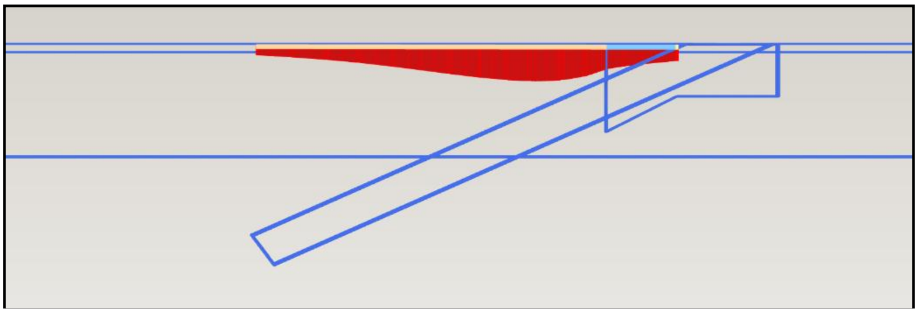


b

Fig. 6. Distribution of vertical movements (subsidence) during construction of Spasskaya underground station escalator tunnel:

a – without additional solid consolidation

b – with additional quaternary deposits consolidation.



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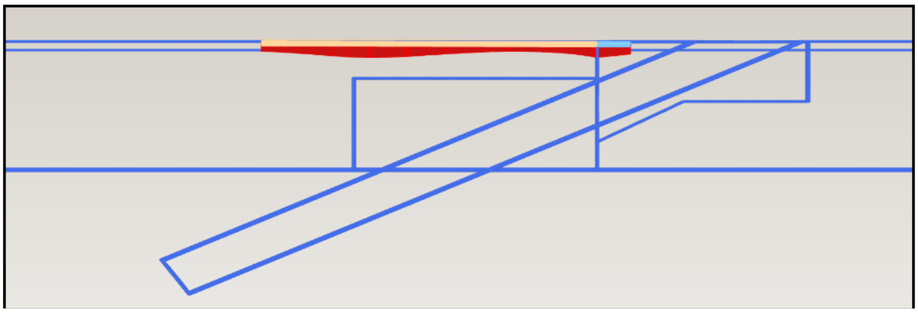

b

Fig. 7. Trough movement during construction of Spasskaya underground station escalator tunnel:

a – without additional solid consolidation

b – with additional quaternary deposits consolidation.

consolidation area boundaries; this is due to relatively active movements within these boundaries (Fig. 7).

The performed studies allowed a number of conclusions to be formulated.

Escalator tunnel PTBM boring with active face pressure support makes it possible to significantly decrease the harmful impact on buildings and structures during mining operations.

As experience has shown, despite a range of modern PTBM methods, it is impossible to fully prevent occurrence and movement of rock mass in strata and on the surface.

Major surface movement occurs above the escalator tunnel top section where the depth of the inclined working is not too great while the bearing strata are water-logged and unstable. In the bottom part of the tunnel, at the entrance to the bed rock, the deformation processes are practically zero.

Soil consolidation by jet grouting at the escalator tunnel mouth area ensures a significant decrease in the movement and deformation rate on the surface and during the construction process. This is proved by the field survey data and displacement mathematical modelling. The surface subsidence levels do not exceed several millimeters in the grouted area.

The positive effect of jet grouting (notable improvement of soil deformation and strength characteristics) which helps decrease deformations during boring may come to nothing due to major soil surface swelling during high pressure grouting.

This work proposes additional soil consolidation in the deep tunnel section, up to the bed rock. However, these works should be performed only in the deep tunnel section, without grouting the upper strata. The model calculation results show that such consolidation can dramatically decrease the main subsidence above the tunnel and minimize the harmful impact of jet grouting.

