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Improving Coal Mine Safety via Aerodynamic Resistance Control in Longwall Faces: A Quasi-Network Model Approach

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ABSTRACT

The problem of eliminating gas contamination in mine workings and reliably forecasting accident probability remains relevant in the coal industry. This article presents scientific and technical developments for managing the aerodynamic parameters of working areas and collapsed rock massifs to combat gas contamination at the production face. The authors propose controlling gas emissions in the working area by regulating air leaks through the mined-out space of the longwall. The study considered the working area as a quasi-network model. Based on this model, theoretical and experimental studies were conducted on air leaks from the longwall and its inflows into the supported ventilation workings. These studies focused on working areas with a direct-flow ventilation scheme in the Karaganda coal basin mines. Numerical experiments were carried out for three methods of gas emission control. As a result, a mathematical model was developed to calculate aerodynamic resistance, taking into account the length of the column, longwall mining, and the distribution of airflow in the mining section (Q_1/Q_2). This model enables the estimation of the probability of forming an explosive concentration of methane, thus improving safety measures in coal mine operations.

1. INTRODUCTION

Present day coal mines are technically and technologically complex production facilities. Their operation is associated with numerous risks, the management of which requires strict compliance with all the established safety standards. The legislative framework for ensuring safety in the coal industry includes both international standards and national legislation developed taking into account the characteristics and specifics of the industry.

To improve the safety of work in the mining industry, on June 22, 1995, the International Labor Organization adopted Convention No. 176 "On Safety and Health in Mines" in Geneva [1]. This convention proposes to prevent all the fatal accidents, injuries or diseases of workers that are caused by industrial activities in the mining industry. Article 7 of Convention No. 176 recommends that employers take all the necessary measures to eliminate and to minimize risks to life and health in mines.

Coal mines are considered hazardous production facilities. The most dangerous production factor is emission of methane from the coal massif in the course of stoping and preparatory work and formation of an explosive methane-air mixture in the mine workings. A methane explosion, including one involving coal dust, is the most dangerous emergency situation in a coal mine. Workers in the explosion zone are exposed to flame and high temperature, shock waves and toxic explosion products. Explosions are also accompanied by damage to the support of the mine workings, mechanisms and equipment located in them and cable networks.

Methane emissions and explosions in coal mines in various countries occur annually. Table 1 provides the information of emissions and explosions of methane-air mixtures that occurred in the period from 2014 to 2023 in the world [2-4].

It can be seen from Table 1 that the consequences of methane outbursts and explosions are always tragic. All the tragic incidents that have occurred, point to a common problem in the coal industry related to the insufficient provision of safety measures during underground mining operations. All the accidents were caused by ineffective gas emission management during coal mining processes and unsatisfactory monitoring of the mine atmosphere, which led to the accumulation of methane with the formation of an explosive gas-air mixture. The use of obsolete and outdated equipment in mines also increases the risk of accidents and the occurrence of a source of ignition of the gas-air mixture. The accidents that have occurred, point to the need to improve gas emission management methods and the methane content monitoring, as well as to modernize mine equipment and to improve labor safety standards to prevent similar tragedies in the future.

For a methane-air mixture to explode, two conditions must be present simultaneously. The first condition is the methane-air mixture formation of a sufficient concentration for an explosion. For methane, this concentration is approximately 5–15% by volume. The second condition is the occurrence of an ignition source with a sufficient temperature or energy to initiate the methane combustion reaction. This can be a spark (electrical or mechanical), an open flame, or a high temperature of a heated surface. If two of these conditions are met simultaneously, an explosion occurs, releasing a large amount of heat, carbon monoxide, carbon dioxide, and water vapor. To reduce the risk of a methane-air mixture explosion, one of the above factors must be eliminated. Therefore, optimizing ventilation parameters is critical to preventing methane accumulation.

Mine workings contamination with gas can be the result of sudden coal and gas emissions, vent emissions, layered and local methane accumulations. All the cases of exceeding the established norms of methane concentration in mine workings constitute an emergency situation in the mine. Methane accumulations can also be caused by emergency stops of main and local ventilation fans, improper degassing of mine workings, incorrectly operating gas measuring equipment, as well as engineering errors in calculating the required amount of air and unsatisfactory control over ventilation of mine workings. Any contamination of mine workings in coal mines can form conditions for the formation of an explosive methane-air environment, which, in the presence of an ignition source, can lead to an explosion with significant human casualties and material losses.

Currently, coal mines use various special measures to combat methane to prevent dangerous methane accumulations and its ignition. However, the main measure to prevent the formation of dangerous concentrations of methane is effective ventilation, as a result of which the permissible methane content is maintained throughout the entire network of operating mine workings.

A significant number of scientific papers deal with the issues of ventilation of coal mines. The problem of increasing reliability and efficiency of ventilation of mine workings is of considerable interest and is the subject of studies in numerous scientific papers. This explains the growing attention and interest of scientists in the issues of ensuring safety during mining operations. This problem is studied especially intensely in China.

