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THE DETERMINATION OF THE INFLUENCE DEGREE OF MINING-GEOLOGICAL AND MINING-TECHNICAL FACTORS ON THE SAFETY OF THE DEGASSING SYSTEM

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Abstract. This article presents the results of the calculation of methane emission from the development of coal seam and enclosing rocks in the roof on Mine Management "Sukhodolska-Skhidna". The required degassing efficiency and the degassing method were determined. The degree of influence of mining -technical and mining-geological factors on the safety of the degassing system has established. The determination of the influence degree of mining factors on the safety of the degassing system can be used both at the design stage mine, mining and preparing it, which allow to predict and prevent the formation of an explosive concentration of methane-air mixture.

Introduction. It is known that in underground conditions the most important factors, which influencing on the safety of gas production in the coal mining, are methane concentration, flow rate and emission sources. The gas factor is the main reason, limiting the productivity of working areas. The presence of a "gas barrier" in mines with high gas content and difficulties, connected with it, reduce the efficiency of coal mining technology. Earlier in the coal industry, the main attention was paid to improving the technology and techniques of coal mining, and issues of gas control were given a secondary role. Nowadays, the main task is to increase the mining safety operations and the associated extraction of mine methane [1].

According to [2], 42 % of DTEK's mines use degassing. Nine mines of them are classified as super-category (methane emissions over 15 m^3/ton or with blower gas emissions) and 7 mines are assigned to category 3 (methane emissions is 10-15 m^3/ton).

Thus, the purpose of the work is to estimate the influence degree of mining-geological and mining-technical factors on the safety of the degassing system, especially during designing mines [3] and predicting methane emissions.

To determine the factors, affecting on the safety of gas production in the coal mining [4-7], as an example, let's consider the conditions in 24th West sloping workface (24th WSW) of the coal seam i_3^1 at level 915 m of the Mine Management "Sukhodolska-Skhidna".

The exploitation of the 24th WSW with a regulatory capacity of 1250 ton per day is planned. The ventilation scheme of the workface is direct-flow with a cooling of the outgoing air stream. The direction of the air flow along the workface is upward.

The workface is equipped with a mechanized complex 3KD-90T, combine 2GSH-68B, conveyor SP-326. Shipment of rock mass from a mining site is carried out by belt conveyors 1 LT - 1000, 1 L - 1000UD, through which coal is transported to a central coal bunker and delivered by skips on the surface.

The enclosing rocks of the immediate roof are composed of siltstone, sometimes overlaid with sandstone with a thickness of 1-4 m. The main roof is represented by sandstone with a thickness up to 25.2 m. The direct and main soil is represented by siltstone with a thickness up to 15 m.The sandstones of the roof and coal seam i_3^1 are outburst and gas-bearing. In accordance with the forecast of the sudden breakthrough of methane from the soil, the 24 WSW is not dangerous.

Sources of methane emission in the mining working are the worked coal seam, the neighboring coal seams (satellites) and containing rocks. In conditions of the 24th WSW gas emission sources are:

– the thickness of worked coal seam i_3^1 up to 1.7 – 2.5 m;

– sandstone, lying in the roof of the seam i_3^1 thickness up to 25.15 m.

Methods. The calculation of the expected methane abundance of 24th WSW by methane emission sources was made in accordance with the guide on coal mine ventilation design [8].The methane emission from the worked coal seam and enclosing rocks in the roof and the necessary degassing coefficient were calculated. The influence of degree of factors, when they increase on the degassing coefficient, as well as the determination of the influence percentage of each factor were obtained by using the statistical methods of results processing.

Methane emission from the worked seam.

