THE CORRELATION BETWEEN ASCENDING MINING
AND STABILITY OF THE OVERLYING STRATA

By
H.C. Li

ABSTRACT

In the process of mining the lower seam in a multi-seam operation, a lot of oblique fractures appeared in the upper seam which was in the equilibrium zone. These fractures closed soon after the lower face line passed. Despite the occurrence of some degree of undulations, and at step-like unevenness, the upper seam was subsequently extracted without difficulty. When attempting multi-seam extraction, from the height of the equilibrium zone, the minimum distance between seams in ascending mining should be predicted. By analysing roof convergence data and the geometrical configuration of subsidence profile, it was found that the minimum safe distance for ascending multi-seam mining depends on the mining height of the lower seam and the properties and structure of the inter-stratified rocks.

INTRODUCTION

Ascending mining is not a common method of mining in general. If this method is adopted, no additional capital or equipment is needed. In addition, it has a distinct efficiency in extracting coal under some particular conditions, such as the case of a competent roof and an overburden of rockbursts.

In order to investigate the movement mode of surrounding rocks, with the possibility of introducing of ascending mining methods, an underground observation was made at Kongshaung Colliery, Barton. The upper seam in this mine has igneous intrusions and is, therefore, partially unmineable. Its roof is a competent sandstone layer with severe periodic weighting problems. Its first weighting interval was 50 to 70m. The induced fracturing within the roof can be increased when ascending mining method is used. This would reduce a danger from a first weighting on subsequent mining of the upper seam.

An experiment in extracting the lower seam first was made in Kongshaung mine. Fig. 1 shows the experimental panel layout and its geological column. The extracted height of the coal face was 1.8m, and the inclination was 25 degrees. The lengths of the longwall face and of the panel were 112m and 425m respectively. The interval between two seams is 25m.

An entry in the upper seam was used as a monitoring station. This roadway was located above the middle of the working longwall face and in the direction of the face advance. 21 observation points were set up in the entry. Surveys were made with level transit at each of these points. A multi-anchor wire

---

1. H.C. Li, Associate Professor, Dept. of Mining Engineering, China Institute of Mining and Technology, Xuzhou, China.

The AusIMM Illawarra Branch, Ground Movement and Control related to Coal Mining Symposium August 1986

232
extensometers were installed at an interval of 8 m along the strike of the entry. Fourteen boreholes, each 2 m long were drilled in the walls and floor of the entry and some flat jacks were installed to measure the pressure within surrounding rock. Ten couples of reference points were installed in the roof, floor and both ribs to measure the displacement and its velocity in both horizontal and vertical directions. Fractures in the entry and changes of water table in boreholes were recorded. Roof conditions of the working face of the lower seam were mapped. All the observations lasted for 204 days. The conditions of the monitored roadway after undermining are shown in Fig. 2.

Fig. 2 - Monitored entry after undermining

DEVELOPMENT OF FRACTURES IN OVERLYING STRATA

With the advance of the lower seam face, two types of fractures, natural and induced, were exposed in both ribs of the monitored roadway. A sketch of these fractures is shown in Fig. 3. The rose diagram shows quantity and distribution of fracture.

These induced fractures have the following properties:

1. The mining induced fractures can be divided into two groups, these are: bed separation and subvertical fractures. The latter was more abundant than the former, and the angles between the subvertical fractures and the seam plane range from 60 to 85 degrees.

2. Most mining induced fractures were developed from natural fissures, joints and weak planes. The newly produced fractures were less than 10% and their occurrence is similar to that of the induced ones.

3. The fractures appeared more evident on the surfaces of shotcrete on the two sides of the entry. Its average density was 1.03 to 1.13 L/m (line/m) and the average width of opening was 12.6 to 14.2 mm. The maximum width of the bed separation was generally 30 to 40 mm.

Fig. 3 - Fracture patterns on both sides of the monitored entry

4. The incipient induced fractures appeared at the position 4.4m ahead of the coal face. Its width was only a few millimeters and it began widening significantly at the position 11.6m ahead of the coal face and began closing gradually at the position 31 to 45.7m behind the coal face, and it basically closed at the position 60m behind the coal face. No step-like slide occurred. Widths of typical bed separations near point D, are listed in Table 1.

