Geomechanical issues in longwall mining

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1 Introduction

Coal mining today is one of the most important energy and fuel supplies in the world. In 2013, worldwide coal production exceeded 7823 Mt (World Coal Association, 2014). Mining methods can generally be subdivided into surface (open-pit) and underground mining. Today, around 40 % of coal is mined open-pit and around 60 % underground (Junker et al., 2006).

2 Mining technologies

Underground mining is conducted using a number of different methods. Longwall Mining and Room-and-Pillar-Mining are the most frequently used ones. Longwall Mining is the most popular and also most productive method. However, the selection of the most suitable extraction technology always depends on factors like geology, mining depth or quality of the seams. As an introduction to this chapter a short overview is given about these two most popular mining methods.

2.1 Room and pillar mining

Room and pillar mining is one of the oldest mining methods. As the name suggests, safety pillars (coal blocks) are left in the panels for support, whereas rooms in between are the empty areas which have already been mined. The maximum depth is limited by the use of room-and-pillar mining, because with greater depths larger pillars are needed, which would decrease the recovery rate of the coal. The spacing intervals and dimensions of the pillars are generally depended of the mine depth and geotechnical parameters. The pillar width can be up to 25 m; normally it is around 6m. The height normally equals the seam thickness. The pillars do not only support the roof above but also the adjacent entries and crosscuts.

For room-and-pillar mining, continuous mining is the most common method. A continuous mining machine excavates the coal while at the same time the coal is loaded onto conveyors or transport shuttle cars. Despite being called “continuous”, after some advancement roof bolting is required to prevent the overlying strata from collapsing. The mining process in room-and-pillar mining is shown in Fig. 1.

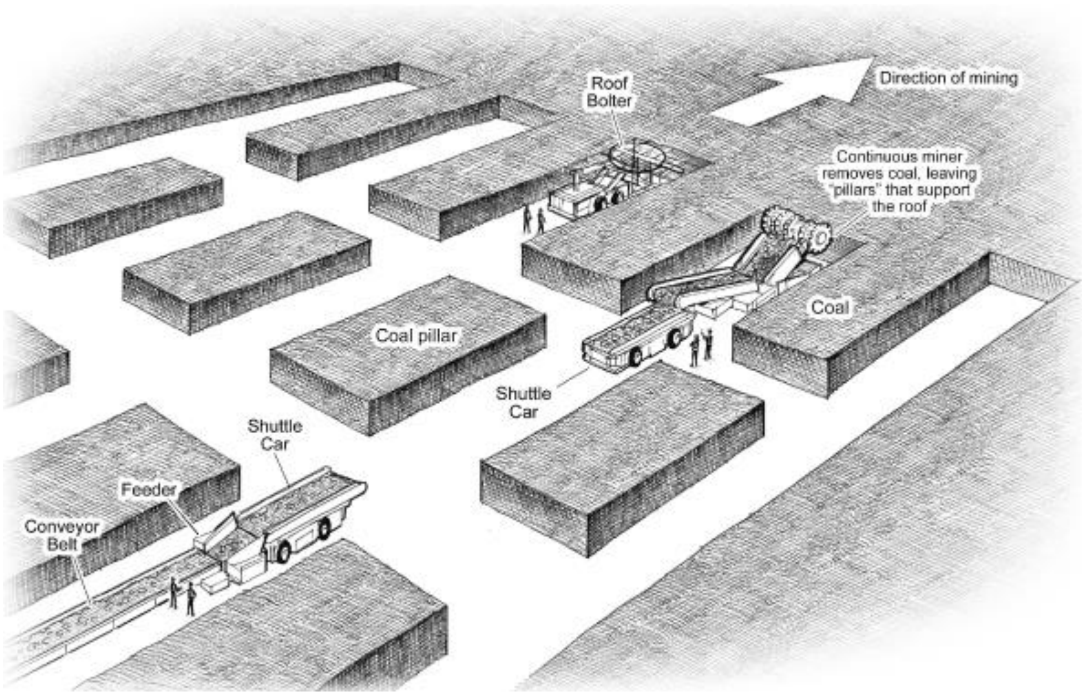

Fig. 1: Room and pillar mining (Arch Coal Inc., 2010).

Mining is conducted cyclical and step-by-step. The basic principle consists of mined-out rooms and pillars left for stability. Mining is usually done simultaneously in different rooms. After blasting the coal in the rooms is transported by shuttle cars to the conveyors. Bolting of the roof is important to prevent roof falls. Often in a final stage additional coal is mined through removal of some of the pillars, called retreat mining. Under certain conditions this can cause major subsidence damage at the surface. Room-and-pillar mining is not only used for coal but also for other resources, such as salt and potash deposits.

2.2 Longwall Mining

Longwall mining is used for flat-lying and tabular coal deposits. It enables mining of nearly the complete resource. The principle of longwall mining is to allow mined-out sections to collapse (Fig. 2). Gate roads are excavated around a panel with pillars left untouched to support the overlying strata there. These roads are connected at the ends by so called bleeders. The entry of the panel is called head gate, the exit respectively tailgate. The gate roads are secured using roof bolting. Panel widths can be up to 260 m, with panel lengths up to 2000m and panel heights up to 2.5 m (Wenzel, 2012). Vertical shafts provide transportation of air, miners and supplies.

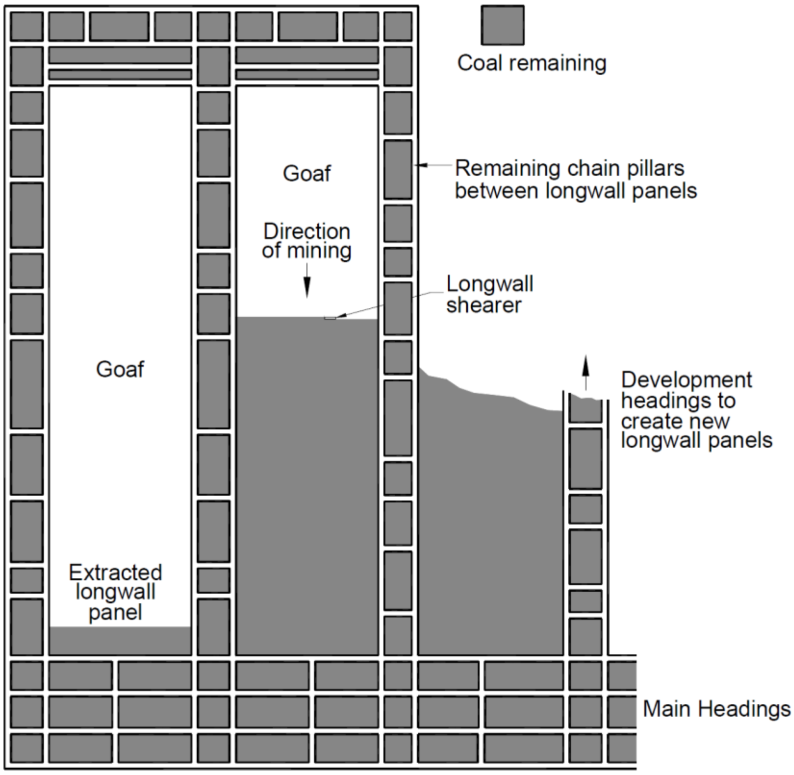

Fig. 2: Typical plan view of longwall panels (MSEC, 2007).

