9 Longwall Mining

Abstract

Since the early 1980s, the longwall method has developed into the safest, highest producing and most productive form of underground coal mining, rivalling the performance of many surface mining operations. This is due in large part to the rapid uptake of computer based technologies for automation and monitoring; improved reliability and performance of longwall mining equipment; and the adoption of plant management and loss control principles. This situation has many important implications for the geoscience and geotechnical engineering professions. For example, lost opportunity costs associated with loss of ground control are now so high that many of the simple observational and empirical approaches traditionally applied to geotechnical designs and operational aspects in longwall mining are no longer commensurate with the business risks that have to be managed. There is an increased need for geotechnical input to be based on sound engineering principles that encapsulate measured ground behaviour, applied mechanics, and numerical modelling. Ongoing research is important to support this need.

This chapter addresses geotechnical principles and practices relevant to satisfying these engineering requirements, making extensive use of figures and photographs to illustrate important concepts. It considers panel layout options and associated chain pillar design; traces the history of powered support design to draw learnings about their static and kinematic requirements; identifies and assesses operational variables, including cutting and support techniques, powered support maintenance, and face operational practices. It then reviews face behaviour and ground control requirements and practices; and evaluates the design and support of installation roadways and longwall recovery roadways, including pre-driven roadways.

Keywords

Abutment loading • Bidirectional cutting • Caliper shield • Canopy ratio • Chain pillar design • Chain pillar function • Chock shield • Conventional mode • Cutting technique • Cyclic loading • Double chock • Floor heave • Gateroad orientation • Ground control • Ground response curve • Half web

J.M. Galvin, *Ground Engineering - Principles and Practices for Underground Coal Mining*,
DOI 10.1007/978-3-319-25005-2_9

cutting • Horizontal stress • Lemniscate linkage • Longwall • Longwall installation roadway • Longwall recovery • Longwall top coal caving • Miniwall • One web back • Periodic weighting • Powered support • Powered support hydraulics • Pre-driven roadway • Sacrificial roadway • Shield • Skew roof • Strata control • Stress notch • Stress relief • Support resistance • Tailgate abutment loading • Tip-to-face • Unidirectional cutting • Yield pillar

9.1 Introduction

Since the early 1980s, longwall mining has developed into the safest, highest producing and most productive form of underground coal mining, rivalling the performance of many surface mining operations. In Australia, for example, advances in technology, geotechnical engineering and work practices over that period have resulted in more than a three-fold increase in the average productivity of longwall mining, with some newer operations achieving up to an eight-fold increase. Subject to adequate coal reserves, environmental constraints, and access to capital, it is the method of choice for new underground operations.

The significant increases in longwall productivity are due in large part to rapid uptake of computer based technologies for automation and monitoring; improved reliability and performance of longwall mining equipment; and the adoption of plant management and loss control principles, leading to both increased rates of production and reduced labour requirements. In terms of production, average daily output of the top performers in Australia has increased from, typically, 5,000 t/day in 1985, to 20,000 t/day in 2013. Incremental cost savings associated with these rates of production, supported by increased coal selling prices, meant that delayed profit opportunity for each day of unplanned stoppage of a longwall face increased from as little as A$10,000/day in 1985, to in excess of A$500,000/day for some premium coking coal operations in 2013, with non-recoverable fixed costs sometimes being of the order of A$150,000/day.

This has important implications for the geoscience and geotechnical engineering professions. Firstly, operations are now in a better position to justify the engagement of geotechnical professionals, with the annual salary of one such person being recouped if their input avoids just one or two days of lost production per annum. Secondly, the lost opportunity costs are now so high that many of the simple observational and empirical approaches traditionally applied to geotechnical designs and operational aspects in longwall mining are no longer commensurate with the business risks that have to be managed. There is an increased need for geotechnical input to be based on sound engineering principles that encapsulate measured ground behaviour, applied mechanics, and numerical modelling. Ongoing research is required to support this need. This chapter addresses geotechnical principles and practices relevant to satisfying these engineering requirements.

9.2 Panel Layout

9.2.1 Basic Longwall Mining Methods

There are two basic types of longwall mining, namely, 'longwall mining on the advance' and 'longwall mining on the retreat'. Longwall mining on the advance involves developing the maingate and tailgate entries just ahead of the longwall face as it is being advanced, with these gateroads being maintained in the goaf of the panel using various combinations of pack walls and arch support systems. The primary advantage of the method is that the single entry gateroads are located in stress relieved zones. Disadvantages include slow mining rates due to gateroad advance and longwall face advance being interdependent; restricted access for ventilation and supplies; ongoing roadway maintenance requirements; increased propensity for spontaneous combustion due to air ingress into the goaf; and no second independent means for egress in an emergency situation.

Longwall mining on the retreat is the most common type of longwall mining. It involves driving one, two or three gateroads down both flanks of a panel to its extremity, and then connecting these two sets of gateroads (Fig. 2.3). The longwall equipment is installed in this connecting roadway and the block is progressively extracted on the retreat. The method is not subject to many of the impediments associated with longwall mining on the advance and has a lower exposure to others. However, serious ground control difficulties can be associated with supporting and maintaining gateroads, cut-throughs and pillar ribs. Longwall mining on the retreat finds extensive application in Australia, South Africa and the USA at depths ranging from as low as 15 m, down to around 700 m. In these countries, it is premised on multiple entry longwall development that requires leaving one or two rows of interpanel pillars (chain pillars) between longwall panels. The method finds application using single entry gateroads at depths exceeding 1,200 m in Europe. It is also used extensively in China, including to recover coal from the goaf in thick seams (see Sect. 9.9.1).

The number of gateroads utilised in longwall mining is a function of many factors including egress requirements, ground conditions, gas and ventilation regimes, panel dimensions, production rate and propensity to spontaneous combustion. Three gateroads are always required when there is a requirement for two independent means of egress from the mining face. High gas regimes may also require three gateroads in order to provide a sufficient quantity of air to dilute the gas to safe and prescribed levels. Gas pre-drainage does not necessarily remove this requirement because rib emissions in the gateroads can still result in gas content in intake airways exceeding permissible levels (typically, no more than 0.25 % CH_4 equivalent in intake roadways) before the air reaches the mining face. This situation is aggravated in long panels, some of which can exceed three kilometres, due to the increased roadway surface area.

In any case, air quantity requirements at the working face may require three gateroads in very long panels in order to compensate for reduced air flow due to increased resistance associated with surface friction and shock losses. The relationship between ventilation pressure, air quantity and airway resistance is given by Eq. 9.1:

$$P = RQ^2 \tag{9.1}$$

where

P = *fan pressure* (pa)
Q = *air flow quantity* (m^3/s)
R = *roadway resistance* (Ns^2/m^8)

The installation of standing support in roadways, especially tailgates, increases the resistance of the ventilation circuit considerably and can make the difference between requiring two or three gateroads in order to deliver the required quantity of air to the face. This is especially the case in coal seams prone to spontaneous combustion, as increasing fan pressure to compensate for increased airway resistance encourages air leakage between intake and return airways and across goaves, thereby promoting the development of spontaneous combustion. The basic principles for managing spontaneous combustion in these situations are presented in a range of literature, including Humphreys and Richmond (1986), Cliff et al. (1996) and MDG-1006 (2011).

Production rate also has a significant bearing on the number of gateroads required for longwall production. Gas emissions, dust make and heat generation increase with rate of retreat and, therefore, a higher quantity of air at an adequate velocity is required to safely manage these factors. The trend towards wider and higher longwall faces and higher capacity longwall production equipment also has a significant effect on heat production at the working face, with the power requirements of some installations exceeding 6 MW. Where conditions permit, consideration can be given to constructing a small diameter shaft at the inbye end of each panel as an alternative to driving extra gateroads for ventilation purposes.

Multiple gateroads aggravate ground control difficulties in longwall mining because the second and subsequent gateroads are exposed to longwall abutment stress. This necessitates that these roadways are either sufficiently remote from a

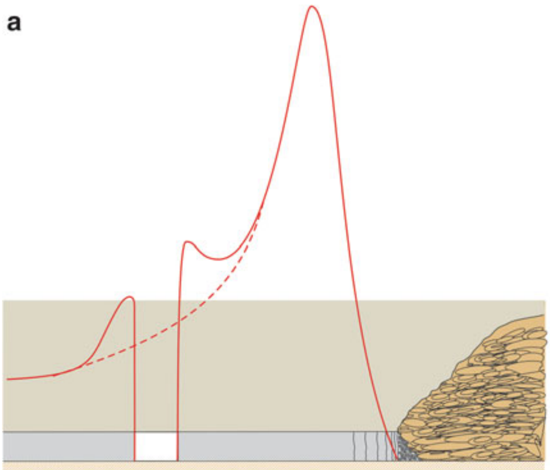


Fig. 9.1 Schematic options for locating twin entry gateroads within an abutment stress zone. (**a**) Wide chain pillar in order to locate tailgate away from high abutment stress, (**b**) Very narrow chain pillar in order to induce controlled pillar yield so that gateroads are then located in a stress relieved zone

longwall panel that abutment stress impacts on them can be safely and productively managed or else, in the case of a twin gateroad situation, the second maingate roadway is located in the yield zone of the abutment stress profile. These options, illustrated in Fig. 9.1, determine the type and width of the chain pillars. Hence, chain pillars can range from squat pillars in the former case to yield pillars in the latter case.

A raft of additional ground control difficulties can be experienced in driving and supporting cut-throughs. Problems can arise on development when preference is given to orientating headings rather than cut-throughs in the more favourable direction for managing horizontal stress. They can arise on extraction because the cut-throughs are located within the abutment stress front. For these reasons and in order to minimise roadway drivage, the distance between cut-throughs is usually maximised, thus resulting in the length of chain pillars typically being two to five times their width.

Many of the basic ground engineering principles relevant to designing roadways, pillars and support systems in these circumstances are presented in Chaps. 2, 3, 4, 5, and 6. Aspects of these specific to longwall interpanel pillars (chain pillars) are developed in more detail in this chapter.

9.2.2 Gateroad Direction and Layout

Factors which need to be considered in optimising gateroad direction and layout include surface constraints; lease boundaries; coal quality consistency; coal thickness consistency, dip and dip direction; cleat intensity and direction; and horizontal stress magnitude and direction. Conflicts between these factors require design compromises, with high horizontal stress tending to be the most important and dominant factor controlling design. Figure 9.2 shows the three general design options for managing this factor.

For the purpose of this text, the layout shown in Fig. 9.2a is referred to as a 0/90/90 layout, indicating that the gateroads are orientated parallel to the major horizontal stress direction and the cut-throughs and longwall installation face are at right angles to this stress direction. This layout minimises the adverse impact of horizontal stress on the gateroads and maximises it on the cut-throughs and installation face. Therefore, it has the advantage of optimising conditions during longwall retreat at the expense of potentially difficult development conditions when driving cut-throughs and the longwall installation roadway. Conditions can be particularly adverse at the point of backholing a cut-through that has been developed from both directions.

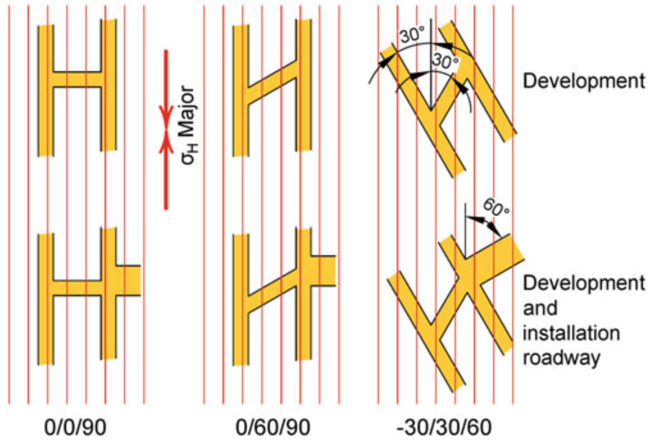


Fig. 9.2 Orientation options for longwall gateroads, cut-throughs and installation roadway in a high horizontal stress field

Figure 9.2b depicts a 0/60/90 layout in which the gateroads continue to be orientated in the optimum direction and the installation face in the most adverse direction, but the cut-throughs are now orientated at 60° to the horizontal stress field to mitigate its impacts on them. The benefits of this option have to be weighed up against a range of operational impediments that can be associated with angling cut-throughs in this manner. These include:

- Cut-throughs can only be driven from one direction. This introduces scheduling restrictions and reduced operational flexibility, both of which may retard advance rates.
- Mobile plant cannot turn left and right into and out of cut-throughs, again reducing operational flexibility.
- There is a higher likelihood that one mining direction will not be optimum for controlling the impact of cleating and jointing, thereby elevating the risk of injury and causing an increase in mining spans due to rib spall.
- Two corners of the chain pillars are acute, resulting in them having a reduced strength, being susceptible to damage by equipment, and prone to fall along cleat and joint planes. In turn, these factors elevate the risk of injuries due to risk of rib fall and result in increased intersection spans and, hence, exposure to roof control problems.

The compromise situation of a −30/30/60 layout, in which horizontal stress impacts on headings, cut-throughs and the installation roadway are all reduced but not eliminated, is depicted in Fig. 9.2c. This layout can be difficult to implement due to other competing mining considerations and constraints, such as lease boundaries, seam dip, and gas management.

Other stand-alone or complementary options for managing horizontal stress in roadways during gateroad drivage and longwall extraction include primary and secondary support strategies and stress relief roadways. These aspects are discussed in more detail in Sects. 3.4 and 5.2 and Chaps. 6 and 7.

9.2.3 Chain Pillar Life Cycle

Longwall interpanel pillars, or chain pillars, can perform a variety of functions, some of which change over the life of the pillars. Essentially, the pillars are required to remain structurally stable and functional until at least the passage of the second longwall. In designing a chain pillar, consideration should be given to its life cycle, which can be broken down into five stages on the basis of pillar loading (Fig. 9.3). The stages are described in respect of a two heading development but the principles also hold true for a three heading development.

- Stage I. Two gateroads are developed into what, for practical purposes, could be considered virgin ground. Pillars are surrounded by solid in all four quadrants (that is, for 360°).

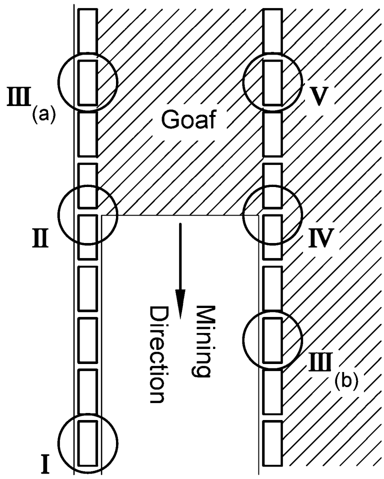


Fig. 9.3 Loading stages in the life cycle of a chain pillar

Unless the depth of mining is very shallow, the overall width-to-depth ratio, W/H, of the gateroad panel is too small to result in full tributary load acting on the chain pillars. Depending on depth and horizontal stress magnitude and direction, the goaf of an adjacent longwall panel may cause the gateroad development panel to be shielded to some extent from horizontal stress.

- Stage II. The longwall face reaches the chain pillar and the pillar is now surrounded by solid in only three quadrants (270°). The abutment load of the goaf in the fourth quadrant is distributed between the longwall block and the side abutment, which includes the chain pillar, resulting in an increase in chain pillar load.
- Stage III(a). As the longwall retreats past a chain pillar, the abutment load carried by the longwall face is progressively transferred onto the chain pillar, or side abutment. Ultimately, the side abutment falls beyond the influence of the longwall face and the chain pillar is subjected to full **maingate abutment loading** arising from the goaf, which now occupies two adjacent quadrants of the pillar (180°). This situation is referred to as **single sided abutment loading**. The magnitude and distribution of the abutment stress profile and the width of the chain pillar determine the proportion of abutment load carried by the chain pillar.
- Stage III(b). The chain pillar occupies the same relative position as in stage III(a) and so, in theory, the abutment loading acting on it should be unchanged. However, because the strength of rock can decrease over time, especially under high load, the stability of both the chain pillar and the adjacent gateroad (tailgate) may deteriorate in the time interval between the passage of the two longwalls. Ongoing mining-induced changes in the stress field can also contribute to this deterioration.
- Stage IV. The chain pillar is now surrounded by solid in only one quadrant (90°). The abutment load of the second goaf is distributed between the second longwall block and the chain pillar, resulting in a further increase in chain pillar load. This situation is sometimes referred to as **tailgate abutment loading**.
- Stage V. As the second longwall retreats beyond the chain pillar, the abutment load carried by the longwall face is again progressively transferred onto the tailgate side abutment, with the chain pillar now surrounded by goaf in all four quadrants. This situation is referred to as **double sided abutment loading**. Due to the stiffness of the superincumbent strata, the load acting on a double abutment loaded chain pillar may not initially be double that for a single sided abutment loading situation, especially at depth. A number of additional panels may have to be extracted before this state is reached, as evident from the profiles of vertical surface displacement depicted in Figs. 3.16 and 3.19.

9.2.4 Chain Pillar Design

Chain pillars constitute interpanel pillars, with the basic principles pertaining to their function and design introduced in Sect. 5.2. There is no single correct design method for longwall chain pillars, particularly since the roles of chain pillars in a mine layout may be quite diverse (Hebblewhite and Galvin 1996). Nevertheless, in nearly all cases, a primary function of chain pillars is to provide a buffer of sufficient width between the goaf of the previous longwall panel and the gateroads of the current longwall panel in order to shield the gateroads from high abutment stress. Therefore, chain pillar design should include consideration of the abutment stress that gateroads can tolerate, having regard to the local geology, the type and density of support to be installed in the gateroads, and the level of serviceability required of them (Galvin et al. 1982).

A variety of empirical and numerical approaches are currently utilised to design chain pillars. Many of the empirical approaches rely on the concepts of angle of break and abutment angle and a single sided abutment load multiplying factor to estimate pillar load at the tailgate/face corner. This load is then compared to pillar strength calculated using an empirical equation derived for bord and pillar mining situations. The two most common means of calculating chain pillar width utilising these empirical approaches are:

- To work backwards from a design pillar safety factor that is judged to produce a safe working environment, acceptable tailgate conditions and, where required, adequate surface subsidence control. In some circumstances where only minimal surface subsidence is tolerable, UNSW power safety factors of 2.2 or more based on double sided abutment loading have been used with the intent of preventing pillar failure in the long term. Otherwise, the design safety factor is usually based on the notion of preventing pillar failure until after the second longwall face has passed by the pillar. in which case the maximum pillar load is taken to be that acting on the chain pillar at the tailgate end.
- To work backwards from a design stability factor selected on the basis of its empirical relationship to some measure of tailgate serviceability. The stability factor is equated to the ratio of chain pillar strength to chain pillar load, with the latter usually estimated at the tailgate corner. A number of permutations of abutment angle and single sided abutment loading multiplication factors can be associated with the calculation of pillar load in some approaches, for example, ALTS (M. G. Colwell et al. 1999). The philosophy of NIOSH (2008) in respect of ALPS and ARMPS needs to be borne in mind, this being that since these are empirical models derived from real-world data, they do not require a full understanding of the mechanics of pillar behaviour. This is an important consideration when applying the formulations at sites other than from where the data was sourced. Risk is always associated with situations where there may be a lack of understanding of the mechanics underpinning behaviour or where loading conditions are significantly different to the cases in the underpinning databases (for example, in multiseam mining situations).

Limitations are associated with both empirical approaches. For example, the concept of an abutment angle does not reflect the mechanics of overburden behaviour as depth increases (Sect. 3.2), while none of the empirical pillar strength formulae applied in the various design procedures were derived on the basis of the behaviour of pillars that abutted caved ground or for pillars in the high width-to-height ratio range of many chain pillars.

In Sect. 5.2.2 it was noted that numerical modelling has been promoted for designing chain pillars since at least the early 1980s. Nevertheless, limitations can still be associated with these approaches, especially in regard to quantifying pillar load, the effect of caving on pillar strength, and goaf reconsolidation characteristics. Notwithstanding this, the cost of undertaking parametric and sensitivity analysis utilising sensible numerical models in order to give confidence to chain pillar design is minor to trivial in comparison to the adverse safety, productivity and financial risks associated with a poor chain pillar design in longwall mining. The various ways in which numerical modelling finds application to chain pillar and gateroad design and support are reflected, for example, in the approaches of Salamon (1991), Gale (2004), Peng (2008), and Esterhuizen et al. (2010b).

As the depth of mining increases, strength considerations result in an increase in chain pillar width-to-height ratio. This has implications for both the pillar width required to provide an adequate buffer from abutment stress and for the propensity for pressure bursts within the chain pillars. A situation is also reached where, irrespective of the width of the pillar, induced stress levels at the pillar ribsides result in deformations sufficient to threaten safety and the serviceability of the gateroad. Longwall mining on the advance is uneconomic for mitigating these impacts. Hence, the concept of **yield pillars** has found application in designing chain pillars in deep longwall retreat operations in attempts to ameliorate pressure bursts, severe rib spall, and pillar punching of the roof and floor strata. The concept is also used in the USA to minimise coal sterilisation and to provide optimum geometries for place changing in three heading developments, and in South Africa to provide more uniform surface subsidence profiles.

The concept of a yielding coal pillar is based on the controlled unloading of a coal pillar once its peak load carrying capacity has been exceeded. It has been applied in the USA in two, three and four heading gateroad layouts. It relies on utilising the post-failure strength of a yielded pillar to provide local ground support, while transferring (shedding) the majority of the overburden and abutment load to adjacent, stiffer, non–yielding pillars. The terminology is sometimes confused, with a yield pillar also referred to as a **crush pillar**. Hebblewhite and Galvin (1996) report that many so-called yield pillars are, in fact, stable load-bearing pillars of very low height in benign roof strata conditions. It is important to appreciate the distinction since the penalty for poor design is severe in the form of sudden and unpredictable pillar collapse.

A review of gateroad yield pillar design approaches and applications in USA longwall operations by NSA Engineering (2000) found that yield pillars at that time were generally 6–9 m wide and ranged in width-to-height ratio from 3 to 5. No 'entirely successful' yield pillar designs were achieved when width-to-height ratio exceeded 5, nor were any 'operationally successful' full-yielding gateroad systems achieved in ground where the CMRR was less than 50. Measurements suggested that full yielding of a pillar seldom occurred until after the first adjacent panel has been extracted well outbye of the pillar and the majority of peak side abutment stress has been attained. Badr et al. (2002) reported similar findings, noting that previous designs have enjoyed mixed success and that load shedding requires three criteria to be satisfied, namely:

- there are load bearing areas (unmined seam or compacting goaf) nearby which can sustain the transferred load;
- the roof and floor are sufficiently competent to facilitate the load transfer without debilitating roof falls or floor heave; and
- the stiffness of the surrounding rock mass is sufficiently high to ensure that the equilibrium of the rocks remains stable.

Salamon et al. (2003) undertook numerical simulation of longwall chain pillars of width-to-height ratios 3, 5 and 10 at a depth of 700 m. The authors noted that their discussion of results deliberately avoided the quantification of the terms 'narrow', 'intermediate' and 'squat'. The research indicated that for narrow pillars, pillar deformation is controlled; the yielding zones progress towards the centre of the pillar smoothly; and a pillar that is yielding throughout its width can readily be created. If the depth of mining is great, this full yielding state can be reached during the development of the gateroad entries. The desktop analysis concluded that such narrow pillars make ideal yield pillars, their only shortcoming being that their load bearing capacity is low. This limits the spans over which they should be applied.

The study concluded that it appears pillars with an intermediate width-to-height ratio cannot be brought into a fully yielding state because their failure process becomes unstable when yielding penetrates to a certain depth. This critical depth could be reached either during primary development or during secondary longwall extraction. The instability may induce a pressure burst-like event and even a total collapse of the pillar. Therefore, the utilisation of yield pillars of this size should be restricted to relatively shallow cover.

Salamon et al. (2003) also concluded that squat pillars have the potential for sudden seismic events in their outer zone but, because of the large width of the remaining inner core, the rubble around the pillar sides provides sufficient confinement to enable stability to be re-established. These pillars were not considered ideal chain pillars in deep longwall mining situations.

A feature of most successful yield pillar and crush pillar outcomes to date has been the presence of very stiff immediate roof strata. This is not surprising, as the high stiffness of this strata regulates both the magnitude and rate of load transfer to pillars adjacent to the longwall block, thus controlling the rate of yield and failure mode of these pillars.

