Analyses and prevention of coal spontaneous combustion risk in gobs of coal mine during withdrawal period

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Analyses and prevention of coal spontaneous combustion risk in gobs of coal mine during withdrawal period

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ABSTRACT
Coal spontaneous combustion (CSC) in gob causes notorious safety issues to workers, especially during withdrawal period. Withdrawal period was normally divided into three stages (initial stage, expanding back channel stage and equipment returning stage) when evaluating and preventing the risk of CSC. This study first analyzed the characteristics of ventilation system at each stage, such as oxidized zone movement, local fire wind pressure and air leakage. Then, principles and specialized measurements were applied to minimize the ranges and effects of CSC in gob. In detail, this includes reducing the total air volume flow of mining face in initial stage, adding windscreen in outlet way and local fan in inlet way to increase the pressure of working face, and lowering external air leakage. Considering the working face ventilation area shrinks at equipment returning stage, keeping it bigger than windscreen’s areas can help to reduce air leakage. Further, foamed gel was grouted into gob to narrow and stop the moving forward of oxidized zone. Field application was applied in Zhang Shuang-lou coal mine. Findings of these studies could be used in other coal mine with similar conditions during withdrawal period.

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KEYWORDS
Coal spontaneous combustion; withdrawal; fire wind pressure; ventilation system; foamed gel

1. Introduction
Coal is currently the most commonly used energy source in the world (Jennifer et al. 2018; Li et al. 2018; Peña et al. 2018; Reisen et al. 2017; Shi et al. 2018). According to state statistics, coal, with 3.433 billion tons production, accounts for 61.83% of disposable energy consumption in 2016, far more than 18.9% of petroleum consumption (Ren et al. 2009; Wang 2010; Yu 2014). However, lethal disasters exist in the process of mining. Coal spontaneous combustion (CSC) is among the leading causes of tragedies, such as gas and coal dust explosion and mortality of workers, and the potential...
economic loss and environmental pollution (Ren and Wang 2012; Ren et al. 2016; Bhattacharjee et al. 2012). In March 2013, two continuous gas explosions triggered by CSC in gob killed 53 people at Babao Coal Mine in Jilin northeast province of China. Incomplete statistics from 2001 to 2014 suggested that there were about 32 cases of gas explosion or fire disasters underground Coal Mines resulting from CSC that led up to 614 death (Zhou 2012; Wang et al. 2014). The Chinese government and research institutes have devoted considerable financial and material efforts on CSC suppression in gobs (Song and Hao 2008; Ye et al. 2017).

Withdrawal of working face is one of the key and must processes for coal mines (Hao et al. 2012; Wen et al. 2014). Equipment, such as shearer, scraper conveyor, crusher, hydraulic support and other devices, is required to be returned for maintenance after finishing working face, and recycled for the next working face. In China, a 150 m long working face, the total value is near to ¥0.15 billion ($23.2 million), which accounts for an important part of coal mine interest. During withdrawal, CSC, roof collapse, gas transfinite and ventilation disorder are the four high-triggered hazards. On 6 January 2018, a worker was killed by a sudden broken wire rope during this withdrawal period in Hai Shiwan coal mine Gansu province. Qinyuan coal mine in Shanxi province had sealed four working faces because of the CSC occurring withdrawal period and hydraulic supports not able to get back, which resulted in millions loss in 2015–2016.

