ORE DILUTION IN SUBLEVEL STOPING

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Abstract

The steeply dipping vein orebody of Bousquet 2 mine is extracted by sublevel open stope method. The presence of structural discontinuities, high induced stress distribution, and narrow stope widths are major factors which can seriously affect ore dilution. Therefore, strict ore dilution control is necessary in order to keep the mine competitive. The thesis describes the selective mining method adapted to this type of orebody, and subsequent measures taken to minimize ore dilution. For this study, the locations of stress relaxation are taken as failures zones. Therefore, these zones are more important since they are subjected to very low stresses which can provoke rock block sliding or local wall caving (tension failure). Tracing the zone of tensile stress in the stope wall gives reasonable prediction of dilution and permits to calibrate the numerical model using the field data obtained from the cavity Monitoring System (CMS). Stope mining is made in several cuts according to the width of the ore zone. For an improved recovery, some stopes are mined using primary and secondary stoping. Blasthole pattern and mining sequences are closely linked to the ore zone configuration. The main causes of ore dilution are summarized. They are a combination of several factors such as ground conditions, blasting damage, state of stresses around the stope, and stope design. In order to minimize ore dilution from hanging-wall and foot-wall, cable support is installed. Numerical modelling is carried out for a typical cable bolt pattern. The results of modelling have demonstrated that the cable bolts were necessary to control hanging wall caving. This is accompanied by a reduction of ore dilution. Consequently the study has shown the effectiveness of cables bolt as pre-support of the schistose walls in Bousquet sublevel stoping environment.

Résumé

Les caractéristiques associées au pendage prononcé des veines minéralisées de la mine Bousquet imposent leur extraction par sous-niveaux dont le chantier demeure vide. La presence de discontinuitée sructurales, de contraintes induites elevées, ainsi que l'ouverture des chantiers d'abattage sont des facteurs majeurs qui peuvent provoquer la dilution du minerai. Ainsi, un contrôle strict de la dilution est necessaire pour maintenir la mine competitive. Cette thèse décrit la méthode sélective adaptée pour ce type de minéralisation, et relate les mesures à prendre pour minimiser la dilution. Pour cette étude, les zones de contraintes de relaxation ont été considérées comme des lieux ayant une potentielle de rupture, par conséquent, elles sont considérées puisque ces zones sont soumises à des contraintes de cisaillement qui peuvent provoquer des glissements de blocks ou des effondrements (tension de rupture). En reportant les zones de tension sur les schemas des chantier il a été possible de predire les zone de dilution; et de calibre le model numerique en utilisant les données de terrain obtenues á apartir du moniteur de mesure du chantier (CMS). D'autre part, les zones de concentration de contraintes induites peuvent être affectées par des engrènements, dans le cas majeur par des coups de terrain (contrainte de rupture). L'extraction du minerai est faite en plusieurs séquences dependenment de la largeur de la zone mineralisée. Pour un meilleur rendement, quelques chantiers d'abattage sont repartis en chantier primaire et chantier secondaire. Les patrons de trous de sautage et les séquences de sautages sont étroitement liés à la configuration de la zone minéralisée. Les causes principales de la dilution du minerai se résumées. Elles sont une combinaison de plusieurs facteurs tels que les conditions de terrain, les dommages du sautage, l'état des contraintes autour du chantier, et le design du chantier. Pour minimiser la dilution du minerai provenant de l'épontes supérieure,des câbles d'ancrage ont été installées. La modélisation numérique est faite sur le patron de câblage typique de la mine. Les résultats ont montré que des câbles étaient necessaires pour stabiliser l'eponte superieure. Cela s'est accompagné par une réduction de la dilution. Par consequent, l'étude a montré l'éfficacité des cables d'anchrage utilisées comme pre-support pour des roches schisteuse des epontes de la mine Bousquet.

Table of Contents

List of Tables

List of Figures

Chapter 1: Introduction

1.1 General

Ore dilution problems in hard rock mines, epecially in Abitibi gold vein mining, are mainly attributed to induced failures that involve in the mining process (stress, blasting, excavation, haulage, backfilling)and to the rock mass quality (discontinuities, rock mass stiffness, rock mass shear strength, etc.). Therefore, the failures of waste rock within the stope are the direct consequence of wall instability. The inelastic and jointed nature of the rock in archean areas make that this instability of ground the greatest cause of underground mining problems. Thus, the effect of unpredicted ground failure due to the close excavations and mining operation, is a matter of great economic importance. Waste dilution increases the cost of production in that money will be spent in handling the waste rock causing dilution, and that ore processing facilities will be engaged for material which contributes very little to final useful metal or mineral production. Because of that, effort will be made to at least minimize this unexpected add of waste rock. The solution of that requires the understanding of all the effects that directly or indirectly provoke wall rock caving in the stope. This introduces an interdependence between scientist and mine operator in the selection of principles which are of value in developing a working knowledge of cause and effect. Because the acceptability of the scientific approach of ground failure depends upon how accurately it can simulate conditions to be observed in practice, thus, through a better knowledge of the science of rock mechanics, practical applications directed towards the study of cause, prediction, and control of rock failure could be done. However, these principles are often obscured by local conditions. Therefore the geomechanical implications are the cause of ore dilution problems in sublevel open stoping. These are related to the physical properties and structure of rock and their reaction to stress distribution arising with the geometry of openings. The relationship can be visualized as a shear/strength ratio and the critical failure come essentially from shear sliding and rarely from compressive buckling. This means to test (by numerical modelling) the rock mass instantaneous response to mining. The consequence of poor performance of the rock mass involves instantaneous waste block sliding, or even important waste rock caving; which unfortunately are added to the ore. Consequently, the factors to be considered in the design of the stope are the state of stress at the excavation boundary and in the interior of the rock medium, compared with the strength of the rock mass; and stability of the immediate walls. Good judgement thus depends upon a judicious blending of the practical with the theoretical.

1.2 Study problem

Prediction and control of ore dilution is not easy; it means both the analysis of instantaneous failure behaviour of immediate wall stope, and the precise knowledge of the geology of the orebody and the surrounding rocks. The phenomena accompanying the mining process and the variety of uncertainties on the geological knowledge of the deposit make that ore dilution problems remain difficult to handle. By understanding and controlling these phenomena there are some means permitting to considerable reduce ore dilution. Many characteristics associated to these deposits make their mining economical difficult. In effect, the rocks of Canadian Archean Shield had been submitted to several tectonic brittle forces. As result, while these rocks are strong, they remain difficult to mine economically because of the existence of some negative aspects of associated ground conditions:

- The rocks are highly foliated; which foliation constitutes the weakness planes in open stope conditions;
- These rocks are affected by several set of joints of which the orientations and characteristics change with the space;
- The area is locally affected by brittle failures, faults and its associated shear zones; the effect of the most prominent structural features of the host rocks (east-west trending, steeply south-dipping), can not be ignored;
- The principal stress is perpendicular to the weakness planes (north-south) and the vein orebody direction.

Presently the mine has attained a high level of ore extraction with 17 000 tonnes per day. On the other hand, poor ground conditions and spatial distribution of rock stress remain a challenge, especially for the recovery of planed stopes. The mining methods involve by the parameters of these orebodies much require for these rocks already weakened. From a rock mechanics perspective, the hanging wall and footwall rock as well as ore zone are competent but highly jointed with evident foliation joint spacing (near the wall contacts) and regional schistosity. Thus, the combination of the nature of the rock, the state of stress and the design of openings could be the main causes of **ore dilution problems**. in sublevel open stoping ore dilution come from the caving of wall rocks.

1.3 Objectives and scope

The ground conditions associated with the above problems demonstrate the difficulty to efficiency extract the ore in narrow vein area. Therefore, the problematic of ore dilution as ground conditions directly come to define all the aspects linked to ore dilution problems. So, the principal objectives of this project are:

1) To identify and understand the ore dilution problems;
2) To create links between the unexpected loss of grade in ore mucked and both ground conditions (induced stress distributions and rock mass quality) and blasting effects;
3) To analyse the measurements taken for reducing the dilution.

As the immediate host rocks are often highly sheared about the hanging wall and footwall contacts, our attention has been particularly oriented to induced instability of wall rock within the stope which lead to immediate caving. To further understand the problem, we has emphazed our study on Bousquet 2 mine.

The Complex of Bousquet mine is owned and operated by Barrick Gold Corporation. It is located between the cities of Val d'Or and Rouyn-Noranda in North-western Quebec. The original Bousquet mine began in July 1979. In 1986, exploration on the eastern side of the property led to development of what was to become Bousquet 2

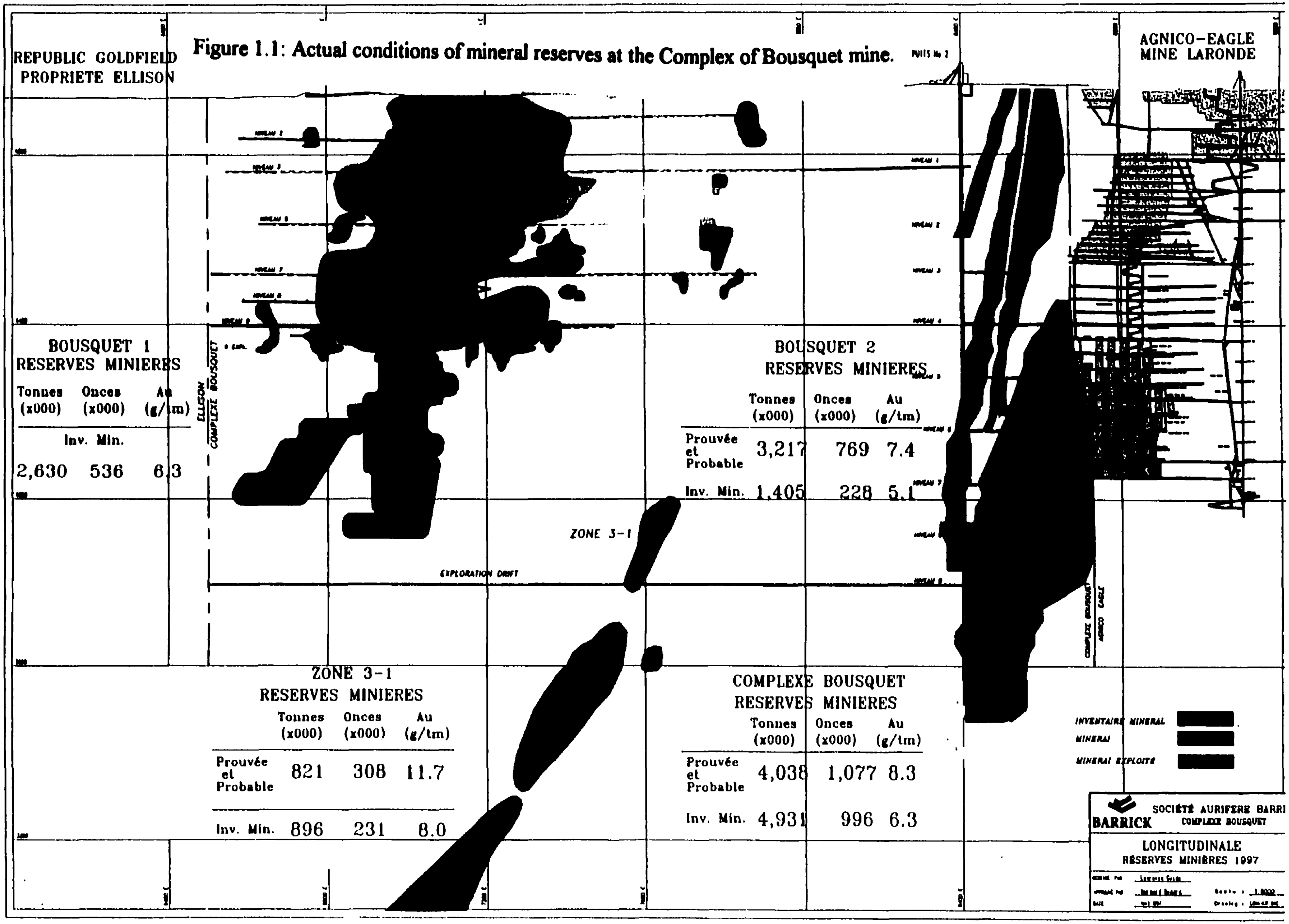


Figure 1.1: Actual conditions of mineral reserves at the Complex of Bousquet mine.

mine, which opened in 1989. The mine has 1.1 million ounces of reserves, grading 0.25 ounces of gold per ton. An additional one million ounces of gold mineralized orebody, grading 0.186 ounces of gold per ton, have been identified. In 1996, exploration of a zone between Bousquet 1 and 2 was followed by development work to bring this zone into production in 1999. There is also further potential to increase reserves at depth and between the two mines. Since the beginning of mining operations (1989), above 2.8 millions tons of ore with average grade of 10.3 g/t Au and 0.95% Cu, had been extracted The actual conditions of mineral reserves at the Complex of Bousquet is shown in Figure 1.1 (August 1997)

At Bousquet 2, the improvement of mine operation efficiency is a prime importance for going toward highest profitability. Therefore, it is not only to focus the efforts on great tonnage extraction, but also to control the factors which maintain the quality of ore produced, without compromising the human safety. Actually, most of metal mines in production become deeper and deeper, and the new mine discovered or will be discovered are located far from the surface. Consequently, the potential damage and/or risk provoked from material removed are not ignored. This reality led to take account all the associated ground effects susceptible to affect mining performance. The ground effects which must be considered are mainly the factors associated to the stress and those associated to the ground rock itself. The effects of stresses are function of depth and their orientation and distribution consequently are much critical. Therefore the performance of mining process is assessed through the following production criteria: Rate of production, Ore recovery, **Ore dilution**, and Ore losses. The consequences involve each of these productivity parameters can negatively affect mine life. However, the qualitative controls of all these parameters are practically impossible due to the inevitable lack of precision in deposit knowledge, orebody estimation and mine planing, due to production constraints.

1.4 Bousquet mine geological setting

The Bousquet mine is located within the southern margin of Abitibi greenstone belt. The steeply dipping gold-bearing zones is a deformed pyritic polymetallic (gold and copper) archean deposit located within a zone of intense ductile-brittle deformation. The orebody consists of massive sulphide veins occurring between mafic tuff (hanging wall) and rhyolite tuff (footwall). Figure 1.2 shows the geology of the vertical section of the mine. The orebody represents the west depth extension of Dumagami deposit. This district includes lot of mines (Bousquet 1 and 2, Doyon, and Dumagami) and approximately represents about 43 % of Quebec gold production. In this area the rock mass are competent but affected many orogeny events that provoke deformations, faults and other structural discontinuities (joints and foliations). In the scale of the mine, the orebody and the wall rock show the evidence of four major deformations (D1, D2, D3, D4). As result of these events, the wall rocks ore highly foliated paralletly to the orebody; and the hanging wall is affected by a major fault. The rock mass quality, characteristics and properties are described in geomechanical section.

