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Research on the Reasonable Spacing of Holes in Gas Drainage along Coal Seams in Consideration of the Superimposed Effect of Drainage

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Abstract

o determine the reasonable spacing of holes for gas drainage, an experimental study was conducted on the effective influence radius of drilling hole through the pressure drop method based on the actual condition of coal seams. The coal seam that contains gas is regarded as elastic-plastic dual media. The governing equation of gas transport is established by analyzing the different flow forms of gas in the pore and fissure systems as well as by considering the mass exchange capacity in the pore-fissure system. The equation is embedded into COMSOL Multiphysics (COMSOL) software to simulate the gas drainage effect further by drilling along a coal seam under a 3-D space. Upon confirming the effective radius and drainage influence radius of a single pore, the holes for gas drainage can be reasonably spaced along the coal seam by analyzing the change features of coal permeability around the borehole and the functional mechanism of the superimposed effect of drainage, namely, $2r \le L \le R$. The study results can reliably guide practical gas drainage theoretically and can also effectively lower the cost of gas drainage as well as ensure the safe production in mines.

Keywords: Coupling model, Gas drainage borehole, Numerical simulation, Spacing of holes, Superimposed effect

1. Introduction

Production in underground coal mines has entered a stage of deep mining as the demand for coal increases. Dynamic coal rock disasters frequently occur because of elevated ground stress and gas content; such disasters have severely affected the safe production of coal in underground mines [1, 2]. The pre-drainage of gas in the coal seam by drilling along the coal seam is a key technical measure to reduce and eliminate the risk of outburst rapidly. Hence, the effective extraction of gas in the coal seam is significant to control gas hazards.

Previous research on the reasonable spacing of holes to drain the gas along coal seams mainly determines the drilling radius for drainage through field measurement and numerical simulation. Most studies regard the results of these methods as the basis for gas drainage design. The obtained radius of drainage is highly erroneous because of the limited number of investigated boreholes and the possible manual operation error incurred during operation [3, 4]. Wang et al. simplified the field of gas flow around the borehole into a 2-D plane model to numerically simulate gas drainage. Nonetheless, a few recent works focus on the gas flow under a 3-D space after draining the coal-rock mass; this approach can reflect the gas flow effectively [5-11]. Previous research on gas drainage along coal seams generally consider a single borehole to be the subject of study, whereas the superimposed effect of drainage is observed between adjacent boreholes in practice. At present, the mechanism of the superimposed effect and how to confirm the reasonable spacing of holes are rarely studied [12-16].

The present research regards the coal rock containing gas as dual media based on the gas flow and elastic-plastic mechanics. The elastic-plastic coupling model of coal rock containing gas is established as well, and this model is then incorporated into the COMSOL multi-physics field in partial differential form to simulate the effect of gas drainage along the coal seam under a 3-D space. The migration law of gas in a coal seam and the mechanism of the superimposed effect of drainage are also studied to confirm the reasonable spacing of boreholes and to reliably guide gas drainage along coal seams theoretically.

2. Field Test Analysis

2.1 Field Test

To determine the reasonable spacing of boreholes for gas drainage along coal seams, the pressure drop method was adopted in this study to conduct a field test on the drainage radius. The field test was conducted on the heading face without an existing geological structure. Furthermore, outburst prevention measures have not been implemented in this area. The test site must be proximal to the working face at which the coal seam is stable. The boreholes were arranged according to the designed construction program, which is shown in Fig. 1. First, four investigated holes, namely, 1#, 2#, 3#, and 4#, were drilled at the same level in the vertical direction of the coal wall. Gas pressure was tested through the initiative manometric method. Once the pressure on the investigated holes was stabilized, hole 5#

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was drilled at the reserved position. After construction, the holes were sealed immediately and connected to a pipe network to drain gas. The total drilling diameter was 94 mm, with a hole depth of 50 m. The negative pressure for drainage was 13 KPa. Holes 1#, 2#, 3#, and 4# were 0.5, 1.0, 1.5, and 2.0 m away from hole 5#, respectively. During the test, the change in the gas pressure of each investigated hole was detected continuously for 120 days.

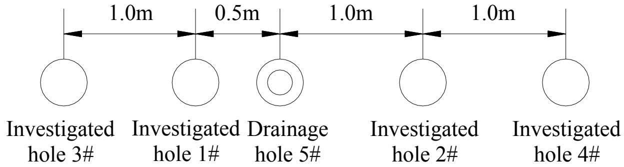


Fig. 1. Arrangement of boreholes

2.2 Analysis of Results

The basis for determining the effective radius and the influence radius for drainage is that the gas drainage rate reaches 30% and 10%, respectively, and that the corresponding gas pressure decreases by 51% and 19%, respectively[17]. Fig. 2 depicts the changing trend of the gas pressure within the investigated hole during the test.

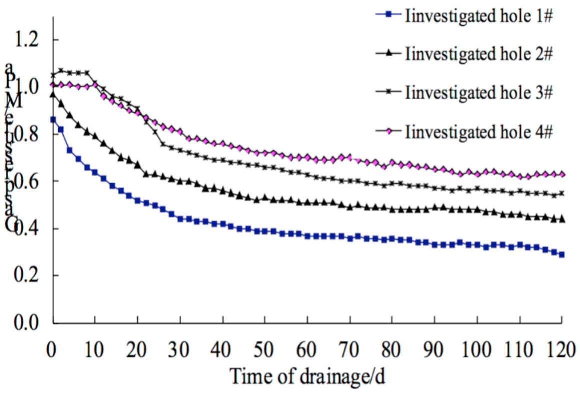


Fig. 2. Curve of gas pressure variation

The gas pressure in hole 1# decreased constantly and evidently from 0.86 MPa to 0.29 MPa. This result indicates that the position of this hole was significantly affected, that the fracture within this scope was well developed, and that the permeability of the coal body was high. Thus, gas pressure declined sharply. The gas pressures in hole 1# on days 6 and 38 were 0.69 and 0.42 MPa, respectively; that is, the corresponding gas pressure decreased by more than 19% and 51%. Thus, the radius of influence and the effective radius for drilling on these days were confirmed to be 0.5 m.

