

3.3.5. These charts should be used with discretion: they span a wide, but by no means exhaustive range of conditions, and should be used to give a first approximation of the size of bracket pillar that might be appropriate in a given scenario.

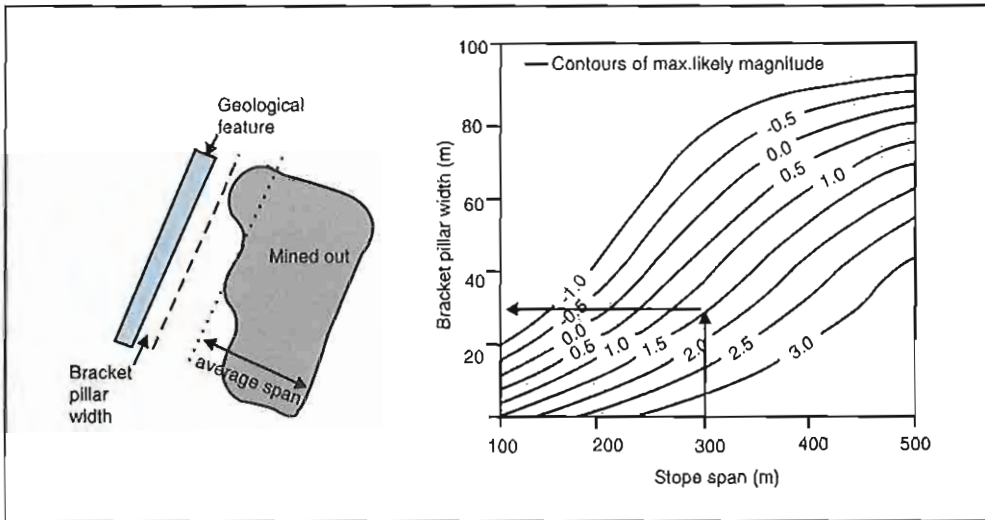


Figure 3.3.5 Example of a chart for assisting in design of bracket pillar width. A 'tolerable' event magnitude M is selected, and an indicative bracket pillar width corresponding to a given final mined span is then read off.

3.3.4 Concrete Pillars

The use of concrete pillars has been proposed as an alternative regional support structure; specifically, as an alternative to conventional strike stabilizing pillars, and an initial assessment of this scenario has been completed. The assessment also applies, in principle, to the use of concrete in place of other regional reef pillars, such as those used in 'sequential grid' layouts.

The potential advantages of such a system include:

- increased extraction, resulting in greater profitability and life of mine
- an engineered, high stiffness permanent support
- placement of concrete in *advanced headings* facilitates better identification and location of faults, and definition of the ore grade and its areal distribution
- reduction of seismic events associated with foundation failure of conventional pillars.

It is important that the concrete has a Young's modulus of at least 10 GPa, to ensure that regional ERRs are not negatively affected by the use of concrete pillars. The modulus of the concrete is largely controlled by the aggregate used, and it is feasible that this aggregate could be produced by crushing waste rock underground. The concrete mix should contain sufficient fines to allow the wet concrete to be pumped from the mixing plant to the stope for placement, and it is feasible that these fines could be obtained from existing tailings-based backfill materials or milled from waste rock. Full engineering studies on concrete mixes, pumping requirements and practical placement strategies have been completed: concrete mixes can be tailored with respect to strength, Young's modulus and pumpability for each application.

Indications are that the uniaxial compressive strength of the concrete does not need to be particularly high: in the order of 20 to 40 MPa. For the particular concrete mix tested and discussed below, the UCS was 35 MPa, with friction angle and cohesion of 27° and 11 MPa respectively.

Numerical modelling has shown that concrete pillars can reduce the regional inelastic closure in the hanging and footwall (Figure 3.3.6). In the case modelled, the reef pillar was 48 m wide on an extraction ratio of 85 %. The concrete pillar was assumed to be placed in two adjacent panels, resulting in two widths of 30 m each, with a gully between. The APS was lower in each of the concrete ribs, compared to that in the equivalent reef pillar. The ultimate face ERR value was also lower when concrete pillars were modelled compared to rock pillars, by some 22 % and 15 % for the Mohr-Coulomb and strain-softening models respectively, for the particular rock and concrete material properties modelled. This is not likely to change significantly with different material parameters, and probably gives a realistic view of the potential of this form of regional support.

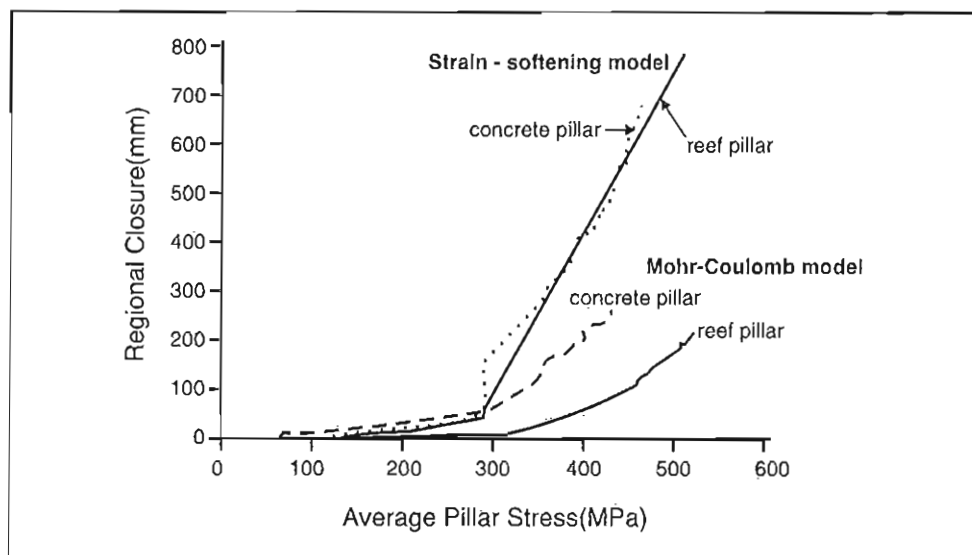


Figure 3.3.6 Modelled regional closure as a function of average pillar stress for reef and concrete pillars; Mohr-Coulomb and strain-softening material constitutive laws.

Although some yielding of the foundations occurred for both concrete and reef pillars in the modelling, this would be likely to occur between six months and two years later for the concrete pillars, depending on the mining rate. Concrete pillars, being softer and weaker than intact rock, are thus likely to lead to less punching damage in the foundations; both locally, adjacent to the pillar edge, and in the rock mass. The levels of pillar-related seismicity would thus be expected to be reduced. For this reason, it is thought to be advantageous to use a concrete of normal construction strength, about 20 to 40 MPa.

The above analyses excluded the possibility of providing additional confinement at the edge of the concrete pillar (for example, by the placing of a relatively narrow 'skin' of conventional backfill). This would increase both the stiffness and stability of the concrete pillar significantly.

An underground trial of a concrete block of width:height ratio of 4.5, showed that concrete is able to retain its laboratory performance in underground conditions. The block was able to retain stability after the area had sustained seismic events, and it was found that a significant amount of support resistance was generated even under excessive loading. While the edge of the block had failed, the core remained intact and was able to continue bearing a considerable load.

There has as yet been no full-scale implementation of concrete pillars in the industry, and the concept, while obviously showing considerable potential, is thus not yet fully proven. Important considerations in a full scale implementation would be:

- provision of sufficient width of concrete in the dip direction (for strike-oriented pillars)
- ensuring that the material and placement quality of the concrete is of a consistent, high standard
- ensuring that the concrete is placed as soon as possible to ensure minimum closure prior to and during curing, so avoiding premature destruction of the cementitious bonds; and ensuring that the concrete is placed in advance, or away from, current mining
- the early placement of concrete in a 'sequential grid' (SMDP) type layout between dip reef pillars would capitalize on an advantageously small-span low-closure environment, and would permit significant increases in mining spans and extraction ratios.

3.3.5 Backfill As A Support Medium

Backfilling is an important mining strategy to alleviate rock pressure problems in deep South African mines. Modern backfilling techniques were introduced at the beginning of the 1980s and the volume of backfill placed reached a maximum of about 4 million tons per year in 1995, but dropped to 3.5 million tons in 1997 commensurate with the decrease in mining production.

Backfill is used for both regional and local support in deep mines. As **regional support**, the backfill is required to reduce the volumetric convergence in the mined out areas and the high stresses at the face, so that the occurrence of seismic events can be reduced by achieving lower levels of energy release rate (ERR) and excess shear stress (ESS). The percentage extraction of the reef can also be increased when backfill is used as regional support in place of or to supplement stabilizing pillars. As **local support**, backfill is required to maintain the integrity of the fractured rock mass, to improve face and gully conditions, to reduce rockburst damage, and to provide better stope width control - Chapter 4.4.8.

Most of the backfill systems in operation in South Africa are based on the use of on-mine tailings. In order to maximise metal recoveries, there is a trend in the industry towards finer ore grinding. This, coupled with the rising demands on backfill quality from the viewpoint of local support, places great emphasis on the need for maintenance of stringent backfill quality controls. The *particle size distribution and slurry relative density* are the two most important variables that must be controlled in any backfill system, in order to maintain backfill quality in relation to the load bearing requirements, and the practicalities of preparation, hydraulic transportation and placement. Particularly when cementitious or other additives are used, a monitoring system is required to ensure that the composition is kept to specifications.

(a) Backfill types in South Africa.

South African gold, platinum and base metal mines use a variety of backfill types. These include:

- (i) cemented full plant tailings (Cem FPT);
- (ii) uncemented classified tailings (CCT);
- (iii) cemented classified tailings (Cem CCT);
- (iv) tailings-based backfills blended with aggregate (e.g. CCT / 70% Agg);
- (v) cemented equivalents of tailings/aggregate fills (e.g. Cem CCT / 70% Agg);
- (vi) comminuted waste backfill < 10 mm max. particle size (CMW).

At present, uncemented classified tailings (CCT) is the most common type of backfill placed in South African gold and platinum mines. Figures 3.3.7a and b show the cumulative particle size distributions of typical South African backfill materials. The particle size distribution is an important parameter in the design of any backfill system, as this is related not only to the backfill support capabilities but also to the hydraulic transportation and placement behaviour of the backfill.

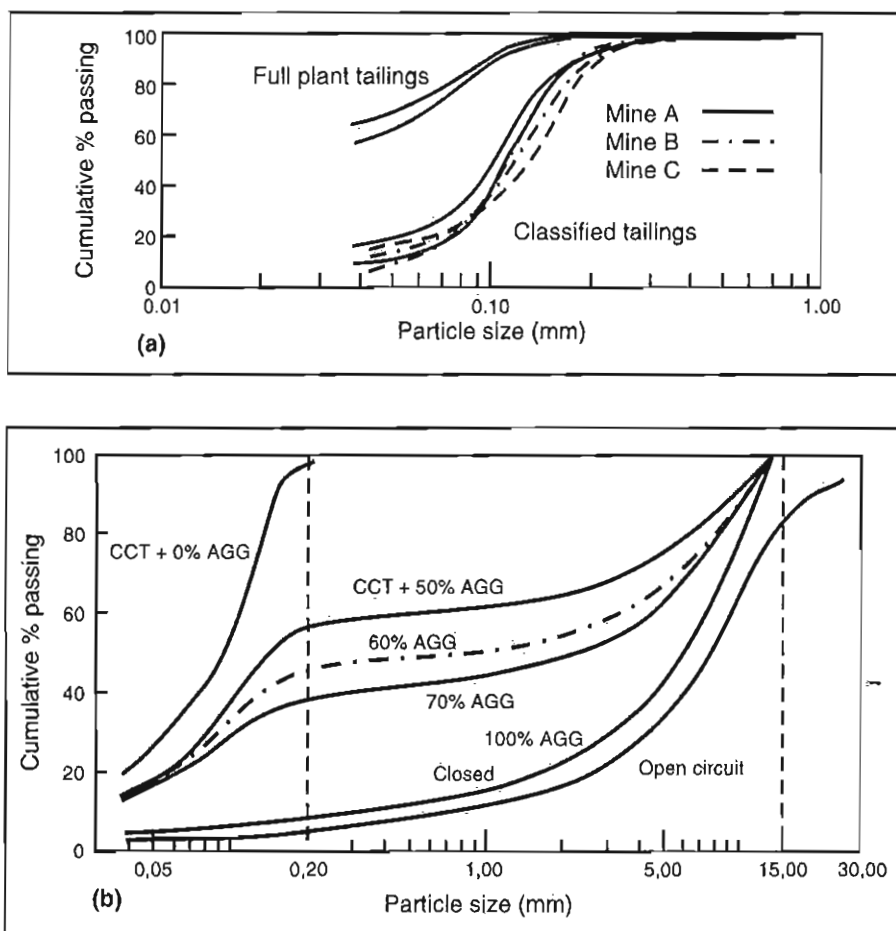


Figure 3.3.7 Particle size distribution curves of **a:** FPT, CCT backfills; **b:** tailings/aggregate backfills.

(b) Backfill porosity and load bearing properties.

There is a direct relationship between the particle size distribution of a backfill material and its minimum porosity ($n_{\min}$) as prepared in the laboratory. The porosity in this context is defined as the volume of voids (water plus air) contained in the interstices between the solid particles, divided by the total volume (water plus air plus solids) - Figure 3.3.8.

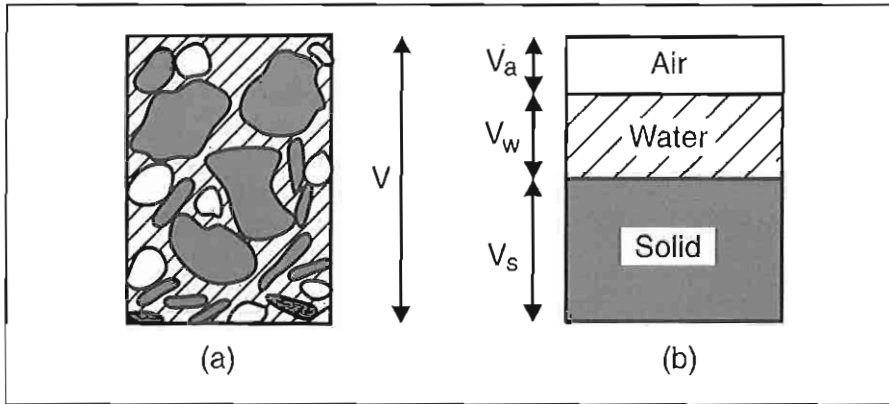


Figure 3.3.8 Porosity of backfill

There is a further direct relationship between the minimum porosity of a backfill and its load bearing capabilities. This is demonstrated in Figure 3.3.9 which shows the stress-strain response of various backfill types with different starting porosities corresponding to the different starting particle size distributions. It can be seen that the backfills with low porosities take stress with considerably less strain than do those with the higher porosities.

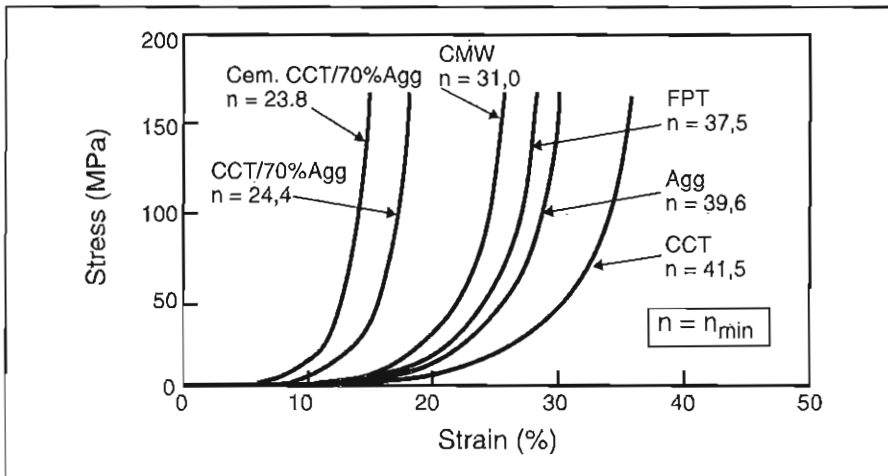


Figure 3.3.9 Backfill compression curves

Figure 3.3.10 shows the loading behaviour of cemented backfill, as compared to a normal 'cohesionless' backfill (a CCT in this example). It can be seen that the

cemented backfill provides a measure of compressive strength at low deformation. This initial strength provides support to the hangingwall and self-standing ability when cemented backfills are used in high stoping widths. The development of this initial strength relates to the type and amount of binder added to the backfill and the curing period available. As stope closure takes place in deep level mining, the cementitious bonds are broken (typically at about 1 MPa), and the backfill then behaves as if it were an uncemented fill having a higher fines content depending on the quantity of binder involved.

Accelerators, e.g. sodium silicate, are sometimes added to develop strength quickly. However, the use of certain accelerators in cemented backfills, while improving strength and reducing immediate water drainage into the stope, cause droplets of water to be locked within the cellular binder structure – thereby increasing the possibility of *liquefaction* of the backfill upon rapid (seismic) loading.

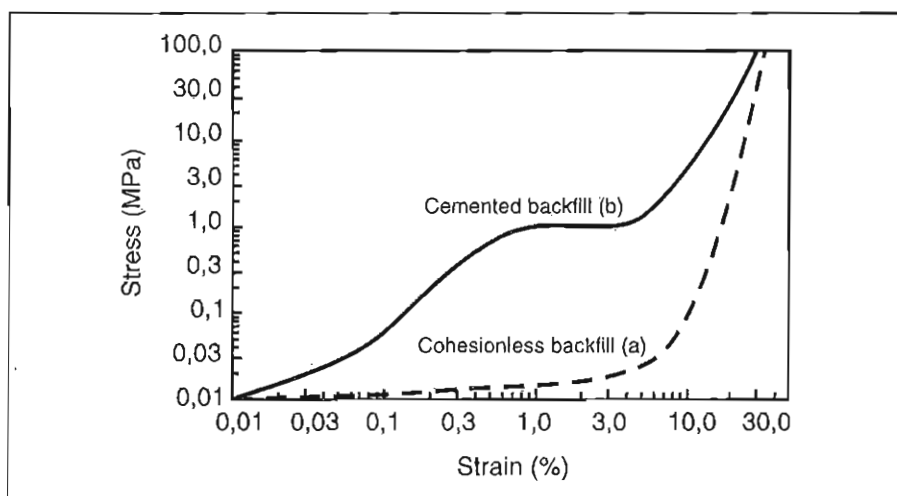


Figure 3.3.10 Compression curves of cemented and uncemented backfills plotted on a log scale

(c) *Hydraulic Transportation*. There is a conflict between the properties of backfill that allow practical and cost-effective hydraulic transportation, and those properties which best satisfy the requirements of placement. Backfills displaying slow settlement behaviour, i.e. a relatively high proportion of ultra fine material, may be beneficial for cost-effective hydraulic transportation, but can seriously impair placement and drainage. In the design of backfill systems it is important that satisfactory compromises be made between these conflicting requirements.

Figure 3.3.11 illustrates the effect of the particle size distributions of FPT and CCT backfills on pipeline pressure losses, as a function of mean slurry flow velocities. It can be seen that at equivalent mean velocities and solids concentration by volume (or slurry relative density, e.g. $RD = 1.85$), pipeline pressure losses for the FPT are up to five times higher than for the CCT. In general, Full Plant Tailings type backfills, with uniform high fines content, exhibit pipeline pressure losses that are considerably higher than their classified tailings counterparts.

For typical classified tailings, the effect of the backfill slurry relative density on pipeline pressure losses for a constant flow rate (2 m/s) and pipe diameter, is shown in Figure 3.3.12. The rapid rise in pipeline pressure losses above the indicated 'critical transition density' RD_t is indicative of a change of slurry flow behaviour from a settling flow regime to a non-settling or homogeneous flow regime. RD_t plays a vital role in the selection of practical hydraulic transportation parameters, allowing decisions relating to the transportation of backfills at the highest possible slurry relative densities, yet within practical pipeline pressure loss ranges.

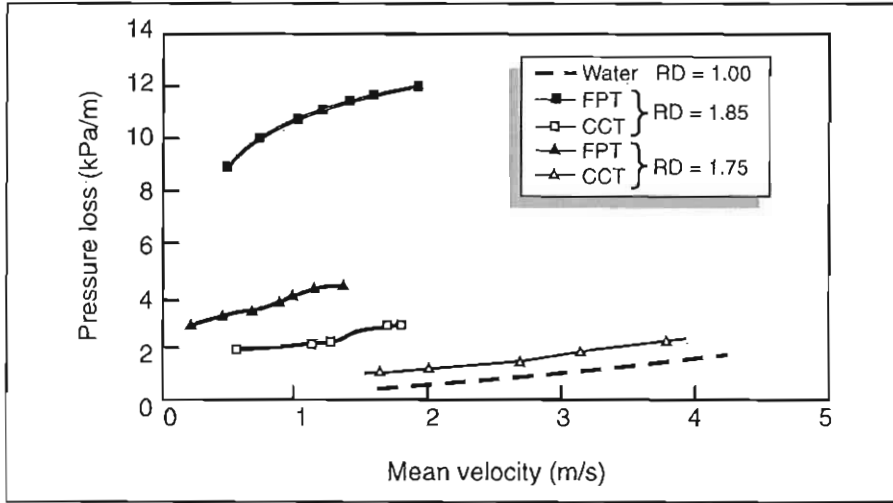


Figure 3.3.11 Dependency of pipeline pressure losses on backfill particle size distribution, relative density, and slurry velocity.

A relationship exists between RD_t and the minimum porosity n_{min} of backfills. This relationship provides a useful guide for predicting RD_t from n_{min} as measured in the laboratory.

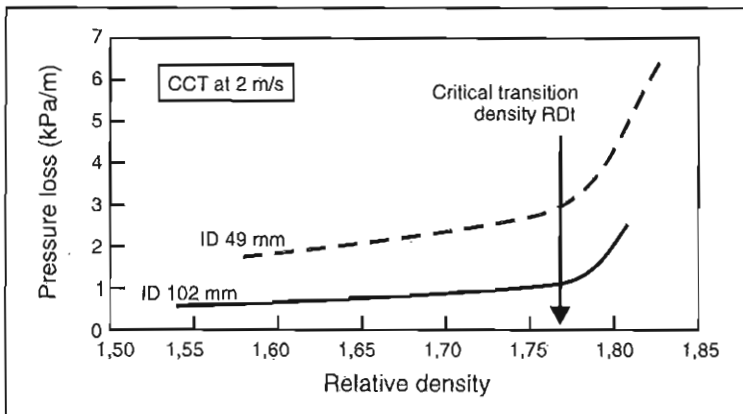


Figure 3.3.12 Dependency of pipeline pressure losses on backfill slurry RD

There is a distinct incentive to place backfill at the highest possible slurry relative density: to reduce solids losses to a minimum, to avoid having to pump large quantities of water to surface after placement, and to minimise the safety hazard of mud and water in the stopes, gullies and boxholes.

(d) Backfill for regional support.

The introduction of backfill material into the back-areas of a stope is a strategy for accomplishing regional support. Figure 3.3.13 shows the calculated beneficial effects of backfill on ERR and ESS levels. The behaviour of backfill as a regional support material is largely governed by its placed porosity n_0 : the lower the better. Alternatively, a parameter known as the 'figure of merit' M_f (defined as the mean strain undergone by a fill in reaching its design load) directly describes a fill's quality in terms of its ERR reduction potential as shown in Figures 3.3.14a and b. 'Stiff' or 'high-quality' fills are characterised by low values of porosity or M_f , and have the desirable property of rapidly building up large reaction stresses under small deformations. Fill quality can, to a great extent, be engineered by the appropriate blending and grading of raw materials, and by the use of binders.

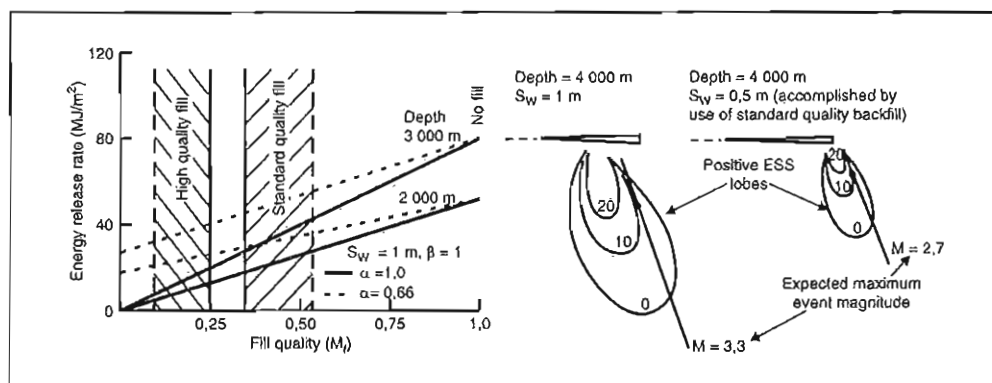


Figure 3.3.13 Calculated effect of backfill on ERR and ESS levels (symbols α and β are explained in the text)

Apart from a fill material's intrinsic quality, three other variables control a fill's performance in practice:

- (i) The fraction α of the stoping width at which the fill is placed. Low values result if gaps are left between the fill and hangingwall, and particularly if the fill is placed too far back from the face such that premature closure limits the fill width. Figure 3.3.14b illustrates the sensitivity to this parameter and the *importance of keeping fill placement as close to the face as possible*.
- (ii) The fraction β of the area of the stope covered by fill. Figure 3.3.14b shows that there is little penalty in using fill ribs covering as little as 20% of the mined area. However, modern practice, from the point of view of local support benefits, favours 60% filling or more.
- (iii) The stoping width S_w . The smaller the stoping width, the more effective backfill becomes in reducing ERR to acceptable levels. In practice, standard quality fills offer reduced regional support benefits (compared to partial extraction methods) for stoping widths greater than 2 m, though the *local support* potential continues to be high.

These variables affect the performance of any fill system, whether it is intended for ERR control, for other aspects of regional support such as in shaft protection, for middling stabilisation in multireef extraction situations, or as local support. Control of placed porosity, avoidance of gaps or shrinkage and maintenance of placement close to the working face are all important operational requirements. The local support aspects of fill are discussed in Chapter 4.4.8.

(e) *Hybrid pillar-backfill systems.* For mines planning to extract extensive blocks of ground (>1000 m span) at depths greater than about 3000 m, backfill alone cannot achieve satisfactory regional support or ERR control as is shown in Figure 3.3.15. Theoretical studies have however indicated that hybrid systems involving combinations of stabilizing pillars and backfill can offer significant advantages in terms of ERR reduction and increased extraction in ultra-deep mining situations, as illustrated in Figure 3.3.15.

