

Discontinuous Modelling of Stratum Cave-in in a Longwall Coal Mine in the Arctic Area

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Abstract This paper presents a discontinuous numerical approach for studying roof cave-in mechanisms and obtaining the required support capacity of longwall shields in a case study site, the Svea Nord coal mine in Svalbard. The block size in the roof strata and the mechanical parameters of the discontinuities for the numerical model were obtained through back-calculations. The back-calculations were conducted with a statistical method of design of experiment. Numerical simulations revealed that voussoir jointed beams are formed before the first cave-in occurs. The maximum deflection of a roof stratum in the study site prior to the first cave-in is about 70 % of the stratum thickness. The maximum span of the roof strata prior to the first cave-in depends upon the in situ horizontal stress state. The roof beams have a large stable span when they are subjected to high horizontal stress; but horizontal stress would increase the possibility of rock crushing in deflected roof beams. The simulations and field measurements show no periodic weighting on the longwall shields in the study site. Stiff and strong roof

beams would result in large first and periodic cave-in distances. As a consequence of having large cave-in distances, the longwall shields must have high load capacity, which can be calculated by the presented numerical approach.

Keywords Coal mining · Longwall mining · Numerical modelling · Cave-in · Cavability · Voussoir beam · Longwall shield

1 Introduction

In longwall mining, mined out voids are filled by cave-in materials. Stratum cave-in improves the stability of the longwall face owing to reductions in both the load on the longwall shields and the stresses in the abutments of the longwall face. Delayed roof cave-in could lead to severe consequences, such as face jamming, rock burst on the face and air burst. Therefore, a thorough understanding of the roof cave-in mechanism is helpful for ground control design in the planning stage; for instance, in determination of the load capacity of longwall shields and the length of the longwall faces.

A number of analytical, empirical and numerical methods have been developed to assess the cavability of roof strata in longwall mining. Analytical solutions are typically based on either theoretical approaches such as beam theory or on conceptual parameters like

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bulking factor. For instance, Obert and Duvall (1967) utilised beam theory to predict the length of the void where the first cave-in event occurs—called the first cave-in distance—as well as the subsequent periodic cave-in distances. However, analytical methods usually oversimplify the case and thus bring about deviations from the reality in complicated situations.

Some empirical methods were developed in India's coal mines to investigate the cavability of roof strata. Sheorey (1984) modified the rock mass classification system Q (Rock Tunnelling Quality Index, Barton et al. 1974) and used it to determine the first cave-in distance. Ghose and Dutta (1987) proposed a classification system for roof cavability in longwall mining by applying fuzzy set theory to a rock mass classification approach. Das (2000) proposed a classification system for roof cavability using data collected in some mechanised longwall panels in India. All the proposed approaches were developed on the basis of specific conditions, for instance a limited number of data, specific geomechanical conditions and panel widths less than 150 m.

In past years, a number of attempts have been made to study roof cavability using numerical methods. Vakili and Hebblewhite (2010) developed a cave-in criterion for Top Coal Caving Longwall Mining based on statistics of numerical results. They assumed in their model that the thick coal layer in the roof of the longwall panel contained sub-vertical and horizontal joints. Extrapolation of this method should not be done with ordinary longwall mines which have non coal roof strata. Singh and Singh (2009) studied the cavability of roof strata and the load on longwall shields through FLAC^(2D) modelling. The mining panels are not fully filled by cave-in materials in their model. Shabanimashcool and Li (2012) developed an algorithm for numerical modelling of longwall mining using FLAC^{3D}. This modelling takes into account roof cave-in, fracturing and consolidation of caved-in materials as well as the dynamic process of cave-in. The authors used a critical cave-in strain to outline the boundary of the cave-in zone. A critical cave-in strain needs to be determined for a given mine in the algorithm.

Other approaches were proposed to study the interaction between longwall shields and roof strata. They include the detached block model, the bulking factor method, empirical methods, monitoring of the cycling of the loads on longwall shields, neural

network and the ground response curve (Trueman et al. 2009). All of these approaches have their limitations in interpreting rock-shield interaction (discussed in detail by Trueman et al. 2009).

Cavability of roof strata and the required support capacity of shields in longwall mining, particularly in competent rock masses, are still not well understood even though the aforementioned studies exist. Cave-in is a progressive process which starts when the excavation reaches a critical length. Cave-in ceases when the void underneath is fully filled by caved materials. The roof rocks behind the excavation face load the shields via either downward deformation or the dead weight of the cave-in materials. Strong and massive roof strata do not cave in immediately behind the face, leaving a long overhang. Cave-in events occur periodically so that a periodic weighting is placed on the longwall shields. The cavability of a rock mass depends not only on its mechanical properties, but also on the height of the mined out void and the spacing of the bedding planes in the roof.

Roof strata behaviour is classified into four categories from the prospect of support loading, as shown in Fig. 1, these are: the main-roof convergence, the periodic cave-in, the detached roof block and the deflection of immediate roof (Barczak 1992). In the case of *main-roof convergence*, the immediate roof is composed of a thick and strong stratum. The support shields cannot provide the necessary load to prevent the roof fall as long as the stratum starts to move downward. The roof convergence ceases when the cave-in materials become consolidated enough. The *periodic cave-in* occurs when the roof strata are stiff and strong and the mined out void underneath is not fully filled by cave-in materials. *Detached blocks* are formed when the spacing of the cross-cut joints is short enough. The roof blocks distress after being mined out underneath and move downward freely. *Deflection of the immediate roof* occurs when the roof is composed of thin strata. The stiffness and the resistance of the longwall shields are important in this case. The shield support prevents the thin roof beams from separation.

In order to know how the longwall shields are loaded, it is necessary to find out which of the above cave-in categories are valid for a given mine. Numerical simulation, since it has the ability to consider a larger number of parameters than analytical methods, is an appropriate tool to investigate the mechanism of roof cave-in. Furthermore, discontinuous models can

