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Chapter 6

Rock fragmentation and optimization of drilling tools

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Abstract

In this chapter possibilities of the improvement of drilling tool efficiency on the basis of the analysis of mechanisms of rock destruction and some methods of the information theory are considered. Physical mechanisms of destruction of hard and plastic rocks under different conditions of mechanical loading are reviewed. Some physical effects which allow to improve the efficiency of rock drilling tool are listed and ways of tool improvement are discussed. It is shown experimentally that the efficiency of drilling by a multicutter drilling tool may be improved by using the interaction between crack systems from neighboring cuts. It was suggested to apply methods of the theory of information to model complex processes of rock destruction and rock/tool interaction. Several examples of using of the information theory methods are given (stress distribution in disordered rocks, indexing effect, tool wear, optimization of complex tool construction). The general principle of tool improvement is formulated on the basis of the informational models.

1 Introduction

Tools for mechanical destruction of rocks are used on almost al1 stages and processes of mining. The improvement of the efficiency of destructing tools is one of most important sources of the improvement of mining and is therefore among first areas of application of the fracture mechanics of rocks. In this work, the following problems are treated: mechanisms of rock fracture under mechanical loading (how it proceeds in simplest and not so simple cases), rock cutting, physical effects which can be used to improve the destructing tool, limitations of tool improvement (the main of them is the tool wear) and some advanced methods of the rock destruction model1ing and the improvement of rock breaking tool.

Some materials from [2,9, 23, 26] are used and the Table 2 from [26] is re- printe
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2 Damage and Fracture under Mechanical Loading

Consider first the mechanism of rock fragmentation under indentation of axisymmetric indenter into a surface of elastobrittle rock. The destruction of rock under the indenter proceeds as follows [1-3]: firstly, the rock in the vicinity of contact surface is deformed and small surface cracks or microcracks appear a layer of destructed rock is formed under the contact surface and then the layer is crushed thereafter, the volume destruction of rock begins. A zone of inelastic deformation is formed under contact surface (this zone corresponds to the area of hydrostatic pressure) and the Hertzian cone crack initiates and grows. The rock in the zone of hydrostatic pressure under the indenter is destructed mainly through local shears and then powdered. The destructed rock under the indenter begins to expand and transmits the load to the rest of rock. Thereafter, the volume of rock bounded by the cone crack is failed and the axial cracks are formed. The cone crack changes the direction of its growth and/or branches what leads finally to spalling out of some volume of rock. After some volume of rock is spalled out, the powdered rock as well. as broken rock fly apart, and the next cycle begins, however, the rock under tool contains the axial cracks from previous loading and some volume of crushed rock from the crushed zone which has bounced apart. That is why the rock behavior during the next cycles differ quantitatively (but not qualitatively) from this in the first cycle of loading.

This process is shown on the Figures 1a (deformation of rock and crack formation) and 1b (further crack evolution and rock fragmentation).

Consider now the effects of rock plasticity on the mechanisms of rock fragmentation. As was shown by Eigheles [4], the plasticity of rock changes the mechanism of fracture drastical1y. He has shown that there are general1y two zones of ultimate state in rock under indenter: at the contour of the contact surface and at some depth under the contact surface. The growth of first zone leads to the formation of cone cracks, the growth of second one leads to the creation of a crescent plastic zone under the contact surface. The first zone has a dominant role in fragmentation of brittle rocks, the second zone in fragmentation of plastic rocks. The destruction of plastic rock in axisymmetric indentation proceeds as fol1ows [4,5]: the zone of irreversible deformation is formed under the contact surface at a depth which is approximately equal to the radius of contact surface. The zone grows and takes a crescent form. Simultaneously, the rock in the zone is crushed. Then, the zone reaches the free surface and that leads to the chipping as wel1. In this case the cone crack forms as wel1 but it remains rather smal1.

The shape of indenter influences the mechanism of rock fragmentation to a

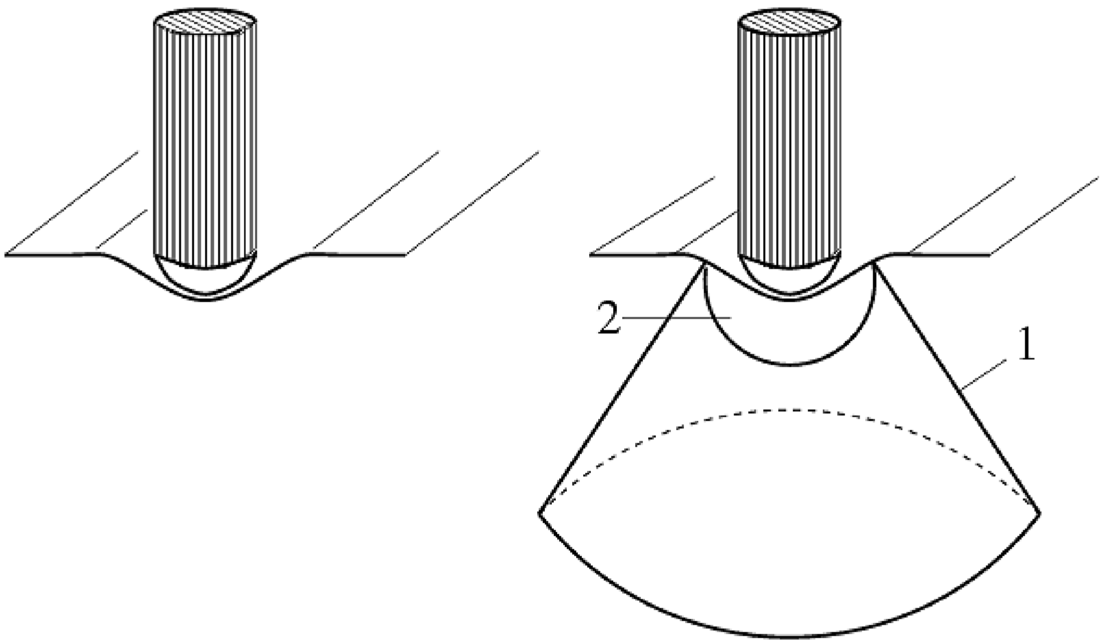


Figure la: Rock deformation and formation of crack system under vertical indentation. 1 -Hertzian cone crack
2 -zone of confining pressure

large extent. Rock fragmentation proceeds most intensively when the hard brittle rock is loaded by a spherical indenter, the hard plastic rock by conical one or the weak rock by wedge-shaped one (for static loading) [6]. At small impact load, the conical impactor makes the maximal crater as compared with other shapes of indenters at the same load. At great impact load however, the most intensive fragmentation of rock is observed under the loading by a cylindrical impactor [6].

In the indentation of spherical bit the cracks are initiated in the center of contact surface and not on the contour of contact surface, as differentiated from the indentation of cylindrical bits [7]. The prismatic bits lead to a more intensive rock fragmentation for hard, viscous, non-cracked rocks, whereas the cylindrical ones are preferable for cracked, brittle, weak rocks [7].

In pressing of the indenter with rectangular section into hard and plastic rocks, it was observed that. penny-shaped cracks are initiated in the vicinity of shorter side of the contact surface, grow and then form a large elliptic crack, which can be considered as corresponding to the cone crack in axisymmetric indentation [8]. The formation of the cracks in indentation of a rectangular indenter is shown on Figure 2.

Therefore, the destruction of rock under mechanical loading is a very complex process, which is caused by the interaction between. many levels of damage (local shears, growth of many macrocracks, spalling, etc) and is influenced by many

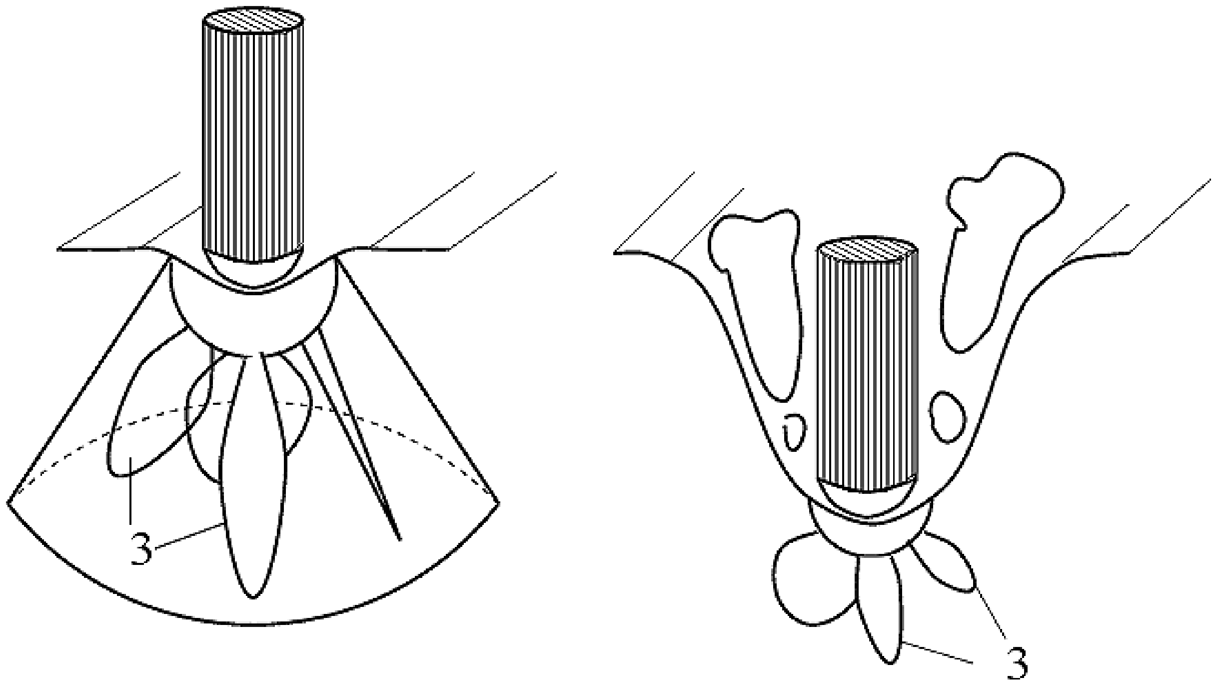


Figure 1b. Rock fragmentation under indenter . 3 -axial cracks.

factors, including random ones, like the composition of rock, the availability of microdefects, etc. Variations of the shape of indenter, rock properties and other parameters lead to drastic changes in the mechanism of rock destruction.

