A Review of Sublevel Stoping

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ABSTRACT

A review of the factors controlling stope wall behaviour such as excavation geometry, rockmass strength, induced stress, ground support, blast damage and drill drive layout is undertaken. In addition, the advantages and disadvantages of single lift stope geometries linked to vertical crater retreat are also analysed. Finally, key components of multiple lift stoping such as cut-off slots, production rings, stope undercuts and drawpoints are studied in detail.

INTRODUCTION

Sublevel open stoping methods are used to extract large massive or tabular, steeply-dipping competent orebodies surrounded by competent host rocks which in general have few constraints regarding the shape, size and continuity of the mineralisation. The success of the method relies on the stability of large (mainly un-reinforced) stope walls and crowns as well as the stability of any fill masses exposed. In general, open stopes are relatively large excavations in which ring drilling is the main method of rock breakage. Ore dilution consisting of low-grade, waste rock or minefill materials may occur at the stope boundaries. In addition, ore loss due to insufficient breakage can also occur within at the stope boundaries.

The method offers several advantages including, low cost and efficient non-entry production operations, utilisation of highly mechanised, mobile drilling and loading production equipment, high production rates with a minimum level of personnel, furthermore, production operations are concentrated into few locations such as ring drilling, blasting and drawpoint mucking. The disadvantages include a requirement for a significant level of development infrastructure before production starts, thus incurring a high initial capital investment. However, most of the development occurs within the orebody. In addition, the stopes must be designed with regular boundaries and internal waste pockets cannot be separated within the broken ore. Similarly, delineated ore cannot be recovered beyond a designed stope boundary.

Technical developments in the understanding of the rock mass and fill behaviour in conjunction with dilution measuring techniques, improved blasting, equipment, ventilation and ground support practices currently allows for a successful application of this method in increasingly complex geological and mining situations. In particular, an increased understanding of the method is required to facilitate improved stope access configurations and optimised extraction sequences leading to full orebody recovery while achieving dilution control. The complexity of the method and the current depth of the orebodies being extracted worldwide, suggest that adequate planning and control of the operations are critical to the successful

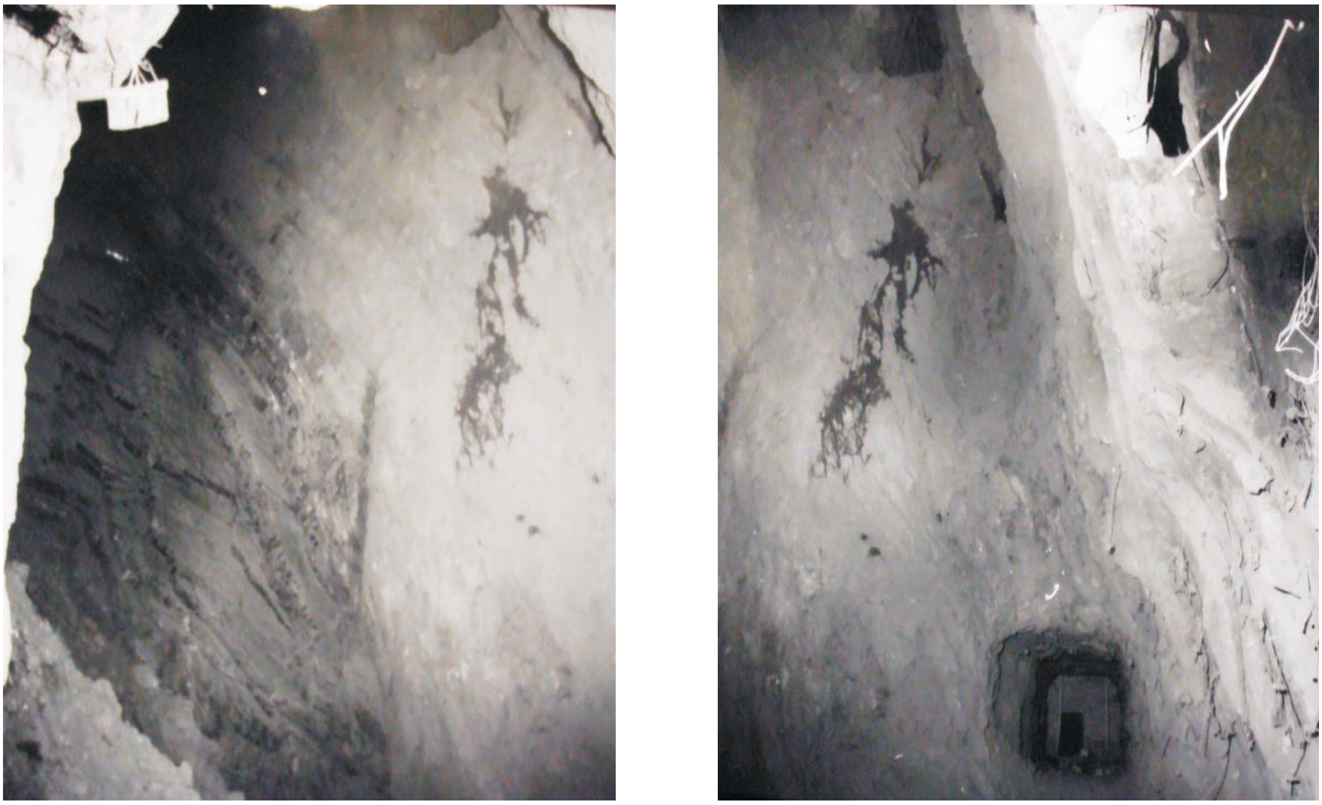

FIG 1 - A view inside an open stope.

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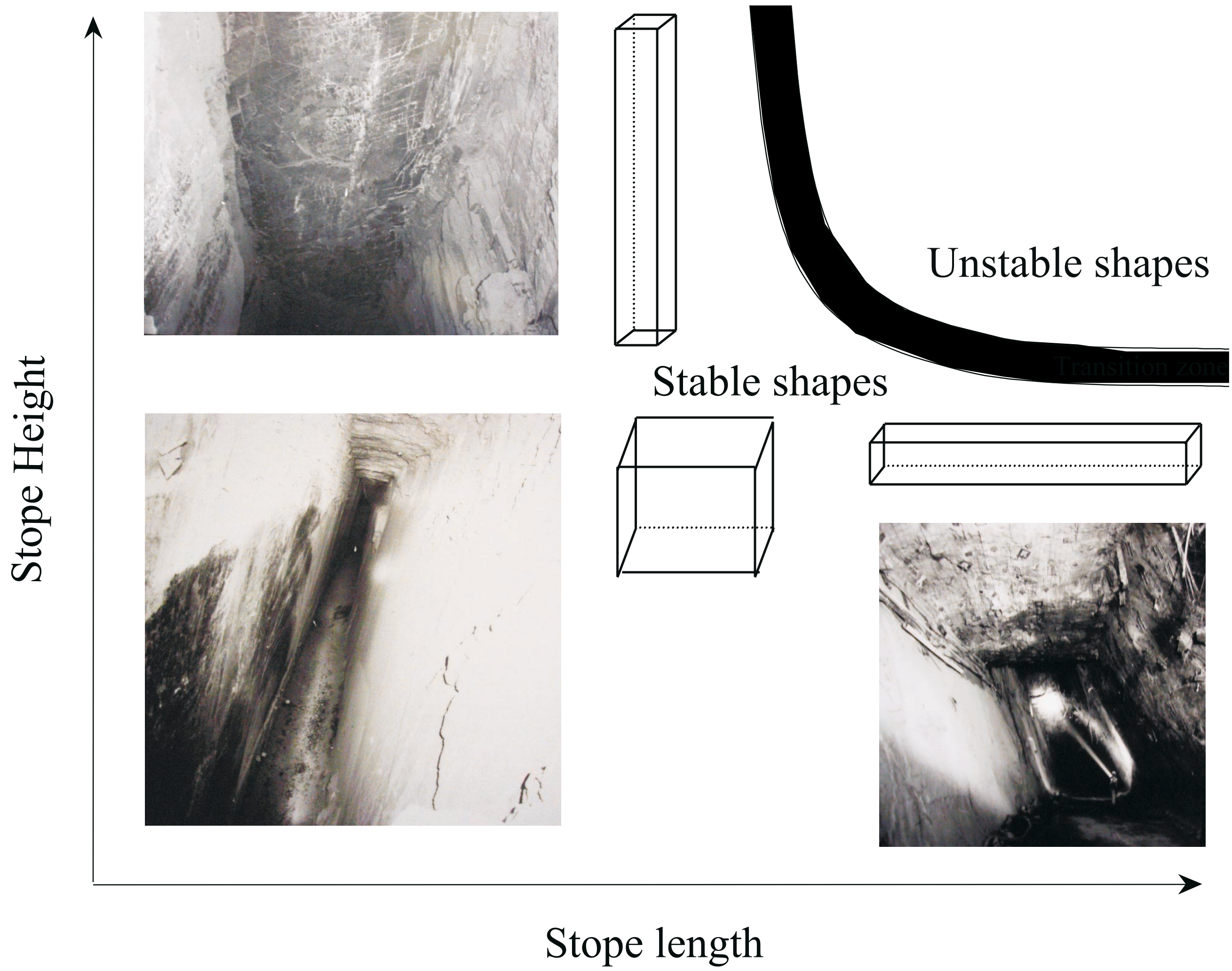