Unlike in hole 1#, the gas pressure in hole 2# decreased slowly during the test from 0.97 MPa to 0.44 MPa, thus indicating that the position of this hole was weakly affected by hole 5# than hole 1# was. The gas pressures on days 11 and 102 are 0.78 and 0.47 MPa, respectively; in other words, the corresponding gas pressures decreased to 19% and 51%. The radius of influence and the effective radius for drilling on these days were therefore confirmed to be 1.0 m.

The initial gas pressure in holes 3# and 4# was high on day 30. Subsequently, the gas pressure dropped from 1.05 and 1.04 MPa to 0.55 and 0.63 MPa, respectively, thereby suggesting that the positions of these holes were weakly affected by hole 5#. This result also demonstrates that the void and fracture of the coal body closed under pressure and that permeability decreased after stress concentration. According to the actual measurement, the gas pressures of holes 3# and 4# were 0.85 and 0.81 MPa on days 22 and 30, respectively. Hence, the corresponding gas pressure decreases by 19%. The influence radii of drainage on these days were confirmed to be 1.5 and 2.0 m, respectively; nonetheless, the gas pressure in these holes did not decrease below 51% during the test.

2.3 Features of the Permeability Change in the Coal Body around a Borehole

A regional change law in the permeability of the coal body around a borehole was presented according to the results of the actual measurement with the pressure drop method. The analysis results indicate that the construction of boreholes changed the stress-state of original coal mass and that a certain pressure-relief zone formed around the borehole. In this zone, the pore and the fracture were connected and the coal permeability coefficient increased. Thus, an open zone of seepage was formed and gas pressure was significantly reduced. With an increase in the distance from the borehole, the stress on the coal body gradually increased and was concentrated. Under load constraint, the pore and the fracture within the coal body were subject to pressure; moreover, the space for seepage shrinked, and permeability declined sharply.

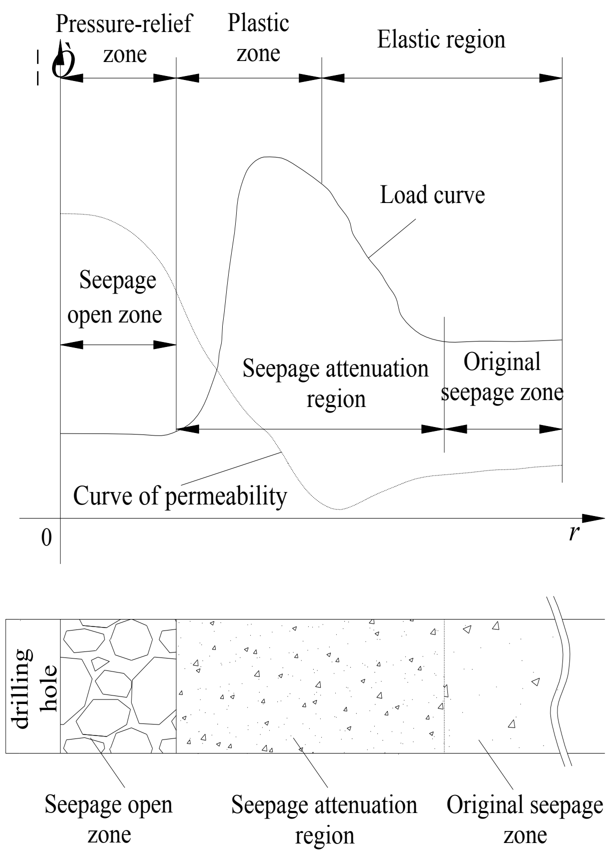


Fig. 3. Variety of permeability around the borehole

Thus, the attenuation area of seepage was generated and the gas pressure within this scope slowly declined. Deep in the coal seam, stress gradually recovered to the level of that on the primary rock. The coal body almost retained its original permeability, which belonged to the initial seepage zone. As the analysis results described above, three zones can be determined from the wall of the hole to coal seam depth, based on the entire stress-strain process of the coal body around the borehole and the gas flow. These zones include the open seepage zone, attenuation area of seepage, and original seepage zone, as depicted in Fig.3.

3. Elastic-plastic Fluid-structure Coupling Model

To analyze the migration law of gas in the coal body around a borehole, this research focused on the theoretical derivation of a fluid-solid coupling model around the borehole.

3.1 Governing Equation for Coal Rock Deformation

In accordance with the elastic-plastic features of coal rock that contains gas and the Terzaghi principle of effective stress, the balance equation for the coal body is expressed by the formula below:

$$\sigma'_{ij,j} + (\alpha p \delta_{ij})_{,j} + F_i = 0 \tag{1}$$

where $\sigma'_{ij,j}$ is effective stress, MPa; α is Biot's coefficient; δ_{ij} is the Kronecker symbol; and F_i is the tensor of body force, N/m³.

The geometric equation for the deformation of coal rock that contains gas can be expressed as follows:

$$\varepsilon_{i,j} = \frac{1}{2}(u_{i,j} + u_{j,i}) \tag{2}$$

where u_i is the displacement component, m.

The constitutive equation of coal rock deformation adopts elastic-plastic constitutive equations. The increment form of such equations is

$$\mathrm{d}\sigma'_{ij} = \left[\lambda\delta_{ij}\delta_{kl} + G\left(\delta_{ik}\delta_{jl} + \delta_{il}\delta_{jk}\right)\right]\left(\mathrm{d}\varepsilon_{kl} - \mathrm{d}\varepsilon^{\mathrm{p}}_{kl}\right) \tag{3}$$

where λ is the Lame constant.

