Evaluation of Mine Scale Longwall Top Coal Caving
Parameters using Continuum Analysis

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Abstract

A mine-scale analysis of Longwall Top Coal Caving (LTCC) is performed using a continuum mechanics finite element solver called COSFLOW. The uniqueness of cosflow is that it incorporates Cosserat continuum theory in its formulation for describing the load deformation of bedded rocks.

It is shown that such a continuum based code is valuable for assessing the feasibility of introducing LTCC in any mine. Various LTCC parameters, for example chock convergences, top coal failure behaviour, strata caving mechanism, abutment stresses and vertical stresses, were evaluated for a mine using COSFLOW.

Key word: LTCC, COSFLOW, Cosserat, FEM, Modelling and Simulation

1. Introduction

Longwall Top Coal Caving (LTCC) is an underground mining method developed for thick seam extraction. Depending on the coal seam condition various layouts can be considered for the LTCC [1]. The full seam height LTCC mining layout has been chosen for this work as it takes full advantage of support pressure in fracturing and breaking top coal which increases caveability and drawability of top coal, less gateroad development and maintenance, simple mining system and less equipment [1].

In full height LTCC layout, the thick coal seam is divided into two parts - bottom and top parts, Figure 1. The shearer slices the bottom of the seam and the top coal is allowed to break under gravity. Normally, the bottom coal has a uniform thickness. The bottom part is extracted using a normal longwall
extraction technique with 2.0m - 3.5m face height. The extracted coal obtained from the face is transferred via a conveyor belt, also known as Armoured Face Conveyor (AFC), installed in front of the hydraulic support near the cutting face. An additional conveyor is attached at the rear of the support to collect the fractured coal falling from upper section of the seam.

The LTCC technique is used extensively in China with over 100 faces producing over 200Mtpa in conditions ranging from soft (<10MPa UCS coal) to hard (>50MPa UCS coal) [2]. A lot of mines in China report that the LTCC is providing high productivity and efficiency in application. It is also cost effective because the shearer slices only the bottom part of the seam and the top coal fractures due to gravity [4-8]. The only additional cost will be to add the rear conveyor and slightly modify the chocks on the existing normal longwall equipment. But there are numerous intrinsic and non-intrinsic parameters which govern the feasibility of LTCC in any mine, and have to be evaluated properly. The intrinsic parameters are thickness of the coal seam, coal strength and deformation properties, inclination of the coal seam, roof sandstone strength and deformation properties and coal geology. The non-intrinsic parameters are existing equipment/support for the normal longwall extraction, life of the mine, financial health of the mine and a detailed geological study of the mine.

The LTCC method can be used to successfully extract up to 12 meters in thick seams. A successful application of LTCC depends on the caving mechanics/layout and gas and dust control options over the rear AFC [2,3]. Under favourable conditions, LTCC is an economical underground mining technique. The top coal recovery depends on the top coal fragmentation mechanism and fragment size distribution generated during normal longwall extraction of the bottom coal. The classical continuum mechanics is well suited for the stress and deformation study of normal longwall and LTCC.

The in-situ stresses should be considered properly to assess the risk and efficiency of a modern day longwall mine. In addition, the stresses released and breakage mechanism of top coal should also be considered for a
feasibility analysis of LTCC. The better implementation of the LTCC may be achieved through past experience of mining in identical geological and excavation situations or from a detailed numerical modelling of the LTCC using comprehensive and accurate mine parameters. The past experience from a mine can be a challenging task for a new mine as none of the mines are same. Similarly a detailed modelling of a mine is also a challenging work as mine as a whole can be visualized as a heterogeneous structure with built-in imperfections. Compared to analytical modelling, numerical modelling provides relatively accurate results. In the past, different two dimensional [9,10] and three dimensional [11] numerical simulations were being used to analyse the feasibility study of LTCC in different mines. This paper presents a detailed three dimensional analysis of various parameters which may impact successful implementation of LTCC. Support (chock) convergences, top coal failure mechanism and recovery estimation for top coal, roof caving mechanism, abutment stress and vertical stresses were examined for a mine using continuum mechanics modelling [12,13].

