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Improvement of thick coal seam gas drainage efficiency using highly pressurized multi-discharge CO₂ gas fracturing technique

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Abstract

Thick coal seam fracture stimulations were carried out to improve pre gas drainage efficiency using the highly pressurized multi-discharge carbon dioxide gas fracturing technique. The paper presents research results obtained from field work and numerical simulation. The field data demonstrates that the multi-discharge fracturing technique can effectively enhance thick coal seam permeability, so as to improve thick coal seam gas drainage efficiency. On the other hand, regardless of numbers of discharge set used, the effective gas drainage radius remains unchanged, which is inconsistent with the traditional coal seam fracture stimulation, so the causes are discussed in detail. Numerical simulation indicates that the technique provides two basic functions in improving efficiency of coal seam gas drainage: 1) improving coal seam permeability and 2) forming gas pressure gradient of coal seam. The former creates fracture network of coal seam, and the latter, as a driving force, drives coal seam gas from a fracturing borehole to the drainage boreholes through the network developed.

Key words: fracture stimulation; gas drainage; multi-discharge carbon dioxide gas fracturing technique; field monitoring; modeling.

1. Introduction

Changping colliery is located in south of Qinshuei coal field, Jincheng city, Shanxi province, China. Coal seam 3# is a main recoverable seam with average thickness of 5.58m. Affected by regional geological structure, the coal seam is characterized by compactness and poor permeability. Historically, the gas drainability in some part of Qinshuei coal field is found to be particularly poor, which posed a significant challenge to pre gas drainage.

On the other hand, with the increased capability of modern mining machinery to achieve high production rates, it is critical to the success of mine operating in these conditions to ensure the both pre drainage and post drainage gas management systems are efficient and capable of supporting the planned production targets(Black and Aziz, 2008).

To meet the high production rate requirements, the preferred method of gas drainage programs typically involved drilling boreholes within the coal seam, from one set of development heading, across the proposed longwall block extending some 15-50metres beyond the next adjacent development heading.

The inherent limitation with this method is that it is linked to the mining cycle, that is, drilling cannot commence until the gateroad has been developed, and the drainage lead time is dictated by the rate of advance of the adjacent gateroad. Therefore as the rate of gateroad advance increases, the effective gas drainage lead time reduces.

As the coal seam 3# in Changping Colliery has quite low permeability, it results in failure of operations to effectively manage longwall gas emission, therefore, it is often occurred that the gas liberated exceeds the dilution capacity of the mine ventilation system. In such situations, the gas concentration in the mine airway exceeds the prescribed statutory limit resulting in a production delay until such time as the concentration is reduced.

Where such difficult drainage conditions exist, it is impossible to remove sufficient gas from coal seam by using a traditional drainage method in a relatively short period of time. A typical response of colliery operator in such case is to employ the highly pressurized multi-discharge carbon dioxide gas fracturing technique to increase coal seam permeability and improve single borehole gas drainage rate and prolong time of stable gas drainage in coal panel.

The highly pressurized multi-discharge carbon dioxide gas fracturing technique is modified from the Cardox system, which is designed to break or aerate materials by discharge carbon dioxide at high pressure into the material. In order to operate the system in any material, a range of tubes (containers) are available. In turn, these provide a range of discharge pressures. The combination of various tubes, discharge pressures and chemical energizers (heaters) allow over twenty different discharge characteristics, which gives flexibility to different characteristics of coal seam and typical applications in coal mines(Cardox User Manual), such as, coal seam fracture stimulation, outburst control, rock burst control and coal/rock strata distress, etc.

As the technique utilizes a released high pressure carbon dioxide gas to break coal/rock, the same gas used in fire extinguishers, so, it is suitable for use with hazardous, flammable or combustible materials and environments, including underground of coal mine. The rapid release of carbon dioxide gas produces a powerful heaving force that pushes on the solid material, effectively breaking it up into smaller particle sizes or creating fractures in the solid material, for instance, the coal seam. This has led to claims that it can reduce noise, vibration, dust and flyrock, as well as other advantages.

According to literature review, it is noted that the Cardox system was first used in coal mines as an alternative to explosives in the 1930's. The detailed application history has been presented in previous publication (Lu, et al, 2015).

2. Characters of highly pressurized multi-discharge carbon dioxide gas fracturing technique

The highly pressurized single discharge carbon dioxide gas fracturing system has been used in Chinese coal mining industry for sometime, and its function and effectiveness have been proved in the past practice (Lu, et al, 2017).

However, during the underground practices, it is noted that the single discharge system - a standard configuration of the original Cardox system, can not satisfy the requirement of thick coal seam fracture stimulation; therefore, a highly pressurized multi-discharge carbon dioxide gas fracturing system is developed.

2.1 Multi-discharge system

The multi-discharge fracturing system is composed of multiple single discharge fracturing systems, in which each single discharge fracturing system is connected in series mechanically and electronically(Figure 1). During the coal seam fracture stimulation operation, each individual single discharge system is discharged simultaneously along the length of single borehole, so as to achieve the purpose of thick coal seam fracture stimulation.

Figure 1 Structure of multi-discharge system(2 discharge sets)

3. Coal seam fracture stimulation using highly pressurized multi-discharge CO₂ gas fracturing technique

3.1 Field condition

The field work is conducted in Changping Colliery, one of the collieries of Jingcheng Coal

Group located in Changping County, Shangxi province, China. Currently, the Colliery produces 500mt of coal annually.

According to colliery's geologic report, there are five faults in the region, including four normal faults and one reversed fault. The minimum fault throw is about 10m and the maximum is about 50m.

There are 19 coal seams with total thickness varying from 135.97 to 171.13m, in which two coal seams 2# and 3# are recoverable. The distance between two coal seams 2# and 3# varies from 9.40 to 25.69m, and the average thicknesses of coal seam 2# and 3# are 0.73m and 5.76m respectively.

Current field work is related to the coal seam 3#. The roof of coal seam 3# is consisted of siltstone, sandy mudstone, mudstone and medium grained sandstone, etc, the total thickness varies from 1.1 to 10.8m. The floor of coal seam 3# mainly contains sandy mudstone and fine grained sandstone with some siltstone respectively. The physical properties of roof and floor rocks of coal seam 3# are given in Table 1.

Table1 Roof and floor physical properties of coal seam 3#

3.2 Gas characteristics of coal seam 3#

According to the gas data of Changping Colliery, the average maximum absolute coal gas emission rate was 17.48m³/min since 2003. Recent years, the coal gas emission rates are increased as shown in Table 2.

Table 2 Coal gas emission rates of Chiangping Colliery in recent years

In order to clarify the methane characteristics, 16 core samples have been collected from coal seam 3# and tested in laboratory. According to laboratory results, the coal seam 3# can be classified as a high gas content coal seam. The detailed gas composition and parameters of the coal seam 3# are given in Tables 3 and 4 respectively.

Table 3 Gas composition of coal seam 3#

Table 4 Gas parameters of coal seam 3#

From the gas data of coal seam 3# given above, it is noted that the coal seam 3# in trial section presents the characteristics of soft, low permeability and rapid attenuation of gas flow, etc, obviously, an extremely difficult drain coal seam has been encountered. In order to overcome such difficulty, the highly pressurized multi-discharge carbon dioxide gas fracturing technique was selected by colliery for improvement of coal seam gas drainage.

3.3 Coal seam fracture stimulation using multi-discharge highly pressurized gas fracturing technique

Considering current mining layout as well as coal seam thickness, the under panel (panel 4306, coal seam 3#) cross strata drilling and coal seam fracture stimulation have been proposed and implemented in the under panel gas drainage roadway, which is a dedicated roadway for operation of fracture stimulation and gas drainage of coal seam.

3.3.1 Under panel cross strata borehole layout and fracture stimulation procedure

The under panel cross strata drilling and fracture stimulation procedure are designed for few purposes, including 1) estimation of drainage radius induced by the highly pressurized multi-discharge carbon dioxide gas fracturing technique, with varying numbers of discharge sets (1,3,5 and 7) used in a single borehole; 2) determination of borehole gas flow attenuation character before and after coal seam stimulation; 3) comparison of short and long term gas drainage efficiency with and without using the technique.

Figure 2a shows borehole drilling layout, where 19 groups of borehole with 5m interval are set up along 100m trial section. The detailed under penal cross strata drilling design for each group is presented in Figure 2b.

a) Trial section arrangement (top view)

b) Under panel 4306 cross strata drilling design(A-A section)

Figure 2 Coal seam fracture stimulation arrangement

The operating procedure of coal seam fracture stimulation and gas data monitoring is given as follows:

- 1) Each group of borehole is drilled along the length of trail section,
- 2) Gas data, including gas concentration, total gas quantity and pure gas quantity of each borehole are collected immediately and then the data will be recorded once a day,
- 3) When attenuation of borehole gas concentration occurs, first fracturing borehole is drilled between borehole groups 4# and 5# (Figure 3a),
- 4) Then, coal seam fracture stimulation with single discharge set is performed after formation of the fracturing borehole, and followed by gas data collecting and monitoring for all boreholes along the length of trail section,
- 5) When attenuation of borehole gas concentration occurs again, the similar operating procedure will be implemented between groups 7# and 8# with three discharge sets used, and between groups 10# and 11# with five discharge sets used, as well as between groups 17# and 18# with seven discharge sets used respectively. The details of fracturing borehole and stimulation operation are given in Table 5.

To make sure all the influential factors are included and identified objectively, all data from each borehole along the length of trail section are collected and analyzed.

Table 5 Fracturing borehole parameters with discharge sets used

3.3.2 Determination of effective drainage radius with various numbers of discharge set

For any coal seam fracture stimulation method, two parameters are particular interesting to the field engineers, such as, 1) effective drainage radius and 2) gas attenuation character of borehole. First one determines how many gas drainage boreholes are to be drilled for a typical mining panel and second one indicates time period of a borehole can be used to drain the coal seam gas. These two parameters will be discussed in current and following sections associated with various numbers of discharge set used.

The effective drainage radius determines a boundary where gas drainage quantity begins to significantly decline, and it implies that the distance between two adjacent gas drainage boreholes should be selected during the gas drainage designation. Obviously, effective drainage radius is not

only important parameter for gas drainage operation, but also a parameter that can be used to compare the advantage and disadvantage of different fracture stimulation technologies.