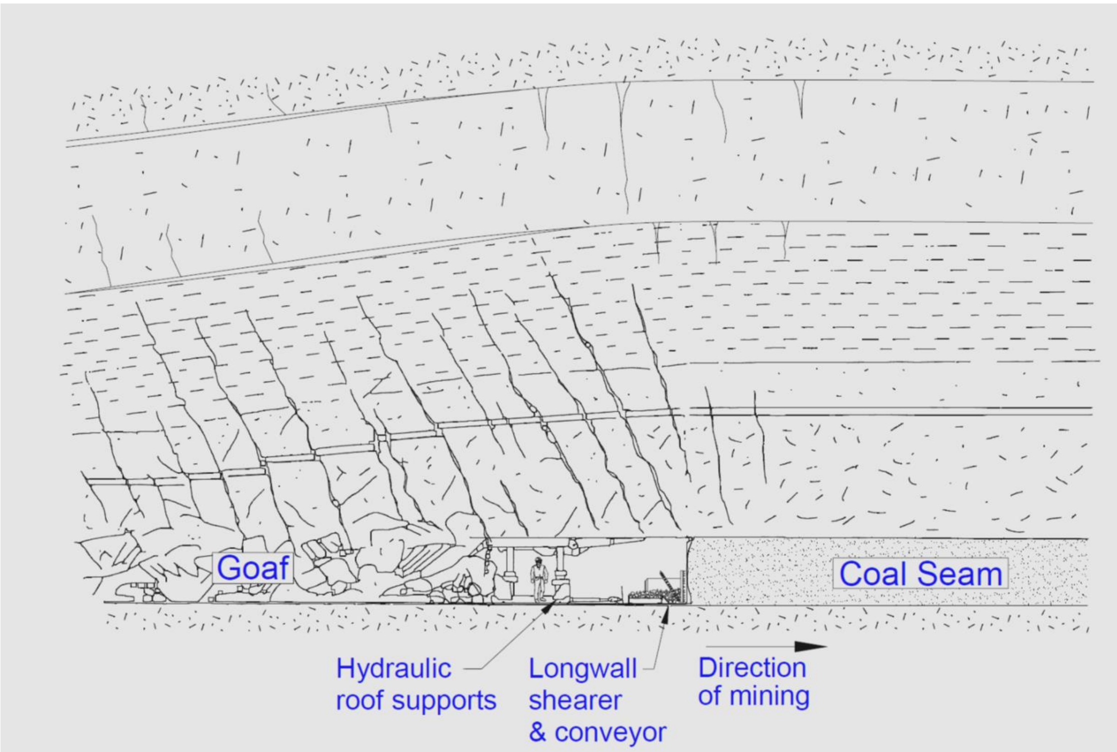

Fig. 3: Typical cross section of longwall face (MSEC, 2007).

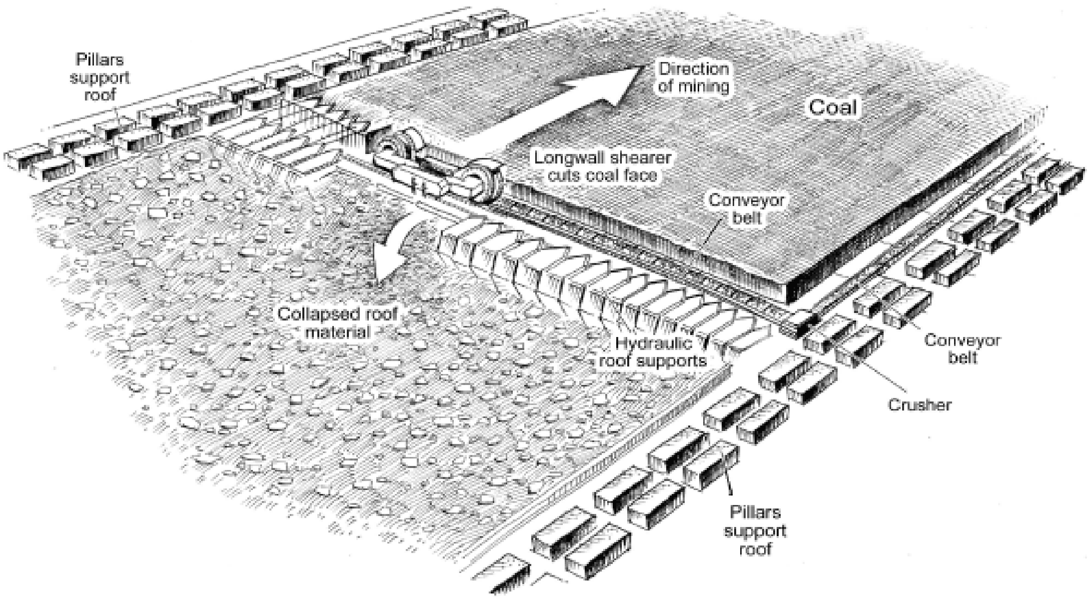

Fig. 4: Longwall mining (Arch Coal Inc., 2010).

During the extraction process material is being mined out by longwall shearers. The width of the cut is about 0.76-1.07 m (Peng, 2006). This is normally a continuous process. A conveyor belt transports the mined material to the side passageways.
When the shearer reaches the end of the panel, the excavation direction is reversed and the shearer excavates the next panel. While the shearer is moving the roof support advance closer to the new face, supporting the roof above and preventing collapse. The distance between the face and the canopy tip of the shield is called unsupported distance. For recovery of the equipment, when the face has advanced close to the mains, a recovery room is set up.
The roof behind the active face is allowed to collapse. As a side-effect of this, eventually the strata movement causes surface subsidence. Normally this happens systematically and does not cause serious problems, however below sensible surface constructions more attention has to be paid. It can also affect the groundwater flow by bedding separations or strata breakage. Additional problems can arise in case of large surface subsidence (in some areas subsidence of 10 m and higher are observed), if water level rises after mine flooding.

3 Powered supports

Both, in Room-and-Pillar and in Longwall Mining the use of support elements is necessary to prevent the collapse of the roof. Various types and methods of roof support exist, such as bolting, cribs, hydraulic props or powered roof supports, each of them with advantages and disadvantages. Below bolting and powered supports are described exemplary.

Roof bolting is used for mine entries and crosscuts. Bolts vary in bolt type, length and capacity. The bolt spacing is dependent on the rock type (strength, joint pattern etc.) and geomechanical conditions (stress state, water pressure, width of opening, expected lifetime etc.). Narrow spacing is used for weaker roofs. Shorter bolts (< 3 m) are considered as primary support because of their quicker installation in cycle with coal cutting (Peng, 2006), while longer bolts (> 3 m) are installed after coal cutting, also called supplementary support.

Powered support shields are mainly used in longwall mining. Because of their compact dimensions and high support pressure they are the most common support type. They consist of four major components – canopy, caving shield, legs and base plate (Fig. 5). Without caving shield the support is classified as *frame* or *chock* type. The support capacity is proportional to the number of hydraulic legs. Today most shields consist of two or four legs. Shield types are divided into frame, shield, lemniscate and chock shields. Today lemniscate shields are the most common used ones.

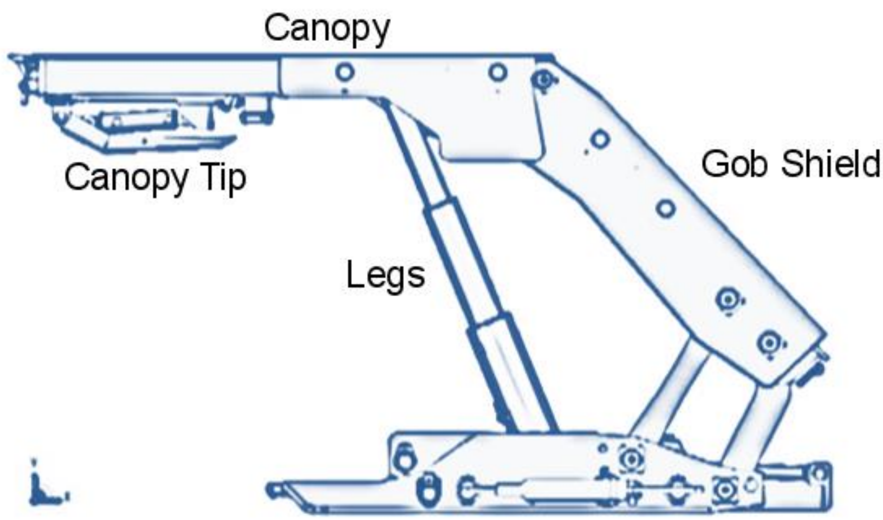

Fig. 5: Basic parts of a powered roof support. The image shows a typical lemniscate shield.

The shield canopy has direct contact with the roof and is usually made of steel plates. It has two tasks: convey load to the roof, and resist loading from the roof. Canopies can consist of one single piece, several parts or an extensible construction. The base transfers the roof loading to the floor, therefore, similar to the canopy it must be strong and stiff. A gob shield protects the support from being damaged by the collapsed backfill, the so-called goaf.

The hydraulic legs have to maintain pressure over long time to resist the roof loading. When setting the support the legs are slowly filled with the pressurized fluid so that the legs rise until the canopy reaches the roof. When the working pressure of the pump is reached the fluid gets locked in the legs. This hydraulic pressure is called setting pressure of the support.

4 Rock mechanics issues

Ground control in mining has to deal with several key problems: subsidence of the overlying strata, pressure of overlying strata in gate roads and shafts, and stress changes and load pressure in panels. At the face short-term roof control is essential, while at gate roads long-term stability has to be considered.

