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Methods to Mine the Ultra-Deep Tabular Gold-Bearing Reefs of the Witwatersrand Basin, South Africa

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CHAPTER 1

Methods to Mine the Ultra-Deep Tabular Gold-Bearing Reefs of the Witwatersrand Basin, South Africa

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1.1 ABSTRACT

The steady depletion of the gold reserves of the Witwatersrand Basin has led mining companies to consider extending mining to ultra-depth (3500 m to 5000 m below surface), where a gold resource of 50 million oz at a grades exceeding 13 g/ton is believed to exist. For several decades the deep tabular reefs of the Central, West and Far West Rand were largely mined using layouts characterised by longwalls, with strike pillars providing regional support. During the past few years a new generation of layouts has been introduced, based on the concept of scattered mining with dip pillars for regional support.

The DEEPMINE research programme seeks to establish whether or not these layouts are suitable for mining at ultra-depth. A realistic model of a ultra-deep Ventersdorp Contact Reef orebody is used to analyse and compare the performance of four mine designs: longwalling with strike stabilising pillars, sequential grid mining, sequential down dip and closely-spaced dip pillar mining. This paper highlights the relevant design considerations of these methods, and gives pointers to some of the major concerns in mining at ultra-depth.

1.2 INTRODUCTION

Gold was discovered in 1886 in Archean conglomerates, which outcrop near Johannesburg. Since then the Witwatersrand Basin has produced about half the gold ever mined, worldwide. The conglomerates, known locally as reefs, vary in thickness from a few centimetres to several metres, and extend over wide areas. Consequently, the mining excavations (stopes) formed during the extraction of the ore have a tabular geometry. The reefs were traced around the perimeter of the basin, where they are mostly concealed by younger strata, leading to the discovery of several new goldfields. The reefs were also followed down-dip, the mines becoming the deepest in the world, with stoping currently taking place at depths exceeding 3500 m. Each year about 20 million square metres of reef is mined, and some 800 km of new tunnels are developed in addition to the 10,000 km in use.

1.2.1 Scattered mining methods

Scattered mining methods were initially employed to mine the gold-bearing reefs. A typical layout is shown in Figure 1.1. As the depth of mining increased, so the stress-related problems of fracturing, mobilisation, and overall instability worsened, especially in cases where relatively small areas of unmined reef, known as remnants, were formed.

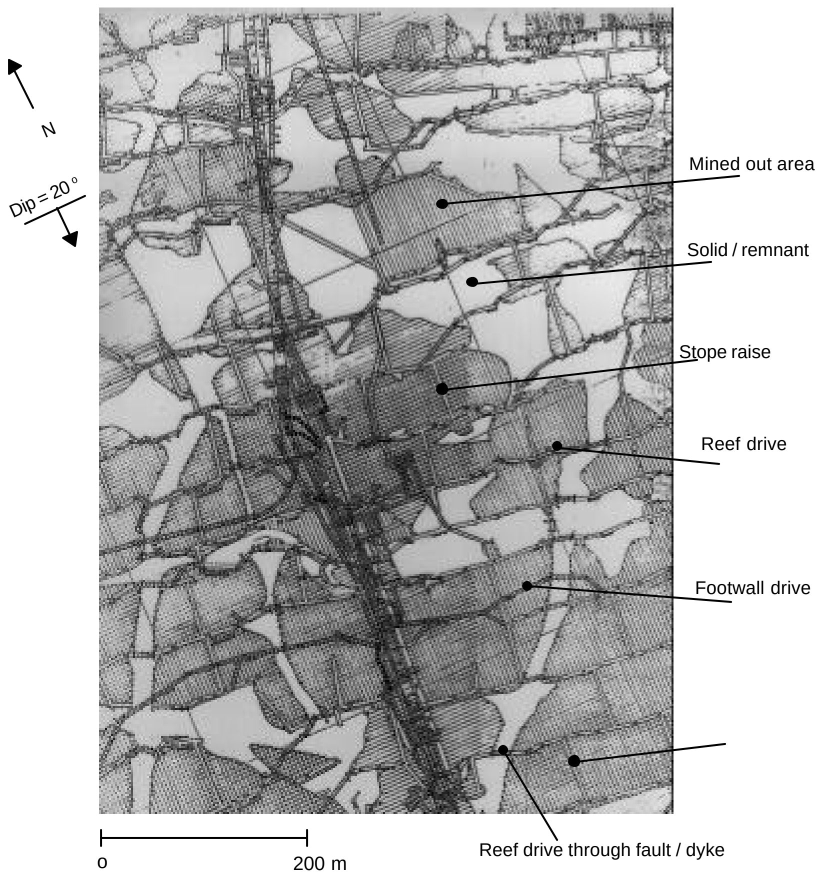


Figure 1.1 Plan showing early scattered mining at a depth of about 1800 m below surface in a mine on the Witwatersrand Basin

In some cases tremors occurred as a result of the violent failure of the stressed rock, causing damage to the excavations and injury to workers. In the mid-1920's a statutory Rockburst Committee issued a number of recommendations aimed at reducing the incidence of rockbursts in South African gold mines. The most significant recommendations, which remain valid today, are:

?? the formation of isolated areas of reef (i.e. remnants) should be avoided;

?? stoping operations should be concentrated as far as practicable; and

?? working faces should be advanced rapidly and continuously.

1.2.2 Longwall mining

The longwall method was devised in an attempt to improve mining conditions. The first systematic application of the method was at the East Rand Proprietary Mines (ERPM) during 1941 (Ortlepp and Steele, 1975). Early ERPM longwalls were designed to have overall face lengths between 900 and 1200 m (Figure 1.2). However, extensive falls of ground occurred, and it soon became apparent that these very long longwalls did not solve the rockburst problems at depth, because stress concentrations at stope faces increased as the longwall stope spans increased, enhancing the probability of hangingwall failure and also of rockbursts.

The ideal orebody for the longwall mining method is one that has a consistent grade distribution, with few disruptions of the reef by faulting and intrusions, conditions that do not always occur. Amongst the benefits of longwalling were a reduction of number of remnants; a reduction of the number of lead-lags between faces, which were shown to be associated with rockbursts; better utilization of ventilation; and the concentration of mining activities leading to improvements in supervision and the provision of services (Beck et al, 1961).

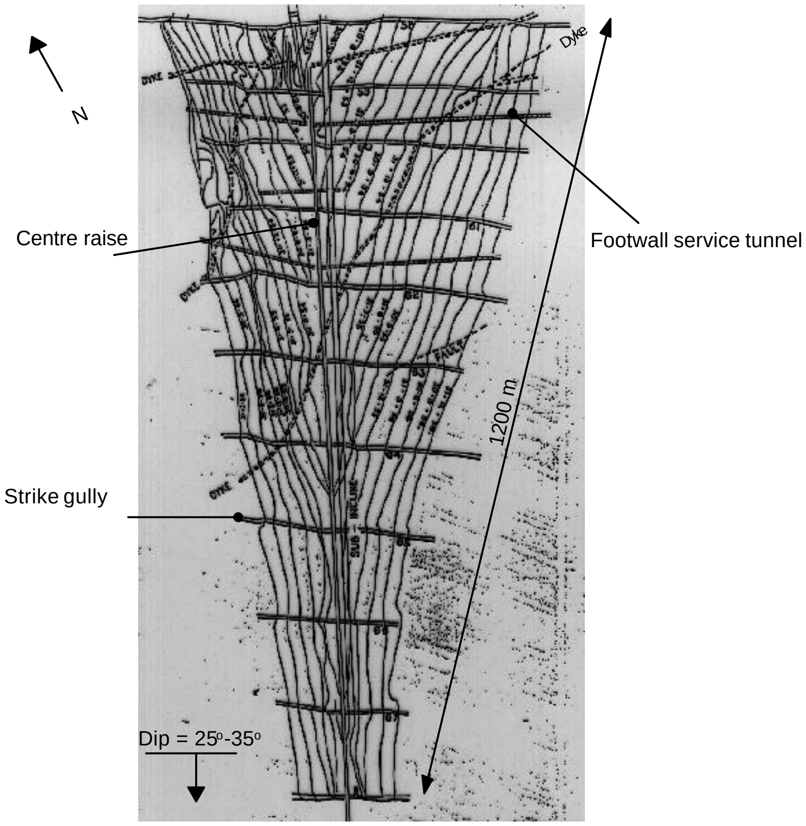


Figure 1.2 Typical longwall section at ERPM during late 1950's at a depth of about 2100 m below surface. Note the extremely long overall face length (1200 m).

1.2.3 Strike stabilising pillars

The next significant advance in mine design came through the realisation that the rockburst hazard could be reduced either by increasing the rate at which energy is dissipated in a non-violent manner, or by reducing the rate at which the energy is released (Cook et al., 1966). Three practical methods were proposed that would limit convergence and, consequently, also limit the rate of energy release:

1. **Reduction in the stope width.** Operational constraints, however, dictated that the stoping width could not be made significantly less than 1 m.

2. **Filling of the stope.** Early attempts at waste filling had not been effective in limiting closure, but it was believed that hydraulic backfilling could lead to appreciable benefits.

3. **Partial extraction of reef**. The cutting of pillars offered, by far, the best chance of reducing the amount of energy released (Cook and Salamon, 1966).

Strike stabilising pillars were introduced at ERPM in the 1960's, where stoping was already approaching depths of 3000 m. These pillars were about 80 m wide, with an accompanying longwall dip span of 270 m. In the late 1970's this design was reviewed, and the pillar width reduced to 60 m. This design afforded a high extraction ratio while limiting the average pillar stress to 400 MPa. Strike stabilising pillars were subsequently introduced on several other deep Witwatersrand gold mines (e.g. Blyvooruitzicht, Western Deep Levels, Kloof). A typical longwall layout incorporating strike stabilising pillars in a Carbon Leader Reef horizon (CLR) is shown in Figure 1.3.

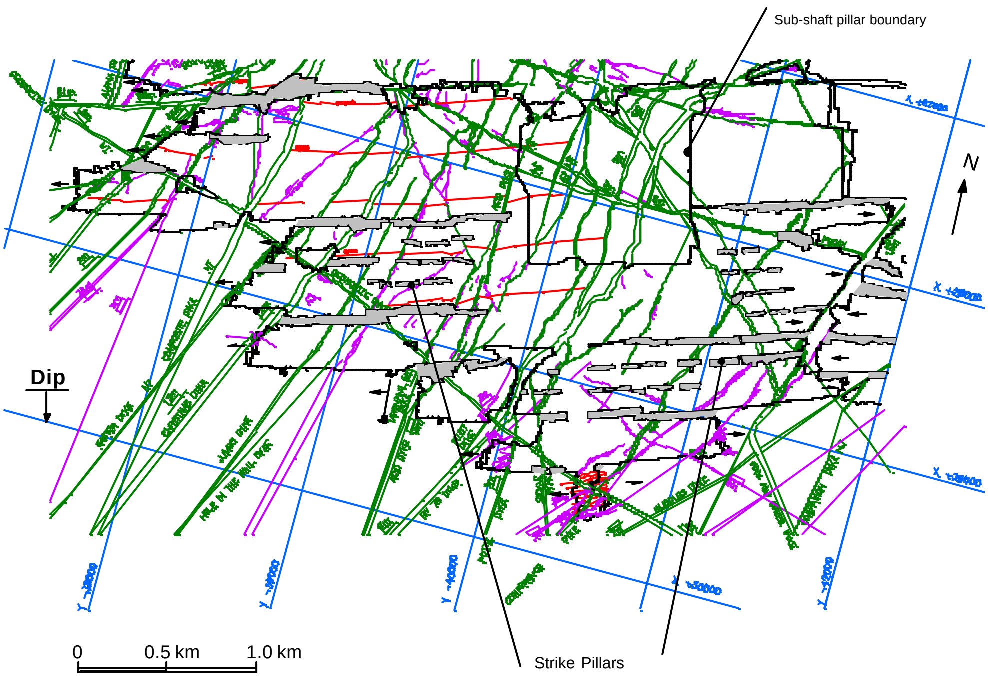


Figure 1.3 Actual longwall layout with strike stabilizing pillar at Western Deep Levels, Carbon Leader Reef. Note the complexity of geology on which the method is implemented.

Empirical rock engineering criteria are applied to the design of regional stability pillars. Parameters commonly considered are the average pillar stress (APS), the width:height ratio and the energy release rate (ERR). Pillar stability is improved by keeping foundation stresses and stress changes to the minimum. The understanding is that the APS should not exceed 2.5 times the uniaxial compressive strength of the country rock. The probability of pillar failure increases sharply for width to height ratios less than 15:1 (Salamon and Wagner, 1979). To avoid failure by punching, the minimum principle stress, $?_3$, in the zones above and below the pillar must remain compressive, and

$$?_1 - k?_3 < ?_c \qquad \text{Equation 1.1}$$

where $?_1$ is the maximum principle stress, $?_c$ is the uniaxial compressive strength (UCS), and k is a factor that describes the effect of confining stress on rock strength (typically $6 < k < 10$).

Case history: Western Deep Levels

A comprehensive review of the development and implementation of longwalling with strike stabilising pillars at Western Deep Level mine is given by Hagan (1987) and Diering (1987). In 1979 fatalities on the mine were running at double the industry average, and rockbursts accounted for a large proportion of fatalities and accidents. It was estimated that the average energy release rate (ERR) for the mine was 40 MJ/m². It was decided to strive to halve the ERR by introducing stabilising pillars in order to improve mining conditions. The pillar width:height rule of 15:1 was applied. On the Carbon Leader Reef (CLR), where the stoping height is of the order of 1 m, 20 m wide pillars were spaced at 133 m centre-to-centre, while on the Ventersdorp Contact Reef (VCR), where the stoping height may be as great as 1.5 m, 30 m pillars were implemented. Pillar widths of 20 m or more also met the criterion that the average pillar stress should be less than 2.5 times the uniaxial compressive strength of the surrounding strata to avoid the risk of pillar punching (Cook *et al*, 1973). These layouts yielded an extraction ratio of about 85 per cent.

