STUDY ON THE OXIDATION AND HEATING CHARACTERISTICS OF RESIDUAL COAL IN GOAFS UNDER DIFFERENT AIR-LEAKAGE CONDITIONS

by

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The spontaneous combustion of broken coal is one of the main causes for mine safety accidents in goafs. To determine the effect of different air-leakage conditions on the spontaneous combustion of leftover coal, the air-leakage passage of the goaf was designed based on the principle of Sudoku grid in an inflammable coal seam. The temperature rise during the auto-ignition oxidation of coal was studied using a self-built experimental platform. By changing the air-flow rate, the laws of the change in the oxygen consumption rate and the heat release intensity with the coal temperature were analyzed. Results show that the oxygen consumption rate had three obvious peaks at 48 °C, 75 °C, and 105 °C, respectively. Above 80 °C, spontaneously combustion of the experimental coal samples began. The exothermic intensity increased exponentially with the rise of temperature. Furthermore, an exponential relationship was observed between the air supply at the working face and the spontaneous combustion of broken coal in the goaf. In addition, the increase in air supply in the fully-mechanized mining face increased the width of the oxidation zone.

Key words*: goaf, air-leakage conditions, oxidation characteristics*

Introduction

Spontaneous combustion is one of the five major disasters in the goaf of coal mines. It has always been a key focus in the field of mine safety [1-3]. Air leakage is the most direct cause of spontaneous combustion of coal in the goaf. As the goaf is semi-open and dynamically changes with the advancement of the working face, the air-leakage channel is constantly changing. Existing research results on the spontaneous combustion of coal and its oxidation characteristics cannot be used to explain the correlation between spontaneous combustion and oxidation temperature rise of broken coal with working face advancement in the goaf, and nor can they be used to prevent and control the catastrophes caused by the air-leakage flow field in the dynamic goaf. Therefore, it is necessary to conduct research on the oxidation and temperature-rise characteristics of the leftover coal in the goaf to provide a basis for the prediction and control of the spontaneous combustion of coal in the goaf.

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Many previous studies have explored the spontaneous combustion of coal in the goaf. A spontaneous coal-combustion event at the Wangtaipu Coal Mine [4] was analyzed and the occurrence conditions, mechanisms, and risk of spontaneous combustion were analyzed in a closed goaf. The influence of preheating with different oxygen concentrations on the spontaneous combustion behavior of secondary coal was investigated in [5]. During the secondary oxidation of coal, preheating at 80 °C promoted the physical adsorption of oxygen at low temperatures, however, it reduced the oxidation at high temperatures. The intersection point temperature (CPT) and isothermal oxidation characteristics of coal were analyzed [6], and it is reported that the upgraded low grade coal was less sensitive to low temperature oxidation and spontaneous combustion. The particle flow code in 3-D (PFC3-D) framework was used to establish a numerical model based on short-distance coal seam mining, and it was found that the air-leakage passage scope rapidly expanded with the roof rupture of the lower coal seam in the goaf [7]. With the periodic breakage of the roof, the air-leakage channel kept moving forward. A model of gas leakage and diffusion was established in the goaf of the third seam top coal caving mining of the Jishan Coal Mine [8]. An experimental study [9] found that coal samples would undergo an exothermic oxidation reaction and support the continuous expansion of the fire area in high temperature, oxygen-poor environment. It was also pointed out [10] that the specific surface area of coal sample had a direct effect on the coal oxidation rate, when the coal sample particle size was greater than 0.1 mm. Air-leakage channel, porosity and oxygen leakage expansion law of the goaf roof in the mined-out area near a thin belt was investigated through numerical model [11]. It was found that the air-leakage channel was constantly moving forward, prompting the lower coal seam to leak oxygen into the upper coal seam goaf. The spontaneous combustion of the remaining coal was intensified in the upper coal seam goaf. A total of 3308 working faces at the Liangbaosi Coal Mine in China were examined [12], and PFC3-D was used to simulate the overburden collapse. It was found that the central and nearby goafs had relatively large porosity, and that, as the height of the mined-out area increased, the porosity of the two cross-shaped wellheads was first larger than that of the middle part, and then smaller. The main air leakage occurred in the range of 0-10 m along the slope of the working surface. The concept of pressure gradient matrix and related calculation methods to analyze potential air-leakage paths was proposed [13]. Electron spin resonance spectroscopy was used to measure directly the changing laws of different types and particle sizes of coal, and it was proposed that the fracture process was the key factor to generate and initiate free radical reactions [14]. The 1-D mine ventilation network (MVN) was coupled with a 2-D/3-D gob flow field (GFF), and MVN analysis was used to evaluate the boundary pressure of the GFF simulation [15]. The FTM fed back the GFF result to the coupling network.