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“Gornye nauki i tehnologii”/ “Mining science and technology”, 2016, № 1, pp. 65–70	
Title:	**Evaluation of jet grouting efficiency as a means of minimizing the harmful impact of escalator tunnel construction**
Author 1	Name&Surname: **Volohov E.M.** Company: **National mineral resources university (Mining university)** Work Position: **Associate Professor** Contacts: **volohov@spmi.ru**
Author 2	Name&Surname: **Novozhenin S.U.** Company: **National mineral resources university (Mining university)** Work Position: **Assistant Lecturer** Contacts: **snovx@mail.ru**
DOI:	**http://dx.doi.org/10.17073/2500-0632-2016-1-67-72**
Abstract:	The article describes the escalator tunnel construction using tunnel boring machines in St. Petersburg. To reduce the harmful effects of underground construction on the Earth's surface in the mouth of the tunnel technology is used jet grouting rock mass. Based on the analysis of field data of the earth surface subsidence and rock strata proved the effectiveness of the jet grouting technology. Subsidence of the surface and within the grouting area, are approximately null. The fact that the development of subsidence occurs mainly outside the zone of jet grouting soil is shown. A method of reducing subsidence by grouting additional unstable rocks in the massif, which will reduce the harmful effects of underground construction for an massif and the earth's surface, is offered. To assess the efficiency of the proposed method, a numerical modeling based on the finite element method is carried out. According to the modeling results it is revealed that the proposed method will reduce the subsidence at the surface by 2-3 times.
Keywords:	displacements and deformations, escalator tunnel, jet grouting, the data of field observations, modeling, finite element method.
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INNOVATIVE, INTERACTIVE AND LANGUAGE DETERMINANTS OF INTERNATIONAL INTEGRATION OF RUSSIAN HIGHER MINING EDUCATION

Russian industry embarking on an innovative development path reflects world trends of international cooperation among higher education institutions dealing with engineering staff training and integration of higher technical education. Russia's joining the Bologna Process requires that higher schools prepare graduates, on the one hand, for an immediate start of professional activity at enterprises and, on the other, to ensure their mobility on the global labor market.

At the same time, Russian higher education institutions should be capable of training engineers from foreign countries, attaining the international level of skills training. The training of mining engineers in Russian technical universities cannot stand apart from the international integration of the educational process.

The generality of technological, production and administrative processes of the mining industry in different countries makes the international integration of the higher education of mining engineers in Russia especially topical. However, the low standing of Russian technical universities in the world ratings, the limited acknowledgment of Russian diplomas abroad, insufficient study motivation of students and poor foreign language skills all serve to constrain the process of international integration of Russian higher mining education.

The response to the challenges placed by globalization of the higher education system and the compelling need for the international expansion of Russian higher schools should become innovative, interactive and language determinants of this process. They represent the conditions that determine the prospects for integration of mining engineers' higher training in the international economic community. The given determinants should be implemented at the level of specific higher education establishments by using innovative pedagogical technologies, such as binary lectures, role-plays, brainstorming, case study, and project training in the educational process.

Keywords: educational integration, mining engineers' education, innovation, interactive, language determinants.

The prospects of Russian industry development have been determined as a transition to the postindustrial development stage of public production [7]. It is characterized by a high share of deep raw material processing, mass intellectual product generation on the one hand, and by "knowledge-driven economy" development and transformation of engineering universities into innovation-driven development flagships on the other hand [14, 17, 19, 20]. The scientific community has been actively discussing Russia's advantages, required for innovation-driven development: these are the comparatively high urbanization level, the USSR's considerable scientific and industrial heritage, including the developed sector of raw materials, and the relatively modern educational system for engineering staff [4, 5, 9, 13, 18].

The tendency towards the integration of industrial entities and higher engineering education institutions has been intensifying lately along with the globalization of national educational complexes and the development of international university cooperation. All of this has become part of the production and educational globalization process. Formation of the global educational system is fully reflected in the Bologna Agreements.

The following objectives can be set when translating the imperatives that the Bologna Process places upon Russian higher education in respect of mining engineers [1, 8, 15, 16]:

1. Not to extend the higher education period while cutting government expenditure for students and preserving the specialist education quality. This is of special significance for the

higher mining education, its quality determined by the competitiveness of the national raw material sector on the one hand and by occupational safety on the other.

2. To ensure the qualification level of graduates, allowing for their immediate engagement in the manufacturing process and control of complex processes applied at the modern mining enterprise.

