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Research on Optimization of Drill Hole Gas Pre-Drainage Process in Low Permeability Outburst Coal Seam

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To study the drill hole gas pre-drainage technology in low-permeability outburst coal seam, this paper selects the 1031 working face of the first coal seam in the Songhe Coal Mine as the research and test area, adopts Ansys numerical simulation software for the simulated calculation of different drill hole spacing and different drainage time, different drainage effects are obtained, and the test results show that: when the hole spacing is 2.0 m, the drainage result is the best, the residual gas content in the coal seam of the working face drops to less than 5.92 m^3/t, and the residual gas pressure drops to less than 0.58 MPa, the gas drainage rate is increased to 61.48%, and the outburst risk is also reduced. This optimization method has a guiding significance for the gas drainage and utilization of other low-permeability outburst coal seams.

1. Introduction

According to statistics, over 95% of China's high gas and outburst mines are low-permeability coal seams, directly leading to difficult gas drainage and extremely low utilization rates (Gong et al., 2012). There are two main technical difficulties: first, low permeability makes the gas difficult to be extracted; second, since the gas concentration is extremely low, it cannot be used effectively. Relevant studies have shown that, for the low-permeability outburst coal seams, if we pre-drain the gas in the coal seams according to current arrangement of conventional hole spacing, the results would be long drainage time and low efficiency, which fails to achieve good drainage results, let along reducing the danger of outburst coal seams. Therefore, by arranging the drill holes using the optimized method we can increase the permeability of the coal seams, realize high-efficiency gas drainage (Fang et al., 2015), and reduce risks of the outburst coal seams. The Songhe Coal Mine is an outburst mine of coal and gas, when the No. 3 coal seam was mined for the first time, during the excavation process, there occurred a few accidents, it is a low low-permeability outburst coal seam. The author selects the 1031 coal seam working face of No. 3 coal seam in Songhe Coal Mine as the test object to study the influence of hole spacing on the drainage effect and the outburst risk, which is of great significance to eliminate the coal and gas outburst risks of No. 3 coal seam.

2. Working face overview

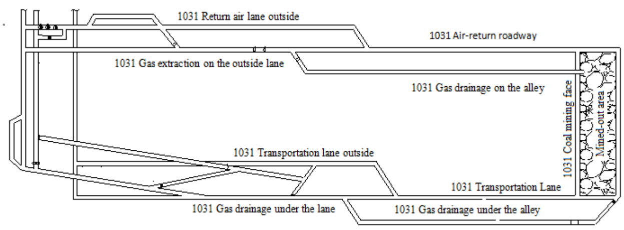


Figure 1: Schematic view of the 1031 working face

The 1031 working face is located in the first section of the central mining district and is located in No. 3 coal seam, as shown in Figure 1. The No. 3 coal seam has a coal thickness of 1.5m to 3.5m, an average coal thickness of 2.54m, and a coal seam inclination of 24° to 31° with an average of 27°. The elevation of 1031 working face haulage gate is +1490m, the elevation of air-return gate is +1594m, the working face strike length is 2095m, adopt the longwall retreating mining on the strike, and use full-seam mining carving method for the roof control, combining with mechanized coal mining process. The coal seam gas parameters are shown in Table 1.

Table 1: Gas parameters in No. 3 coal seam

Coal seam	Pressure measuring point	Buried depth (m)	Gas pressure (MPa)	Gas content (m^3/t)	Permeability coefficient ($m^2/MPa^2.d$)	Drill hole flow attenuation coefficient (d^{-1})
3#	In the 1031 haulage roadway 50 meters away from the 1031 connection roadway	520	2.23	11.67	1.7589	0.043

3. Existing control measures

3.1 Pre-drainage gas drill hole arrangement

According to the on-site survey data, the pre-drainage drill hole arrangement parameters for the 3# coal seam are shown in Table 2 in detail.