It can be seen from the literature survey that the relationship between air-leakage conditions and coal oxidation characteristics has so far been discussed considering only a single air-leakage channel with the air leakage assumed to be uniform. In other words, the possibility of multiple air-leakage channels was ignored. Such an approach cannot obviously address the characteristics of several concurrent and non-uniform air-leakage channels in the goaf. Based on this observation, the influence of different air-leakage conditions on the spontaneous coal combustion in the goaf of combustible coal seams in deep mines needs to be studied urgently. To that end, a coal spontaneous combustion test system with a nine-square grid was built to study multi-channel air leakage. Three types of uneven air-leakage channels are designed. The experiments examined coal spontaneous combustion temperature increase and oxidation. The heating and oxidation process and law of coal left are discussed for different air-leakage conditions in the goaf. The influence of different air-leakage conditions on the spontaneous combustion characteristics of coal in the goaf is examined.

Experimental system

Preparation of experimental coal samples

The experimental coal sample was taken from mine No. 1 of the Yungaishan Coal Mine, *Yunmei No. 1 Mine*, belonging to the Yongjin Energy Company, Henan Province, China. Yunmei No. 1 Mine is located in Foshan Village, Mojie Town, Yuzhou City, Xuchang City, Henan Province. The mine was evaluated as a high gas mine.

The coal sample was pre-processed and screened using a mesh screen prior to the experiment. The parameters of the coal sample are shown in tab. 1.

Table 1. Parameters of the experimental coal sample

Parameter	Value
Average particle size [mm]	3.629
Coal pile height [cm]	80
Coal pile bottom area, [cm^2]	225
Void ratio [–]	0.3528
Air supply flow [m^3s^{-1}]	$95 \cdot 10^{-6}$-$300 \cdot 10^{-6}$
Initial temperature [°C]	21
Lump coal density [gcm^{-3}]	1.43
Bulk coal density [gcm^{-3}]	0.948
Specific heat [$Jm^{-3}°C^{-1}$]	$5.12 \cdot 10^5$
Thermal conductivity [$Wm^{-1}°C^{-1}$]	1.72

Parameters of experimental system components

The components and parameters of the coal spontaneous combustion experimental system – 1 were 15 cm × 15 cm × 10 cm, three-leg combustion platform (h = 15 cm), asbestos net (d = 20 cm), 5 – closed cover with micro-holes (16 × 16 × 5), 3 – three thermocouples, 2 – a CO gas concentration detector, and 4 – an air supply device. The schematic diagram of the experimental system is shown in fig. 1.

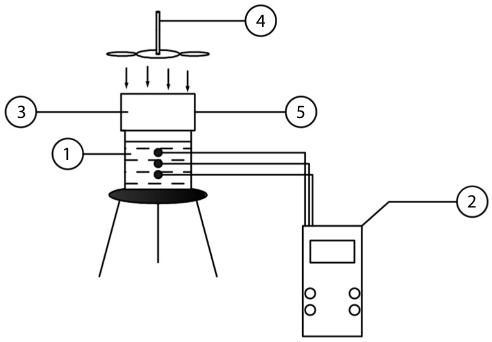

Figure 1. The experimental system in the present study

Experimental process

– The test coal sample was formed into a long cuboid (15 cm × 15 cm × 8 cm). Temperature measuring probes were set every 2 cm along the neutral surface inside the cuboid coal sample. The probes were denoted as A, B, and C from the bottom to the top, and were evenly distributed in the coal sample to measure accurately the temperature change of the coal sample during the experiment. The locations of the temperature probes are shown in fig. 2.