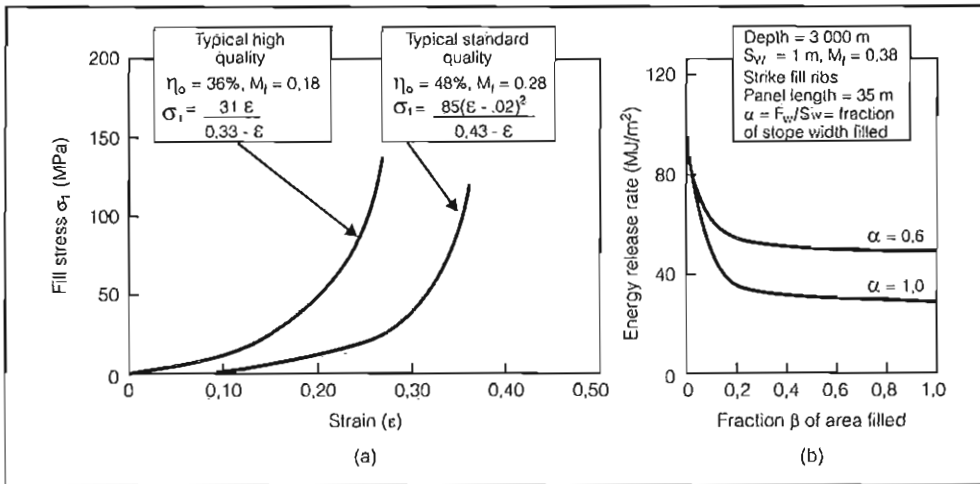


Figure 3.3.14 a: Fill stress-strain behaviour; b: Effect of fill placement parameters

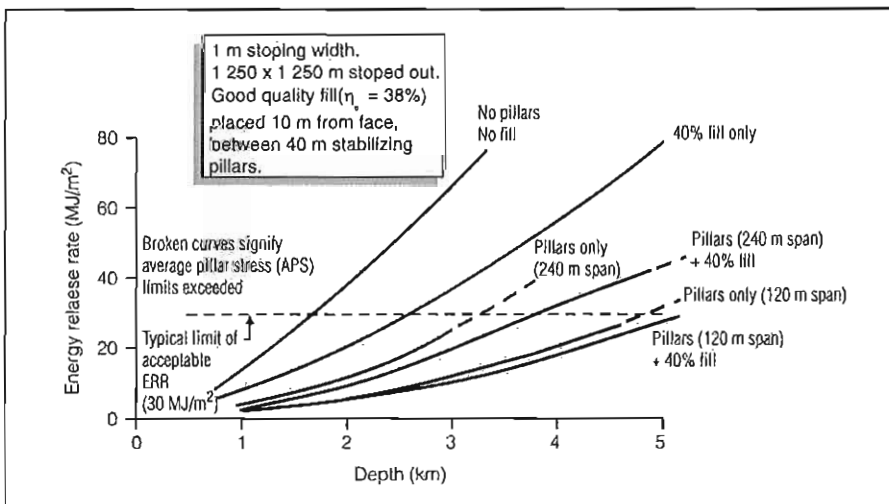


Figure 3.3.15 Hybrid pillar-backfill combinations

A second, more radical concept could conceivably permit replacement of stabilizing pillars with high-quality (cemented or concrete) backfill at great depth. However, in order to maintain a high fill width fraction α and to facilitate adequate curing of the cement, advance development of reef headings into virgin ground would be required, preferably at low stope width, into which high-quality fill would be introduced in this relatively low closure rate environment. Normal mining would then follow behind, protected by these strike (or dip) 'stabilizing pillars' of stiff cemented fill material. This concept was discussed in the placement of concrete pillars (section 3.3.4).

(f) In situ effectiveness of backfill as a regional support medium

Although theoretical studies indicate that backfill has significant potential as regional support to reduce seismicity, it has been very difficult to prove this in practice. Case studies were carried out on three mines in which the seismic 'b' value parameter (the ratio of large to small events) and γ (a measure of the seismic deformation associated with stope closure due to mining) were used to quantify the difference in seismicity levels between filled and unfilled areas. To show that backfilled areas experience a reduction in seismicity or seismic-energy release compared with unfilled mining panels, the effective changes in these parameters would be demonstrated by an increase in the 'b' value and a reduction in γ . The results of these case studies were inconclusive. It was found that geology and geological discontinuities, which strongly determine the level of seismicity, varied significantly in filled and unfilled areas where these case studies were conducted. At present, the results obtained indicate that, for backfill to provide regional support so that the release of seismic energy is to be significantly reduced, it is necessary for mines to place reasonably high quality (low porosity) backfill routinely over long periods of time as mining takes place. These requirements have not yet been fulfilled. Nevertheless, evidence is accumulating that indicates a reduction in the severity of seismicity associated with stabilizing pillar foundation failure where backfill is placed in the stopes between pillars. Good quality placement is required to enhance this benefit, particularly in the panels adjacent to the pillars.

3.4 STANDARD LAYOUTS

As introduced in Chapter 2, the formats of **standard** mining layouts are dictated to a major extent by the **effective mining depths** in question. Other issues (such as the requirement for wide- or multi-reef mining, or the negotiation of adverse geological situations) will demand modified, or even radically different, mining layouts for their most effective resolution, and these cases will be discussed in sections 3.5 and 3.6.

3.4.1 Shallow/Overstoped Areas (effective mining depth <1000 m)

Scattered mining methods are used extensively in shallow environments, where detailed stoping layouts tend to have little direct effect on the incidence of rockfalls, assuming ground conditions are reasonable and that appropriate in-stope pillar layout and support strategies have been adopted. There is considerable latitude of choice in the orientation of stopes and faces and in the layout of gullies; but there are some important provisos:

- Shallow layouts must take cognisance of any *major geological weaknesses*, especially faults/joints or dykes that cut the hangingwall beam. To avoid severe

collapses, these structures generally need to be supported by additional pillars (on the footwall side of dipping planes of weakness), protected by bracket pillars, or enveloped into the mine's barrier pillar systems.

- In the case of *minor geological weaknesses*, faces should be adequately supported and preferably swung to avoid breasting through these structures. In severe panel-collapse situations a diametrically different strategy can be tried: to switch the mining direction so that the troublesome joints are *parallel* to the face, thus reducing the span of joint-bounded hangingwall beams to just the distance between in-stope chain pillars. If such a strategy is implemented, the potential for FOGs close to the face is increased and thus face area support needs to be upgraded accordingly.
- Special layout precautions are generally required in remnant extraction (Section 3.5.1), and in wide-reef or multi-reef extraction exercises (section 3.6).

As outlined in Chapter 2.2.1, because of the presence of extensive tensile zones and intrinsically unstable hangingwall conditions (which, if improperly controlled, can lead to problems ranging from large FOGs, through severe hangingwall beam unravelling, to massive stope backbreaks), shallow mines must use **in-stope pillars** as an integral part of their overall layout. The following sections discuss the design and disposition of these structures and other aspects of shallow mine layouts.

(a) *In-stope pillar system design*

The design of shallow mining pillar layouts needs to address the entire pillar *system*: the span between pillars, the pillars themselves including their foundations, and the (worst-case) loading to which the pillars will be subjected.

A basic consideration is the **safe span** to be used between the pillars. This is mainly dictated by the intrinsic rock mass condition of the hangingwall beam. In near-surface operations the rock is usually *weathered*, and this can be exacerbated by ingress of water through opened-up jointing. Elsewhere, the hangingwall may be heavily *laminated* (banded ironstones, for example), or densely permeated with low-friction/cohesion *joint sets*. In these conditions (and always taking into account that appropriate local support systems will be in place) safe spans may be no more than 4 to 10 m, and room and pillar layouts will be appropriate. In many hard-rock situations, however, conditions are less hostile, and panel spans of 30 m or more provide a safe and productive mining environment. The definition of appropriate in-stope spans (for given geotechnical conditions) is currently largely empirical. Ongoing research is aimed at providing more quantitative means for the determination of stable spans between pillars, entailing the development of a rock mass rating scheme tailored for narrow tabular stoping environments – Chapter 10.3.5.

As mining depths increase, the pillar design requirements in shallow mines change significantly – Figure 3.4.1. In **very shallow** stopes (depth less than some 300–400 m), in-stope pillars need to be designed to carry the full weight of overburden *up to surface*, with a safety factor of not less than 1.6. These in-stope pillars are **non-yield pillars** – Figure 3.4.2. Sections 3.2.7/3.2.8 summarised the criteria and concepts involved in their design. **Barrier pillars**, which are not essential ingredients provided adequate factors of safety exist on the in-stope pillars, nevertheless form a type of insurance policy against unforeseen pillar runs and potential mine-wide collapses. Their use in very shallow operations is therefore advocated.

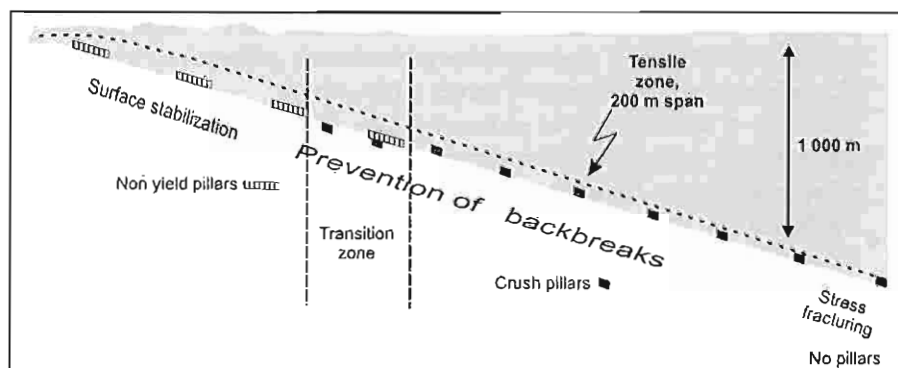


Figure 3.4.1 In-stope pillar types appropriate to different (shallow) mining depths.

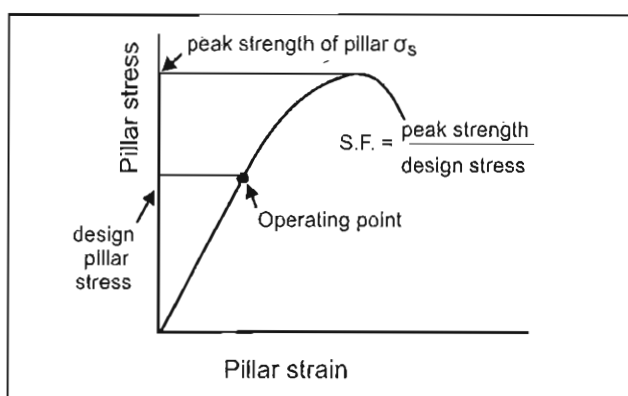


Figure 3.4.2
Design of non-yield
pillars.

In **deeper stopes** (say 600-1000 m BS), the continued use of non-yield pillars is possible but only at reduced extraction ratios due to the heavier cover loads that need to be borne. An attractive alternative is to use **barrier pillars** to reduce the regional spans and thus the height of the tensile zones – Figure 3.4.3. The in-stope pillars now need only support a reduced thickness of hangingwall strata (at least up to any potentially unstable weak parting plane), and this is easily accomplished by exploiting the residual strength of small **crush pillars** – Figure 3.4.4. These pillars, which typically have a nominal width:height ratio of 2:1, have no potential for instability as they should already be crushed and at their residual level *when cut* from the face. The appropriate use of crush pillars prevents backbreaks, and leads generally to stable conditions with significant improvements in extraction ratio. However, higher rates of stope closure can be expected. These need to be determined by monitoring, and taken into account in the design of panel support.

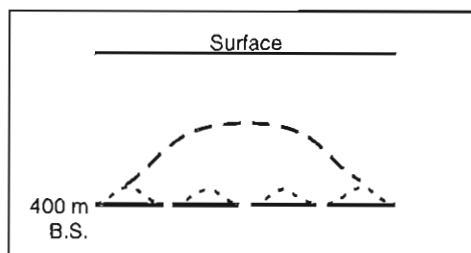


Figure 3.4.3
Height of potential tensile zone (dashed) over unsupported 860 m span stope at depth of 400 m. Height of tensile zones (dotted), if 10-20 m wide barrier pillars introduced.

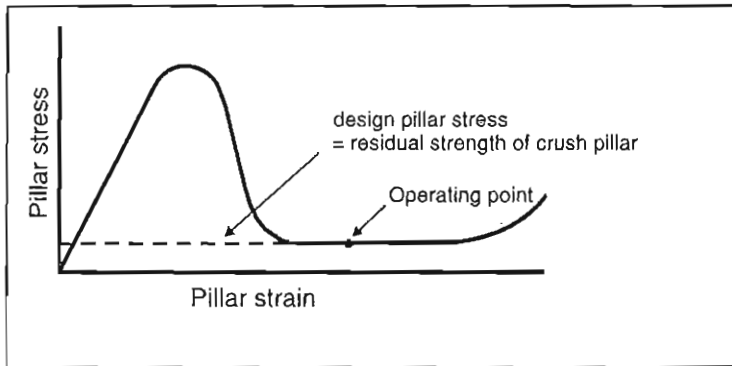


Figure 3.4.4 Design of crush pillars.

In a **transition zone** (approximately 300-600 m BS), crush pillars cannot readily be used since they may be intact when cut and only crush when loaded in the back areas, triggering violent failure or a potentially hazardous pillar run in the panel. Experiments with the use of **yield pillars** (squat pillars with $w:h$ ratio of about 3 - 5, which were thought to be not only strong but to yield in a stable fashion – Figure 3.4.5) proved in the end to be unsatisfactory. Such pillars when fully loaded were prone to occasional bursting or, more commonly, to punching into the footwall with excessive footwall heave. While mining in the transition zone, it is probably wisest to retain the use of non-yield pillars until a depth is reached at which cautiously-controlled experiments demonstrate that crush pillars can safely be introduced. The shallower workings should then be insulated by leaving a substantial *strike barrier pillar* in place, incorporating fault losses or other unpay ground where available.

Other in-stope pillars are occasionally required for **local hangingwall strata control**. For example, a fault between two lines of pillars can be stabilized by inserting one or more small pillars on the footwall side of the fault. When poor ground conditions (serpentinised jointing, 'cooling dome' structures, etc.) are encountered, it may similarly be advantageous to halve the panel spans by leaving occasional small pillars. This type of application of properly-designed 'crush' pillars is by no means restricted to shallow mining operations.

(b) Barrier pillar design

Barrier pillars act to compartmentalise a mine (facilitating ventilation and water-ingress control, and restricting the extent of any major pillar collapse situations). They also act to reduce the height of potential tensile zones above the stopes, and increase the effective hangingwall strata stiffness (which means that should pillar failures occur, they will happen in a less violent fashion). Barrier pillars need to be **squat** in order to give them the necessary long-term strength, and should thus be cut with a width:height ratio of not less than about 10:1. The appropriate **span** between barrier pillars is contentious. An old (and conservative) rule-of-thumb, that this span should not exceed about one-quarter of the depth BS, is qualitatively supported by a number of theoretical considerations; for example, reduction of the tensile zone and stiffening of the hangingwall beam. In practice, however, spans of about **one-half of the depth**, with well-designed in-stope pillars in place, have proved completely satisfactory in numerous mines. Barrier pillars can incorporate or comprise blocks of

unpay ground (dykes, 'potholes', etc), and can also be used as bracket pillars around potentially dangerous structures such as faults or joint swarms – see Figure 2.2.2.

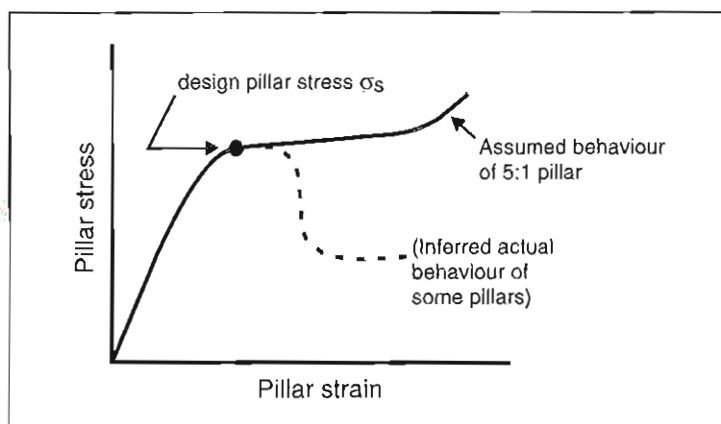


Figure 3.4.5 Behaviour of yield pillars

As with all heavily loaded pillars, barrier pillar designs need to be checked for possible foundation-failure problems (APS criterion, section 3.2.7).

(c) Common shallow mining layouts

The layouts commonly being employed in shallow mining situations illustrate the above principles, and are variations on one of three main types. They all make use of in-stope pillars with extraction ratios ranging from 75 to 95 %, depending on depth, the nature of the reef, and hangingwall conditions.

In **breast mining** layouts, the mining is carried out over typically a 28 m long face advancing in the strike direction – see Figure 2.2.2. The stope support comprises mainly stiff, pre-loaded support units at the face, and a row of rectangular pillars left along the down-dip side of the strike gully. In moderate to poor ground conditions, blast-on props placed close to the face may be required, although end-anchored bolts are also sometimes used to control hangingwall beams of up to about 1 m thickness. At *shallower depths* (<400 m), pillars need to be of the 'non-yield' type with a safety factor of 1.6 or more (and the use of barrier pillars is recommended). In cases where the hangingwall is poor, such as the presence of 'cooling domes' or large blocks formed by discontinuities, consideration should be given to leaving additional small pillars to support these structures. Strike gullies developed with an apparent dip of 2° to 5° are free of stress fracturing and can be excavated 3 to 5 m ahead of the face. Pre-developed on-reef drives are also an option. The lead/lag distance between panels can be determined from production requirements rather than rock mechanics considerations. Pillars can be left in line with gully sidewalls, but the increase in the pillar height on the gully side must be accounted for when calculating strength of the pillars. At depths *greater than about 500 m*, the strike-parallel pillars can be designed to 'crush' as long as their residual support resistance is sufficient for effective hangingwall beam control. A gully siding of 1.5 m is usually required to protect gully sidewalls from relatively high pillar foundation stresses.

Up dip or down dip mining with mainly non-yield dip pillars has been successfully used at depths ranging from less than a hundred metres down to 800 m. Mining takes place both sides of the raises developed typically 30 m apart, leaving a dip pil-

lar at the mid-distance between the raises - Figure 3.4.6. Dip pillars provide back area support against backbreaks and good protection to the faces. Up dip mining is particularly suitable for good ground conditions where throw blasting can be practised over face lengths of up to 13 m on each side of the centre gully, and excellent face advance rates can be achieved. Poor ground conditions necessitate shorter spans; support needs to be installed closer to the face, and this can cause blasting out of support units and choking of the throw. The stability of hangingwall over the raise area requires special attention in up dip mining, since the raises are used extensively for man and material transport over the duration of their life. In poor rock mass conditions, forming the dip pillars next to the raises increases the stability of hangingwall over the raise area. Down dip mining provides better protection for raises, since the operational part of the raise is always in the unmined area. At greater depths, the increased pillar stresses can give rise to foundation failures, and this needs to be considered in the overall layout design.

Where the maximum stable hangingwall span is in the order of 4 to 12 m (e.g. in manganese, asbestos and some shallow chrome mines), **room and pillar mining** is employed. The pillars are designed as 'non-yield' at all times, and the use of barrier pillars is particularly recommended unless the in-stope pillars have safety factors of 2 or more. This method is also suitable for undermining surface structures.

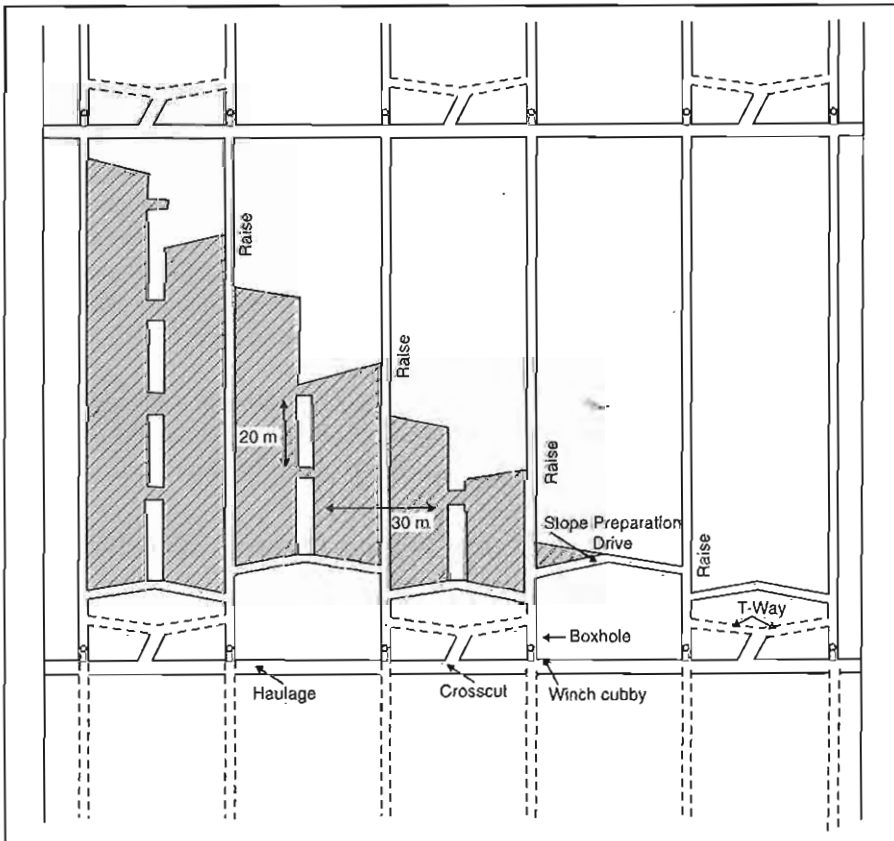


Figure 3.4.6 Typical shallow up dip mining layout.

(d) Overstoped situations

Overstoped ('stress-relieved') situations at any depth often present similar settings to those of very shallow mining. If total closure has not occurred in the overstoped area, the middling stability is best achieved using non-yield pillars. Otherwise, total closure can revert the vertical stresses to near virgin levels and crush pillars can be used to ensure stability of middling up to a thickness of 30 m. When middling heights are less than 10 m, pillars may need to be spaced closer to each other for better hangingwall control; cemented backfill can be a good alternative provided it has the required stiffness. Section 3.6.2 discusses further issues involved in multi-reef mining.

(e) Access tunnel layouts – shallow mining

The access developments for shallow scattered mining usually utilise the familiar patterns of main crosscut / main footwall haulage with multiple crosscuts to reef (spaced at 120-150 m) / raises / boxholes. Under normal conditions, field stresses during development and subsequent mining-induced stress changes, are too low to cause significant damage to tunnels, which may therefore be sited wherever they are most suited to the stoping method in use. Certain situations, nevertheless, should be avoided or treated with appropriate caution:

- Tunnels which pass within about 30 m directly under (or over) pillars or other remnants, can suffer stress-related damage. Numerical modelling should be used to determine safe distances, using the RCF criterion for tunnels in the brittle quartzites of the Witwatersrand Basin [no fracturing if $RCF < 0.7$, very difficult conditions if $RCF > 1.4$]. Re-siting, overstoping, or upgrading of support standards, are all potential solutions.
- In the somewhat more ductile igneous rocks of the Bushveld Complex, the RCF criterion is less well substantiated and local experience is currently the best guide. In particular, in areas of high virgin horizontal stresses where 'Gothic-arch' fracturing can arise, special development and support procedures need to be adopted (over-stoping is strongly *undesirable* in this specific environment).
- Tunnels in close proximity to major geological structures can encounter poor ground conditions as well as enhanced risk of intersecting ground water.
- Haulages sited in weak or serpentinised strata will suffer higher FOG risks and require more stringent support requirements.
- Crosscuts in shallow-dipping strata form potentially unstable wedges of hangingwall at the reef intersections; these can be avoided by stopping the crosscuts short of reef and developing inclined travellingways to the stope horizon.

3.4.2 Medium Depth (Scattered) Mining (approx. 1000-2250 m)

In medium depth stoping, *face fracturing* and resulting dilational movements generate stresses which help to stabilise the hangingwall beam. *Scattered* mining methods can continue to be used, and conditions tend to be relatively congenial: in-stope pillars are no longer required, and conventional timber-based support systems or backfill provide adequate hangingwall stabilization. Certain layout-related problems are nevertheless routinely encountered:

- Unfavourable low-angle fracturing can affect the stability of certain excavations, especially raises and gullies – sections 3.4.6 and 3.4.7.
- Advance footwall development tunnels can suffer high field stresses under large-span mining abutments, to the extent that overstoping protection may need to be resorted to.

- Seismicity associated with geological structures starts to emerge in deeper scattered mines, and remedial measures such as the use of bracket pillars – section 3.3.3 – may need to be employed.
- Seismicity and rockbursting associated with extraction of the numerous remnants which arise in scattered mining becomes a serious problem at greater depths. Remnant extraction precautions (section 3.5.1) are imperative; and at depths exceeding about 2000 m, consideration needs to be given to radically changing the mining method to, for example, 'sequential grid' mining – section 3.4.4.

The following discussion relates to scattered mining layouts at medium depths, but some of the principles introduced apply, with equal relevance, to other mining layouts at comparable or greater depths.

(a) Mining configuration.

Mining configurations may be breast, underhand or overhand, and may generally be freely chosen without intrinsic effect on the incidence of rockfalls or rockbursts, provided the principle of 'mining towards solid' (Figure 3.4.7) is adhered to. The effect that stopping configurations will have on service excavations (such as main haulages) for the duration of mining should also be considered (Chapter 5), particularly with regard to the remnant stage where high induced stress concentrations can give rise to significant problems.

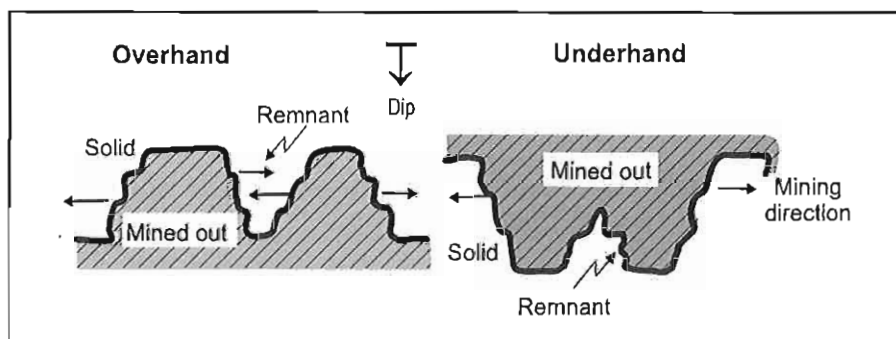


Figure 3.4.7 Principle of 'mining towards solid': overall mining directed towards nearest unmined ground, and final remnants extracted against solid abutments.

The relationship that the mining configuration bears to a given geological environment can be important. Thus, the orientation of panels, or groups of panels, should not be parallel to any large geological structures or sets of repetitive features such as joints, dykes or faults that respond unfavourably to a certain direction of mining, particularly if these are seismically active. The mining configuration should, where practicable, approach such structures as obliquely as possible ($> 35^\circ$ from breast-on). Alternatively, the principle of 'mining away from geological structures' – Figure 3.4.8 – is a strategy that aims to place dangerous formations into the back areas as soon as possible, thus blunting their damaging potential. For example, if extensive areas along geological structures are likely to be subjected to positive excess shear stresses (section 3.2.3), remedial measures such as mining on retreat, or use of bracket pillars or other forms of regional support (section 3.3) need to be considered.

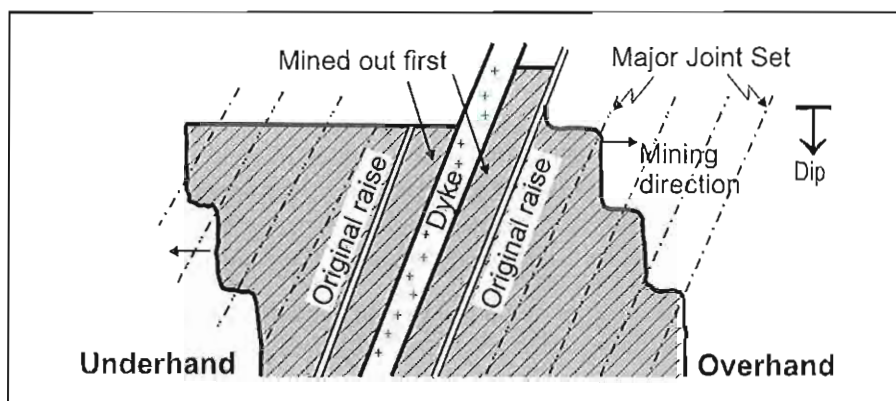


Figure 3.4.8 Principle of 'mining away' from seismically-active structures, and obliquely through pervasive geological weaknesses.