Literature reviews indicated that majority studies focused on implementation of safety strategy during the process of mining (Liu et al. 2014; Jin et al. 2015; Li et al. 2016; Zhang et al. 2016). However, withdrawal period faced the highest chance of CSC in gobs and was much more complicated due to unstable ventilation system and longtime stopping (Ren et al. 2016). Further, with the breakthrough of deep mining, high ground stress and geothermal are more favourable for CSC events. XIE et al applied local ventilation flow control to prevent the diffusion of CO during withdrawal period, but they ignored the instability of ventilation system and its limitation in fire prevention (Xie et al. 2002). Chen and Hou presented comprehensive methods to prevent CSC in easy-ignition coal seam gobs, but the working face was single seam and they did not analyze how air leakage affected the developing of CSC (Chen and Hou 2012). Liu and Qin (2017a, 2017b) indicated that CSC in gob was the result of interaction of multi-physical fields, including air seepage and heat transfer. Among those factors, ventilation system plays the most significant role, because the stability of ventilation system is the premise for safety mining. Some researches divided the gob into three zones, where the area with the risk of CSC is identified as ‘oxidized zone’, but it is not possible to locate an accurate fire position for prevention (Ren et al. 2015; Cheng et al. 2017; Wang et al. 2014). In summary, there are still some severe issues need to be considered on CSC prevention during withdrawal period, for example the integrated technique to follow. In addition, challenges still exist in stabilizing the ventilation system and control CSC. As a result, the main objective of our study is to identify the ventilation change and choose a feasible way to control CSC during withdrawal period. The findings probably could be used in other coal mine under similar conditions during withdrawal period.

In this study, we analyzed the causes of CSC during withdrawal period. The whole process of withdrawal was divided into three parts: initial stage, expanding back
channel stage and equipment returning stage. Ventilation changing laws correspond-
ing to the three steps were presented. Meanwhile, risk area of gob was narrowed
within 20 m back the working face. Foamed gel technology was used for CSC preven-
tion. Field application was implemented in Zhang Shuang-lou Coal Mine in Xuzhou
Jiangsu province.

2. Characters of CSC during withdrawal period

2.1. Causes for CSC

Coal seams with spontaneous combustion tendency in developing into self-ignition
when gathering four factors: plenty of broken coal; continuous air supplying; heat
storage and enough time for coal’s oxidation (Gulyaeva and Arikan 2017). Causes for
CSC during withdrawal period are as Table 1.

First, at the final time of mining, top coal, around 50 m, is not caved to keep the
roof stable, resulting in plenty of coal in gob ignition resource. Second, it takes a
long time to excavate returning path and return equipment, which provides sufficient
time for coal oxidation. Further, the heat generated by oxidation accumulates and
reinforces coal oxidation in return. At last, ventilation system may disorder with roof
collapse caused by hydraulic supports moving. According to our empirical study,
CSC in gob could be set three grades based on withdrawing time (T) and spontan-
eous combustion period (t) as Table 1.

We should predict the process of CSC and take specific methods according to the
grades. The most affected aspects are ventilation system and CSC areas.

2.2. Movement of oxidized zone

Gob is generally divided into three zones based on O₂ concentration (φ). They are
non-spontaneous combustion zone (φ > 15%), oxidized zone (15% > φ > 5%) and
asphyxiation zone (φ < 5%). Oxidized zone is the most dangerous area for possible
developing CSC (Wang 2010; Zhang et al. 1998; Mishra et al. 2011). The ranges of
oxidized zone vary with gobs, but we assume that it is constant for a given gob.
However, oxidized zone would move forward if the working face stopped a long
time, which make it worse for CSC control and prevention.

Permeability distribution of gob determines the flow-diffusion law of oxygen,
affects the range of oxidized zone, and further applies to the oxidation exothermic of
coal, and finally determines the distribution of temperature field.

The relationship between broken expansion coefficient α and porosity k of gob is:

\[ k = 1 - 1/\alpha \]  (1)

Which means k and α are positive correlation (Zhang et al. 2012).

<table>
<thead>
<tr>
<th>T/d</th>
<th>0.8t</th>
<th>1t–1.2t</th>
<th>&gt; 2t</th>
</tr>
</thead>
<tbody>
<tr>
<td>Grade</td>
<td>Safe</td>
<td>Dangerous</td>
<td>Disaster</td>
</tr>
</tbody>
</table>
Based on the previous studies, $\alpha$ can be expressed as Equation 2:

$$\alpha = \alpha_0 + \lambda \frac{1}{\tau}$$

The expression of permeability and porosity in gob is as Equation 3:

$$k = \frac{D_p^2 n^3}{150 (1-n)^2}$$

Where: $\alpha_0$: The minimum broken expansion coefficient. $\lambda$: Correction coefficient. $\tau$: Time of withdrawal. $D_p$: The average diameter distribution of coal and rock in gob; $n$: Permeability of gob.