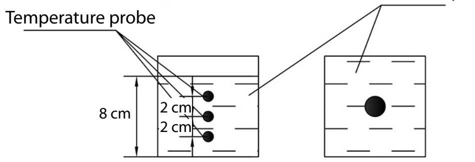

Figure 2. Temperature measurement points

– The spontaneous coal-combustion test bench was placed in the goaf under a sealing cover with micro-holes. The number of micro-holes was set gradually in accordance with the number of grids in the nine-square grid. Each small grid area was provided with three micro-holes for air supply.
– The coal sample was indirectly heated by a thermoelectric rod. At the same time, the ventilator was turned on supply air to the microspores. Three types of air leakage were simulated. The changes in the internal temperature and CO concentration were measured.
– When the temperature was raised to the spontaneous ignition point of 80 °C at Point A at the lower part of the coal sample, the external heating was stopped.
– The changes in such parameters as temperature, CO concentration, and O_2 concentration of the experimental coal sample were recorded every 1 second for 1 hour. There were 3600 data for each parameter in the experiment. When processing the data, one was selected every 100 seconds. There were a total of 36 data.

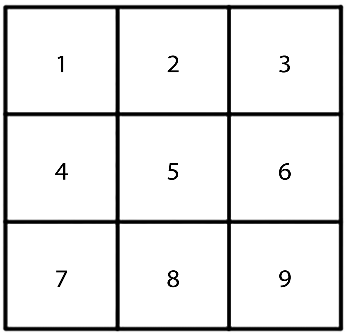

Figure 3. Grid of micro-hole distribution sin airtight cover

Design of air-leakage channel based on the principle of nine-square grid

The previous experimental process shows that the size of the air leakage must be changed by gradually changing the micro pore distribution in the cover. The micropore airtight cover was divided into nine grids, as shown in fig. 3. In the experiment, three areas were selected to simulate different air-leakage channels: Area 1 comprised grids 4-5-6, Area 2 grids 1-3-5-7-9, and Area 3 grids 1-2-3-4-5-6-7-8-9, respectively. An anemometer was used to measure the average wind speed of each cell, and then the average wind speed of each area was calculated, as shown in tab. 2.

Table 2. Air-leakage passage wind velocity of each cross-section

	1	2	3	4	5	Average wind velocity (of experiments)
1	3.2	3.4	3.4	3.5	3.3	3.36
2	3.4	3.6	3.5	3.4	3.5	3.48
3	3.6	3.6	3.3	3.7	3.5	3.24
4 Grid No.	3.4	3.4	3.5	3.5	3.4	3.44
5	4.1	4.3	4.5	4.5	4.5	4.38
6	3.3	3.6	3.5	3.4	3.5	3.46
7	3.4	3.3	3.2	3.3	3.4	3.32
8	3.6	3.3	3.3	3.2	3.5	3.44
9	3.5	3.2	3.4	3.3	3.2	3.32
Average wind velocity (of grids)	3.5	3.5	3.5	3.5	3.5	3.5

The diameter of micro pore in the airtight cover was 2 mm, and there were three micro pores in each cell on average. The air-leakage area, S [m^2], of each cell can be calculated:

$$S = 3.14 \times R^2 \times 3 = 9.42 \cdot 10^{-6} \tag{1}$$

The simulated air leakage, Q [m^{3}s^{-1}], can be expressed:

$$Q = vS \tag{2}$$

During the experiment, the amount of air leakage was changed by opening different numbers of micro pores in the airtight cover. The amount of air leakage is shown in tab. 3.

Table 3. Determination of the air leakage rate

Number	v [ms^{-1}]	Q [m^{3}s^{-1}]
Area1	3.46	$97.78 \cdot 10^{-6}$ (small)
Area 2	3.52	$165.80 \cdot 10^{-6}$ (medium)
Area 3	3.49	$295.88 \cdot 10^{-6}$ (large)

Results and discussion

To study the characteristics of spontaneous coal combustion with changing air leakage, it is necessary to start with oxygen consumption rate and heat-release intensity [16].