Cutting of 20 m wide pillars on the CLR longwalls commenced early in 1980. Initially the pillars had 8 m wide ventilation slots cut through them at 120 m intervals, but these slots soon proved to be too narrow and were increased to 16 m. The position of stabilising pillars had also to be taken into account when placing service tunnels, travelling ways, and other auxiliary developments. Twenty-six new pillars were cut on the CLR in the six years following the introduction of the new design. Although the rockburst incidence was reduced, a number of serious problems were associated with the narrow (20 m) pillars (Diering, 1987; Hagan 1987; Lenhardt and Hagan, 1990):

1. The area of the stope immediately up-dip of the pillar suffered severe and regular damage from rockfalls and rockbursts.
2. From within the ventilation slots it was observed that the pillars were pervasively fractured, leaving no solid core, and that appreciable deformation was taking place.
3. Cutting of the ventilation slots proved very difficult and rockbursts often occurred when doing so.
4. It was difficult to maintain pillar widths, and pillars often ended up being less than 20 m wide.
5. Some of the pillars failed violently, with the seismic source parameters indicating a crush-type failure mechanism.

As a result of these problems, the pillar width was increased to 40 m, with a 280 m centre-to-centre spacing. This layout design still enabled an 85 per cent extraction ratio to be achieved. It was observed that the 40 m pillars had an unfailed core, indicative of greater stability. Furthermore, the layout using 40 m pillars is less development intensive, and eliminates the need for ventilation slots to be cut into the pillars.

1.2.4 Backfill

The need to apply good quality backfill in deep level mines was highlighted by Diering (1987). It was thought that backfill could partially replace pillars and increase the extraction ratio. Filling was recognised as having the added advantages of improving ventilation and reducing the fire risk. A system of 40 m wide strike pillars at 280 m centres can maintain energy release rates (ERR) below 30 MJ/m^2 at working faces of a longwall up to 3200 m below surface (Hagan, 1987). At 4000 m below surface, 90 m wide strike pillars centred at 280 m are required to maintain ERRs below 30 MJ/m^2, but the extraction ratio would be only

68 per cent. Hagan (1987) concluded that maximum benefit would be a system integrating pillars and backfill.

Hydraulic backfill became widely used in the early 1980s, and it was observed that backfilled stopes had significantly lower closure rates than unfilled stopes (Gürtunca et al, 1989). The placement of backfill is is now well established in a number of South African deep mines. Piper and Ryder (1988) state that the use of backfill in combination with stabilizing pillars can provide a reduction of up to 30 per cent in both the rate of energy released and average pillar stresses of regional layouts.

1.2.5 Dip pillar systems

Dip orientated pillar systems were first used in the South African gold mining industry during the 1950s. For example, a method called *wide-end stoping* was successfully used to limit subsidence at shallow depth (Figure 1.4).

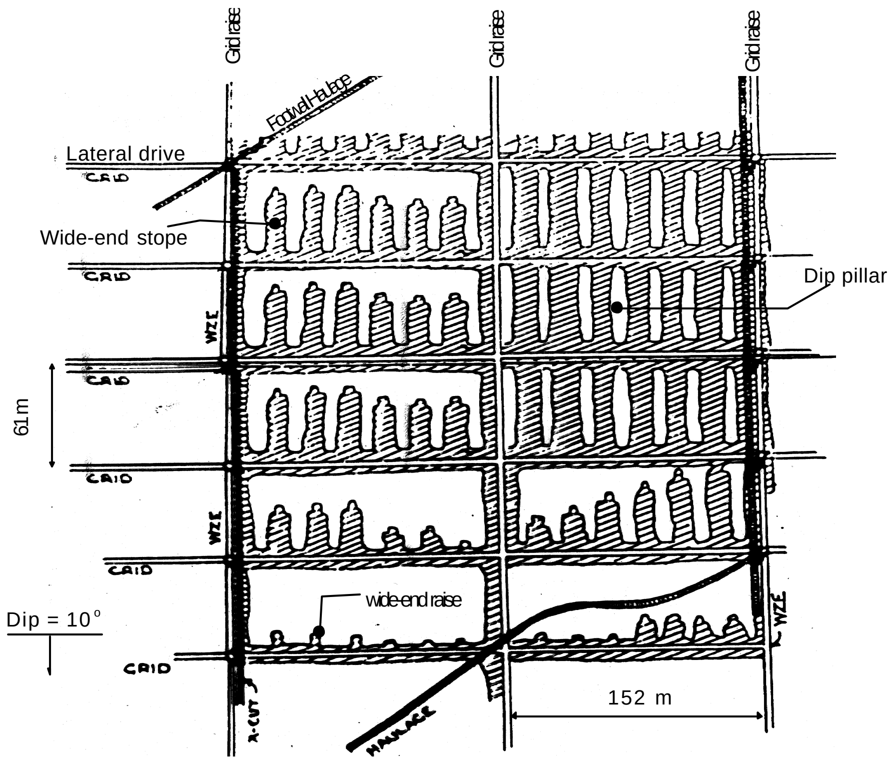


Figure 1.4 ***Grid*** **of primary development and reef development with stoping in progress in a** ***wide-end stoping*** **layout with dip stability pillars, applied by the Government Mining Areas (after Hindle, 1957). Note that mining proceeded up dip.**

The problem of subsidence became less relevant as the mining depth increased, and these early dip pillar type layouts were discontinued. In 1987 another dip orientated pillar system was introduced in a section of the Buffelsfontein Gold Mine that was experiencing acute seismic activity related to a major dyke. The scattered mining practised at Buffelsfontein was considered to be unsuitable for stoping the ground adjacent to this active structure, and a dip orientated pillar system was designed which provided sufficient regional support and also stabilised the dyke (Figure 1.5) by providing a system of buttress pillars.

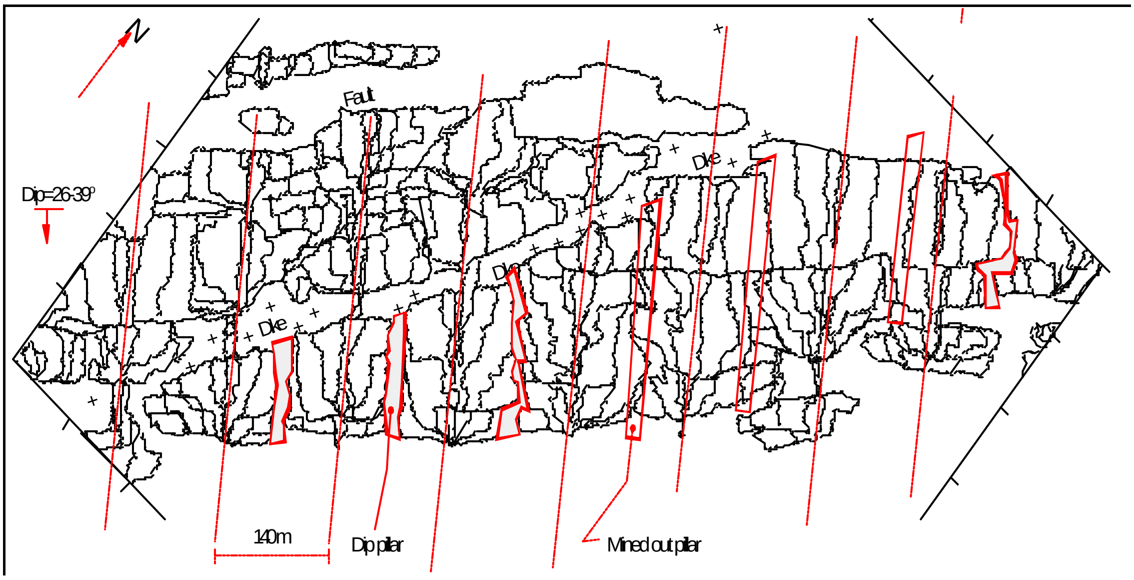


Figure 1.5 **Dip pillars in the Strathmore-South area (SMS), Buffelsfontein Gold Mine. Face outlines as of February 1998.**

A new type of dip pillar system was introduced at Elandsrand Gold Mine in the 1990's. The large number of dykes and faults and the erratic grade distribution led to the conclusion that longwall layouts were not optimal for mining of the ore body. A mining method was devised which utilised a regular grid of dip pillars for both regional support and to bracket most faults and dykes. A grid of footwall tunnels is developed prior to stoping. Haulages are typically placed between 70 and 80 m vertically below reef, and crosscuts to reef are developed at roughly 200 m intervals, followed by raises off each crosscut. These raises are ledged and equipped prior to the commencement of full stoping operations. This method, termed **sequential grid mining** (Applegate, 1997) was first implemented 2200 m below surface and has since been extended to a depth of 2800 m (Figure 1.6).

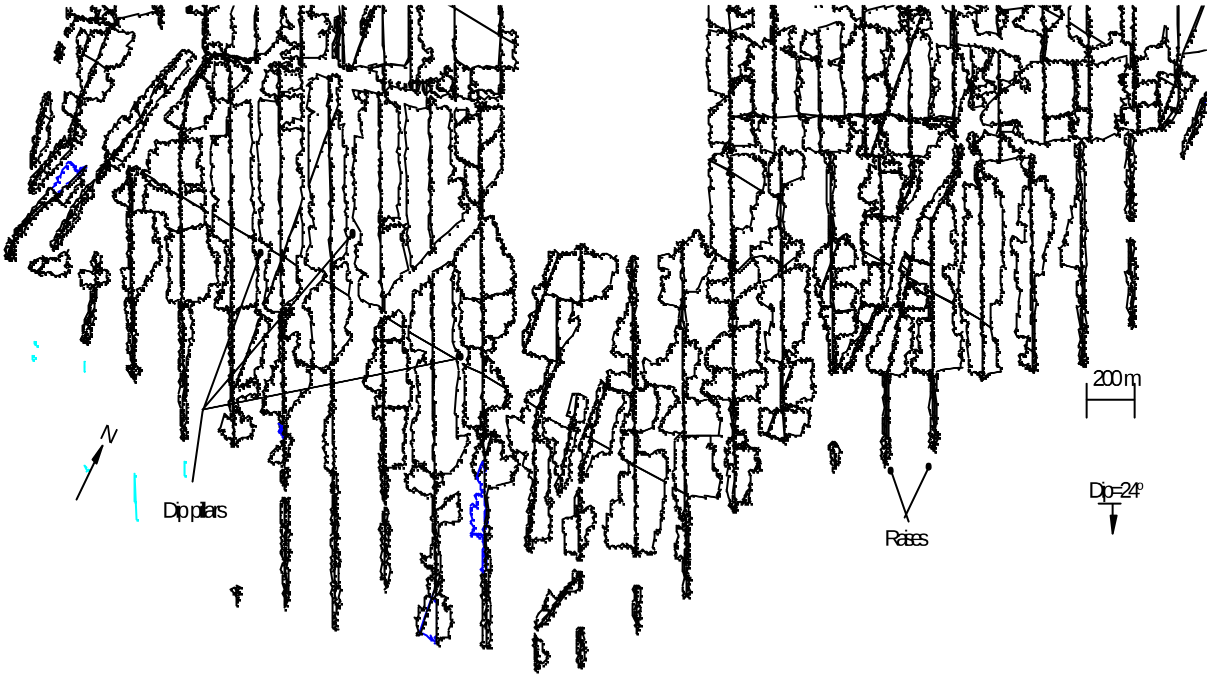


Figure 1.6 Macro plan of the sequential grid mining layout with dip stabilising pillars at Elandsrand. Face outlines as of March 1998 are shown.

Two new methods that employ closely-spaced narrow dip pillars to keep stope closure and energy release rates low have been implemented in the late-1990s. One method, employing down-dip mining (termed the **sequential down-dip** mining method), has been implemented at Kloof Gold Mine to extract a Ventersdorp Contact Reef orebody. Strike spans are small enough (about 75 m) to limit closure to the point where it is possible to dispense with backfill. The second method, employing breast mining (termed the **closely-spaced dip pillar** mining method), has been implemented at Driefontein Consolidated Gold Mine to extract a Carbon

Leader Reef ore body. This method allows the ore body to be accessed rapidly, as it is possible to commence stoping immediately the reef is intersected by a crosscut driven from a hangingwall access haulage.

1.3 MINING AT ULTRA-DEPTH

1.3.1 DEEPMINE Research Programme

The Witwatersrand Basin of South Africa has produced over 50,000 tons of gold since the discovery of the auriferous reefs in 1886. Annual gold production reached a peak of about 1,000 tons in 1970, but has since halved to less than 500 tons. The main reasons for this decline in production are a decrease in the gold price coupled with an increase in production costs, owing to, amongst other factors, the depletion of the shallower and high-grade reserves.

Reflection seismic imaging, deep drilling and the extrapolation of payshoots from areas of current mining indicate that the gold-bearing horizons persist to depths in excess of 5 km. If a favourable gold price is maintained, and challenges such as high virgin rock stress and temperature are overcome, it is predicted that the depths at which mining takes place will steadily increase. While only about 5% of production currently occurs below three kilometres, this could increase to over 40% by 2015 (Willis, 1997). It has been estimated that the gold resource that resides at these depths is similar to the amount of gold recovered from the reefs of the Witwatersrand basin during the last century. The great challenges of mining at ultra-depth has led to unprecedented co-operation between mining companies, research institutions, government, labour and universities and the establishment of the DEEPMINE Collaborative Research Programme in 1998 (Diering, 1997; Gürtunca, 1998).

The DEEPMINE Programme aims to create a technological and human resources platform that will make it possible to mine gold safely and profitably at depths of 3 to 5 km (the term “ultra-deep” is used for this depth range). For several decades the deep tabular reefs of the Central, West and Far West Rand were largely mined using layouts characterised by longwalls, with strike pillars providing regional support. During the past few years a new generation of layouts has been introduced, based on the concept of scattered mining with dip pillars for regional support. One of the most important issues to be addressed is the choice of mining method to be adopted at ultra-depth, and a study was undertaken to analyse and compare the performance of these mining methods.

1.3.2 Assessment of alternative layouts

A model of a typical ultra-deep Ventersdorp Contact Reef (VCR) orebody, derived from the interpretation of a 3D reflection seismic survey constrained by information from shallower mining and boreholes (Gibson et al, 1999), was used for this study. Several reflectors representing major stratigraphic breaks were mapped. Minor internal reflectors were also interpreted and together helped constrain fault geometry. Faults with throws greater than 18 m clearly offset the reflectors and can easily be mapped, while faults with throws less than 6 m cannot be detected. Faults with intermediate throws are identified using seismic attribute analysis. The orebody considered is 24 square kilometres in extent, ranging from 3800 m to 5000 m below surface, with an average dip of 23° (Figure 1.7a).