A 3D finite element code called COSFLOW is used in this study. A unique feature of COSFLOW is the incorporation of Cosserat continuum theory [12] in its formulation. In the Cosserat model, inter-layer interfaces (joints, bedding planes) are considered to be smeared across the mass, i.e. the effects of interfaces are incorporated implicitly in the choice of stress-strain model formulation. An important feature of the Cosserat model is that it incorporates bending rigidity of individual layers in its formulation and this makes it different from other conventional implicit models.

2. Model Development

Existence of easily caving main roof and maximum recovery of the top coal are the pre-requisites for a successful longwall top coal caving. The roof strata that do not cave during longwall extraction may cause severe geotechnical and safety problems such as face instability, roof guttering, windblasts. Thus it is imperative that roof caveability is appropriately assessed. COSFLOW was used to analyse the caveability of immediate sandstone roofs at the mine.

A model was developed with a plan area of approximately 9km², which comes from the Singarani mine. It was necessary to use such a large area to minimize the boundary effects. The plan view of the model can be seen in Figure 2. The plan view consists of 11418 elements. The mine plans for a mine formed a basis for the geometry of the model. The mesh was then rotated to align with the principal horizontal stress direction. The close up view of the region (fine mesh area) where the chocks were installed is shown in Figure 2.
In this study first 750m retreat along a 250m wide panel is considered. The coal seam was divided into bottom and top parts. The bottom part was assigned a uniform height of 3.5m and the top coal was assigned with remaining height of the coal seam. The bottom part was extracted using the normal longwall extraction method. The top coal was extracted in such a way that it mimics the real top coal caving mechanism, i.e. top coals located behind the chocks were excavated as soon as the chocks were advanced.

The longwall panel was extracted in steps to minimize the dynamic response of the model. First 500m was extracted with coarser but gradually fining steps. Next 50m was extracted with further fining steps. Then, the last 200m was extracted with uniform steps of 0.8m. The chocks are 1.75m wide and 4.8m in length. These steps are shown by dark lines running parallel to the excavation face in the figures discussed in the results and discussion section.

In the model, the top coal is extracted using the forward diagonal method as shown in Figure 1. When the ‘n’ excavation step of the bottom coal is being extracted the fractured coal generated in the top coal at ‘n-1’ excavation step is also being collected as shown in the figure.

A number of detailed 3D COSFLOW simulations were conducted to assess the caveability, chock capacity and behaviour of top coal for the high capacity longwall top coal panels at the mine. The models were prescribed roller boundaries on the four sides and the base. Initial stress field equal to the in situ stress was prescribed.

In this study, 1100t chocks and 250m wide panels were selected. Each model was discretised using approximately 1.5 million finite elements. The parallel version of COSFLOW was used. The problem region was split into 32
different regions and analysed using 32 processors. The run time for each simulation was approximately 8 weeks.

The constitutive model employed for the rock blocks was the elastic perfectly plastic Mohr-Coulomb model. The softening/hardening model was also used to compare with the standard models. The parameters which are a function of plastic strain were assumed based on the experience. The constitutive model used for the joints were the standard Mohr-Coulomb slip model. A typical core log of the model is shown in Figure 3. The figure also shows the zoom-in area of the core log.

A number of models were simulated to study the effects of variation on strength of sandstones and coal. Table 3 presents the various rock mass strength properties used in the COSFLOW simulations.
Case 1 represents the properties shown in Table 1 and Table 2. In this case all the sandstones were massive without any bedding planes; however, planes of weaknesses were introduced in-between SS40 and IIA, and IIA and SS50.

Table 1 Rock mass properties obtained from laboratory study.