In order to accurately determine the effective gas drainage radius, the gas concentration data of equidistant groups, including 2.5m, 7.5m and 12.5m, on both sides of the fracturing borehole are calculated and analyzed. Figure 3 shows the average incremental percentage of gas concentration at the same distance on both sides of the fracturing borehole with one, three, five and seven discharge sets used respectively.

a) One discharge set

b) Three discharge sets

c) Five discharge sets

d) Seven discharges sets

Figure 3 Incremental percentages of methane concentration with various numbers of discharge sets

From Figure 3, it is noted that the effective drainage radius is around 12m under current geological and geomechanical conditions of coal seam. Particularly, the value of effective drainage radius does not change significantly with increasing numbers of discharge set, and the explanations for such phenomenon are given as follows:

Firstly, it is due to the structure of multi-discharge system. Unlike blasting operation, the explosive rolls can be placed together in a borehole, so, the energy released during the explosion can concentrate on a point/a section. Accordingly, with the increase of explosive usage, the fractural range of coal seam will be increased. For a multi-discharge system, however, it is consisted of several single systems connected mechanically as shown in Figure 1, and there is a gap between the discharge heads, normally, varying from 1.5 to 2.5m. Because of this gap, the discharge energy released by the multi-discharge system acts separately, it results in the fractural range of coal seam remains same as that induced by single discharge system.

Secondly, it is due to limited energy generated by the system. According the TNT equivalent of the technique calculated in previous section, the discharge energy is limited; it is difficult to break the coal seam between the two discharge sets. Therefore, although a multi-discharge system is adopted, the energy generated by each discharge set cannot be combined together to form greater energy, so the fractural range of coal seam still remains same as that induced by single discharge system.

Thirdly, it is due to geological characteristics of coal seam. Coal, as a combustible sedimentary rock, occurs as layers or seams, which presents strong anisotropic characteristics - relatively weaker in horizontal direction. Because of this character, the failure of coal seam may easily occur and propagate in horizontal direction rather than in vertical direction under the high pressure gas loading generated by the system. As the result, the energy generated by adjacent discharge sets may not be able to accumulated, so, the fractural range of coal seam remains same as that induced by single discharge system.

Because of above mentioned three main factors, there is not significant change in the effective drainage radius, regardless of numbers of discharge set used during the coal seam fracture stimulation.

Based on the theoretical equations proposed previously (Lu et al, 2015), as well as the current geological and geomechanical data of coal seam, the crushing radius(r_c) of borehole, which represents the zone of yielded and extensively broken or crushed around the borehole and the fragmental radius(r_f), which indicates the effective influential zone caused by the high pressure gas impact loading can be determined as:

$$a = 2 + \frac{\mu}{1 - \mu} \quad (1)$$

$$r_c = \left(\frac{P}{K_c \sigma_c} \right)^{1/a} r \quad (2)$$

Where, P , the gas pressure generated by the high pressure gas system, is equal to 276MPa, K_c , the dynamic compressive strength coefficient of coal, is equal to 9, and r is the radius of fracturing borehole, is 0.037m, σ_c is the uniaxial compressive strength of coal, which is 0.5MPa in current case, a is a stress attenuation index, and μ is the dynamic Poisson's ratio of coal, 0.1. The crushed radius(r_c) can be determined as 0.25m according to the Eqs. (1) and (2).

$$b = \frac{\mu}{1 - \mu} \quad (3)$$

$$r_f = \left(\frac{bP}{K_t \sigma_t} \right)^{1/a} r_c \quad (4)$$

Where, K_t , the dynamic tensile strength coefficient of coal, is equal to 0.95, σ_t , the static tensile strength of coal, is equal to 0.0085, and b , the lateral stress coefficient. Accordingly, the fracture radius(r_f) is determined as 12.4m.

It is noted that the fragmental radius(r_f) calculated by Eqs.(3) and (4) is close to the effective drainage radius(12m) determined from field monitoring data.

3.3.3 Decline characteristics of borehole gas concentration

Decline of borehole gas concentration/gas flow is one of the important characters in a gas drainage operation, it determines the pre drainage time of a borehole after completion of drilling, it also indicates the pre gas drainage efficiency of a single borehole before mining.

The colliery operates in the coal seam 3# have relied on the use of extensive underground to in-seam drilling to drain gas before longwall extraction (pre drainage), so as to reduce the gas content of the coal seam to be extracted to meet the requirements of safety regulation. However, according to the monitoring data, it is noted that the decline of gas concentration occurs within a relative short period of time – around 10 days during the idle period (Figure 4), which is consistent with the previous description of coal seam characteristics, that is, low permeability and difficult drain.

To improve gas drainage efficiency for such coal seam, four coal seam fracture stimulating operations, with discharge sets 1, 3, 5, and 7, have been implemented along the coal seam 3# within 100m trial section respectively. The typical gas data collected from three borehole groups located at front, middle and rear of 100m trial section is presented in Figure 4.

Figure 4a shows the variation of gas concentration collected from borehole 2, group 2 located at the front of trial section. In here, the borehole is only affected by the first stimulating operation with single discharge set. Comparing variation characteristics of gas concentration before and after coal seam fracturing stimulation, it is noted that the effective drainage time is increased from 7 days to 40 days.

Figure 4b presents the variation character of gas concentration collected from borehole 2, group 7 located at the middle of trial section. In here, the borehole is affected by the first, second and third

fracture stimulating operations with one, three and five discharge sets respectively. It is noted that the gas concentration has been raised steadily with time due to fracture stimulating operations implemented periodically along the length of coal seam.

Figure 4c gives the variation character of gas concentration of borehole 1, group 17 located at the real of trial section. In here, the borehole is affected by third and fourth fracture stimulating operations with five and seven discharge sets respectively. It is noted that the fracture stimulating operation is able to effectively maintain the stable increase of gas drainage efficiency in such difficult drain coal seam.

Comparing before and after implementation of coal seam fracture stimulation, the borehole gas concentration can be improved and maintained in a relatively higher level up to 40 days or more. It implies that the highly pressurized multi-discharge carbon dioxide gas fracturing technique can effectively improve gas drainage efficiency for a thick and difficult drain coal seam.

a) Borehole 2, Group 2

b) Borehole 3, Group 7

c) Borehole 1, Group 17

Figure 4 Effect of time on coal seam gas drainage

3.3.4 Effect of numbers of discharge set used on borehole gas drainage

Figure 5 shows the average incremental percentage of gas concentration on both sides of the fracturing borehole with different numbers of discharge set used. It indicates that, although increasing the number of discharge set may not be able to increase the effective drainage radius (in horizontal direction); however, influential region along the length of borehole is increased (in vertical direction).

Under the current geological and geomechanical conditions of coal seam, the empirical relationship between average incremental percentage of gas concentration and numbers of discharge set used in a single borehole can be defined as:

$$g = 7.58n - 3.53 \quad (5)$$

Where, g denotes average incremental percentage of gas concentration collected from the drainage borehole groups near the fracturing borehole (within 5m radius) and n is number of discharge set used in a fracturing borehole.

Also, during the data fitting, it is noted that the power exponent function fits the data much better than the linear function does, but it is believed that the fracturing region along the length of borehole is proportional to the numbers of discharge set used, thus the linear function is selected to describe this relationship. The differences between raw data and fitted line may be due to the error of data measurement and different geomechanical characteristics of coal seam that the boreholes located.

Figure 5 Discharge set number vs incremental percentage of methane concentration

4. Effect of operational, geological and geomechanical factors on fracture stimulation

Currently, the highly pressurized carbon dioxide gas fracturing technique has been accepted in Chinese coal mining industry for coal seam fracture stimulation and improvement of gas drainage

efficiency. The results of engineering practice indicated that the technique is safe and effective for coal seam fracture stimulation in underground coal mine. However, comparing with the field applications, the effect of operational factors, geological and geomechanical factors on effectiveness of coal seam fracture stimulation is not clear. Thus, in following sections, these controllable and uncontrollable factors will be studied, and the influence of these factors on coal seam fracture stimulation and improvement of gas drainage efficiency will be discussed, including:

1) Effect of discharge pressure on coal seam deformation(volumetric strain of coal body), failure(plastic failure around borehole) and gas pressure gradient of coal seam, as well as their effect on coal seam permeability,

2) Effect of space between discharge heads on coal seam deformation and gas pressure gradient of coal seam,

3) Effect of number of discharge set on coal seam deformation and gas pressure gradient of coal seam, and

4) Effect of mining depth and coal body strength on deformation and gas pressure gradient of coal seam.

4.1 Approach and purposes

A multi-physical gas and solid coupled three dimensional numerical model associated with the field data collected is developed using a commercial software, COMSOL multiphysics, so as to gain insights into the interaction of high pressure gas impact loading and coal seam, and clarify the effect of controllable and uncontrollable factors on coal seam fracture stimulation, coal seam permeability and efficiency of coal seam gas drainage.

4.2 About COMSOL multiphysics and control equations

The coupling between gas flow and solid deformation in porous media, which is a principal geo-material to deal with in underground coal mining, has received a considerable attention because of its importance in the area of multi-physical field analysis, contaminant transport and gas outburst during coal mining.

COMSOL is a multiphysics modeling tool that solves various coupled physical problems based on Finite Element Analysis and Partial Differential Equations, which provides a user-friendly interface for mesh generation, equations configuration, and results visualization(Shao, et al 2014). The model can be built with user-defined equations or discipline-specific modules, in here, coupled gas and solid model with dynamic impact loading has been selected. The Subsurface Flow Module and Solid Mechanics Module are applied for modeling the coupled hydrological and soil mechanical system respectively.

4.2.1 Governing equations for gas-solid coupling model

The gas and solid coupling modeling is conducted to study the interaction of in-seam gas distribution characteristics (fluid mechanics), porosity and permeability of coal seam, as well as deformation and failure of borehole wall(solid mechanics) with the high pressure gas impact loading, therefore, the model involves fluid mechanics governing equations and solid mechanics governing equations in porous medium.

For fluid mechanic governing equations, which are used in here to control in-seam gas movement and gas pressure distribution, there are two types of control equations included, that is, the seepage control equation and the gas state equation.

The seepage control equation. There are two assumptions have been taken into consideration,

including 1) the gas movement within coal seam obeys Darcy's law, and 2) ideal gas has been considered for coal seam gas and the gas movement is treated as an isothermal process, so there is:

$$\frac{\partial Q}{\partial t} + \nabla \cdot (\rho \cdot v) = 0 \quad (6)$$

or

$$v_D = -\frac{k}{\mu} \nabla p = -\frac{k_j}{\mu} p \quad (7)$$

Where, ρ is density of coal seam gas, kg/m^3 , v is velocity of seepage, m/s , Q is gas content of coal seam, kg/m^3 , μ is kinetic viscosity of coal seam gas, v_D is Darcy seepage velocity tensor, and k is permeability tensor.

