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RESEARCH OF INFLUENCE OF GEOMECHANICAL FACTORS AND DEVELOPMENT OF METHODS OF INCREASING ROCK OUTCROP STABILITY OF DRIFT EXCAVATION

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The questions of influence of geomechanical factors on rock outcrop stability in the working zone of excavations at their carrying out moment are examined. Methods, which allow increasing their steadiness, are numerated.

The loss of rock outcrop stability results to excavation speed reduction on 38...43 %, to interior packing consumption increase, and to decrease of safety drifting operation. Over 45 % of accidents during rock-first workings happen because of stability loss and coal, rock, side excavation failure, at the same time, over 70 % of stability loss account for in-seam working with false and unsteady roof.

According to mine observations, the main part of working area falls in producible excavations happened due to mining, geological and technological factors, influencing the rock outcrop stability in time, during outcrop space choice. As a result of analysis of the forms of stability loss, the following forms were distinguished: parabolic – occur in homogeneous rocks with pressing breaking point σ_{pr} over 30 MPa; fractured – with fracture distance L_{fr} 0,01...0,1 m (more than 8 fractures on 1 m; vaulted circular – forms inrush either equal or more than half-span of excavation, in homogeneous rocks σ_{pr}=25...45 mPa; multischistose, chinked, L_{fr}=0,1...0,2 m; vaulted semicircular – forms inrush either equal or more than half-span of excavation σ_{pr}=30...40 mPa, L_{tr}=0,2...0,3 m; trapeziform – forms due to small cohesion between layers m_{cl}, with thickness 0,2...0,6 m, σ_{pr}=25...40 MPa [1].

However, in most cases, vaulted and close to it forms are marked, moreover, we distinguished that the basic factors, influencing their stability, are fissuring, humidity and cleavage.

Analysis of results of natural observations carried out, proved that coal and coaly mudstones, broken and narrow-cleavage with stratum capacity m_{st} less than 0,1 m, tend to spontaneous exfoliation during 10...15 min, the rest rocks with breaking point more 60 MPa, at the space of outcrop 5 m^2 calve in 35 min or more.

During natural and laboratory research conducted by us and received data analysis, there was determined, that at σ_{pr}=50...60 and stratum capacity more than 0,8 m rock roof exposure keep steady condition during more than 2 hours. At m_{st}=0,1...0,4 m and rocks strength up to 40 MPa, the time of steady state retains within1 hour. If stratum capacity is less then 0,1 m, which is peculiar for false roof, steady state time comes to 10...20 min [2].

According to laboratory analysis results and shaft observations, it was discovered that overwhelming majority of fractures in rocks are filled with argillaceous, carbonate, coaly and other materials. At the crack face in siltstone one can find thin coating of clay and lime matters. In rocks, crack's width reaches 8 mm, though the most abundant is 2 mm width.

Basing on results of received data handling, during the research, it was found out that in strongly fractured, broken ground overlaying coal-seams (mainly false roof), with fracture distance from 0,01...0,2 m, their stability usually do not prevail 20 min; weak, strongly fractured rocks from 0,3...0,5 m, with breaking point in pressing 20...40 MPa are steady during 0,5...1,5 hours; massive, fractured rocks 0,6...1 m, σ_{pr}=40...50 MPa, are steady during 2...3,5 hours.

During humidity analysis, the following results were reached: at humidity increase, sandstones on carbonate cement loss strength property approximately on 5 %; siltstones on siliceous and carbonate-siliceous cement – on 14 %; siltstones with clayey cement – on 20...30 %; mudstones and coaly mudstones – up to 80 %.

Basing on fulfilled researches and received results generalization of mining and geological factors, we suggest rocks classification according to stability (table) [3].

Table. *Rocks classification according to stability*

Group	Stability degree	Rock characteristic	Stra-tum capa-ci-ty, m_{st}, m	Dist-ance between frac-tures, m	Rock solidity at pressing σ_{pr}, MPa	Rocks humi-di-ty, %	Steady rock roof outcrop square (single roof), m^2	Time of rock outcrop without lining stabi-lity preservation, m
I	Very steady	Very steady and steady ho-mogeneous sandstone and siltstone of massive structure	>4	>3	>100	>0,5	>200	>540
II	Steady	Massive and homogeneous and weak-laminated sand-stone and siltstone	2...4	1,8...3	70...100	0,5...1	80...200	300...540
III	Average stea-dy	Massive, laminated, fractu-red grounds (sandstone and siltstone, mudstone, etc.)	1...2	1...1,8	50...70	1...2	40...80	200...300
IV	Weak steady	Massive, laminated, fractured	0,6...1	0,6...1	35...50	2...4	10...15	120...200
V	Unsteady	Weak- laminated, thin-layer, strongly-fractured rocks, ha-ving and other structural, te-xture defects (coal layers, argil-laceous fraction, furrow slip)	0,3...0,6	0,3...0,5	20...35	4...6	3...5	30...100
VI	Very unstea-dy	Very unsteady, strongly-frac-tured, thin-layer, broken rocks, overlaying coal-seam (mainly false roof)	<0,3	<0,2	<20	6...8	<2	5...20

Basing on received data and mine observation results, we concluded the necessity to regard mining factors influence on rock outcrop stability.

