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Evaluation of the energy efficiency of rotary percussive drilling using dimensionless energy index

V.N. Oparin, V.N. Karpov, V.V. Timonin, A.I. Konurin*

Chinakal Institute of Mining, Siberian Branch, Russian Academy of Sciences, Krasnyi Prospect 54, Novosibirsk, 630091, Russia

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ABSTRACT

This paper sets forth a geomechanics framework for assessing the energy efficiency of rotary percussive drilling using the energy criterion, which has been proposed by Victor Oparin for volumetric destruction of high-stress rocks having nonuniform physico-mechanical properties. We review the long-term research and development in the specified area of science and technology, including research and development projects implemented at the Institute of Mining, Siberian Branch of the Russian Academy of Sciences. A new modified expression of Oparin's dimensionless energy criterion of volumetric rock destruction k is introduced. The range of in situ values is determined for the energy criterion of volumetric rock destruction at the optimized energy efficiency of rotary percussive drilling. The temporo-spatial intervals of geotechnical monitoring are found to control pneumatic drilling energy efficiency at subsoil use objects in Russia. The integrated experimental, theoretical and geotechnical approach to the comprehensive investigation of real-time processes of rock fracture in rotary percussive drilling using the energy concept possesses the necessary geomechanical performance-and-technology potential to create the next level geotechnical monitoring of drilling systems for various purposes, including determination of physico-mechanical properties and the stress-strain analysis of rock mass in full-scale drilling.

1. Introduction

The critical link of a technology chain in mineral mining is drilling for implementing exploration, blasting, gas/water drainage, processing and mine rescue procedures. The highest percentage in the scope of drilling in the subsoil use belongs to drilling of blast holes which reach hundreds of thousand meters in total length annually in mines. Drilling in hard rocks is carried out primarily by rotary percussion in Russia and abroad, using drilling machines equipped with offset or down-the-hole hammering tools (e.g. Tanaino and Lipin, 2004; Wijk, 2008; Fox, 2011). High productivity of drilling, at an optimized cost of direct expenses in specific geological and geotechnical conditions of mineral deposits, is a challenging issue both for large mines and small agencies engaged in the provision of drilling services in Russia (Eremenko et al., 2015; Sobolevskyi et al., 2017).

An essential condition of reducing the cost of drilling and, accordingly, in the cost of minerals and engineering materials is lessening the energy input of rock fracture. To this effect, it is possible to improve the design of a drilling tool, or to enhance its impact capacity at efficient coupling of impact energy and frequency. Consequently, an important condition of rotary percussion drilling effectiveness is the correct feasibility study of drilling within a specific mine infrastructure. It is crucial to determine effectual drilling modes for specific physico-mechanical properties.

The rotary percussion drilling practices are currently provided with ample guidelines on the selection of offset and down-the-hole (DTH) hammering and drilling tools and the determination of operation conditions. However, designers of drilling systems emphasize the greatly generalized nature of these guidelines (Atlas Copco, 2002; Halco Rock Tools, 2016). In many ways, such generalization is conditioned by rocks' various physico-mechanical properties at drilling sites. Accordingly, it is routinely required to adapt drilling machines to essentially varying geological and geotechnical conditions of the subsoil use in Russia. Moreover, it is necessary to train drilling equipment operators to employ the machines skillfully. In this respect, considerable attention is paid to

* Corresponding author.
E-mail address: akonurin@yandex.ru (A.I. Konurin).
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understanding the mechanism of rock fracture in rotary percussion drilling. Hartman (1959)'s law of rock fracture is often used (Fig. 1).

The sequence in Fig. 1 graphically proves that percussion is the basic mode of drilling (Kwon et al., 2013; Hung et al., 2016). At the same time, it is known from practice that an increase in the rotation frequency (f_{rot}) over a certain critical value leads to an increase in the wellbore area per each drill bit owing to the higher rotational angle between impacts (γ), which is connected with increased wear of indenters (drill bits). A 'super-high' impact (N) results in rock cutting, which also accelerates the failure of carbide indenters. Consequently, given the irrational choice of the parameters f_r and N, the above effects significantly contribute to the wear of carbide drill bits and overlap during drilling. Thus, the operating conditions in rotary percussion drilling using offset and DTH hammering tools should be carefully selected, especially in the case of large diameter drilling (Fox, 2011; Atlas Copco, 2015; Halco Rock Tools, 2016).

This means that drilling procedures should be adapted to specific geomechanical conditions of solid mineral mining, both at the beginning and at the close of each working shift, to ensure an optimum penetration rate at a minimized flow rate of energy source and lowest wear of rock breaking tool.

Accordingly, modern drilling technology and equipment require a theoretical and applied framework for the adaptive quantification of efficiency of different drilling systems in operation in variable full-scale conditions. Such framework is also necessary for the firmware engineering for self-testing of drilling machines to identify an energy-effective mode of rock fracture towards enhanced drilling performance in variable geological and geotechnical conditions of solid mineral mining.

2. New approaches to a procedural framework for in situ geotechnical monitoring of drilling performance

Researchers from different countries have been striving to optimize fracture conditions in rock drilling in various geological and geotechnical conditions for many decades. Mining involves a wide range of technical challenges to be met, starting from the surface and underground mine engineering to mine rescue operations (Lukosavich, 2010; Ren et al., 2018; Zhang et al., 2021). Continued expansion is observed in the research aiming to enhance energy efficiency and output of drilling, including technologies connected with real-time determination of physico-mechanical properties of rocks (strength, jointing, abrasiveness, etc.) during operation of drilling machines at various objects of the subsoil use (Ghosh et al., 2017; Abu Bakar et al., 2018; He et al., 2018; Li and Zhan, 2018; Xiao et al., 2018a; Regotunov and Sukhov, 2019; Zhang et al., 2019).

As sensing devices came into use in experimental data recording in mineral exploration, drilling and extraction, the mining and petroleum industries have accumulated ample experimental evidence of rocks' physical, mechanical and acoustic properties. These data are applicable in the analysis of seismic and microseismic information for the feasibility analysis of geotechnical monitoring, performance optimization of drilling machines, and the enhancement of safety of the work environment in mines (Mohammadpoor and Torabi, 2020).

The modern instrumentation used in measurement while drilling (MWD) provides real-time information on rocks' physico-mechanical properties. This information is essential for engineering design and online decision-making on adjusting conditions of technological processes (Isheyskiy and Sanchidrián, 2020). The standard method of physical information acquisition is the measurement of acoustic emission in rocks during drilling. In this process, it is possible to control the dynamic displacement of the drill bit–rock interface, static and dynamic loads on the drill bit, and the penetration rate using appropriate laser sensors (Xiao et al., 2018b).

The artificial intelligence techniques are successfully applied to solving a wide range of problems in rock mechanics as the conventional (empirical, mechanical–mathematical and statistic) methods (Lawal and Kwon, 2021). The express-analysis of experimental data can combine the artificial neural network method, binary classification, and borehole monitoring of drilling tool parameters to model and predict work-related accidents. Changing the work mode of a drilling tool in a borehole can be an early warning of an accident and can help prevent it (Muojeke et al., 2020).

In Russia, specific emphasis is laid on the necessity for integrated studies into drilling processes with due regard to a wide range of physico-mechanical properties of high-stress rock masses for substantiation of engineering factors and process variables of drilling machines. The point is that foreign and domestic drilling equipment, which are expensive as a rule, offers poor performance in the actual mining conditions in Russia. From practical evidence, the cumulative quantitative indicator of drilling equipment efficiency in full-scale mining conditions in Russia is the total energy input of accident-free drilling under conditions of highly variable physico-mechanical properties of rocks and their lithological types often inadequately predicted from structural features. In such situations, the study of highly complex and nonlinear interaction process between rock-breaking tools of drilling machines and high-stress rock mass structured as a hierarchy of blocks from supermolecular to macro-scale acquires strategic importance for science and technology.

Primarily, this relates to advanced super-long deep-hole drilling for various purposes. In this case, the simplified scheme of rock fracture under the dynamic impact of solid indenters (Fig. 1) becomes inadequate for describing the interaction between the medium and the geotechnical system. The current scientific achievements in nonlinear geomechanics and geophysics of rock fracture at different scales in the zones of increased concentration of elastic energy generated by various sources (earthquakes, rockbursts, explosions, etc.) implicate their effectiveness in new-generation drilling systems (Adushkin and Oparin, 2012, 2013, 2014, 2016; Oparin and Adushkin, 2018, 2019).

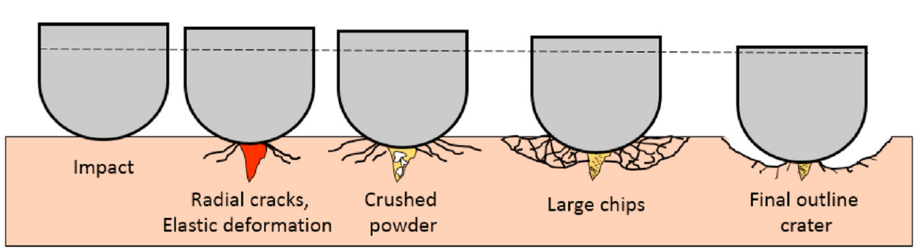


Fig. 1. Rock fracture during center formation in rapid penetration of indenter.

Let us take, for instance, the generalized analysis of the theoretical and experimental researches in order to understand the influence of physico-mechanical properties of rocks on drilling efficiency from the viewpoint of energy. The analysis addresses efficiency evaluation of different drilling techniques with regard to the physico-mechanical properties of rocks. Such studies can be used to determine modes and process variables of drilling, to examine feasibility of drilling paths, or to develop a mixed-type classification of rocks by their physico-mechanical properties and drillability, with the adaptable scale of the latter.

The physico-mechanical properties of high-stress rocks greatly influence the drilling quality. This fact is commonly known. However, there is no recognized approach to representing these properties in an integrated geomechanical–geotechnical classification. With this end in view, the multi-factor approach has been proposed in Tanaino (2008) to rock drillability classification based on a canonical scale put forward by Oparin and Adushkin (2018, 2019) for the hierarchical representation of physical and structural properties of rocks and rock masses by dimensionless indicators of the resistance power of rocks uniformly based on their strength, jointing, grain size, hardness and porosity. The resultant analytical expression is based on the fundamental dependence of rock fracture quality on the hierarchical structure of rocks and rock mass at different scales (Vikulin and Ivanchin, 2013) according to the scale effect in the phenomenon of zonal disintegration of rocks around underground voids and excavations.