Ideally, scattered or other forms of mining should aim for levels of average energy release and stress at the moving faces that remain as near constant as possible during the extraction of an entire area. The creation of excessive lags (>10 m) should be avoided wherever possible. Treatment of the large numbers of temporary remnants inherent in scattered mining needs special attention: whether to abandon, or (with appropriate 'remnant precautions') to extract.

Numerical modelling provides useful tools for assessing and optimising these situational problems – Chapter 11.

(b) Stope faces.

The layout of individual stope faces may be considered in terms of face shape, rate of advance and mining direction.

Stope face shape may be breast, underhand or overhand, up dip or down dip, without generally influencing the incidence of rockfalls and rockbursts. Stope configurations should however take into account extraneous factors such as the geological environment (Figure 3.4.8) and overall mining geometry constraints. Leads and lags should be kept to a minimum because of their tendency to generate adverse low-angle fracturing. Over a wide range, the *rate of face advance* probably does not affect rock conditions greatly, provided that in achieving high rates of advance, support quality is not compromised and excessive blast damage is not incurred. However, very low rates of face advance and, in the extreme case, idle faces, can lead to significant time-dependent deterioration of the rock mass condition. This increases with increasing depth, and necessitates installation of stiff support close to the stope face (Chapter 4.7.3).

The *direction of mining* can generally be decided purely on practical grounds. Occasionally, the geological environment has to be taken into account, for example the direction of cross-bedding or jointing (Figure 3.4.8); and in some difficult mining circumstances, such as in remnant extraction, a change in mining direction is often advantageous and should be seriously considered (section 3.5.1). Gully layouts are discussed in section 3.4.7.

(c) Access tunnel layouts – medium depth mining

In tunnels developed to access medium to deep scattered mining, *stress fracturing* and exposure to *seismic hazards* become increasingly prevalent, with significant safety and cost implications. The cautionary provisos of shallow tunnel layouts itemized in section 3.4.1(e), therefore, need to be stringently applied. In particular, the siting of tunnels with respect to the stoping environment (*stress levels*, notably under remnants and large-span abutments), and their proximity to geological structures (*seismic hazards*), are matters which demand serious attention:

- Wherever possible, tunnels should not pass closely above, or closely below, any *highly-stressed* pillar, abutment or permanent remnant. In particular, strike haulages should be laid out at practical distances below the temporary remnants which arise in scattered mining scenarios, such that the value of the RCF criterion remains at acceptable levels [normal support required if $RCF < 1.0$, greatly enhanced support standards required if RCF rises above 1.4]. Often, the presence of a relatively strong stratigraphic unit can be exploited for optimal tunnel siting. In general, an economic trade-off has to be drawn between high support costs in a high-RCF environment, or lower support costs but higher crosscut/boxhole development costs in a lower-RCF environment further from the reef horizon.
- When occasions demand that development be taken through areas of present or future very high field stresses, *overstopping* (either on-reef or in waste rock) may provide a solution. This option is, however, both costly and potentially hazardous; and should be avoided if at all possible by proper foresight and pre-planning of the entire mining of a given area.
- Tunnels should avoid running parallel to and less than about 50 m from any structure assessed to be *seismically active*, and in particular, the *intersection* of two such structures. For example, tunnels sited in fault losses can be subjected to high seismic hazards over unacceptably large distances. The specific design distance at which a tunnel should be kept from a particular active structure can be established from recorded past experience, analysis of appropriate seismic data including estimates of the potential PPV (peak particle velocity), and/or numerical estimation of the seismic risk using the ESS criterion.
- Where seismic hazards are nevertheless seen to be high, appropriate upgrading of the support standards becomes imperative. In extreme situations, the affected section of tunnel may need to be declared a special area with defined usage procedures enforced; or appropriate overtopping or re-development measures invoked.
- Seismically-active structures, or zones of inherently poor ground conditions (for example, 'running dykes' or other highly incompetent or jointed rock) should be intersected at as *high an angle* as possible to minimize the length of tunnel exposed to the hazardous conditions.
- The inherent flexibility and facility for pre-exploration offered by scattered mining layouts needs to be exploited to the utmost in medium to deep mining environments, in order to expedite the types of specific layout changes mentioned above; for example to put in place the infrastructure required for 'mining away from geological structures'. Thus in particular, *geological structures* encountered during pre-development need to be mapped and characterized in appropriate detail (strike, dip, throw, contact properties for seismic potential). *Geotechnical properties* (strength, joint spacings, parting characteristics) of relevant stratigraphic horizons both on and off-reef need also to be established so as to facilitate tunnel sitings and the delineation of ground control districts in the mine.

(d) Summary

Scattered layouts, where practicable, provide the mining engineer with probably the most convenient and flexible of all tabular mining methods. Advance exploration and opening-up of new working areas, selective mining, and the negotiation of geological disturbances are all facilitated.

Unfortunately, rock engineering problems associated with conventional scattered mining become increasingly severe with increasing mining depth: in particular, the problems of high *abutment stresses* in advance haulages and the hazards of frequent *remnant extraction*.

3.4.3 Deep Longwall Mining (2000-3500 m)

Longwalls were introduced into the deeper gold mines in an effort to address the problems of scattered mining mentioned above.

(a) Longwall mining configurations

In the longwall mining system, mining proceeds outwards from an initial central raise - Figure 3.4.9. These raises may be as much as 2 to 3 km apart, and longwall back lengths can exceed one kilometre. Strike *stabilizing pillars* (often enhanced with close-in backfilling) are used to provide regional support. Typical extraction ratios are 80-85 %, and the pillars are made wide (40 m or more) in order to inhibit through-going fracturing, thus maintaining their stiffness and ability to control ERR at the working faces. The presence of wide pillars also helps to break up extensive ESS lobes ahead of the stope face, thus reducing the potential for long face-parallel ruptures and corresponding large seismic events - see Figure 3.3.2.

Dykes and faults with throws of less than about 10 m are generally treated by simply 'mining them out', but serious seismic problems often occur at these times. Faults with significant throw are sometimes handled by waste overstoping and subsequent re-raising - see Figure 3.5.1 - an arduous and costly operation. Due to the relative inflexibility of longwall mining, and the general lack of advance geological and geotechnical information, there are also often difficulties with approaching hazardous structures with the required $> 35^\circ$ orientation.

A longwall configuration with straight faces will minimise the formation of localised areas of high stress and of adverse fracturing patterns at the corners of large leads. In practice, however, a number of factors militate against this idealised geometry; in particular, the tendency for rockbursts to damage extensive lengths of a very straight longwall due to the presence of an extended lobe of positive ESS there. This can be overcome by staggering individual groups of panels within the longwall or, in a stabilizing pillar environment, simply by using sufficiently wide pillars (section 3.3.2).

Staggered layouts continue to form overall breast, overhand or underhand configurations. Provided geological considerations do not play an overriding role, the advantages and disadvantages of each configuration may be assessed in terms of the surrounding stoping geometry and the position and form of the final remnant. Two approaching breast longwalls will form a long, thin remnant that would exacerbate rockfall and rockburst problems. A similar situation will result from two approaching longwalls if one has an overhand configuration and the other an underhand configuration. Employing uniform configurations will result in triangular remnants sited

updip in the case of overhand and downdip in the case of underhand layouts. Obviously, if the updip area is extensively mined it would be better to employ an underhand configuration which would position the remnant downdip; mining would then be towards the solid with no highly stressed island remnant left. These principles, which also apply to scattered mining, are illustrated in Fig. 3.4.10.

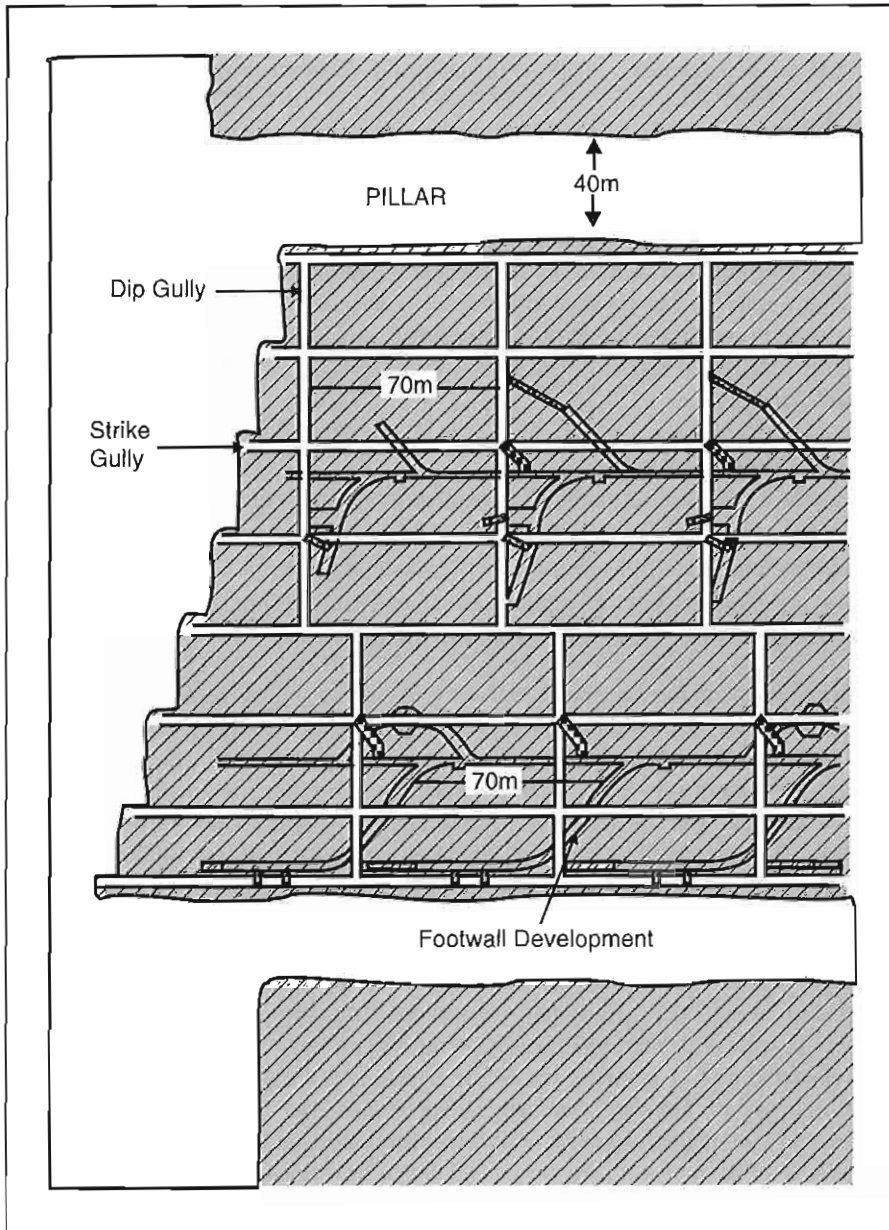


Figure 3.4.9 Schematic of a typical longwall mining layout

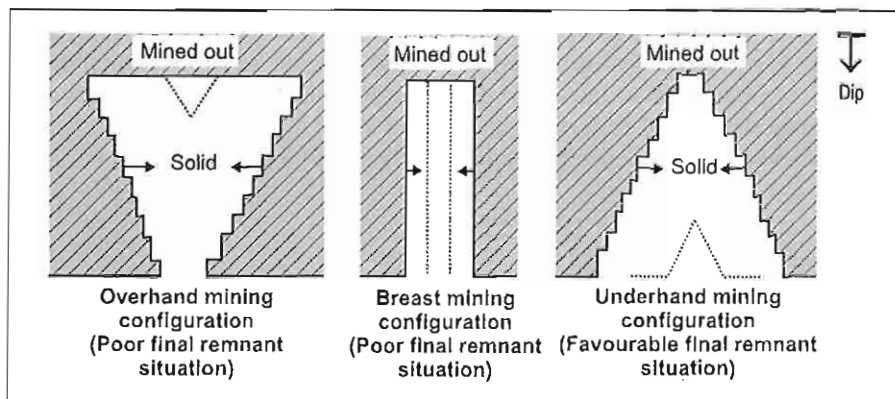


Figure 3.4.10 Form of final remnant, depending on overall stoping configuration.

There are, unfortunately, practical problems associated with underhand mining, including cleaning and ventilation difficulties and poor strata control disposition of the bottom gully and associated follow-behind haulages. These problems may to some extent be overcome by adopting an *overall underhand* longwall configuration, but using a staggered layout of breast or slightly overhand mini-longwalls between the stabilizing pillars.

(b) Access tunnel layouts

In longwall mining, access tunnels (better known as *follow-behind haulages*) are generally sited only about 20 m below the reef plane and are kept no less than 20 m behind the stope face (following the '45° overstoping rule'). They thus remain in *destressed* ground at all times, and this is one of the more important rock engineering advantages offered by standard longwall mining.

In order to maintain high standards of layout efficiency, regular reviews are required of the positioning of footwall excavations with respect to the reef and stabilizing pillars: ensuring that crosscuts avoid faulted or otherwise poor quality ground, and that tunnels do not approach the zones of high abutment or pillar stresses too closely. In addition the quality of development, equipping and support needs to be of a high standard. Regular maintenance is an important issue since access tunnels are subjected to significant stress changes and deformations over their long operational lives (as much as 15 years in the case of follow-behind haulages).

A layout suitable for a *standard* mini-longwall separated by stabilizing pillars is depicted in Figure 3.4.9. [However, *non-standard* back lengths are often forced onto the mine planner due to the spacing and orientation of faults and dykes, or other mining considerations]. The layout shows two haulages at different elevations servicing the mini-longwall. The upper haulage is often an *interlevel* developed at an elevation most suited to the circumstances. The lower level is carried approximately 20 m in the footwall of the reef, with cross-cuts and dip gullies spaced at 70 m intervals. Also shown, as an extension to the cross-cut, is a shallow footwall drive to service the bottom gully of the longwall and to provide bottom entry to the stope. Boxholes are required to intersect the overlying gullies; these often pose a survey problem as both the stopes and tunnels experience heavy closure and ride deformations. Furthermore,

the provision of necessary facilities for the cooling of ventilation air before it enters the stope requires appropriate chambers to be cut and supported. The air, entering the stope at the bottom, may be re-cooled at an upper level before being reintroduced to the stope.

A major benefit of this mining layout is that the tunnels are developed in stress-relieved ground behind the stope face, and thus do not experience the effects of very high abutment stresses as is the case with scattered mining. No stress fracturing associated with the driving of the tunnels occurs, but the tunnels do intersect stress fractures related to the stope above. These dip at angles of 55° - 65° against the direction of the haulages, and cut cross them at high angles. In the crosscuts, the fractures run sub-parallel to the tunnels, and need to be taken into account in the design of support. Also to be considered is the fact that some stress-relaxation will continue until total closure of the stope above occurs, at which stage compressive stresses will increase possibly up to virgin stress levels. This process will be accelerated where the stopes are backfilled.

There can be other significant mining difficulties: principally that the infrastructure required to service the stope face area is often too far behind the face - more than the optimum maximum single scraper distance of 70 m. Unfortunately, there is no compromise to the requirement that the tunnels be kept from advancing closer to the stope face than their depth below the reef.

Severe rock engineering problems arise in the negotiation of large faults. In these situations, the development has to be driven ahead of the stope face through the highly-stressed abutment, and the whole infrastructure then re-established. The likelihood of seismic loading of tunnels in these situations is high. Two methods have been used:

- (i) the shallow footwall drives are re-established behind the current stope face at a position suitable for the displaced reef on the other side of the fault, and then advanced on-strike through the fault under the protection of off-reef overstoping, or
- (ii) the haulages are turned to cross-cut to a position sufficiently deep in the footwall such that when the tunnels pass under the abutment, the field stress is tolerable. Numerical modelling (in terms, for example, of the RCF criterion) is necessary in the first case to determine the dimensions and extent of the overstoping, and in the second to determine the depth below reef that the haulage must be before crossing the abutment.

A comparable situation arises when a longwall has to be incrementally deepened. Here, wide winzing with immediate stoping to form an underhand arrowhead configuration provides the overstoping cover under which the crosscut can be advanced. Furthermore, if long-term *replacement haulages* need to be developed deeper in the footwall, a similar problem arises when these tunnels have to swing below the highly-stressed ground under a stabilizing pillar in order to link with the existing shallow follow-behind haulage system. Several tunnel shapes and support schemes have been tried to control the highly stressed rock, with various degrees of success - Chapters 5 and 6. In spite of these precautions, the potential for nearby seismic events is ever present and all such development should be treated as special areas, particularly when traversing strong brittle dykes.

The shallow footwall drive adjacent to the pillar shown in Figure 3.4.9 is not well protected by overstoping and is usually highly-stressed and vulnerable to rockbursting from seismic events located either at the stope face or along the pillar. This aspect of the layout can be hazardous and not cost effective, and is therefore not recommended. Similarly the layout of any long-life haulage running parallel to an abutment or pillar should be treated conservatively and kept further from the solid ground than the '45° rule' would require, so as to further distance the tunnel from repeated dynamic loading from seismic events located along the abutment.

It is seen, therefore, that with respect to development there are significant advantages and disadvantages of the longwall system with strike stabilizing pillars. As the complexity and frequency of dislocation of the reef increases so do the safety hazards and negative effects on profitability. Under such adverse conditions other mine layout options should be given serious consideration.

(c) Summary

Where there is a low incidence of geological disturbances, longwalling is an attractive deep mining method, providing the potential for good ERR control and footwall development protection in a simple consistent layout. In fact for many years, longwall mining was considered to be *the ideal* method for all deep mining (2000-3500 m). This is now questioned, particularly in more geologically complex areas; the main problems being seismicity and adverse fracturing associated with the stabilizing pillars themselves and, overwhelmingly, the hazards and delays of 'blind' mining up to, and through, dangerous geological structures. Other layouts, for example 'Sequential Grid' and 'Room and Pillar' methods, are being considered and need to be actively evaluated.

3.4.4 Deep 'Sequential Grid' Mining

This controlled adaptation of *scattered* mining layouts to deep conditions has considerable potential. The basic idea is to leave permanent dip-pillars in a scattered layout as 'planned remnants', which act as coverload-carrying stabilizing pillars and which permit significant advance exploration to be carried out. The method has been used successfully at Elandsrand Gold Mine and appears to offer major advantages over longwall mining at depths down to at least 2800 m at this mine.

The mining follows a planned sequence outward from the shaft pillar on a 'grid' of consecutive raise lines. The name 'Sequential Grid Mining' is used locally as a result; though an alternative name could be 'Scattered Mining with Dip-Pillars' (SMDP). The raise lines are 200 m apart, and 30 m wide strips of reef are left between the raises as dip stabilizing pillars – Figure 3.4.11. The resultant 85 % extraction is appropriate at the depths involved (≈ 2500 m), but would need to be reviewed if the method were considered for greater depths.

The *access/ventilation tunnels* are located deep (80 m) below the reef plane in a strong quartzite formation, and are excavated out ahead of the mining operations. Long crosscuts every 200 m link the main haulages to the reef plane where the raises are developed on-reef. Levels are about 70 m apart, and two are usually stoped at a time.

Mining from a particular raise proceeds first in the direction towards the shaft pillar - Step 1 in Fig. 3.4.11. The dip-pillar position is reached with only a limited span of mining behind, thus ensuring lower stresses in the pillar and improved pillar stability at that stage. Once the pillar position has been reached by about four panels, Step 2 may begin and proceed outwards in the direction away from the shaft pillar. After say four panels of Step 2 have reached the planned pillar position, then mining can commence on the next raise and so on, sequentially mining out raise lines all the way to the mine boundary.

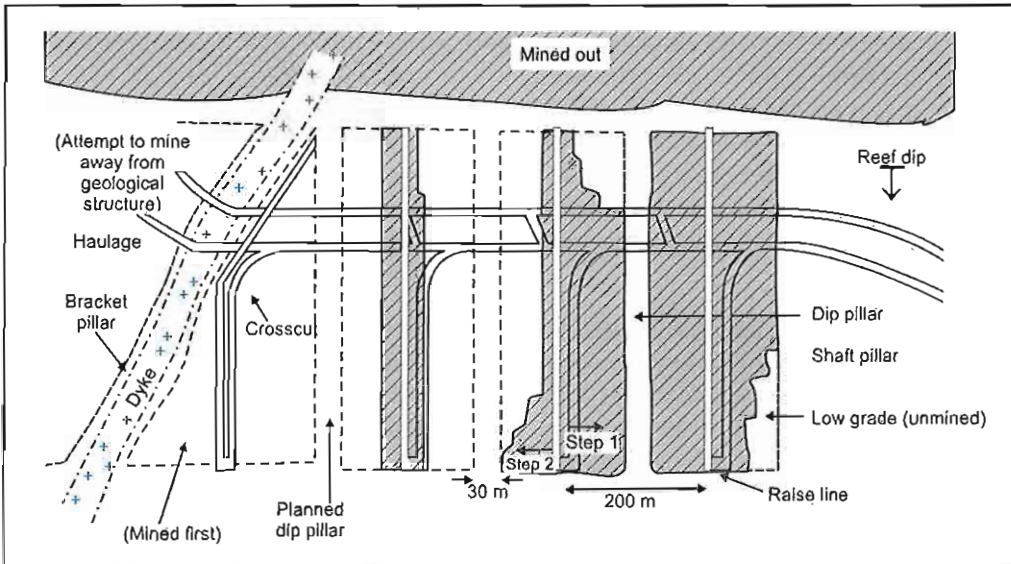


Figure 3.4.11 Sequential grid (SMDP) mining method.

When a fault or dyke is encountered, the planned pillar position can be shifted to incorporate the feature, thus serving as a bracket pillar as well as part of the overall stabilizing pillar system. Low-grade areas can also be left in situ and so form part of the stabilizing pillar system. Backfilling may be practised to improve general stability and provide additional regional support, as well as providing effective local support when integrated with RYHPs (hydraulic props) or pre-stressed elongates.

The advantages and disadvantages of SMDP methods, as compared to longwall mining, are listed below.

Advantages:

- Improved pre-knowledge and hence control of potentially hazardous faults and dykes. The spacing or orientation of the 'dip' pillars is not rigidly fixed, and so the need to mine through dykes and faults is greatly reduced. Many of these features can be left intact with adequate bracketing, and made to form part of the overall stabilizing pillar system. Pre-development allows accurate determination of the position and orientation of faults and dykes, thus greatly facilitating the planning process and enabling the flexibility of this mining method to be fully exploited.
- Dip pillars seem to be inherently more stable than strike pillars, thus reducing the incidence of pillar foundation failures. The reason for this may be that the major ride component (down-dip) is parallel rather than perpendicular to the pil-

lar orientation. Major foundation failures are sometimes experienced on strike stabilizing pillars, resulting in extensive damage to nearby strike footwall development. In addition, strata control problems often experienced in gullies immediately up-dip of strike stabilizing pillars are eliminated.

- In SMDP layouts, the *extent* of potential damage to footwall development (running, as it does, perpendicular rather than parallel to the pillars) is significantly reduced [though significant *localised* problems, including high stress and seismicity levels under the pillars, can by no means be ruled out in general – see ‘disadvantages’ below].
- The effectiveness of backfill as a regional support is better in the case of SMDP methods. It is placed initially in a low closure environment, close to the raise, and its potential to provide substantial support at later stages of mining is improved. [The placement of cementitious or concrete backfills would also be greatly facilitated.]
- The need for slotting highly stressed pillars to destress footwall development close to the reef plane is generally not necessary with SMDP methods. Indeed this practice, which is very hazardous, has been virtually discontinued in recent years in mines using conventional strike stabilizing pillars.

Disadvantages:

- * Rock stresses on the working face increase from a low level as one mines away from the raise line towards the dip pillar (whereas in longwall mining, face stresses and ERRs tend to be higher but more constant as mining progresses). However, correct sequencing and control of mining spans, inherent in the method, can ensure that these stresses are kept to within safe limits.
- * In SMDP, footwall tunnels and boxholes are pre-developed in high virgin stress conditions with immediate fracturing of their rockwalls, and are later subjected to a changing stress regime as mining takes place (in longwall mining, on the other hand, most of the footwall accessways are developed below the stoped-out reef and remain in stress-relieved ground). Refer to Chapters 5 and 6 for the adverse consequences of high field stresses and stress changes in tunnels. Careful numerical modelling of stress levels and stress regeneration (particularly under pillars and where stiff backfill is emplaced around the raise) is essential for the proper design of detailed layout, ledging and support standards.
- * Heavy pillar stress concentrations, and the effects of any incipient foundation failure events, are directed onto portions of the haulages below reef. The latter therefore need to be sited deep below (or above) the reef plane in *good-quality strata* to alleviate the problem, and be appropriately supported; or else (as an as-yet untried alternative) be sited close to the reef horizon but be overstoped by early extraction of appropriate slots out of the dip pillar locations.
- * Thus, crucial to the successful implementation of ‘sequential grid’ methods to *ultra-deep environments* will be the discovery of viable methods for developing and stabilizing tunnels under high virgin and field stress environments. In situations where heavy seismicity seems to be unavoidable, aggressive strategies such as *overall retreat* SMDP mining may need to be implemented.

3.4.5 Deep (and Ultra-deep) ‘Room and Pillar’ and other Mining Possibilities

Squat pillars with a width to height ratio of say 20:1 are potentially very strong and, theoretically at least, should be able to carry heavy cover loads without undue rock

engineering problems. Numerical modelling of a basic 'room and pillar' layout with 20x20 m internal pillars on 20 m bords (75 % extraction) showed acceptable APS values of 320 MPa, and very low ERR levels of 1-4 MJ/m² at 3000 m below surface. In addition, global field stresses did not rise significantly above virgin levels, and advance on-reef or deep off-reef development were not subjected to high abutment or face stresses [crosscuts to reef would, nevertheless, require special treatment].

There are, of course, serious uncertainties concerning the ventilation aspects, economics and safety of mining this way at great depth: notably the stability of the pillars and their reaction to transient seismic waves. The concept nevertheless warrants further investigation, and might best be evaluated on an actual field trial basis.

Other mining methods that exploit similar **low-extraction/low-span** principles are conceivable. For example, 20 m wide *rib pillars* (with small ventilation holings, or larger overtopping slots) oriented preferably on dip, could separate pairs of conventional 30 m panels, giving similar extraction, APS and ERR levels to the above. An actual example (an SMDP variant), utilising 60 m total span down-dip oblique-faced panels with 30 m dip pillars, is currently being tried in the field, and is showing promise as a practicable deep and ultra-deep mining layout.

Further issues that will need to be addressed, if ultra-deep mining is to become practicable, were reviewed in Chapter 2.4.3.

3.4.6 Ledging and Raising

One of the most important operations which impacts on the long term viability of a stope in a scattered or SMDP layouts is **ledging**, as this often determines whether the centre gully – the artery of subsequent stoping operations – remains stable over its lifetime. Rock-related accident and fatality statistics indicate that centre gullies and associated ledging operations can pose significant safety problems.

In creating a **raise** in a block of ground, stresses will be induced in the rock surrounding the opening and, if these rise high enough and exceed the strength of the rock, significant damage will occur. In light of this, it is important that the ledging sequence minimises the further build-up of stress around the new centre gully, and the sequence of mining should also limit the amount of additional stress cross-fracturing.

Several methods of ledging have been used in the past to cater for various ground conditions. These include breast and down-dip methods, single or double-sided ledging, or a compromise of partial ledging on one side with installation of support, followed by partial ledging and supporting on the other side prior to completion of the original ledging. In poor ground conditions, 'chequerboard' ledging has been successfully implemented, in which stope-height bays ('cubbies') are cut alternately on either side of the raise and support installed, prior to removing the intervening reef. Where the rock mass is mobile, full height ledging has been tried, in some cases with a row of packs down the centreline of the widened raise. [Where ledging is carried out at the full height of the raise, the stiffness of the support along the gully will be reduced, compared with the case where packs are installed along the shoulders of a normal gully, allowing more flexure of the hangingwall beam and possibly less sta-

ble conditions. Reinforcement by roofbolts should be seriously considered to counter such deterioration; noting the geotechnical composition of the hangingwall in deciding on the anchor length, appropriate anchor type and the drilling and installation method.]