9.2.5 Chain Pillar/Gateroad Behaviour

9.2.5.1 Stage I – Development

During gateroad development, roadway and pillar behaviour are governed by the same principles that apply to bord and pillar mining. The main difference between bord and pillar main development and gateroad development, which is unlikely to be detectable in practice, is that the load acting on the gateroad pillars may be lower because the narrower gateroad panel width results in a smaller reduction in the stiffness of the overburden. Nevertheless, this load can still be expected to result in extensive fracturing of gateroad sidewalls as depth of mining increases.

In the case of intermediate depth longwall panels, actual pillar stresses may be comparable to those encountered at the shallower depths of typical bord and pillar mining. In deeper longwall situations, pre-mining rock stress will already be higher than pillar stresses normally encountered in bord and pillar mining and, therefore, extensive fracturing of gateroad sidewalls can be expected even at narrow panel widths.

9.2.5.2 Stage II – Maingate/Face Corner

As the goaf approaches a chain pillar at the maingate face corner, the pillar is subjected to increased abutment load. The presence of the goaf also causes a change in the state of horizontal stress in the immediate roof and floor of the gateroad. Primary factors that determine the magnitude of this change include the direction of the major horizontal stress relative to the gateroad direction; the composition of the immediate roof strata; the elastic modulus and Poisson's ratio of the immediate roof strata; and whether caving develops at the face or is delayed. Figure 9.4 shows an example of the manner in which a horizontal stress notch developed around the maingate end of a longwall face when the in situ major horizontal stress was approximately twice the primitive vertical stress high and orientated at around 30° to the maingate.

A widely-employed relationship between the horizontal stress concentration factor and the angle of the maingate to the major horizontal stress direction is plotted in Fig. 9.5a. It is based on stress measurements in stone strata some 5 m over the rib of chain pillars and 2.5 m above the mining horizon. Horizontal stress is shown to peak at almost 2.2 times primitive (virgin) stress at this horizon when the gateroad is orientated at approximately 65° to the major horizontal stress direction. The end user of the relationship shown in Fig. 9.5a needs to be cognisant that it is based on limited data, with the shape at higher stress angles being determined by just one data point (Cook Colliery). Furthermore, it has no regard to the impact on ground behaviour of the ratio between the two principal lateral stress directions.

An update of this relationship developed by Gale (2014) is shown in Fig. 9.5b. This revision has regard to some new data, face position and numerical modelling. Both relationships shown in Fig. 9.5 indicate that maingate stress conditions will be optimised when the gateroads

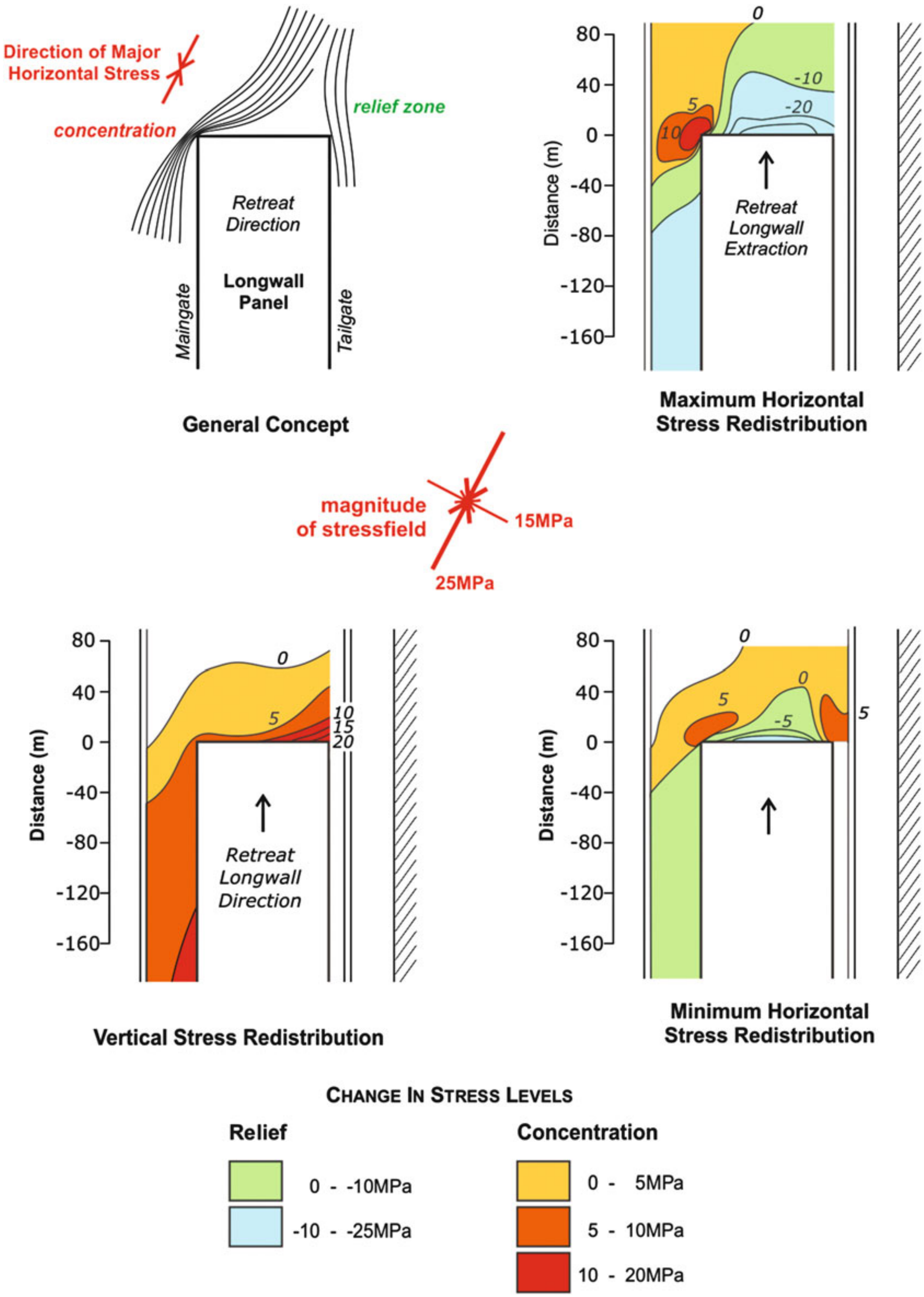


Fig. 9.4 Vertical and horizontal stress distribution about a longwall panel as determined from monitoring at a depth of around 500 m in the circumstances noted in the figure (After Gale and Matthews 1993; Gale 2014)

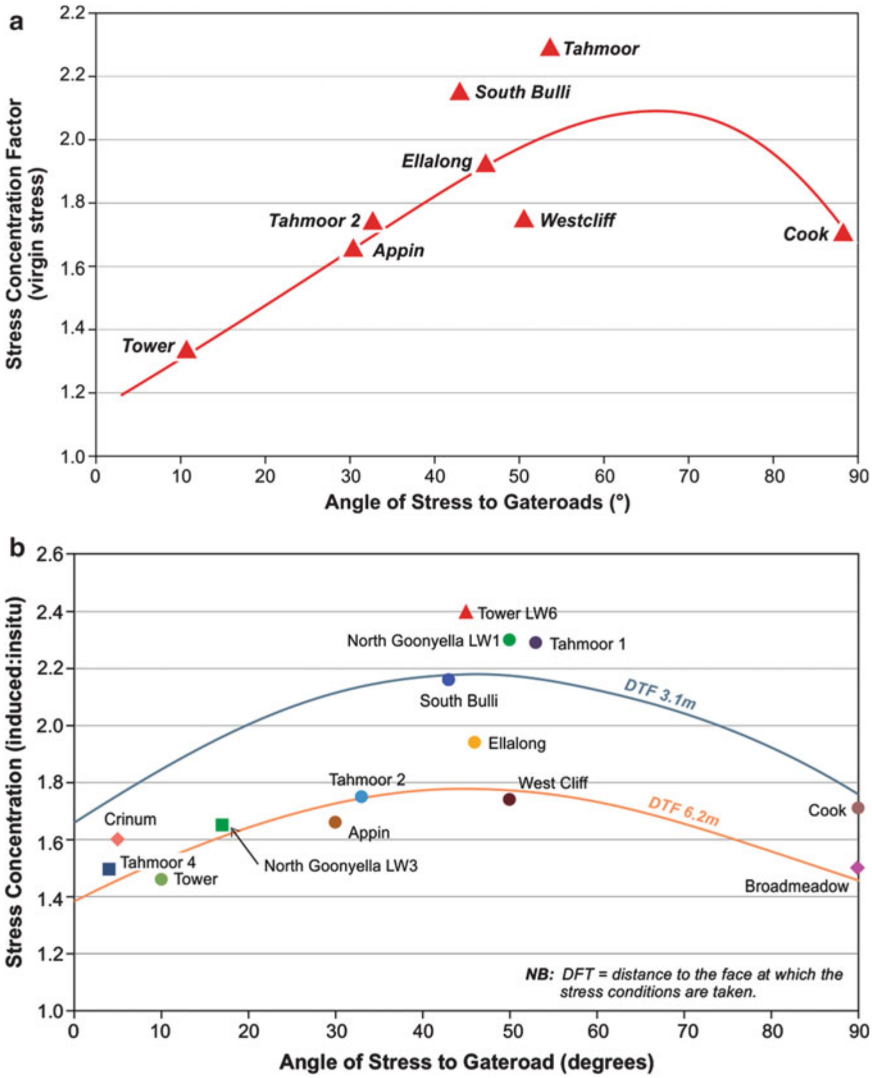


Fig. 9.5 Relationships between orientation of gateroads relative to the major horizontal stress direction and horizontal stress concentration factor (After Gale and Matthews 1993; Gale 2014). (**a**) Relationship reported by Gale and Matthews (1993), (**b**) Updated relationship reported by Gale (2014)

are orientated within approximately 25° of the direction of the major horizontal stress. Stress relief is very well developed in stiff materials but not well developed in thick coal or soft materials (Gale 2014).

Stress notching tends to develop once the extraction face approaches to within about 30 m of an intersection but can be present over the full panel length and, if the stress is sufficiently high, extend into the companion gateroad (travelling road). In some instances, the intermediate/minor horizontal stress may also be of sufficient magnitude to result in stress notching. There is usually a marked increase in the impact of a stress notch when the extraction face is within 20 m of a maingate intersection, before the stress is relieved at the intersection. This stress relief is conducive to the unravelling of the strata fractured by the notching, resulting in intersection roof falls to a considerable height if pre-emptive support measures are not in place. Typically, these measures need to include long cables installed well ahead of the stress notch together with some form of surface support system. It is preferable to install this additional support prior to advancing the conveyor belt during gateroad development so that the placement of the support is not constrained at a later date by equipment, lack of access, or lack of space.

The floor is also subjected to the stress notching, which can result in floor heave making a substantial contribution to convergence. This may necessitate the installation of standing support, the leaving of additional coal in the floor, concreting of the floor and, in some cases, the bolting of the floor.

9.2.5.3 Stage III – Travel Road/Tailgate Single Abutment

The significantly increased loading on the pillars and the presence of the adjacent goaf create the potential for a number of interactive behaviour modes to impact on pillar and roadway stability in single abutment loading situations. In the first instance, the increased pillar load results in compression of the pillar and its roof and floor strata. The strata adjacent to the goaf are free to dilate but this freedom progressively reduces with distance from the goaf edge back into solid abutment as self-confinement is restored. This generates an increase in horizontal stress in the immediate roof and floor strata due to the Poisson's effect. Depending on mining geometry, the Poisson's effect may be substantially recovered at the site of the travelling road (maingate companion road, which subsequently becomes the tailgate), thus subjecting the roof and floor strata of this roadway to elevated horizontal stress from this contributing factor.

If the increase in abutment stress is sufficiently large, it can initiate or aggravate yielding and crushing of the outer portions of the coal pillars. This results in an increase in the effective span of the travel road, thus reducing the resistance of both the roadway roof and floor to bending and buckling forces.

Crushing and yielding of the outer portions of a pillar give rise to a second source of induced horizontal stress in the roof and floor strata of the travel road/tailgate. The confined core concept for explaining pillar strength (Chap. 4) is premised on the outer crushed and yielding zones of a coal pillar providing confinement to the elastic core of the pillar (Fig. 4.22). In accordance with Newton's law of action and reaction, these restraining forces have to be balanced by compressive forces induced in the roof and floor strata. These compressive forces may induce the buckling and failure of the roof strata and/or the buckling and heaving of the floor beds (Salamon 1991) (Fig. 9.6).

In addition to the coal pillar element of the pillar system, consideration has to be given to the mechanical properties of the immediate roof and floor strata of the pillar system. Possible behaviour modes under the effect of high abutment stress include bearing capacity failure of the roof or floor strata and extrusion of soft roof or floor layers.

It has been suggested by some researchers that the immediate roof of a longwall travelling road/tailgate can be put into tension following the formation of the first adjacent longwall goaf. They attribute this to horizontal stress relief resulting from one or a combination of the presence of the goaf and differential pillar compression either side of the gateroad. The concept of horizontal stress relief due to the formation of a goaf is discussed in Sect. 5.2.5 and illustrated in Figs. 5.12, 5.13, and 5.14. The deviation of the in situ stress field around the zone of softening overlying a goaf may result in a reduction in lateral stress in the roof and floor of a travel/tailgate roadway but, in most cases, the residual component of lateral stress is still likely to be significant.

The concept that differential pillar compression associated with yielding coal pillars could contribute to the immediate roof of a tailgate being placed in tension (for example, as proposed by R. W. Seedsman 2012) appears to have its origins in stability concepts put forward by Diederichs and Kaiser (1999b) in relation to mining beneath blocky hanging walls in hard rock mines. The end-user is advised to carefully review the source publication to determine the relevance of the concept to their conditions.

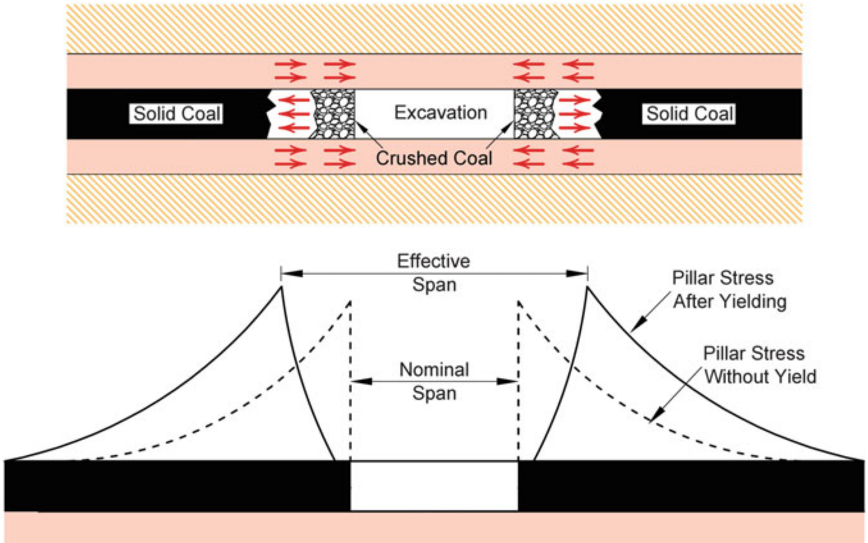


Fig. 9.6 A schematic of pillar edge crushing showing the influence of the associated dilation and yielding on the lateral loading of the immediate roof and floor strata (Modified from Salamon 1991)

In some situations, roof displacement and floor heave can be associated with the failure and dilation of stiff, thin layers within the immediate roof or floor strata. These strata attract stress because of their stiffness before shearing and bulking so as to drive the roof down or the floor up, often in a rapid manner. This behaviour mode can be difficult to distinguish from pure buckling failure and from general bearing capacity failure. Two or more modes may be interactively in play at the same time.

Figure 9.7 shows an example of floor heave that developed dynamically in a tailgate companion roadway. The coal roof and pillar ribs were bolted and strapped. As the pillars dilated under high abutment load, the supported rib line remained intact but started to ride over the W straps and rock bolts installed in the roof. Ultimately and without warning, the coal floor uplifted. Vasundhara et al. (2003) provide more detailed discussion on weak floor failure mechanisms associated with longwall mining operations.

A range of operational benefits is associated with not installing standing support in a longwall travelling road/tailgate. These relate to ventilation efficiency, inspections, material and

Fig. 9.7 Dynamic heave of a coal floor beam in a gateroad located adjacent to a highly loaded chain pillar

equipment access, and labour requirements. Hence, there has been a focus on replacing standing support systems with long tendons. This has met with success at some mines. However, standing support continues to be required in those mining environments where floor uplift constitutes a significant component of seam convergence and where the effectiveness of tendon support systems is adversely affected by shear displacement on bedding planes. The timing of the installation of standing support is a matter for site management, as dictated by local conditions and mining priorities. However, when standing support is required, it is strongly advisable to always have it in position for 100 m outbye of the longwall face and, preferably, for at least 200 m.

9.2.5.4 Stage IV – Tailgate/face

Ground behaviour in the vicinity of the tailgate end is complicated further by two factors. Firstly, the chain pillars are subjected to additional increases in abutment stress. Secondly, the immediate and upper roof strata have another degree of freedom, with the opportunity to displace both transversely into the existing adjacent goaf and longitudinally into the approaching new goaf. Weak bedding planes in the roof facilitate large scale slip of the roof strata towards the goaf, as measured for example by Fabjancyk et al. (2006). Assessment of strata deformation modes and impacts in these environments falls outside the scope of empirical and semi-empirical approaches to pillar stability assessment.

One type of behaviour specific to this environment is the so-called **skew roof mechanism,** which Tarrant (2005a) and Fabjancyk et al. (2006) associate with a change in the profile of a tailgate from rectangular to rhomboidal as shown in Fig. 9.8. The behaviour, which can vary in magnitude and direction between mine sites, has been attributed to the reorientation of the stress field around the goaf generating shear couples on bedding planes and other structures in the roof and floor. These shear couples result in differential shear within the strata, leading to high levels of strata failure in the roadway. According to Fabjancyk et al. (2006), once a skew roof mechanism is initiated, it is likely to extend a substantial distance into the goaf. The magnitude and direction of pre-mining horizontal stress is believed to have a major impact on the direction of the skew and the extent that the skew process impacts on the roadway. Tarrant (2005a) lists the key factors driving skew roof behaviour as:

- the absolute and relative magnitudes of the vertical and horizontal stresses;
- the shear modulus of the strata pile (shear deformability); and
- the extent of overburden bridging.

Tarrant credits shear stress damage due to skew roof behaviour with being capable of destroying intrinsic support, including cable bolts. Hence, standing support rather than cables is considered the most appropriate stabilisation strategy, with Tarrant (2005b) providing a range of recommendations in that regard.

Fabjancyk et al. (2006) provide further discussion of the skew roof mechanism, concluding that the range of strata deformation mechanisms that can occur around goaves warrants that the positioning of roadways in the vicinity of goaves is based on a higher level of assessment than that used for traditional pillar stability approaches. Moodie and Anderson (2011) report on similar behaviour associated with longwall top coal caving at Austar Coal Mine, Australia, where change in vertical stress was measured to be higher on the travel road side of a chain pillar rather than on the goaf side.

In summary, pillar and roadway behaviour about a tailgate can be complex. It can involve a range of stress paths and mechanisms. All may have application in some situations but none are exclusive. Different mechanisms and combinations of mechanisms operate in different environments and at different points in time in the mining process. As already noted in Sect. 5.2.5, each situation should be individually assessed, with consideration given to utilising

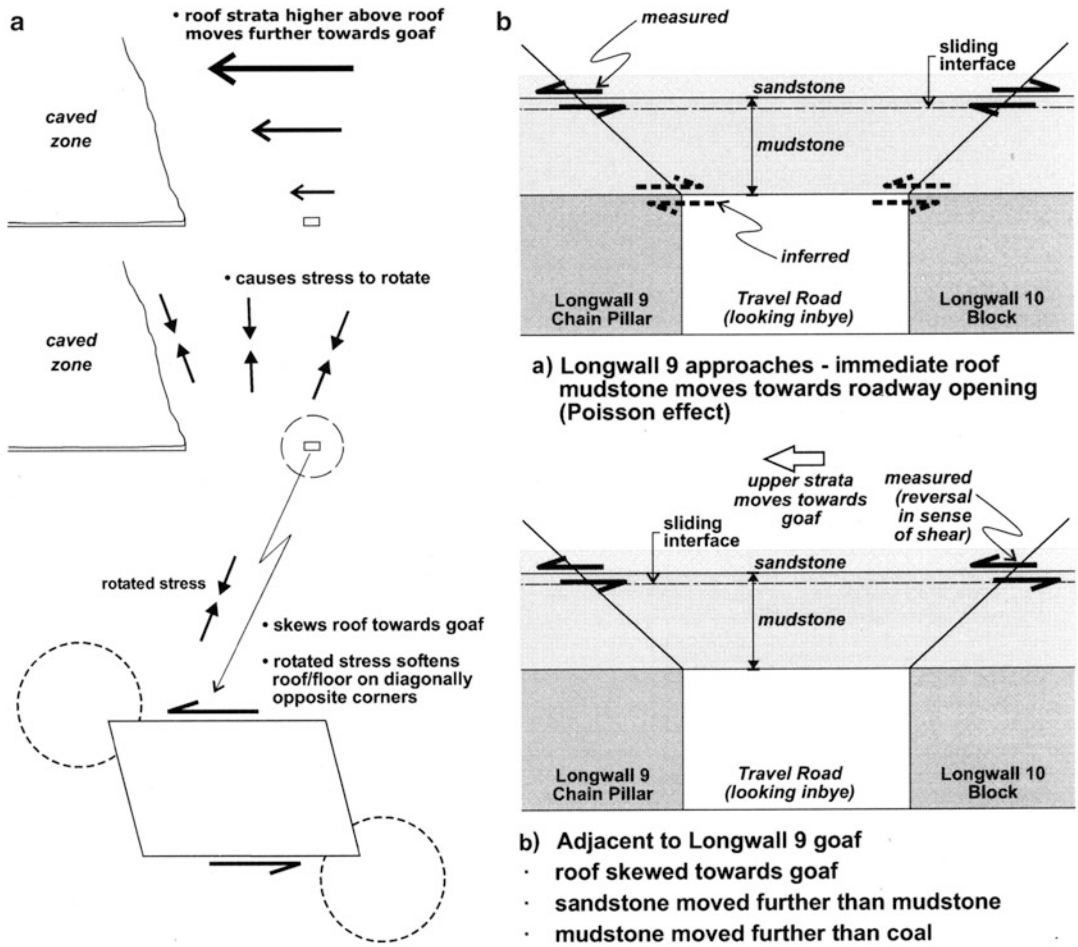


Fig. 9.8 Concepts developed by Tarrant (2005a) relating to the skew roof mechanism in longwall tailgates. (**a**) Simplified model of stress/displacement changes adjacent to goaf. (**b**) Relative movement as monitored at Metropolitan Colliery, Australia

numerical modelling to predict principal stress magnitudes and profiles and to give insight into strata behaviour modes.

9.2.5.5 Stage V – Double Abutment Loading

Full double abutment loading situations are not usually of interest other than if the chain pillars have a role to play in restricting surface subsidence or if multiseam mining is contemplated. In the case of surface subsidence, the structural integrity of the coal pillar and the compression of the chain pillars and surrounding strata take on added significance. It should be noted that surface subsidence above a chain pillar is not necessarily an indicator of the state of stability of the pillar. This is because elastic strata compression, especially at depth, can make a major contribution to surface subsidence.

Particular care is required when using surface subsidence behaviour above chain pillars to draw conclusions about their state of stability. Some vertical surface displacement will occur over any chain pillar simply due to elastic compression of the coal pillar and surrounding strata in response to mining-induced stress. This compression can be quite considerable at depth. Lateral displacement of the overburden towards the goaf is another factor that contributes to the development of vertical displacement above chain pillars.

9.3 Longwall Powered Supports

9.3.1 Development

Longwall mining of coal originated in the 1800s as a so-called ‘hand-got’ mining method (pick and shovel) using timber props with headboards as face support. Hand-got mining was progressively replaced with shotfiring, ploughs and shearers. Support progressed to rows of friction props with connecting bars that were leapfrogged forward with face advance. Hydraulic props were introduced in the 1940s, followed by the Eastern European concept of a shield support comprised of a rigid, half-arch, frame to protect the face from goaf flushing. These early support systems provided the basis for the first longwall **powered supports**, so-called because they were connected to a hydraulic power supply and capable of self advancing.

