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Ensuring Stability of Undermining Inclined Drainage Holes During Intensive Development of Multiple Gas-Bearing Coal Layers

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At high rates of production face advance, requirements towards reliable operation of undermining drainage holes get raised. The issue of maintaining high intensity of gaseous seams development under naturally increasing gas content, mining depth and capacity of production equipment poses a problem. The greatest threat comes from the loss of hole stability in the bearing pressure affected zone (in front of the face) and in the intensive shift area of overhanging rock corbels (behind the face). Intensification of air leaks due to deformation of borehole channel leads to impoverishment of removed methane-air mixture and an increasing risk to disturb safe aerogas regime in the mining area.

The paper describes a mechanism of how coal-face operations affect the state of underground holes and formation of overhanging rock corbels. A typification of basic kinds of borehole deformations is presented. Authors point out critical disadvantages of the most widely-used technological schemes of gaseous seams development under high load on the production face, which hinder normal operation of a gas drainage system.

As a result of research, a dependency of shot hole number, as well as the distance between shot hole axes and the borehole, on the stress state of the borehole outline has been defined more precisely. Basing on that, a formula to calculate drilling parameters of the discharge hole system has been suggested. Implementation of these measures will allow to increase the efficiency of underground gas drainage and to maintain growing intensity of gaseous coal seam development.

***Key words*:** underground holes; stability; intensification of coal extraction; discharge shot holes; decreasing methane content in the extraction block

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Introduction. By the year 2030 coal industry production will have to reach 480 million tons/year. With this in mind, coal extraction from deep-lying gaseous seams under worsening mining and geological conditions is predicted to increase. Increasing extraction depth will activate negative manifestations of rock pressure, which can become the reason of drainage hole failure even before the production face reaches the wellhead. The growing rate of face advance increases the probability of mined-out space gasification due to heterogeneous sedimentation of main roof rocks and formation of larger rock corbels. This is indirectly confirmed by non-linear dynamics of methane release during the development of deep-lying gaseous seams in the Donetsk basin [16]. Predicting patterns and defining general non-linear character of aerogas processes in undermining holes get significantly harder, if mined-out spaces contain coal pillars.

In order to compensate for the influence of pillars, some authors suggest using mining systems without them [4].

Analysis of foreign and Russian sources [3, 7, 14, 15, 17, 18] demonstrated that stability problems of undermining drainage holes are still relevant. The reasons why boreholes decrease their production capacity are the following: occurrence of small rock inrush from the hole walls [14], justification of protection measures with no regard towards water saturation of fractures in the surrounding rock mass [3]; inadequacy of casing parameters [19] or parameters of borehole spatial orientation [15]; justification of parameters and drilling regime [7] (including zones of high rock pressure [18]).

The study carried out by M.G. Mustafin also justifies the necessity to perform additional measures to drain gas-bearing areas – «gas bags», occurring in front of zones with elevated bearing pressure, but there are no specific solutions to this problem [8]. Besides, existing methods, used to calculate the boundaries of zones with elevated mining pressure, do not always take into account the specifics of technological schemes applied in the intensive development of contiguous gas-bearing seams in the «mine – production face» mode [5]. Therefore, increasing borehole stability in the bearing pressure affected zones is a relevant research task, highly important for the improvement of underground gas drainage technology. Low efficiency of existing solutions can aggravate a problem of environmental safety of undermined rocks masses left in the process of coal mine closure [17].

The purpose of current study is to improve the technology of underground gas drainage from the position of undermining holes stability during development of deep-lying gaseous coal seams.

Theoretical results and discussions. In the existing practice of mining production there is a widely used technology of rock mass gas drainage, which implies drilling holes into the coal-rock massif, casing and sealing a part of the borehole, as well as hooking up their wellheads to the local pipeline with subsequent removal of methane to the surface [11]. Unfortunately, this technology has a set of drawbacks, which do not allow to fully realize its potential (reach standard performance indicators). The main disadvantage is limited capacity of pipe casing to stand against various deformations of the hole shank and its cross section.