As the seismic survey could only detect faults with throws exceeding 6m, several smaller faults were added to the model. To give more realism to the geological model, and to test the ability of the different methods to mine difficult geologic conditions, a graben type structure was created by manipulating the displacement directions of two adjacent major faults. This model orebody with a total mineral resource area of 19.7×10^6 m^2 was named "Iponeleng", a Setswana word meaning "discover me" or "explore me".

This model (Figure 1.7a) was used to evaluate the four mining methods currently in use in an ultra-deep environment. A reef thickness of 1.5 m was assumed, typical of the Ventersdorp Contact Reef. These methods are: longwalling with strike stability pillars (LSP), sequential grid mining (SGM), and the sequential down-dip (SDD) and the closely-spaced dip pillar (CSDP) methods.

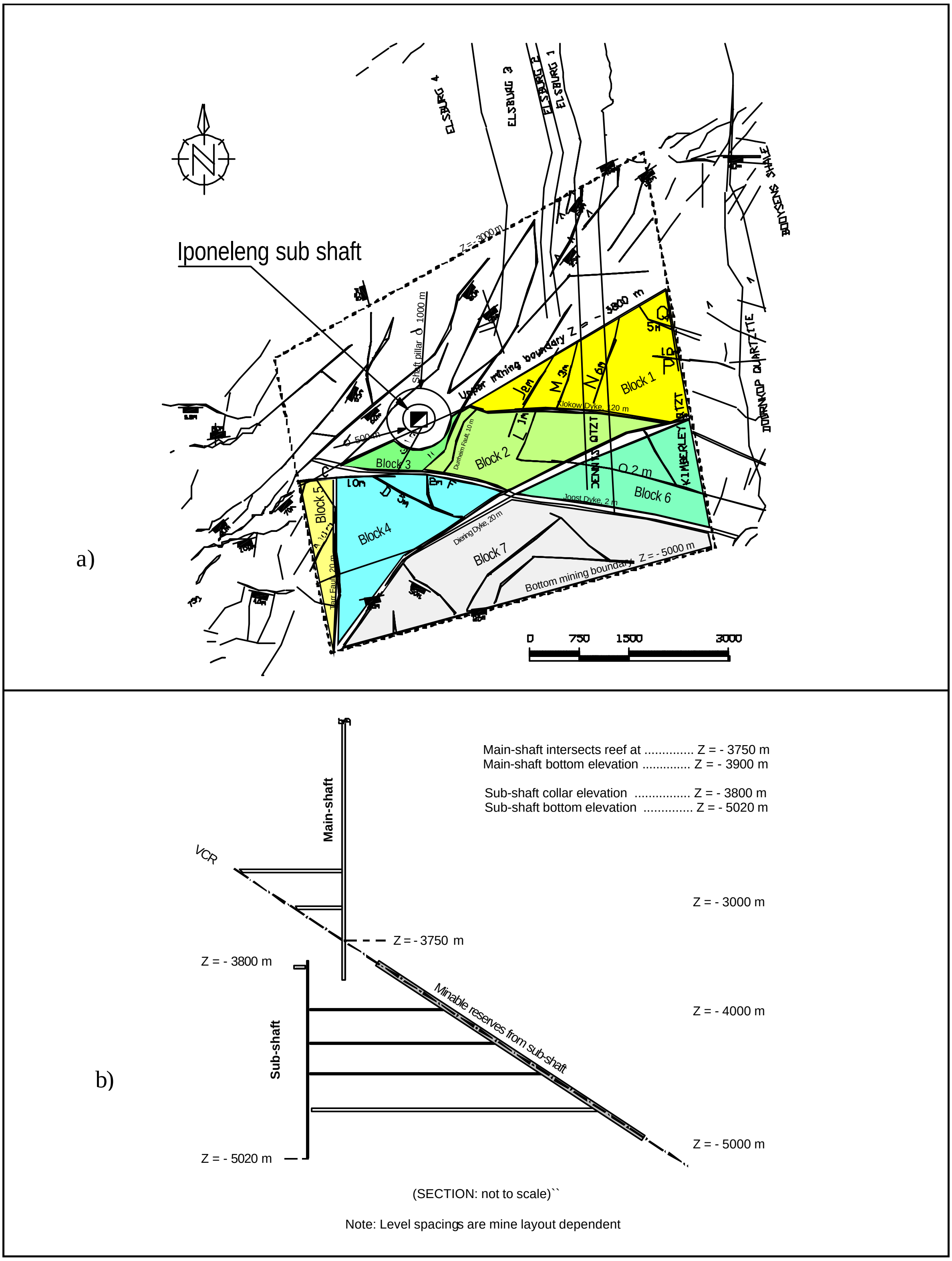


Figure 1.7 Iponeleng mine geological model with identified mining blocks (a) and vertical limits of the minable reserves (b).

Some operational assumptions were made and applied consistently to all four layout assessments (Table 1-2). Firstly access to the ultra-deep reserves is through an existing main-shaft and a newly sunk sub-shaft system, the latter being the infrastructure requiring new capital investment. The position for the sub-shaft infrastructure, assumed to be common for all mining methods, was selected on the following considerations (Figure 1.7b):

?? The shaft pillar is circular with a radius of 500 m, and considered to be mined out and with all station infrastructures completed.

?? The sub-shaft site should have as few geological disturbances as possible.

?? The main shaft intersects the reef at a depth of –3500 m. The full sub-shaft column would thus exist from a depth between –3500 m to –5020 m.

The mining rules, scheduling rates, performance criteria, and design specifications for each layout on both micro and macro scale were defined by teams of experts (Table 1.3). A **micro layout** consists of a plan and a section of a raise connection and its footwall infrastructure showing all the components of an operational raise connection and its service elements e.g. timber bays, travelling ways, boxholes, cooling loops and cubbies, ventholes, etc. This was to provide an understanding of how the layout actually works. The design parameters include level spacing, back length, crosscut spacing, pillar sizes, panel lengths, panel shape, mining direction, face orientation (overhand, underhand), leads-lags etc. A **macro layout** expands the micro mine design over four or five mining levels to enable the mine designers to determine whether or not a mine-wide layout is indeed practicable e.g. whether or not the desired air flow is actually possible, or whether the overall mining sequence is achievable for a realistic orebody such as the Iponeleng model.

Once the micro and macro concepts of the layouts and scheduling rates were defined, the mining of the ultra-deep entire Iponeleng orebody was simulated using the CADSmine package. A production target of 30,000 m^2 per month was initially set (equivalent to approximately 150,000 tons), which is similar to the output of some current deep level mines, and subsequently raised to 45,000 m^2 per month in view of preliminary economic considerations. The four conceptual mine designs were applied to arrive at an extraction path of the Iponeleng orebody and then subjected to exhaustive evaluation with respect to:

- Efficiency and cost of providing ventilation and refrigeration.
- Efficiency and cost of transporting men, material and rock.
- The degree to which rock engineering criteria such the energy release rates, average pillar stress, excess shear stress and stresses in footwall excavations were met.

It was assumed that the ultra-deep virgin stress regime is determined solely by the weight of overburden rock, and that the rock mass is an isotropic continuum. The virgin stress conditions at a point below surface are then defined by vertical σ_v and the horizontal σ_h components, given by Equation 1.2 and Equation 1.3 respectively,

$$\sigma_v = \rho h g \qquad \text{Equation 1.2}$$

$$\sigma_h = k\sigma_v \qquad \text{Equation 1.3}$$

where ρ is the rock density, h the distance below surface and g the acceleration due to gravity and k the ratio of horizontal to vertical stress. A *k-ratio* of 0.5 was used to model the rock of the Witwatersrand Basin. The virgin stress at the depths considered in the current analysis are indicated in Table 1.1. The Energy Release Rate (ERR) distribution ahead of mining faces was calculated using the MINSIM 3-D elastic boundary element method (Napier and Stephenson, 1987). ERRs ahead of mining faces were calculated assuming that no backfill had been placed.

Table 1.1. Theoretical virgin stress values in the far field at various ultra-deep elevations (assuming a rock density of 2780 kgm^{-3} and g = 9.81 ms^{-2})

Depth [m]	σ_v [MPa]	σ_h [MPa]
3000	82	41

4000	109	55
5000	136	68

1.3.3 Longwall with strike stabilising pillars (LSP)

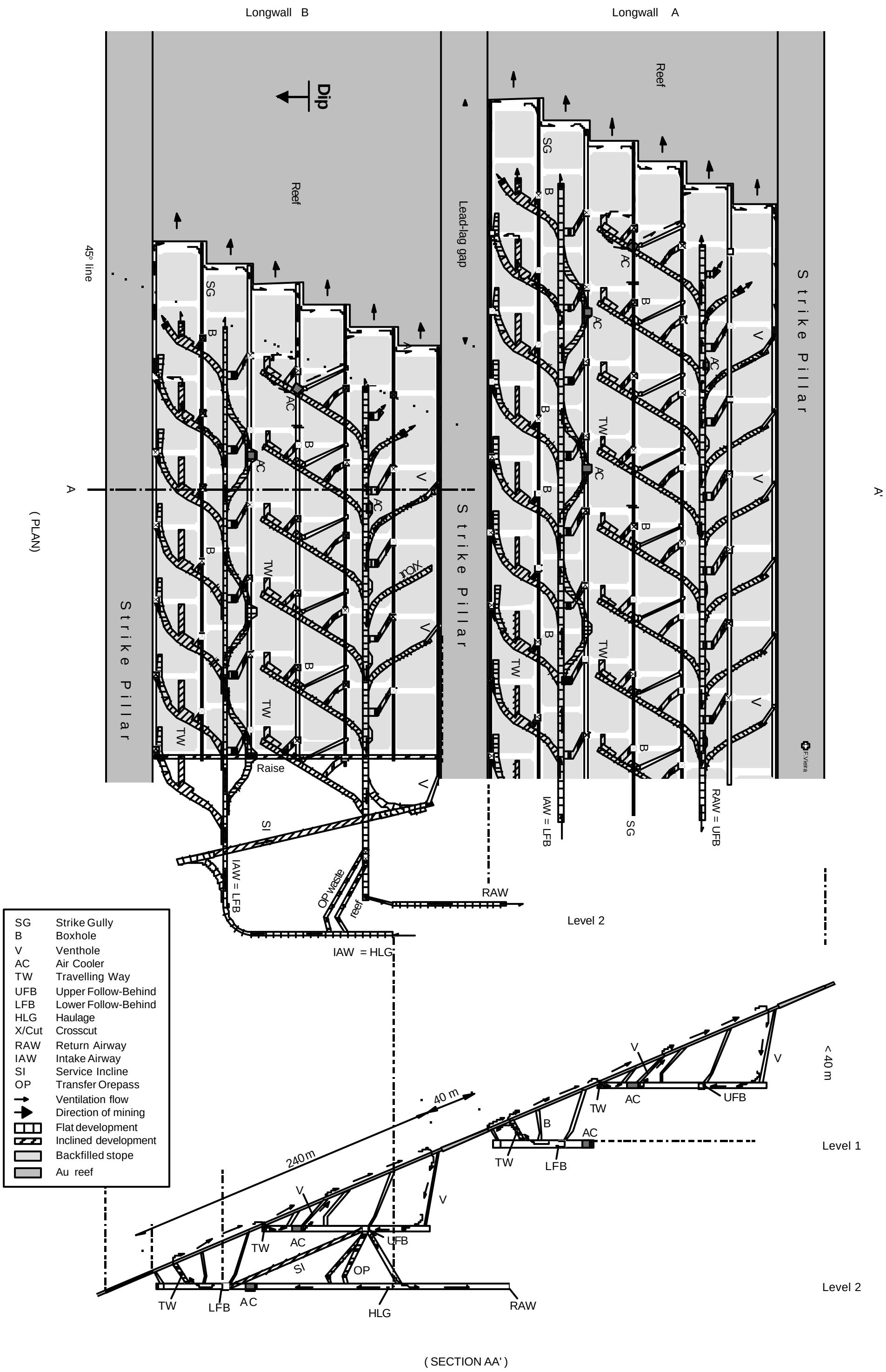


Figure 1.8 Micro concept of a longwall layout with strike stabilising pillars. Longwall A is depicted as a mature longwall production unit. Longwall B is depicted with all footwall infrastructure required for longwall establishment.

Micro layout

Twin haulages, one for intake air (IAW) and the other for return air (RAW), are driven from the sub-shaft to the block of ground to be mined (Figure 1-8). In order to establish a longwall, a single on-reef raise is developed between strike pillars, providing the platform for ledging and stoping. The panels are mined in the strike direction, with the lower panels leading, yielding an overhand configuration. A service incline (SI) is established from the level of the intake haulage to an intermediate elevation (see section of longwall B in Figure 1.8). Two flat and parallel tunnels are established from the flat connections of the service incline and are driven along strike beneath, and within 40 m of, the mined out reef plane. These are known as **follow-behind** (FB) tunnels as they lag sufficiently far behind the advancing faces to avoid the extreme abutment stresses. The lower follow-behind (LFB) is on the same level as the main access haulage (IAW or HLG). Crosscuts are established at 63 m intervals on either side of the follow-behind tunnels. Travelling-ways are established at the end of the reef crosscut to provide access to the stope. The broken rock is moved from the face area to the strike gully developed at the foot of each panel either by scraping or water-jetting, and then scraped along the gully to the nearest boxhole. From the boxholes it is transported by train to the main transfer orepass.

The lower and upper follow-behinds are used as intake and return airways to the stopes. Loops for bulk air-cooling are developed along the lower follow-behind after every second crosscut. It is necessary to re-cool air in both the lower and upper crosscuts as return air from the lower follow-behind headings is mixed with the intake air to stopes, and the return air from stopes is used for ventilating the upper follow-behind headings. Careful attention must be given to airflow at the top of a longwall to ensure that no methane accumulates. The top of the longwall is connected to the upper crosscut by a box- or venthole. The high virgin rock temperatures at ultra-depth will probably make in-stope cooling necessary.