The Bousquet orebody, through its mineral components and its structural elements, belongs to massive sulphide with a volcaneuous source. The occurrence of Bousquet district deposit along and near contacts between rocks of metasedimentary and metavolcanic-volcaniclastic explains the synvolcanic origin of gold mineralization. The mode of deposition of gold and the copper in Bousquet area still complicated since the temporal and genetic relationship with the massive sulphide. The principal minerals associated with the deposit are sulphidic minerals: pyrite, chalcopyrite, pyrrhotite, sphalerite, galena; hydrothermal minerals silicate alteration products: quartz, chlorite, magnesium, carbonates, etc.; and metamorphic mineral: biotite, cordierite, talc, kyanite, etc. The genetic model for massive sulphide in Abitibian archean greenstone belts emphases on the combination of hydrothermal activities and the volcanic events. Then, those mineralisations are formed around the discharged vents of submarine hot springs in the tectonically active, high heat flows environment of felsic volcanic centers (Franklin et al., 1981). According to this hypothesis, the presence of the mineralisation is linked to hydrothermal deposition of gold along fissures and fault systems. In some localities probably related to emplacement of mafic, intermediate or felsic intrusions. therefore such mineralisation still controlled loci of a variety of volcanic and hydrothermal effects ranging in age from pre- to post-mineralisation. Thus the location of many individual orebodies is controlled by faults and fissures. The guides of exploration for those types of deposits must be firstly on the specific area selection. For that, wide variety of features which are meaningful must be collected such as volcanic history of the area and all parameters controlling ore deposition such as favourable structures (faults and fissures), favourable rocks (volcanic rocks) and favourable mineralogy (presence of sulphide minerals). Recently, the application of geophysical methods (indirect method) has further increased the possibility to find these types of deposits.

1.5 Thesis outline

The thesis is divided in six main chapters. In all the chapters the topic begin by the general definitions and/or general assumptions, and later from data provided by Bousquet 2 mine it has been possible to identify important items that can help in the development of our conceptual ideas of the mechanism of failure in the hanging wall and the footwall. Chapter 1 gives the general information, the problem of narrow steeply dipping vein mining in Abitibi area and geological conditions. The description of mining methods is presented in chapter 2. Chapter 3 is focused on the effects of mining operations on ore dilution. Chapter 4 is devoted to numerical modelling of failure mechanism on typical stope of Bousquet 2 mine. The following (chapter 5) explains the estimation method of dilution and the measures taken to reduce it. Chapter 6 discusses the relation between the failure zones predicted by the numerical modelling and the stope boundary obseved by the CMS. The conclusions of the work including some recommendations are given in chapter 7.

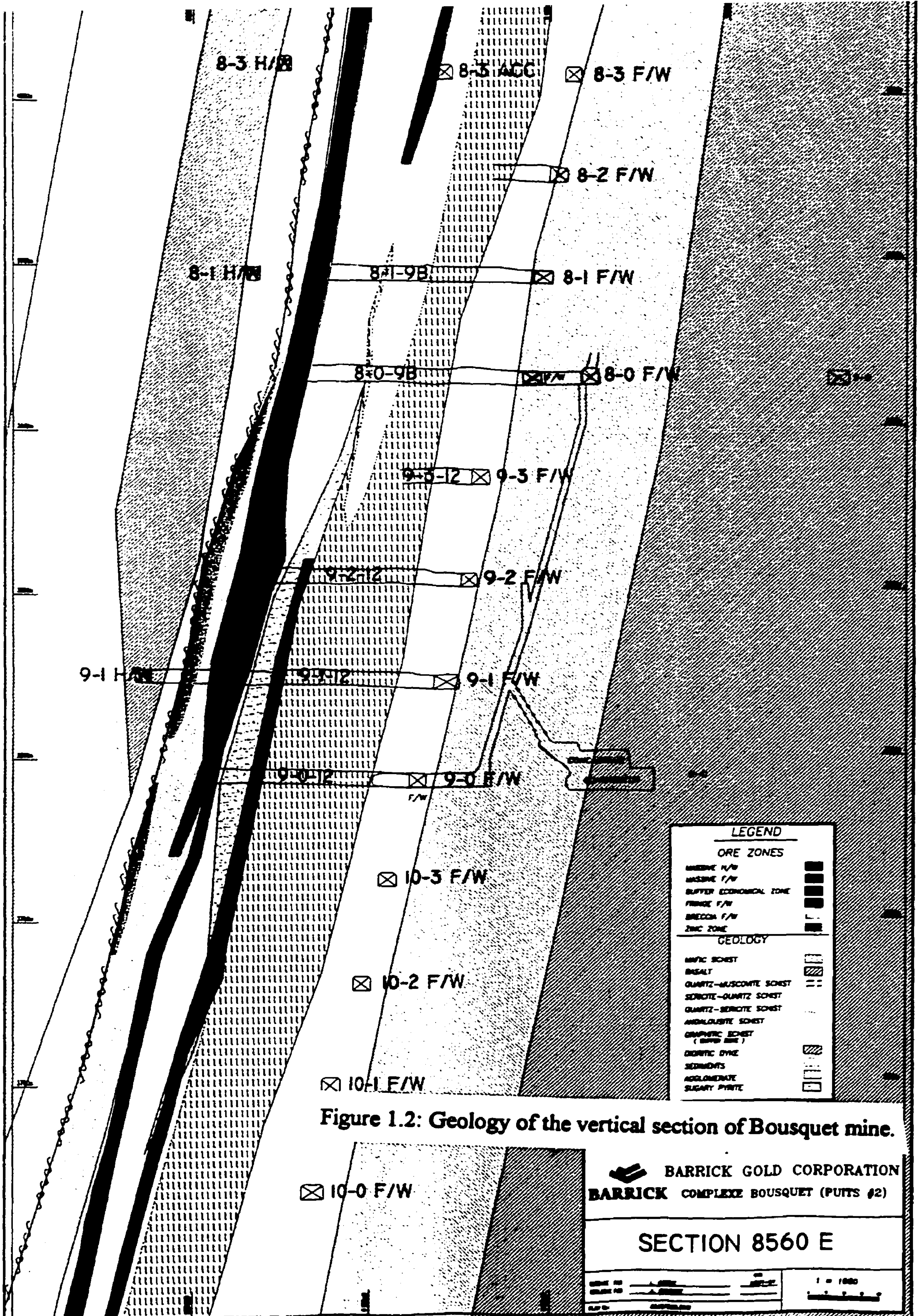


Figure 1.2: Geology of the vertical section of Bousquet mine.

Chapter 2: Sublevel Mining methods

2.1 General

The choice for a specific mining method result from target knowledge on orebody characteristics as well as ground conditions. According to Folinsbee and Clarke study (1981), the procedure of choice should began with the preliminary selection of a mining method or methods on which an engineering evaluation will be carried out. Naturally for steeply dipping vein orebody (where its configuration including size, shape, grade distribution is provided through the geological data), the preliminary choice must be carried out on some sublevel mining methods (cut and fill mining, shrinkage stoping, etc.). Then conceptual and engineering studies are done for each method , taken account the technical and economical optimization. The factors considered for the evaluation are multiple:

- Geomechanical data: competence of the ore and host rocks, joints systems, foliation, faults, groundwater, etc.;
- In situ stresses: Orientation of principal stresses, stress levels;
- Economical data: grade of the mineable ore, ore grade distribution throughout the orebody, the value of the ore in the ground;
- Geographical data: natural and societal conditions

Although experience and engineering judgment still provide major input into the selection of mining method, subtle differences in the characteristics of each deposit, which may affect the method chosen or the mine design, can usually be perceived only through analysis of measured characteristics. Sublevel mining is a large-scale mine stoping method and can be divided in five mains types of stoping methods: Sublevel Caving, Shrinkage Stoping, Block Caving, Cut and Fill, and Sublevel Open Stoping. Among these mining methods, the sublevel open stoping is the most common method used in canada. In fact, the open stoping method provides significant advantages in term of operational performance. As it can be seen in table 2.1, open stoping method with efficient cable bolt implementation offer relativelly lower dilution (5 to 10 %) with better recovery (rather

97% of recovery). Howevere sublevel open stope mining is applied for vein orebodies with the following characteristics:

•Steep dip: the inclination of the foot-wall must exceed the angle of repose of broken rock by some suitable margin. This is required to promote free flow of fragmented rock to the loading cross cuts.

•Strong hanging-wall and foot-wall: since open stoping is unsupported from inside, the strength of orebody and country rock must be sufficient to provide stable walls, faces and crown for excavation.

•Competence of ore and Regularity of orebody boundaries: the orebody boundaries must be fairly regular, since selective mining is precluded by the requirement for regular stope outlines, which are associated with the used of long blast holes.

These characteristics define the nature and the quality of orebody and host rocks as illustrated in figure 2.1 and 2.2. Hence sublevel mining method is suited to low grade orebodies with regular stope outlines. Rock reinforcement may be used in particular areas, but is not required as a routine operation as in cut-and-fill mining.

Mining Method	Ore Dilution (%)	Ore Recovery (%)
Sublevel Stoping	5 - 10	95 - 97
Shrinkage Stoping	10 - 15	93 - 95
Cut and Fill	15 - 30	93 - 95
Sublevel Caving	10 - 15	85 - 88
Block Caving	15 - 10	80 - 85

Table 2.1: Sublevel mining methods with their recorded performance (Eustace W.,1983).

Under the need to continually improve efficiency in mining operations, more selective is opted while a dilution action plan currently targets solutions to dilution and cavity monitoring measuring system provides surveys of blasted stopes that can be compared to planned outlines. As a result, much effort is made on cablebolting process to reduce ore dilution. Consequently this presents in first stage the sublevel stoping methodology applicable to the steeply dipping orebody of Bousquet mine. Any particularity which lead to modify stope geometries. In the second stage, the use of cablebolt techniques to minimize the dilution coming from wall sides.

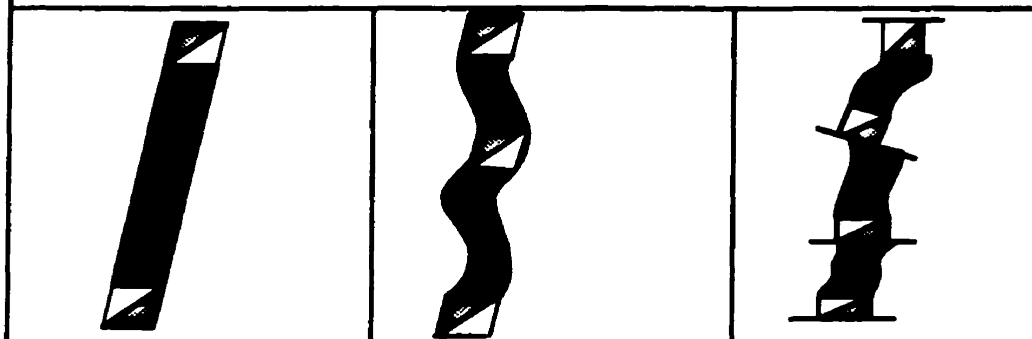
Nature of deposit	
Regular	Irregular to discontinous
Vertical cross section	
	
Delimitation of orebody	
- Easy	-Difficult: density and total length of prospecting holes must be increased.
Length and Orientation of blast holes	
-Parallel long holes	-Drilling and length of blast holes adapted to the irregularities and discontinuities of the oreboby.
Distance betwen sublevels	
-Possibility to increase the distance between the sublevels and application of bulk mining method.	-Short, adapted to the irregularities of walls and discontinuities.
	Ore dilution
-Low, if rock quality is adequate.	-High, for sub-level and bulk mining methods -Can be decreased through selective mining methods

Figure 2.1: Nature of orebody and mining techniques (orebody dipping > 45°). After Potvin (1988)

As it can be noted above, greater detail rock mechanics information at Bousquet district must be used to provide realistic estimates of underground opening design, amount of support, orientation of opening. If ground control or operational problems should be encountered, modifications could be made. Also other parameters that must be examined include:

Geometry and grade distribution of the deposit

Mining costs and capitalization requirements

Environmental concerns and other site -specific considerations.

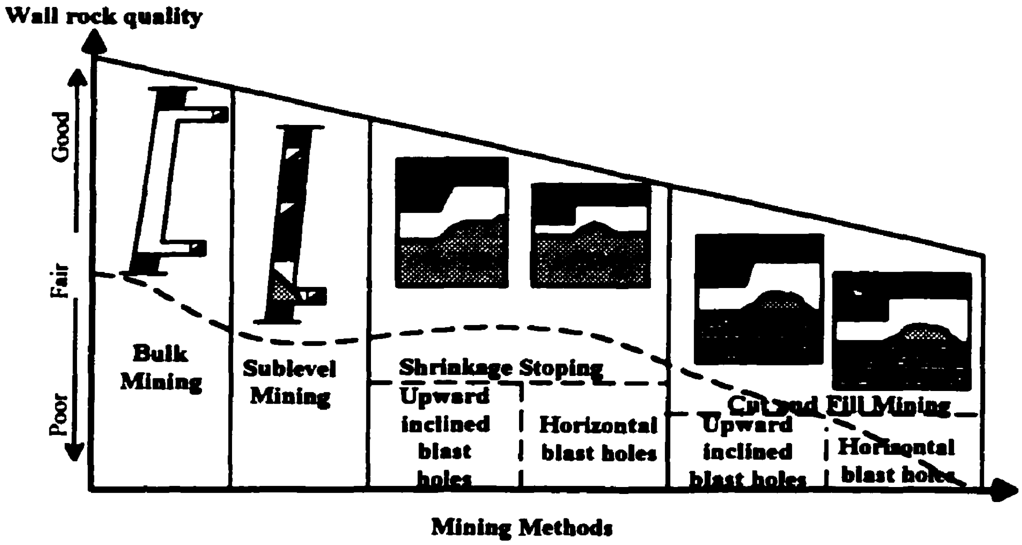


Figure 2.2: Influence of rock wall quality on the choice of mining method. Potvin (1988)

Basically the method entails providing access to the orebody at various subintervals between the main haulage levels in order to drill and blast the intervening ore. Stope drilling is carried out from drilling drifts on the sublevels, and the ore is blasted in slices towards an open face, which generally is vertical on the downholes and may be inclined towards the open face for the up holes. The blasted ore gravitated to the bottom of the stope and is collected through drawpoints. Dilution with waste rock may occur if ore boundaries are irregular or if caving occurs (that currently happens in weak zones); but 100% of the ore within the stope usually is recovered. While pillar recovery sometimes is a problem.

Sublevel open stope mining methods can be classified into three methods, because of local rock quality, depending of extraction sense of orebody:

- Vertical Crater Retreat (VCR) or Vertical retreat (figure 2.3);
  - Transverse Blasthole (figure 2.3a)
  - Longitudinal Blasthole (figure 2.3b)
- Horizontal Retreat (figure 2.4);
- Sublevel Retreat (figure 2.5);
  - Longitudinal Longhole (figure 2.6)

This classification is based on following parameters :

◊ Extraction direction:

⇒ Longitudinal: advancement of blasting moves towards the extension of the orebody;

⇒ Transversal: generally consist to develop mining in way to advance blasting sequence from a wall side towards the other.

◊ Use of pillar and backfill

⇒ No pillar non backfill;

⇒ With pillars but no backfill (medium recovery, for example low grade orebody);

⇒ With pillars and backfill (use of primary and secondary stopes).