The calculation of methane emission from the worked seam q_s (m^3/ton) was made according to the formula [8]

$$q_s = q_{wm} + q_{cm} + k_{ol}\left(x - x_o\right), \tag{1}$$

where q_{wm} – relative methane emission from the workface of development coals, m^3/ton; q_{cm} – relative methane release from extracted coal, m^3/ton; k_{ol} = 0 – the coefficient of the operational loss of coal within the mining site (it was not calculated because of the absence of operational losses); x – the natural methane content of the seam, m^3/ton; x_o – the residual methane content of coal which remains in the worked-out area, m^3/ton.

$$q_{wm} = 0.85 \cdot x \cdot k_{ms} \cdot \exp\left(-n\right), \tag{2}$$

where k_{ms} – the influence of the mining system coefficient on the methane emission from coal seam; n – the parameter, which depends on the rate of the working face advancement, the release of volatile substances from coal and the depth development.

$$k_{ms} = \frac{L_w - 2b_{wy}}{L_w}, \tag{3}$$

where L_w = 345 m – the workface length; b_{wy} = 14 m – the coefficient, depending on the release of volatile substances [8];

$$x = x_g \cdot k_{WAa}, \tag{4}$$

where x_g = 22.5 m^3/ton of dry ashless mass– the natural methane content of the seam,

$$k_{WAa} = \frac{100 - W - A_a}{100}, \tag{5}$$

where k_{WAa} – the coefficient of methane content recalculation on coal seam; W = 3 % – the coal seam humidity; A_a = 28.9 % – the coal ash content,

$$n = a_1 \cdot V_w \cdot exp(\text{ - } 0.001 \cdot H + b_1 \cdot V^{\ daf}), \tag{6}$$

where a_1 = 0.152, b_1 = 0.051 – empirical coefficients [8]; H = 1000 m – seam depth; V_w =1.26 m/day – workface movement speed; V^{daf} = 26.3 % – the yield of volatile substances.

$$q_{cm} = q'_{mw} + q''_{mc}, \tag{7}$$

where q'_{mw} – relative methane release from coal extracted workface, m^3/ton; q''_{mc} – relative methane release from coal extracted in a belt entry, m^3/ton.

$$q'_{mw} = x \cdot k_{ms} \left[1 - 0.85 \cdot \exp(-n)\right] \cdot (b_2 \cdot k_{mb} + b_3 \cdot k'_{mw}), \tag{8}$$

$$q''_{mc} = x \cdot k_{ms} \left[1 - 0.85 \cdot exp(-n)\right] \cdot b_2 \cdot k''_c, \tag{9}$$

where b_2 = 0.6, b_3 = 0.4 – empirical coefficients, which take into consideration the proportion of repulsed coal respectively on the conveyor and left on the soil in the workface at one-side coal extraction scheme [8]; k_{mb}, k'_{mw}, k''_c – coefficients, which take into consideration the degassing degree of repulsed coal in the stope entry on the conveyor, on the soil in the workface and on the conveyor in the mining site:

$$k_{mb} = a \cdot T_w^b, \tag{10}$$

$$k'_{mw} = a \cdot T_c^b, \tag{11}$$

$$k''_c = a \cdot T_c^b - a \cdot T_w^b, \tag{12}$$

where T_w – the spending time of repulsed coal on the conveyor in the workface, min; a = 0.118, b = 0.25 – empirical coefficients, which characterizing gas recovery from repulsed coal [8],

$$T_w = \frac{l_w}{60 \cdot V_c}, \tag{13}$$

where V_c = 1 m/s – coal transportation speed in workface.

$$T_c = \frac{\sum_{i=1}^{n_i} l_{mi}}{60 \cdot V_{mi}}, \tag{14}$$

where T_c – the spending time of repulsed coal on the conveyor entry within the mining site, min; $n_i = 2$ – the number of mining sections l_{mi} at different speeds of coal; l_{mi} = 1000 m – the length of working with conveyor (SP-250 – 100 m; 1LT1000 – 900 m); V_{mi} = 1.6 m/s – coal transportation speed on the conveyor belt 1LT1000.

Methane emission from enclosing rocks in the roof.

In accordance with the geological and forecasting passport of the mining site of 24th WSW the gas-bearing sandstones are deposited in the roof of the worked seam.

