Main development trends and some technical decisions on mining tools equipped with super-hard composite materials inserts

LT Dvornikov¹, PD Krestovskyodvihensky², SM Nikitenko², VA Korneyev¹ and PA Korneyev¹

¹Federal State Educational Institution of Higher Professional Education, Siberian State Industrial University, Novokuznetsk, Russia
²Federal Research Center for Coal and Coal Chemistry, Siberian Branch of the Russian Academy of Sciences, Kemerovo, Russia

E-mail: ¹tmmiok@yandex.ru

Abstract. Directions of a significant increase in effectiveness of rock destruction with tools equipped with super-hard composite material inserts are reviewed and justified. Drilling tool designs with the cutting insert in the form of elliptical Cassinian oval and the asymmetric ring cleaves are suggested. Versions of laboratory stand designs in order to determine the power consumption of rock destruction are developed.

Rock crushing is the main process of mineral deposit development. The mechanical crushing method widely used for this purpose at mining enterprises needs creation of high strength wear-resistant tools which can stand increasing power loads conditioned by steady increase in drive power of rock crushing machines. Technical solutions embodied in the design of the existing mining tools, most of them have been proposed in the second half of the twentieth century and are in need of modernization in accordance with the results of scientific and technical progress.

The authors conducted theoretical studies in order to justify new designs of rock crushing tools, reinforcement materials for their cutting inserts, and also tested the experimental samples of mining tools within the project framework of the Federal Target Program on Development of Experimental Combined Tool Designs Using Super-Hard Composites for Effective Crushing of Rocks.

The results received allow identifying the most promising modernization trends of the existing mining tools nomenclature that enhance their productivity, wear resistance, and also reduce specific energy consumption in the process of mine rock crushing. Rock crushing tool efficiency increase can be achieved by various ways: the tool geometry change, the use of new materials in the manufacture of cutting inserts, as well as the use of new, more productive and energy efficient modes of rock crushing.

The first direction includes research on the development of various types of mining tool structures providing guaranteed turnability of revolving cutters, prevention of rear cutter face from lowering, more rational arming of bladeless tool, as well as radial cutters use range expanding according to the rocks strength without changing the cutting insert geometry.

Based on the results of the fulfilled researches new drill bits designs are developed: with a cutting insert of Cassinian oval elliptical shaped [1] and also with asymmetric ring positioning [2]. The cutter...
with a cutting insert of elliptical Cassinian oval shape is shown in Figure 1a. It consists of a body 1, two styluses with hard alloy inserts 2 of elliptical Cassinian oval shape, blades 3 set along the big oval axis wherein the major axes of the ovals are inclined to the tool axis at an acute angle $\alpha$.

One of the properties of an elliptical Cassinian oval is the presence of two points with a curvature tending to infinity, which allows the full contact when interacting with the face at these points. This feature of the cutter helps to reduce cutter blades blunting 3, in contrast to the existing cutters, in which the interaction of the blades with the face is carried out along the stress concentrators placed on them.

In order to reduce the energy consumption of rock drilling, by providing their destruction with major cut-offs within the frames of the study the drilling bit with asymmetric ring positioning was designed (Figure 1b). The drilling bit was created on the basis of RKS bit design, consisting of a body and discontinuous blades placed symmetrically to the bit axis [3]. The disadvantage of the RKS tool is the fact that both inner semi-blades cut on the face of the borehole concentric strips, easily imposed on each other, which does not lead to active face destruction between the cutting strips, i.e. power consumption of destruction process is high.

![Figure 1. Drilling bits with the cutting insert of elliptical Cassinian oval shape (a) and asymmetric ring positioning (b).](image)

Considering this drawback of PKS cutter, in the design of the cutter with asymmetric ring positioning the inner semi-blades are located asymmetrically with respect to the axis of the cutter rotation (Figure 1b). The developed cutter design with asymmetrical ring positioning allows different semi-blade combinations depending on borehole drilling conditions.

Drilling tool with asymmetrical ring positioning (Figure 1b) consists of a body 1, two styluses with discontinuous blades constructed as peripheral 2 and the internal 2' and 2'' semi-blades while internal semi-blades 2' and 2'' are set asymmetrically with respect to the cutter axis, at the distances $L_1$ and $L_2$. During drilling, the cutter is rotated according to the arrow in Figure 1b with simultaneous axial advance to the face. Thus semi-blades 2, 2' and 2'' start cutting rock from the face. Due to the asymmetric installation of the internal semi-blades 2' and 2'' the cut strips overlap each other thus providing the drilled borehole face destruction with big pieces, resulting in reduced power consumption of the drilling process.