The gas state equation. This equation defines the relationship amount gas density, pressure and temperature. Under the condition of constant temperature, the coefficient of gas compressibility is determined as:

$$\alpha_f = -\frac{1}{V_f} \frac{\partial V_f}{\partial p} \quad (8)$$

Where, V_f is gas volume, and p is gas pressure. According to mass conservation law, $m_f = \rho_f V_f = \text{constant}$, $dm_f = V_f d\rho_f + \rho_f dV_f = 0$, there is:

$$dm_f = -V_f \frac{d\rho_f}{\rho_f} = dV_f = 0$$

Combining Eqs.(7) and (8) :

$$\alpha_f = \frac{1}{\rho_f} \frac{\partial \rho_f}{\partial p} \quad (9)$$

Integrating Eq.(9), the relationship between gas pressure and gas density at constant temperature, also called state equation of gas, can be derived as:

$$\rho_f = \rho_{f0} \exp[\alpha_f (p - p_0)] \quad (10)$$

Where, ρ_{f0} is gas density with initial gas pressure p_0 . On the other hand, the coefficient of gas compressibility can be represented by using bulk modulus of elasticity as:

$$K_f = \frac{1}{\alpha_f} \quad (11)$$

Combining Eqs(12) and (13), the gas state equation of coal seam can be written as:

$$\rho_f = \rho_{f0} \exp\left[\frac{1}{K_f}(p - p_0)\right] \quad (12)$$

For solid mechanics governing equations, which are used to control and analyze the variation of porosity and permeability of coal seam as well as deformation and failure of borehole wall and coal seam under high pressure gas impact loading. There are four types of control equations included, such as, 1) solid equation of internal structure, 2) porosity control equation, 3) theoretical model of permeability and 4) deformation equation of coal and rock mass.

Solid equation of internal structure. According to the solid mechanics, the state equation of internal structure of coal body containing methane can be expressed as (Li and Kong, 2003):

$$\rho_s = \rho_{s0} \exp[a_s(p - p_0)] = \rho_{s0} \exp\left(\frac{p - p_0}{K_s}\right) \quad (13)$$

Where, ρ_{s0} is density of internal structure of coal body under pressure p_0 , 10kg/m^3 , p is coal seam gas pressure, MPa, K_s is bulk elastic modulus of gas, Pa, a_s is compressibility coefficient of gas.

Porosity control equation. It is assumed that the coal body contains a single phase saturated gas, and the linear elastic deformation occurs when the loading is applied on the coal body. Accordingly, the porosity (φ) of coal body can be defined as:

$$\varphi = 1 - \frac{(1 - \varphi_0)}{1 + \varepsilon_v} \left(1 + \frac{\Delta V_s}{V_{s0}}\right) \quad (14)$$

Where, V_s is solid structure volume of porous media, m^3 , ΔV_s represents the variation of V_s , m^3 , φ_0 is initial porosity of coal seam, and ε_v is volumetric strain of coal body, m^3 .

Theoretical model of permeability. Permeability is a parameter, which defines the difficulty of gas movement within coal seam. Normally, the permeability is considered as a constant value in the seepage model, which did not involve the internal structure deformation and variation of porosity of coal body. However, when an impact loading is applied on the coal body, the internal structure and porosity of coal body may be changed significantly, which results in the variation of permeability of coal seam. Under such circumstances, the theoretical control equation for permeability, which defines the relationship between porosity and permeability of coal seam can be described using Kozeny-Carman equation as (Lu and Shen, 2002):

$$k = \frac{\varphi}{k_z S_p^2} = \frac{\varphi^3}{k_z S_v^2} \quad (15)$$

Where, k_z is constant and equals 5, φ is permeability, S_v is surface area of pores in unit volume of porous media, cm^2 , $S_p = A_s/V_p$, A_s is total surface area of coal, cm^2 , V_p is pore volume of coal, cm^3 .

Deformation equation of coal and rock mass. Under an impact loading, the deformation or failure of coal body may occur. Such deformation and failure is defined by using rock mechanics deformation control equation, including geometric equation and constitutive equation respectively.

For geometric equation of coal body deformation, it can be expressed as:

$$\varepsilon_{ij} = \frac{1}{2}(u_{i,j} + u_{j,i}) \quad (16)$$

and the volumetric strain is:

$$\varepsilon_v = u_{11} + u_{22} + u_{33}$$

Where, ε_{ij} is stress tensor, ε_v is volumetric strain of coal, u is deformation, m.

For the constitutive equation of deformation and failure of coal body, the deformation is described herein by two parts, elastic strain increment($d\varepsilon_{ij}^e$) and plastic strain increment($d\varepsilon_{ij}^p$), that is, the plastic flow rule:

$$d\varepsilon_{ij} = d\varepsilon_{ij}^e + d\varepsilon_{ij}^p \quad (17)$$

The increments between elastic stress and elastic strain are connected by elastic matrix D , and the increment of plastic strain is represented by plastic potential energy, so we have:

$$d\varepsilon_{ij}^p = \lambda \frac{\partial g}{\partial \sigma_{ij}} \quad (18)$$

Where, λ is positive undetermined limited value. For a stable plastic deformation, normally the hardened material g can be represented by similar form of yield function f , so the plastic flow rule can be given as:

$$d\varepsilon_{ij}^p = \lambda \frac{\partial f}{\partial \sigma_{ij}} \quad (19)$$

and total increment of strain is:

$$d\varepsilon_{ij} = D^{-1}d\sigma_{ij}^e + \lambda \frac{\partial f}{\partial \sigma_{ij}} \quad (20)$$

Where, D is elastic matrix, $d\sigma_{ij}^e$ is increment of elastic stress. The yield criteria of coal body can be described by using Drucker-Prager Law as follows(Zhou, Li, 2008):

$$f(I_1, \sqrt{J_2}) = \alpha I_1 + \sqrt{J_2} - K \quad (21)$$

Where, I_1 and J_2 are first invariant of principal stress, and second invariant of deviatoric stress, and a , k are constants related to the cohesion(c) and friction angle(ϕ) of geo-material, which can be determined as:

$$\alpha = \frac{2\sin\phi}{\sqrt{3}(3-\sin\phi)}, \quad \kappa = \frac{6c\cos\phi}{\sqrt{3}(3-\sin\phi)} \quad (22)$$

According to the strengthening criterion of Drucker-Prager Law:

$$\sigma_p = \sigma_p^0 + \frac{(\sigma_p^m - \sigma_p^o)\varepsilon^{ep}}{A + \varepsilon^{ep}} \quad (23)$$

Where, σ_p^o initial yield stress, MPa; σ_p^m is maximum value of reinforcement function, MPa; ε^{ep} is equivalent strain, and A is controlled constant of plastic hardening rate.

4.3 Model development and boundary conditions

During the model development, there are several aspects are taken into consideration, including 1) dimension of model, 2) boundary conditions of model, 3) parameters assigned to the model, 4) dynamic impact loading.

First of all, considering possible influential range of high pressure gas impact loading detected from field work, the dimension of model is built with 20m x 20m x 30m as shown in Figure 6.

When the boundary conditions of the model are concerned, except for the symmetric face, non-reflecting boundaries were enforced to the other faces of the model to simulate the infinite rock mass, where no external loading is involved, and the top face of model is loaded uniformly distributed load, which is determined by the depth of coal seam(γh). In addition, the parameters obtained during the field work are assigned to the model as shown in Table 6.

Finally, for the impact loading, a uniformly distributed dynamic impact loading is applied on the borehole wall located at the middle of symmetric face of model with 0.5m in length, which is determined by the length of discharge head of highly pressurized carbon dioxide gas fracturing system. The dynamic loading is defined by the relation of impact loading(p) and acting time(t), the detailed $p - t$ relationship will be discussed in the following section.

Figure 6 Model and mesh

Table 6 Parameters assigned to model

4.4 The $p - t$ relationship

Dynamic loading in current case is related to physical expansion. After discharge of the highly pressurized carbon dioxide gas fracturing system, the high pressure gas is released from the sealed compressed gas cylinder, then the pressure of gas reduces with time elapsed.

To simulate such dynamic impact loading, the relationship between the discharge pressure (p) generated by the highly pressurized carbon dioxide gas fracturing system and the duration of loading (t), in other words, the decay function of discharge pressure with time, must be determined. In there, there are two parameters involved, first, the maximum gas pressure applied on the borehole wall after discharge and second, the decay time of the gas pressure.

It is noted that the maximum discharge pressure generated by the system is 276MPa, which is the failure pressure of rupture disc. However, this pressure may be changed due to: 1) the geometry of discharge head, 2) the numbers of nozzle installed on the discharge head and nozzle distribution pattern, and 3) the geometry of nozzle. However, considering current research does not focus on these issues, for simplicity, the maximum pressure generated by the system, 276MPa, is used in current numerical modeling.

From literature review, some decay functions, which were used to describe the blasting pressure and duration of loading in different types of media, have been found and these functions are summarized as follows:

Firstly, the pressure decay function in steel structure. To obtain the decay function, an experimental study was carried out on steel beam under blast loading. The experimentally measured blast waves were conventionally idealized by Eq(26), where P_+ is peak force in positive duration t_d , b is a shape parameter having major effects especially for the shape of negative phase(Figuli, et al, 2017)

$$P(t) = P_+ \left(1 - \frac{t}{t_d}\right) e^{-b \frac{t}{t_d}} \quad (24)$$

Figure 7a shows the pressure decay function of blast loaded on steel beam. Throughout the pressure-time profile, two main phases can be observed, including the positive phase of duration and the negative phase of duration. The negative phase is of a longer duration and a lower intensity than the positive duration.

Secondly, the pressure decay function in rock. Use of explosives is the most cost effective and widely used rock excavation method in coal mine, such as the drilling and blast. The damage induced by blasting is mainly results of explosive stress waves and subsequent explosive gas expansion.

The blast pressure loaded on the borehole wall can be determined as(Yanga, et al, 2007):

$$P_w = \frac{\rho_e V_d^2}{2(\gamma + 1)} \left(\frac{d_c}{d_b}\right)^{2\lambda} \quad (25)$$

where ρ_e is the explosive density, V_d is the velocity of detonation, d_c is the charge diameter, d_b is the blast hole diameter, γ is the specific heat ratio, and λ is the explosive's adiabatic expansion constant, normally, $\gamma=3$.

The borehole wall pressure-time history is adopted to approximate the borehole pressure-time history, which was proposed as(Yang, et al, 2017):

$$P_w(t) = 4P_w \left(e^{-\beta t / \sqrt{2}} - e^{-\sqrt{2}\beta t} \right) \quad (26)$$

where, $P_w(t)$ is the borehole wall pressure-time history, t is time, P_w is borehole wall pressure, and β is a damping factor that is determined on the basis of rising time of borehole pressure to it peak. According to Eq.(29), the peak pressure occurs at $t_r = -\sqrt{2} \ln(1/2) / \beta$, so, the damping factor can be expressed as $\beta = -\sqrt{2} \ln(1/2) / t_r$. Figure 7b shows the borehole wall pressure-time history in rock material.

Thirdly, the pressure decay function in cellular material. Cellular materials, such as open and closed foams, honeycombs, and metal hollow spheres, are new classes of ultra-light multi-functional materials that can be withstand large deformation at nearly constant plateaus stress. These materials can absorb a large amount of kinetic energy before collapsing to a more stable configuration. Cellular materials are attached as sacrificial layers to protect structures, machines, and infrastructure against dynamic events(Liang, et al, 2017).

