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M. T. Biletsky[1], B. T. Ratov[1], A. A. Kozhevnykov[2], A. R. Baiboz[3], D. N. Delikesheva[3]

[1]Caspian University, Almaty, Kazakhstan,
[2]State Higher Education Institution "National Mining University", Dnipro, Ukraine,
[3]NAO Kazakh National Research Technical University named after K. I. Satpayev, Almaty, Kazakhstan.
E-mail: biletsky@yandex.ru, ratov69@mail.ru, aak2@ua.fm, baiboztegi@gmail.com, delikesheva@mail.ru

UPDATING THE THEORETIC MODEL OF ROCK DESTRUCTION IN THE COURSE OF DRILLING

Abstract. Improvement of the rock destruction model during drilling with the aim of using it in the development of new types of crowns and bits and estimating the expected performance of their work.

The known model of vertical and horizontal displacements of the indenter over the rock surface is detailed with reference to the drilling of holes by crowns and bits of a cutting type in rocks with different physical and mechanical properties. Computer programs were developed that allow to calculate the work of the incisors using the methods of computational mathematics.

The choice of the type of the rock destruction model is substantiated with respect to drilling with a rock cutting tool and, in particular, with PDC bits. In accordance with the chosen model, an algorithm has been developed that makes it possible to perform specific calculations of the design parameters of crowns and bits and to estimate the expected performance of their work. An algorithm is developed for calculating the distribution of loads along the cutting edge of the bladed bits as a function of the distance from the axis of rotation. The design of a blade bit is proposed, which excludes its "hovering" in areas close to the axis of rotation, and a quantitative estimate of the expected effect is made.

The algorithm for achieving the above goal, developed by us, is original and, as far as we know, nowhere else published.

The algorithm developed by us can be used as the basis for creating a new rock-destroying tool. It allows to estimate in advance the possible effect of its application. With the help of this algorithm, the phenomenon of "hovering" of the bit in its central section was quantitatively evaluated and a method for eliminating this problem was proposed, which would provide a significant increase in the rates of the wells deepening.

Keywords: drilling wells, rock destruction, algorithm, computer model, rock destruction tool, designing, quantitative evaluation.

Introduction. Updating the existing technology must be based on the theory of processes considered. Availability of such a theory makes it possible:

- to find a most effective direction of the work;
- to anticipate a possible effect of the outputs;
- to evaluate the actual results and the extent of their approach to optimum;
- to find the cause of difference between the actual result and the expected one,;
- to find the ways of improvement.

Currently, two theoretical models [1] of rock destruction at drilling are prevalent.

Methods: ***The first model*** [2-4], considers drilling ascombination of two consecutive processes:

– the process of imbedding the assumedlyimmobile cutter into the well face;

– the process of spreading by rotation the destruction zone from the indentation cup all over the well face, removing the rock layer the indention cup thick.

The imbedding process is looked upon as the moreimportant and energy expending one, and as such is explored well enough [2-4]. The spreading over the well face process is considered an auxiliary one, 70–80 % of its energy expenditure being ascribed to the cutter's friction against the well face.

Our opinion is, that such a model can prove to be adequate on occasions, when in use is a drilling mode, consisting prevalently of imbeddings. It is the case with drilling with cone rock bits. Their cones in the course of rolling make their teeth alternatively plunge intoclosely situated sectors of the well face.

The similar situation takes place at rotary percussion drilling, where rock destruction is performed at the expense of impacts of cutters, while rotation only serves to uniformly distribute the impacts against a well face

It should also be pointed out, that the theory of the cutter rock imbedding is developed mostlywith relation to friableand friablyductilecrystalline rocks. With relation to ductile and porous formations (clays, loams, sandy clays etc.) the theory is not so well elaborated. However,now it is generally recognized [3, 4], that with anincrease of the well's depth under influence of the geostatic, and the hydrostatic pressures, the friably ductile rocks gradually turn into ductile ones.

What with the cutting type drilling with taking off a layer of rock by the moving along the well face cutter, – here the adequacy of the above-described model brings about serious doubts. It does not explain the reason, why once imbedded cutter during its displacement along the well face does not keep its initial depth, but continues penetrating. And it is just the penetrating in the course of "horizontal" movement, that represents the principal feature of the cutting type drilling.