Table 1. The information of emissions and explosions of methane-air mixtures that occurred within the period from 2014 to 2023 in the world

Date	Place	Type	Consequences, Number of Victims
April 21, 2014	Private coal mine in southwestern Yunnan Province in Qujing City (China)	Gas explosion	13 miners
July 5, 2014	Coal mine in Xinjiang Uyghur Autonomous Region (China)	Gas explosion	17 miners
November 27, 2014	Coal mine in Guizhou Province (China)	Gas explosion	11 miners
December 14, 2014	Coal mine in Jixi City of northeastern Heilongjiang Province (China)	Gas explosion	10 miners
March 4, 2015	Zasyadko Coal Mine in Donetsk People's Republic	Gas explosion	30 miners
August 12, 2015	Coal mine in southern Guizhou Province (China)	Gas outburst	10 miners
February 25, 2016	Severnaya Coal Mine, Vorkuta (Russia)	Gas explosion	36 miners (inc. mine rescuers)
March 6, 2016	Coal mine in Tonghua County, Jilin Province (China)	Gas outburst	12 miners
October 31, 2016	Coal mine in Inner Mongolia Autonomous Region of northern China	Gas explosion	33 miners
December 3, 2016	Coal mine in Laisu City, Chongqing Municipality (China)	Gas explosion	17 miners
December 5, 2016	Coal mine in Inner Mongolia Autonomous Region of northern China	Gas outburst	11 miners
May 3, 2017	Coal mine near Azadshahr City, Golestan Province (Iran)	Gas explosion	42 miners
May 7, 2017	Xinjia Coal Mine in Hubei Province (China)	Gas outburst	18 miners
August 31, 2017	Coal mine "Kazakhstanskaya" in the Karaganda region (Kazakhstan)	Gas outburst	3 miners
August 6, 2018	Coal mine "Zimujia", Panzhou city (China)	Gas outburst	13 miners
August 12, 2018	Coal mine in western province of Balochistan (Pakistan)	Gas explosion	12 miners
December 20, 2018	Coal mine ČSM-north near Karvina city (Czech Republic)	Gas explosion	13 miners
December 17, 2019	Coal mine in Guizhou province (China)	Gas outburst	14 miners
September 27, 2020	Coal mine in Chongqing city (China)	Gas outburst	16 miners
November 7, 2021	Abayskaya coal mine in Karaganda region (Kazakhstan)	Gas outburst	8 miners
November 25, 2021	Listvyazhnaya coal mine in Kemerovo region (Russia)	Gas explosion	51 горняк
November 3, 2022	Lenin coal mine in the Karaganda region (Kazakhstan)	Gas outburst	5 miners
April 20, 2022	Pniowek coal mine in Silesian Voivodeship (Poland)	Gas explosion	14 miners
October 14, 2022	Coal mine in northern Turkey	Gas explosion	41 miners
December 9, 2022	Coal mine in West Sumatra province (Indonesia)	Gas explosion	10 miners
March 14, 2023	Coal mine in Sutatausa municipality (Colombia)	Gas explosion	21 miners
October 28, 2023	Kostenko coal mine in the Karaganda region (Kazakhstan)	Gas explosion	46 miners

The analysis of mine ventilation in work [5] shows that in the period from 2010 to 2023, the studies in this area went through three stages: stable development, slow growth and rapid growth. The main areas of research are in the modeling of air flows, gas dynamics and the development of ventilation control systems. For example, in work [6], scientists built an experimental model of a three-dimensional ventilation network with an independent working face, developed a platform for monitoring ventilation parameters and fan control. Reference [7] discusses possible correlations between the characteristics of ventilation air and underground mining operations. In their research paper [8], Wang et al. present a method of air quantity distribution in the design of coal mine ventilation, and propose a method to eliminate air leakage through ventilation structures in the ventilation system. Zhou et al. [9] improved the ventilation network calibration method by applying sensor location correction factors to obtain a more accurate value of the average air velocity. In their paper, Pei et al. [6] reported the experimental results of the fact that the gas concentration in the working face has a power-law relationship with the operating frequency of the fan, and also confirm that increasing the air quantity and vacuum is an effective means to solve the problem of unbalanced gas emissions in the working face.

Numerous studies focused on the simplification of complex mine ventilation networks [10-13]. Network simplification involves reducing its size by reducing the number of nodes and branches. This method allows reducing the branching ratio of the ventilation network, significantly improving the simplicity of the ventilation network and has a higher applicability and greater flexibility than traditional methods. The use of network simplification technology significantly accelerates the analysis and reduces the time of developing a complex mine ventilation network. Simplifications of ventilation networks allow designing and building an experimental platform for monitoring and automatic regulation of air flow parameters in the face. For example, in work [12], the authors present an algorithm for equivalent simplification of complex ventilation networks based on the degree of entry and exit of nodes, as well as on the inflow and outflow nodes of branches, and also proposed an optimization strategy for a set of pairs of nodes requiring deep search in the network based on the principle of balance of flows in the incoming and outgoing branches of the subnetwork nodes. In work [13], scientists, using the method of equivalent transformation of the topological relationship of the network, proposed a method of solving the problem of constructing a Q-H graph for a network with unidirectional contours, and also developed a simplified mathematical model of the network based on the method of independent trajectories and the theory of adjacency matrices of nodes.