FIG 2 - Stable shapes for sublevel stoping.

implementation of optimum stope sizes and sequences of extraction. The method is commonly known throughout the world as open stoping, sublevel stoping, longhole or blasthole stoping. The essential common elements of sublevel stoping are (Bridges, 1982):

- the stopes are open at some stage, without substantial wall collapse or caving;
- the stopes extend from sublevel to sublevel, with operations carried out only at these sublevels;
- the blasted rock moves by gravity alone to the stope drawpoints;
- the method uses long blastholes for rock breakage;
- the blastholes are located within planes called rings;
- the holes are drilled mostly downwards;
- the initial expansion slot is located on the side or bottom of each stope; and
- the method is non-entry, and personnel do not have access to the open portion of a stope (see Figure 1).

FACTORS CONTROLLING STOPE WALL BEHAVIOUR

Excavation geometry

In sublevel stoping, drilling and blasting is undertaken from drilling drives located at each sublevel along the height of a stope. Because of the limited cablebolt reinforcement that can be provided at the exposed stope walls, the excavations are usually designed to be inherently stable. In this regard, experience has shown that in most cases, it is possible to achieve stope wall stability (with minimal dilution) by either excavating openings having high vertical and short horizontal dimensions, or openings having long horizontal and short vertical dimensions (see Figure 2).

The shape of the stability curve is hyperbolic and suggests that for multiple lift sublevel open stopes (excavations with walls that have high vertical and short horizontal dimensions) the critical spans are either the exposed horizontal lengths or the stope widths. Length and width are the critical stope dimensions, as they also control the dimension of the stope crowns. Bench stopes are excavations where the longest dimension is the strike length and the critical spans are usually the exposed heights as the orebody width is usually narrow.

Rock mass strength

It is generally accepted that behaviour at the stope walls is largely controlled by the strength of the rock mass surrounding a stope, which in turn depends upon the geometrical nature and strength of the geological discontinuities as well as the physical properties of the intact rock bridges. Single or combinations of major discontinuities (usually continuous on the scale of a stoping block) such as faults, shears and dykes usually have very low shearing strength and, if oriented unfavorably, provide a failure surface when exposed by the stope walls (see Figure 3). Such geological discontinuities largely control overbreak and stability around exposed stope walls. In particular, those including platy and water susceptible minerals such as talc, chlorite, sericite and kaolin (Bridges, 1982).

FIG 3 - Stope hangingwall behaviour controlled by bedding and joint frequency.

In some cases, stoping activities have been linked to instability in concurrent voids along the strike or dip of main geological features such as fault zones (Logan *et al*, 1992). The location of these main geological discontinuities is well defined and most mines have a three-dimensional model of the local fault/shear network (see Figure 4). These features can also be seismically active, further increasing fall-off at the excavation boundaries. Overbreak is usually very difficult to control in this case, regardless of the blasting practices, and can only be minimised by stope sequencing.

Stope wall behaviour is also a function of the number, size, frequency and orientation of the minor scale geological discontinuities. Such discontinuity networks usually control the nature and amount of overbreak at the stope boundaries. Rock mass characterisation techniques can be used to estimate the shape and size of blocks likely to be exposed at the final stope walls (Villaescusa and Brown, 1991; Villaescusa, 1992). The geometrical discontinuity set characteristics (size, frequency, orientation, etc) relative to the stope walls largely control the amount of dilution experienced at those walls (see Figure 5). Individual joints have a limited size and they may either terminate in intact rock (forming an intact rock bridge) or against another structure within a discontinuity network. These intact rock bridges are significantly stronger than the naturally occurring discontinuities and provide a higher resistance to failure within a rock mass.

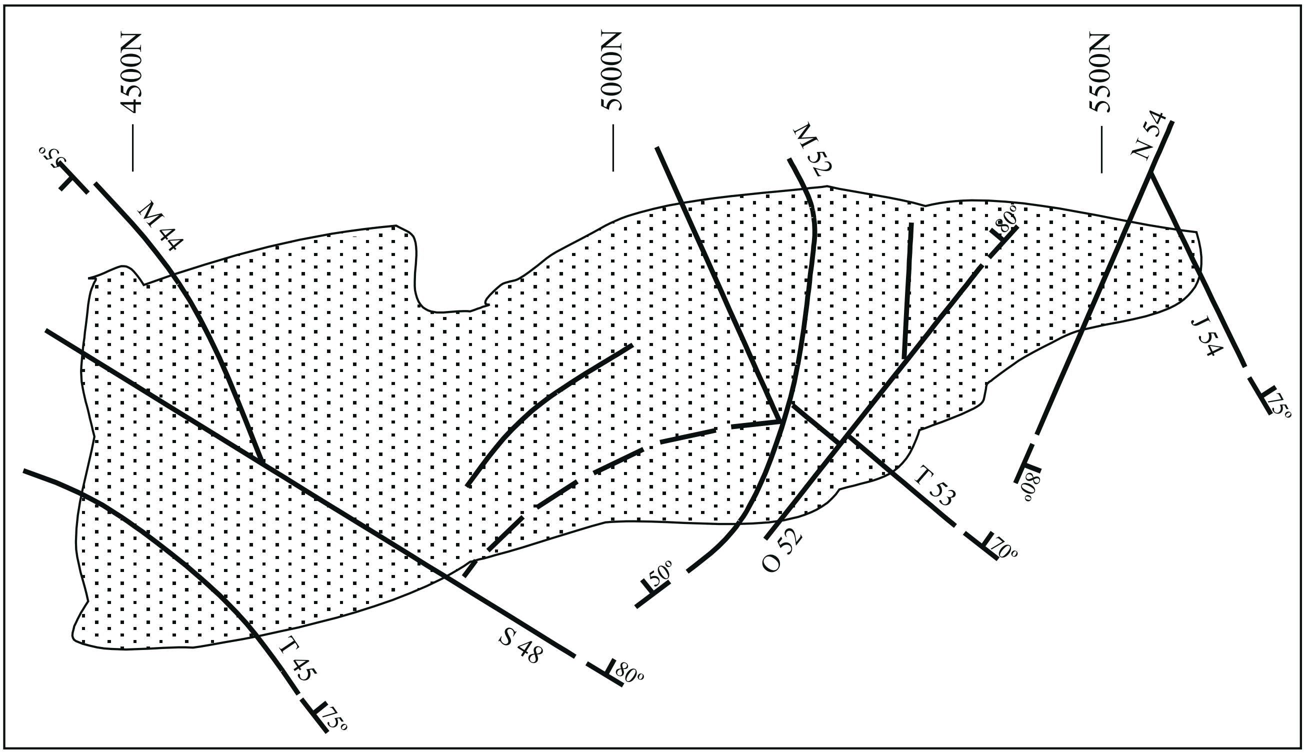

FIG 4 - Major structures affecting sublevel stoping at the 1100 orebody, Mount Isa Mines (from Alexander and Fabjanczyk, 1982).

FIG 5 - Footwall and hangingwall fall-off.

Induced stresses

Extraction within a stoping block, can generate large concentrations of stress around the excavation boundaries. If the local (induced) stresses increase beyond the shear strength of the rock mass, changes in the quality of the rock mass around the stope will occur, and localised failures are likely to be experienced either following discontinuity surfaces or directly through intact rock. Where movement through discontinuities occurs, stresses are relieved, however this may in turn lead to overbreak, dilution or large failures (see Figure 6).

The change in rock quality (before failure) around the boundaries of a stope is called pre-conditioning of a rock mass. This is a very complex process, which in most cases results from a combination of stress re-distributions, near field blast damage and the effects of the excavation itself. In cases where stope wall failures do not occur due to the initial pre-conditioning by the stresses, vibration and gases from nearby blasting may damage the intact rock bridges, which define (and interlock) the *in situ* rock blocks, causing overbreak or dilution at the stope boundaries. Furthermore, the dynamic behaviour of an unsupported wall is directly proportional to the amount of intact rock available within the rock mass. The less intact rock available, the more cracking, stabbing and visible stope wall displacement will result from the blasting process. The larger the openings, the larger the excavation-related deformations that are expected at the boundaries, making the walls more susceptible to damage from stress re-distribution and/or blasting.