An experienced driller, when resuming the interrupted drilling process, would never before starting rotation prop the bit against the well face to obtain maximum penetration. It may bring about breakage of the cutters to say the least. In fact, the rotation is started with no bit load (that is with no penetration) at all, and only after rotation starts the load and the penetration are gradually brought to the planned values

Currently, an active process of the cone rock bitsforcing out by the PDC bits is underway. The drilling by PDC bits belongs to cutting type drilling, for which the above model is inadequate.

The second model, developed by V. Vladislavlev [5], represents drilling as a combination of longitudinal and transverse displacements of the cutter. The idea being, that an imbedded cutter in the course of its movement along the well face is continuously leaving the compressed with the longitudinal load sector (impossible for further penetration). While displacing the opposing wall of the groove, the cutter comes over to the neighboring sector, not yet deformed and compressed. For that reason on the new sector under the same load, the cutter is gaining more penetration. The process is illustrated inthe figure 1 [5].

An indenter 1 with a flat end, which width is δ. is penetrating a ductile rock, which preserves the shape and dimensions of the imbedded part of the indenter. An assumption is made, that the indenter, while displaced by the value of δ, comes over from one position to another without time expenditures.

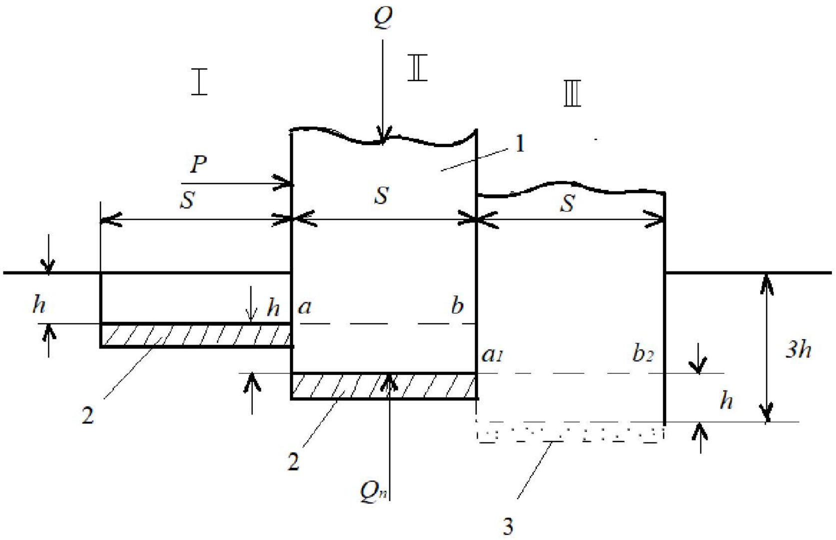

Figure 1 – A model of the indenter's penetrating, while moving along the well face

From position I, where it was in anhdeep groove, the indenter along the $a - b$ line comes to position II. Thereafter it plunges by the value h, which corresponds to the longitudinal load Q and resistance Q_p of the compressed layer 2 of therock.

Under the force P the indenter, displacing the 2h thick opposing layer is traveling along $a_1 - b_1$ line by the distance δ to position III, where the rock has not yet been subjected to deformation and the resisting force $Q_R = 0$. The balance $Q = Q_R$ is broken again, and under the load Q the indenter again plunges by the same value h, bringing about restoration of the balance $Q = Q_R$, but this time with the groove 3 h deep.

The process of vertical h and horizontal δ displacements goes on as long as the forces P and Q are not removed

When δis tending to zero, h is also tending to zero ($\delta \to 0$, $h \to 0$). So instead of a stepwise trajectory a direct line will be formed, which inclination to the well face plain is characterized with tangent

$$tg\alpha = \frac{h}{\delta}. \tag{1}$$

Because of rotation, the cutter's inclined direct line trajectory turns into a helix. In figure 2 [5] an unfolding of its two first revolutions is shown. At the part of the unfolding, related to the first revolution, the penetration y of the cutter and the thickness of the layer, removed by it, increases with the course along x. After having made the first revolution with $x = x_{ob}$ and gainingy = H penetration,the cutter returns to its initial position (although at a greater depth). Henceforward in the course of the second and all the further revolutions, the thickness of the layer preservesthe constant value H [6-10].