Drilling various purpose boreholes is a process widely applied in mining, geological exploration, construction, and other industries. Nonetheless, this art of engineering is still in many ways based on personal experience (Aldred et al., 2012). Researchers, engineers and designers face a complicated problem: finding and utilizing a 'feedback' between nonlinear geomechanical processes of high-stress rock fracture and process flows involved in rock fracture at optimized energy consumption (Oparin et al., 2016; Neskoromnykh and Popova, 2019).

Thus, the novel approaches to integrated geomechanical and geotechnical monitoring for the drilling equipment efficiency rating in terms of energy in real-life mining is a critical and strategic course of drilling improvement. At present, the Institute of Mining, Siberian Branch, Russian Academy of Sciences is developing new methods and techniques to assess the efficiency of various drilling systems in real-life mining environments (Oparin et al., 2017; Sklyanov et al., 2018; Kondratenko, 2020) based on the energy consumption of this process concerning strength and the hierarchical block structure of high-stress rocks and rock masses.

To this end, the authors have developed an integrated procedure of energy efficiency estimation in rotary percussive drilling with DTH. The process considers the stress-strain behavior of rock mass and the physico-mechanical properties of rocks. A feature of the procedure is the integrated energy-based estimate of the drilling process, including possible (stochastic) effects of natural, technical and technological factors in the aggregate as they directly relate to the mechanism of volumetric rock destruction.

The energy efficiency of rock fracture is estimated in the framework of nonlinear geomechanics in connection with the discovered phenomena of the alternating response of rocks to dynamic impacts and nonlinear pendulum-type elastic waves (Oparin and Adushkin, 2019; Wang et al., 2019). The discovery of these phenomena provided the scientific ground for their application in the analysis of complex processes of rock fracture in concentration zones of higher stresses and strains (physical shocks, explosions, earthquakes, rockbursts) with a view to rockburst-hazard prediction and prevention, mine support design, etc. (Qian et al., 2012; Oparin et al., 2015; Yuan and Xu, 2018). The application of these phenomena in the estimate of energy efficiency of rotary percussive drilling is possible through Oparin's dimensionless energy criterion of volumetric rock destruction (k).

The theoretical and experimental research findings (Oparin et al., 2017; Karpov et al., 2019) point out the criterion's efficiency in shaping an energy concept for the various purpose of geotechnical monitoring. The authors of this study describe the modern approaches to geotechnical monitoring systems based on long-term investigation results obtained in commercial approval of the new procedure for DTH drilling energy efficiency estimation in mines using the criterion mentioned above.

In this regard, we connect the potential advance with the dimensionless energy criterion of volumetric destruction of high-stress rocks having various physico-mechanical properties and nonuniform internal structure. The criterion was derived by Oparin and Adushkin (2018, 2019) from generalization of energy conditions of earlier discovered nonlinear pendulum-type elastic waves, subnormal friction in geomedia and 'geomechanical quasi-resonances' in rock failure in concentration zones of higher stresses and strains during earthquakes, rockbursts and explosions of different forces (up to nuclear). The evolution of this concept in the theoretical and experimental researches is sufficiently amply described in Adushkin and Oparin (2012, 2013, 2014, 2016), Oparin and Adushkin (2018, 2019), Shemyakin et al. (1992), Karpov and Timonin (2018), and Kurlenya et al. (1998). Below in this paper, we will briefly expound some notions required to understand the essence of modern achievements of nonlinear geomechanics in the context of energy framework for the next-level multipurpose drilling equipment engineering.

3. Energy conditions of volumetric rock destruction in high-stress concentration zones

For the description of complex and dynamic processes of rock fracture in concentration zones of high stresses and strains (mechanical shocks, explosions, earthquakes, rockbursts) in high-stress rock masses of hierarchical block structure, with regard to their physico-mechanical properties, Victor Oparin has introduced a generalized dimensionless energy criterion $\hbar$ of nonlinear pendulum-type elastic waves and geomechanical quasi-resonances (Kurlenya et al., 1997; Adushkin and Oparin, 2012, 2013, 2014, 2016; Oparin and Adushkin, 2018, 2019):

$$\hbar = \frac{W}{M\nu_{\mathrm{p}}^2} = (1-4)\times 10^{-9} \tag{1}$$

$$M = \rho V,\ W = \alpha(V)(U_0 + W_{\mathrm{k}}) \tag{2}$$

where $U_0 + W_{\mathrm{k}}$ is the sum of elastic (potential) and kinetic energies of structural elements of rocks in the fracture source zone (J), $\alpha(V)$ is the coefficient of load induced by earthquakes and blasts (Sadovsky and Nersesov, 1974), V is the volume of rock destruction zone (focus) (m^3), ρ is the density of rocks ($\mathrm{kg/m}^3$), ν_{p} is the P-wave velocity (m/s), M is the mass of rocks in the focus of volumetric destruction, and W is the energy characteristic of rock fracture zone. The experimental values of ν_{p} can be replaced by the known analytical expressions for P-wave velocities, including physico-mechanical characteristics of rocks (Young's modulus, Poisson's ratio and bulk density) and three-dimensional configuration of waveguide structures (Oparin and Adushkin, 2019).

Eqs. (1) and (2) were verified using the experimental and theoretical data on earthquakes of different energy classes, blasts with explosives of widely ranged weights (Shemyakin et al., 1986) and rockbursts (Lovchikov, 1997). This data sampling covers a range of energy from tens to 4.2×10^{12} J, and is quite representative and valid for different depths and regions of the world. The reference sources sometimes lack data on P-wave velocities and densities of geo-substances, which are necessary for using Eq. (1). For this reason, we performed calculations with the averaged values of these characteristics: $v_p = (3-5) \times 10^3$ m/s and $\rho \approx 2.8 \times 10^3$ kg/m^3.

3.1. Crustal earthquakes and underground explosions

Tsuboi (1956) first cognized a simple relationship between the earthquake energy E_c and the earthquake volume V_c:

$$E_c = aV_c \tag{3}$$

where a is the density of seismic energy.

Later on, Sadovsky and Nersesov (1974) derived the same relation at $a = 1 \times 10^3$ (with energy in erg (1 erg = 1×10^{-7} J) and volume in cm^3). The comparative assessment of the dynamic parameters of crustal earthquakes and underground explosions was undertaken in Sadovsky and Nersesov (1974) at the assumed equality of a at these sources of seismic vibrations (E_c, V_c and the rupture length L_c depending on the total source energy E_0). The congruent values were obtained for the earthquake and explosion sources with the energy $E_0 = 4.2 \times 10^{19}$ erg, 4.2×10^{20} erg, 4.2×10^{21} erg and 4.2×10^{22} erg, or Y = 1 kt, 10 kt, 100 kt and 1000 kt of TNT equivalence, from the formulae below.

For dense rocks, we have

$$\left.\begin{array}{l} \log_{10}E_c = 1.451\log_{10}E_0 - 11.5 \ (\log_{10}E_c < 20.5) \\ \log_{10}E_c = 1.031\log_{10}E_0 - 2.06 \ (\log_{10}E_c \geq 20.5) \end{array}\right\} \tag{4}$$

For loose rocks (explosions), we have

$$\log_{10}E_c = 1.181\log_{10}E_0 - 6.55 \ (\log_{10}E_c \approx 20.5) \tag{5}$$

These data are compiled in Table 1. Considering mathematical notation assumed in Sadovsky and Nersesov (1974), Eq. (1) takes following form:

$$k = \frac{\alpha E_0}{\rho V_c v_p^2} \tag{6}$$

The value of dimensionless energy criterion of volumetric rock destruction k from Eq. (6) using the tabulated data, $E_0 = 4.2 \times 10^{19}$ erg $= 4.2 \times 10^{12}$ J and $v_p = (3-5) \times 10^3$ m/s is given by

Table 1
Dynamic parameters of crustal earthquakes and underground explosions (Sadovsky and Nersesov, 1974).

Y (kt)	E_0 (erg)	$\log_{10}E_c$ (erg)	V_c (cm^3)	L_c (km)	Estimated coefficient of underground blast-induced load (%)		
					α (for any rocks)	α_1 (for loose rocks)	α_2 (for medium-compact and dense rocks)
1	4.2×10^{19}	16.6–17 (16.8)	6.3×10^{13}	0.7	0.15	0.1	0.3
10	4.2×10^{20}	17.8–18.4 (18.1)	1.25×10^{15}	1.8	0.3	0.15	0.8–1
100	4.2×10^{21}	19–19.8 (19.4)	2.5×10^{16}	5	0.6	0.2	2–4
1000	4.2×10^{22}	20.2–21.2 (20.7)	5×10^{17}	13.5	1.2	0.3	3–5

Note: In the third column, the range of the values corresponds to the loose and dense rocks, and the number in the round brackets represents the average value.

$$k = \frac{0.15 \times 10^{-2} \times 4.2 \times 10^{12}}{2.8 \times 10^3 \times 6.3 \times 10^7 \times (9-25) \times 10^6} \approx \begin{cases} 3.97 \times 10^{-9} \ \left(v_p = 3 \times 10^3 \text{ m/s}\right) \\ 1.43 \times 10^{-9} \ \left(v_p = 5 \times 10^3 \text{ m/s}\right) \end{cases}$$

We can obtain that $k = (1.4-4) \times 10^{-9}$. It is readily proved that the same range of k is obtained from Eq. (6) with $\alpha = 0.3 \times 10^{-2}$, 0.6×10^{-2} and 1.2×10^{-2} and $E_0 = 4.2 \times 10^{20}$ erg, 4.2×10^{21} erg and 4.2×10^{22} erg.

Another example is the Benham nuclear explosion with a yield of 1100 kt ($\sim 4.6 \times 10^{15}$ J) at the Nevada Test Site in the USA (Crowiey and Germain, 1971). The aftershock zone size was $V_* \approx$ 6 km $\times$ 20.5 km $\times$ 28 km $=$ 3444 km^3, and 99% of aftershock concentrated in the area of 15 km $\times$ 9 km (or $V_0 \approx 810$ km^3). Using induced load coefficient value $\alpha \approx 1.2 \times 10^{-2}$ (fits $E_0 \approx 4.2 \times 10^{22}$ erg in Table 1) and v_p = 3–5 km/s, Eq. (6), with replacement of V_c by V_* and V_0, produces $k_* = (2.3/6.4) \times 10^{-10}$ for V_* and $k_0 = (1/2.7) \times 10^{-9}$ for V_0.