FIG 6 - A large stress-related footwall buckling failure.

In addition, failures due to stress changes of a tensional nature can also be experienced (Bywater *et al*, 1983). Stope extraction in a destressed orebody may lead to normal stresses of very low magnitude across some of the exposed walls. Buckling type failures may occur in depending upon the frequency of discontinuities parallel to a stope wall, the size and frequency of any cross discontinuities and the size and shape of the exposed spans (see Figure 7).

FIG 7 - A large structurally-controlled hangingwall failure.

Ground support

Reinforcement by cable bolting provided at selected locations (usually constrained by the distance between drilling sublevels) can be used to reduce the deformations experienced at the final stope boundaries (crowns, walls and rib pillars). The stope walls are pre-reinforced prior to any stope firings and, in most cases, the cablebolts are installed from rings drilled within the stope access drives. Thus, stope wall reinforcement tends to be localized in continuous bands that are separated by the distance between the sublevel intervals. The function of such arrangements is to divide the stope walls into a number of stable stope wall spans as well as arresting up-dip hangingwall failures (see Figure 8).

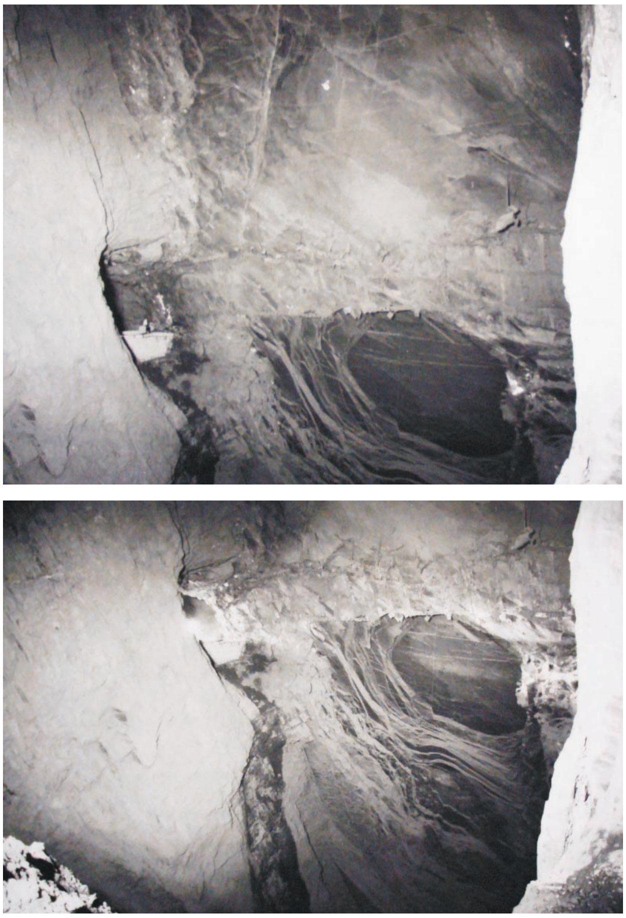

FIG 8 - A large stope hangingwall failure arrested by a row of cablebolts installed prior to stope firings.

As an alternative to installing cablebolts from stope drill drives, special drives can be developed around a stoping block solely for cablebolt installation. To decrease cost and increase their reinforcing effectiveness, such horizontal drives are usually located at the same vertical horizon as the drilling sublevels and 10 to 15 m away from a planned stope wall location (see Figure 9).

Support from minefill can also be used to minimise the deformations experienced by the stope walls and also to provide restraint of any adjacent rock masses. In general, cemented fill is needed to recover ore from secondary stopes, where stable fill exposures are required to minimise dilution. Cemented fill is essential in chequerboard extraction patterns within massive orebodies (Bloss, 1996), while uncemented rockfill is normally used in conjunction with bench stoping operations (Villaescusa and Kuganathan, 1998). An example of a bench stoping extraction strategy linked to backfill is shown in Figure 10 where the exposed walls are usually limited to a critical length value, which is defined by the distance between the minefill and an advancing bench brow (Villaescusa *et al*, 1994).

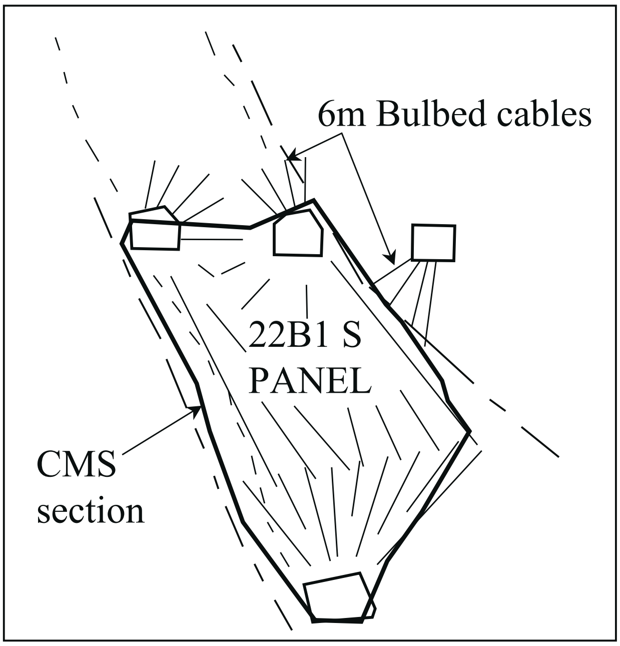

FIG 9 - Stope hangingwall cablebolting from a remote drive (Villaescusa *et al*, 1995).

Blast damage

Damage to a blasted rock mass refers to any strength deterioration of the remaining rock due to the presence of blast induced cracks and to the opening, shearing and extension of pre-existing or newly generated planes of weakness (see Figure 11). It is generally accepted that the damage is caused by expanding gases through the geological discontinuities and to the vibrations experienced from the blasting process. However, it is not easy to establish the approximate contribution to damage caused by the expanding gases, as it is difficult to measure their path within a rock mass discontinuity network (McKenzie, 1999). Nevertheless, significant backbreak may be regularly observed when the explosive gases are well confined within a volume of rock, and in some cases the gases can travel well beyond the explosive charges.

Damage by the shock energy from an explosive charge close to a blast can be related to the level of vibrations measured around the blasted volume. Repetitive blastings also impose a dynamic loading to the exposed stope walls away from a blasted volume, and may trigger structurally controlled fall-off and ultimately overbreak. Conventional blast monitoring and simple geophysical techniques can be used to measure the effects of blasting in the near field. Vibrations and frequency levels from the shock wave can be measured reasonably accurately. This can be related to damage provided the contribution (to the total damage) from the shock energy can be estimated. Vibration and frequency levels at the mid-span of instrumented stope walls can be used to characterise the dynamic response to blasting at the stope boundaries (Villaescusa and Neindorf, 2000).

Drill drive layout

Additional factors such as poorly located (or pre-existing drives) which undercut the stope walls, also contribute to dilution or fall-off at the stope boundaries. In general, the number and location of drilling drifts in open stoping is usually a function of the width of the orebody. In wide orebodies hangingwall and footwall drill drives can be used to minimise the impact of blasting at the stope boundaries. In such cases, drilling and

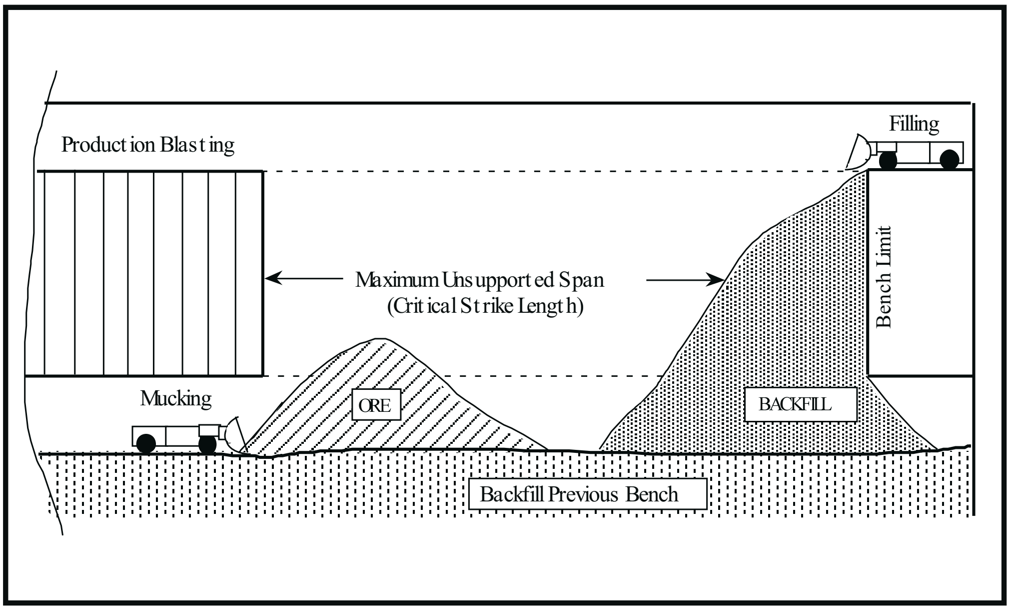

FIG 10 - Continuous filling operations in bench stoping.