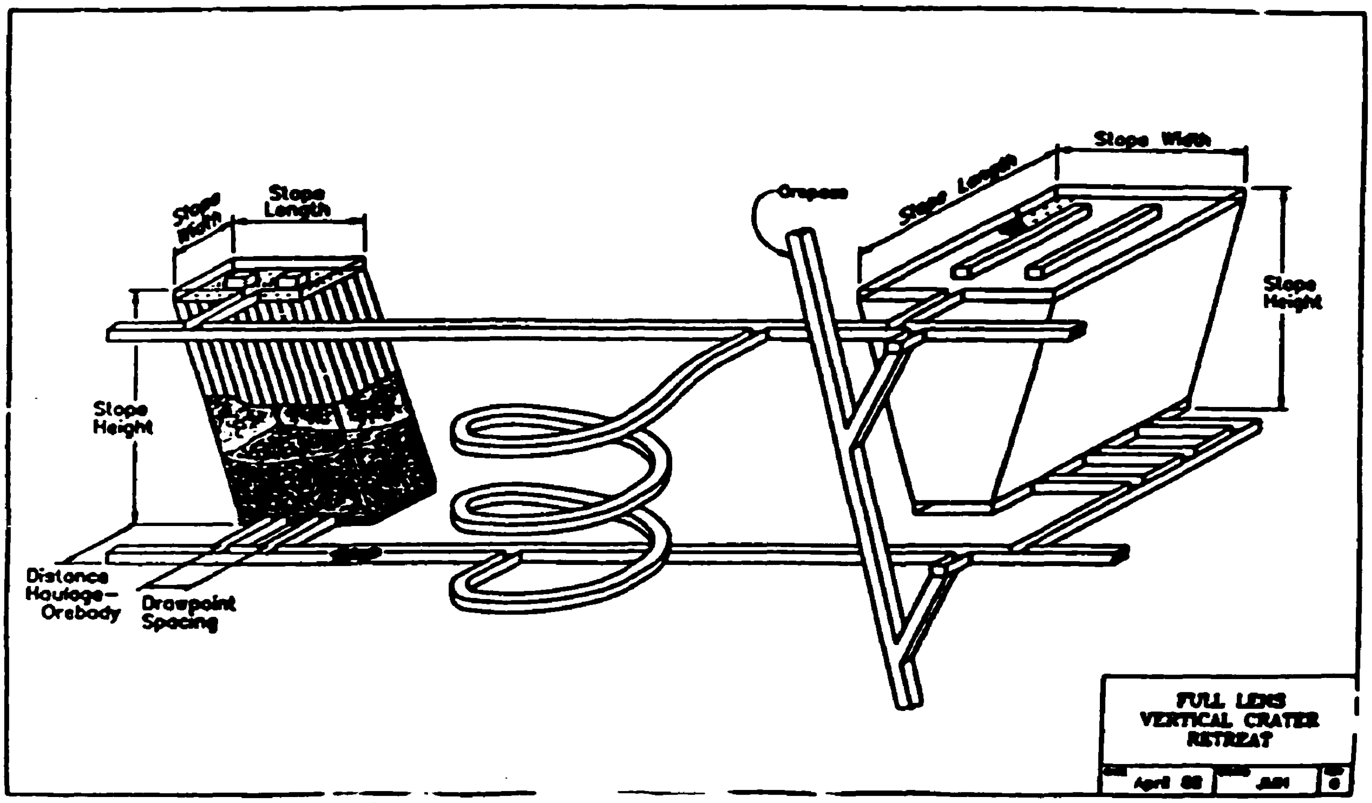


Figure 2.3: Vertical Crater Retreat (VCR). Potvin (1988)

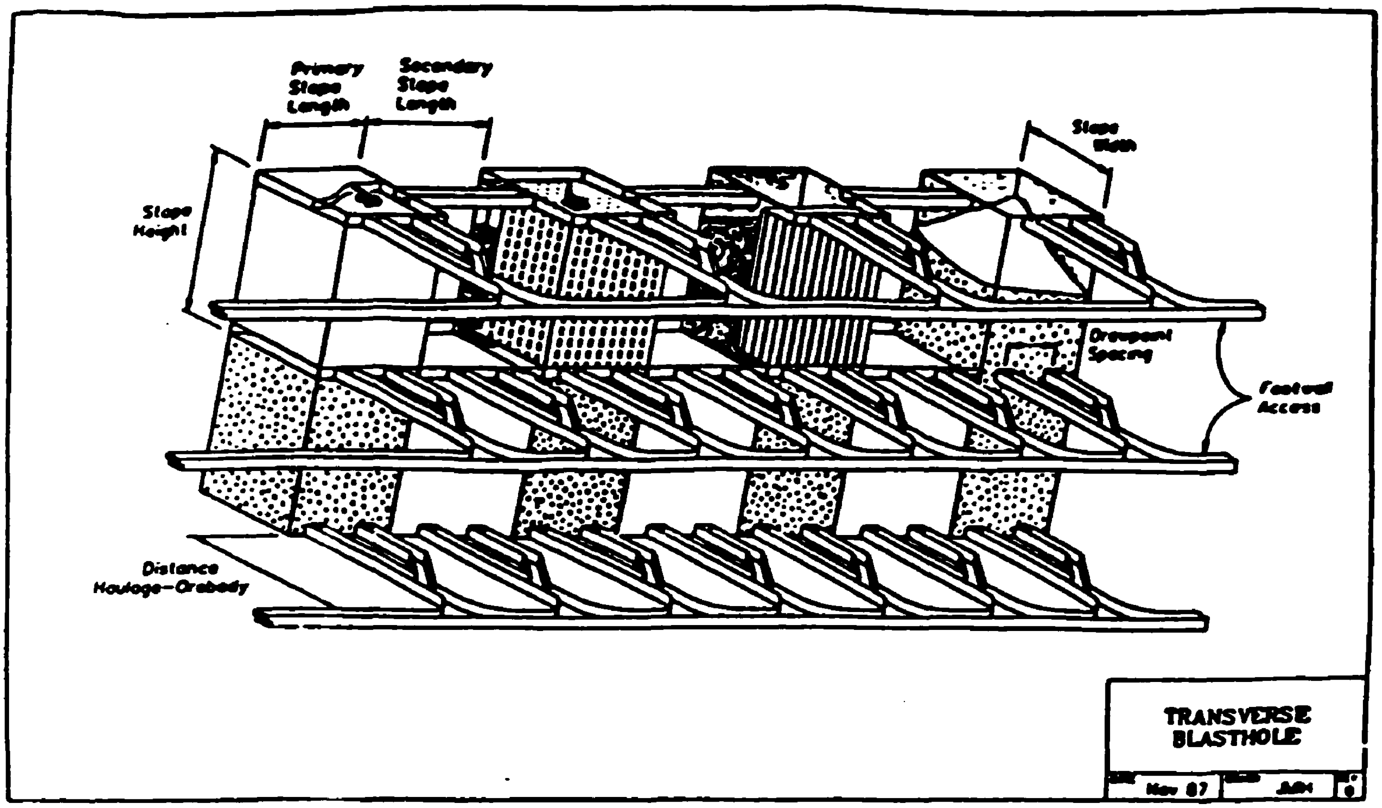


Figure 2.3a: Transverse blasthole in sublevel vertical retreat. Potvin (1988).

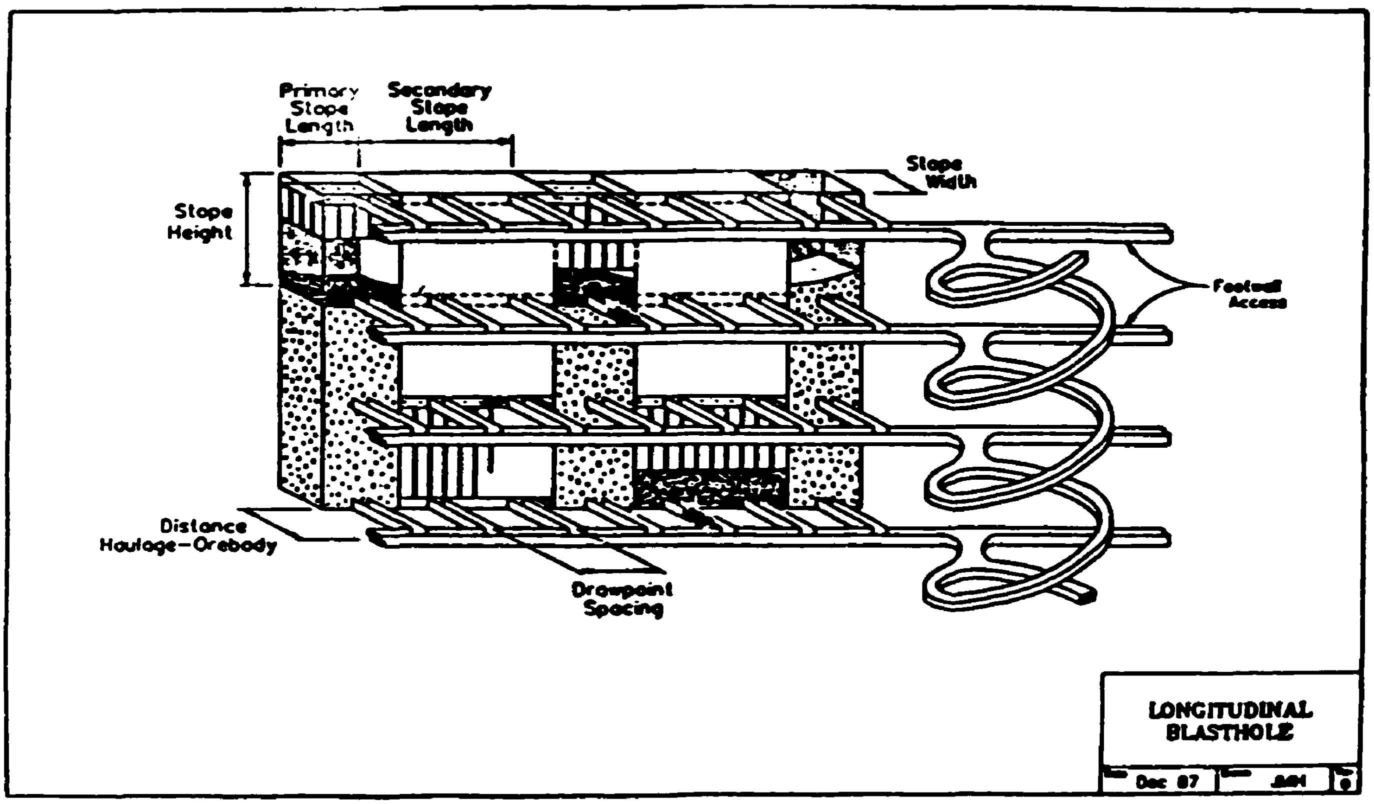


Figure 2.3b: Longitudinal blasthole in sublevel vertical retreat. Potvin (1988).

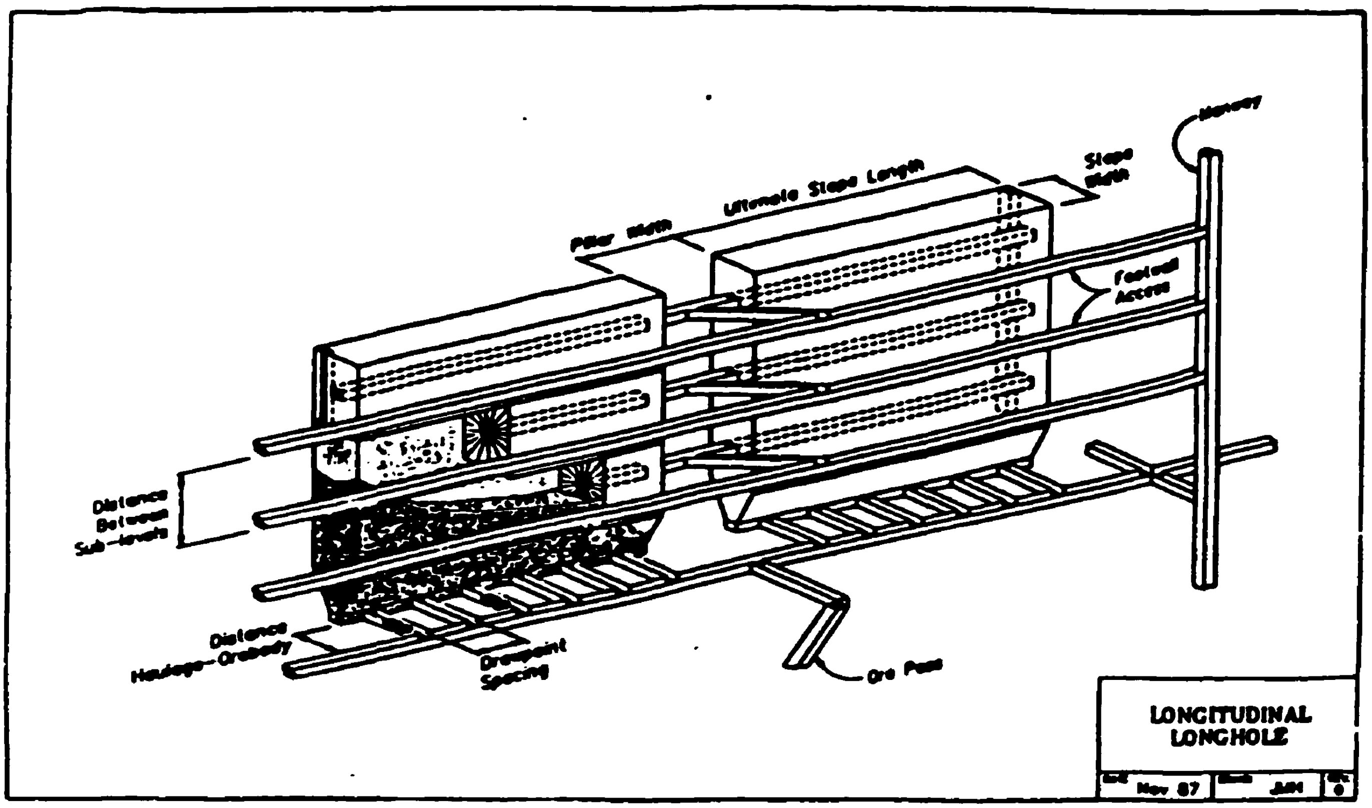


Figure 2.4: Longitudinal longhole mining Horizontal Retreat. Potvin (1988).

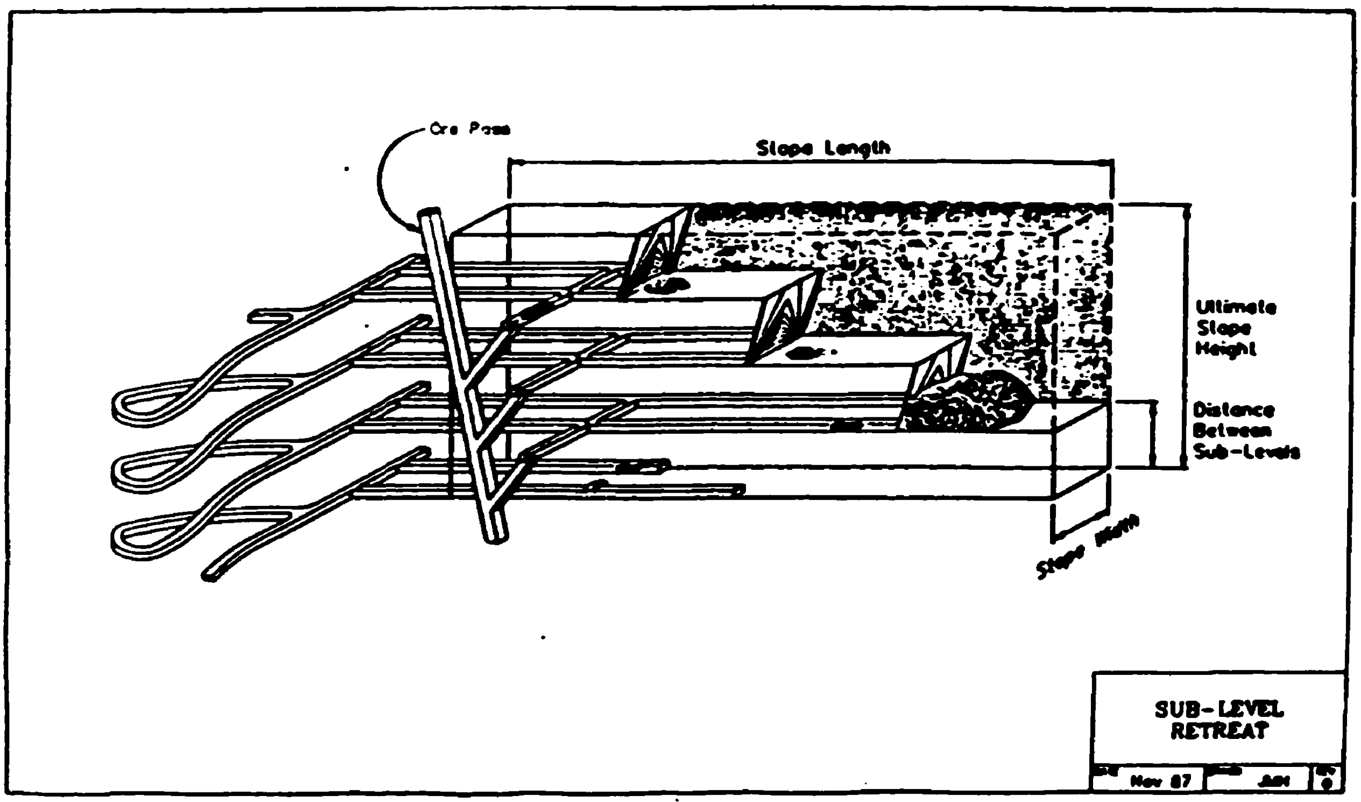


Figure 2.5: Typical sublevel retreat mining. Potvin (1988).