As it was previously noted, when considering ways to improving efficiency of destruction of rocks by drilling cutters, one of the most urgent problems to be solved during the design process is to prevent the collapse of the rock face with the cutter rear side.

Theoretical studies carried out in the framework of the Federal Target Program allowed justifying fundamentally new designs of drilling bit cutting blades.
Figure 2 depicts the bit movement geometry, performing rotational movement around the axis $0^0$ with the speed $\omega$ and the simultaneous forward advance to the depth $h$ per one revolution.

![Figure 2. Drilling bit movement geometry.](image)

It is seen in Figure 2 that each point of the cutter blade $0_1$, $0_n$ remote from the tool axis at a distance $r_i$ passes the path $0_10_1', 0_i0_i,'$, … $0_n0_n'$ during the cutter turnover, respectively.

Helical trajectory scan of each point $0_1$, $0_i$, … $0_n$ in Figure 2 represents the hypotenuse $S$ of a right triangle the legs of which are the distances of the drilling tool advance $h$ and the circumference length with $r_i$ radius scribed with a corresponding point relatively to the axis $0^0$ (Figure 3a). In this way the trajectory of each cutter blade point is the angle of inclination $\alpha_i$ with the cutting plane of the cutter at zero advance (Figure 3a). This angle can be found as:

The value of the angle $\alpha_i$ is variable and decreases from its inner edge to the outer edge. This fact imposes certain restrictions on the clearance value of the rear cutters angle $\beta$ since achieving equality $\alpha_i=\beta$ drilling cutter stops cutting, causing the collapse of the rock.

![Figure 3. Helical trajectory scan of the cutter point (a) and cutting geometry (b).](image)

Usually, in practice [3], mining tool rear angle is within the $5^\circ$–$20^\circ$. With positive front angle rear angle increase leads to weakening of the cutting edge of the tool and may lead to its failure. With negative front angle, as the studies have shown, rear angle can be increased up to $20^\circ$–$30^\circ$, as the strength of the cutting edge is not reduced, and the wear area have less values.

It is obvious that the cutter cutting edge strength depends directly on the sharpening of its angle which is composed of the front $\alpha'$ and rear $\alpha''$ sharpening angles (Figure 3b). At the same time, to avoid rock collapsing with the rear cutter blade, it is required that $\Delta\alpha$ equal the difference between the cutting angle $\alpha$ and the rear sharpening angle in accordance with condition (1)

$$\Delta\alpha = (90 - \alpha'') - \alpha. 
(1)$$

The cutter design corresponding to condition (1) can be obtained when using variable values of back sharpening angle $\alpha''$ over the entire length of the blade [4]. Considering the plots in Figure 3 and using drilling cutter advance meaning $h$, we can calculate the sharpening angle value $\alpha''$ for any point of the blade (Figure 4).
Thus, the implementation of variable adjustable sharpening angle $\alpha_||$ of the cutting edge across the length of the blade in the design of the tool helps to prevent collapse of rocks with the cutter back blade by providing difference $\Delta \alpha$ between the cutting angle $\alpha$ and rear sharpening angle.

Reducing the sharpening angle from the periphery to the center can be justified by the fact that the moment of cutting resistance at the points of the tool from the periphery to the center is reduced due to reducing of these points radii with respect to the tool axis.

Apart from the considered case, avoiding rock crushing with the cutting tool back blade can be implemented in the design of the drilling bit with constant blade sharpening angle under the following condition

$$\alpha^I + \alpha_|| = \alpha_|| = \text{const},$$  \hspace{1cm} (2)

where $\alpha_||$— sharpening angle of the bit cutting insert.

Cutting insert design [5] corresponding to the conditions (1) and (2) is realized through the provision of variable rear and front angles of sharpening in compliance with the constancy of the cutter sharpening angle (Figure 5). The advantage of this design is the same strength of the cutting insert along the entire length of the blade.

Figure 4. The rear cutter sharpening angle changing from the center to periphery.

Figure 5. Cutter sharpening angle changing from center to periphery preserving sharpening angle constancy.