In modern research, innovative solutions such as remote ventilation control using sensors and the use of artificial intelligence for data analysis are recommended to effectively improve the optimal use of air supply. The growth of digitalization and intelligent technologies in the mining industry increased the research interest and importance of mine ventilation in recent years. The intelligent ventilation system using real-time monitoring data can immediately correct the inefficient distribution of air volume, realize automatic power adjustment, remote control and monitoring of the facility, and identify dangerous areas [14-16].

Despite the existence of a number of scientific studies in the field of mine ventilation, the application of innovative technologies in ventilation and improving the performance of modern fans, the problem of eliminating methane explosions in coal mines has not yet been solved. As the accident statistics presented in Table 1 show, explosions in mines with tragic consequences still occur.

The formation and diffusion processes of the gas-air mixture in the mined-out areas remain poorly understood during the ventilation of working sections using a direct-flow ventilation system with refreshed exhaust airflow. Previous studies [6-10] and the obtained patterns of air leakage through the mined-out space assume a constant value of its specific aerodynamic resistance. At the same time, according to other studies [17, 18], the specific aerodynamic resistance changes as the distance from the longwall face increases, which complicates the task of determining the filtration velocity and flow direction in the collapsed space.

The inaccessibility of the mined-out space for direct observations and measurements of air leakage and permeability has led to the use of a quasi-network model.

Assessing, managing, and reducing the risk of methane-air mixture explosion in working areas make an integral part of the industrial safety management system of mines and a relevant scientific problem. The authors of the article propose to manage gas emission in the working area by regulating air leaks through the mined-out space of the longwall to solve this scientific problem. In the study, the working area is considered as a quasi-network model, the structure of which is as close as possible to the real rock massif. The studies were carried out in working areas with a direct-flow ventilation scheme of the Karaganda coal basin mines.

This study aims to create a mathematical model of the resistance in the mined-out space to optimize ventilation parameters in the working area and evaluate the likelihood of explosive methane concentrations forming in the workings under a direct-flow ventilation scheme.

The study results enabled the identification of the optimal ratio between the airflow directed to the longwall and the additional airflow in the supported ventilation roadway, which prevents the formation of a gas accumulation zone at the working face and lowers the risk of a methane-air mixture explosion. Numerical experiments were conducted for three methods of gas emission control using the example of mines in the Karaganda coal basin: changing the air flow rates in the longwall and ventilation working; using insulation of the mined-out space; blasting the ventilation working.

2. MATERIALS AND RESEARCH METHODS

Direct measurements of aerodynamic parameters of the developed space in mine conditions are associated with certain difficulties. Therefore, theoretical and experimental studies of air leaks from the longwall and its inflows into the supported ventilation working were carried out in this study using a quasi-network model.

To assess the effect of the developed space state of the longwall on the air inflows into the supported ventilation working, a series of gas-air measurements were conducted in the mines of the Karaganda coal basin. The studies and necessary measurements were carried out at five working areas with a direct-flow ventilation scheme. The methodology of conducting gas-air surveys was as follows. The general part of the studied ventilation working was divided into sections 25 m long. Air and depression surveys were carried out for each section in accordance with the current regulatory documents on industrial safety. The air inflow through the developed space to the studied section was determined as the difference between the air flow rates in adjacent flows.

The obtained measurement results were subjected to mathematical processing using the correlation-regression analysis. This method of mathematical statistics allowed identifying the relationships between the studied values. The analysis and assessment of the closeness of the relationship between the values of the dependences was carried out using the standard deviation and the correlation coefficient. To assess the strength of the relationship between the studied parameters, the Chaddock scale of English statistics was used.

3. QUAI-NETWORK MODEL OF THE LONGWALL WORKED-OUT SPACE

The application of pillarless coal seam mining technology combined with a direct-flow ventilation system in the Karaganda coal basin has proven to be highly effective. It allows increasing the load on the working faces, reducing the volume of development workings, eliminating coal losses in the near-cut pillars, and forming favorable conditions for the isolated liquefaction and removal of methane at the sources of its intake. At the same time, a complicating factor in the widespread use of a direct-flow ventilation scheme for the working area is the need for reliable control of the aerogas dynamics of the mined-out space. Here the determining role is played by air leaks through the caving zones, which play an important role in the formation of the gas balance of the working area. On the one hand, due to leaks, a significant part of methane is removed directly to the ventilation working, bypassing the stoping working. On the other hand, leaks contribute to decreasing the volume of air used for ventilation of the face space and to a possible excess of permissible methane content standards in the working area of the longwall.