According to the plasticity-associated flow rule, the strain increment of plasticity is given by the potential function of plasticity g. Given strain-hardening material with stable plasticity, g generally takes the same form as the following yield function f. That is, when $g = f$, the plasticity flow rule can be expressed as follows:

$$\mathrm{d}\varepsilon^{\mathrm{p}}_{ij} = \lambda\frac{\partial g}{\partial \sigma'_{ij}} = \lambda\frac{\partial f}{\partial \sigma'_{ij}} \tag{4}$$

The yield criterion of coal rock deformation adopts the Drucker-Prager criterion, which is particularly suitable for the mechanical property of rock and soil.

$$f\ (I'_1, \sqrt{J'_2}) = \alpha' I'_1 + \sqrt{J'_2} - k' = 0 \tag{5}$$

where I'_1 is the first invariant of the effective stress tensor; J'_2 is the second invariant of the effective stress deviator; and α' and k' are the experimental constants that are related to frictional angle φ and cohesive force c', respectively. The latter two variables are determined as follows:

$$\alpha' = \frac{2\sin\phi}{\sqrt{3}(3-\sin\phi)}\ ,\ k' = \frac{6c'\cos\phi}{\sqrt{3}(3-\sin\phi)}$$

In accordance with the definition of the isotropic yield criterion[18]:

$$\sigma_p = \sigma_p^0 + \frac{\left(\sigma_p^{\mathrm{m}} - \sigma_p^0\right)\varepsilon^{\mathrm{ep}}}{A + \varepsilon^{\mathrm{ep}}} \tag{6}$$

where σ^0_{p} is the initial yield stress, MPa; $\sigma^{\mathrm{m}}_{\mathrm{p}}$ is the maximum value of the hardening function, MPa; $\varepsilon^{\mathrm{ep}}$ is equivalent plastic strain; and A is the constant controlling the plasticity enhancement rate.

The *Von Mise* flow rule is adopted in this study and is expressed as

$$g = \sigma'_i \tag{7}$$

where σ'_i is the strength of effective stress, MPa.

3.2 Governing Equation of Gas Migration

Gas migration in a coal seam is a complicated process. Coal rock that contains gas is a type of dual media that exhibits pores and fractures. On the basis of the occurrence state of gas in the coal body, gas migration can be classified into gas in free and adsorbed states. Upon disrupting the original adsorption equilibrium state of a coal seam through extraction, the gas at free state within the fracture system flows to the extraction space through seepage. The speed of gas seepage is significantly greater than the diffusion velocity. Thus, pressure varies between the fracture and pore systems. The gas at the adsorbed state in the latter first diffuses to the surface of the coal particle, passes through the adsorption film, and is then desorbed into the fracture system. Mass exchange capacity q changes between the two systems.

In the coal body, gas in the free state is mainly detected in the fracture. The seepage under the pressure gradient conforms to Darcy's law. Based on the mass conservation equation at the gaseous phase, the migration equation of the fracture can be obtained as follows:

$$p\frac{\partial\varphi}{\partial t} + \varphi\frac{\partial p}{\partial t} + \nabla\left[\frac{k}{2\mu}\nabla p^2\right] = RTq \tag{8}$$

where p is the pressure of gas under a free state, Pa; φ is the porosity of the coal body; ∇ is the Hamiltonian operator; k is the permeability of the coal rock fracture, m^2; μ is the dynamic viscosity of gas, Pa·s; R is the Platts gas constant of the coal seam gas, 8.3145J/(kg·K); T is the temperature of the coal body, K; and q is the mass exchange capacity, kg/(m^3·s).

The large amount of pores in the coal body generates the area that contains gas at the adsorbed state. After the original adsorption equilibrium state is disrupted, the gas at the adsorbed state is first desorbed from the coal particle, diffuses to the coal particle surface, passes through the coal particle film, and then enters into large pores and fractures. According to Fick's law and the law of mass conservation at the gaseous phase, we can obtain the migration equation of pores.

$$\frac{\partial C}{\partial t} = D\nabla^2 C - q \tag{9}$$

where C is the mass concentration of the gas at the adsorbed state, kg/m^3; D is diffusion coefficient, m²/s.

Within the pore system, the content of gas in the adsorbed state satisfies the Langmuir isotherm adsorption equation. The content of gas in the adsorbed state per unit volume is determined as follows:

$$C = \frac{abcp_n p}{(1+bp)RT} \tag{10}$$

where a is the limit of the adsorbing capacity of a unit mass of coal, m^3/kg; b is the adsorption constant of coal, MPa^{-1}; p_n is the gas pressure under standard condition, 101325 Pa; and c is the combustible mass in a unit volume of coal, kg/m^3.

By substituting Eq.(10) into Eq.(9) and omitting the trace at the second order, so the migration equation of the gas in the pore system expressed by gas pressure can be derived as follows:

$$\frac{\partial p}{\partial t} = D\nabla^2 p - \frac{(1+bp)^2 RT}{abcp_n} q \tag{11}$$

According to the principle of the dual media structure of the coal containing gas, the flow velocities of gas are different within the pore and fracture systems, which are attributed to the pressure difference, and mass exchange capacity q exits between these systems. According to Zhang et al. [19], mass exchange capacity q can be expressed as

$$q = \frac{3}{r_0^3}\tau(C_0 - C_p) \tag{12}$$

where,

$$C_0 = \frac{abc\rho_n p_0}{1+bp_0}, C_p = \frac{abc\rho_n p}{1+bp}$$

where r_0 is the limit of the coal particle radius, m; τ is the film coefficient expressed in a solid phase, m/s; C_0 is the initial mass concentration of the gas at the adsorbed state, kg/m^3; C_p is the gas at the adsorbed state that balances with p, kg/m^3; and p_0 is the gas pressure at free state, Pa.