4.1 Underground rock deformation

Underground rock deformation includes plastic, elastic and disjunctive deformation. Along gate roads disjunctive deformation (cave-in of the roof) is most common. The process of roof cave-in includes fracturing of the rock, disintegration and (downward) movement. Above the roof three zones of disturbances in the overburden strata can be distinguished (Peng & Chiang 1984): a caving zone, a fractured zone and a continuous deformation zone (Fig. 6). They differ mainly in the induced degree of damage.

The caving zone is the region where the immediately overlying strata fall into the panel. This zone has a thickness of two to eight times the mining height. Fracturing and large deformation dominates in this zone. Strata are broken into smaller irregular and platy blocks of various sizes. The volume of the caved roof is normally larger than that of the intact roof. The corresponding ratio is called bulking factor *K*:

$$K = \frac{V_E}{V_0} > 1$$

Where V_E is the Volume of the caved roof and V_0 is the Volume of the former intact roof. With a high bulking factor the caved rock can support the roof and distribute the subsidence more evenly. In the fractured zone rocks are intensively fractured and lose their cohesion. Blocks are separated by vertical fractures and horizontal cracks caused by bed separation, which leads to the formation of cavities between the layers.

In the continuous deformation zone no fractures occur, but plastic deformation takes place. Also, in this zone cavities between different layers can develop. This zone reaches until the surface and can be considered as one continuous medium. Peng et al. (2006) also included a thin soil zone immediately below the surface, which is characterized by unconsolidated rock.

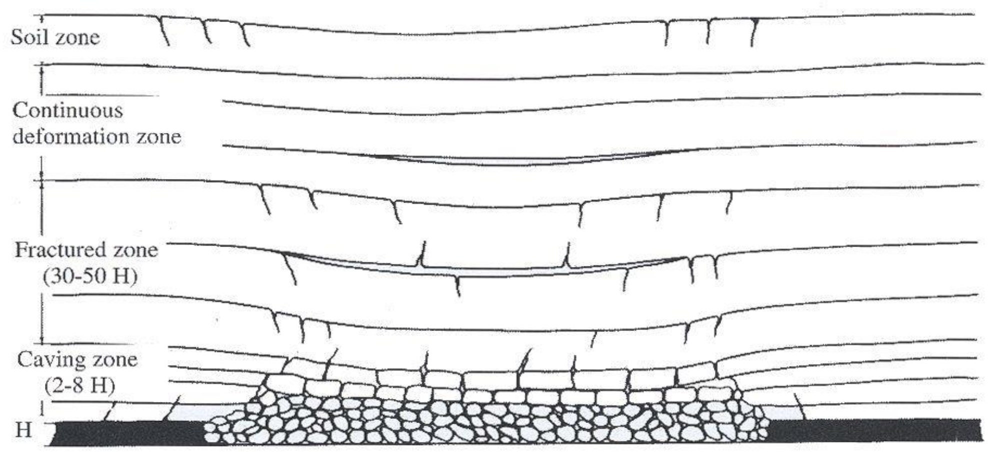

Fig. 6: Classification of overlying strata (Peng & Chiang, 1984).

The caving zone also called immediate roof has the biggest effect on roof control at the longwall face. The roof cannot transmit horizontal force along the mining direction, because it is broken, therefore the powered roof support must completely support its weight (Peng & Chiang, 1984).

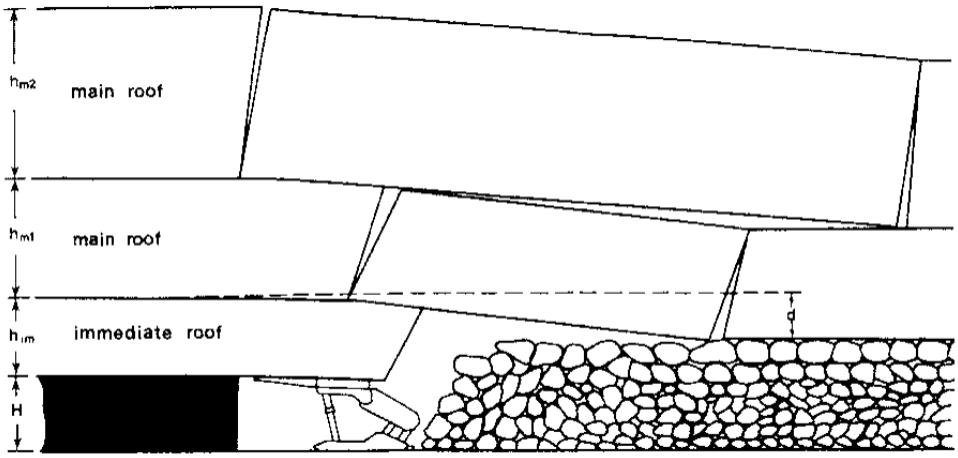

Fig. 7: Calculation of the immediate roof height (Peng & Chiang, 1984).

The thickness of the immediate roof depends on various factors, such as mining height, mining method and rock properties. A rough estimation of the height h_{im} of the immediate roof (Fig. 7) can be made using the following formula:

$$h_{\text{im}} = \frac{H-d}{K-1}$$

with h_{im}, the height of the immediate roof, H the Mining height and d the settlement of the lower main roof. The settlement coefficient c is calculated using the following equation:

$$c = \frac{d}{H} = \frac{h_{\text{im}} + H - h_E}{H}$$

Therein h_E is the height of the current layer. The main roof refers to the lower part of the fractured zone with the slightly broken but uncaved strata. Jacobi (1976) showed that roof cave-in occurs in a periodic cycle. In the immediate and the main roof two phases of overburden movement can be distinguished, the first roof weighting interval and the periodic roof weighting interval.

The first phase begins with the immediate roof collapsing within a large area, followed by the breakage and caving of the upper main roof. The roof layers are supported by both, the stope and the powered supports. The maximum roof pressure during this time is called first weighting. The weighting interval L_0 is the distance from the setup entry to the first weighting defined as follows (Fig. 8):

$$L_0 = \sqrt{\frac{2h_m \cdot \sigma_{t,m}}{\gamma_m}}$$

Where $\sigma_{t,m}$ is the tensile strength and γ_m is the unit weight.

In the second phase the roof layers are only supported by the powered supports. Breakage of the roof occurs periodically, which leads to increasing and decreasing of the pressure. This is called periodic roof weighting.

The periodic roof weighting interval (Fig. 8) is calculated as a Periodic weighting resistance P_{tm} :

$$P_{tm} = c_1 + c_2 \cdot L_p$$

and the corresponding load difference Δp:

$$\Delta p = \frac{p \cdot \left(c_3 \cdot L_p - c_4\right)}{L_p}$$

with the load *p,* different settlement rates $c_1 \ldots c_4$ and the periodic weighting interval L_p.

For supporting the roof with shields, it is important to know the weight of the roof. Peng (2006) developed a simple method for calculating the support load. Length *L* and width of the roof block h_{im} are calculated by:

$$L = L_W + L_U + L_C + L_O$$

and

$$h_{\text{im}} = \frac{H - d}{K - 1}$$

where:

- K – Bulking factor of caved rocks
- d – Sagging of the lowest uncaved stratum
- L_W – Width of cut
- L_U – Unsupported Distance
- L_C – Shield canopy length
- L_0 – Rear overhang.

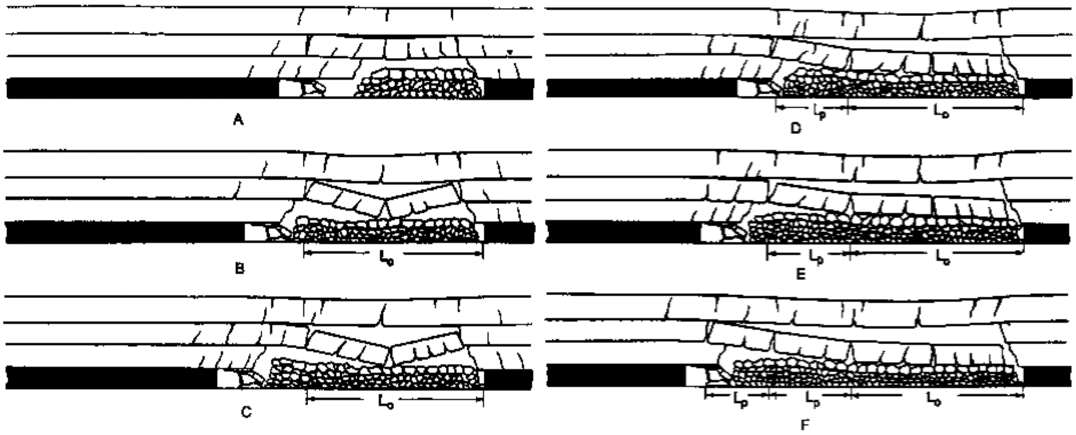

Fig. 8: Length of roof caving (Peng, 2006). The length L_0 is the first weighting interval, L_1 describes the periodic roof weighting.