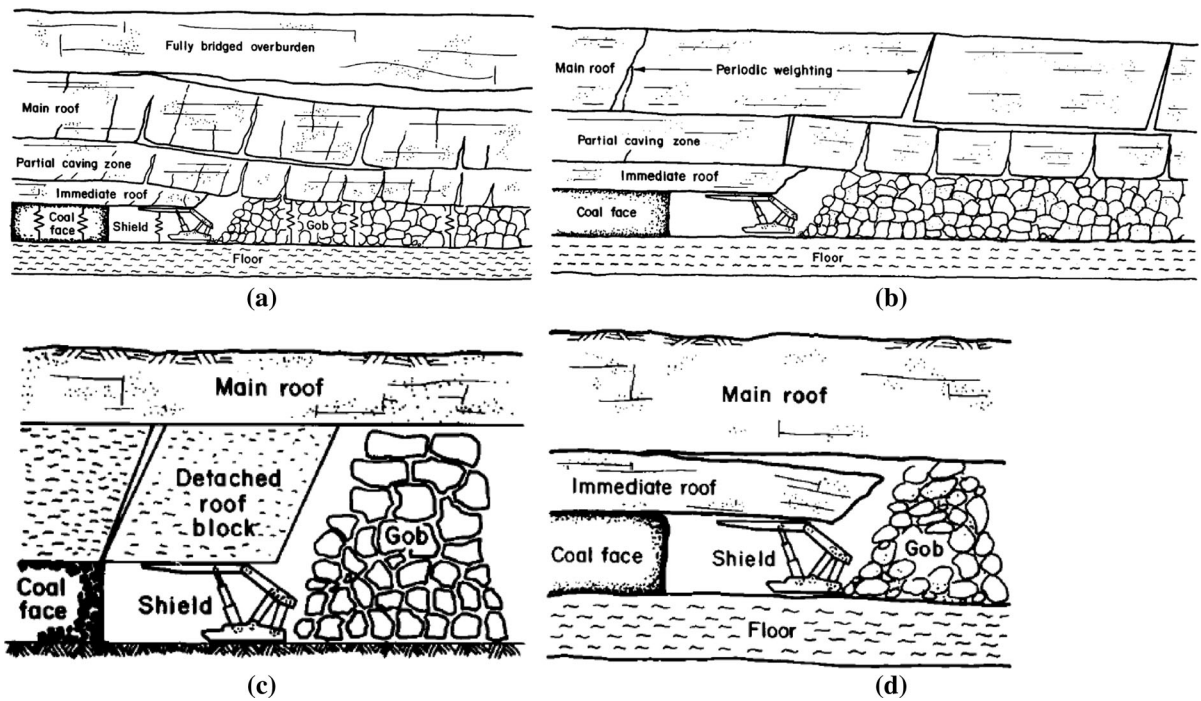


Fig. 1 Strata behaviour pertaining to shield loading: **a** main roof convergence, **b** periodic weighting, **c** detached roof block and **d** deflection of immediate roof (Barczak 1992)

simulate the behaviour of rock masses more realistically than continuous models, and therefore they are more useful for studying the roof cave-in mechanism.

The major objective of the study presented in this paper is to improve knowledge of roof cave-in related to longwall mining through discontinuous numerical modelling using UDEC. The study site is the Svea Nord coal mine in Svalbard. A back-calculation method, based on a statistical method of design of experiment (DOE), was implemented to obtain the required parameters for the modelling. The simulated loads on the longwall shields were then compared with field measurement data in order to calibrate the model. Beam theory was used to explain the cave-in mechanism. Sensitivity study of relevant geological and mechanical parameters was conducted and the desirable loading capacity of the longwall shields was obtained.

2 Cave-in Mechanisms

The cave-in of panel roof is an important stability issue in longwall mining. Roof cave-in is a complex

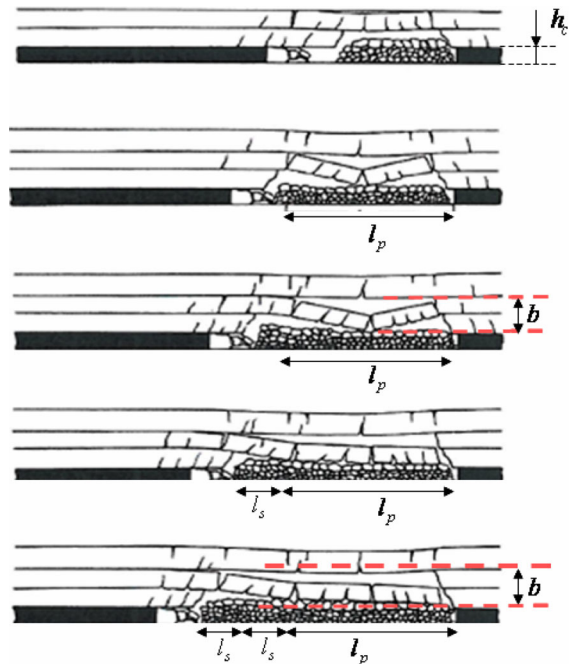


Fig. 2 A cave-in scenario in longwall mining (After Peng and Chiang 1984). h_c and b refer to the mining height and the maximum height of the cave-in zone, respectively

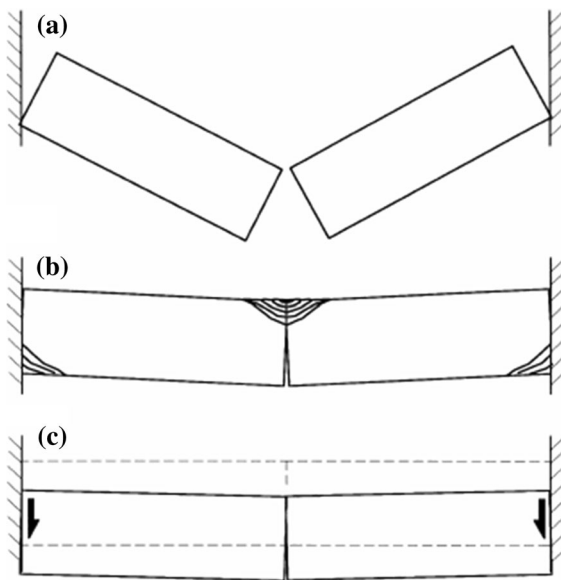


Fig. 3 Failure modes of voussoir beams: **a** snap-through or buckling; **b** crushing and **c** sliding (Diederichs and Kaiser 1999)

dynamic process involving rock fracturing, disintegration and movement. Roof cave-in results in a reduction in the stresses at the abutments and ahead of the longwall face, which is favourable for the stability of both the gates and the face. Peng and Chiang (1984) developed a scenario illustrating the process of roof cave-in based on field observations (Fig. 2). In their scenario, the first cave-in event occurs when the length of the mined out zone reaches a critical distance l_p . The caved height in the roof reaches its highest in the middle of the caved zone where the panel void is fully filled by the cave-in materials, whilst it decreases toward the longwall face, leaving a dome-shaped roof composed of cantilever beams of laminated rock. Cave-in periodically occurs behind the face after every critical advance of l_s until mining is completed in the panel. The cavability of a panel roof is represented by the critical lengths of l_s and l_p . The longer the lengths l_s and l_p , the more violent the cave-in events are. The dynamic impacts of cave-in events are harmful to the stability of the longwall face.

The first cave-in event involves shear fracture ahead of the face, bedding separation above the mined void, deflection and collapse of voussoir beams. The periodic cave-in events involve similar processes to those in the first cave-in event, but it is cantilever beams instead of voussoir beams that collapse with every cave-in event.