The migration equation of the gas of the pore-fracture dual media can be obtained by uniting Eqs.(8), (11), and (12) simultaneously as follows:

$$\begin{aligned}&\varphi D\nabla^2 p + p\frac{\partial \varphi}{\partial t} + \nabla\left[\frac{k}{2\mu}\left(1+\frac{m}{p}\right)\nabla p^2\right] = \\ &\frac{3\tau abcp_n}{r_0}\frac{p_0 - p}{(1+bp_0)(1+bp)}\left[1+\frac{\varphi(1+bp)^2}{abcp_n}\right]\end{aligned} \tag{13}$$

3.3 Governing Equation of Porosity and Permeability

During gas drainage, the stress state, pore pressure, and damage type of the coal body affect the variation of porosity in coal body; as a result, permeability is diversified. The porosity equation, which considers the volumetric strain of coal rock, gas pressure, and adsorption swelling effect, can be expressed as follows[20]:

$$\varphi = 1 - \frac{1-\varphi_0}{1+\varepsilon_v}\left[1 - K_Y(p-p_0) + \frac{2a\rho_v RTK_Y}{9V_m(1-\varphi_0)}\ln\frac{1+bp}{1+bp_0}\right] \tag{14}$$

Where φ_0 is the initial porosity of the coal body; ε_v is the volumetric strain of coal rock; K_Y is the coefficient of volume compressibility; ρ_v is the apparent density of the coal body, t/m^3; and V_m is the air molar volume, m^3/mol.

According to Jing et al. [21], an exponential relationship exists between porosity and permeability; this relationship is expressed as follows:

$$k = k_0 \exp[22.2(\varphi/\varphi_0 - 1)] \tag{15}$$

Where k_0 is the initial permeability of the coal body.

As stated previously, the elastic-plastic coupling model of the coal rock containing gas comprises the governing equations of coal-rock mass deformation, the gas application equation, and the governing equation of porosity and permeability.

3. 4 D Numerical Simulation of Gas Drainage

In this study, the COMSOL multi-physics coupling analysis software is used to investigate the aforementioned model in the form of $y = f(x)$ into the partial differential equation for the system and the block of solid mechanics to realize the secondary development of the modification process. Thus, the user-defined control process can be implemented. Furthermore, a solution is got quickly through the finite element method.

4.1 Establishment of the Geometric Model

A 3-D geometric model is established to represent the practical stress distribution of the coal body around the borehole. The model presents a coal seam that measures 100 m × 60 m × 5 m (length × width × height) with a gravity load on top that simulates of the overlying rock. The baseboard at the bottom of the coal seam is a fixed boundary, and the constraint mode around the coal seam is supported by a roller. The model itself bears a self-weight load. The borehole for drainage within the coal seam has a diameter of 94 mm, and the negative pressure for drainage is 13 KPa. A geometric diagram of the model is illustrated in Fig. 4, and the main design parameters of the model are presented in Table 1.

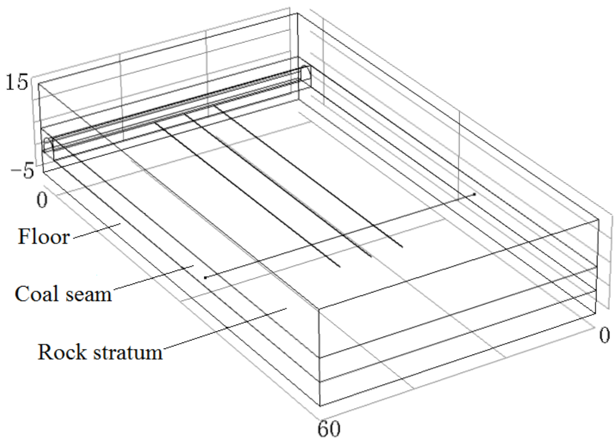


Fig. 4. Geometric model

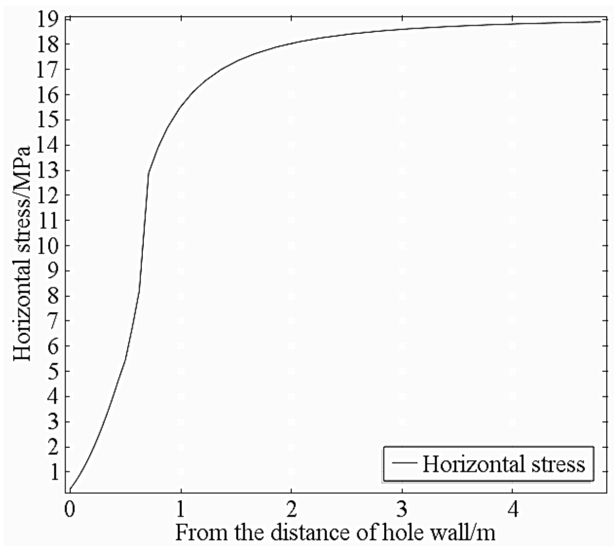
Table 1. Main parameters of the model

Parameters	Values
Initial gas pressure p	1.0 MPa
Initial porosity of the coal rock containing gas φ_0	0.0016
Initial permeability of the coal rock containing gas k_0	1.2×10^{-19}m^2
Apparent density of the coal rock containing gas ρ_v	1.46 t·m^{-3}
Dynamic viscosity coefficient of gas μ	1.08×10^{-5}Pa·s
Poisson's ratio of the coal rock containing gas υ	0.31
Adsorption constant a	33.771 m^3/t
Adsorption constant b	0.646 MPa^{-1}
Ash content of the coal rock containing gas A	0.1441
Water content of the coal rock containing gas M	0.01606
Universal gas constant R	8.3145J/(kg·K)
Molar volume of gas Vm	0.0224 m^3/mol