According to previous research work conducted, the linear decay function is proposed and used to describe the relationship between blast pressure and time in cellular material:

$$P(t) = P_0 \left(1 - \frac{t}{t_0}\right) \quad (27)$$

Figure 9 Variations of simulated permeability and monitored gas concentration

4.6 Effect of discharge pressure on deformation, failure and gas pressure gradient of coal seam

Of operational factors considered, discharge pressure, distance between discharge heads, and number of discharge set used, are simulated to clarify how these factors influence the coal seam fracture stimulation and the gas drainage efficiency of coal seam.

Firstly, five levels of discharge pressure, which is the most important operational factor for stimulating operation, are selected to study its effect on coal seam deformation and failure as well as gas pressure gradient of coal seam.

Table 7 shows plastic failure around borehole wall associating with different discharge pressures. Obviously, the value of failure radius varies logically with different discharge pressure applied. Apart from plastic failure around borehole wall, the internal deformation and/or failure of coal seam induced by the high pressure gas impact loading may directly influence porosity structure and permeability of coal seam.

Table 7 Failure around borehole with different discharge pressures

The gas permeability of a coalbed is influenced during gas production not only by the simultaneous changes in effective stress and gas slippage, but also by the volumetric strain of the coal matrix that is associated with gas desorption (Harpalani and Chen, 1997), as well as the degree of damage, which is related to the deformation of coal (Yang, et al, 2011). The theoretical relation between permeability and volumetric strain is proposed (Sun, et al, 2018):

$$K = k_0 \times \left(\frac{1 + \Delta\varepsilon}{n} \right)^2 \quad (28)$$

where, K is permeability, k_0 is initial permeability, n is porosity of media, and $\Delta\varepsilon$ is increment of volumetric strain.

Figure 10 demonstrates the variation of internal coal seam deformation associated with the different failure radius of borehole wall(r) and discharge pressure(P) (Figure 10a) and the effect of different levels of high pressure gas impact loading on coal seam deformation (Figure 10b), as well as the deformation profile within the coal seam under maximum discharge pressure (Figure 10c).

a) Volumetric strain distribution in coal seam with various discharge pressures

b) Volumetric strain vs. discharge pressures

c) Volumetric strain distribution in coal seam with max discharge pressure

Figure 10 Volumetric strain of coal body vs discharge pressure

Based on Eq.(28) and numerical result obtained in Figure 10b, the relationship between permeability(K) of coal seam and discharge pressure(P) generated by the highly pressurized carbon dioxide gas fracturing technique can be proposed as:

$$K = k_0 \times \left(\frac{1 + 0.028e^{0.0057P}}{n} \right)^2 \quad (29)$$

Also, the permeability of coal seam reduces with an increasing of distance from the fracturing

borehole. For the maximum discharge pressure, 274MPa, the internal permeability of coal seam, in other words, at a particular distance from the fracturing borehole, can be determined as:

$$K = k_0 \times \left(\frac{1 + 0.12e^{-0.3x}}{n} \right)^2 \quad (30)$$

where, x is a distance from fracturing borehole, m.

On the other hand, to improve the gas drainage efficiency of coal seam, apart from coal seam permeability, the gas pressure gradient of coal seam is another important factor.

After discharge of the system, the high pressure gas impact loading is applied on the borehole wall, and it not only causes significant deformation/failure of coal seam, but also forms local high pressure gas zone around the fracturing borehole, thus, the gas pressure gradient of coal seam is built up. Figure 11 gives the gas pressure distribution characteristics of coal seam under the maximum gas impact loading (274MPa).

Figure 11 Gas pressure distribution of coal seam after discharge

Considering Darcy flow theory, owing to discharge pressure causes a rise of pore-pressure of coal seam around fracturing borehole, and the flow of coal seam gas to drainage borehole takes place through the fractural network. As a result, an increment of gas concentration/flow can be detected.

Theoretically, the coal seam gas density(ρ_g) can be expressed as (Liu, et al, 2017):

$$\rho_g = \frac{M_c}{RT} p \quad (31)$$

where, M_c is molar mass of methane, kg/mol, p is gas pressure of coal, MPa, R is universal gas constant, J/(mol·K), T is temperature of coal, K.

Based on the gas pressure distribution of coal seam (Figure 11), the gas density at different location within the coal seam (under the maximum discharge pressure, 276MPa) can be determined:

$$\rho_g = \frac{M_c}{RT} (-1.8Ln(x) + 4.8) \quad (32)$$

where x is distance from fracturing borehole, m.

4.7 Effect of space of discharge heads and number of discharge set on deformation and gas pressure gradient of coal seam

The space of two discharge heads and numbers of discharge set are two other important operational parameters for this technique.

During the field operation, it is noted that the effective gas drainage radius does not change significantly with an increasing of discharge number. One of the reasons caused is due to the space between two discharge heads, which prevents energy accumulation of two adjacent discharge heads. However, if the space is small enough, can it improve the effective gas drainage radius? To answer this question, the different spaces between two discharge heads varying from 1-5m (under 276MPa) are simulated respectively.

According to the modeling results, one meter space gives the best result comparing with others, and it seems to have some impact on the effective gas drainage radius(Figure 12). From manufacture point of view, it is possible to reduce the space between two discharge heads to one meter by shortening the length of CO₂ storage; however, the quantity of liquid CO₂ will be compromised. Therefore, it is no necessary to take serious consideration about this issue.

a) 1m

b) 5m

Figure 12 Effect of space between two discharge heads on coal seam deformation

Following above results obtained, one meter space between two discharge heads is used to simulate the effect of numbers of discharge set on coal seam deformation. The result is given in Figure 13, which reveals the internal deformation characteristics of coal seam with different numbers of discharge set used. It shows that, within five meters from the fracturing borehole, increasing numbers of discharge set has significant impact on the deformation/permeability of coal seam. After five meters, the impact gradually decreases, and no impact can be found after ten meters, regardless of how many discharge sets used.

Again, the modelling result gives the similar conclusion as that of field observation, that is, the gas drainage radius will not increase with increasing number of discharge set.

Figure 13 Number of discharge set vs coal seam deformation

4.8 Effect of geomechanical parameters on deformation, failure and gas pressure gradient of coal seam

Apart from operational parameters discussed above, the geological and geomechanical factors, such as, coal seam depth and coal body strength, also play an important role in coal seam fracture stimulation and gas drainage operations.

The depth of mineral resources like coal continuously increases due to the exhaustion of shallow resources, and the characteristics of high ground stress in deep ground inevitably affect fracture of rock blasting (Yang, et al, 2018).

A common approach for determining the in-situ stress field is to assume that the pre-mining principal stresses are vertical and horizontal. Then, the vertical stress component can be estimated by the weight of the overburden, with $\sigma_v = 0.025 - 0.027 h$ (h is depth), even this assumption is not always true.

To understand the effect of mining depth on coal seam fracture stimulation using the highly pressurized carbon dioxide gas fracturing system, four mining depths are selected to simulate their effect on the deformation and failure as well as the gas pressure gradient of the coal seam.

Figure 14a shows the failure around borehole wall under high pressure gas impact loading with different mining depths. The effect of mining depth on the coal seam failure around the borehole wall is obvious, that is, the failure radius around borehole wall reduces linearly with an increasing of mining depth. It implies that the failure and fracture propagation around borehole wall induced by high pressure gas impact loading is restrained by the increased stress of coal seam.

Figure 14b shows the volumetric strain distribution of the coal seam associated with various mining depth. It demonstrates the difference of volumetric strain of coal seam is relatively small, accordingly, it may be claimed that the effect of mining depth on deformation of the coal seam is limited under high pressure gas impact loading. Similarly, the variation of gas pressure gradient of the coal seam is also limited (Fig.14c).

a) Failure character vs mining depth

b) Coal seam volumetric strain vs mining depth

c) Internal gas pressure vs mining depth

Figure 14 Effect of mining depth on borehole failure and gas pressure within coal seam

On the other hand, the strength of coal body, as another geomechanical parameter of coal seam, should be taken into consideration during the coal seam fracture stimulation and gas drainage operations. Logically, it is understood that the higher the strength of coal, the smaller the fracture propagation of coal seam, however, it has always been of interest to mining engineers to precisely define the relationship between coal seam strength and coal seam fracture/failure radius in coal seam fracture stimulation design and operation.

The theoretical relationship between crushing radius of rock mass after blasting and rock strength is given below (Liu et al, 2017):

$$r_c = \left(\frac{\sqrt{2}\sigma_c \xi^{1/3}}{Cp_c} \right)^{\frac{1}{\alpha}} r_b \quad (33)$$

where, r_c is crushing radius (plastic failure) around borehole, m, σ_c is uniaxial compressive strength of rock, MPa, ξ is loading strain rate of rock, C is intermediate variable, p_c is initial pressure of shock wave applied on the borehole wall, MPa, r_b is radius of borehole, m. α the coefficient of attenuation of shock wave.

Similarly, it is also important to demonstrate the effect of coal strength on deformation, failure and gas pressure gradient under the high pressure gas impact loading. So, four levels of the Protodyakonov coefficient of coal body, including $f = 0.5, 1, 2, 3.8$, which represents low, moderate and high coal strength, are simulated respectively.

a) Coal seam failure vs strength

b) Coal seam deformation vs strength

c) Internal gas pressure distribution vs coal strength

Figure 15 Effect of coal strength on deformation and failure

According to the modeling results, the plastic failure radius around borehole wall reduces with an increasing of coal body strength linearly (Figure 15a), and corresponding to different coal strength, the internal deformation of coal seam varies significantly (Figure 15b), which implies that the coal strength have a significant impact on fracture development and propagation of coal seam. For different coal strength, the gas pressure gradient built up in coal seam is different within 5m(Figure 15c). Overall, the modeling results are consistent with the traditional concept, that is, high coal strength will limit fracture development of coal seam.

Comparing with these two uncontrollable parameters, the coal strength is a more sensitive parameter for the fracture formation and propagation during the coal seam fracture stimulating operation.

5. Conclusions

From Figure 4, it is noted that the continued use of traditional underground gas drainage method alone will not be capable of draining sufficient gas in the relatively short time to support

high production rates by longwall mining in gassy and low permeability conditions. Thus, coal seam fracture stimulation is conducted using the highly pressurized multi-discharge carbon dioxide gas technique to improve coal seam gas drainage efficiency. Based on this work, the relevant research is carried out to gain insight into the functions of coal seam fracture stimulation using this technique, as well as their effect on improvement of coal seam gas drainage efficiency. Accordingly, the research results can be drawn as follows:

1) Due to thick coal seam condition, the multi-discharge system has been used with various numbers of discharge set 1,3,5,7 respectively. The monitoring data indicated that the effective gas drainage radius is around 12m (Figure 3), that is, it remains unchanged regardless of numbers of discharge set used. Obviously, the result is inconsistent with the traditional coal seam gas drainage practices, and it may be caused by few reasons as follows:

- The structure of the multi-discharge system,
- Limited expansion energy generated by the technique, and
- Geological characteristics of coal seam.