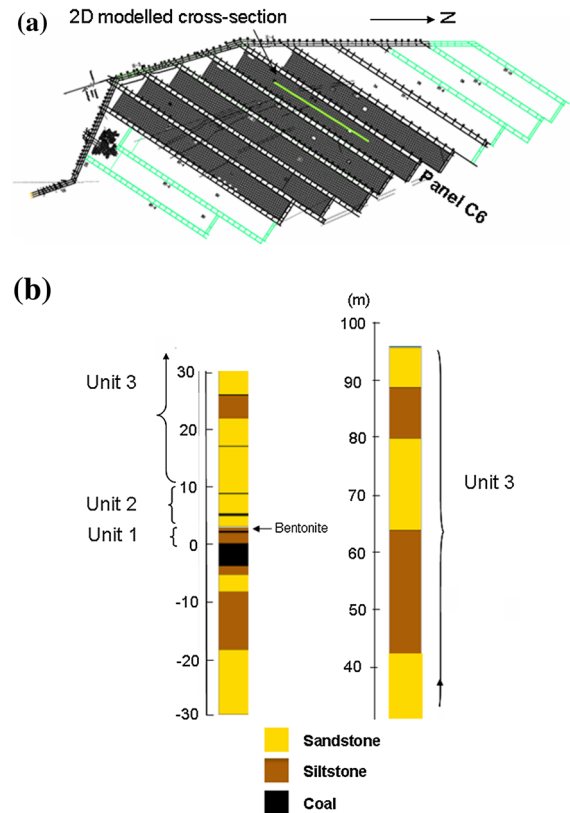


Fig. 4 **a** Mine map and **b** geological stratigraphy of the mine

Shear fracture occurs in the roof strata ahead of the face. The strata behind the face experience tensile fracturing and extension, resulting in separation of bedding planes. The stratified roof strata are cross-cut by sub-vertical joints which are either original or mining-induced. The rock blocks rotate and interlock each other, forming a voussoir beam above the mined out void. The deadweight of the beam as well as the ground pressure above is transferred to the abutments via the voussoir beam.

The primary failure modes of a voussoir beam are buckling, compressive failure (crushing) in the middle span and at the abutments, and slippage at the abutments (Fig. 3) (Diederichs and Kaiser 1999). Slippage occurs in thick beams of low span-to-thickness ratios, while crushing and buckling occur in slender beams with high span-to-thickness ratio. The voussoir beams are usually much more slender compared to the length of the panel. Therefore the first two are the dominant failure modes in longwall mines.

Voussoir beams are statistically undetermined. Some analytical methods have been developed for

voussoir beams, e.g. Sofianos (1996) and Diederichs and Kaiser (1999). Nomikos et al. (2002) and Tsesarsky and Hatzor (2006) have claimed that discontinuous numerical modelling, such as by DDA and DEM, is an appropriate approach to study such an undetermined problem.

3 Study Site

In the Svea Nord coal mine, the longwall panels are approximately 250 m wide and 2,500 m long. The overburden varies from 10 to 400 m in thickness, while most of the ground surface above the panels is covered by glaciers up to 250 m thick. A mine map showing the layout of the longwall panels can be seen in Fig. 4.

In-situ rock stresses were measured in six locations in the mine via the overcoring relief method. The principal stresses in the mine area are vertical and horizontal. The vertical stress σ_v is equal to the weight of the overburden, while the horizontal stresses in the central mining area are $\sigma_H = 10$ MPa and $\sigma_h = 8$ MPa. The horizontal stresses σ_H and σ_h are approximately parallel with and perpendicular to panel length, respectively. A number of thrust faults are present in the mining region which strike roughly perpendicular to the panels, but have limited persistence.

Figure 4b shows the geological sequence of the laminated sedimentary coal/rock layers below and above the coal seam currently being mined; the panel roof is located at 0 on the scale ruler. The roof strata are composed of siltstone and finely grained sandstone, interblended with thin coal and bentonite layers. A single bentonite layer lies around 2–3 m above the mined coal seam. The roof strata can be divided into three units: unit 1 ranges from 0 to 2.5 m above the coal seam and is composed of siltstone with thin coal interlayers; unit 2 lies from 2.5 to 10 m above the coal seam and is composed of sandstone and siltstone with coal interlayers; and finally unit 3 refers to rock strata more than 10 m above the mined coal seam and is mainly composed of relatively massive sandstone and siltstone. Unit 1 forms the immediate roof of the panels, with its upper border marked by a bentonite layer.

The rock mass is generally dry in the mining region, but wet in the cave-in area being fed by water from the

glacier above. However, the presence of water is not a pressing issue in the mine.

4 Numerical Modelling

A two-dimensional discrete element code, UDEC, was used to model the roof cave-in in the study site. The code is capable of simulating large displacement, separation and rotation of rock blocks. A cross-section in the middle of panel C6 was selected for modelling in order to fulfil the plane strain condition in a 2D model, Fig. 4a. The model consists of discrete blocks representing the discontinuous medium of the rock masses surrounding the panel. In the model, the rock blocks are deformable and obey a linear elastic stress–strain relationship. Discontinuities obey the Mohr–Coulomb constitutive law. The mechanical properties of the rock mass and discontinuities are presented in Table 1. The dimensions and boundary conditions of the model are illustrated in Fig. 5. The average spacing of bedding planes is assumed 1 m in the roof

Table 1 Mechanical properties of rock blocks and discontinuities in the UDEC model

	Young's modulus (GPa)	Poisson ratio	Density (Kg/m ³)
Rock blocks	12	0.25	2,500
	Cohesion (MPa)	Friction (Degree)	Tensile strength (MPa)
Discontinuities	0	30	0

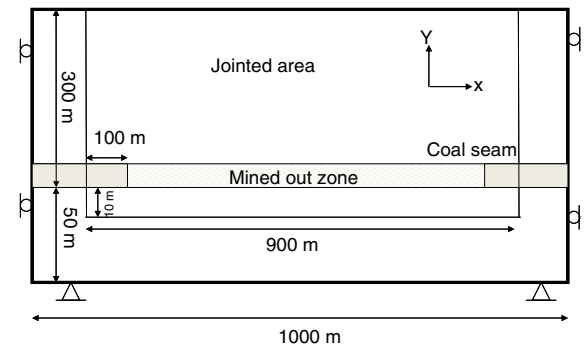


Fig. 5 Geometry and boundary conditions of the numerical model

strata. Most of the cross-joints in the roof strata are created by mining so that the spacing of the cross joints cannot be predetermined before mining starts. This was determined in the modelling by back-calculations at the moment of the first cave-in.

The normal and shear stiffness of joints, K_n and K_s , are also important in the simulation of both the cave-in process and the load on longwall shields. K_n and K_s are associated with the length of joints and the normal stress. K_n and K_s were determined by back-calculations at the moment of the first cave-in. In addition, the numerical results of the longwall shield load during subsequent cave-in events in panel C6 were compared with field measurement data for the sake of verification of both the numerical results and the applicability of the model in investigation of longwall shield loading.

The longwall shields in the study site have two hydraulic legs with total load capacity of 700 tonnes. The setting pressure of the shields is 70 % of their total load capacity. The shields have width of 1.7 m and their length is about 5 m. In order to maintain the statically equilibrium of the shields two stabilising hydraulic rams were embedded in the caving shields. Those rams maintain the stability of the shields against high vertical loading at the tip of the canopy. A realistic presentation of such a complex structure in the numerical modelling is important. Since the shields are mainly loaded vertically, their vertical load–deformation characteristics are considered in the numerical modelling. Therefore, a structure with same vertical stiffness as the shields was introduced to the numerical models. In addition, the statically stability of the simulated shield should be ensured under the vertical loading. The longwall shields were simulated as two stiff blocks connected by two support elements between the roof and floor in the model. The support elements are one-dimensional and thus could only sustain load–deformation in their axial direction. Two support elements with symmetric distance from the middle of the shield were assigned in the numerical models in order to maintain the statically stability of the simulated shield. The support elements have distance of 1 and 3 m from the longwall face. The two stiff blocks were stretched at the areas where canopy and base of the shields cover. Those stiff blocks transfer the vertical load induced by the roof/floor convergence uniformly to the support elements. Hence, the total vertical force provided by those

support elements is constant regardless of their location as far as the simulated shield is in a statically equilibrium. However, the vertical load in each individual support elements is dependent on their location.