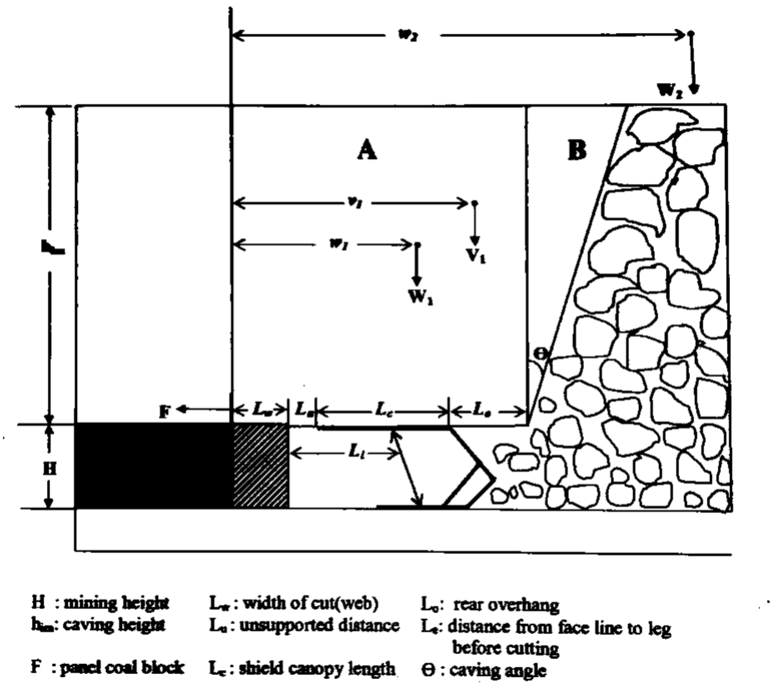

Fig. 9: Dimensions of the mining blocks. (Peng, 2006)

For solid and weak roof material there are different formulas for calculation of the loading:

<u>Solid roof:</u>

$$W_1 = h_{\mathrm{im}} \cdot \gamma \cdot b_S \cdot \left(L + 0{,}5 \cdot h_{\mathrm{im}} \cdot b_S \cdot \tan\theta\right)$$
$$w_1 = 0{,}5 \cdot \left(l_{\mathrm{ges}} + h_{\mathrm{im}} \cdot \tan\theta\right) > L_{\mathrm{Stempel}}$$

Where:

- W_1 – vertical weight
- w_1 – width of the roof

- b_s – support spacing
- θ – angle of fracture surface
- l_{ges} – Length of the whole cut
- $L_{Stempel}$ – length of the canopy

Weak roof

$$V = W_1 + W_2$$

$$\nu \cdot V = w_1 \cdot W_1 + w_2 \cdot W_2$$

4.2 Roof falls

Rock failure happens in general if applied stress exceeds the strength of the rock mass. However, the detailed reasons for cave-in of roof strata can be manifold. Many roof falls have been attributed to high horizontal stresses (Peng, 2008). Material parameters of the rocks, especially tensile strength, have great influence. Roof falls tend be occur more in weak roofs than in strong roofs like for example massive sandstone. Also, discontinuities in the rock mass can reduce stability. Roof stability is depended on the thickness of the layers, with thicker roofs being generally worse (Peng, 2008). Clayey roof exposed to ventilation air and therefore to wet and dry cycles is highly sensitive to weathering, and could crumble fast. The so-called *stack rock* made from thin layers of sandstone/sandy shale interbedded with carbonaceous material has been responsible for many roof falls, as the thin carbonaceous films decompose easily. Barczak (1992) distinguished four categories of roof fall mechanisms according to Fig. 10.

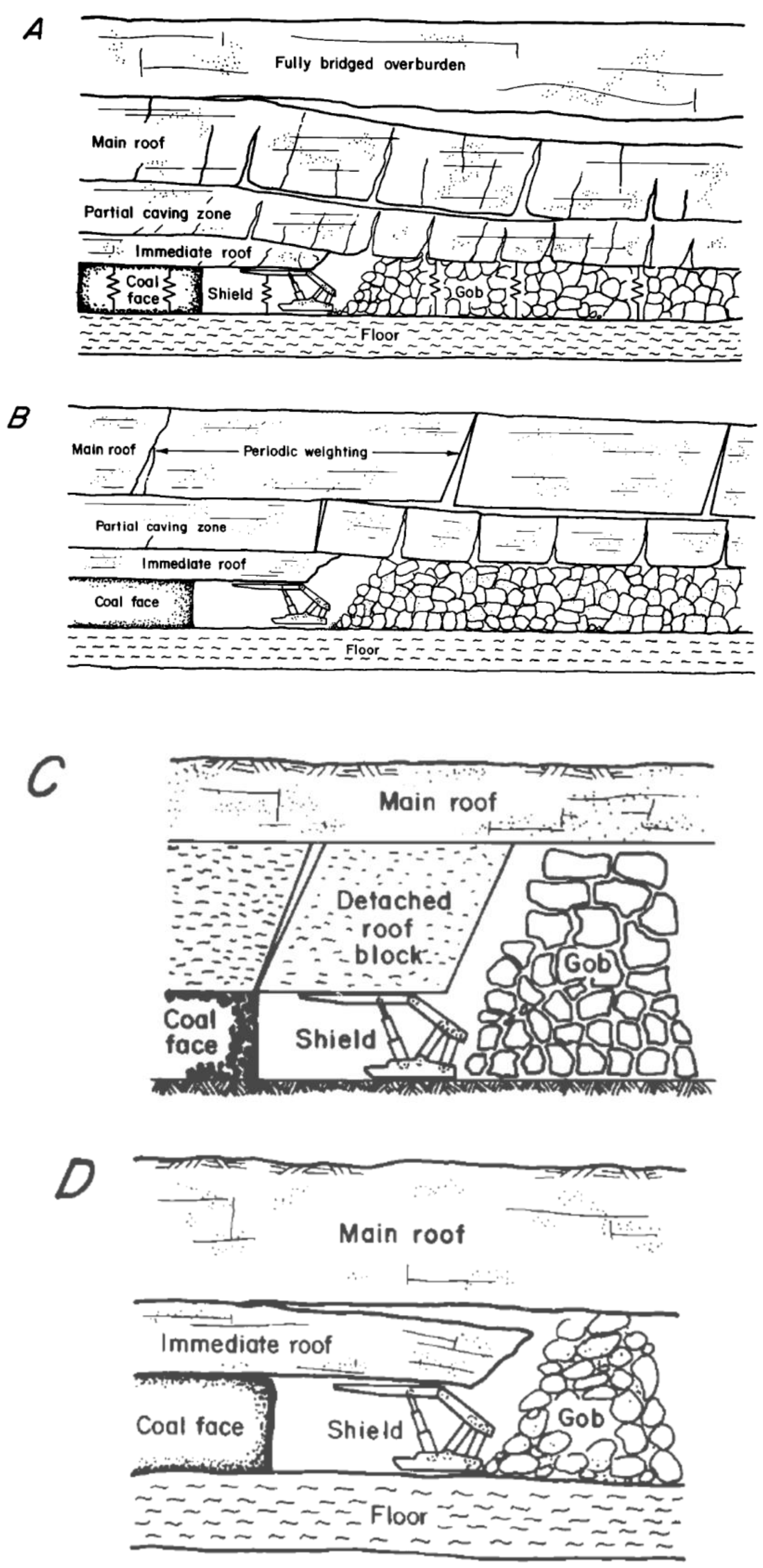

Fig. 10: Different types of roof fall. A – Main roof convergence. B – Periodic weighting. C – Detached immediate roof. D – Deflection of immediate roof. (Barczak, 1992)

In geomechanics it is a common approach to use physical models to investigate the stability of underground openings (e.g. Jacobi 1976; Ju & Xu 2015). Fig. 11 shows two of such models.