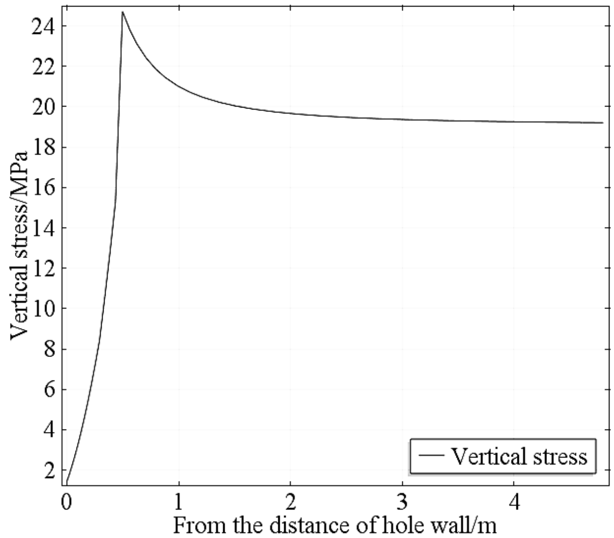
4.2 Analysis of the Simulation Results of a Single Borehole

4.2.1 Analysis of the Stress on the Coal Body around the Borehole

Fig. 5 indicates the curve of stress variation around the borehole. Horizontal stress increases from 0 MPa on the wall of the hole to the in-situ rock stress of 19.2 MPa. Meanwhile, vertical stress increases with the distance from the wall of the hole and peaks at 24.7 MPa at 0.5 m from the hole wall. The vertical stress in a deep coal seam gradually reduces to the initial stress of 19.2 MPa. The simulation results indicate that the stress loaded on the coal body from 0.00-0.45 m around the borehole is less than the original stress, thereby the area is in a state of pressure relief. The stress in an area 0.45-5.00 m from the wall of the hole is greater than original rock stress, which is located within the stress concentration zone. In the areas over 5 m from the wall of the hole, the coal body almost maintains the original rock stress state, which is located in the elastic region.



(a)Horizontal stress



(b) Vertical stress

Fig. 5. Curve of stress change around the borehole

4.2.2 Rule of Gas Pressure Change around a Borehole

Fig. 6 presents the gas pressure of a single borehole. The gas pressure of a coal seam is lower near the wall of the hole; within an area 0.5 m away from the wall of the hole, gas pressure decreases quickly, gradually increases with distance from the borehole, and recovers to the primary gas pressure at coal seam depth (1.0 MPa). The simulation results of the stress around the borehole indicate that gas pressure decreases rapidly and the permeability is relatively higher in the pressure relief area. In the plastic zone, the coal body is subject to pressure due to an increase in stress; thus, the pores and fractures in the coal body gradually close. Permeability is low, and gas pressure decreases slowly. In the elastic region, the coal body is weakly affected by the borehole, and gas pressure gradually recovers to the initial level.

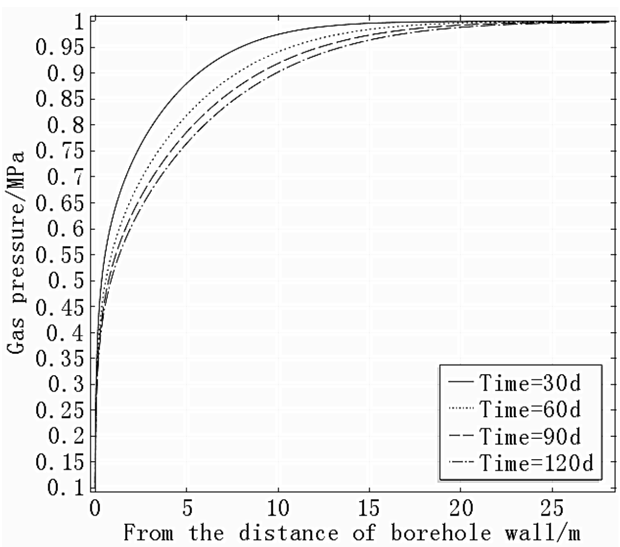


Fig. 6. Gas pressure distribution curves

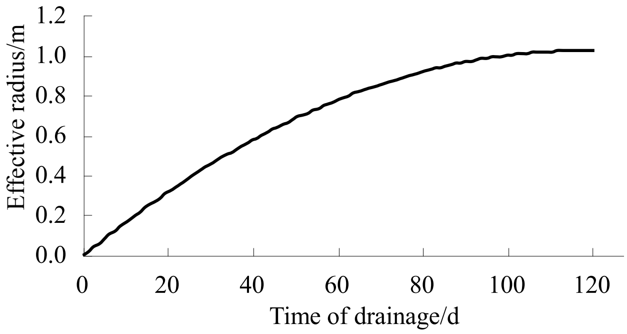
4.2.3 Rule of Changes in the Radius of Drainage through the Borehole

Gas drainage along coal seams is a key method of draining gas in coal mines, and the radius of drainage is a significant parameter in this process. Effective radius r and the radius of influence for drainage R at different drainage times can be obtained from the simulation results of gas pressure in a single borehole (Table 2). Fig. 7 presents the changing curve of the radius of drainage with time.

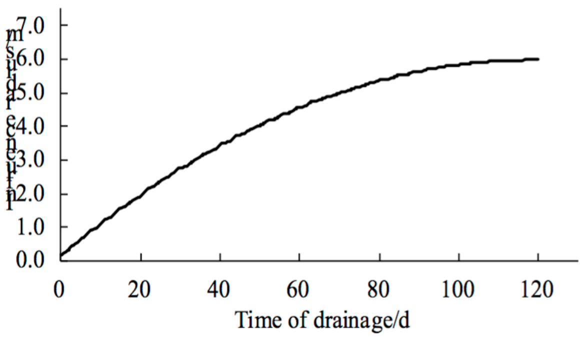
Table 2. Drilling extraction radius

Time of drainage/d	Effective radius r/m	Influence radius R/m
30	0.45	3.31
60	0.62	4.62
90	0.81	5.45
120	0.91	6.03