The stiffness of the longwall shields in the mine is assumed 120 MN/m which was determined using the measurements conducted by Barczak and Schawemmer (1988). A longwall shield is about 1.7 m wide. In the model, the support elements had an out-of-plane spacing of 1 m and thus the stiffness of the support elements are 35 MN/m. The setting pressure of the longwall shields was applied as triangularly distributed stress along the canopy with the maximum stress at the end of canopy close by the gob side while it decreases to zero at the canopy tip, as suggested by Barczak and Gearhart (1992). The yielding of the shields did not included in simulations in order to clearly assess the required support capacity and periodic weighting of the shields.

4.1 Back-Calculations

Back-calculation is often used to indirectly determine rock mass properties. As mentioned above, the normal and shear stiffness of rock joints, as well as the spacing of the cross-cut joints in the roof strata were determined via back-calculations in this study. Field observations revealed that the first cave-in distance in the longwall panels was around 36 m; this distance was used to back-calculate the unknown parameters.

Generally, an iteration method is employed for back calculations. However, the iteration method is time consuming when several parameters are back calculated. A statistical method, called the design of experiment (DOE) technique (NIST/SEMANTech 2012), was used in this study for back calculations. DOE can be used to qualitatively evaluate the relationship between input variables (factors) and the output of experiments.

In using the DOE method, it is necessary to define the experimental objective and select the input/output variables. In our case, the unknown parameters are the normal and shear stiffness of rock joints, K_n and K_s , and the spacing of the cross-cut joints. The first cave-in distance (l_p) is the experimental response (i.e. output). Furthermore, the variation range of each parameter needs to be pre-defined before experiments start. Joint shear stiffness K_s is associated with joint

Table 2 Data sets proposed by CDM to obtain the RSM

Group number	K_n (GPa/m)	K_s (GPa/m)	S_j (m)
1	0.5	0.005	1
2	0.5	0.005	2
3	0.5	2	1
4	0.5	2	2
5	20	0.005	1
6	20	0.005	2
7	20	2	2
8	0.005	1.0025	1.5
9	26.9	1	1.5
10	10	0.001	1.5
11	10	2.7	1.5
12	10	1	0.4
13	10	1	2.7
14	10	1	1.5
15	10	1	1.5
16	10	1	1.5
17	10	1	1.5
18	10	1	1.5
19	10	1	1.5
20	10	1	1.5
21	10	1	1.5
22	10	1	1.5
23	10	1	1.5

K_n is the normal stiffness and K_s the shear stiffness of joints in the model. S_j denotes the spacing of the cross-cut joints

dilation and friction angle, joint length, and normal stress on the joint plane (Barton and Choubey 1977). Bandis et al. (1983) produced a graph based on their test results and in situ observations. The length of cross-cut joints was assumed to be 1 m in the present study. An elastic numerical simulation revealed that the maximum horizontal stress in the roof of the mined out zone barely exceeds 12 MPa. Shear stiffness K_s was thus assumed to be in the range of 0.005–2 GPa/m. Joint normal stiffness (K_n) can be calculated from the joint shear stiffness and the normal stress on the joints (Bandis et al. 1983). During cave-in the normal stress on the cross-cut joints is very small and therefore the ratio of K_n/K_s in the present study was assumed to be a value in the range of 10–100, that is, K_n was in the range of 0.5–20 GPa/m.

Price (1966) developed a theory suggesting that the spacing of cross-cut joints in laminated sedimentary rocks is about the same as the thickness of the stratum,

while Narr and Suppe (1991) argued that such spacing is around 1–3 times the stratum thickness. The mine in this study is characterised by the presence of a number of mining-induced fractures/joints. The joint spacing used in this study was assumed to be 1–2 times the stratum thickness.

After definition of the variation ranges of the parameters, DOE experiments started using a number of data sets in the ranges given above. A linear, quadratic, or high order function relationship, denoted as the response surface method (RSM) (NIST/SEMANTECH 2012), was fitted to the experimental output. A RSM was then established to predict the nonlinear relationships between the input parameters and the outputs of the experiments. In the present study, the input parameters of the experiments are the normal and shear stiffness of rock joints, K_n and K_s , and spacing of the cross-cut joints (S_j). The output is the first cave-in distances (l_p).

Formation of the data sets requires special techniques based on the number of input parameters and also the number of experiments. The data sets, in Table 2, were selected with the help of the central decomposition method (CDM) (NIST/SEMANTECH 2012), which is suitable for experiments comprised of three input parameters.

The simulation results based on the data sets were then used to derive a quadratic function describing the relationship between the input parameters and the output. The response surface has the form of:

$$l_p = -4.656 + 4.0499K_n + 39.85K_s + 11.32S_j + 0.59K_nK_s - 0.67K_nS_j - 11.50K_sS_j - 0.10K_n^2 - 7.29K_s^2 - 2.05S_j^2 \quad (1)$$

where K_n and K_s are in MPa/m and S_j and l_p in m.

The back-calculations finally revealed that the most favourable parameters for the study site were $K_n = 18$ GPa/m, $K_s = 0.05$ GPa/m and $S_j = 1$ m. These data were used for the numerical modelling presented in the following section.

4.2 Modelling Results

The excavation sequence in the model was the same as the real one, that is, a 5 m-long advance for every excavation round. The stepwise excavation makes it possible to evaluate the process of cave-in and the load-deformation behaviour of the roof strata.