3 Cutting of Hard Rocks

A very important case of mechanical breakage of rocks by a tool is the rock cutting. Although the loss of energy due to the friction and wear in cutting is sufficiently more than that in impact destruction of rocks, cutting is a very perspective method of rock removal. It is determined by the fact that the cutting process goes continuously or almost continuously, rock is removed mainly due to the cutter movement., not. due to secondary effects like the crack interaction and the loss of energy due to dynamical effects (wave effects, vibrations, etc) is sufficiently less than in impact destruction of materials. The main disadvantage of cutting, i.e. fast wear and dulling of tool due to the continuous tool/rock interaction may be overcome in short time, in connection with the development of new wear-resistant and high strength materials, or using self-sharpening tools.

3.1 Mechanisms of Rock Cutting

Among the peculiarities of cutting as differentiated from the indentation, one can mention non-vertical direction of loading, non-axisymmetric shape of tool,

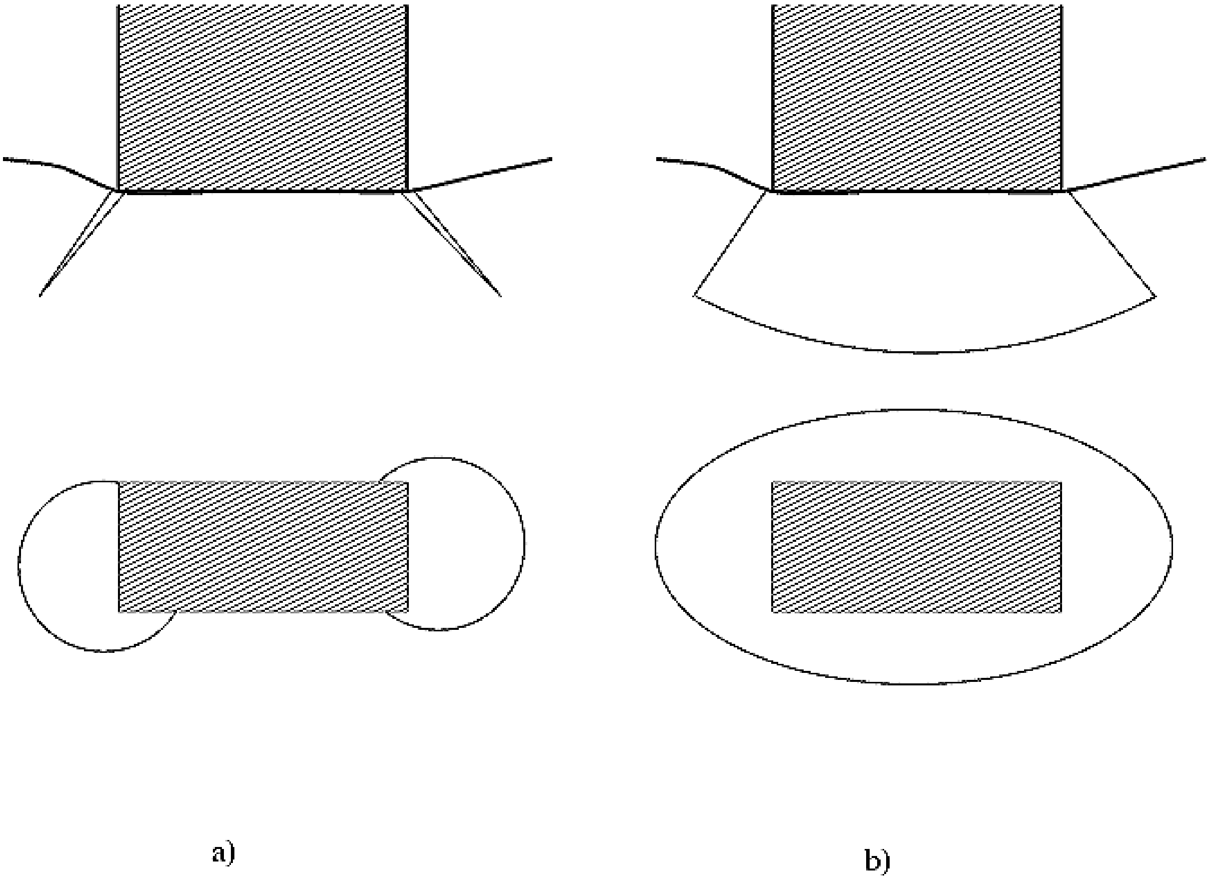


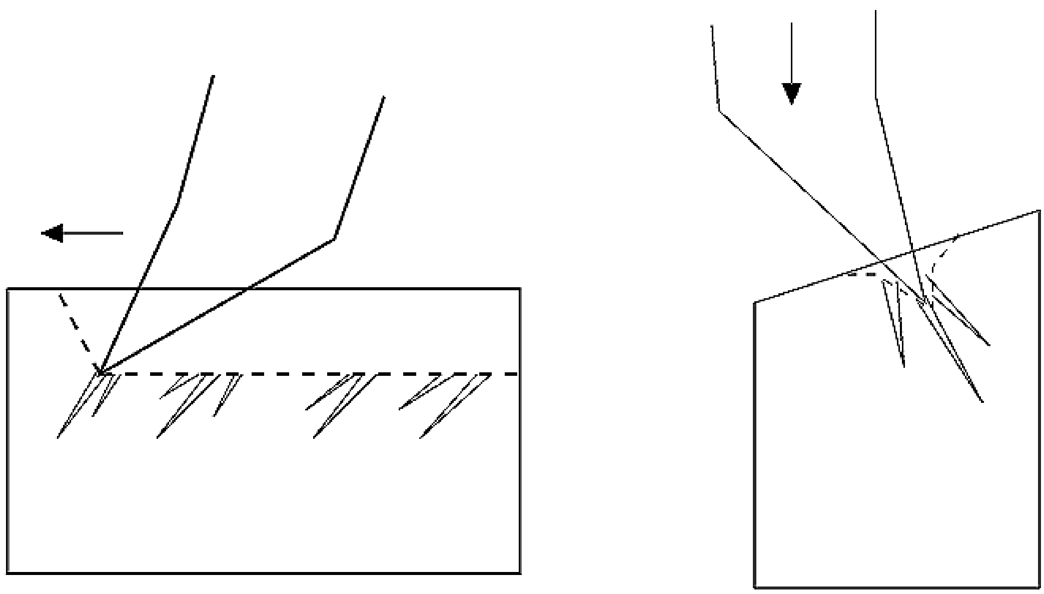
Figure 2. Formation of cracks in indentation of right-angled indenter . a) Crack initiation, b) Formation of a large elliptic crack

interaction of as-formed crack system with the free surface, continuity of the process of chips removal (although the chip is discontinuous). All these factors change the mechanisms of rock destruction sufficiently. In order to investigate the peculiarities of crack formation in cutting caused by above-listed factors, a set of experiments was carried out. The experiments were conducted on glass to make possible the observation of the crack distribution.

Blocks of glass with a trapeziform section (the upper side of the block was inclined at angles of *16°* and *8°* to the horizontal plane) were cut using cutters with lentils-shaped hard alloy inserts [9]. On the block with inclination *8°*, a second cut was made on the distance 28 mm from the axis of the first cut; it was done in order to appraise the influence of availability of a cut on the cutting force in neighboring points. The cutting rate was 21 cm/s and the depth of cut was 1 mm. The cutting force was also measured.

The appearance of cracks formed in glass in cutting is shown on Figure 3. The distance between the large cracks measured along the cut axis was 4-6 mm, which was approximately equal to the distance between the peaks which were seen on the cutting force oscillogram and corresponded to the instants of chip spalling.

It follows herefrom that the formation of the cracks in the damaged zone under cuts occurred simultaneously with the spalling of the chip segment. The angle of the force vector with the vector of the cutting rate, which was determined as arctangent of the tangential cutting force divided by the normal cutting force, is approximately equal to the angle made by the cracks in the damaged zone to the cut surface (i.e. 30 *-40°).* So, the direction of maximal crack under the cut coincided with the cutting force direction. Despite the fact that an inclined surface ,vas cut, the cracks under cuts did not deflect any more to one side of the cut axis than to the other (see Mishnaevsky Jr [9]).



a) b)

Figure 3: Cracks in glass in cutting (a) and in indentation of a cutter (b),

The peculiarities of cutting as compared with the indentation may be divided into the peculiarities caused by the non-axysimmetric shape of cutters and the ones caused by the translational continuous movement of a cutter. To isolate the peculiarities of the crack configuration caused just by the cutter shape and not by the cutting process as such, a vertical indentation of the cutter was carried out as well. The rate of indentation was 1 mm/s. The indenters have been forced down till first spalling. The indentation depth was approximately 2 mm. The appearance of cracks formed in glass in indentation is shown on Figure 3 as well. One can see that in both cutting and cutter indentation processes, neither Hertzian cone cracks nor circumferential cracks are formed. and all cracks are penny-shaped ones.

Therefore, one can conclude that a zone containing penny-shaped cracks (damaged zone) is formed under the cutter in cutting of brittle materials. The depth of the zone may be several times greater than the depth of the cut. From the above observation and the review by Mishnaevsky Jr [2], one can conclude that the crack system in cutting includes a crack which reaches a free surface (crack of chip spalling; the main crack which ensures the removal of rock), cracks which remain under cut (cracks of damaged zone) and zone of high damaged (powdered) material. So, the crack system which is formed in a brittle rock before the chip spalling looks like shown on Figure 4.

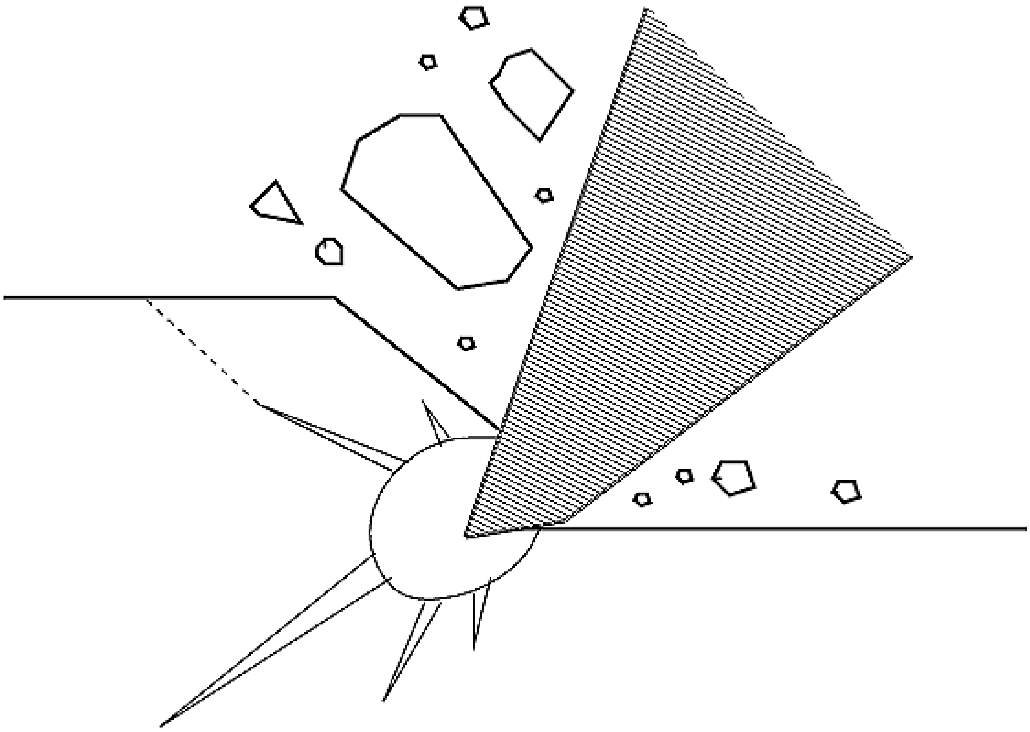


Figure 4. Crack system in brittle rock at chip spalling.