2) Comparing decline characteristics of borehole gas concentration before and after coal seam fracture stimulation, it is noted that the borehole gas concentration can be improved and maintained in a relatively higher level up to 40 days or more(Figure 4), and the results imply that this technique can effectively improve gas drainage efficiency for a thick and difficult drain coal seam.

3) Although increasing the number of discharge set may not be able to increase the effective drainage radius, however, influential region along the length of borehole is increased(Figure 5), it indicates that the multi-discharge fracturing technique may benefit the fracture stimulating operation and improve the borehole gas drainage efficiency in thick and low permeability coal seam.

On the other hand, based on field geological and geomechanical conditions, the numerical modeling is carried out to study the effect of controllable and uncontrollable factors on deformation, failure and gas pressure gradient of coal seam. The work allows understanding how these factors contribute to the coal seam fracture stimulation and the mechanisms involved in improving gas drainage efficiency of this technique.

1) From the $p-t$ diagram in Figure 7b, it is noted that the pressure on the borehole wall is applied instantly and then it lasts for a certain period of time. Such a high pressure gas induces borehole wall failure and coal body fragmentation. The interaction of various levels high pressure gas impact loading with coal seam causes varying degrees of plastic failure around borehole, where the coal between the radial cracks is sheared and many researchers identify this zone as the crushing zone(Yorbica and Lapcevic, 2018). In this part of study,

- The linear relationship between discharge pressure and crushing radius of fracturing borehole is identified (Table 7), and
- The volumetric strain of coal seam induced by various levels of discharge pressure is presented(Figure 10a), accordingly, an empirical relations between discharge pressure and permeability of coal seam are proposed (Eqs.31 and 32),
- Actually, the discharge pressure not only causes deformation and failure of coal body, but also forms high gas pressure zone around fracturing borehole(Figure 11), because of the result, a gas pressure gradient in coal seam is built up. Theoretically, the gas density at different location within the coal seam (under the maximum discharge pressure, 276MPa) can be determined using Eq. 34.

2) Apart from the discharge pressure, other controllable parameters, such as, the space between discharge heads and the numbers of discharge set used in a single borehole are also studied.

- The results show that 1m space between two discharge heads gives the best fracture stimulation outcome (Figure 12), but the size of liquid CO₂ storage will be compromised,
- Under such spacing conditions, the increase of the number of discharge set will have a significant impact on the deformation/permeability of coal seam within 5m from the fracturing borehole, and the impact reduces gradually from 5 to 10m, and no impact can be found after 10m (Figure 13). It is proved that the gas drainage radius may not be changed significantly, whether reducing the space between two discharge heads or increasing the number of discharge set.

3) For the uncontrollable factors, such as, the mining depth and the coal strength are simulated to see how they are able to affect the results of coal seam fracture stimulation. In this part of the work, it is noted that:

- The variation of mining depth does not cause significant difference of deformation and gas pressure in coal seam (Figure 14), however,
- The deformation and gas pressure level of coal seam reduce with an increasing of coal strength (Figure 15). Comparatively, the coal seam strength is more sensitive parameter than the mining depth during the coal seam fracture stimulating operation.

4) In this research, the analysis of interaction is made of the effect of high pressure gas impact loading on coal seam deformation and failure, as well as gas pressure gradient of the coal seam; the work may lead to some of the results as follows:

- The high pressure impact loading results in significant change of internal structure of coal body, because of various levels of volumetric strain occurs in coal seam, which implies that the fracture network in coal seam may be expanded further and the coal seam permeability may be improved.
- The high pressure impact loading causes coal seam relaxation, and the adsorption pressure in coal seam is reduced, so the gas adsorbed in the coal body may be released and the efficiency of coal seam gas drainage can be improved.
- The high pressure impact loading leads to formation of gas pressure gradient within the coal seam, because of this, the movement of free gas in the porosity structure of coal body occurs.

5) In summary, the highly pressurized multi-discharge carbon dioxide gas fracturing technique provides two basic functions to improve the efficiency of coal seam gas drainage, that is, 1)improving permeability of coal seam and 2)forming gas pressure gradient within the coal seam.

- The permeability is improved due to significant change of internal coal body structure induced by the high pressure gas impact loading, those changes include: 1) an increasing in the number of fractures or channels, 2) extension of existing fractures or channels, 3) opening or dilation of fractures within the coal seam, and 4) a creation of void space within the coal seam. All of these will lead to increase the permeability of coal seam.
- On the other hand, a high gas pressure zone around fracturing borehole is developed, after discharge of the gas fracturing system. As a result, the gas pressure gradient is formed between the fracturing borehole and the drainage boreholes within the coal seam, and as a driving force, it drives the coal seam methane from the fracturing borehole to the drainage boreholes, through the network developed/improved.

References

- Black, D.J. Aziz, N.I., Gas drainage improvement in underground coal mines, APCBM 2008 Symposium, Brisbane, September, 2008, Page 1-7.
- Cardox User Manual, User manual, Cardox International Pty Ltd.

- Chatterjee, R., Pal, P. K. Estimation of stress magnitude and physical properties for coal seam of Rangamati area, Raniganj coalfield, India, *International Journal of Coal Geology*, 2010,81: 25–36.
- Figuli, L., Bedon, C., Zvakova, Z., Jangl, S, Kavicky, V. Dynamic analysis of a blast loaded steel structure. *Procedia Engineering*, 2017, 119: 2463-2469.
- Harpalani, S., and Chen, G.L., Influence of gas production induced volumetric strain on permeability of coal, *Geotechnical & Geological Engineering*, 1997, Vol. 15, Iss. 4, pp. 303-325.
- Harvey, C. R. Analysis and management of outbursts: with particular reference to the Bulli coal seam, PhD thesis, Faculty of Engineering, University of Wollongong, 2001, pp.110 – 114.
- Heunis, R. The development of rock-burst control strategies for South African gold mines, *Journal of the South African Institute of Mining and Metallurgy*, April, 1980, pp. 139 – 150.
- Karacan, C., Felicia, A.R, Cotè, M., Phipps, S. Coal mine methane: A review of capture and utilization practices with benefits to mining safety and to greenhouse gas reduction, *International Journal of Coal Geology*, 2011Vol. 86, Iss. 2 - 3, pp. 121-156.
- Konicek, P., Konecny, P., Ptacek, J. Destress rock blasting as a rockburst control technique, *Harmonising rock engineering and the environment – Qian & Zhou (eds)*, Taylor and Francis Group, London, ISBN 978-0-415-80444-8, 2012, pp. 1221 – 1226.
- Li P.C, Kong X.Y. Mathematical model of fluid solid coupling seepage in saturated porous media, *Journal of hydrodynamics*, 2003, Vol.18, No.4, pp. 420 - 426.
- Liang, M., Li, Z., Lu, F., Li, X. Theoretical and numerical investigation of blast responses of continuous-density graded cellular materials. *Composite Structures*, 2017,164:170-179.
- Liu, Z.D., Cheng, Y.P., Jiang, J.G., Li, W., Jin, K. Interactions between coal seam gas drainage boreholes and the impact of such on borehole patterns, *Journal of Natural Gas Science and Engineering*, 2017, 38:597-607.
- Liu, C. Y., Yang, J.X., Yu. B. Rock-breaking mechanism and experimental analysis of confined blasting of borehole surrounding rock, *International Journal of Mining Science and Technology* Volume 27, Issue 5, September 2017, Pages 795-801, doi.org/10.1016/j.ijmst.2017.07.016.
- Lu, P. Shen X.W. Characterization and experimental study on permeability of rock mass in process of stress-strain, *Journal of China University of Science and Technology*, 2002, No.6. pp. 679-681.
- Lu, T.K, Sun,H.L, Yang, X.L. : TNT equivalence and fragmentation mechanisms of high pressure gas fracturing technique, *Blasting and Fragmentation*, 2017, 11 (1): 65-84.
- Lu, T.K., Wang, Z.F., Yang, H. M., Yang, P.J, Han, Y.B., Sun, X.M. Improvement of coal seam gas drainage by under-panel cross-strata stimulation using highly pressurized gas, *International Journal of Rock Mechanics & Mining Sciences*, 2015, 77: 300 - 312.
- Shao, W., Bogaard,T., Bakker, M. How to Use COMSOL Multiphysics for coupled dual-permeability hydrological and slope stability modeling , *Procedia Earth and Planetary Science* 9 (2014) 83 - 90.
- Sun, J, Wang, L.G., Zhao, G.G. Failure characteristics and confined permeability of an inclined coal seam floor in Fluid-solid coupling, *Advances in Civil Engineering*, 2018(4):1-12, dio:10.1155/2018/2356390.
- United Nations, Best practice guidance for effective methane drainage and use in coal mines, ECE Energy Series No.31, 2010, New York and Geneva.
- Unver, B. Effect of residual tectonic stresses on roadway stability in an underground coal mine, *The Journal of The South African Institute of Mining and Metallurgy*, MAY/JUNE, 1999, pp. 167 –

- Yang, L.Y., Ding, C.X., Yang, R.S., Wang, J. Full field strain analysis of blasting under high stress condition based on digital image correlation method, *Shock and Vibration*, 2018(16):1-7, DOI: 10.1155/2018/4894078.
- Yang, W., Lin, B.Q., Qu, Y.A., Zhao, S., Zhai, C., Jia, L.L, Zhao, W.Q. Mechanism of strata deformation under protective seam and its application for relieved methane control, *International Journal of Coal Geology*, 2011, Vol. 85, Issues 3-4, pp. 300-306.
- Yang, J.H., Yao, C., Jiang, Q.H., Lu, W.B., Jiang, S.H. 2D numerical analysis of rock damage induced by dynamic in-situ stress redistribution and blast loading in underground blasting excavation, *Tunnelling and Underground Space Technology*, 70 (2017) 221–232.
- Yanga, J.H., Yao, C. Jiang, Q.H., Lu, W.B., Jiang, S.H. 2D numerical analysis of rock damage induced by dynamic in-situ stress redistribution and blast loading in underground blasting excavation. *Tunnelling and Underground Space Technology*, 2007, 70:221-232.
- Yorbica, S. and Lapcevic, V., Rock fracturing mechanisms by blasting, *Underground Mining Engineering*, 2018, 32:15 - 31.
- Zhou, F.X., Li.S.R. Drucker-Prager strength criteria, *Soil mechanics*, 2008, Vol.29, No. 3, pp. 748-749.