Table 2: Pre-drainage drill hole arrangement parameters

Construction site	Hole number	Position /°	Inclination /°	Drill hole diameter /mm	Hole depth /m	Average hole depth /m	Hole spacing /m
Upside the 1031 air-return inclined roadway, 185 meters away down the grade change point	1-10	62~72	3~10	75	78~96	85.6	1.4,1.5
Downside the 1031 air-return inclined roadway, at the opening of the manhole	1-8	11~25	15~21	75	84~107	98.1	1.6,1.8
Downside the 1031 air-return inclined roadway, inside the level workings	1-12	17~28	8~16	75	52~87	71	1.3,1.7
120 meters away from the junction of 1031 air-return roadway and the outer segment of 1031 air-return roadway	1-11	43~55	0~9	75	65~83	76.4	1.7,1.8

3.2 Drainage effect analysis

According to the above parameters, we can see that the drill holes are arranged at different positions in the air-return roadway, drill holes down the seam to extract the gas in the 1031 working face air-return roadway, the diameter of the drill holes are all Φ75 mm, the drill holes are arranged along the strike of the coal seam or along the inclined direction of the coal seam, the hole spacing is between 1.3m and 1.8m.

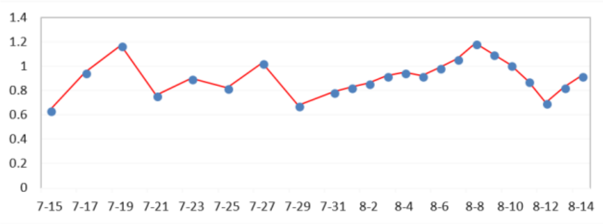


Figure 2: Gas concentration in 1031 air-return roadway in 2017 (%)

In the air-return roadway16 meters away from the open-off cut, a gas concentration monitoring system is set up to monitor the gas concentration of the 1031 air-return roadway. The maximum value of the monitoring system is taken as the daily value to draw a gas concentration map as shown in Figure 2, the results show that: the gas concentration in the air-return roadway is in an over-limit state for a long time, and the amount of gas emission increases during the excavation, when the excavation is stopped, the emission amount is relatively smaller, which is very unfavorable for the safe mining of the mining face. After calculation, the gas drainage rate is only 26.37%, so the optimization of the existing solution is imperative.

4. Outburst prevention technology optimization

The drainage effect of this working face is not ideal, its basic reason is that the spacing of drill holes is not reasonable, which directly leads to the failure of achieving the purpose of eliminating the outburst. The gas concentration in the air-return roadway exceeds the limit for a long time. In this regard, numerical simulation studies have been conducted to optimize the spacing of drill holes in the coal seam (Fang et al., 2012).

4.1 Construction of Ansys model

In order to achieve efficient drainage and reduce the risk of coal and gas outburst, the author selects 1031 working face of 3# coal seam in Songhe Coal Mine as a numerical simulation object, and uses Ansys finite element software to establish a digital model (Fang et al., 2012). The different gas drainage effects are simulated and calculated for different hole spacings and gas drainage times, and comparative analysis (Sun, 1993) is conducted to guide the drill hole arrangement and drainage time setting of the 1031 working face.

4.1.1 Basic assumptions of coal seam gas movement

The movement of gas in coal seams is very complicated. In order to facilitate the study of the main laws of gas movement, necessary and rational simplification of the actual movement environment must be carried out:
(1) Assume that the characteristics of the coal seam in the test area are isotropic and continuous with a horizontal distribution, and the faults, folds and cracks in the coal body are ignored;
(2) Only some of the drill holes in the intercepted section are studied. Both sides can be freely extended and present as free surfaces.
After simplifying the model, the gas flow equation is derived from the following assumptions (Sun, 1993):
(1) Assuming that the top and bottom roof of the coal seam are insulating layers, the gas cannot pass through the insulating layer and diffuse freely;
(2) The gas is assumed to be ideal gas, which conforms to the ideal gas state equation (Clapeyron equation):

$$\rho = \frac{P}{RT} \tag{1}$$

Where, ρ is the density of the gas, RT is the gas constant, P is the pore pressure, the same below;
(3) Gas in the coal seam exists in two forms: free state and adsorbed state, and its content conforms to the Langmuir Equation, i.e.:

$$c = n\rho + \frac{a_1\rho}{1+bP} \tag{2}$$

Where, the gas analysis in the coal seam is completed in an instant; (n is porosity, a_1 and *b* are the adsorption constants);
(4) The seepage of gas in the coal seam is regarded as non-linear seepage, and its value is a function of pore pressure and volume stress acting on the coal body (Han, 2007;):

$$q_i = k_i p_i \tag{3}$$

(5) The flow of gas in the coal seam obeys Darcy's law (Liu, 2002);

$$V = \frac{K(h_2 - h_1)}{L} \tag{4}$$

Where, V represents the flow rate of seepage, *K* is the coefficient of permeability, $(h_2-h_1)/L$ represents the seepage flow gradient of the porous media.

4.1.2 Determination of model parameters

According to coal seam condition and mining conditions of 1031 working face in Songhe Coal Mine, for the

height of the model, takes 2.5m as the thickness of the coal seam and 25m as the width, the k-ε model is selected as the gas turbulent calculation model, the movement of the gas in the coal seam conforms to the porous media model (Zhang, 2001), the parameters of 1031 working face is shown in Table 3, the hole spacing takes 1.6m, 2.0m, and 2.4m, respectively, the boundary conditions are set as follows:
(1) The top and bottom roof rock layers are impermeable barriers, and the left and right boundaries are free interfaces (Fang et al., 2015); (2) Drill hole drainage pressure takes: -17KPa.

Table 3: 1031 working face parameters

Coal seam original gas pressure(MPa)	Coal seam thickness(m)	Drill hole diameter(mm)	Drainage pressure(KPa)	Coal seam permeability coefficient($m^2/MPa^2 \cdot d$)
2.23	2.5	75	-17	1.7589

The Ansys intelligent mesh is used to divide the quadrilateral units and the triangular units (drill hole local refinement area), which has a total of 8056 units, 10379 nodes, and the mesh model is shown in Figure 3.

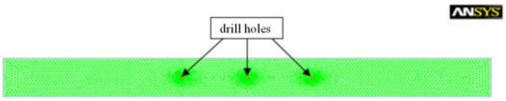


Figure 3: Schematic diagram of the model

4.2 Results analysis

Through simulated calculation of the above model, the following results are obtained:
(1) The gas pressure distribution is shown in Figures 4 and 5 when the hole spacing is 1.6m.

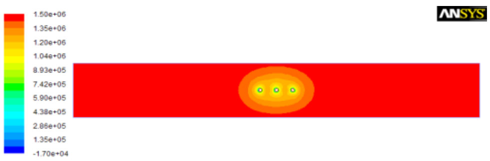


Figure 4: Gas pressure in surrounding coal seams after 30-days drainage

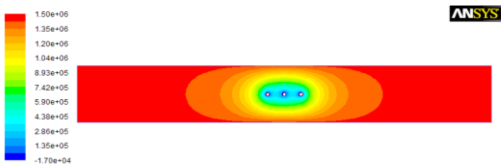


Figure 5: Gas pressure in surrounding coal seams after 150-days drainage

(2) The gas pressure distribution is shown in Figures 6 and 7 when the hole spacing is 2.0m.

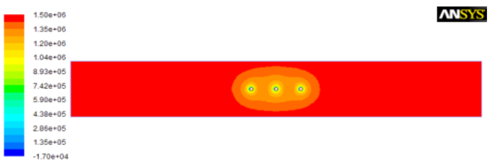


Figure 6: Gas pressure in surrounding coal seams after 30-days drainage

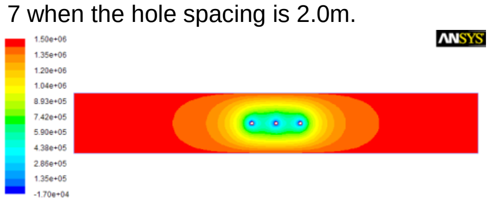


Figure 7: Gas pressure in surrounding coal seams after 160-days drainage

(3) The gas pressure distribution is shown in Figures 8 and 9 when the hole spacing is 2.4m.