Figure 6. A three-dimensional view of cutting inserts for drilling bit with a variable sharpening angle (a) and a constant sharpening angle (b).
Three-dimensional view of the cutting inserts with variable and constant sharpening angles is shown in Figures 6a and 6b, respectively. Figure 6 depicts: 1—insert body, 2—blade, 3—rear insert edge, X–X geometric drilled borehole axis, arrows a and b show the insert rotation direction with respect to X–X and its advance—deepening of the cutting insert into the produced borehole. Points A, B, C, D, are used to indicate the tops of the insert, forming its rear edge, L—thickness of the cutting insert, α||—the angle of its sharpening.

During the research under the Federal Target Program, possible ways to increasing efficiency of rock crushing by changing the indenter geometry and layouts in the edgeless drill bits were also reviewed. It is known that edgeless cylindrical spherical hard alloy inserts are usually located at the end of the bits according to different layouts which quite often do not correspond to the rock crushed and the face geometry. Thus, an excessive concentration of the inserts in the geometric center of the face can result in rock overgrinding which in its turn leads to increased power consumption for drilling.

One of the options for ensuring the reduction of drilling energy consumption may be execution of a concentric recess on the bit casing striking surface (Figure 7). Such drill bit construction [6] consists of a casing 1, its striking surface 2, indenters 3 of the given diameter d with the given radius r of their working bulge h. On the striking surface 2 a concentric recess 4 is made with depth h not less than the indenter working bulge h (h_1 ≥ h), and the concentric recess 4 diameter d_1 is at least twice the diameter of the indenter d, (d_1 ≥ 2d).

![Figure 7. Edgeless drill bit construction with a concentric recess on the striking surface.](image)

When such bit impacts the borehole face and because its striking surface 2 has a concentric recess 4, a core is formed. The formed core while drilling further is easily sheared off due to the efforts from the indenters located on the edge of concentric recess 4, because of the occurring forces F, directed normally to the surface of the indenters.

All currently used edgeless inserts are built on the principle of the surface curvature constancy of the insert in the area of its interaction with the rock, i.e. the directly impacting surface is manufactured of a spherical shape. The disadvantage of this design is the fact that when impacting the rocks such indenters have no directional effect, i.e. destruction can occur equally in all directions from the point of contact. To have the opportunity to manage the process of destruction in a preferred direction, it is advisable to create indenters with different curvatures in main directions.

One of such insert variant [7] is shown in Figure 8: 1—the body of the insert; 2—the working portion of the insert; P—the insert working portion surface; M—the contact point of the insert working area P surface with the rock; K—contacting P surface at M point the tangent plane—the plane of the destruction face.

![Figure 8. Edgeless drill bit construction with a concentric recess on the striking surface.](image)

It is known from the theory of surfaces that through the surface and the tangent plane contact point one can draw two main direction lines I–I and II–II, which, being mutually perpendicular, define in their cross sections the largest k_1 and the smallest k_2 of curvature, i.e. in section I–I there appears plane curve with a radius of curvature \( \rho_1 \), and in section II–II—plane curve with a radius of curvature \( \rho_2 \). In this case the relations \( \frac{1}{\rho_1} = k_1 \) and \( \frac{1}{\rho_2} = k_2 \) become curvatures.
The suggested insert for drill bits is manufactured with different main curvatures \( k_1 \neq k_2 \), i.e. at the insert cross-section with the plane perpendicular to the axis y-y insert elliptic curve appears with different geometric axes. In Figure 8 an example of a special case of such insert is shown for the ratio \( \frac{k_1}{k_2} = 2 \).

The suggested insert impact principle on a face consists in the fact that the axial force acting in different directions inside the main directions creates various states of stress in the crushed medium and leads to the destruction along the axis of the greatest curvature thus controlling the destruction process.

![Figure 8. Insert geometry for an edgeless drill bit which provides the mine rock crushing process control.](image)

In addition to using effective geometry in the construction of the tool blade and body, it becomes important for the mining tool performance improvement to provide the tool cutting inserts with better wear resistance. Thus, the results of the statistical analysis of drill and combine cutters wear types, conducted in 2014 in the first phase of the Federal Target Program implementation, show that the vast majority of the tool failure causes happen due to the blunting of the cutting inserts.