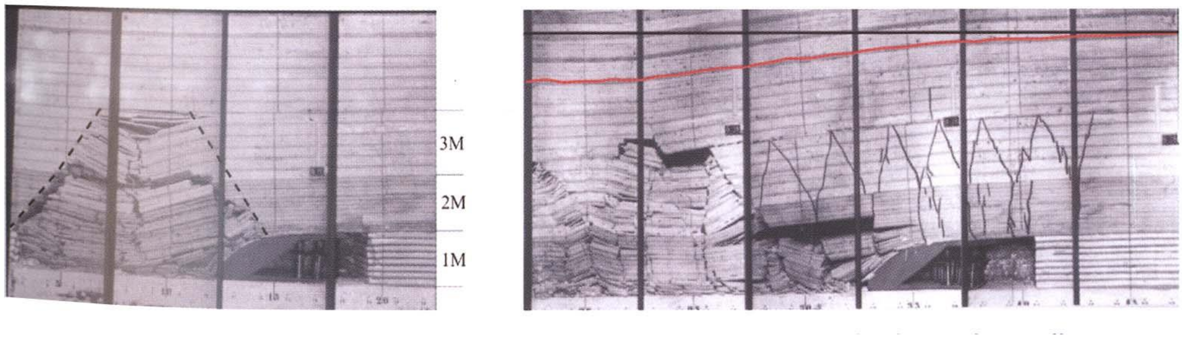

Fig. 11: Small-scale models to investigate roof cave-in. (Jacobi, 1976)

Layers were coloured differently to distinguish different strata. The left image shows typical deformation and fracture pattern shortly after start of excavation. In the right image periodic fracture patterns can be observed (fully developed longwall mining). Based on such models, parameters like angle of breaking, periodic weighting length and height of the immediate roof can be obtained. Support elements can also be included into the model to investigate their performance and influence. However, one has to bear in mind that model setup is quite time-consuming and careful consideration of scaling material parameters is necessary. An alternative approach is the use of numerical modelling techniques (e.g. FDM, FEM or DEM). Exemplarily, Fig. 10 shows two stages of a numerical simulation of a coal caving process. The rock mass is represented by deformable Voronoi blocks. Fractures can propagate along the edges of the blocks and can lead to complete disintegration. Such a procedure is suited to simulate the fracture and damage processes including stress redistribution process, especially in the roof layers.

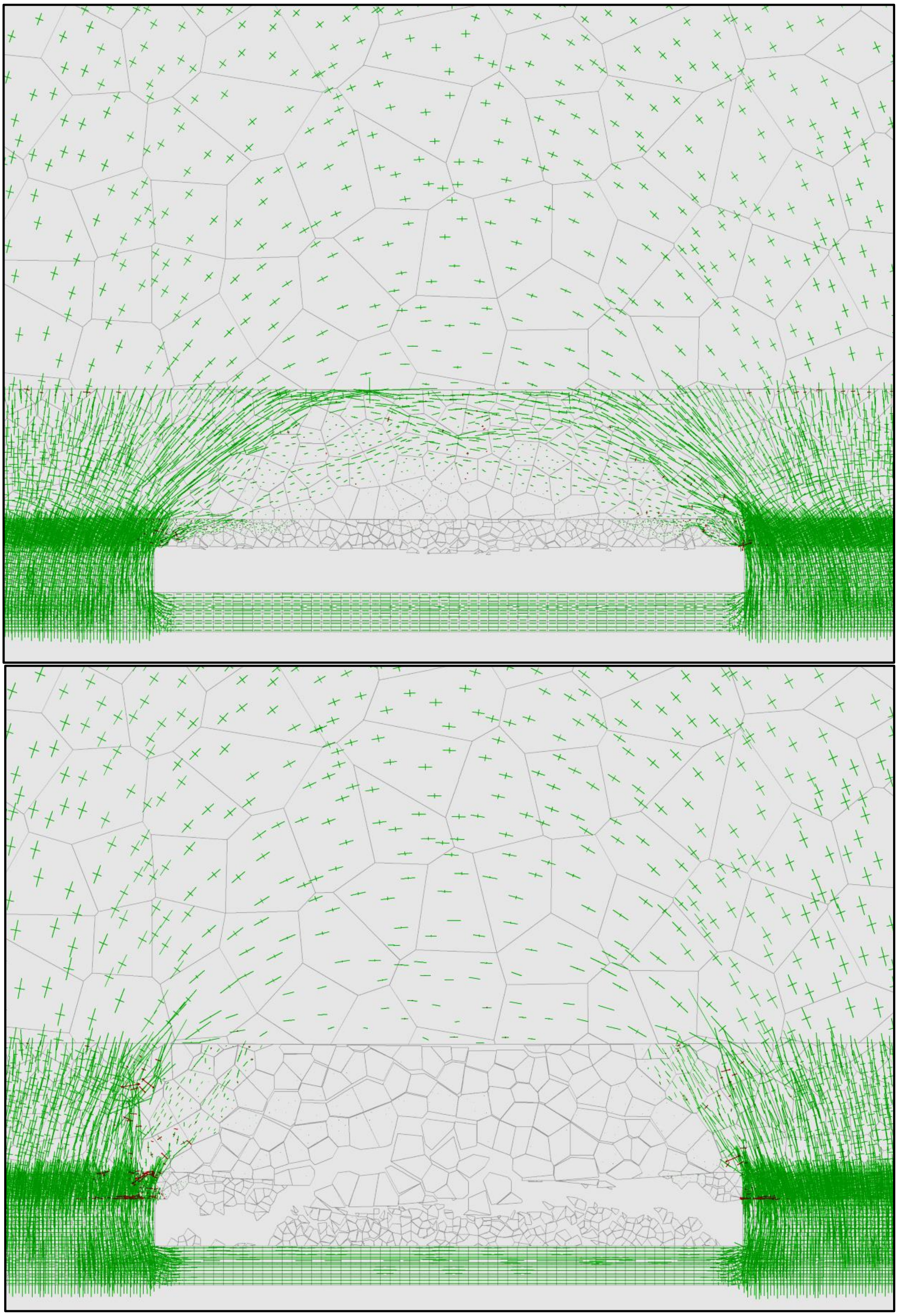

Fig. 12: DEM simulation of longwall coal mining with roof collapse in a stratified rock mass (block structure and principal stresses for two different stages).

4.3 Stress distribution

Underground mining dramatically changes the in-situ stress conditions in the rock mass. Layout of the panels and mining sequence affect the redistribution of rock stresses. Also, cave-in of the roof strata results in stress changes. These stress re-distributions have pronounced impact on stability of gates, pillars and drifts.
Generally, vertical stress concentrations increase with longer panels, also changing the horizontal stresses.

Fig. 13 shows the stress distribution along a longwall panel. Approaching the face (from left to right) the stress increases very quickly. A few meters ahead of the face an abutment peak is reached. Both at the face and at the rib the vertical stress is zero. With increasing distance from the face the stress level returns to cover load.

More realistic stress distribution is shown in Fig. 14 together with specification of different stress types. In the unaffected zones denoted `virgin coal´ and behind the goaf the vertical stress approaches the virgin state. Behind the face the stress reaches its maximum, going to nearly zero directly at the face. Another stress drop appears at the begin of the goaf.

The position of the pressure maximum depends on the rock hardness (stiffness and strength), as shown in Fig. 15. For very hard rocks, the maximum of the pressure is located at 3 x M to 6 x M ahead of the face. For weak rocks location of the maximum can be up to 15 x M.

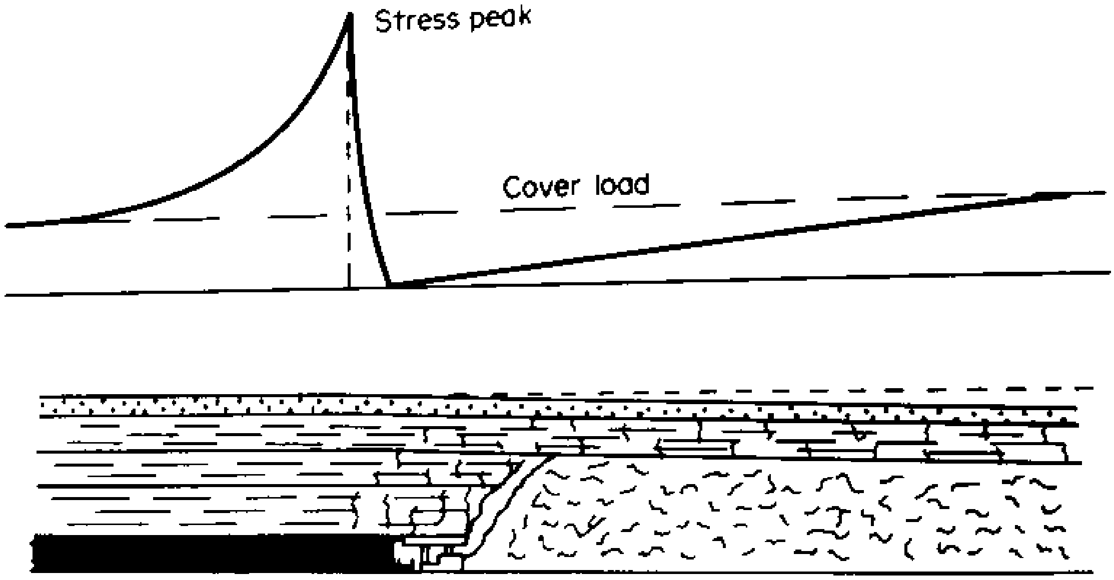

Fig. 13: Vertical stress distribution at a longwall panel from stope to goaf (Hudson, 1993).