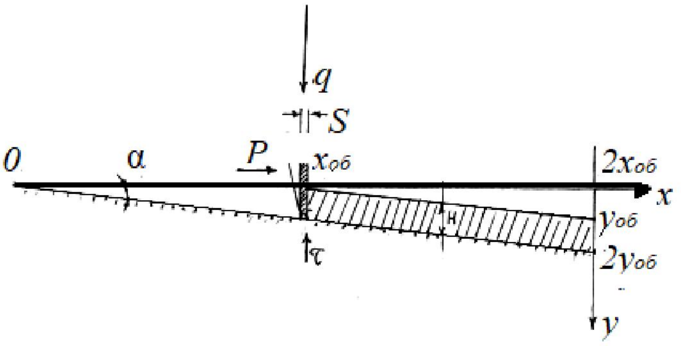

Figure 2 – The single isolated cutter's performance on the well face (unfolding – with two revolutions have been made): x – abscissa; y – ordinate; x_{ob} – the length of revolution; $y_{ob} = H$ – penetration for revolution; α – the cutting line inclination to the well face; δ – the cutter's width; q – longitudinal load; P – transverse load; τ – friction force

In figure 3 an unfolding of one revolution of a core bit, supplied with 3 cutters is shown. The view 3a shows the cutters in their initial position; various hatching being provided to the layers of rock, removed by various cutters.

The cutter #1, after having performed 1/3 of revolution (from its initial position) runs into an H tall wall, left by the cutter #2. The same will occur with the cutter #2: it will run into an H tall wall, left by the cutter #3 and that one in turn – in an H tall wall, left by the cutter #1. Thus after 1/3 of revolution, all 3 cutters (and the bit) will penetrate by the depth $H = y_{ob}/3$ (see the view 3b, position I).

At the view 3b in positions II and III shown are the cutters, completing 2/3 and a full revolution. After completing the full revolution all the cutters and the whole of the drill stem will penetrate by $y_{ob} = 3H$.

The conclusions from the above are as follows:

– The number of cutters does no influence the penetration per revolution. A single cutter with the same load q = Q/m (Q – a bit load, m –number of cutters), after having made one revolution, will be penetrated by some depth $y_{об}$, as m cutters.

– The reason why the bit is supplied not with a single one, but with several cutters, consist in the fact (as it is shown in figure 3), that the removed by one revolution layer of rock $y_{OБ}$isdividedbythe number of cutters m.