One of the most promising ways of improving wear resistance of the rock cutting tool is the use of inserts of super-hard composite materials, which are soldered or mechanically fastened on its bearing steel part. The polycrystalline super-hard materials (PSTM) are industrially manufactured in large quantities and are divided into two big groups: made on the basis of cubic boron nitride and diamond-based (Table 1). Polycrystalline cubic boron nitride PCBN is obtained by reactive sintering in a high-pressure apparatus (HPA) under pressures from 4.0 to 6.0 GPa and temperatures from 1300 to 1500°C. Cubic boron nitride is characterized by quite high hardness and, compared with diamond composite materials, possesses excellent chemical resistance to materials containing iron, nickel, and carbon. For this reason PCBN is more widely used in the metal-working industry as a cutting tool for continuous and impact sharpening of quenched steels and irons. The wear-resistance of the composite PCBN depends first and foremost on the content of cubic boron nitride in the composite as well as on the composition and heat resistance of the bonding phase. Tool wear at the cutting boundary is due to partial recrystallization of the material and embrittlement of the cutting edge owing to the interaction with the worked material.

Composites based on polycrystalline diamond PCD, which are obtained by thermobaric synthesis, are widely used for working non-ferrous metals and their alloys as well as in areas associated with the development of deposits and mining minerals. Owing to high hardness diamond composites are used predominately in operations where the main mechanism of tool wear is abrasive wear. However, polycrystalline diamond has a number of limitations in operation that significantly degrade the tool efficiency. One such limitation is the impossibility of using this composite at high temperatures (> 400°C).
is well known that increasing the operating temperature of the tool to 400°C and the expansion coefficients (CLTE) of the metal or carbide bonding phase and the base diamond phase. It

diamond, synthetic diamond single crystal and hard tungsten
crystal and hard tungsten

| Table 1. Comparison of the physical properties of different composites based on polycrystalline diamond, synthetic diamond single crystal and hard tungsten–cobalt alloy. |
|---|---|---|---|
| Index | PCD (PDC) | Heat resistant PCD (TSPD) | Synthetic single crystal | Sintered hard alloy VK15 |
| Density, g/cm³ | 3.90–4.10 | 3.42–3.46 | 3.51–3.52 | 13.80–14.10 |
| Microhardness, GPa | 50–60 | 50–55 | 70–105 | 15–18 |
| Strength in compression, GPa | 7.5–8.5 | 4.0–5.5 | 7.0–9.0 | 12.0–14.0 |
| Fracture toughness, MPa m¹/² | 9.0–10.0 | 7.0–8.0 | 3.0–4.0 | 11.0–12.0 |
| Young’s modulus, GPa | 810–850 | 920–960 | 1000–1150 | 70–100 |
| Thermal conductivity, W m⁻¹ K⁻¹ | 550–750 | 120–250 | 700–1500 | 70–100 |
| Wear resistance, mm | 0.15–0.20 | 0.20–0.25 | – | > 2.0 |
| Heat resistance, °C | 700 | 1200 | 1200 | 1400 |

The first reason for this is structural degradation owing to the difference of the linear thermal expansion coefficients (CLTE) of the metal or carbide bonding phase and the base diamond phase. It is well known that increasing the operating temperature of the tool to 400°C and the different CLTE of the components of the composite material promote partial breaking of the diamond–diamond bond at the interphase boundary, which results in crack formation and subsequent chipping of the cutting edge of the PCD composite during operation.

The second reason is the thermal degradation of the structure during operation owing to the presence of bonding materials, viz. metallic cobalt, which is classic for PCD, in the intermediate layers of the composite and gradual dissolution of the diamond grains in the cobalt at heightened temperatures.

Different additional treatments of sintered materials (etching cobalt out of the near surface layer) as well as the development of fundamentally new recipes (systems with close CLTE, for example, diamond– silicon carbide) and hybrid materials (for example, layered composites of the type WC–Co–diamond–SiC) are being proposed in order to solve the problems of the low heat resistance of classic PCD composites.

Today, bilayer PCD composites obtained under pressures 6–8 GPa and temperatures 1400–1600°C are widely used for resource exploration and mining (drilling industry). A diamond-containing layer is formed owing to the bonding material Co–WC–C, which infiltrates from the substrate based on sintered tungsten carbide. The high mass content of cobalt in the diamond layer (from 6 to 10%) greatly lowers the heat resistance of the diamond insert. Owing to an increase of the cutting speed the temperature on the cutting edge of the composite increases to 700°C, which results in catastrophic thermal and chemical wear of PCD inserts.