<table>
<thead>
<tr>
<th>Stratigraphic Unit</th>
<th>Density (kg/m³)</th>
<th>σ_T (MPa)</th>
<th>UCS (MPa)</th>
<th>E (GPa)</th>
<th>φ</th>
<th>K (GPa)</th>
<th>G (GPa)</th>
<th>C (MPa)</th>
<th>ψ</th>
</tr>
</thead>
<tbody>
<tr>
<td>SS60</td>
<td>2192.0</td>
<td>0.75</td>
<td>15.10</td>
<td>7.17</td>
<td>35.5</td>
<td>3.33</td>
<td>3.14</td>
<td>3.88</td>
<td>5.00</td>
</tr>
<tr>
<td>IIIB</td>
<td>2080.0</td>
<td>0.48</td>
<td>3.84</td>
<td>3.00</td>
<td>40.0</td>
<td>1.43</td>
<td>1.30</td>
<td>0.90</td>
<td>7.50</td>
</tr>
<tr>
<td>SS50</td>
<td>2234.0</td>
<td>0.78</td>
<td>15.62</td>
<td>6.66</td>
<td>39.8</td>
<td>3.43</td>
<td>2.83</td>
<td>3.66</td>
<td>5.00</td>
</tr>
<tr>
<td>IIA</td>
<td>2290.0</td>
<td>1.01</td>
<td>8.16</td>
<td>3.00</td>
<td>40.0</td>
<td>3.33</td>
<td>1.11</td>
<td>1.90</td>
<td>7.50</td>
</tr>
<tr>
<td>SS40</td>
<td>2187.0</td>
<td>0.58</td>
<td>13.00</td>
<td>5.70</td>
<td>39.3</td>
<td>3.09</td>
<td>2.39</td>
<td>2.74</td>
<td>5.00</td>
</tr>
</tbody>
</table>

Where, σ_T is tensile strength, UCS is Unconfined Compressive Stress, E is Elastic Modulus, φ is friction angle, K is stiffness, G is bulk modulus, C is cohesive strength and ψ dilation angle.

Other stratigraphic Units

BASE

Table 2 Coal mass properties used in the model.

<table>
<thead>
<tr>
<th>Stratigraphic Unit</th>
<th>Density (kg/m³)</th>
<th>σ_T (MPa)</th>
<th>UCS (MPa)</th>
<th>E (GPa)</th>
<th>φ</th>
<th>K (GPa)</th>
<th>G (GPa)</th>
<th>C (MPa)</th>
<th>ψ</th>
</tr>
</thead>
<tbody>
<tr>
<td>Top coal</td>
<td>1529.0</td>
<td>1.07</td>
<td>12.00</td>
<td>3.00</td>
<td>40.0</td>
<td>1.25</td>
<td>1.36</td>
<td>2.80</td>
<td>7.50</td>
</tr>
<tr>
<td>Bottom coal</td>
<td>1529.0</td>
<td>1.07</td>
<td>12.00</td>
<td>3.00</td>
<td>40.0</td>
<td>1.25</td>
<td>1.36</td>
<td>2.80</td>
<td>7.50</td>
</tr>
</tbody>
</table>

Where, σ_T is tensile strength, UCS is Unconfined Compressive Stress, E is Elastic Modulus, φ is friction angle, K is stiffness, G is bulk modulus, C is cohesive strength and ψ dilation angle.

In Case 2, the coal strength was doubled compared to Case 1. The elastic modulus was same as in Case 1. In Case 3, in addition to doubling the strength the elastic modulus of the coal seam was also doubled. In Case 4 strengths of all sandstone units were doubled. Case 5 was a strain softening model with residual coal strength of 0.4 MPa and the slope of the softening curve of 3 GPa.

Table 3 Different cases of the model.

<table>
<thead>
<tr>
<th>Case</th>
<th>Property</th>
</tr>
</thead>
<tbody>
<tr>
<td>Case 1</td>
<td>Massive sandstones, I, A, IV</td>
</tr>
<tr>
<td>Case 2</td>
<td>Case 1 + double coal strength</td>
</tr>
<tr>
<td>Case 3</td>
<td>Case 1 + double coal strength and double Elastic modulus</td>
</tr>
<tr>
<td>Case 4</td>
<td>Case 1 + double strength of all sandstones</td>
</tr>
<tr>
<td>Case 5 (strain softening)</td>
<td>Case 1 + softening slope of 3 GPa and residual strength of 0.4MPa</td>
</tr>
</tbody>
</table>
The chocks were modelled as finite elements. The deformation at each node of the elements was noted and averaged out to calculate the average vertical displacements of the elements. The element expansion and shrinkage were also noted and considered in the averaging process. The average deformation of the elements was considered as chock convergence.