An analysis of modern notions and theoretical foundations of rock pressure and shifting of overlying rock demonstrates that modifying the size of overhanging rock corbels in the main and immediate roof is a key factor which defines the dynamics of stress-strain state of the drained masses in the bearing pressure affected zone (Fig.1) [1]. Analysis of Fig. 1 shows that in the area of full shifts angle ψ forms a gas-conducting cavity, which on the one hand intensifies methane release into the hole, and on the other can cause its destruction. The height of gas-conducting fracture h_{sh} is calculated by dividing the thickness of a mined-out seam m on the loosening rate of the caved rocks (taken into account through a loosening coefficient k_l).

The length of gas-conducting fracture is proportionate to the hypotenuse in a right-angle triangle shaped by the cavity, lying wall and h_{sh} height.

The process of roof subsidence into the mined-out space determines development of anthropogenic fracturing, «enveloping surface» in front of the face (Labas theoty), and the general shift plane – «anthropogenic cavity», suitable for accumulation of released gas. The angle of this fracture-plane is proportionate to the full shift angle (for a low dip it can vary between 60 and 70°). The main parameters of an anthropogenic cavity need to be identified considering rock loosening in the zone of active shifts in the top cover, value of horizontal thrust and mined-out seam thickness [13]. Partial location of the borehole in the area of such fracture is responsible for significant risks of the shank stability, as well as for the need to take into account specific requirements towards roof control when justifying protective measures at the hook-up between the face and mined-out area in the proximity of the tunnel. E.g., in order to prevent rock convergence, artificial constructions are used –

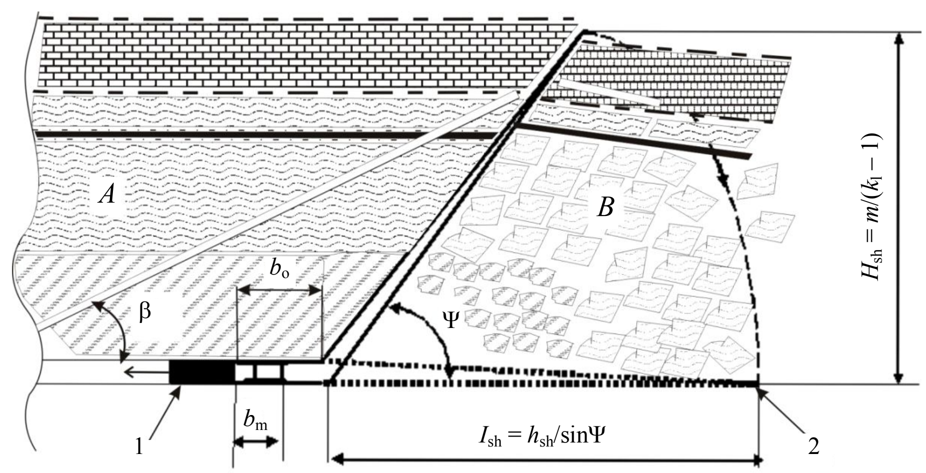

Fig.1. Scheme of borehole distortion during rock corbel formation [13]

1 – coal seam; 2 – mined-out space; A – coal-rock mass in the areas of a roof sag; B – massif area in the zone of active roof shifts; b_o – overhanging corbel width; b_m – unit width of mechanized support

cast strips, erected in the mined-out space. However, parameters, engineering specifics and working patterns stated in the existing regulations do not always prevent destruction of pipe casing.

It has been suggested that the above mentioned specifics have been taken into account in the method used for roof rock gas drainage in the reverse development of extraction columns [10]. Production cycle of this technology includes the following operations: borehole sinking into the coal-rock mass parallel to the face; introduction of steel casing segments into the cavity of the borehole and annular cement sealing; pipeline hook-up and gas removal. Beside that, judging from the sizes of borehole shank projection it is additionally offered to leave an assembly of wooden chock frames in the mined-out space.

