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Simulation of the hazard arising from the coupling of gas explosions and spontaneously combustible coal due to the gas drainage of a gob

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ABSTRACT

The spontaneous combustion of residual coal and potentially subsequent gas explosions that occur during gas drainage of the gob were investigated. The 1262 fully-mechanized mining face of the Dingji Coal Mine (Huainan, China) was used as a research benchmark. The methane and oxygen distributions were simulated as well as the temperature field associated with different gas-drainage models. Using a steady-state simulation method for temperature cut-off, linear superimposition of various parameters was used to determine the hazardous zones arising from the coupling of gas and spontaneously combustible coal. The results show that when gas drainage is applied, the range of oxidation zone is different with different gas drainage modes. Among three different drainage modes, when applying the buried pipes and cross-measure boreholes mode, the methane concentration near the working face and in the upper corner of the return airway not only can be effectively reduced, but the size of the oxidation zone in the gob is the smallest. Based on the model of buried pipes, cross-measure boreholes and surface wells, the coupled gas–coal hazard zones were derived. The results show that the coupled hazard zone shifts to the deeper parts of the gob as a result of gas drainage, and the scope of the hazard zone is enlarged in the strike, dip, and vertical directions to different extents. The cooling effect is better when the nitrogen is injected into the deep part of the gob rather than the shallow part. Furthermore, nitrogen injected from double boreholes in the deep part appears to exert the largest cooling effect.

1. Introduction

Methane (CH_4) in coal seams is one of the foremost factors leading to major underground accidents in coal mines (Jiang et al., 2015; Xia et al., 2015a; Karacan et al., 2011; Yan et al., 2016). The first three quarters of 2015 witnessed 25 gas explosion accidents in Chinese coal mines and the death of 89 workers (Tang et al., 2015). If the CH_4 in the coal masses ahead of a working face is not effectively drained in the coal mining process, coal and gas outburst (hereinafter referred to as 'outburst') accidents are more likely to occur, and the large amount of CH_4 gushing out during an outburst will likely result in an explosion if there is an ignition source present (Qu et al., 2016; Zheng et al., 2016; Guo and Yuan, 2015; Yan et al., 2018). In mines where the coal has a high gas content, the residual CH_4 in the gob is one of the primary sources of CH_4 gushing out from the working face (Ediz and Edwards, 1991; Luo and Li, 1998; Chu et al., 2010). CH_4 is likely to accumulate in the upper corners of the working face and its concentration will therefore surpass the allowed limit in the return airflow if the drainage of gas from the gob does not meet the required standards. Under these conditions, a gas explosion is very likely to occur if there is an ignition source (Karacan, 2015; Yuan and Smith, 2008). Unfortunately, this problem has become extremely serious in recent years. With the constant need to mine for coal resources, the mining depth employed has increased at the rate of 10–20 m annually in China (and even up to 50 m yearly in some regions) (Wang et al., 2017; Fan et al., 2018; Wang et al., 2018). An increase in mining depth undoubtedly aggravates the severity of gas hazards, especially in seams where the coal has high gas content and is prone to spontaneous combustion.

More than one third of the large-scale coal mines in China contain coal with high gas content and a tendency to undergo spontaneous combustion. This proportion is likely to increase as the mining depth increases (Zhou, 2012; Liang et al., 2018). Gas explosions frequently occur due to the tendency of the residual coal in

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the coal seam to undergo spontaneous combustion. For example, in 2012, 48 workers were killed by a gas explosion in a coal mine in Sichuan province (China) which was triggered by the spontaneous combustion of coal (Yang et al., 2014). The risk of a gas explosion being induced by the spontaneous combustion of residual coal in the gob is most likely to be ignored and is also one of the most difficult problems to deal with (Ren and Balusu, 2009). The hazard arising due to the coupling of gas in the gob and spontaneous combustion of coal has become a recipe for disaster in coal mines and has led to the occurrence of some major accidents (Luo and Li, 1998). At present, various approaches have been used to drain the gas from the gob. The main methods involve the use of buried pipes (BPs), which are applied to the upper corners of the face and drilling cross-measure (CM) boreholes and surface wells (SWs) for gas drainage, depending on the amount of gas emitted (Qu et al., 2016; Schatzel et al., 2012; Whittles et al., 2006; Karacan et al., 2007; Guo et al., 2015; Xu et al., 2010; Chu et al., 2011).