3. To integrate the Russian mining engineer educational system into the international labor market with a view to reproducing modern specialists in the oil-gas, coal, mining, ferrous and nonferrous, energy, oil-chemistry and coal-chemistry industries.

4. To provide for the mobility of graduates with a view to advanced training and possible employment in any country, including in the European Union, through a system of educational grants for students from other countries.

5. To ensure compatibility and comparability of the national systems of higher engineering education and establish uniform educational standards.

Besides the above mentioned objectives, the Bologna Agreements established distinct requirements for contemporary higher education in the globalization environment, entailing the following changes. First of all, work has begun on introducing the European Credit Transfer System (ECTS). Second, university graduates will soon have to obtain a European diploma supplement, containing details about all the student's academic activities together with a letter rating of the student's knowledge, ranging from A (excellent) to F (unsatisfactory). Third, we are now observing the first steps in pursuing the policy of international openness of higher education institutions that should result in the growth of student and teacher mobility. And finally, there is the introduction of a point rating system for the evaluation of students' knowledge and self-educational activities.

However, the principal requirements placed on contemporary higher education are dictated by the international community of primary producers, where barriers on the means of international circulation of products and investments are relaxed and where international migration of specialists accelerates with every year.

Many graduates of modern engineering universities leave for on-the-job training to Germany, France, Great Britain, or China and find themselves in a cross-cultural and educational multi-platform community with coexisting modern and traditional knowledge transfer techniques, various educational rules and regulations, corporate culture and social responsibility standards.

The Russian higher education institutions are now at the initial stage of integration into the global educational community. At the same time, the low standing of Russian universities in the international rankings is a serious setback to such integration. Thus, for example, not one Russian university appeared in the top 200 world universities in the Times Higher Education-QS World University Rankings 2012–2013, the first positions being taken by the universities of the USA and Great Britain. This list included universities, for example, from China, Singapore, Australia, Belgium, South Korea and elsewhere [10].

Another problem for the integration of Russian engineering education is the limited recognition of domestic university diplomas when seeking employment abroad. Russian graduates have to confirm the level or their knowledge and skills through a system of additional tests. At the same time, the qualification of higher education specialists, including mining engineers, has been gradually declining in Russia. One of the reasons for this is the low learning motivation and insufficient preparedness of applicants for studying at higher education institutions. In fact, the Russian higher mining education will have to overcome the divergence of interests of higher education institutions as the sellers of educational services, enterprises as the educational customers and the government as the owner of institutes and universities, and pass through several levels of globalization of the educational sphere. The following package of integration processes is currently underway at Russian engineering universities:

1. At the academic level of a given university, a wider introduction of international practical engineering experience into the educational process has been implemented in the course of the project-based and case studies, and interactive communication of teachers and students.

2. There has been more extensive adaptation of curricula and educational programs to the requirements of specific enterprises acting as educational customers and target education of engineering staff has been developed. The entry of international-level manufacturers into the Russian markets imposes world-class requirements on the universities as regards the quality of engineering staff education, pushes universities to cooperate with scientific organizations, initiates the elaboration of original courses, and gives impetus for development of intracompany on-site training.

3. At the regional level, conditions for the development of interuniversity cooperation, scientific and research collaboration have been formed. At the same time, improvement of competition in the Russian and international markets of educational services raises the question of enhancing the competitiveness of engineering universities. Its primary factors are the innovative educational activity and ability of universities to participate in the development of the national innovative system.

4. At the national level, the system of Russian universities is now being adapted to international requirements regulating the quality of engineering staff education, to the competence level of graduates, and to technical and academic support of the educational process. To date, the integration processes at the national level have involved higher school legislation and laid the foundation for development of a number of strategic and policy documents. Here, the Education Development Federal Targeted Programme for 2011-2015 should be mentioned, which is aimed at "…ensuring the accessibility of high-quality education complying with the requirements of the innovation-driven socially-oriented development of the Russian Federation" [12]. Without doubt, the leading role is this process is played by the engineering universities, which should preserve and increase the scientific and production potential of the Russian economy. The discussion of problems facing the higher education system for engineering staff in Russia is aimed at generating a proposal to the Government of the Russian Federation and Ministry of Science and Education of the Russian Federation regarding modernization of the engineering education system which is much-needed for staffing support of the import substitution and rehabilitation of the industry on a new technological base.