From Equations 1 and 2, we know that when the porosity is small, permeability decreases with porosity (Peña et al. 2017; Oliveira et al. 2018; Zhou 2017). $O_2$ distribution in gob is influenced by porosity. Air leakage velocity is affected by permeability. Moreover, CSC in gob is a result of oxidation of coal residue under suitable air leakage velocity $(v_{\text{air}})$ and oxygen concentration $(\phi)$. Thus the oxidized zone in gob shrinks and moves forward as the reduction of air leakage velocity and $O_2$ concentration (Figure 1). It is confirmed by field applications.

### 2.3. Ventilation system changing laws

#### 2.3.1. Air leakage

In ventilation system (Figure 2), the most important factor is air leakage ($\Delta Q$) in gob, including external leakage ($\Delta Q_E$) and internal air leakage ($\Delta Q_I$) (Equation 4).

$$\Delta Q = \Delta Q_E + \Delta Q_I$$

In ventilation system, air volume follows Equation 5:

$$Q = Q_W + \Delta Q_I$$

Where: $Q$, intake air volume; $Q_W$, air volume in working face. Air leakage varies with withdrawal periods.
1. Initial stage: the airflow ($Q$) is smaller than in process of mining, so air leakage is small.

2. Expanding back channel stage: Ventilation resistance decreases significantly in the second stage. Based on ventilation resistance law, it can be easily deduced that $Q_W$ increases. As $Q$ is constant, $\Delta Q_I$ would decrease correspondingly. However, the reduction of total pressure in working area leads to the increase of $\Delta Q_E$, which promotes the risk of CSC in gobs.

3. Equipment returning stage: Collapsed coal and rock are silted the working face at this stage, which increase the ventilation resistance and pressure of working area. As a result, $\Delta Q_E$ would decease, $\Delta Q_I$ increases on the contrary, which leads to severe CSC risk.

2.3.2. Local fire wind pressure

Local fire wind pressure is another potential disaster during withdrawal period. Shown as Figure 3, there are two ways for ventilation in declined working face: upward and downward ventilation.
In inclined roadway, Local fire wind pressure can be calculated as Equations 6 and 7:

\[ H = gz(q_1 - q_2) \]  
\[ \Delta h = h - H \]

Where: 
- \( H \): Local fire wind pressure, Pa. 
- \( g \): Gravitational acceleration, m/s\(^2\). 
- \( z \): Height difference, m. 
- \( q_i \): Air density at position \( i \), kg/m\(^3\). 
- \( h \): Ventilation pressure, Pa. 
- \( \theta \): Angle, \(^\circ\). 
- \( \Delta h \): Pressure difference, Pa.

Pressure difference determines the flow direction, and it will not be changed in upward ventilation by local fire wind pressure. However, for downward ventilation, if \( \Delta h < 0 \), flow reversal would happen and the mine ventilation system will be disrupted.

To minimize the influences from local fire wind pressure, lower air leakage, and reduce the risk of CSC, pressure balancing technology was used. Foamed gel was also grouted into gob to seal air channel and stop the movement of the oxidized zone.

### 3. Pressure balancing technology and foamed gel technology

#### 3.1. Theoretical analyses based on the process of withdrawal

Ventilation system of mining face is simplified as Figure 4. The explanations of branches are shown in Table 2. Premises are follows:

1. External air leakage channel is regarded as a ventilation branch;
2. Mining face is divided into two parallel branches.