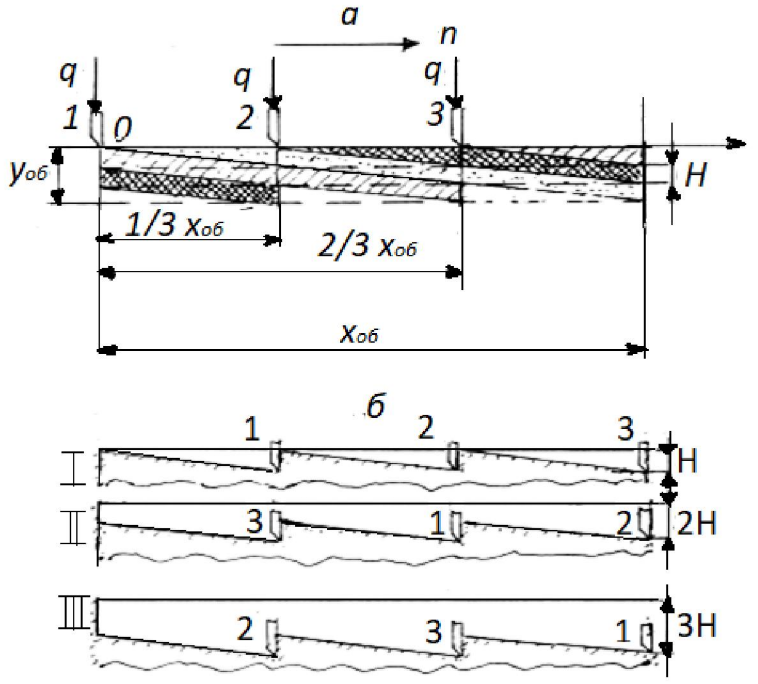

Figure 3 – Performance of the core bit, supplied with 3 cutters:
a – the layers of the rock, removed with the cutters; b – positions I, II, III of the cutters after layer depth H tuple penetration;
$x_{об}$ – the length of one revolution; $y_{об}$ – the penetration per revolution; n – rotation direction; q – the cutter load

$$H = y_{ob}/m, \tag{2}$$

Reducing H, in turn, reduces the acting on each cutter force P (see figure 2), necessary for horizontal displacement of the rock – thus reducing the cutters breakage risk.

Besides, existence of several cutters reduces vibrations and makes the position of the bit more stable, decreasing the well deviation risk.

The figures 2 and 3 demonstrate, that the drilling process is more efficient, the greater is inclination angle α of the cutting line. According to V. Vladislavlev [5]

$$tg\alpha = \frac{2q}{a\delta}, \tag{3}$$

where δ is dimension of the indenter along the line of its movement (figure 1); q – the intensity of the vertical load, a – rigidity of the indenter – rock couple.

According to [4] we assumed:

$$a = \frac{E}{1-\mu^2}, \tag{4}$$

where E –is the rock's Young's modulus; μ – its Poisson's constant.

The indenter's rigidity being by an order higher, than that of the rock, the indenter can be considered as an absolutely hard body, and hence the ***a*** can be taken as the rigidity of the rock.

As applied to conditions of a well in the course of drilling:

$$q = \frac{Q}{mб}, \tag{5}$$

where Q – is a bit load; m – the number of cutters, overspanning the width of the well face; б – the width of the well face along its radius.

Basing on the formulas (3) – (5), we obtain a formulation of penetration per one revolution of the bit:

$$y_{Ob} = x_{ob}\frac{2Q(1-\mu^2)}{бmE\delta\lambda}, \tag{6}$$

where $x_{об}$ – is the length of a circumference, related to the cutter's middle, λ – a coefficient, allowing for the constrained conditions of the cutters' work on the well face. According to [5], λ = 1.38 .

The drilling penetration speed:

$$V_M = x_{OБ}\frac{2Q(1-\mu^2)}{бmE\delta\lambda}n, \tag{7}$$

n – being rotation frequency.

Table 1 presents an example of the above theory application for conditions of drilling limestone. For that rock E = 50000 MPa, μ = 0.3 [3]. The drilling is performed with a core bit, diameter of which is equal to 112 mm, supplied with m= 4 cutters 9 mm wide (b = 9 mm). The mean circumference of the bit $x_{об}$ = 326 mm, the bit load Q = 10000 H, rotation frequency n = 100 r/min.

Adependency of penetration per revolution $y_{об}$ and penetration speed $V_{м}$ ondimension δ of the cutter's flat end is shown in the table. That dimension may belongto a self-sharpening cutter in the shape of a thin hard alloy plate, supported with an iron element.

Table 1 – The dependency of penetration per revolution and penetration speedon the cutter's blunting area width δ

Dimension δ, mm	2	1.5	1	0.8	0.6	0.4	0.2
Penetration y_{ob}, mm	1.2	1.58	2.38	2.98	3.96	5.96	11.9
Penetration rate $V_{м}$, m/h	7.2	9.5	14.3	17.9	23.8	35.8	71.4

While moving along the well face the cutter, meeting the rock's resistance, acts with its front facet on the wall of the groove with the force [5]:

$$P = \sigma_{СД} * F, \tag{8}$$

where $\sigma_{СД}$ – is breaking strength of the rock at shearing, and F – the area of the groove's front wall.

$$F = б * H, \tag{9}$$

where H – is a height of the wall (fig. 2 and 3), found by formula (2).

