Review of Gas Emission Prediction and Control Methods for Multi-seam Mining in Chinese Coal Mines

Zongwei Wang
University of Wollongong

Ting Ren
University of Wollongong, tren@uow.edu.au

Lei Zhang
University of Wollongong

Publication Details
REVIEW OF GAS EMISSION PREDICTION AND CONTROL METHODS FOR MULTI-SEAM MINING IN CHINESE COAL MINES

Zhongwei Wang, Ting Ren and Lei Zhang

ABSTRACT: In China, the practice of multi-seam mining with high gas emission rates is common, and many coal mines are suffering from major gas management issues. To deal with these problems, empirical formulas for gas emission prediction and gas control methods suitable for site-specific conditions are developed. Specifically, the statistical method and split-source-method are widely used for gas emission prediction. In addition to ventilation, various gas control methods, including cross-measure boreholes, directional long-holes, dedicated roadways and surface goaf wells are used for gas drainage. Techniques for integrated extraction of coal and coal seam gas have been developed in these coal mines, resulting in significant economic and environmental benefits. A detailed review of gas emission prediction and control methods adopted in highly gassy Chinese coal mines is presented.

INTRODUCTION

Gas-related disasters have long been recognised as one of the most serious threats to mine safety in underground coal mines in China, especially for those extracting multi-seams where gas migration from adjacent seam and gas-bearing strata may result in unexpected or uncontrolled gas issues. According to a recent survey by the Ministry of Land and Natural Resources (http://www.to-gd.com/webpage/2010-07-12/gd559381.htm), China has a mineable coal reserve of 204 billion tons, most of which is associated with multi-seam conditions.

Due to various specific coalifications, some mining areas, typically including Yangquan, Huainan, Tiefa and Pingdingshan, have been suffering from high gas emission rates. To deal with the gas issues in these gassy mines, an accurate gas emission prediction is essential for mine ventilation design and the deployment of suitable gas control methods. These mining areas have developed empirical formulations under their site conditions, and the gas control methods vary from site to site as a result of various geological conditions, reservoir characteristics and mining methods.

Although gas control techniques have been widely improved worldwide either by ventilation or gas drainage, they may not suit for Chinese high gassy coal mines due to the complex site conditions characterized by low permeability of the coal seams, deep burial or high geostress. Many coal mines in the east of China have developed into a deep mining stage with mining depth exceeding 1000 m. It is also estimated that the mining depth will increase at a rate of 8-12 m annually providing a great challenge to the existing gas control techniques. Table 1 describes the general exploitation conditions of typical high gassy multi-seam mining areas in China. It can be seen from the table that permeability of coal seams is critically low. For the other mining areas, the permeability of coal seams is generally between 10^-7-10^-4 md, leading to poor performance of pre-drainage technique and the development of post-drainage methods either through cross-measure boreholes or dedicated gas drainage roadways.

Table 1 - Gas related parameters in typical high gassy mining areas (You, 2008a; Yuan, 2008a; Li and Liu, 2005; Huang and Yang)

<table>
<thead>
<tr>
<th>Mining area</th>
<th>No. of coal seams /minable coal seams</th>
<th>Overburden (m)</th>
<th>Gas content (m^3/t)</th>
<th>Gas pressure (MPa)</th>
<th>Permeability (md)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Yangquan</td>
<td>16/3</td>
<td>150-500</td>
<td>7.13-21.73</td>
<td>0.25-2.3</td>
<td>3.75*10^-4</td>
</tr>
<tr>
<td>Huainan</td>
<td>/8-15</td>
<td>400-1500</td>
<td>10-36</td>
<td>4-6</td>
<td>2.75*10^-5</td>
</tr>
<tr>
<td>Tiefa</td>
<td>/13</td>
<td>600-1200</td>
<td>8.13-24.8</td>
<td>0.85-3.92</td>
<td>2.75*10^-6</td>
</tr>
<tr>
<td>Pingdingshan</td>
<td>/8</td>
<td>300-1200</td>
<td>6-15</td>
<td>0.6-2.55</td>
<td>-</td>
</tr>
<tr>
<td>Huaibei</td>
<td>8/4</td>
<td>400-1300</td>
<td>6-22</td>
<td>0.1-4.5</td>
<td>0.02-0.3</td>
</tr>
<tr>
<td>Songzao</td>
<td>12/3</td>
<td>200-500</td>
<td>17.1-29</td>
<td>1.2-4.5</td>
<td>1.43<em>10^-7-7.98</em>10^-4</td>
</tr>
</tbody>
</table>

