Conditions for a start of sudden coal and gas outbursts in the breakage faces of coal mines

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Abstract. Theoretical studies of the conditions of a start of sudden coal and gas outbursts in the breakage faces of coal seams were carried out and, as a result, expressions for the gradients of the active and passive forces of a sudden outburst were obtained. Their ratio makes it possible to determine the potential for the occurrence of this gas-dynamic phenomenon by the calculation method. On this basis, by means of mathematical transformations, a structural indicator was obtained, which allows the real outburst hazard of the coal mass to be directly monitored at the face, using operational methods for instrumental determination of the parameters characterizing its gas-dynamic state.

1. Introduction

In the recent past, the problem of preventing sudden coal and gas outbursts in the faces of coal mines was relevant in Russia only for the mines of the Eastern Donbass. Approximately 29 outbursts were recorded during the coal mining operations in the region. At the mines of the eastern and northern basins of the country, emissions occurred mainly in the development seam workings and during the coal-seams uncovering, in the faces there were only isolated cases of this phenomenon. To date, sudden outbursts in the stopes in the eastern and northern regions have been registered at six mines – in the Kuznetsk basin in Severnaya and Berezovskaya mines, in the Pechorsky basin – in mines No. 1 and Severnaya, in Primorsky Krai – in the Podgorodnenskaya and Tsentralnaya mines. The problem of forecasting and preventing sudden outbursts in the faces is much more complicated than in the uncovering and development workings for the reason that it is necessary to control the degree of outburst hazard of the bottomhole part of the massif over a long distance and to carry out timely blowout treatment of hazardous areas. For a successful solution of the problem, the tasks of reliable determination of the state of the bottomhole part of the massif in terms of the possibility of a start of sudden outbursts of coal and gas must be solved. The present paper is devoted to the solution of these problems.

2. Force condition for a start of a sudden coal and gas outburst

To determine the conditions for a start of a sudden coal and gas outburst and its flow to the final stage in the working face of a coal mine, it is necessary to consider the possibility of the occurrence of each stage of this phenomenon.
The first stage is preparatory. At this stage, under the influence of rock pressure, significant deformations occur in the zone of influence of the rock massif, a system of intensive development of quasi-parallel to the planes of outcropping of cracks, filled with free gas under high pressure, is formed. As a result of gas pressure on the walls of the cracks, an active force of a sudden outburst arises. When it exceeds the passive force of resistance of coal outburst, the second stage of the phenomenon occurs – the squeezing of the most unloaded and degassed part of the mine from the rock mass to the bottomhole part of the working, accompanied by the destruction of coal and increased desorption of gas from the rock mass. According to [1], active $F_a$ and passive $F_p$ forces are determined by the following dependencies:

$$F_a = \frac{\sqrt{(n + \beta h)^3} \left(P_0^3 - P_1^3\right) \Delta x \times 10^3}{2P_1(n + \beta h)^2 \left(\frac{1}{\beta^2} \ln \left[\frac{1}{\frac{m + \beta h}{m + \beta h}}\right]\right)}; \tag{1}$$

$$F_p = (2m + 2)\Delta x (K_i + K_i \mu g \rho_i) \times 10^3, \tag{2}$$

where $n$ – the fracture porosity of the coal massif behind the zone of working influence; $\beta$ – intensity of stress growth in the area of the limiting stress state; $h$ – the value of the horizontal coal squeezing in the mine, m; $P_0$ – seam gas pressure, MPa; $P_1$ – gas pressure at the bottomhole edge (0.1 MPa); $S$ – the specific cross-sectional area of the mine, equal to $m \times 1$ m, where $m$ is the thickness of the seam; $\Delta x$ – the length of the area of plastic deformation, in which the stresses do not exceed the static rock pressure, m; $x$ – the length of the area of the ultimate stress state of the seam, m; $K_i$ – coefficient of adhesion of coal at the edge of the face, MPa; $\nu_i$ – coefficient depending on the angle of internal friction of coal at the edge of the face $\rho_i$, deg.

Dependences (1) and (2) were refined for the purpose of simplification, taking into account the hysteresis of methane sorption by coal and the power and strength of a potentially outburst-hazardous pack (set of packs) of coal. Curvilinear dependences are replaced by straight ones, while, as it can be seen from figure 1, in case of replacement the condition is satisfied not to underestimate the maximum value of the ratio of active to passive force. Equations (1) and (2) are divided into $\Delta x$, which made it possible, instead of the forces $F_a$ and $F_p$, to obtain gradients of their change when deepening into the massif.