A **wide raise** which carries the ledging along with the advance of the gully has a number of advantages:

- the stress fractures are created perpendicular to the gully direction, an inherently more stable orientation and a more stable foundation for the gully support [however, shallow-dipping fractures arise in the gully hangingwall, which require special attention – Chapter 4.4.9]
- if the gully is created by footwall lifting as a secondary operation, the blast damage to the gully sidewalls is reduced
- stresses are removed from the gully edges immediately
- the permanent support can be installed at smaller spans
- hydraulic props may be used to assist in getting support as close to the face as possible.

The relative safety and productivity aspects of these various methods have not been thoroughly evaluated for different ground conditions, and the choice of method is thus best left to local experience.

Bearing in mind the large amounts of stope closure that may be anticipated in the vicinity of a centre gully, the permanent supports should have sufficient long term yieldability and stability, but also not be so strong that they damage the gully sidewalls.

Where ledging is carried out as a secondary operation after the raise has been developed, it is advisable to minimise the damage to the rock by taking short blasts. If possible the ledging should be carried out parallel to the raise, to minimise the development of differently-oriented fracturing which could give rise to a blocky rock mass adjacent to the gully. In general, the ledging operation should be carried out as soon after creating the raise as possible, so that time-dependent failure does not occur in the rock surrounding the centre gully. If stoping is delayed, moreover, particularly good support needs to be installed in the ledged area.

Consideration needs to be given to installing rock reinforcement (with or without fabric support) during the development of the raise, which is adequate for the excavation once it becomes the centre gully. This will not only reduce time-dependent deterioration, but will also eliminate the hazards and costs of later supporting ground which has become dangerously deteriorated. In situations where excessive sidewall closure is a problem, the early installation of yielding tendons has cost-effective safety advantages [alternatively, the use of sacrificial footwall-lifted gullies parallel to the centre gully can be tried, Figure 3.4.19].

3.4.7 Gully Layouts

Stope gullies and the immediately adjacent stope faces are often the most dangerous areas when stoping a deep tabular orebody. Unfortunately, an important contributing factor is a widespread lack of appreciation of the rôle that poor gully geometries can have in the generation of adverse fracture patterns. Unfavourably oriented fractures,

often combined with relatively low convergence rates as well as cleaning constraints, make the design of appropriate support particularly difficult.

Where *fracturing is absent*, as in shallow mines or in stress relieved situations, certain simple gully configurations can be adopted without increasing the risk of instability. These configurations give rise to added flexibility in the stopping operation and should be employed wherever they can be used to advantage - Figure 3.4.12.

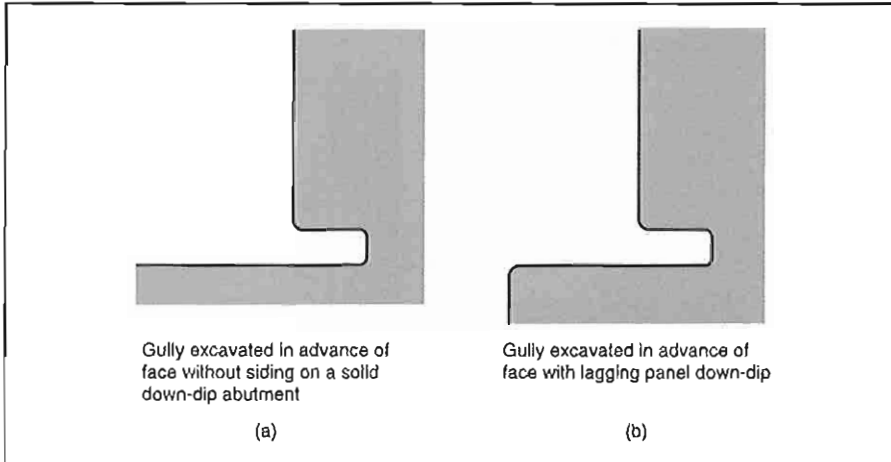


Figure 3.4.12 Gully configurations suitable only for low-stress environments.

In mines where in-stope pillars are used as part of the support system, the issue arises as to whether the pillars should butt directly against the strike gully (thus reducing their effective width:height ratio and thereby their strength), or should be cut some distance down-dip leaving a siding (thus maintaining their full strength). If pillars or gully sidewalls appear to be spalling prematurely, the cutting of sidings should be considered; otherwise there is no reason to change from the more convenient mining option.

Under conditions where *stress fracturing* of the hangingwall or gully shoulders occurs, much greater attention needs to be paid to the gully geometry. The extent and intensity of fracturing increases with increasing stress. In order to achieve good hangingwall conditions, the gully needs to be excavated *within* the fracture zone created by the stopping excavation, and should not be positioned such that the gully causes additional adverse fractures to develop. Figure 3.4.13 shows a section of a deep stope with its associated fracture pattern, as well as a plan of the hangingwall fracture trace and dip around a bottom abutment. It will be noted that fairly stable hangingwall conditions with steeply dipping fracture planes occur at a distance of 3 m or more from the abutment. Thus, if the gully is situated at least 3 m from the abutment (i.e. using a dip siding of >3m), relatively good hangingwall conditions will be encountered.

A gully developed too far ahead of the stope face fracture zone will generate its own fracture envelope (Figure 3.4.14) which, combined with the stope's fracturing, will give rise to unstable blocky ground. Normal support is commonly inadequate in these conditions. In particular, a lack of sufficient area coverage or interaction between the tendons in the highly fractured and blocky ground can lead to serious falls of ground, especially where dynamic loading is involved. In addition, the gully

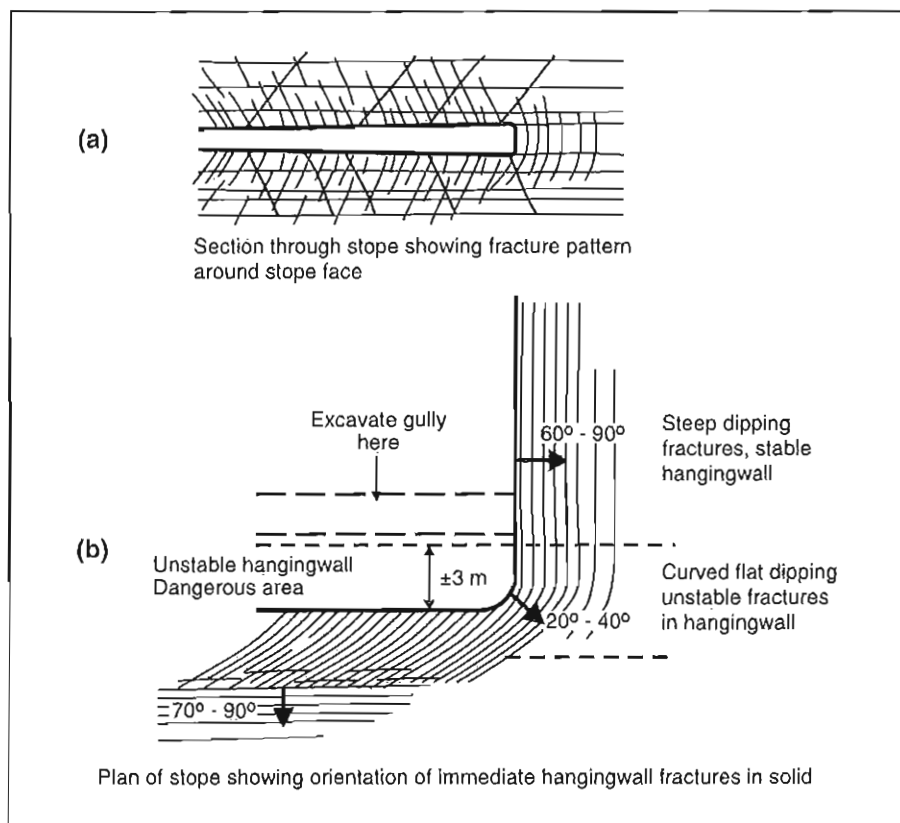


Figure 3.4.13 Fracturing around a deep stope near a down-dip abutment.

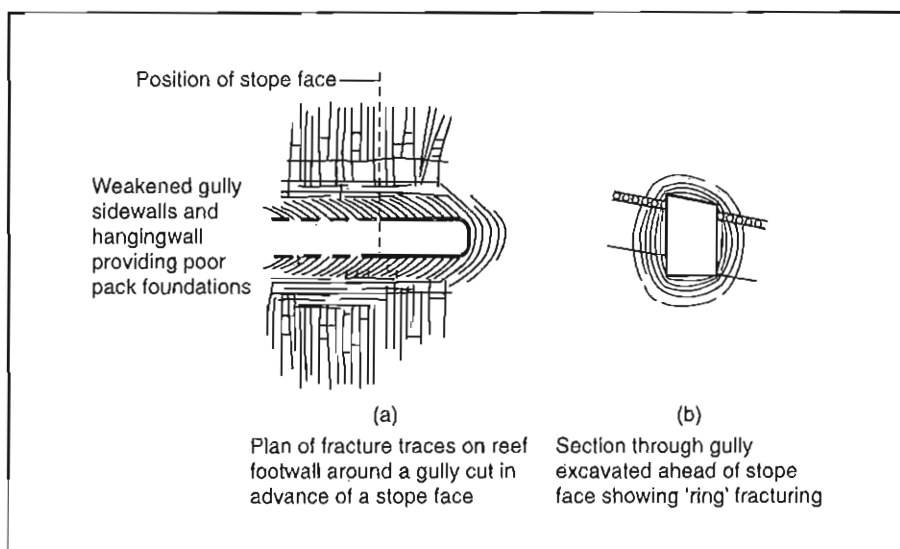


Figure 3.4.14 Adverse fracturing generated by a deep ASG with too large an advance.

sidewall fracturing is oriented at an acute angle to the direction of advance: these conditions provide a poor foundation for support erected on the ledge and often lead to instability and increased risk of injury.

Thus under *high stress* conditions, gully options are restricted to those configurations which do not create additional stress fractures. Generally, these include only those where the gully is excavated *in line with or behind the stope face*. The inclusion of a *supported siding*, advanced in line with the face and the gully, is generally considered to be good practice. This is possible to achieve without danger of blast cutoffs, if strict attention is given to achieving correct blast timing. Appropriate gully configurations, which comply with these high-stress conditions, are illustrated in Figure 3.4.15.

Sidings must always be excavated *in the reef horizon*. Sidings which are excavated at a flatter dip (to facilitate cleaning) destroy the integrity of the hangingwall and give rise to dangerous conditions, particularly in well-bedded formations or where a weak rock unit occurs less than 2 m above the reef. Appropriate means for cleaning this troublesome area need to be provided: e.g. a second winch with chain providing multiple snatchblock attachments. In steeply dipping reefs (dip > about 50°), the cutting of sidings becomes, however, impractical. In these circumstances, a pattern of closely-spaced reinforcing tendons is required to cover both the hangingwall and sidewalls of highly-stressed gullies – Figure 3.4.16.

Wide heading strike gullies illustrate a number of the above-mentioned principles. It was common practice when introducing strike stabilizing pillars on mines in the 1970s and 80s to use advanced strike gullies (ASGs) in the bottom panel of the shortwall adjacent to the pillar. Serious ground control problems were experienced in these gullies, leading to numerous falls of ground and rockburst casualties – Figure 3.4.17a. Rehabilitation where fall thicknesses of 4 m or more were experienced, was a costly and dangerous procedure, and truly safe conditions were seldom re-established. This had further adverse consequences, in that the delays caused by the rehabilitation resulted in unfavourable face shapes developing on the shortwalls.

To ameliorate this problem, *wide headings* were introduced. These were kept approximately 8 m in advance of the adjacent panel and were about 8 m wide with double pack support on either ledge. In fact, one unfavourable fracture pattern was substituted by another – note the flat fracturing in Figure 3.4.17b – but in the new layout the laminated rock could be better supported by the four packs and by roof-bolting, and the severity of the FOG problem was reduced. In order to improve conditions still further, experience indicates that the wide heading should best be developed *within* the stope fracture zone; that is, it should not be in advance of the panel by more than about 2 m – Figure 3.4.17c.

Wide raises and dip gullies also require special attention. Conventional raises or dip gullies of rectangular cross-section developed under *high stress conditions* generate a fracture envelope around themselves which gives rise to weak sidewalls, as well as to dangerous curved fractures in the hangingwall (c.f. figure 3.4.14b). The sidewall fracturing can cause undermining of the pack support when the raise is ledged, and, in addition, meshing of the hangingwall is required to make such raises safe.

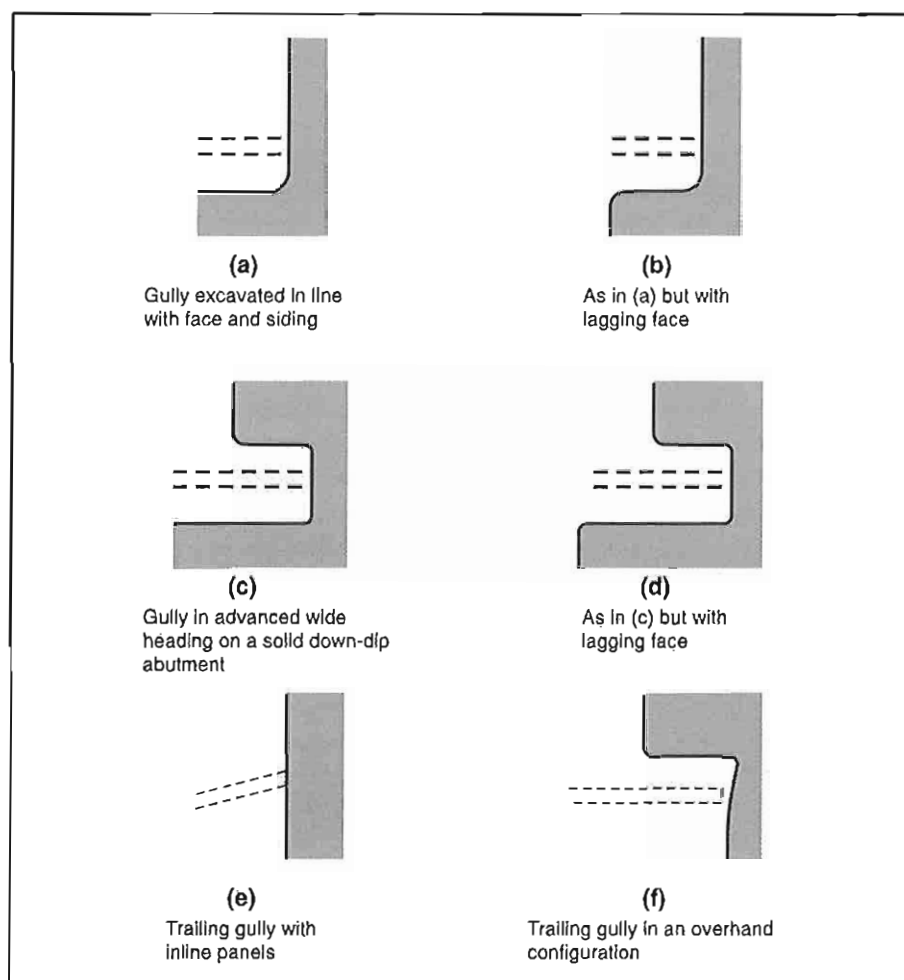


Figure 3.4.15 Appropriate gully configurations for *high-stress* conditions.

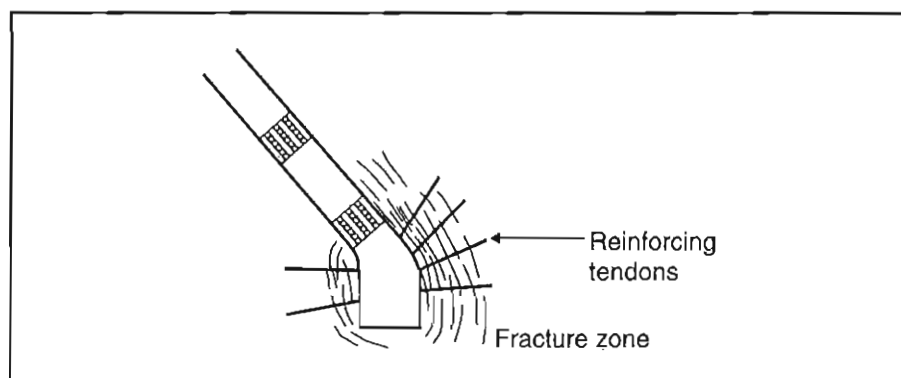


Figure 3.4.16 Support of deep steep-dip gullies, without sidings. Note orientation of tendons.

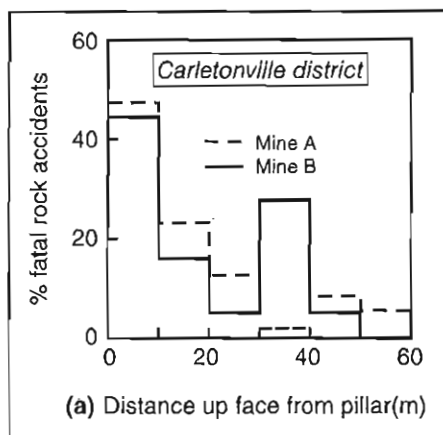
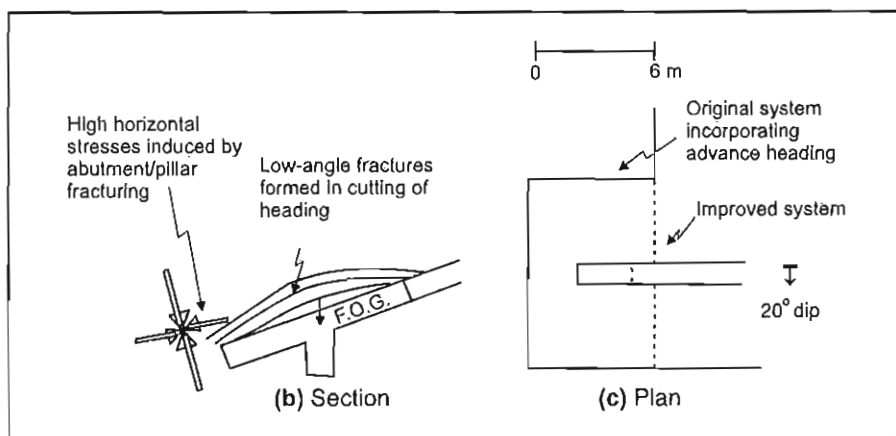


Figure 3.4.17

a, b: Gully problems immediately up-dip of highly-stressed stabilizing pillars;
c: partial solution by removing or greatly shortening advance heading.



In order to avoid these conditions, *shoulders* are often excavated on reef either side of an advance gully or raise. Properly shaped **wide raises** excavated under high stress conditions swing the orientation of the fractures in the gully sidewalls to nearly perpendicular to the direction of advance, creating more stable conditions on which to construct packs. The fractures in the hangingwall are flat dipping but straight and can therefore be supported more effectively by rockbolts or reinforcing tendons. In wide raising, the shoulders are often excavated at a natural angle of about 70° to the direction of advance – Figure 3.4.18a – which also helps to direct the blast throw to facilitate cleaning. Fractures are thereby created which strike into the gully sidewalls at the angle of the face; the hangingwall fractures have a similar strike and, although they curve over the stoped out area, are easily secured (if joints parallel or slightly oblique to dip, do not occur too frequently).

It is appropriate to blast the full gully face simultaneously with the shoulders. Lifting the footwall behind the face to form the gully is a possibility, but has been known to destabilize the gully sidewalls if pack support has not been installed ahead of the footwall lifting. Under no circumstances should a heading be allowed to advance ahead of the shoulders, as this creates very dangerous fracture conditions with the possibility of serious falls of hanging – Figure 3.4.18b. Convergence in a wide raise is small, therefore during the development stage the temporary support must be stiff, preferably

utilizing props or stiff elongates [EPS support units provide both early stiffness and long-term yieldability, as do the latest generation of low-mass concrete packs].

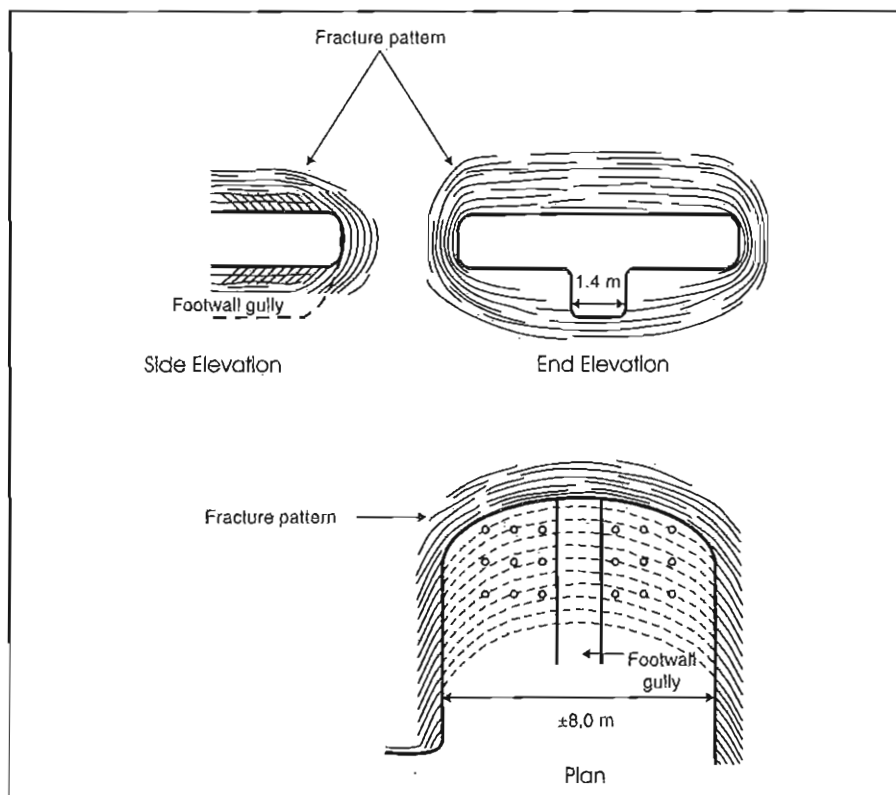


Figure 3.4.18a Wide raising in high-stress conditions.

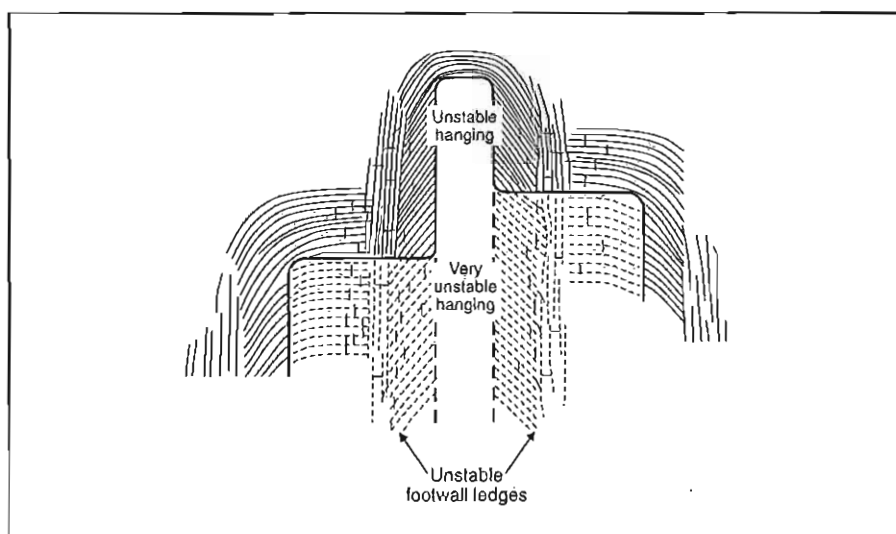


Figure 3.4.18b Wide raising in high-stress conditions – POOR layout.

Where *bulk movement* of the sidewalls of dip gullies or hangingwall buckling failures manifest in the more ductile rock masses, these can be inhibited by cutting slots in the footwall or hangingwall adjacent and parallel to the gully – Figure 3.4.19.

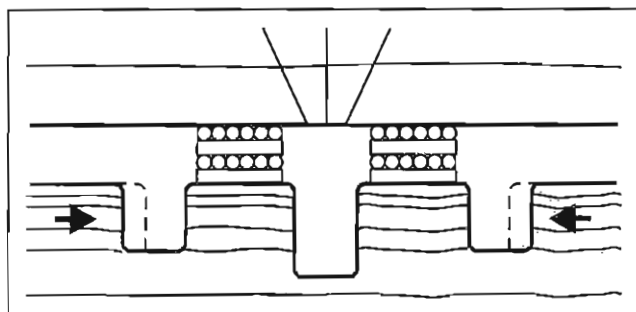


Figure 3.4.19 Cutting sacrificial slots to relieve excessive dip gully sidewall closure in ductile ('plastic') footwall rock types.

Sound **blasting practices** are particularly critical in the excavation of gullies of all types. Severe gully sidewall damage can arise with overburdened, misaligned or mis-fired blast holes. By particular attention to the use of appropriate burdens, accurate drilling and correct detonation sequencing, a significant reduction in gully stability problems can be anticipated.

3.5 LAYOUTS IN SPECIAL MINING AREAS

'Special area' situations which arise during the course of normal mining operations include remnant extraction, the negotiation of geological structures, anomalous stress conditions, the re-opening of collapsed or rockburst damaged panels, and environments requiring the use of caving procedures.

The official 1996 'Guidelines for the compilation of codes of practice' define special areas as follows: *'During the course of routine mining an increased risk of rockfalls or rockbursts may develop. Such areas requiring additional attention and precautions must be designated special areas'*. Thus, mines are required to put in place a strategy to identify and deal with the increased risk of rockfalls and rockbursts which may develop during the course of routine mining operations. Having identified such special areas, the mine is required to make appropriate and properly documented modifications to mining methods and support.

3.5.1 Remnant Extraction

Remnants are blocks of ground surrounded by extensive mining, usually created during the final stages of the extraction of a mining area. As remnants become smaller and pass through critical minimum dimensions, stress and ERR levels, stress-induced fracturing and seismicity all tend to increase. Despite accounts of improved mining conditions during the *late* stages of remnant extraction, remnant precautions need to be maintained until extraction is completed.

Removing a highly-stressed remnant from within an extensively mined-out area will result in the transfer of this stress onto peripheral solid areas or geological weaknesses, and this can lead to adverse seismic side-effects in these areas. These possibilities need to be considered and their severity assessed by well thought-through and appropriate computer modelling.

Several layout considerations apply to remnant mining:

- (a) The stoping layout should be such as to mine toward the largest or closest solid area. These alternatives should be assessed with the assistance of suitable numerical modelling.
- (b) In general, the stoping layout should attempt to minimise seismic or ground control hazards. This may be achieved by mining away from geological features, or by approaching minor features obliquely (angle of approach $> 35^\circ$). Particular care should be taken when mining near structures known to be hazardous.
- (c) Remnants should not be split and mined in two different directions.
- (d) Panels should not approach each other from either side of the remnant. One side should be stopped and suitably supported.
- (e) Large leads/lags between panels should be avoided.
- (f) When remnants are elongated, consideration should be given to mining in the direction of elongation. This reduces the size of ESS lobes and thus the likely magnitude of seismic activity.
- (g) Particular consideration should be given to changing the direction of mining during the final stages of remnant extraction, for example, final up-dip mining of a dip-oriented remnant.
- (h) Joint sets and fracturing from previous mining should be considered in the choice of mining direction and in the specification of support, in particular the spacing and areal coverage requirements.
- (i) Faces should be blasted regularly to limit time-dependent deterioration. The advance per blast should be kept as small as practically possible. This will facilitate rapid cleaning and adequate access for support installation, as well as limiting hangingwall damage.
- (j) At least two separate routes for access and egress should be established for each face, and kept clear at all times.
- (k) Several serious crush-type rockbursts have occurred in rectangular remnants whose least width was in the order of 10-20 m. To prevent this, the use of *preconditioning* should be seriously considered, and the faces aligned to form an arrow-shaped apex to the remnant which would then crush in a more stable manner.