Gas emission prediction and control methods adopted in typical high gassy mining areas in China are discussed.
GAS EMISSION PREDICTION

Gas emission prediction is a basic parameter required for the design of mine ventilation systems and corresponding panel layouts, and the selection of proper and effective gas drainage technique. The accurate prediction of gas emission therefore plays a significant role especially for multi-seam mining where gas emission into a working face may increase rapidly compared with single seam mining. Furthermore, for production mines, gas emission must be checked regularly to ensure that its ventilation system and gas drainage system are sufficient enough to deal with the gas issues due to increasing mining depth and changing geological conditions.

Generally, two main methods are employed in the process of gas emission prediction: a statistical method and a split-source method, specifically, a statistical method is usually used for predicting gas emission of an entire coal mine, whilst a split-source method can be used for both a coal mine and a particular active longwall face.

Statistical method

Statistical methods have been widely adopted in China for mine gas emission prediction. This method involves the collection of gas data from previous experience of a mine, a particular area of a mine or of neighboring mines where similar mining methods are being used in similar geological conditions, thus establishing the relation between relative gas emission and mining depth by regression analysis. Figure 1 shows some typical curves obtained using this method (Yu, 2005).

\[
q = \begin{cases} 
0.072H^{-2.96} & H \text{ in depth} \\
9.27+0.028H & \text{otherwise}
\end{cases}
\]

Based on this relationship, the gas emission for a further mining depth can be predicted empirically by extending the line to a proper depth provided that the relationship has been obtained. The accuracy strongly depends on the precision of previous data from which the relationship is founded and the similarity of geological and mining conditions. It is noted that this method is most applicable for the methane-bearing zone, and the extended prediction for seams at greater depth should be discounted with the decrease of seam dip within 100 to 200 m, otherwise, significant errors may occur (Yu, 1992).

Split-source method

Gas emission can also be predicted by the analysis of emission contributing sources. A typical grouping of gas sources in a coal mine is shown in Figure 2, from which it can be seen that gas liberated into an active longwall face can be divided into two main sources, the mined coal seam and adjacent gas-bearing strata (overlying and underlying) (State Administration of Work Safety, 2006). Thus, most of the split-source methods for longwall gas emission are similar, what makes the difference is the various methods of determining the quantity of gas released from overlying and underlying seams or gas emission degree.

Yu et al. (2000) demonstrated that gas released from rib, excavated coal and goaf were the main contributors to longwall face emission. For the first two parts which can be attributed to emission from the mined seam, the following equation was proposed:

\[
q_m = C \cdot X
\]  

(1)
Where, \( q_m \) is the relative gas emission from mined coal seam, in m³/t; \( X \) is the gas content of mined coal seam, in m³/t; \( C \) is the degassing coefficient of mined coal seam, and is generally between 0.4-0.8 for thin and medium thickness coal seams.

\[
q_g = C_g \times a \times v \times \frac{m \times \delta}{24 \times 60} \sum_{i=1}^{n} \frac{m_i}{m} x_i \eta_i
\]  

(2)

Where, \( C_g \) is the coefficient determined by ventilation system (\( \leq 1 \));
\( v \) is advance rate of face, m/d;
\( av^2 \) is the correction factor taking into account the lag of roof and floor break and the displacement relative to time and space;
\( m \) is the mining height, in m;
\( L \) is face length, in m;
\( \delta \) is coal density, in t/m³;
\( m_i, x_i \) and \( \eta_i \) are thickness, gas content and emission degree of adjacent gas-bearing strata, respectively.

Figure 2 - Gas sources in a coal mine (State Administration of Work Safety, 2006)

Gas emitted from adjacent strata into the goaf is dependent on the ventilation system adopted, the gas content of adjacent strata, the advance rate of face, the thickness of gas-bearing strata and the corresponding emission degree, thus, it can be stated as:

\[
q = q_m + q_a = k \frac{h}{m} (x_0 - x_c) + \sum_{i=1}^{n} \frac{m_i}{m} x_i \eta_i (x_{oi} - X_{ci})
\]  

(3)

By taking into account the residual gas content, the following formula was proposed (Yu, 2005; State Administration of Work Safety, 2006; Cheng, et al., 2006):