In Russia, the rigid shield was developed into a hydraulic **shield support** by pinning a canopy to the flushing shield and connecting the flushing shield to a base by a simple hinge and one or two hydraulic legs (Figs. 9.9a and 9.10). These supports were often referred to as **caliper** or **arc shields** since the canopy tip followed a circular pathway as the support was raised or lowered. The concept was developed further in Germany in the 1960s, with these shield supports being characterised by rear-facing angled legs; a relatively high tip load capacity; a low rear load carrying capacity; good protection against goaf flushing; and a high longitudinal stiffness to resist horizontal displacement towards the goaf. Because the legs of a shield support are angled, the vertical support provided to the roof is less than the rated capacities of the legs and reduces as the legs become more inclined when the support yields.

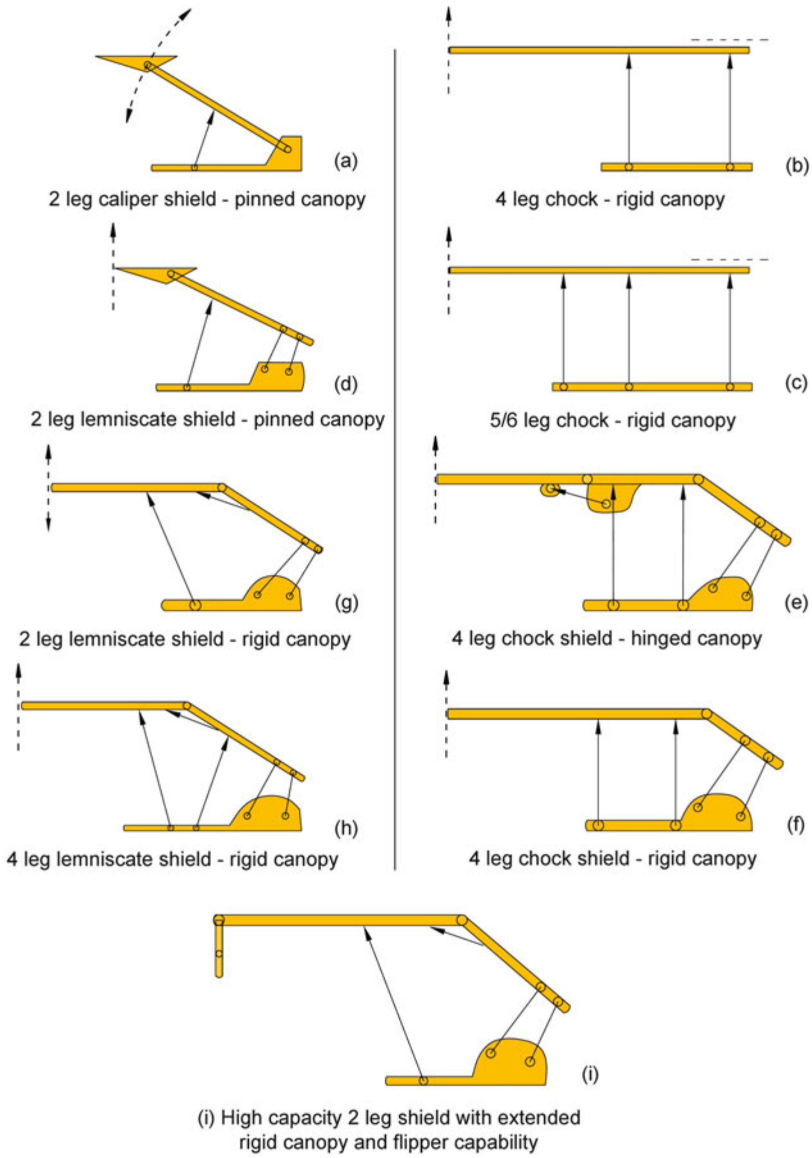
The first hydraulic supports in Britain were installed in 1951. These comprised hydraulic legs mounted on a base, with connecting bars replaced with a solid canopy to produce a so-called hydraulic **chock support** (Fig. 9.9b). The British National Coal Board (NCB) stipulated a maximum distance between the coal face and the front support leg of 2 m, which effectively precluded the use of 2 leg shield supports. The NCB also dictated that the full face height had to be cut in one pass (bi-directional shearing) and that the face was operated in so-called **conventional mode**, whereby the supports were set up against the armoured face conveyor (AFC) prior to the passage of the shearer and advanced immediately after its passage.

The British powered supports were characterised by four or more vertical hydraulic legs, a high rear load carrying capacity, a low tip load carrying capacity, poor protection from flushing of the goaf, and poor longitudinal stability, the latter making them prone to collapse in a scissor-like manner as the immediate roof strata moved towards the goaf. In order to address the low tip load capacity and to satisfy NCB specifications, additional hydraulic legs were fitted to the front of some supports (Figs. 9.9c and 9.11).

A major design development in longwall hydraulic supports occurred when the German coal industry replaced the simple hinge on the shield support with a **lemniscate linkage**. The lemniscate linkage caused the canopy tip to travel in a near vertical plane as the support was raised and lowered and imparted high longitudinal stability to the support (Fig. 9.9d). This addressed the concern that the arc motion of a caliper shield resulted in an unfavourable reduction in confinement to the immediate roof when the support was set and a favourable increase in confinement as the support yielded and converged. Subsequently, the concept was incorporated into chock supports to prevent them from collapsing into the goaf, thereby giving rise to the **chock shield** (Fig. 9.9e).

In the meantime, shield supports were fitted with a rigid canopy instead of a pinned canopy and forward angled legs became the norm, with both features increasing tip load capacity (Figs. 9.9g and 9.12). In the 1970s, in response

Fig. 9.9 Chronology of the development of longwall powered supports



primarily to a marketing ploy by some chock shield manufacturers that supports with four legs 'obviously' had to offer support benefits over shields with only two legs, an additional pair of legs began to be incorporated into shield supports (Fig. 9.9h). These supports were short-lived, with engineering analysis by McKay (1978) and others of the heavier, more complex and more costly 4 leg supports, concluding that the additional legs offered little, if any, additional support benefits and, in some instances, reduced performance.

Up until the late 1970s, many chock shields employed a hinged canopy, also referred to as a split or cantilevered canopy. The location of the hinge point ranged from between the front and back legs to well in front of the front legs, although the hinge point immediately in front of the front legs was most common (Figs. 9.9e and 9.13). This allowed the canopy to adapt better to

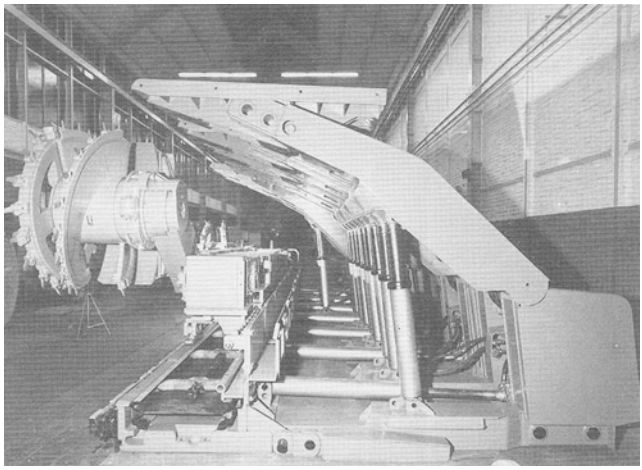


Fig. 9.10 2 leg caliper shield powered supports, shearer and AFC for the longwall face reported by Cloete (1980) to have set a world record of a monthly production of 162,557 tonnes at Sigma Colliery, South Africa, in June 1980

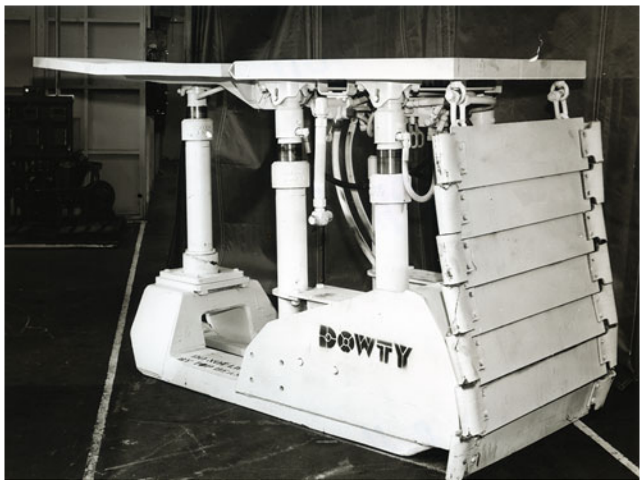


Fig. 9.11 A five leg, cantilevered canopy, chock powered support

the shape of the roof, which often contained vertical steps because the shearer cutting horizon was controlled manually. The tip load capacity of these supports was independent of the leg capacity, being determined by the cantilever cylinder capacity and its lever arm distance. The supports generated low tip loads, typically one-tenth of the nominal support capacity and, with the onset of yielding, the cantilever extensions were prone to a domino collapse along the face. A failure of this type involving supports with a tip capacity of only 100 kN (10 t) occurred on the first longwall face at Coalbrook Colliery, South Africa, in 1979 (Henderson 1980; personal experience). These types of incidents contributed to cantilevered canopies being phased out in favour of rigid canopies (Fig. 9.14).

Major advances in extrusion technologies in the 1990s enabled hydraulic leg capacity to be more than doubled, from typically 2–5 MN (~200 to 500 t). Leg capacity has continued to increase, approaching 0.9 MN (900 t) by 2010, with corresponding increases in tip load capacity and in leg stiffness due to the larger bore area. The configuration and high tip load capacity of shield supports has enabled the length of the rigid canopy section to be extended while maintaining a very high tip load capacity, so that longwall faces are now operated routinely with the powered supports set back from the AFC a

Fig. 9.12 A 2 leg, rigid canopy, shield powered support fitted with a lemniscate linkage

Fig. 9.13 A 4 leg, cantilevered canopy, chock shield powered support

distance of one cutting web. The additional space gained in this so-called **one-web back** mode provides a number of operational benefits, such as improved ventilation, larger face conveyors, and a second travel (walk) way along the face. In the event that face conditions deteriorate and/or the face spalls excessively, the option still exists to ‘close up the face’ by advancing the supports a distance of up to one web, although care is then required when taking the next shear to avoid cutting into the support canopies. Closing up the face in this manner is referred to as **double chocking.**

The substantial improvements that have been achieved in tip support capacity and minimisation of the area unsupported between support tips and the face can be negated if the coal face spalls, especially in thicker seams. To address this problem, it is now very common for powered supports utilised in thicker coal seams to be fitted with an hydraulically activated extension, or **flipper**, that can be deployed as either or both an extension to the canopy to confine the immediate roof and a face sprag to confine the coal face (Figs. 9.9i and 9.15).

In order to accommodate larger diameter legs and longer canopies and flippers and to improve the lateral and torsional stability of shield supports in thick seam mining operations, the width of powered supports has progressively

Fig. 9.14 A 4 leg, rigid canopy, chock shield powered support

Fig. 9.15 A 2 leg, 17.5 MN capacity, shield powered support fitted with an articulated flipper

increased from the traditional 1.5 m to upwards of 2.0 m. Hence, shield supports now offer advantages over chock shield supports in terms of higher tip load capacity; increased vertical stiffness; reduced number of components; less structural complexity; and reduced size and reduced weight, while at least matching the rear support capacity of chock shields. Shield supports are standard on all new longwall faces in Australia and the USA. However, 4 leg chock shields continue to be utilised in longwall top coal caving operations in thick seams.

9.3.2 Basic Functions

Effectively, powered supports are located in the goaf and, therefore, are surrounded by strata that have already been impacted by mining-induced fracturing. The basic ground control functions of a powered support are to maintain this fractured strata in a confined and interlocked state; control convergence in the face area to limit further localised fracturing and bedding plane movement; and provide a goaf break off line. These functions are not mutually exclusive.

Excessive convergence generates additional fracturing and leads to increased rib spall and guttering at the face, and bed separation and block detachment above the supports, all of which are conducive to roof falls on the longwall face. Face spall increases the unsupported distance between the tip of the longwall supports and the face. Guttering results in roof cavities and, together with bed separation and block detachment, increases the load acting on the supports. In turn, this increases the likelihood and magnitude of support yield, resulting in more convergence. The process can become self perpetuating as yielding of the powered supports results in increased face load, bed separation, and fracturing. Failure to induce caving of the immediate roof at the rear of the powered supports aggravates these conditions.

In addition to providing support to the roof, a powered support assists in sustaining horizontal stress in the immediate roof strata to confine the fractured rock and maintain it in an interlocked state so that it does not unravel on the face line. This is accomplished by sandwiching the immediate roof between the support canopy and the upper strata, thereby maintaining bedding planes in a clamped state to resist horizontal displacement and dilation as the strata subsides onto the goaf pile. This function requires the support to have the capacity to transfer horizontal thrust to the floor, which is achieved through the lemniscate linkages.

9.3.3 Static and Kinematic Characteristics

The performance of a powered support is dependent on its static and kinematic characteristics. Two conditions must be satisfied for a powered support to be in a state of equilibrium, namely, the algebraic sum of all forces acting on it must be zero, and the algebraic sum of all moments about any point must be zero. Other parameters of particular importance to shield performance are:

- total roof support resistance of the support;
- support resistance of each load bearing member;
- stiffness of the support;
- canopy ratio (or canopy balance, being as discussed later, the ratio of canopy face tip to leg distance to canopy rear end to leg distance);
- capacity to vary canopy attitude;
- immediate roof and floor bearing pressure and capacity; and
- the kinematic properties of the support for adapting to various roof geometries.

The computation of load acting on a longwall powered support is complex and statically indeterminate. It is a function of the stiffness of the powered support and the stiffness of the surrounding strata, both of which can vary during the mining process and be time dependent. There are numerous permutations in the factors that determine the system stiffness. These include:

- depth of mining;
- mining height;
- composition, thickness and caveability of the immediate roof strata;
- composition, thickness and caveability of the upper roof strata;
- relative location and thickness of particularly weak, strong or extrusive strata;
- strength of the floor strata;
- joint direction, dip and density;
- density of mining-induced fracturing;
- configuration of the powered support;
- stiffness of the powered support; and,
- setting and yield pressure of the powered support.

Hence, no geotechnical model finds universal application and each site has to be assessed in its own right using tools such as surface to seam displacement instrumentation; microseismic monitoring; powered support pressure and convergence monitoring; surface subsidence monitoring; numerical modelling; and observation

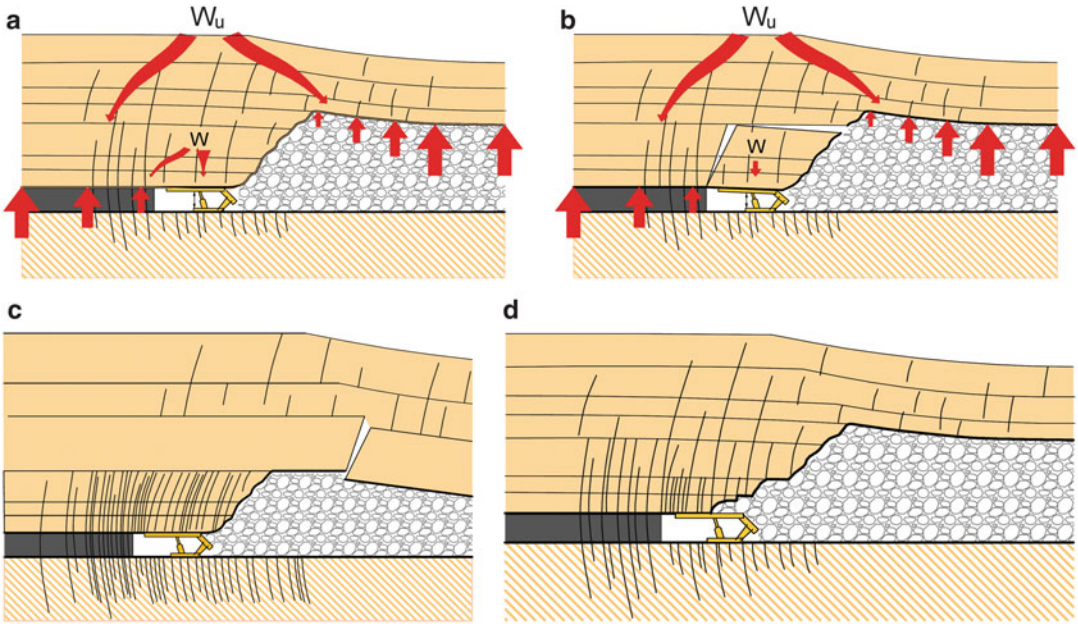


Fig. 9.16 Conceptual loading models for longwall powered supports. (**a**) Bulking model, (**b**) Detached block model, (**c**) Periodic weighting model, (**d**) Unconfined model

and deduction. The four models depicted in Fig. 9.16 provide a basis for conceptualising powered support statics across the range of conditions most commonly encountered.

There is no unique definition of **total roof support resistance**, also referred to as **total support density**, but it is most commonly defined as the ratio of the total normal thrust applied by the support to the roof, to the area of roof supported by each powered support unit. This area is measured from the coal face to the edge of the last supporting element on the goaf side of the face. The total thrust is based on the sum of the nominal yield loads of all the hydraulic support elements in the system. The support resistance is a minimum immediately after the passage of the shearer and a maximum once the support has been advanced.

Early developments in determining total support resistance were based around British views that support resistance need only be sufficient to prevent excessive convergence and European views that it had to be as high as possible to prevent bed-separation over the face area. Many of the European views related support resistance to mining height, reasoning that the greater the mining height, the greater the caving height and, therefore the thicker the strata bed resting on the longwall face supports. At the time, minimum support resistances on installed faces ranged from 100 kN/m^2 to 1.2 MN/m^2 (10–120 t/m^2), the latter associated with strong massive roof strata situations. With the benefit of hindsight, it appears that the difference between the two philosophies was simply a reflection that typical British strata behaved in a more plastic manner and, therefore, was more tolerant of convergence than the more massive and brittle strata associated with European conditions.

Since the early 1990s, it has become common practice in weak to moderately strong roof strata in Australia and the USA to operate powered supports at a set pressure of 0.6–0.8 MN/m^2 (60–80 t/m^2) and a yield pressure of 1–1.1 MN/m^2 (100–110 t/m^2). Some operations use a higher set pressure of 90 % of yield pressure. These appear to be limiting values when the contact strength of the roof strata is taken into account and to be supported by numerical modelling outcomes. Gale (2009), for example,

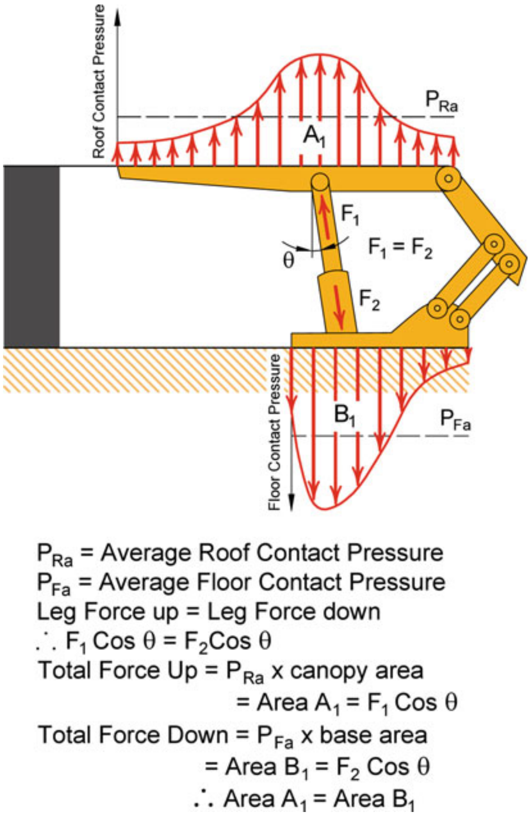


Fig. 9.17 Idealised distribution of roof and floor contact pressure about a powered support

has concluded on the basis of numerical modelling that support resistances over 1.2 MN/m^2 (120 t/m^2) would be considered excessive and not required in weak environments. In stronger and more massive roof strata, higher values for support resistance, setting and yield pressure prevail. As at 2010, the highest capacity shield supports in the world had a total support resistance of 1.6 MN/m^2 (160 t/m^2) (Winter et al. 2010). These were employed at Moranbah North Mine, Australia, beneath a weak immediate roof overlain by a strong massive roof prone to periodic weighting.

It is important to appreciate that the total support resistance is not uniformly distributed over the roof and the values quoted earlier and those provided in manufacturer's specification sheets are averaged over the full roof area. In the ideal case of the powered support being sandwiched between two rigid plates, the maximum support resistance is generated at the end points of the hydraulic legs, as illustrated in Fig. 9.17. Load transfer to the roof reduces with distance along the canopy from the legs. A similar load transfer profile exists in the floor. This situation approximates to that associated with the bulking model depicted in Fig. 9.16a.

The bulking model can be conceptualised as a displacement controlled system, with irresistible strata convergence of the upper roof strata loading the coal face, the powered supports and the goaf. The powered supports represent very soft springs located between stiff springs, being the adjacent goaf, and very stiff springs, being the coal face. The stiffness of the legs of the powered supports and their setting load determine the overall stiffness of the powered support and, therefore, the amount of convergence that can occur prior to the supports reaching yield. In this setting, powered supports only have the capacity to influence strata behaviour in their immediate vicinity. The concept of 'bigger is better', in terms of support resistance, does not necessarily deliver improved face control. Rather, the more critical controlling factors may be the point of application of support resistance; load distribution within the canopy and the base; support stiffness; the integrity of the immediate roof to function as a fractured but interlocked beam or cantilever; and roof and floor contact strengths.

The behaviour of the bulking model is changed significantly if a face break occurs, resulting in a detached block above the powered support (Fig. 9.16b). The detached block causes the system to revert from being displacement controlled to being load controlled. Ashwin et al. (1970), Whittaker (1974) and Wilson (1975) proposed similar simple analytical models for determining the distribution of forces and moments for this situation. While there are a number of limitations associated with these models (see for example, Smart et al. 1982; Aziz and Porter 1985) which have resulted in modifications by Smart and Redfern (1986), Barczak and Tadolini (2007), and others, they

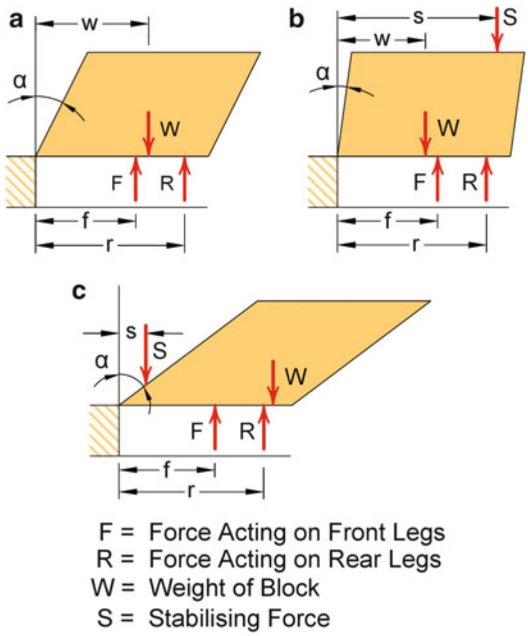


Fig. 9.18 Detached block model geometries for a 4 leg rigid canopy chock shield powered support

still give valuable insight into the basic behaviour of powered supports and provide a reasonably accurate analysis of one extreme condition.

The detached block models assume that the legs of the powered support are rigid and that the support is uniformly loaded by the dead weight of the detached block of strata of mass, W. The block can be of any size and shape, although some models define its geometry on the basis of caving height, caving angle and overhang distance into the goaf. Geometry determines where the centre of gravity, or centroid, of the detached block acts on the powered support. This may be on the face side of the legs, the goaf side of the legs, directly over the legs of a shield support, or between the front and back legs of a chock shield. In order to prevent rotation, a fictitious balancing or stabilising force, S, has to be introduced to mimic the resistance to rotation provided by the roof strata (Fig. 9.18).