It follows that for 99% of the aftershock zone, the criterion of geomechanical quasi-resonances in geomaterials holds true: $k = (1-4) \times 10^{-9}$.

Finally, verifying the hypothesized generalizibility of energy criterion (Eqs. (1) and (2)) to embrace crustal earthquakes and underground explosions yields encouraging results. In this context, Eq. (6) can be assumed as a generalization for phenomenological relations (Eqs. (4) and (5)).

3.2. Rockbursts and induced earthquakes

A similar analysis was performed using the data from Tsuboi (1956), Lovchikov (1997) and Eremenko et al. (2005). Table 2 offers a fragment of the energy-based classification of rockbursts from Lovchikov (1997).

Application of Eqs. (1) and (6) to the specified energy range required to correlate the volumes and induced seismic energies of rockbursts. Our analysis is attitudinal, and we think it reasonable to choose the unknown relations to be the relations between the upper limit values of the area S_i, volume V_i and energy E_i ranges in Table 2 (i = 1–6):

$$\left.\begin{array}{l} E_1 \approx 10^4 \text{ J} \leftrightarrow S_1 \approx 55 \text{ m}^2 \leftrightarrow V_1 \approx 300 \text{ m}^3 \\ E_2 \approx 10^5 \text{ J} \leftrightarrow S_2 \approx 260 \text{ m}^2 \leftrightarrow V_2 \approx 3 \times 10^3 \text{ m}^3 \\ E_6 \approx 10^9 \text{ J} \leftrightarrow S_6 \approx 1.2 \times 10^5 \text{ m}^2 \leftrightarrow V_6 \approx 3 \times 10^7 \text{ m}^3 \end{array}\right\} \tag{7}$$

In mining practice, a visible damage area is determined sufficiently accurately, as a rule. Accordingly, alongside V_i, we also used the volume characteristics of rockburst-induced damages on the

Table 2
Classification of rockbursts by energy (Shemyakin et al., 1986).

Class	Rockburst	Shock energy (J)	Geometrical characteristics	
			Area of visible effects within a horizon (m^2)	Volume of visible effects (m^3)
I	Extremely weak	$<10^4$	<55	$<3 \times 10^2$
II	Weak	10^4–10^5	55–260	3×10^2–3×10^3
III	Strong	10^5–10^6	260–1200	3×10^3–3×10^4
IV	Very strong	10^6–10^7	1200–5600	3×10^4–3×10^5
V	Large	10^7–10^8	5600-26,000	3×10^5–3×10^6
VI	Extremely large	10^8–10^9	26,000–120,000	3×10^6–3×10^7
VII	Induced	$>10^9$	$>1.2 \times 10^5$	$>3 \times 10^7$

assumption of cubic and spherical symmetries of the rockburst volumes, V_i^c and V_i^s (i = 1–6), respectively. In this case, V_i^c and V_i^s for S_i are given by

$$V_i^c = S_i\sqrt{S_i} \quad (8)$$

$$V_i^s = \frac{4}{3}S_i\sqrt{\frac{S_i}{\pi}} \quad (9)$$

Eqs. (8) and (9) with S_i from Table 2 yield V_i^c and V_i^s as follows:

$$\left.\begin{array}{l} V_1^c = 4.1\times10^2\ \text{m}^3,\ V_2^c = 4.2\times10^3\ \text{m}^3,\ V_3^c = 4.2\times10^4\ \text{m}^3 \\ V_4^c = 4.2\times10^5\ \text{m}^3,\ V_5^c = 4.2\times10^6\ \text{m}^3,\ V_6^c = 4.2\times10^7\ \text{m}^3 \end{array}\right\} \quad (10)$$

$$\left.\begin{array}{l} V_1^s = 3.1\times10^2\ \text{m}^3,\ V_2^s = 3.2\times10^3\ \text{m}^3,\ V_3^s = 3.1\times10^4\ \text{m}^3 \\ V_4^s = 3.2\times10^5\ \text{m}^3,\ V_5^s = 3.2\times10^6\ \text{m}^3,\ V_6^s = 3.1\times10^7\ \text{m}^3 \end{array}\right\} \quad (11)$$

In Eq. (6), we substitute V_c for V_i from Table 2, and for the above values of V_i^c and V_i^s from Eqs. (10) and (11) by turn, replace αE_0 by E_i from Eq. (7) for the appropriate i, and obtain the ranges of the energy characteristic k (Table 3). It follows from Table 3 that, despite a weak difference in the influence of approximated geometries of rockbursts with the specified energy (from weak rockbursts to high-capacity induced earthquakes), the range of the criterion k for rockbursts corresponds to the range of this criterion for earthquakes and high-power explosions: (0.4–1.3) $\times$ 10^{-9} and (1–4) $\times$ 10^{-9}.

Two more examples from Lovchikov (1997) are adduced here. The first case is a tectonic rockburst with energy of ~10^9 J in Umbozero mine, Apatity. The induced damage covered an area up to 8 $\times$ 10^4 m^2 (fracturing and rockfalls). Using Eqs. (8) and (9), we have the estimates of the rockburst volume as follows: $V^c \approx 2.3\times10^7$ m^3 and $V^s \approx 1.7\times10^7$ m^3. From Eq. (1) at $\rho \approx 2.8\times10^3$ kg/m^3 and υ_p = (3–5) $\times$ 10^3 m/s, we have k_V^c = (0.6–1.7) $\times$ 10^{-9} and k_V^s = (0.8–2.3) $\times$ 10^{-9}, i.e. k = (0.6–2.3) $\times$ 10^{-9}.

The other example is a case study of the extensive in situ testing data on characteristics of damages of much smaller size than above. According to the energy-based classification of rockbursts from Lovchikov (1997), rockbursts are the dynamic events that can cause a collapse of one or two timber support frames and rockfalls (ore and coal) of 1–2 m^3 in volume. Such events have energy of $\leq 10^2$ J. Using the above estimates, we have k = (0.7–2) $\times$ 10^{-9} for $V \approx 2$ m^3 and (1.4–4) $\times$ 10^{-9} for 1 m^3, i.e. (1–4) $\times$ 10^{-9}. Naturally, this fact offers grounds for continuing with the analysis of high-stress rock fracture processes during drilling.

Table 3
Ranges of energy characteristic k for various strength rockbursts.

E_i (J)	S_i (m^2)	k_{V_i} (10^9)	$k_{V_i^c}$ (10^9)	$k_{V_i^s}$ (10^9)	Remark
10^4	55	0.5–1.3	0.4–1	0.5–1.3	k_{V_i} for V_i in Eq. (7)
10^5	260	0.5–1.3	0.4–0.9	0.5–1.2	
10^6	1200	0.5–1.3	0.4–0.9	0.5–1.3	$k_{V_i^c}$ for V_i^c in Eq. (10)
10^7	5600	0.5–1.3	0.4–0.9	0.5–1.2	
10^8	26,000	0.5–1.3	0.4–0.9	0.5–1.2	$k_{V_i^s}$ for V_i^s in Eq. (11)
10^9	120,000	0.5–1.3	0.4–0.9	0.5–1.3	

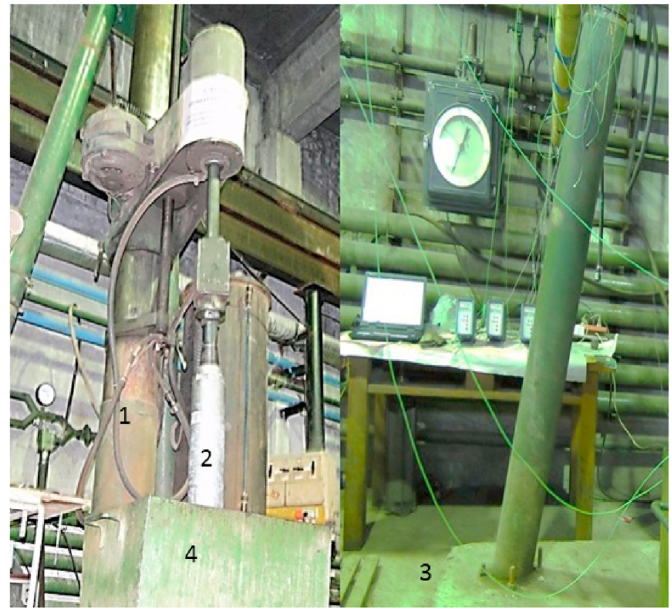


Fig. 2. Test installation for volumetric destruction of geomaterials.

3.3. Energy condition of optimum rock-breaking tool–geomaterial interaction in drilling

It is known that rock fracture and its energy content during drilling depend on the value of unit load (pressure) applied to the bottom hole. A high-force impact can induce either fatigue or less energy-consuming volumetric destruction and fragmentation. The key objective of the energy efficiency enhancement in drilling is to determine the optimum conditions of volumetric rock destruction in terms of fraction composition, the identification of the appropriate rock-breaking tool parameters and their combination to produce the optimum energy input. This is also valid for heading machines.

The optimum energy input is understood as the minimum essential energy of mechanical effect generated by the rock-breaking tool on the bottom hole or fore-breast to be advanced. The optimum energy input is required to be known to implement rock fracture, as well as to prevent overgrinding and oversize production. In this respect, the dimensionless criterion of volumetric destruction of rocks and rock masses (Eq. (1) and (2)) can be applied.

The study (Karpov et al., 2019) may be of interest in this context. The energy criterion of mechanical action exerted by a rock-breaking tool on rock mass, considering the variability of physico-mechanical properties and stress-strain behavior of the latter, can be presented in a more general form (without setting the ranges of the coefficients γ and θ):

$$k = \theta \times 10^{-\gamma} \quad (12)$$

where θ and γ are positive and vary within limited ranges.

According to Adushkin and Oparin (2012, 2013, 2014, 2016), in 'free condition' of contact geoblocks having the mass M, pendulum waves can only arise under the external energy inputs W such that θ = 1–4 and γ = 9 as per Eqs. (1) and (2).