3.2 Cutting Force

Consider now the cutting force. The rock removal in cutting is determined by the discontinuous process of the spalling chip segments. To determine the energy needed for the spalling one can use the approximate formula from [9] obtained on the basis of the probabilistic analysis of crack propagation from the contact point to the free surface:

$$A=16.7\ Ga^2/K_{br} \qquad (1)$$

where a - depth of cutting, K_{br} is the brittleness coefficient of rock (which was introduced by Rzhevsky & Novik [10] as a characteristic of plastic/brittle properties of rock, and is defined as the ratio between the energy of the formation of new surface and the overall deformation energy in mechanical loading of rock) and G is the specific surface energy of rock. The mean cutting force Pc can be determined as the cutting energy (which is a sum of the chip spalling energy *A* and the frictional energy over the distance equal to the spalling step) divided by the corresponding distance (spalling step l_s -the distance between the two consecutive points in which the chip elements are spalled), which is calculated by the formula [1, 9]:

$$l_s = 8.2\ (G/g_{cr}\ B\ K_{br})^{1/2}$$

where g_{cr} is the specific energy of crushing the rock, and B is the cut width. The frictional component of force can be found on the basis of following reasoning. Sulakshin [11] has shown that in cutting of brittle rocks, a layer of finely crushed material is formed under the clearance face. This powdered brittle material plays the role of a lubricant between the cutter and rock. So, one can use the model of elastic friction (i.e. the friction force is supposed to be proportional to the vertical component of loading force). From the equations (1) and (2), one deduces the following formulas for the components of cutting force:

$$Pz=2a\ [g_{cr}\, GB/\ K_{br}\,(\cos\psi+\mu\sin\psi)]^{1/2} \qquad (3)$$

$$Py = 2a\ (g_{cr}\, GB/\ K_{br}\,\sin\psi)^{1/2} \qquad (4)$$

where μ-elastic friction coefficient, ψ -angle between the direction of the summary force and the horizontal surface.

3.3 Interaction between Crack Systems from Neighboring Cutters

As shown in [1, 9] the volume of detached rock under indentation of several indenters can be increased by the using of the interaction of crack systems from neighboring indenters without an increase in loading energy (see Section 4.2). This effect is determined by the fact that two large cracks, the distance between which is of the same order as their size, can join together due to the interaction of their stress fields [1,8].

The large cracks are formed under cuts as well. It is reasonable to assume that during the cutting process, the barrier between cuts can be also spalled owing to the interaction between neighboring systems of cracks. It means that there is a possibility to increase the removing capacity of multicutter tool by using the interaction between the damaged zones. Taking into account the appearance of crack systems in the cutting process (see Figures 3,4), it is possible to present the process of barrier spalling, as shown in Figures 5a and 5b. The condition of barrier spalling can be formulated as follows: the barrier between neighboring cuts is spalled due to the interaction between cracks provided that the largest cracks under neighboring cuts are very close one to another or even touch. If the cracks are penny-shaped and each of them is symmetric about the axis of corresponding cut, the condition of barrier spalling can be stated also in the following way: the barrier is spalled, if the distance between the axes of neighboring cuts is no more than the linear size of maximal cracks under cuts (i.e., the depth of damaged zone, approximately) .

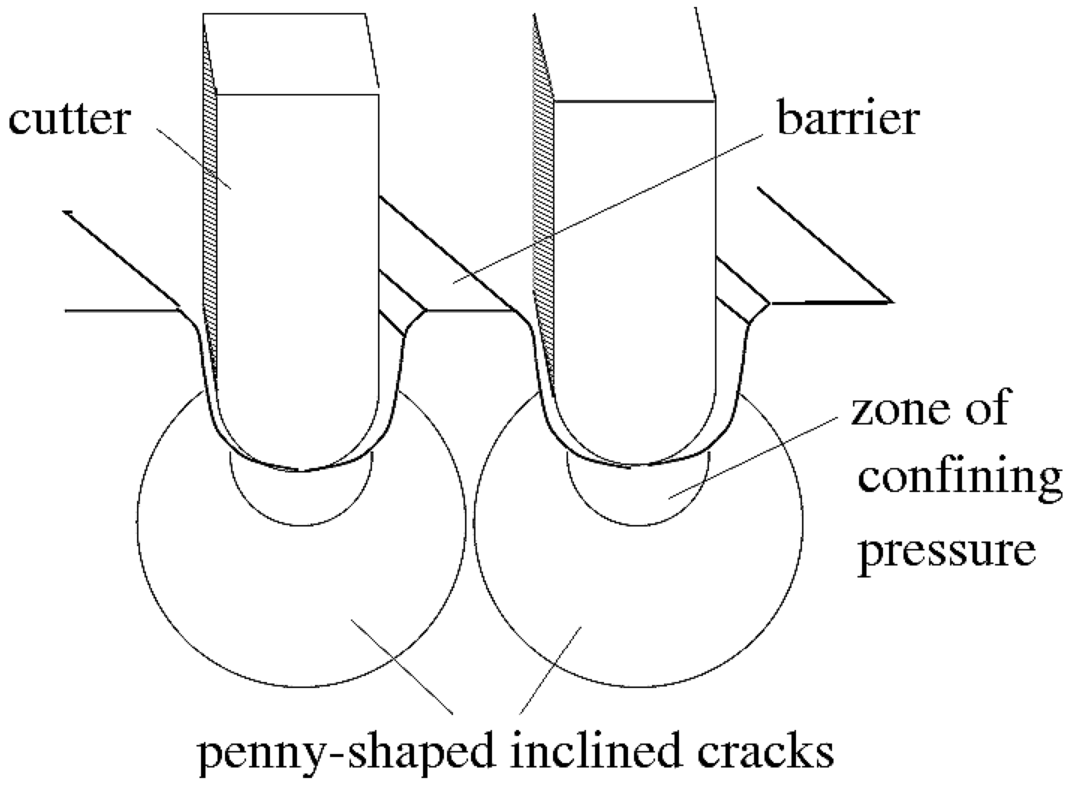


Figure 5a: Spalling of barrier between cuts. Crack interaction.

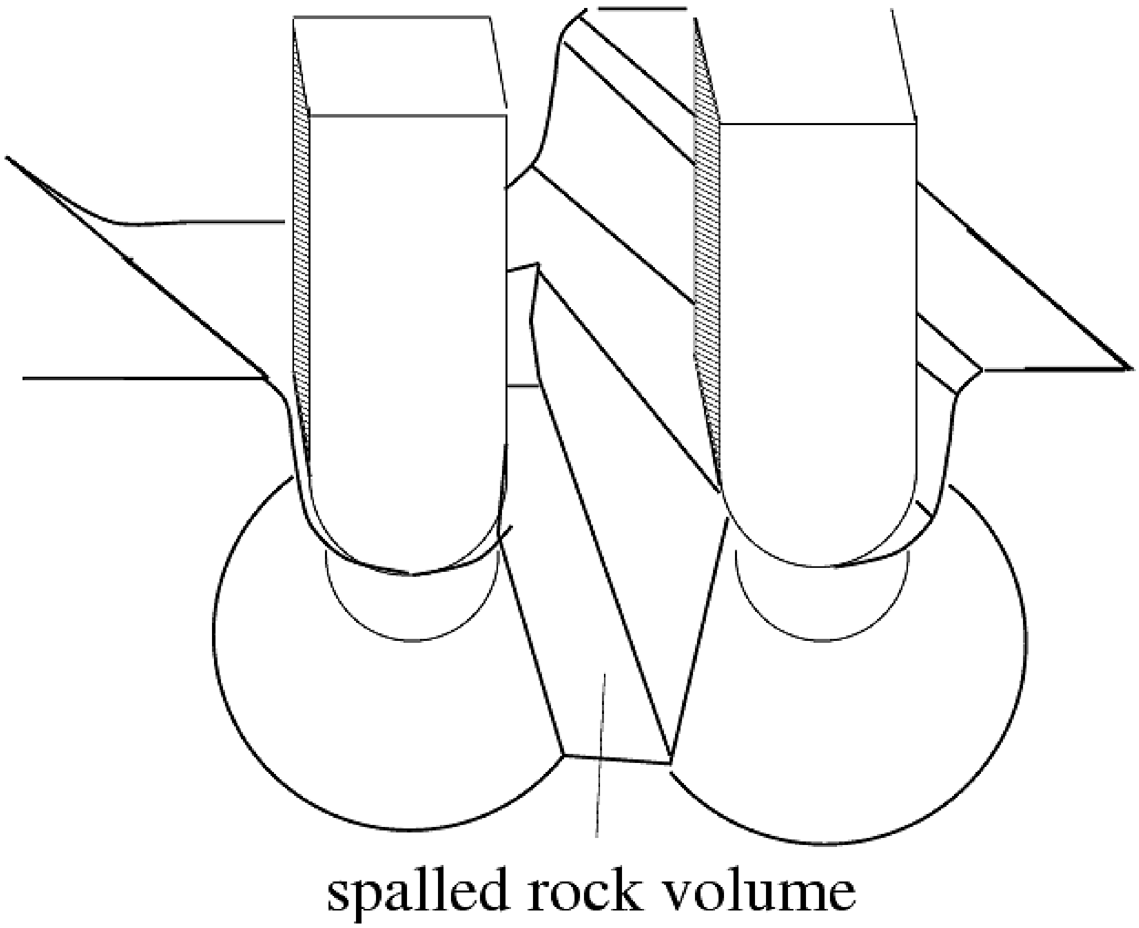


Figure 5b: Spalling of barrier between cuts. Removed volume of rock.