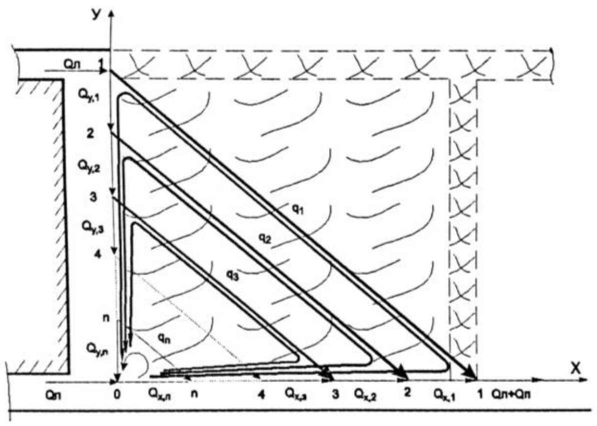

Figure 1. Quasi-network representation of the mined-out space of the longwall

Considering that the working face in a direct-flow ventilation scheme is a diagonal element, i.e., it is located in a potentially unstable ventilation zone, it is important to study the process of filtering air leaks through the mined-out space. To investigate how the aerodynamic properties of the collapsed rock mass and the existing workings within the working area influence the distribution of airflow, a quasi-network model of the longwall's mined-out space, as shown in Figure 1 [19], is proposed for use.

In the conditions of an active stope, the origin of coordinates is the point located in the area of the stope line and the wall of the supported working junction on the side of the mined-out space. The *OX* axis is directed along the wall of the supported stope. The *OY* axis runs along the direction of the stope, while the *OZ* axis is oriented perpendicular to the coal seam plane. The number of branches connecting the nodes in the longwall and the ventilation working and simulating the directions of air movement through the collapsed space varies from 1 to n depending on the size of the mined-out space. Thus, this model represents a complex ventilation system system comprising 3n branches, 2n + 1 nodes, and n independent circuits.

The structure of the quasi-network model maximally reflects the real processes occurring with the rock massif in the conditions of an active stope. Therefore, this model allows simulating and solving complex ventilation problems.

The system of equations describing the studied circuit, for a given direction of bypassing the circuits, has a closed system with respect to the unknown air leakages q_i, $i = \overline{1, n}$ and has the form [20, 21]:

$$\begin{cases} \sum_{i=1}^{n} R_{y,i} \left(Q_л - \sum_{j=1}^{i} q_j \right)^2 + \sum_{i=1}^{n} R_{x,i} \left[\left(Q_л + Q_п \right) - \sum_{j=1}^{i} q_j \right]^2 - r_1 q_1 = 0; \\ \sum_{i=2}^{n} R_{y,i} \left(Q_л - \sum_{j=1}^{i} q_j \right)^2 + \sum_{i=2}^{n} R_{x,i} \left[\left(Q_л + Q_п \right) - \sum_{j=1}^{i} q_j \right]^2 - r_2 q_2 = 0; \\ \sum_{i=3}^{n} R_{y,i} \left(Q_л - \sum_{j=1}^{i} q_j \right)^2 + \sum_{i=3}^{n} R_{x,i} \left[\left(Q_л + Q_п \right) - \sum_{j=1}^{i} q_j \right]^2 - r_3 q_3 = 0; \\ \cdots \\ \sum_{i=n}^{n} R_{y,i} \left(Q_л - \sum_{j=1}^{i} q_j \right)^2 + \sum_{i=n}^{n} R_{x,i} \left[\left(Q_л + Q_п \right) - \sum_{j=1}^{i} q_j \right]^2 - r_n q_n = 0. \end{cases} \tag{1}$$

This system of equations is nonlinear and is solved by the Newtonian linearization method [19]. The essence of this method is the linear approximation of nonlinear network laws. As a result of solving Eq. (1), the calculated values of air flow for each section of the longwall are determined. However, they do not always correspond to the actual data.

This is because, under a direct-flow ventilation scheme, the airflow changes along the length of the working area. Moreover, air leakage through the mined-out space has a significant impact on the turbulence level of airflow, as well as on parameters such as the Reynolds number and the aerodynamic resistance coefficient. As a result, the calculated airflow values need to be adjusted to account for air leakage through the mined-out space.

4. STUDYING AIR LEAKS FROM THE LONGWALL AND ITS INFLOWS INTO THE SUPPORTED VENTILATION WORKINGS BASED ON A QUASI-NETWORK MODEL

The studies of air leaks from the longwall and its inflows into the supported workings carried out on the basis of the quasi-network model, show their interval correspondence. If the longwall and supported workings are divided into n intervals, then with the longwall length l, the length of the interval along the *OY* axis will be equal to $\Delta y = l_{lw}/n$ and, accordingly, with the length of the worked-out part of the seam l_{wo}, the length of the interval along the *OX* axis will be equal to $\Delta x = l_{wo}/n$.

The distances along the flow lines along the *OX* and *OY* axes can be taken to be equal only in cases where the total air leaks from the longwall $\sum q_{y,i}$ along the intervals $\Delta y_i = \overline{1,n}$ are equal to the inflow value $\sum q_{x,i}$.

The analysis of the obtained experimental data shows that the air leaks in the longwall coincide with similar inflows into the supported working along the corresponding intervals of the *OX* and *OY* axes. In the calculated ventilation scheme, these leaks can be replaced by a network of diagonal branches, which confirms the validity of using a quasi-network model to solve problems of managing the aerogas situation in working faces with a direct-flow ventilation scheme.