◊ blasthole diameters:

⇒ Small diameters 50 to 64 mm (2 - 2.5 inch.); usually adapted for longhole methods;

⇒ Big diameters of 100 to 200 mm (4 - 8 inch.), for blasthole methods.

2.2 Stability Aspects in Sublevel Mining

Analysis to determine rock mass response to mining is apparently rarely undertaken explicitly in mine design. To understand stability considerations in sublevel mining, it is necessary to emphasize upon some indispensable geomechanical aspects. Attention may be concentrated on predicting ground performance around blocks of stopes, and on establishing an extraction sequence which will minimize any risks arising from instability. Increasing depth of mining, or the need for increased extraction ratios from near-surface orebodies, increases the potential for unstable rock mass behaviour. Sublevel stoping excavation design is distinguished by uncertainties concerning the in-situ strength of rock, and frequently by the need to control an a large scale the performance of overstressed or failed rock. Sometime adverse induced stress redistribution allow propagation of cracks formed by overstressing, and the formation of tension cracking or opening up of pre-existing structures adjacent to faults (currently associated to the ore zones) showing large deformation. Borton (1997) found that joints at a low angle to the wall of an excavation result in greater wall deflections than joints at high angle. High stress effects can be also apparent in the bored cut out raises and in long blast holes. These blast holes have to be sometime pre-charged immediately after drilling and several months before blasting. This has several undesirable effects like desensitization of explosive, cut offs in the line of explosive and consequent hang-ups. In addition, the most undesirable effect is the premature detonation of precharged holes due to rock movements under high stresses. In sublevel steeply ore mining like Bousquet mine, small to large scale displacements are induced in the rock mass and the energy is dissipated by slip and crushing of wall sides. Then the evaluation of orebody ground conditions should start before opening up the ground for development. Two important parameters in the economics of a stoping method for which a rock mechanics study can provide estimates are the with of the stopes and the size of the pillar (Nicholas, 1981). In the sublevel stoping, the width of the a stope is a function of the immediate and intermediate roof (Alder and Sun, 1968). The immediate roof is characterized with the pressure arch concept. In the pressure arch concept, the rock is considered to have maximum distance that it can transfer the load. The ability of the rock to transfer a vertical stress in a lateral direction over an underground opening depends on the shear strength of the rock, the horizontal stress, and the strength of the rock pillars. The maximum stope width is twice the maximum pressure arch. The pillars spaced this distance must be able to carry tributary-area-load. Joint orientation, spacing, and length can be used to defined the stope width. The pillars within twice the maximum transfer distance do not have to carry tributary-area-load, but rather the load under the pressure arch, half way to the next support.

It can be concluded that, in sublevel mining several of the instabilities in crown and rib pillars are associated with the development openings which caused high stress concentrations. More attention appears desirable in mine planning, and also during mining operations, to the possible stress concentration around such openings. An numerical analysis 2D or 3D appears adequate to define zone of possible failure (confined and relaxation zones). Consequently, to remedy unexpected wall caving, ultimate site investigation is needed for optimum stope design. The contribution of blast stresses to pillars failure near opening, and to the dilation of pillar and walls generally must warrant investigations.

2.3 Bousquet Sublevel stoping

2.3.1 Mine final design

Mine design is conceived as a process oriented towards technical and economical optimization. Every major design and engineering decision is taken as the result of technical feasible alternatives. The most relevant vertical design plane of bousquet 2 mine is shown in Appendix A. Actually the mine search about 1350 m depth divided in 9 levels along the orebody. Each planned stopes, now being mined or planned to be mined with sublevel stoping, can be seen. A vertical shaft (shaft n° 2) gives access to different levels. Originally it is planned with main levels at 30 m intervals. Main levels for personnel material transport and haulage connect the shaft with the production blocks. Eventually the main design parameters to think about in Bousquet mine should be situated in 4 questions; 1) Sublevel distance; 2) sublevel orientation; 4) ore recovery; and ore dilution. A plan view of level 7 -0 of the mine is shown in Appendix A. Access to ore zone and its transportation has involved several drift and ore-pass. Much data about geology, rock mass quality and geostructural data (discussed in last term topic) are available in this figure.

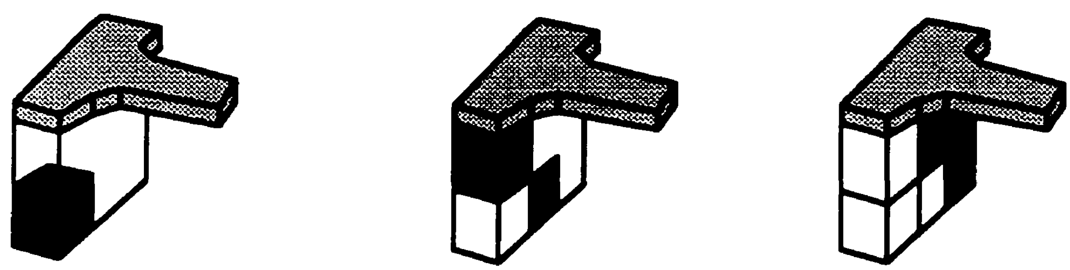
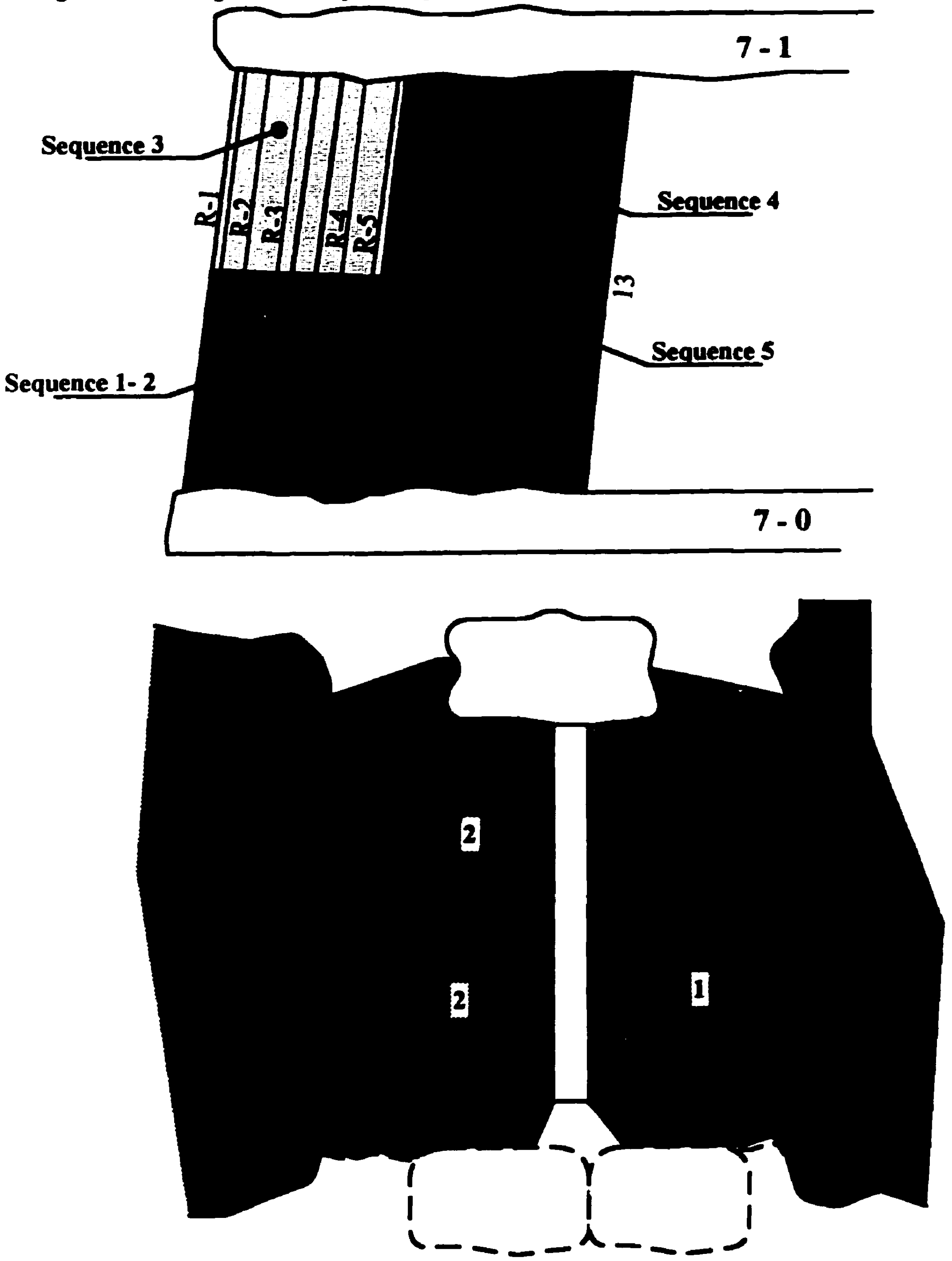


Figure 2.6: The mining sequence for the stope 9-1-15 stope (Hyett, et al. 1997)

2.3.2 Bousquet 2 stoping

At Bousquet the ore zone is a steeply and dipping (80°) vein of disseminated sulphide. The shape of this orebody , its geometry and geomechanical characteristics of host rock at Bousquet district, impose its extraction by Sublevel Open Stoping Method. The open stope mining referred to here is the mining of large opening by non-entry mining methods. This method of mining is initially development intensive, however, this is compensated by more flexibility (combination of other mining methods if necessary) and of appropriated equipment. At the beginning of stoping, an access is developed in the ful length of the planned stope (15 m); because to be able to drill production blasthole. The diameter of those hole is 10 cm; and their pattern is 2 meter (fardeau) by 2.5 meter (width); they are drilled parallel or in eventail depending to local shape of the orebody. Also the length and the orientation of each blasthole is adapted to the local shape of the orebody. Each stope have a rise hole with a diameter of 1.2 meter used as initial opening for the need of effective blasting. Generally, the explosive used for production blasting is ANFO for the blastholes situated in the center as well as those situated close to the footwall; while, blasthole along the hanging wall are loaded by fairest energy explosive. This blasting design result from specific controlled blasting, and is used to minimized ore dilution coming from the hanging wall. For efficient productivity, vertical bulk mining is used for single stope. But depending to planned stope size, mining of the stope may be made in several blasting sequences the Figure 2.6 show the mining sequence for stope 9-1-15. Notice that stope number is established using number of the main level, the number of the sublevel following by stope number. Sometimes, if geological and geomechanical permit, largest stope can be planned and extracted. That is the case of stope 7-1-8 where 21 300 tonnes of ore material are mined. Mining such a large stope has necessitated five blasting sequences with much that 2 445 meters of blasthole drilled, as shown in figure 2.7. These blasthole are grouped in 13 ring (R-1 to R-13): characteristics of blasting sequences and patterns (type of explosive, weight of explosive charge, Basting delay) are shown in Appendix A. Usually the first three sequences are upwards blasted, that create free space necessary for efficient blasting of the next sequences. Next sequences are vertical bulk blasting sequences implying larger volume of ore mucked. This process is made somewhat the removal of the entire stope. The stopes must remain open until the ore is removed and the opening backfilled. This sublevel vertical retreat leaves the possibility that any major dilution from the walls will fall into mined out area. It also allows for sill or rib pillar to be quickly established if ground conditions become unfavorable. Blast holes are long-holes drilled from upper level and its pattern depend of local ore zone configuration. In regular ore zone blast holes are drilled rather underlying level, parallel to the wall; therefore inclined at 284° downwards. While in particular irregular or large ore

Figure 2.7: Design of Bousquet stope blastholes and blasting sequences



zone, blast hole patterns are disposed in fan of 16°. For this a relatively high powder factor and drill factors are required in order to minimize wall failure mechanism and to ensure good fragmentation. The mining longitudinal sequence is east-west. As at Hemlo Gold mine, the development of cable bolt drifts increases induced instability; hence stress conditions prevailing narrow the stopes coupled to existing development configuration render the recovery of these stopes hazardous and difficult.

The relative economy of mining in this manner (compared with cut and fill mining) has become increasingly attractive at moderate depths with improvements in equipment for long-hole drilling and ore loading, and increasingly large blasts are being used. The development of extension steels and special long-hole rockdrills and more recently, the large-hole blasting techniques, has made sublevel stoping a method of increased popularity. The complicated and comprehensive development may be seen as a drawback but is compensated by efficient ore production. The drilling, blasting, and loading operations can be performed independently of each other, offering the potential for high utilization of equipment, and high output with few machine units and operators. Knowledge of the of the geology, the ore boundaries and a careful control of hole alignment in the long-hole pattern are key factors for the successful application.

2.4 Conclusion

The sublevel open stoping remains an improved mining method for Bousquet 2 orebody characteristics. The presence of several close openings, in jointed volcanic rock, required by the mining laid the challenge of stress and instability distribution along the stope walls. Open stope required lot of development works, but because of that many type of equipment can be used for increasing the production rate. The experiences from this mining method are positive, that involve that the method in the will increase in popularity, replacing methods as shrinkage and cut fill mining if geomechanical data are not very unfavorable. However, the use of cable bolts as a pre- reinforcement method of ground control must be efficient to minimize ore dilution from stope wall rocks.

Chapter 3: Ore Dilution

This chapter presents a study of various factors that may cause ore dilution in sublevel mining.

3.1 Overview

Ore dilution is the addition of waste rock or non-ore material to the ore during the mining process. The addition of waste rock decreases ore grade and increases the mined tonnage for a given geological reserve. The adverse effects of this remain a major problem especially in gold vein underground mining. The nature of these types of mines is such that extraction is always accompanied by the extraction of a certain amount of waste rock. Sometimes the tonnage of waste rock mined is larger than the strict allowable, and the grade of the run of the mine ore will be lower than the estimated in-situ grade of the deposit. For this reason geostatistians refer to dilution problem as ore losses. In fact, if mine operators expect a grade **x** and they recover y, there is a loss **x-y**, a relative loss of $\frac{\{x-y\}}{x}$; and it can be said the dilution is $\frac{\{x-y\}}{x}$. On tonnage, if they expect **x** and end up with y, the harmful gain is **x-y**, or relatively $\frac{\{x-y\}}{x}$. So, ore dilution is an important mining parameter for evaluating the efficiency of a mining method adopted for a particular orebody. There are four main factors which can result in ore dilution:

- Weak ground conditions;
- Mining method employed;
- Nature of orebody;
- Degree of operation controlled.

There are two types of ore dilution: intentional and unintentional.