This condition may be written as follows:

$$Bc < h- K_s B \quad (5)$$

where B_c is the barrier width (or the distance between cuts), *h* -linear size of the crack formed under the cutter (this value can be calculated by formulas from [1,9]), B -cutter width, K_s is the coefficient of the cutter shape. The coefficient K_s characterizes a probability of initiation of a crack in different points through the cutter width. This coefficient can be determined in the following way: when stress concentrators are located close to the side faces of cutter (i.e. the cutter is rectangular in face section), the cracks are nucleated rather near the side faces of the cutter, and this coefficient K_s can be taken to be zero. When the stress concentrators of cutter are located on the cut axis (i.e. the cutter is triangular in face section), the cracks are nucleated rather near the cut axis and K_s is equal to 1.

To verify the formula (5), the following experiments were performed. Marble

Table 1: Depths of cut at which barriers between cuts are spalled spontaneously (mm) and the distances between cuts (mm)

Distance between cuts	19	20	21	22	23	24
Depth of cut at which barrier is spalled	1.6	1.7	1.8	2.2	2.2	3.0

blocks were cut by a cutter with a rectangular section [9]. The cutting rate was 21 cm/s and the depth of cut was varied between O and 3 mm. To gain the variation of the depth of cut, the blocks were inclined with respect to the vector of cutter movement. The cutter width is 21 mm. After the first cut, a second cut was performed close to the first one. The distance between the cuts varied from 18 to 26 mm. The depth of cut at which the barriers were spalled was recorded. The experimental and calculated data on the depth of cut at which the barrier was spalled and the barrier width are given in Table 1.

One can see that the experiments confirm our assumption about possibility of spalling of barriers between cuts due to the interaction between cracks under cuts. It can be seen also that the greater the depth of cut, the greater is the allowable width of barrier between cuts.

Hence, the removing capacity of cutting by multiteeth tool can be increased using the interaction between the damaged zones under neighboring cuts. This may be achieved by locating the cutters at such a distance that the barriers between them are spalled. The distance between the cutters at which the barriers are spalled can be calculated using equation (5).

In this Section, the method of the improvement of the efficiency of multicutter tools is developed on the basis of the experimental study of the mechanisms of rock fracture in cutting. In the next Section, some other methods of tool improvement based on studies of mechanisms of rock fracture are described.

4. Physical Effects Which May Be Used in Tool Improvement

There are physical effects in mechanical loading or rocks which are often used in the improvement or tool and drilling process. Some or the effects are considered below.

4.1 Low Tensile Strength

The tensile strength of non-plastic rocks is sufficiently lower than their compressive or shear strength. It means that the efficiency of destructing tool may be increased if the tool is constructed or oriented in such a way, that large zones of high tensile stress are formed in rock.

The larger the zones of high tensile stresses as compared with the compression and shear zones, the more intensive and less energy consuming is the rock destruction (see Artsimovich[13], Moscalev et al [15]). This rule gives a number of possibilities of the improvement of constructions of drilling tools on the basis of a simple elastic simulation of stress distribution in rocks: actually, on the basis of numerically obtained stress distributions in loaded rock one can say how intensive the rock will be destructed, and which improvements of the tool construction would be desirable.

The effect of low tensile strength of rock promotes the rock fragmentation in impact drilling as well: under dynamical loading, the degree of brittleness of rocks increases, and therefore the sensitivity of plastic rock to tensile stresses increases as well. Practically, it means that one may intensify the fragmentation of non-brittle rock by imposing the impact load on the cutting force, which causes the embrittlement of the rock. It was shown by Artsimovich et al [12] that the superimposing the impact component on the cutting force leads to the formation of larger cracks under cuts, than at a simple cutting.

4.1. Crack System Interaction and Indexing Effect

The volume of removed rock at the same energy of loading may be increased if the destructing elements are located on the tool in such a way that the crack systems from neighboring elements interact. As applied to cutting, this effect was investigated in the Section 3. Consider the effect of crack joining here in wider sense.

The joining together of cone cracks from neighboring indenters was observed by Artsimovich [8, 13] in indentation in glass and marble specimens. In so doing, the volume of spalled rock from the joined crack systems was sufficiently greater than the sum of two independently spalled volumes from single indenters, The volume of rock spalled by two indenters was maximal when the distance between the indentation points was approximately 3.5-4 diameters of the indenters; the spalled volume was 1.3-1.4 (for marble) and 1.5-1.7 (for glass) times larger than the volume of rock removed by one indenter multiplied by the amount of indenters. In these experiments, the indentation forces were not changed; the increase in the volume of removed rock was achieved only through appropriate arrangement of indenters. Therefore, this effect can be very useful in optimization of constructions of multiteeth tools: the reduction of energy per unit volume of removed rock can be achieved by only optimal arrangement of destructing elements on a drilling tool, without any additional input of energy. The optimal arrangement of the destructing elements means here such distance between them, that the cracks from neighboring elements can join together. The spalling of large volumes of rock due to the joining of cracks from neighboring cuts without additional energy can be also ensured if radial (facing) and tangential cutters are located on the same tool: in this case, the as-formed cracks are directed along perpendicular planes, and they can simply join together .

Sometimes, the effect of interaction of crack systems is confused with the indexing effect (these effects are observed usually together and are difficult to separate). The indexing effect implies that if two indenters or cutters deform rocks one near another and their stress fields are superimposed, it leads to more intensive microcracking in rocks. This effect can be observed separately from the crack interaction on I y at initial stages of deformation of weakly damaged rocks. Mavlyutov [14] has studied the effect of several rectangular indenters on the energy consumption in rock fragmentation. He has shown that the energy consumption at simultaneous indentation of several indenters is lower than in successive indentation. In so doing, this effect becomes weaker with increasing of the amount of indenters. It was shown also that such an orientation of two rectangular indenters at which their shorter sides are parallel to the line between them ensures lower energy consumption in indentation than in the case, when their longer sides are parallel to the line between the indenters.

The indexing effect is studied theoretically in the Section 5.2.

4.3. Effect or Free Surfaces

One of very important effects in mechanical fragmentation of rocks is the effect of free surfaces available near the loaded volume. The specific energy of rock fragmentation in vertical indentation (which can be defined as the energy to spalling divided by the spalled volume of rock) near a vertical free surface (see Fig. 6c) can be less than the energy in simple indentation by a factor 30 to 100 [13]. Reversely, the specific energy of rock fragmentation in vertical indentation near a wall (see Fig. 6a) can be 20-50 times greater than the energy of simple indentation. For example, in indentation in marble, the ratios between the spal1ing forces near the wall P_l, in simple indentation P_2 and near the free surface P_3 (see Figure 6) are as fol1ows [13]:

$$P_1/P_2=0.55; \qquad P_3/P_2=1.53 \tag{6}$$

(the distances both between the wall and indentation point, and the free surface and indentation point were 6 mm in these experiments). The availability of free surfaces in the vicinity of a loaded volume reduces the degree of triaxiality of stressed state and the volume of crushed rock under the indenter w hat makes easier the removal of rock.

In order to use this effect in the real conditions- of multicutter drilling, a tool with two groups of cutters can be produced: first, leading cutters, which make

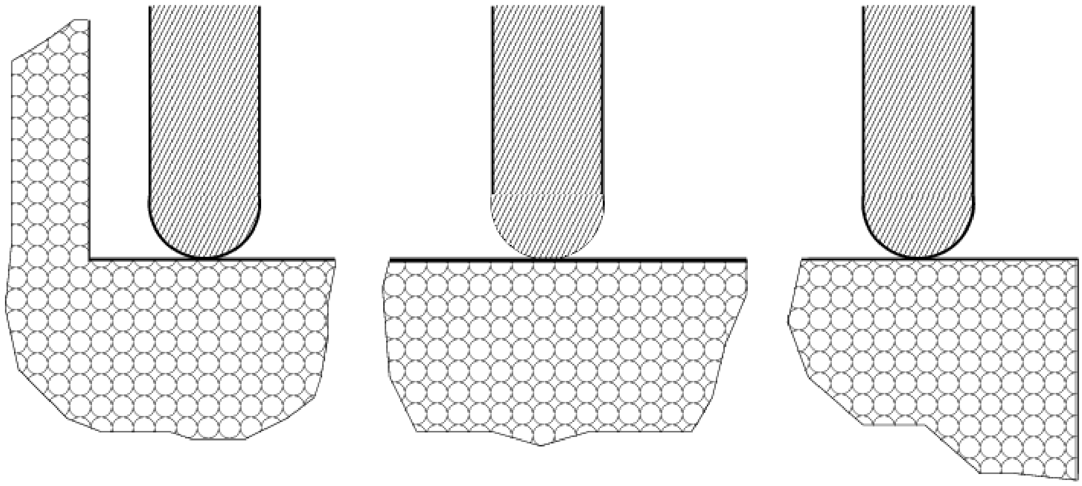


Figure 6: Indentation near a wall, into a Hat surface and near a free surface.

narrow cuts in the rock volume to be removed, and the second, main cutters, which move the rock weakened by the free surfaces (cuts made earlier) [15]. The summary energy of rock removal in this case is less than at the usual arrangement of cutters (Moscalev et al [15]). It was suggested also to create vertical cracks on the boundaries of the tool/rock contact area with the use of tangential cutters, w hat would facilitate the rock removal inside of the zone limited by the cracks [8]. Then, the effect of free surface is used when the drilling tool has a form of a cone with several steps: in this case, the cutters which remove rock first, are located on the more extending part of tool than the cutters which go after them, and the first cutters create free surface which facilitate the rock removal for the subsequent cutters [13].

Hence, the effect of free surface in the multicutter tool improvement can be used practically as follows: some cutters on the complex tool should create additional free surfaces (cuts, steps) in the rock to be removed, and therefore, facilitate the rock removal for the other cutters; the first group of cutters may be very narrow or non-symmetric, in order to reduce the cutting forces on the cutters, which do not seek to remove rock, but just to weaken it.

4.4 Rock Weakening by Available Microcracks

The strength of rock decreases drastically when some micro- or macrocracks are available in rock. When a cutter or bit interacts with rock, it causes not only the chipping out and removal of some volume of the rock, but also produces micro- and macrocracks outside the removed volume. When the next cutter or tooth interacts with this rock volume, the rock is weaker and may be removed by applying lower force.

The first cutters (or teeth) remove some rock volume and simultaneously produce cracks in the rest of the massif, which will be removed later by other cutters or teeth. Therefore, if the cutters or teeth are arranged in such a way that the damage caused in non-removed rock by the cutters which have gone earlier is used in order to reduce the cutting force of subsequent cuts, the summary energy consumption of drilling is reduced.