The curve of the variation in the radius of drainage (Fig. 7) indicates that both the effective radius and the radius of influence for drainage extend with drainage time. Within 60 days of drainage, this extension velocity is large as drainage time increases, the expansion velocity of these radii decreases gradually because the pressure gradient of the gas in the coal seam at the beginning of the drainage process is large. The decrease in gas pressure is notable. As the gas pressure in the coal seam decreases continuously, the effective stress on the coal seam increases and permeability drops; moreover, the decrease in gas pressure decelerates. After 120 days of draining, the effective radius and the radius of influence for drainage tend to almost. stabilize



(a) Effective radius



(b) Influence radius

Fig. 7. Curve of extraction radius change

4.3 Reasonable Spacing of Holes for Gas Drainage along Coal Seams

4.3.1 Mechanism of the Superimposed Effect of Drainage

Numerous actual measurements of gas drainage along coal seams indicate that the superimposed effect of drainage is observed between adjacent boreholes. The decrease in gas pressure between adjacent boreholes is more evident in this case than in drainage through a single borehole. Based on the flow regime of gas in the different regions around the borehole and the simulation results of a single borehole, the permeability of the coal body within the scope of effective radius r for drainage improves because of the influence of pressure relief. Furthermore, this area belongs to the open zone of gas seepage and is the main area affect the decrease in gas pressure. Nevertheless, within the scope of the radius of influence R; thus, coal body is subject to stress concentration and permeability is low. This area is within the attenuation area of gas seepage, and is the secondary area that affects the drop in gas pressure. In the elastic zone, the coal body is weakly affected by the borehole. Thus, the gas flow under the original permeability is hardly affected. This area constitutes the original gas seepage zone.

The rule followed in determining the spacing of holes for gas drainage along coal seams is that excessive spacing should be avoided in case of forming the drainage blind area, as should narrow spacing that heightens drainage costs. To confirm the reasonable spacing of holes, the variation characteristics of different permeability in the open area and in the seepage zone of the coal body around the borehole should be considered to prevent the crossing and repeated effect of high permeability in the open seepage zone. Hence, the high permeability area of the coal body in the open zone should be considered along with the attenuation zone to determine the hole arrangement in gas drainage. The seepage attenuation zone between adjacent holes should be overlapped to accelerate the drop in gas pressure. On the basis of the different permeability of the coal bodies around a single hole, the superimposed effect of drainage can be divided into three conditions,

(1)For the spacing of a hole characterized by $2r \le L < R$: superposition of one decay zone and two decay zones of seepage;

(2)For the spacing of a hole at $R \le L < R + r$: superposition of one decay zone and one transition zone of seepage;

(3) For the spacing of a hole characterized by $R < L < 2R$: superposition of two attenuation zones; the hole arrangement plan is illustrated in Fig. 8.

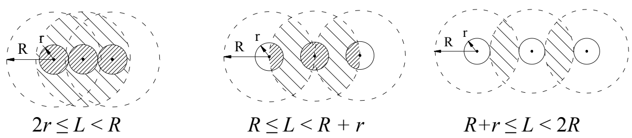
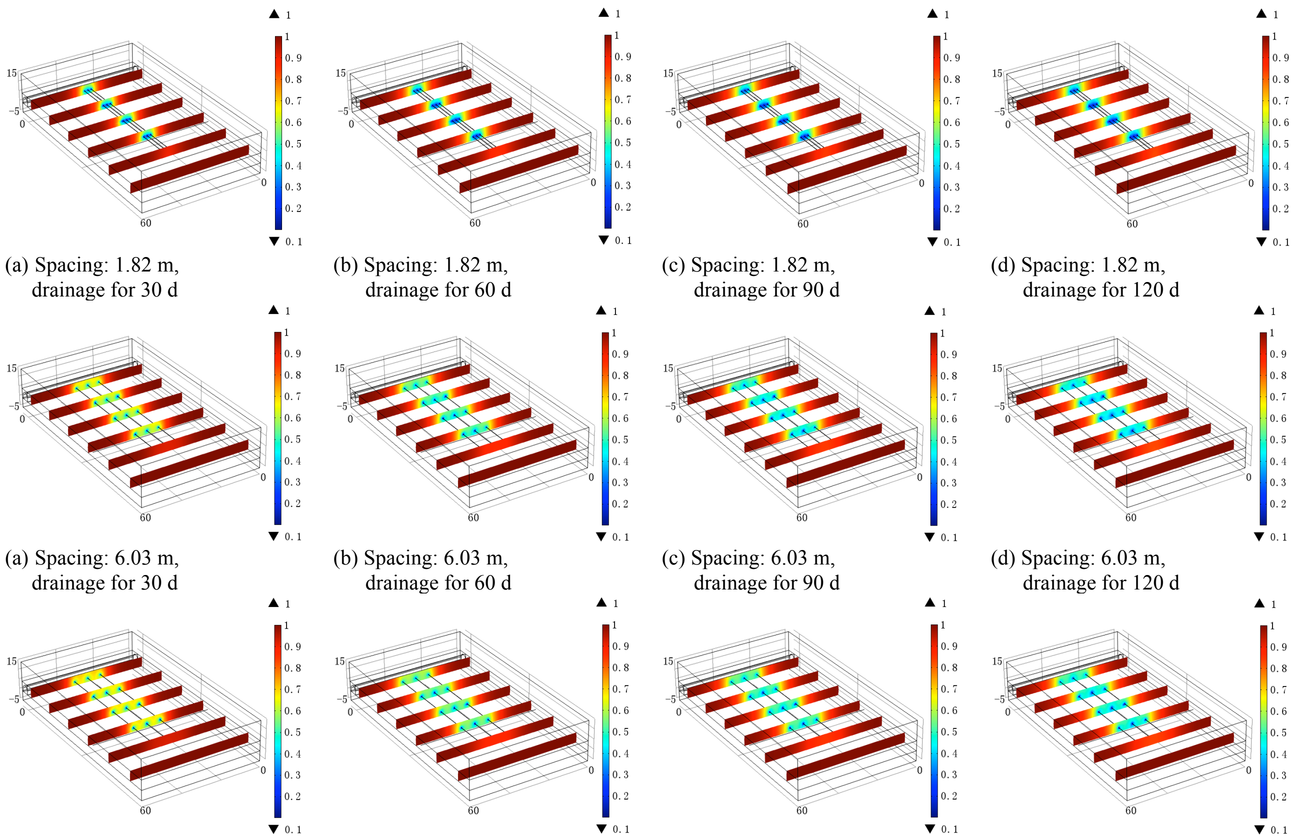


Fig. 8. Diagram of the space between two boreholes

4.3.2 Simulation Results of Gas Drainage under Different Hole Spacings

Based on the analysis of the three types of superimposed effect under different hole spacing, the simulation implemented three hole arrangement plans ($L = 2r$, $L = R$, and $L = R + r$, with hole spacing of 1.82, 6.03, and 6.94 m, respectively) to calculate and determine the superimposed effect of drainage.