3. Results and Discussion

3.1 Chock convergence

The comparison of chock convergences for a 250m wide panel with Case1 properties at different locations along the mining width is shown in Figure 4 (top left). The chock convergence plots are analysed only in the fine mesh area as shown in Figure 2. In other words, the abscissa values start from 550 from the start line, ie 20m in the fine mesh area is equivalent to the 570m (550m + 20m) from the start line. It can be seen from the figure that the middle part is sagging and yielding higher chock convergence compared to the chocks located towards the panel ends. A cyclic loading can be observed in the middle chock. The figure (top right) also shows an effect of coal seam modulus in convergence i.e Case2 and Case3 results. Case3 has double the coal seam modulus than Case2. From the figure it seems the modulus of the coal layer does not have much effect on the convergence. Similarly the figure (bottom left) shows the effect of coal seam strength in chock convergence. Case2 has double the coal strength than Case1. It is interesting to note that the coal strength affects the loadings on the chock with fewer fluctuations and relatively lesser convergence compared to the weaker coal seam (Case1).

Figure 4 (bottom right) presents the effect of strength of sandstones (Case1 and Case4) and the strain softening models (Case5). Compared to the standard models, strain softening models show some peaks with higher convergences suggesting that the top coal with the strain softening characteristics (Case5) may have caused breaking of the overlying sandstone unit (SS40) just above the chocks or in close proximity to the chocks. It is interesting to note that in almost all the models (standard and strain softening) maximum peaks are obtained at around 80m-900m and 140m-145m longwall retreat distances. This may be attributed to the local variation in mine geology.
Case1 and Case2.

Analysis of excessive chock convergence

In the convergence graphs discussed above, chock convergence can be seen to attain maximum values at some specific locations; this is especially true for the case with strain softening coal. Figure 5 shows the fracturing of SS40 (for the strain softening coal, Case5) as well as the thickness distribution for SS40 plotted on vertical cross-sections taken across the longwall face at a number of locations. The left pictures show fracturing pattern and the right pictures show thickness distributions. As can be seen from the figure SS40 seems to be breaking ahead of the mining face at locations 40m and 100m as marked in (Figure 4); whereas at location marked 80m in (Figure 4), the failure of SS40 can be seen to occur very close to the face almost right above the chocks. SS40 at this 80m location is thickest (about 29m thick) compared to its thicknesses at other locations. This observation infers that the mine geology (thickness) is a major factor to cause the change in the fracturing location of SS40 bringing it very close to the face line resulting in higher chock loading. Figure 6 shows the thickness of SS40 along the mining direction, which clearly indicates that SS40 attains a maximum thickness at this location. In most of the cases the thickness of SS40 varies between 22 to 26m (less than 28) yielding relatively less chock convergence values. This indicates that at about 29m thickness SS40 may act critically exerting excessive load and inducing rapid chock convergence. It is worthwhile to note that in the case with strain softening coal, even the upper lying sandstone units (SS50 and SS50) seems to fracture right above the chock positions (see Figure 10 and discussed in section 3.3) thus exerting excessive load on the chocks and yielding relatively much higher chock convergence as noted in Figure 4.
3.2 Top Coal Caving Behaviour

Top Coal of a thick seam was discretised into seven elemental layers. These different elemental layers were necessary to investigate the nature and evolution of failure profile within the top coal. It was assumed that the extraction of bottom coal can have impact on top coal as well as overlaying strata on fracture. For the considered geological condition, top coal, SS40, SS50 and SS60 can be affected during the excavation of the bottom coal.

The yield plots are taken along the vertical cross-section passing through the mining face (i.e. perpendicular to the mining direction). On the plot except for the case of zero yield (i.e. coal never yielded and remained elastic), every
other values indicate that the coal has undergone one or another form of yielding (i.e. either tensile or shear or combination of both). On the figure red colour shows the yielded/fractured coal and blue colour shows the intact coal.