For earthquakes and rockbursts of any energy class, W is a value of seismic energy emission from the involved rock mass volumes V. The portion of the accumulated potential energy U_0 in the emitted seismic energy is yet disputable, though according to Table 1, it can be assessed using the value of α which is the seismic load induced by explosions of adequate energy (comparable with the class of an earthquake or a rockburst), i.e. $W = \alpha(V)U_0$. Here, it is assumed that $U_0 >> W_k$ by Eq. (2).

It can be seen in Table 1 that an increase in the earthquake or blast energy by three orders of magnitude increases the induced seismic load coefficient $\alpha(V)$ by an order merely. For this reason, it can be expected that in the destruction of rocks, the coefficient $\alpha \approx 1.5 \times 10^{-6}$ at the energy $E_0 \approx 10^3$ J and $\alpha \approx 1.5 \times 10^{-7}$ when $E_0 \approx 1$ J.

These estimates certainly need experimental verification, at least at the specified levels of the smallness of destruction volumes. This is important, among other things, because of the difference between the coefficients α_1 and α_2 for loose and dense rocks in Table 1 (supposedly, for any rocks). Consequently, it can be expected that the underground blast-induced load coefficient (more generally, the pulsed mechanical action coefficient), along with the broken volume V, will be influenced by the stress-strain behavior of rocks in the zone of future destruction.

No systematic research (especially tests) was undertaken in this area. Therefore, to estimate the coefficient α (in this case, the rock-breaking tool impact on rocks) and optimize the spacing between indenters of a drilling tool, the experimental studies were performed by Karpov and Timonin (2018).

The volumetric destruction testing installation (Fig. 2) consists of frame 1 to hold impact loading facility 2 and slab 3 with fixed geomaterial block 4 to simulate rock mass. The impact loading facility is pneumatic hammer P105. The pneumatic hammer was equipped with a unique rock-breaking tool and tungsten carbide bit inserts. The rock-breaking tool in the tests had one-bit inserts, two-bit inserts spaced at 15 mm, 20 mm and 25 mm, and three-bit inserts arranged at the corners of equilateral triangles with the sides of 15 mm, 20 mm and 25 mm on the tool face (Fig. 3).

The impact targets were blocks made of organic glass, marble and granite, and 200 mm × 200 mm × 200 mm in size. The tests allowed visual observation of fracturing and stress interaction under penetration of the indenters. The surfaces of the blocks were carefully polished. The volume of destruction and the applied mechanical energy were determined using a special procedure.

The efficiency of the applied pulse energy was assessed in terms of the volumetric fracture energy content A_v by

$$A_v = A_b/V \tag{13}$$

where A_b is the blow energy (hammer) (J).

The tests were implemented in following three stages:

(1) Dynamic penetration of one spherical indenter with a diameter of 10 mm;

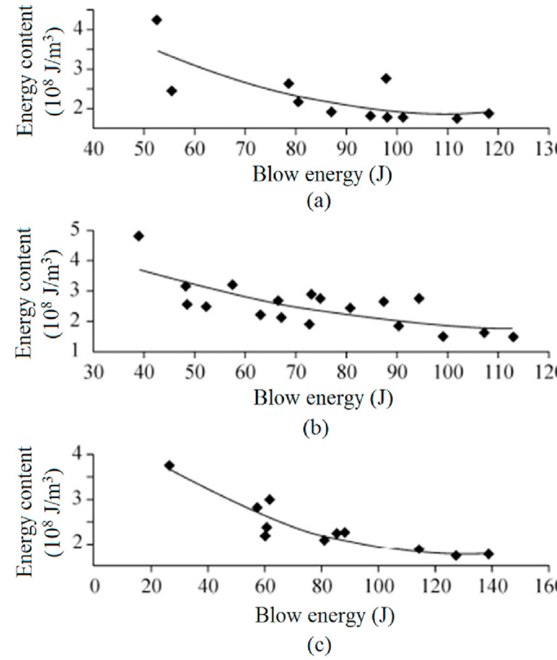


Fig. 4. The energy content of granite destruction versus unit blow energy of rock-breaking tool equipped with three-bit inserts spaced at (a) 15 mm, (b) 20 mm and (c) 25 mm.

(2) Penetration of a tool with two indenters spaced at 15 mm, 20 mm and 25 mm; and

(3) Penetration of a tool with three indenters arranged at the corners of equilateral triangles with the sides of 15 mm, 20 mm and 25 mm on the tool face.

As the energy of blows was increased in the tests, a few modes of fracture gradually appeared as

(1) Spots after contact with the indenters;
(2) Three cavities cut by three disjoint indenters;
(3) Junction of these cavities via ‘channels’; and
(4) A chip cut out from the volume limited by the ‘channels’, reflective of essential interaction of stress and strain fields generated by neighbor indenters.

Starting from a certain threshold, the further increase in the blow energy resulted in no deeper penetration of the rock-breaking tool. The test data were used to plot the relationship between the

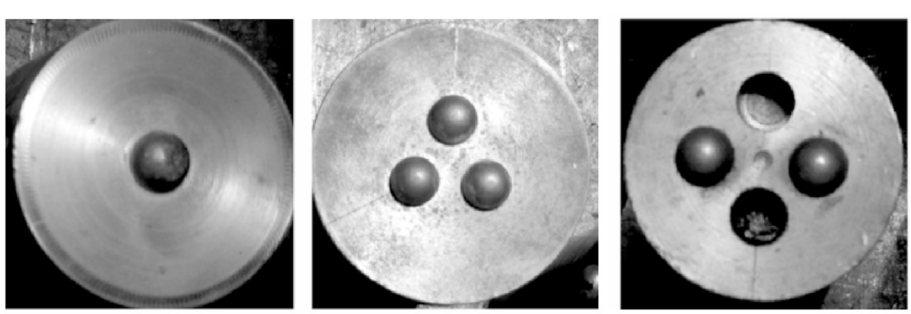


Fig. 3. Rock-breaking tool with different sets of indenters in the tests.

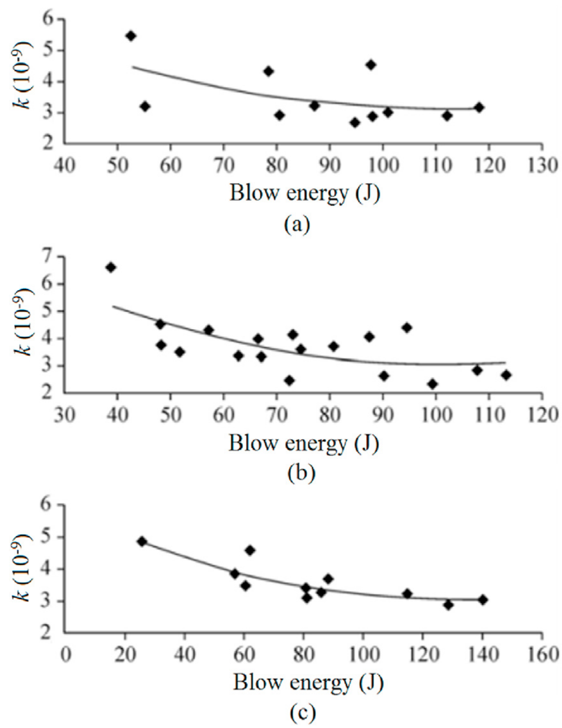


Fig. 5. Dimensionless criterion k versus unit blow energy of rock-breaking tool equipped with three-bit inserts spaced at (a) 15 mm, (b) 20 mm and (c) 25 mm.

energy content of fracture and the unit blow energy for each set of indenters on the face of the rock-breaking tool (Figs. 4 and 5).

The energy content of rock fracture by tungsten carbide inserts versus the unit blow energy is presented in Fig. 4. Fig. 5 demonstrates the values of k versus the unit blow energy at different sets of indenters on the rock-breaking tool. To this end, Eq. (1) was modified to fit with the experimental conditions:

$$k = \frac{\alpha E_0}{\rho V {v_p}^2} = \theta \times 10^{-\gamma} \tag{14}$$

The values of ρ, V and v_p were determined experimentally in the tests of geomaterials, and the values of E_0 varied within a preset range (Figs. 4 and 5). It appears that at $\theta = 1$–4 and $\gamma = 9$, the coefficient α has a value of 1.5×10^{-6}. This is a satisfactory estimate for the actual volumes of chips cut out from the test blocks, considering the above-predicted orders of values of α (using extrapolation of Sadovsky's data from Table 1). Allowance was made because the test blocks were in 'free state', on the ground surface and, consequently, experienced no essential internal stresses governed by the action of the confining pressure in deep rock mass.

The highest destructive effect of the impact is achieved when the dimensionless criterion k is similar to the minimum values of the reduced quasi-parabolic curves. Thus, it follows from the plots in Figs. 4 and 5 that the minimum energy contents of fracture and the minimum values of k almost coincide. This proves the conclusion that mechanical energy applied to rock mass destruction at the minimum energy content is mainly spent for breakage and separation of fragments from rock mass.

The tests used rock-breaking tools equipped with indenters with spherical faces. The coincidence of the criterion k and the energy condition of nonlinear pendulum-type elastic waves (Kurlenya et al., 1998) means that nonlinear geomechanical quasi-resonance events arise in the zone of minimum energy content (Kurlenya et al., 1997).

3.4. An approach to maximum stress assessment in rock damage zones

At present, it is almost impossible to determine a volumetric ultimate strength of rock mass in the earthquake and rockburst zones by direct measurements for known reasons. However, given the above, it seems to be feasible to evaluate average stresses σ in rock damage zones with the help of seismic records of earthquakes and rockbursts using the coefficient α of seismic load induced by earthquakes and underground blasts of preset energies by Sadovsky (Table 1): $\alpha = \alpha(V)$.

The above results make it possible to conclude that under a dynamic event with the energy W (from the seismic record), in the destruction volume V, it should hold that (it is assumed that $U_0 >> W_k$ for Eqs. (1) and (2)):

$$W = \alpha U_0, \; U_0 = \frac{3}{2}(1-2\nu)\frac{\sigma^2}{E}V \tag{15}$$

where E is the Young's modulus, and ν is the Poisson's ratio.