This rock fracturing is caused by deformational forces and the fragile nature of rock material. The maximum deformation of the surrounding rock (sandstone) at the position 25 m away from the gob area was 6.6 mm/m based on the measured data. The critical strain of the sandstone is only 0.35 mm/m. This will certainly make the rock split apart along the weak planes and form subvertical fractures. The magnitude of horizontal deformation was approximately equal to the total width of the subvertical fractures within one unit length. Bending and convergence of rock can also be described by the development of subvertical fractures and bed separation. Therefore, the overlying strata especially the hard and thick layers can be considered as a fractured rock beam which had been broken into blocks.

EQUILIBRIUM OF OVERLYING STRATA AFTER UNDERMINING

The curves in Fig. 4 show the convergence between the roof and floor of the entry. Three different zones of strata behaviour can be distinguished from this graph.

The AusIMM Illawarra Branch, Ground Movement and Control related to Coal Mining Symposium August 1986
TABLE 1  WIDTHS OF THE BED-SEPARATED CRACKS (D.B.)

<table>
<thead>
<tr>
<th>Distance from coal face (m)</th>
<th>18</th>
<th>4.5</th>
<th>-13.5</th>
<th>-21.4</th>
<th>-25.5</th>
<th>-26.4</th>
<th>-26.8</th>
<th>-30.1</th>
<th>-34.1</th>
<th>-42.3</th>
<th>-46.3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Width (mm)</td>
<td>5</td>
<td>50</td>
<td>100</td>
<td>120</td>
<td>116</td>
<td>114</td>
<td>100</td>
<td>90</td>
<td>80</td>
<td>60</td>
<td>close</td>
</tr>
<tr>
<td>Date</td>
<td>5/2</td>
<td>3/3</td>
<td>25/3</td>
<td>1/4</td>
<td>6/4</td>
<td>7/4</td>
<td>8/4</td>
<td>11/4</td>
<td>15/4</td>
<td>22/4</td>
<td>27/4</td>
</tr>
</tbody>
</table>

Zone 3 was supported by broken rocks that had been consolidating gradually. The bed separation had basically closed within this zone. The convergence of the roof and floor of this entry increased steadily. A local acceleration in the rate of convergence in zone 3 indicates that the rear abutment pressures of polyfractured rock beams were formed by increasing load from the lower to the upper part gradually.

Parameters measured in each of the three zones were shown in Table 2.

After undermining, the maximum subsidence of the monitored roadway reached 1.1 m. The maximum vertical and horizontal convergences were approximately equal to about 400 mm. Thus the monitored roadway can be used for mining the upper seam after being maintained.

As mentioned above, the overlying rock zone distance above the lower seam is practically a fractured-rock beam supported by foundations with different properties and heights. This is because the whole rock was supported by the coal seam and the goaf.

Fig. 5 is the curve of convergence for the entry floor and deep borehole.

The subsidence curve of the fractured-rock beam is a negative exponential equation as follows:

$$W_x = W_0 \left(1 - e^{-az^2}\right)$$

where: $W_x$ - the convergence at a distance $x$ from the coal face;
$W_0$ - the convergence at a distance $L$ from the coal face;
$L$ - the distance between the coal face and the point where it is basically stable;
$z = \frac{x}{L}$

The coefficient $a$ is the parameter expressing the bending property of rocks. In this experimental panel, $a$ is nearly equal to 4.

The coefficient $b$ is the parameter expressing the difference between properties of front and rear foundations. The higher the rock is above the coal seam, the greater the value $b$.

The AusIMM Illawarra Branch, Ground Movement and Control related to Coal Mining Symposium August 1986

234
TABLE 2  PARAMETER MEASURED IN THREE ZONES AT THE MONITORED ENTRY

<table>
<thead>
<tr>
<th>Name of Monitored Point</th>
<th>( D_3 )</th>
<th>( D_4 )</th>
<th>( D_5 )</th>
</tr>
</thead>
<tbody>
<tr>
<td>supporting by coal seam zone 1</td>
<td>region (m)</td>
<td>6-12</td>
<td>15-9</td>
</tr>
<tr>
<td></td>
<td>max. location (m)</td>
<td>-6</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>conver. rate (mm/day)</td>
<td>2.8</td>
<td>13</td>
</tr>
<tr>
<td>bed-separated zone 11</td>
<td>region (m)</td>
<td>-12-31</td>
<td>-9-44</td>
</tr>
<tr>
<td></td>
<td>width (m)</td>
<td>19</td>
<td>35</td>
</tr>
<tr>
<td></td>
<td>convergence (mm)</td>
<td>16</td>
<td>30</td>
</tr>
<tr>
<td>broken rocks compacted zone 11</td>
<td>region (m)</td>
<td>-31-45</td>
<td>-45-63</td>
</tr>
<tr>
<td></td>
<td>location of max. conver. rate (m)</td>
<td>-42</td>
<td>-49</td>
</tr>
<tr>
<td></td>
<td>conver. rate (mm/day)</td>
<td>2.0</td>
<td>1.7</td>
</tr>
<tr>
<td>sum value of conver. (mm)</td>
<td>63.85</td>
<td>71.78</td>
<td>209.68</td>
</tr>
<tr>
<td>dynamic pressure</td>
<td>before first weight obvious not obvious</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