Analysis of oxygen consumption rate

The current methods of calculating the oxygen consumption rate differ depending on whether porosity. One was the formula derived by Qin *et al*. [17], and the other was the formula derived by Xu [18]. This paper used eq. (3) to calculate $V(T)$. It can be expressed [17]:

$$V\left(T\right) = \frac{Q(C_1 - C_2)}{sLn} \tag{3}$$

The experimental data was substituted into eq. (3). The relationship between the coal sample temperature and the oxygen consumption rate was obtained for different air-leakage rates, as is shown in fig. 4.

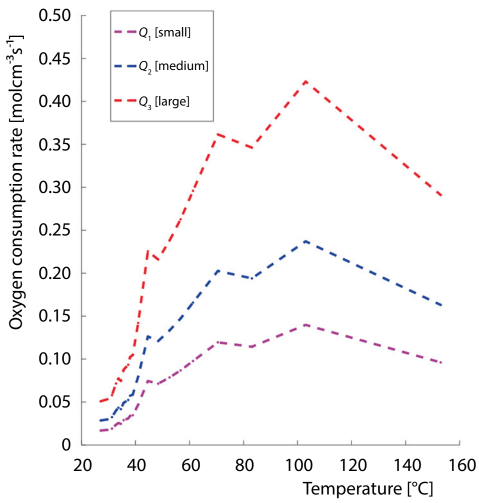

Figure 4. Change in oxygen consumption rate for different air-leakage rates

Figure 4 shows that there was a close correlation between oxygen consumption rate and air leakage. There are three obvious peaks. The first peak is around 48 °C. In the initial stage of coal oxidation in the experimental chamber, the heat generated was less than the heat released to the outside. The temperature of the second peak was about 75 °C. At this time, the experimental coal sample entered the second stage of oxidation – the spontaneous combustion stage. A change in oxygen consumption rate can clearly be seen. With the influence of the air leakage in the goaf, the experimental coal sample did not directly enter the third-stage combustion. Instead, the oxygen consumption rate showed a decline, because the incoming air-flow reduced the temperature in the experimental chamber. While the oxidation heat release of coal dominated, the third peak occured at a temperature of about 105 °C. The experimental coal sample began to burn, and as the coal sample was being consumed, the heat release to the outside environment was dominant. As a result, the oxygen consumption rate declined sharply.

With a small air-flow rate, the change in the peak value was gentler, and the temperatures were the lowest at the three peaks. Compared to other air-flow rates, the temperature of the peak changed only a little. It was found that the change in the air-flow had an impact on the peak oxygen consumption rate, but its impact on volatility was not obvious. With different air-flow rates, the peak values of three oxygen consumption rates were all closely related to the air-flow rate. Foe larger air-flow rates, the peak oxygen consumption rates were higher and the peak differences were larger. Because the air-flow velocity was large, the oxygen supply was sufficient, and this was conducive to the oxidation of the experimental coal samples.

For a single air-leakage channel, it is believed [17] that different air-flows had little effect on the growth trend of the standard oxygen consumption rate. Also, the growth trend was assumed to be exponential. With was multi-channel air leakage, the relationship was a step increase between the oxygen consumption rate and the air leakage. For larger air-flow rates, the peak oxygen consumption rates were higher and the peak differences larger. With small air leakage, the change of air-flow had no obvious influence on the fluctuation of oxygen consumption peak rate.