Intentional ore dilution results from waste rock within the stope design boundaries. Hence this type of ore dilution is planned and the coming ore grade justify its extraction; figure 3.1 illustrates ore dilution problem in stope design. The improvement of mine productivity resulted in heavy mechanization and bulk mining methods. The mining engineer must consider the parameters associated with these choices. The design of planned dilution is observed both in development works (drifting) and in

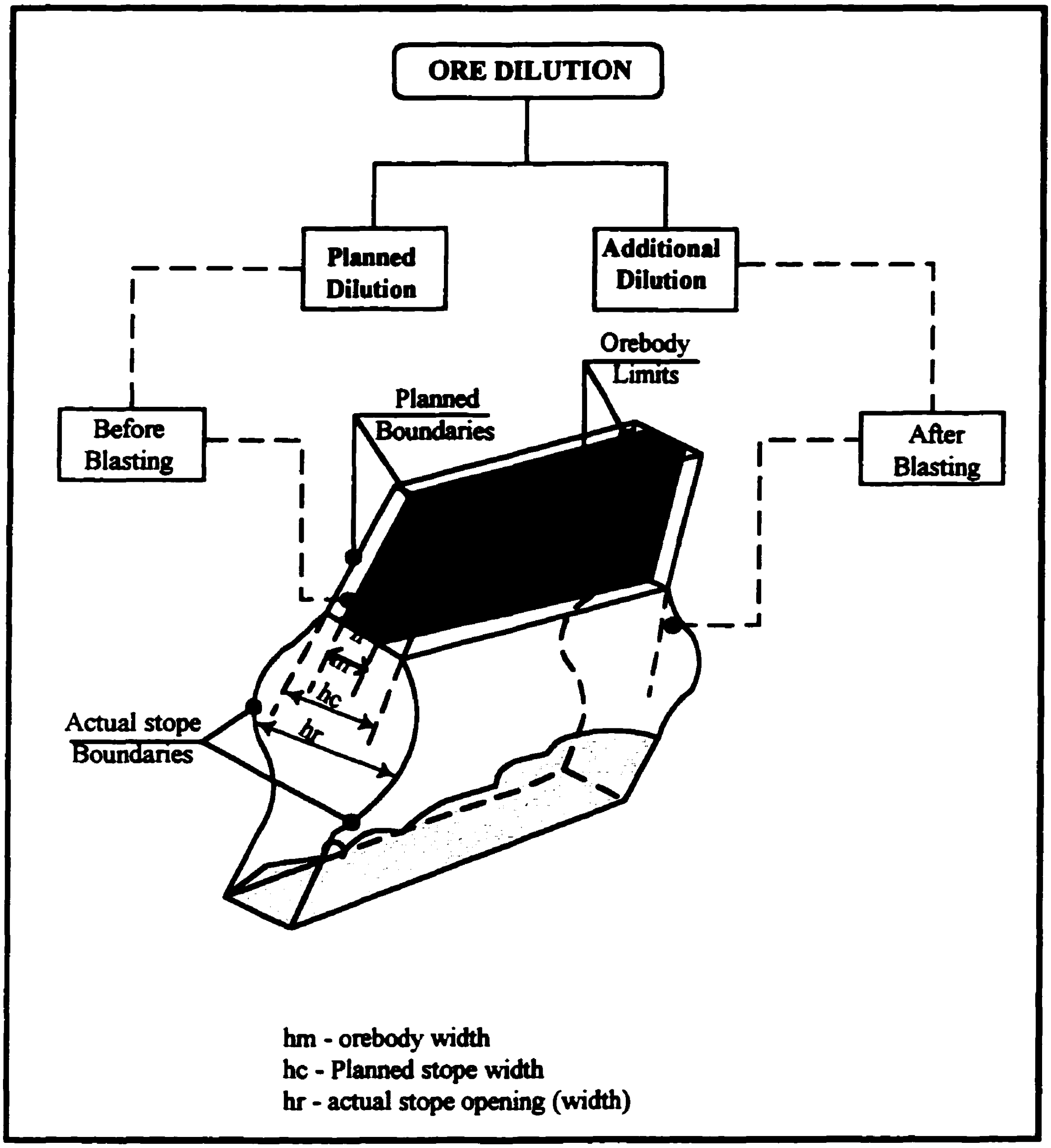


Figure 3.1: Illustration of different types of ore dilution in a production stope in steeply dipping vein mining (Bourgoin et al., 1991).

stope mining. In drift development within the orebody, dilution is linked to the size of drifts imposed by the width of equipment used. Ore dilution resulting from stoping is much more important, because stoping remains the main purpose of mining activity. Consequently, the design of the stope shape, especially in narrow vein extraction, may not include more than the strict minimum waste rock (planned ore dilution). While the incentive for high productivity allows for the use of bulk mining and wider mechanical equipment; it is not simple to arrive at the final stope design in vein orebody mining. Figure 3.2 shows the intentional (planned) and the unexpected additional ore dilution in stope mining. It takes into account the addition of certain quantity of waste rock in ore reserve calculation; consequently this expected addition of waste material is due to a combination of ore deposit discontinuity, its configuration and poor rock mass quality.

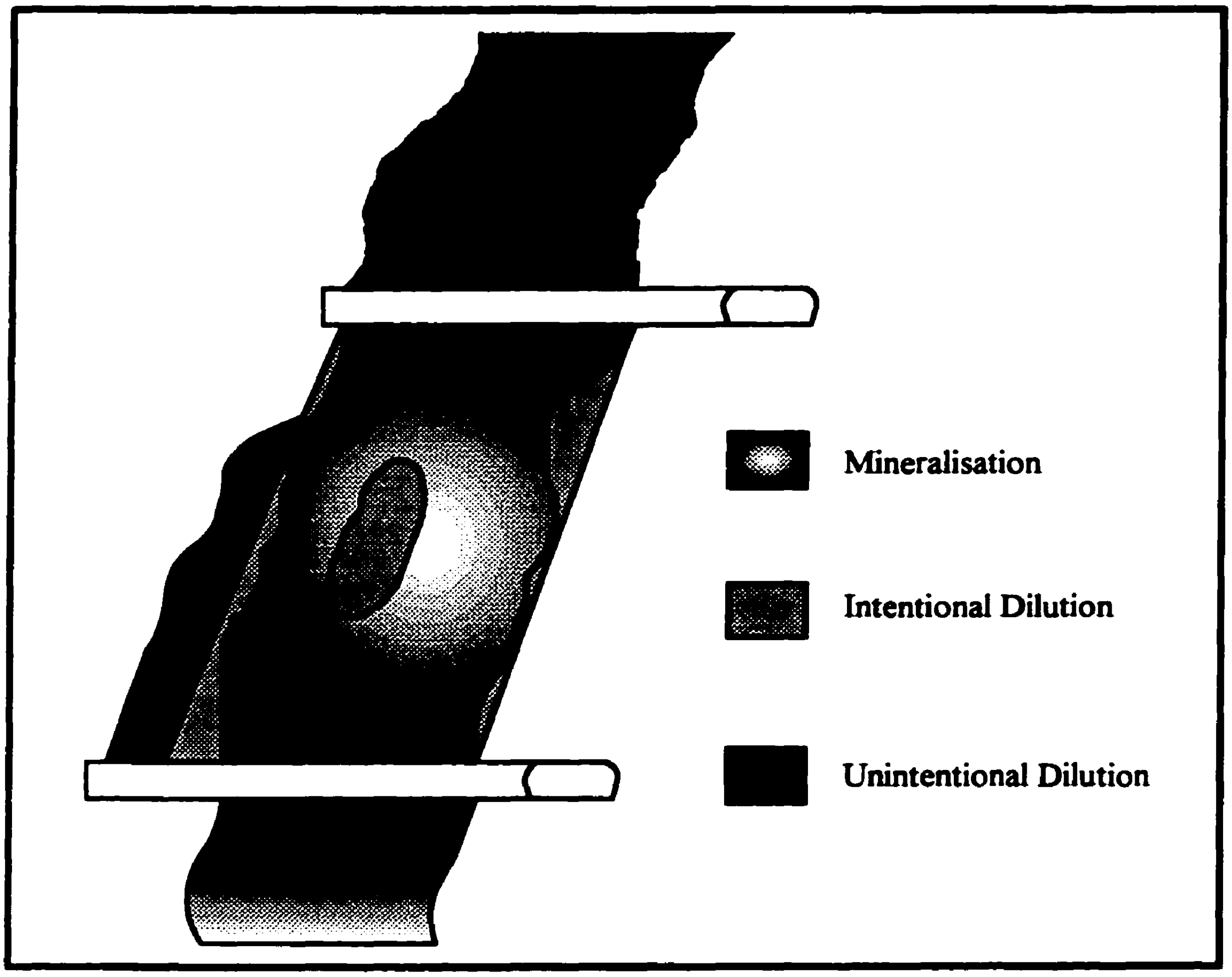


Figure 3.2: Example of stope design showing intentional and unintentional ore dilution in sublevel mining.

Unintentional ore dilution is the unexpected addition of waste rock not included within the stope design. The impact of this unintentional addition of waste rock is that it is mixed with stoped material and produced as ore. The unintentional dilution is hence, an additional ore contamination which could be mainly due to wall rock caving. Figure 3.3 illustrates the sources of additional ore dilution. As can be seen, the geomechanical aspects pay a major role. In general, main parameters to consider are:

- Wall rock qualities: competency of hanging and foot walls. Tendency of the walls to cave or slip, depending on stope geometry and its production (blasting effect);
- State of stress: mining process involves induced stresses which directly or indirectly lead to rock mass fracturing and displacements throughout the zone of influence. This implies local instability of rock around individual stopes and other excavations.
- Nature of orebody: mining of narrow vein is the challenge against ore dilution. Also, orebody local discontinuities and irregularities remain critical. The irregularities of the orbody make it difficult to mine; while orebody continuity is an essential element of its quality control. Deposit estimation depends on continuity appraisals at two levels: 1) definition of the geometry or physical form of a mineral-bearing, geological structure, and 2) spatial variability of a value or quality measured (e.g. grade, thickness) within a mineralized zone. Another aspect linked to orebody nature is the facility or not to identify the ore zones directly in the mining field.

Other causes are directly linked to blast induced damage, bad stope sequencing and lack of good knowledge of the orebody. As can be observed, geomechanical aspects are the main sources of unintentional ore dilution. For this reason the present thesis is mainly focused on rock mechanics aspects to understand unexpected ore dilution. Stoping in fair ground conditions implies to process, at all mining levels, for additional, yet useless tonnage. Two other major parameters influence ore dilution: choice of stoping method and mining technique or operational controls. Large deposits with relatively homogeneous, mineralization are often mined with stope excavation sized smaller than the deposit dimensions. Intentional dilution may be included in the recovered stope; but unintentional dilution is frequently excluded in this instance.

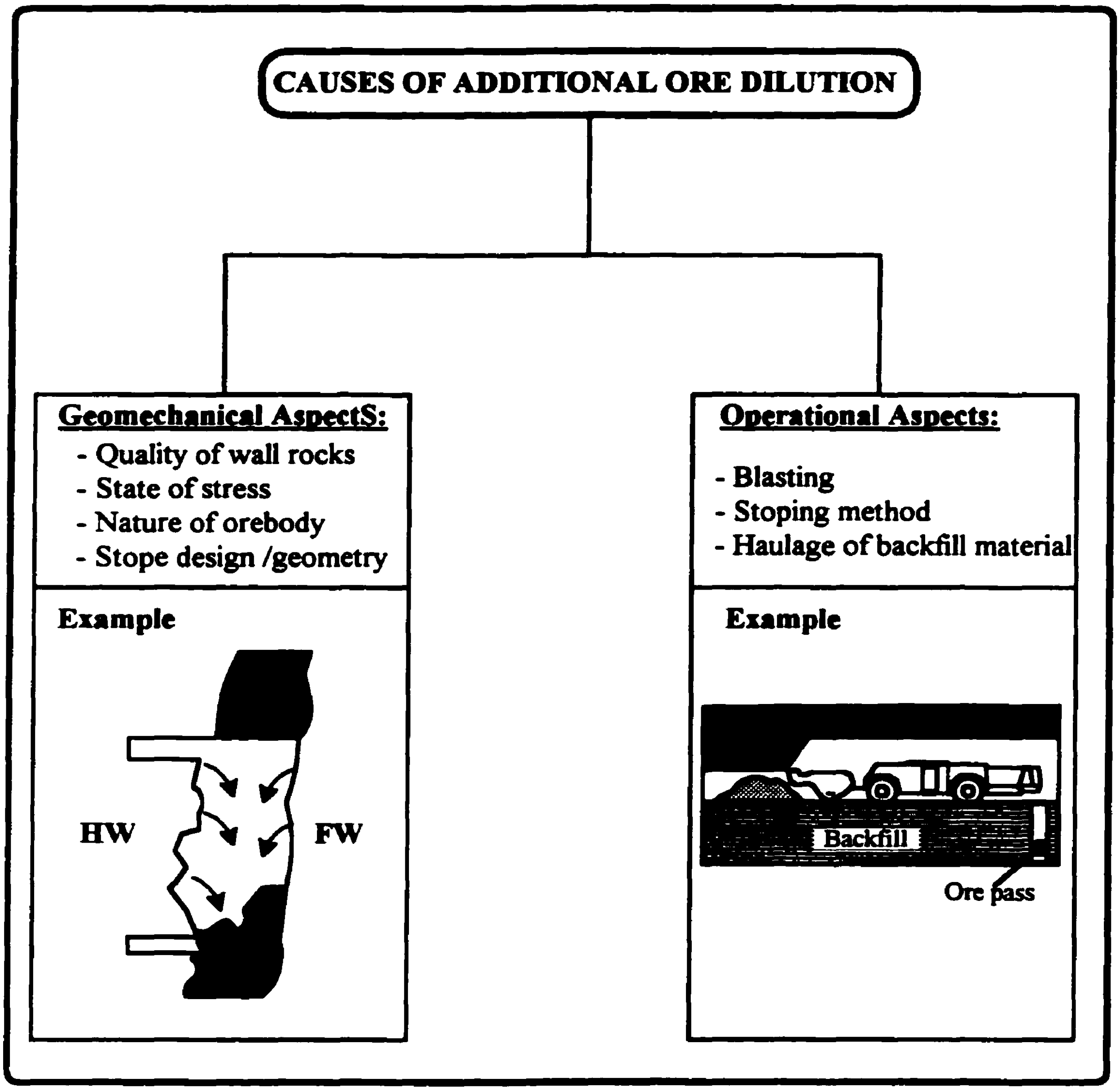


Figure 3.3 Causes of additional ore dilution (Planeta, note course, 1992).

For example, dilution may take place as result of hauling with backfill during the recovery of pillars adjacent to filled stopes, stoping adjacent to physical grade boundaries.

Knowing the tonnage of the mucked rock and its appropriate grade extracted is not simple. An oversight in orebody grade calculation is the omission of mineable grade and implicitly the mineable tonnage. In effect orebody estimation is based on geological basic information. Data provided by geological investigations give only the average grade of the whole orebody, and geological tonnage (calculated geometrically). Often the ore produced from mining process has, its own grade and tonnage values which are different from geological estimation. Mineable grade and tonnage include intentional dilution and are estimated by adjusting the reserve outline to reflect the material within an economic cut-off grade. Grade and tonnage found are imposed by the mining method and other parameters linked at them (e. g. rock mass quality, stope design, equipment, etc.). The effects of these parameters involve both dilution and reduced recovery rate. The ore recovery is linked to the lost of ore in the stope:

$$\%\text{Recovery} = \left[\frac{\{\text{Planned tonnes} - \text{Ore lost in stope}\}}{\text{Planned tonnes}} \right] \times 100$$

On the other hand, dilution is linked to the decrease of ore grade; the amount of material extracted will be larger than estimated to obtain the same equivalent metal content, and the grade of the run-of mine ore will be lower. This can be expressed by:

$$\%\ \text{Dilution} = \left[\frac{\text{Waste tonnes} + \text{Caving tonnes}}{\text{Planned tonnes}} \right] \times 100$$

For the economic aspects, ore dilution should not exceed 10 % (lower is better); while mining recovery must be ranked between 65 % and 85 %, depending on the type of deposit and the mining method employed.