4.5 Fragment Size Distribution and the Drilling Efficiency

The energy consumption in drilling is the greater the greater the share of small and smallest fragments of rock in the destructed rock [15]. In order to reduce the energy consumption in rock fragmentation, one should minimize the volume of crushed and powdered rock, and ensure the removal of rock rather by the propagation of macrocracks. The crushing of rock occurs in the zone of confining pressure under cutters; so, the volume of the zone should be minimized. The further overcrushing of rock proceeds if the fragments of destructed rock are not removed from the area of rock/tool interaction (see the Section 4.8).

Then, the removal of rock by growth of macrocracks, which reach free surfaces or join together with other macrocracks, presents also a preferential way of rock removal: the new surface formed in rock is relative small as compared with crushing whereas the volume of removed rock is large. So, the construction of tool should ensure minimal volume of confining pressure in the loaded rock, and the spalling of relative large rock volumes due to the macrocrack propagation.

4.6 Surfactants in Drilling Fluids

Let us look now at some effects which are not related directly with the construction of tool, but can be used in order to improve the drilling efficiency as well.

The energy consumption in rock fragmentation depends on the fracture toughness of rock [1]. The addition of surface-active chemical additives in drilling fluids may reduce the fracture toughness of rock [16, 17]. Kim & Staroselsky [17] have shown that sizes of cracks under cuts are sufficiently greater in cutting some types of rocks with surfactants as compared with the dry cutting. They demonstrated also experimentally that the resistance to cutting with surfactants is sufficiently

(45% -55%) lower than that in dry cutting. Therefore, the addition of surface-active additives to drilling fluids may intensify the rock fragmentation in drilling.

4.7 Tangential Motion of Indenter

If the indentation force is tilted with respect to the rock surface (it is the case in cutting, tangential motion of indenter or indentation into an inclined surface), the mechanism and intensity of rock fracture are changed due to the availability of a large zone of tensile stresses behind the tangentially moving cutter/indenter [13]. Due to the low tensile strength of rocks the availability of the large zone of tensile stresses may lead to the sufficient increase in the intensity of rock fragmentation. The size of the tensile stress zone behind the cutter is the more the greater is the ratio between normal and tangential components of cutting force [13] .Another effect of tilted force is that the zone of powdered rock is not axisymmetric, but tilted to the rock surface; therefore, the spalling of chip in front of the powdered zone proceeds at lower applied force [9]. So, the loading of rock with translational movement of the loading element ensures more effective removal of rock and lower energy consumption as compared with the vertical indentation, due to the more intensive fragmentation of rock in the tensile stress zone behind the cutter and simpler spalling of rock in front of the powdered zone under the cutter .

From the interrelation of the ratio between normal and tangential components of cutting force with the size of the tensile stress zone behind the cutter (and consequently, with the intensity of rock fragmentation) one may draw a paradoxical conclusion: decreasing the friction coefficient between tool and rock may have an adverse effects on the intensity of rock fragmentation (since the decrease of the friction coefficient causes also the decrease of the ratio between normal and tangential components of cutting force, and consequently decreases the tensile stress zone behind the cutter). Some experimental justifications of this conclusion may be found in [18] ; however, the changes of the friction coefficient influence the rock fragmentation in a complex manner, and other factors may suppress this effect.

4.8 Effect of Remains of Destructed Rock On the Course of Rock Fragmentation

The fragments of removed rock, which remains on the surface of rock after the chipping out influence the rock fragmentation efficiency as well.

Mavlyutov [14] has shown that the mechanism of rock destruction in a static indentation of a cylindrical indenter through a layer of powdered rock is fully similar to the destruction mechanism in the indentation of a spherical indenter . The specific energy of rock fragmentation in dynamic indentation through the layer of powdered rock is 18 % more than that in the indentation in the pure surface f181.

The accumulation of fragments of destructed rock in front of the cutter face leads to the overcrushing of rock, and to additional losses of the energy. This can be avoided by using of the cutters with one-sided bevel on the cutter face [15].

So, the rapid removal of remains of destructed rock from the area of rock/tool interaction (by the means of constructive modifications of the tool, or by other means, like drilling fluids) is a very important for the efficiency of drilling .

4.9 Effect of Impact Energy on the Efficiency of Rock Fragmentation

The dependence of the specific energy of rock fragmentation under dynamical loading on the impact energy is non-linear. The specific energy of rock fragmentation (which may be defined as the impact energy A divided by the volume of rock removed by the impact V) decreases with increasing the impact energy, yet, the function A/V versus A has a local maximum [14]. It means that although one may recommend high loading energy in order to decrease the energy consumption in drilling in general (the wear of tool and technological problems of high impact energy apart) , in the small ranges of energy variation the increase of the energy may lead even to the decrease of the efficiency of rock fragmentation.

The physical effects considered in this Section may be used in order to increase the efficiency of drilling tools and drilling processes. The described recommendations for the tool improvement are based mainly on experimental and in some cases on numerical data. However, general recipes of tool improvement are required for efficient (and also computer-aided) tool design. Such general solutions may be obtained on the basis of comprehensive theoretical models of rock fragmentation in drilling, not just on studies of some aspects or processes. The direct way to construct such a comprehensive theoretical model is to use the methods of continuum, fracture and damage mechanics. However, the development of the model is fraught with a number of theoretical and numerical difficulties. In the next Section, some other approaches to the modeling of rock destruction and to the tool improvement, which are based on the informational methods are discussed.

5 Informational Methods in Modeling of Damage and Fracture

As discussed above the rock destruction under mechanical loading is a very complex process, with the formation of interacting complex patterns (crushed zone, cone crack, etc). The traditional approaches (theories of elasticity and plasticity, fracture and damage mechanics, etc) describe basically elementary processes of rock destruction, like elastic deformation or crack growth, or their relative simple combinations. However, in order to optimize drilling tools, the model of whole process of rock fragmentation which takes into account the complex effects and interrelations between elementary processes is required.

That is why it is reasonable to apply to the modeling and optimization of rock destruction not only the direct approach which anticipates the development of models of elementary processes, like elastic deformation or fracture, and then assembly /combination of the elementary models to describe the full complex process and optimize it, but also other methods based on the theory of information [1, 19, 20].

This approach can be described shortly in the following way. Any complex system with known input and output "signals" and unknown structure is considered as a "black box" .If it is known how the elementary processes interact, the system may be presented as consisting from several interrelated subsystems (which can be "black boxes" in turn). If the processes to be considered are time- dependent, the dynamical system approach is used [1, 22]. In many cases, the system contains also feedbacks: for instance, in wear (changing of shape of contacting bodies due to the wear influences the rate of wear [23]) or in damage growth (the energy dissipation due to the damage formation slows down the damage growth [24, 25]). Any non-homogeneity or asymmetry, which can be described by a set of (non-rectangularly distributed) random values, may be considered as a sequence of signals which carry some information [19]. An interaction between two systems, which leads to a change in the structure or properties of the systems can be described as an information (signal) transmission. For instance, an inter- action between a tool and a specimen, after which a crater replicating the tool shape remains on the specimen surface, can be taken as a transmission of a signal about the tool shape from the tool to the specimen. Clearly that this model can be applied also in more complex cases (for example, cutting, drilling or other technological processes), when the correspondence between the process of signal transmission (indentation) and the content of "message" (in this case, shapes of tool and crater) is not so evident [26]. Then, the informational entropy of distribution of some parameters of a system characterizes uncertainty, role of random factors as compared with deterministic ones in the behavior of the system [27]. Using the postulate about the equivalency of negentropy and information, one can set up a correspondence between the informational and thermodynamical models of damage and fracture [20, 24].

The methods of the information theory (or more generally, the theory of complex systems) allow to describe the complex process of rock destruction in the cases in which using of the continuum and fracture mechanics approaches may lead to great numerical difficulties [20]. These methods can efficiently amplify the traditional methods of modeling of the material destruction.

In order to illustrate the possibilities of this approach we present here some simple examples of the application of the theory of information to the modeling of rock/tool interaction, which can serve as a basis for the development of a more general theory.

5.1 Informational Description of Tool Shape

The intensity of rock destruction in indentation is determined by the shape of indenters. Let us look at three simplest forms of indenters: spherical, conical and cylindrical ones. Although the shapes of indenters differ evidently and the peculiarities of destruction for each of the indenters have been well investigated, there is no quantitative parameter ("input" or apriopi characteristic) which can characterize the form, serve as a criterion for their comparison and which may be generalized for more complex cases of rock/tool interaction.

The experiments on the indentation of differently shaped indenters, described in [6], have shown that the volume of craters of spalled rock is maximal for conical, minimal for spherical and medium for cylindrical indenters. If one compares the result with the contact stress distributions for these cases [29], one can see that the maximal volume of crater corresponds to the most sharp curve of contact stress distribution, whereas the minimal volume corresponds to the most homogeneous contact stress distribution.

One can suppose that the "sharpness" (i.e. non-homogeneity) of contact stress distribution is a parameter which determines the intensity of rock destruction (in this case, the volume of crater). To characterize this "sharpness" of distributions quantitatively for arbitrary tool shape (including, for example, a non-axisymmetric drilling bit with many teeth), one can use an informational entropy of contact stress distribution [1, 26].

Let us suppose that a contact stress distribution function is given in following general form:

$$\sigma_c = F(x, y, z) \tag{7}$$

where σ_c is the contact stress in a point, x, y, z -coordinates or a contact point. The function (7) is determined by the tool shape and the stress-strain relationship for a given rock. Peaks or this function correspond to stress concentrators on the tool surface. Quantifying the range or contact stress variation, one can obtain from eq. (7) the probability distribution or contact stress over the contact surface:

$$p(\sigma_c) = (1/N_L) \Sigma Y[F(x,y,z); \sigma_c] \tag{8}$$

where $p(\sigma_c)$ is the probability that the contact stress in a point is equal to the value σ_c, N_L -the amount of quantisation levels of σ_c, j -the number of a contact point, Y[] -step function, Y[X1; X2] = 1, when Xl = X2 and is equal to 0, otherwise.

The informational entropy of contact stress distribution can be calculated by the formula:

$$Hc = -\int_{\sigma c} p(\sigma_c) \ln p(\sigma_c) \, d\sigma_c \tag{9}$$

The greater the parameter Hc, the more non-homogeneous the contact stress distribution. This value characterizes the "sharpness" (non-homogeneity) of con- tact stress distribution for arbitrary function F, and thus, for arbitrary drilling bit shape. This parameter does not depend on any kind of symmetry of tool, and may be used for tool shapes of any complexity.

In order to investigate the influence of the parameter Hc on the intensity of rock destruction, a series of contact stress distributions with different parameters Hc was taken. Each of the distributions corresponds to some shape of indenter. The initial damage parameter in rock for each value Hc was calculated.

The damage evolution in contact interaction proceeds as follows: first, the surface damage is formed in the vicinity of the contact surface, and then the damage density begins to grow. The damage growth rate is the more the greater the damage parameter [25, 29]; therefore, the initial damage (in this case, surface damage) determines the damage in the rock at later stage of destruction. That is why we use here the surface damage as a characteristic of the damaged state of the rock.