(a) Yangquan mining area

(b) Huainan mining area

Figure 3 - Gas emission degree of adjacent strata developed in different mining areas (You, et al., 2008a; Cheng, et al., 2006)
Where, $q$, $q_m$ and $q_a$ are the gas emission quantities of longwall face, mined seam and adjacent strata, in m$^3$/t, respectively; $k$ is the influencing coefficient of gas emission from mined seam, and it is control by three factors ($k = k_1 \cdot k_2 \cdot k_3$, where $k_1$ is coefficient of gas emission from surrounding strata, and $k_1$=1.2 for fully caving; $k_2$ is coefficient of gas emission from coal left in face, and it is the inverse of the face recovery ratio; $k_3$ is the influencing coefficient of pre-gas emission in the process of roadway development, and for retreat longwall mining, $k_3 = (L - 2b)/L$, where $L$ is face length, $b$ is the width of pre-gas emission in roadway); $m$ and $M$ are the mining height and seam thickness respectively, in m; $X_0$ is the initial gas content of mined coal seam, in m$^3$/t; $X_c$ is residual gas content of mined coal seam, in m$^3$/t; $M_i$ is thickness of the $i$ adjacent seam, $X_{0i}$ is the initial gas content of the $i$ adjacent seam, in m$^3$/t; $X_{ci}$ is the residual gas content of the $i$ adjacent seam, m$^3$/t; $\eta_i$ is the percentage of gas emitted from adjacent seam $i$.

Obviously, the accuracy of the two methods depends on the coefficients adopted which are different in different coal mines, and the quality of the gas data-set is critical to the prediction accuracy.

**GAS CONTROL METHODS**

It is acknowledged that ventilation is the most basic and common method for gas control, and for less gassy mines, ventilation is normally sufficient to dilute the gas concentration below the statutory safety limit; however, efficient drainage strategies are necessary for highly gassy coal mines where large quantities of gas may migrate into underground workings from adjacent strata and ventilation dilution capacity is limited in this situation. Therefore, a reasonable combination of a well-designed ventilation system and an efficient gas drainage system is critical to ensure a safe working environment. Figure 4 shows some of the gas control techniques used for multi-seam mining.

![Figure 4 - Gas control methods developed in Chinese coal mines for multi-seam mining](image)

**'U+L' ventilation scheme based gas control technique**

Based on the 'U+L' ventilation scheme, cross-measure boreholes can be drilled towards the fractured zone from the exterior tailgate (the second return roadway) with an interval of 15-35 m, as shown in Figure 5.

Firstly, gas released from the underlying strata and coal left in the goaf will flow through the cut through kept open behind face instead of coming back to the face which usually results in high gas concentration at the intersection of longwall face and return roadway. In other words, this ventilation scheme mainly deals with goaf gas emitted from underlying seams, mined seam and overlying seams within the caved zone. It is noted that the interval between cut throughs impacts on the performance of the exterior tailgate as a result of re-compaction in the goaf, and large quantities of goaf gas may report to face again when the cut through behind face is too far from face line.
Thus, cross measure boreholes are drilled to intercept gas desorbed from seams in the fractured zone before migrating into goaf and working area. It is suggested that the interval of boreholes should be in line with the periodic weighing interval of roof caving, ensuring that at least one borehole is working when large quantities of gas migrate into the working face immediately after periodical roof collapse. The effective drainage radius of boreholes should also be considered while designing intervals (Yu, 1992). The diameter of boreholes is generally between 73 mm and 300 mm according to available drilling equipment on site, the borehole length depends on the distance of the gas-bearing strata from the mined seam. The elevation angle, as shown in Figure 6, is another significant parameter, affecting performance of a borehole, and can be determined by,

$$\tan(\alpha + \beta) = \frac{h}{h \cot(\varphi + \alpha) + b}$$  (4)

Where, $\alpha$ is the seam angle, $\beta$ is the angle between borehole and horizontal, $h$ is the vertical distance between the adjacent seam and the mined seam, $\varphi$ is the de-stressing angle of strata after excavation, and $b$ is the pillar width.

Operational experiences indicates that borehole suction pressure should be kept between 6.7 and 13.3 kPa, and sealing length no less than 2-5 m.

Field trials carried out at Yangquan mining area demonstrated that the gas recovery ratio of adjacent coal seams could reach 60-70% using boreholes with diameter of 200 mm (Zhang and Cheng). It should be pointed out that although the exterior tailgate has a great significance in the dilution of gas, this in turn leads to a major disadvantage of this technique, i.e., the self-heating or oxidation of coal in the goaf due to air leakage through the cut through behind the face, consequently, this technique is limited in its use for spontaneous combustion prone coal seams. Slotted casing must be used where soft rock especially water-swelling mudstone exists in the roof, the water discharge system is also critical in this case.