It resulted to carrying out of instrumental and field observations in excavations with particular mining conditions. To determine a dependence of mining factors influence on rock outcrop stability and their quantitative evaluation in mining basin of excavations with typical condition.

According to field observations results and mining and technological factors analysis, which, from our point of view, condition the main influence on rock outcrop stability, the following results were find out [4].

With increase of excavation width from 4 to 6 m, excavation displacement increases on 23...28 %.

Increase of location depth from 150 to 600 m in layers with 1,6...2,5 m capacity resulted in increasing of cover displacement in rocks with σ_{pr}=45...80 MPa – in 2...2,4 times.

Basing on preceding, methods of stability guarantee of rocks outcrop in drifts were developed by us [5].

Loss of rocks stability occurs mainly because of G weight is more then cohesion P in the contact with rocks overlying:

$$G \geq P. \quad (1)$$

False rocks cover weight (easily falling rocks) within pitch face can be calculated with formula:

$$G = K_{fr} h \ell b \gamma,$$

where K_{fr} is the coefficient, considering layer's false roof capacity variation on the excavation length; h is the false roof capacity, m; b is the excavation width in drifting, m; ℓ – is rated excavation length (ℓ=1), m; γ is the volume weight of falling rocks, кN/m^3.

Reasoning from practice data, it is recommended to take for K_{fr}=1,15...1,2.

Cohesion P, on the whole surface of failure:

$$P = b \ln c,$$

where c is the cohesion of specific rocks (unit surface), kPa.

Rocks cohesion is greatly influenced by coaly substances coating, decreasing cohesion in 40...54 times, fissuring in 15...18 times, rocks cleavage in 5...8 times.

With condition (1) fulfillment, it is recommended to sink working with cutting-off and excavation of false cover or easily-falling bottom slices. If $G<P$, excavations should be fulfilled with cover saving, i.e. without cutting-off. In these conditions, dangerous deformations and rock cover destruction occurs under the influence of changing density during the process of excavation exploiting. Rock stability loss happens when durability is less than rock density.

If false rocks cover or easily-falling rocks loose their stability due to small cohesion, excavation should be fulfilled with cutting-off and false cover excavation of easily-falling rock. With density value less than rock cohesion, excavation should be fulfilled with cover saving, i.e. without cutting-off.

To evaluate cost-effectiveness of drift and mine slopes with and without cutting-off of false cover and easily-falling bottom slices of coal rock cover, one should know and compare costs of roadheading and excavation exploiting.

Analysis of mines of Belovskiy, Leninskiy and Kemerovskiy region proved the following: excavations, fulfilled with false cover saving (on beds: 4, Breevskiy, Zhurinskiy, 12 etc.). are propped with carriage pitch plant mainly 0,5 m, and mine working with cutting-off and these rock excavation – with carriage pitch plant 0,8 m, i.e. 1,6 times more; in each section of 100...120 length in these excavations there are 1...1,5 of roof fall of 0,4...0,5 m height, during drifting or exploiting; 0,4...0,5 m^3 of timber are necessary on their sealing-off; additionally, one have to repair and excavations, where basic deformations and poor solid formation roof collapse are revealed, with plant on a section of 3–5 new carriages. Moreover, exploiting safety deteriorates severely in excavation exploiting in the area of second working abutment pressure.

Basing on data from drifting operations, working rules and prices on mine workings and interior packings, there were evaluated costs for 1 m drifting with cutting-off and easily-falling rock excavation and without cutting-off. Calculations were fulfilled for mining drift of 5 m width, 2,6 m height, stratum capacity 1,8...2 m; fulfilling by combines without easily-falling rock cutting-off, and for mining drifts of the same width with cover cutting-off capacity from 0,1 m to 0,6 m.

Cost of 1 m excavation without cover cutting-off is averagely 2 times more, then with cutting-off; it is conditioned by costs on excavations propping and repair.