For the case where the centre of gravity acts between the face and the front legs of a rigid canopy chock shield (Fig. 9.18a), the maximum weight, W, of loosened strata that can be supported is found by taking moments about the rear of the support and is given by Eq. 9.2.

$$W = F\left(\frac{s-f}{s-w}\right) \tag{9.2}$$

where

F = normal component of combined capacity of front legs
B = normal component of combined capacity of rear legs
s, w, f and r = lever arm distances

Assuming that the yield ratings of the front and back legs of the chock shield are equal, it follows from Equation 9.2 that because (s-w) is greater than (s-f), the actual support resistance is less than one half of the nominal support resistance. A similar static analysis can be performed for 2 leg shields and for 4 leg chock shields when the centre of gravity of the load acts between the two sets of legs or behind the rear set of legs. The periodic weighting model (Fig. 9.16c) represents an extreme case of the detached model in which the centre of gravity of the load acting on the support is some distance back into the goaf.

The model demonstrates that the total thrust of a support system is only ever equal to the sum of the nominal thrust of the system components when the centre of gravity of the load acts directly over the legs on a 2 leg shield support or at the mid-point between the front and back legs on a four leg chock shield. Longwall support manufacturers utilise the detached block model to compute and specify the tip and rear load capacities of powered supports, examples of which are presented in Table 9.1. This table shows that when the centre of gravity of the supported load acts at the tip, the actual load carrying capacity of the powered support is of the order of only 25 % of the its nominal support capacity.

The detached model highlights the importance of considering not only total support

Table 9.1 A selection of manufacturer's specifications for longwall powered supports

Support type	Total leg support capacity (MN)	Pre-cut support resistance (MN/m^2)	Maximum support capacity when centre of gravity acts at: Tip (MN)	Rear (MN)	Average roof bearing pressure at yield (MN/m^2)	Average floor bearing pressure at yield (MN/m^2)
4 leg chock shield	8.0	0.77	1.91	6.07	1.17	3.13
4 leg chock shield	9.0	0.87	2.13	6.83	1.26	2.35
2 leg shield	9.8	1.05	2.56	7.29	1.30	3.59
2 leg shield	12	1.30	3.49	8.56	1.45	2.68
2 leg shield	17.48	1.50	5.06	12.42	1.60	3.20

resistance when selecting powered supports but also the location and distribution of turning moments that may be generated within the support. However, the model has limitations, as becomes evident when it is applied to high capacity shield supports. The model cannot cause the supports to yield under any realistic detached block configuration other than one which cantilevers at least 10–15 m into the goaf, such as encountered in some periodic weighting situations. The model is unable to account in its own right for other situations in which shield supports yield. This partially reflects the fact that powered supports do not have the capacity to resist all mining-induced convergence, with the level of convergence required to cause yield decreasing with increase in powered support stiffness associated with higher set pressures and stiffer hydraulic legs.

The unconfined model represents the situation where the caving line progresses over the top of a powered support (Fig. 9.16d). This is more likely to occur at larger mining heights in weak strata environments. It results in a relaxation in lateral confining stress at the face, allowing the fractured strata between the tip of the support and the face to unravel. Factors which aggravate the situation include the presence in the immediate roof of low friction bands and bands prone to extrude under load; an irregular roof cutting profile; the presence of a cavity associated with a previous face fall; sloppy lemniscate linkages; and inadequate support resistance.

The progression of the caving line towards the face increases the turning moments at the tip of the support because it simultaneously removes counter balance from the rear of the canopy and moves the centre of gravity of the load towards the tip. Once the cave line reaches the front legs, the canopy is free to rotate about these legs, allowing the tip to drop into the working place and reducing tip capacity to zero. Face falls are inevitable without intervention to fill voids and reconsolidate the fractured strata.

The load distribution profile, maximum tip capacity, and maximum rear capacity of a shield support are very sensitive to the **canopy ratio**, or **canopy balance**, defined by Eq. 9.3 as:

$$\begin{aligned} &Canopy\ Ratio,\ or\ \ Canopy\ Balance \\ &= \frac{Distance\ from\ tip\ to\ legs}{Distance\ from\ legs\ to\ rear} \end{aligned} \tag{9.3}$$

A misconception sometimes associated with a shield support is that angling of the legs towards the face introduces a horizontal component of stress to confine the immediate roof, with this confinement increasing as the support yields.

This is not the case as the lemniscate linkage causes the support canopy to travel in a straight vertical trajectory (Fig. 9.9).

In addition to maintaining forces and moments in equilibrium, the capacity of a support to control convergence depends on the stiffness of its hydraulic system and on its setting and yield loads. Hydraulic system stiffness is determined primarily by the height and area of the fluid column in the legs, with a component also associated with expansion of the leg tubes and hoses. In accordance with Eq. 2.3, everything else remaining unchanged, the higher the fluid column in the legs, the less pressure (or support resistance) developed per unit of convergence. The setting load corresponds to a prestress applied to resist convergence, while the yield load determines the peak resistance to convergence. Although longwall mining height has increased substantially and now approaches 6 m, the corresponding reduction in leg stiffness has been offset to some degree by the larger bore diameter of the hydraulic legs associated with modern thick seam supports. In the case of double telescopic legs, the load generated by the support is determined by the cross-sectional area of the smallest cylinder in the telescopic leg.

Care has to be exercised in relying on some stiffness values and concepts for longwall supports presented in the literature as there is a mix of definitions of stiffness, some computations are flawed, and some concepts are confused. Typically, a load increment of 1 MN (100 t) with its centre of gravity acting in the thrust line of the legs of a modern 2 leg shield support extended to 3 m will result in 5–7 mm of convergence up to the yield point of the support, corresponding to a support stiffness of 0.14–0.2 MN/mm.

However, a lower load is required to produce the same convergence if the centre of gravity of the load acts in front of or behind the thrust line of the legs, or if the support operates at a greater height. If the effective area supported by a 2 leg shield is approximated to be 10 m^2, then a 1 MN (100 t) load increment acting over the same shape and size area on a 3 m high coal face would result in around only 0.6 mm of convergence. Hence, the effective stiffness of a powered support is an order of magnitude less than that of the coal that it replaces, meaning that even in the most favourable circumstances, a powered support only makes a small contribution to controlling the overall stress and convergence distribution about a longwall face.

If debris accumulates over or under a powered support, it acts as a soft inclusion and can negate the benefit of high leg stiffness to control convergence. Good housekeeping to minimise the accumulation of this material, high setting pressures, and maintenance of setting pressures to compact the material are important in minimising convergence. Skimming the roof with the canopy of a powered support as it is advanced also assists in minimising debris on top of the canopy.

In specifying the support resistance for a longwall powered support, careful consideration needs to be given to the contact strength and bearing capacity of the immediate roof and floor strata and to the loading profile of the support canopy and base. Contact pressures are higher at the floor than at the roof due to the smaller load bearing area of the support base and the effect of turning moments (Fig. 9.17). The combination of leg configuration and high tip load capacity of a shield support can generate high turning moments and, therefore, concentrate loadings at the toe of these types of supports. Hence, the bearing capacity of the floor is an important consideration when designing a powered support for a specific site or assessing if a powered support is suitable to a different site. It has a significant influence on powered support design in respect of:

- the overall geometry of the powered support and AFC so that base loading profiles do not exceed the bearing capacity of the floor;
- the type of base fitted to the support (solid or split); and
- the fabrication of the base to tolerate bending and torsion over its planned operating life.

A range of approaches can be adopted to assessing the bearing capacity of the floor, with the most common being the application of bearing capacity formulae of the type presented in Appendix 4 and numerical modelling. Solid bases to maximise load carrying area and base lifting rams to raise the front (toes) of the powered supports when advancing them are two controls utilised in weak floor strata to mitigate against bearing capacity failure. If the bearing capacity of the floor is exceeded, powered supports start to rotate towards the face, resulting in the unloading of the canopy at its tip. In these situations, shield supports are prone to topple towards the face, especially once mining height exceeds about 3 m. On the other hand, uneven, hard floor conditions can subject the base of a support to excessive bending and torsion, leading to the failure of welds. Split base support systems offer some advantages in these conditions because torsional forces on the base are greatly reduced.

The computation of load and turning moments acting on a longwall support is complicated because in addition to being statically indeterminate, it is also time dependent. Medhurst (2005) proposed that the ground response curve concept provided a convenient means to graphically show ground behaviour, its relationship to powered support performance, and roof stability. The basis of this approach is shown in Fig. 9.19. It is premised on roof behaviour being convergence controlled, with a unique ground response curve applying to each combination of mining conditions (geology, depth, geometry etc.).

In practice, considerable uncertainty is associated with the calculation of a ground response curve for a longwall face environment because of the numerous complex permutations of strata behaviour about a longwall face, their time dependency, and a lack of data over the full range of a ground response curve. Medhurst (2005) proposed that a strata-support interaction relationship of the type defined by the curve AD in Fig. 9.19 could be derived by considering:

- routine geotechnical data;
- leg convergence/stiffness test results;

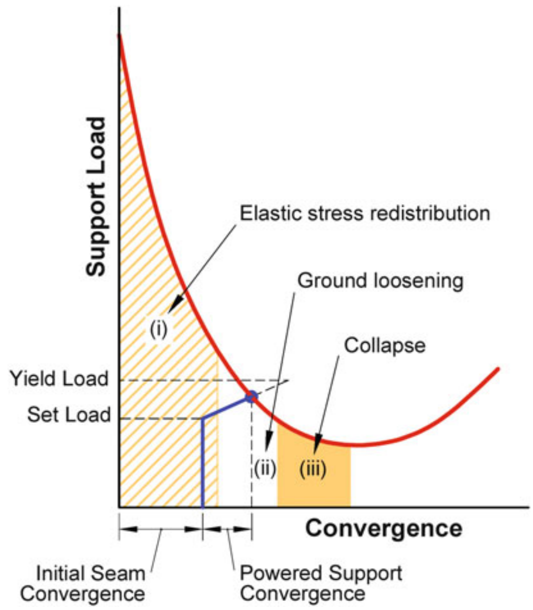


Fig. 9.19 Ground support interaction curve applied to a longwall face (Adapted from Medhurst (2005) and Gale (2009))

- monitoring data relating to leg pressures, surface subsidence, surface to seam extensometers and microseismics; and
- underground observations.

A limitation with this approach, as recognised by Barczak (2006) and others, is that the mine loading conditions are not sufficiently comprehensive and controlled to develop a full curve. Numerical modelling can assist but it is still constrained by the variable and complex behaviour modes and failure states of strata that fall within the zone of influence of a powered support. Gale (2009) utilised the ground response curve concept in a general form to define three stages in the ground response to longwall mining, shown imprinted on Fig. 9.19. These stages are:

- Stage (i) – An elastic “intact” mass whereby the amount of support to resist ground movement is well outside the capability of any face support.
- Stage (ii) – A fractured interlocked mass that has some remaining confined strength but is

typically still outside the capacity of face supports.

- Stage (iii) – Fractured ground which starts to lose its interlocking nature, resulting in a reduction in confinement leading to unravelling and falls of ground.

The role of powered supports in this scenario is envisaged as maintaining the remaining ground strength in Stage (ii) and stopping the transition to Stage (iii). Typically, setting load is intended to provide adequate control against progression to Stage (iii).

As the fractured rock mass unravels, the situation progressively changes to a load controlled system. The stiffness of the powered supports then becomes variable, depending on where the centre of gravity of the load acts on the support. At that point, the application of a strata-support interaction curve of the type shown in Fig. 9.19 becomes problematic.

Irrespective of its static characteristics and capabilities, a longwall powered support system has limited practical value if the geometric proportions of the system and its kinematic characteristics limit contact of the canopy with the roof. It is important that the support is in good contact with the roof and that the span between the coal face and the area of application of the main thrust of the support system is small. The introduction of shearer technology that senses and remembers mining profiles along the face has aided in reducing the frequency of large vertical steps in the floor and roof caused by loss of horizon control. However, poor roof contact conditions can still occur in the presence of geological disturbances, excessive loose material on the canopy of a powered support, and roof cavities.

Rigid canopies limit the options for maximising canopy contact area when the roof profile is irregular and for applying support where it may be most needed. The two leg configuration of a shield support in association with the compensating, or canopy tilt, cylinder connecting the canopy to the flushing shield provides some potential to optimise canopy orientation and, therefore, contact area. However, this canopy orientation may not be maintained under load. It must be remembered that the effective tip distance is that distance from the face back to where the immediate roof strata comes into contact with the canopy of the powered support. Some support designs promote tip contact by curving the canopy tip upwards.

9.4 Operational Variables

In addition to equipment selection and mine design, there are a range of operational variables that are important for managing ground control about a longwall face. The timely and effective use of these is vulnerable to the vagaries of human performance. Therefore, they need to be underpinned by a robust Trigger Action Response Plan (TARP).

9.4.1 Cutting Technique and Support Configuration

There are three basic techniques for cutting coal from a longwall face, namely, **bidirectional** (**bi-di**), **unidirectional** (**uni-di**) and **half web**, and a range of permutations within each.

In bi-di mode, the face is cut from both directions to its full height and width (one web) with each pass of the shearer. This enables the AFC to be advanced immediately behind the shearer. The AFC has a limited degree of articulation and so is advanced incrementally over a distance of 15–20 powered supports, with this transition section being referred as the **snake**. It also enables the face to be double chocked (closed up) in poor ground conditions immediately after each pass of the shearer. The potential disadvantages of this cutting technique are loss of floor horizon control because it is not easy to see and monitor this horizon when cutting; poor floor cleanup leading to debris ingress under the powered supports; and extended time for the shearer to double shuffle at each end of the longwall face in order to cut out the bottom section of the face right up to the gate end.

Uni-di cutting entails mining the top section of the face from one direction and the bottom section from the other direction. This removes the need for the shearer to double shuffle at the gate ends, thereby providing for faster turn-around times, and minimises the need for operators to work in dust on the return airway side of the shearer. Ground control benefits are associated with improved horizon control and a cleaner floor. More uniform coal loading and increased cutting speed can result in cycle times that approach or exceed that of bi-di cycle cutting on longwall faces shorter than around 250 m.

Historically, the main ground control disadvantage with uni-di cutting was related to not being able to advance the powered supports until after the shearer had taken the bottom pass. The advent of powered supports that can be operated in one web back mode while still generating a high tip load and be advanced immediately after the passage of the shearer has removed this disadvantage, other than when ground conditions are so poor that the face needs to be closed up and double chocked.

Half web cutting modes involve variations on undercutting the face in uni-di mode at mid height over the middle sector of the longwall face and cutting the gate end sectors in bi-di mode using half web advances. Improvements in cycle times can translate to improved ground control. However, in weak coal, the undercut is prone to fall and to increase the tip-to-face distance.

9.4.2 Powered Support System Maintenance

Maintenance of the powered support system is critical to ground control on a longwall face. Matters of particular importance are:

- Condition of the hydraulic legs. The total support resistance of powered supports on a longwall face reduces in direct proportion to the number of non-functional hydraulic legs on the face. It is not uncommon for major longwall face falls to have been associated with leg fault rates exceeding 20 % (e.g. Galvin 1997b). Trueman et al. (2008) report that up to 10 % of shield legs had faults on a typical Australian longwall face. This is sufficient to adversely affect strata stability along the full length of the face. Excessive convergence, guttering and cavities can also develop on a localised scale due to load transfer from an under-performing support to its adjacent supports. The move from 4 leg chock shields to 2 leg shields has had the benefit of minimising the number of legs that have to be maintained on a longwall face. However, support performance is now more sensitive to an underperforming leg.
- Valve maintenance. Valves control a number of functions crucial to ground control on a longwall face including set pressure, yield pressure, activation of leg stages, activation of adjacent supports, positive set and positive set reactivation. Over time, they can become clogged and scoured, resulting in them operating at lower pressures than design. On a number of occasions, the poor state of valving has only become apparent after a rapid loading event when upwards of 100 or more yield valves designed to control such events have failed.
- Pressures and volumes. In theory, the hydraulic reticulation system should be capable of supplying sufficient volumes of fluid at sufficient pressure to all areas of the face. In practice, however, fluctuations in line pressure occur at times of peak demand. The midpoint of the face, where it is most critical that powered supports operate at design pressure, is the most vulnerable to insufficient supply pressure in some installations. In others, it is the tailgate third of the longwall face. Pumping rates need to be sufficient to keep up with setting times and powered support advance rates (determined by shearer cutting speed). Positive set reactivation is important to correct situations where legs may not have reached set pressure due to peak demands on the supply system. This should not be

tolerated on more than a sporadic basis as uneven set pressure distributions can induce roof instability.

- Lemniscate linkages. Lateral thrust on powered supports can cause the lemniscate linkages to become sloppy through racking and wear, allowing significant horizontal movement prior to the canopy bedding into the roof. This movement reduces and or removes lateral confinement of the immediate roof strata, increasing the potential for this strata to unravel in the tip-to-face region.
- Structural integrity. Powered support components can be subjected to eccentric loadings, concentrated loadings, point loadings, impulse loadings, cyclic loadings, and corrosive environments, all of which are conducive to deformation, wear, and fatigue failure at critical load bearing points in the structure. Often, these points may not be visible or accessible until the longwall face is salvaged. In any case, when structural failures are detected during operation, they cannot usually be remedied on the longwall face. Hence, design, fabrication techniques, inspections and maintenance of powered supports are also fundamentally important to effective strata control on a longwall face.

For reasons of both safety and productivity, it is advisable that an engineering maintenance scheme which addresses these types of issues is an integral element of the overall mine management scheme.

9.4.3 Face Operating Practices

Ground control on a longwall face can also be influenced significantly by operating practices and operating discipline. The following are particularly important and warrant carefull consideration when preparing a Face Management Trigger Action Response Plan:

- Rate of retreat. It is long established from total extraction mining operations that the strength of highly loaded rock, particularly sedimentary rock, can decrease over time and, therefore, the speed of extraction is a critical parameter, especially during periodic weighting events and when negotiating structurally disturbed ground. Table 9.2 summarises convergence limits and rates suggested by Medhurst (2005) as being typical for most Australian longwall mining operations.

Table 9.2 Summary of powered support convergence limits and rates proposed by Medhurst (2005)

Event	Convergence
Initiation of face spall	15–20 mm
Cavity development	>30–50 mm
Overlying strata broken	>100 mm
Heavily weighted environments	10 mm/h
Periodic weighting cycle	>20 mm/h

Based on these figures and a consideration of the extent of fracturing ahead of a longwall face, Medhurst (2005) concluded that a minimum retreat rate of 5 m/day should be maintained when mining at a height of 2–3 m, increasing to 10 m/day when operating in thicker weak coal seams.

- Face alignment. Maintaining a straight face alignment has long been considered important for preventing the formation of local stress raisers on the longwall face. It has a secondary strata control benefit in that it reduces the likelihood of a face stoppage due to damage to the AFC. However, some operators of faces over 250 m long report a benefit in advancing the middle third of a longwall face in periodic weighting situations. This may be related to the trajectory of mining-induced fracturing along the face.
- Horizon control. Steps in the roof and floor associated with poor horizon control can present obstructions to advancing the AFC and powered supports and prevent the support canopies from making full contact with the roof. Loss of contact with the roof effectively equates to an increase in the tip-to-face distance. Roof steps can give rise to point loads that exceed the contact strength of the roof. In

hard floor environments, floor steps can generate point loads and flexing that are of sufficient magnitude to result in structural damage to the support bases. Automatic horizon control on the shearer is an aid in managing this risk but does not eliminate it.

- Powered support advance. Support advance must not be permitted to lag behind the shearer. Automatic initiation of support advance by the shearer is a valuable control, provided that sufficient hydraulic volume and pressure are available. Programmable control circuits constitute another control, enabling powered supports to be advanced individually or in 'banks' or 'blocks' that typically comprise between two and five supports. Advances in coal cutting and clearance technologies have resulted in a significant increase in shearer speed, to the point where it is difficult to keep up with the shearer when advancing the powered supports individually. Block advance, or bank push, assists in addressing this problem but it has the disadvantage of not enabling the roof to be supported immediately upon exposure. Hence, in poor ground conditions it is advisable to slow the shearer down if necessary to enable the powered supports to be advanced on an individual basis immediately behind the shearer.

  If the immediate roof is already in a fractured state or contains the lip of a cavity that needs to be 'caught', there can be benefits in maintaining some load on the roof as the powered support is advanced. This operating procedure is referred to as **contact advance**. It can increase the time taken to advance each support and, therefore, may also require a reduction in the speed of the shearer to enable freshly exposed roof to be supported immediately.
- Setting and maintaining leg pressure. Powered supports need to be reset to the correct setting pressure after being advanced and not be permitted to drop below this pressure during a cutting cycle. Positive set and positive set reactivation are of assistance in this regard, aided by having a separate hydraulic circuit for set reactivation. Some operations employ a second 'high-set' hydraulic circuit in any case in order to increase initial set pressure to an intermediate value between nominal set and yield.
- Debris. Compaction of loose material over the top of or beneath a powered support results in additional convergence and, therefore, a reduction in support stiffness. Debris on top of the canopy can also generate point loads and reduce the area of roof that is actively supported. Debris on the floor may cause the powered supports and AFC to ride up on the loose floor material, leading to a loss of horizon control. Positive set reactivation is a control for managing these types of situations. However, a more effective control is to eliminate the debris by means such as contact advance, cutting to a different horizon, improving the dozing capability of the AFC, and clearing loose material from the floor.
- Negotiating weak roof and cavities. Risk management procedures, preferably encapsulated in Trigger Action Response Plans, should contain provisions for reverting to double chocking, conventional mode, bi-di shearing, and/or reducing mining height in a timely manner when ground conditions deteriorate. When negotiating cavities, it may be necessary to turn off the positive set system in order to maintain the attitude of the canopy. Operators need to be aware that this can result in poor set pressures across the face and no compensation for pressure loss due to leaks in the hydraulic circuitry.

Operating discipline is particularly important when it comes to stopping the face in order to install secondary support such as rock bolts, long tendons, spiles, strata binders and void fillers. Experience attests to the risk associated with continuing to mine in an attempt to 'catch the lip', rather than stopping and taking remedial action (Galvin 1996; Payne 2008). Face falls associated with attempting to outrun a situation are often vertically and laterally extensive in nature, which not only makes their recovery more time consuming but may also expose those working on the recovery operation to greater risk of injury.

- Real time monitoring of longwall leg pressure trends and yielding behaviour offers significant potential benefits in this regard because it assists in quantifying the state of face stability and provides an immediate and objective basis for risk management decision making. Hoyer (2011) and Wiklund et al. (2011) describe applications of a software package utilised for providing early warning of the development of periodic weighting and roof cavities on the basis of a leg pressure algorithm. Such algorithms can be based around average support pressures, support loading rates (pressure increase/unit time), and yield frequency per cutting cycle.
- Extended downtime. When a longwall face is to be idle for an extended period, typically more than a shift, standard work procedures should be available that detail the requirements for setting flippers and closing up the powered supports. These should be encapsulated in a Face Management Trigger Action Response Plan, which also constitutes a control for these situations.

9.5 Longwall Face Strata Control

9.5.1 Introduction

In addition to face operating practices (Sect. 9.4.3), strata control on a longwall face is a function of a range of other interactive factors that include:

- lithology and sedimentology;
- pore pressure;
- mining height;
- panel span;
- panel depth;
- interpanel pillar width;
- the static and kinematic characteristics of the powered supports;
- engineering maintenance standards; and
- the presence and nature of workings in adjacent seams.

Many of the seminal concepts of strata behaviour around a longwall face, such as those developed by Potts (1957), Salamon et al. (1972), Wagner and Steijn (1979) and Galvin et al. (1982) were based on surface to seam extensometers; surface subsidence measurements; monitoring of leg pressures and convergence on longwall faces; and observations of goaf behaviour. Subsequently, these concepts have been developed and enhanced by Kelly and Gale (1999), Gale (2004), Gale (2009) and others utilising advances in microseismic monitoring, computational techniques and stress measurement to give more detailed insight into the location and nature of rock fracturing about a longwall face.

9.5.2 Coal Face

The stability of the coal face is particularly sensitive to the direction, dip and density of cleats, joints and mining-induced fractures; mining height; abutment stress magnitude; and rate of mining. Cleats and joints provide pre-existing failure surfaces for face spall; delineate coal slabs and columns that are conducive to bending and buckling failure under load; and create the potential for slabs to topple onto face equipment and into the work area. While orientating the longwall face line parallel to the natural cleat and jointing direction is sometimes suggested and utilised as a control for inducing massive roof strata to cave, experience confirms that this can result in an unsafe local mining environment. It increases the risk of rib spall on the longwall face and in gateroad cut-throughs and, if a conjugate cleat or joint set is present, in the gateroad headings. It also increases the potential for face breaks and for large blocks to fall out of the roof in front of the longwall supports. Hence, for safety and operational reasons, it is generally preferable to orientate drivages at an angle of at least 20° to natural cleat and joint systems, albeit that this may aggravate spalling of pillar corners.