In the geological context, the grade of the ore deposit is the value of ore taken inside the orebody; and the configuration of this (orebody) is found by considering the mineralized zone which has concentrations above the cut off grade of the desired metal. Hence rock material around the orebody can be completely waste or may have some metal concentration below the cut off grade. By definition, material below the economic cut-off grade will surround the economic mineable reserve. Material below cut-off grade will also be included within this reserve as non-mineralized or lower grade inclusions resulting from geological or structural effects. If this material cannot be mined separately, it must be included in the calculation as intentional dilution (see figure 3.2). Consequently final dilution (intentional dilution + unintentional dilution), with some concentration of desired metals, is not much worrying (rather some proportion). But for the archean vein orebody of Bousquet, volcanic rocks which are in contact with the orebody, do not have gold concentration; in this condition any unexpected addition of host rocks has considerable negative consequences. However, using cavity Monitoring measurement combined with a knowledge of rock properties, ground conditions, and previous experience with the stoping method, an estimate of unintentional stoping dilution can be made. Any grades attributed to such material would result from its anticipated source. Ore dilution due to waste rock caving from the hanging wall or other source may be added as a percentage of the designed economic mineable reserves or as a constant related to the stope width from footwall to hanging wall. The estimated unintentional dilution is added to the mineable reserve to give the total tonnage mined from a stope.

3.2 Factors causing ore dilution in sublevel mining

In sublevel mining, the nature of the orebody and several processes involved to the extraction of ore zones are sometime the sources of ore dilution. Dilution due to the nature of the orebody is called structural ore dilution. This dilution is inherent in the occurrence of the mineral deposit (Wright, 1982). Two examples are: - the presence of several barren formations within a deposit in such a way that selective mining is not possible; - in-situ leaching caused by penetrating underground water (usually the case with some copper and uranium ores). In vein and lense orebodies, structural dilution can be due to irregular ore/waste contacts, but may also, be due to the ripping of waste rock to provide sufficient working space in the stope (intentional dilution). On the other hand, production dilution is generally unintentional dilution and has two sources: - in stoping operation, wall waste rock can cave in the stope and dilute the broken ore; - and in filling operations, the mixing of the filling material with the broken ore. Production dilution is more unexpected, consequently, the understanding of the effects causing related ore dilution should be useful. There are mainly five effects leading ore dilution in sublevel stoping: the effects of ground conditions, effects of in-situ stress, effects of stope geometry, effects of geology and the effects of blasting.

3.2.1 Effects of ground conditions

In open stope mining the most important aspect of ore dilution control is the knowledge of ground response to mining. This implies the requirement of some knowledge of rock mechanic properties in compression, tension, and shear, under the numerous conditions of loading which may exist and also of the various environmental factors influencing the strength/stress relationship such as confinement, relaxation, and structural conditions of the rock mass. For their needs, mine operators could refer to the classifications based on these physical properties, rock mass quality, and evaluate the rock mass behaviour under ground stress conditions. Data of many type of rocks are available in rock mechanics literature. Particularly, the instability risks in Abitibian jointed rock, is hence due to poor rock mass quality (see table 4.1) while the wall behaviour is closely linked to joint systems and joint conditions. Among the rock mass properties which are most relevant to be considered are: elastic modulus, Poison's ratio, Uniaxial Compressive Strength, tensile and shear strength, cohesion and density. The rock in this felsic volcanic area has medium to high strength with Elastic Moduli in the order of 43 to 100 GPa and a Poisson's ratio ranging from 0.16 to 0.30. The local instability of rock around individual stopes are hence controlled by a combination of the rock properties and constitutive stress-strain relations.

The effects of near-field ground performance is controlled by pillar design, while the stope peripheral rock performance is controlled by the stope design itself (cf. Section 3.2.3). The near-field ground control is achieved by the development of load bearing pillar between the production stopes. The instability inside the open stope is usually provoked by unfavorable rock properties (low tensile strength, low cohesion, etc.) and unfavorable rock mass nature (structural discontinuities, soft rock, etc.). The rock property affect on stope wall instability can be understand using numerical modelling; while discontinuity approach is based on field observations. Hence, poor ground conditions mean inevitably the poor efficiency of pillar (transverse and longitudinal) left between the stopes. The rock mass displacement generated under poor performance of pillars can be the sources of rock instability along the hanging wall and the footwall. Then, the ore dilution coming from the wall caving could be linked to rock mass self-support capacity. The instantaneous weakness along the stope wall have its source in the failures of pillars which must sustain the imposed states of stress, and may result in the extensive collapse of the adjacent near field rock. If the volume of the unfilled mined void is high, the risk is that collapse may propagate through the pillar structure (Brady and Brown, 1985). Consequently, the design of stope and pillars is very significant in open stope mining (cf. 3.2.3); its design must take account of ground conditions in order to minimize dilution problems.

3.2.2 Effects of in-situ tresses

In Abitibian jointed rock mines, the vertical stress approaches the overburden load while the horizontal stresses generally exceed the vertical ones (Arjang and Herget, 1997). The majority of maximum and intermediate principal compressive stresses (δ_1 and δ_2) are aligned in a horizontal to sub-horizontal plane. The magnitude of principal compressive stress increases with depth (Figure 3.4a). The increase of vertical stress component with depth is shown in Figure 3.4b. While the ratios of the maximum/minimum horizontal stresses show a large scatter at shallow depth, such scatter decreases with depth exceeding about 1000 m (Arjang and Herget, 1997). The relationship between stress ratios and the configuration of mine openings suggests that most mining excavations are subjected to high stress conditions.

The instability associated with stresses, in open stoping linked to post-mining stress distribution. There are four stress-related factors which may influence ore dilution. those are:

1. Stress concentrations and relaxations around mine stopes;
2. Stand-up time of the rock mass subsequent to a production blast;

3. Magnitude and orientation of in-situ stresses;
4. Vertical stress due to gravity which is proportional to depth. The rock elasticity implying a resultant horizontal stress.

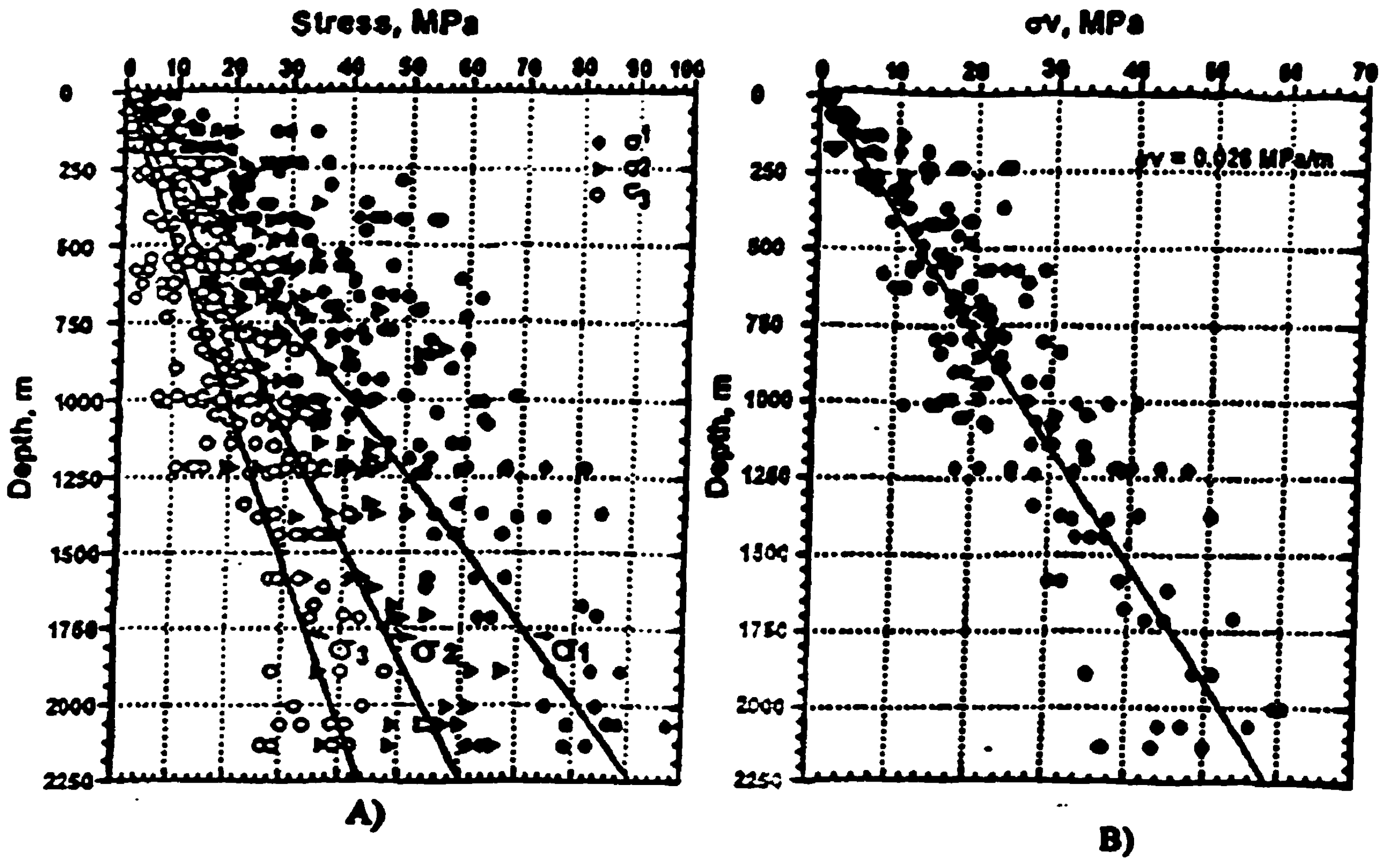


Figure 3.4: A) Increase of principal compressive stresses with depth; B) Increase of vertical stresses with depth in Abitibi shield rock (Arjang, 1997).

As can be anticipated, ore dilution problems due to stress conditions emanate mainly among other things from shear sliding failure along set of inclined joints (associated with relaxed zone); or rock crushing in the confined stress areas (stope back for example). Figure 3.5 illustrates the possible post-mining with the induced failures provoking both ore dilution and strainburst. Tensile stress associated with the relaxed zone produces induced failures as well as the opening of existing discontinuities (joints, faults). As the relaxation zones occur along the contact zones (hanging wall and foot wall), the leading instability causes ore dilution by hanging wall caving and foot wall sloughing. On the other hand, the compression stresses are concentrated at the roof and bottom of the stope as well as in the sill pillars. These confined zones are submitted to progressive or brittle deformation depending to the rock mass properties. Generally, in these zones, potential strainburst is caused by rock crushing or shear failure. Therefore, it is important to identify highly stressed and relaxed areas, so that mining engineers can evaluate the mining cycle and determine the best approach to stope sequencing, or use destressing or preconditioning techniques to relieve the stress.

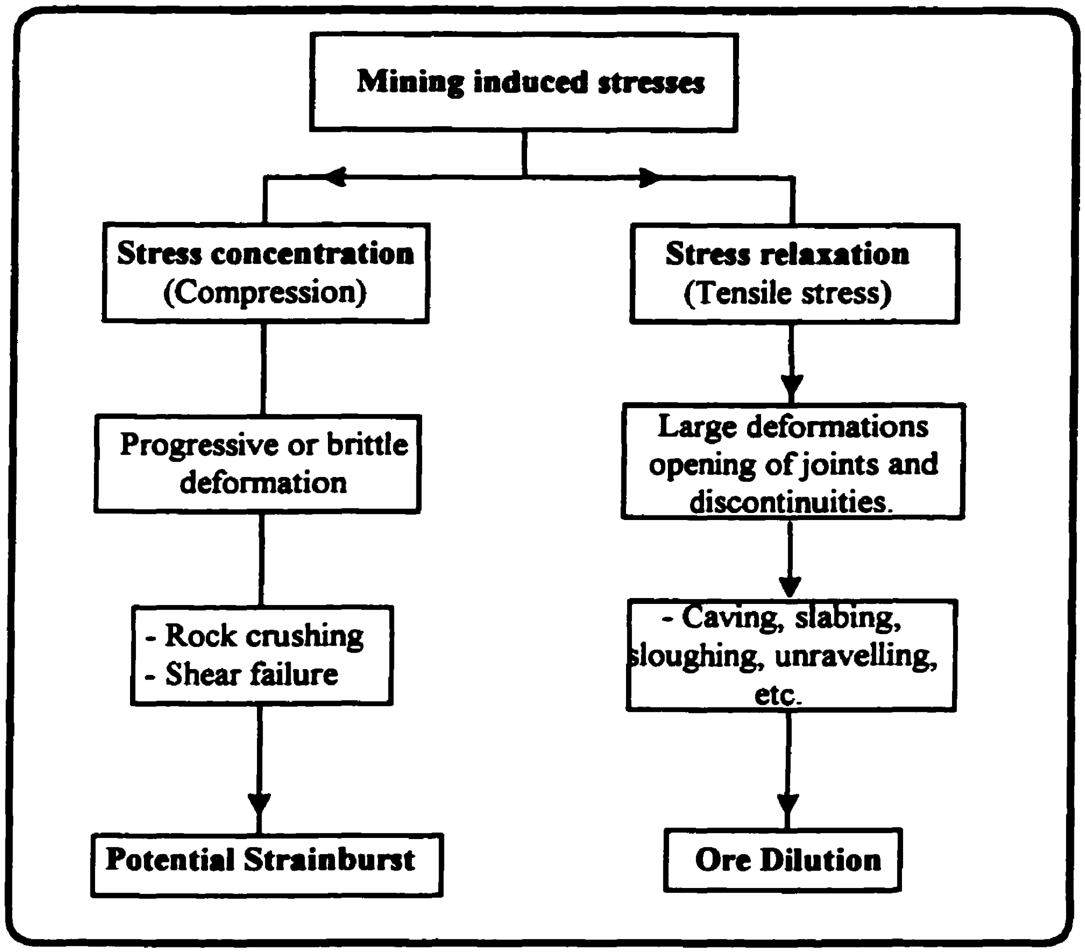


Figure 3.5: Effects of mining induced stress in open stope.

According to data on the joint systems provided by geological investigations, and those from the laboratory and in-situ tests on the joints and wall rock, the failure mode of shear sliding should be verified prior to the commencement of mining operation. to prevent the shear sliding it is suggested that when the joints have different shear proprieties, the minimum shear strength should be taken as the critical shear strength.

The process of waste rock crushing in the stope is mainly linked to the compressive failure. But this process occurs, usually in the crown pillars (e. g. in cut and fill mining). Thus, high compressive stress directly governs the behaviour of the rock mass.

3.2.3 Effect of stope Design

The stability of the rock mass at the boundary of the stope is closely linked to the stope design. The design of each stope must be optimized as a function of local rock mass quality. Larger individual stopes generally have high productivity however, at higher risk of instability. One of the useful design tools developed for the dimensioning of open stopes is the empirical 'stability graph method' originally proposed by Mathews et al. (1981); and later by Diederichs and Kaiser (1996) to determine stope critical dimensions (i.e., maximum), expressed in terms of the hydraulic radius of the roof and wall, to ensure stability of the excavation. The hydraulic radius (HR) of a stope face is calculated as the face surface area divided by the face perimeter. This stability of the open stope is expressed in stability number N' and illustrated in figure 3.6.

$$N' = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times A \times B \times C \text{ ; where}$$

RQD is rock quality designation (Deere and al. 1967);

J_n is the joint set number from the rock tunneling quality index, Q (Barton et al. 1974);

J_r is the joint roughness number (Barton and al. 1974) for the kinematically critical joint set;

J_a is the joint alteration number (Barton et al. 1974) for the kinematically critical joint set; and

A, *B*, and *C* are respectively parameters which are: rock stress factor, joint orientation factor and Gravity Adjustment Factor (see figure 3.7).