Deformation and rock outcrop condition of average stability is connected with whether they are fulfilled with or without cover cutting-off of coal-bed. These rock strength, with usually more than 40...45 MPa pressure and cutting-off by roadheader or drill and fire system, results in considerable cover asperity (unlike false cover cutting-off) of 60...80 m height. Due to this reason, under the influence of increasing density during mining works fractions opening and development occurs in line rock cover, sometimes with dangerous deformations and failure on lining.

To observe rock cover displacement (upheaval was not mentioned) from the moment of outcrop on each drift section, in the middle in 3 sections, in every 4 m were placed contour reference points on 0,3 m depth.

Conveyor lava drift № 18–21 on Tolmachevskiy bed (mine «Polisaevskaya») fulfilled with 11,8 m^2 net area in drifting. Bed capacity is 2,2 m, hade is 5...6°. The very bed cover is formed by stratiform siltstone, stratum capacity is 0,4...0,5 m, ultimate compression rock strength is 46...50 MPa, soil rock – homogeneous siltstones, σ_{pr}=50...57 MPa.

A part of drifting was made without rock cover cutting-off, another part – with cutting-off on 0,5 m. Steel-polymeric anchor of 2 m length was used to excavation strengthening. Anchors plant density in the roof is 1,25 items/m^2.

Combine GPKS fulfilled excavation, outcropping cover surface for a cycle is 36...40 kN.

From the observation results follows that rock displacement in the section, passed by with cover cutting-off is averagely 1,4 times more, than in the section passed by without cover cutting-off. At the first section, rock displacement occurs during 3 months after outcrop, at the second section – during 2 months.

Cover crippling and stratum connection deformation negatively influence rock deformation and displacement at the first section. Cover irregularities worsen sharply work of lining, especially bolting, due to gapping of grab anchor. Grab absence between these elements of lining with cover results to unequal deformation development and inrush on lining. This occurs most frequently between anchors in cover layered rock with ultimate compression strength 40...45 MPa. Cover irregularities and gapping of grab embarrass anchors procrastinating during plant and result in unloading of some anchors because of inrush.

Thus, realizing mining drift with cutting-off of average strength of rock cover complicate maintaining of coupling of lava and drift, due to different length of these excavations and in some cases lead to rock failure in coupling.

Depending on scheme of mine field horizon preparation and positional relationship of field and stratal drifts, cross-cuts can be realized in two directions – according to traversable rock and coal-beds angle of deposit, and in opposite direction.

To evaluate the influence of crosscutting direction regarding rock occurrence on cover stability special observations were fulfilled in intermediate cross-cuts on 20 m of F.E. Dzerzhinskiy mine and on 40 m of Prokopyevskiy region mines 5–6.

Intermediate cross-cut, east side on 20 m (F.E. Dzerzhinskiy mine), was fulfilled with drill and fire system, section area – 14 m^2 on depth of 280...300 m. Metal, arched support made from special profile SVP-22, support pitch – 0,8 m. Rocks, crossed by cross-cut, are formed by siltstones alternation, sandstones and partly by mudstones of 0,3...5 m capacity with ultimate compression strength from 35 to 70 MPa, rocks deposit angle is 45...48°. Excavation was fulfilled in the direction of rocks failure.

Observations for rock outcrop during crosscutting proved that sandstones and siltstones with σ_{pr}=40...45 MPa on the outcrop surface up to 10...12 m^2 were at the stable condition during 1...1,5 h, with σ_{pr}=60...70 MPa – more than 7...8 h. Only here and there failure of weak fractured siltstones in cover took place after 20...30 min of their outcrop.

Observations were fulfilled in intermediate cross-cut on the same horizon – 20, which were fulfilled towards deposit angle of crossed rock. Conditions of crosscutting are practically the same, as a cross-cut in east side.

During crosscutting occur frequent and dangerous deformations and siltstones cover falls σ_{pr}=40...45 kN through 30...40 min after outcrop. This reason to decreasing of drifting pitch from 2,4 to 1,6 m (in comparison with cross-cut fulfilled along the rock fall). Rock stratified and fell along the rock stratification. Dangerous rock cover deformations revealed more intensively in the places of approach and crossing of coal-beds and coals seam.

From field observation results follows that regarding conditions of crosscutting along deposit angle of crossed rock mass guarantee rock cover displacement increasing in 2...3 times as against their fulfillment against rock cover displacement direction.

In mine 5–6 observation realized in intermediate cross-cuts № 108, № 110, № 111 in horizon of 40 m. A part of them were constructed from the side of dip, a part – from the side of seam rise. Excavation section area is 12,5...13 m², metal support from SVP-22, installation of carriage pitch is 0,8 m. Crossed rock are presented by separate siltstones, sandstones masses and alternation of these rock, deposit angle is 70...72°, predominant rock competency is 50...60 MPa.