Figures

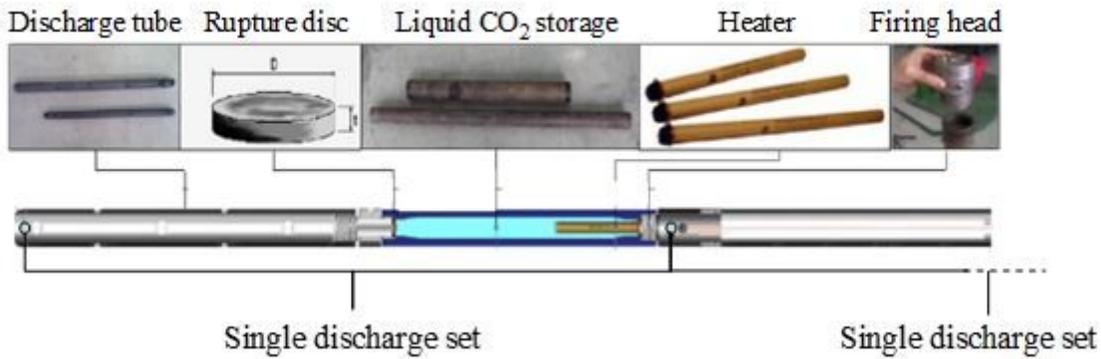
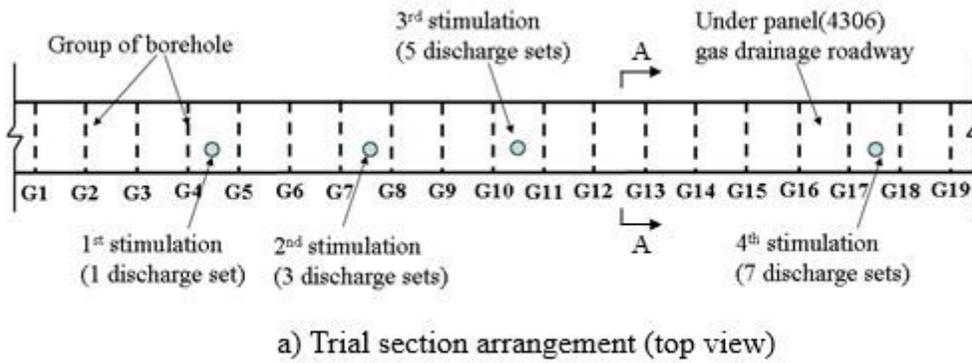
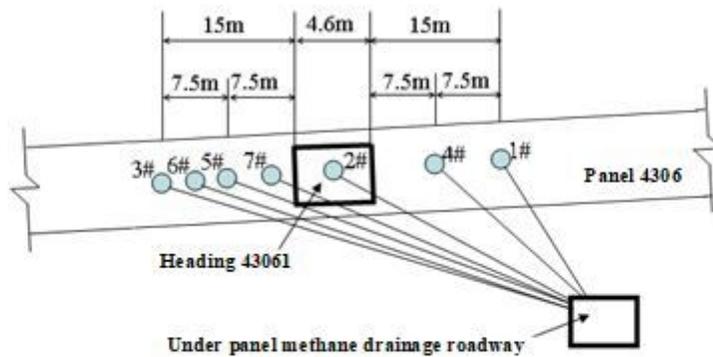


Figure 1

Structure of multi-discharge set (2 discharge sets)



a) Trial section arrangement (top view)



b) Under panel 4306 cross strata drilling design(A-A section)

Figure 2

Coal seam fracture stimulation arrangement

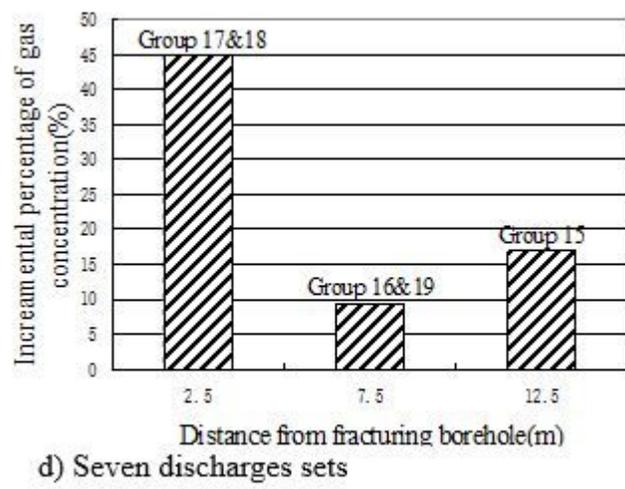
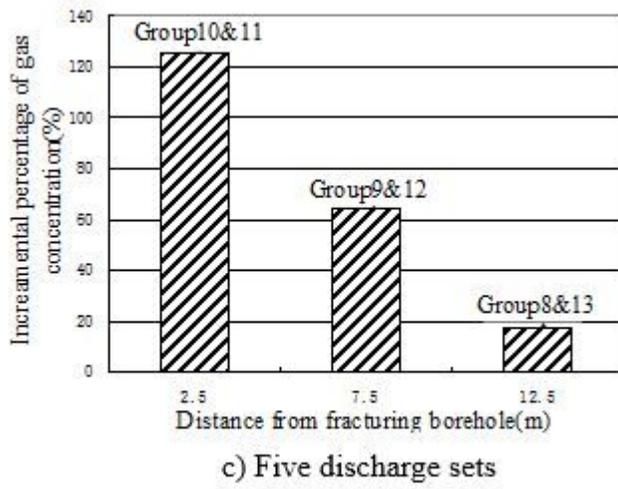
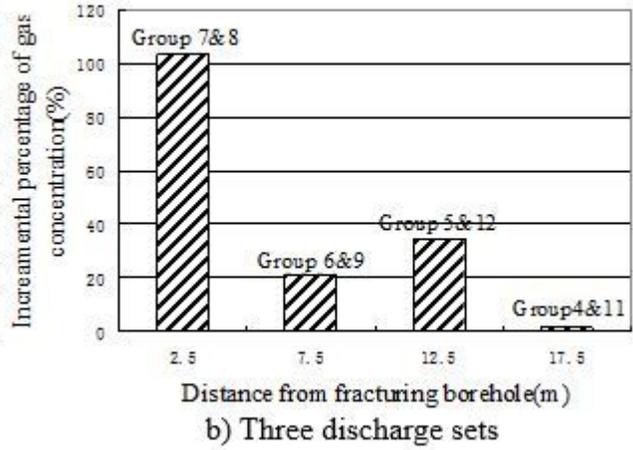
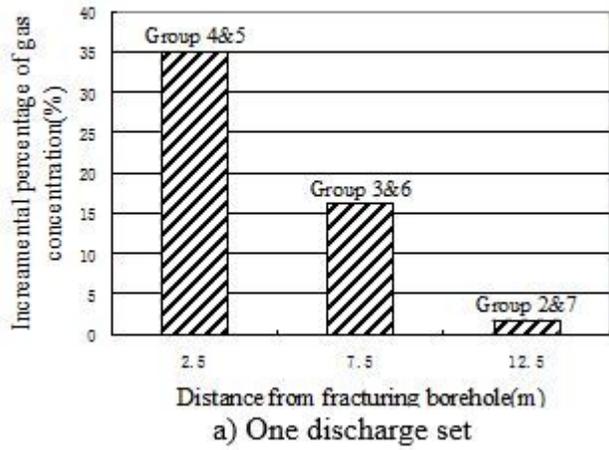
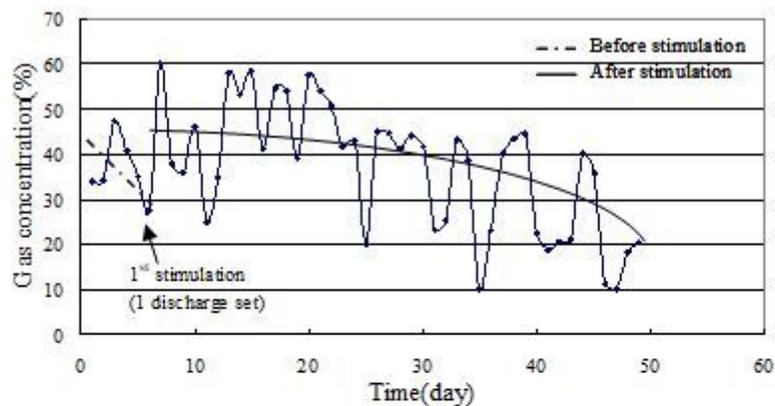
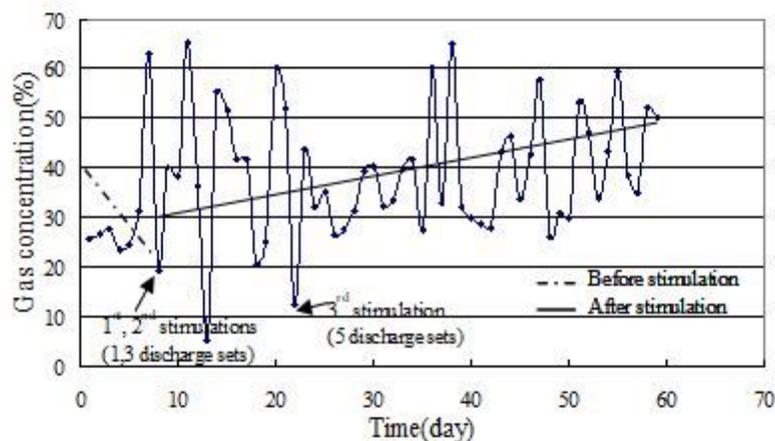


Figure 3

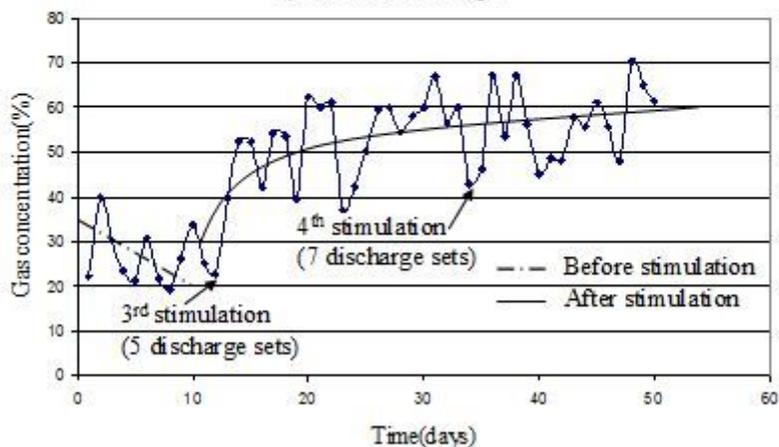
Incremental percentages of methane concentration with various numbers of discharge sets