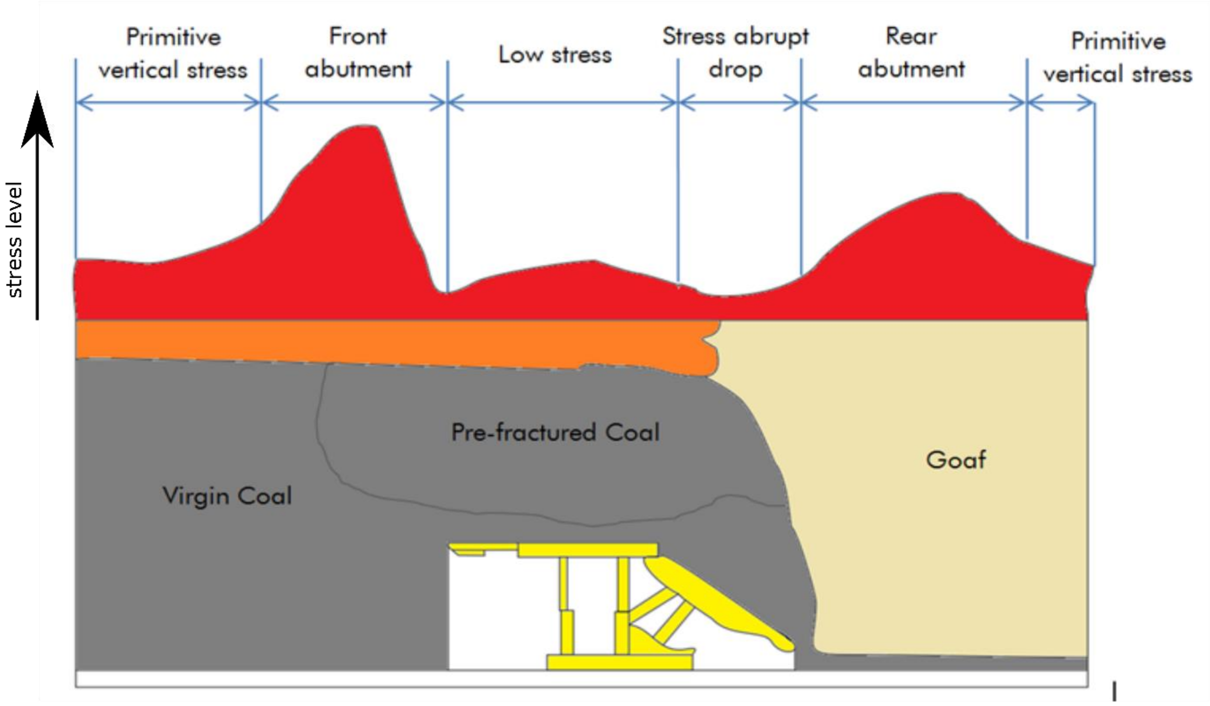

Fig. 14: Different types of stress in the roof rocks (Modified after Alehossein & Poulsen, 2010).

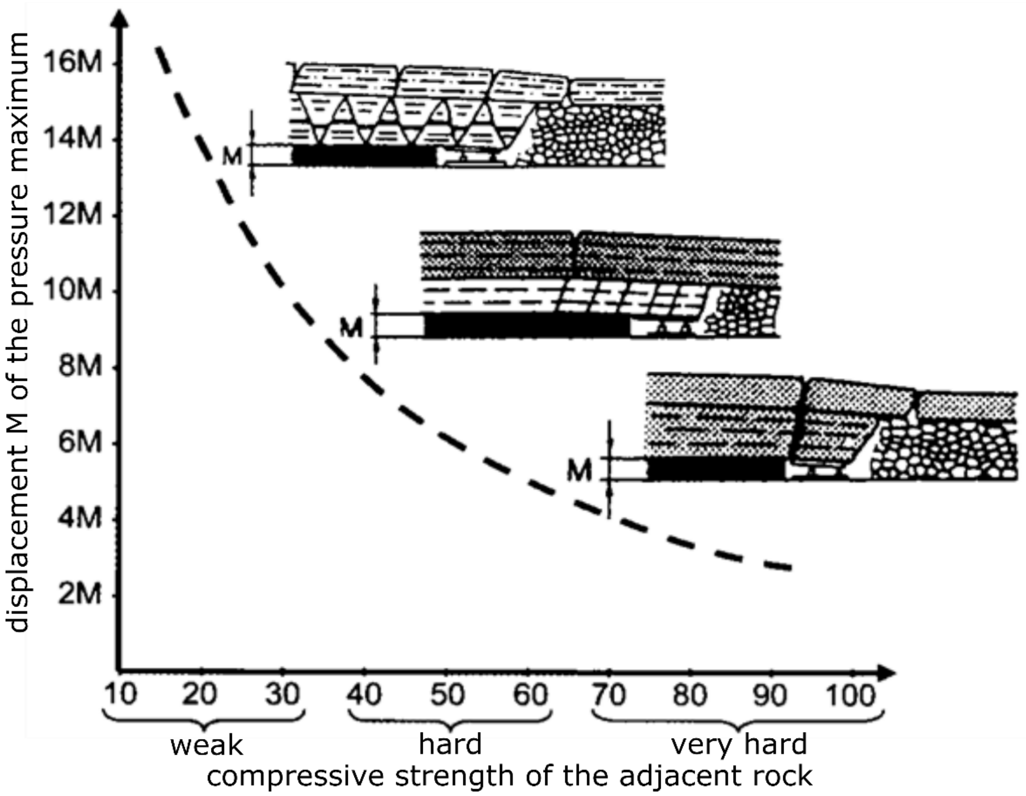

Fig. 15: Redistribution of the pressure maximum in relation to rock hardness. (After Juncker et al., 2006)

4.4 *Overburden and surface subsidence*

Due to the removal of large amount of coal, the whole overburden from the coal seam up to the surface can be affected by subsidence. In longwall mining, when the goaf collapses, rocks above lose support and sag into the void. This can change surface topography as well as surface water and groundwater flow. Surface disturbance can appear as subsidence trough, sinkholes or cracks. With a sufficient length of a panel excavated, the roof strata will begin to form a subsidence trough with development of compression and tension zones, separated by an inflection point. Fig. 16 – 18 illustrate this deformation process.

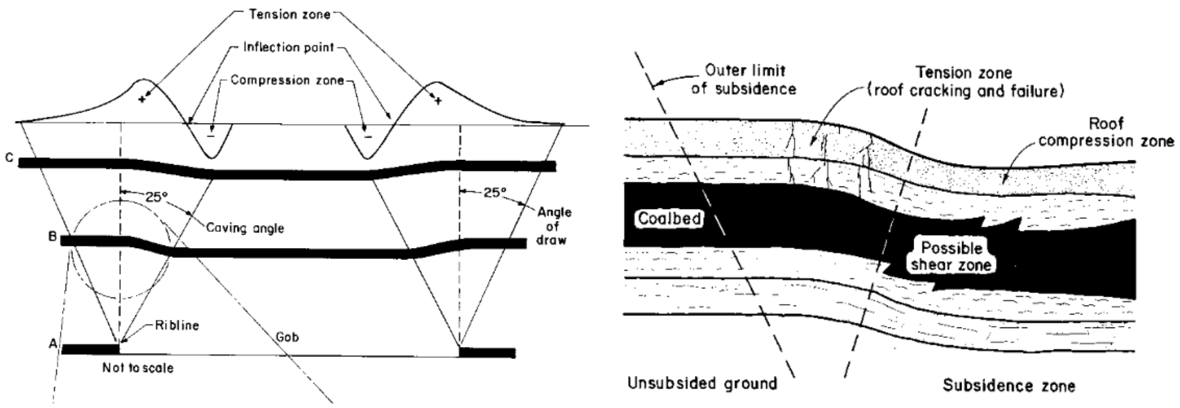

Fig. 16: Formation of a subsidence trough above a mined-out panel (Haycocks et al., 1982)