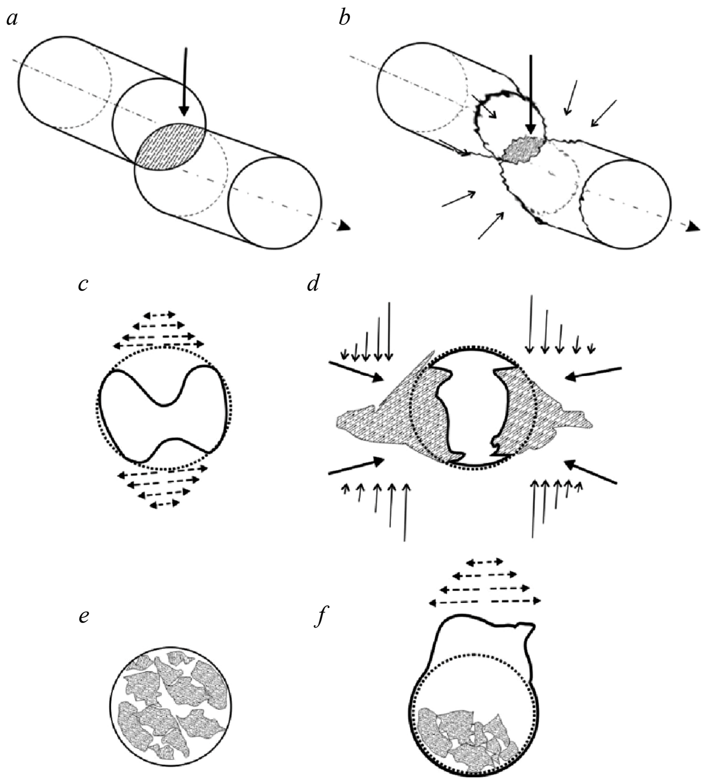

Fig.2. Crippling examples of undermining drainage holes: *a* – distortions caused by borehole shearing; *b* – shearing coupled with all-round compression; *c* – distortion from tension stresses; *d* – rock inrush from compressive stress concentration; *e* – formation of a rock plug; *f* – leaching of the cross core section

Due to the lack of efficient methods to control the state of the surrounding rock mass (low support capacity of the chock), application of this drainage method can be associated with the loss of more than a third of all boreholes. It should be noted that the method is very labour-intensive, especially in the process of chock backing (when using rubble chocks).

As a result of negative rock pressure manifestations (Fig.2) and partial distortion of undermining hole shanks, the losses of vacuum-gauge pressure (non-productive energy losses) increase and air leaks intensify. The loss of one or several boreholes has an impact on the performance of the gas-drainage network of the site, reduces safety of mining operations and cuts down additional income from using coalmine methane as an alternative energy source (combusted in cogeneration units). Special actuality of this problem requires a search for alternative solutions to maintain borehole stability.

This gas drainage technology of the undermined coal-rock mass [12], which allows to increase borehole stability and safety of mining operations, centers around a well-known effect of boost voltages compensation using discharge cavities. A special characteristic of shot hole discharge in this method is the fact that the impact area of rock failure zone, forming around the borehole itself, is taken into consideration. It is implied that in addition to the operations, used for realization of existing technologies [10, 11], around the axis of the hole shank a system of shot holes is evenly distributed, and its parameters allow to transfer the major share of stress from the borehole contour to the area between the shot holes (Fig.3, *a*).

Implementation of this method also implies construction of a cast strip from quick-setting cement, geometrical proportions of which are s elected basing on lithological composition of the overlying rocks.

Formation of the local protective zone is ensured by optimal selection of shot hole discharge parameters (Fig.3, *b*), parallel drilling, even distribution, equal distance between shot holes 6 and their extension in the same direction. To prevent pressure losses in the annular space, the cavity of each shot hole needs to be filled with bonding material on a polyurethane basis.