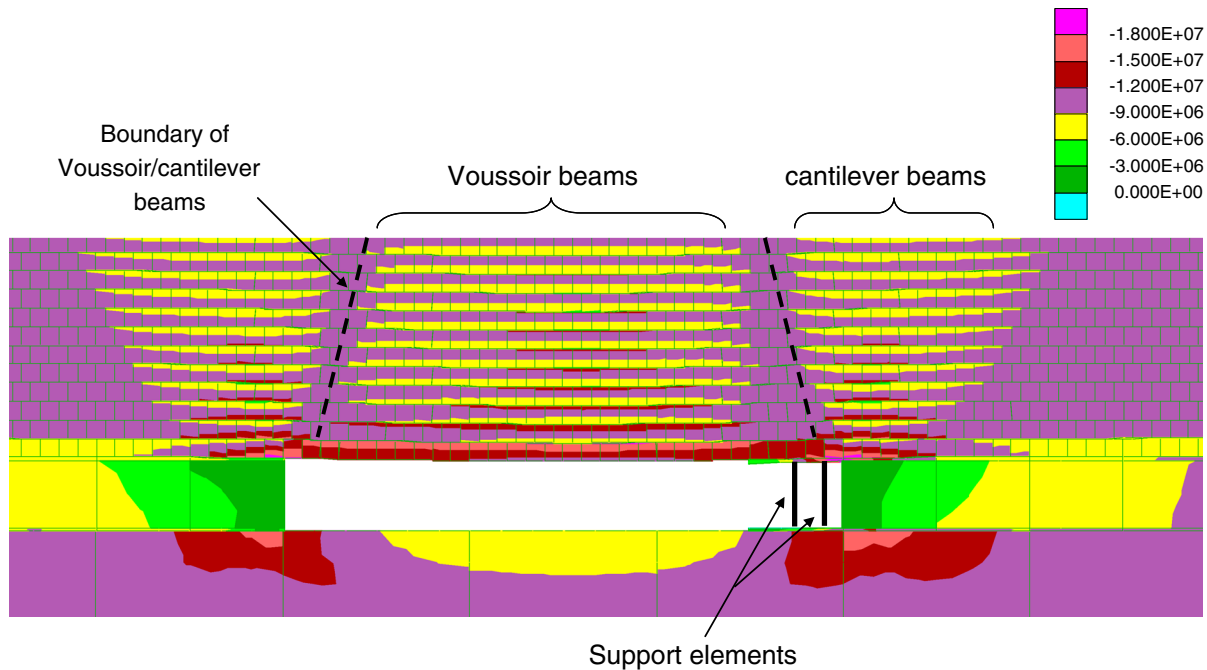


Fig. 6 Horizontal stresses in the roof strata when the length of the mined out panel is 30 m

With longwall face advances the vertical stress is elevated compared with its original state in the abutments. The roof strata above the mined-out zone separate from each other at the bedding planes under the gravity, deflect and even fail in buckling. The vertical stress concentration in the abutments, pillars and roof strata above the gates, affects the stability of the longwall face, gates and pillars.

The roof cave-in is associated with the separation of roof strata and the horizontal stress. The separation length decreases with the distance from the stope roof. The strata experience tensile fracturing, separation and rotation in the region between the roof surface and a critical depth where the bedding plane separation is negligible.

Figure 6 depicts the horizontal stress distribution in the panel roof just before the first cave-in, when the face has advanced 30 m since the start of mining. Because the horizontal stress is higher than the vertical stress, it may be expected that the maximum horizontal stress concentration should occur within the roof strata. However, the periodic fluctuations in the horizontal stress imply that the separated roof strata behave like beams. Beam bending induces stress fluctuations in individual roof strata. Above the

mined-out zone, the roof strata are separated from the host rock, and the rock blocks in the strata rotate and form a voussoir beam. Beam theory suggests that beam deflection will result in tensile stresses in the lower portion of the beam, with compressive stresses in the upper portion. The total horizontal stress in the beam is the superposition of the bending stress and the in situ horizontal stress in the strata. It is always compressive in the upper portion of the beam, but it may be compressive or tensile (or zero in the case of joint separation) depending on the magnitude of the in situ stress. In extreme cases, the beam deflection is so large that the beam could collapse either due to crushing or buckling. However, the UDEC model is not able to simulate rock crushing.

The horizontal stress in the roof strata in regions close to the ends of the mined out zone has a pattern different from that in the middle region (Fig. 6). The roof strata behave like cantilever beams with the fixed ends sitting deeply in the host rock. The transition zones between the voussoir beams and the cantilever beams incline toward the centre of the voussoir beam region. The cantilever beams are formed owing to the downward deflection of the voussoir beams above the mined-out zone.

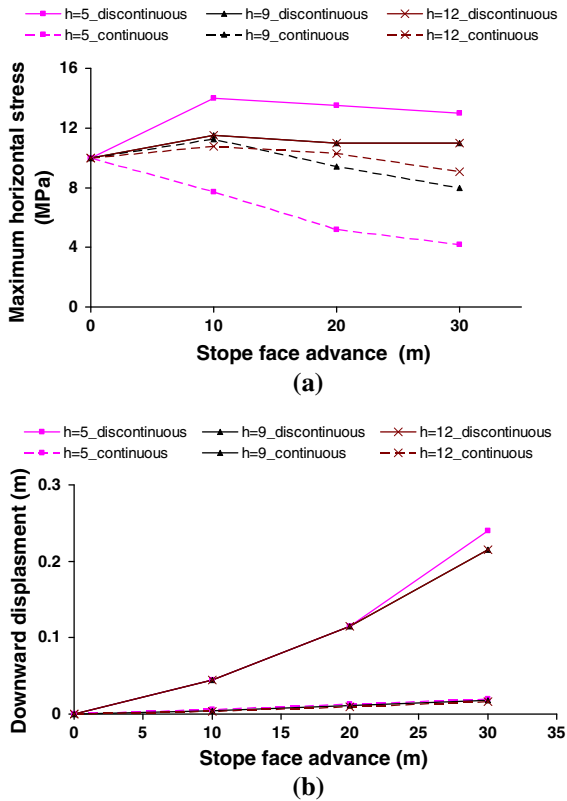


Fig. 7 a Horizontal stress and b downward displacement of the roof versus longwall face advance. *h* refers to the height of the roof strata/rock above the floor of the coal seam

After the first cave-in, the voussoir beams collapse and fill the mined void. The in situ horizontal stresses in the host rock surrounding the mined void then become elevated. An additional consequence of the roof cave-in is that the horizontal stresses in the cantilever beam regions become dramatically decreased.

In this paper, the deflection of a roof stratum is expressed as the ratio of its vertical deformation to its thickness. Based on the analytical methods by Diederichs and Kaiser (1999), the middle deflection of a voussoir beam deviates from the linear behaviour upon reaching 10 %. The beam collapses at a deflection of 25–35 % (Diederichs and Kaiser 1999; Sofianos 1996). In these methods, it is assumed that the confining horizontal stresses in the beam are induced by the rotation of rock blocks. As a result, the maximum sustainable deflection of a beam increases with the horizontal stress.

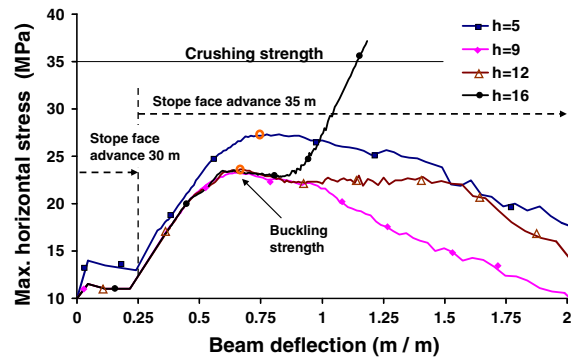


Fig. 8 Maximum horizontal stress versus deflection of the roof strata. *h* refers to the height of the roof strata above the panel floor

In order to investigate the beam formation and the bending process, the maximum horizontal stress at the middle span of a stratum beam was monitored in the model versus the downward deformation (Fig. 7). Plotted in the figure are also the roof displacement in a continuous homogenous medium, which are the results from a boundary element modelling using Examine2D (Rocscience). The numbers, 5, 9 and 12, mean the depths of the monitoring positions in the panel roof. The deflection increases and also the stresses in the roof strata are changed, with the longwall face advance. Figure 7a shows that the stress change starts as soon as mining begins. Roof deflection causes higher stress in the laminated strata than in the homogenous rock masses. The stress increment depends upon the extent of strata bending. It increases with the advance of the longwall face. The roof deflection is approximately 25 % when the face advances 30 m.