Observations showed that in deposit angle $\alpha \geq 70°$, direction of excavation fulfillment relative to rock attitude practically do not influence the rock outcrop stability.

Similar results were received after factual material examining on dangerous deformations and cover fall in cross-cuts, passed in various directions relative to crossed massif masses, on separate mines of Belov and Andzhersk regions.

Quite different character and degree of deformations of stratified, fractured and other macro faulty rock cover in cross-cutts are conditioned by great difference between composing powers F_{pr} and F_{sc} power $P_в$ (Fig. 2), where P_v is the vertical power of rock pressure; F_{pr} is cohesion (pryout force), acting on normal to rock stratification; F_{sp} is the slide power, acting along surfaces of layer stratification.

$$F_{pr}=P_v\cos\alpha;\ F_{sp}=P_v\sin\alpha;\ F_{fr}=F_{pf}f_{fr}, \qquad (2)$$

where α is the angle of rock deposit; $F_н$ is normal power, acting on this surface, kN.

During crosscutting from the side of hanging layer, i.e. according to scheme, Fig. 1; rock displacement would not occur in acting powers ratio

$$F_{dis}<F_{pf}f_{fr}+cS,$$

where F_{dis} is the displacement power, acting along the surface of slipping down (sliding), kH; f_r is the constant of friction.

With low cohesion power, i.e. with c=0 rock slipping down into excavation contour part do not occur if $F_{dis}<F_{pf}f_{fr}$.

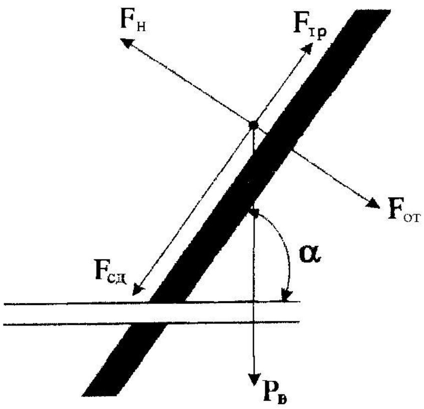

Fig. 1. *Power scheme, acting in the contour part of rock cover cross-cut*

From expression (2) follows that F_{pr} and F_{fr} power ratio depends on angel α rock deposit, with α increase, other things being equal, power F_{pr} decreases, and power F_{fr} increases.

During crosscutting building, break and failure occur most frequently from the side of bottom layer, on weakened deposit contact on normal, i.e. on condition that $F_н\sin\alpha>c$.

During research it was proved that the most rock outcrop stability is saved in deposit angle of 45...50°, in deposit 50...70° rock stability decreases dramatically, and with deposit angle more than 70° drifting direction, other things being equal, do not influence their stability and failure in the cross-cut bottomhole area. When rock deposit angle is 45...50°, and excavation fulfilled from the side of hanging layer, bottomhole area of massif reacts the essential part of density of rock outcrop within cut pitch, this results in the lessen power display of $F_н$ and F_{fr}.

During excavation fulfillment in the opposite directions, they are deprived of this bearing, and elevated drift performs this function. Observations carrying out prove that crosscutting in the direction of hanging layer, depositing mainly under the angle of 50°, provides essential stability increase of cross-cuts rock outcrop. Beds with deposit angle up to 30...40° one can mine up to 90 %.

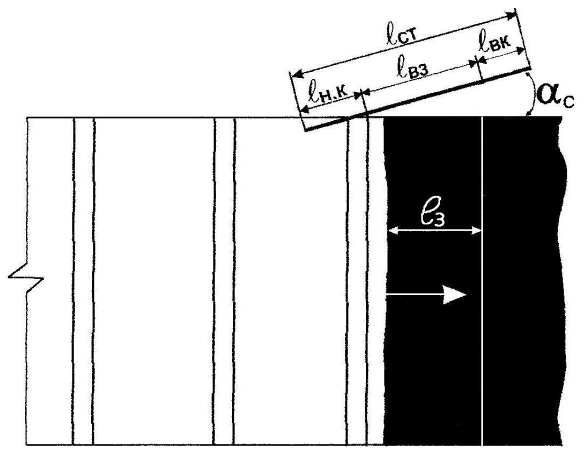

Fig. 2. *Length definition of fastening rods*

Excavation fulfillment in the areas of geological violations connected with various degree of complexity. In the Kuzbas mines a part of them constitute rock cover of coal-bed violation in massif, traversed by cross-cuts. Length of violation areas constitutes from 5...10 to 150 m and more. Minor violations of small length are fixated poorly or are nit fixated by exploration work; usually they reveal during drifting operation or second working within mining field or column.