As mining height increases, the stiffness of both the coal face and the powered supports is reduced, resulting in increased roof convergence. The stability of the powered supports may also be reduced. There is an increased potential for face spall due to a reduction in coal strength, and for this spall to extend to a greater depth into the coal face, resulting in a substantial increase in tip-to-face distance and, hence, unsupported roof span. The dip of geological features takes on added significance because it controls the sizes of blocks that may spall from the face. Operators and equipment are exposed to higher levels of gravitational energy from face spall as mining height and block size increase. All of these impacts are magnified with increase in abutment stress. Such increases may be associated with increased depth of mining, cyclic caving or interaction with workings in the same or adjacent coal seams.

Because the depth of spall is almost invariably less at the bottom of the coal face than the top, the AFC and powered supports can be prevented from being advanced to support the increased area of exposed roof until the toe of the face has been mined. The associated time delay and potential for this operation to initiate additional face spall can aggravate the situation. Control options for safely and effectively managing these circumstances include:

- maintaining a straight face line so as to avoid localised stress concentrations (noting that some operators have reported benefits with curved faces in periodic weighting situations);
- mining to the correct horizon;
- incorporating 'double knuckle' flippers into powered supports to function as face sprags with an extended reach;
- closing up the face (double chocking) immediately after passage of the shearer;
- limiting abutment stress magnitude by the judicious selection of panel orientation and geometry, particularly panel width, W;
- stopping to support and consolidate the face and immediate roof before the tip-to-face distance becomes excessive (which requires operating discipline, fit-for-purpose equipment that is on-hand, and robust safe working procedures);
- incorporating facilities in thick seam powered supports for accessing the roof line to undertake consolidation and secondary support; and
- in all cases, safe work procedures that prevent operator exposure to rib spall and roof falls on a longwall face.

9.5.3 Floor

Abutment stress induces fracturing of the floor ahead of the face, with fractures traversing bedding and dipping back under the goaf and also running along bedding planes. Abutment stress impacts are more likely and greater when the floor strata is soft or weak or contains bands that are prone to extrude under load or to swell or disintegrate in the presence of moisture, leading to bearing capacity failure. High toe loadings on powered supports can also induce bearing capacity failure of the floor. Associated floor heave can obstruct the advance of the supports and, in the extreme case, result in supports rotating to an extent that they become unstable and topple towards the face. Floor heave can also have a serious impact on the operation of the AFC and shearer, causing the AFC to rise, relay bars to bend, and the shearer to topple towards the longwall supports. If the shearer is still able to traverse the face, horizon control may be lost. Fracturing of the floor strata can also significantly increase the potential for release of gas into the workplace from deeper seams.

Control options for safely and effectively managing these situations include:

- Optimising the design of powered supports to avoid high toe pressures. Options include solid bases and varying the canopy ratio. Floor pressure profiles may be the defining factor in determining powered support capacity.
- Incorporating base lifting rams in the powered supports.
- Leaving bottom coal to protect the floor.
- Limiting abutment stress magnitudes, by the judicious selection of panel orientation and geometry, particularly panel width, W.
- Maintaining a relatively rapid rate of face retreat.
- Limiting ingress of water into the face area by mining up dip and utilising efficient water management systems.
- Limiting influx of gas by pre-draining the mining seam and adjacent seams.
- Diluting gas make by utilising an effective ventilation system.

9.5.4 Immediate and Upper Roof Strata

There are no unique definitions of what constitutes immediate and upper roof strata, which may be comprised of numerous combinations of strata type, thickness and properties. However, when discussing strata response in longwall mining, it is convenient to consider the immediate roof strata as comprising the strata that constitutes the caving zone and to classify the strength of the immediate and upper roof strata as either ‘weak to moderate’ or ‘moderate to strong’.

Causes of roof cavities on a longwall face include:

- geological features;
- excessive tip-to-face distance;
- inadequate setting pressure;
- poor setting geometry, especially supports set with their tip down;
- cleat parallel to face;
- loss of horizon control;
- periodic weighting; and
- loose material above chock canopies.

9.5.4.1 Weak to Moderate Strength Roof Strata

Field observations and studies supported by numerical simulations confirm that mining-induced stress can cause fracturing of weak and/or laminated strata well ahead of the longwall face, with fracture network intensity increasing towards the face and resulting in the strata caving readily immediately behind the longwall supports. This is confirmed by the outcomes of seismic monitoring, shown in Fig. 3.16d, that was undertaken by Hatherly et al. (1995) and Kelly and Gale (1999) in a weak roof and floor environment about a longwall panel. Gale (2004) utilised a two-dimensional FLAC model to simulate the behaviour of strata and fluid pressure along a longitudinal plane running down the centre of this longwall panel. The derived fracture network is shown in Fig. 9.20. It was concluded from these combined studies that:

- fracturing of the roof and floor strata occurs well ahead of the mining face and is not related to the caving process behind the supports;
- the dominant initiating failure modes in weak strata ahead of the face are shear fracture of rock mass and shear along bedding (Fig. 9.20);
- the extent of bedding plane shear ahead of the longwall face is variable but typically extends over large distances, often in excess of 100 m;
- in general, fracture size is variable, however, shear fractures tend to be limited to less than a couple of metres and form in an incremental manner rather than in one large event;
- tensile initiated fractures may develop ahead of the face in response to bending moments but these fractures are generally confined to stronger upper strata;

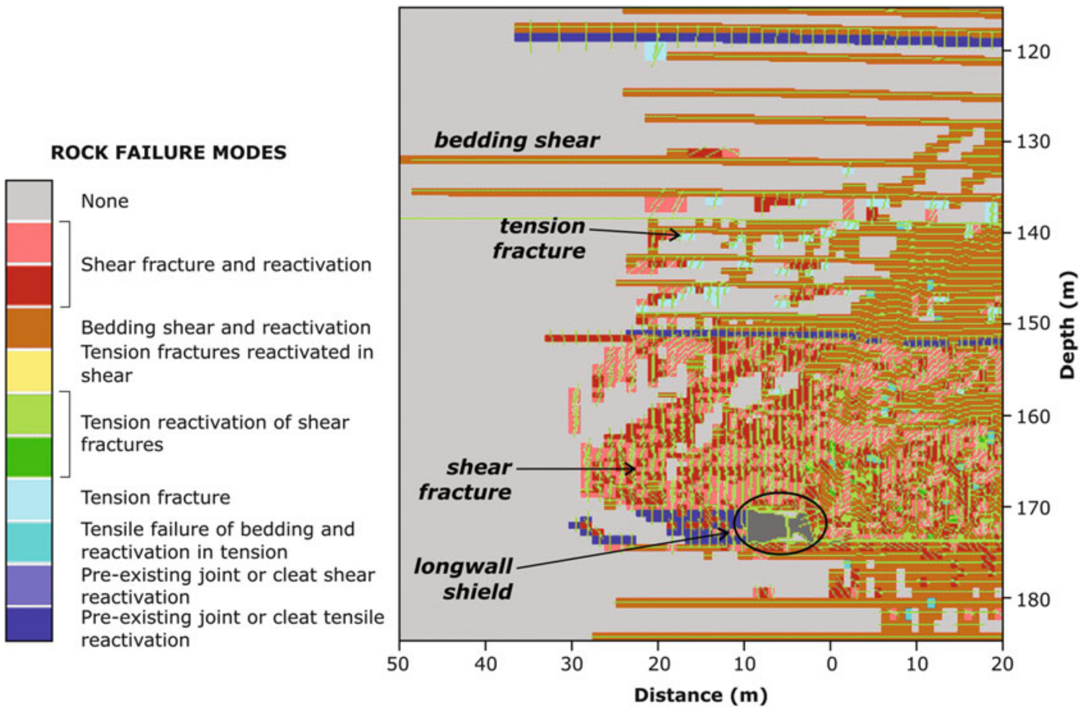


Fig. 9.20 Rock failure modes and fracture orientations in weak to moderate strength roof as predicted by two-dimensional numerical modelling (After Gale 2004)

- few new fractures are generated during the caving process; and
- the high frequency of fracturing prevents the accumulation of large amounts of stored strain energy, as reflected by the low seismic magnitude of fracture events (typically less than −1 on the Richter scale).

The various monitoring and analyses highlight that good forward roof support is critical in weak strata conditions. In subsequent analysis of the performance of a 2 leg shield in weak to moderate roof sections, Gale (2009) concluded that:

- The yield capacity to control the caving line and provide confinement to the fractured material is recommended to be in the 1–1.1 MN/m^2 (100–110 t/m^2) range, with a set of approximately 0.8 MN/m^2 (80 t/m^2).
- The canopy balance is recommended to be less than 2.4 and, preferably, less than 2. In most instances this relates to a tip to leg distance of 3–3.5 m, corresponding to the reaction point of the legs being less than 0.7, and preferably less than 0.6, of the canopy length back from the face.
- The tip-to-face distance to maximise roof integrity and limit dilation should be less than approximately 0.6 m. The smaller the distance, the better the result.
- A rigid canopy offers benefits over a hinged cantilevered canopy.

Increased convergence arising from the powered supports going into yield is conducive to roof scaling, slabbing roof, and guttering, resulting in cavities over the top of the powered supports as they are advanced. These cavities can limit the extent to which the canopies come into contact with the newly exposed roof and prevent the supports being set at their specified pressure, thus encouraging the formation of further cavities. Slow rates of retreat also compound this situation.

Control options for safely and effectively managing weak and friable immediate roof situations mirror many of those for managing coal face stability and include:

- restricting tip-to-face distance to a minimum;
- powered supports with a high tip capacity;
- contact advance of powered supports;
- flippers with a tilting capacity to provide immediate forward support to the roof and/or face;
- leaving top coal to prevent slabbing of weak, friable immediate roof strata;
- closing up the face (double chocking);
- limiting abutment stress magnitude by the judicious selection of panel orientation and geometry, particularly panel width, W;
- stopping to support and consolidate the face and immediate roof before the tip-to-face distance becomes excessive; and
- incorporating facilities in thick seam powered supports for accessing the roof line to undertake ground consolidation and secondary support.

9.5.4.2 Moderate to Strong Strata

The presence of stronger strata units in the immediate roof might reasonably be expected to result in improved longwall face conditions. However, should these units or strata higher up in the roof sequence be sufficiently massive to result in cyclic caving, then periodic weighting becomes a concern. These cycles typically occur at intervals of 10–30 m, but may exceed 70 m in some circumstances. Periodic weighting may also develop in overburden sections that have a relatively uniform shear strength sufficient to allow a limited span between the longwall face and the goaf to develop (Gale 2001).

The impact of a massive stratum on the severity and frequency of cyclic loading, face conditions and surface subsidence is a function of the thickness and material properties of the massive stratum; its distance above the mining horizon; its depth below surface; the width of the extraction panel; and face control measures. Generally, the closer a massive unit is to the extraction horizon, the less thick it needs to be to result in periodic weighting. Periodic weighting can be influenced by the behaviour of competent beds up to 70 m or more above the seam (reference, for example, Wagner and Steijn 1979; Mills and O'Grady 1998; Trueman et al. 2008; Wiklund et al. 2011).

Periodic weighting gives rise to zones of intense fracturing in the coal face and immediate roof and floor strata and slabbing of the coal face. Significant convergence of the powered supports is associated with caving of the cantilevered strata. Slabbing of the coal face both removes support to the immediate roof and increases its unsupported span, thereby increasing the risk of local roof falls.

The risk of roof falls is elevated further because periodic weighting is also usually associated with discontinuous subsidence, whereby a gap develops at the base of the bridging strata. This results in the goaf strata being compressed only by the weight of the parting and not by the total weight of the overburden, resulting in a significant reduction in the lateral constraint provided to fractured strata in the vicinity of the face. The combination of high face stress, extensive fracturing of the coal seam and roof strata, and the low lateral stress in the goaf, leads to a potentially dangerous situation whereby massive blocks formed by mining-induced fractures can slide out of the roof on the longwall face (Wagner 1994). The problem becomes more severe with increase in mining height. In some instances, a detached block can fall onto the back of the powered supports during a weighting event. The resultant force of the slab hitting the goaf shield has the capacity to push the support forwards into the AFC and face (Hookham 2004).

Creech (1996) observed that following a periodic weighting event, mining-induced shear planes dipping back over the powered supports were present for the next four to six metres of extraction. Operational experience confirms that the risk of a fall of ground is not immediately reduced once the strata caves. On the contrary, the risk is often elevated until the face has been advanced several metres through the mining-induced fracture zone because the caving event results in removal of confinement to the shattered and failed roof and coal face material in this zone, allowing it to unravel.

A number of models have been proposed to explain periodic weighting. Following on from Beer and Meek (1982), Wold and Pala (1986) applied voussoir beam theory to analysing periodic weighting associated with a massive sandstone overlying a relatively friable 10–15 m thick lower roof at Ellalong Colliery. Frith and Creech (1997) initially proposed a form of detached block model but later subscribed to a voussoir beam model proposed by Seedsman and Stewart (1996). Gale (2004, 2009) utilised the FLAC computational code to model behaviour down the centreline of extraction panels. All are two-dimensional approaches and have their limitations but, nevertheless, are useful for conceptualising behaviour and undertaking parametric analysis of periodic weighting, which is fundamentally a three-dimensional behaviour.

Gale (2004) simulated the nature of fracturing and caving associated with a massive sandstone unit having a UCS of approximately 40 MPa that immediately overlaid a coal seam (Fig. 9.21a). This was complemented with measurements of cyclic loading of supports (Fig. 9.21b) and monitoring of overburden caving using surface extensometers when the seam was mined. It was concluded that:

- a block of massive strata begins to form in the immediate roof early in the caving process;
- the massive block develops bending stresses in response to overburden subsidence onto the goaf;
- failure is initiated in the upper section of the cantilevering block and then progresses rapidly down towards the seam;
- resistance to convergence is lost;
- face convergence may be ‘instantaneous’ or occur over a number of shears, depending on lithology; and
- overburden ‘rebound’ rather than gravity dropout of an isolated block may be the

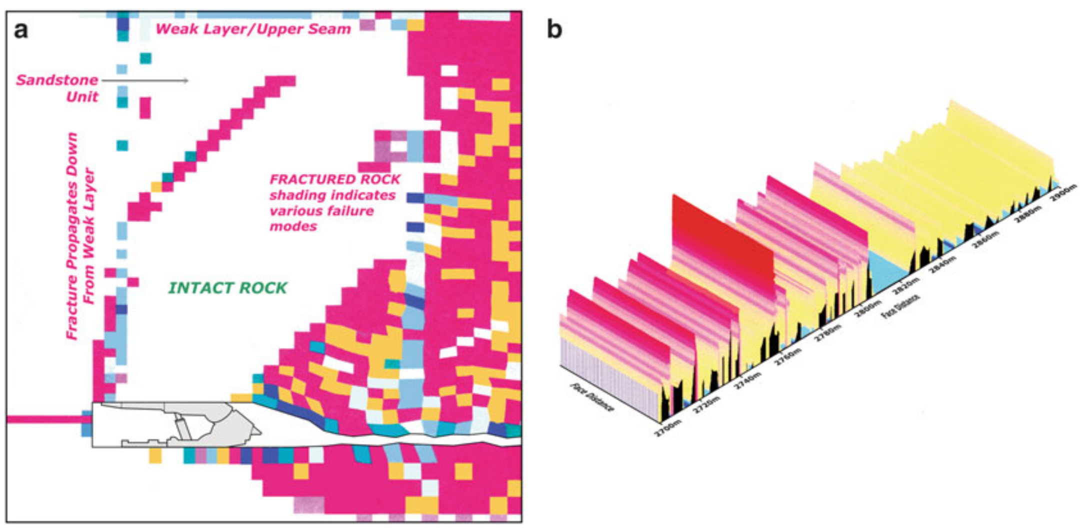


Fig. 9.21 An example of two-dimensional numerical modelling applied to the simulation of fracture mode and orientation associated with longwall mining beneath a massive sandstone unit, complemented with monitoring of longwall powered support pressures when mining beneath the unit (After Gale 2004). (**a**) The nature of fracturing and caving as determined by numerical modelling. (**b**) Monitored rates of increase in powered support pressures for the situation modelled numerically in (**a**), with the red bands corresponding with episodes of cyclic weighting

principal driver of convergence, thereby resulting in a finite displacement, typically in the range of 0.1–0.6 m.

The significant influence of the composition of the immediate and upper roof strata on strata behaviour around the longwall face is illustrated by comparing Fig. 9.20 with Fig. 9.21. Field observations and microseismic monitoring (Fig. 3.19b) revealed a strong trend for periodic weighting events to concentrate around cut-throughs. Frith (2005) reported that at South Bulga Colliery, Australia, no major weightings were ever experienced outside of a few metres from a chain pillar cut-through.

The impacts of periodic weighting on face stability and equipment can range from nuisance value to complete loss of the face. During the mining of Longwall 5 beneath a massive immediate roof at Newstan Colliery, Australia, periodic weighting resulted in 14 falls of ground at 35–40 m intervals that extended up to 70 m along the face, up to 6 m ahead of the face, and over 10 m into the roof (Hebblewhite and Simpson 1997). Blocks in excess of 6 m long and 1 m wide fell onto the AFC or bridged between the face and the powered supports, which on occasions converged 1–1.5 m over a period of one to two shifts. Phalen Colliery, Canada, experienced over 1 m of convergence in less than four hours on a longwall face (MacDonald 1997), while a convergence rate of 15 mm/s was recorded during a dynamic event at Churcha West Colliery, India, that resulted in some 1.5 m of closure in less than one hour and the destruction of 23 chock shield supports (Gupta and Ghose 1992).

In an attempt to control periodic weighting, some operators have replaced powered supports with higher capacity units. While this offers benefits, the improvements are generally limited and, because of lever arm effects, are not in proportion to the increase in nominal support capacity. For example, analysis by Galvin (1997b) showed that when the capacity of 4 leg chock shields working under a 6.3 m thick massive immediate roof was increased from 8 MN to 9 MN (800 t to 900 t), the extra overhang that could be supported before the powered supports went into yield was only 1 m.

Effective mine design controls for mitigating periodic weighting are either to increase panel width such that the massive strata caves soon after the commencement of panel extraction and then at very short and regular intervals thereafter, or else to limit panel span such that the massive strata bridges the panel indefinitely. The manipulation of panel width to control periodic weighting has been applied very successfully for decades in South Africa on the basis of the Galvin dolerite sill failure span formula (Galvin 1983). However, if panel span is deliberately restricted, care is required to ensure that this does not result in excessive abutment stress throughout the life of the panel or in windblasts. At Coalbrook Colliery, South Africa, longwall panel span was designed to be either greater than 200 m in order to induce failure of the overlying dolerite sill, or else less than 120 m in order to control abutment stress (Henderson 1980). While changing longwall span at Newstan Colliery from 226.5 m to, initially, 90 m and, subsequently, 150 m proved very successful in mitigating severe face instability associated with periodic weighting, it resulted in shallow caving of the nether roof that generated violent windblasts (Hebblewhite and Simpson 1997).

The trend in longwall mining is towards wider faces, or panel spans, made possible by advances in coal clearance technology, in particular synchronised multiple drive motors for AFCs. Wider faces are attractive because they reduce gateroad drivage metres and down time associated with longwall moves and the extent of the surface affected by differential subsidence. However, they increase financial risk associated with an underperforming installation. From a ground engineering perspective, it might be concluded that once panel width-to-depth ratio becomes supercritical, an increase in face width will have little impact on ground behaviour other than that the length of longwall face subjected to maximum abutment stress may increase and the longer time between shears increases the opportunity for the face to deteriorate. However, early experience with a 400 m wide longwall face in Australia indicates that abutment stress impacts extend further outbye of the face than those associated with narrower supercritical width faces.

Otherwise, once the span of a longwall reaches the critical width, little detailed consideration is usually given to the impacts of further increases in longwall panel width. However, careful consideration has to be given to a reduction in longwall panel width below its critical span in order to avoid the operation being subjected to high abutment stress throughout the life of the panel. Furthermore, a relatively small change in panel width-to-depth ratio or geology can result in a step change in surface subsidence, as evidenced in Fig. 3.14.

The reader is referred to the range of approaches to determining panel span presented in Sect. 3.3.3. Once again, the situation is similar to that in pillar extraction in that semi-empirical and analytical models can provide reasonably accurate estimates of the span required to induce full caving and subsidence if calibrated to site-specific data. Appropriately chosen and constructed numerical models can be valuable for quantifying abutment stress magnitudes and distributions as a basis for selecting mining span, but outputs can also be unreliable. Therefore, numerical modelling outcomes should be used as an aid and supported by parametric and sensitivity analysis, rather than being accepted as absolute and correct. Irrespective of the desktop approach taken to design, historical field performance and local operational experience are invaluable for determining mining span, especially in situations where there is potential for cyclic loading and/or windblast.

Precursors to cyclic loading events can include:

- audible noise or ‘bumping’ of surrounding strata, sometimes correlating with significant coal face spalling or ejection of coal from the face and with proximity to a cut-through;
- guttering at the face/roof intersection;
- face spall;
- roof spall;
- water make from the roof;
- an increased rate of rise from set to yield pressure in powered supports;
- an increased rate of convergence of powered supports; and
- a dynamic loading event.

Controls to minimise the occurrence and impacts of periodic weighting are:

- Mine design
  - The minimum dimension (width) of an extraction panel should be sufficiently large to induce caving of massive strata very soon after commencement of extraction and at short intervals thereafter, or else sufficiently narrow to prevent the onset of caving and to limit abutment stress.
- Powered support design
  - The supports should make provision for minimising the unsupported span from the point of effective support load application to the coal face. This may require some form of articulation at the front of the support.
  - Supports should have a high tip support capacity.
  - Support resistance should be maximised by minimising the total supported roof area.
  - Supports should incorporate face sprags where working height permits.
  - Yield valves should be of a rapid release type.
- Powered support operation and maintenance
  - Support hydraulics should be maintained to a high standard with minimal hydraulic leaks.
  - Hydraulic pumps should not be shut down during a periodic weighting event.
  - Adequate volumes of hydraulic fluid at the correct pressure need to be available.
  - Supports should incorporate guaranteed set.
  - Supports need to be set at optimum pressure being, typically, at least 80 % of yield.
  - Positive set should only be turned off in areas affected by cavities.

  - Yield valves should be maintained in good condition and operate at design relief pressure.
- Face operation
  - Since the strength of rock can reduce over time, it is important to retreat the face regularly so that abutment stresses have less opportunity to cause fracturing of the coal face and the immediate roof and floor strata in the vicinity of the coal face.
  - The face should be maintained in a straight alignment (noting once again that some operators report a benefit in the centre of the face being in advance of the gate-ends).
  - At high mining height, flippers (sprags) should be used at all times. These will not only stabilise the face, but also prevent broken material from flushing onto the AFC.
  - Subject to not inducing excessive face spall, mining height should be maximised prior to an anticipated weighting event in order to accommodate yielding of powered supports.
  - Rate of face retreat should be maximised but only to the extent that it remains regular and controlled.
  - Maintenance should not be scheduled and the face should be worked around the clock during the event.
  - Powered supports should be advanced individually just behind the shearer so as provide support immediately to newly exposed roof. Bank push is not advisable.
  - A rapid rate of face retreat should be maintained for at least 3–6 m after relaxation of face pressure in order to prevent unravelling of shattered, unconfined face and roof strata.
  - If face spalling or fallen roof material is excessive or retreat rates are rapid, coal clearance capacity may need to be maximised by shutting down other belt systems.
  - In the event of excessive face spall or the face having to stand for any period of time, the face should be closed up.
  - Adequate supplies of suitable secondary support, strata consolidation products and void fillers should be on hand in the event that the face has to be stopped; the tip-to-face distance becomes excessive; or face or roof control is lost.
  - Secondary support measures need to be implemented as soon as face control begins to be lost. Support pressure monitoring algorithms based on factors such as loading rates, yield frequencies, time weighted average pressures and number of affected powered supports (for example, that described by Hoyer 2011), can provide early warning in this regard.
- Outbye services
  - Allocate labour to outbye services to minimise disruption to longwall face operations, especially those caused by conveyor belt stoppages and loss of electric and hydraulic power supplies.