a) Borehole 2, Group 2



b) Borehole 3, Group 7



c) Borehole 1, Group 17

Figure 4

Effect of time on coal seam gas drainage

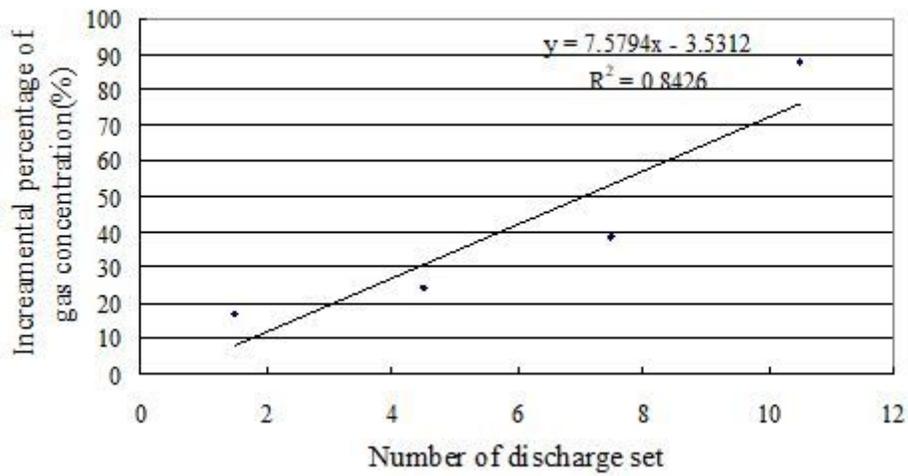


Figure 5

Discharge set number vs incremental percentage of methane concentration

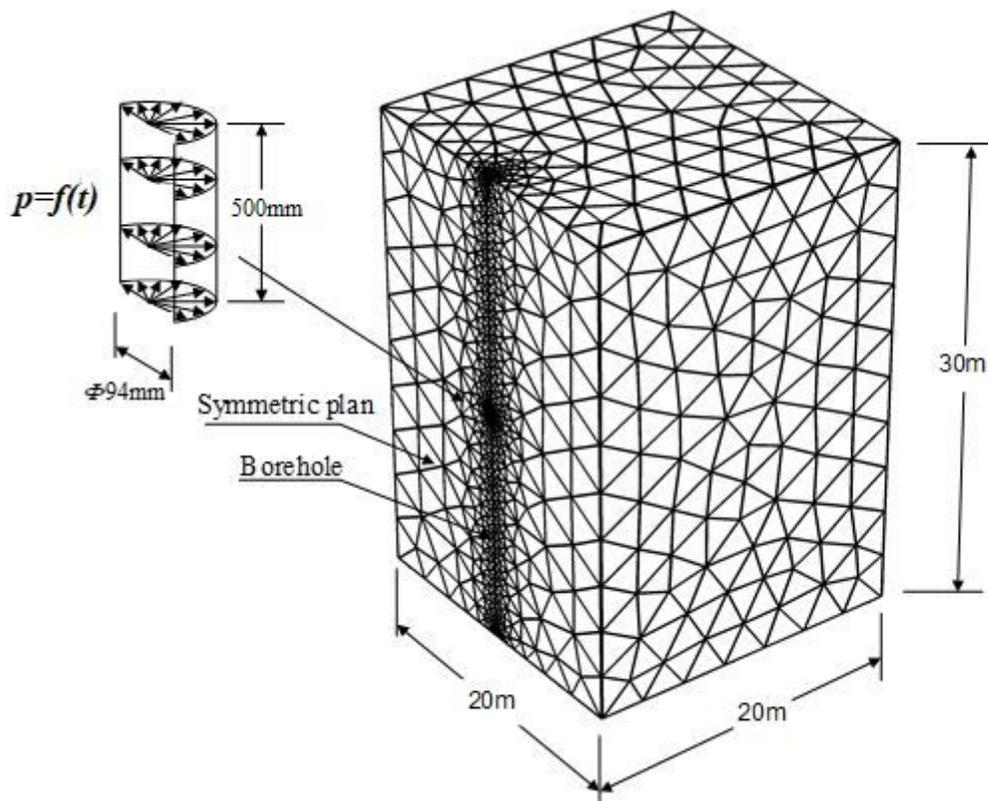


Figure 6

Model and mesh

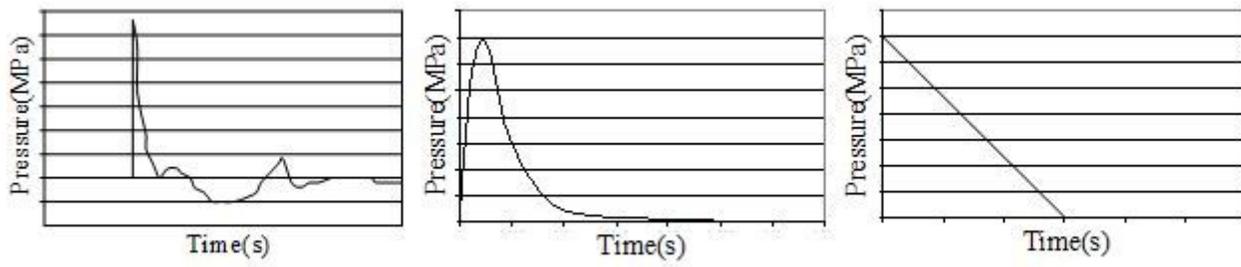


Figure 7

Decaying paths in different types of media

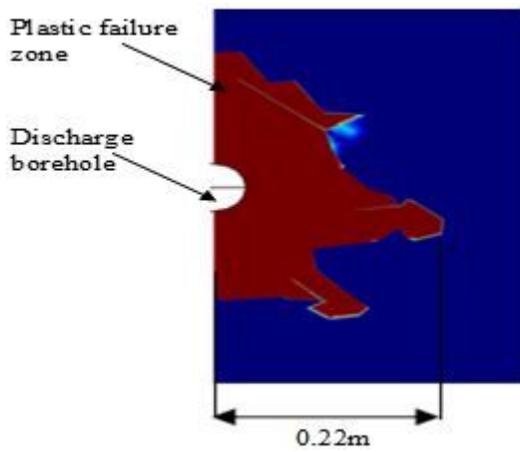


Figure 8

Plastic failure around borehole after discharge

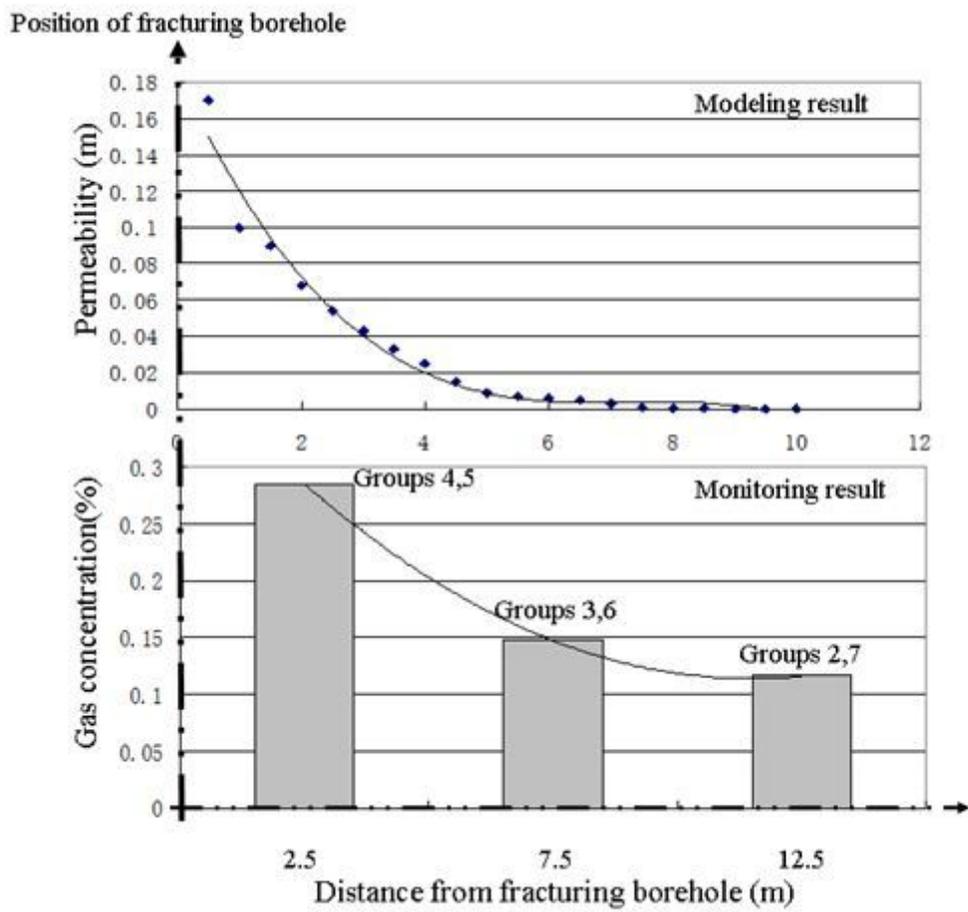
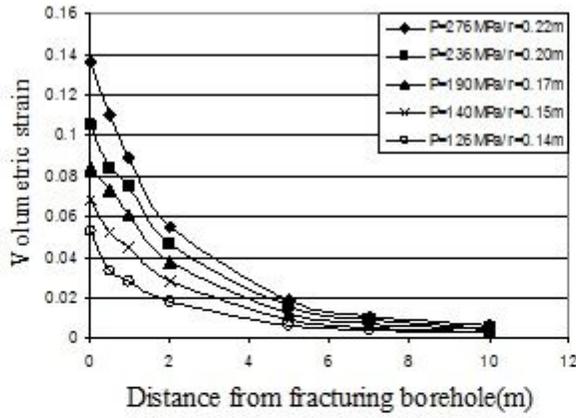
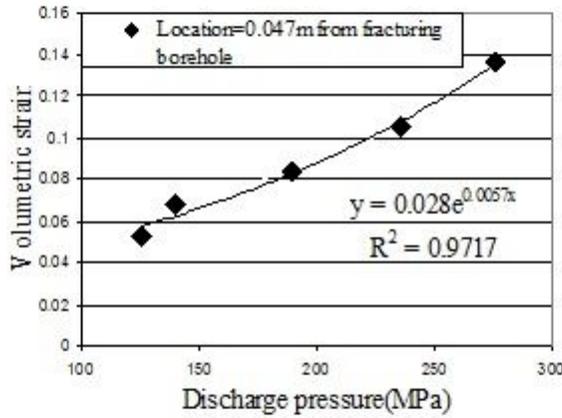