Studying air filtration in the mined-out space is complicated by the fact that the equation of the *i*-th branch, characterizing the path of leakage movement, is unknown. Estimating the possible direction of leakage filtration through the collapsed massif, we come to the conclusion that the equation of the *i*-th branch can be described by one of the equations of the power type:

$$y = y_i[1 - (x/x_i)]^{n_i} \tag{2}$$

where, y_i is the numerical value of the ordinate of the *i*-th branch with $x=0$;

x_i is the numerical value of the abscissa of the *i*-th branch with $y=0$;

x is the current value of the abscissa in the interval of 0, x_i;

n_i is the function index for the *i*-th branch.

The graphical interpretation of dependence (2) is presented in Figure 2.

With $n_i<1$ function (2) is convex and corresponds to the shape of flow line 1 (Figure 2). With $n_i=1$ Eq. (2) describes a straight line 2. With $n_i>1$, dependence (2) is concave curve 3. Curves 2 and 3 usually describe the flow lines near point 0, i.e. in the zone of junction of the longwall with the supported working.

An important issue in the formation of the database in the process of preparation for the calculation of the quasi-network model is the determination of the aerodynamic resistance of all the *i*-th branches simulating the filtration paths of the gas-air mixture through the collapsed massif.

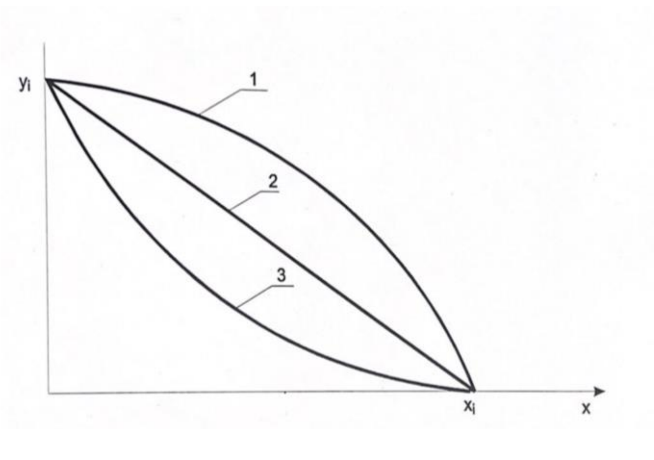

Figure 2. Graphs of Eq. (2) for various function indices for the *i*-th branch

The specific aerodynamic resistance of collapsed rocks *r* at a distance *x* from the face can be found by the formula:

$$r = r_0 \times e^{B \cdot x^n} \tag{3}$$

where, r_0 and *B* are experimental coefficients.

Knowing the value of the indices of function (2), we can find the resistance of each branch. The specific resistance for the mines of the Karaganda coal basin can be represented as function (3) provided that $n=1$. In this case, the resistance of the *i*-th branch can be found through the specific aerodynamic resistance of the collapsed rocks after introducing the arc differential *dl*, i.e.

$$R_{B,i} = \int_0^{x_i} r dl. \tag{4}$$

Since $dl = \sqrt{y_i^2 + 1dx}$, resistance of the *i*-th branch in Eq. (4) after substituting the corresponding data will be determined by the formula:

$$R_{B,1} = \int^{x_i} r_0 e^{Bx} \sqrt{\frac{y_i^2 \cdot n_i^2 \cdot x^{2(n-1)}}{x_i^{2ni}} + 1dx} \tag{5}$$

The solution of Eq. (5) is associated with the determination of the coefficients r_0 and *B* included in the integrand. Experimental studies were conducted for this purpose.

In-kind measurements were carried out in the mines of the Karaganda coal basin. Three series of measurements were taken. The first was before the initial settlement of the main roof, the second was after the initial settlement of the main roof, and the third was under established rock pressure. During each series of measurements, the volume of air supplied to the longwall Q_{lw} and the air flow rate supplied to light the outgoing longwall jet Q_l were recorded. To determine the air flow rates Q_i, the cross-sectional area *S* and the average air flow velocity *V*, as well as the pressure losses *h* in all sections, were measured in each *i*-th section of the calculation scheme.

According to the scheme (Figure 3), the values of the reduced aerodynamic resistances $R_{B,i}$ $i=1,5$ were determined taking into account the fulfillment of the second law of networks and air leakage q_i through the mined-out space as the difference between the air flow rates in adjacent sections. The measurement results are presented in Table 2.

Statistical processing of the q_i and $R_{B,i}$ data (Table 2) shows that the $R_B(x)$ dependence has a pronounced exponential character. For the first series of experimental data, the regression equation has the form:

$$R_B = 1.667\ e^{0,0115x} + 4.04. \tag{6}$$

The second characterizes the jump in aerodynamic resistance in the primary landing zone and is associated with the process of displacement of the immediate roof. Here the dependence is described by formula (7):

$$R_B = 1.667\ e^{0,0146x} \tag{7}$$

The third series of experiments falls on the period with established rock pressure, for which the aerodynamic resistance of the collapsed massif is characterized by a dependence of the form:

$$R_B = 1.667\ e^{0,012x} + 4.04. \tag{8}$$

The value of 1.667 in dependencies (6-8) characterizes the value of resistance r_0 of the collapsed massif adjacent to the stope. In the zone of primary roof settlement, the coefficient *b* is equal to zero, and at steady rock pressure it is equal to 4.04. The value of *B* changed insignificantly within the range of 0.0115 to 0.0146. The correlation coefficients of the obtained equations of the form (6-8) are high (r>0.89), which indicates the presence of a close statistical relationship between the studied values.