The calculation of methane emission from enclosing rocks in the roof q_p (m^3/ton) was made according to the formula [8],

$$q_p = 1.14 \cdot V_w^{-0.4} \left(x - x_0\right) \cdot k_r \left(H_1 - H_0\right), \tag{15}$$

where k_r = 0.00106 – the empirical coefficient, taking into consideration the influence of the method of roof control and the lithological composition of rocks [8]; H_1 = 1150 m – the depth development; H_0 = 150 m – the depth of the upper boundary of the methane gas zone; x_0 – residual methane content of coal, left in the worked-out area in pillars and non-removable packs, m^3/ton

$$x_0 = x_{rm} \cdot k_{WAa} , \tag{16}$$

where x_{rm} = 2.6 m^3/ton of dry ashless mass – the residual methane content of coal.

Calculation of the required efficiency of degassing.

Let’s determine the required degassing efficiency

$$K_d \geq \frac{I_c - I_a}{I_c} \cdot 100 , \% \tag{17}$$

where I_c – average expected methane release on the mining site, m^3/min; I_a – average methane rate, which can be diluted with air to a safe level, m^3/min.

$$I_a = 0.0078 \cdot Q_c , \tag{18}$$

where Q_c – air flow rate for airing the mining site, m^3/min.

The maximum coefficient of advance degassing without fluid fracturing k_{ds} is equal 0.2 [9]. In order to achieve the required degassing efficiency of 24th WSW its calculation is supposed to be carried out according to the formula

$$K_d \leq \left(k_{ds} + \left(1 - k_{os}\right) \cdot \left(d_r \cdot k_{dr} + d_s \cdot k_s\right) + k_{os}\left(d_r + d_s\right)\right) \cdot 100, \tag{19}$$

where k_{os}, k_{dr}, k_s – the efficiency coefficients of the degassing of the worked-out area, roof and soil, fraction of units; d_r, d_s = 0 – the percentage of methane emissions from the roof and soil in the total gas balance of this area.

Thus, the technological process of gas extraction from the coal deposits development is based on abidance of the safety conditions of miners, located in the mining site.

Results and discussion. The results of the influence of mining-geological and mining-technical factors on the safety of gas extraction in the coal mine are described in Table 1. For the convenience of calculations, the values of all parameters increased by 5%, and the results were entered in Table 1, while their effect on safety conditions was recorded using computer simulation.

Table 1 – The values of the increased dates of the mining -technical and mining-geological factors

Factor number	Factor	Value	Increased dates				
			5 %	10 %	15 %	20 %	25 %
1	L_w	345.00	362.25	379.5	396.75	414.00	431.25
2	l_{mi}	1000	1050	1100	1150	1200	1250
3	V_{mi}	1.60	1.68	1.76	1.84	1.92	2.00
4	T_w	240	252	264	276	288	300
5	k_{ol}	0	0.05	0.10	0.15	0.20	0.25
6	V_c	1.00	1.05	1.10	1.15	1.20	1.25
7	H_0	150.0	157.5	165.0	172.5	180.0	187.5
8	Q_c	2907.00	3052.35	3197.70	3343.05	3488.40	3633.75
9	x_{rm}	2.60	2.73	2.86	2.99	3.12	3.25
10	H_1	1150.0	1207.5	1265.00	1322.50	1380.00	1437.50
11	x_g	22.50	23.625	24.75	25.88	27.00	28.13
12	W	3.00	3.15	3.30	3.45	3.60	3.75
13	A_a	28.9	30.345	31.79	33.24	34.68	36.13
14	H	1000	1050	1100	1150	1200	1250
15	V_w	1.260	1.323	1.386	1.450	1.512	1.580

A graph of the change of the degassing coefficient from the values of mining-geological and mining-technical factors was showed in Figure 1.