The maximum horizontal stress (σ_{xx}^{\max}) at the middle span is plotted versus the beam deflection prior to the occurrence of the first cave-in, shown in Fig. 8. The stress σ_{xx}^{\max} starts to dramatically increase when the longwall face advances to a distance of 30 m and reaches its ultimate at 35 m. The beam strata then fail in buckling. Buckling of the roof strata occurs at a deflection of 60–70 %.

The voussoir beams can fail in the form of crushing (Fig. 3b) when the maximum horizontal stress in the beam at the middle span reaches the compressive strength of the rock material (Diederichs and Kaiser 1999; Sofianos 1996). 50 % of the uniaxial compressive strength (UCS) of the rock material was used as

the crushing strength of the rock beams in this study. The UCS of the host rock in the study site is about 70 MPa; 50 % of the UCS, i.e. 35 MPa, was taken as the crushing strength of the voussoir beams. It can be seen in Fig. 8 that the maximum horizontal stress in the first 16 m roof strata drops after its ultimate value, marking buckling failure there. The stress continues to increase with deflection in the roof strata above 16 m, some portion of the rock there may fail in crushing when the stress is beyond the crushing strength of the rock.

The periodic cave-in length (l_s) is associated with the strength of the cantilever beams. The load on the longwall shields is related to the beam deflection and the deadweight of the cantilever beams in the worst case. The mechanical behaviour of a cantilever beam depends on the block size and mechanical properties of the cross-cut rock joints in the beams. A strong cantilever beam would result in a large overhang over the longwall shields, which may collapse in a violent manner when it fails. Weak cantilever beams are more favourable than strong ones for longwall mining.

The bulking effect of the cave-in materials cannot be taken into account in UDEC modelling. Therefore the mined void is not fully filled in the simulations and the UDEC modelling may overestimate the deadweight of the cantilever beams formed during periodic cave-ins. Figure 9 presents the simulation results of the load on the longwall shield and the field measurement data. The field measurements of the shield load were conducted when the longwall face had reached approximately the middle of panel C6 (Fig. 4a) and they were measured after every advances of the longwall face. The longwall face advance in the figure is the total length of the mined-out panel in the model. As aforementioned, the longwall shield was modelled by two support elements (El_1 and El_2) connecting two stiff blocks in the roof and floor of the face. The support elements of El_1 and El_2 are located at 1 and 3 m, respectively, behind the longwall face. Element El_2 is loaded more than El_1 because of a larger beam deflection occurring at El_2. The total load of the two simulated support elements, obtained by accumulation of the loads in El_1 and El_2, agrees quite well with the field measurement data. The approximately constant load on the shields implies that overhang of roof strata does not exist over the

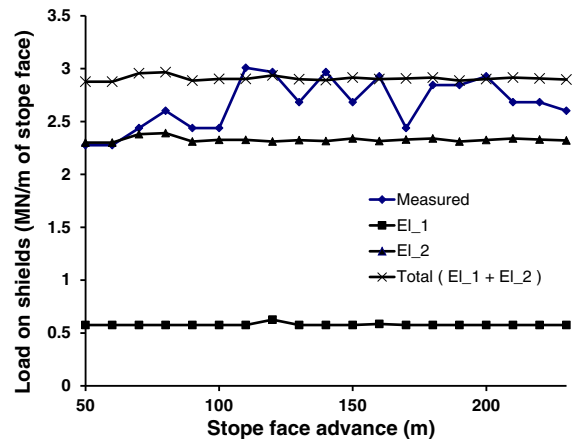


Fig. 9 Simulated loads on longwall shields and the field measurement data

longwall shields and the roof cave-in continuously occurs with advance of the longwall face.

5 Discussion

The cavability of the roof strata is associated with the mechanical properties of the strata, the block size, the in situ stress state and the height of the mined-out void. Beam forming and beam deflection are the major concerns in studying roof cave-ins. The aim of this section is to understand the cave-in mechanism through sensitivity analyses of the mechanical properties of the roof strata and the in situ stress state. In the simulations for sensitivity study, the input parameters varied in the following ranges: 5, 10 and 50 GPa for the E modulus of the strata; 1, 2 and 3 m for the spacing of cross-cut joints; and ($K = 0.33, 1$ and 2 for the in situ stress ratio. The normal and shear stiffness of the joints, K_n and K_s , were kept constant and the cohesion and dilation angle of joints were zero in the simulations. The mechanical characteristics of the longwall shields were also kept constant.

Recommendations regarding the implementation of the presented numerical method in other mines have also been presented. Advantages and shortcomings of the presented method were also discussed. The recommendations consider how to implement the statistical method of the DOE in coupling the

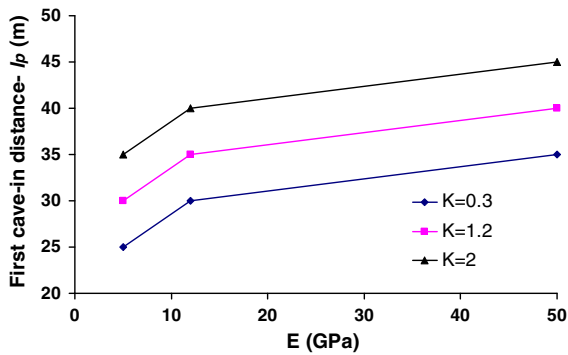


Fig. 10 First cave-in distance versus the E modulus of the rock blocks in the roof

numerical simulations with UDEC to study the cavability of the roof strata and obtain the loading on the longwall shields.

5.1 On the Cave-in Mechanism

The mining causes the roof strata to separate at the bedding planes. Rotation of the rock blocks form voussoir beams in the roof strata. The bending stress in the voussoir beams is related to the Young’s modulus of the rock, the stiffness of rock joints and also the number (or spacing) of rock joints. Large spacing of cross-cut joints and high Young’s modulus of rock would result in a stiff roof beam. The numerical results reveal that stiff roof beams can sustain high load (Fig. 10). As a consequence, the first cave-in distance (l_p) increases with the Young’s modulus of the rock blocks.

Furthermore, the horizontal in situ stress is helpful to roof stabilization. It prevents rock blocks from separating from each other whilst the roof deflects towards the mined out void underneath. Accordingly, the first cave-in length increases with the stress ratio K (Fig. 10).