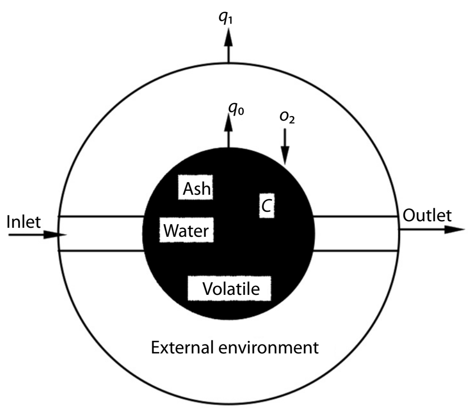

Figure 5. Heat exchange diagram of experimental chamber

Heat-release intensity

For coal at low temperatures, there are two methods to calculate the exothermic strength, $q(T)$, namely, the heat balance method, $q_0(T)$, and the bond energy estimation method, $q_{\min}(T)$, $q_{\max}(T)$. The heat balance calculation method can be used to calculate the heat-release, intensity. The bond energy estimation method only serves as a verification result [19]. Therefore, this paper used the heat balance method to derive the heat-release intensity, $q_0(T)$, of coal oxidation at low temperature.

Assuming that T_{air} that flows through the coal pile in this experiment cavity is equal to T_{coal}, the air flew along the longitudinal axis. The only heat exchange only considered was coal oxidation heat release, cavity wall conduction heat dissipation and the convection heat dissipation of the air-flow in the cavity during the coal oxidation process, and other heat exchanges were ignored. The heat field exchange in the experimental chamber is shown in fig. 5.

At the center of the experimental cavity, the heat balance equation can be expressed [19]:

$$q(T) = \rho_e c_e \frac{\partial T}{\partial t} + \bar{Q}\rho_g c_g \frac{\partial T}{\partial t} - \lambda_e \left[\frac{\partial^2 T}{\partial r^2} + \frac{\partial^2 T}{\partial z^2} \right] \tag{4}$$

$$\frac{1}{\lambda_e} = \frac{1}{\lambda_g} + (1-n)\frac{1}{\lambda_m} \tag{5}$$

$$c_e = nc_g + (1-n)c_m \tag{6}$$

$$\rho_e = n\rho_g + (1-n)\rho_m \tag{7}$$

During the low temperature oxidation stage, the oxidation exothermic intensity of coal, $q(T)$, is proportional to the oxygen mass concentration, C [19]. The relationship between temperature and $q_0(T)$:

$$q_0(T) = \left(\frac{C_0}{C_1}\right) q(T) \tag{8}$$

and shown in fig. 6.

From fig. 6, when the temperature reaches about 80 °C, the exothermic intensity calculated by the heat balance method exhibits an exponential increase as temperature rises. Below 80 °C, the heat release of coal samples was very limited and was hardly changing. In that stage the accumulated heat and the released heat competed with each other.

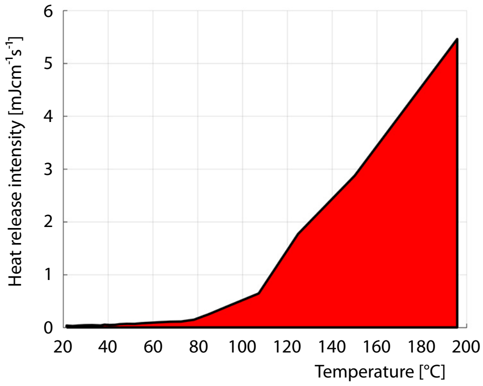

Figure 6. Variation of exothermic intensity of experimental coal sample

Relationship analysis between air supply and air leakage

The amount of air leakage is depended on the amount of air supplied to the working face in the goaf. When other conditions remain unchanged, the amount of air supplied becomes larger at the working face, and the pressure drop increases. It is thus obvious that the amount of air leakage must be increased. The relationship between the working face pressure drop, Δh, and air supply can be expressed [12]:

$$\Delta h = \frac{R\left(Q_{\text{in}} + Q_{\text{out}}\right)^2}{4} \tag{9}$$

The actual air leakage at the working face, ΔQ, can be expressed:

$$\Delta Q = Q_{\text{in}} - Q_{\text{out}} \tag{10}$$

The air leakage per unit area in the goaf, ΔQ_{leak}, can be expressed:

$$\Delta Q_{\text{leak}} = \frac{4\left(Q_{\text{in}} - Q_{\text{out}}\right)}{LH\left(Q_{\text{in}} + Q_{\text{out}}\right)^2} Q^2 \tag{11}$$

According to the actual measurement at the fully mechanized coal face, $Q_{\text{in}} = 915$, $Q_{\text{out}} = 828$, $L = 120$, $H = 3.5$, $\Delta h = 24.5$. The air-leakage rates at the fully mechanized coal face are shown in tab. 4. The relationship between the air-leakage rate and the air supply rate is also shown in fig. 7.