5. At the international level, globalization of the educational system is a kind of response to integration processes in the world economy, formation of a common information space, and international labor migration. For the Russian engineering universities, educational globalization provides an opportunity to act within the framework of a space and knowledge paradigm of social and economic development. This means that higher engineering education, owing to its applied nature, is becoming an important factor in production and generates a guaranteed income for its owner irrespective of the country where the engineer is employed.

Thus, the Russian engineering universities that train mining industry personnel have now encountered two fundamentally new international integration challenges.

On the one hand, globalization of the educational sphere as a response to economic globalization makes mining education unitary by imposing similar educational requirements on engineering staff in different countries. This process is facilitated by the application of similar processes and technologies, industrial control and management techniques at the mining enterprises in Russia, countries of Eastern and Western Europe, North and South America, Africa, Australia, China, etc. This gives new impetus to universities in various countries to integrate their scientific and educational activities and implement new projects aimed at improving the quality of higher mining education.

On the other hand, integration of the Russian engineering universities that train mining engineers into the global educational system is hindered by problems that decrease their competitiveness in the global educational market. These problems relate to the long-term isolation of Soviet and Russian higher education from worldwide standards, degradation of the academic science during the period of reforms, marginalization of pedagogic work and the loss of its social status, and the non-innovative nature of the Russian economy. As a result, many engineering universities are busy simply surviving on the educational market by providing non-core educational services (economic, managerial, judicial) and have no staff, financial, material or technical resources for transformation into flagships for promoting the development of the innovation-driven process in the industry.

Along with this, the teaching of mining engineers in Russia is characterized by insufficient implementation of the intrauniversity capabilities for integration into the global educational space: familiarization with the latest technological innovations in the industry, language proficiency, and the practice of international student exchange with leading mining education centers in the USA, France, Germany, and China.

The fact that most Russian universities training mining engineers are still not able to address adequately the above challenges hinders their transformation into national innovation centers, lowers their positions in the international university rankings, and complicates the formation of teaching staff to satisfy international requirements.

Therefore, conditions should be created to facilitate the integration of Russian engineering universities into the international environment of higher mining education. These governing conditions – determinants of the integration process – relate not just to the external environment (joining the Bologna Agreements, international student exchange and education of foreign students). The starting point should be the introduction of international requirements for the higher educational process proper. We believe the determinants for integration of Russian universities into the international mining education system that relate to the educational process proper are the innovative, interactive forms of its organization and the development of linguistic competence up to a level enabling Russian graduates to be equally competitive on the global labor market.

The innovative determinant includes a continuous creative search of university teachers, familiarization with advanced processes and technologies, and diffusion of the new knowledge within the teacher community. In fact, we are talking about creating a sound innovation teaching unit in the mining education system. An important part in this process should be given to the innovative technologies of teaching proper, in particular, the switch to new forms of lecture material presentation – problem-based and binary.

At the contemporary stage of mining science, development the traditional lecture is less effective in communicating new information as compared to scientific publications, the sharing of experience on introducing new processes and technologies within the professional mining community, patenting of inventions and diffusion of innovations within industries. Therefore, the traditional form of lectures – monotonous presentation of information by the teacher – should be replaced with lectures engaging students in collective collaboration with the teacher and each other, in discussions of problems and prospects of introducing innovations into mining operations.

For instance, during a problem-based lecture the teacher and students are involved in an active scientific and educational process, provided the lecture is conducted in the form of a live dialog. The subject of such dialog may be the discussion of an effect resulting from the introduction of innovations, replacement of obsolete equipment, or the use of new progressive forms of industrial management.

Thus, for students of the major discipline 21.05.04 "Mining engineering" of the "Open-pit mining" specialism, the issues selected for problem-based lecture discussion can be the objectives of improving the environmental friendliness of mining operations and cleaning of polluted water bodies [11, 21], increasing equipment capacity, and enhancing product quality.