The microseismic monitoring associated with Fig. 3.19 gives insight into a number of aspects of strata behaviour for the case of multiple 200 m wide longwall faces located at a depth of around 500 m. Aspects of particular note include:

- the majority of fracturing (low frequency events) extended to a height of 50–70 m above the seam and to a depth of 80–90 m into the floor;
- cyclic failure was not symmetric about the longwall face but was biased from mid-face to the tailgate;
- mining reactivated strata failure beneath the chain pillars of the previously extracted panel, up to 300 m away; and
- mining activated a strike-slip structure in the maingate (high frequency events) when it was still more than 300 m from the structure.

9.6 Installation Roadways

Geological structure, horizontal stress, seam dip, cleating and jointing are some of the in-seam factors that give rise to preferred mining directions. In longwall mining, it is usual to orientate the panels so that neither the headings nor the cut-throughs are orientated in the least preferred direction. Headings can be biased towards the favoured mining direction while still limiting exposure of the cut-throughs to the poorest ground conditions if the cut-throughs are not driven at 90° to the headings (reference Sect. 9.2.2). At the inbye end of the longwall panel, however, there is no option but to drive the longwall face installation roadway at 90° to the headings. To provide sufficient space to safely manoeuvre the longwall equipment, the width of this roadway typically ranges from 7 to 11 m, depending on the size of the longwall face equipment.

The drivage and support of a wide roadway presents an elevated risk due to the increased likelihood of ground instability and the potentially high safety and business related consequences associated with instability of such a critical roadway. Therefore, drivage methodology and support design warrant careful consideration.

In benign ground conditions, the installation roadway may be driven in a single pass. This offers considerable operational advantages associated with coal clearance, ventilation, advancing services, installing support, and maintaining a flat roof horizon. Most often, however, strata control considerations require the roadway to be driven in two passes, and sometimes three, in order to restrict unsupported span. This usually requires subsequent passes to be cut to a lower roof horizon to avoid damaging roof support already installed in the roadway.

If the installation roadway has to be driven at some acute angle to the direction of an elevated horizontal stress field, a lateral stress shadow will be induced in both flanks of the drivage. The protection that this provides to the second pass depends on:

- stress magnitude, strata strength and strata stiffness;
- which side of the first pass the second pass is driven; and
- the direction of drivage of the second pass.

If, in the example shown in Fig. 9.22, the second pass was to be driven in the same direction as the first pass and to the right of it, the right hand corner of the new drivage would once again be the leading corner in penetrating the lateral stress field and so might be impacted, albeit perhaps not to the same extent as during drivage of the first pass. On the other hand, if the second pass is driven to the left of the first pass, the leading corner is immediately adjacent to the void of the first pass and, therefore, is well within a lateral stress shadow and protected by ground support already installed in the first pass. The left hand side of the second pass does not come to be exposed to the lateral stress since mining on the right hand side causes this stress to be redistributed in advance of mining the left hand side. If the second pass is driven from the opposite direction to that of the first pass, the situation just described is reversed.

Consideration also needs to be given to the ultimate location of the primary damage zone in the completed installation roadway, with the options being for it to be located at the face ribside; towards the centre of the roadway; or, preferably, at the goaf ribside. The location of the primary damage zone towards the centre of the installation road can introduce additional operational and safety issues for face workers when mining through and supporting the damaged zone on the second pass. In particular:

- it is problematic if the free edge of the roof of the second pass will remain intact in the structurally disturbed conditions;
- there are operational and quality assurance challenges associated with drilling long holes and achieving effective anchorage in ground that is already fractured; and
- a roof fall can undermine and render ineffective the support installed in the damaged zone during mining of the first pass.

A factor not to be overlooked is damage to the floor, which can cause serious operational problems given the large ground forces and multiple movements of equipment associated with relocating longwall equipment, especially in the presence of water. These problems are most severe if the primary damage zone is located

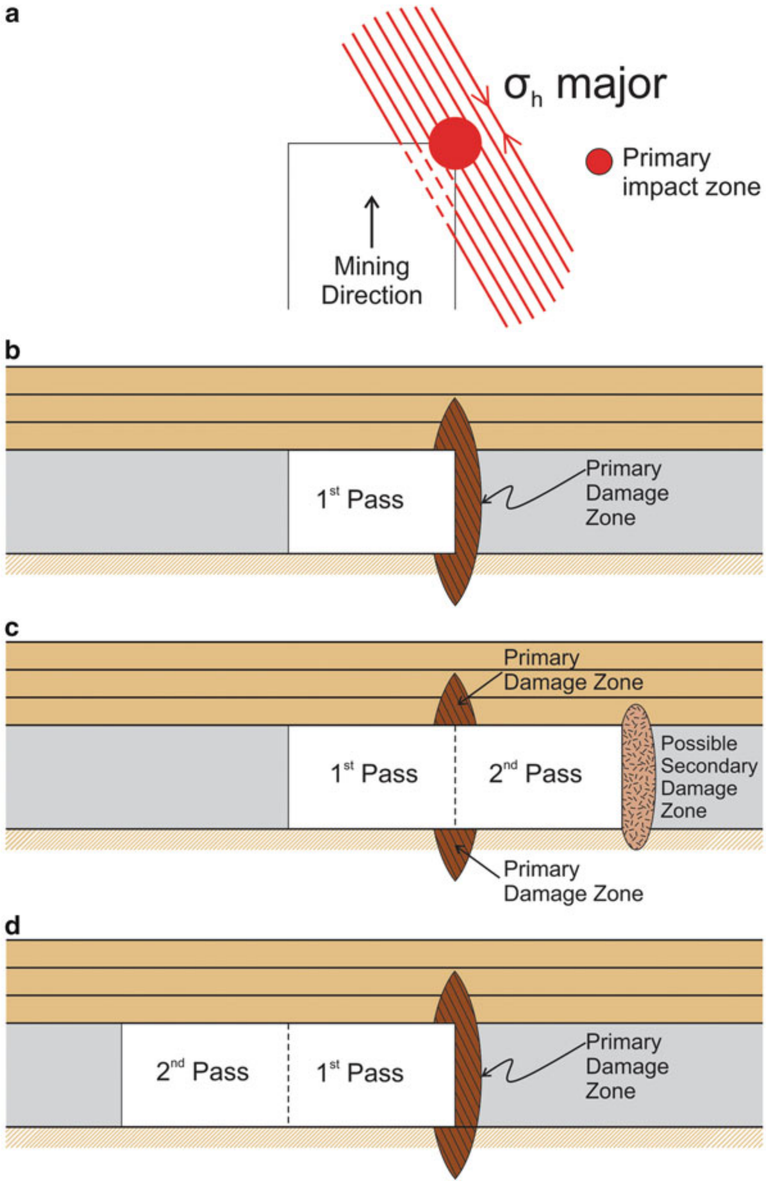


Fig. 9.22 Location of potential damage zones in an installation roadway driven at an acute angle to the direction of an elevated horizontal stress field. (**a**) Plan view of 1st pass, (**b**) Cross-section through 1st pass, (**c**) 2nd pass driven adjacent to stress field, (**d**) 2nd pass driven in stress shadow

towards the centre of the completed installation roadway.

A higher degree of uncertainty is associated with the design and performance of roadway reinforcement systems when the primary damage zone is located in the centre of an installation road rather than in a supported and partially confined state in a ribside. If the situation deteriorates to the point where standing support is required, this support needs to be installed towards the centre of the roadway, where it then presents a serious obstruction to the installation of the face equipment and an additional risk when the time comes to remove it. Hence, ideally, the installation roadway should be driven in a direction that results in the lateral stress induced damage being located at the rear of the powered supports. For the example shown in Fig. 9.22, the optimum situation would be to drive the second pass on the left hand side of the first pass, with longwall mining retreating to the left.

Caution is required if support design is based purely on empirical data sourced from other mines and when the design process has limited regard to behaviour mechanics. The amount of convergence that develops during mining of the first pass is an important consideration when determining support requirements for the second pass. However, regard must also be had to the potential for ground behaviour mechanisms to change during subsequent mining passes. Some support design procedures have a reliance on criteria that have limited, if any, regard to stress paths or to the mechanical behaviour of the ground support elements and the surrounding rock mass. Reinforcement Density Index, discussed in Sect. 7.3.5, is an example of one of these criteria.

Controls to assist in managing ground stability when driving installation roadways include:

- Optimising mining direction to minimise exposure of installation roadways to elevated horizontal stresses.
- Minimising installation roadway width. This includes avoiding the holing of an installation roadway and, therefore, the formation of an intersection at any point along its length.
- Driving the installation roadway in at least two passes so that it is already in a partially reinforced state when full span is achieved.
- Numerical modelling supported by monitoring data to aid in identifying behaviour mechanisms, stress paths and support requirements, and to support empirical design procedures.
- Monitoring of strata response during each phase of the drivage process and timely processing and evaluation of monitoring data.
- Installing powered supports as soon as possible after the completion of drivage.
- Delaying the driving of the second pass or installing a higher density of support to counteract any creep behaviour if the installation roadway is to stand for an extended period of time.
- Trigger Action Response Plans which provide for timely identification and response to deviations from anticipated behaviour during each pass.
- Contingency Plans which provide for the necessary materials, equipment and competent personnel being on hand to respond to triggers.
- Utilising powered supports as a temporary support measure in critical situations by installing them in a longitudinal line down the centre of the installation roadway.

Sometimes it is unavoidable that an installation roadway has to be orientated in an adverse direction to a high horizontal stress field. A control option in these situations is to place the roadway in the stress shadow of an adjacent drivage (Figs. 5.4, 5.5, and 5.6). This drivage may take one of three forms which, in order of increasing reliability and effectiveness, are:

- A conventional roadway supported in the standard manner for the mine. This approach has met with limited success.
- A conventional roadway with minimal support such that it remains in a safe condition during drivage but promotes roof softening and may fall after mining has ceased in the area.
- A sacrificial roadway comprising a conventional roadway supported in the standard manner and then lifted off (widened without installing additional support) on the retreat so as to encourage caving to a substantial height, typically to at least two-thirds of the overall roadway width (Fig. 9.23).

The concept of a sacrificial roadway, or stress relief roadway, has proven highly effective at some mines (reference, for example, Galvin 1996 and Doyle and Gale 2004). Figure 9.24a shows the condition of the cut-through leading to a longwall installation roadway. In this particular case, stiff stone bands interbedded with coal plies were prone to shear and dilate under the effect of high horizontal stress and rapidly drive down the

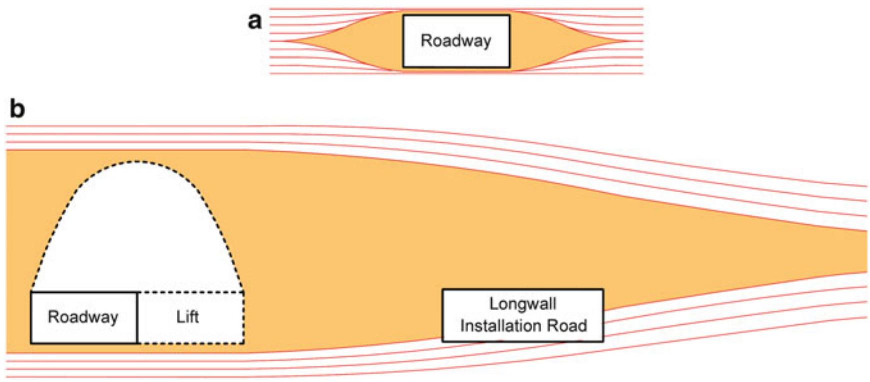


Fig. 9.23 An illustration of the concept of driving an installation roadway in the stress shadow of a sacrificial roadway. (**a**) Horizontal stress contours around a roadway, (**b**) Location of longwall installation roadway in a zone of reduced horizontal stress due to shadowing effect of sacrificial roadway

Fig. 9.24 An example of the beneficial effect on ground control of utilising a sacrificial roadway to create a lateral stress shadow in a high horizontal stress environment. (**a**) A 4.8 m wide cut-through driven at ~65° to the regional major horizontal stress, which was σ_1 (**b**) An 8 m wide longwall installation roadway driven from the cut-through shown in (**a**) after the formation of a 10 m wide caved sacrificial roadway some 9 m further inbye

immediate roof. Drivage of the installation roadway was delayed until after a parallel roadway had been driven, widened and caved (lifted off) some 9 m inbye, so as to place the installation roadway site in a stress shadow. Figure 9.24b shows the remarkable improvement in conditions and ground support requirements when the installation roadway was driven.

While sacrificial roadways are effective in redirecting horizontal stress away from an installation roadway, this can introduce a new risk in the form of tensile failure, especially in jointed ground. Removal of the horizontal stress results in unclamping of the jointed ground, enabling it to fall without warning. Abnormal water make from the roof can be one of the few warning signs of impending tensile failure. This failure mode can present an elevated risk because:

- most often, roof reinforcement patterns in underground coal mining are not designed to control a tensile environment;
- failure is more likely to develop suddenly due to the absence of secondary reinforcement in the form of long tendons;
- the destressed installation roadway is often driven at full face width in a single pass rather than in two or more passes, therefore resulting is a lower likelihood of detecting signs of impending failure in time to respond effectively;
- standard monitoring instrumentation may not detect the onset of tensile failure at all, or in time; and
- mine workers are likely to be conditioned to recognising and responding to signs of compressive stress rather than tensile stress.

9.7 Pre-driven Roadways Within a Longwall Block

Pre-driven roadways can comprise:

- Stubs driven a limited distance into a longwall block.
- Existing bord and pillar workings.
- An excavation formed as an outcome of extracting a dyke ahead of a longwall face.
- One or more roadways driven across the full width of a longwall block. Sometimes these are pre-planned to facilitate ventilation and gateroad drivage from multiple locations in very long longwall panels and are aligned so as not to trend parallel to the longwall face. Often they are a legacy of past mining and trend parallel to the longwall face.
- A longwall recovery roadway driven parallel to the longwall face over its full length.
- Short longwall recovery stubs driven some distance in from each gate end.

Fundamentally, longwall mining into a pre-existing excavation is a practice that is contrary to ground control principles. It is undertaken because, if successful, it may offer high financial rewards in terms of continuity of coal production; cost savings in not having to relocate the longwall face around the excavation; and, in some situations, increased resource recovery. However, the risks can be high, especially when mining into a pre-driven longwall face recovery roadway. These risks relate not only to financial loss arising from equipment damage and extended loss of production if face stability is lost but, most importantly, to the health and safety of the mine workers, both at the time of losing ground control and during recovery operations.

International experience suggests that the failure rate associated with mining into pre-driven longwall recovery roadways is of the order of 10 %. This is high and all the more noteworthy because of the high consequences associated with failure. Past troublesome and/or unsuccessful cases such as those discussed by Gardner (1987), McKensey (1988) and Klenowski et al. (1990), and subsequent unpublicised events, highlight the need to carefully consider the risks associated with this practice.

Some unsuccessful outcomes have been associated with design methodologies that do not recognise or properly evaluate all the primary controlling variables because of their empirical nature, especially when they rely to a considerable degree on curve fitting to empirical data. In these later cases, shortcomings have been compounded on occasions by misplaced confidence in statistical analysis. Given the risk profile of pre-driven roadways (probability of a strata failure and consequences of this failure), it is strongly recommended that the design of a pre-driven longwall recovery roadway is underpinned by an applied mechanics approach, supported by appropriate numerical modelling.

9.7.1 Generic Types and Mining Practices

Stabilisation of pre-driven roadways in preparation for longwall mining consists of one or a combination of:

- reinforcement of the roof and ribs using short and long tendons and mesh;
- standing support, such as large diameter timber legs, timber and cementitious based chocks and packs, and monolithic cementitious cylinders; and
- backfilling, most often with a weak cohesive mixture of cement, flyash and sand or coal.

Mining into an excavation formed as a result of dyke extraction warrants additional care because the dyke contact surfaces constitute discontinuities extending well into the upper roof strata. The discontinuities disrupt the transmission of mining-induced stress, giving rise to stress concentrations, and providing potential failure planes for large wedges to drop out of the roof.

Van der Merwe (1988) reported on three instances at a mine in South Africa where a dolerite dyke was extracted ahead of 200 m wide longwall faces at a depth of around 140 m, corresponding to a panel width-to-depth ratio, W/H, of 1.4. The dyke meandered between 0° and 15° off the line of the longwall face, such that only portions of the dyke were exposed at any point in time. The immediate roof of the coal seam was weak and highly laminated. On the first occasion, the excavation was supported successfully with timber chocks (packs). The second occasion was also successful, this time utilising a reduced pack spacing. On the third occasion, the excavation was supported with timber props. The floor of the longwall was some 300 mm lower than that of the pre-driven excavation when it was intersected, necessitating secondary blasting. The slow progress resulted in steadily worsening conditions and eventually a face break occurred. Although the longwall face was recovered, very high costs and production losses were incurred.

Subsequently, the same dyke was removed ahead of a longwall face in a lower seam. The roof of the excavation was cable bolted and the ribs were supported with wooden dowels before being backfilled. Despite mining only 10–12 m directly beneath the interpanel pillars in the upper seam and several extended production delays of up to a week during extraction through this area, no instability problems were encountered.

Minney (1999) reported on the successful extraction of a dyke trending sub-parallel (~8°) to two longwall faces in a competent immediate and upper sandstone roof environment at New Denmark Colliery, South Africa. The longwall face width was reduced from 200 to 120 m to modify the behaviour of the upper roof strata which contained a massive sandstone unit some 21 m thick. The dyke excavation was supported with fully encapsulated cables and the tailgate was kept 10 m in advance of the maingate so as to hole into the excavation progressively. Success was attributed in part to the presence of massive sandstone roof.

Other experiences at this mine serve to illustrate the critical role that the stiffness of the mining system plays in determining the success of mining into pre-driven roadways. These include the longwall mining of nominally 80 m wide panels of standing bord and pillar workings reported by Galvin et al. (1991) and Bruins (1997), shown in Fig. 8.18 and discussed in Sect. 8.3.2.4, and the utilisation of only 9 m wide chain pillars between longwall panels (Galvin 1997a; Minney and Karparov 1999). Figure 8.18 reflects the benefit of maintaining a stiff loading environment by restricting the panel width-to-depth ratio.

Jones (2008) reported on the successful extraction of an igneous plug ahead of a longwall face in Australia, where longwall mining of standing pillars has also been undertaken on a small scale. At Homestead Colliery, a 3 heading development that extended about two-thirds of the way across a 200 m wide longwall block was backfilled with a cement, sand and flyash mix prior to the successful passage of the longwall face (Grice et al. 1999).

A number of mines have successfully extended the width of longwall panels by extracting adjacent standing pillars. Van der Merwe (1989) reported on a longwall operation at Bosjesspruit Colliery, South Africa, that extracted a row of pillars at the tailgate, with success being attributed to the presence of a very competent sandstone roof.

Cordeaux Colliery, Australia, successfully extended the width of a longwall panel some 45 m by also extracting a line of standing pillars at the tailgate end of the face. The roof of the 15 year old workings was re-supported with a combination of monolithic cementitious cylinders and pretensioned cables up to 6 m in length. The ribs were re-supported with 1.2 m long cuttable rib bolts and synthetic mesh and the maingate face end was maintained in advance of the tailgate end. Fisher (2001) reports that excessive noise and rib convergence occurred in the 8 m to 4 m zone from holing and, at a fender width of 2 m, the cut-throughs exhibited signs of failure. Floor heave occurred in the roadways about to be holed. Once the fender was removed, there was a marked acceleration in roof displacement of up to 10 mm/h while the powered supports were in yield. Displacement tapered off when the face had advanced about half way across the pre-driven roadways.

All these case studies involve loading environments that vary substantially from those applying to most pre-driven longwall recovery roadways. For example, panel width-to-depth ratio, W/H, was deliberately restricted in some cases. Others involve pre-driven roadways that were narrow; and/or exposed to abutment stress in a confined state; and/or located towards one end of the longwall face. It is these types of considerations that make it strongly advisable from a risk management perspective for design to be based on a mechanistic approach supported by sensible numerical modelling rather than only on a purely empirical approach.

9.7.2 Pre-driven Longwall Recovery Roadways

A pre-driven longwall recovery roadway can comprise a short stub driven from a gate end or a roadway that trends parallel or very near parallel to the face line across the full width of a longwall panel. Driving a longwall into a pre-driven roadway presents an elevated risk, especially in the case of a roadway that extends over the full width of the longwall panel, because:

- The pre-driven roadway causes an increase in abutment stress.
- There is potential for a higher density of mining-induced fracturing around the longwall recovery roadway because the strata are effectively unconfined when subjected to the approaching abutment stress front of the longwall.
- The width-to-height ratio of the pillar, or fender, between the longwall face and the pre-driven roadway is progressively reduced along the full width of the longwall face, such that:
  - the stiffness of the fender is also progressively reduced, resulting in increased seam convergence and, therefore, increased mining-induced fracturing and load transfer onto the powered supports and the outbye coal face;
  - face spall leading up to fender failure increases the tip-to-face distance at roof level but may leave a wedge of material at floor level that prevents the powered supports immediately being advanced to control this situation;
  - when the fender ultimately fails, there is a step increase in effective tip-to-face distance, increasing the potential for unravelling of the immediate roof and face falls;

  - conditions may be conducive to the fender failing in a sudden and violent manner; and
  - loss of strata control may extend along the full length of the face.
- Prior to the fender failing or being extracted, fender stress may initiate failure of the fender foundations, resulting in the fenders punching into the roof or floor and inducing roof falls and floor heave that present an impediment to advancing the powered supports.
- In the event of a face break, the centre of gravity of the load acting on the powered supports can migrate rapidly to the front of the supports, with the resulting moment arm resulting in a rapid and significant reduction in the total load carrying capacity of the powered supports and a step increase in floor pressure under the toes of the powered support bases.
- There is a loss of face height, resulting from some or all of the above factors.
- Stability and face advance are highly dependent on the longwall face holing into the pre-driven roadway at or very close to the same floor and roof elevation and this can be difficult to achieve, especially in the presence of floor heave, seam convergence and roof instability.
- The consequences of any downtime are higher, given that the strength of stressed rock, particularly sedimentary rock, can be time dependent.
- There is a higher likelihood of unplanned downtime due to the impact of many of the preceding factors on face operations, with the highest probability of downtime coinciding with the critical stage of holing through when the consequences of downtime are highest.

Impacts commonly associated with these factors are:

- roof falls on the longwall face due to the increased tip-to-face distance;
- a face break, which then results in the fender and powered supports being loaded, often rapidly, by a detached block;
- dynamic and violent failure of the fender, sufficient in one instance to have caused serious damage to hydraulic circuitry on the powered supports;
- loss of horizon control and clearance for the shearer on the longwall face due to the AFC being lifted and tilted by floor heave;
- inability to advance the powered supports due to a difference in floor or roof horizon between the pre-driven roadway and the longwall face;
- inability to generate tip support due to a difference in roof horizon between the pre-driven roadway and the longwall face;
- reduce powered support capacity due to damage to hydraulic circuitry; and
- trapped and iron bound equipment.

Case studies provide insight into strata behaviour around pre-driven roadways and the factors that influence success. Simpson et al. (1991) report on a series of successful pre-driven roadways in three different seams at Newstan Colliery, Australia. The powered supports were 4 leg chock shields, face width ranged from 118 to 201 m, extracted height from 3.0 to 3.4 m, and depth from 20 to 90 m, except for one case at a depth of 300 m. The immediate roof of each of the pre-driven roadways was supported by various combinations of rock bolts, straps, mesh, 10 m long fully encapsulated cables, and monolithic cylinders. The outbye riblines were supported with 1.8 m long steel bolts and mesh and the inbye riblines with various cuttable bolts and dowels (up to 8 m in length) and strata binders. The face operation mode reverted to conventional (effectively, double chocked) as the pre-driven roadway was approached, with the maingate leading by up to 8 m.

Floor heave occurred in all instances, with up to 1 m occurring within one hour in the two shallowest seams. It is reported that this was easily cut and loaded out by the shearer.

Extensive monitoring of the pre-driven roadway developed at a depth of 300 m revealed that although there was a rapid increase in roof to floor convergence as the fender width was reduced from 8 to 4 m, there was no bed separation within the first 11 m of the immediate roof. It was only during the last 4 m of extraction when the fender failed that the immediate roof developed partings up to the 4 m horizon. The controlled nature of the fender failure in the shallower seams was believed to be associated with the bearing failure of the floor. However, this could not account for controlled failure in the deeper seam where the floor was a 100 MPa strength shale.