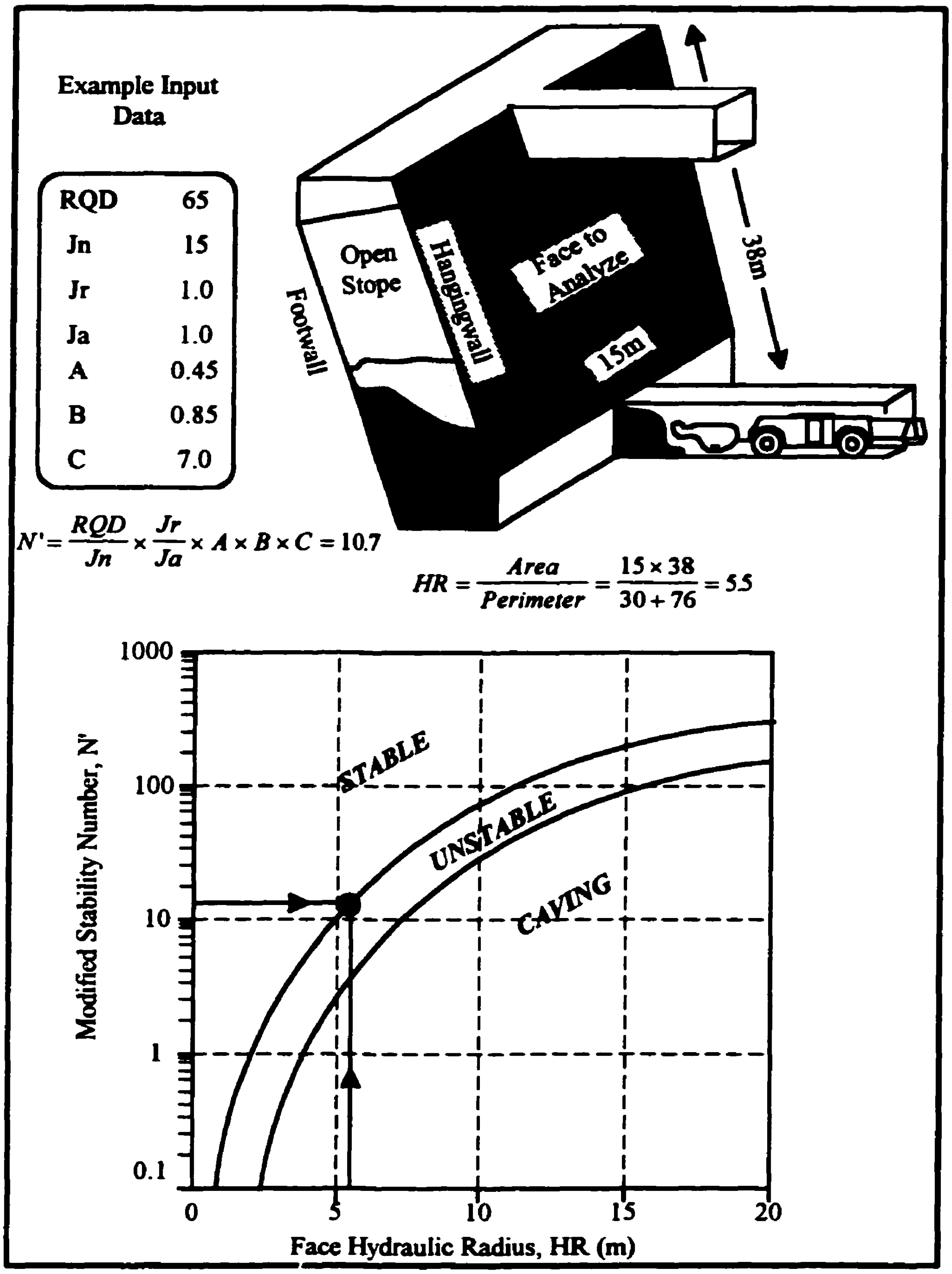


Figure 3.7: Stability assessment at Bousquet 2 mine using the modifie Mathiew stability graph.

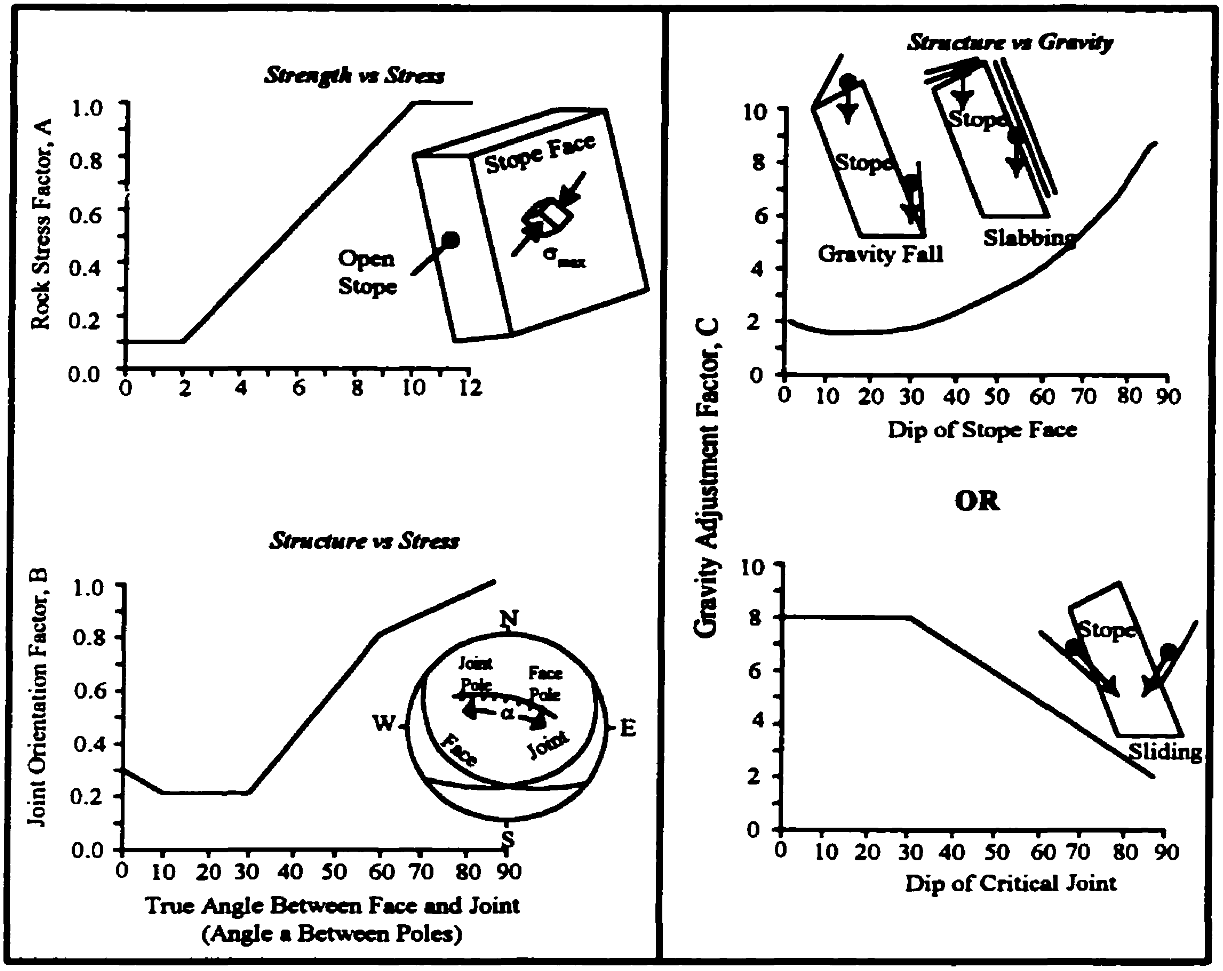


Figure 3.8: Manner to found input parameters for the design of open stope by graph method (Diederichs and Kaiser, 1996).

Consequently, the stability of an open stope is evaluated taking into account excavation geometry, rock mass quality, and induced stress. If the rock quality is too low or if the stope is too large (high HR), instability may occur, which leads to dilution of barren wall rock into the blasted ore. If the instability is severe, it can prevent the stope from being mined and prevent safe access to future mining areas. Using the above method and available data, it is possible to evaluate Bousquet's stope stability. At Bousquet mine, the stopes are excavated in a fair quality rock mass (RQD = 60) with four dominant set of joints (Jn = 15) which are smooth planar and unaltered (Ja = 1.0; Jr = 1.0). The sulphide ore has a compressive strength of 120 MPa., while the maximum induced wall stress determined from modelling is approximately 24 MPa in 1100 m deep. From figure 3.7 the

Rock Stress Factor is A = 0.45. The critical joint set forms an angle of 85° with the stope face (which is E-W), giving a Joint Orientation factor B = 0.2 and the Gravity Adjustment Factor C = 7.0. Hence, the resulting stability number N' of Bousquet's stope (at level 7) is :

$$N' = \frac{60}{15} \times \frac{1.0}{1.0} \times 0.45 \times 0.85 \times 7.0 = 10.7$$

The hydraulic radius, HR, of Bousquet stope can easily be found from figure 9. The stope analyzed is 15 m along striked and 38 m down dip. The resulting HR is:

$$HR = \frac{Area}{Perimeter} = \frac{38 \times 15}{76 + 30} \approx 5.5$$

The associated point (N', HR) plotted on the stability graph as shown in figure 3.6, is located at the boundary between the stable and the unstable zones proposed by Potvin (1988). This indicates that a hydraulic radius of 5.5 represents the maximum prudent unsupported dimension to avoid instability and ore dilution in this stope.

Expected approach of underground mine opts for large open stope (desirably unsupported). But according to Dierderichs and Kaiser (1996) probability studies, a stope design like at Bousquet mine (which have hydraulic radius of 5.5, and stability number of 10.7) will have a 35% probability of instability and a 20% chance of major caving. Ore dilution risk associated with these instability and caving for each stope could be intolerable. Because of this, significant effort must be made for the implementation of an efficient wall support system (section 6).

3.2.1 Effects of geology

Abitibi archean rock mass is not really intact rock; these are consolidated volcanic rocks, which are jointed and are affected by shear zones and by both major and minors faults provoked by tectonic evens. Consequently, the effects of geology on the stability are strongly linked to the nature and distribution of structural features within the rock mass. Joints are breaks of geological origin, when grouped in parallel are called joint set, and joint sets intersect to form a joint system. Joints may be open, filled or healed. Jointed hard rocks are subjected to wedge failure. This instability occurs when a set of joints strike obliquely across the stope face. The shear zones represent zones of stress relief in which fractured surfaces may be coated with low -friction materials (produced by stress relief process or weathering). Like faults, shear zones have low shear resistance but they may be much more difficult to identify visually. The faults are recognised by the relative displacement of the rock on opposite sides of the fault plane. They may be pervasive features which traverse a mining area or they may be of relatively limited local extent on the scale of meters; they often occur in echelon or in groups. Their thickness may contain weak materials such as gouge (clay), fault breccia. The ground adjacent to the fault may be disturbed and weakened by associated structures such as drag folds or secondary faulting. These factors result in faults being zones of low strength on which slip may readily occur. The spatial disposition and orientation (dip azimuths) of these discontinuities relative to the faces of excavations have a dominant effect on the potential for instability due to falls of blocks of rock or slip on the discontinuities. At Bousquet mine the main behaviour of the rock mass is controlled by weathered foliation. In fact, this rock is highly foliated and as seen in the table3.1, there are 4 different types of joints identifiable by their characteristics.

Structure	Dip Direction (degree)	Dip (degree)	Joint spacing (m)
D1 (foliation)	185	85	0.01 to 0.1
D2	95	45 to 85	0.5
D3	270	20 to 60	0.5
D4 (sub-horizontal)	345	5 to 25	1

Table 3.1: Typical joint sets at Bousquet mine (Henning, 1997)

Instability causing dilution problem is also preoccupying in soft and sedimentary rocks. Soft and soil rocks are very weak and are very heavily jointed or broken, as in a waste rock dump; hence the dilution is provoked by circular failures which are defined as single discontinuities but tend to follow circular failure paths. On the other hand sedimentary rocks are formed by several strata separated by bedding planes. These are generally immediate failure planes because on these planes shear resistance would be purely frictional.

3.2.5 Effects of blasting

High production extraction in hard ore requires (at the present) high energy of explosive. This blasting mechanism produces vibration energies, a major part of which fragments surrounding rock material while a residual part travels beyond the far zone before it finally drops below the background noise level. In hard vein blasting, like at Bousquet 2 stopes, the vibration level can be determined by observations (by camera and/or monitoring systems) of damage along the wall faces. Under fair or poor ground wall conditions, the scale of blast damage could be translated into induced failures, and even sometimes, wall rock overbreak (which means ore dilution). To keep the mine profitable, blasting process needs to be performed in a more efficient way to minimize ground and induced structural vibration. On the other hand, the dilemma is that the power must remained high enough for maintaining good the fragmentation product; because the size of blasted muck will control ore haulage. Therefore, blast engineer must refer on the maximum and minimum ground vibration velocity for many combinations of interhole and interrow delays. The spectrum of the blast sequence is the result of Fourier transform. Effective delay times for each blast hole within multihole blast can be calculated by the formula:

$$T_i = \frac{\sqrt{(X - X_i)^2 + (Y - Y_i)^2 + (Z - Z_i)^2}}{V} + Tp_i$$

where

X,Y and Z are coordinates of the vibration monitoring station

V is the propagation velocity of the seismic waves,

Tp_i is the cumulative nominal (pyrotechnic) delay time, and

T_i is the cumulative effective delay time.

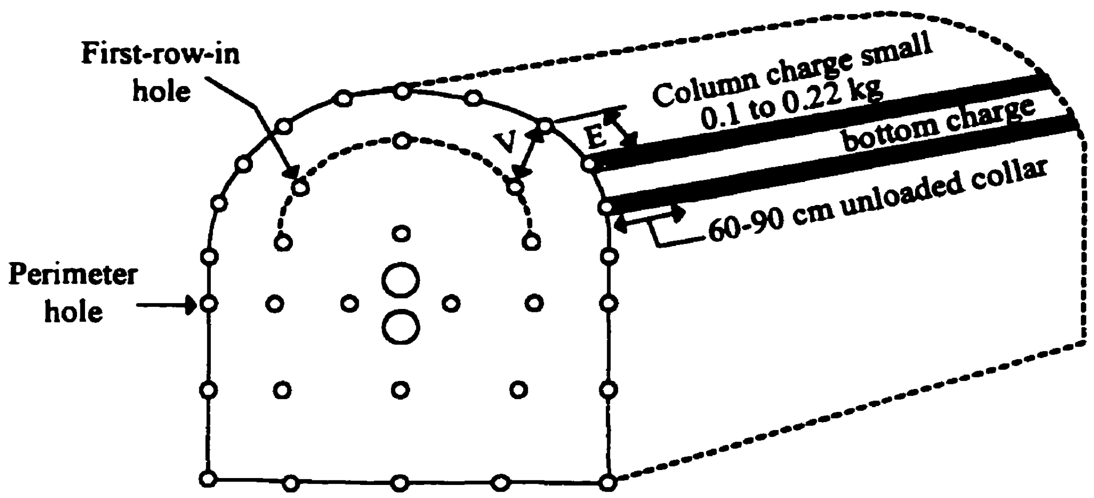


Figure 3.9: Design of smooth blasting pattern (McKown, 1986).