When production of a longwall panel increases or gas emission from adjacent coal seams is too large and cross measure boreholes are not sufficient to prevent gas from migrating into working panels, a special roadway, which is driven as an inclined high level roadway for gas drainage, is developed from the exterior tailgate. While reaching the target drainage seam level in the fractured zone, it is developed along the seam and extended about 25 to 40 m with a cross section of 3-4 m$^2$. Then the roadway is sealed at the bottom and a pipeline installed. A typical layout and parameters of roadway involved in this technique can be found in Figure 7, from which the following formula can be obtained,

$$h_1 = b \ast \tan \beta = b \ast (h_c + \Delta h_c)/(a + b)$$  (5)
Where, $h_1$ is the height that the inclined roadway exceeding the pillar, $b$ is pillar width about 20-25 m, $\beta$ is the elevation angle of roadway about 40-45°, $h_c$ is height of caved zone, $\Delta h_c$ is a safety height preventing the horizontal roadway from broken, and is taken as 1-1.5 times of mining height, $L$ and $L'$ are safety distance ensuring the inclined part to be intact, $\varphi$ and $\gamma$ are destressing angle and broken angle of strata after excavation respectively (You, Li and Zhang, 2008a).

The interval between inclined roadways is much greater than that between cross-measure boreholes due to its large cross section and varies between 150 and 250 m. Field monitoring data of this drainage technique indicates that the average gas concentration captured could reach 71.79%, and the average gas flow rate 31.14 m$^3$/min, leading to a 74.28% gas recovery ratio from adjacent coal seams.

Compared with the ‘U’ ventilation scheme, the ‘U+L’ ventilation scheme gas drainage technique is superior in terms of the large dilution capacity of the exterior tailgate, which could account for 57.67% of the total gas handled by ventilation, and the ease of drilling cross-measure boreholes or inclined high level roadway development and pipeline installation in the exterior tailgate. The occurrence of gas concentration exceeding the statutory limit at the upper corner has been improved greatly. However, a serious problem induced by this technique is the increased risk of oxidation in the goaf especially when cut-throughs behind the panel are not sealed tightly and timely.

‘U+I’ ventilation scheme based gas control technique

To overcome the shortcoming of ‘U+L’ ventilation scheme, the innovative ‘U+I’ ventilation scheme is adopted during the exploitation of No.15 coal seam which is a spontaneous combustion prone seam in Yangquan mining area. It can be seen from Figure 8 that the average thickness of No.15 coal seam is 6.52 m, and within 60 m above the seam, is the K$_3$ K$_4$ limestone, where fractures and Karst caves are well developed, and are filled with gas. The gas content of No.12 coal seam is up to 14.75 m$^3$/t, which also poses a threat to the operation of No.15 seam. It should be pointed out that the top coal caving process is another factor influencing the gas control method in the panel. Under this condition, a strike high level roadway along the panel and interior tailgate are developed to deal with gas emitted from the overlying gas bearing strata and the top coal or goaf, respectively.

A general layout of this technique is illustrated in Figure 9. The interior tailgate is developed in the top coal along the roof about 15-25 m away from the return roadway, and the strike high level roadway parallel to the return roadway is developed in the fractured zone generally 40-60 m above the mined seam and about a third of the face length away from the return roadway horizontally (Zhu, et al., 1997).
The interior tailgate provides a new or negative pressure outlet besides the return roadway along the panel, and takes advantage of the collapse as the panel advances but is at least 5 m behind the coal caving line as it is developed in the top coal; consequently, this interior tailgate always performs better at collecting gas emitted from goaf compared with the return roadway and even the exterior tailgate developed in the ‘U+L’ ventilation scheme, and this has been verified by field data showing a dilution capacity of 75.24% total ventilation gas, 17% higher than that of the exterior tailgate [You, et al., 2008a]. In addition, the interior tailgate is easier to maintain than the exterior tailgate, cut throughs are neglected under this scheme. The strike high level roadway developed above the mined seam with a cross section of 4 to 5 m² is superior to the inclined high level roadway in terms of capture of gas desorbed from the adjacent gas-bearing strata as it works with a relatively stable and efficient drainage all the time during panel retreat.
Numerous field trials have been carried out in the Yangquan mining area and this technique has now developed into a main pattern of integrated exploitation of coal and coal bed methane for multi-seam mining in high gassiness and spontaneous combustion prone seams. In these field trials, gas recovery of the strike high level roadway could rise to 80-90%, and an average gas flow rate of 40-60 m³/min was achievable (Zhao, 1996). However, one limitation of the technique is that the interior tailgate will be difficult to maintain (or develop) in soft (or thin) coal seams where it should be developed, i.e., this technique is applicable for a seam thickness of 5.5 m. Meanwhile, it is strongly recommended that the mining height under the interior tailgate be carefully controlled and the flipper of chocks be extended in time to avoid its collapse ahead of the face line. Another factor that should be taken into account is the gas issues encountered in development of the strike high level roadway, the absence of a suitable adjacent seam where it can be developed also challenges this technique because of the large rock roadway development work and the relatively high cost, but it is justified for creating a safe working environment.