On the front facet of the cutter, the force P brings about anupward friction force, counteracting the vertical load Q

$$\tau = Pf, \tag{10}$$

where f –is a friction coefficient.

Thus the actual value of the bit load:

$$Q_i = Q - \tau \tag{11}$$

To find by that formula $Q_{И}$ and then, using it, to determine by formula (6) the actual value of penetration per revolution $y_{ОБ}$, one has already to know that yet unknown $y_{ОБ}$ to put it in formulae (2) , (8) – (11), which represents a "vicious circle" and looks inadmissible.

However, the problem can be solved, by numerical techniques [6], enabling to find an approximate solution at any required precision.

Table 2 presents an example of results, obtained by application of that technique for solving the above problem.

The conditions of work of the core bit are the same as for the table 1. In addition, there were introduced: the friction coefficient hard alloy – rockf = 0.55 [4]. The lime stone breaking strength at shearing, measured at laboratory conditions is equalto 12 MPa [3], but allowingfor geostatic and hydrostatic pressure on the wellface a value of $\sigma_{СД}$ = 48 MPa was adopted. The calculation was performed for the cutter's bluntingarea width δ = 0.2 mm. The maximum admissible error of bit load is 50 H.

Table 2 – The dependency with consideration for friction forces of the actual bit load, penetration per revolution and penetration speed on the cutter's blunting area width

Dimension δ, mm	2	1.5	1	0.8	0.6	0.4	0.2
Bit load Q, H	9724	9636	9465	9337	9135	8754	7786
Penetration y_{ob}, mm	1.16	1.54	2.26	2.79	3.64	5.23	9.30
Force P, H	502	663	977	1204	1571	2258	4017
Friction forceτ, H	276	365	537	662	864	1242	2209
Penetration rate $V_м$, m/h	6.97	9.21	13.6	16.7	21.02	31.4	55.8

Comparison of the tables 1 and 2 enables to make a conclusion, that the sharper is the cutter, the greater is reduction in its penetration rate, caused by thereduction of the bit load brought about by friction forces. So if for the cutters with blunting 2 mm the penetration rate drops for that reason by only 3.2 % (from 7.2 to 6.97 m/h) for the cutters with δ = 0.2 mm that drop attains as much as 22 % (from 71.4 to 55.8 m/h).

It is possible to ascertain, that due to proportional dependency (see formula (7)), the drop in penetration is caused by the same relative drop of the bit load.

While drilling with full-hole bits a problem arises of its penetration slowing down on the central point of the well face. Using the adopted fell face destruction model (see figure 1) it is possible to see the problem in more detail, by making corresponding computations.

Let us consider the performance of a drag-type bit, cutting edge of which has K cutters on each blade. The width of each cutter equals:

$$b = \frac{D}{2K}, \tag{12}$$

where D – is the bit diameter.

If each cutter is supplied with its own number i (counting from the bit's longitudunal axis), then the distance from the middle of each cutter to the axis equals:

$$R_i = 0.5\text{б} + \text{б}(i-1) \tag{13}$$

If all the cutters had worked independently, then, due to the fact that each cutter has its own $X_{OБi}$. according to formulae (6) and (7), each of them wouldhaveits own penetration per revolution $y_{obi}$and penetration rate V_{Mi}. But it is impossible, all the cutters representingawhole (belong to the same bit).

Therefore all the cutters are plunging at the same speed V_M and removing the same thick layer of the rock:

$$H = \frac{V_M}{nm}, \tag{14}$$

where n – is rotation frequency, and m – the number of blades of the drag bit.

So, instead of distribution of penetration rates, because of different distances from rotationcentre, a distribution of longitudinal loads along the bit's cutting edge is taking place. It can be seen at the transformed formula (7)

$$Q = \frac{\text{б}E\delta\lambda m V_м}{2x_{ob}(1-\mu^2)n} \tag{15}$$

According to formula (15), the lowest load is acting on the remotest from the axis part of the bit edge, and the highest one – on the closest.