Consequently, using the energy criterion k of a seismic emission event (for definiteness, it is assumed that $k = \theta \times 10^{-9}$, where $\theta = 1$–4), Eq. (1) yields the relation:

$$\frac{\frac{3}{2}\alpha(1-2\nu)\frac{\sigma^2}{E}V}{\rho V {v_p}^2} = \theta \times 10^{-9} \tag{16}$$

or, with regard to $\alpha = \alpha(V)$, we have

$$\frac{\frac{3}{2}\alpha(V)(1-2\nu)\sigma^2}{\rho {v_p}^2 E} = \theta \times 10^{-9} \tag{17}$$

Eq. (17) produces

$$\sigma \approx 10^{-4} v_p \sqrt{\frac{\theta \rho E}{15(1-2\nu)\alpha(V)}} \tag{18}$$

The value of σ obtained from Eq. (18), considering the dynamic event that has occurred, can simultaneously be taken as the volumetric strength characteristic of rock mass in the destruction zone. It can readily be understood that Eq. (18) possesses an essential potential for geotechnical monitoring while drilling, not only in terms of σ. Under certain conditions, similar expressions can be derived for other expected characteristics of rocks, given the rest of the parameters involved are known.

4. Adapting Oparin's energy criterion of volumetric rock destruction to rotary percussive drilling process

The scientific framework used in the practical analyses of rock disintegration in drilling was the phenomenon of zonal rock disintegration around underground excavations (USSR Discovery No. 400) (Shemyakin et al., 1992) and the phenomenon of the alternating response of rocks to blast-induced (dynamic) effects (Oparin and Adushkin, 2019).

The transition to the application of Eqs. (1) and (2) in drilling was made based on understanding the mechanism of rock breakage per unit blows of piston on impact tool equipped with rock-breaking elements (Oparin et al., 2017; Primychkin et al., 2017). The accomplished research evaluated the V-dependent coefficient α in case of the impulse destructive effect of a percussive tool on rocks (Eq. (17)). As a result, the modified expression was obtained for application of the volumetric rock destruction energy criterion (VDEC) in the engineering of rock-breaking tools for down-the-hole rotary percussive drills:

$$k = \frac{E_0\alpha}{V\rho\upsilon_{\mathrm{p}}^2} = (1-4)\times 10^{-9} \quad (19)$$

where α is the coefficient of impulse rock-breaking effect exerted by percussive tool on rock mass: $\alpha = 1 \times 10^{-6}$ for loose rocks, $\alpha = 1.5 \times 10^{-6}$ for rocks on average, and $\alpha = 3 \times 10^{-6}$ for medium- and high-density rocks (Karpov et al., 2019).

The relevant outcome of the research was the discovery that the dimensionless energy criterion of volumetric rock destruction and the energy criterion of pendulum waves coincided in the zone of minimum energy consumption of rock breakage. Those discovery-initiated studies aimed at the in situ integrated evaluation of energy efficiency of rock destruction in rotary percussive drilling by down-the-hole hammers.

Considering the specific nature of rock destruction in rotary percussive drilling, Eq. (19), as a derived relation from Eq. (1), is transformed into a modified expression (Karpov et al., 2019):

$$k = k_{\mathrm{c}}\eta = \frac{E_{\mathrm{air}}(t)a}{V_{\mathrm{b}}(t)\rho\upsilon_{\mathrm{p}}^2}\eta = (1-4)\times 10^{-9} \quad (20)$$

where k_{c} is the dimensionless energy index (DEI), $E_{\mathrm{air}}(t)$ is the energy of air fed to drill string (J), $V_{\mathrm{b}}(t)$ is the volume of broken rock in the time t of the test section drilling (m^3), and η is the DTH hammer efficiency.

In rotary percussive drilling in mines, it is difficult to calculate factual energy fed to the machine, and then, via the drill bit, to the bottom hole because of energy-source leakage along the drill string and due to wear of parts in the air-distribution system and rock-breaking tool in DTH hammers. For this reason, the VDEC is determined as the product of the DEI k_{c} and DTH hammer efficiency η found from rig tests and used as an empirical coefficient of energy transfer from drill rig to the drilling tool (Oparin et al., 2017; Karpov and Timonin, 2018).

The compressed-air energy fed to drill rig is calculated by (Karpov and Petreev, 2021):

$$E_{\mathrm{air}} = W_{\mathrm{air}}t = \frac{RT}{\mu}Q_{\mathrm{air}}\ln\frac{p}{p_{\mathrm{atm}}}t \quad (21)$$

where W_{air} is the power supply to ensure the operation of DTH hammer and removal of chips from hole (W); t is the time of rock destruction in the test section in rotary percussive drilling (s); R is the universal gas constant equal to 8.31 J/(mol K); T is the absolute air temperature (K); μ is the air molar weight, and $\mu = 0.02896$ kg/mol; Q_{air} is the airflow rate (kg/s); p is the working (rated) pressure of DTH hammer (Pa); and p_{atm} is the atmospheric pressure (Pa).

It is also valid to characterize the drilling energy efficiency using dimensional quantities, in particular, index of energy consumption of rock fracture. The energy consumption in air-percussion drilling is found without α, ρ and υ_{p} from Eq. (20) and is given by

$$A = A_{\mathrm{h}}\eta = \frac{E_{\mathrm{air}}(t)}{V_{\mathrm{b}}(t)}\eta \quad (22)$$

where A is the energy consumption of rock breakage with regard to the energy fed to drill string and broken volume of rock in the test drilling section (J/m^3), and A_{h} is the energy consumption of rock fracture with regard to the energy spent by DTH hammer to break rocks in the test section of the drill hole (J/m^3).

The efficiency η is included in Eq. (22) as a coefficient of energy transfer from drill string to bottom hole. The energy consumption of rock fracture in drilling lacks the explicit connection with physico-mechanical properties of rocks for the absence of ρ and υ_{p} in its formula, as well as owing to deficiency of the energy efficiency landmark in the range of $k = (1–4) \times 10^{-9}$, where rock disintegration process approaches an ideal condition as k is close to 1×10^{-9} (Oparin et al., 2016). For this reason, the in situ research in Russian mines preferred estimating energy efficiency of rotary percussive drilling with the dimensionless values k and k_{c}.

By way of illustration, we discuss the case study of energy efficiency estimation in DTH drilling in iron-poor quartzite layers at the Korobkovo iron ore deposit (Table 4) (Oparin et al., 2017).

The maximal energy efficiency of rotary percussive drilling with the preliminarily identified range of the minimum energy consumption k was ensured in successive steps. The solutions aiming at higher energy efficiency of drilling included natural, technical and process factors that had an influence on the drillability of rocks during air-hammer drilling (Oparin et al., 2016).

Research stage I included design of an air hammer with the higher energy of unit blow and low flow characteristics in operation at a compressed air pressure of 0.5–0.7 MPa. The feature of designed air hammer PP105EN is the elastic ring valve in the air distribution system. The rig tests of the air hammer in the compressed air pressure range of 0.35–0.6 MPa showed high energy per unit blow (from 90 J to 180 J). It was found that at the rated air pressure higher than 0.6 MPa, about the studies into dynamic penetration of indenters in rock specimens, the machine had energy potential for volumetric destruction of hard rocks during drilling (Xiao et al., 2018a). Research stage II successfully tested the air hammer operability and the ability to detect hidden defects. Stage III designed and manufactured a rock-breaking tool before evaluating drilling energy efficiency with air hammer PP105EN.

For high-quality removal of drilling chips, the bit has an exhaust channel. Aiming to enlarge cut-out volume in the rock by the bit subjected to cyclic impact, the bit is equipped with ballistic-shape indenters made of grade VK15 alloy for air-percussion drilling in hard rocks. The base matrix of the drill bit was the series-produced model KNSH105. One more design modification in the bit was its spline joint and against the keyed joint in the series-produced machine. The new design was governed by the requirement to ensure higher unit blow energy, which needed a more flexible and tear-proof connection with an air hammer for torque transmission when loaded. The latter is important for adjusting the bottom hole capacity of the air hammer in drilling within various ranges of drilling string rotations per minute. More details on the design solutions implemented in air hammer model PP105EN are given in Karpov and Timonin (2018).

Table 4
Physico-mechanical properties of iron-poor quartzite.

Property	Value
Density, ρ $^{\mathrm{(kg/m3)}}$	2810
Uniaxial compressive strength, σ_{c} (MPa)	193
Tensile strength, σ_{t} (MPa)	13
Young's modulus, E (10^4 MPa)	12.1
Poisson's ratio, μ	0.22
Gravimetric rock humidity, $\omega_{\mathrm{в}}$ (%)	0.05
Porosity, P (%)	9.6
Coefficient of brittleness by Baron (Yarali and Soyer, 2011), B	0.6
Jointing category, J	III
Abrasiveness class, K_{abr}	VII
Rock drillability category, K_{dril}	XIX
P-wave velocity, υ_{p} (m/s)	5950

The full-scale tests of drilling energy efficiency were carried out in mines using air hammers PP105EN and P105PM (production model) as well as a pilot model air hammer PP ST manufactured by Staltrest. Using drawings provided by the Institute of Mining, Siberian Branch, Russian Academy of Sciences, the latter manufacturer also made an original drill bit for PP105EN and a bit for its own-produced air hammer PP ST. The characteristics of the machines are compiled in Table 5. Fig. 6 demonstrates the models of drill bits for DTH hammers from Table 5.

The rig test data (Table 5) show that air hammer PP105EN has the highest capacity and the lowest specific airflow rate amongst the test machines. This fact is important for the improvement of drilling energy efficiency. In terms of dynamic load per indenter, PP105EN is also advantageous: at the pressure of 0.5–0.6 MPa, energy from 14 J to 16 J falls at one indenter (without regard to energy loss in travel along the bit to the indenters). These values may be higher by 10%–20% in actual drilling since not all indenters are simultaneously engaged in the process of rock destruction.

Research stage IV was the refinement of physico-mechanical data of rocks from the assigned test site at the geological department of the mine (Table 2). Drilling sites often lack accurate data on the strength, jointing and abrasiveness of rocks. This affects the objectiveness of testing results. Under the production environment, such a situation ends with the problematic determination of output standards and time of drilling, which reduces productivity (Buyalich and Khusnutdinov, 2017; Aksenov et al., 2020).