The ‘U+I’ ventilation scheme should be improved when the face width increases significantly. Field trials of K8206 panel in Yangquan 3rd mine, where the face width reached 252.2 m and gas concentration in the return roadway was always 1.0-1.2%, indicated that the single interior tailgate was not sufficient enough to dilute gas emitted from the goaf because of the increase of destressing zones both horizontally and vertically. The increased air quantity and ventilation pressure also leads to the flushing of a high volume of gas emission from the goaf. As a result, another interior tailgate can be developed, forming the ‘U+II’ ventilation scheme. Field trials had been carried out at K8205 panel, two interior tailgates with a distance of 28 m and 89 m away from the return roadway respectively were developed in the top coal along the immediate roof. Gas monitoring data demonstrated that exceeding the limit of gas concentration in the return roadway rarely occurred, and the dilution capacity of the tailgate was enhanced.

‘Y’ ventilation scheme based gas control technique

The gas control methods mentioned above may be not applicable in the Huainan mining area where coal and gas outbursts have become a major threat due to the high gas pressure (4-6 MPa) and burial depth (700-800 m). Low permeability (generally 2.75 x 10⁻⁷ md) and high geostress greatly limits the effectiveness of a pre-drainage strategy, thus, various post-drainage techniques have been developed.

One of the most efficient and cost effective methods was the ‘Y’ ventilation scheme based gas control technique without a coal pillar. From the perspective of outburst prevention, it is a protective mining method, which involves firstly mining a seam with low gas pressure and content to destress the outburst prone seams above and below, capture as much gas as possible, and finally remove the outburst risk of the protected seams (Yu, 1992).

It can be seen from Figure 10 that the ‘Y’ ventilation scheme is a variant of ‘U’ scheme, a bleeder road behind the panel is maintained for air return, and the two other roadways ahead of the panel are used for intake air. The gas flow and air leakage patterns in the goaf under the ‘Y’ ventilation scheme vary from the previously mentioned ‘U+L’ and ‘U+I’ scheme. Firstly, as the two roadways ahead of face act as intakes, there is less air exchange between the goaf and face, thus less gas migrates into the face compared with other schemes, protecting the panel from the threat of goaf gas. Gas accumulation issues around the tailgate corner, which generally accounts for a large proportion of production delay can be well handled (Yuan, 2008b); secondly, as a result of lower ventilation pressure and the goaf gas buoyancy effect, large quantities of gas accumulate along the retained roadway in the goaf, benefiting long term high purity gas drainage.

Figure 10 - ‘Y’ ventilation scheme based gas control technique (Yuan, 2008b)
The retained roadway provides a gallery from which cross-measure boreholes can be drilled to capture large quantity of desorbed gas. Two or three upward boreholes can be drilled in each drilling gallery to reach target seams where vertical fractures are not well developed in the bending zone, so the borehole length depends on the vertical distance and elevation angle, both of which can be determined through Figure 5, borehole diameter is generally more than 90 mm, and the spacing between galleries about 20 m. Downward boreholes should also be drilled to drain gas desorbed from underlying seams; however, the destressing effect may not be as good as that for overlying seams the same distance from the mined seam. Field investigations show that the destressed region in the Huainan mining area may extend up to 130-150 m in the roof and 80-100 m in the floor of the mined seam respectively (Yuan, 2008b), therefore a shorter drainage gallery interval of 10 m is recommended. This ‘Y’ ventilation scheme based gas control technique has been adopted in the Xinzhuangzi mine and the Guqiao mine.