Based on ventilation resistance law, we have:

\[ h = RQ^2 \]
In Figure 4, the air volume matches Equation 9:

\[ Q_6 = Q_2 + Q_5 = Q_3 + Q_4 + Q_5 \]  

(9)

According to ventilation resistance law:

\[ Q_5 \propto \Delta h_5 \]  

(10)

Where \( Q_i \) (\( i = 1, 2, 3 \ldots 7 \)) represents air flow volume of each branch. \( \Delta h_5 \): Pressure difference between upper gob and gob.

### 3.1.1. Expanding back channel stage

As described above, external air leakage will increase in this stage, which aggravates the risk of CSC in gobs of mining face.

To reduce the external air leakage is to add pressure of working face. So a regulation windscreen was installed in branch 6, and a local fan was installed in branch 2. Pressure distributions in working face are shown in Figure 5.

### 3.1.2. Equipment returning stage

In this stage, wooden cribs were used temporarily for roof supporting which can lower the ventilation areas in branch 3, and its ventilation resistance increased while branch 4 kept constant. Air flow differences of branch 2, 3 and 4 matched with Equation 11, which indicate that internal air leakage increased with the ventilation in branch 3.

\[ \Delta Q_4 = \Delta Q_2 - \Delta Q_3 > 0 \]  

(11)

To balance the ventilation system, we took following two measures:

i. Expand ventilation areas of branch 3.
ii. Use regulation windscreen in branch 6 to enhance the pressure of working face.
The key point is how to control the ventilation areas of branch 3 and 6. After taking i and ii measures, air flow volume changes as Equation 12:

$$\Delta Q_1 = \Delta Q_3 + \Delta Q_4 + \Delta Q_5$$  \hspace{1cm} (12)

The reduction of ventilation resistance between node 2 and 5 (whole working face) is caused only by measure i:

$$\Delta R_{2-5} = R_{2-5}' - R_{2-5}$$  \hspace{1cm} (13)

After taking measure ii, the ventilation resistance increases between node 2 and 5:

$$\Delta R_{2-5}'' = R_{2-5}'' - R_{2-5}$$  \hspace{1cm} (14)

where: $R_{2-5}$ is the original ventilation resistance before taking measure i; $R_{2-5}'$ is the ventilation resistance after taking measure i; $\Delta R_{2-5}'$ is the difference value; $R_{2-5}''$ is the ventilation resistance after taking measure ii; $\Delta R_{2-5}''$ is the difference value.

Air leakage reduction can be explained by following expressions:

1. $\Delta Q_4 < 0$, represents the reduction of internal air leakage.
2. $\Delta Q_5 < 0$, represents the reduction of external air leakage.

Reducing internal air leakage is easily achieved by expanding the ventilation area of branch 3, just as described in measure i.
For $\Delta Q_5 < 0$, we should keep:
\[
\Delta R_{2-5} + \Delta R_{2-5}'' > 0 \tag{15}
\]

Ventilation resistances of each branch match Equation 16:
\[
\begin{align*}
R_{2-5} &= R_2 + R_7 + \frac{R_3 R_4}{R_3 + R_4} \\
R_{2-5}' &= R_2 + R_7 + \frac{R_3' + R_4}{R_3' R_4} \\
R_{2-5}'' &= R_2 + R_7' + \frac{R_3'' R_4}{R_3'' + R_4}
\end{align*} \tag{16}
\]

Where: $R_i$ is the ventilation resistance of each branch; $R_i'$ and $R_i''$ represent the ventilation resistance of each branch after taking measures i and ii, $i = 1, 2, 3 \ldots 7$.