As the gas is drained, a negative pressure gradient is produced which causes an increase in air leaking back into the system. As a result, the residual coal becomes more prone to spontaneous combustion (Qin et al., 2016; Zhang et al., 2013; Yang et al., 2018). As the coupled gas–coal combustion hazard is caused by the combined effect of the coal fracture field, temperature field, and gas concentration fields (of CH_4 and O_2) in the gob, the whole disaster process involves physical changes and chemical reactions (Zhou, 2012; Liang et al., 2017). Therefore, simply carrying out physical experiments (as is commonly done) will not take into account all of the factors of influence. In contrast, using computational fluid dynamics (CFD) to construct a model of the gob is known to be an effective research method (Yuan and Smith, 2009; Xu et al., 2010; Li, 2008). Some outstanding research results have been obtained concerning the mechanism of occurrence and factors influencing coupled gas explosion–coal spontaneous combustion hazard using the CFD technique. Previous research has clearly shown that increasing the ventilation volume dilutes the CH_4 concentration in the return airway. However, it also augments air leakage back into the gob (Xia et al., 2016; Yuan and Smith, 2008). Furthermore, the larger the air leakage is, the higher the temperature of the coal due to self-heating, and the faster the oxidation rate. This gives a larger oxidation zone and the time needed to induce the spontaneous combustion of the coal is shortened (Xia et al., 2016; Xia et al., 2015b; Qin et al., 2012; Yuan and Smith, 2009; Tan et al., 2011). Although increasing the speed of advance of the working face will significantly decrease the time for oxidation of coal, slowing down the rate of spontaneous combustion of residual coal, the CH_4 concentration is apt to exceed the limit at the upper corners of the return side due to increasing CH_4 leakage (Xia et al., 2016; Taraba and Michalec, 2011). In China, if the methane concentration increases up to 2 v/v% in a volume larger than 0.5 m^3, mining operation must be stopped (CSAWS, 2016). Draining gas from the gob can effectively solve the problem of above-limit levels of gas in the return airflow and at the upper corner. However, it also increases the number of air-leakage points in the gob and therefore enlarges the rate of air leakage (Chu et al., 2010; Zhang et al., 2013; Chu et al., 2012; Xia et al., 2017). This further increases the risk of spontaneous combustion of the coal and can trigger gas explosions at certain CH_4 concentrations. Nevertheless, in order to help prevent gas disasters in coal mines with high gas content, it is essential that gas is extracted from the gob. Additionally, a different approach is often taken: to inhibit the spontaneous combustion of the residual coal by injecting certain substances (generally water, grout, gel, or inert gases) into the gob from the working face during gas drainage (Cheng et al., 2016; Cheng et al., 2017; Song and Kuenzer, 2014; Wang et al., 2016). Among these substances, nitrogen is the one that has been extensively used in coal mines due to its effectiveness and low cost (Luo et al., 1997; Li et al., 2005; Ren and Balusu, 2009).

At present, the problem of preventing the occurrence of hazards due to gas–coal coupling is in need of extensive research. When multiple drainage methods are simultaneously used, the hazard zones arising due to the coupling between the coal fracture field, temperature field, and gas concentration fields (of CH_4 and O_2) are bound to change significantly as a result of the drainage process. Currently, however, gas-drainage simulations mainly consider just a single drainage mode. Therefore, the distribution of coupled gas–coal hazard zones in the gob needs to be further studied when multiple gas-drainage modes are employed. Proper fire prevention and extinguishing methods also need to be developed (based on nitrogen-injection and other methods) according to the distributions of gas and spontaneously combustible coal in coupled hazard zones. In this study, a 3D CFD model was established for the gas drainage system used in the 1262 working face of the Dingji Coal Mine operated by the Huainan Mining Group (Huainan, China). On this basis, the distribution of CH_4 and O_2 was analyzed, as well as the temperature field in the gob, when different gas drainage models were employed. Moreover, the distribution of the hazardous zones in the gob was studied using a method of linear superposition of various physical parameters including temperature, and gas concentration (of CH_4 and O_2). Nitrogen-injection schemes for the prevention and extinguishing of fires were also discussed.

2. Establishing a CFD model for the gob

2.1. Background information

The Dingji Coal Mine is located in the northwest of Huainan city. It has two main mineable coal seams: 13-1 and 11-2. The 13-1 coal seam is 0.50–10.68 m thick (3.7 m on average), in which the maximum gas content is 15.86 m^3/t which largely exceeds the critical value (8 m^3/t) of gas content of outburst-prone coal seam (CSAWS, 2009), presenting a high level of outburst risk. Additionally, the seam has a low gas-permeability coefficient, so it is difficult to predrain the gas. If the 11-2 coal seam is first mined, the permeability of the 13-1 coal seam will greatly increase due to the effect of stress relief induced by the mining action. This thus improves the gas drainage efficiency, which will reduce the gas content of the coal seam, and then decrease the level of outburst risk. Therefore, as a preventive measure, the 11-2 coal seam was mined prior to the 13-1 coal seam and thus the former was acting sort of as a protective layer. (With a distance of 75 m between the two coal seams, the situation more formally corresponds to long-distance protective-layer mining.) The 11-2 coal seam is 0.36–6.05 m thick (2.19 m on average) and has a simple structure in which the properties of the coal change only slightly. The coal seam here is also prone to spontaneous combustion and the spontaneous ignition period is 35 d. As for the primary mining face of the 11-2 coal seam, the 1262 working face is 500 and 203 m long in the strike and dip directions, respectively. Its burial depth is in the range −790–−825 m. As the entire working face is located at a broad and gentle syncline, the coal seam in the region hardly fluctuates with dip angles in the range 0–8° (4° on average), in which the average coal thickness is 3.3 m and its gas content is ~6.2 m^3/t; the absolute gas emission rate is 50 m^3/min. A U-type full negative-pressure ventilation regime was adopted in the working face with an air supply amounting to 2160 m^3/min. To ensure gas was extracted efficiently, multiple gas drainage modes were employed in the gob. Two surface wells (SW1 and SW2) for gas drainage were drilled in the gob of the 1262 working face. The bottom of SW1 was located 90 m from the open-off cut, 76 m from the return airway, and 127 m from the intake airway. The highest methane drainage concentration and daily pure methane content