The principles governing remnant mining may be applied to the extraction or partial extraction (where required) of stabilizing/boundary/barrier pillars, and in the cutting of ventilation or overstoping slots through very high-stress ground. Comprehensive numerical modelling is desirable to determine not only the changing conditions on the pillar during extraction, but also on other regional pillars and remnants in the vicinity.

3.5.2 Mining Adjacent to and Negotiation of Geological Structures

Geological structures are generally recognised as potentially hazardous in respect of both rockfalls and rockbursts and require special attention in mining at all depths.

Smaller features may be stabilised by the judicious use of local support. However, larger or repetitive features may require the modification of stopping layouts in order to minimise the hazards posed by these structures.

Faults, dykes and other geological discontinuities create **rockfall** hazards by defining or contributing to the formation of unstable hangingwall blocks, particularly at shallow and intermediate depths and in de-stressed environments. These blocks can vary in size from small to very large. Geological discontinuities in the footwall may also adversely affect stability particularly in and around gullies. In addition, major structures are often accompanied by adjacent sympathetic features which can lead to friable ground conditions at all depths. Thus, where possible, both mining panels and the overall face configuration should approach geological structures obliquely ($> 35^\circ$) to avoid exposing large contiguous areas of potentially weak or unstable ground and thereby reducing the possibility of large falls of ground or hangingwall collapse. Additional support precautions are also indicated.

In very high-stress environments it may become necessary to *de-stress* re-development operations by over- or under-stopping. The decision to mine through a structure or to re-establish the layout is largely determined by the dip, throw, width and sense of the feature and the geometry of the mining situation. Particular care must be taken when geological structures make a shallow angle relative to the reef plane. A re-development layout exploiting protective under-stopping is shown in Figure 3.5.1.

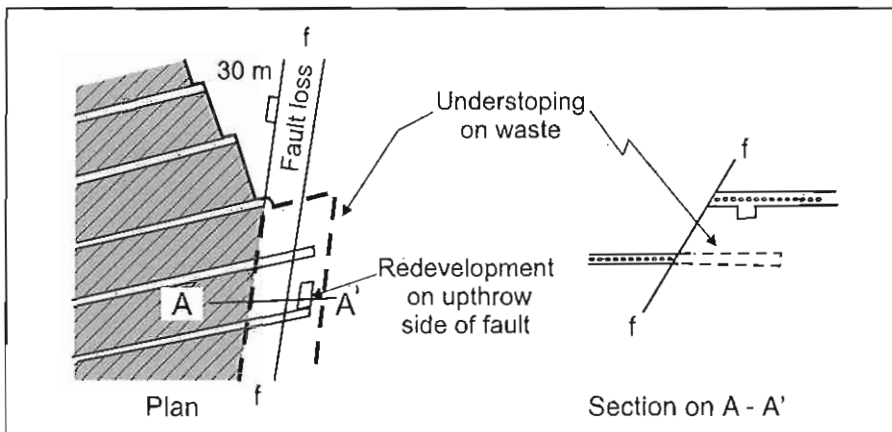


Figure 3.5.1 Negotiation of an upthrow fault loss in a deep mine.

The presence of geological disturbances has an even more significant impact on **rockburst** incidence. The level of understanding of the behaviour of geological structures is often limited, and emphasis should be placed on gathering relevant data on their composition and properties, as well as their seismic history. Overall strategies to reduce seismicity, such as the use of regional support systems, the mining out of geological features or use of bracket pillars (section 3.3.3), and the principle of 'mining away' from such structures (Figure 3.4.8), are vitally important. Mining should, wherever possible, approach geological structures in such a manner as to limit the magnitude and extent of stress and ESS increases, which could lead to violent failure. The 35° obliquity rule (Figure 3.4.8) is an illustration of this principle.

As in other special mining situations, small mining spans are desirable, and additional support precautions are a necessity. Increasing the intensity of coverage of seismic monitoring, with daily analyses, is also a desirable course of action.

3.5.3 Re-Establishing Panels

Re-establishing panels after massive falls of ground and rockbursts is commonly achieved by open-raising within the confines of the existing excavation, or by re-raising beyond the collapsed panel.

The positioning of on-reef re-development is determined largely by the magnitude of the stresses ahead of the stope, the fracture zone and the general ground conditions. At all depths, the size of any pillar left between the re-raise and the stope face should be small enough for it to yield in a stable manner, or be large enough not to become burst-prone. Local knowledge, past experience and computer modelling can be used to help determine pillar size, excavation dimensions and development geometry for each situation. A further general consideration is that mining on neighbouring panels should not be such as to affect re-development operations unfavourably.

Re-raising, and re-development in high stress situations, should preferably be by means of well-supported wide-ends. Heading leads, if any, should be kept to a minimum. These considerations are illustrated in Figures 3.5.2 and 3.5.3.

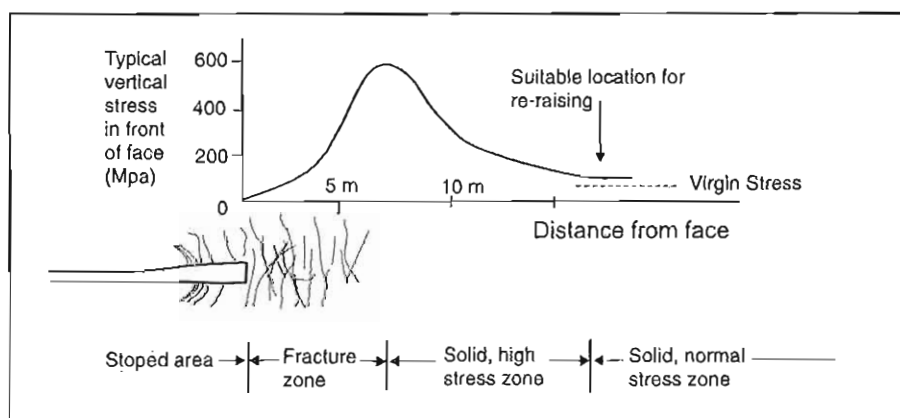


Figure 3.5.2 Fracture zone and stresses in front of typical deep face.

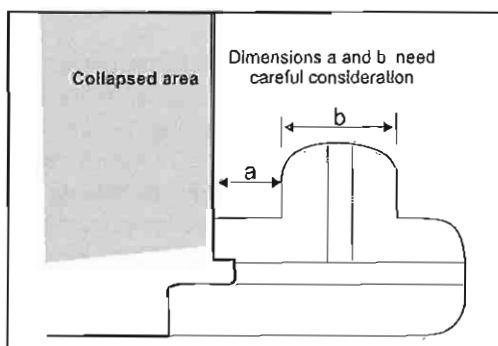


Figure 3.5.3
Re-developing on reef in high stress situations.

3.5.4 Mining In Anomalous Stress Conditions

Experience has shown that a number of mining areas experience anomalous virgin stress fields, usually associated with the presence of major geological features (Chapter 1.3.1). Observations of atypical stress fracturing in tunnels or unusual dog-eared in boreholes will often be the first indications of the presence of such changed stress environments. In these circumstances it may be critical to establish, by direct measurement, the actual stress regime that is present; as the stability of pillars, tunnels, faults and dykes in the vicinity may be adversely affected.

The measured anomalous virgin stress tensor should obviously be used in any numerical modelling of the area, and appropriate changes may need to be made to the mining layouts involved. As an example, if a low minimum horizontal stress ratio was measured ($k_{\text{min}} < 0.4$) in a particular direction, ESS levels and likely seismicity would be high on faces mining in this direction and serious consideration would need to be taken to swinging them into a more oblique orientation, or implementing other forms of regional control.

3.5.5 Caving

Cave mining is the term used to describe the practice of supporting only the immediate working area of advancing panels, and allowing the unsupported back area to collapse in a more or less controlled fashion. Conditions which need to be met to allow this efficient mining method to be successfully applied seem to be the presence of a well-layered and mobile hangingwall and an argillaceous ('ductile') reef environment, mined at least at medium depths. When these conditions are encountered, caving in fact seems to be the *only* viable mining method.

(a) Caving practice

Cave mining is currently practiced at only one mine (Hartebeestfontein Gold Mine), but has, since 1940, been applied with varying degrees of success at some 9 other gold mines. Two similar versions of the method are currently in use at Hartebeestfontein. These are: 'stick caving' where new rows of mine poles are added as the face advances while the fifth row is blasted out to allow the cave to advance, and 'prop caving' in which three rows of rapid-yield hydraulic-props are employed with the back row being removed remotely and leap-frogged forward in a systematic fashion as the panel advances. Successful cave mining in general requires the presence of active, strong support in the face area.

(b) Caving theory

Cave mining theory is founded on the concept of 'macro-plastic' deformation of a well-bedded rockmass, in which the interfaces between individual beds have a low resistance to shear movement and permit the rockmass as a whole to 'flow' into the stopping excavation.

In deep tabular excavations, the load previously carried by a stoped-out area is transferred to the abutments, causing failure of rock at the face and transfer of the stress peak deeper into the rock mass. If macro-plastic flow is permitted to occur, further

lateral restraint is lost as rock moves towards the stoped out area. The stresses become more smoothed-out: the stress peak is lower and is shifted further into the abutment and stress-induced face fracturing becomes less intense.

If, however, macro-plastic flow is opposed or restricted, the stress distribution ahead of the face tends to revert back towards the brittle condition, resulting in higher stress levels and more intense fracturing at the face. This in turn leads to higher horizontal stresses being generated in the hangingwall and footwall of the stope excavation. As a consequence, footwall heave and hangingwall buckling occurs, and, if not relieved, serious deterioration in conditions and collapse of the face area can follow.

In addition to the intensity of fracturing, caving can modify the geometry of the curved fracture front. The inflection plane, which in non-cave stopes lies close to the horizontal plane of symmetry of the excavation, is shifted up into the hangingwall in caved stopes, to coincide more closely with a plane of symmetry that includes the cave region. The curvature of the fractures is also less sharp. Hangingwall fractures at the face are thus more steeply oriented, and may even dip towards the back area. These modifications are thought to have a beneficial influence on the stability of the hangingwall.

Rock failure and the macro-plastic flow processes which occur ahead of the face, and which are responsible for inducing horizontal stresses in the hangingwall and footwall, are time-dependent phenomena. The strategy of shedding a portion of horizontal hangingwall clamping stress for the purpose of reducing and transferring stress from the immediate face area is highly dependent upon the continual regeneration of horizontal stress by the attainment of a regular and adequate rate of face advance. If regeneration does not take place, back-area caving will result in the unclamping of the vertically sliced hangingwall strata at the face. The method is for this reason an anathema for deep mines in brittle rock where high horizontal clamping stresses are vital for the stability of the stope hangingwall.

The arguments explaining the mechanisms of caving can be extended to explain why the use of backfill in pseudo-plastic mining environments can result in serious deleterious consequences associated with the build up of high horizontal stresses in the weak well-bedded hangingwall and footwall strata. The two approaches (caving vs backfill) are thus mutually exclusive.

The role of face-to-support distance in the maintenance of stable hangingwall conditions at the working face is as vital in caving stopes as it is in non-caving stopes. The spacing and installation of support as a function of face advance is particularly important where the immediate hangingwall layer is intrinsically weak.

(c) Cave Performance Indicators

The achievement of the delicate balance between inhibiting bedding plane separation, maintaining adequate lateral compression of the hangingwall, and allowing sufficient lateral movement of strata to reduce stress levels ahead of the face is a relatively straightforward matter in practice. It involves mining personnel monitoring the following well-defined 'malfunction indicators' on a daily basis, and taking the necessary action (induction of a cave, slot-cutting, or increasing the rate of face advance) to correct any detected imbalances:

- *Fracture inclination.* The inclination of face-parallel fractures in the middle of cave panels should be vertical or, better still, should dip steeply towards the back area. Fractures that are inclined over the back area result from excessive stress on the face, indicating that an adequate cave has not taken place.
- *Footwall heave.* The occurrence of heaving or doming of the footwall behind and between the first or second rows of support (which differs noticeably from punching of support units into the footwall) is also caused by increased stress at the face, indicating that lateral movement is being restricted in either the footwall, the hangingwall, or both.
- *Excessive barring.* One of the first indications of excessive stress build-up at the face is an unusual increase in the amount of barring or dressing required.
- *Jumpers sticking.* Percussion holes drilled into the stope face are prone to rapid scaling under high stress conditions, and take on a flat elliptical shape. In addition, closely spaced face-parallel fractures tend to form offset steps in drill holes, with difficulty in extraction of the drill steels.
- *Excessive closure.* Abnormally rapid closure is an indication of high stress at the face, and soon causes premature failure of support units: sticks start to break in the second or third row, instead of in the fourth row; or props run out of travel and start to freeze.
- *Hangingwall buckling.* Buckling of the hangingwall between the front row of support and the face is an indication that excessive support resistance is preventing shear movement between hangingwall layers, with incipient instability and falls of ground at the face. The condition is fairly rare, as other factors mentioned above will have clearly indicated the need for stress-relief measures long before this stage is reached.

3.6 LAYOUTS FOR SPECIAL OREBODY GEOMETRIES

Special mining methods which arise as a result of the nature of the orebody itself include wide-reef, multi-reef, and steeply dipping operations.

3.6.1 Wide-Reef Mining Layouts

Wide-reef mining is generally accepted as that in which the stope width exceeds 2.5 m, and is currently practised in South Africa at all mining depths. There are a number of methods of wide-reef mining, for example conventional mining, pillar mining, double-cut, multi-cut, massive mining, to name a few. The choice of method depends on the depth and geotechnical conditions involved.

Conventional narrow-reef stoping methods become inappropriate at a limit (typically about 2.5 m) which is dictated largely by the decrease in efficiency of conventional support units with increasing stoping width.

At depths down to about 1200 m, **pillar mining** is the most common method of extracting reefs in the width range 2–6 m. This may take several forms: room and pillar, rib pillar or 'drift' mining. An important consideration in pillar mining is the use of *barrier pillars* to prevent large-scale instabilities and to help limit the necessary size of in-stope pillars (section 3.4.1). Spans between barrier pillars should be

several times less than the mining depth, and should not exceed about 400 m.

Pillar design requirements for wide-reef pillar mining in hard rock types have not been thoroughly investigated, and strength formulae have not been as reliably determined as in the case of coal mining - section 3.2.8. Consequently, stable geometries for these workings for given depths, mining spans and mining heights are usually based on local experience supplemented by laboratory test results on simulated pillars, or by in situ back analyses. Underground monitoring (Chapter 10.3) provides essential input in assessing the performance of pillars, hangingwall conditions, and the control of closure. The design of *crush pillar* systems at high stope widths is particularly uncertain, and in situ monitoring of these layouts is imperative.

The use of self-standing *cemented fill* in drift or primary stopes may allow later partial or complete extraction of the in-stope pillars. Backfill for such applications needs to be designed to suit the particular conditions and mining geometry.

Cognisance should be taken of proximity to highly stressed in-stope and barrier pillars when siting travelling-ways, crosscuts, and other accessways. This is of particular importance if *trackless mining* methods are employed, as these large excavations are vulnerable to falls of ground if high field stresses due to pillars are present.

Partial extraction (pillar mining) of wide reefs is not generally practised below 1500 m due to increasingly unacceptable extraction ratios with increasing depth. An effective method of extracting up to about 5.0 m wide orebodies at depth is to divide a wide stope into two cuts in which the first (upper) cut leads the lower to provide space for support and practical mining. This geometry is illustrated in Figure 3.6.1. Layout considerations for **double-cut mining** are otherwise very similar to those for conventional narrow stopes. An option which can be considered in deep situations is **multi-cut mining**, in which successive portions of a payable channel can be extracted by conventional methods once the first cut has fully closed and virgin stresses have virtually regenerated.

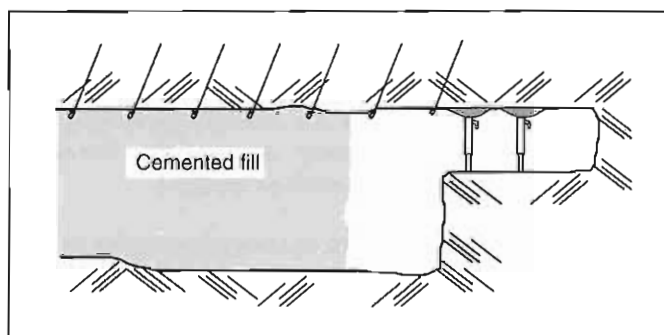


Figure 3.6.1 Section through double-cut mining panel, showing layout and support

When orebody geometry is favourable, or when conventional or pillar mining cannot be applied because of dip, width or depth limitations, common base metal **massive**

mining methods can be employed. Layouts and choice of mining method are very dependent on orebody geometry and thus require individual assessment.

In wide-reef mining at all depths, the use of appropriately designed hangingwall bolting is strongly recommended (elongate and pack supports become increasingly ineffective at stope widths exceeding about 2 m). As depths increase, careful control of spans and stope face-to-fill distances need to be maintained even if hangingwall bolting is in use.

3.6.2 Multi-reef Mining Layouts

When designing stoping layouts for multi-reef mining, a number of additional variables need to be considered. These include the distance between reefs, the effect of pillars and remnants and the influence of the surrounding rock type. In addition, mining depth strongly influences layout requirements. A good general rule, applicable in most situations, is to plan to mine in a **top-down direction**, i.e. to mine the upper reefs first – this will greatly assist in maintaining normal support integrity on a stable footwall.

However, due to the large number of variables which determine extraction sequence and support requirements when planning multi-reef stoping, each case needs to be approached as a unique problem. Computer modelling can be of considerable assistance here, but, because of the complex numerical and geometrical problems involved, particular care needs to be exercised in the interpretation and application of numerical results.

Two basic situations are encountered in multi-reef mining: where the second and successive reefs are mined *subsequent* to the extraction of the first, and where two or more reefs are mined *simultaneously*. The layouts appropriate to these situations differ according to mining depth, and the cases that arise are discussed individually below.

(a) Subsequent extraction (shallow mining conditions)

In shallow mining conditions, superimposed regional pillars will normally be used, but when the second reef is mined, a middling is formed whose stability becomes an important consideration. As a general rule, middling stability decreases with increasing mining span on the second reef, until at a span:middling ratio of some 3:1 (this figure is very dependent on local conditions), consideration needs to be given to the use of appropriate counter-measures.

Thus, if a **lower** reef is being mined, stope pillars or cemented backfill should be used to limit spans and improve middling stability. Gullies should not be sited beneath upper reef remnants or pillars. If an **upper** reef is being mined, and if middling stability is a problem (causing footwall deformation into the lower stope with consequent lethally-dangerous removal of support resistance from conventional support units in the upper stope), consideration should be given to first backfilling the previously mined lower reef horizon, which is usually accessible at shallow depths. In general, the desirability of **subsequent** extraction of the **lower** reef in shallow narrow-middling situations cannot be over-emphasised.

(b) Subsequent extraction (intermediate mining conditions)

At intermediate depth, total stope closure can occur over large areas of the previously-extracted reef horizon. The magnitude of regenerated stresses can approach or even exceed virgin stress levels (Figure 3.6.2). The orientation of these stresses at the second reef horizon can be significantly different from the original stress orientations, depending on stope geometries and the middling between reefs.

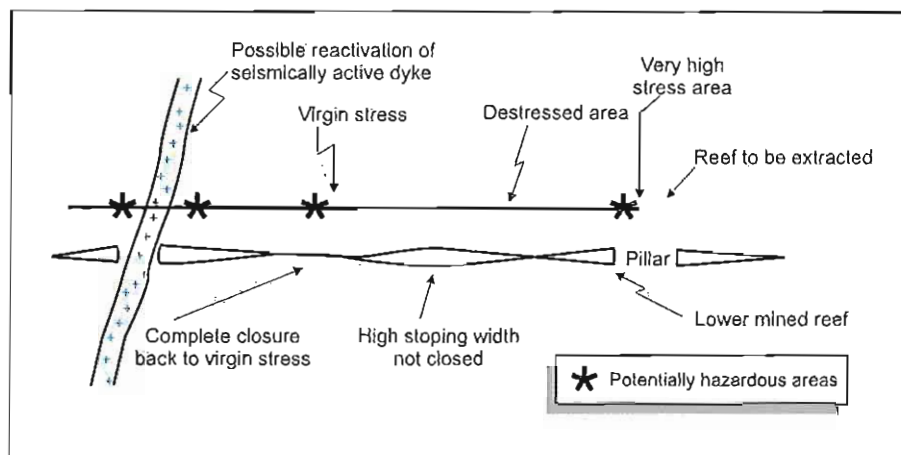


Figure 3.6.2 Conditions encountered during the mining of a second reef.

If the stope middling is small (less than about 20 m), stoping on the second reef may intersect the stress fractures formed during the extraction of the first reef. An oblique direction of secondary mining in relation to previous mining fracture orientations and any dominant jointing can significantly improve hangingwall conditions.

Where pillars, remnants, fault losses, localised filling or narrow stoping widths are present on the reef mined first, stresses well in excess of virgin stress can be regenerated and these will significantly affect mining conditions during the extraction of the second reef. Such stress concentrations on the second reef should be determined by prior numerical modelling, and the feasibility of the extraction assessed. There may be numerous areas where the middling distance and stress levels are such that mining on the second reef horizon will be difficult or impossible, and superimposed pillars of unmined ground will have to be left. If mining is judged to be feasible, then 'remnant' mining procedures (section 3.5.1) should be applied. Stoping operations on the second reef horizon should not commence in the vicinity of such stressed areas, nor should service excavations traverse these without special precautionary measures being taken.

There is evidence that the mining of a second reef can reactivate seismic activity, particularly on transecting geological structures, in which case implementation of rock-burst control measures will be necessary.

(c) Subsequent extraction (deep mining conditions)

In deep mining conditions, total stope closure is common over large areas of the extracted reef horizon. Stress levels approaching the original virgin stress magni-

tudes are regenerated through these closed areas and will exceed virgin stress levels in localised areas of low stopping width, under pillars, or in backfilled areas where uniform area filling was not achieved. If partial extraction was previously practised, the stress levels in the vicinity of the stabilizing pillars will be extremely high. The considerations discussed for intermediate conditions (see (b) above) apply with increased emphasis. In general, extreme caution should be exercised in the extraction of highly stressed zones on the second reef, and best practice would be (guided by numerical modelling of stress and ERR levels) to leave these unmined in the form of superimposed pillars. Their location, particularly as depth increases, is critical and should be regularly checked by survey. The siting of tunnels and service excavations should avoid traversing such areas. Seismic activity on geological features will also often be reinitiated by subsequent reef extraction activities, requiring use of appropriate counter-measures (section 3.5.2).

(d) Simultaneous extraction (shallow mining conditions)

In shallow mining conditions, the lead or lag of extraction of one reef on the other is of secondary importance provided that the middling between the reefs remain stable. This is primarily dependent upon the middling thickness and the mining spans, as in (a) above, and consideration should be given to the use of pillars or cemented backfill in the lower stope to maintain middling stability.

(e) Simultaneous extraction (intermediate mining conditions)

In these conditions, the interaction of the stress concentrations at each of the stopping faces begins to become important. To minimise these interactions, panels should not be mined in line. One of the reefs should lead the other by a distance such that the abutment stress from the leading reef does not affect the face of the trailing reef. This distance can be determined by numerical modelling of each specific situation. In these circumstances the trailing reef will be destressed and will enjoy favourable mining conditions. Overall stope orientations should be similar, but face orientations on the two reefs should be oblique to one another if the middling is narrow to avoid encountering adverse fracture patterns in the trailing stope.

The considerations regarding the presence of local stress concentrators (pillars, backfill, variable stopping widths) discussed in (b) above also apply to simultaneous extraction. Thus, for example, if pillars or remnants are left in the leading reef, serious consideration should be given to leaving superimposed pillars in the trailing reef to avoid the hazards of mining in severely over-stressed ground.

(f) Simultaneous extraction (deep mining conditions)

Although no examples exist in practice, the considerations given in (e) above should apply with increased emphasis to deep mining situations. Numerical modelling together with actual measurements of closure rates on the leading reef will allow estimation of most favourable lag distances, taking into account the presence of regional support (pillars and backfill) used in the leading stopes.

3.6.3 Steep Mining Layouts

Stopping is considered to be **steep** when the dip exceeds 35° . Only a brief treatment of the specific problems of steep mining layouts will be given in this book.

A steeply dipping reef imposes practical constraints on layouts which do not arise with flatter dipping reefs. For example, cleaning problems arise with sidings on the down-dip side of gullies, and consideration should be given to using well-supported versions of the layouts illustrated in Figure 3.4.12 with minimal ASG dimensions.

Face geometries (overhand, breast or underhand), and the disposition of haulages and crosscuts, depend on local conditions rather than on specific steep-dip considerations. However, secondary footwall drives developed close to the reef plane to facilitate boxholing can be damaged at depth if they are advanced beyond the stopping abutments. The geometry of stope faces in relation to these tunnels is an important layout consideration (Figure 3.6.3).

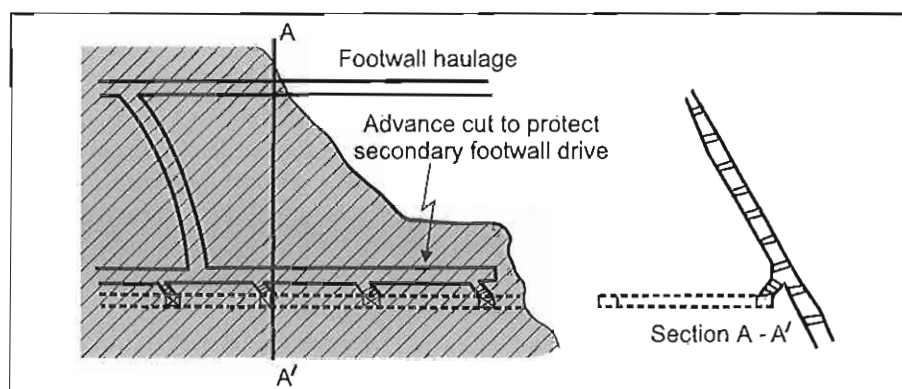


Figure 3.6.3 Typical steep reef face: drive layout.

A related common pitfall is to underestimate the extent of overstoping required to protect service excavations in deep steep-reef situations. The following rough guide may be used in some situations: overstop by distances equal to the vertical distance between the service excavation and the reef horizon, measuring from the point vertically above the service excavation.

In steeply dipping reefs, the virgin horizontal stresses become increasingly important in controlling mining conditions. Their magnitude should preferably be established by in situ measurements to facilitate the application of numerical modelling to mine layout design.

4.1 INTRODUCTION

Following on the design of good mine layouts and regional support systems, and the implementation of sound strata control practices, **stope support** is the ultimate strategy to combat the hazards of rockfalls and rockbursts. The primary function of stope support is to stabilise the rock mass immediately surrounding stoping excavations, that is, the zone of rock subjected to tensile stresses around shallow and intermediate stopes, and the zone of fractured rock which behaves inelastically around intermediate and deep stopes.

The great variability in the stope support design problem in the gold and platinum mines is exemplified by the fact that over 130 different support systems are currently employed on the mines, comprising different types, combinations and spacings of units. The reasons for this variability stem from the large number and range of factors which objectively contribute to support requirements. Nevertheless, it is probable that not all of these support systems are optimal, either from a safety or an economic perspective, and that there is considerable scope for *rationalisation* of support designs.