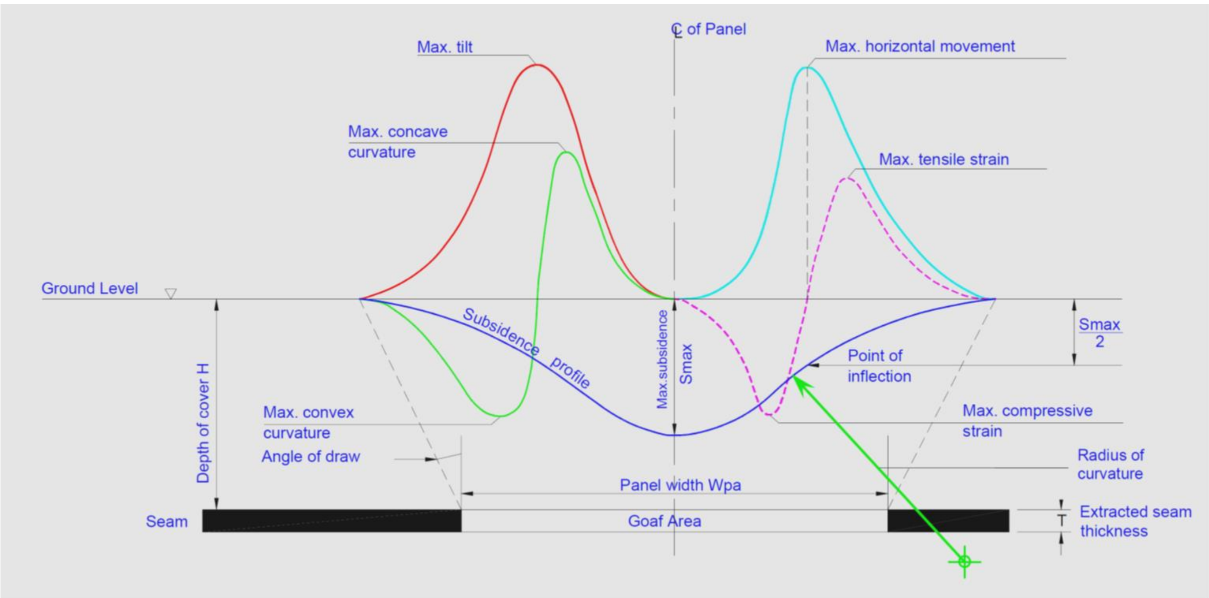

Fig. 17: Detailed geodetic characterization of subsidence trough above a mined-out panel (MSEC, 2007)

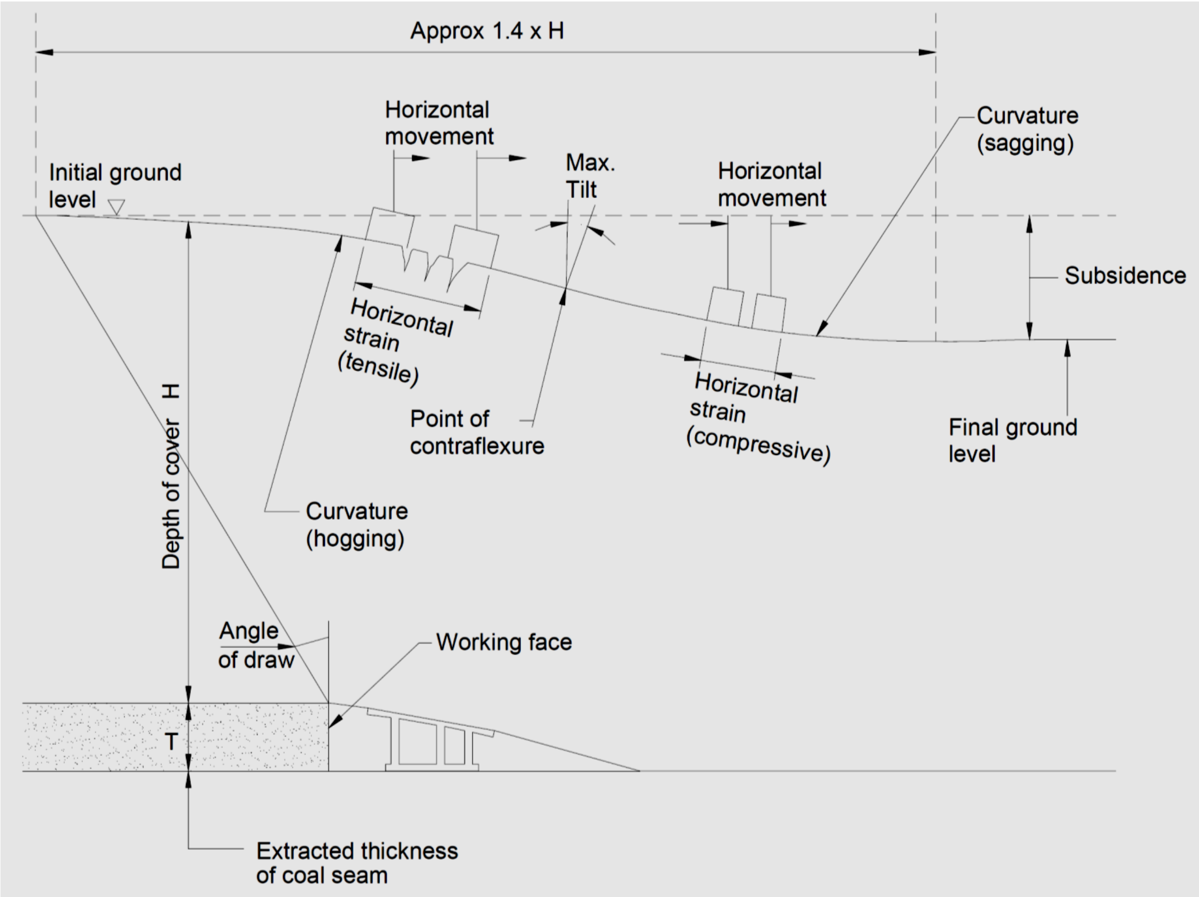

Fig. 18: Development of subsidence trough above longwall mining area (MSEC, 2007)

Looking closer to the strata immediately adjacent to the coal seam, one can recognize that the compression zone over the mined-out panel causes only minor problems related to ground control. This zone develops towards the center of the panel. However, inside the tension zone fractures and cracks develop easily because tensile strength of rock is much lower than compressive strength. The angle of draw corresponds to the angle at which the subsidence spreads out towards its limits at the surface. A subsidence of less than 2 cm at the surface is generally seen as insignificant. The subsidence at the surface is always smaller than the thickness of the mined coal due to volumetric increase of broken rock mass compared to intact rock (bulking factor).

Several methods for prediction of subsidence exist, such as the influence function method, profile function method and zone area method. The influence function method is became more widely used throughout the world, because it can be more easily applied to a larger range of situations (Luo, 1997). Nevertheless, classical geodetic models are more and more replaced by numerical simulations based on profound physical laws. Such model are able to simulate the whole deformation and fracture process including the interaction with support measures. Subsidence values over mined-out panels can reach several meters and can cause huge effects in terms of groundwater management after flooding.

4.5 Performance of powered supports

Powered supports temporarily hold up the roof strata. To evaluate the performance of powered supports, it is necessary to know the yield and setting load. The yield load for each support leg is:

$$p_y = \frac{\pi \cdot d^2 \cdot \sigma_y \cdot 175.1076}{4 \cdot 2000}$$

with

- p_y - yield load in tons
- d - inner diameter of the leg cylinder in meter
- σ_y - yield pressure in pascal.

The setting load P_S can be calculated as following (after Peng, 1984):

$$P_S = \frac{4.448 n \cdot \eta \cdot \pi d^2 \cdot \sigma_S}{4 \cdot 2000}$$

with

- n - number of legs
- η - support efficiency
- σ_S - setting or pump pressure in pascal

It is also possible to include shields into the numerical models to investigate the interaction between shield and rock mass and to determine the actual load on the shields including the corresponding load distribution. Important parameters are geometrical dimensions (e.g. canopy length, base length, max. height etc.) and geotechnical parameters (stress state, deformation and strength parameters, layer thickness etc.). Fig. 19 shows a specific longwall mining model with explicit consideration of hydraulic support. Fig. 20 and 21 illustrate the stress distribution on the canopy and the force transmission to the base.