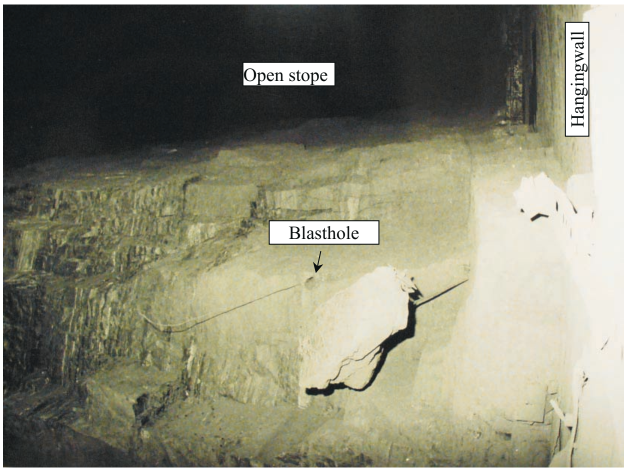

FIG 11 - Structurally controlled damage around a hole in an open stope brow.

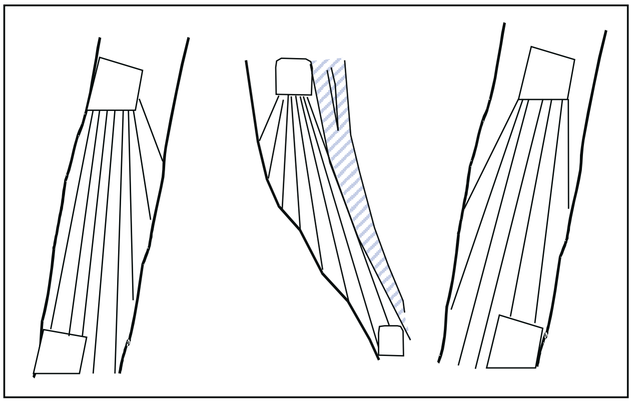

FIG 12 - Holes toeing into the stope boundaries.

blasting can be done on a plane parallel to the final stope walls or to any exposed backfill masses. Suitable values of stand-off distance (parallel to the stope boundary) for the perimeter holes can be determined, depending upon the rock type and the hole size being used (Villaescusa *et al*, 1994).

On the other hand, excessive wall damage, dilution and ore loss may be experienced in the cases where stoping requires drilling holes at an angle to a planned backfill exposure or a stope boundary (see Figure 12). In this example, hole deviation at the toes may create an uneven stope surface, thereby preventing effective rilling of the broken material to the stope drawpoints. In addition, hole deviation may cause excessive confinement at the hole toes, thus causing breakage beyond the orebody boundaries.

SINGLE LIFT STOPING

A single lift design is the most basic arrangement for a sublevel stope extraction. The stope shape and size is constrained by two sublevels; the extraction and the drilling horizons. Access to the stope is via cross-cuts off a permanent access drive parallel to the orebody. The method requires a 'moving' drawpoint system as the stoping progresses upward. Following the backfilling of a stope void, a drilling horizon becomes the next extraction level (see Figure 13).

In order to optimise mucking productivity, up to two access

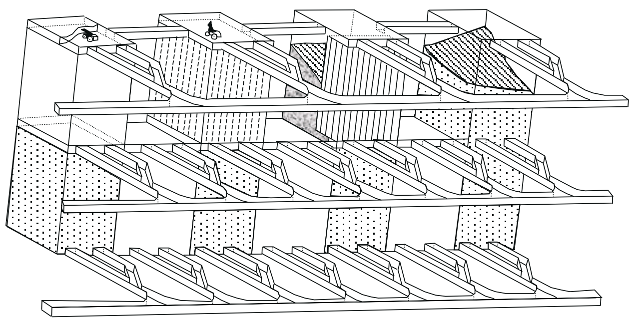

FIG 13 - Three-dimensional view of single lift sublevel stoping (Potvin *et al*, 1989).

cross-cuts per stope may be required at each sublevel interval. This actually increases the overall access development in waste to actual stoping ratio. The method requires very good control of the stope back and brow stability, especially in a highly stressed environment. Stress re-distribution due to the stoping sequence itself can create significant back failures, especially if shallow dipping discontinuities are present within a rockmass.

Extended backs and pendant pillars and highly stressed brows are likely to be formed somewhere within the stoping sequence, and full cablebolting coverage is required to minimise the potential failures at each sublevel location. Full cablebolting coverage requires stripping the orebody access to the full stope width, thereby minimising the size of stopes that can be safely developed. As a result, single lift stopes tend to be relatively small openings, compared with multiple lift stoping.

Primary development requires extending the access cross-cut to a proposed hangingwall location, where both the drill and the extraction sublevels are completely silled out to allow the installation of cablebolt reinforcement. In addition, the drilling of parallel blastholes is also facilitated with full stope undercut and overcut geometries. Parallel holes is the preferred way of drilling and blasting in vertical retreat stoping, which is linked to single lift stoping. The method requires a significant amount of remote mucking due to the flat-bottom nature of the single lift stope geometries, thereby increasing the overall mining cost compared to a multiple lift stoping geometry.

In wide orebodies a number of stopes may occur across the strike in a given area, and in all cases, adjacent primary stopes are extracted to a level above that of a secondary stope. This type of sequence creates what is called a pendant pillar. A pendant pillar is a solid piece of ground that has many degrees of freedom for movement, as most stopes around it have been extracted (see Figure 14). Large pillar failures may be experienced in such stoping geometries (Milne and Gendron, 1990).

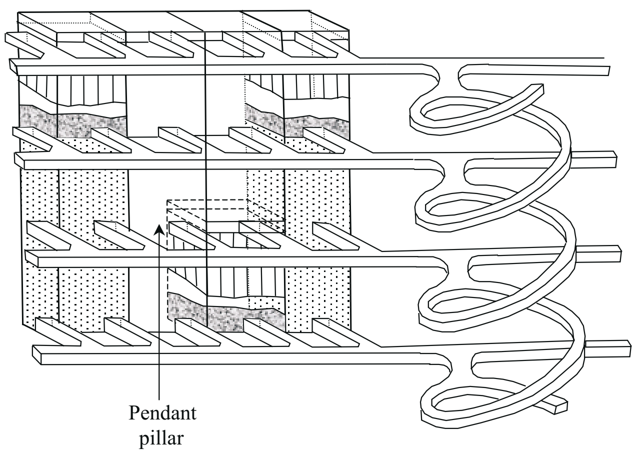

FIG 14 - Idealised stoping sequence for single stopes in a 1-4-7 extraction sequence.

Conventional vertical retreat stoping

Vertical crater retreat (VCR) is a single lift stoping method where the stopes shape is defined by a lower (undercut) and upper (overcut) horizon. Large diameter holes are drilled in order to minimise deviation, and the holes are charged from the overcut and blasted by means of horizontal slices of ore progressing from the bottom level to the top level (see Figure 15). The separation between the undercut and overcut is a function of stope wall competency, nature of the orebody and drilling accuracy.

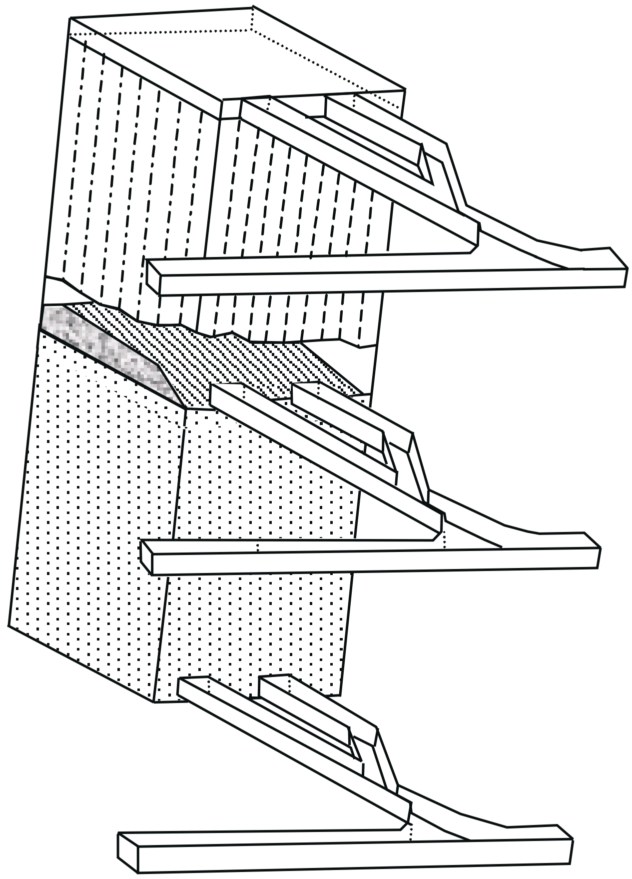

FIG 15 - Vertical crater retreat within a single lift stope.