The sustainable deflection of a beam refers the deflection at which buckling starts (Fig. 11). It increases with the stress ratio K . When K is smaller than 0.3, the sustainable deflection is about 25 %, which is consistent with the analytical solutions by Diederichs and Kaiser (1999). The sustainable deflection could be larger than 100 % when the stress ratio K is large enough, for instance $K > 2$. It should also be noted that rock crushing could occur for high stress ratios. For instance, at $K = 2$, the dominant failure mode could be crushing when the deflection of roof

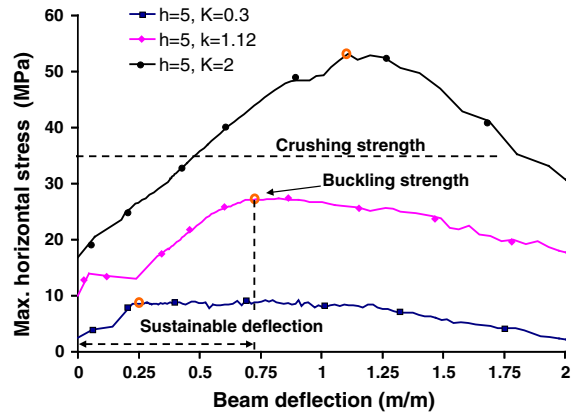


Fig. 11 Maximum horizontal stress in a stratum versus deflection for different stress K ratios

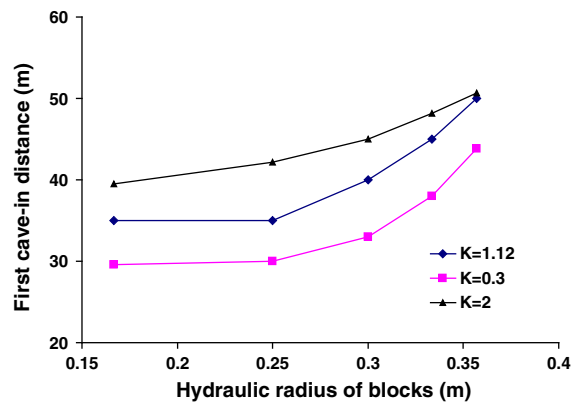


Fig. 12 First cave-in distance versus block hydraulic radius

strata is beyond 50 % for a crushing strength of 35 MPa (Fig. 11). Hence, not only would the longwall panels subjected to high in situ horizontal stress have a large cave-in distance, but it is also possible that roof strata would cave-in abruptly while their downward deflection is small.

Figure 12 shows the influence of block size on the first cave-in distance. In the figure, the hydraulic radius of the block (R_b), defined as the area of the block divided by the perimeter, is used to represent the block size. The stiffness of roof beams increases with R_b and the roof deflection would be smaller. However, an increase in the horizontal in situ stress makes the first cave-in distance less sensitive to the block size.

The load on longwall shields during the first cave-in is equal to the deadweight of a portion of the collapsed roof rock. Accordingly, a large first cave-in distance would result in a large shield load. During the periodic

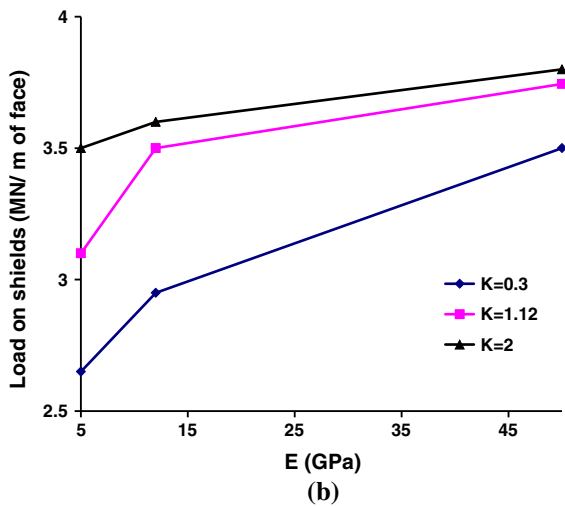
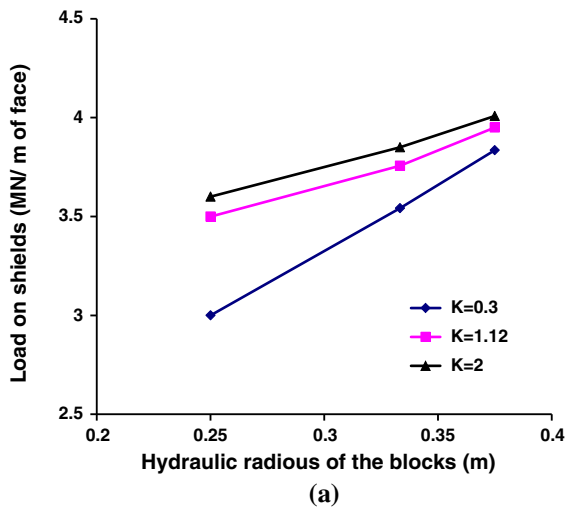


Fig. 13 Load on longwall shields versus: **a** Block hydraulic radius, for the case that the E modulus of the roof blocks is 12 GPa, and **b** the E modulus of the roof blocks

cave-ins, the shields are loaded by both the deflection of the cantilever roof beams and the deadweight of the collapsed roof rock. The shield load increases with increases in the block size and in the E modulus of the roof rock (Fig. 13). An increase in the horizontal in situ stress slightly increases the shield load during the periodic cave-ins.

5.2 Comparison of the Continuous and Discontinuous Numerical Modelling Results

In the FLAC^(3D) continuous model by Shabani-mashcool and Li (2012), a critical tensile strain

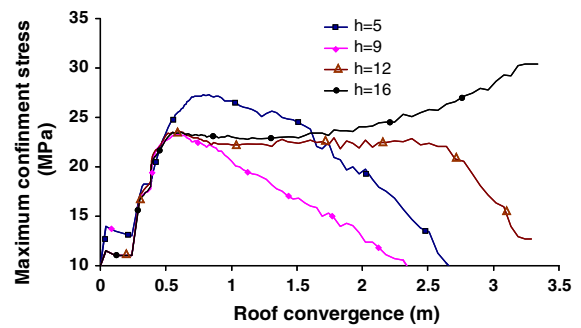


Fig. 14 Maximum horizontal stress in the roof strata versus roof convergence when the longwall face advanced from 0 (mining start) to 35 m in the panel. *h* refers to the height of the roof strata above the panel floor

value was utilised to identify the boundary of the cave-in region. The authors found that the roof strata above the mined out void tend to cave in when the maximum principal strain (in tension) is equal to, or larger than 5%. The height of the cave-in zone is a maximum of about 16 m above the coal seam. In this section we address the similarities and differences between the FLAC^(3D) continuous model and the UDEC discontinuous model of the study site.

In the continuous model, equivalent mechanical properties were applied to the rock mass and a strain softening constitutive model was used. The uniaxial compressive strength for the rock mass was 8 MPa. Discontinuity elements were added in the model to simulate bedding planes in the rock mass. Rock block rotation were not considered in the FLAC^(3D) model.

Figure 14 presents the maximum horizontal stress in the roof beams versus the middle-span convergence of the roof during the first cave-in when the advance distance reaches $l_p = 35$ m in the UDEC model. It can be seen in the figure that the maximum horizontal stress in the first 16 m roof strata drops after its ultimate value, implying that the roof strata fail in buckling, whilst the stress in the strata above continues to increase with the roof convergence, indicating that they are stable. By considering the bulking factor, the volume of the caved-in materials from the first 16 m roof strata is large enough to fully fill the mined out void. Therefore, the caved materials could provide support to the strata above and protect them from cave-in. The results of the UDEC modelling agree with the results of the FLAC^(3D) modelling.