From Equation 16, we get:
\[
\Delta R_{2-5} + \Delta R_{2-5}'' = R_4^2 \frac{\Delta R_3'}{(R_4 + R_3')(R_4 + R_3) + \Delta R_7'} \quad \tag{17}
\]

Combine the Equations 15 and 17:
\[
\frac{R_4^2}{(R_4 + R_3')(R_4 + R_3)} < \frac{\Delta R_7'}{\Delta R_3'} \tag{18}
\]

Where, $\Delta R_3'<0$. And
\[
\frac{R_4^2}{(R_4 + R_3')(R_4 + R_3)} = \frac{1}{(1 + R_3'/R_4)(1 + R_3/R_4)} < 1 \tag{19}
\]

If $\Delta R_7'/|\Delta R_3'| > 1$, Equation 20 is correct:
\[
\frac{R_4^2}{(R_4 + R_3')(R_4 + R_3)} = \frac{1}{(1 + R_3'/R_4)(1 + R_3/R_4)} < 1 < \frac{\Delta R_7'}{|\Delta R_3'|} \tag{20}
\]

Then $\Delta Q_5 < 0$.

In field application of withdrawal process, $\eta = \Delta R_7'/|\Delta R_3'|$ could be a standard that indicates the change of external air leakage, there are two situations:
\[
\eta = \begin{cases} 
> 1 & \text{Reduction of external air leakage} \\
\leq 1 & \text{Increase of external air leakage} 
\end{cases} \tag{21}
\]

In ventilation system, $R = \pi LU/S^2$.

Where: $R$: Friction, (kg/m$^2$); $\pi$: Friction factor, (kg/m$^3$); $L$: Length of roadway, (m); $U$: Circumference of roadway, (m); $S$: Ventilation area of roadway, (m$^2$).

For an intact roadway in coal mine, $S$ is the easiest factor to change for regulating wind pressure and air flow in field application.
3.2. Parameters of foamed gel

Foamed gel was used to seal air leakage channel and prevent the moving of oxidized zone. This material consists of free liquid, gel particles, and foam, which integrates the performance of both foam and gel on fire control. Gel is transported into the interspace of gobs by taking foam as a carrier, which has excellent performances of holding water, spraying, covering, and sealing air leakage channel.

Its field application flow chart was shown as Figure 6. The operating procedures are as follows. First, mixture with additive A/B is pumped proportion to the pipeline, flowing through a flow regulation windscreen to foam device. High-speed jet flow forms when the premixed solution flows through the nozzle in the foaming device. Meanwhile, nitrogen is injected into the device through nitrogen inlet, and mixes with the liquid jet generating a large amount of foamed gel.

4. Field application and effect

4.1. Developing of CSC

Field application was conducted in Zhang Shuang-lou Coal Mine, Jiangsu province in China. The roadway layout, downward ventilation system and fire areas of mining face are shown in Figure 7. Average coal seam thickness and length of fully mechanized 9421 working face, down to 7425 mining face, are 6.4 m and 320 m. Its average inclination angle is 25°, which is the main cause of slowing mining speed. Igneous rock, another induced factor of CSC, with areas of 478 m², intrudes the partial roof of 9421 working face.

The development of CSC was divided into four stages:

1. The mining distance was only 1.8m per month in 12/2013, which provided sufficient time for coal oxidation and heat producing. Grouting liquid CO₂ in sealed gobs was taken to control the risk, which has limited effect.
2. On April 2014, CO concentration increased straightly to $720 \times 10^{-6}$ in the rear of supports. Air leakage exacerbated CSC risk.
3. 9421 mining face was unsealed in May 2015, but it was stopped again due to the restriction of igneous rock, which lowered the mining velocity to 4m/s in one month and aggravated the coal self-heating. The mining face was sealed and grouted liquid CO$_2$ again.

4. As the high temperature areas were not eliminated entirely due to the insufficient sealing time, CO concentration increased fast to $700 \times 10^{-6}$ after the second unseal of 9421 mining face.

In the application process, local fire wind pressure occurred near upper corner, the velocity of air flow was near 0.1m/s, opposite to original flow direction, which reflected that CSC endangered working face severely.

4.2. Application of pressure balancing technology and effects

Based on above analyses, following measures were then used:

1. Expanding back channel stage: install windscreen in outlet way and local fan in inlet way to reduce external air leakage.

2. Equipment returning stage: First, clean working face to add ventilation areas. Second, based on the first stage, keep local fan’s working conditions constant, and regulate windscreen step by step. To guarantee $\eta > 1$ according to Equation 21, ensure that ventilation areas of outlet way is smaller than the areas of working face.