Support requirements in essence are defined by the thickness of potentially unstable strata that needs to be supported, its blockiness, the amount of quasi-static deformation (closure) of this strata, and whether it is likely to be subjected to dynamic (seismic) ground velocities or not. With a knowledge of these factors, the support design process can proceed. **Support design** is a process resulting in a support system which is both practical and meets or exceeds by a factor of safety the support requirements for a particular geotechnical environment.

The main factors which control support requirements are:

- The stratigraphy of the immediate hangingwall (and sometimes footwall);
- Minor geological structures such as joints and bedding planes or partings;
- Mining depth, which defines the stress regime around stopes and roughly correlates with stress concentrations around excavations and therefore with stress fracturing and the rate of closure;
- The presence or lack of seismicity.

Special mining environments, such as wide-reef mining, proximate multi-reef mining or steep dip mining, certainly influence support requirements, but are considered as special cases and are treated separately later in this chapter. Other factors, includ-

ing stoping width and stope closure rate, do not significantly affect support requirements as such, but do influence the design of the support to meet those requirements.

The support design process involves a logical sequence of actions starting with the delineation of geotechnical areas ('ground control districts'), and ending with the monitoring and evaluation of the implemented system. This process is depicted in the flow chart in Figure 4.1.1. This chapter discusses the factors governing the design of stope support systems, general characteristics of support systems, and support requirements under various stress conditions, and then elaborates on the design process. The final sections describe typical support systems, and go on to discuss support for special mining situations and special orebody geometries.

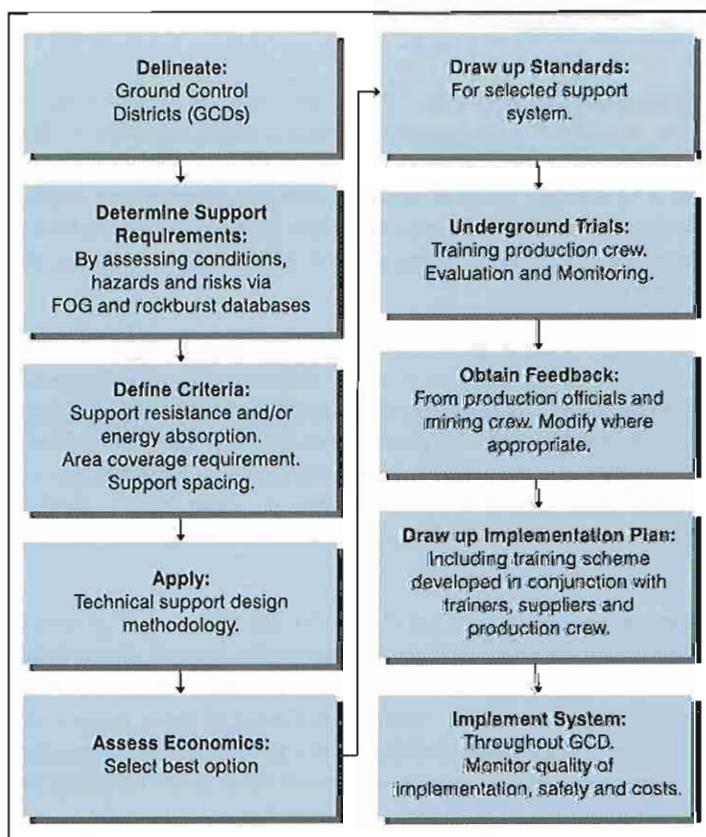


Figure 4.1.1 The Support Design Process

4.2 FACTORS GOVERNING THE DESIGN OF STOPE SUPPORT SYSTEMS

The design (Figure 4.2.1) of any support system depends upon knowledge of three fundamental aspects:

- The nature and extent of the rock to be supported.

- The deformations to which the support system will be subjected, normally defined by the **stope closure** profile, and whether dynamic closure (**rockbursting**) is likely.
- The generated force (load) **characteristics of the support** systems available for installation, and the reinforcing (and sometimes damaging) effect this has on the strata against which the support has to bear.

The factors influencing these aspects are discussed in the following sections.

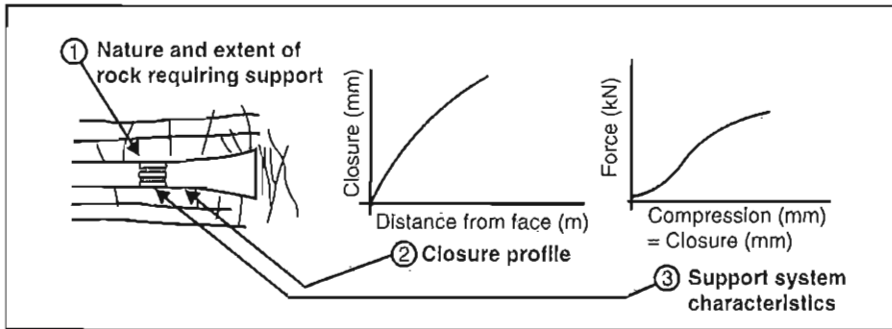


Figure 4.2.1 Aspects governing stope support design.

4.2.1 Mining Depth

Significant changes in the stress conditions, to which mining excavations are subjected, occur as the depth of mining increases from shallow to ultra deep. The total ('absolute') stresses in the rock mass adjacent to mine excavations are the sum of the virgin stresses and the induced stresses resulting from the size and shape of the excavations concerned. Of the three total principal stresses in the immediate hanging-wall, two or more may be tensile in shallow stopes and in a discontinuous rock mass this can lead to major instabilities. In deeper stopes, due to inelastic effects (mainly crushing and dilation of rock in front of the face), all stresses in the hangingwall may become weakly or strongly compressive. Which of these conditions exists, significantly influences the behaviour and stability of the discontinuous rock mass surrounding mining excavations, and thus strongly influences support requirements.

In addition, the magnitude of the principal stresses will determine whether stress fracturing will actually occur in the rock and, if so, the pattern and orientation of the fracturing: also an important factor in support design.

As mining proceeds to greater depths, the stress concentrations and strain energy stored in the rock mass increase, leading to situations where rapid and violent instabilities will inevitably occur. The depth at which the resulting seismic events become a problem depends on the geological structure, stratigraphy and percentage extraction of a particular area or mining field. Once the problem of significant *seismicity* and rockbursting is recognised, the influence of dynamic loading on excavation stability must be considered as the *dominant* factor in support design.

The influence of mining depth on stress conditions, stress fracturing and rockburst potential is discussed in the following sections.

4.2.1.1 Stress conditions

Two aspects of the virgin stress are strongly influenced by depth below surface. These are the magnitude of the **vertical stress**, and secondly the ratio of horizontal stress to vertical stress - the **k-ratio**. Stress measurements carried out on mines in the Bushveld Complex and in the Witwatersrand Basin show that at depths of less than about 700 m the k-ratio is usually greater than 1 and can exceed 2. Particularly in the platinum mines, the k-ratio can vary significantly over distances of as little as a few hundred metres, both geographically and stratigraphically. As depth increases below 1000 m, the k-ratio tends to about 0.5, with a horizontal stress anisotropy of commonly about 50%.

It is of importance to note, however, that anomalously high or low k-ratios can occur at great depth (>2500 m), usually in the vicinity of particular dykes or faults. The major impact of these anomalies is on mine layout design and tunnel support, and only to a lesser extent on stope support. The k-ratio per se has little influence on stope support design, its main influence being in the location and orientation of fracturing with respect to a given excavation. For example, in areas of high k-ratio in some platinum mines, shallow-dipping fractures emanate from abutments (including the stope face) and from pillars, and curve over to become parallel to the hanging-wall. The hangingwall thus becomes slabbed, and is a factor to be taken into account in the design of support - Figure 4.2.2.

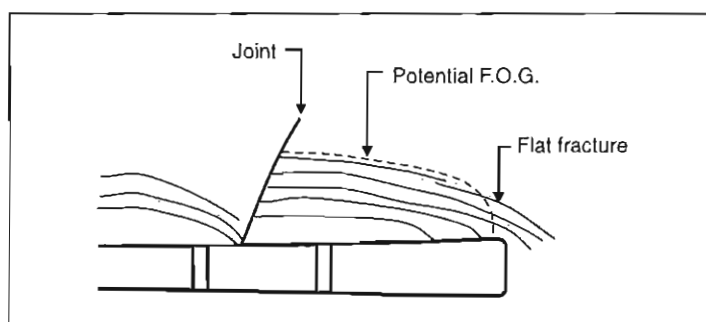


Figure 4.2.2 Flat fracturing in the hangingwall due to high virgin horizontal stresses (high k-ratio)

The magnitude of the vertical stress, which increases linearly with depth, is centrally important. The ratio of this stress to the strength of the rock mass, taking into account stress concentrations caused by the excavation, determines at what depth below surface stress fracturing will occur. Thus stress fracturing is initiated at depths slightly in excess of 1 km in Witwatersrand quartzites, and at somewhat greater depths in the Bushveld. The types and orientations of fracturing around stopes is the subject discussed in the following section.

With respect to stope support, the major impact that depth and stress conditions have on support requirements is whether the hangingwall is in a state of tension or compression. According to elastic theory, the vertical (and sometimes the horizontal) stresses are tensile. This implies that blocks in the hangingwall formed by the intersection of discontinuities of unfavourable orientation, are inherently unstable, and the

function of support is to hold in place these unstable blocks, thus retaining the integrity of the hangingwall.

Another aspect of the hangingwall tensile zone that needs to be considered is the influence of tensile stresses on continuous hangingwall partings, sub-parallel to the stope. If the stresses are greater than the cohesive strength of the parting then bed separation will occur, and if the support resistance of the support system is insufficient to carry the deadweight of the hangingwall up to the parting, a large collapse can occur.

It should be borne in mind that the height of the tensile zone is a function of the stope span and that at spans in excess of 150 m, the height can exceed 40 m. Under these circumstances and usually in the presence of unfavourably oriented faults, the potential for large-scale stope collapses or **backbreaks** exists. Backbreaks involve the collapse of millions of tons of rock, and can only be prevented by the use of pillars or backfill.

The tensile zone and its influence on support design, as described above, applies only to mines where the host rock mass behaves essentially in an elastic manner. Where the stresses are such that fracturing and associated dilation occurs, the rock peripheral to excavations behaves inelastically, causing compressive stresses in the hangingwall which very significantly change the rock mass behaviour and consequent support requirements. The depth at which this transition occurs depends on a number of variables, but can be taken at between 1000 m and 1500 m in strong brittle rocks.

The consequence is that the highly fractured hangingwall rock is clamped horizontally and is substantially more stable than the same rock mass in a tensile state of stress. There is thus the apparent anomaly that, at shallow depth, blocks in the hangingwall are more likely to be dislodged and the height of potential instability is greater, requiring support systems of much greater support resistance and stiffness. However, at mining depths where seismicity is expected, the function of support is changed from merely holding in place loose blocks to one of absorbing the energy imparted to unstable strata by the seismic 'strong ground motions'.

4.2.1.2 Fracturing and associated dilation

In **shallow mines** stress-induced fracturing occurs only under abnormal situations and, in general, the discontinuities in the rock mass which influence support design are the inherent geological partings such as joints, faults and bedding planes. It is therefore necessary to have a reasonably detailed knowledge of the occurrence, orientation, spacing and physical properties of these discontinuities and how they vary across a mine. Where these variations are such that they result in differences in rock mass response or failure mechanisms and hence support requirements, different 'ground control districts' need to be defined, requiring individual designs of support. In making these assessments, the direction of mining with respect to the sets of discontinuities must be taken into account.

Stress fracturing can occur in shallow mines when the k-ratio is abnormally high, Figure 4.2.2. 'Unexpected' stress fracturing can also occur in multi-reef situations where mining on the second reef passes under or over abutments or pillars left on the first reef. Where the two stopes mined are within a few stope widths of each other, the change in conditions occurs within a few metres of face advance and can be dramatic, from a normal shallow mining condition to one equivalent to mining at great depth.

Where excessive spans are mined without the protection of adequate pillars, the first signs of instability are the development of fractures in the hangingwall, dipping at shallow angles towards the face. If these are accompanied by large, typically irregular, tensile fractures near the centre of the span and running on dip, then the onset of a backbreak is imminent and immediate steps, such as stopping mining to leave a barrier pillar, are required.

It is necessary in shallow mining to quantify inherent geological discontinuities for support design purposes, and to have an awareness of any stress-induced fracturing as indicators of abnormal instabilities.

In **deep mines**, stress fracturing regularly occurs in the abutments of stopes wherever the magnitude of the stresses exceeds the strength of the rock mass. Wherever this fracturing occurs, the intensity is such that the stress fractures usually become the dominant set of discontinuities in the rock mass - the average spacing being of the order of 60 mm.

Several types and sets of stress fractures occur; these sets being best identified in terms of their orientation. Intersections of different sets of fractures within the immediate few metres of hangingwall strata or intersections of fractures and geological discontinuities can result in the formation of unstable blocks, wedges or slabs which need to be supported. It is necessary, for the determination of fall of ground mechanisms and hence support requirements, that a clear understanding be gained of the types and patterns of fracturing that occur on a particular mine or reef. The three types of fractures that occur are:

Extension Fractures, which are the most common. They are formed in induced tension, but within a wholly compressive stress field. They develop in the σ_1/σ_2 plane (the plane normal to the minor principal stress σ_3) and are sensitive to changes in the orientation of this plane, to the extent that they will follow any irregularity in the outline of the stope face and curve around from being parallel to the stope face to being parallel to sidings. Extension fractures are clean and planar, and often either initiate or have their propagation terminated on bedding partings or joints. Their strike length seldom extends more than about 8 m.

Shear Fractures, which are 'mining-induced faults', develop in a plane dipping approximately 25° to 35° off the plane of σ_1/σ_2 . They are characterised by the presence of fine comminuted rock flour up to 20 mm thick, the result of crushing of a fine pattern of cross fractures between the bounding fracture surfaces. The sense of displacement is equivalent to that of normal faults, and throws of up to 140 mm have been recorded; however around 20 mm is more typical. Shear fractures are typically spaced at 1-3 m, and cut through bedding partings or other discontinuities. They can extend 20 m or more into the hangingwall or footwall, often as en-echelon arranged segments, and strike lengths of more than 30 m have been recorded.

It is of interest to note that the majority of shear fractures develop without the emission of significant seismic energy, the explanation being that the ultimate displacement observed is the result of the accumulation of many very small displacements which occur as the stope face advances towards the shear fracture. On the other hand

there are several recorded examples of '**burst fractures**' where shear fractures have been associated with seismic events of magnitude > 2 .

Genuine shear fractures should not be confused with extension fractures which have suffered displacement subsequent to their formation. The latter will also have fine rock powder between their surfaces, but the characteristic associated minor fracturing of true shear fractures is missing, and the orientations are those of simple extension fractures.

Tensile Fractures: Although rock is considerably weaker in tension than in compression, the occurrence of fractures resulting from direct tension is relatively rare, particularly in deep mines. The reasons for this are that most of the rock mass is under the influence of compressive stresses, and that, where tensile stresses occur, the magnitude is usually less than is required to fail intact rock.

Vertical tensile stresses in bedded sedimentary or stratified igneous rocks will exploit the pre-existing strata-parallel weaknesses in the rock mass, with the result that bed-separation type failure will occur preferentially to failure of the rockmass. However, there are situations where the bending of beams or plates results in tensile stresses developing on the convex surfaces sufficient to fail the rock in tension. These tensile fractures are usually open, with a clearly visible aperture and are characterised by having an irregular trace in contrast to extension fractures which are straight and planar.

Fracturing in front of stope faces. Petroscope observations in boreholes drilled into the rock ahead of stope faces have given some insight into the processes of fracturing. Figure 4.2.3 is an example of these observations. The upper depiction of the borehole and fractures represents the initial observation, whilst the lower illustrates the fracturing after the face had advanced 2.3 m.

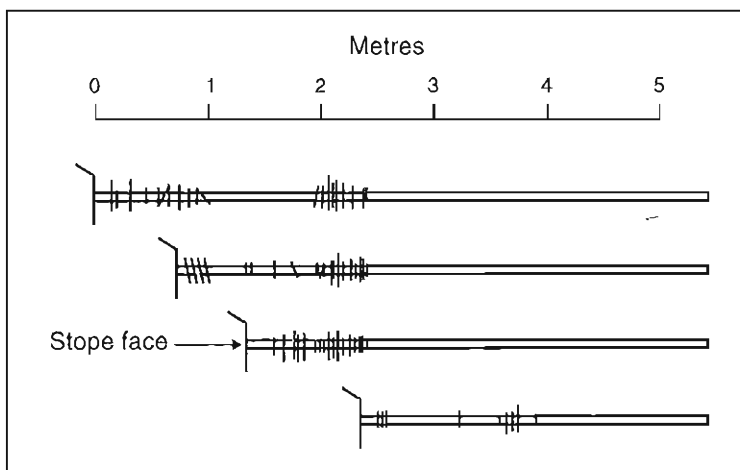


Figure 4.2.3 Petroscope observations of fracturing ahead of an advancing stope face.

What is remarkable about these observations, and which is typical, is that the **fracturing is localised into zones** approximately 1 m wide and separated from adja-

cent zones by virtually unfractured rock also about 1 m in width. Observations in boreholes spaced at 1 m intervals along the stope face indicated that these zones are continuous for at least 8 m down dip, parallel to the face. It was also noted that, while the face was static, very few new fractures were formed although existing fractures might propagate or dilate; the inference being that time-dependent creation of *fresh* fractures is uncommon.

Other observations associated with advance of the face include two significant phenomena. Firstly, a new group of fractures may be formed deeper into the rock mass than the deepest pre-existing zone and separated from it by a band of unfractured rock. These can be considered as **primary fractures** and consist of both shear and extension fractures. Secondly, if the face has advanced into one of the unfractured bands of rock, this may result in **secondary fracturing** close to the face as illustrated in Figure 4.2.3. However, in about 10% of cases this secondary fracturing does not occur and this has a number of consequences:

- (i) subsequent fracturing in the hangingwall over the 'hard patch' takes the form of extension fractures dipping towards the face at $20^\circ - 40^\circ$, leading to unstable wedges as shown in Figure 4.2.4. This type of hard patch formation and preservation is enhanced where joints run sub-parallel to stope faces, with resultant frequent flat fractures, poor ground conditions and 'factory roof' type fall-out profiles.
- (ii) the probability of face strain bursting is enhanced
- (iii) production blast design must take into account the presence of both fractured and sporadic intact rock on the face.
- (iv) mechanised rock breaking must also be capable of mining both fractured and strong intact rock.

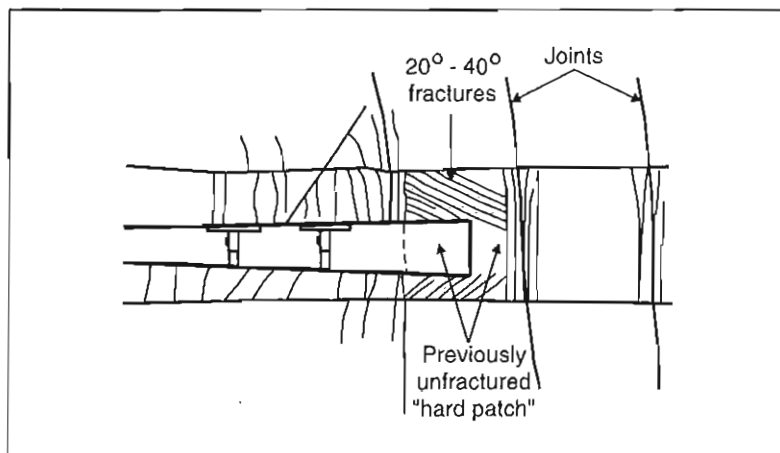


Figure 4.2.4 Development of shallow-dipping fractures over the stope face, in a previous 'hard patch'

The depth of fracturing ahead of the face is directly related to the ERR. At an ERR of about 8 MJ/m² a single shallow zone of face fractures occurs, while at 70 MJ/m² the depth of fracturing increases to about 10 m. There is little increase in the

intensity of primary fracturing with increase in ERR. However, some decrease in spacing of secondary fractures has been noted.

Observations of fractures in stopes and gullies have revealed their extension in the third dimension. Figure 4.2.5 is the classic representation of fracturing around stopes where the immediate hangingwall and footwall comprise similar strong, brittle quartzites.

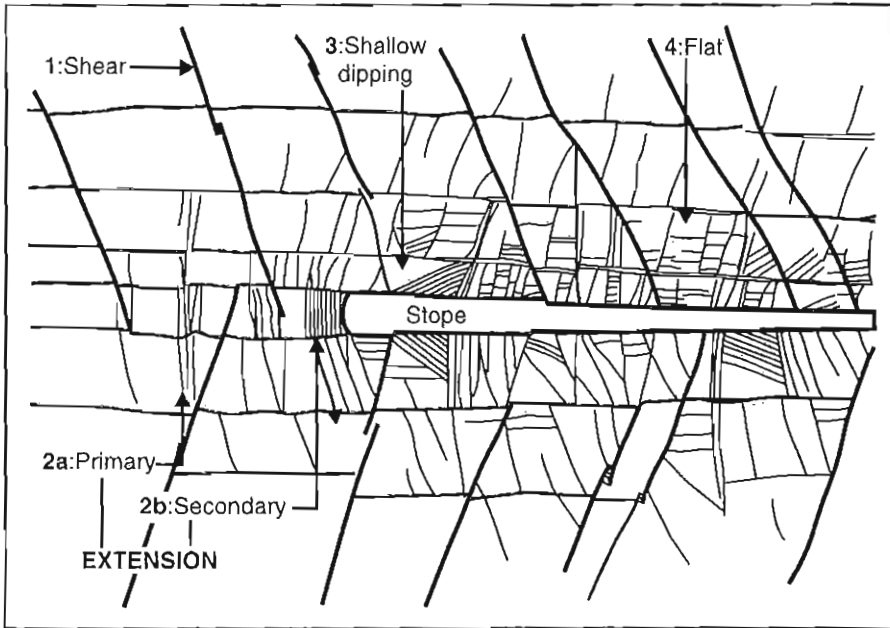


Figure 4.2.5 Stress fracturing around a deep stope mined in strong, brittle quartzites.

The diagram illustrates the remarkable symmetry of the fracturing that often occurs about the reef plane. Three sets of extension fractures numbered **2a**, **2b**, **3** and **4**, and the orientation of the shear fractures numbered **1** are shown.

1. The *shear fractures* develop at the extremity of the fracture zone where the normal stresses and confining stresses are such as to exceed the triaxial strength of the rock. The hangingwall and footwall fractures are mirror images and seldom penetrate across the reef plane to any extent.

2a. *Primary extension fractures* also occur at the extremity of the fracture zone as a result (it is thought) of the reduction in σ_3 due to deformation of the fracture zone closer to the stope face, caused by an advance of the face. These fractures are near vertical in orientation.

2b. *Secondary extension fractures* develop within 2 m of the stope face, principally in the intact rock between bands of primary fractures. Ahead of the face they are almost vertical, but in the hangingwall curl over to dip towards the face at angles down to about 70° .

3. *Low-inclination fractures* dipping at 20° to 40° towards the face in the hangingwall. These young fractures develop primarily close to the stope face where intact 'hard patch' bands of rock are preserved at the stope face, and follow the σ_1/σ_2 plane that would be expected under those circumstances. They usually occur sporadically in patches elongated in the dip direction and measure a few metres in length. They

terminate against steep-dipping fractures indicating the relative time of their development. The flat wedges so formed are inherently unstable and require *special attention* when designing support. An unusual development of this type of fracture has been noted at a number of rockburst sites, where fractures of this nature emanate from the top corner of the face and extend 5 m or so up and back into the hangingwall. The length along the slope of some of these fractures is more than 40 m. They are characterised by a peculiar fracture surface comprised of curved interlocking fragments about 10-20 mm thick and 100-150 mm long. It is assumed that they formed as a result of dynamic loading.

4. *Fractures in the plane of stratification.* The relatively latest fractures to develop in the whole fracture process around stopes are those that form parallel to the stope hangingwall. They occur within the steep slabs bounded by the face-parallel fractures, and terminate against these pre-existing fractures. They thus have limited areal extent, but can be numerous such that the hangingwall is further fragmented and rendered more difficult to support. They develop over or slightly behind the stope face as a result of build-up of horizontal stresses or, possibly, of repeated dynamic loading. Figure 4.2.6 is reproduced from a tracing of fractures in a mosaic of photographs taken in a slot cut into the hangingwall above a Carbon Leader Reef stope. The diagram illustrates, among other features, the intensity of the hangingwall-parallel fractures that can occur.

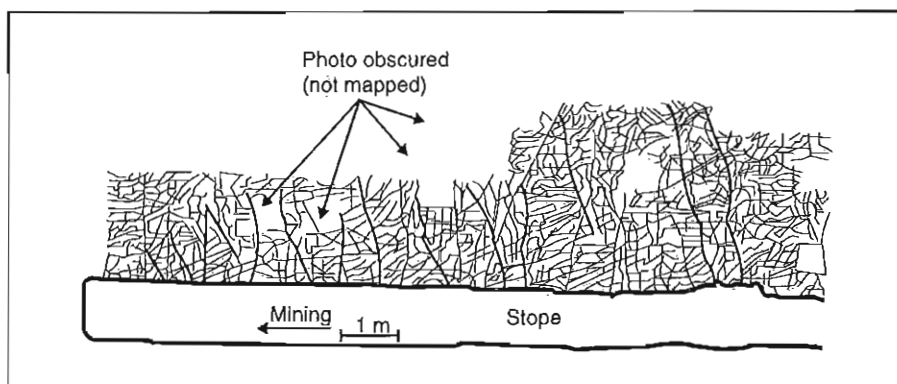


Figure 4.2.6 An example of intense fracturing parallel to bedding.

Figures 4.2.5 and 4.2.6 illustrate the various types and sets of fractures that can occur in a specific geotechnical environment. The relative proportion of the different fractures varies from place to place. In addition, the rock types forming the hangingwall and footwall of stopes can have a strong influence on the orientation and predominant type of fracturing that occurs. For example, the hangingwall of the VCR is either one of two types of lava and, due to an unconformable relationship, overlies a range of rock types. Figure 4.2.7 illustrates the observed fracturing around typical VCR stopes in the different stratigraphies to be encountered. Figures b and c show the presence of an unusual set of hangingwall fractures dipping at shallow angles towards the *back area*.

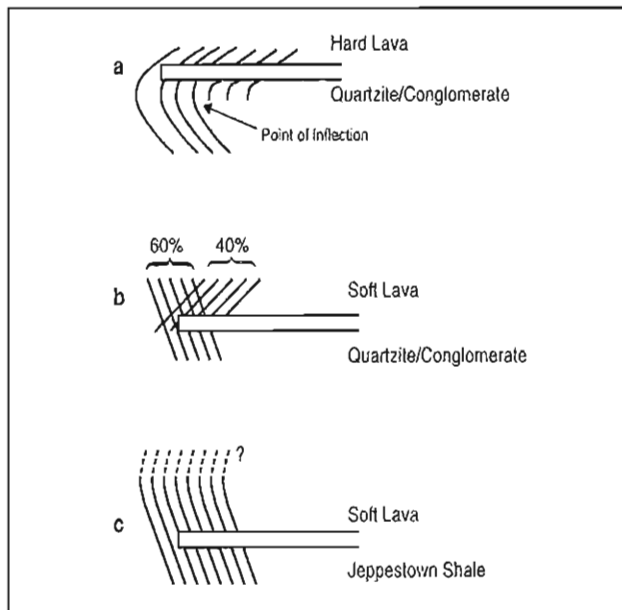


Figure 4.2.7 Influence of h/w and f/w rock types on patterns of stress fracturing (VCR stopes)

In plan, the extension fractures closely parallel the outline of the stopes, shallowing in dip where they curve around corners such as in a leading panel. Fractures in this area can curve over the corner to ultimately daylight in the stope below the gully, with the result that there are flat fractures over this gully. In the same way, fractures curve over wide headings and advanced gullies, which therefore require special support design, including roofbolting to fulfil a beam building function.