The teacher raises issues for discussion involving the entire audience by putting questions to the attendees and finding answers together with them. While answering, the students develop their engineering thinking, display responsibility and defend their point of view. The principal organizing role in conducting a problem-based dialog lecture lies with the teacher as it is his/her ability to hold a discussion and pursue interaction with the students that has a decisive impact on their active study of the innovation-driven process in the industry.

Another example of teaching technology implementing problem-based and dialog-oriented principles is a binary lecture. This is essentially the work of two teachers who simultaneously conduct a lecture in the same subject and interact one with another and with the audience on the basis of problem-organized material. During the dialog between the teachers and students, objectives are set and a problem situation is analyzed, hypotheses are made, proved or rejected, contradictions are solved, and solutions are found.

To prepare mining engineers for open-pit mining operations, it is important that the practitioners – the employees of surface coal mines, processing plants, and power engineers – are involved in the educational activity. It is these practitioners who can get the message about the horizons of innovation-driven activity in the mining industry, challenges and prospects of introducing innovations at certain enterprises across to students and share their views. However, practicing mining engineers usually have no teaching or lecturing experience and are therefore reluctant to participate in the university educational process.

This is precisely why binary lectures conducted by a university teacher jointly with an engineer or a manager from a mining enterprise satisfy to the maximum extent the conditions governing the inclusion of the educational process into the innovative process. The binary lecture process as part of mining engineer training implies the following:

– selection of a relevant subject, disclosing both traditional and innovative issues of mining technology or different points of view;

– selection of two teachers who are compatible both in terms of thinking style and means of interacting;

– elaboration of the lecture scenario (thesis plan, content blocks, time distribution, etc.).

Of course, the lecture as a form of teaching plays an important part in the university education of mining engineers. At the same time, we should note the significant importance of practical studies, which are often conducted in a traditional manner – solving calculation problems and providing answers to questions.

Therefore, the interactive determinant of the integration process in the university education of mining engineers includes the implementation of specific teaching techniques aimed at enhancing the students' acquisition of professional competences and promoting their interest in innovations. Such methods include organizational-activity games, brainstorming, and case studies.

A specific feature of an organizational-activity game is that it is aimed mainly at obtaining a certain final result in discussing a problem. This result can be the development of a new equipment introduction project, elaboration of new mining operational plans, the introduction of a new bonus system for employees of an enterprise, implementation of occupational safety improvement measures, and so on.

Therefore, the primary characteristics of such games are maximum approximation to actual industrial management problems, the conventional nature of roles, a shared objective for the whole group, and decision-making teamwork.

Another equally important interactive method in university engineering personnel teaching is brainstorming. It enables the generation of a significant number of solutions for a professional problem and their critical review within a short period of time. After a detailed review by the participants of their own ideas, they are to find the best one for the given situation.

The range of problems that future specialists in open-pit mining operations can solve through brainstorming include the response to emergency situations, increasing labor productivity, the selection of new domestic and foreign equipment, improvement of product quality, and so on.

It is customary in scientific literature to highlight the following brainstorming mechanism: formulation of the problem to be brought up for discussion; selection of the brainstorming leader; selection of two secretaries to record the proposed ideas; the brainstorming session; review of the results and selection of the final problem solution. The brainstorming session should be conducted in three stages: introductory briefing (5 to 10 min), working session (10 to 20 min), final stage – selection and discussion of the final option (10 to 15 min).

The case study technique, i.e. the study of practical situations that have predominantly occurred in real life, is based on the discussion by a group of students of the problems actually encountered by mining enterprises and finding solutions. It is desirable that these problems are formulated with the involvement of practicing mining engineers, in order to convey the message to students on the importance of taking weighted and reasoned engineering decisions and on the corresponding responsibility.

Each problem situation (case) should contain the following: a detailed description of the source of the problem and comments related thereto as well as the role of certain managers and engineering staff. In the context of mining enterprises, problem situations may pertain to staff relations, adaptation of employees to new duties and positions, measures to introduce innovations, install new software, and on problems arising in the course of equipment operation, and during mining activities [3].