It follows from the formulae (8) – (16), that for the cutters, located on the same radius of the drag bit, the load equals

$$Q_i = \frac{mV_M}{n}\left(\frac{E\delta\lambda}{4\pi(i-0.5)(1-\mu^2)} + \frac{Dcdf}{2K}\right) \tag{16}$$

The part of the formula after "plus" is an additional load, caused by friction forces between the cutter and the opposing wall of the groove (formulae (8) –(10)).

The total bit load:

$$Q_Д = \sum_{1}^{K} Q_i, \tag{17}$$

In table 3 an example of the above theory is presented. It relates to drilling limestone. For that rock E = 50 000 MPa;μ = 0.3, $\sigma_{СД}$ = 48 MPa, constrained conditions factor λ = 1.38, friction coefficient f= 0.55. Drilling is performed with a full-hole drag bit, of D= 190 mm diameter, supplied with m = 2 blades; the cutting edge of each blade has K = 10 cutters with blunting area width δ = 0.2 mm;the rotation frequencyn = 100 r/min; the speed $V_М$ =10 m/h.

Table 3 shows, that the cutter load quickly drops with the distance from the rotation axis; the load on the cutters (of both blades) number i = 1 being 8045 H, and that on the bits number 10–1259 H or 6.4 times less. Total load on the first 5 cutters of the blades equals17 723 H, or 72 %of the bit load (24 686 H).

Table 3 – Distribution of the loads along the cutting edges of the bit

i	1	2	3	4	5	6	7	8	9	10
R_I, мм	4.7	14.1	23.5	32.9	42.3	51.7	61.1	70.5	79.9	89.3
Q_i, H	8045	3518	2445	1985	1730	1567	1455	1372	1309	1259
$\sum Q_i$, H					17723					24686

As it was mentioned, the data of table 3 are related to penetration speed 10 m/h. To double that speed (to raise it up to 20 m/h)the bit load has according to formula (16) also be doubled and brought up to 49 kH, which may cause problems like an inadmissible borehole curving

From the table 3it can be seen that if by some procedure the central zone of the cutting edge would be removed, the same penetration rate may be obtained at many tines smaller bit load. Thus, by removing cutters number 1 – 5 we could bring the bit load required for the same penetration rate 10 m/hby 72 % – that is up to 7 kH. That would sharply bring down curving and cutters wear and provide for higher drilling speed per run

The reverse is also true: higher penetration speeds can be achieved at the same or even lower bit loads. In the case under consideration, a speed of 20 m/h can be obtained at the bit load 14 kH.

The solution of the problem can be achieved as follows:

At the site of the removed central cutters between the blades, a gap is to be left for a core to form. After the core gains a certain height, its diameter can be reduced to zero (or to a size guaranteeing its destruction by vibrations and the down flow of the flush liquid.

In order to avoid chaotic core destruction with formation of big fractures, for gradual reduction of the core's diameter, on the inner side of the blades cutters are to be stepwise located with the distance between them narrowing bottom-upwards. The corehaving a relatively small diameter and being surrounded by free surface, the forces applied to those cutters will be by an order of magnitude lower, than those applied to the cutters remaining on the bit cutting edge (in fact they will be negligibly small).

Conclusions.

1. The updating of the borehole drilling technology must be based upon an adequate theory of the drilled rock destruction.

2. Two drilling process theoretical models are currently in use:

– a model, according to which the cutter's imbeddingprocess is preceding the process of spreading the destruction zone all over the well face;

– a model, according to which both processes are proceeding at the same time.

3. The first model is adequate as applied to the rotary percussion drilling and drilling with the cone rock bits.

4. The second model is adequate as applied to the cutting type drilling with the cutter removing from the well face a layer of rock.

5. Drilling by the PDC bits gainingcurrently more and more preference, it is to be recognized, that only the second model provides its adequate theoretic description.

6. The second model obtained in the article a detailed consideration with respect to its application under specific geologic and technological conditions.
7. The problem of assessing the role of friction forces was solved by methods of computer mathematics with an error not exceeding an assigned value.
8. Basing on the second rock destruction model an algorithm of calculating distribution of vertical loads along the drag bit's cutting edge was proposed.
9. Basing on the algorithm of the clause 8, a drag bit design, solving the problem of penetration slow down on central part of the cutting edge, was proposed, promising a sharp increase in drilling speeds.
10. The elaborated algorithmsare illustrated by examples, including concrete numerical assessments.