The vertical extent of fracturing above and below stopes is related to the prevailing ERR. Around stopes having ERRs of about 30 MJ/m², fairly intense fracturing extends 5 - 8 m from the stope with the intensity then falling off, but stope-related fractures can still be identified 30 - 40 m from the stope where they dip at approximately 50°.

Deformation of the Fracture Zone. Associated with the fracturing and compression of the face rock by high vertical forces and relaxation of horizontal confining forces as the stope advances, are significant amounts of horizontal dilation. Measurements taken into the stope face indicate horizontal displacement towards the stope of up to 60 mm per metre of face advance. In the hangingwall and footwall these movements are resisted and are obviously less, but an important consequence is that horizontal forces are generated which act to clamp the predominant steep fracture-bound slabs, rendering the highly fractured rock mass remarkably self-supporting and more stable. In essence, these compressive clamping forces tend to more than cancel out the effects of the tensile stresses typical of shallow (purely elastic) mining conditions. In this book, therefore, the criterion used to distinguish between 'shallow' and 'medium/deep' mining conditions is where sufficient fracturing occurs to generate significant *compressive* horizontal stresses in the hangingwall.

A further important consequence of horizontal dilation is that, because differential displacement tends to occur in adjacent strata, *horizontal shearing* occurs on partings separating the beds. This shearing destroys any cohesion the partings may have had (resulting in unattached blocks overlying the stope), with accompanying significant vertical dilation. This process may occur up to several metres into the hangingwall, and provides a partial explanation of the large inelastic closure that occurs in deep stopes. As the shearing takes place, the lower stratum is forced downwards as it overrides asperities or undulations on the upper surface. This opening or bed separation is largely irrecoverable. The mechanisms causing dilation and the consequences thereof are illustrated in Figure 4.2.8.

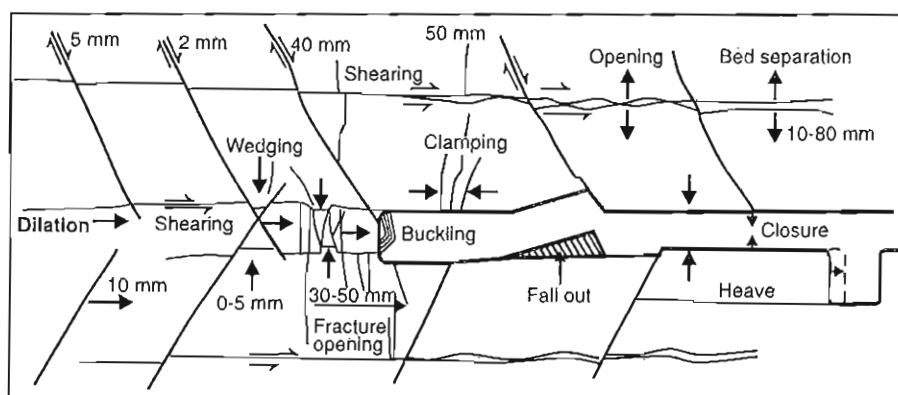


Figure 4.2.8 Deformation of the rock mass around deep stopes.

4.2.1.3 Potential for seismicity

Since the vertical virgin stress increases linearly with depth below surface, the stress changes brought about by mining at increasing depths result in proportionately more significant consequences and hazards. Principal amongst these hazards is the potential for mining-induced seismicity and rockbursts.

The main criteria to be considered in assessing the potential for seismicity in a particular area include the levels of ERR (which give a rough general indication of this potential), the magnitude of the ESSs that will be generated by the mining, the volume of rock that may be subjected to significant levels of ESS, and the presence and orientation of geological structures. Where the planes of maximum ESS generated by the mining coincide reasonably closely with planes of geological structures, the potential for seismicity is strongly enhanced. Given these facts, some first estimate of the potential for seismicity can be made, bearing in mind the great complexity of the problem. In areas where the horizontal:vertical stress ratio k is > 1 , the ESS needs to be examined on shallow dipping structures as well.

In the past, rough relationships between seismic activity or rockbursting and levels of ERR have been derived on certain mines. Although the cause and effect relationship is less direct than with ESS, the more easily calculable ERR can be useful; but the relationships need to be derived on individual mines and for specific reefs by back analysis for different geological conditions. Examples of these and other means of evaluating the seismic hazard are presented in Chapters 3.2, 8 and 9.

From a support design point of view, a fundamental issue is: under what conditions, or related to which criteria, should support be changed from a system to prevent rock-falls only, to one which conforms with criteria necessary to withstand seismic loading? In practice most Witwatersrand mines, mining at depths in excess of about 1700 m, are monitored by seismic networks. Judgement needs to be made on the results of analysis of the seismic data, as well as direct analysis of rockburst incidences, as to what areas on such mines are becoming prone to rockbursting. Table 4.2.1 indicates typical relationships between magnitude of seismic events and proportion of rockbursts, the distribution of rockbursts with respect to event magnitudes, and the cumulative damage in days lost per magnitude range. The data set involved 1844 events mainly from a mine using longwalls, in which the use of rockburst-resistant support was clearly necessary. Such data is obviously site-specific, and mines need to compile similar information for their own situations.

Table 4.2.1 Relationship between seismic events (magnitude M) and rockbursts on a specific mine.

M	Rockbursts/ 100 events	%Distribution of Rockbursts	% Days Lost
0-0.9	1	2	2
1-1.9	11	40	25
2-2.9	32	46	52
3-3.9	75	12	19
>4	100	<1	2

4.2.2 Minor Geological Features

For a fall of ground to occur the block or blocks that fall out, of whatever shape, must have intersecting **release surfaces on all sides** to provide the instability. In most instances the release surfaces are present prior to the fallout, and it is usually only in the case of some large collapses or as a result of dynamic loading that new fractures form to trigger the collapse. The bounding surfaces comprise stress fractures, geological structures, or, in deep mines, a combination of both. In this section geological structures commonly associated with FOGs are discussed. These are of particular importance in shallow mines which, by definition, do not have stress fractures and are subjected to tensile stresses in the hangingwall. However, the significance of these structures in deep mines should not be underestimated, because stress fractures are usually sub-parallel to the stope face and, for an unstable block to form, other discontinuities must be present running in the direction of mining. It is thus important in both shallow and deep mines to have a good knowledge of the pervasive, minor geological structures which contribute to hangingwall instability.

Inherent discontinuities in the rock mass are categorised into stratigraphic and structural features. Alone or in combination they can form potentially unstable release surfaces which can initiate falls of ground and therefore need to be carefully considered in the design of support. The properties which increase the instability of the discontinuities are low cohesion, low friction and limited roughness.

Discontinuities not only provide the unstable geometries which are directly associated with potential rockfalls but can indirectly contribute by forming weak planes along which shearing can occur, causing inelastic deformation and loosening of the peripheral rock mass. They also influence the pattern of stress-induced fracturing, on occasions adversely.

From a general strata control point of view, it is often found that mining in a certain direction is significantly more difficult than mining in the opposite direction, or that mining at right angles to the originally preferred direction is easier and safer. This is usually caused by the presence of a set of joints adversely oriented with respect to the stope faces, or cross-bedding in the immediate hangingwall strata (mining in the direction of dip of the crossbeds or troughs being more hazardous).

4.2.2.1 Bedding partings

Bedding planes are the contact surfaces between adjacent strata in a sequence of sedimentary rocks. These surfaces can vary from gradational changes from one rock type to another, through abrupt changes in particle grain size across the boundary, to a situation where there was a change in sedimentation and a layer of argillaceous material was deposited and preserved between stronger beds. It is mainly the latter situation which results in **bedding partings**. The argillaceous layer typically comprises mica and clay platelets which settled out to form an anisotropic fabric of micro layers, parallel to the overall bedding, producing a layer of fissile rock. Because of the mineralogical composition and fabric, the cohesion or tensile strength normal to the layering is low, as is the frictional resistance to shearing parallel to bedding.

The presence of such partings can thus lead to shearing, loss of cohesion and bed separation. Their thickness can range from a mere veneer to layers tens of millimetres thick. In areal extent they vary from tens of square metres to tens of thousands of square metres. Vertical separation of the partings can be measured in centimetres to a metre or two. In a fairly argillaceous sequence of quartzites, the vertical spacing is commonly 0.3 - 0.7 m.

Cross-bedding is subsidiary bedding within a stratum. The dip of this bedding is 20° - 40° steeper than the regional dip and often has a preferred orientation in a particular area. Planar cross-bedding is formed by the advance of a sandbank with deposition of a new material on the face slope of the bank. Trough cross-bedding is formed by the scouring and infilling of temporary channels in a depositional environment. Argillaceous material is more frequently deposited to form partings on the base of the trough cross-beds. They are particularly hazardous as they form cusped wedges in the hangingwall and their exact location and height into the hangingwall cannot be predicted.

In the layered igneous Bushveld Complex, interlayer partings are much less common than in the sedimentary rocks of the Witwatersrand sequence. Above the Merensky Reef the contact between rock types is usually gradational with no specific parting (with the exception of certain areas of the so-called 'Bastard Merensky' horizon). However, in the hangingwall of the UG2 Reef and above some of the mined chromitite seams, very persistent thin chromitite layers can occur which have very sharp basal contacts. These constitute 'good' partings and the strata below them need to be adequately supported. Sometimes too, the cooling magma has resulted in a strength anisotropy, the rock being somewhat weaker in tension normal to stratification, or when loaded in compression parallel to stratification. Thus, in areas of stress con-

centration such as around pillars, fracturing in the plane of stratification can occur producing 'bedding partings'. Flat jointing can play a similar role.

An important aspect of bedding partings is that, when non-intersecting steep discontinuities cut the same parting, a potentially unstable block will result. It is thus important that, in the description of ground control districts, the disposition of bedding partings be clearly stated and taken into account in the setting of support design criteria.

4.2.2.2 Joints

Joints are naturally occurring planar geological discontinuities across which there is no displacement. They commonly have some infilling material, which defines their cohesion and friction properties. Joints usually occur in sets, a set being defined as a group of joints with common orientation. Two, three or four joint sets usually occur in a rock mass with one of the sets predominant in intensity in a specific area. Sets of joints developed in response to major tectonic events and thus their occurrence can be expected over large areas.

In the goldfields, joint sets normally occur parallel to the major geological structures, the frequency of jointing often increasing as a fault or dyke is approached. Complex jointing also occurs in igneous rocks as a result of contraction due to cooling of the magma. Thus, in the Bushveld Complex, three sets of joints occur which are associated with tectonism, and further complex jointing is associated with 'potholes' in the reef. The occurrence of peculiar domical arrangements of joints, forming structures ranging in size from a few metres to tens of metres, results in unpredictable, poor ground conditions. Also of importance to hangingwall stability and support requirements is the presence of flat joints in the hangingwall. As these do not daylight into the excavations, exploration diamond drill holes may be extended into the hangingwall to ascertain if they are present and at what height in the hangingwall. In areas where flat joints are a problem, short geotechnical exploration holes drilled from raises are also able to provide valuable information.

It is important to determine the orientation and average joint spacing of the dominant joint sets in different areas. As mentioned, joints striking sub-parallel to mining faces can influence the pattern of stress fracturing leading in some cases to poor, difficult to support, ground conditions. In these situations consideration needs to be given to changing the direction of mining to eliminate the hazard. This has been necessary, for example, in some VCR longwalls where westward-mining breast faces experiencing this problem were converted to up-dip mining - Figure 4.2.9. In other situations, overhand longwalls were converted to breast or underhand layouts.

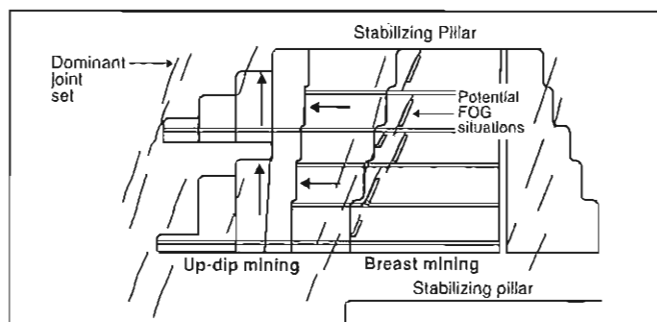


Figure 4.2.9
Change in mining direction to reduce fall of ground hazard due to influence of jointing on fracturing.

In shallow mines using in-stope pillars, predominant and persistent joints running parallel to the rows of chain pillars can lead to frequent large collapses between the joints once a critical strike span has been exceeded. This support problem and safety hazard is also eliminated by changing the mining direction such that the joint-bound beams are supported by the pillars - Figure 4.2.10. However this solution to the collapse problem may introduce a fall of ground hazard in the face area which needs to be addressed by keeping panel support of adequate design as close to the face as possible.

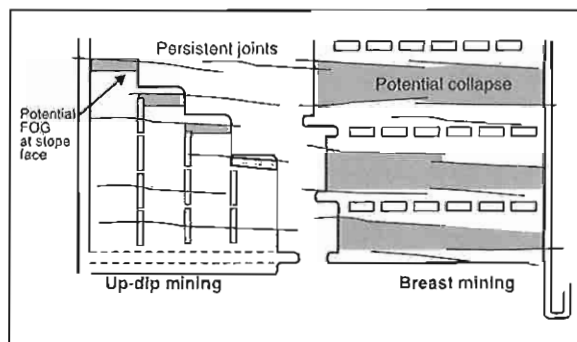


Figure 4.2.10
Effect of persistent joints
on stability of pillar-
supported stopes.

In low stress, shallow mining conditions the most common hazard resulting from jointing is the formation of unstable blocks. Intersecting joints of unfavourable geometry (under the influence of low or even tensile confining stresses as is typical in shallow mining) give rise to keyblocks which may support adjacent blocks susceptible to falling, or to isolated blocks in a state of unstable equilibrium. If these blocks do not fall out spontaneously as they are exposed by the advance of the stope face, their presence in the hangingwall is extremely dangerous. They are sensitive to minor deformation of the rock mass and shock loading due to mining activities, and unless adequately supported can fall out without warning. What aggravates the situation further is demonstrated by the result of a preliminary assessment of fatal fall of ground accidents that occurred on some platinum mines. The data indicate that almost 50% of the incidents were associated with either combinations of steep and flat joints or partings in the hangingwall, jointed dome-like structures, or combinations of steep and shallow dipping joints. All of these structures are difficult to recognise, which, combined with the statistic that 60% of all the fatalities in stopes occurred less than 4 m from the working face, strongly suggests that the first row of support should be installed as close to the face as possible.

This requirement is therefore the same as that for deep gold mines, where the hangingwall is much more damaged by stress fracturing and appears less stable. This is certainly **not** the case and, for the prevention of rockfalls, the **support requirements with regard to installation distance to the face in shallow mines are at least equivalent to that of considerably deeper mines.**

For support spacing design, joint data together with provisional support unit spacing can be used as input data in the Jblock computer program to assess the adequacy of the design (Chapter 11.4.4).

The *classification* of rock masses based on the occurrence and quality of jointing, for the purpose of delineating areas of different stabilities and hence support require-

ments, would seem to be a useful endeavour. Attempts to apply established techniques (such as the Q and RMR systems used in civil engineering tunnelling and other shallow underground excavations, Chapter 10.3.5) to hard rock stoping situations have met with little success. Even the Laubcher scheme, devised mainly for mines exploiting massive ore bodies, has not been successfully applied in tabular stopes. The main deficiencies of these systems seems to be that they do not account adequately for the orientation of joints with respect to the stope faces nor the mining-induced stress changes experienced by the rock mass. For example, joints or stress fractures sub-parallel to the hangingwall in stopes are important structures controlling stability and their influence is significantly under-estimated by the conventional characterization systems.

Some mines have developed their own classification systems with the main objective of delineating ground control districts. These appear to have been applied fairly successfully, but the result is that there are now several systems in use aiming to achieve the same objectives. It would appear therefore that there is a strong need for the development of a unified practical rock mass characterisation system for stopes, which takes into account support requirements, that could be applied throughout the gold, platinum and chromitite mines.

Apart from their influence on stope hangingwall stability, the effect of joints on the strength or stability of other rock engineering structures needs to be considered. For example, the strength of the small in-panel pillars used in shallow mines can be significantly degraded in the presence of unfavourably oriented joints. The integrity of the sidewalls of gullies can be compromised by a joint set slightly oblique to the direction of the gully. For this reason and the fact that joints striking sub-parallel to the axis of a gully increase the probability of unstable blocks being formed, **gullies should be aligned at as high angles as is practicable to the dominant joint direction.** Where this is not possible, additional roof bolts should be installed in the hangingwall and the bolting of sidewalls considered.

Furthermore, jointing can influence the location and severity of face-related seismic events. For example, stopes advancing breast-on through a joint set can experience enhanced seismicity (c.f. Chapter 1, Figure 1.9.3). In addition, the presence of joint swarms can be associated with more events than in adjacent less jointed ground.

4.2.2.3 Presence of incompetent strata

Incompetent strata in the near hangingwall of stopes pose a major strata control and support design problem. In the gold mines, these strata are typically well-bedded argillaceous or fissile, fine-grained rock types. Good examples of these are the Khaki Shale overlying the Basal Quartzite of the Basal Reef, and tuffs of the Westonia Formation which directly overly parts of the Ventersdorp Contact Reef. The Green Bar, which occurs about 2 m above the Carbon Leader Reef, is a similar rock type but causes serious problems only when it is brought closer to the reef by minor faulting or channelling, or is exposed by rockbursts. In the Bushveld, hangingwall jointing and serpentinization can be severe in places; so severe that conditions can come to resemble those in a 'running dyke'.

Characteristic of these rock types is that, once a small fall occurs, it is likely to expand rapidly and without warning if adequate remedial action is not timeously

taken. It is thus important, to the prevention of the development of hazardous conditions and costly rehabilitation or re-establishment of stope faces, that support of the highest quality be designed for such areas and that the installation standards for the support be strictly adhered to. In addition, the design of production blasts should be such as to significantly limit damage of the hangingwall, and conservative face advance rate targets should be set. In severe circumstances other strata control measures may have to be tried. Where the incompetent strata is thin enough it could be mined together with the reef (open stoping) or, at greater thicknesses, resue stoping could be considered. Spray-on fabric coatings are a further option. Nevertheless extreme situations exist where the quality of the rock is so poor and its thickness so great that, with current technology, *no* viable mining method may be possible.

4.2.3 Rock Mass Deformation

The deformation of the rock mass surrounding stopes is generally a combination of both elastic and inelastic mechanisms. The deformation level controls the performance of support units in terms of support force generated, it defines the yieldability and stiffness requirements of support systems and influences both negatively and positively the stability of the hangingwall and hence the design criteria for support systems.

4.2.3.1 Elastic and inelastic phenomena

Underground excavations deform in response to both elastic relaxation of the rock mass and inelastic shearing, sliding and opening mechanisms driven by abutment stresses, dilation and gravitational forces in the nearby surrounding strata. Elastic deformation is essentially instantaneous, while inelastic displacements have a strong time-dependent component but are initiated by changes brought about in the rock mass by advances of the stope face. The term **convergence** is used to describe the elastic deformation which can be calculated by numerical codes or by analytical means. The term **closure** describes the combined elastic and inelastic components of deformation experienced by an excavation, and has been measured to be up to five times the theoretical convergence. Closure in deep stopes cannot at present be modelled reliably because of the many and variable factors that can play a role in inelastic behaviour. In shallow stopes adequately protected by in-panel pillars on the other hand, closures usually have a negligible inelastic component and can be accurately predicted by standard elastic numerical modelling codes.

It is important for support design that, where inelastic mechanisms are encountered, the **closures should be measured** for each ground control district and for the different rates of mining pertaining on the mine.

The elastic deformation (convergence) is hardly affected by the presence of conventional support, and depends only on the in situ modulus of the rock, and on the mining depth, geometry and span. The ERR measure (Chapter 3.2.1) compactly sums up the effect of these parameters. For an ERR = 30 MJ/m² environment, the elastic convergence rate 10 m back from the face is about 10 mm/m of advance. In deep mines actual closure rates of 20 mm/m of face advance are common and can be as high as 40 mm/m. At these rates, visible total closure of narrow stopes will occur a few tens of metres behind the face. The closure rate is reduced by backfilling - once the vertical stresses in the backfill have reached 1 to 2 MPa, the closure rate is reduced to almost the elastic convergence rate.

The mechanisms that contribute to inelastic deformation were illustrated in Figure 4.2.8 and discussed in Section 4.2.1.2. They include displacement on shear fractures, shearing on uneven bedding partings, buckling of fractured beams, and thrusting along shallow incline parting surfaces. In shallow mines at spans that are approaching criticality in terms of large collapses, gravitational loading of beams or plates can cause rapid **bed separation**.

Recent continuous underground measurements of closure in stopes have shown a significant difference in **time-dependent behaviour** between different rock types. Where the surrounding rock mass comprises strong unbedded strata with a high Young's modulus, most of the closure that takes place between production blasts occurs within minutes of the blast and the rate of continuing time-dependent closure is low. Where the rock mass is weaker, more argillaceous and has many bedding partings, less than 50% of the closure is associated directly with the blast but the time-dependent rate of closure is high. [It has been conjectured that this difference in behaviour could be used to assess the face strainburst hazard, and when to implement preconditioning rockburst control measures.]

The differences in mode of closure also need to be taken into account when considering support. In cases where the time-dependent component is large, it is found that this behaviour can continue for 10 days or more after mining has ceased. If a panel is stopped for a period while mining in such a rock mass, the continuing closure will use up the yieldability of the support and, when mining resumes, its effectiveness may be compromised.

Ride is the differential lateral movement between hangingwall and footwall and is affected by factors such as reef dip, stope geometry, varying hanging and footwall rock quality, and movement of rock into hangingwall or footwall slots or gullies. Ride generally has a deleterious effect on support performance.

In addition to the quasi-elastic deformations discussed so far, stoping excavations can be subjected to **dynamic (rockburst)** deformations due to seismic events. The ground velocity which support units have to withstand varies from mm/s where the causative seismic event is distant, to probably more than 3 m/s on rare occasions where the excavation is in the source region (near field) of a particularly severe seismic event (Chapter 1.4.4). Apart from separation distance, factors which could influence dynamic ground velocities include magnitude of the seismic event, ambient stresses in the source region, geometry of the slip plane, whether other events have occurred in a similar position on the slip structure, and the position of the excavation with respect to the direction of slip. Few of these factors are currently well understood and research is ongoing. In addition, the rock conditions around a stope can significantly influence the ground velocity and extent of damage (so-called 'site effect'). With the current state of knowledge, it is required that support units for rockburst-prone areas be designed to withstand closure rates of 3 m/s. The amount of dynamic closure is also known to be highly variable, not only between different seismic events, but as a result of an individual event. This variability is probably influenced by the same factors which influence the quasi-static rate of closure, but the conditions around the stope and the type of support installed could be of greater influence.

The dynamic closure rate and the amount of tolerable closure determine the energy content associated with a volume of rock which has to be supported. An evaluation of these parameters determines the design criterion for support in seismically active areas - sections 4.3.4 and 4.4.4.

4.2.3.2 Rockfall and rockburst mechanisms

To be able to design effective support systems, it is necessary to understand the mechanisms that drive falls of ground, collapses, backbreaks and rockburst damage.

The most common and simple fall of ground results from gravitational forces, acting on a block of unfavourable geometry, which exceed the cohesion, roughness and friction on the release surfaces. The consequence is either the fall of an isolated block, the collapse of a brow, or the release of a key block leading to subsequent falls of adjacent blocks. In these cases, direct support of the loose block is required. Some small clamping forces are generated in the hangingwall by support units, but their magnitude and orientation are dependent on the orientation of discontinuities in the rock mass.

In well-bedded or stratified strata, beam or plate failure can occur and several mechanisms can be involved. The susceptibility to beam failure is a function of the length:width aspect ratio. Modes of beam failure such as snap-through, shear at abutments and cantilever failure are all represented in the underground situation. The beams and plates are usually discontinuous, being cut by joints or faults and stress fractures in deep mines. These discontinuities materially downgrade the strength and deformation characteristics of the beams and corresponding stable spans.

The differences in mine layout between shallow and deep mines dictate a difference in scale of the potentially unstable beams that occur in the two situations. In shallow pillar-supported stopes, plates of rock span across panels, typically 30 m between pillars. The thickness of the plates is usually determined by pre-existing partings provided by bedding planes in sedimentary sequences, or chrome seams and flat faults in the Bushveld mines. On occasion, under particular stress or stratigraphic conditions, it appears that flat fractures emanating from the pillars can propagate parallel to the hangingwall to form the plate. It is fortunate that in most instances, the failure of these large plates (up to 150 m by 30 m) is accompanied by ample warning in the form of progressive failure of the support and loud cracking sounds. Initial signs of failure can be noticed days or occasionally weeks before the final collapse, which can take hours to occur.

A special case of beam failure occurs in Bushveld mines, where punching of pillars into the footwall results in **footwall heave**, which in turn consumes the limited yield capability of support units selected for shallow, normally low closure-rate conditions. Thus support, designed to adequately resist the weight of a well-defined beam with a factor of safety, can be caused to fail. Obviously, the forces involved in the footwall heave are greater than the resistance of the support system. A similar phenomenon caused by buckling of the hangingwall beam is possible, but has not yet been recorded. It is necessary to monitor closure in areas where footwall heave occurs, in order to select support units with sufficient yieldability.

Two other issues need to be considered when attempting to elucidate the cause of a beam failure. Firstly excessive bending of a fractured beam may preferentially compress, beyond their peak strain level, support units located at the position of maximum beam deflection. Adjacent support units would thus have to carry the additional load. As the beam deflects, further progressive failure of support units can be envisaged, until the support resistance of the entire remaining support system is exceeded and the beam collapses.

Secondly, the possibility of columnar support units (various forms of props) **failing by ride action and toppling** is often mooted. In about 17% of over 150 fatal accident inquiries in platinum mines, toppling of support units was mentioned as the cause or contributed to the cause of the fall of ground or collapse. The mechanism whereby large blocks or cantilevered beams slide out on inclined joints causing lateral movement to the top of props and subsequent toppling is readily envisaged, particularly in shallow stopes where closure rates are low and the potential support resistance has not been fully realised close to the face. Criteria to prevent this type of support failure have not yet been established. The use of support units that are less inherently susceptible to toppling failure in geotechnical conditions where sliding of blocks is likely, is obviously a part solution, particularly in the vicinity of shallow dipping joints. Pre-stressing normally passive support units would also help in this regard. The use of the block theory program Jblock (Chapter 11.4.4) could assist in determining the likelihood of sliding failure for different combinations of discontinuity sets.

Backbreaks are huge collapses in shallow mines spanning whole stopes. They can involve millions of tons of rock and extend to heights above the stope of 40 m or so. The mechanism, in strata with well-developed bedding partings, is probably a successive upward migration of beam failures within the tensile zone. In many instances the span at which the massive failure occurs is reduced in the presence of faults or dykes that are inclined over the stope abutments. In more massive strata, unfavourably oriented faults are probably a prerequisite. Spans at which backbreaks occur vary from about 120 m to 240 m, depending on local geological conditions. The only methods that prevent backbreaks are to limit spans by barrier pillars to less than the critical span if this is known; to use in-stope pillars (the most commonly used and successful method); or to use stiff backfill.

Rockburst mechanisms, and mechanisms responsible for underlying seismic events, were reviewed in Chapter 1.4 and specific examples can be found in Chapter 8. Seismic monitoring and seismic source quantification is dealt with in depth in Chapter 9.