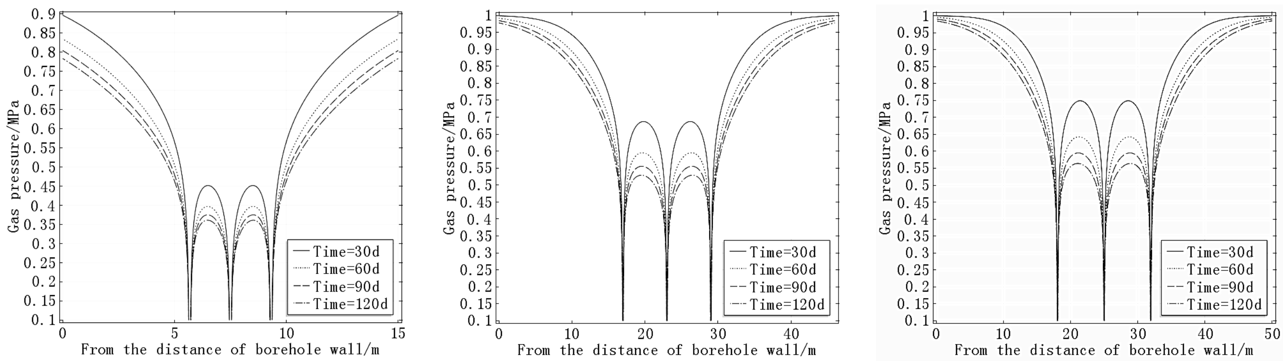
(a) Spacing: 1.82 m, drainage for 30 d
(b) Spacing: 1.82 m, drainage for 60 d
(c) Spacing: 1.82 m, drainage for 90 d
(d) Spacing: 1.82 m, drainage for 120 d

(a) Spacing: 6.03 m, drainage for 30 d
(b) Spacing: 6.03 m, drainage for 60 d
(c) Spacing: 6.03 m, drainage for 90 d
(d) Spacing: 6.03 m, drainage for 120 d

(a) Spacing: 6.94 m, drainage for 30 d
(b) Spacing: 6.94 m, drainage for 60 d
(c) Spacing: 6.94 m, drainage for 90 d
(d) Spacing: 6.94 m, drainage for 120 d

Fig. 9. Gas pressure cloud with different borehole spacing

Fig.9 displays a cloud image of gas pressure under different hole spacing. The gas pressure around holes with the same spacing declines continuously over drainage time. The scope of drilling influence extends continuously; with the progression of drainage, the effect of gas drainage gradually stabilizes. Based on the comparison between the effects of gas drainage under the same moment but with different hole spacing, the larger the hole spacing is, the more gradually the gas pressure between the adjacent holes decreases. The gas pressure between two holes reduces when the space between two holes increases, thus indicating that the strength of the superimposed effect is inversely proportional to the spacing between two boreholes.



(a) Gas pressure curve with a spacing of 1.82 m
(b) Gas pressure curve with a spacing of 6.03 m
(c) Gas pressure curve with a spacing of 6.03 m

Fig.10. Gas pressure curves with different borehole spacings

Fig. 10 presents the curve of gas pressure under different hole spacing, which indicates that the smaller the spacing is, the more significant the drop in the curve of the gas pressure between adjacent boreholes is. Under hole spacing of 1.82, 6.03, and 6.94 m as well as 120 days of drainage, the gas pressures at the center of the interval between adjacent boreholes are 0.361, 0.533, and 0.567 MPa, respectively; the corresponding gas drainage rates are 42.71%, 30.39%, and 28.2%. Under hole spacing of $L = 2r$ and $L = R$, the gas drainage rate is greater than 30%, thereby reaching the drainage standard. Under the hole spacing of $L=R+r$, this rate is less than 30% and fails to reach the drainage standard. This outcome indicates that the superposition of two open seepage zones between adjacent boreholes can improve gas drainage efficiency, compared with that of the single borehole. The superpositioning effect of one open zone and one attenuation zone is almost equal to the drainage effect in the open seepage zone of a single hole. Moreover, the superposition between two attenuation zones of seepage does not improve drainage efficiency. As per the results of many simulation experiments, the reasonable spacing of holes along coal seams for gas drainage should be within twice the scope of effective radius r of drainage through a single hole to the radius of influence R of drainage through a single hole, that is, $2r \leq L \leq R$.

It can be confirmed that the reasonable spacing of holes and drainage time should be determined in consideration of the scope of $2r \leq L \leq R$ for practical gas drainage, according to the deployment plan of mine excavation. When not interfering with the mining alternating, the spacing of holes and drainage time can be increased appropriately to guarantee safe production underground and to reduce drainage cost.

5. Conclusions

(1) This study regards the coal rock containing gas as a type of dual elastic-plastic media. By adopting the Drucker-Prager rule in accordance with rock and soil mechanics, the governing equation of coal rock mass deformation is established. The governing equation of gas migration is established by analyzing the different flow modes of gas in the pore and fracture systems and by considering mass exchange capacity q. The gas migration equation, porosity equation and permeability equation together constitute the elastic-plastic coupling model of the coal rock containing gas.