The behaviour of the immediate roof at Newstan Colliery contrasts with that associated with the unsuccessful cases shown in Fig. 9.25a, c, d. The Longwall Panel 6 failure in 1987 at Pacific Colliery, Australia, depicted in Fig. 9.25a, was associated with a 138.5 m wide panel being extracted at a height of 2.6–2.7 m utilising 2 leg, 5.6 MN (560 t) capacity, shield supports. The immediate roof comprised weak tuffaceous sediments (claystone) overlain by stronger mudstone and sandstone strata. The panel had been subject to periodic weighting that was attributed to the presence of the sandstone. The pre-driven roadway was supported by a combination of 2.4 and 2.7 m long fully resin encapsulated bolts installed through straps, and 8 and 10 m long cementitious grouted cables. No standing support was installed in the recovery roadway.

Salient points associated with this failure as described by Gardner (1987) are:

- The ranging arm on the shearer failed when the face was some 22 m away from the recovery roadway, resulting in a 16 h stoppage.
- The fender yielded at a width of 7 m, generating an 18 m long cantilever.
- 1:30 am, 19/7/87: Fender width 3 m. W straps in recovery road roof started to buckle, and small amounts of guttering were apparent in outbye corner of the roadway.
- 5:00 am: Powered supports along face went onto continuous yield, at the same time as the roof in the recovery roadway underwent *'dramatic deterioration with much fretting between W-straps and around cable bolt plates'*.
- 7:00 am: Upon the shearer holing the maingate end of the recovery road, it was found that the roof of the recovery roadway was much lower than the longwall face. The powered supports still had about 150 mm of leg travel remaining. *'At this time the remaining coal fender, about 2 m thick, literally exploded into the recovery roadway.'*
- 7:30 am: Shearer was unable to pass under support canopies. Nearly all legs had less than 50 mm of travel remaining.
- 9:30 am: All roof movement appeared to have ceased. Powered supports were now iron bound. Most of the recovery roadway was still intact but in very poor condition. During the next 6–8 h, some sections did fall.

One of the causes of the incident proposed at the time was the failure of a sandstone cantilever ahead of the face that resulted in the powered supports being loaded well in excess of their capacity. This mechanism is similar to that concluded from instrumentation by Klenowski et al. (1990) in regard to difficulties experienced in holing into the pre-developed recovery roadway for Longwall Panel 303 at Central Colliery in 1990 (Fig. 9.25b). Salient points associated with this experience were:

- The pre-driven recovery roadway was supported with bolts and straps that were subsequently augmented with 8 m cables in the roof and dowels in the fender.
- The support design was based mainly on the performance of an instrumented stub heading which extended 15 m into the panel, complemented with finite element modelling. The stub fender yielded at a width of 2.7 m.
- The pre-driven roadway fender yielded at a width of 5 m. However, it did not fail until there was one shear remaining, when maximum recorded face convergence was almost 260 mm.

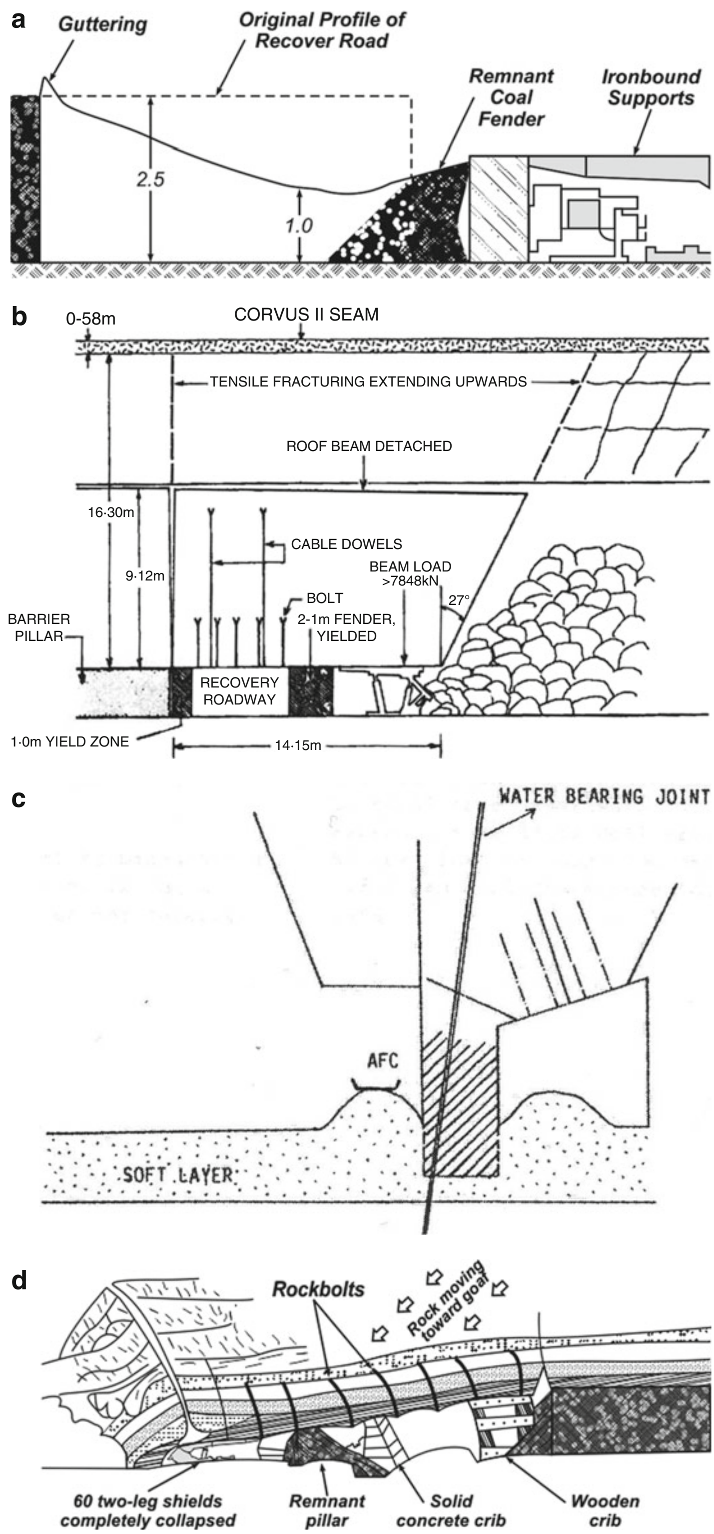


Fig. 9.25 Cross-sections through three unstable (**a**, **c**, **d**) and one troublesome (**b**) pre-driven longwall recovery roadways. (**a**) Pacific Colliery, NSW, Australia, 1987 (After L. Gardner 1987), (**b**) Central Colliery, Qld, Australia (After Klenowski et al. 1990), (**c**) Sasol Colliery, South Africa (After Van der Merwe 1989) (**d**) Colorado, USA (Adapted from Oyler et al. 1999, from Pulse 1990)

- Convergence increased to at least 420 mm at the shearer as the last shear was being taken.
- At the time of holing in, guttering had developed along the outbye ribline over a 10 m distance at mid-face.
- Convergence was greater on the longwall face side of the fender than the outbye side, requiring the setting height of the powered supports to be increased immediately after holing. Insufficient hydraulic fluid was available for this to occur and so powered support advance ceased until the pumps reservoirs were refilled. It appears that gas yield valves commenced to malfunction at this time.
- Back-analysis indicated that a 9.3 m thick roof beam with an average length of some 21 m detached from the outbye side of the recovery roadway and commenced to crush the fender when it was 5 m wide. Virtually the full weight of the beam was taken by the powered supports. At a fender width of 2.1 m, the height of the detached beam increased, with its pivot point being over the outbye pillar. At the completion of holing in, a second roof beam located 16.3 m above the seam had also detached. More than 104 powered support legs were defective by that stage and four timber props had to be set under the canopy of each support.

It was concluded at the time that roof-to-floor convergence continued and timber props progressively failed because the total resistance provided by the functional support legs and timber props was 10.1 MN (1,010 t) per support, compared to a beam load of 10.68 MN (1,068 t). Greater insight can be gained today by considering where the centre of gravity of this load acted and the need to also balance moments in order to maintain stability. Application of the detached model presented in Sect. 9.3.3 and defined by Eq. 9.2 indicates that the maximum load capacity of the powered supports for the geometry depicted in Fig. 9.25b was only of the order of 950 kN (95 t). Hence, aggressive support yield was inevitable. The front legs of the powered supports would have had to have a combined load carrying capacity in excess of 43 MN (4300 t) to avoid the support going into yield. Based on static analysis, the maximum thickness of detached block that could have been sustained without the powered supports going into yield was only about 2 m.

Van der Merwe (1989) reported on an incident in South Africa in which the conveyor belt tore when the face was only 3 m from holing. The repair took 8 h, during which time the face pillar (fender) slowly punched some 1.5 m into the floor. The floor heave lifted the AFC, which in turn, lifted the shearer so high that it could not pass beneath the powered supports (Fig. 9.25c).

Figure 9.25d shows a cross-section through a failed pre-developed roadway in Colorado, USA, as reported by Oyler et al. (1999) and attributed to Pulse (1990). The fender was between one and two metres thick when the AFC pan line became stuck, causing face advance to stop for 6 h. The mechanics of the detached block are similar to those shown in Fig. 9.25a, b and to a failure in Australia in 2011, shown in Fig. 9.26, where the fender and the standing support in the pre-driven roadway punched into the floor under the deadweight of a detached block.

Hanson et al. (2014) report on two successful pre-driven longwall recovery roadways at a depth of 70 m at Bull Mountains Mine No. 1 in the USA. The first pre-driven roadway was some 10 m wide and supported with a combination of rock bolts, cable bolts and cuttable and non-cuttable cementitious cribs. Problems installing the recovery mesh over the longwall powered supports caused extended delays which allowed the roof to deteriorate. During this time, there was significant load transfer to the cribs and powered supports.

The second pre-driven roadway was almost 13 m wide and driven in two 6.5 m wide passes. The first pass was supported with bolts, steel mesh and 5 and 8 m long cables of 55 tonne capacity, before then installing the longwall recovery mesh. Next, the roadway was completely backfilled with a cuttable, low-density, 5.5 MPa concrete. After the concrete had cured the outbye second pass was driven, bolted, meshed and also completely backfilled. Full contact of the concrete with the roof was verified, with voids being filled with a 5.5 MPa

Fig. 9.26 Failure of a pre-developed roadway in 2011 at an Australian colliery, associated with punching of fender and standing supports into soft and weak floor under deadweight load of a detached block. (**a**) Pivot point of detached roof block close to edge of outbye pillar. (**b**) Standing support starting to punch floor. (**c**) Floor heaving as bearing failure occurs beneath standing support. (**d**) Extent of punching of standing support into floor strata

polyurethane. When the fender pillar was 3 m wide, load transfer occurred onto the backfill and the longwall powered supports, with the shearer having no trouble cutting through both the coal and the concrete. An approximately 150 mm thick layer of concrete was left against the roof and this peeled off easily after being undercut to expose the pre-installed roof support and permit the recovery mesh to lay down on the shields.

Experience demonstrates that the risk of ground instability associated with excessive tip-to-face distance is elevated significantly when driving into any pre-driven roadway. This risk is magnified significantly if a detached block develops in the roof during this process. Static analysis highlights that moments, rather than forces, are the dominant factor determining stability in these situations. Hence, the capacity of the powered supports has limited influence on the outcome.

Instability is initiated by failure of the fender pillar system, either through bearing capacity failure of the fender foundations or yielding of the coal seam element. The consequences of this failure are determined by the stiffness of the loading system relative to the stiffness of the coal pillar system. As the stiffness of the surrounding strata is a function of elastic modulus and span, the probability of rapid loading leading to dynamic failure is reduced in the presence of more competent roof strata, provided that a face break does not develop towards the outbye rib of the pre-developed roadway. A face break in any type of strata immediately reduces the stiffness of the detached strata to zero, resulting in a high likelihood of uncontrolled failure when the strength of the fender pillar system is exceeded. As periodic weighting is prone to produce face breaks, there is an elevated risk of failure in periodic weighting environments.

A consideration of moments highlights that a detached block of only a few metres in thickness is sufficient to cause yielding of powered supports. Tendon reinforcement systems have limited influence on the development of a detached block, both because their capacity is insufficient to resist the forces generated by the turning moment of the detached block and because the height of the detached block extends beyond the reach of the tendons.

Panel span and distance from the panel corners impacts on the stiffness of the surrounding strata and, therefore, on the vertical and lateral extent of a detached block and the rate of loading of the fender system. Hence, limited reliance should be based on trial excavations that are of restricted extent and/or located to one side of a panel.

Given the complex combination of factors that contribute to behaviour when holing into a pre-driven roadway and the opportunities and threats associated with this practice, it is important that design is premised on an applied mechanics approach and that this is underpinned with a good understanding of the surrounding geology, material properties and stress environment. The insight provided by appropriate numerical modelling is illustrated by studies such as those of Tadolini and Barczak (2004) and Zhang et al. (2006). Tadolini and Barczak (2004) utilised a calibrated three-dimensional finite element model, developed in conjunction with an underground test site, to undertake a parametric study of the critical components and design principles relevant to a pre-driven roadway. The modelling identified critical stress distributions, failure modes, and failure locations which then provided a basis for selecting support types, patterns and densities and for anticipating critical events such as yielding of the fender.

Experience supported by numerical modelling highlights that careful consideration needs to be given to the following when planning to mine into a pre-driven longwall recovery roadway:

- selecting standing support with appropriate stiffness, strength and yielding properties, and erecting it to a pattern and density such that it is capable of sustaining both convergence and the deadweight load of a detached block without disintegrating or punching the roof or floor strata;
- reinforcing and suspending the immediate roof of the pre-driven roadway with long cables angled over both the outbye rib and the fender as well as installed vertically;
- binding the immediate roof together with bolts and mesh to provide confinement to the overlying strata and to prevent unravelling and local roof falls;
- reinforcing the fender with cuttable tendons and the outbye rib with steel tendons;
- ensuring a high level of integrity of the hydraulic circuitry for powered supports;
- consistent and rapid rate of retreat which, in turn, relies on addressing a range of other factors such as coal clearance capacity, maintenance schedules and labour availability;
- operation of the longwall face in a closed up state (doubled chocked);
- recognition that the fender may fail in a sudden and violent manner and implementing controls to safely manage such a situation;
- driving the face into the pre-driven roadway at an angle to avoid weakening the whole face in one pass and to reduce the likelihood of pillar failure rapidly developing along the full length of the face; and
- maintaining horizon control such that the longwall face holes in at the same roof and floor elevation as the pre-driven roadway.

Backfilling of a pre-driven roadway tightly to the roof with a suitably stiff material can reduce or eliminate the risk associated with some of these factors.

9.8 Longwall Face Recovery

The relocation of a longwall face presents a number of ground control challenges, particularly because the powered supports have to be extricated from a goaf in circumstances where the tip-to-face distance has been deliberately increased to the order of 2.5 m to provide sufficient working space and because the strata has time to deteriorate due to the duration of the process. Operations may also have to contend with factors that are unknown prior to the commencement of recovery, such as geological features and the state of loading in a periodic weighting cycle. Success is contingent on safe, speedy recovery of face equipment which, in turn, is very dependent on the standard of preparation of the recovery face and the roadways leading to it; monitoring of strata deterioration in these excavations during recovery operations and implementation of timely and effective responses; and powered support withdrawal procedures. These should be elements of a comprehensive risk assessment of all aspects associated with relocating a longwall face.

Basically, face equipment may be recovered via one or both gate ends and/or from roadways, or **chutes**, that hole into the recovery roadway at various points along its length (Fig. 9.27). Sometimes, the first 20–50 m of one or both ends of the recovery road may be pre-driven. Stability of intersections on the recovery face line is critical, particularly at the maingate end if the face is recovered predominantly through that access point. Cable bolting of these areas should be a matter of routine.

As the longwall approaches the pull-off point, the immediate roof needs to be treated in order to control goaf flushing during the extraction of the powered supports. This is usually achieved by commencing to screen and rock bolt the roof when the longwall approaches within 10–15 m of its final position. The screen often comprises polyester geotextile that has been stitched together before being taken underground so as to form one continuous sheet that ultimately extends out from under the goaf pile and over the top of the powered supports and partially down the face (Fig. 9.28). Pattern bolting of the roof should extend back to at least the rear of the powered supports, typically at no more than 1 m

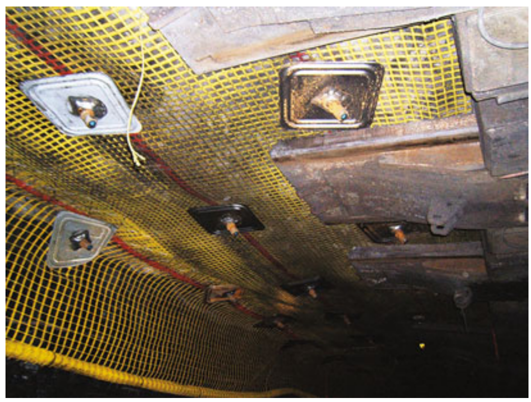


Fig. 9.28 A longwall recovery roadway that has been supported with polyester screen and bolts

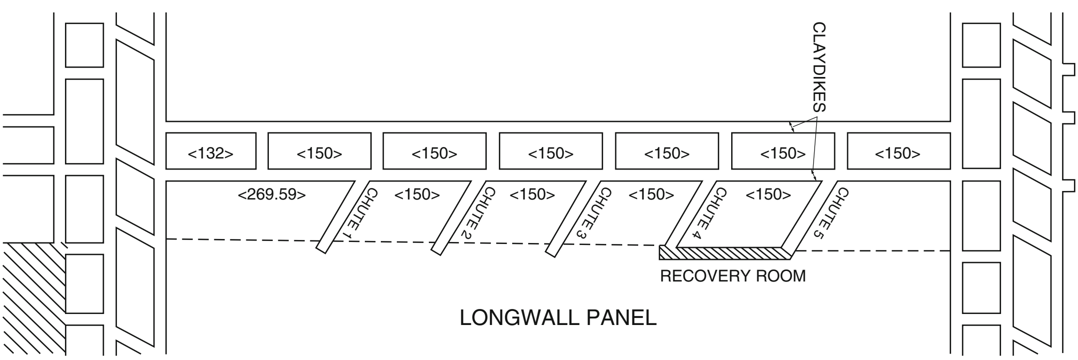


Fig. 9.27 An example of longwall recovery roadway layout utilising chutes (After Tadolini and Barczak 2004)

centres over the supports. In the tip-to-face region, bolt density may exceed one bolt per square metre. If these bolts are made from high tensile steel, caution is required that the bolt tails do not become projectiles if sheared off by the canopies of the powered supports as they are advanced. Some mines specify mild steel bolts for longwall face bolt up to mitigate this stored energy risk. In shallow mines, the recovery area may be pre-supported by installing cable bolts from the surface (reference, for example, R. Butcher and Kirsten 1999).

In weak to moderate strength roof strata environments, consideration should be given to installing one to two rows of cable bolts along the final face recovery position, with the face-side row angled over the ribline. It is advisable that cable bolts be at least 6 m long, with longer and higher capacity bolts recommended in areas affected by periodic weighting and geological features.

As mine personnel and equipment are exposed to the coal face in a confined space in circumstances where the ribline is subjected to abutment stress over an extended period of time, it is essential that the coal face is supported. Extending the roof screen down the ribline and pinning it with rib bolts is one commonly employed method (Fig. 9.29). Payne (2008) reported that at Crinum Mine, Australia, a flexible rib spray product in combination with friction bolts worked reasonably well in conditions where face spall had been a problem when using synthetic grid mesh and resin anchored rib bolts.

The capacity of the floor strata to sustain abutment stress, horizontal stress, and high, repetitive vehicle axle loads is another important geotechnical consideration when planning a longwall recovery. Deterioration in floor conditions can directly and indirectly lead to delays in recovering equipment when, from a geotechnical perspective, time may be of the essence. Therefore, a roadway maintenance scheme that includes provision for concreting poor sections of roadways prior to commencing to recover the longwall is advisable.

Having designed and implemented support systems in and about the longwall recovery face, these need to be complemented with monitoring instrumentation and timely data processing procedures in order to detect any deviations from as-designed and to underpin a robust Trigger Action Response Plan. Commencement of recovery of longwall face equipment should be commensurate on all of the preceding measures being in place. Ground control problems during the extrication of the powered supports usually arise from one or a combination of goaf flushing into the working area; roof convergence in and about the recovery site; physical interference between supports during extraction; and roof deterioration along the egress path.

The manner and sequence in which the powered supports are extricated from the goaf can vary, depending on factors such as ventilation, number of take-off points, type of recovery equipment, ground conditions, and room to manoeuvre. Within a recovery length, supports

Fig. 9.29 Looking inbye on a longwall recovery roadway showing meshed coal face and roof, rock bolts and cable bolts, walker (buttress) chocks and Link-n-Lock® timber chocks to maintain a return airway

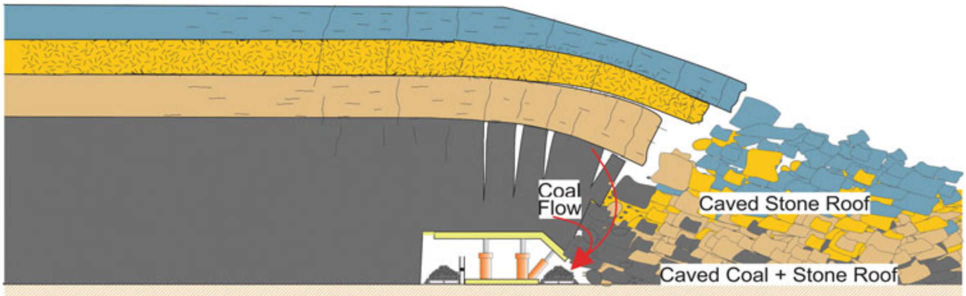


Fig. 9.30 A conceptual vertical cross-section through a LTCC face

may be recovered in a linear or alternate sequence. It is common to set two or three powered supports longitudinally to the coal face at the site of support recovery in order to provide increased protection from goaf flush and to act as a break off point for goaf falls (Fig. 9.29). These are referred to variously as **buttress chocks**, **walker chocks** or **walking shields** and sometimes comprise mobile roof supports (MRS). If a return airway has to be kept open along the recovery face, timber chocks and props may also need to be installed. More in-depth appraisals of the geotechnical planning and design to support longwall face recovery operations are provided by Hill (2006, 2010).

9.9 Other Longwall Variants

9.9.1 Longwall Top Coal Caving

Longwall Top Coal Caving (LTCC) involves longwall mining the lower portion of a thick coal seam and drawing off the upper portion as it caves into the goaf (Fig. 9.30). The method evolved in Europe, where it was known as **soutirage** or **integrated longwall mining with sublevel caving**, and incorporated either a chute in the canopy of each powered support to direct caving coal on the face conveyor or a retractable gate in the rear shield plates to control coal flow onto a second conveyor at the rear of the supports (Fig. 9.31). In many early operations, a second conventional longwall face extracted a slice from the top of the thick seam at least one month in advance of LTCC operations, in a method known as **non-integrated longwall mining with sub-level caving**.

Fig. 9.31 A Hemscheidt LTCC face with gates incorporated into the shield plates, operating at Velenje, Slovenia (After Galvin 1978)

The concept of LTCC was introduced into China in 1982 (Cai et al. 2004). The Chinese made significant improvements to the rear loading technology by installing supplementary, hydraulically activated, shield plates behind the supports to protect the rear conveyor, to enable it to be retracted independently of the front conveyor, and to provide better control over the drawing of coal. An example of such a support is shown in Fig. 9.32. The method now finds application to seam thicknesses typically ranging from 4.5 to 12 m, with the upper extraction height being constrained by geotechnical considerations and the reach distance of the rear conveyor into the goaf to recover coal flow from greater heights.

A considerable amount of the early research into LTCC was undertaken in France in the 1960s and 1970s and produced the model of coal flow and corresponding displacements shown in Fig. 9.33 (Adams 1976). A range of

Fig. 9.32 A modern Chinese LTCC powered support

investigations at Chinese mines has produced similar generic conclusions.