The contact stress distribution (i.e. the function (7)) can be presented as a power function in a rather general case:

$$\sigma_c (x) = q_1 (x/a_c)^{q_2} \tag{10}$$

where $2a_c$ is the width of contact area, q_2 is a power coefficient which determines the appearance of contact stress distribution (when $q_2 > 1$, tool is extremely sharp; when $0 < q_2 < 1$ it corresponds to the more realistic case when the tool has a convex surface), q_l -a coefficient which depends on the applied load, x- a distance between a point and the axis of tool. The applied force is supposed to be constant. The multiplier q_l is determined by integrating the equation (10): $q_l = (q_2 + 1)P_a/(a^{q2+1})$, where P_a is the applied load. The values of initial damage R_0 and the contact stress entropy were calculated by formulas (9) and (8). The damage parameter was determined by the formula (14). The coefficient q_2 was varied from 0.25 to 1.8. The following input data were used: $2a_c = 10$; $P_a = 25$, the number of quantifying levels for stress $N_L = 800$, the step of discretisation of contact stress was 0.1, the contact surface was discretised for 1000 elements, the average local strength of the rock a_{cr} is equal to 170. A plot of surface damage R_0 versus the contact stress entropy obtained as a result of the calculations is presented in Figure 7. From this Figure one can see that the surface damage increases monotonically with increasing the entropy of contact stresses (at constant load).

Therefore, one can conclude, that the destruction ability of tool is increased with increasing the informational entropy of contact stress distribution.

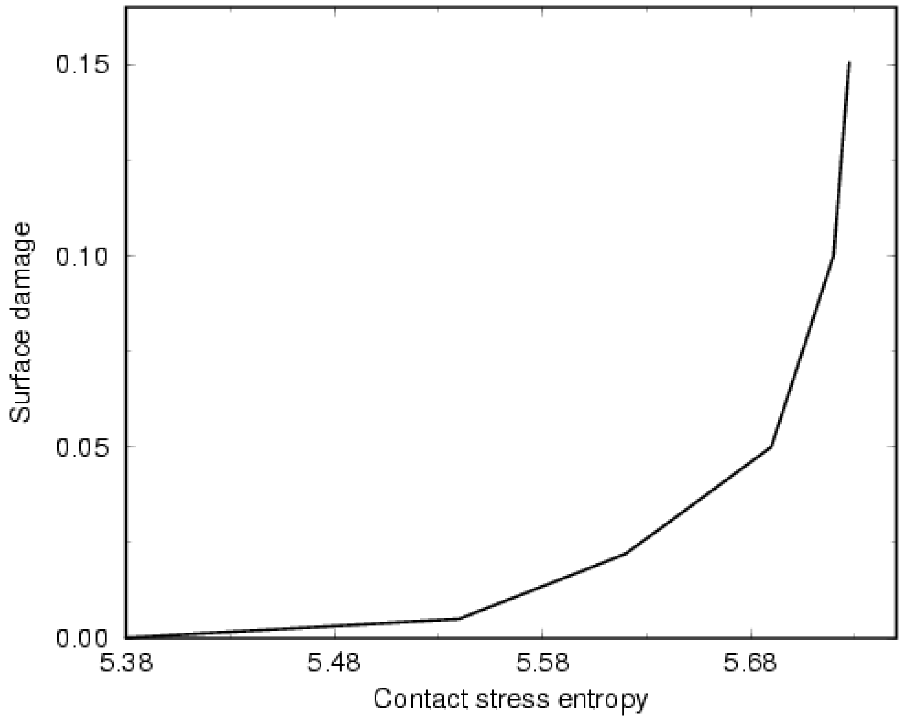


Figure 7: Surface damage in rock plotted versus the contact stress entropy [26].

5.2 **Entropy Maximization** and. **Indexing Effect**

The stress distribution in rocks depends on a number of random factors: spatial distribution of minerals, distributions of the grain sizes, shapes, properties and orientations, superposition of stress field caused by each stress concentrators in materials, variability of properties and types of local behavior, cooperative effects, etc. These effects cause high variability of the local stresses from the values which are calculated with the use of the averaged constants and elastic or plastic solutions. Lippmann [30] noted that the stress in disordered materials presents a sum of an averaged stress and a microstress caused by the local microstructure and heterogeneities. This microstress can be considered as a local fluctuation of stress field. The microstructure of heterogeneous rock, distribution of minerals, their sizes and orientation are usually unknown. Mishnaevsky J r & Schmauder [25] have shown, that a material becomes more heterogeneous with increasing damage parameter and approaching to failure. It leads also to the increase of stochastic component in the local stress in the materials.

So, the problem of description of stress state in disordered materials is a problem of determination of parameters under conditions of the lack of required information. In order to solve such problems, one can use the maximum entropy method [21]. Consider the stress distribution in a loaded rock. A stress in a point is determined by the interaction between stress fields of ail nearest stress concentrators (inclusions, hard particles, microcracks, pile-ups of dislocations). The stress distribution which is determined by the superposition of stress fields of a wealth of randomly distributed stress concentrators can be taken as random and may be described by a probability function, for instance, by a probability function of the von Mises equivalent stress (for simplicity, we shall use here this scalar value, but this approach can be applied to describe a distribution of vector components as well). In the case of homogeneous rock, this probability function is reduced to the Dirac delta function. For the case of disordered materials, this probability function is determined from the condition of maximal likelihood of a given stress state, which corresponds to a maximal entropy of the system. In order to determine the probability function which describes the stress state, all available data about the stress distribution are introduced into the corresponding Lagrangian, and the probability function is determined on the basis of the condition of the maximum of this Lagrangian.

For example, consider the case when the rock is extremely heterogeneous and almost no information, except for the energy of loading A and strained volume V is given. One should note here that this model may be applicable to thermic loading, but not to mechanical loading at which conditions and direction of loading are known always; the other case in which the model can be applied is that when there is two levels process (for example, micro- and macrofracture) in a problem to be considered and one may neglect geometrical effects on one of the levels.

The restriction on possible probability distributions of stress is minimal and involves only a prescribed average specific elastic energy a_e: $\sum p_a(a_e)\, a_e = A/V$ where $p_a(a_e)$ -probability function of the specific elastic energy, A -energy of loading, V - strained volume. (This equation is written for the distribution of specific elastic energy $a_e = k_p\sigma^2$, and not stress for the sake of simplicity; here $k_p = (1+\nu)/3E$, σ- von Mises equivalent stress, E- Young modulus, ν -Poisson's ratio). The condition of maximum entropy is written in such a way:

$$-\Sigma\, p_a(a_e) \ln p_a(a_e) \to \max, \tag{11}$$

The Langangian of the system looks as follows [21]:

$$Z = \lambda_1 \left[1-\Sigma\, p_a(a_e)\right] + \lambda_2 \left[A/V - \sum p_a(a_e)\, a_e\right] - \\ - \lambda_3\, \Sigma\, p_a(a_e) \ln p_a(a_e), \tag{12}$$

where λ_1, λ_2, λ_3- coefficients. Minimizing the Lagrangian ($dZ/\, p_a(a_e) = 0$), one can obtain:

$$p(\sigma)= \lambda_1\sigma(2k_p)^{1/2} \exp(-0.5\, k_p\, \lambda_1\, \sigma^2) \qquad (13)$$

where $\lambda_1= \lambda_2= \lambda_3= V/A$, $p(\sigma)$ -probability function of von Mises equivalent stress.

If one states the condition of local failure (the critical level of equivalent stress, for instance), one can determine the probability of microcrack formation (i.e., damage parameter) as well:

$$R=\int_{\sigma c} p(\sigma)d\sigma= \exp(-k_p\, \lambda_1\, \sigma^2{}_{cr}) \qquad (14)$$

where σ_{cr} -critical level of the equivalent stress. This formula relates the micro- crack density and the probability function of stress distribution (which depends on the heterogeneity of rock). One should note here that the value σ_{cr} in such a medium should be a random value; therefore, the formula (14) can be used only as a rough approximation (the value σ_{cr} means therefore an averaged critical level of the equivalent stress over the loaded volume).

One can introduce into the Lagrangian (12) also other boundary conditions than the simplest ones used above, what may allow us to take into account the structure of the material and the strain compatibility. For example, one can exert into the Lagrangian the correlation function of stress in different points of strained volume (that can be considered as an analogue of strain compatibility equations as applied to the brittle disordered materials), or the average deviation of stress in strained volume points from the stress being calculated by an elastic problem solution (with averaged elastic constants).

Thus, the developed method makes it possible to obtain the probabilistic description of stress distribution and damage in disordered material, taking into account the lack of information about the material structure and the available (insufficient) information about it.

Practically, it means that the problem of stress state modeling is solved from the "other end": usually, one supposes that an ideally elastic material (or a material with other fully known properties) is considered, and then one begins to introduce corrections, which should allow for heterogeneity and other deviations of the material from the ideal case; here, we accept initially, that the information about the material structure and behavior is lacking, and then begin to introduce any available information into the Lagrangian.

In order to illustrate possibilities of practical using of developed model, consider the influence of superposition of stress fields and the effect of indexing on the damage in rock.

Consider simultaneous and successive loadings of a rock by several (in this case, two) indenters. In first case (i.e. the simultaneous indentation), there are the superposition of stress fields from the indenters (it is supposed that the indentations are made on rather small distance, which is about 1... 10 indenter radii). In the second case (the successive loading), the rock is unloaded during the time interval between indentations and the stress fields do not interact (the influence of as-formed damage on the formation of new damage is neglected).