Figure 9

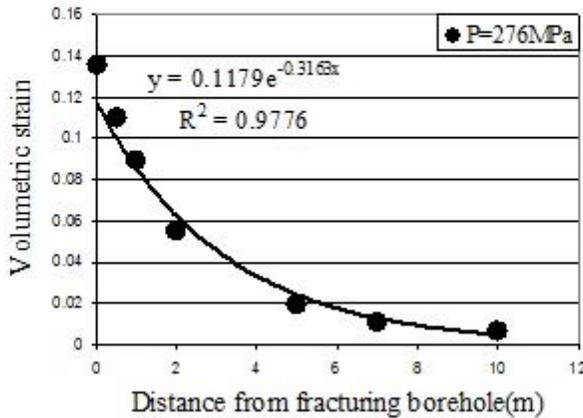
Variations of simulated permeability and monitored gas concentration



a) Volumetric strain distribution in coal seam with various discharge pressures



b) Volumetric strain vs. discharge pressures



c) Volumetric strain distribution in coal seam with max discharge pressure

Figure 10

Volumetric strain of coal body vs discharge pressure

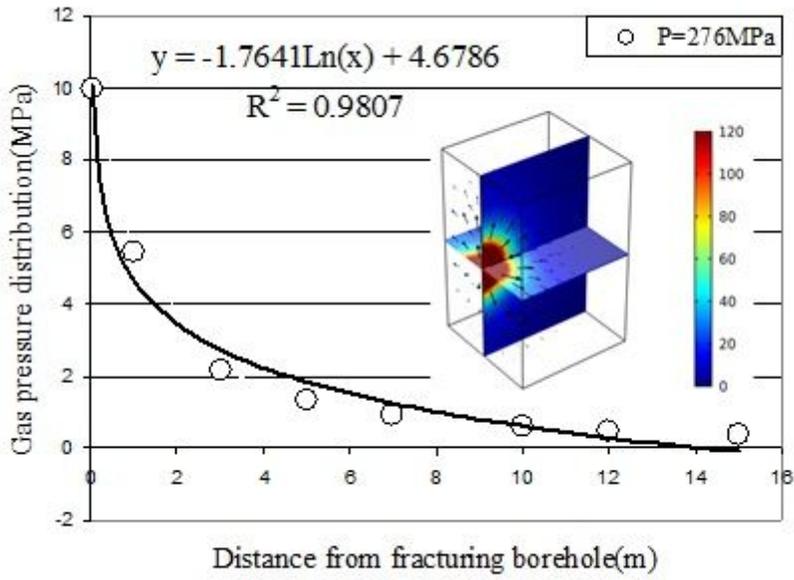


Figure 11

Gas pressure distribution of coal seam after discharge

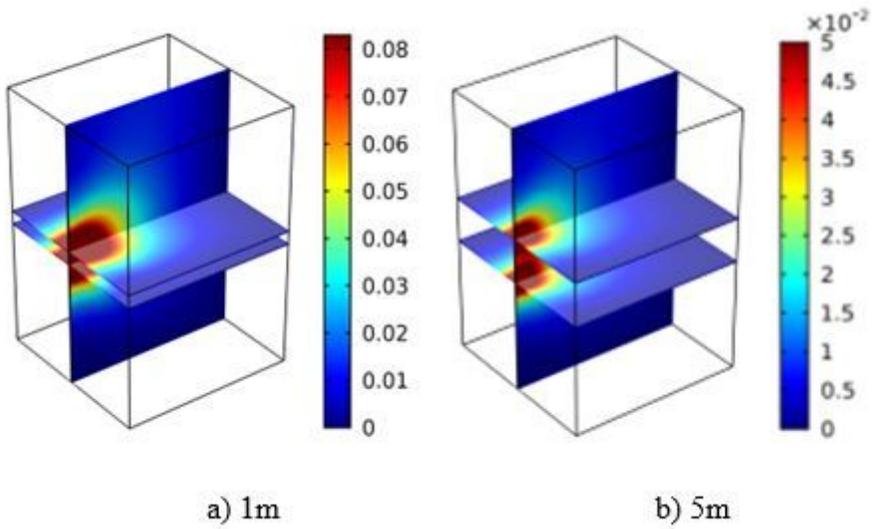


Figure 12

Effect of space between two discharge heads on coal seam deformation

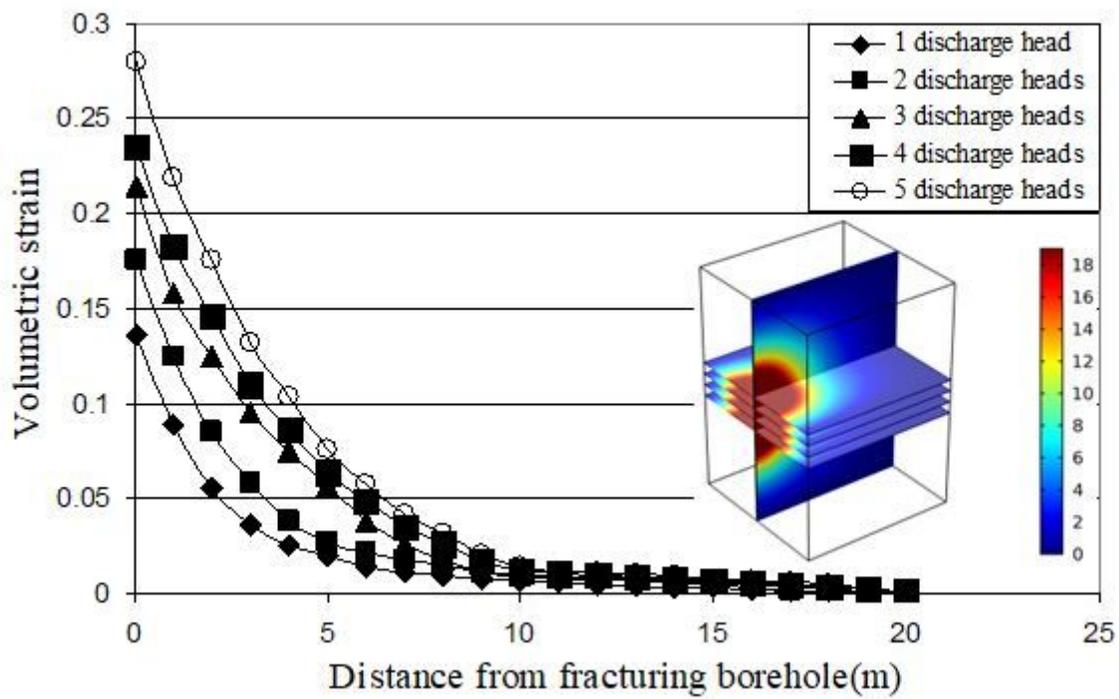
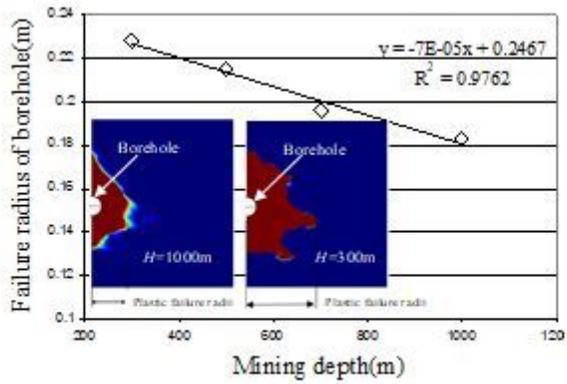
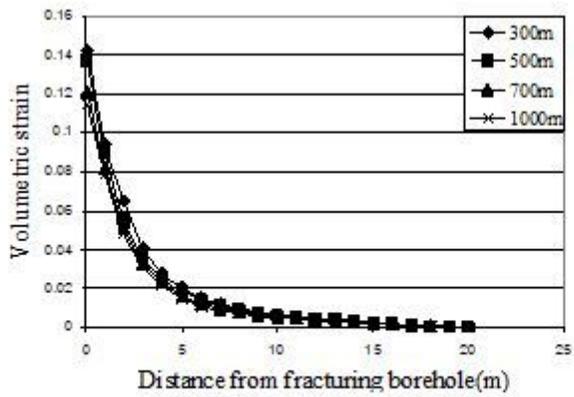


Figure 13

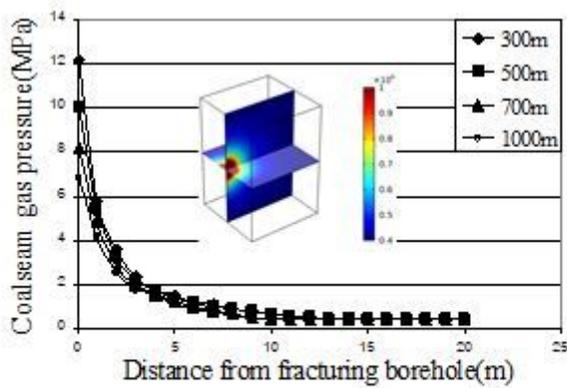
Discharge numbers vs coal seam deformation



a) Failure character vs mining depth



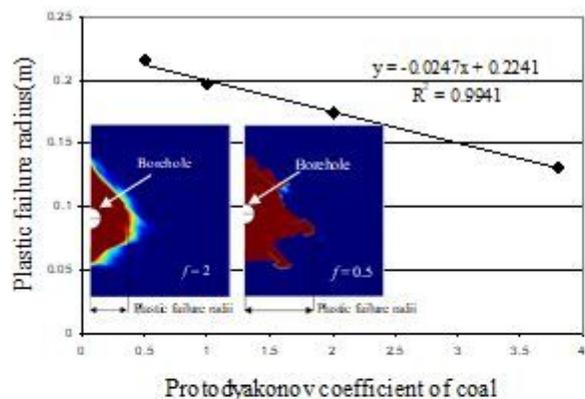
b) Coal seam volumetric strain vs mining depth



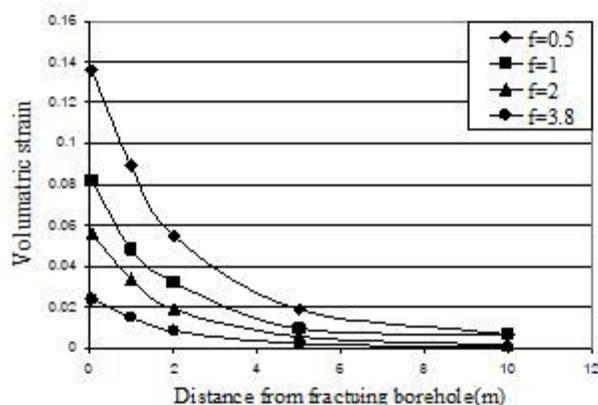
c) Internal gas pressure vs mining depth

Figure 14

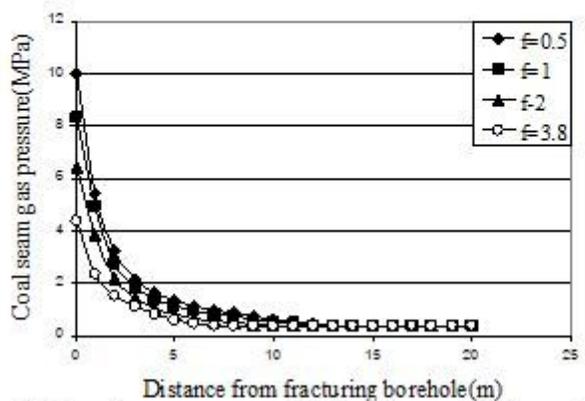
Effect of mining depth on borehole failure and gas pressure within coal seam



a) Coal seam failure vs strength



b) Coal seam deformation vs strength



c) Internal gas pressure distribution vs coal strength

Figure 15

Effect of coal strength on deformation and failure