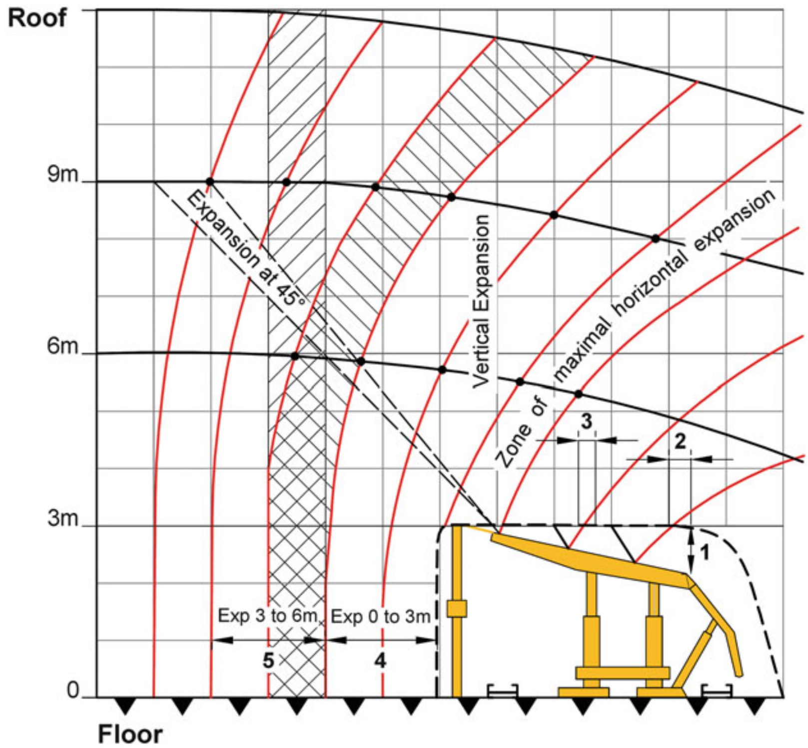
The success of LTCC lies in the friability and caveability of the coal, which is a function of coal material strength; cover depth; structural inclusions (joints, cleat etc.); stone bands within the top coal; and the nature of the overlying roof strata. Depth, as encapsulated in face abutment stress, provides the driving force to fracture the coal. The lack of abutment stress when mining beneath the goaf of an upper slice is one of the main reasons why this version of the technique is no longer utilised. In-seam bands can impede caving and block draw points.

The nature of the immediate stone roof determines the frequency and area of caving. LTCC effectively extracts the parting and removes the cushioning to subsiding upper strata. Risks of windblast and equipment damage arise if caving of this material does not follow closely behind face advance. Delayed caving of the immediate and/or upper stone roof also gives rise to the risk of periodic weighting. Moodie and Anderson (2011) reported that during periodic weighting episodes at Austar Coal Mine, Australia, caving operations were temporarily suspended and the speed of retreat was increased. This resulted in less intense loading events spread over a longer period of retreat. The authors also reported that a stress rotation occurred in the tailgate, consistent with the skew roof model of Tarrant (2005a), which may be exacerbated by the increased extraction associated with LTCC.

Caving must be restricted towards the ends of the longwall face to maintain the integrity of the gate ends, interpanel pillars and structures installed within cut-throughs, such as ventilation stoppings and seals. It must also not be permitted to approach or over-run the face, with removal of lateral confinement being conducive to face falls. The gate ends require very careful management from a local ground control perspective as the second conveyor is lagging in the goaf and persons require safe access to this equipment.

9.9.2 Miniwall

A **miniwall** is a longwall mining variant in which a single ranging arm shearer works to a blind end, with ventilation being returned over the goaf. Face length is restricted to the order of 50 m and two panels may be extracted from the one set of main developments. The method has found application in shallow situations in Australia where panel width has had to be restricted in order to limit surface subsidence. Its success is dependent on a strong immediate roof strata which does not cave to the extent of choking off return ventilation over the goaf and does not present a windblast risk if caving is delayed. More detailed information is to be found in McKendry and Simes (1987), Simes (1989) and Hedley and McDonald (1993).



Characteristics						Results of Measurements					
Mines	**Face**	**Seams**	**Depth (m)**	**Thickness (m)**	**Cutting Height (m)**	**1**	**2**	**3**	**4**	**5**	**Expansion Total %**
Darcy	Taille D	4th	800	12.3	2.8	1155	254	120	0.38	3.35	20
Rozelay	T.3b	2nd	310	12	2.4	660	160	38		2.1	9
Ricard	Taille 2	1st	840	4.5	2.5	587	131		0	0.5	

Notes: Numbers in the Figure are the locations where the measurements were taken:

1. Convergence mm;
2. Convergence per metre of face advance;
3. Displacement of roof beams between setting and advancing;
4. Expansion 3 to 6m;
5. Expansion 0 to 3m

Fig. 9.33 Outcomes of research undertaken in France in the 1960s and 1970s into coal flow associated with LTCC (After Adams 1976)

References

Adams, R. (1976). *French thick seams mining practices* (pp. 41–50). Paper presented at the symposium on thick seam mining by underground methods, Central Queensland Branch. AusIMM.

Ashwin, D. P., Campbell, S. G., Kibble, J. D., Haskayne, J. D., Moore, J. F. A., & Shepherd, R. (1970). Some fundamental aspects of face powered support design. *The Mining Engineer, 129*, 659–671.

Aziz, N. I., & Porter, I. (1985). *Strata control investigations at a 2.7 m high longwall face using 4/900 tonnes chock shield supports* (pp. 16–26). Paper presented at the international symposium on mining technology and science, Xuzhou, Jiangsu, China, Clausthal-Zellerfeld: Trans. Technical Publications.

Badr, S., Schissler, A., Salamon, M. D. G., & Ozbay, U. (2002). *Numerical modelling of yielding chain pillars in longwall mines* (pp. 99–107). Paper presented at the 5th North American rock mechanics symposium, Toronto, Canada, University of Toronto Press.

Barczak, T. M. (2006). *A retrospective assessment of longwall roof support with a focus on challenging accepted roof support concepts and design premises* (pp. 232–244). Paper presented at the 25th international conference ground control in mining, Morgantown, West Virginia University.

Barczak, T. M., & Tadolini, S. C. (2007). *Longwall shield and standing gate road support designs – Is bigger better?* Paper presented at the Longwall USA conference, Pittsburgh, 26.

Beer, G., & Meek, J. L. (1982). Design curves for roofs and hanging-walls in bedded rock based on "voussoir" beam and plate solutions. *Transactions of the Institution of Mining and Metallurgy, 91*, A18–A22.

Bruins, G. R. (1997). *Longwall experience and performance at New Denmark Colliery* (pp. 85–92). Paper presented at the 1st international underground coal conference, Sydney, UNSW/ACIRL.

Butcher, R., & Kirsten, M. (1999). *Experience with very shallow cover longwalls, Baal Bone Colliery* (pp. 91–105). Paper presented at the 2nd international underground coal conference, Sydney, UNSW/ACIRL.

Cai, Y., Hebblewhite, B. K., Onder, U., Xu, B., Kelly, M., Wright, B., & Kraemer, I. (2004). *Application of longwall top coaling to Australian operations* (p. 135). ACARP end of project report C11040. Brisbane: Australian Coal Association Research Program (ACARP).

Cliff, D., Rowlands, D., & Sleeman, J. (1996). *Spontaneous combustion in Australian coal mines* (C. Bofinger Ed.). Brisbane: Safety in Mines Testing & Research Station (SIMTARS).

Cloete, C. J. (1980). Contribution to paper on experience in longwall mining at Coalbrook Collieries. *Journal of South African Institute Mining and Metallurgy, 80*(1), 34–36.

Colwell, M. G., Frith, R. C., & Mark, C. (1999). *Calibration of the Analysis of Longwall Pillar Stability (ALPS) chain pillar design methodology for Australian conditions* (pp. 282–290). Paper presented at the 18th international conference ground control in mining, Morgantown, West Virginia University.

Creech, M. (1996). *Microseismic monitoring, windblasts and longwall mining under massive strata at Newstan Colliery* (pp. 239–249). Paper presented at the symposium on geology in longwall mining, Sydney, Coalfield Geology Council of New South Wales.

Diederichs, M. S., & Kaiser, P. K. (1999). Tensile strength and abutment relaxation as failure control mechanisms in underground excavations. *International Journal of Rock Mechanics and Mining Science, 36*(1), 69–96.

Doyle, R., & Gale, W. (2004). *Successful use of a stress relief roadway at Appin Colliery* (pp. 233–238). Paper presented at the Coal 2004: Coal operators' conference, Wollongong. AusIMM/University of Wollongong.

Esterhuizen, G. S., Mark, C., & Murphy, M. M. (2010). *Numerical model calibration for simulating coal pillars, gob and overburden responses* (pp. 46–57). Paper presented at the 29th international conference ground control in mining, Morgantown, West Virginia University.

Fabjancyk, M., Tarrant, G., Nemcik, J., & Gale, W. (2006). *Review of barrier pillar, bleeder, chain pillars in weak strata and thick coal* (p. 239). ACARP end of grant report C12011. Brisbane: Australian Coal Association Research Program (ACARP).

Fisher, M. (2001). *Longwall mining through old pillar workings; A case study*. Paper presented at the 3rd international underground coal conference, Sydney, 15. UNSW/ACIRL.

Frith, R. (2005). *Half a career trying to understand why the roof along the longwall face falls in from time to time* (pp. 33–43). Paper presented at the 24th international conference ground control in mining, Morgantown, West Virginia University.

Frith, R., & Creech, M. (1997). *Face width optimisation in both longwall and shortwall caving environments* (p. 96). ACARP end of grant report C5015. Brisbane: Australian Coal Association Research Program (ACARP).

Gale, W. J. (2001). *Prediction and management of adverse caving effects on longwall faces* (p. 124). ACARP end of grant report C7020. Brisbane: Australian Coal Association Research Program (ACARP).

Gale, W. J. (2004). *Rock fracture, caving and interaction of face supports under different geological environments. Experience from Australian coal mines* (pp. 11–19). Paper presented at the 23rd international conference ground control in mining., Morgantown, West Virginia University.

Gale, W. J. (2009). *Matching longwall support design to industry requirements in weak ground* (p. 55). ACARP end of grant report C14030. Brisbane: Australian Coal Association Research Program (ACARP).

Gale, W. J. (2014). *Update of stress concentration effects about longwall panels for improved mine planning* (p. 103). ACARP end of grant report C20031. Brisbane: Australian Coal Association Research Program (ACARP).

Gale, W. J., & Matthews, S. M. (1993). *Stress control method for optimised development and extraction operations* (p. 200). NERDDP report no. P282N, ACARP end of grant report C1301. Brisbane: Australian Coal Association Research Program (ACARP).

Galvin, J. M. (1978). *Report on overseas visit to thick seam operations in Canada, Austria and Yugoslavia* (Internal report). Johannesburg: Chamber of Mines of South Africa Research Organisation.

Galvin, J. M. (1983). *The development of the theory of dolerite sill behaviour* (pp. E127–E132). Paper presented at the 5th congress of International Society Rock Mechanics Melbourne, A.A. Balkema.

Galvin, J. M. (1996). *Impact of geology on longwall mining: A 20 year case study* (pp. 21). Paper presented at the symposium on geology in longwall mining, Coalfield Geology Council of New South Wales.

Galvin, J. M. (1997a). *Longwall support analysis, Southern Colliery* (Consulting report to Capricorn Coal Management). Sydney: Unisearch, UNSW.

Galvin, J. M. (1997b). Task report: Pillar design. *International Committee for Coal Research and the Australian Coal Association research mission to South Africa* (Vol. I & II, 0 7334 1560 1, pp. Vol. I – 55, Vol. II – 109). Sydney: International Committee for Coal Research (ICCR) & Australian Coal Association (ACA). ICCR/ACA.

Galvin, J. M., Steijn, J. J., & Wagner, H. (1982). Chapter 4: Rock mechanics of total extraction. In C. J. Fauconnier & R. W. O. Kirsten (Eds.), *Increased underground extraction of coal* (Monograph series 4, pp. 69–110). Johannesburg: SAIMM.

Galvin, J. M., Anderson, I., & Stothard, R. (1991). *Report of study tour investigation into pillar extraction at shallow cover, South Africa and United States of America* (p. 114). Sydney: NSW Coal Association.

Gardner, L. (1987). *Recovery of longwall 6 face at Pacific Colliery*. Presentation/Lecture to mine managers, mine operators and students of University of Newcastle, Australia.

Grice, A. G., Finn, T., & Smith, P. A. (1999). *Innovative use of backfill to enable continuous longwall mining through old headings at Homestead Colliery, Wambo Mining Corporation* (pp. 75–83). Paper presented at the 2nd international underground coal conference, Sydney, UNSW/ACIRL.

Gupta, R. N., & Ghose, A. K. (1992). *Strata support interaction on a powered support longwall face under a massive dolerite sill – A study* (pp. 140–149). Paper presented at the 11th international conference ground control in mining, Wollongong, AusIMM.

Hanson, B., Ochsner, R., Stankus, J. C., & Li, X. (2014, September). Signal Peak Energy makes longwall recovery history. *Coal Age*, pp. 22–25.

Hatherly, P. J., Luo, X., Dixon, R., McKavanagh, B., Barry, M., Jecny, Z., & Bugden, C. (1995). *Roof and goaf monitoring for strata control in longwall mining* (p. 76). ACARP end of grant report C3067. Brisbane: Australian Coal Association Research Program (ACARP).

Hebblewhite, B. K., & Galvin, J. M. (1996). *Issues affecting longwall chain pillar design* (pp. 193–197). Paper presented at the symposium on geology in longwall mining, Sydney, Coalfield Geology Council of New South Wales.

Hebblewhite, B. K., & Simpson, J. B. (1997). *Wider longwall faces may not be better* (pp. 201–218). Paper presented at the 11th international conference coal research. Republished 2001 Transactions of Institute Mining and Metallurgy, Section A, 110 (A11–A17), Calgary, International Committee Coal Research.

Hedley, T. P., & McDonald, A. (1993). Chapter 22: Longwall and shortwall. In A. J. Hargraves & C. H. Martin (Eds.), *Australasian coal mining practice* (Monograph series no. 12, pp. 359–390). Melbourne: AusIMM.

Henderson, P. G. (1980). Experience in longwall mining at Coalbrook Collieries. *Journal of South African Institute of Mining and Metallurgy, 80*(1), 22–33.

Hill, D. (2006). *Improving the efficiency of longwall face recoveries by managing the geotechnical threats* (p. 98). ACARP end of grant report C13022. Brisbane: Australian Coal Association Research Program (ACARP).

Hill, D. (2010). *Managing the geotechnical aspects of longwall face recovery* (pp. 84–87). Paper presented at the Coal 2010: Coal operators' conference, Wollongong, Illawarra Branch AusIMM.

Hookham, M. (2004, 13 August). Mining under sandstone. *International Longwall News*. Aspermont.

Hoyer, D. I. (2011). *Early warning of longwall weighting events and roof cavities using LVA software* (pp. 207–213). Paper presented at the 30th international conference ground control in mining, Morgantown, West Virginia University.

Humphreys, D., & Richmond, A. (1986). *Mining and ventilation practice in coal mines liable to spontaneous combustion*. Sydney: ACIRL.

Jones, G. (2008). *Longwall mining through an igneous plug and associated roadways*. Paper presented at the Longwall 2008, Hunter Valley, 24 Slides. IIR Conferences.

Kelly, M., & Gale, W. (1999). *Ground behaviour about longwall faces and its effect on mining*. ACARP end of grant report C5017. Brisbane: Australian Coal Association Research Program (ACARP).

Klenowski, G., Philliips, R. N., & Ward, B. (1990). *Strata control techniques for chock salvage, German Creek Mines, Queensland* (pp. 114–122). Paper presented at the Bowen Basin symposium 1990, Mackay, Qld, Qld Division, Geological Society of Australia.

MacDonald, R. J. (1997). *Significant weighting events on the longwalls in Phalen Colliery* (pp. 68–78). Paper presented at the 16th international conference ground control in mining, Morgantown, West Virginia University.

McKay, S. D. (1978). *Static analysis comparison of 2 leg versus 4 leg longwall supports*. Internal report (Unpublished). Chamber of Mines of South Africa Research Organisation.

McKendry, J. B., & Simes, M. W. (1987). *Gretley Colliery miniwall mining system* (pp. 126–149). Paper presented at the 1987 ACIRL/ACA underground mining seminar on factors influencing underground productivity, Sydney, ACIRL/ACA.

McKensey, B. K. (1988). *Installation roadway failure, Pacific Colliery*. Contribution to discussion at annual ACIRL/ACA underground operator's seminar. Sydney.

MDG-1006. (2011). *Spontaneous combustion management – Technical reference*. Sydney: NSW Government.

Medhurst, T. P. (2005). *Practical considerations in longwall support behaviour and ground response* (pp. 49–58). Paper presented at the Coal 2005: Coal operators' conference, Wollongong, Illawarra Branch AusIMM.

Mills, K. W., & O'Grady, P. (1998). *Impact of longwall width on overburden behaviour* (pp. 147–155). Paper presented at the Coal 1998: Coal operators' conference, Wollongong, Illawarra Branch AusIMM.

Minney, D. (1999). *Amcoal experience of longwalling through dykes* (pp. 85–90). Paper presented at the 2nd international underground coal conference, Sydney, UNSW/ACIRL.

Minney, D., & Karparov, K. (1999). *South African experience with mined chain pillars and remnant crush pillars* (pp. 107–114). Paper presented at the 2nd international underground coal conference, Sydney, UNSW/ACIRL.

Moodie, A., & Anderson, J. (2011). *Geotechnical considerations for longwall top coal caving at Austar Coal Mine* (pp. 29–39). Paper presented at the Coal 2011: Coal operators' conference, Wollongong, Illawarra Branch AusIMM.

NIOSH. (2008). *NIOSH Crandall Canyon report*. 24. Retrieved from website: http://www.cdc.gov/niosh/mining/NIOSHCrandallCanyonReport.pdf

NSA Engineering. (2000). *Review of current yielding gate road design approaches and applications in U.S. longwall operations*. Appendix II of Volume 1 of ACARP end of grant report no. C9018. Brisbane: School of Mining Engineering, University of New South Wales. ACARP.

Oyler, D., Frith, R., Dolinar, D., & Mark, C. (1999). *International experience with longwall mining into pre-driven rooms* (pp. 44–53). Paper presented at the 17th international conference ground control in mining, Morgantown, West Virginia University.

Payne, D. A. (2008). *Crinum mine, 15 longwalls 40 million tonnes 45 roof falls – What did we learn?* (pp. 22–43). Paper presented at the Coal 2008: Coal operators' conference, Wollongong, Illawarra Branch AusIMM.

Peng, S. S. (2008). *Coal mine ground control*. Morgantown: West Virginia University.

Potts, E. L. J. (1957). Underground instrumentation. *Quarterly. Colorado School of Mines, 52*(3), 135–182.

Pulse, R. (1990). *Ground stability evaluation around collapsed longwall face* (Memo to B.D. Owens, MSHA file #6017, p. 2).

Salamon, M. D. G. (1991). Behaviour and design of coal pillars. *The Coal Journal: Australian Journal of Coal Mining Technology and Research, 32*, 11–22.

Salamon, M. D. G., Oravecz, K. I., & Hardman, D. (1972). *Rock mechanics problems associated with longwall trials in South Africa* (p. 35). Research report no. 6/72. Johannesburg: Chamber of Mines of South Africa Research Organisation.

Salamon, M. D. G., Badr, S., Mendoza, R., & Ozbay, M. U. (2003). *Pillar failure in deep coal seams: Numerical simulation*. Paper presented at the 10th congress International Society Rock Mechanics, Sandton, South Africa, 8. ISRM.

Seedsman, R. W. (2012). *The development and application of a logical framework for specifying roof and rib support/reinforcement in Australian underground coal mines* (pp. 153–161). Paper presented at the 7th international symposium on rock bolting & rock mechanics, Aachen.

Seedsman, R., & Stewart, A. (1996). *A design method to assess massive strata for longwall face and subsidence behaviour* (pp. 227–231). Paper presented at the symposium on geology in longwall mining, Sydney, Coalfield Geology Council of New South Wales.

Simes, M. W. (1989). *Controlling surface subsidence using a miniwall – Gretley Colliery*. Paper presented at the 1989 ACIRL/ACA coal operators' seminar on longwalling in the 90's, Sydney, 20. ACIRL.

Simpson, J. B., Neal, P., & Frith, R. C. (1991). *The use of pre-driven roadways for longwall recoveries* (pp. 250–275). Paper presented at the 1991 ACIRL/ACA seminar on continuous miner systems – The future, Sydney, ACIRL.

Smart, B. G. D., & Redfern, A. (1986). *The evaluation of powered support specifications from geological and mining practice information* (pp. 367–377). Paper presented at the 27th US rock mechanics symposium.

Smart, B. G. D., Davies, D. O., & Isaacson, A. K. (1982). Conclusions from strata mechanics investigations conducted by Cardiff and Strathclyde Universities at longwall faces. *Mining and Engineering, 141*(246), 539–546.

Tadolini, S. C., & Barczak, T. M. (2004). *Design parameters of roof support systems for pre-driven longwall recovery rooms* (pp. 1–13). Paper presented at the 2004 SME annual meeting, Denver, Society of Mining, Metallurgy & Exploration.

Tarrant, G. (2005a). *Skew roof deformation mechanism in longwall gateroads – Concepts and consequence* (pp. 73–88). Paper presented at the Coal 2005: Coal operators' conference, Brisbane, Illawarra Branch AusIMM.

Tarrant, G. (2005b). *Standing supports: Its time for an engineering solution* (p. 374). ACARP end of grant report C12006. Brisbane: Australian Coal Association Research Program (ACARP).

Trueman, R., Lyman, G., & Cocker, A. (2008). *Managing roof control problems on a longwall face* (pp. 10–21). Paper presented at the Coal 2008: Coal operators' conference, Wollongong, Illawarra Branch AusIMM.

Van der Merwe, J. N. (1988). *Strata control aspects of the backfilling of a dyke ahead of a longwall face underneath incomplete pillar extraction at Sigma Colliery* (pp. 135–142). Paper presented at the SANGORN Symposium: Rock Mechanics in Africa, Johannesburg, South African National Group on Rock Mechanics.

Van der Merwe, J. N. (1989). *Strata control considerations and experience gained at the collieries of Sasol Coal* (pp. 23–36). Paper presented at the symposium on advances in rock mechanics in underground coal mining, Witbank, South African National Group on Rock Mechanics.

Vasundhara, C. J., Galvin, H. M., & Hebblewhite, B. K. (2003). *Failure mechanisms of weak floor strata near longwall panels and their impact on mining* (p. 110). ARC-SPIRT project C 89917615, UMRC research report RR2/03. Sydney: School of Mining Engineering, University of New South Wales and Capricorn Coal.

Wagner, H. (1994). *Coal strata control handbook*. Unpublished.

Wagner, H., & Steijn, J. J. (1979). *Effect of local strata conditions on the support requirements on longwall faces* (pp. 557–564). Paper presented at the 4th congress of International Society Rock Mechanics, Montreux. A.A. Balkema.

Whittaker, B. N. (1974). An appraisal of strata control practice. *The Mining Engineer, 134*(166), 9–22.

Wiklund, B., Kizil, M. S., & Canbulat, I. (2011). *Development of a cavity prediction model for longwall mining* (pp. 48–59). Paper presented at the Coal 2011: Coal operators' conference, Wollongong. Illawarra Branch AusIMM.

Wilson, A. H. (1975). Support load requirements on longwall faces. *The Mining Engineer, 134*, 479–491.

Winter, S., Robertson, B., Canbulat, I., & Laws, A. (2010). *Effectiveness of 1750t shields at Moranbah North Mine*. Paper presented at the 9th annual longwall conference, Hunter Valley, NSW.

Wold, M. B., & Pala, J. (1986). *The response of powered supports and pillars to initial longwalling under a strong main roof* (pp. 194–203). Paper presented at the conference on ground movement and control related to coal mining, Wollongong, AusIMM.

Zhang, Y., Mishra, J., Trackemas, J., Zeglen, E., Huff, C., Peng, S. S., & Chen, J. S. (2006). *Pre-driven longwall recovery room under weak roof conditions – Design, evaluation, and monitoring* (pp. 221–228). Paper presented at the 25th international conference ground control in mining, Morgantown, West Virginia University.