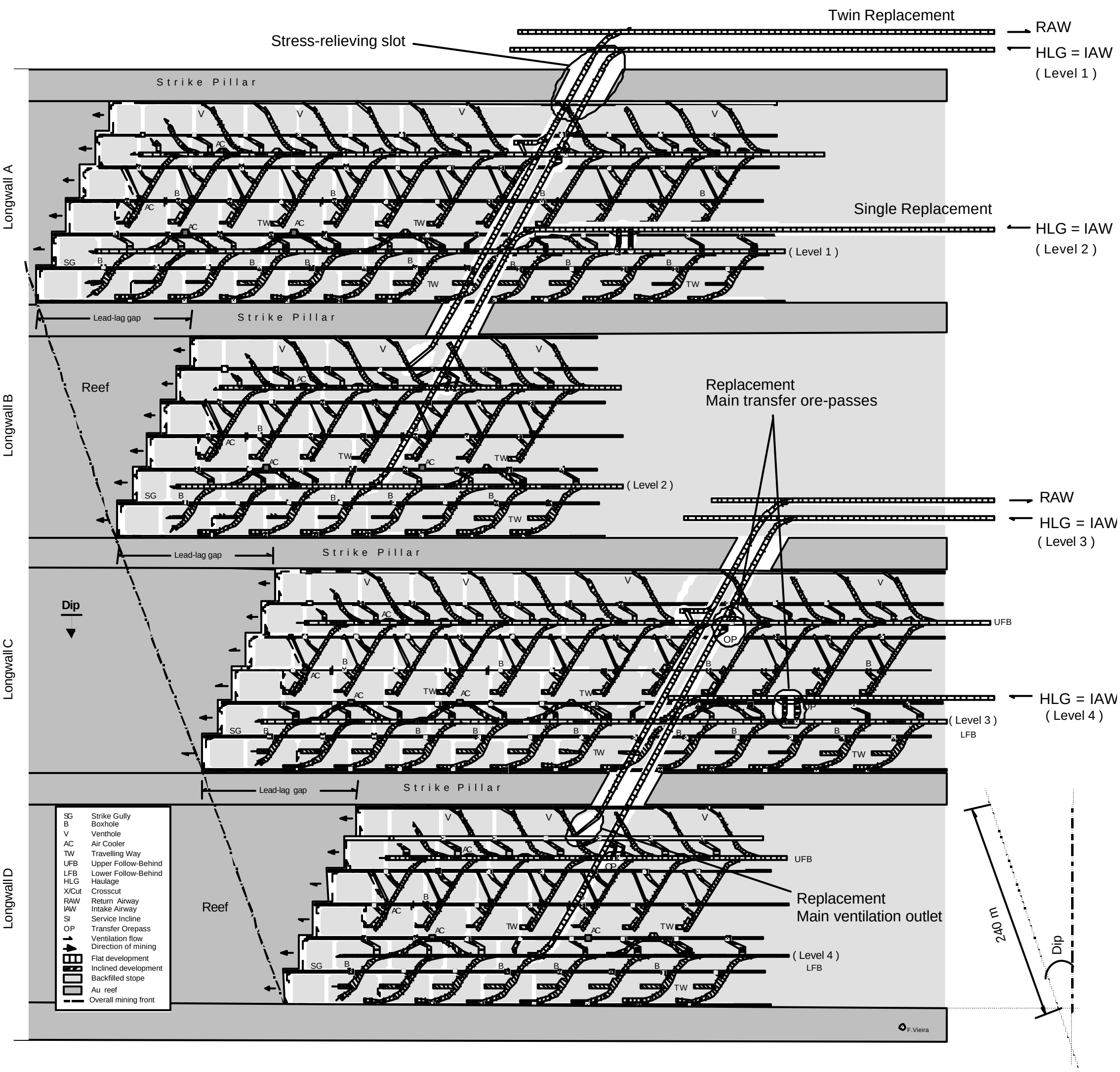
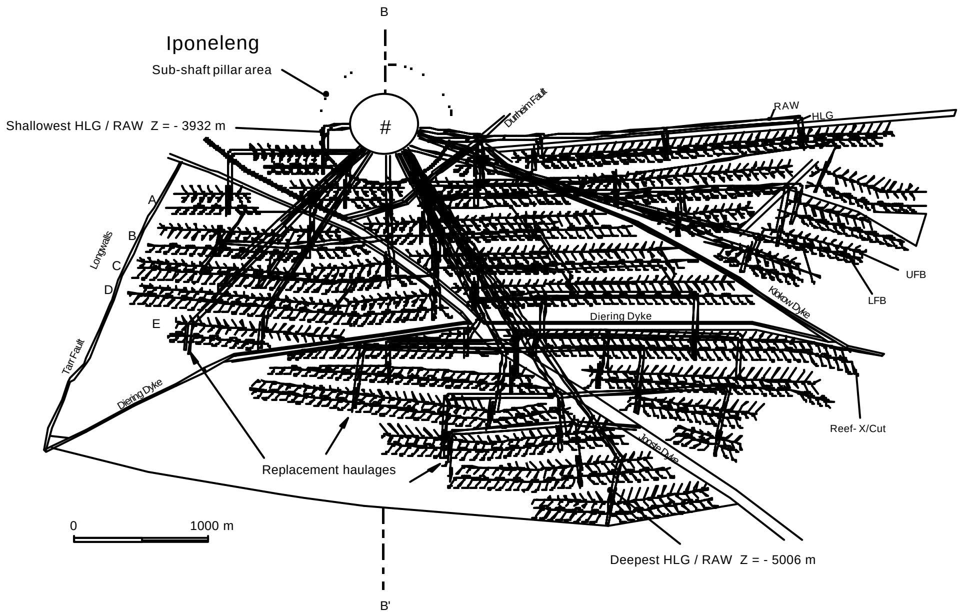


Figure 1.9 Macro concept of a longwall layout with strike stabilising pillars. Three longwall units are depicted. Replacement haulages are shown, these being developed roughly every 750 m.

Macro layout

The upper longwalls advance ahead of lower longwalls, yielding an overall underhand mining configuration (Figure 1-9). Twin replacement tunnels are typically placed 145 m below reef, largely beyond the influence of mining induced stresses. Slots are cut into the strike pillars to reduce the stress on the replacement tunnels passing beneath them. Each longwall unit is an independent ventilation district with its own intake and return airways, giving a fair amount of flexibility to the ventilation system as borehole connections can be used to adapt the system as and when required.

a)



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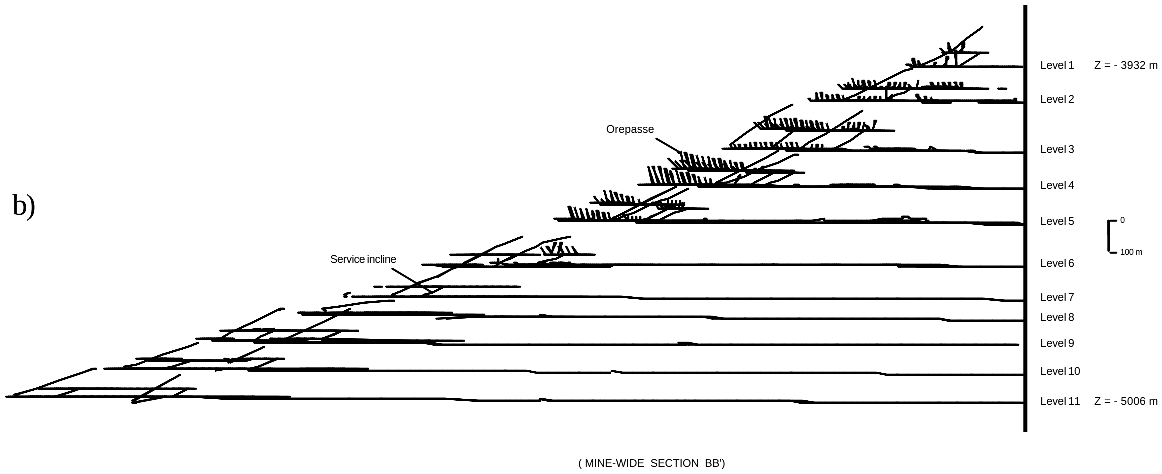


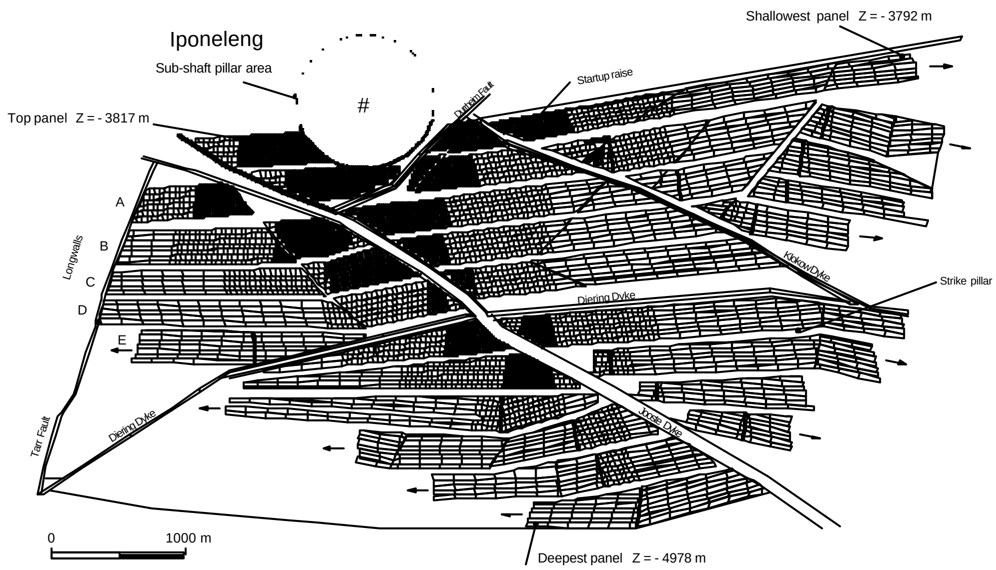
c) 

Figure 1.10 Application of the LSP micro and macro layout design concepts to the extraction of the Iponeleng orebody. Outlines represent Year 20 of 45 years of life of mine (LOM). (a) depicts the extent of the development infrastructure, (b) shows the number and relative position of service levels (c) the shows the regional stoping pattern.

Simulated mining of the Iponeleng model orebody

The orebody model in Figure 1.7 is first used to define viable longwall blocks. Note that it is assumed that the shaft and all service excavations within the shaft pillar are already established. The major decisions to be made by the design team are: the span between pillars, the number of panels (and hence the length of each panel), and the width of the strike stabilising pillars. The geometry of the longwall unit will also define the shaft station vertical interval. The design team specified a longwall layout comprised of six 40 m long panels between 3 and 4 km depth, and 35 m long panels between 4 and 5 km depth; 40 m wide strike stabilising pillars between 3 and 4 km depth, and 50 m wide between 4 and 5 km depth (Table 1.3 and Figure 1.8). This design gives a back length of about 240 m. The Energy Release Rate (ERR) ahead of the mining faces is a function of the local geometry of the longwall and depth. At 3000 m depth the probability for the ERR to exceed 30 MJ/m^2 on any face is 0.2, while at 5000 m the probability is 0.6, indicating in an increase in the potential for rockbursting. The placement of backfill is considered essential to reduce the ERR values. Furthermore, backfill assists in ventilation control and also reduces the potential heat load originating from worked-out areas. The average pillar stress (APS) on the reef plane ranges from 450 MPa at 3000 m to 695 MPa at 5000 m. At great depth the APS exceeds 2.5 times the UCS of the host rock, and the stability of the pillar may be compromised. Tunnels passing near or under pillars or abutments experience stresses between 140 MPa at 3000 m and 260 MPa at 5000 m.

Eleven levels are required to extract the model orebody. The life of mine is expected to be 45 years, in which time a total area of 13.1×10^6 m^2 out of a 19.7×10^6 m^2 resource is mined, yielding an overall extraction ratio 67 per cent, with 4.1×10^6 m^2 being left locked up in stablising pillars and ground adjacent to geological features. The total length of off-reef development is 552 km.

Evaluation

The longwall method works well where the grade is consistent, and dykes and faults are sparse. Ventilation is also easy to manage, very little re-conditioning of the ventilation air is necessary, and only an average amount of ventilation air is required.

There are several disadvantages to the longwall method. Firstly, the servicing of an advancing longwall demands the development of a substantial infrastructure (follow-behind tunnels, **crosscut**s, loops and cubbies for air coolers, timber bays, boxholes, etc) that takes a significant time and cost to establish. Furthermore, follow-behind tunnels must lag behind the advancing face to ensure that they are situated in stress-relieved ground, and the maximum practical pulling distance for a scraper from face to boxhole is considered to be 100 m. These considerations, if current technology is applied, limit the stoping face advance rate in a longwall unit to 12 m/month/face. Secondly, as the longwall method cannot mine selectively, much low-grade ore may be mined in areas of erratic grade distribution. Lastly, it is a major effort to establish a new longwall, with the result that it is often necessary to mine through potentially hazardous structures such as dykes and faults. If there is a significant throw on the fault, substantial time and effort must be spent to re-establish the stope on the reef horizon.