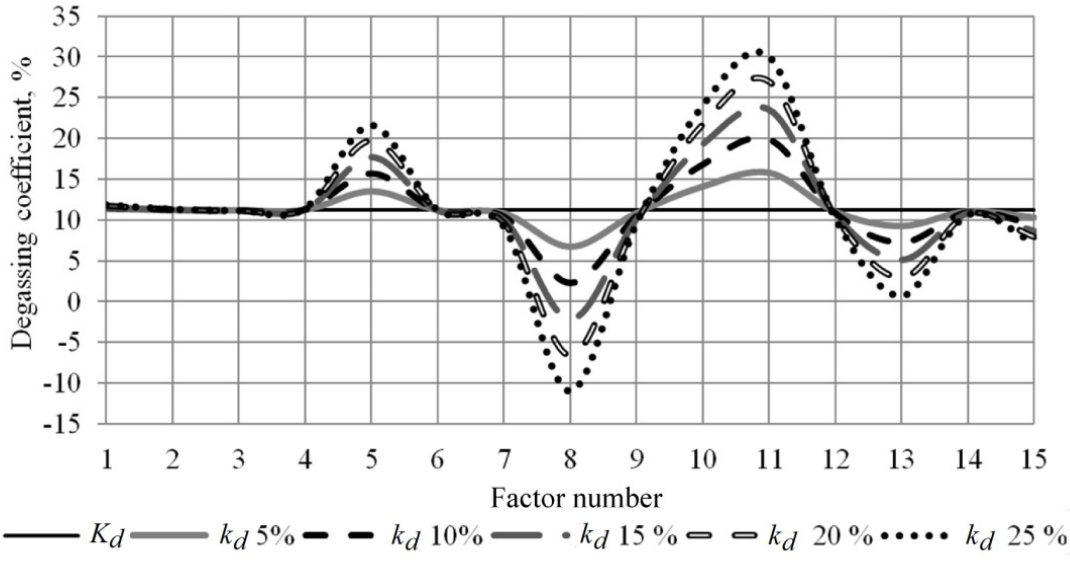

Figure 1 – The change of the degassing coefficient from increasing values of mine factors

According to Figure 1, the degassing factor changes with an increase of the following mine factors: k_{ol} (factor 5), Q_c (factor 8), H_1 (factor 10), x_g (factor 11), A_a

(factor 13), V_w (factor 15). An increase of other factors does not change the degassing coefficient significantly. Therefore, below we will consider only these six factors.

In the tables below the results of calculation of the degassing rate with an increase in mine values by the corresponding percentage (Table 2) and in recalculation by 1 % (Table 3) were presented.

Table 2 – The calculation results of the degassing coefficient with an increase of mining - technical and mining-geological factors by the corresponding percentage

Factor number	Factor	Degassing coefficient increased value k_d (K_d = 11.2167 % by an initial degassing coefficient)				
		5 %	10 %	15 %	20 %	25 %
5	k_{ol}	13.5114	15.6905	17.7624	19.7349	21.6151
8	Q_c	6.7776	2.3384	-2.1007	-6.5399	-10.9791
10	H_1	14.1041	16.8095	19.3497	21.7394	23.9915
11	x_g	15.7509	19.8445	23.5587	26.9439	30.0421
13	A_a	9.2920	7.2820	5.1809	2.9823	0.6794
15	V_w	10.3300	9.4900	8.6900	7.9500	7.2300

With an increase of mine factors by 5 % (such as k_{ol} (factor 5), H_l (factor 10), x_g (factor 11)), the degassing coefficient rises accordingly by 20 %, 25.7 %, 40.4 % and with an increase by 5 % (such as Q_c (factor 8), A_a (factor 13), V_w (factor 15)) degassing coefficient decreases accordingly by 39.6 %, 17.2 %, 7.9 %. The change of the degassing coefficient from the mine factors occurs by linear dependence (Table 2).

Table 3 – The calculation results of the degassing coefficient change with increasing mining-technical and mining-geological factors in recalculation by 1%

Factor number	Factor	The share of change in the degassing coefficient with an increase in the factor values in recalculation by 1%, S				
		5%	10%	15%	20%	25%
5	k_{ol}	2.7023	1.5690	1.1842	0.9867	0.8646
8	Q_c	1.3555	0.2338	-0.1400	-0.3270	-0.4392
10	H_1	2.8208	1.6809	1.2900	1.0870	0.9597
11	x_g	3.1502	1.9845	1.5706	1.3472	1.2017
13	A_a	1.8584	0.7282	0.3454	0.1491	0.0272
15	V_w	2.0664	0.9495	0.5801	0.3974	0.2893

The change of the degassing coefficient with an increase of the factor value in recalculation by 1 % was showed in Figure 2.