4.2.4 Mining Environments and Ground Control Districts

For many years, support systems in South Africa were deployed on a purely empirical basis. In order to take changes in local geological conditions more explicitly into account, the concept of subdividing mines into 'geotechnical areas' was introduced in the early 1990s; and the first attempts at doing this involved defining areas of like combinations of hangingwall and footwall types. A regulatory requirement to define formal geotechnical areas has since been included in the Chief Inspector of Mines'

guidelines for the compilation of codes of practice (Chapter 2.6). In this document, a 'geotechnical area' is defined as a portion of a mine in which a given set of geological conditions, with associated rock-related hazards, permits a *common* set of rock engineering strategies to be employed to minimize the risks.

Unfortunately, interpretation of this definition has varied widely across the industry, the root cause being most probably uncertainty as to the basic purpose of declaring 'geotechnical areas'. Were these to be areas where different *major* mining and rock engineering strategies were to be applied, or merely where different strata control and *support* techniques were to be implemented? In many instances, unsuccessful attempts were made to combine these two concepts.

It has become apparent that the 'geotechnical area' subdivision of a mine needs, in general, to be carried out at *two* levels, namely '*mining environments*' for the regional subdivisions requiring major strategic decisions, and '*ground control districts (GCDs)*' for distinguishing areas requiring specific strata control procedures and support designs.

Thus, '*mining environments*' is the term used in this book to describe *major* subdivisions of a tabular orebody which dictate specific fundamental extraction strategies such as mining method, overall mine layout, and the need or otherwise for the use of regional support systems. Coverage of such major subdivisions was given in Chapters 2 and 3 - the factors involved including effective *depth* ('shallow', 'intermediate', 'deep', 'ultra-deep'), degree of faulting, normal or '*wide reef*', normal or '*multi-reef*', and normal or '*steep dip*' of reef. Mining environments occur on a district scale and thus cross mine boundaries. In most instances, delineation of the applicable mining environment is the first step in defining relevant GCDs.

'*Ground control districts (GCDs)*' are further subdivisions of orebodies on a mine, based on like geotechnical conditions that occur, into areas in which the *same standard strata control methods, mining direction and support system designs can be implemented*. Thus, if significant variations in geological structural features / discontinuities / stress fracture patterns occur, a single 'mining environment' should be divided into more than one GCD.

There are many stratigraphic, structural and stress-induced features, together with the potential for seismicity, which in combination could potentially define a particular GCD. In Table 4.2.2 these factors are listed and categorised into inherent geological, stress-induced and other mining-related groups. It is not necessary that all of these features be evaluated in the delineation of GCDs. What the rock engineer must decide is, which *two, three or four features are the most important* in contributing to the relative stability/instability of the hangingwall on the particular mine, and thus would in combination define areas *requiring different support and/or strata control strategies*. In addition the presence of weak or mobile strata in the footwall needs to be taken into account if they allow punching of support units, yielding of in-stope pillars, or cause footwall heave such that abnormally high rates of closure occur. The factors leading to such conditions include inherently weak strata such as shale or metasomatically altered rock, relatively thick argillaceous partings at critical distances in the footwall, or intensely jointed or fractured rock.

Table 4.2.2 Factors which govern support system choice, and hence delineate specific ground control districts (GCDs) within a given mining environment

Support system specification	Governing geotechnical factors
Rockbursts likely/ not likely to occur	Seismic history. Proximity to hazardous geological structures (faults, dykes, joints).
Required Energy Absorption Capacity/ Support Resistance	Likely fallout thickness - highest <i>unstable</i> plane in h/w (dynamic/ static). [Database of fallout thicknesses. Nature of stress fracturing. Geotechnical study of parting-plane and joint properties].
Support stiffness/ yieldability	Expected slope closure rate. [Measured, estimated].
Support spacing. Need for load-spreaders (areal coverage).	H/w condition and blockiness. [Strength, stress fracturing, jointing, presence of weak strata, level of horizontal stresses]. Mechanism of failure. F/w condition. [Strength].

An example of GCD distinctions based on hangingwall conditions is provided by the unstable Khaki Shale which overlies much of the Basal Reef in the Freestate gold-field. Isopach maps of the thickness of the Basal Quartzite occurring between the reef and the shale could be used as a basis for delineating GCDs. Where this interval is less than a particular value based on local experience, an area could be simply and immediately defined wherein open or rescue stoping would be indicated with a concomitant change in stoping width and support requirements. In addition this area could be further subdivided on the basis of the thickness of the Khaki Shale. Where this is greater than a certain value, mining becomes uneconomic with current techniques and therefore such areas would be classified as a separate ground control district. In the other major Basal Reef geotechnical categories, where the shale can be undercut, or does not occur, further subdivision into more specific ground control districts can be made. The basis for this could be the intensity of faulting, the dip of the reef, the likelihood of seismicity, particularly in the presence of geological structures with a known history of associated seismicity, whether the area has been or is likely to be overstoped, as obvious examples of what needs to be considered. However local knowledge and experience will determine which factors are responsible for poor ground conditions and thus are significant to ground control district definition. This requires an understanding of the fundamental rock mass behaviours on a mine and of the failure mechanisms leading to falls of ground and rockbursts. It is then necessary to determine criteria or significant values for these parameters which govern different types or degree of rock mass response to mining and which lead to less or more hangingwall failures or severity of failure.

The important factors which need to be considered when setting these criteria are how they will influence the three factors fundamental to support design - Table 4.2.2. These are: the potential fallout thickness (support resistance or energy absorption requirement), the closure rate (stiffness and yieldability requirement of the support units), and the block size of the unstable hangingwall (support spacing and areal coverage requirement). Seismic activity will have been considered previously, in the delineation of mining environments. Having decided the criteria, the positions where

these limits occur on the mine are best depicted on mine plans as a set of *contours*. The overlays of the contours for the various parameters then need to be examined and areas of like conditions outlined. Using this method it is likely that, in addition to defining large significant GCDs, several areas with the same conditions but of small extent will be indicated. These would be impractical GCDs, and judgement needs to be made as to which other GCD these small areas should be attached. The results of the exercise then need to be studied to see whether they correlate with experience of ground behaviour and support effectiveness in the different areas. If not, the process should be repeated using different values for the criteria.

On the other hand, large fairly uniform GCDs may have narrow zones of a different geotechnical condition cutting through them. These may be, for example, sedimentary channels in the reef or a zone of disturbed ground parallel to a dyke or fault. If, at the time of preparing the GCD plan, the exact location of such features is not known but their occurrence is expected, then a GCD description of the features should be made and a mining and support strategy defined. Such specifications could be documented either as a separate formal GCD, or simply as a codicil to the code of practice for an existing, more general, GCD.

A similar method to the above is to use a matrix along one axis of which distinct geographical areas on a mine are listed, and along the other axis the previously identified criteria or parameters are listed. Each area, which could be a large fault block or separate longwalls, is then rated according to the criteria. The rating system could be simple: such as high or low; high, medium or low; present or not present; depending on the number of parameters used.

In this way different geographical areas with like geotechnical conditions will be identified and grouped into a single GCD. Conversely, separate GCDs will be identified. A potential deficiency in this method is that, if the geographical areas selected are too large or ill-considered, smaller yet significant GCDs within the geographical area may be missed. Also if this particular method is used, regular re-evaluation of conditions in current mining areas must be undertaken to determine whether conditions have changed significantly or not.

Another approach to defining GCDs is to use rock mass characterisation techniques, particularly in shallow mines where geological discontinuities are the dominant factors controlling rock mass behaviour and instability, and the influence of stress-related factors is limited. Attempts at using conventional classification systems, designed primarily for use in shallow tunnels, have been unsuccessful mainly because factors such as orientation of discontinuities with respect to mining direction, persistence of discontinuities and the influence of stress changes are not adequately addressed, and due to the large amount of data that has to be collected, processed and updated. However, characterisation methods can be devised for stope conditions which quantify those factors which are relevant to the local situation. The characterisation of jointing and other hangingwall disturbances could be the most important aspect of such a system, but would be adjusted by other factors deemed significant on a particular mine. Such systems can be made relatively simple and practical to implement, but need to be evaluated and perhaps modified with respect to current support design practice and experience of ground conditions. The benefit of such a system is that there would be an unbiased numerical output for each area that can be compared to other situations.

The rate of closure needs to be determined for each of the GCDs defined, bearing in mind that the rate of face advance could influence the results of the monitoring. It has been found on the VCR, for example, that the higher the face advance rate the lower the closure rate expressed in mm/m of advance. This effect should not be used in defining GCDs, but needs to be considered in the design of support.

It should be the aim of the definition of GCDs that less than about five geotechnical conditions be identified per reef. It is also possible that a particular geotechnical condition will not occur in one unified area but could occur in several isolated areas separated by ground having distinctly different geotechnical conditions. These separate areas would however all be grouped into a single GCD definition requiring the same mining and support strategy. The final product of a GCD exercise should be a mine plan delineating the extent of the various GCDs, extrapolated into unmined ground. The plans should be reviewed on an annual basis and updated when new information becomes available. The important defining parameters and associated potential hazards should be described in the Code of Practice for each GCD identified.

The concept of GCDs could be extrapolated to off-reef development. Here, the basis for definition could be rock type or stratigraphic formation, but would be modified to take into account expected stress changes due to stoping, using possibly the RCF criterion, proximity to seismically hazardous structures or changes in k-ratio. In shallow mines, conventional or simplified rock mass characterisation techniques could be used.

It is apparent that in the definition of GCDs, a great deal of basic geological information is required. It would seem prudent therefore that a *formal and good working relationship be established between the rock engineering and geology departments on a mine*. This collaboration should extend beyond GCDs and would enrich both departments, and benefit safety as a whole on the mine.

4.2.5 Characteristics Of Support Units

The functions of stope support systems are to provide, with a factor of safety, sufficient support resistance or energy absorbing capacity and areal coverage or support unit interaction to maintain the integrity of a discontinuous hangingwall, which may be subjected to seismically generated ground velocities. To achieve these requirements for a range of geotechnical environments and mining conditions, various types of support units are necessary and are generally available.

The factors that are important in the design of a support unit for a particular set of conditions are **strength, stiffness, yieldability, energy absorption ability**, and whether the unit is capable of being pre-stressed or tensioned or not, i.e. whether it is an **active or passive** support. Important factors which also need to be considered, particularly in South Africa where the majority of stope support units are constructed wholly or in part of timber, are the effects of loading rate and creep on the above properties.

Other issues which at times need to be considered are: the contact stresses between the support unit and hangingwall or footwall where punching into weak or friable rock is a potential problem; buckling resistance (particularly of long timber or steel

props including any extension pieces which may be used); buckling and shearing resistance of slender packs; the resistance to toppling, particularly where potential falls of ground comprise large blocks; blast resistance and the capability of being installed within 2 m of a working face without being significantly damaged or dislodged by the blast; ease of installation which obviously influences productivity but can also affect quality of installation with rock engineering consequences; flammability and toxic fume emission if burnt; and resistance to corrosion or fungal attack. Figure 4.2.11 shows four force-deformation curves of typical support units and illustrates differences in the prime requirements of a support unit.

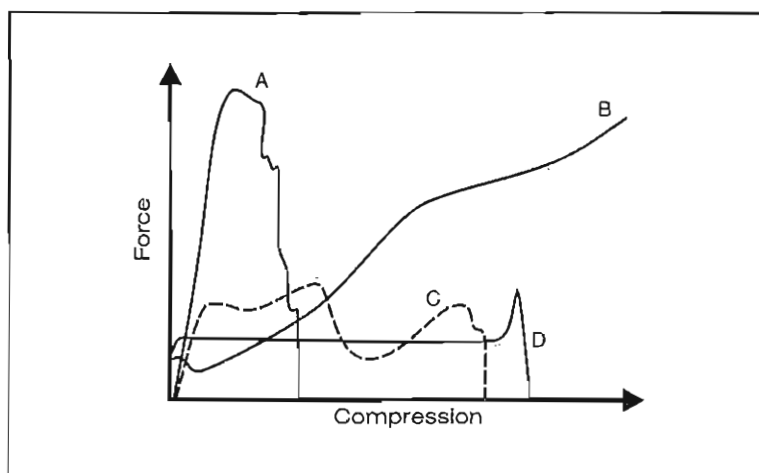


Figure 4.2.11 Force-deformation curves of typical support units.
A: grout pack, B: prestressed timber pack, C: elongate prop,
D: hydraulic prop.

Types A and C have high initial stiffnesses, i.e. the slope of the initial part of the curve is steep. The low initial stiffness of type B can be somewhat compensated for by its ability to be prestressed, but there is then a period of negative stiffness as creep effects reduce the initial load bearing capacity.

Types B and D are **prestressed** ('active') - i.e. they exert a positive support force on installation.

Type D can be considered to have infinite initial stiffness, as it is pre-loaded. Support with high stiffness or active support is essential for stopes with low rates of closure, and generally to enhance the blast resistance of units.

Peak strength is an obvious characteristic. Type A attains this after minimum closure and is tailored for stopes with low rates of closure, while Type B is more suitable for stopes with higher closure rates. A high strength unit such as B is useful where large deadweight requirements may occur such as the support of middlings between stopes, however this capability is often not necessary in normal stopes.

Stiffness is the initial slope of the force-deformation curve. High stiffness is important in low closure rate (shallow or de-stressed) stopes in order that the support units build up support resistance sufficiently rapidly. **Prestressing** is a way to generate effective

high stiffness in otherwise low stiffness units. However, field data indicate that for normal mat packs for example, creep effects tend to rapidly nullify the benefits; thus pre-stressing should be motivated mainly on grounds of preventing blasting-out of packs or other supports, or of streamlining the operations of support installation.

Yieldability is defined as the amount of deformation a unit can withstand while sustaining a significant load bearing capability. Type A has little yieldability thus limiting its application to areas where low amounts of closure are expected. B is capable of yielding by more than 50% of its installed height. This type of performance is required in stopes where the rates of closure are high or where long-term support is necessary in areas with moderate rates of closure. Types C and D typically have yield capacities of 250 - 400 mm. This combined with their high yield force is usually sufficient to provide enough energy-absorbing ability for the working area of a stope in a seismically active region. However, where closure rates are unusually high or where the working area is excessively large, the work available to be done by units in the back rows of the support system could be too low. This can readily be assessed by using the support design analysis program SDA (section 4.4).

Yield Force is the load bearing capacity of a unit at various amounts of compression. Usually, but not always (see Type B), these are the forces which develop after the unit has reached a peak load carrying capacity. The yield forces of Type A rapidly decrease with compression ('strain softening'). Such units should therefore be used in situations where they will not be subjected to loads exceeding their peak strength or to deformation that exceeds the strain at peak strength. Type B is 'strain hardening' thus developing ever-increasing yield forces with compression. These high forces will be generated in the back areas of stopes where they are often not needed. More seriously, when used as gully packs they can cause damage to gully shoulders weakened by unfavourable stress fracturing or jointing. This is particularly the case when subjected to dynamic loading. Type C typically has an irregular yield force profile. If the fluctuations in yield force are not excessive this is not considered a serious deficiency. Nevertheless, Type D has the ideal, constant yield force characteristic.

There are scores of support units on the market from which the rock engineer can select for a support system design. Most of these fall into the categories shown in Figure 4.2.11. Units made from timber with the grain parallel to the surfaces to be supported provide good yield but poor stiffness characteristics. Where the grain is parallel to the direction of compression, higher stiffness but less yield is achieved. Concrete and grout based units have high stiffness and strength but generally poor yieldability. Steel units, whether mechanical or hydraulic, have very reliable and regular performance characteristics and can be designed to provide a range of performance specifications.

In fact, with modern technology and expertise most manufacturers of any type of support can, with innovative design and combination of materials, supply units to cover a wide range of performance requirements. Examples of these are various proportions, geometries and constructions of end and parallel grain components in a timber pack unit; various combinations of timber and concrete in packs; various concrete strengths, internal reinforcement and geometry of blocks for concrete packs; geometries and confining mechanisms of yielding timber props (elongates); and mechanical and hydraulic design variations in steel props. In addition, various methods of pre-stressing both props and packs are available.

The following diagrams reflect the force-deformation behaviour of some typical support units, and illustrate some further important issues. Figures 4.2.12 and 4.2.13 show the effects of loading rate, height or length on the performance of a number of common support types. The following should be noted:

(i) There is a major difference between laboratory (or some manufacturer-leaflet) behaviour and actual *underground* performance. This is due to effects of the *slow* underground loading rate and irregular hangingwall or footwall contacts. [Conversely, under dynamic (seismic) loading, timber-based supports tend to show significantly higher support resistance than indicated in conventional laboratory testing].

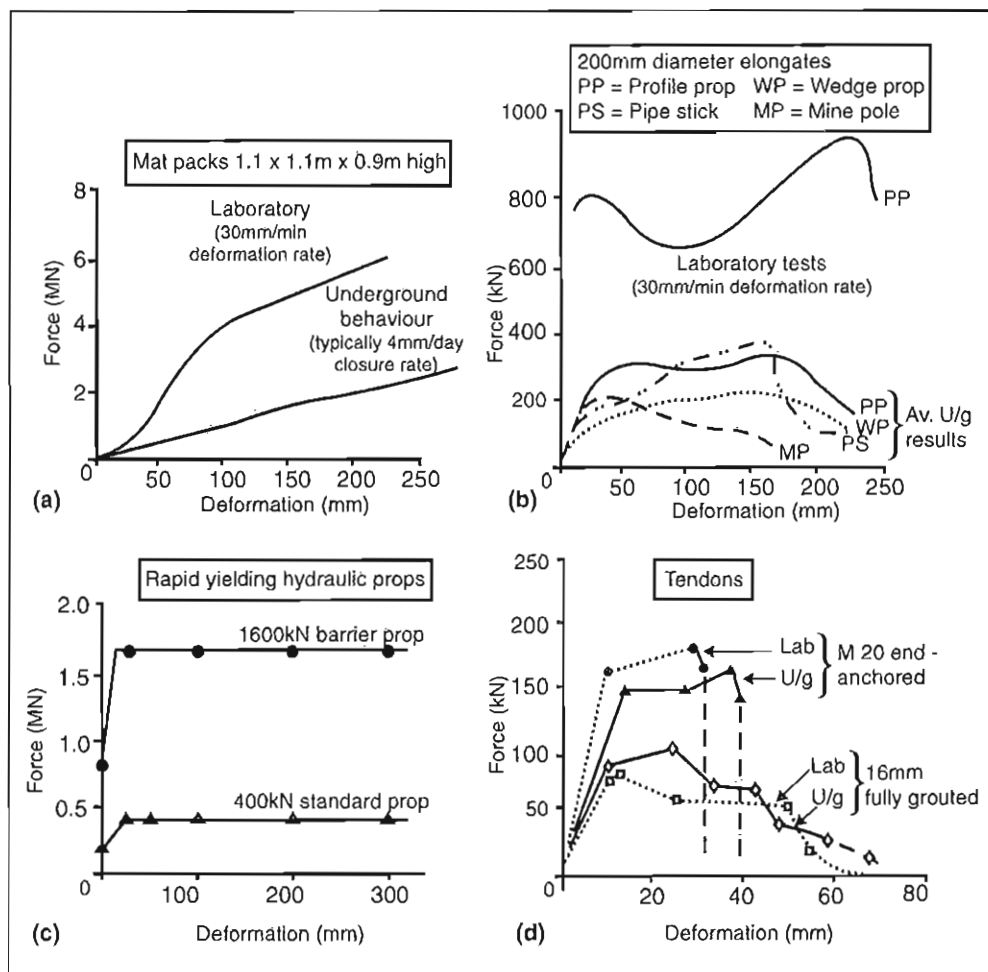


Figure 4.2.12 Effect of loading rate on force-deformation behaviour

(ii) Performance is also affected by unit length; thus for example, packs or elongates in a 2 m stope width environment have to undergo nearly twice the deformation (and be subjected to a greatly increased chance of premature buckling) than 1 m elements to attain the same load - Figure 4.2.13. Thus, packs should not normally be constructed with a height:width ratio exceeding about 2:1, nor should elongates normally have a height:width ratio greater than about 10:1.

These effects are less marked or are absent in the case of steel-based support elements such as hydraulic props or tendons.

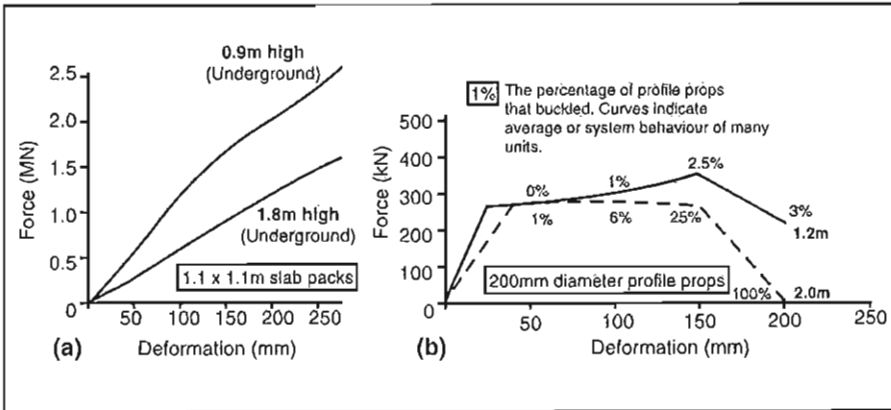


Figure 4.2.13 Effect of element length/height on force-deformation behaviour

4.3 SUPPORT SYSTEMS - GENERAL CHARACTERISTICS

4.3.1 General Definitions

Stope support systems comprise an integrated design of support units which fulfil the support requirements of the working area, back area, gullies (sections 3.4.7, 4.4.9) and sidings in a stope. In addition separate attention needs to be given to the support of centre gullies, the support during ledging (section 3.4.6) and to special areas such as wide headings or raises, gully intersections and winch bays as examples.

The **working area** encompasses that area of a stope where routine mining operations take place and extends from the face back to the sweeping line. This is a variable distance ranging from 8 m to 12 m or more, particularly in up-dip mining. The width should be minimised to ensure that support resistance and energy absorption criteria are met by the support in this total area, particularly when support units with limited yield capacity are used in stopes with high rates of closure. The Support Design Analysis (SDA) program described in section 4.4 is a simple method for evaluating whether proposed support systems meet the established criteria.

The working area has a subdivision termed the **face area** which extends from the face back a distance of about 4 m. From the safety point of view this is the crucial area on a mine. Most working man-hours on a mine are spent in this area and most fall of ground casualties occur in this region of the stopes. Support in this area can be either temporary, semi-permanent (hydraulic props) or permanent, but is usually a combination of these.

The **back area** is that between the sweeping line and the original on-reef access tunnel. The function of back area support, and particularly its influence on the stability of the working area, is not always clear. Different effects occur in different ground

control districts. For example in many situations, measurements indicate that back-fill increases the horizontal clamping stresses over the working area compared to conventional support, the inference being that stability is improved. However, where the hangingwall is well-bedded with many partings, the same forces appear to promote buckling of the relatively thin plates causing less stable conditions. In fact, in this environment, no back area support is necessary and the radically different 'caving' mining method has been successfully employed - Chapter 3.5.5.

Similarly, where elongates are used as permanent support in stopes with high closure rates there is, in effect, virtually no back area support more than 30 m behind the face except along the gullies. In this situation there is no apparent deleterious consequence on stability in the working area. Based on this experience, field trials were carried out in a deep longwall where three to four rows of rapid-yield hydraulic props (RYHPs) were used in conjunction with 1600 kN rapid yield barrier chocks in the back row, with *no* back area support. From a rock engineering stability perspective these (admittedly small-scale) trials were highly successful, to the extent that rock-burst damage occurred in adjacent conventionally supported panels while none occurred in the experimental panels, even in the back areas.

With some important exceptions it appears, therefore, that in many **deep mine** situations the requirements of back area support are limited and that the high forces generated by some pack systems in these situations are unnecessary. The main function of permanent support (which ends up as the back area support) is the support it provides when initially installed in the working area.

The main exceptions are where the deadweight of middlings in multireef situations have to be carried, and in potential poor ground conditions such as where weak strata (e.g. Khaki Shale) occur in the near hangingwall. In the latter situation a fall in the back area could lead to a collapse which overruns the working area. Backfill is also a major exception, in that there is evidence to show that it can provide superior support to the face area and to gullies particularly in rockburst situations, and that it plays a dual role by also providing regional support.

The above observations and experiences are suggestive of relatively minimal back area support requirements. However little thorough research has been undertaken in this area and, until a proper understanding of the functions of back area support is obtained, significant changes to currently successful designs should not be contemplated lightly.

In **shallow mines**, back area support of adequate design is essential. Below the usually stable arch between in-panel pillars is a thickness of jointed strata which is not in compression and is potentially unstable. The thickness of this 'tensile' zone needs to be supported, otherwise large panel collapses could occur. Similarly in bedded strata with well-developed partings, the potential deadweight of the beams or plates needs to be catered for.

Particular attention needs to be given to ensuring the stability of **gullies** (sections 3.4.7, 4.4.9), as these are long-life excavations in stopes which are subjected to relatively large amounts of closure, and in which personnel routinely congregate, travel and work. Apart from their support, the stability of gullies is particularly sensitive to

strata control practices such as the implementation of good blasting practice, and appropriate layout design - shape and size, particularly in regard to sidings and maximum distance of advance allowed ahead of the stope face. Dip sidings should generally be cut; failing which large falls in gullies may occur which will be difficult to rehabilitate and will lead to additional hazards which have to be traversed frequently. It is thus important that diligence be applied to the design and support of gullies and that these designs be strictly and lineously adhered to.

Factors in gullies which have to be considered are complex, including shallow-dipping fracturing; large amounts of closure; the forces and vibrations caused by snatch block anchors which tend to loosen or even pull out rock; and the frequent weakening or undermining of gully shoulders. Special attention needs to be given to the integration of siding support in overhand layouts with the requirements of cleaning systems and the ultimate gully support. Deficiencies in these aspects have led to the area of intersection of stope face and gully being the scene of many FOG accidents. Support design for gullies therefore requires special attention. Roofbolting is often required for beam reinforcement of flat fracture-bound slabs, and where faults or long joints strike sub-parallel through the gullies. In more blocky conditions, area support in the form of mesh may also be required. Further details of gully layouts are given in Chapter 3.4.7, and gully support in section 4.4.9.

4.3.2 Area Coverage of Support Systems and Support Spacing

The predominant cause of falls of ground in stopes is inadequate areal coverage or interaction between support units. The majority of FOGs occur between the face and the first row of support, indicating less than adequate interaction between the support provided by the face abutment and the support, or between the support units themselves. Support units seldom fail purely mechanically. The question of support interaction, support spacing and areal coverage is thus critical to reducing FOG casualties.

A basic measure of the areal coverage of a support system is the percentage that the contact area of the support units makes of the total area of hangingwall to be supported. For typical prop and elongate systems this figure is $< 1\%$, and for pack systems $< 10\%$. Except for purposes of comparison the value of such statistics is limited.

What is of importance, in **shallow mines** in particular, is the probability of the pattern and size of support units and accessories directly supporting individual loose blocks or keyblocks. The ability of various proposed designs in achieving acceptable probabilities can be tested by evaluation with the Jblock program (Chapter 11.4.4) for various geotechnical circumstances. However, two other factors need to be considered. The first question is how do the forces generated by adjacent support units interact to provide support to blocks not directly supported? Preliminary studies, using the finite element code ELFEN, indicate that force trajectories are strongly dependent on the orientation of dominant fracture or joint sets (Figure 4.3.3; see also Chapter 6.3 and 6.4 on tunnel support design). It is envisaged that design charts will become available from this work to indicate optimal support spacing, or the need for additional areal coverage to be provided by the support system. It is also obvious that the support interaction between passive support units is initially *minimal*, particularly in the important working area where almost all support is installed. The effect of