Despite positive experience of super-hard materials use, the biggest market share in the cutting insert constructions based on tungsten cobalt hard alloys. One of the main reasons for this is a significant cost of super-hard materials, which for cubic boron nitride is from 1000 USD per kilogram, against USD 100 per kilogram for classic hard alloy. With this in mind, the use of super-hard materials must be selective, that is, they can be used as reinforced inserts of the of hard alloy blades.

In this case it is important to justify the most critical sections of the blades, the installation in which the super-hard elements can be justified in terms of the required crushing performance and the cutters wear resistance. Thus, the rotary drilling field expansion in rocks with the strength of 100–120 MPa is possible through the use of the combined cutting inserts equipped with elements made of super-hard materials of PCBN or PCD type in the design of cutting blades. The advantage of such designs is the selective reinforcement of inserts, allowing, on the one hand, using a super-hard material exclusively in areas of high stress concentration, and on the other considerably save expensive composites.
One of the variants of a cutter design with combined cutting inserts [8] developed with the implementation of applied scientific research, is shown in Figure 9. It consists of a cutter body 1, blades 2 and cutting inserts 3. Wherein the inner 4' and outer 4'' with respect to the cutter rotation axis x–x trihedral corners of the cutting inserts, are the inserts stress concentrators, and are made in the form of super-hard material element—PCBN or PCD.

![Figure 9](image1.png)

**Figure 9.** Drilling cutter with combined cutting inserts made of tungsten cobalt super-hard alloy and composite materials.

The idea of selective reinforcement of rock cutting tool inserts with super-hard materials also found its application in the design of experimental configurations of tangential rotary cutters (Figure 10). The developed cutting insert [9] is a two-stage form structure with the fixing stage, designed as a cone with self-retarding corners, and a cutting step performed in axial section according to the law of the power function $y = xn$ with $n = 2$. Cutting insert surface is reinforced with indenters made of composite wear-resistant material disposed in the plane along the spiral of Archimedes. The proposed design of the cutting insert will provide a definite cutter turning effect, due to the asymmetry of the load application.

![Figure 10](image2.png)

**Figure 10.** Various options for the location of super-hard materials of the tangential rotary cutter cutting insert surface made of tungsten cobalt alloy.

Rock crushing efficiency increase can also be achieved through the use of new, more productive and energy efficient modes of rock crushing. With respect to the rotary hole drilling it is a fundamental improvement of the drilling method for the purpose of its use in rocks of medium and above medium strength, i.e. in rocks with the strength factor $> 10$, where rotary drilling method is traditionally considered unsuitable due to insufficient strength of cutters and first of all because of the low wear resistance of the cutters. To expand the scope of rotary drilling method and for drilling speed increase super-hard materials must be used for the cutting insert reinforcement, and also more powerful drilling machines must be created.

The task of a substantial (by a few times) increase in drilling productivity, which means the drilling rate, can be expressed as follows

$$V = h \cdot n,$$

where $V$—drilling rate; $h$—advance per one bit rotation; $n$—number of bit rotations.
It is known that the relationship between the drilling rate $V$ and the tool number of revolutions $n$ is not linear in view of the optimal tool rotation frequency for rocks with different strength factor [3]. Accordingly, the drilling performance resource may be found only in the cutter advance increase, resulting in a significant increase of a cutting angle.

Table 2 shows the calculated data on the cutting angle changing depending on the cutter point radius remoteness from the blade axis and the length of its advance. A graphical representation of the calculated data is shown in Figure 11: it is seen that the cutting angle decreases from the inner edge to the periphery, while with the advance increasing the values increase. Thus, the increased advance drilling requires the use of cutters with reduced cutting-point angle in order to avoid touching the crushed rock with the back edge of the cutter. Decreasing the cutting-point angle of tools reduces their strength under the existing blade manufacturing technology, which in turn imposes limitations on the increase in the cutter advance rate. The design developed in the framework of the conducted applied research and shown in Figure 6b can serve as the way out from this situation. A characteristic feature of this design is the presence of variable front and rear cutting-point angles.