1. The closer the value \( b \) approaches to 3, the more gentle the slope of the curve becomes.

2. The closer to the coal seam the fractured-rock beam is, the steeper the angle of slope for the maximum convergence of the beam will be. Based on the statistical data the slope angle, however, is not a constant.

3. At the same time, the convergence curves at different heights separated clearly, and the difference between slope angles increased.

4. The slope angle at the inflectional point of the convergence curve of the fractured-rock beam was much steeper than those at other points, as Table 3 shows.

It is known from the geological column that two layers of sandstone 4.05 m and 4.6 m thick respectively are between the two coal seams. Ascending mining method cannot be used if a serious instability and a displacement of the sandstone occurred, and hence the upper seam is disturbed by extracting the lower seam.

It is known from the collected data that the fractured-rock beam with a different height has different instability conditions.

As a consequence, the overlying strata can be divided vertically upward, into three zones:

1. **Equilibrium-lust Zone.** In this zone, the coal face roof was in a state of solid cantilever. And it caved in the gob area where the horizontal force could not be transmitted. The height of caving
Fig. 6 - The inclination of rock beams with different height.

1. **Equilibrium Zone.** In this zone, the thick and competent roof rock is rotating and forming fractures so that the rock beam with certain bearing capacity has a sufficient horizontal supporting force to form the equilibrium structure. Influence of bearing capacity of the supports on the equilibrium of rock within this zone was not clear.

SAFETY DISTANCE IN ASCENDING MINING

The further the fracture beam reaches upward above the lower coal seam, the lower the slope of its subsidence profile (Table 4). Hence the equilibrium and continuity of the rock beam will be more easily maintained. Therefore the criteria of adopting ascending mining should be:

1. **Height** of the equilibrium lost zone is as much as 2.4 to 3.4 times the mining height in this experimental area.

2. Semi-equilibrium Zone. The fractured rock beam consisting of a thick and competent rock sometimes suffers from a lose of equilibrium. This results in weighting phenomena and formation of steps in the prop free area at the face. Therefore equilibrium within the...
TABLE 4 PARAMETERS OF CONVERGENCE CURVES

<table>
<thead>
<tr>
<th>Height above coal seam (m)</th>
<th>Coefficient a</th>
<th>Coefficient b</th>
<th>Location of inflexion (m)</th>
<th>Max. incline of draw</th>
<th>Max. angle of draw</th>
</tr>
</thead>
<tbody>
<tr>
<td>5</td>
<td>4</td>
<td>1.2</td>
<td>-3.5</td>
<td>5.7⁰</td>
<td>60⁰</td>
</tr>
<tr>
<td>10</td>
<td>4.1</td>
<td>1.0</td>
<td>-9.4</td>
<td>5.1⁰</td>
<td>60⁰</td>
</tr>
<tr>
<td>15</td>
<td>4.1</td>
<td>2.0</td>
<td>-17.5</td>
<td>1.4⁰</td>
<td>7⁰</td>
</tr>
<tr>
<td>25</td>
<td>3.3</td>
<td>2.4</td>
<td>21</td>
<td>0.6⁰</td>
<td>5⁰</td>
</tr>
</tbody>
</table>

\[ \frac{Q}{T} > \tan (\theta - \theta) \]

where:
- \( Q \) = the weight of roof block;
- \( T \) = horizontal force;
- \( \theta \) = angle of friction;
- \( \theta \) = angle of fracture surface from the vertical plane;
- \( \alpha \) = angle of inclination of the main roof block.

It will lose stability and steps will occur in the prop free front area.

The weight of the cantilever beam of immediate roof is supported partly by the resistance of coal face supports. The remaining part of the resistance will contribute to the balance of the strata in the semi-equilibrium zone. It will also control the inclination angle of the main roof block, utilizing the arching function of the equilibrium structure compensating the friction force to resist shear force. It will prevent clockwise rotation of the main roof block and steps in the roof.

The extreme tilt angle can determine the geometry deformation conditions of the contacting blocks.