Bearing in mind that the first layer of coal between the point of application of the maximum active force and the bottom does not necessarily collapse, it would be more correct to use the ratio of not the forces themselves, but their gradients to assess the possibility of disrupting the equilibrium state of the massif. As a result, the force condition for disrupting the equilibrium state (force criterion of outburst hazard) of the coal massif will be written in the form

$$R = \frac{\text{grad } F_a}{\text{grad } F_a} \geq 1. \tag{3}$$

![Figure 1. Replacement of $F_a$ and $F_p$ with straight lines tangent to them.](a) (b)
3. Accounting for the hysteresis of methane sorption by coal

Formulas (1) and (2) should take into account the effect of increasing gas pressure in the seam due to the hysteresis of the methane sorption by coal, which occurs due to the oscillatory motion of the roof rocks during its collapse [2]. For this purpose, an amendment has been introduced to the formula for calculating the active force, according to which the value of the $P_k$ used in the formula for the intervals of the bottom movement in the zone, where the roof may collapse, should be determined from the expression:

$$P_r = P_0 + \Delta P,$$

where $P_r$ is the seam gas pressure; $\Delta P$ – the increase in gas pressure due to the phenomenon of hysteresis in the methane sorption by coal due to fluctuations in the main roof during its collapse.

The addition to the gas pressure due to the sorption hysteresis is calculated on the assumption that the frequency of pressure fluctuations in the zone of the greatest seam roof bend ahead of the face and the grouping of the most intensively developed quasi-parallel fracture systems to the face plane, in the so-called “gas bag” [2], coincides with the frequency of roof vibrations after caving.

The gas pressure in the unmined massif at a depth $H$ from the day surface is presented through the stress plot in the massif $\sigma_{Ht}$, where $(x)$ is the distance from the face, and the stress $\sigma_{Hb}$ at the lower boundary of the gas weathering zone $H_w$:

$$P = \gamma_w \left( \sigma_{Ht}(x) - \sigma_{Hb} \right),$$

where $\gamma$ and $\gamma_w$ – are the specific gravity of rock and water, respectively.

The vibration amplitude will be:

$$P = PU,$$

where $U$ is the deformation of the seam edge during dynamic roof vibrations.

The relative addition to the gas pressure due to the sorption hysteresis is presented as:

$$\frac{\delta P}{P} = \frac{0.5U}{1 + B \frac{\gamma_w}{\gamma} (\sigma_{Ht}(l) - \sigma_{Hb})},$$

where $B$ is the parameter of the methane desorption isotherm.

Deformation $U$ was estimated using the formula for thick plates bending:

$$U = 0.1 \frac{\sigma_{Ht} L^3}{E_{roof} h^2 m},$$

where $E_{roof}$ is Young’s modulus of the roof; $h$ is its power; $m$ – reservoir thickness; $\sigma_{Hb}$ – average rock pressure at a depth of $H_t$; $L$ is the length of the roof caving step.

Based on the hydrostatic law $\sigma_{Ht} = \gamma_0 H$ it was obtained:

$$\frac{\delta P}{P} = \frac{A(H / H_0)}{1 + B(H / H_0 - 1)}.$$

The dimensionless coefficients $A$ and $B$ included in (9) are determined by the expressions:
The form of the graph of the change in \( \Delta P/P \) due to the dimensionless variable \( H/H_0 \) is shown in figure 2. It can be seen from the graph that an increase in depth leads to a decrease in the relative contribution to the total gas pressure of the additive due to the sorption hysteresis.

\[
A = 0.05 \frac{L}{E_{\text{roof}}} h^2 \gamma \frac{H_0}{m}; \\
B = B(\lambda H H_0).
\]  

Figure 2. Graph of the relative increase in gas the pressure \( \Delta P/P \) due to “hysteresis sorption” of methane by coal with a depth of immersion of the formation \( H/H_0 \).

Figure 3 shows the graphs of the calculations of the relative increase in gas pressure \( \Delta P/P \) due to the sorption hysteresis with the depth of immersion of the seam \( H \) at various depths values of the gas weathering zone \( H_0 \).

Figure 3. Change in the relative increase in the gas pressure \( \Delta P/P \) due to sorption hysteresis with the depth of the seam immersion \( H \).
As it can be seen from the graphs, when the reservoir is immersed at great depths – 800 m and more, the hysteresis effect of methane sorption by coal becomes almost imperceptible. Apparently, at great depths, the roof rocks above the seam near the face are clamped by rock pressure so much that bending of the roof with the formation of a “gas bag” becomes impossible.