The primary advantages of the case study technique as part of the university education of mining engineers are as follows:

1. The ability to “immerse” students in an actual complicated situation, which is typical of future professional activities.

2. Improvement of teaching efficiency through a more intensive digestion of educational materials as a consequence of indicative visualization of the problem.

3. Emotional involvement of students in the teaching process and increased motivation to study the subject due to its obvious practical usefulness.

4. Enhanced formation of practical skills and professional competences in the course of case-study teaching.

Along with the above said, the most important factor of the international integration of Russian mining education is the mobility of students and teachers of engineering universities. This increases the significance of the language determinant of the integration process, which is a package of skills enabling an engineer to solve the personal and professional tasks of communicating in a foreign language. Implementation of the language determinant enables a student to “absorb” innovative technologies, obtain data on advanced equipment and its application prospects, and procure information on improving the mining machinery and manufacturing processes. A significant component of the linguistic determinant is the lexical and grammatical knowledge necessary for obtaining information from foreign-language sources [2].

The international transfer of new technologies, universal employment of the equipment produced by world's leading manufacturers, entry of the Russian engineering universities into the global market [22] cumulatively resulted in the need for preparing the mining engineers with profound foreign language proficiency. However, despite the undeniable importance of knowing a foreign language, the Russian universities engaged in teaching mining engineers reduce the number of academic classes for studying foreign languages.

In this case, the problem of implementing the interactive determinant arises (application of game-based, brainstorm, and case-study techniques) during practical training on foreign-language subjects. Due to the reduction of the number of classroom hours, this process is shifting towards self-educational and non-classroom activities of students, where the project-based techniques are one of the forms.

The project-based educational technique involves non-classroom activities of students who acquire production information in the foreign language on the professional topics set by the teacher and then analyze this information. The result of the project-based activities is the elaboration of projects for the development of a mining enterprise and its site and presentation of these projects. On the one hand, the importance of preparing such projects in a foreign language is conditioned by the need to use the scientific and production potential accumulated in the mining engineering development abroad. On the other hand, it is precisely the “project” thinking that enables university graduates to successfully find employment and climb the career ladder at international industrial companies.

Thus, the objectives that the Bologna Process sets for the Russian higher education institutions involved in teaching mining engineers are inseparably associated with the integration of the domestic higher engineering education system into the international system. Successive solution of the said objectives is constrained by such problems as the degradation of academic science during the period of reforms, long-term isolation from worldwide standards, and the underfunding of universities. Therefore, an important driver in the international integration of the mining engineer education is the international requirements – innovative, interactive and language determinants. Their implementation requires the development of educational techniques at university level and a focus on language competences as well as involvement of practicing mining engineers in the educational process.

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DOI	**http://dx.doi.org/10.17073/2500-0632-2016-1-74-81**

Abstract:	Russian industry embarking on an innovative development path reflects world trends of international cooperation among higher education institutions dealing with engineering staff training and integration of higher technical education. Russia's joining the Bologna Process requires that higher schools prepare graduates, on the one hand, for an immediate start of professional activity at enterprises and, on the other, to ensure their mobility on the global labor market. At the same time, Russian higher education institutions should be capable of training engineers from foreign countries, attaining the international level of skills training. The training of mining engineers in Russian technical universities cannot stand apart from the international integration of the educational process. The generality of technological, production and administrative processes of the mining industry in different countries makes the international integration of the higher education of mining engineers in Russia especially topical. However, the low standing of Russian technical universities in the world ratings, the limited acknowledgment of Russian diplomas abroad, insufficient study motivation of students and poor foreign language skills all serve to constrain the process of international integration of Russian higher mining education. The response to the challenges placed by globalization of the higher education system and the compelling need for the international expansion of Russian higher schools should become innovative, interactive and language determinants of this process. They represent the conditions that determine the prospects for integration of mining engineers' higher training in the international economic community. The given determinants should be implemented at the level of specific higher education establishments by using innovative pedagogical technologies, such as binary lectures, role-plays, brainstorming, case study, and project training in the educational process.
Keywords:	educational integration, mining engineers' education, innovation, interactive, language determinants.
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