1.3.4 Sequential grid method (SGM)

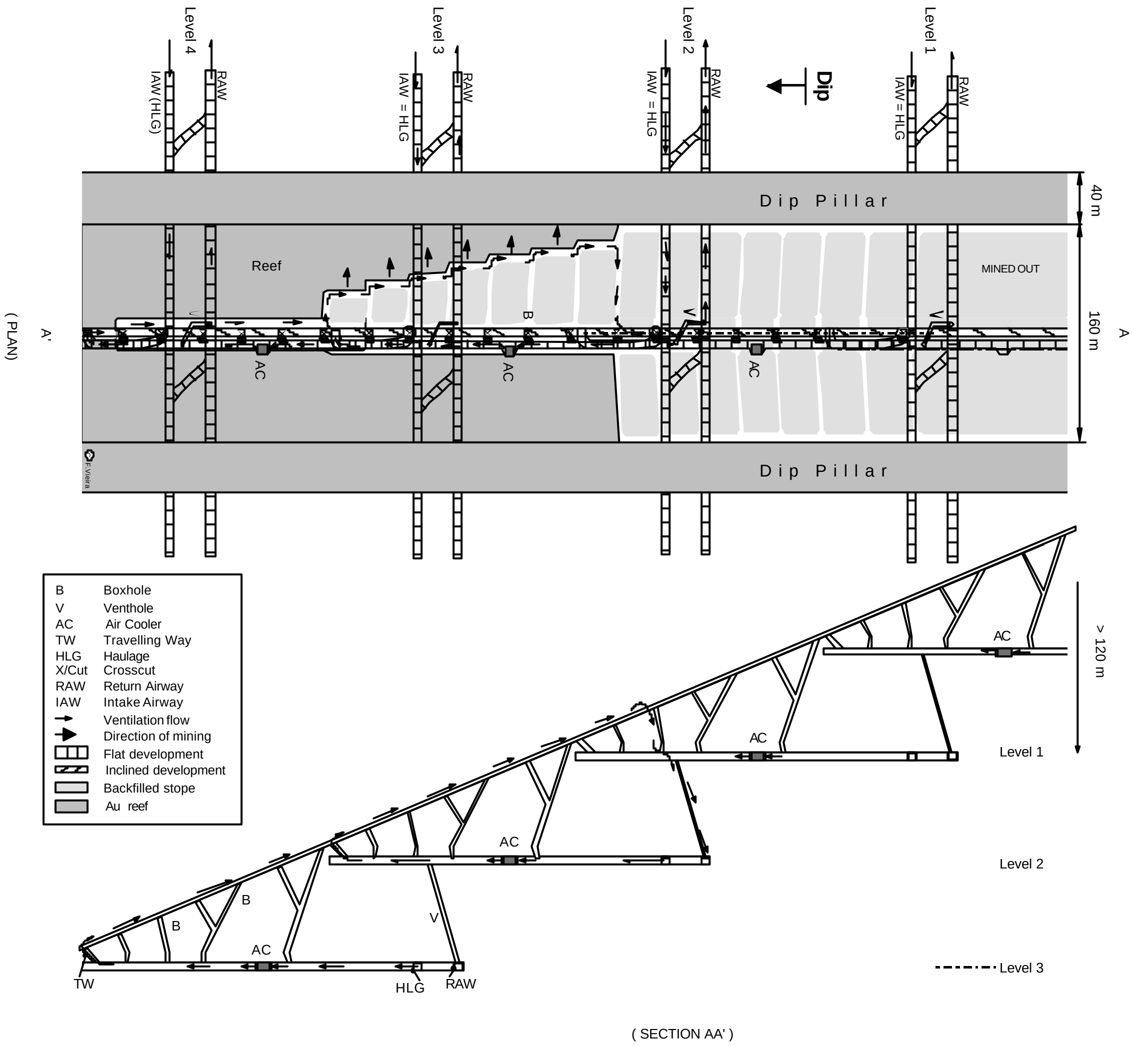


Figure 1.11 Micro concept of a sequential grid mining layout with dip orientated pillars

<u>Micro layout</u>

As for the longwall method, twin haulages are driven from the shaft to access the block of ground to be mined (Figure 1-11). These haulages, as well as the reef-access crosscuts, are developed ahead of stoping, enabling faults and dykes to be located prior to stoping, and pillars to be designed to bracket any hazardous structures. Haulages are situated deep in the footwall (about 120 m) in order to reduce their vulnerability to the strain and stress changes associated with stoping.

Crosscuts are developed at regular intervals to give access to the orebody, and the raises are developed, ledged and equipped. Mining is carried out according to a strict sequence to ensure low energy release rates and pillar stresses (Applegate, 1997). Panels are first mined, in an underhand configuration, from the raise line towards the planned pillar position closest to the shaft, and only once this is complete is the other half of the stope area mined. This is termed “single side mining”, to distinguish it from similar methods where mining may take place concurrently on both sides of the raise. The broken rock is moved from the face area to the strike gully at the foot of the panel either by scraping or water-jetting, and then scraped along the gully to the nearest boxhole. One of the most demanding aspects of the sequential grid method is the development of the ore passes servicing the uppermost part of the stope, as the vertical distance from crosscut to reef may be as great as 100 m, depending on the dip of the reef and the back length.

The micro ventilation system is quite simple, with a dedicated intake at the bottom of the stope, a maximum of seven panels, and a return airflow through a ventilation boxhole at the top of the stope to the lower level return airway. Ventilation control is relatively easy due to the required use of backfill and a maximum airflow strike distance of only 80 m (i.e. single-side distance). Although backfill will reduce the heat load from worked-out areas, **in-stope cooling** will still be required at ultra-depth, which might complicate the control of ventilation air in stopes. The SGM layout requires ventilation air to be returned from the top of a stope to the lower level return airway via a disused boxhole, which may be a restriction in the ventilation system, and care must be taken to avoid the accumulation of methane in the top section of the stope.

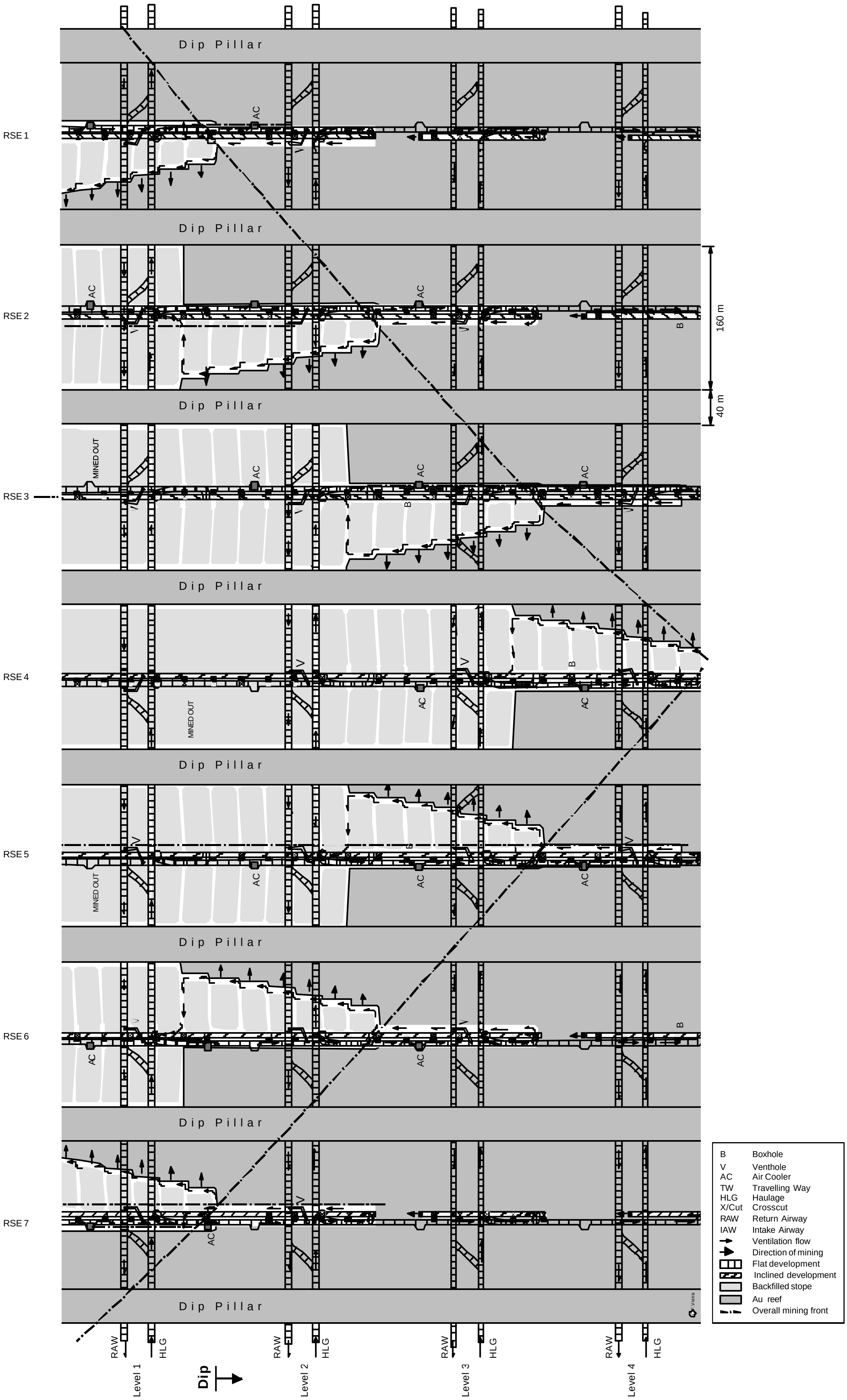
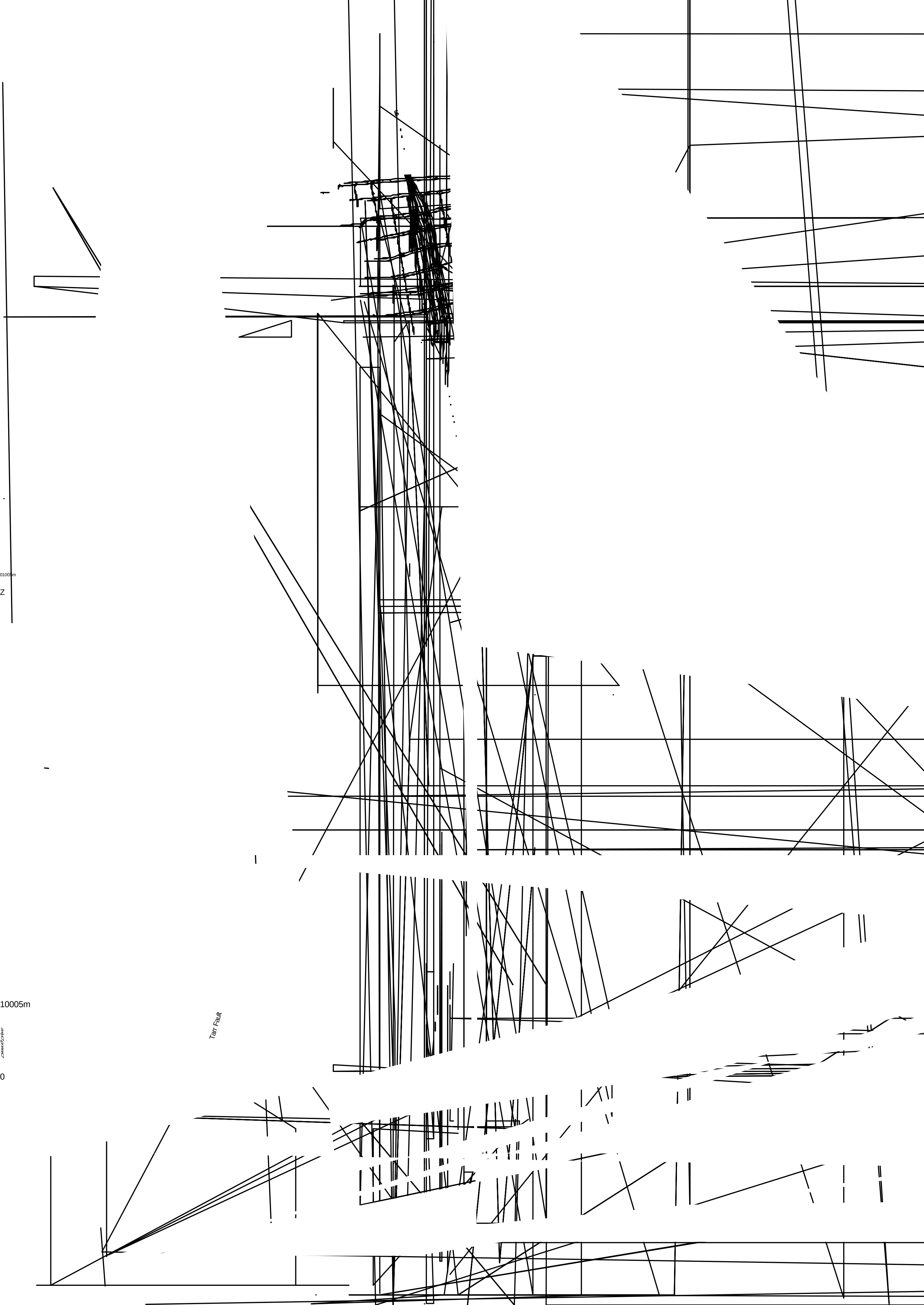


Figure 1.12 Macro concept of a sequential grid mining layout with dip stabilising pillars.

Macro layout

Stoping may take place concurrently on two adjacent raise lines, while the next raise is being ledged, and a fourth raise being developed or equipped (Figure 1-12). If stoping is taking place in two adjacent raise lines, it is important that the faces where mining is taking place concurrently are never closer than 70 m to avoid abnormal loading of pillars.

The primary ventilation system consists of twin airways (intake and return) on each level connected to each stope through **crosscut**s, box- or ventholes. Bulk air-cooling is required at the top of the sub-shaft to ensure acceptable conditions at the stations and sealed-off HLG-RAW connecting crosscuts are used as cubbies for closed-circuit coolers in the intake airways, as and when required.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extent of the development infrastructure, (b) shows the number and relative position of service levels (c) the shows the regional stoping pattern.

Simulated mining of the Iponeleng model orebody

The orebody model in Figure 1.7 is first used to define viable mining blocks for this method. The major decisions to be made by the design team are: the span between pillars (which dictates crosscut spacing), the width of the dip stabilising pillars, the length of back (which dictates the position of haulages), and the depth of the main haulages beneath reef. The design team specified a 200 m crosscut spacing; seven panels per raise, each with a face length of 30 to 40 m, yielding a back length of about 240 m; stabilising pillars 40 m wide between 3 and 4 km depth, and 50 m wide between 4 and 5 km depth; and main haulages at least 120 m below the reef plane (Table 1.3 and Figure 1.11). It was calculated that there is a probability of 0.1 for the ERR ahead of any stope face to exceed 30 MJ/m^2 at a depth of 3000 m, increasing to 0.35 at 5000 m. The placement of backfill is considered essential to reduce the ERR values ahead of working faces. The footwall tunnels were placed 146 m below reef, and all pass beneath dip pillars where the near-field stress ranges from 93 MPa at 3000 m to 170 MPa at 5000 m. The surface stress concentrations on the tunnel surface are higher (Figure 1.14). The average pillar stress on the reef plane ranges from 320 MPa at 3000 m to 520 MPa at 5000 m.