The performed studies show that dynamic oscillations of the main roof during cavings, subsidence and other similar phenomena in the longwalls can significantly disturb the balance of active and passive forces in the bottomhole part of the seam due to short-term excess pressure of free gas as a result of an effect called hysteresis of methane sorption by coal.

The $\Delta P$ value can be determined using the graph in figure 3.

The computational model was also refined to take into account the power and strength of a potentially outburst-hazardous band (a set of adjacent bands) of coal in the face section, which largely determines the face outburst hazard.

Taking into account the said above, the dependences for determining $\text{grad } F_a$ and $\text{grad } F_p$ will look as follows:

$$\text{grad } F_a = \frac{1}{2} \frac{(P^2_s - P^2_t) \cdot n_s \cdot S \cdot 10^6}{P_s (n_s + \beta h_s)^{1/3}} \left[ L_s + \frac{1}{\beta} \ln \frac{n_s + \beta h_s e^{\beta n_s}}{n_s + \beta h_s} \right], \text{MN/m;}$$

$$\text{grad } F_p = 2(m_o + 1) \cdot (K_o + K_v f_{GP}) \cdot 10^5, \text{ MN/m,}$$

where $n_s$ is the fracture porosity of the rock mass in the section of a potentially outburst-hazardous unit behind the zone of influence of the mine; MPa; $S$ is the specific cross-sectional area of a potentially outburst-hazardous pack (a set of adjacent bands), equal to $m_o \times 1$ m, where $m_o$ – the power of this band (a set of bands); $L_s$ – the length of the area of the ultimate stress state of the formation, m; $K_o$ – coefficient of adhesion of the damaged structure of coal at the edge of the face, MPa; $\nu_o$ – coefficient depending on the angle of internal friction of coal of a potentially outburst-hazardous structure at the edge of the face $\rho_o$, deg.

Ratio $R$ (formula 3) and dependencies (12) and (13) can be used to determine non-outburst-hazardous parameters of bottomhole advancement (ratio of bottomhole advancement rate and depth of entry), with which it is possible to prevent the manifestation of outburst hazard [3].

4. Evaluation of the possibility of subsequent stages of a sudden outburst in the breakage faces

To assess the possibility of the next stage of a sudden coal and gas outburst, it is necessary to determine whether sufficient energy is accumulated in the bottomhole massif for the occurrence of a layer-by-layer separation wave, that is, whether the condition is fulfilled

$$W + E \geq F + U,$$

where $W$ is the potential energy of coal; $E$ – kinetic energy of rocks; $F$ – the work required to shift coal towards the mine; $U$ – the work required to break up coal in a sudden burst.

To determine this energy, various calculation methods are proposed [4-8, etc.]. However, most of them are not suitable for practical purposes due to the laboriousness of determining the
parameters used and the large amount of time spent on this process. The most acceptable method seems to be developed as a result of research by G.N. Feiyt [9-11]. The method is based on many years of research carried out at the National Mining Research Center – A.A. Skochinsky Institute of Mining and is a continuation of the energy theory. Its distinctive feature is the ability to perform a practical calculation of the potential energy of the massif with differentiation by gas and rock pressure factors, which increases the reliability of the forecast and allows, moreover, safe technological parameters to be reached. For the calculation, geological exploration data are used, supplemented by the results of observations in mine workings.

In accordance with this method, the gas energy is determined from the equation:

\[ W_g = \delta h \cdot b_{30} \cdot X \cdot \eta \cdot d, \]  

(15)

where \( \delta h \) is the change in the specific enthalpy of the gas; \( b_{30} \) is a coefficient that determines the fraction of gas evolved over 30 s (average outburst time); \( X \) is the gas content of the coal seam; \( \eta = 0.717 \) is the density of methane in the gas phase; \( d \) – specific gravity of coal.

The change in the specific enthalpy of a gas during its expansion is found using the dependence:

\[ \delta h = (X + 0.0027) \cdot 10. \]  

(16)

The \( b_{30} \) value is determined from the following correlation:

\[ b_{30} = 0.165 - 0.15 f_k, \]  

(17)

where \( f_k \) is the strength index calculated from caliper data.