When a rock block turns anticlockwise by an angle \( \alpha \) (Fig. 8), its left upper tip \( T \) moves backwards (towards the goaf) by a horizontal distance of \( h \) \[\cot \theta - \alpha \] \[\cot \theta - \frac{L(1 - \cos \alpha)}{2}\].

The sum of expand values appears in compressive deformation \( \Delta \) at front and rear contact portion of the blocks located near the inflexion.

\[ \Delta = \frac{h}{2} \left[ \cot \theta - \alpha - \cot \theta - L(1 - \cos \alpha) \right] \]

If the length of the contact face in deformation zone is \( \delta \);

\[ \delta = \Delta \tan (\theta - \alpha) \]

Assume the distribution of thrust force on the contact face has a triangular form. Then the horizontal force, \( T \) is:

\[ T = \delta \frac{g}{2} \]

Fig. 7 - Instability of fractured rock beam

Fig. 8 - Deformation and tilting of a block

where: \( g \) is the uniaxial compressive strength of rock. The vertical distance between two horizontal forces, \( A \), is:

\[ A = \frac{h - Z_{max} - \frac{2\delta}{3}}{3} \]

The balance moment resisting rotation of the block is \( T\alpha \). In the strata separation zone the rock layer was under the action of its weight and part of the overlying load. As this force increases, the slope angle of rock block also increases with the resulting increase of the horizontal force \( T \) and decrease of the distance \( A \). When the moment \( T\alpha \) reaches the maximum value, the slope is at the extreme incline and if the load increases further, the rock block will accelerate rotating until it loses its stability. These aspects were proven by model examination of the fracture beam.
Fig. 9 shows the recorded change in force T and vertical displacement A. The extreme balance angle can be determined by the following formula:

$$dT/dA = 0$$

As an example, the parameters in this experiment panel are: L=15 m, w=75°, h=4.05 m, $v_c = 7.5 \text{m}^2/\text{min}$. The calculated extreme incline angle is about 30°.

Therefore, based on the extreme incline angle, the residual bulking factor at the inflexion $K_I$ (caved rock zone) and $K_2$ (regular caved zone) the ascending mining safety distance between seams, $h$, can be determined by the following formula from the geometric conditions of extractions.

$$h > h' + \frac{n-h(k_1-k_2)}{k_2-1}$$

Where: $h'$ - the thickness of the lowest balance layer, m.

According to the measured data in experimental panel: $m=1.8$, $h'=4.05$, $v_c=0.3$, $K_1=1.32$, $K_2=1.15$, $K_2=1.06$, the calculated value of $h$ is: $h=13.9$ m.

Therefore it is found that ascending mining can be adopted where the distance between seams is greater than 13.9 m for similar geological conditions.

The safety distance for ascending mining depends on the mining height of the lower seam, and the properties and structures of laterally stratified rocks. The latter factor includes the existence or non-existence of the strong thick layer parameters K and L. Under normal conditions the stronger or thicker the strata, or the smaller the bulking factor, the greater the safety distance must be.

**WATER TRANSFERENCE LAW IN UNDERMINING ROCK MASS**

There were 6 deep boreholes (a1 a6) downwards to lower seam and 11 shallow boreholes (CI CI1) at the floor of the monitor entry. There was no water dripping from the roof of entry before undermining water accumulated as deep as 0.3 m in the entry, and all boreholes were filled up with water.

When the lower coal face advanced by 50.4 m, the first weighting of the main roof occurred, and one oblique fracture appeared on the half of the entry. This fracture was extending towards the water bearing sandstone roof of the upper seam. The flow rate through this fracture was 2 m³/hr. for 10 days. It is clear that the height of permeable fractured zone is not constant and is higher in the period of first weighting than that of normal mining.

The condition of water filling in deep-boreholes is shown in Table 3. In general, water flow disappeared at the distance of some 11.4 m in front of the coal face and restored 20.3 m behind. These distances are similar to the location of extending fractures and inflexion of convergence curves. The average width of previous extraction is 33m and the horizontal strata is 200 m.

The relationship between variations of water filling and the advance of a coal face in shallow-boreholes differ from those in deep-boreholes. For example, the water flow disappeared at the position 26 m ahead of the coal face on March 9th in deep-boreholes, in contrast to only 2.5 m ahead in shallow-boreholes. When the coal face had advanced 91 m from the setup entry although the water level in the shallow boreholes varied, the water table remained stable until 70 m behind coal face, i.e. the end of the monitoring program. Therefore the thickness of pervious fracture zone did not reach this level.