Twelve levels were required to extract Iponeleng ore body, and the life of mine is expected to be 31 years. A total area of 11.7×10^6 m^2 of the 19.7×10^6 m^2 resource is mined during this period, yielding an overall extraction ratio 59 per cent, with 7.8×10^6 m^2 being left locked up in pillars and ground adjacent to geological features. The total length of off-reef development required is 290 km.

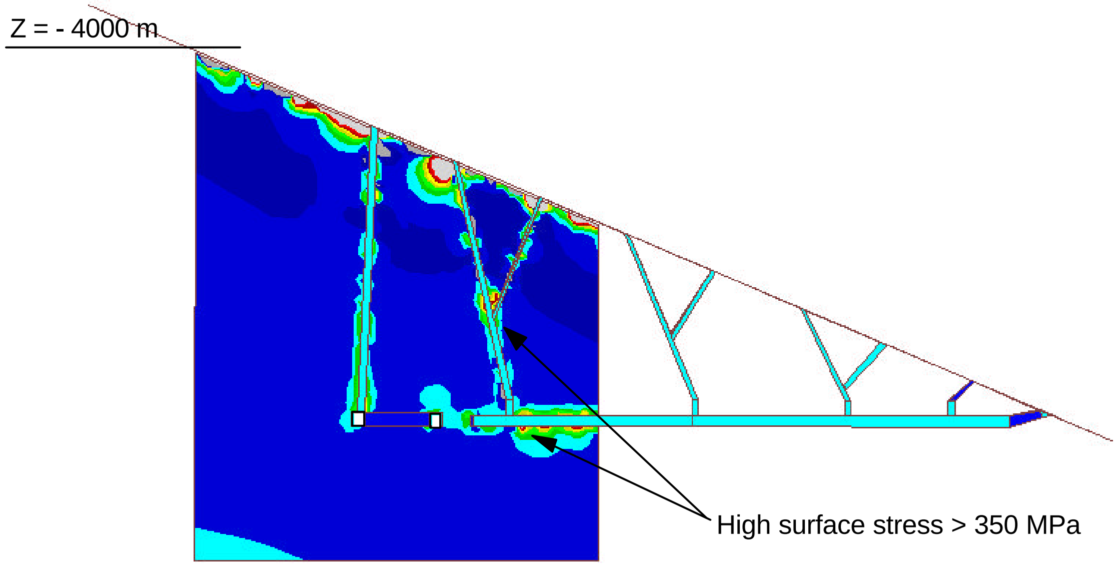


Figure 1.14. Stress environment in footwall excavation surfaces when placed underneath a SGM raise at a depth of 4000 m.

<u>Evaluation</u>

The main advantage of the sequential grid method is that the development of tunnels ahead of stoping gives provides foreknowledge of potentially hazardous structures, and enables them to be incorporated into stabilizing pillars. It is also possible to practice selective mining, leaving behind areas of low grade ore. One major disadvantage of the layout is that the service excavations connecting the crosscut and reef are subject to large stress changes, which may compromise their stability. Changes as great as from 190 MPa to 230 MPa are possible at a depth of 4000 m.

The sequential grid method requires a relatively high volume of ventilation air. This system requires fairly long ventilation holes, usually boxholes or boreholes, from the top of each stope or from the crosscut on the upper level to the return airway at the intake level (Figure 1.11). The ventilation system is, however, flexible and relatively easy to manage.

1.3.5 Sequential down dip method (SDD)

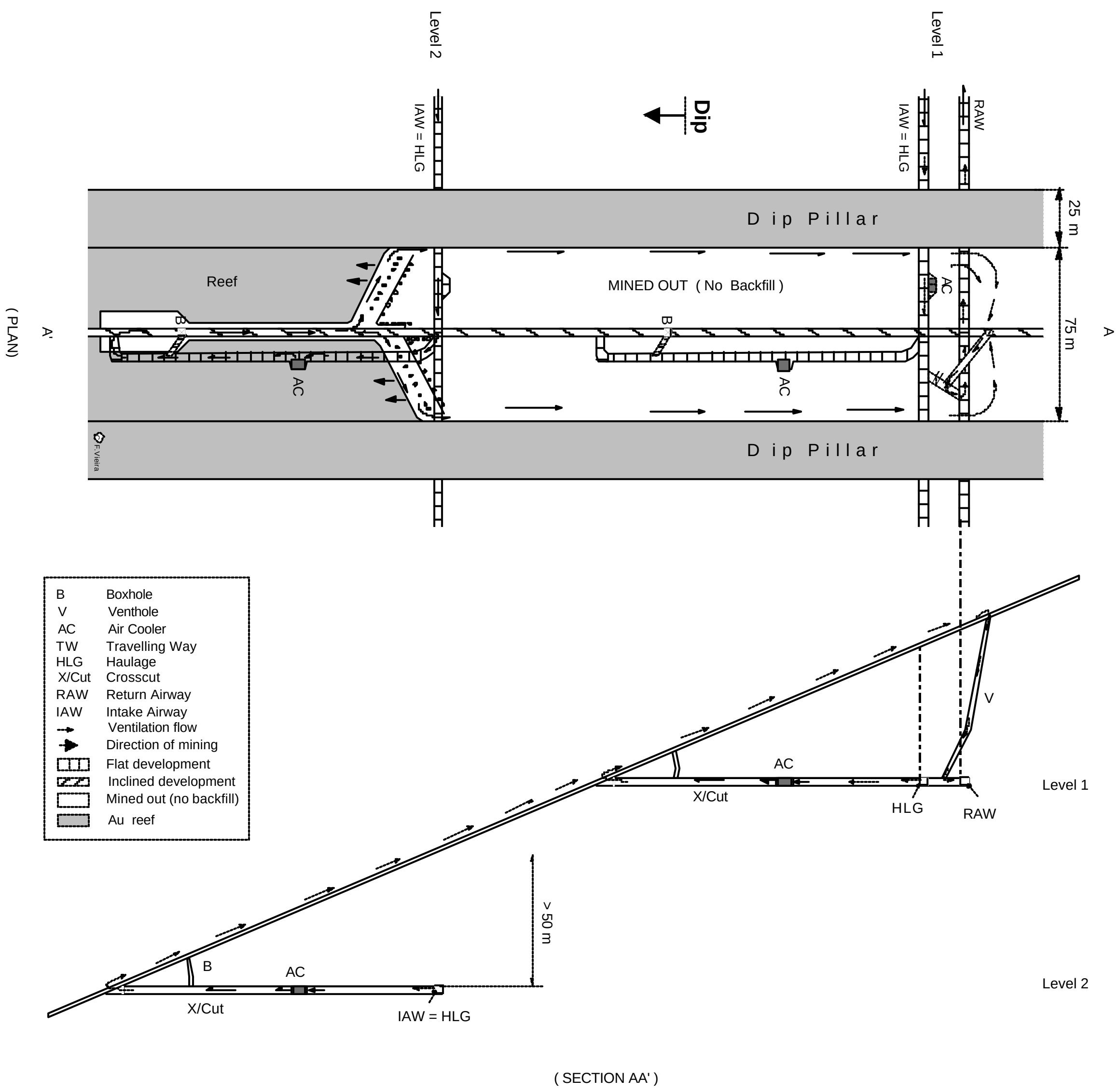


Figure 1.15 Micro concept of a sequential down-dip mining layout with dip orientated pillars.

<u>Micro layout</u>

Haulages are driven on each level from the shaft to the block of ground to be mined (Figure 1-15). Alternate levels have both intake and return airway tunnels, while other levels have only intake airways. These haulages are developed ahead of stoping, enabling faults and dykes to be located prior to stoping, and pillars to be designed to bracket any hazardous structures.

Crosscuts are driven until reef is intersected at the foot of the planned stope. Reef drives are developed for 10 m along strike on either side of the raise line. A 20 m wide raise is mined updip, overstoping the area above the timber bay and cooling car bay. Once the raise is 6 m beyond the box hole, the span of the wide raise is reduced to 12 m. The 240 m long wide raise is mined at a rate of 20 m/month, until the crosscut on the level above is reached. A further 2 months is allowed for services such as winches to be installed before down-dip stoping begins. Two panels, one on each side of the raise and each with a face length 46 m, are mined down dip at a face advance rate of 15 m/month/face. The broken rock is moved from the face area to the wide raise either by scraping or water-jetting. It is then scraped to the boxhole at the foot of the raise. Men and materials enter the stope from the upper crosscut.

The micro ventilation system is fairly simple. It consists of a dedicated intake at the bottom of the stope through a 12 m wide raise, a 46 m long down-dip panel to each side of the raise, and a return through the worked-out top section of the stope to the upper level return airway. Ventilation control is somewhat difficult as faces are underhand and no backfill is used, making it difficult to channel the airflow close to the face. The build-up of pollutants in SDD stopes is not a major problem.

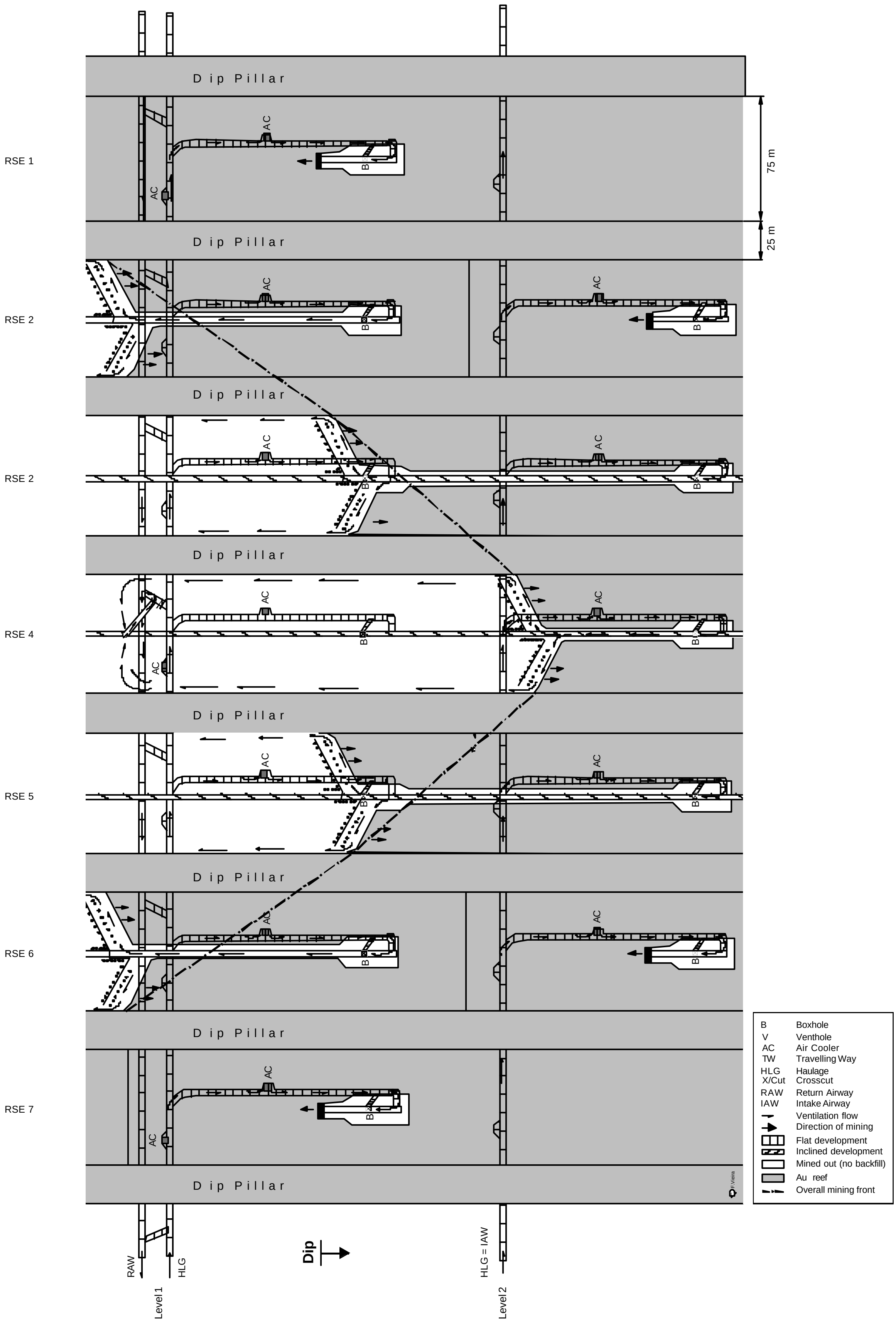


Figure 1.16 Macro concept of a sequential down dip mining layout with dip stabilising pillars.

Macro layout

A macro layout is shown in Figure 1.16. The mining rules do not permit stoping to take place concurrently on the same level on adjacent raise lines. While one raise line is being stoped, the next crosscut is developed. As it cannot be assumed that the orebody is planar, development of the haulage beyond the crosscut breakaway is halted until a prospecting borehole has been drilled and the reef located, so that the precise position of the next crosscut breakaway is established. This process limits the rate at which haulages are developed, and new raise lines opened. The optimum global mining configuration is underhand, with stoping starting first on the upper levels, and expanding either side of the centre of the mining block.

The macro ventilation system is largely dictated by the production requirements. The airway system is fairly rigid and is relatively difficult to manage.