Gas content \( X \) is determined depending on the depth \( H \) by the formula:

\[ X = (0.475 \cdot H - 7.05) \cdot 10^{-4}. \]  

(18)

Taking into account (17) and (18), formula (15) can be written in the form:

\[ W_g = X(X+0.0027)(1.65-1.5 f_k) \cdot \eta \cdot d. \]  

(19)

To determine the elastic recovery energy of a coal seam \( W_e \), the following formula is used:

\[ W_e = k H^2 / 2E, \]  

(20)

where \( k \) is the stress concentration factor; \( \gamma \) – average volumetric weight of rocks of the overlying strata; \( H \) is the depth of the seam; \( E \) is the elasticity modulus of coal.

To determine the value of \( E \), a correlation equation is proposed:

\[ E = (1.28 f_k - 0.087) \cdot 10^{-4}. \]  

(21)

To determine the outburst hazard of an untouched mining section of the seam when performing the joint work by A.A. Skochinsky Institute of Mining and VostNII Scientific Center for Industrial and Environmental Safety in the Mining Industry [12, 13] the following energy criterion for outburst hazard was obtained:

\[ E = W_g + 0.33 W_s - 0.18 \geq 0. \]  

(22)
Later, performing the next joint work (14), the energy criterion was adapted for the conditions of the face and took the form:

$$E_o = (W_g + 0.33W_s) l_o/l_g - 0.18 \geq 0,$$

(23)

where $l_o$ – bottomhole advancement in one cycle; $l_g$ is the length of the unloading zone of the bottomhole part of the seam, determined at the beginning of the cycle. For this criterion, the strength index can be determined directly at the bottomhole, which will significantly improve the accuracy of calculations. It takes into account the rate of bottomhole advance, which affects the value of the gradient of gas and rock pressure and, accordingly, the possibility of gas crushing of coal.

Conditions (3), (22), (23) are quite acceptable for practical use. However, their disadvantage is that they do not take into account the process of gas overflow from the coal massif in front of the face through the cracks formed in the roof bending zone into the gob. The mechanism of this phenomenon is very complex and has not yet been described. And because of this, a significant error of the 2nd kind is possible (the classification of zones that are actually not dangerous as dangerous ones) is possible.

The solution was the use of an additional indicator reflecting the actual presence of gas in the bottomhole part of the formation. Let us consider the possibility of obtaining such an indicator.

For an element of a coal massif adjacent to the face plane with a length along the face movement equal to $\Delta x$, according to [15], we can write:

$$\sqrt[m]{m_1^2 \left( \frac{dP}{dx} \right)} \geq \frac{\tau \cdot Per}{S},$$

(24)

where $m_1$ – the porosity of the coal mass in the zone of influence of the working; $\left( \frac{dP}{dx} \right)$ – gas pressure gradient; $\tau$ – shear stresses along the working perimeter; $Per$ and $S$ – respectively, the perimeter and area of the exposed coal massif in the section of the working.

This condition can be written as:

$$\left( P_s - P_f \right) \sqrt[m]{m_1^2 S} \geq 1,$$

(25)

where $P_s$ – gas pressure in the seam at a distance $x$ from the bottomhole; $P_f$ – gas pressure in the working at the contact with the face.

Taking into account the fact that the cavity of a sudden outburst develops in a tectonically disturbed coal, condition (25) can be specified:

$$\left( P_s - P_f \right) \sqrt[m]{m_1^2 S_o} \geq 1,$$

(26)

where $Per_o$ and $S_o$ are the same parameters of a potentially outburst-hazardous coal band (set of bands).

The $S_o/Per_o$ ratio for a production face can be written as follows:

$$\frac{S_o}{Per_o} = \frac{L m_o}{2(L + m_o)},$$

(27)

where $L$ is the length of the production face; $m_o$ is the capacity of a potentially outburst-hazardous band (a set of adjacent bands).
Since we are talking about mechanized faces in shallow seams, the value of $m_0$ is two to three orders of magnitude less than $L$. Therefore, the following approximate formula can be written:

$$r = \frac{S_i}{P \sigma_0} \approx \frac{L m_i}{2L} = \frac{m_i}{2}$$  \hspace{1cm} (28)

As a result, inequality (26) is written in the form:

$$\frac{m_i \left( P_x - P_f \right) \sqrt{m_i^2}}{2 \pi} \geq \frac{1}{10}.$$  \hspace{1cm} (29)

Let us pay attention to the expression $\left( P_x - P_f \right) \sqrt{m_i^2}$. Let us compare it with the dependence obtained in [16], which expresses the initial gas release rate in the interval of a well of unit length at a distance $x$ from the face:

$$g_{i,x} = 10 \pi k_s \left( P_x - 0.1^2 \right),$$  \hspace{1cm} (30)

where $\eta$ – coefficient, which can be considered constant; $k_s$ – permeability coefficient of the massif at a distance $x$ from the bottom.