The pilot tests of the air hammers involved drilling rig series NKR100 M. Drilling was carried out both upward and downward. The preliminary analysis of drilling by series-produced P105PM on the drilling rigs with high and non-adjustable rotation frequency (more than 1.25 Hz) shows that the high and non-adjustable rotation frequency is one of the major causes of low endurance exhibited by the air hammer in hard rock drilling. For this reason, air hammers PP105EN and PP ST were tested on drilling rigs with adjustable rotation frequency, while the energy efficiency of P105PM was evaluated on each drilling rig model.

At process stage V, the drilling modes were selected with regard to minimum and sufficient water content of water-and-air mixture governing capacity of the air hammers. For the test machine size range, the water flow rate was not higher than 5–7 L/min. In hard rock drilling, it is sufficient for efficient dust suppression.

Fig. 7 presents the energy efficiency evaluation results for drilling with DTH hammers based on the dimensionless energy criterion of volumetric rock destruction under the rated compressed air pressure of 0.6 MPa.

Table 5
Technical characteristics of DTH air hammers.

Technical characteristic	P105PM	PP105EN	PP ST[a]
Valve in the air distribution system	Yes	Yes	No
Back valve	No	No	Yes
Blow energy A at $p = 0.5$ MPa (J)	118	162	103
Blow frequency, f_{bl} (Hz)	24	19,5	21
Impact power, W (kW)	2.8	3.2	2.2
Specific airflow rate, q (m^3/(kW s))	0.045	0.036	0.052
Piston weight (kg)	4	4.1	3.9
Drill bit	KNSH105	KNSH105 M	KNSH ST
Shape of indenters	Spherical	Ballistic	Ballistic
Exhaust holes in bit matrix	No	Yes	Yes
Number of indenters	12	12	15

[a] Capacity characteristics are taken from the analog air hammer P110A.

The implemented technical stages of the research aimed to ensure that the minimum energy consumption range $k = (1-4) \times 10^{-9}$ in air-percussion drilling in iron-poor quartzite strata proved experimentally the analytically ascertained feasibility.

The lowest energy efficiency is observed in drilling with hammer PP ST due to low unit blow energy (Table 5) and because of too many indenters installed on the bit (Fig. 6c), which take up dynamic stress twice less than the standard value. The rock-breaking insert fell out of the bit during drilling, which caused extra energy consumption. The same situation but with three fallen out indenters took place in drilling with hammer P105PM on the rig with non-adjustable rotation frequency (1.3 Hz): an increasing trend was observed in k. The value of k twice approached the upper limit of the minimum energy consumption range because of the decrease in the water content of the air-and-water mixture down to a minimum (5–6 L/min) and due to the increased feed force by 20%.

In view of the high correlation of the dimensionless energy characteristics k and k_c with specific compressed air cost C_{air} in air-percussion drilling, Table 6 presents the specific air costs (without water) for each tested machine together with the penetration rates for each test interval of drilling path (Fig. 7).

The obtained results visualize the advantages of energy-efficient drilling in hard rocks. In process of the investigation, the rational rotation frequency mode was determined for drill strings at the feed forces sufficient to balance the backflow of DTH hammers (without overshoot). For hammer P105PM, the setting (initial) range of rational rotation frequencies after spudding of holes was 0.56–0.63 Hz in downward drilling. In this range, air-percussion drilling features minimum energy consumption, low wear of rock-breaking elements and maximum penetration rates. For DTH hammer PP105EN, this range is 0.63–0.7 Hz. Substantiation of rational drilling modes for downhole machines in terms of power on bottom hole is in detail described in Aksenov et al. (2020).

The accomplished research findings were applied in developing a systematic framework for energy efficiency estimation in air percussion drilling. They launched further studies into the breakage resistance of rocks in rotary percussive drilling.

Fig. 6. Drill bits: (a) KNSH105, (b) KNSH105 M, and (c) KNSH ST.

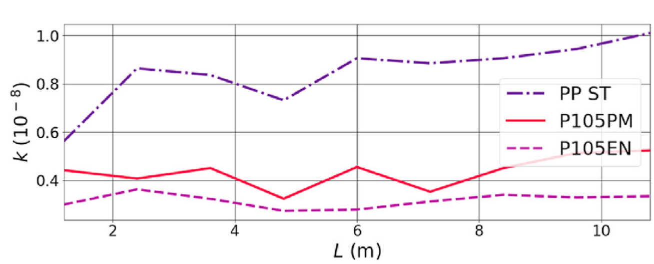


Fig. 7. Energy efficiency evaluation of DTH hammer drilling by VDEC k: (a) PP ST, $f_{rot} = 0.58$ Hz; (b) P105PM, $f_{rot} = 1.3$ Hz; and (c) PP105EN, $f_{rot} = 0.7$ Hz.

5. Main elements of geotechnical monitoring of air percussion drilling in specific conditions of a mineral deposit

The advanced technology and equipment for rotary percussive drilling are adaptive approaches. This means that drilling machines, rock-breaking tools and drilling modes are selected soundly for specific conditions of a mineral deposit. It is well known that the change in the technical parameters and process variables of drilling or geological and geotechnical conditions requires adjusting drilling modes to maintain high drilling efficiency at the minimum wear of the rock-breaking tool and lowest cost of energy source. Modern rotary percussive drilling facilities are equipped with instruments for recording drilling mode parameters and are highly functionally admissive in terms of choice and attachment of such instruments. On the other hand, the practice shows that the presence of such instrumentation is not a guarantee of the energy efficiency of drilling. The principal cause is that drilling modes are selected based on penetration rates and general recommendations on their determination. Thus, approaches to drilling mode selection are empirical. Furthermore, when drilling in the stratified or highly jointed rock mass, the jumps in penetration rates often end with accidents. There are no timely control regulations for power on the bottom hole in the operation of DTH hammers when it is set to reduce impact load on the bottom hole in zones of heavy disintegration or to increase impact load in case of elevated breakage resistance of rocks. Accordingly, interference into drilling processes for adjustment of drilling modes should be adaptive, well-timed, continuous and objective. Thus, the quantitative assessment of the safety and energy efficiency of drilling requires monitoring to be undertaken.

The experimental research aiming at in situ verification of the energy characteristics k and k_c in rotary percussive drilling demonstrates that the process control is possible through adjustment of these values. The quantitative assessment of the ranges for safe and energy-efficient drilling was accompanied by appropriate geomechanical monitoring. Such approach features the applied relevance as a drilling mode preset by a drill unit operator is unaltered in the test interval of borehole: drilling is carried out using a specific model of air hammer–rock-breaking tool–rock interaction at a set of the known physico-mechanical parameters of rocks (sufficient to determine k or k_c). In the case under analysis, *monitoring is geotechnical* and aims to find an optimal drilling mode subject to the bit load and rotation frequency constraints. This is a special type of monitoring for in-service inspection of drilling. Such monitoring can be carried out simultaneously with the fulfilment of current production tasks by the drilling unit operator toward optimizing the energy efficiency of drilling.

Currently, the operation of air-percussion drilling machines requires the direct participation of an operator. Moreover, the machines are unfitted with adaptive in-service geomechanical monitoring. The reason is that a real rock mass is nonuniform and isotropic in terms of physico-mechanical properties, has micro- and macro-cracks, and is plastic sometimes. Consequently, the operator cannot respond to considerable alterations in the drilling process in due time, even with available instrumentation. This is valid both for a short-term estimate of the drilling process quality and a long-term assessment when the operator should respond to a gradual drop in rock drillability and decide either on technology (change of the drilling mode, intensive borehole cleanout) or process alternatives (replacement of rock-breaking tool or downhole drilling machine).

In connection with this, large-scale drilling can only be energy efficient within rational ranges of drilling modes. Optimal values are typical of short intervals in drill holes and, thus, can be maintained solely in the mode of short-term geomechanical monitoring with prompt adjustment of power on the bottom hole and targeted measures of borehole cleanout. That is, the process control in drilling means fulfilment both of *basic* operations towards adjustment of operating conditions in case of drilling on the bottom, and well-timed *auxiliary* operations connected with borehole cleanout.

The following shows the case study on the alternating response of rocks to cyclic impact breakage in air-percussion drilling in the Borok quarry, Novosibirsk Region. Geotechnical monitoring was implemented in the comparative appraisal tests of DTH hammer models COP64.2 (Atlas Copco) and PV170 M (Institute of Mining, Siberian Branch, Russian Academy of Sciences) (Fig. 8).

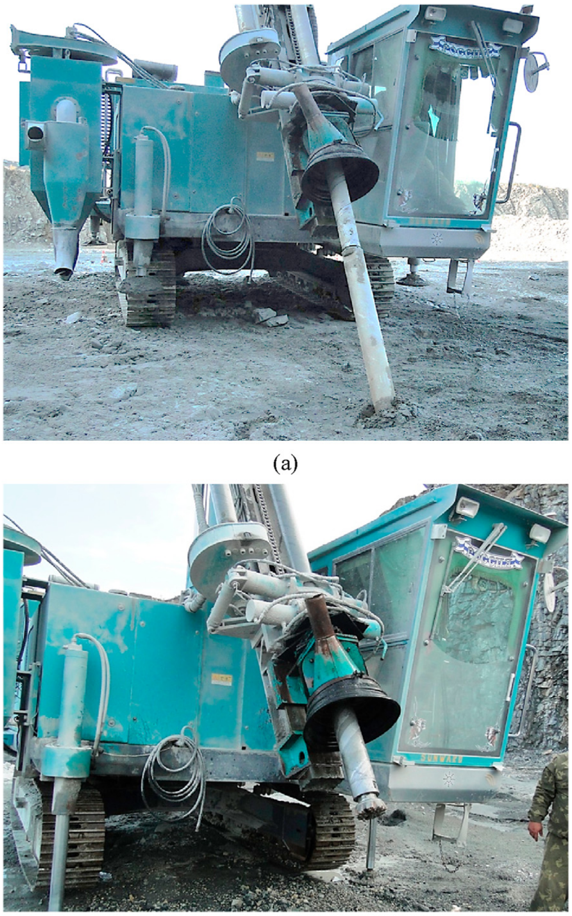
The performance capabilities of the test machines are listed in Table A1 in the Appendix. The comparative appraisal of the air hammers was performed in a highly jointed rock mass (jointing category I). Table 7 reports the results of the comparative appraisal of energy efficiency in drilling with DTH hammers with in-service analysis of the effect exerted by rock-breaking tools on rock mass.