Cumulative experience and technical-economical analysis prove that during increasing of excavation length of violated rock, effectiveness of rock strengthening by with solution through advance bores increases significantly. Usage of this method in the areas of 15...20 m length, especially inside mining fields and columns, is economically pointless. For these conditions we advice to use advance, safety lining.

As advance lining, on the Kuzbas mines, reinforce bars are used. The system of lining load determination, depending on rock deformation condition and basic lining parameters is poorly based.

Basing on practical experience analysis, bar, plain, reinforcing steel is recommended to use as advance lining for basin mines, and the following building technology. Blast-holes, for rod location, are drilled from the last carriage, directly near pit-face, uniformly on fixed roof contour, sides at the distance, determined from textured and structural conditions of rock to be secured, support load and calculated, bearing capacity, on-the-mitre from such a calculation, for bottomhole carriage to be bearing of outboard end of a rod; and rock massif serve as bearing of inboard end of a rod. Distance between rods of advanced lining on fixed contour of excavation is taken from the conditions of lifting and failure exclusion in the width between them.

Rods length ℓ_{rd} is determined from the condition (Fig. 2):

$$\ell_{cm} = \ell_{н.к}\ell_{вз}\ell_{вк} = \ell_{н\,к}\,\frac{\ell_{з}}{\cos\alpha}\ell_{вк},$$

where $\ell_{п.к}$ is the length of the rod end, obtrusive in excavation, m; $\ell_{вз}$ is the length of rod part, directly over the cut, within cut length, m; $\ell_{вк}$ is the length of the rod end, leaning on the massive, m; ℓ_c is the rod angle of slope to longitudinal of excavation, degree; $\ell_з$ is the cut length, m.

To provide the reliability of advanced cut lining work is recommended not less than 0,4...0,5 m. Bore-holes for cut should be drilled on-the-mitre α_c=16...18°.

Bore-holes of advanced lining works on a bend, for technical calculations, they can be regarded as beams, free-ended on terminal poles – carriages and massifs. Therefore, bending moment in a carriage (in area of $\ell_{вз}$ length) from rock pressure influence (violated rock weight):

$$M_{из} = \frac{P_в \ell^2_{вз}}{8},$$

where, $P_в$ is vertical, uniform power of rock pressure on a carriage.

The moment of carriage strength, with d diameter is determined according to the formula:

$$W_c = \frac{\pi d^2}{32} \approx 0{,}1d^2 \text{ или } W_c = \frac{M_{из}}{\sigma_{из}}.$$

According to definite values of $M_{из}$ and W_c factual tension is determined on a bend $\sigma_{из}$ in a carriage, which should be $\sigma_{из} \le [\sigma_{из}]$, where $[\sigma_{из}]$ is accessible voltage (design resistance on a steel bend, from which a carriage is made).

As a result of data handling of excavation examination and factual materials on excavations, fulfilled in the areas of geological violations, on the Kuzbas mines was determined , that vertical load $P_в$ from the cover side on advanced, guard lining can be determined according to P.M. Tsimbarevich method, considering the dependence from rock violation and weakening degree:

$$P_в = h_в \cdot \gamma, \text{ кПа},$$

where $h_в$ is the height of possible rock cover failure, m; γ is the volume rock weight within limits of probable inrush, kN/m^3;

$$h_в = \frac{a_1}{fK_н},$$

a_1 is the half-span of rock cover failure (probable), m; f is rock density coefficient according to m.M. Protodyakonov; $K_н$ – coefficient, concerning the influence of texture rock weakening violation according to the height of inrush $h_в$, is evaluated quantitatively with basic block values, and distance between surfaces of rock weakening;

$$a_1 = a + h\,\mathrm{tg}\varphi_к,$$

where a is excavation half-span in drifting, m; h is the excavation length in inrush, m; $\varphi_к$ is apparent dip of rock inner friction ($\varphi_к$=arctg$f$), degree.

As a result of data handling of full-scale observations for the Kuzbas mines, results of coefficient $K_н$ are received depending on distance between basic surfaces of rock weakening (blocks size) and its strength: $K_н$=0,2...0,3 – for rock with distance between weakening surfaces ≤0,15 m, strength coefficient f=0,8...2; $K_н$=0,3...0,35 – for rock with distance between weakening surface 0,16...0,4 m, f=2...3.

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