Table 4. Relationship between air supply and air leakage

Variables \ Air supply	800	900	1000	1100	1200	1300
ΔQ_{leak}	0.1745	0.2209	0.2727	0.33	0.3927	0.4609
ΔQ	73.31	92.78	114.55	138.60	164.95	193.59

From fig. 7, the air leakage per unit area of the goaf and the air leakage at the fully mechanized mining face are approximately in a linear relationship with the air supply at the working face. The greater the air supply at the working face, the greater the air leakage per

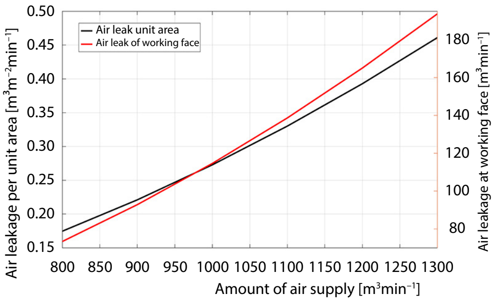

Figure 7. Relationship between air leakage and air supply

unit area of the goaf and the air leakage at the fully mechanized mining face were. This is because increasing air supply to the working face improved the pressure drop at both ends of the working face. If no measures are taken in the air-leakage channel, the air leakage would be increased in the goaf.

Relationship between air supply to working face and width of oxidation zone

The oxygen consumption rate was $3.74 \cdot 10^{-2}$ molm^{-3}min^{-1} when the oxygen concentration was 20.9%. It was assumed that there was no temperature difference in the goaf. The width of the oxidation zone can be expressed [13]:

$$w = \int_{[O_2]_{min}}^{[O_2]_1} \Delta Q_{leak} \frac{d[O_2]}{v} \tag{12}$$

Combining eqs. (10) and (11), the relationship between the width of the oxidation zone and the air volume at the fully mechanized caving face can be obtained:

$$w = \frac{1000}{22.4} Q^2 \frac{4(Q_{in} - Q_{out})[O_2]_0}{LH(Q_{in} + Q_{out})^2 v_0} \ln \frac{[O_2]_1}{[O_2]_{min}} \tag{13}$$

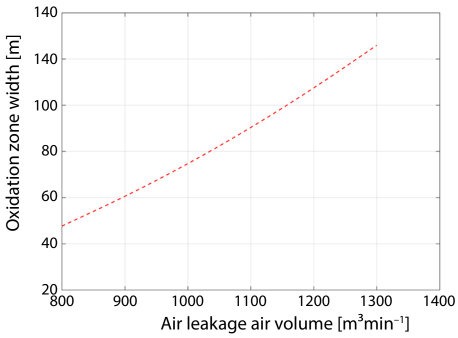

Figure 8. Relationship between air supply and oxidation zone width

The relationship between the air supply and the width of the oxidation zone is shown in fig. 8.

As fig. 8 shows, for a fully mechanized coal mining face of the combustible coal seam, the air leakage in the goaf and the air leakage on the fully-machanized mining face are linearly correlated to the air supply to the working face. The increase of air supply in fully mechanized mining face would increase the width of oxidation zone. The air supply reached a certain value, and the air leakage meet the coal spontaneous combustion requirements. The thickness of the broken coal at the fully mechanized mining face was thicker in the goaf, if the top coal mining technology was selected. This can reduce the advancing speed of the working face. During the minimum coal spontaneous combustion period, when the advancing distance of the working face was greater than the width of the oxidation zone in the goaf, the coal body could not combust spontaneously. On the contrary, it was more likely to cause spontaneous combustion of into in the oxidation zone.