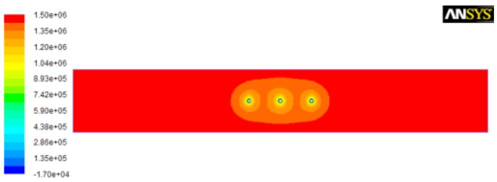


Figure 8: Gas pressure in surrounding coal seams after 30-days drainage

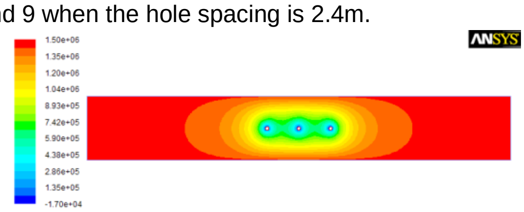


Figure 9: Gas pressure in surrounding coal seams after 185-days drainage

Figures 4, 6 and 8 shows that when the drainage time reaches 30 days, the adjacent two drill holes have influenced each other's gas pressure around them, and the smaller the spacing, the more obvious the influence, but this does not affect the top and bottom roof of the coal seam. Due to the fact that the average thickness of the coal seam in the test working face is only 2.5m, it also limits the arrangement of the drill holes, only single-row of holes can be arranged, therefore, for the short-term drainage, gas of the top and bottom roof of the coal seam has become a major factor restricting the drainage effect.

Figures 5, 7 and 9 shows that, when the hole spacing is 1.6m, 2.0m, and 2.4m respectively, the time required for the gas drainage radius to reach the top and bottom roof is 150 days, 160 days, and 185 days, respectively. After the drainage radius reaches the top and bottom roof, continuing the drainage has little effect on reducing the gas pressure around the drill holes of the coal seam. Compared with the time required for the three drill-hole spacings, considering only the drainage progress factors, the shorter the drainage time, the better, that is, the shorter the construction period, the higher the efficiency. The drill hole spacing in this test area is better chosen to be 1.6m and 2.0m.

Comparing Figures 5 and 7, the time required for the gas to reach the top and bottom roof is 150 days and 160 days respectively, the difference is small. However, the impact range of the drill holes with spacing of 2.0m on the gas pressure along the coal seam is larger than that of the drill holes with spacing of 1.6m. From this point of view, it is more appropriate to choose drill holes with larger spacings. In addition, from the perspective of coal mine economics, for stoping of the same spacing, the larger the spacing, the less drill holes needed, and more drilling cost is saved, and the more feasible economically. In addition, according to the current coal seam condition and mining conditions in the 1031 working face of the Songhe Coal Mine, it is more appropriate to select drill holes with a spacing of 2.0m in this test area.

4.3 Drainage effect inspection

According to the actual situation of the working face, 8 drill holes are arranged on the 1031 coal mining face to inspect the drainage effect (Xie et al., 2013). The detailed parameters of the drill holes are shown in Table 4. 3 sets of samples are taken from each drill hole, after on-site sampling and laboratory analysis, the residual gas content and residual gas pressure in coal seams in different drill holes are measured, the coal seam samples are averaged. The test results are shown in Table 5.

Table 4: Drill hole parameters of 1031 working face inspection

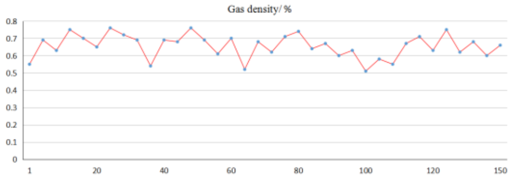
Drilling number	Coal seam	Orifice location	Drilling parameters		
			Position/°	Inclination /°	Hole depth /m
3-1	3	K0+60m in the 1031 return wind alley	47	-7	65
3-2	3	K0+175m in the 1031 return wind alley	48	-6	69
3-3	3	K0+290m in the 1031 return wind alley	43	-9	60
3-4	3	K0+418m in the 1031 return wind alley	45	-5	72
3-5	3	+335m in the 1031 return wind alley	223	16	60
3-6	3	+126m in the 1031 return wind alley	221	15	70
3-7	3	+79m in the 1031 return wind alley	225	15	73
3-8	3	+62m in the 1031 return wind alley	228	15	70