Following blasting, only a slight amount of broken ore is mucked, so that enough room is available for a subsequent blast to break into. This keeps the stope full of broken rock, thereby providing passive support to the exposed stope walls, until blasting to the stope overcut is complete. Once blasting is completed and all the ore within the stope is mucked, the undercut accesses are closed off and the stope is filled. As mining progresses upwards, the stope overcut becomes the next mucking horizon in the sequence.

The method has a number of perceived advantages including the requirement for few large diameter blastholes, likely to reduce the overall in-the-stope drilling. Large holes enable a larger sublevel interval, thus reducing the overall sublevel development cost. The cost of raising and slashing to create a slot is eliminated, and all the drilling and loading operations are carried from the overcut, thereby increasing safety.

The disadvantage of this method is the potential for blast damage from crater blasting at the stope boundaries. Small diameter holes are not used due to hole closure caused by ground movment following the individual stope blasts (Hills and Gearing, 1993). In addition, this method may be susceptible to poor fragmentation (fall-off) from the unsupported areas defined by blasting, especially if an uneven back is formed and high stresses are subsequently redistributed upwards. Blast damage from cratering is even more detrimental when shallow dipping geological discontinuities are present within a rockmass.

Modified vertical retreat stoping

A modified vertical retreat method uses a winze or a raisebored hole, which is located near the middle of the stope, into which a radiating pattern of blastholes is sequentially fired in horizontal lifts. The raise is used to overcome the limited free face available in a conventional vertical retreat stope. In order to facilitate the initial blasting, the method requires close spacing of the holes near the raise (see Figure 16). All the holes in a horizontal lift are fired, and some danger of collar damage exists when the inner holes near the raise do not perform. In addition, hole damage (closure, requiring re-drilling) within the last lift in the stope may be continuously experienced with this method (Hills and Gearing, 1993). On the other hand, the method is considered a relatively safe method because no vertical opening is made within the stope until the last firing.

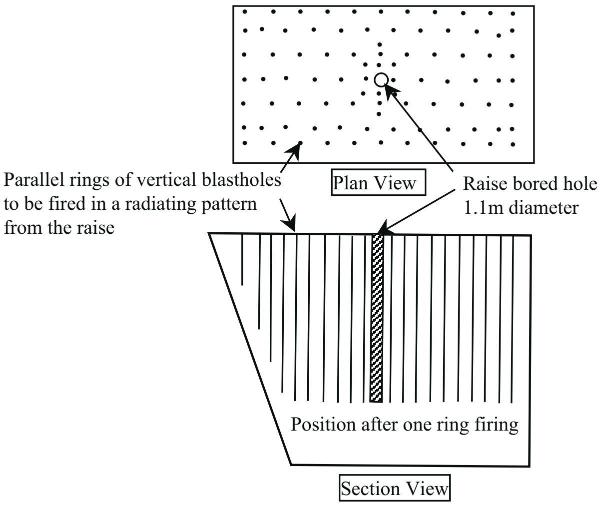

FIG 16 - Typical blast layout for a modified vertical retreat stope in the Porgera Mine (Hills and Gearing, 1993).

MULTIPLE LIFT STOPING

Multiple lift stopes extend vertically over a number of sublevel intervals, in some cases exceeding hundred of metres in vertical extension. Stope extraction starts by creating a cut-off raise, which is expanded to a cut-off slot and extended over the full stope height. Main production rings are then fired progressively into the void created by the cut-off slot until the stope is completed. Ore breakage is achieved by rings of parallel or fanned blastholes, depending upon the type of drilling access used. Trough undercuts are developed at the base of the stopes in order to direct the broken ore into the drawpoints for extraction.

The number of drawpoints is usually a function of the stope size, but in most cases at least two drawpoints are designed. Because the drawpoint location is fixed, permanent reinforcement can be afforded at minimum cost per unit of ore extracted. Access to the stope on each of the other sublevel locations is required for drilling, blasting and backfilling purposes. Usually, a single crosscut access is required at each sublevel, significantly decreasing development in waste (see Figure 17).

In general, multiple lift stopes minimise back cable-bolting within the intermediate sublevels because a permanent back (full area) is only exposed at the actual crown of the stope. Consequently, stope crown stability is facilitated as reinforcement may only be required within a finite area on the top of the stope. Cablebolting coverage at the stope crown is a function of the degree of development within the top sublevel. In addition, the requirement for permanent reinforcement within any intermediate sublevel is minimised by the fact that all the back exposures within the drill drives are consumed by the stoping process itself. The stopes are usually sequentially sliced

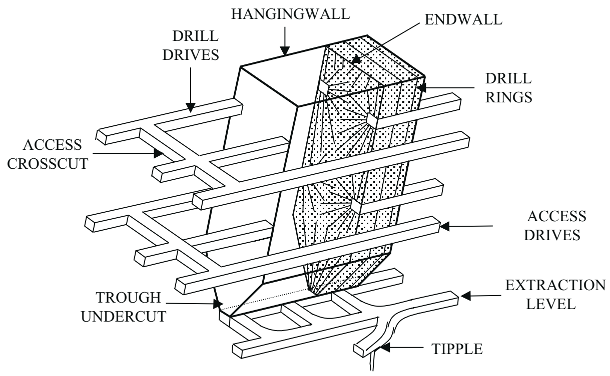

FIG 17 - Idealised three-dimensional view of a multiple lift sublevel stope.

up, firing rings towards an open slot, from sublevel to sublevel. The blasting sequences can be designed to minimise undercutting the individual sublevels during the stope blastings. A straight face is kept along the entire stope height by firing a similar number of rings at each level. The firing sequence advances upward as shown in Figure 18. Maintaining a straight retreating face along the entire open stope height minimises the creation of brows, which can be highly stressed and contribute to stope fall-off, which in turn can severely affect productivity during the subsequent mucking operations.

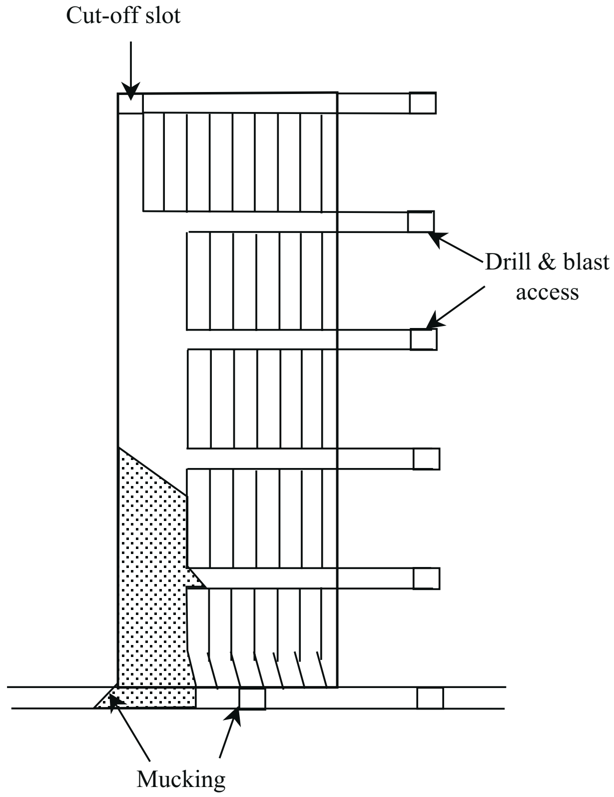

FIG 18 - A long section view of a multiple lift stope extraction.

The outline for multiple lift sublevel stopes in tabular orebodies is usually associated with the use of long blastholes drilled from drives parallel to the orebodies. Depending upon orebody width these drill drives are either full orebody width or located at the boundaries of the orebodies. In such orebodies the stope boundaries are usually well defined by the orebody itself. Crown, hangingwall, footwall, end-walls and a drawpoint can be defined for each stope. The stability of stope crowns and hangingwalls is usually the most critical factor in the design and extraction sequences. A conventional design usually consists of multiple drilling sublevels with a single mucking horizon at the bottom of the stope as shown in Figure 19.