For the fully mechanized top coal caving face of the coal mine studied in this example, the air supply was 915 m^3 per minute, and the oxidation zone width was calculated to be 62.36 m. The shortest fire period was measured to be 35 days, while the average advancement of the face was 2 m per day. The advancing distance was 70 m in the shortest spontaneous ignition period. The width of the oxidation zone was smaller than the advancing distance in the shortest spontaneous ignition period. Obviously, the risk of spontaneous combustion was relatively high in the goaf. If the air supply volume of the mining face was adjusted to 100 mm^3 per minute, the width of the oxidation zone was calculated to be 74.49 m. The width of the oxidation zone was greater than the advancing distance during the shortest spontaneous combustion period. Overall, the risk of spontaneous combustion was relatively low in the goaf.

Conclusions

- With the influence of multiple air-leakage channels, if the void ratio of the coal sample was considered, the oxygen consumption rate exhibited three obvious peaks, and the peak values increased with the increasing air-leakage. Compared to large air-flow, in small air-flow the changes of the peak values were relatively gentle, and three peaks of temperature were the lowest. The greater the air-flow, the larger the peak oxygen consumption rate and the greater the peak difference.
- Below 80 °C, the difference was not large between the heat release and heat dissipation of the coal sample, and the heat-release intensity hardly changed. Above 80 °C, the heat generated by the experimental coal sample began to dominate, and became far greater than the heat exchange to the outside. The relationship was exponential.
- The air-leakage rate per unit area of the goaf and the air-leakage rate at the fully mechanized mining face exhibited an approximately linear change with the air supply rate at the working face. The width of the oxidation zone had an exponential relationship with the air supply in the goaf. Specifically, for fully mechanized caving mining the remaining coal, the advancement of the working face was slower than the safe speed. When the air-leakage channels were increased, this could easily cause spontaneous combustion of coal in the goaf.
- By analyzing the relationship between different air-leakage conditions and the spontaneous temperature-rise characteristics of floating coal in the goaf, combined with the data measured and obtained in the goaf and experimental measurements, the risk of spontaneous combustion of the coal seam in the goaf could be predicted more accurately. This could guide the formulation of on-site fire-fighting technical measures.

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Nomenclature

C_1 – oxygen concentration at inlet, [= 20.9%]
C_2 – oxygen concentration at outlet, [%]
c_m, c_g – experimental specific heat capacity of coal sample and air, [$Jg^{-1}°C^{-1}$]
Δh – working face pressure drop, [Pa]
L, H – length and mining height of fully mechanized mining face, [m]
n – void ratio of experimental coal sample, [= 0.45]
$[O_2]_0$ – the oxygen concentration when the oxygen consumption rate is v_0 [= 20.9%]
$[O_2]_1$ – oxygen concentration upper limit, [= 20.8%]
$[O_2]_{min}$ – oxygen concentration lower limit, [= 6.96%]
Q – air leakage air volume, [m^3min^{-1}]
Q_{in}, Q_{out} – intake and return air volume of mining face, [m^3min^{-1}]
ΔQ_{leak} – air leakage per unit area, [$m^3m^{-2}min^{-1}$]
$q(T)$ – heat-release intensity at temperature T, [$Jcm^{-1}s^{-1}$]
$q_0(T)$ – oxidation exothermic intensity at temperature T, [$Jcm^{-1}s^{-1}$]
R – radius of micro pores, [m]
r – transverse length of experimental cavity, [m]

S – air-leakage area of cell, [m^2]
s – cross-sectional area of experimental goaf, [m^2]
T – time, [s]
v – average wind speed through section, [ms^{-1}]
$V(T)$ – oxygen consumption rate, [molcm^{-3}s^{-1}]
z – length of experimental cavity, [cm]

Greek symbols

λ_m, λ_g – thermal conductivity of experimental coal sample and air, [Jcm^{-1}s^{-1}°C^{-1}]
ρ_g, ρ_m – equivalent density of air and experimental coal sample, [gcm^{-3}]

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