Table 2. Cutting angle dependence on blade point remoteness radius from the cutter axis and the length of its advance.

| Advance, mm | Blade point remoteness radius, mm |
|------------|----------------------------------|
|            | 5      | 7      | 9      | 11     | 13     | 15     | 17     | 19     | 21     |
| 3          | 5.46   | 3.91   | 3.04   | 2.49   | 2.11   | 1.83   | 1.61   | 1.44   | 1.30   |
| 4          | 7.26   | 5.20   | 4.05   | 3.32   | 2.81   | 2.43   | 2.15   | 1.92   | 1.74   |
| 5          | 9.05   | 6.49   | 5.06   | 4.14   | 3.51   | 3.04   | 2.68   | 2.40   | 2.17   |
| 6          | 10.82  | 7.78   | 6.06   | 4.97   | 4.21   | 3.65   | 3.22   | 2.88   | 2.61   |
| 7          | 12.57  | 9.05   | 7.06   | 5.79   | 4.90   | 4.25   | 3.75   | 3.36   | 3.04   |
| 8          | 14.30  | 10.32  | 8.06   | 6.61   | 5.60   | 4.86   | 4.29   | 3.84   | 3.47   |
| 9          | 16.00  | 11.58  | 9.05   | 7.43   | 6.29   | 5.46   | 4.82   | 4.32   | 3.91   |
| 10         | 17.67  | 12.82  | 10.04  | 8.24   | 6.99   | 6.06   | 5.35   | 4.79   | 4.34   |

Figure 11. The angle of cutting blade points depending on the distance from the drilling cutter rotation axis for different advance values.

An additional advantage of drilling with the increased advance is to reduce the rock abrasion impact on the cutter due to reduction of the $S$ path traveled by points placed on the cutter. In Figure 3a it is seen that the path $S$ traversed by the cutter blade point depends on the size of the specific advance. The total path travelled by the cutter blade point in $n$ revolutions, can be found as

$$S_{\sum} = S \cdot n$$

where $S_{\sum}$ —the total path travelled by the cutter blade point; $S$—the path traversed by the cutter point at one revolution; $n$—number of the cutter’s revolutions.
It is obvious that during drilling a drill with higher specific advance $h$ needs to make less cutter revolutions $n$ when drilling to the same depth. Then, according to the relation (4) reduction of the cutter revolution number $n$ will reduce the total path traveled by the cutter blade point.

It should be noted that physical phenomena accompanying the drilling process impose limits both to specific advance and to the tool revolution frequency. In addition, the parameters $h$ and $n$ are interdependent values, i.e. $n$ change causes naturally change of $h$. When drilling boreholes by rotary method using the existing machines a specific advance $h$ can vary from zero up to 10–15 mm/rev. Therefore, these values can be considered achievable in principle. At the same time there is objective limit for advance $h$ due to the tool blades limited strength.

Such restrictions are reduced to the limit values of axial forces and torques, which can withstand the cutters, without being destroyed. The strength of the existing drilling tools is limited by axial forces $P = 15–20$ kN. When you create a more durable material for cutting tools and increase axial forces up to 30 kN and above, you can expect a significant increase in drilling performance. Generalizations show that by using the existing tools one can stably provide averaged values of $h = 2.5–4.5$ mm/rev. The presence of cracks and interstices in rock mass necessitates control of specific advance for the purpose of the tool protection against overloads and failures (it is the simplest way to provide constant specific advance of the tool).

Attention should be also paid to the rotary drilling method dynamic features. The frequency of the drilling rod rotation and its torsional stiffness directly affects the speed of drilling. Specially conducted experiments showed that the rate of drilling with the same axial force and the same rotational frequency changes depending on the rigidity of the rods. Thus, after 20 times reduction of rod rigidity advance rate is reduced by almost half.

Under real conditions of borehole drilling rod rigidness may change more than 10 times, and during well drilling—100 times. The torsional stiffness of the rods directly affects the thickness of the removable chips. The smaller is the rigidity of rod, the smaller is the removable chip thickness.

In addition, it can be affirmed that the rock like a fragile medium under crushing in the mode of alternating chips is the source of vibrations in the rotating machine–rod–cutter system. Vibration parameters are determined by the elastic characteristic of the system and above all by the drill rod characteristic. Rod rigidity increase leads to higher frequency vibrations with small amplitudes without actually and finally affecting the drilling process result. The increased torsional stiffness of the rods and transmissions as well as reduced inertia of the advancing mechanism positively influences drilling efficiency.

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