One of the advantages of this design is that drilling and blasting can be done on a plane parallel to the final stope walls. Hangingwall and footwall drill drives are used to minimise the impact of blasting at the stope boundaries greatly decreasing the likelihood of dilution due to blast damage. In addition, the method reduces stope development in waste, given that except for the mucking horizon, a single stope drilling access is actually required at each sublevel location. The stability is enhanced when all the sublevel accesses are excavated after the adjacent (previously extracted) stopes have been backfilled.

Open stoping in large, massive orebodies consists of a mining sequence that requires several stages of stoping in conjunction with the application of delayed minefill methods to enable pillar recovery. Usually, this requires that a number of stopes are designed between the orebody boundaries. In such cases, stoping comprises a number of stages which include primary, secondary and tertiary stopes which are usually extracted using a checker board sequence (Alexander and Fabjanczyk, 1982). The number of fill exposures ranges from none (in a primary stope) to up to three exposures within the late stages of stoping.

Large vertical dimensions can be designed with the height of the stopes usually constrained by the orebody thickness or by the stability of the exposed fill masses required for secondary and tertiary stope extraction. Stope dimensions in plan view are usually constrained by stope crown instability. The broken ore is also extracted in the bottom part of the stope (see Figure 20).

In cases where the ground conditions are favourable, stope dimensions can be very large in plan, with full orebody height extraction achieved in a single stope (Bloss, 1996). Drilling and blasting is carried out from a series of sublevel locations ranging from 40 - 60 m apart. Blastholes are mainly drilled downwards, with some short upholes drilled within the trough undercuts and sometimes at the stope crown when a top access is not available.

Following pillar extraction (secondary and tertiary stopes), a number of backfill exposures are created depending upon the location of the stope in the mining sequence. Pillar stope nomenclature is usually based on the number of exposed backfill masses. As an example a two SLOS stope has two fill exposures. Early in the life of a massive orebody primary stopes usually account for a significant part of the production. As an orebody extraction increases, the shifting to pillar mining as the primary method of extraction is evident. In such cases, the stability of the fill exposures is of primary importance to achieve target production (Bloss and Morland, 1996).

The stope cut-off slot

Sublevel stopes are created by the sequential blasting of production rings into an initial opening that is called the cut-off slot. The cut-off slot is located on a side of the stope either transversally (across) or longitudinally with respect to the strike of an orebody. The critical point relates to whether the cut-off blastings will expose a critical stope wall (such a hangingwall or a backfill mass) very early in a stope blasting sequence. The slot

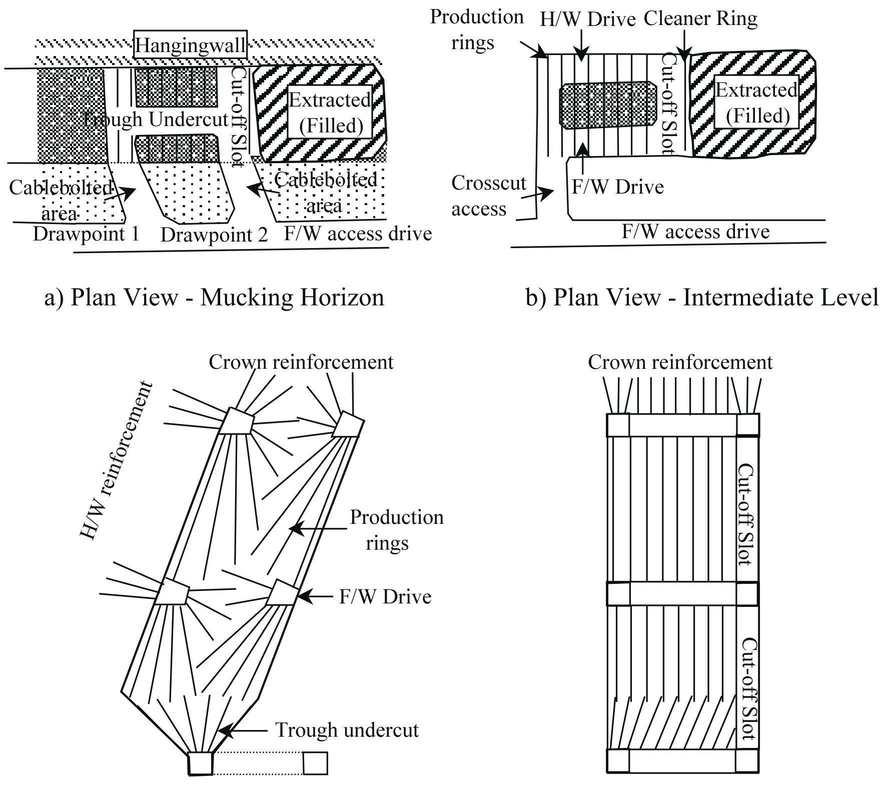

FIG 19 - Sublevel stoping in a tabular orebody.

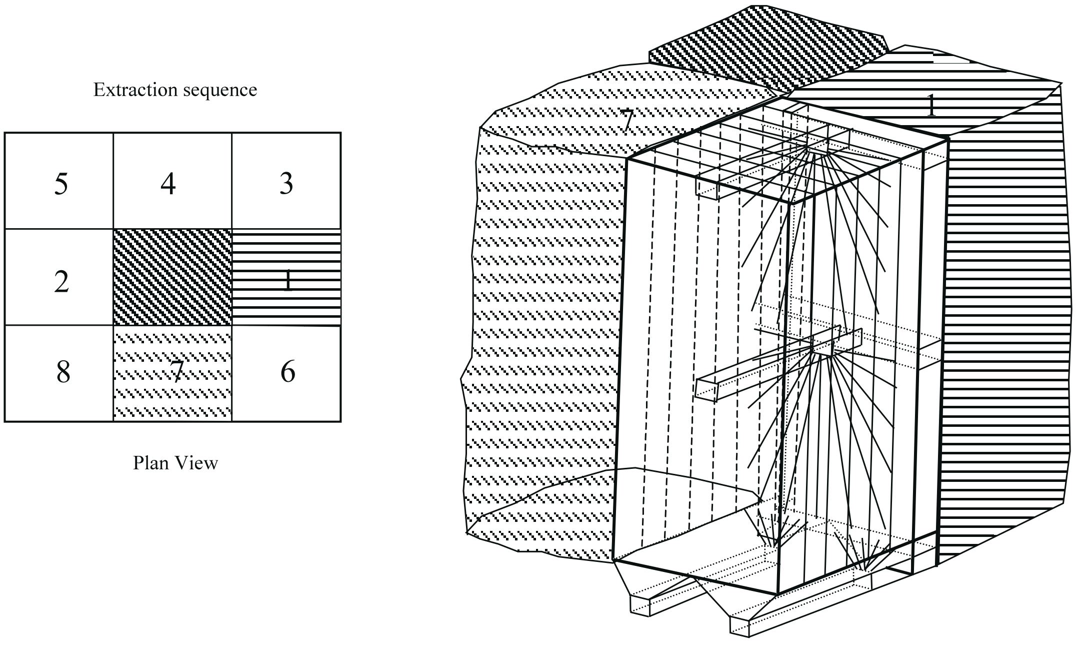

FIG 20 - Sublevel stoping in a massive orebody.

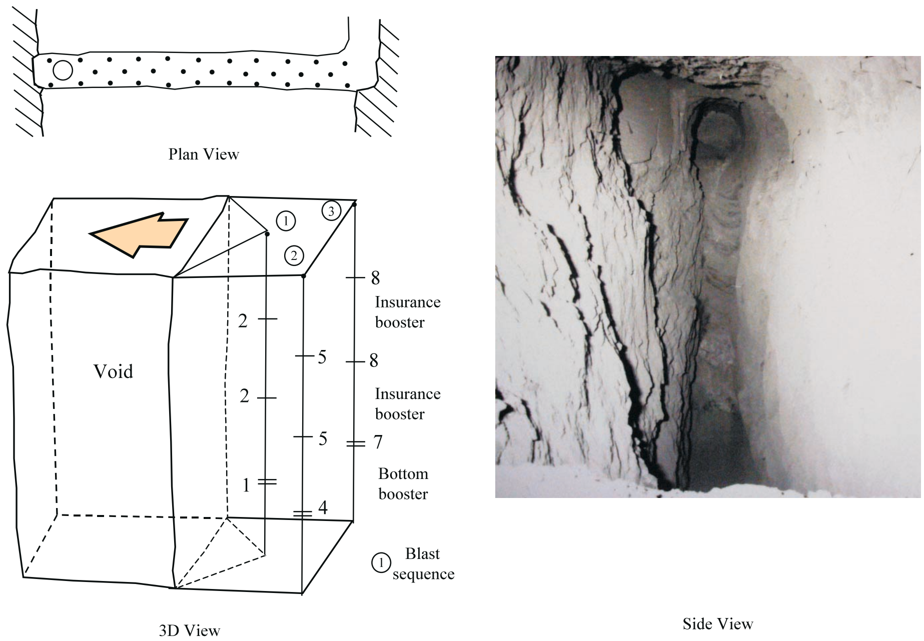

FIG 21 - Sequential blasting of a cut-off slot.

raises are blasted upwards from sublevel to sublevel in order to expose a full stope height. At each level the slots are formed by sequentially blasting parallel holes into a long-hole winze (LHW) or a raised bored hole. The slot must be expanded to the full width of the production holes that will be subsequently blasted into this initial opening (see Figure 21).