Differentiating Eq. (8) with respect to X with subsequent equating the obtained value to the integrand of formula (5), it is clear that the equality condition will be fulfilled with the exponent $n_i=1$, which indicates a linear dependence of function (2) on X.

Taking into account the above, it can be assumed that the current lines through the collapsed massif are parallel. In this case, the following statement is true:

$$\frac{Y_i}{X_i} = \frac{l_{lw}}{l_{wo}} \tag{9}$$

where, l_{lw} is the longwall length, m;

l_{wo} the total length of the worked-out part of the massif, m.

After the appropriate transformation of formula (5) with consideration of (9), we obtain:

$$R_{B,i} = \int_0^{x_i} r_0 \cdot e^{Bx} \sqrt{\frac{l_{\text{отр}}^2 + l_{\text{л}}^2}{l_{\text{отр}}^2}}\, dx \tag{10}$$

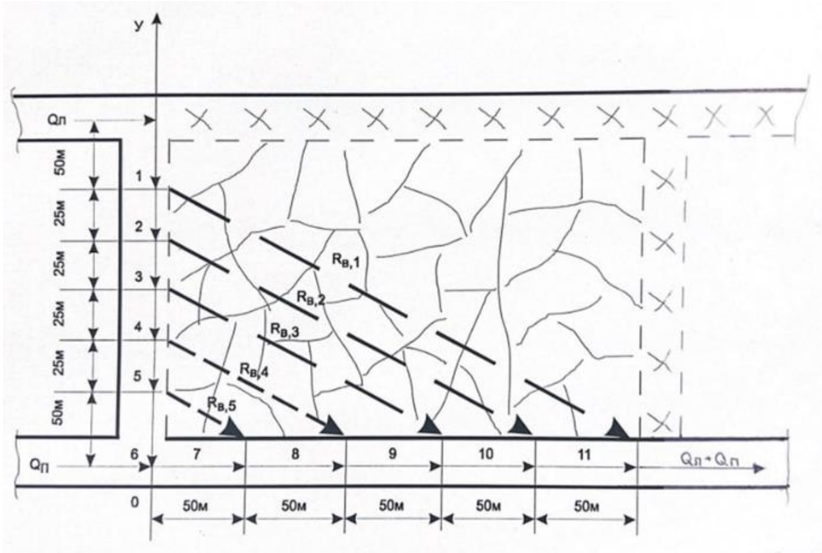

Figure 3. Layout of measurement points for determining resistance of the collapsed massif

Table 2. Results of calculations for determining air leaks and aerodynamic resistance of the mined massif

Branch No.	Series of Observations					
	I		II		III	
	q_i, м³/с	$R_{B,i}$, даПА с/м³	q_i, м³/с	$R_{B,i}$, даПА с/м³	q_i, м³/с	$R_{B,i}$, даПА с/м³
1	0.75	38.70	0.50	64.40	0.70	34.93
2	0.95	23.50	0.80	27.25	0.70	26.79
3	0.95	16.53	0.96	18.44	0.80	16.69
4	0.26	38.80	0.28	41.43	0.70	11.50
5	1.64	3.23	1.64	3.48	0.80	5.00

Table 3. Actual and calculated values of aerodynamic resistance $R_{B,i}$ of mined-out spaces for different longwall faces

Longwall	Disrance from the Working Face x_i, m	Aerodynamic Resistance $R_{B,i}$, ∂aPa s/m³		
		Actual	Calculated	Deviations
	100	24.72	23.20	-6.15
Longwall 1	175	55.50	55.56	+0.11
	252	111.00	111.06	+0.08
	50	5.0	4.5	-10.0
	100	11.5	10.0	-13.0
Longwall 2	150	16.69	16.71	+0.1
	200	26.79	24.92	-7.0
	250	34.93	34.93	0
	100	25.04	27.6	+10.20
Longwall 3	150	51.3	51.3	0
	250	135.0	135.1	+0.07
	100	22.65	22.65	0
Longwall 4	150	50.65	47.5	-6.2
	250	157.4	165.5	+5.1
	100	21.9	22.1	+0.91
Longwall 5	150	42.1	42.01	-0.21
	250	116.6	116.4	-0.17

It follows that the value of resistance $R_{B,i}$ will be determined from expression (11):

$$R_{B,i} = \frac{r_0}{B}\sqrt{\frac{l_{wo}^2 + l_{lw}^2}{l_{wo}^2}}\,(e^{Bx_i} - 1),\, \partial aPa\ s/m^3 \qquad (11)$$

Table 3 presents the actual and calculated values of the aerodynamic drag $R_{B,i}$ determined with the use of formula (11).

A comparative analysis of the values of aerodynamic resistance $R_{B,i}$ found from the measurement data in mines at different distances from the face of the longwall and obtained using formula (11) shows that the calculated values have minor deviations (Table 3). Thus, formula (11) should be used when preparing the initial data for calculating air leaks through the collapsed massif.