5.3 Application and Limitations of the Presented Numerical Approach in Other Mines

The method presented here could be applicable in other mines at the design stage when there is a requirement to validate the outcomes of the empirical analysis and evaluate roof strata caving mechanism. At this stage since there are no field measurements and observations, back-calculation of the required mechanical parameters of the rock masses are not possible. However, with site investigations, for instance geological surveys and core logging data from exploration boreholes, it is possible to find most of the required parameters for the numerical simulations. The most important parameters for evaluating the cavability of the roof strata is the spacing of the cross-cut joints since they significantly influence the roof cave-in mechanism and cavability as well as the loading on the longwall shields. The spacing of the cross-cut joints can also be obtained from geological survey results if there is visible exposure of the roof strata. However, some of these cross-cut joints might be generated during mining due to the stress concentration in the roof strata ahead of the longwall face. Therefore, the spacing of the cross-cut joints may be shorter than the original natural value. Based on a geological model developed by Price (1966), the cross-cut joint spacing is around 1–3 times that of a stratum thickness. In order to have a safety margin on the obtained results, it is recommended in the numerical modelling to assume that spacing of the cross-cut joint is in the range of 1–3 times that of a stratum thickness. Later on, by implementing the DOE method and conducting a number of numerical simulations, a range for the first/periodic cave-in distances and loading of the longwall shields could be found.

The presented numerical approach does not consider the bulking of the caved materials. Therefore, the volume of the overhanging rocks from the longwall face and shields is more than the reality. As a result, the load on the longwall shields is over estimated by this method. Moreover, the numerical modelling does not consider directly the corner crushing of the rock blocks, Fig. 3c, which leads also to overestimate the first caving distance and loading of the longwall shields.

6 Conclusion

A numerical approach for predicting the roof cave-in mechanism and the load on longwall shields was presented through a case study, the Svea Nord coal mine in Svalbard. It is a discontinuous approach taking into account all the bedding planes and cross-cut joints in the roof strata. In the case study, the spacing of the cross-cut joints in the roof strata and their mechanical properties were obtained through back-calculations of the first cave-in distance.

The modelling results of the longwall shield load agree with the field measurements, which indicate that the discontinuous numerical approach is reliable enough for engineering applications.

The simulations for the study site show that voussoir beams are formed in the roof strata above the mined out void before the first cave-in occurs. This beam-forming process causes high horizontal stress concentration in the roof strata owing to bending. Beam buckling is associated with the horizontal in situ stress in the strata. The sustainable deflection of a voussoir beam is about 25 % when the horizontal in situ stress is low, while in high horizontal in situ stress situations, both the sustainable deflection and the buckling strength of the beams are increased significantly.

Sensitivity studies reveal that the rock block size, in situ horizontal stress and mechanical properties of the roof strata have significant influences on the first and periodic cave-in distances and the load on the longwall shields.

Presented numerical approach is applicable for the validation of the traditional empirical analysis and to carry out sensitivity analysis in order to obtain the critical geomechanical issue influencing roof cavability and loading of the longwall shields. Validity of the numerical results is attributed to the reliability of the input parameters. In order to decrease the uncertainty in the numerical results, an acceptable range of input data could be obtained and introduced to the numerical modelling by the presented DOE method. The outcome of such numerical simulations could provide an acceptable range for the required support capacity of the longwall shields.

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References

- Bandis SC, Lumsden AC, Barton NR (1983) Fundamental of rock joint deformation. *Int J Rock Mech Min Sci Geomech Abstr* 6:249–268
- Barczak TM (1992) Examination of design and operation practices for longwall shields. United States Department of The Interior, Bureau of Mines Information Circular, IC 9320. p 14
- Barczak TM, Gearhart DF (1992) Canopy and base load distribution on a longwall shield. United States Department of The Interior, Bureau of Mines Information Circular, RI 9418. p 23
- Barczak TM, Schawemmer DE (1988) Stiffness characteristics of longwall shields. United State Department of The Interior, Bureau of Mines, Report of Investigation, RI9154. p 14
- Barton NR, Choubey V (1977) The shear strength of rock joints in theory and practice. *Rock Mech Rock Eng* 10:1–54
- Barton NR, Lien R, Lunde J (1974) Engineering classification of rock masses for the design of tunnel support. *Rock Mech* 6(4):189–239
- Das SK (2000) Observation and classification of roof strata behaviour over longwall coal mining panels in India. *Int J Rock Mech Min Sci* 37:585–597
- Diederichs MS, Kaiser PK (1999) Stability of large excavation in laminated hard rock masses: the voussoir analogue revisited. *Int J Rock Mech Min Sci* 36:97–117
- Ghose AK, Dutta D (1987) A rock mass classification model for caveing roofs. *Int J Min Geol Eng* 5:257–271
- Narr W, Suppe J (1991) Joint spacing in sedimentary rocks. *J Struct Geol* 11(9):1037–1048
- NIST/SEMANTECH (2012) E-handbook of statistical methods. www.itl.nist.gov/div898/handbook, 15.01.2012
- Nomikos PP, Sofianos AI, Tsoutrelis CA (2002) Structural response of vertically multi-jointed roof rock beams. *Int J Rock Mech Min Sci* 39:79–94
- Obert L, Duvall WI (1967) *Rock mechanics and the design of structures in rock*. Wiley, New York
- Peng SS, Chiang HS (1984) *Longwall mining*. Wiley, New York
- Price NJ (1966) *Fault and joint development in brittle and semi-brittle rock*. Pergamon Press Ltd, London
- Rocscience. *Examine 2D: 2D Stress Analysis for Underground Excavations* version 7.0. Downloaded from: <http://www.rocscience.com/products/11/Examine2D>
- Shabanimashcool M, Li CC (2012) Numerical modeling of longwall mining and stability analysis of the gates in a coal mine. *Int J Rock Mech Min Sci* 51:24–34
- Sheorey PR (1984) Use of rock classification to estimate roof caving span in oblong workings. *Int J Min Eng* 2:133–140
- Singh GSP, Singh UK (2009) A numerical modeling for assessment of progressive caving of strata and performance of hydraulic power support in longwall workings. *Com Geot* 36:1142–1156
- Sofianos AI (1996) Analysis and design of an underground hard rock voussoir beam roof. *Int J Rock Mech Min Sci Geomech* 33(2):153–166
- Trueman R, Lyman G, Cocker A (2009) Longwall roof control through a fundamental understanding of shield-strata interaction. *Int J Rock Mech Min Sci* 46:371–380
- Tsesarsky M, Hatzor Y (2006) Tunnel roof deformation in blocky rock masses as a function of joint spacing and friction—a parametric study using discontinuous deformation analysis (DDA). *Tunn Undergr Space Technol* 21:29–45
- Vakili A, Hebblewhite BK (2010) A new cavability assessment criterion for longwall top coal caving. *Int J Rock Mech Min Sci* 47:1317–1329