The effect after using CFA was evaluated by CO concentration shown in Figure 8.

4.3. Application of foamed gel

The arrangement of high level boreholes was shown in Figure 9. There were two rows of boreholes, covering the areas of suspected fire areas. The inner diameter of borehole was 75 mm and the final positions were about 10 m and 20 m away from hydraulic support. Table 3 are parameters of holes.
The broken rock in gobs gradually compacted under the influence of surrounding rock pressure, and the oxidation space spread from the deep area to the shallow, which caused severe fire risk. To prevent and minimize the damage of coal fire, additional boreholes were installed. Their final borehole position was 0.5–1 m down the coal seam roof, and about 3–5 m reared to the liquid supports. The total number was 15, and the diameter of borehole was 42 mm (Figure 10).

Table 3. High level boreholes parameters.

<table>
<thead>
<tr>
<th>No.</th>
<th>Azimuth angle/°</th>
<th>Inclination angle/°</th>
<th>Length/m</th>
<th>No.</th>
<th>Azimuth angle/°</th>
<th>Inclination angle/°</th>
<th>Length/m</th>
</tr>
</thead>
<tbody>
<tr>
<td>1–1</td>
<td>90°36′</td>
<td>18°7′</td>
<td>94.8</td>
<td>2–1</td>
<td>88°23′</td>
<td>15°5′</td>
<td>90.6</td>
</tr>
<tr>
<td>1–2</td>
<td>88°18′</td>
<td>17°35′</td>
<td>82.4</td>
<td>2–2</td>
<td>84°43′</td>
<td>13°16′</td>
<td>88</td>
</tr>
<tr>
<td>1–3</td>
<td>84°47′</td>
<td>15°54′</td>
<td>89.8</td>
<td>2–3</td>
<td>80°53′</td>
<td>11°18′</td>
<td>85.7</td>
</tr>
<tr>
<td>1–4</td>
<td>81°6′</td>
<td>15°44′</td>
<td>78.5</td>
<td>2–4</td>
<td>76°23′</td>
<td>8°39′</td>
<td>83.7</td>
</tr>
<tr>
<td>1–5</td>
<td>78°9′</td>
<td>11°28′</td>
<td>86</td>
<td>2–5</td>
<td>72°50′</td>
<td>7°0′</td>
<td>82.7</td>
</tr>
</tbody>
</table>
Two isolating strips were formed 10 m and 20 m back to the support after grouting foamed gel. The CO concentration in 9421 working face fell from $400 \times 10^{-6}$ to $50 \times 10^{-6}$ in about 5 days after grouting foamed gel. This working face was withdrawn safely eventually.

5. Conclusion

Coal spontaneous combustion is a main threat to the safety of workers during equipment withdrawal period in coal mines. To prevent this risk, comprehensive methods including pressure balancing and foamed gel were used to suppress coal fire hazard based on ventilation system analyses. Findings of this study could be used in other coal mines with similar conditions. The main findings include:

1. Withdrawal process can be divided into three stages: initial stage, expanding back channel stage and equipment returning stage. Each stage has own features on ventilation system, which influences CSC and CO distribution.

2. The movement of oxidized zone and local fire wind pressure were two potential disasters during withdrawal period. Minimizing the destruction ranges can control ventilation system. In expanding back channel stage, windscreen and local fan were installed in outlet way and inlet way to reduce external air leakage. In equipment returning stage, standard of $\eta$ was set up, which required the ventilation area of windscreen in outlet way to be smaller than that of working face. This standard could widely be used in other coal mines during withdrawal period.

3. To prevent the moving of oxidized zone, foamed gel was used to seal air leakage channel and cool high fire zones. Foamed gel integrated the performance of both foam and gel, and consisted of free liquid, gel particles, and foam. Two rows of drillings were arranged for grouting foamed gel, which could form isolation strip in gobs.

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