**Surface well drainage technique**

Besides various gas control techniques conducted underground, surface well drainage has also recently been developed as an effective method in some mining areas, such as Tiefa, Huabei and Huainan. Initially, the surface well drainage technique was limited in use as a kind of pre-drainage technique due to the low permeability of coal seams in most mines, however, recognising the effect of mining induced stress relief and permeability increase, surface wells are now widely used to capture stress relief gas desorbed from remote protected seams and goaf gas (sang, et al., 2010). Structurally, the surface wells are almost the same as shown in Figure 11, including surface casing, intermediate casing and slotted casing for gas production (Li and Liu, 2005; Sang, et al., 2010). Research results and field tests demonstrated that the ‘O’ shape compaction will be formed in the goaf as panel retreats, and it is suggested that the wells should be located close to the return roadway and the bottom should reach the ‘O’ circle where large volumes of gas accumulated (Xu and Qian, 2000). The bottom elevation of the well is usually 5 to 10 m above the mined coal seam. Depending upon specific site conditions, the diameter of the three borehole casing section is 299 mm, 177.8 mm, and 139.7 mm, respectively (Yuan, 2004; Li and Liu, 2005). It is accepted that the overlap of intermediate casing and slotted casing should be no less than 10 m, and the length of slotted casing should be 35-45 m. Spacing between surface wells is generally 200-400 m depending on their effective radius under specific site conditions, and the first surface well is usually 40-80 m away from the start up line, suction pressure is 40-60 kPa.

![Figure 11 - Typical surface well location and structure (Li and Liu, 2005; Sang, et al., 2010)](image)

Early trials of surface well gas drainage were conducted at N1405 panel of Daxing mine in the Tiefa mining area, where three surface wells with an interval of 150 m were implemented, and they were 50 m away from the return roadway, the first well was 60 m away from the start up line. Results showed that within 280 days about 2.9 Mm³ of methane was captured with a gas concentration of more than 83% (Li and Liu, 2005). However, the results were not always satisfying in some mines where well collapse is the main obstructive factor, for instance, two wells drilled in Zhangbei mine of the Huainan mining area collapsed as the panel passed them by 54 m and 49 m, respectively (Liang, 2007). Therefore, further research on both theory and practice of surface well gas drainage is necessary, especially the stability of the surface well.
Other gas control techniques

There are also some other gas control techniques being used in Chinese coal mines, for instance, cross-measure boreholes are conducted from drilling galleries in return roadways (Figure 12a) (Yu, 1992; Yuan, 2000), and this is applicable under close distance coal seam mining condition where desorbed gas may migrate through the goaf into the working environment; in some outburst prone mines, dedicated roadways are developed to eliminate the outburst risk (Figure 12b) (Huang and Yang); long horizontal boreholes parallel to roadway are also employed in certain mines for the purpose of replacing strike high level roadway or interior tailgate (Figure 12c) (Li and Hu, 2009). However, for various reasons, these gas control techniques are not widely adopted in China.

CONCLUSIONS

In order to effectively manage gas emission hazards, gas emission rates must be well understood and predicted for the choice of suitable gas control techniques. Generally, two main methods: statistical method and split-source method are adopted for gas emission prediction of coal mine and longwall face, and the accuracy of both methods is highly dependent on the coefficients adopted. Based on various ventilation schemes, both cross measure boreholes and dedicated roadways are widely employed to capture gas desorbed from overlying and underlying seams. For cross-measure boreholes, the interval, elevation angle, length and sealing are the main factors influencing its performance. As to the strike and inclined high level roadways, the horizontal and vertical distance from the return roadway and the mined seam are critical for their performance. Surface wells have been developed as an effective drainage method for stress-relief gas desorbed from the protected seam in the bending zone, where vertical fractures are not well developed, and for goaf gas.
REFERENCES


Huang Changwen, Yang Liangzhi. Status and research direction of gas drainage in Songzao Mining Area.


Li Wei, Chen Jiaxiang, Liu Huamin, 2007. Practice of triple use of one surface well for gas control, in *Proceeding of the 32nd international symposium on mine safety*. the 32nd international symposium on mine safety, Beijing.


You Hao, 2008b. The safe and efficient exploitation of high gassy and spontaneous prone seam by enhancing both ventilation and gas drainage simultaneously. *International symposium on theory and practice of safe, efficient, and green mining*. Xuzhou.