High powder factors are normally used during slot blasting in order to ensure breakage and thus to have a free face and a void available where the remainder of the stope is to be blasted. The choice of slot location depends upon rock mass conditions, stope access and the extraction sequence chosen. In a steeply-dipping orebody, where the critical stope boundary is usually the inclined hangingwall, transversally oriented slots are used to ensure a sequential hangingwall exposure by the production rings. In large, massive orebodies, the choice of slot orientation is also controlled by factors such backfill exposures, stress regime and pre-established access.

In general, a slot must be designed so that failure within the main production rings is minimised. In highly stressed pillars a slot can be oriented normal to the main principal stress to shadow the main production holes. This is likely to minimise hole squeezing or dislocation due to stress related damage. In cases where a stope access can be re-designed, the slot can be placed normal to geological features likely to fail and damage the main ring geometries (see Figure 22).

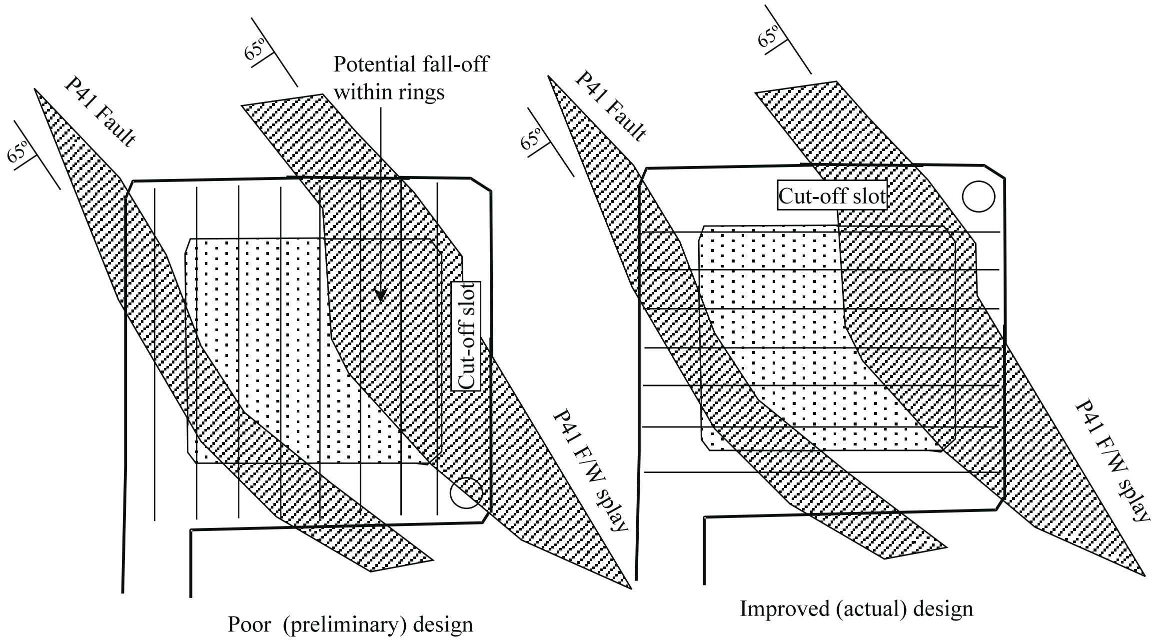

FIG 22 - Exposure of weak geological features by a cut-off slot (After Rosengren and Jones, 1992).

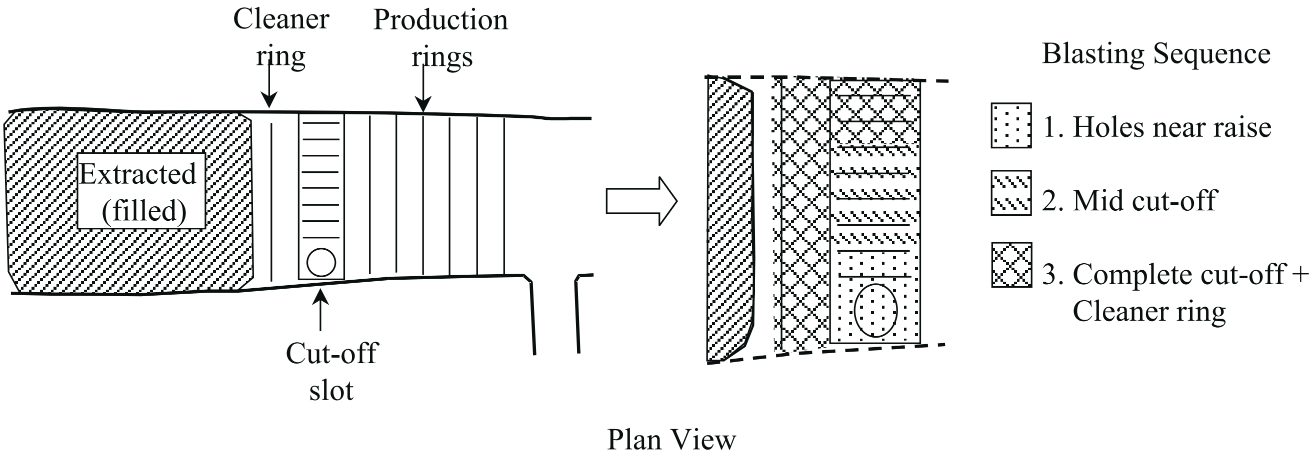

FIG 23 - Cleaner ring geometry to minimise fill damage from blasting.

Damage to fill masses from cut-off slot blastings can be minimised by placing a cleaner ring between the cut-off and the fill boundary (see Figure 23). The rockmass adjacent to a fill mass is usually pre-conditioned by stress re-distributions and it is likely to fail following the cleaner ring blasting. This practice, however, minimises dilution from fill failures, as the cut-slots are blasted away from the fill masses.

Experience suggests that (except for highly stressed pillars) stope hangingwall failures are unlikely to occur during a transversely oriented cut-off slot formation, because the exposed hangingwall planes shapes are likely to be within a stable range of exposures (see Figure 2). The risk of a hangingwall failure during slot blasting increases when the hangingwall is undercut by the stope development or when the stope is undergoing large stress changes. In cases where longitudinal cut-off slots are located parallel (and adjacent) to a stope hangingwall, the slot exposes the full hangingwall plane early in the stope life. This usually limits the size of exposures that can be safely excavated, as this critical wall of the stope may fail when subjected to repetitive dynamic loading by the rest of the stope firings as shown conceptually in Figure 24. In addition, such stoping geometries are likely to toe holes into the adjacent backfill masses, thereby increasing the likelihood of dilution.

On the other hand transversely oriented cut-off slots create a full exposure at one of the stope end-walls early on the stope life. In cases where the stope is located within a highly stressed environment, failures within such walls may occur. Although, strictly speaking, fall-off from stope end-walls does not constitute dilution, the stope mucking productivity can be severely affected, especially if the size of a failure is such that secondary breakage is required (Anderson and Leblenc, 1995).

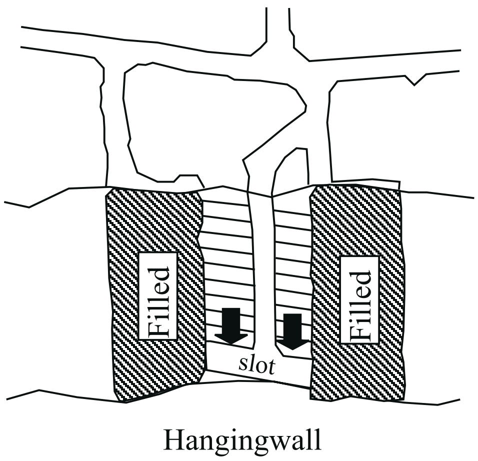

FIG 24 - Dynamic loading of a fully exposed hangingwall plane.

The production rings

A designed stope shape is achieved by sequentially blasting rings of blastholes into the opening created by the cut-off slot. The main rings are sequentially blasted on each sublevel attempting to minimise undercutting of the internal solid portion of a stope. Avoiding undercutting usually reduces fall off from retreating stope brows as damage from stress re-distribution is minimised when a straight face is maintained along the entire stope height (see Figure 18).