Let the function p(σ) for single indentation be denoted by $p_0(\sigma)$.The damage parameter corresponding to the single loading is designated by R_0. Consider the values of damage R for simultaneous and successive loading by two equal loads. In line with the damage accumulation hypothesis, one can write for the case of successive loading: $R_{suc} = 2 R_0$. In the simultaneous loading (when the stress fields are superimposed), the damage is equal to the sum of damage parameters from each indenter plus some value ßR which is determined by stress field superposition: $R_{sim} = 2 R_0 + \Delta R$. By its meaning, the value ΔR is a probability of local failure (i.e. the microcrack formation) due to the stress fields superposition in a point in which the rock does not fail at the successive loading. This probability can be determined as a probability of coincidence of two events: an event A that $\sigma < \sigma_{cr}$ in the point at successive loading, and an event B that 0" > O"cr in the same point at simultaneous loading: $\Delta R = Prob_A \, Prob_B$, where $Prob_{A,B}$ is the probability of the event A or B. Determine the value $Prob_A$ and $Prob_B$. It is clear that $Prob_A = 1- R_{suc}$ and $Prob_B$ is equal to the probability function $p_{sim}(\sigma)$ of stress distribution for simultaneously applied loads integrated over stresses from critical stress to the infinity. Then, the distribution of stress for the case of superimposed fields can be found as a convolution of two distributions $p_0(\sigma)$: If one defines a coefficient of indexing as a ratio between the damage in rock in simultaneous and successive loading by the same load $\eta = R_{sim}/ R_{suc}$, one can calculate this value with the use of eq. (14) as fol1ows:

$$\eta = 1+0.5 \int_{\sigma cr} p_{sim}(\sigma) \, d\sigma$$

Substituting eq. (14) to eq. (15), one can obtain: $Prob_B = (I + \lambda_1 a_{cr}) R_0$. After same rearrangements, one derives:

$$\eta=1+0.5 (1-2 R_0)(1-\ln R_0)$$

Let us take $R_0 = 0.3$ (i.e. the cracked part of loaded volume of rock is considered the value 0.3 presents the critical damage density at which a large crack is formed [29]). Then, we have from eq.(16) : η = 1.44.

To verify this result, one can use the experimental data from [31]. In this work, the maximal distances between indenters which ensures the spalling of barriers between craters in successive and simultaneous indentations of two and three indenters in marble blocks were determined. It was shown that this distance for simultaneous indentation of two indenters is equal approximately to 8 radii of indenters and for the case of the successive indentation presents 6 radii. So, the simultaneity of indentations leads to the increase by 33% in the linear size of as-formed cracks. The area of new surface may be taken as being proportional to the damage parameter to the power 2.42 [25]. So, the linear size of as-formed cracks should increase at the expense of the simultaneity of indentation by the value $\eta^{1.21}$, in accordance with the present model. If we use the value $\eta = 1.44$, it corresponds to the increase by 1.55 in the linear crack size. The deviation is about 16 %. One can see that the developed model gives the results which are rather close to the experimental data despite the rough assumptions used in the model.

So, the intensity of rock destruction increases sufficiently (almost 1.5 times) in simultaneous indentation of several indenters as comparing with the successive indentation of the same indenters. Therefore, one may conclude that the arrangement of cutters or teeth in pairs or in groups, which ensures the superposition of stress fields from neighboring elements, can present a way to improve the efficiency of drilling by complex tools in brittle rocks. This is confirmed by the results of Moscalev et al [15], who have shown that the placement of cutters on a multicutter drill bit in groups ensures lower energy consumption and higher efficiency of rock fragmentation than the uniform arrangement of cutters.

5.3. Dynamical Systems with Feedbacks and their Application to Modeling of Tool Wear

From general reasonings one may suppose that the simplest way to construct a most efficient destructing tool is to make it extremely sharp. However, any tool will change its shape and lose its sharpness in short time due to the wear processes and the duration of this period is the shorter the sharper the tool is. Actually, wear is a main limitation in design of breaking tool, and any concept of tool improvement which is based just on the rock fracture studies and does not take into account the tool wear is incomplete.

Tool wear is determined by a number of mechanisms, including abrasive processes, fatigue, adhesion wear, fretting, oxidization, etc. However, Artsimovich [13] has shown that the main mechanism of wear of tool from composite material in rock drilling is the fatigue failure of small areas on the contact surface (mainly, filler grains) intensively stimulated by the local heating.

The tool from a particulate composite material (like hard alloys, for instance) which is worn by the fatigue failure of filler grains on the contact surface is shown on the Figure 8. On the basis of the assumptions about the fatigue mechanism of drilling tool wear one may obtain the following formula for the rate of local wear [23]:

$$y=2.6\,(N_g T/Q_f\, n_p^{\,3/2}\, T_0)\exp\left[(E_a/k_b)(1/T_0-2/T) - \sigma^2_{WC}/\sigma^2_c\right], \qquad (17)$$

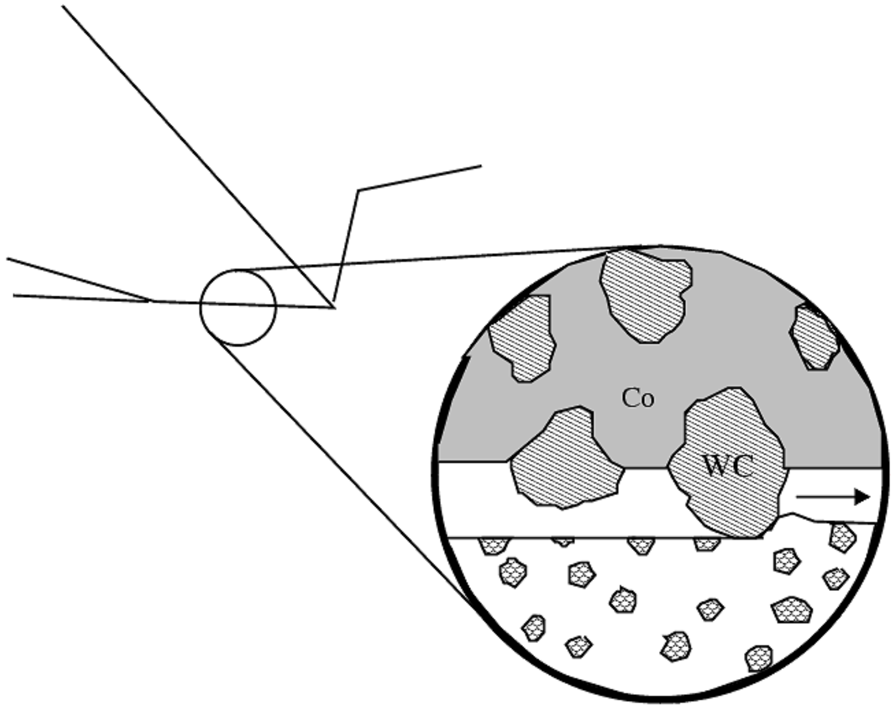


Figure 8. Fatigue wear of composite tool

where y -the average thickness of worn layer over the contact surface of tool divided by the friction (cutting) distance, N_g -the number of hard grains of the rock per unit length, T and T_0 -temperature in the contact area and room temperature, Q_f is the contact area, σ_{WC} is the strength of the carbide particle, n_p is the mean number of the particles per unit of contact area, σ_c -contact stress, E_a is the activation energy, k_b -Boltzmann constant.

After the removal of the particles from the tool surface, the tool shape is changed, the contact stress redistribution occurs and then the redistribution of wear rates over contact surface comes about. That can be considered as a transient in a dynamical system with feedback: the tool shape determines the distribution of wear rates over the contact surface, and in turn this distribution influences the tool shape.

The wearing tool can be represented as the dynamical system in the form

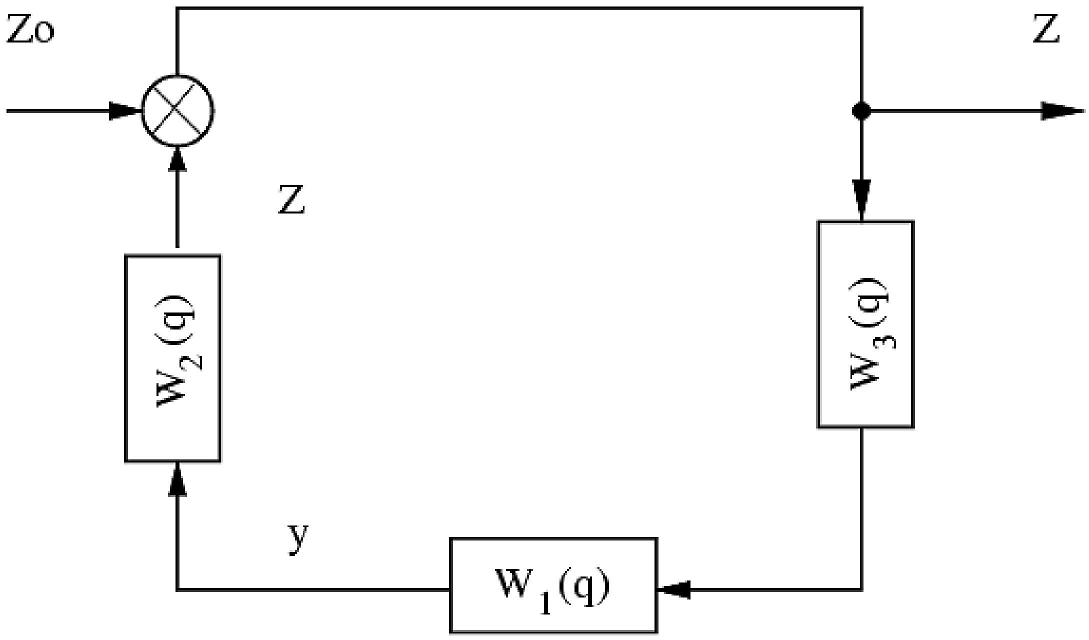


Figure 9. Block diagram of "black box model" of tool wear mechanism [23].

of a block diagram [22, 23]. The system is presented on Figure 9. It is made up of three units which take into account the following factors: influence of the contact stress in point on the intensity of local wear (unit 1); influence of the local wear rate on the tool shape (unit 2); influence of tool shape on the contact stress distribution (unit 3). Each of these units corresponds to some physical mechanism, which is considered as an element of signal processing. Relations between input and output signals of the units are determined by consideration of the physical mechanisms.