The tests have found that the hammer COP64.2, advantageous for higher unit blow energy by 18% and heavier piston by 22% as compared with PV170 M at the rated pressure of 1–1.2 MPa, is faulty in operation in a highly jointed rock mass. Such machines were actually lost together with drill rods in the test quarry. In the conditions of high disintegration of ore and enclosing rock mass with crack networks (predestruction stage), operation of drilling units with high output performance is risky and dictates a reduction in power on bottom hole, feed force, rated pressure or rotation frequency of drill string (in order to decrease impact effects on bottom hole).

The known horsepower formula $W = Tf_{\text{blow}}$ (T is the unit blow energy (J), and f_{blow} is the blow frequency) is used to describe technical characteristics of down-the-hole percussion machines. It only characterizes 'test-bench' impact capacity of a machine (without rotation), or the bottom hole coverage capacity in rotary percussive drilling at the set rotation frequency of 1 Hz. On this basis, finding of the specific power on bottom hole of an air hammer ($W_{2\pi}$, W/rotation) during drilling should take into account

Table 6
Cost–performance ratio of drilling.

PP105EN		P105PM		PP ST	
v_p (m/min)	C_{air} (Rub/m)	v_p (m/min)	C_{air} (Rub/m)	v_p (m/min)	C_{air} (Rub/m)
0.218	43	0.16	57.6	0.053	189
0.184	50.7	0.118	78	0.048	208.32
0.206	45.2	0.106	86.4	0.05	201.6
0.244	38.2	0.148	62	0.057	176.4
0.24	39	0.105	87.2	0.046	218.4
0.214	43.6	0.136	67.6	0.047	213.36
0.196	47.5	0.106	86.4	0.046	218.4
0.203	46	0.093	98.3	0.044	227.64
0.2	46.8	0.091	100.3	0.041	243.6



(a)

(b)

Fig. 8. DTH hammers: (a) COP64.2 and (b) PV170 M.

the rotation frequency of the drill string (at preset feed force) which governs the number of impacts (s, impact/rotation) on bottom hole per one rotation (2π, rotation) and volume of bottom hole breakage ($V_{2\pi}$, m^3/rotation).

For hard rock breakage, it is necessary that $s \geq f_{blow}$ or $f_{blow} \leq$ 1 Hz. The power of air hammer on bottom hole is found from the formula:

$$W_{2\pi} = Ts = \frac{Tt_{2\pi}}{t_{cycle}} \tag{23}$$

where T is the unit blow energy of air hammer (J), s is the number of impacts per one rotation of drill string (impact/rotation), $t_{2\pi}$ is the time of complete rotation of drill string ($t_{cycle} = 1/f_{blow}$) (s), and t_{cycle} is the impact cycle duration (s).

The value of s is determined by

$$s = f_{blow}t_{2\pi}, t_{2\pi} = \frac{60}{n} \tag{24}$$

Taking into account the data on unit blow energy in rig tests of percussive machines and the average values of impact frequency in full-scale conditions (Table 8), we calculate the output performance of the machines at different rotation frequencies of the drill string. The angle between the impact loads in bottom hole coverage by drill bit indenters is determined by

Table 7
Comparative appraisal of energy efficiency in air hammer drilling of hole with 18 m in length.

Characteristic	Value	
Rock strength (MPa)	125	
Rock density (kg/m^3)	2660	
P-wave velocity (in jointing category IV rock mass) (m/s)	5500	
Air hammers	COP64.2	PV170 M
Drill hole diameter (mm)	165	172
Hole length (m)	18	
Rated pressure (MPa)	1.2	
Airflow rate, Q (m^3/min)	11.5	12
Inlet power, W (kW)	50	48
Feed force, N (kN)	12	10
Rotation per minute (r/min)	50	50
Rotation frequency, f_{rot} (Hz)	0.83	
Blow frequency, f_{blow} (Hz)	20.7	23.2
Angle between impact loads on indenters, γ (°)	14.3	12.8
Penetration rate, V_{pent} (m/min)	0.285	0.28
Volumetric drilling rate, V_{vol} (m^3/min)	6×10^{-3}	6.6×10^{-3}
DEI, k_c (10^{-9})	10.76	10.72

$$\gamma = \frac{360^{\circ} f_{rot}}{f_{blow}} \tag{25}$$

As a result, it has been found that the rational range of rotations for PV170 M in drilling in highly jointed rock mass is 0.9–1 Hz, the same range for the other machines is higher than 1 Hz (overrun of the maximum frequency limit, for drill rig SWDB165), and thus, drilling is carried out in the limited mode. In rocks of joint category III, for PV170 M, the frequency range is 0.75–0.83 Hz. Operation in this range ensures maximum capacity and minimum wear of intenders on rock-breaking tool. The information on the sound selection of setup drilling modes and the dimensionless energy characteristics k and k_c for DTH hammers is given in Karpov et al. (2019).

Fig. 9 shows the curve of k in drilling in the increased rock disintegration site in the Borok granite quarry. The values of P-wave velocities are taken from stable rock mass sites (Table 7). In the set drilling mode by DTH hammer PV170 M, the VDEC k was 4.75×10^{-9}. Accordingly, in the analysis of variation in VDEC in the section L, the lower and upper deviations of the energy criterion were assumed as the decrease and increase in breaking resistance of rocks, respectively. In the check section drilling, no intervention was made in the process; the technical parameters of drilling are compiled in Table 7.

It can be seen in Fig. 9 that in the section of 0.8–0.9 m, there is a rather short zone of accident risk; starting from 1 m point, drilling is stable in the zone of increased rock resistance. Considering that the penetration rate of high-pressure air hammers varies from 0.2 m to 1 m in the quarry, the linear range for determining k should be from 0.05 m to 0.1 m.

The data on variation in k per 0.1 m were processed by comparing the same index from the stable section (4.75×10^{-9}) and the nearest sections. The results show that in drilling in a jointed rock mass with a joint spacing of 0.1 m, the highest hazard is the drop in this index relative to the stable standard values of k as well as relative to each other in the monitoring mode of drilling.

The comparison of the values of k relative to each other is more sensitive for monitoring and prediction of accidents (Fig. 10a). The drop in k relative to the nearest value by 20% means entry in the zone of increased jointing. This follows from the fact that the drilling mode is the same, while the volume of broken rocks increases since the bottom hole area is jointed (pre-destructed), which manifests itself in the reduction in the criterion k. The

Table 8
Parameters of power on bottom hole in air hammer drilling within wide rotation range.

Parameter	COP64.2 (25 J per indenter)				PV170 M (15 J per indenter)			
n (rpm)	40	50	60	65[a]	40	50	55	60
$t_{2\pi}$ (s)	1.5	1.2	1	0.85	1.5	1.2	1.09	1
s (impact/rotation)	31.5	24.8	20.8	17.8	34.8	27.8	25.3	23.2
γ (°)	11.4	14.4	17.3	18.7	10.2	12.8	14.1	15.5
$W_{2\pi}$ (W/rotation)	15,649.2	12,519.36	10,483.2	8985.6	14,337.6	11,470.08	10,427.35	8569.6

[a] Drill rig SWDB165 is technically unfitted for drilling at $n = 65$ r/min.

further decrease in k by 23% means entry in the zone of maximum rock mass disintegration in a short accident-risky section followed by the section of higher breaking strength.

Fig. 10a contains the section with the drop in k by 46% with subsequent deviation to 5% and a positive trend of increase in the dynamic breakage resistance. In Fig. 10b, the first section of entry in the disintegration zone is characterized by deviation from normal k by 24%, which is followed by the drop in k. The next section is the entry in the jointing zone with a deviation of k by 22% followed by the drop by 29% and by the increasing trend in k by 19%. In the constant drilling mode, the drop in the energy criterion of volumetric destruction from 20% to 50% means entry in the zone of increased rock mass disintegration, i.e. in the zone of potential breakdowns.

When k grows over 50%, it is required to decrease power on the bottom hole by a reduction in the rated pressure or by increasing the rotation frequency of the drill string until k reaches the expected normal value. Borehole cleanout in highly jointed rock mass causes sloughing of walls and the risk of the drilling machine failure. The risk of failure in drilling in highly jointed rocks enhances in case of negative deviations; for this reason, the lower deviation of k (both relative to the reference value and to the neighbor sections in boreholes) should never exceed 20%.

Fig. 11 presents the interpreted field data obtained in air-percussion drilling in the Tashtagol mine. When the rated pressure in the mining network dropped, an accident took place. The time of the accident is visualized by the change in the criterion k. The positive deviation Δk is related with the reference value $k = 1.99 \times 10^{-8}$. After the rated pressure drop, drilling appears in the accident-risky zone upon the excess of $k = 1.99 \times 10^{-8}$ by 50%; in section 0.12 m long, from 3.56×10^{-8} to 3.81×10^{-8}, the overshoot of the reference value makes from 75% to 100%; and in the next section 0.12 m long, the accident occurs at the jump of k from 250% to 350%.

It follows from these field data that when the value of the VDEC jumps by 50% and more, emergency measures should be undertaken with forced borehole cleanout; in case of the accompanying drop of the rated pressure, it is necessary to increase power on bottom hole up to the rational range. When k continues growing and no effect of emergency measures is observed, it is required to stop drilling and to disassemble the drilling string until the problem is fixed (lost indenters, lost gasket rings, etc.). Otherwise, the drill string itself may be lost, which is inadmissible.

It follows from these experiments that when breaking resistance of rocks in the check sections grows by more than 30%, it is necessary to carry out forced and intensive cleanout of the borehole. Then, drilling can be continued in case that it is ensured that k returns to the normal level. Otherwise, it is necessary to increase power on bottom hole up to the rational mode of drilling. Furthermore, the tests show that estimation of the VDEC in the monitoring mode in mines should use short drilling intervals of 0.03–0.05 m in holes.

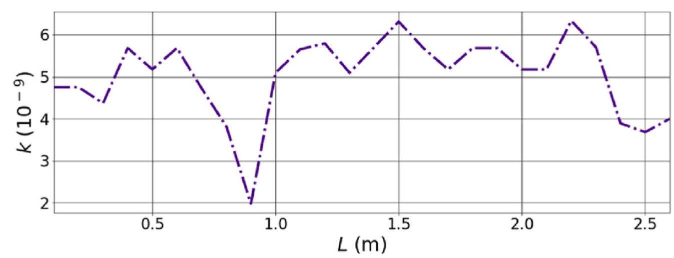


Fig. 9. Alternating response of rocks in rotary percussive drilling by DTH hammer PV170 M.