The thickness of the pervious caved and fractured zone was 32 m which was 12.8 times mining height at the first weighting period, and this zone ranges from 11.4 to 20.5 m in the direction of the coal face.

The principal factor which influences the thickness of a pervious caved and fractured zone is the strength and thickness of rock layer. The mining height and the time period after the lower coal face has passed also play an obvious role. The fractured overlying strata close to a coal face have a stronger water-permeability. Extraction of coal with ascending mining is inconvenient as the coal face of an upper seam approaches this region. Therefore the upper seam ought not to be extracted until the movement of overlying strata induced by extraction of the lower seam has ceased. It means that extraction work ought to progress out of the previously fractured zone.

According to the measured data the upper seam longwall face should start at least 3 months later than the extraction of the lower seam.

It must be pointed out that because of the influence of the angle of draw, the subsidence of the trough's edge after undermining is not in the direction of the seam inclination. For this reason the pillarless protection methods of the development entries or small width of chain pillars during the process of mining lower coal seam should be adopted, which will result in a uniform subsidence of the upper seam.
TABLE 3 THE CONDITION OF WATER FLOWING IN DEEP BOREHOLES

<table>
<thead>
<tr>
<th>Date</th>
<th>distance from coal face (m)</th>
<th>location of begin flow out</th>
<th>location of mature water fill</th>
<th>side of previous</th>
<th>d</th>
</tr>
</thead>
<tbody>
<tr>
<td>9/3</td>
<td>-25.9</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>8/4</td>
<td>- 9.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>14/4</td>
<td>- 4.70</td>
<td>-23.75</td>
<td>-18.5</td>
<td>26</td>
<td></td>
</tr>
<tr>
<td>15/4</td>
<td>-10.8</td>
<td>-25.7</td>
<td>-18.2</td>
<td>36.5</td>
<td></td>
</tr>
<tr>
<td>17/4</td>
<td>- 9.5</td>
<td>-18.1</td>
<td>-16.3</td>
<td>27.8</td>
<td></td>
</tr>
<tr>
<td>4/5</td>
<td>- 5.64</td>
<td>-20.5</td>
<td>-18.3</td>
<td>31.9</td>
<td></td>
</tr>
<tr>
<td>16/5</td>
<td>-13.4</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>average</td>
<td>-11.4</td>
<td>-20.5</td>
<td>-18.3</td>
<td>31.9</td>
<td></td>
</tr>
</tbody>
</table>

MINING PRACTICE OF UPPER SEAM EXTRACTION WITH ASCENDING ORDER

In Hua San colliery, the upper seam was extracted with no difficulties, where the distance between two seams is 19.8 m and the lower seam has been extracted 3 years ago. The length of the coal face of the upper seam was located above the lower seam roof. During mining of the upper seam, a study on rock pressures was conducted simultaneously at two districts of the coal face: under mined and over virgin ground. It is found:

1. The periodic weighting intervals was 8-9 m, in the coal face. During weighting period the dynamic pressure was smaller at the undermined district and the pressure value fluctuation was also small. Therefore, if the roof of the upper seam is a strong layer and is located within the fracture zone of the lower face in ascending mining, roof caving in large areas during mining the upper seam will not occur.

2. The average load of the face support in original district is 37% greater than that in the undermined district and the roof convergence is 23% smaller. The closure of supports is 16% greater.

3. In the undermined district the immediate roof caved easily and its fragments were small. The pressure on supports of entries was also small, and the dynamic pressure was insignificant. There was a small separation of 2-5 m between the coal seam and the roof.

4. The methane emission from the lower working area upwards was noticed and its concentration was 42% greater than that in descending mining.

CONCLUSIONS

1. As the overlying strata bend, subside and stabilize, the oblique fractures and bed separations produced by undermining undergo the "open and close" process. The strong layer in this zone can be considered as fractured block beam.

2. The overlying strata can be divided into three zones: in-equilibrium (unstable), semi-equilibrium and equilibrium (stable). The upper seams located in the equilibrium zone can maintain its bed continuity, and the step-like unevenness will not affect the extraction in ascending order.

3. The safe distance depends on the mining height of the lower seam and the properties and structures of interstratified rocks. This can be determined by the following formula:

   \[ H > \frac{m-h(K_1 - K_2)}{K_3 - 1} \]

4. The upper seam ought to be extracted after the movement of overlying strata (induced by extracting the lower seam) is stabilized. Using the pillarless methods in a lower seam is favourable with resulting uniform subsidence of the upper seam.

REFERENCES


The AusIMM Illawarra Branch, Ground Movement and Control related to Coal Mining Symposium August 1966

239