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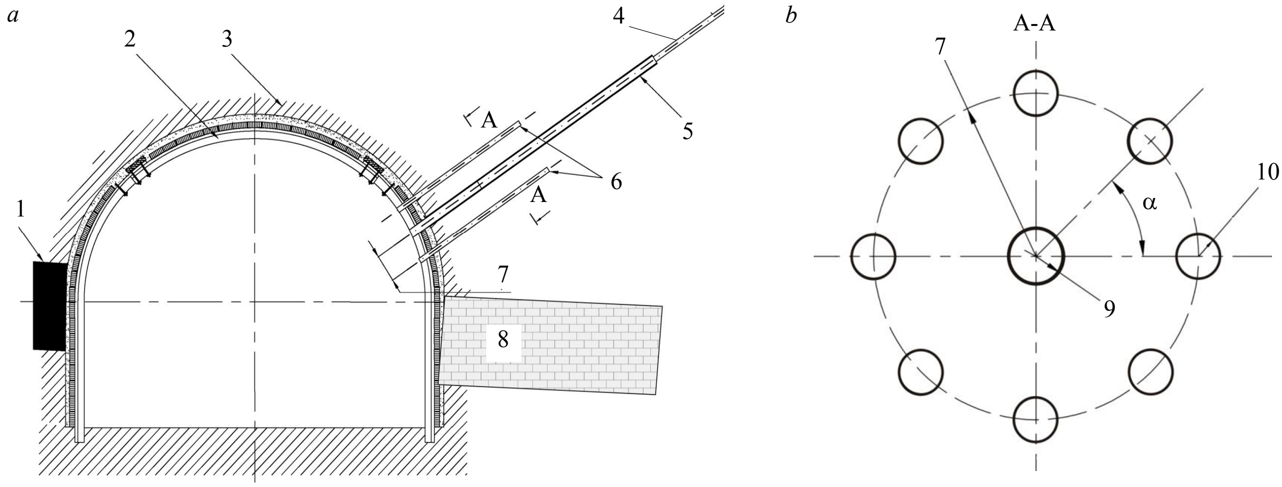

Fig.3. Location plan of holes bored round with the system of discharge shot holes (*a*), and cross-section A-A (*b*):

1 – coal seam; 2 – mining tunnel support; 3 – rock mass; 4 – undermining borehole; 5 – tubing zone; 6 – system of discharge shot holes; 7 – distance from the shot hole axis to the borehole axis (*a*) and distance from the borehole center to the system of shot hole centers $R_{z.}$ (*b*); 8 – cast strip; 9 – diameter of drainage hole; 10 – sizes of discharge shot holes

In order to justify the distance between shot hole centers and the borehole axis, the size of plastic stress range (PSR) is specified for a particular reservoir, basing on the regularities of stress-strain state dynamics in the rock mass around the development mining tunnel. Under the conditions of deep-lying coal seams in the Donetsk basin, its size can be estimated using formula proposed by professor N.N. Kasyan [6]. Subsequent development of experimental-analytical method of PSR size estimation led to its improvement – after the rates of fracture and anthropogenic faults of the surrounding rock mass had been taken into account [2, 13], the formula became the following:

$$R_{\text{PSR}} = r_{\text{t}}\left[1 + 5.7\left(\frac{\gamma H}{k_{\text{r.}} R} - 0.21\right)\right], \qquad (1)$$

where R_{PSR} – PSR size, m; r_{t} – mining tunnel radius, m; γ – bulk density of the rock, MN/m^3; H – distance from the surface to the center of the mining tunnel, m; k_{r} – coefficient, taking into account rates of fracture and anthropogenic faults; R – compression strength of the rocks, MPa.

As a result of studies [9], logarithmic dependence has been established that reflects how the contour stress of underground drainage hole decreases with the distance between the borehole and the system of discharge shot holes, as well as their number. It can be written as follow:

$$k'_{\sigma}(A-E) = 96.05 - 129.00\ln N + 14.58L + 61.28(\ln N)^2 + 5.32L^2 - 19.63L\ln N - ..$$

$$... - 10.09(\ln N)^3 + 0.50L^3 - 3.14L^2\ln N + 6.24L(\ln N)^2, \qquad (2)$$

where $k'_{\sigma}(A-E)$ – contour stresses of the hole as relating to the rate of geostatic stress; N – shot hole number; L – relative width of the bridge, $L = \delta/r_{\text{h}}$; δ – minimal distance between borehole and shot hole contours, mm; $r_{\text{h.}}$ – borehole radius, mm.

The distance between centers of the borehole and the shot holes need to be selected in such a way that the shot holes are located near the PSR boundary, calculated using formulas (1)-(2), and are sufficient to form a local discharge area.

Conclusion. The paper proposes a way to increase development efficiency of deep-lying gaseous coal seams in the Donetsk basin, using the method of shot hole discharge to protect the shanks of gas drainage holes from high rock pressure. It has been identified that stress reduction of borehole contour is characterized by a logarithmic dependency on the increase of bridge size and the number of shot holes. Optimal ratio of the above mentioned parameters can reduce stress concentrations by 30 %.

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