Figure 7 shows the yield of top coal at different excavation stages. As observed from the pictures, almost all of the top coal is yielded. The bottom part of the top coal has yielded in every excavation step for the considered cases. Except in Case2 and Case3, top part of the top coal is also predicted to fracture in all other cases. In Case2 and Case3, top two out of seven elemental layers can be seen to be only partially fractured. It is worthwhile to note that in Case2 and Case3 coal mass strength is assumed to be 24MPa which represents an extremely rare case. Comparison between Case2 and Case3 suggests that although the coal seam strength is same, the elastic modulus can play some role in fragmentation mechanism, i.e., coal with higher stiffness will possibly attract higher stresses with the same amount of deformation of the surrounding rocks thus favourably assisting in the fracturing process.

In most of the cases, except strain softening model Case5, top coal at the right hand top corner is not yielded. Recalling the strength of different cases, Case1 has weakest sandstones, Case2 has stronger coal seam compared to Case1 and Case3 has double the elastic modulus compared to Case2. Case4 is strongest of all the cases. Case5 has a strain softening model. In general, the top coal can be seen to yield in all the cases indicating favourable LTCC conditions.
Figure 7 Comparison of yielding behavior of the top coal at 614m from the start line for different cases. Red colour shows the yielded/fractured coal and blue colour shows the intact coal.

To further investigate the fracture evolution pattern, the top coal has been examined at different distances from the mining face as depicted in Figure 8, which shows the top coal yield at different distances from the mining face at different excavation steps. These vertical cross-sections are again aligned perpendicular to the mining direction, i.e., parallel to the mining face. It can be seen that the top coal is completely yielded above the chock and is partially yielded up to 2.4m ahead of the face. As the front abutment distance increases from the face, the severity of fracture reduces. The model results do not indicate any top coal yield at about 4.0m ahead of the face. This is further verified by Figure 9 where all the top coal above the coal seam can be seen to yield. The top coal ahead of the face can be seen to partially yield up to a certain distance as shown in the magnified picture.
Figure 8: Top coal yield at different distances from the face for Case 4. Red colour shows the yielded/fractured coal and blue colour shows the intact coal.

Figure 9: Vertical cross-section of the coal seam (including top coal) along the mining direction (left to right) for case 4.

It has to be noted here that these quasi-static simulations have some limitations. The dynamic failure of the lower parts of the top coal may facilitate the fracturing of the top parts of top coal. Except zero all values indicate failure.
### 3.3 Strata Caving Behaviour

In addition to chock convergence and top coal caving analyses, the models were also used to better understand the deformation behaviour of overlying strata specially the sandstone layers SS40, SS50 and SS60 during longwall excavation.

**Figure 10** Comparison of yield for different cases at different excavation steps as noted in the respective figures for SS40 (bottom layer of SS40), SS50 (bottom layer of SS40) and SS60 units. Except zero all values indicate failure.

Figure 10 shows a comparison of fracture for different cases at 718m from the start line for SS40, SS50 and SS60 units. The pictures suggest that the SS40
would cave in without any difficulty and failure extend up to the mining face. Depending on the nature and strength of sandstones the severity of the failure may be different. Compared to Case4, other cases have weaker sandstones, hence severity of failure is higher for other cases. The failure pattern of SS50 and SS60 shows that the failure is contained within the mining face however extended beyond the chain pillars. The strongest sandstones on Case4 show the failure lagging relatively behind the mining face and less extended on the side pillars. The SS50 and SS60 would show the arch type failure on the horizontal.