REFERENCES

[1] Martin F. Computer modeling. M.: Sovetskoje radio, 1972. 260 p.

[2] Ostroushko I. The bottom hole processes and tools. M.: Gornoje delo, 1962. 271 p.

[3] Sulakshin S. The technology of drilling exploration bore-holes. M.: Nedra, 1973. 316 p.

[4] Technology of drilling oil and gas wells, edited by A. Spivak. M.: Nedra, 2004. 509 p.

[5] Vladislavlev V. Rock destruction at drilling wells. M.: Gostoptechizdat, 1958. 316 p.

[6] Kalitkin H. The numerical techniques. M.: Hauka, 1978. 511 p.

[7] Biletsky M.T., Ratov B.T., Baiboz A.R. Theoretical justification of an automatic device for drilling mud funnel viscosity measurement // News of the national academy of sciences of the Republic of Kazakhstan, Series of geology and technical sciences. ISSN 2224-5278. 2017. Vol. 4, N 424. P. 123-132.

[8] Ratov B.T., Korgasbekov D.R., Fedorov B.V., Sabirov B.F., Baiboz A.R. Effect of fracturing and properties of drilling mud on a core blocking during the coring from deep wells // 17th International multidisciplinary scientific geoconference (Science and Technologies in Geology, Exploration and Mining / ISSUE 14) / ISBN 978-619-7408-00-3. ISSN 1314-2704, DOI: 10.5593 / sgem 2017/14. 2017. Vol. 17, Issue 14. P. 609-616.

[9] Ratov B.T., Fedorov B.V., Sabirov B.F., Korgasbekov D.R. Research parameters of an ejector knot of device for coring from deep well // News of the national academy of sciences of the Republic of Kazakhstan series of geology and technical sciences / ISSN 2224-5278. 2017. Vol. 3, N 423. P. 143-150.

[10] Ratov B.T., Ayazbay M.D., Ibyldaev M.H., Seksenbay M.Zh. Zhanabayev T.A. Optimization of drilling mud composition // Report on the 16th International Multidisciplinary Scientific GeoConference, Albena, Bulgary, 2016, Conference Proceedings. ISBN 978-619-7105-57-5 / ISSN 1314-2704. 30 June – 6 July, 2016. Book 1. Vol. III. P. 875-882.

М. Т. Билецкий[1], Б. Т. Ратов[1], А. А. Кожевников[2], А. Р. Байбоз[3], Д. Н. Деликешева[3]

[1]Каспий университеті (Каспий қоғамдық университеті – ҚҚУ), Алматы, Қазақстан,
[2]Мемлекеттік жоғары оқу орны «Ұлттық тау-кен университеті», Днепр, Украина,
[3]Қ. И. Сәтбаев атындағы Қазақ ұлттық техникалық зерттеу университеті, Алматы, Қазақстан

ҰҢҒЫ БҰРҒЫЛАУ КЕЗІНДЕ ТАУ ЖЫНЫСТАРЫНЫҢ ТАЛҚАНДАЛУНЫҢ ТЕОРИЯЛЫҚ МОДЕЛІН ЖЕТІЛДІРУ

Аннотация. Ұңғы бұрғылау кезінде тау жыныстарының талқандалу моделін жетілдіру оны коронкалар мен қашаулардың жаңа типін дайындау және олардың күтілетін жұмыс көрсеткіштерін бағалау мақсатында іске асырылады.

Тау жынысының бетінде уақытқа тәуелді тік және көлденең бағыта қатар жүретін индентор қозғалысының белгілі моделі физикалық-механикалық қасиеттері әр түрлі тау жыныстарын коронкалар мен кескіш типті қашаулармен ұңғы бұрғылауға қатысты талданып көрсетілген. Математикалық есептеу әдістерін қолданып кескіштердің жұмысын есептеуге мүмкіндік беретін компьютерлік бағдарламалар құрылған.