Following the method of "black box modeling" [22], we determine the transfer functions of the units. First, consider "input-output relations" of each of the ul1its. Let the tool surface be described by the equation: Z = f(X, Y), where X, Y ,Z are the Cartesian coordinates. The axis Z is so directed that the function f(X, Y) is convex up. This function describes the tool shape. If one accepts that the stress-strain relation is linear or piecewise-linear, one can write: $d\sigma_c = \sigma_m / Z_m \, dZ$, where Z_m is the mean Z-coordinate over the contact surface, σ_m is the mean contact stress over the contact surface, the ratio σ_m / Z_m is proportional to the Young modulus of thc rock. The relation between contact stress and wear rate is given by equation (17). The change in the Z-coordinate of the surface point due to the wear can be found as follows

$$Z(t) = Z_0 - v_c \int y dt$$

where Z_0 is the Z-coordinate of the point in the initial moment of cutting, v_c is the cutting rate. From equations (17)-(18) it is seen that unit 2 can be considered as integrating, and unit 3 as proportional one. The unit 1 can be taken to be proportional one only as a first approximation. In this case, its transfer function $W(q) = L[y(\sigma_c)]/L(\sigma_c)$ becomes a constant value, which is equal to $C_y = dy(\sigma_c)/d\sigma_c$ (i.e. the proportionality coefficient in a piecewise-linear approximation of the function (17)).We have:

$$W_1(q) = L[y(\sigma_c)]/L(\sigma_c) = C_y \qquad (19)$$

where *L()* -Laplace transform, C_y -a coefficient. The value $C_y = dy/d\sigma_c$ can be calculated with the use of the equation (17). Assuming that $\sigma_c \sim \sigma_m$, one obtains:

$$C_y \sim (T\sigma^2_{WC} / T_0\sigma^3_c) \exp(-\sigma^2_{WC} / \sigma^2_c) \qquad (20)$$

The transfer function of the unit 2 (integrating one) presents the Laplace transform of function Z(t), which is given by eq.(18): $W_2(q) = L[Z(t)] = v_c/q$. The transfer function of the proportional unit 3 is equal to the proportionality coefficient between σ_m and Z_m: $W_3(q) = \sigma_m/Z_m$. With the scheme of Figure 9 we can obtain the transfer function of all the system, and, after some rearrangements, the formula describing the variations of tool shape during wear:

$$W(q) = 1/(1+W_1 W_2 W_3) = 1/(1+ C_y \sigma_m v_c q/ Z_m)$$

and

$$Z(t) = Z_0 \exp(-2C_y\sigma_m Z_0 v_c t/ Z_m^2) \qquad (22)$$

where W_1, W_2, $W_3$are the transfer functions of the units 1, 2, 3 correspondingly, the function *Z(t) = f(X, Y,* Z_0*, t)* characterizes the tool shape after a time t, Z(t) is Z-coordinate of a given point of the contact surface at instant t .The formula (22) is obtained from (21) as a reverse Laplace transform of the product of the transfer function (21) by the Laplace transform of the input signal (which is the initial form of the tool; *Z = f(x, Y)).*

From equation (22) it follows that the greater the initial height of a point of the tool surface, the greater the wear rate in this point:

On the basis of the above results one may formulate some ways of the wear control for destructing tools: first of all, the strengthening of the tool material (strength of hard grains, their connectivity, density, etc. [31]) then, the discontinuity of the interaction between tool and rock; and self-sharpening of the tool. The last can be achieved by using such a tool shape that stronger components of tool material are located in extending (and therefore more loaded) parts of the working surface of tool, whereas the weaker elements are located at less loaded parts.

5.4 Self-Sharpening of Tool

Consider now the interaction between the rock destruction, caused by a tool, and the tool wear , caused by the rock.

From the dynamical system model of tool wear, given in the Section 5.3, one can obtain the following formula for the variation of contact stress in some point of contact surface (this formula is derived by simple transformation of the system given on Figure 9; the initial contact stress becomes the input and the current contact stress the output signal of the system):

$$\sigma_c = \sigma_{c0} \exp(-C_w\, t\, \sigma_{c0}),$$

where σ_{c0} is a contact stress in a point of non-worn tool, C_w -a coefficient of wear intensity which depends on the local physical properties of the tool working surface, drilling conditions, etc, t -time. As seen from eq.(24) the tool wear leads to the decrease of the contact stress entropy: the local contact stresses vary (decrease) due to the tool wear, and the rate of contact stress decrease is the more, the greater is the contact stress it means that after a lapse of time the contact stress will be the same over all contact surface. Since the intensity of rock fragmentation is the more the greater the contact stress entropy (as shown above), we obtain a fully evident conclusion, that the wear of tool should lead to the decrease of its destructing ability.

One should note however that the constant wear-resistance over the contact surface (C_w = *const)* was assumed when this conclusion was made. Yet, if the value C_w and contact stress are related by the formula $C_w \sim 1/\sigma_c$ (as it was suggested in the Section 5.2), the contact stress distribution and its entropy can remain constant in drilling (as can be seen from eq.(24) as well). So, the destruction ability of tool remains constant provided the wear-resistances in different points of tool working surface differ and the local wear-resistance is the greater the less is the contact stress in the point.

Let us test this assumption. Consider the case when a tool is made from composite material, and wear-resistance of contact points depends on the local properties of components of tool material (which are randomly distributed). Suppose that the tool consists on 4 components, the values $C_w t$ of which differ and equal to 0.05,0.1,0.15 and 0.2, correspondingly. The volume content of each component was varied from 0.1 to 0.7 and the probability $p((C_w\ t)_j)$ that the wear-resistance in a point is equal to one of the values $((C_w t)j)$ is equal to the volume content of each j-th component. After calculations we obtained a plot of the statistical entropy of contact stress distribution versus the statistical entropy of local wear- resistances of tool surface (i.e. the value $Hw = \sum p(C_w t)j \log p(C_w t)j$, where j =1,2,3,4- the number of constituents in the material, $p(C_w t)j$ -the probability that the local wear-resistance is equal to $(C_w t)j)$ (see Figure 10).

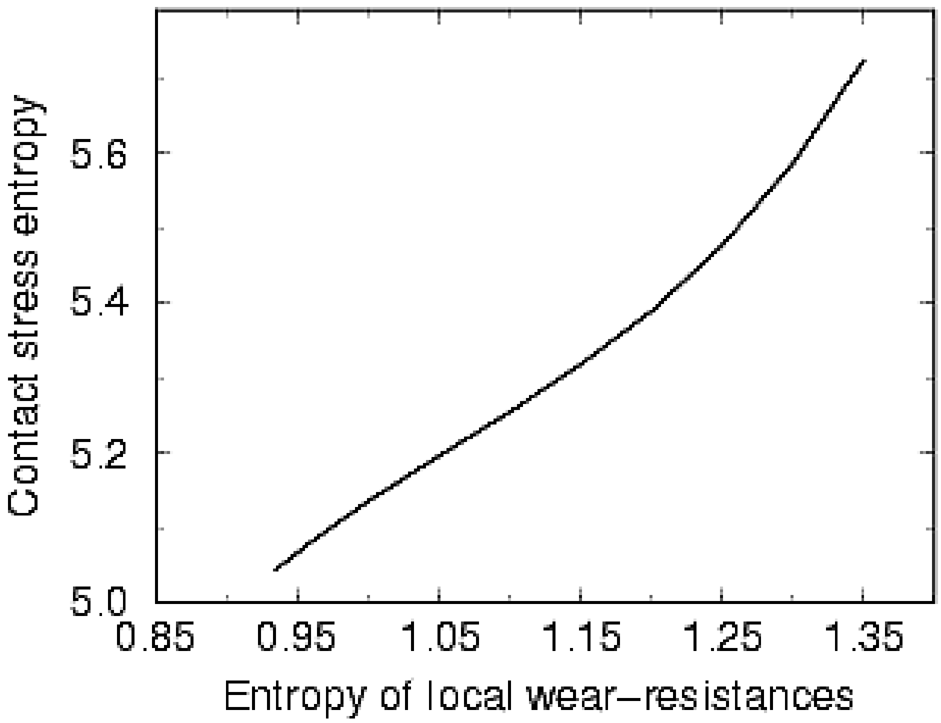


Figure 10: Contact stress entropy plotted versus the statistical entropy of local wear-resistances [26].

One can see that the greater the entropy of local wear-resistances of tool surface, the more the contact stress entropy (i.e. the sharpness of tool). As shown above, the greater is the contact stress entropy, the greater the intensity of rock fragmentation by the tool. Thus, one can conclude that the heterogeneity of local wear-resistances over the contact surface leads to self-sharpening of drilling tool and may ensure also high destructing ability of the tool even after a lapse of time of drilling. Practically, it means that in order to ensure the high efficiency of rock fragmentation by a drilling tool during long time, the tool working surface should consist on a number of components with different wear-resistances.

6. Principle of Tool Design

The conclusion that an increase in the informational entropy of contact stress distribution leads to the improvement of destructing ability of tool was obtained in Section 5.1 on the basis of the consideration of the elementary forms of indenters and then confirmed numerically. In Section 5.4 it was shown that the self-sharpening of the tool can be achieved if the wear-resistances of different points on the tool working surface are very heterogeneous. In Section 5.2 it was shown that the arrangement of teeth on a drilling bit in pairs or in groups ensures higher efficiency of rock fragmentation.

If one generalizes these results, one may assume that introducing some heterogeneity into a construction of tool (for instance, uneven distribution of contact stresses over the contact area, heterogeneity of local wear-resistances or distances between teeth) leads to the improvement of tool efficiency.

In order to verify this assumption, we use the Table 2 from [26], which presents a result of an analysis of about 250 patents in the area of tool improvement. The analysis of the patents included the search of common approaches and ideas in different patents; as a result of the analysis, the patents were divided into seven groups in correspondence with close ideas used in the technical solutions.

Correlating the ideas from all of the groups, one can see that all considered methods of tool improvement consist in introducing some heterogeneity (in other words, information) in tool constructions.

The groups 1 and 2 from this Table correspond fully to the main conclusion obtained in Section 5.1: the increase in the destructing ability of tool is achieved by increasing the informational entropy of contact stress distribution. If one generalizes the results of the patent analysis and the conclusions made in Sections 5.1-5.4, one can formulate the following general principle of drilling tool design: the efficiency of the drilling tool increases with increasing the heterogeneity of distribution of local parameters of tool.

In terms of the information theory it can be formulated as follows: the efficiency of a complex tool can be improved if the informational entropies of distributions of local parameters of the tool are increased.

Table 2. Technical solutions in the area of tool design [26]

Main Ideas	Some Examples
1. Unevenness of the tool work surface: making a cutting face convex or concave, or prismatic or cylindrical lugs on cutting face, stepped working surface; cavities, bevels, slopes	No 1044765A, 1023062A, No. 1323706A1, 623958 (USSR); No.1284539 (UR) No 57- 35357 (Japan)
2. Asymmetry of tool working surface about direction of tool movement: a cutting face or its parts are inclined ta cutting vector Table 2:	723123 and 1046465A (USSR)
3. Using teeth of dissimilar shapes or orientations on the same bit: combination of radial and tangential cutters, different cutting and wedge angles on teeth from one bit, using different materials of inserts, the strength of inserts changes from axis of auger to periphery	No.395559, 153680A1, 1366627 Al (USSR)
4. Irregular arrangement of teeth on a bit: teeth or cutters are placed in pairs or in groups; various distance between teeth	No.3726350, 3158216 (USA) No. 1472623A1 (USSR)
5. Elements with different mechanisms of loading on a bit are combined: combination of mobile and fixed elements, or rotating and progressively moving elements, or cutting and impact elements	No.52-48082 (Japan) 697711 (USSR)
6. Different wear-resistances or different points or tool working surface: layers with different strengths in a cutter; diamond coatings and graded materials; cavities or required shape in tool	No.714003, 281349. 145496, 693000, 609884 (USSR)
7. Self-sharpening and self-organization of tool	No.4230193 (USA) No. 717327, 719192 (USSR)

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