3.5 Abutment Stress

Abutment pressure may be considered as a main parameter affecting the caveability of top coal [1, 9] and the abutment pressure in front of the LTCC face is affected by the coal strength, seam thickness and ratio of top coal drawing. Figure 11 presents the distribution of maximum abutment stress along the centre of the 250m wide panel with 1100t chocks for Case4 and Case5 at a particular excavation step. The pictures on the top row show the abutment pressure measured for the bottom coal and the bottom row shows the abutment pressure measured for the top coal. The “H” on top right corner of each figure shows the location of maximum abutment stress while excavation. Being the model having strongest sandstones Case4 shows the highest abutment stress for the bottom and top coal. This can be attributed to the strength and massive nature of sandstones present in the model. The similar observation has been noted for the chock convergence (discussed in section 3.1). Case4 shows the maximum abutment stress of approximately 65 MPa for the bottom coal and 52 MPa for the top coal, which is much higher than Case1.

For other cases except, strain softening case (Case5), the bottom and top coal abutment pressures were noted between 50-65MPa and 34-53MPa respectively. The stiffness and strength of the coal also seem to have effect on the abutment stresses similar to the observation noted in top coal yield. The Case2 has stronger coal strength but low stiffness compared to Case3, as a result Case2 show lower abutment stress on top and bottom coal seam compared to the Case3.

The stronger and more massive the roof strata the higher will be the abutment stress. At this depth of approximately 400m the in situ vertical stress can be expected to be around 10MPa. Thus it can be seen that the front abutment stress can reach as high as six times the pre-mining stress for the bottom coal and five times the pre-mining stress for the top coal.
Figure 11 Influence of rock mass strength (including coal strength) on the abutment stress in (Pa) for a 250m wide panel with 1100t chocks (BC = Bottom Coal and TC = Top Coal).

3.6 Vertical Stress

Figure 12 presents a plot showing the distribution of vertical stress for 250m wide panel for Case1 with 1100t chocks. The figure is plotted at the distance of 630m from the start line. The plot indicates that the vertical stress in the chain pillar can be more than 40 MPa i.e. more than 4 times the in situ pre-mining stress. This stress estimate can be used in the design of chain pillars. As seen from the colour code, the area above the chock is actively yielding (fractured) at present. The vertical stress above the chock is lower than the stress at the mining face. This indicates the load being carried by the chocks. It has to be noted here that once the lower part of the top coal fractures the stress distribution on the top coal will be changed.
4 Conclusion

**Chock convergence**

The chock convergence was not uniform due to the nature of the sandstones present in the mining zone. As anticipated the assumption of massive strata compared to bedded strata yielded higher chock convergence. The coal seam strength did not seem to affect the chock convergences.

The model predictions indicated that the chock convergence was likely to be between 20-30mm in average whereas it could reach up to maximum of 130mm, if chocks were left standing for a long time i.e. several hours. Thus it would be necessary to maintain a critical minimum retreat rate to mitigate the possibility of face instability.

**Top-Coal caving behaviour**

To investigate the top coal fracture evolution pattern seven elemental layers of the top coal starting from the top of the top coal to the bottom of the top coal for different cases were analysed. In most of the cases, the pictures suggest that the top coal would break easily.

**Strata caving behaviour**

The yield patterns of all the cases can be seen to be similar. This infers that almost all of them could follow a similar failure pattern, similar cave in condition and could show similar trend in chock convergences.

A number of following important observations could be made from strata caving analysis:

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Figure 12 Vertical stress in (Pa) for 250m wide panel for Case1 and 1100t chocks at the middle of bottom and top coal layers, 630m from the start line.
• Failure patterns of the respective sandstones were similar.
• Failure of SS40 is predicted to extend up to the mining face and beyond the mining face occasionally with the strain softening coal.
• In comparison to the fracturing in SS40 and SS50, the fracture front in SS60 (in some cases SS50 as well) seemed to take arch shapes and lag substantially behind the face line.

Abutment pressure

The maximum abutment pressures were measured at middle of the bottom and top coal layers. It was found that the bottom coal abutment pressure could lie between 50-65MPa and top coal abutment pressure lies between 30-53MPa for the standard cases (case1-Case4). Case4 with the strongest sandstones showed the highest abutment stress for the bottom and top coals.

The stronger and more massive the roof strata the higher would be the abutment stress. The front abutment stress is predicted to attain as high as six times the pre-mining stress in the bottom coal and five times the pre-mining stress in the top coal.

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