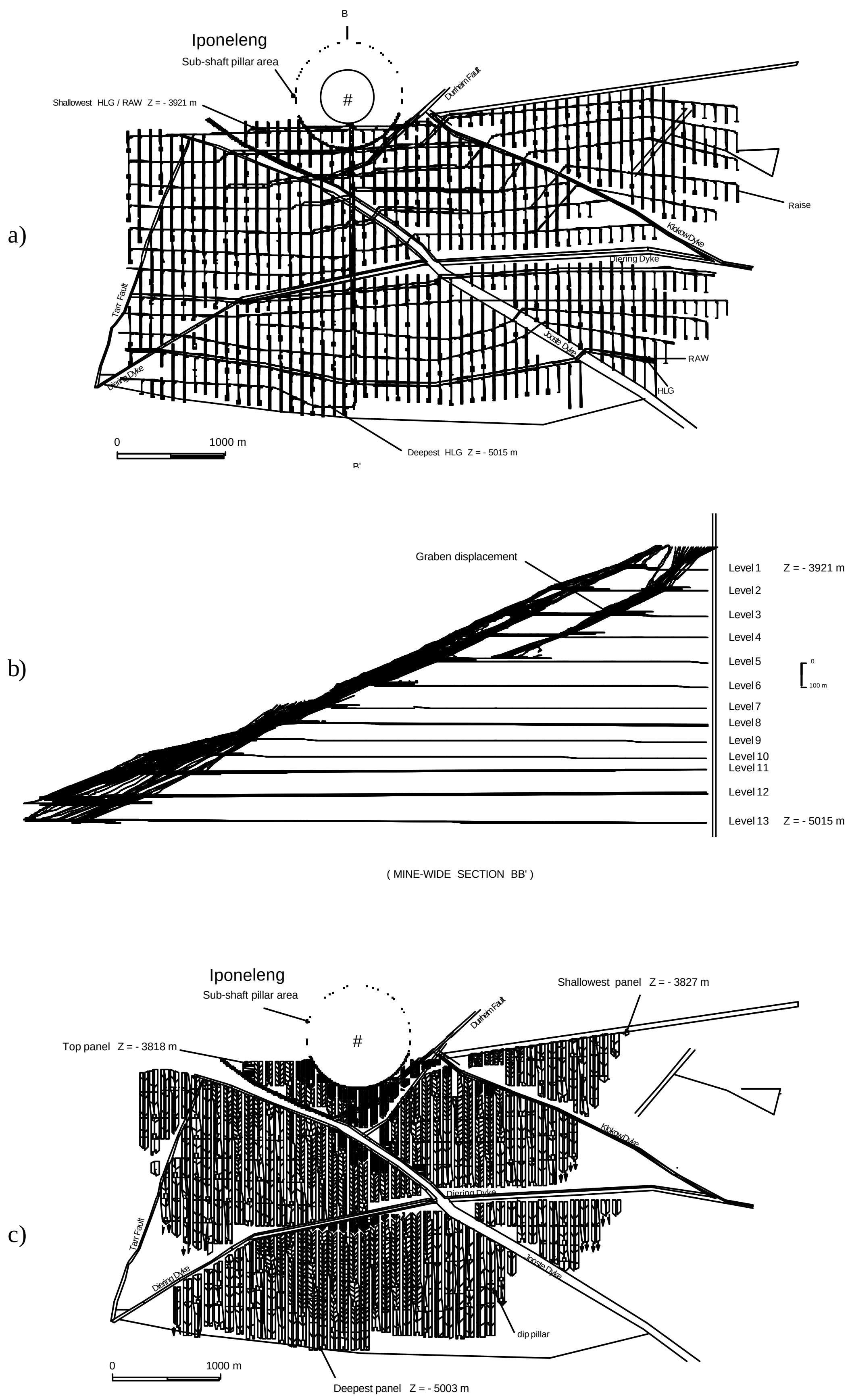


Figure 1.17 Application of the SDD micro and macro layout design concepts to the extraction of the Iponeleng orebody at Year 20 of 37 years of LOM. (a) depicts the

extent of the development infrastructure, (b) shows the number and relative position of service levels (c) the shows the regional stoping pattern.

Simulated mining of the Iponeleng model orebody

The orebody model is first used to define viable mining blocks. The major decisions to be made by the design team are similar to those made for a sequential grid layout: the haulage and crosscut spacing, the width of the dip stabilising pillars, and the depth of main haulages beneath reef. The design team specified a 100 m crosscut spacing; haulages about 220 m apart yielding a back length of about 240 m; 25 m wide dip stabilising pillars; and main haulages at least 50 m below the reef plane (Table 1.3 and Figure 1.15). This layout ensures that ERR will not exceed 30 MJ/m^2. The maximum ERR value at 3000 m depth is 15 MJ/m^2, and only 25 MJ/m^2 at 5000 m. Due to the narrowness of the mined out spans, very low closure rates occur and only increase marginally with depth. It is unlikely that total closure will occur during the phase of mining, unless abnormal strata conditions exist. As a result, backfill is considered to be unnecessary. Haulage tunnels have been placed about 88 m below reef, and may experience some stress changes due to mining. The change in tunnel stresses range from 50 MPa at 3000 m to 91 MPa at 5000 m. While the layout gives rise to significant stress reversals in the footwall rock mass, little infrastructure is affected (Figure 1.15). The average pillar stress on the reef plane ranges from 310 MPa at 3000 m to 510 MPa at 5000 m.

Thirteen levels are required to extract Iponeleng orebody when applying the SDD method. (Figure 1.17). The life of mine planned using this method is expected to be 31 years. During this period a total area of $11.4x10^6$ m^2 of the $19.7x10^6$ m^2 resource is mined, with $4.7x10^6$ m^2 being left locked up in pillars and ground adjacent to geological features, yielding an overall extraction ratio 58 per cent. The total length of off-reef development required is 344 km.

Evaluation

The main advantages of the sequential down dip method are the very low energy release rates, which make backfilling unnecessary, and the allowance for the physical separation of rock transport from men and materials. However, this has serious implications for ventilation: very large volumes of air are required as it is difficult to keep the airflow close to the face in the absence of backfill; and the return airways have to be cooled as they are used as travelling ways and the air exiting the stope would have gained heat while passing through the worked out areas. One of the major disadvantages of this method is the dependency on one single short orepass with limited storage, which may constrain production output.

1.3.6 Closely-spaced dip pillar method

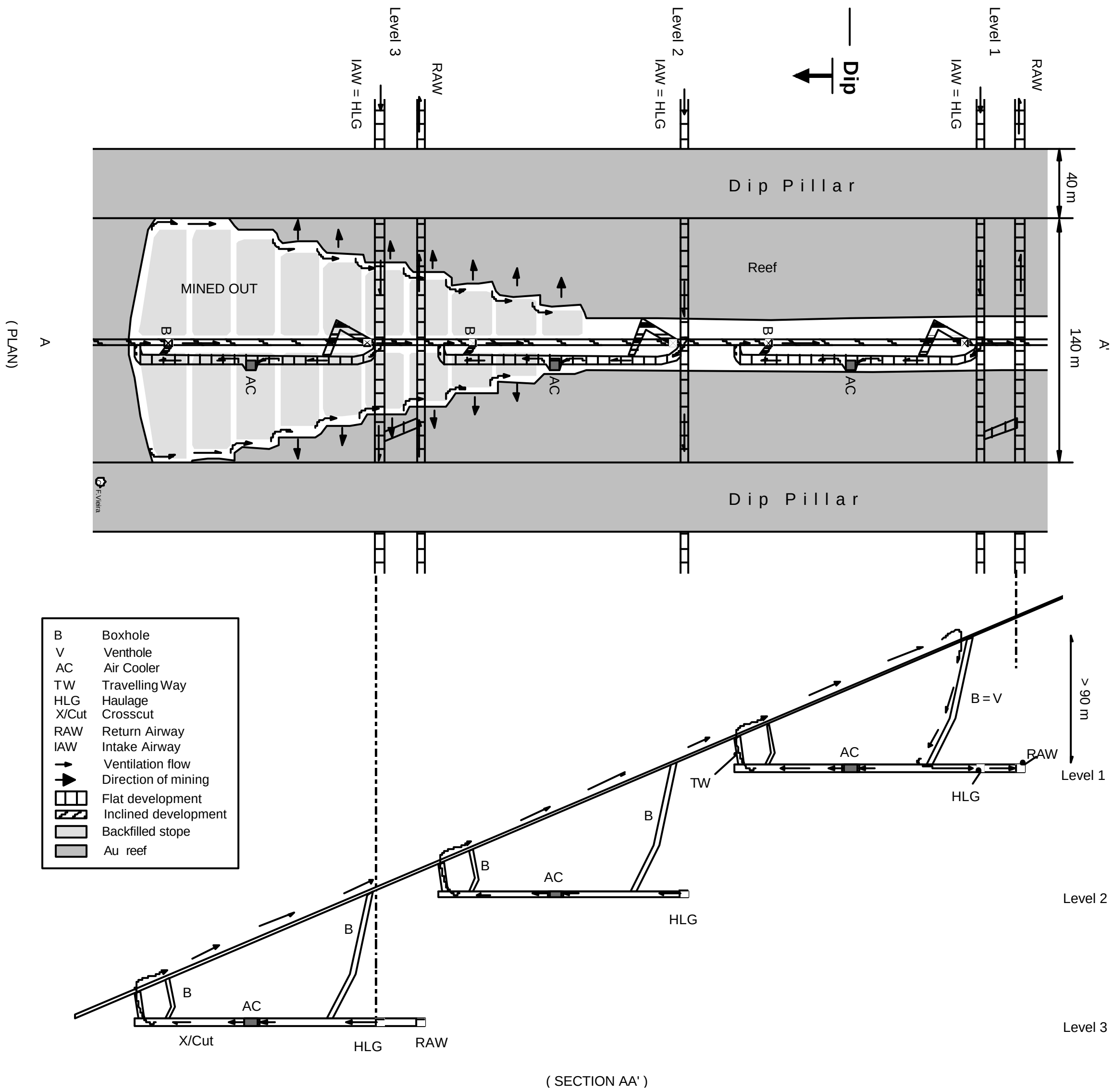


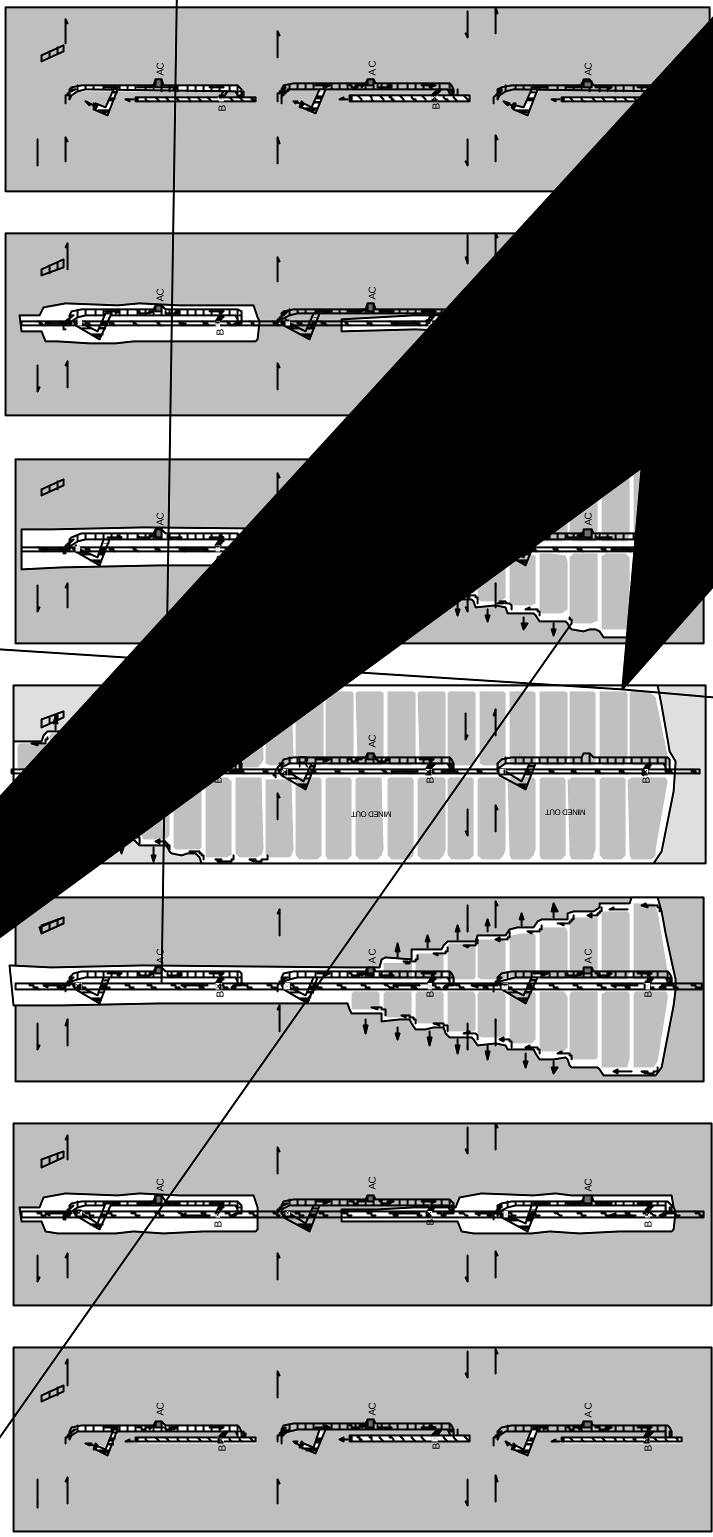
Figure 1.18 Micro concept of a closely-spaced dip pillar mining layout.

<u>Micro layout</u>

As for the preceding methods, haulages are driven from the shaft to the block of ground to be mined. Haulages used as intake airways are developed on every level, with a twin return airway on every second level (Figure 1-18). All footwall tunnels are developed ahead of stoping, enabling faults and dykes to be located prior to stoping, and pillars to be designed to bracket any hazardous structures. Crosscuts are driven until 12 m below reef, with a bay for timber and cooling cars on the side of the crosscut. A travelling way to the reef horizon is excavated, and a 20 m wide raise developed in a down-dip direction. Only two orepasses service the entire back: the boxhole near the foot is only used during the excavation of the raise, while the long orepass at the top of the raise is the passage for all the ore once production begins. Stoping only commences once the entire raise line has been ledged, supported and equipped. Panels 25 m in length are mined on breast towards the planned position of the dip stabilising pillar at a face advance rate of 15 m/month/face. Panels on either side of the raise are mined simultaneously (i.e. double-side mining), commencing at the bottom of the raise. The start of the adjacent up-dip panel is delayed until a 5 m lag has been established, yielding an overall overhand configuration. The broken rock is moved from the face area to the raise either by scraping or water-jetting, and then moved to the boxhole at the top of the raise by means of a continuous scraper pulling up-dip (Figure 1-19). This method is the only one that considers some degree of continuous mechanisation in the cleaning operations.