Table 5: Residual gas content and gas pressure in the coal seam

No.	Test site	The residual gas content(m^3/t)	The residual gas pressure(MPa)
1	K0+60m in the 1031 return wind alley	5.27	0.58
2	K0+175m in the 1031 return wind alley	5.36	0.48
3	K0+290m in the 1031 return wind alley	5.40	0.50
4	K0+418m in the 1031 return wind alley	5.82	0.28
5	+335m in the 1031 return wind alley	5.92	0.51
6	+126m in the 1031 return wind alley	5.64	0.37
7	+79m in the 1031 return wind alley	5.55	0.33
8	+62m in the 1031 return wind alley	5.73	0.47

The tables show that during the stoping, the residual gas content and gas pressure are monitored. The residual gas content in the coal seam of the working face was reduced to less than 5.92 m^3/t, and the residual gas pressure was reduced to less than 0.58 MPa, both lower than limits of 8.0m^3/t and 0.74MPa in the *Basic Indicators for Coal Mine Gas Drainage*. After calculation we get that, the gas drainage rate of 1031 working face increased from 25.46% to 61.48%, which is much higher than the standard drainage rate of 30% specified in the *Coal Mine Safety Regulations* and the *Basic Indicators for Coal Mine Gas Drainage*.

During normal excavation of the working face, perform real-time monitoring of the gas concentration of the 1031 working face upper corner and air-return roadway, the gas concentration is shown in Figure 10 and 11, the data show that the gas concentration in the upper corner is generally higher than the gas concentration in the air-return roadway, the upper corner gas is more likely to become a hidden safety risk, it has become the focus of monitoring and prevention, but no abnormalities or overruns has occurred. The gas drainage effect and the goal of eliminating the overruns have been achieved, meanwhile the key areas for gas prevention and controlling during normal stoping period have also been shown.



Distance from the open-off cut when stoping (m)

Figure 10: Gas concentration in the upper corner of 1031 working face

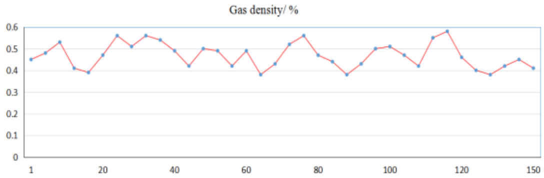


Figure 11: Gas concentration in 1031 air-return roadway

5. Conclusion

(1) When the drainage time is 30 days, the adjacent two drill holes with the hole spacings of 1.6m, 2.0m and 2.4m have already affected each other's surrounding gas pressure. The smaller the spacing, the more obvious the influence, but it does not affect the top and bottom roof of the coal seam, so the gas in the top and bottom roof of the coal seam has become the main factor restricting the drainage effect.

(2) When the hole spacing is 1.6m, 2.0m, and 2.4m, the time required for the gas drainage radius to reach the top and bottom roof is 150 days, 160 days, and 185 days, respectively. The shorter the drainage time, the shorter the construction period and the higher the efficiency. The drill hole spacing in this test area is better chosen to be 1.6m and 2.0m. However, the impact range of the drill holes with spacing of 2.0m on the gas pressure along the coal seam is larger than that of the drill holes with spacing of 1.6m, it is more appropriate to choose drill holes with larger spacings, the larger the spacing, the less drill holes needed, and the more drilling cost is saved, and the more feasible economically, so it is more appropriate to select drill holes with a spacing of 2.0m in this test area.

(3) It has been proved that compared with the previous drill hole spacing of 1.2m, the drill hole spacing was adjusted to 2m for the drainage. The residual gas content in the coal seam of the working face was reduced to less than 5.92 m^3/t, and the residual gas pressure was reduced to less than 0.58MPa. The gas drainage rate increased by 59.89%, indicating that the optimized scheme is economically reasonable, and has great practical significance for the gas drainage and utilization of other low permeability gas outburst coal seams.

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