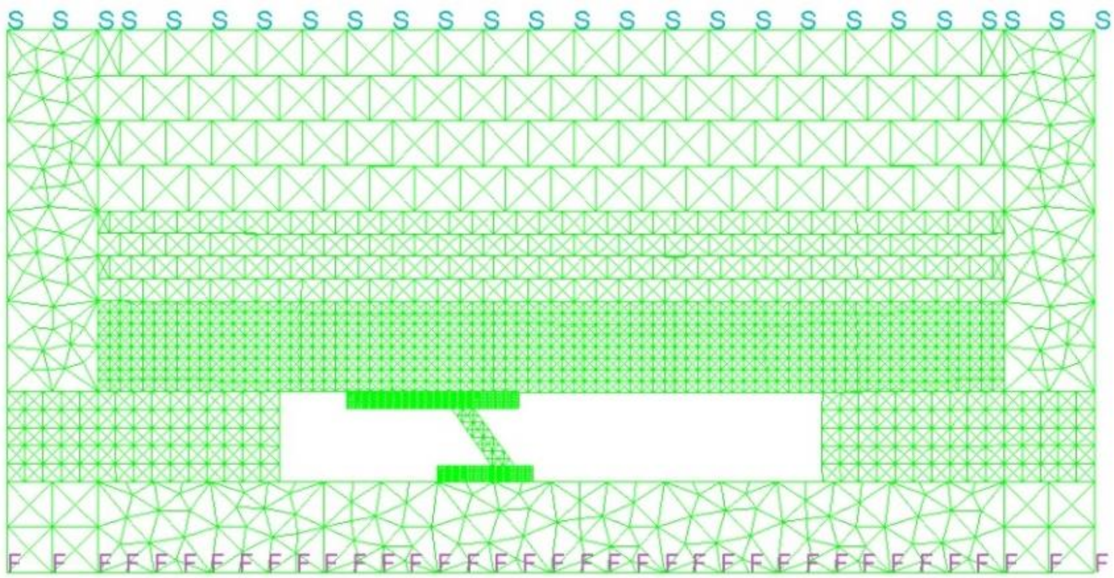

Fig. 19: UDEC shield model (Wenzel, 2012)

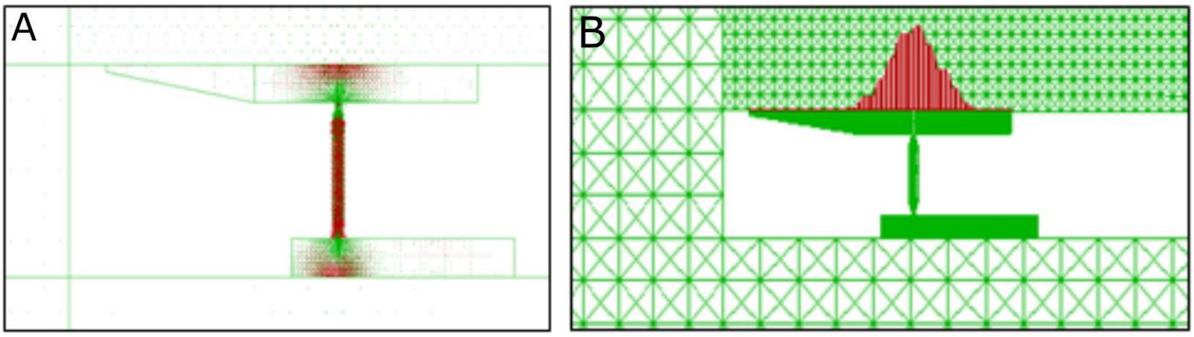

Fig. 20: A - Stress distribution in the shield. B - Stress distribution in the hanging rocks (Wenzel, 2012)

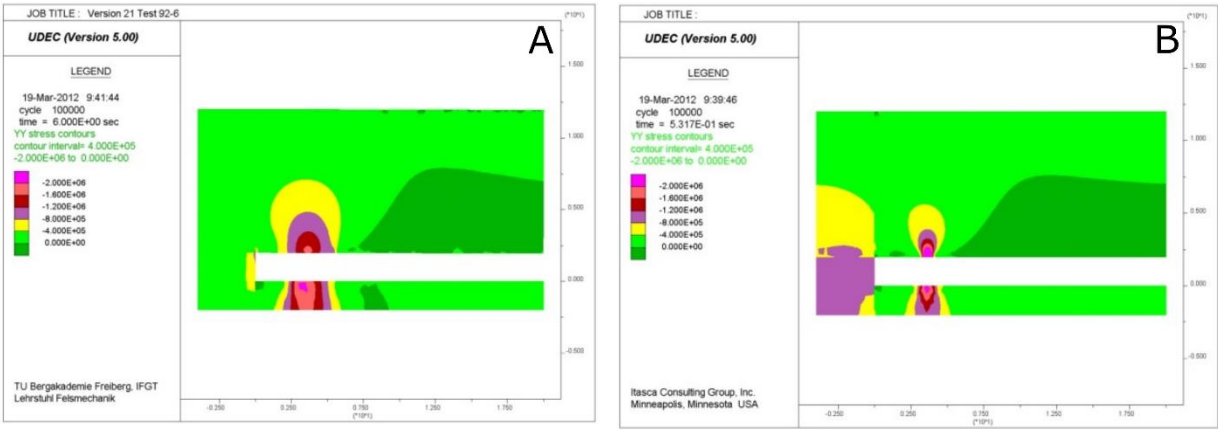

Fig. 21: A - Vertical stresses above weak rocks. B - Vertical stresses above solid rocks (Wenzel, 2012)

The interaction between support and rock mass can be quite different, like illustrated in Fig. 21: within soft rocks the support induced stresses cover a larger area, but they are lower in magnitude. In hard rocks, stresses are very localized with high peak loads.

5 Rock burst hazards

Underground coal mining is often faced with rock burst problems. If overlying strata are characterized by high strength and stiffness, large capacity to store strain energy and brittle failure behavior, several measures have to be taken to avoid high stress concentrations, which can be performed by intelligent overall mine design and mining sequence, water injections at the face, de-stress blasting (Saharan M.R., 2011; Konicek et al., 2011) or the mining of safety seams above or below the actual mined seam (Bräuner, 1992).

The effect of safety seams is illustrated in Fig. 22. Zone (a) is influenced by mining operations and induces a subsidence trough at the surface. Area (b) denotes a zone of reduced pressure. The excavation of a safety seam leads to large-scale stress re-distributions and stress reductions inside a certain area (c), so that over- or underlying seams can be mined under reduced stress level. Maximum subsidence is expected in area (d). Suitable safety seams should be not too far above or below the main seam and should be next to weaker rocks.

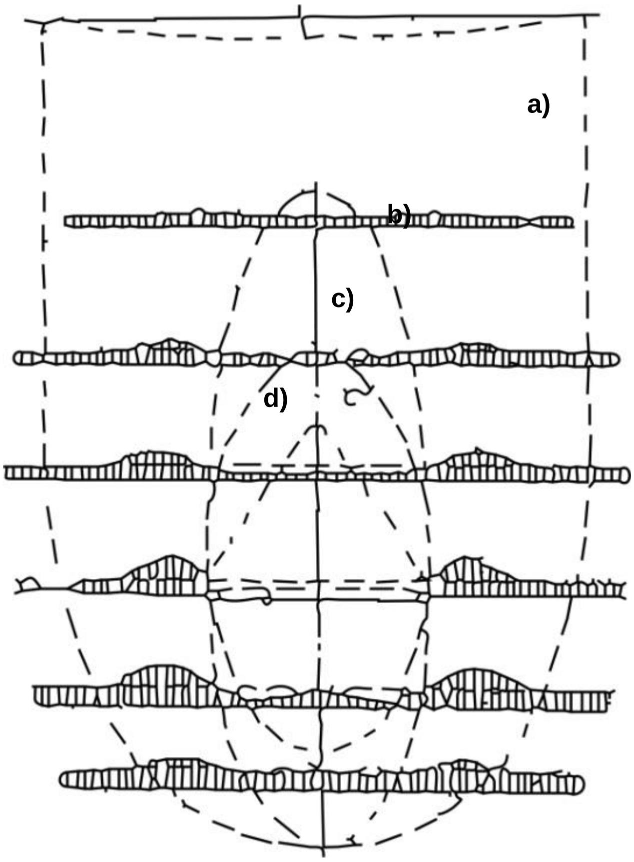

Fig. 22: Zone of influence during longwall mining (Bräuner, 1992).

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