Based on the results of the integrated field research, the ranges of energy-efficient and failure-free drilling are set based on VDEC (DEI) (Fig. 12). The found ranges can be used in engineering of the adaptable control systems for drilling facilities.

The relevant output of the research is the plot for express-estimation of energy efficiency of DTH hammer drilling (Fig. 13). This plot is advised to be used as an interface on the drill rig operator's display.

Similarly, in the diagram of the air-percussion drilling energy-efficiency based on the DEI, the excess of DEI over the energy efficient range $k = (1\text{–}4) \times 10^{-9}$ is governed by the value of machine–rock energy transfer coefficient. Evaluation of drilling energy efficiency by the VDEC (DEI) can be applied with other methods of rock disintegration. This is relevant for the sound selection of a drilling method for solid mineral deposits with highly variable physico-mechanical properties and in the case of drilling facilities implementing various drilling methods.

6. Discussion

It is essential to address a long overdue need for a theory of independent geotechnical monitoring of drilling, including rotary percussive drilling, for the detailed and multi-factor geomechanical analysis of full-scale destruction of rocks using different drilling machines. This article reviews the long-term research and development in the specified area of science and technology, including research and development (R&D) projects implemented at the Institute of Mining, Siberian Branch of the Russian Academy of Sciences.

The described results continue a cycle of studies, which are based on the phenomenon of zonal rock disintegration around underground excavations and the phenomenon of alternating response of rocks to blast-induced (dynamic) effects.

Generalized dimensionless energy criterion $\hbar$ of nonlinear pendulum elastic waves and geomechanical quasi-resonances can be used to describe complex and dynamic processes of rock fracture in concentration zones of high stresses and strains. Previously, it was used mainly to describe crustal earthquakes, un-

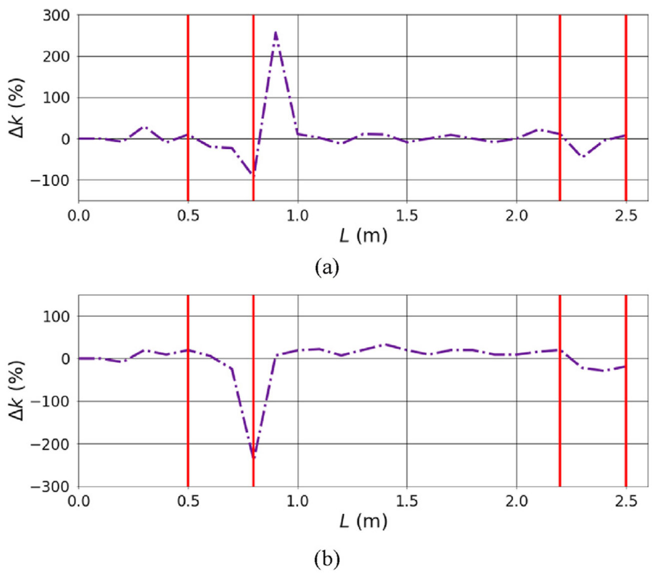


Fig. 10. Monitoring of air-percussion drilling by values of k in the Borok quarry: (a) Relative change of k in the neighbor sections in the drill hole, and (b) Relative change of k as compared with the optimum $k = 4.75 \times 10^{-9}$.

derground explosions, rockbursts and induced earthquakes. Recent studies have proven that it can determine the energy condition of optimum rock-breaking tool–geomaterial interaction in drilling.

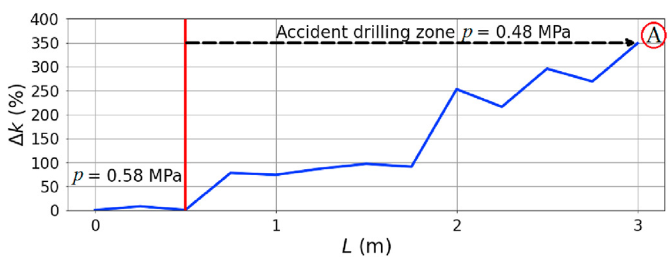


Fig. 11. Change in the energy criterion k in air-percussion drilling monitoring in the Tashtagol mine.

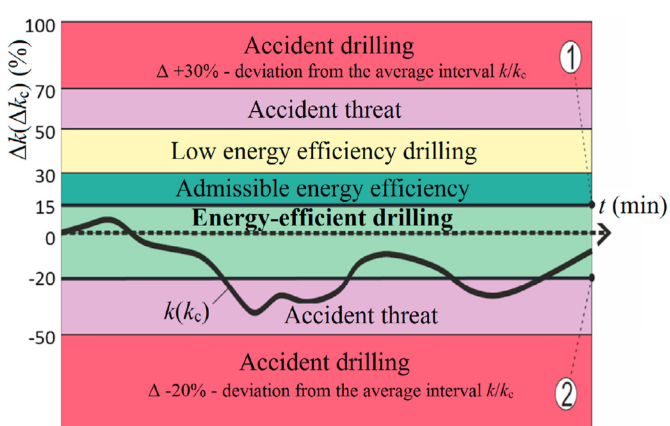


Fig. 12. Safe and efficient ranges for rotary percussive drilling based on the VDEC (DEI).

The change in the work mode of a drilling tool in a borehole can be an early warning of an accident and can help prevent it. These data are applicable in the geotechnical monitoring, performance optimization of drilling machines, as well as for the enhancement of safety of the work environment in mines. The next step to be taken to apply the energy criterion of volumetric rock destruction in mining and construction is the development of express-analysis methods for in situ data of instrumental MWD.

The authors intend to continue the related studies aiming at the design of self-contained instrumentation to control and adjust the performance of rotary percussive drilling towards the maximum capacity of the process at minimum wear of rock-breaking tool and optimum energy consumption as per drilling and blasting patterns. Accuracy and straightness of drilling have an essential effect on fragmentation quality, dilution and mineral extractability. In this respect, the energy efficiency of mining systems used in surface and underground mines defines the route of improvement of drilling equipment.

The integrated experimental, theoretical and geotechnical approach to a comprehensive investigation of real-time processes of rock fracture in rotary percussive drilling using the energy concept possesses the necessary geomechanical performance-and-technology potential to create the next level geotechnical monitoring of drilling systems for various purposes, including determination of physico-mechanical properties and for the stress-strain analysis of rock mass in full-scale drilling.

7. Conclusions

(1) The energy criterion of volumetric rock destruction by Victor Oparin has been adapted to drilling to enable analysis and control of disintegration processes in rock mass during rotary percussive drilling.

(2) The authors have, for the first time, implemented geomechanical and geotechnical monitoring of full-scale rock fracture process in rotary percussive drilling based on Oparin's energy criterion of volumetric rock destruction. It has been found that such an approach allows smart selection of DTH drilling conditions and ensures the rotary percussive drilling performance adjustment and control in terms of maximum drilling capacity, minimum rock-breaking tool wear and optimum energy content.

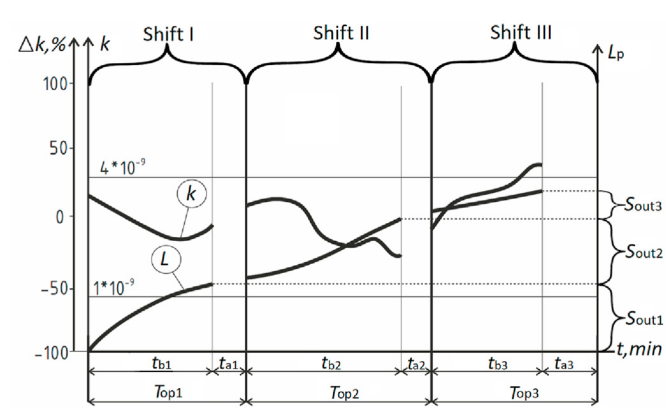


Fig. 13. Estimation plot for safety and energy efficiency of air-percussion drilling. L – headway per drill bit (m); T_{op1}–T_{op3} – drilling operation time per shifts (min); S_{out1}–S_{out3} – outputs per shifts; L_p – drilling length (m); t_{b1}–t_{b3} – main operation times (drilling on bottom) per shifts (min); t_{a1}–t_{a3} – auxiliary operation times per shifts (min).

(3) The authors believe R&D towards wide application of the energy criterion of volumetric rock destruction in full-scale mining should be concerned with the express-analysis methods for monitoring data in instrumental MWD. In particular, advancement is connected with audio analysis, including automated identification of standard drilling operations, digital processing of acoustic signals and real-time reporting of various data on the physico-mechanical properties of rocks intersected while drilling.

(4) Geotechnical monitoring gives in situ information, which is important for engineering design and analysis, prompt decision-making on mining technologies, assessment of rock mass stability and accident prediction and prevention. On-stream adjustability of drilling tool operation ensures early warning and elimination of accidents, and thus, improves various-purpose drilling machines.

Declaration of competing interest

The authors declare that they have no known competing financial interests or personal relationships that could have appeared to influence the work reported in this paper.

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Appendix A. Supplementary data

Supplementary data to this article can be found online at https://doi.org/10.1016/j.jrmge.2021.12.021.

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Viktor N. Oparin obtained his university degree in Novosibirsk State University, Russia, in 1974, and his Candidate of Science and Doctor of Science degrees in Chinakal Institute of Mining, Siberian Branch, Russian Academy of Sciences, in 1978 and 1987, respectively. He was affiliated with Chinakal Institute of Mining, Siberian Branch, Russian Academy of Sciences, as scientific staff from 1978 to 1985, as Head of the Laboratory of Mining Geophysics from 1985 to 2012, as Deputy Director from 1998 to 2003, as Director from 2003 to 2013, and as Head of the Department of Experimental Geomechanics since 2014. He founded the Department of Geomechanics at the Faculty of Geology and Geophysics of Novosibirsk State University and created the Mining Scientific and Educational Center with the participation of nine largest Universities in Siberia, Russia. His research interests include development of theoretical foundations for geomechanical interpretation of geophysical data, creation of complexes of measuring instruments for diagnostics of the stress-strain state of rock masses and control of geomechanical processes, and nonlinear geomechanics and mining geophysics. He has been participated in a large number of international and Russian national projects.