The ventilation system has a dedicated intake at the bottom of the stope, a maximum of 24 overhand panels, and ventilation air return is through a 10 m wide ledged raise to the next return airway. Ventilation control is not difficult due to the use of backfill, a maximum strike distance of 70 m, and the overhand face orientation. Although backfill will reduce the heat load from worked-out areas, in-stope cooling will still be required at ultra-depth, which might complicate the control of ventilation air within the stopes. The build-up of pollutants in CSDP stopes could occur due to long residence time of the ventilation air, but this is not seen as a major problem.

Figure 1.19 Up-dip continuous scraping facility currently on trial in a deep CSDP stope.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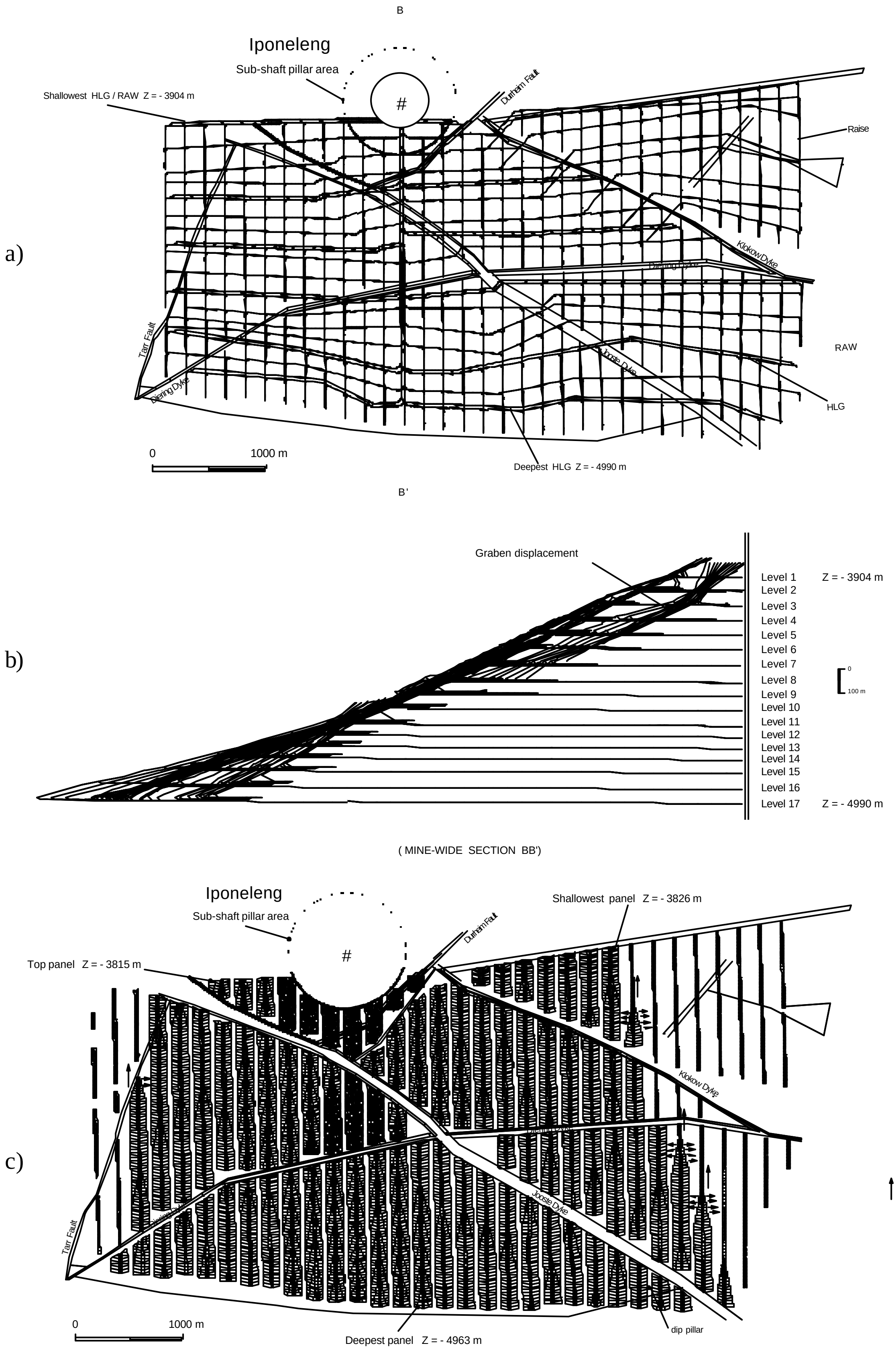

Figure 1.21 Application of the CSDP micro and macro layout design concepts to the extraction of the Iponeleng orebody at Year 20 of 26 years of LOM. (a) depicts the

extent of the development infrastructure, (b) shows the number and relative position of service levels (c) the shows the regional stoping pattern.

Simulated mining of the Iponeleng model orebody

As for all the other methods, the orebody model (Figure 1.7) is first used to define viable mining blocks. The major decisions to be made by the design team are similar to those made for the sequential grid and sequential down dip methods: the haulage and crosscut spacing, the width of the dip stabilising pillars, and the depth of main haulages beneath reef. The design team specified a 180 m crosscut spacing; haulages about 220 m apart on plan yielding a back length of about 240 m; 40 m wide dip stabilising pillars; and main haulages at least 90 m below the reef plane (Table 1.3 and Figure 1.18). The ERRs ahead of the faces were determined. Modelling indicates that the ERR ahead of all working faces will be less than 25 MJ/m^2 at depth of 3000 m, and at a depth of 5000 m the probability is high (0.96) that all ERRs ahead of working faces will be less than 30 MJ/m^2. The placement of backfill is considered essential to reduce the ERR values ahead of working faces. Haulage tunnels have been placed about 94 m below reef, and only experience a small degree of mining induced stress. The tunnels pass beneath dip pillars, where near-field stress regimes between 50 MPa at 3000 m and 91 MPa at 5000 m are possible. The average pillar stress on the reef plane ranges from 300 MPa at 3000 m to 500 MPa at 5000 m.

As many as 17 levels are required to extract the Iponeleng ore body (Figure 1-21). This method required substantially more levels than the other methods because the back length of a CSDP raise is substantially less. The expected life of mine is expected to be the least of all four methods, as only 26 years are required to extract the orebody. During this period a total area of 12.2×10^6 m^2 of the 19.7×10^6 m^2 resource is mined, yielding an extraction ratio 62 per cent, with a total of 5.2×10^6 m^2 being left locked up in pillars and ground adjacent to geological features. The total length of off-reef development required is 381 km.

Evaluation

The closely-spaced dip pillar method is characterized by highly concentrated stoping with mining taking place on both sides of the raise and up to 24 panels per stope. Relatively high production rates are achieved because of double-sided mining on each raise line, and adjacent raise lines being stoped concurrently. Development is also fairly limited due to the limited number of stoping areas required for this method. These factors result in a reasonably low ventilation air requirement, and the management of the primary ventilation system is simple.

1.4 CONCLUSIONS

This study of the mining of the ultra-deep Iponeleng model orebody has identified some strengths and weakness of each mining method. The study has demonstrated the importance of simultaneously evaluating the impact of a particular layout on the efficiency of cooling, the efficiency of transporting men, material and rock, and the meeting of rock engineering criteria. Trade-offs will inevitably have to be made. This study has also highlighted the importance of carrying out multi-disciplinary life-of-mine simulations to assist in determining the economic viability of a mining venture.

The longwall method is ideal for a large mining block with a regular grade distribution, while the other methods offer greater flexibility where faults and dykes are prevalent, or the grade is erratic. The choice of mining method ultimately depends on the nature of the orebody, the critical mine design factors being the size of viable mining blocks and the regularity of the grade. It is conceivable that more than one method could be used on a single mine depending on the characteristics of the different mining blocks.

1.5 ACKNOWLEDGEMENTS

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Table 1.2 Macro design guidelines and assumptions

Orebody structure	based on 3D vibroseismics
Depth Range	3000 m – 5000m
Size of orebody to evaluate (plan distances)	6 km dip x 6 km strike
Assume constant grade of:	15 g/ton
Mine (Shaft) production output	start from 0 building up to 45 000 m^2/month
Average face advance rate	15 m / month
Inter-level spacing:	mining method dependent
Minimum number of levels	5
Development rates (max. values)	twin-ends: 100 m/mth /twin
	raise: 40 m/mth (from 0-50 m long)
	raise: 30 m/mth (from 50-100 m long)
	raise: 25 m/mth (for >100 m long)
Mine out shaft pillars first?	yes
Average stope-width	1.5 m
Average stope face length	to suit the method
Working days per month	25
Areal extraction ratio	method dependent
Length of stope backs	method dependent
Tramming width	1.3/1.8 m

Table 1.3 Expert design specifications for the various layouts.

		Mining methods			
	(units)	LSP	SGM	SDD	CSDP
Generic macro design parameters					
Raise to raise spacing...	m	n/a	200	100	180
Maximum length of back...	m	240	240	240	175
Dip pillar's maximum deviation from true dip (degrees)...........................	degrees	n/a	45		
Dyke and fault bracket pillar width (on either side)	m	20	10	20	20
Preliminary vertical distance between levels	m	<120	92	70	68
Width of dip stabilising pillar for depth range < 4 km	m		30		
Width of dip stabilising pillar for depth rang between 4 km and 5 km....	m		40		
Width of dip stabilising pillar up to a Max. depth of 4.6 km	m			25	40
Width of strike stabilising pillar for depth range between 3 km and 4 km.	m	40			
Width of strike stabilising pillar for depth range between 4 km and 5 km	m	50			
Design of flat development infrastructure					
Minimum vertical distance of haulage / RAW below reef	m	90	120	50	90
Minimum distance to reef of replacement haulages	m	90			
RAW-HLGE connecting crosscut spacing	m	250		200	180
Footwall drives/Haulage and RAW tunnel dimensions (height x width)....	m^2	3.0 x 3.5	4.5 x 4.0	4.5 x 4.0	4.0 x 3.0
Replacement Haulage and RAW tunnel dimensions (height x width)......	m^2	4.0 x 3.5			
Minimum distance from a HLGE - RAW connecting to reef X/cut breakaway	m		25	25	25
Minimum distance between opposite breakaways (longwall)..................	m	15			
Haulage and RAW spacing (centre to centre)...	m	30	30	30	30
Minimum middling of follow-behinds from reef plane	m	20			
Maximum middling of follow-behind from reef plane	m	40			
Minimum middling of incline man & material way from reef plane	m	60			
Follow-behind tunnel dimensions (height x width)..................................	m^2	3.0 x 3.5			
Crosscut-to-crosscut spacing (grid definition)	m	63	200	100	180
Crosscut to reef dimensions (height x width)..	m^2	3.0 x 3.5	3.5 x 3.5	3.5 x 3.5	3.0 x 3.0
Minimum crosscut distance below reef (perpendicular) at ar reef intersection	m	10	12		15
Design of stoping related infrastructure					
Panel face length between 3 km and 4 km	m	40	30-40	46	25
Panel face length between 4 km and 5 km	m	35			
Maximum number of panels per longwall section....................................	unit	6			
Maximum number of panels per raise...	unit	n/a	7	2	7
Average stope width (VCR considered)	m	1.5	1.5	1.5	1.5
Raise dimensions (height x width)..	m^2	2.0 x 1.8	3.0 x 1.5	2	1.8 x 2.5
Strike gullies dimensions (height x width)...	m^2	1.6 x 2.0	1.8 x 2.8		1.8 x 2.0
Ledging distance from centre line (on either side of raise)......................	m		10	6	15
Maximum strike scraper distance...	m	100	80		70
Maximum on-dip distance for continuous-scrapping...............................	m				150
Maximum length of winze	m		75		
Depth of wide raise (on either side from centre of raise)..........................	m			6	

	(units)	LSP	SGM	SDD	CSDP
		Mining methods			
Design of vent control infrastructure					
Bulk-cooling loop dimensions (height x width)..	m^2	3.0 x 3.5			
Cooling car cubby (one side extra width, in excess of x/cut width)	m	4 + 4		5	
Cooling car cubby length..	m	6	15	20	
No. of dedicated ventholes per x/cut..	unit	1	1		
Design of ore-handling infrastructure					
Conventional stope box hole dimensions (height x width).......................	m^2	2.0 x 1.8	1.8 x 1.4	1.4 x 1.5	1.8 x 2.5
Raisebore stope orepass diameter...	m		1.4 m		2.1
Boxhole stub length (extra width in excess of x/cut width).......................	m				5
Boxhole stub length..	m		12		20
Maximum raisebore length	m		120		
No of conventional boxholes/boxfronts per X/cut....................................	unit		2	2	2
No of Y-leg boxholes/boxfronts per X/cut...	unit		3		
Inclination of conventional stope boxhole (degrees)...............................	degrees		55	55	65
Maximum inclination of raisebore orepass (degrees)............................	degrees		45	15	65
Design of access-ways infrastructure					
Incline man and material way (longwall) (height x width).........................	m^2	4.0 x 3.0			
Length of the station for the incline man and material way (longwall)......	m	25			
Travelling way dimensions (height x width)...	m^2	2.8 x 2.0	2.4 x 2.4	2.4 x 2.4	2.0 x 2.5
Travelling way length..	m		15		15
Travelling way inclination (degrees)..	degrees		34	34	34
Design of materials-handling infrastructure					
Timber bay dimensions (height x width)..	m^2		3.0 x 4.0	3.0 x 3.0	3.0 x 2.0
Timber bay length...	m		30	20	30
Timber bay distance from reef	m		20		10
Generic scheduling rates and mining constraints					
Normal Haulage and RAW development	m/mth/end	35	30	35	35
High speed Haulage and RAW development..	m/mth/end	70	70	70	70
X/cut development..	m/mth/end	35	30	35	35
Raise development...	m/mth/end	30	25	12	30
Winze development..	m/mth/end		15		
Boxhole development (conventional)..	m/mth/end	20	10	35	20
Drop-boxhole development (conventional)	m/mth/end				30
Raisebore development..	m/mth/end	35			35
Delay in stripping of services at a raise..	months		1		
Delay attributed to stope equipping	months		6		
Lodging face advance..	m/mth/face		25		
Stoping face advance	m/mth/face	12	15	15	15