Conventional multiple sublevel stoping requires the sequential exposure of high vertical, short horizontal stope walls likely to remain stable and provide undiluted ore. The strike lengths exposed during the initial stoping extraction are unlikely to exceed the critical unstable stope spans. As the excavations are enlarged and several rings are sequentially blasted into the void formed by the cut-off and the initial production rings, confining stresses are reduced, and excess strain energy is induced and displacement of the stope walls is experienced. Depending on the structural nature of the exposed walls, the rock may tend to displace following a sheet-like behaviour, in which a group of layers move together (bedded rock), or the movement may be isolated to individual blocks which partially rotate and slide against each other.

Once enough room is available, the most appropriate way to blast the rest of a stope can be considered, depending on the circumstances such as the level of the induced stresses or production requirements and access constraints (see Figure 25). As production blasting continues towards the final stope geometry (shape and size), the excavation becomes more unstable. Geological discontinuities as well as the dynamic impact of blasting begin to affect stope wall stability and contribute to dilution.

Diaphragm rings consist of rings drilled parallel to a fill exposure. The purpose of a diaphragm ring is to prevent fill failure from a known weak cemented fill mass, to contain uncemented fill in adjacent stopes or to prevent fill failure from exposures of a greater dimension than is considered stable. Experience has shown that although parts of a diaphragm against backfill do fall off, this rarely results in excessive backfill dilution, as the fill mass remains comparatively undisturbed, compared to when blasting takes place next to the backfill (see Figure 26). A diaphragm is not capable of load bearing capacity and it is likely to deform considerably. However, when a large portion of the diaphragm remains intact, this enables clean stope extraction until the diaphragm is either fired or the stope is completed.

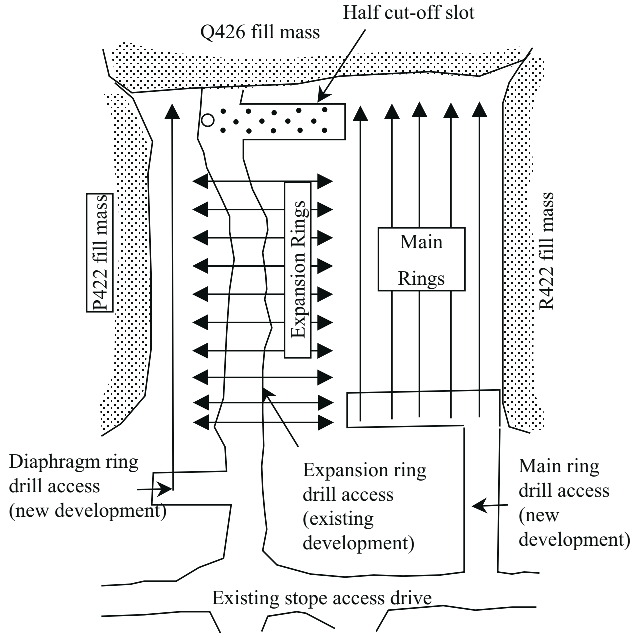

FIG 25 - Influence of existing development and adjacent stoping on a stope firing sequence (Bloss and Morland, 1996).

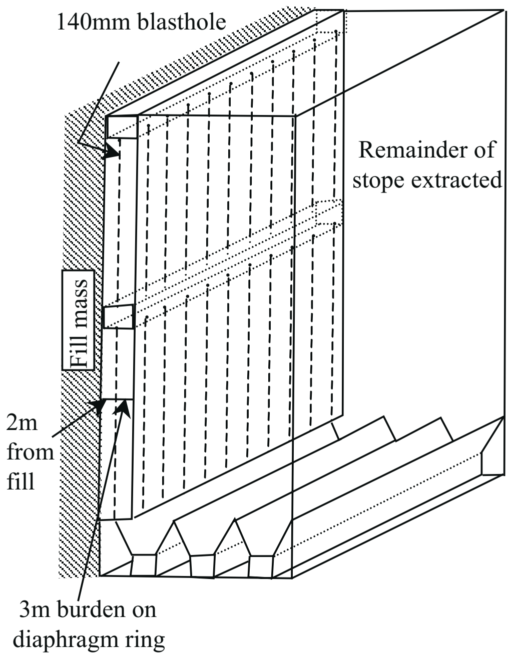

FIG 26 - Idealised sketch showing a diapraghm ring.

The stope undercut

The lower portion of a stope is shaped using trough undercut (TUC) rings in order to facilitate the draw of fragmented ore to the stope drawpoints and to minimise remote mucking. A TUC ring consists of parallel upholes drilled inclined towards the cut-off slot. Usually the toes of the TUC ring interlock with the toes of the main ring downholes from the sublevel above (see Figure 27). Drilling and blasting of the trough undercuts is usually carried out using relatively small diameter holes (70 - 89 mm). An improved explosive distribution likely to minimise rock mass damage around the stope drawpoints is achieved by using such small diameter holes. A disadvantage is the limited drilling length achieved, and the inability to match the burden drilled for the production ringholes immediately above

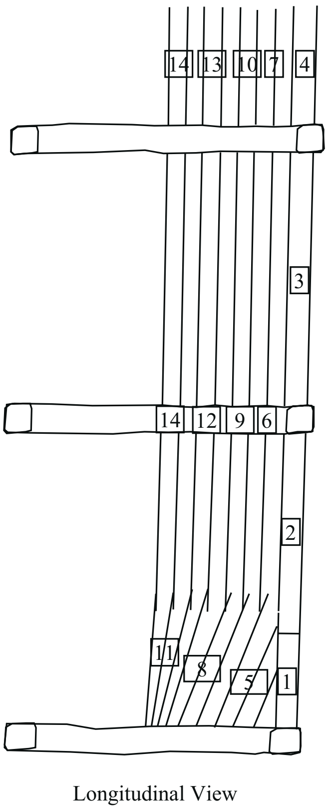

FIG 27 - Firing sequence of a trough undercut with main rings in an open stope.

Because the TUC rings are drilled with a different burden to the main rings, the lower portion of the stope is usually blasted ahead of the main rings. This leads to undercutting of the main rings, which can lead to fall-off, especially in cases where large geological discontinuities are present or in regions of high stress re-distribution.

The drawpoints

Mucking in multiple lift sublevel stoping is usually carried out transversely across the strike of the orebodies. This requires the introduction of fixed and specialised drawpoint geometries that may be located outside the orebody boundary (see Figure 28). The factors considered during drawpoint design, include size of equipment, tramming distance from access drives, gradient and orientation with respect to a stope boundary. The drawpoint dimensions must be sufficient to suit the equipment, but kept as small as possible to minimise instability. Drawpoint access should be straight and restricted to 15 - 20 metres in length from the stope access drive to the stope brow. This will ensure that auxiliary ventilation will not be required while mucking, and also that rear of the mucking unit is inside the drawpoint. Drawpoint spacing is determined by ground conditions and stope geometry. In most cases the minimum spacing used is 10 - 15 m between centre lines.

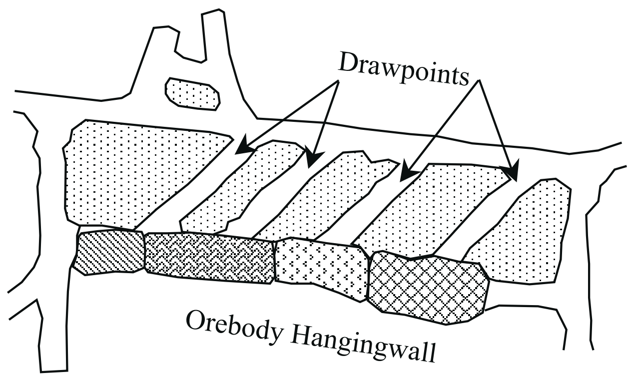

FIG 28 - A plan view of a fixed drawpoint geometry in sublevel stoping.

CONCLUSIONS

An understanding of the different factors controlling stope wall behaviour, as well as the response of a rockmass to the different styles of stoping geometry is critical to the successful application of sublevel stoping. In particular, single lift stoping usually requires significant lateral development where extended highly stressed backs and pendant pillars are likely to be created. Consequently full cablebolt coverage may be needed at each sublevel location, as the method requires moving drawpoints for mucking. In addition, a significant amount of remote mucking is required due to the flat-bottomed nature of the single lift stopes. On the other hand, multiple lift stoping requires significant vertical development while allowing the use of specialised trough undercuts and drawpoint geometries. Furthermore, reinforcement within the stope itself is minimal and limited to the stope crown, hangingwalls and drawpoints, but not necessarily required within the intermediate sublevels. More importantly, this method minimises stope wall dilution and fall-off from rings within the stope itself by firing a similar number of rings on each sublevel and creating inherently stable excavation shapes.

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