As a result of calculated data analysis the approximation equation (Figure 2) in the form of degree functions and two degree polynomial was obtained by means of mathematical statistics methods, which can represented as:

$$S = 0.3144 \cdot k_{ol}^{-0.71},\ R^2 = 0.99;$$

$$S = 0.5123 \cdot x_g^{-0.6},\ R^2 = 0.99;$$

$$S = 0.3683 \cdot H_1^{-0.672},\ R^2 = 0.99;$$

$$S = 0.056 \cdot V_w^{-1.215},\ R^2 = 0.99;$$

$$S = 62.945 \cdot A_a^2 - 27.366 \cdot A_a + 2.9957,\ R^2 = 0.98;$$

$$S = 63.027 \cdot Q_c^2 - 27.209 \cdot Q_c + 2.4847,\ R^2 = 0.98.$$

where R^2- approximation accuracy.

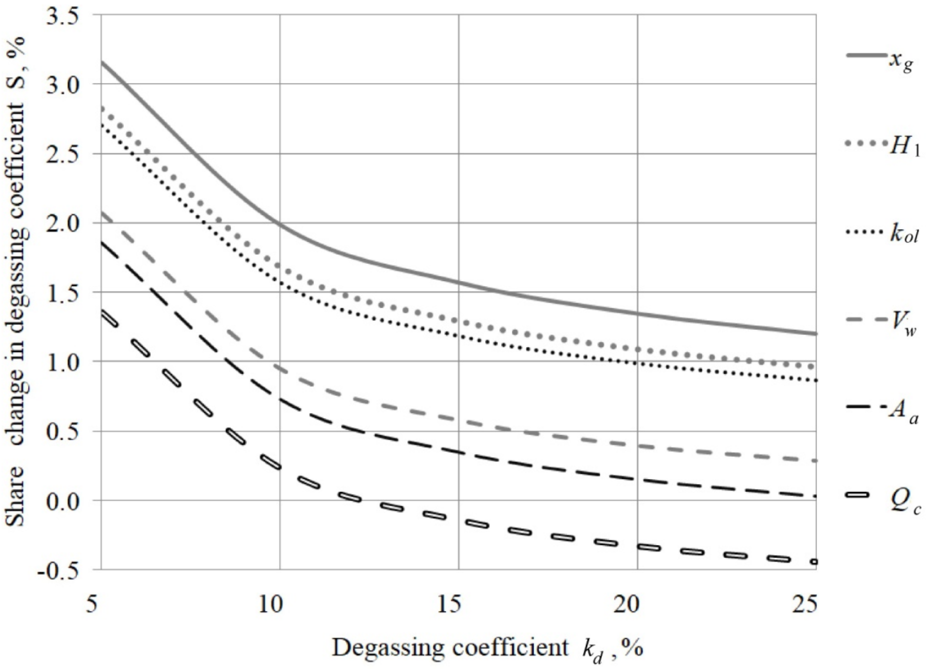

Figure 2 – The change of the degassing coefficient with increasing mine factors in recalculation by 1 %

Conclusion. The influence degree of mining-geological and mining-technical on the change of the degassing coefficient was determined. Among the mining factors that have the greatest degree of influence on the safety of the degassing system in the mining coal are: the coefficient of the operational loss of coal within the mining site k_{ol}; air flow rate for airing the mining site Q_c; workface movement speed V_w. Among the mining-geological factors that have the greatest influence on the safety of gas production the mining coal are: the depth development H_1; the natural methane content of the seam x_g; the ash content of coal A_a

The change of degassing coefficient with an increase of mining-technical and mining-geological factors on 1 % occurs by a two-degree polynomial and degree functions with approximation accuracy $R^2 = 0.98 – 0.99$.

These results can be used both at the design stage mine, mining and preparing it, which allow to predict and prevent the formation of an explosive concentration of methane-air mixture.

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