Кескіш типті тау жынысын талқандаушы құралдар, сонымен қатар PDC қашауларымен бұрғылауда қолданылатын тау жыныстарын талқандау моделінің типін таңдау негізделген. Таңдалған модельге сәйкес коронкалар мен қашаулардың құрылымдық параметрлерін және олардың күтілетін жұмыс көрсеткіштерін нақты есептеуге мүмкіндік беретін алгоритм дайындалды. Айналу осімен арақашықтыққа қатысты қалақшалы қашаулардың кескіш жиегі бойына жүктемені бөлу алгоритмі қарастырылған. Айналу осіне жақын аймақтарда «ілініп қалуды» болдырмайтын қалақшалы қашаулардың конструкциясы ұсынылды және күтілетін эффекттің сандық бағалануы жасалды.

Жоғарыда аталған мақсатқа жету үшін біз жасаған алгоритм ерекше, және біздің пайымдауымызша ешбір жерде жарияланбаған.

Біз дайындаған алгоритм жаңа тау жыныстарын талқандушы құралды жасауға негіз бола алады. Ол өзінің қолданылуындағы мүмкін эффектті алдын ала бағалауға мүмкіндік береді. Бұл алгоритм көмегімен қашаудың «ілініп қалу» көрінісін сандық бағалауға қол жеткізілді және бұл проблеманы жоятын әрі ұңғыларды бұрғылау жылдамдығын арттыруды қамтамасыз ететін тәсіл ұсынылды.

Түйін сөздер: ұңғыларды бұрғылау, тау жыныстардың талқандалуы, алгоритм, компьютерлік модель, тау жынысын талқандаушы құрал, құрастыру, сандық бағалау.

М. Т. Билецкий[1], Б. Т. Ратов[1], А. А. Кожевников[2], А. Р.Байбоз[3], Д. Н. Деликешева[3]

[1]Каспийский Университет (Каспийский общественный университет – КОУ), Алматы, Казахстан,
[2]Государственное высшее учебное заведение «Национальный горный университет», Днепр, Украина,
[3]НАО Казахский национальный исследовательский технический университет им. К. И. Сатпаева,
Алматы, Казахстан

СОВЕРШЕНСТВОВАНИЕ ТЕОРЕТИЧЕСКОЙ МОДЕЛИ РАЗРУШЕНИЯ ГОРНЫХ ПОРОД ПРИ БУРЕНИИ СКВАЖИН

Аннотация. Совершенствование модели разрушения горных пород при бурении с целью ее использования при разработке новых типов коронок и долот и оценки ожидаемых показателей их работы

Известная модель совмещенного во времени вертикального и горизонтального перемещений индентора по поверхности горной породы детализирована применительно к бурению скважин коронками и долотами режущего типа по породам с различными физико-механическими свойствами. Составлены компьютерные программы, позволяющие производить расчеты работы резцов с использованием методов вычислительной математики

Обоснован выбор типа модели разрушения горных пород применительно к бурению породоразрушающим инструментом режущего типа и в частности PDC – долотами. В соответствии с выбранной моделью разработан алгоритм, позволяющий производить конкретные расчеты конструктивных параметров коронок и долот и оценивать ожидаемые показатели их работы. Составлен алгоритм расчета распределения нагрузок вдоль режущей кромки лопастных долот в зависимости от расстояния от оси вращения. Предложена конструкция лопастного долота, исключающая его “зависание” на участках, близких к оси вращения, и сделана количественная оценка ожидаемого эффекта.

Разработанный нами алгоритм достижения указанной выше цели, является оригинальным и, насколько нам известно, нигде больше не опубликован.

Разработанный нами алгоритм может быть положен в основу создания нового породоразрушающего инструмента. Он позволяет заранее оценить возможный эффект от его применения. С помощью этого алгоритма было количественно оценено явление “зависания” долота на его центральном участке и предложен способ устранения этой проблемы, который обеспечит значительное повышение скоростей углубки скважин.

Ключевые слова: бурение скважин, разрушение пород, алгоритм, компьютерная модель, породоразрушающий инструмент, конструирование, количественная оценка.