(2) The practical parameters of the occurrence of coal seam are taken as an example to establish a 3-D model of gas drainage through boreholes. First, the migration law of gas in the coal seam is analyzed by simulating the effect of gas drainage through a single borehole according to the change in stress and permeability of coal at different depths around boreholes. The mechanism of the superimposed effect of drainage is also examined; upon confirming the effective radius r and the radius of influence R of drainage through a single borehole, we can determine the reasonable spacing of holes among the boreholes of gas drainage along coal seams, i.e., $2r \leq L \leq R$.

(3) The research results can reliably guide gas drainage along coal seams theoretically. The reasonable spacing of holes for gas drainage and the drainage time should be confirmed based on safety and cost as well as according to the implemented mine excavation plan in practical gas drainage. When not interfering with the mining alternating, the spacing of holes and drainage time can be increased appropriately to reduce drainage cost, to effectively drain the gas in a coal seam, and to ensure safe production in mines meanwhile.

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References

1. Hu Guozhong, Wang Hongtu, Fan Xiaogang, Li Xiaohong, Deng Yin and Shen Yonghong, "Gas pressure investigation on protection region of up-protective layer of pitching oblique mining", *Journal of China University of Mining and Technology*, 37(3), 2008, pp. 328-332.
2. Hu Weiyue, "Study Orientation and Present Status of Geological Guarantee Technologies to Deep Mine Coal Mining", *Coal Science and Technology*, 41(8), 2013, pp. 1-5.
3. Hao Fuchang, Liu Mingju and Sun Lijuan, "Study on Comparison of Methods to Determine Gas Drainage Radius and Exits Problems", *Coal Science and Technology*, 40(12), 2012, pp. 55-58.
4. Xu Sanmin, "Discussion on method for determining effective methane drainage radius", *Coal Engineer*, (3), 1996, pp. 43-45.
5. Wang Yougang, Liu Jianjun, Yang Jinghe and Zeng Mingsheng, "Simulation of fluid solid coupling flow of coal-bed methane", *Journal of China Coal Society*, 26(3), 2001, pp. 285-289.
6. Qin Yujin, Luo Haizhu, Jiang Wenzhong and Tian Fuchao, "The research of gas migration multi-physics coupling model under the non-isothermal adsorption distortion", *Journal of China Coal Society*, 36(3), 2011, pp. 412-416.
7. Si Hu, Guo Tao and Li Xiaohong, "Analysis and numerical simulation of fluid-structure coupling of gas drainage from boreholes", *Journal of Chongqing University*, 34(11), 2011, pp. 106-110.
8. Liu Jun, Sun Dongling, Sun Haitao and LIN Fujin, "Solid-gas coupling dynamics model and it's application research of coal containing gas", *China Mining Magazine*, 22(11), 2013,pp. 126-135.
9. Liu Feifei, Huang Yuanyue, Xu Quan, Li Shuai, Zhao Pengtao, Qin Zuoya and Huang Bing, "Numerical Simulation of Seepage-heat Similarity for Effective Radius of Drilling Gas Drainage", *Safety in coal mines*, 45(12), 2014, pp. 1-5.
10. Wang Zhaofeng, Li Yanao, Xia Huihui and Tan Ronghui, "Numerical Simulation on Effective Drainage Radius of Drill Hole Along Coal Seam Based on COMSOL", *Safety in Coal Mines*, 43 (10), 2012, pp. 4-6.
11. Zhang Xuebo, Ji Zhongchaon and Gao Jianliang, "Numerical simulation for the influential factors of the effective drainage radius of boring hole", *Journal of Safety and Environment*, 13(2), 2013, pp. 157-160.
12. Li Jianyun, "Optimized Design of Seam Gas Pre-drainage Plan", *Coal Science and Technology*, 39(6), 2011, pp. 46-48.
13. Li Gang, Fan Xisheng, Ma Piliang, Fan Shaowu and Jia Dongxu, "Determination of Rational Pattern Space for Gas Pre-Drainage Borehole in Mining Seam", *Coal Science and Technology*, 39(3), 2011, pp. 56-58.
14. Ma Geng, Su Xianbo and Wei Qingxi, "The determination method of coal gas drainage radius based on methane flow state", *Journal of China Coal Society*, 34(4), 2009, pp. 501-504.
15. Shang Qun, Yang Zhanwei and Jin Jianwei, "Study on Rational Borehole Space of Crossed Boreholes for Gas Pre-drainage", *Coal Science and Technology*, 37(9), 2009, pp. 48-50.
16. Yan Duo, Wang Wencai, Liu Yejiao and Ren Yuhui, "Application of MATLAB in Determining Gas Hole Spacing", *Coal Technology*, 33(5), 2014, pp. 246-248.
17. Liu Sanjun, Ma Geng, Lu Jie and Lin Baiquan, "Relative pressure determination technology for effective radius found on gas content", *Journal of China Coal Society*, 36(10), 2011, pp. 1715-1719.
18. Chaboche J L, "Constitutive equations for cyclic plasticity and cyclic viscoplasticity", *International Journal of Plasticity*, 5(3), 1989, pp. 247-302.
19. Zhang Li, He Xueqiu and Li Houquan, "Coal-bed methane percolation equation and its numerical simulation", *Natural Gas Industry*, 22(1), 2002, pp. 23-26.

20. Hao Fuchang, Liu Mingju and Sun Lijuan, "Determination method of gas drainage radius based on multi-physics coupling", J*ournal of China Coal Society*, 38(s1), 2013, pp. 106-111.

21. Jing L, Tsang C F and Stephansson O, "DECOVALEX-an international co-operative research project on mathematical models of coupled THM processes for safety analysis of radioactive waste repositories", *International Journal of Rock Mechanics and Mining Sciences & Geomechanics Abstracts*, 32(5), 1995, pp. 389-398.