5. MODELING AIR LEAKAGE THROUGH THE MINED-OUT AREA OF THE LONGWALL FACE

The most effective means of preventing local methane accumulations at the longwall face with the supported working junction is its effective ventilation. To determine the optimal ventilation parameters for the working area, air leaks through the mined-out space of the longwall face were simulated. The aerogas situation is controlled by aerodynamic methods, i.e. by changing the aerodynamic parameters of the branches that form the calculated ventilation scheme of the area. This primarily includes changing the ratio of the airflow volume between the longwall face and the ventilation roadway, as well as isolating the mined-out space from the side of the ventilation drift, blasting the ventilation drift in the areas with the greatest deformation of the working cross-section.

Numerical experiments were carried out to simulate the aerogas situation in the mine. This allowed a fairly thorough analysis of the nature of the air flow distribution in the areas under consideration under the action of various aerodynamic factors.

The initial data for calculating the air distribution in the section ventilation network were:

- $S_{x,i}$ – cross-sectional area of the *i*-th segment of the supported roadway,
- $S_{y,i}$ – cross-sectional area of the longwall face in the *i*-th section;
- $R_{B,i}$ – aerodynamic resistance of the mined-out space in the *i*-th direction (according to formula 11).

Taking into account the practice of managing the aerogas situation at the extraction site, the following were selected as the main factors influencing air distribution:

- cross-section of the supported ventilation working S_x;
- aerodynamic resistance of the mined-out space of the longwall R_B;
- the volume of airflow delivered to the longwall Q_1;
- the volume of supplementary air delivered to the supported ventilation roadway Q_2.

The obtained results of numerical experiments with a change in the air flow rate in the longwall showed that with a change in the air supply to the longwall, the nature of ventilation of the unstable zone changes. Thus, with the Q_1/Q_2 ratio of 0.4, which corresponded to the supply of 6.96 m^3/s of air to the longwall and 17.55 m^3/s to the supported ventilation working, in the junction area between the longwall and the ventilation roadway in a section of 17 m, an overturning of the ventilation stream with the flow rate of 2.57 m^3/s is observed. In this situation, the likelihood of methane accumulation in the working area of the longwall increases.

With increasing the volume of air entering the longwall and a decrease in its supply to the supported ventilation working, the normal ventilation mode of the unstable zone is restored. With the Q_1/Q_2 ratio of 0.67, the air flow rate at the exit from the longwall was 0.38 m3/s at a speed of 0.05 m/s. Further increasing the air supply to the longwall with simultaneous reducing its supply to the supported ventilation working ensures stabilization of the longwall ventilation mode, which meets industrial safety requirements. Thus, with Q_1/Q_2 equal to 1.45 and 2.52, which corresponded to a supply of 14 m^3/s and 17.55 m^3/s into the longwall face, with the supply of 10 m^3/s and 6.96 m^3/s into the supported ventilation workings, the air flow rates at the outlet from the longwall face were 5.04 m^3/s (V=0.63 m/s) and 8.1 m^3/s (V=1.01 m/s), respectively.

A similar trend in changing the value of air consumption and leaks through the collapsed massif was observed in the other longwalls of the Karaganda coal basin mines.

Thus, during the numerical experiment, the values of the (Q_1/Q_2) ratio were obtained. They allow estimating the probability of the explosive concentration of methane formation (Table 4).

Table 4. Criteria for assessing the probability of the formation of an explosive concentration of methane in the stope workings of the extraction area

Q_1/Q_2	Probability of Formation of an Explosive Concentration of Methane	Note
$Q_1/Q_2 \leq 0,4$	High	The ventilation stream is overturned. A zone with air speeds close to zero is formed, which leads to methane accumulations in the working area of the longwall above the permissible standards.
$0,4 < Q_1/Q_2 < 0,67$	Averagee	Gradual restoration of the normal ventilation mode of the unstable longwall zone.
$Q_1/Q_2 \geq 0,67$	Low	Stabilization and establishment of a ventilation mode for the working face of the extraction area that meets industrial safety requirements. Effective ventilation of the working face.

One of the indicators that characterizes the reliability of a mine ventilation scheme is the stability of ventilation. which is assessed by the airflow rate and velocity in the workings of the mining area and in the zone of unstable ventilation of the longwall. The minimum permissible air velocity in the working faces of coal mines is regulated by safety requirements and must be at least 0.5 m/s.

Studies on the stability of ventilation were carried out under the conditions of four operating longwall faces of mines in the Karaganda coal basin as they were being mined. Below. Table 5 presents some actual indicators of air velocity measured in these longwall faces.

According to the data in Table 5, it can be seen that with the ratio of air supplied to the longwall and for auxiliary ventilation ranging from 0.27 to 0.67. the air velocity in the longwalls was between 0.8 m/s and 1.2 m/s. while methane concentrations in certain points of the longwall face were recorded at levels of 1.7–3.1%.