We see that the functions $\left( P_x - P_f \right) \sqrt{m_i^2}$ and $g_{i,x}$ depend practically on the same parameters (the value of the gas permeability coefficient is directly related to the porosity of the coal mass).

In [16], the analytical studies were carried out, showing that within the limits of the change in the mining-geological and mining-technical parameters of mining in modern coal seams, these two dependencies are very similar in form and can be written:

$$\left( P_x - P_f \right) \sqrt{m_i^2} \approx a \left( g_{i,max}^* - g_{i,f}^* \right) f_o^2,$$  \hspace{1cm} (31)

where $a$ is a constant; $f_o$ – the strength factor of a potentially outburst-hazardous band (set of bands) according to M.M. Protodyakonov; $g_{i,max}^*$, $g_{i,f}^*$ – respectively the maximum along the length of the well and the initial gas release rate obtained for the first interval of the well, reduced to its design diameter, l/min; $l_{g}^*$ – distance to the middle of the interval where $g_{i,max}^*$ is measured, m.

The given values of the initial gas release rate are determined by the following formula [17]:

$$g_{i,i}^* = g_{i,i} \sqrt{\frac{V_i}{V_f}},$$  \hspace{1cm} (32)

where $g_i$ – values measured in the $i$-th interval of the control well, respectively, of the initial gas release rate and the output of the drill stem, l/min and l; $V_f$ – is the output of the coal dust, corresponding to the design diameter of the well, l.

To measure the indicators $g_{i,i}$ and $V_i$ for the purpose of assessing and monitoring gas-dynamic phenomena, as a rule, auger drilling of control wells with a diameter of 43 mm at intervals of 1 m is used with measurements in each interval after its re-drilling of the indicated parameters. The $V_f$ value for wells of this standard is 2 liters.
In the same work, it was proved that the voltage \( \tau \) can be replaced with an accuracy sufficient for practice by the expression \( b f_o^3 \), where \( b \) is a constant coefficient.

Taking into account the stated above, inequality (29) can be represented as:

\[
\frac{am_o(g_{i,max}^*-g_{i,j}^*)f_o^2}{2f_o^*bf_o^3} \geq 0.
\] (33)

Or, replacing constants \( a, b \) and 2 with one constant \( c \) and making reductions, the outburst hazard indicator for production faces can be expressed by the following formula:

\[
B_o = \frac{m_o}{f_o} \left( \frac{g_{i,max}^*-g_{i,j}^*}{f_o^*} \right) \geq 1.
\] (34)

The structure of the outburst hazard indicator is substantiated by the above theoretical studies. However, taking into account the use of approximate dependencies in obtaining it, the value of the coefficient \( c \) should be obtained using correlation analysis based on the data of mine experimental studies in zones of varying degrees of danger for sudden outbursts.

Based on the stated above, the following approach to assessing the outburst hazard of a coal massif is proposed. The assessment of the potential outburst hazard identified by the results of the forecast in the delineating workings and geophysical additional exploration of the seam sections is carried out according to the \( E \) indicator in accordance with the condition (22). Non-outburst hazardous parameters of face advance are determined by the \( R \) index using condition (3). It is advisable to check these parameters in terms of \( E_o \), taking into account the actual size of the unloading zone before advancing the face in accordance with condition (23).

In order to bring the obtained results closer to specific conditions, an assessment of the real hazard is carried out according to the \( B_o \) indicator, which is determined by the parameters directly measured in the face (including in the massif depth). The ratio of the gradient to the parameter indirectly expresses the ratio of the active force to the passive force, taking into account the real gas factor and can be used to control the outburst hazard when advancing faces in the coal seams.

5. Conclusions

The research results described in this work make it possible to solve the following problems:

1. To assess the potential outburst hazard of tectonically disturbed sections of the coal seam, the energy outburst hazard criterion \( E \), calculated on the basis of geological exploration data, supplemented by observational data in the mine workings outlining the working face.
2. To calculate the degree of outburst hazard of a particular face and select non-outburst-hazardous parameters of its advancement along the gradients of changes in active \( Fa \) and passive \( Fp \) forces of sudden coal and gas outburst and indicator \( R \).
3. Evaluate the real gas-dynamic state of the bottom-hole part of the formation in terms of the outburst hazard \( B_o \), based on the gas-dynamic characteristics of the rock massif directly measured in the face.

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