Table 5. Actual indicators of air velocity in the longwalls

Longwall	Q_1 m³/s	Q_2 m³/s	Q_1/Q_2	V m/s
Longwall 1	11.52	24.96	0.46	1.2
	8.81	26.65	0.33	0.87
	7.82	21.20	0.36	0.92
	9.49	15.66	0.60	0.90
	8.16	13.00	0.62	0.80
	11.62	17.40	0.67	1.21
	11.66	17.02	0.69	1.19
	11.57	17.00	0.68	1.30
	11.61	17.20	0.68	1.29
Longwall 2	9.00	27.54	0.33	1.20
	9.15	28.34	0.32	1.22
	7.82	25.76	0.30	0.92
	6.96	21.68	0.32	1.16
	6.23	17.18	0.36	0.89
	11.25	17.20	0.65	1.50
	11.10	17.22	0.64	1.50
	11.25	17.18	0.66	1.60
	11.68	17.23	0.68	1.60
Longwall 3	12.10	28.92	0.42	1.33
	9.20	28.80	0.32	1.00
	10.20	29.60	0.34	1.00
	8.16	31.31	0.27	0.80
	17.10	25.83	0.66	1.61
	17.32	26.13	0.66	1.64
	17.07	25.76	0.66	1.60
	17.05	25.80	0.66	1.69
Longwall 4	9.94	22.24	0.44	1.08
	9.45	24.27	0.38	1.05
	9.57	20.65	0.46	1.04
	14.91	25.41	0.59	1.42
	14.91	18.48	0.80	1.46
	14.94	18.20	0.82	1.66
	14.98	18.47	0.81	1.40
	14.92	18.48	0.80	1.57

With the ratio of air supplied to the longwall and for auxiliary ventilation ranging from 0.67 to 0.82. The air velocity in the longwalls increased and reached values from 1.2 m/s to 1.7 m/s. As a result. the stability of ventilation and the gas environment in the working face improved.

6. RESULTS AND DISCUSSION

In case of high and medium probability of formation of explosive concentration of methane in the stope of the working area, it is recommended to isolate the worked-out space from the ventilation drift. During the study, three isolation options were considered to assess the effect of isolation of the worked-out space: I_{is}= 20 m, I_{is} = 40 m and I_{is}= 60 m. In the calculation scheme, the isolation options were modeled by increasing the resistance of the branches-leakages through the worked-out space (by a thousand times).

A comparative analysis of the modeling results confirms that in all cases a positive effect is achieved, the ventilation of the unstable zone is stabilized, the air flow at the exit from the longwall increases, and the total air leaks through the collapsed massif are reduced. Thus, if at the ratio Q_1/Q_2=0.4 the total air leaks through the collapsed face massif were 9.89 m³/s, then after the isolation of the ventilation workings on a section 20 m long, their total value decreased by 55% and was 4.5 m³/s. Basically, the leaks in the isolation zone decrease, in other directions they are redistributed with a slight increase.

The study examined the role of blasting (increasing the cross-section) of the ventilation workings as one of the possible options for aerodynamic control of air distribution in the ventilation network of the extraction area and reducing the probability of gas contamination of the workings of the working area. Experiments in this part of the study show that blasting worsens the ventilation mode of the face-ventilation drift interface zone and contributes to increasing air leaks through the collapsed massif. The overall negative impact of blasting on area ventilation becomes more pronounced when it is combined with changes in airflow to the face, additional air supply to the supported ventilation roadway, and partial isolation of the mined-out space.

With a combination of all the gas emission control methods, blasting the ventilation workings also did not have a positive effect on the ventilation of the working area.

Thus, blasting of the ventilation workings cannot be recommended as an option for managing the aerogas situation in the working area. However, as blasting is essential for addressing technological challenges, it must be considered in the analysis to ensure accurate data on airflow distribution within the working area.

7. CONCLUSION

To reduce the risk of methane-air mixture explosion in the working areas, theoretical and experimental studies were carried out to assess the probability of formation of explosive concentrations of methane in the stope. During the studies, air leaks from the longwall and its inflows into the supported ventilation workings through the mined-out rock mass were studied. To examine how the aerodynamic properties of the collapsed rock mass and the active workings within the working area influence airflow distribution, a quasi-network model of the longwall's mined-out space was utilized. During the study, a mathematical model was obtained for calculating the value of aerodynamic resistance, taking into account the effect of the length of the workings of the column and the longwall. The resulting model allows determining air leakage through the caved-in rock mass under the conditions of the mines of the Karaganda coal basin.

The authors considered the methods of managing the aerogas situation in the working area by changing the ratio between the airflow directed to the longwall *(Q_1)* and the supplementary airflow supplied to the supported ventilation roadway *(Q_2),* isolating the mined space from the ventilation drift and blasting the ventilation working. In the course of the numerical experiment, the values of the (Q_1/Q_2) ratio were obtained. This allowed estimating the probability of formation of an explosive concentration of methane. The research results allow organizing stable, effective ventilation of the working area and ensuring safety of mining operations in the conditions of the mines of the Karaganda coal basin. Further research will be aimed at reducing the risk of methane-air mixture explosions during mining operations in coal mines. Particular attention will be paid to improving mine ventilation, including the development of more efficient ventilation schemes, determining and reducing the aerodynamic resistance coefficient of mine workings in the Karaganda coal basin. The research will cover issues of ventilation management to maintain optimal aerodynamic regimes, which will reduce methane concentration in mine workings and enhance the safety of miners.

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