Minimization of blast vibration is restricted by the production requirements, such as good fragmentation and muck pile size. This means short interhole and interrow delay times are not suitable. On the other hand, lower blasting energy condition, the rock fragments remain too large for efficiency haulage; while highest blasting energy can produce large waste slough in the stope. In this way, it is important therefore to predict likely size and the risk of oversize sloughing from the wall. An other option in dipper veins blasting is the stabilization (by pre-installation of cable bolts) of the hangingwall and footwall, and then increase the blast vibration energy to obtain the desired fragment size. In archean jointed, it almost impossible to achieve a perfect design of the stope (minimum dilution); the fragmentation is controlled by the distribution of jointing in the rock mass. Consequently, the control of ore dilution will refers to modify the type of explosive or the placement of particular explosives such that the damage zone created by the perimeter holes does not exceed the damage zone produced by the holes within the perimeter. This process is called smooth wall blasting or controlled blasting largely experimented by

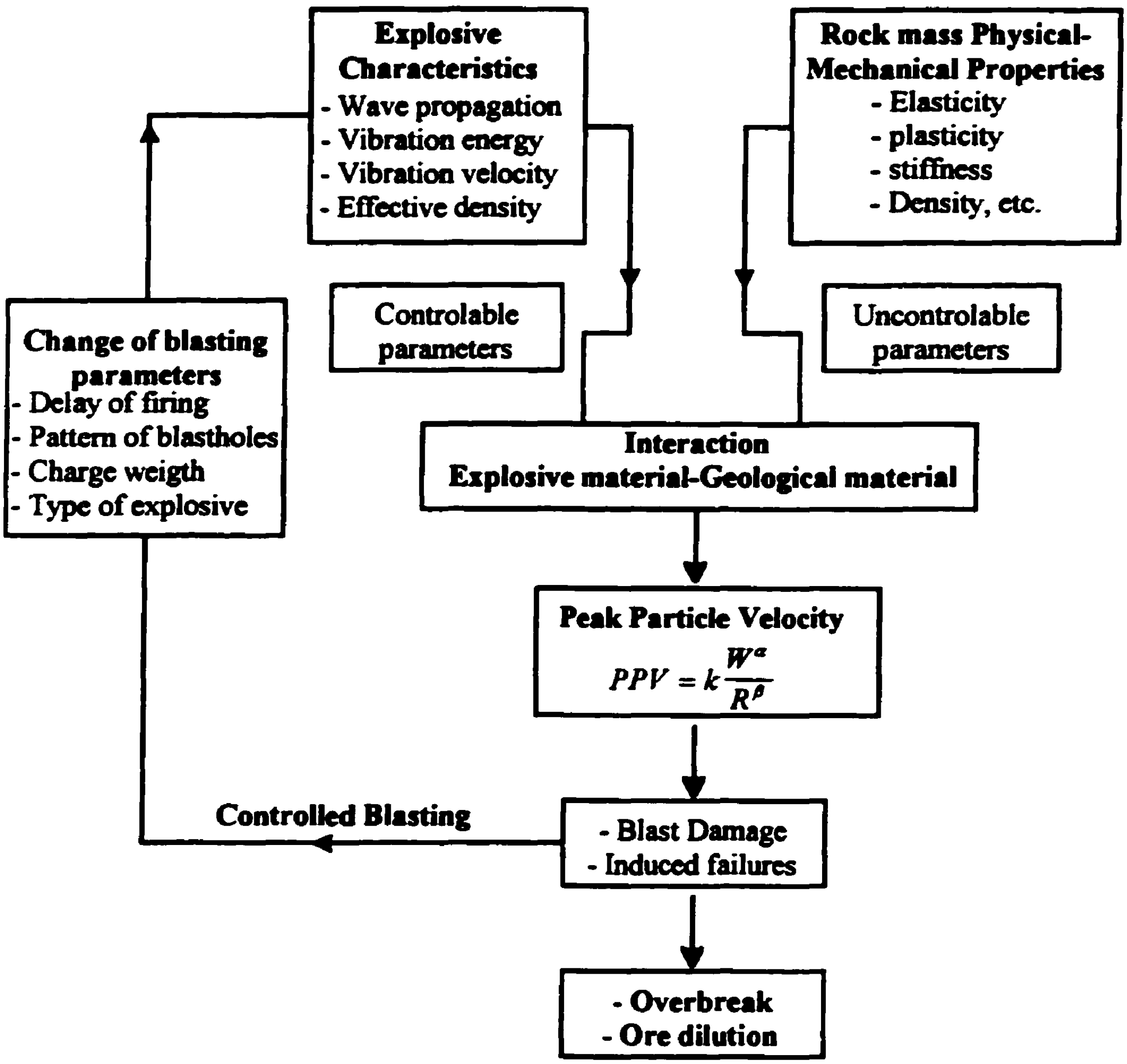


Figure 3.10: Assessment of controlled blasting design against ore dilution caused by overbreak in sublevel open stope.

CAMNET in drift development, Lizotte (1996). The drift for experiment, shown in Figure 3.8, has about 2.4 m by 2.4 m as face dimension with 36 to 43 holes drilled to a depth of 2.4m. Even original objective is to assess a suitable controlled blasting techniques for underground development drift, the damage criteria used could be validly adapted for the control of dilution. There are direct relationship between actual shape of the opening and the level of blast vibration. To understand that, it require to monitor the blastholes. For the case of the Canmet experiment, the blastholes were monitored with a multichannel blast vibration monitor, to which was attached sets of high-frequency geophones and accelerometers. The blasting damage level is linked to both the rock mass physical-mechanical properties (elasticity plasticity, etc.), and its structural components (Joint, foliation, etc.) as well as the power of explosive used. See Figure 3.9. Under fair or poor ground wall conditions, the scale of blast damage could be translated into induced failures, and even sometimes, wall rock overbreak. Controlled blasting is considered a better way to handle this problem. The blasting of the stope involves two aspects: The explosive Characteristics (Wave propagation, vibration energy, vibration velocity, effective density) and the rock mass physical-mechanical properties (elasticity, plasticity, stiffness, density etc.). The interaction between the explosive material and the geological material creates a Peak Particle velocity (PPV) which puts in evidence the behaviour of the rock mass under given blasting parameters, then the blast damage and induced failure are assessed based on the PPV, and the reflective tensile wave. In most cases, the overbreak causing dilution is caused by these blast damage and induced failures. However, it is possible to reduce this problem by controlled blasting. Controlled blasting also called perimeter blasting results from various combinations of blasting design parameters. It involve the following components (single or in combination): modification of the firing sequence; the use of different explosives; modification of the explosives placement procedures; and the modification of the geometry of the volume where the explosives are placed. Also the blasting parameters could be changed (firing delay, blasthole pattern the effective density of the explosive, type of explosive. Generally a suitable result is found after various combinations of these parameters have been tried out. Naturally, inelastic rock material has brittle behaviour under dynamic strain of the explosive. Because of the non-homogenity of geological material, the deformation is rather plastic and the limit of such behaviour correspond to the zone where the strain level has exceeded the strength of the rock mass. From the classical elasticity Hooke's law (linear defomation $= \frac{\delta}{E}$), Holmberg et al. (1984) have demonstrated this deformation is proportional to the strain of the explosive:

$$\varepsilon = \frac{\delta}{E} = \frac{v}{c}$$

where

ε is the strain

δ is the stress

E is the young's modulus of elasticity of the rock.

ν is the vibration velocity, and

c is the wave propagation velocity.

The problem of controlled blasting is considerably complicated; most people focused their attention only on the explosive parameter, forgetting the rock mass itself. However, as is evident from the above relationship, the rock mass physical-mechanical properties plays a great role in controlled blasting. Thus rock mass can be considered damaged when it no longer reacts elastically and the deformation is plastic (Lizotte, 1996).

At Bousquet mine, several triaxial geoghones are used to measure the vibration caused by blasting. The measures have been done for 2 isolated stopes (one is primary stope while the other is a secondary stope) in level 9. This study was made in 1997 by Bousquet mine engineers (J. Henning, P. Gauthier, M. Ruest and C. Provenche). The amplitude of blasting vibration was determined using the field constants for the mine and the quantity of explosive loaded in the blastholes of 9-1-15 and 9-1-11 stopes. The formula of PPV have been used to determine the constants "k" and "a" which depend on the structural and elastic properties of the rock mass. "k" is an indicator of scale of the charge; and "a" is an indicator of vibration attenuation. The amplitude was found to be lowest in the secondary stope than in the primary stope (table 3.2). That is not surprising because there is highest attenuation of vibration at the relaxation zone in the hanging wall.

	Primary stope 9-0-15		Secondary stope 9-0-11	
	K	a	K	a
Total database	498	1.19	498	0.70
Data for AMEX only	283	0.66	283	1.30

Table 3.2: Results of the blasting vibration indicators obtained (Henning et al., 1997).

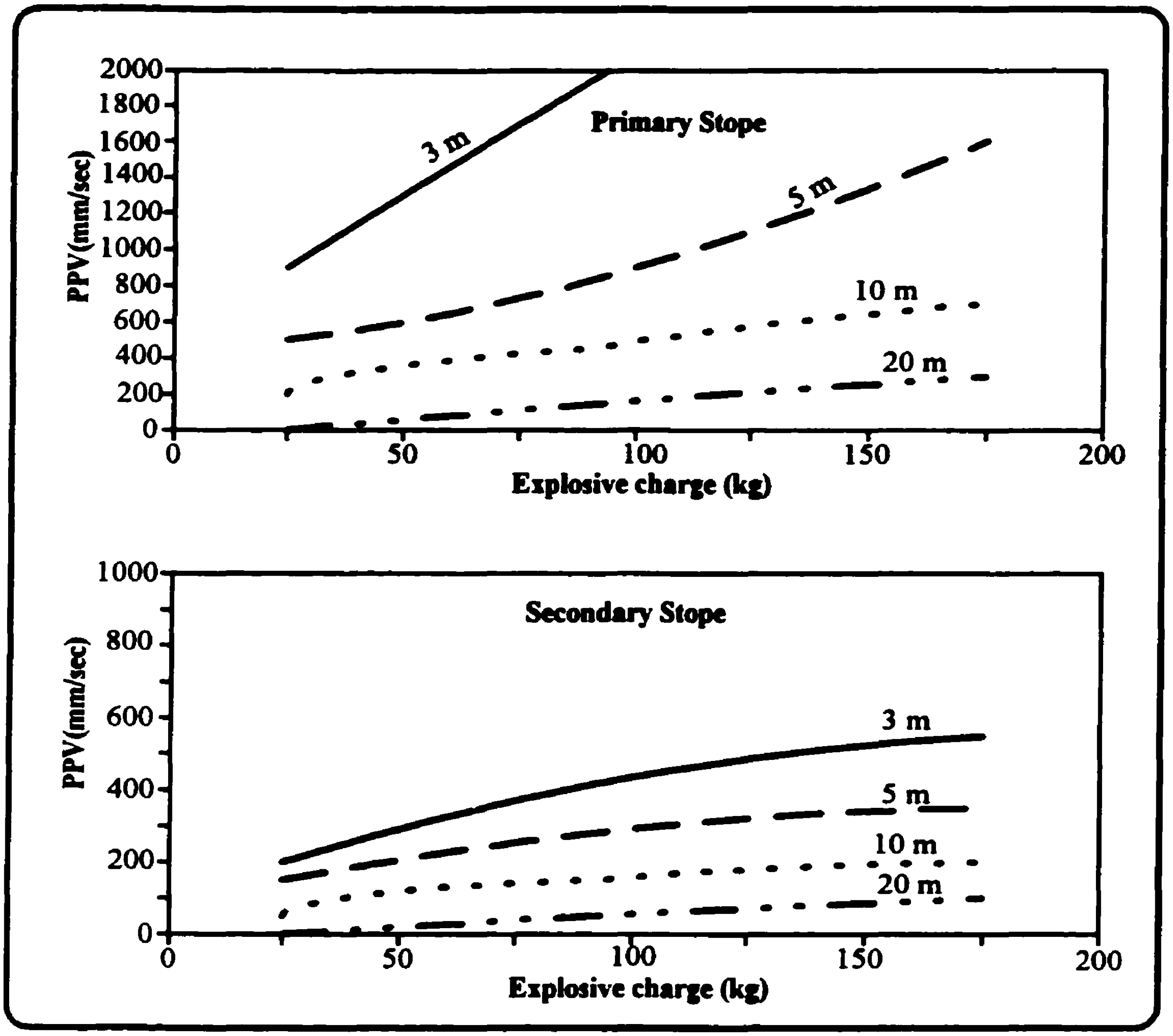


Figure 3.10 Blast vibration in the hanging wall at Bousquet stopes (J. Henning et al., 1997,).

	Maximum linear overbreak	
	Hangingwall	**Footwall**
Primary stope	4.0 m	0.5 m
Average	3.1 m	0.5 m
Secondary Stope	2.0 m	1.5 m
Average	2.2 m	1.0 m

Table 3.3: Results of overbreak measurement in Bousquet stopes hanging wall (J. Henning et al., 1997,).

In Figure 3.10, the curves of explosive quantity used versus maximum blasting vibration predicted, show the effect of these constants (k and a) calculated for primary and secondary stopes (Henning et al., 1997,). About the first five meters around the blasthole, vibrations higher than 1000 mm/sec have been measured for the primary stope. Vibrations calculated for the secondary stope are relatively lowest. This is certainly one of reasons of higher ore dilution in primary stope. To compare the stope overbreak volumes, equivalent linear overbreak factor is used. This is determined by calculating the area between the planned stope boundary and the actual geometry of the stope. As can be noticed from Table 3.3. the primary stope is affected by more overbreak volume than the secondary stope.

Therefore, a good approach for reducing ore dilution is to understand the explosive/rock mass interaction. This means good knowledge of both rock mass and explosive characteristics, and the design of optimal fragmentation blasthole pattern. The challenge of successful controlled blasting remains still to minimize damages associated with the explosive/rock mass interaction while to optimize the fragmentation produced by explosive energy.

Chapter 4: Numerical Modelling

4.1 Introduction

The presence and the distribution of failures which are responsible of ore dilution must be extensively considered before any measure taken against the problem. Any underground opening create varied degrees and magnitudes of stress in both the near and far field area. Thus, the stoping operation being the essence of mining process, put in evidence the rock mass stability behaviour within the orebody and in the hosted rock. This chapter describes the response of stope boundary conditions under effects of stress redistribution in open stoping at Bousquet mine. Numerical modelling techniques such as e-z tools is proven to be effective in the analysis of these types of geomechanical problem. This software of numerical modelling (created and developed at McGill University by. Mitri, 1992) using the finite element method to divide a representative zone into an assembly of discrete, interacting elements (meshes); therefore, to define problem domains surrounding an excavation (a stope for our case study). The mechanical behaviour of rock mass is evaluated in term of displacement, deformation, while, the stresses and the strains in the vicinity of the opening.. As discussed by Brady and Brown (1985), the assumption in such method is that transmission of internal forces between the wedges of adjacent elements is represented by interactions at the nodes of the elements. This geomechanical study is used to analysis the effects that a large open stope has on the stability of the stope walls and on the local stress field. Therefore, the main objective is improved understanding of ore dilution involved by wall instability behaviour under fair ground conditions.

The stope site selected for the numerical modelling analysis is located about 1 156 m under surface. Figure 4.1 shows a two-dimensional view of this stope. Material data such as rock mass characteristics and properties and in situ rock stress are obtained from previous studies carried out by Henning et al., (1996). In order to achieve a realistic analysis of the behaviour of the hangingwall and footwall, the influence of structural geology is used to provide detailed information.

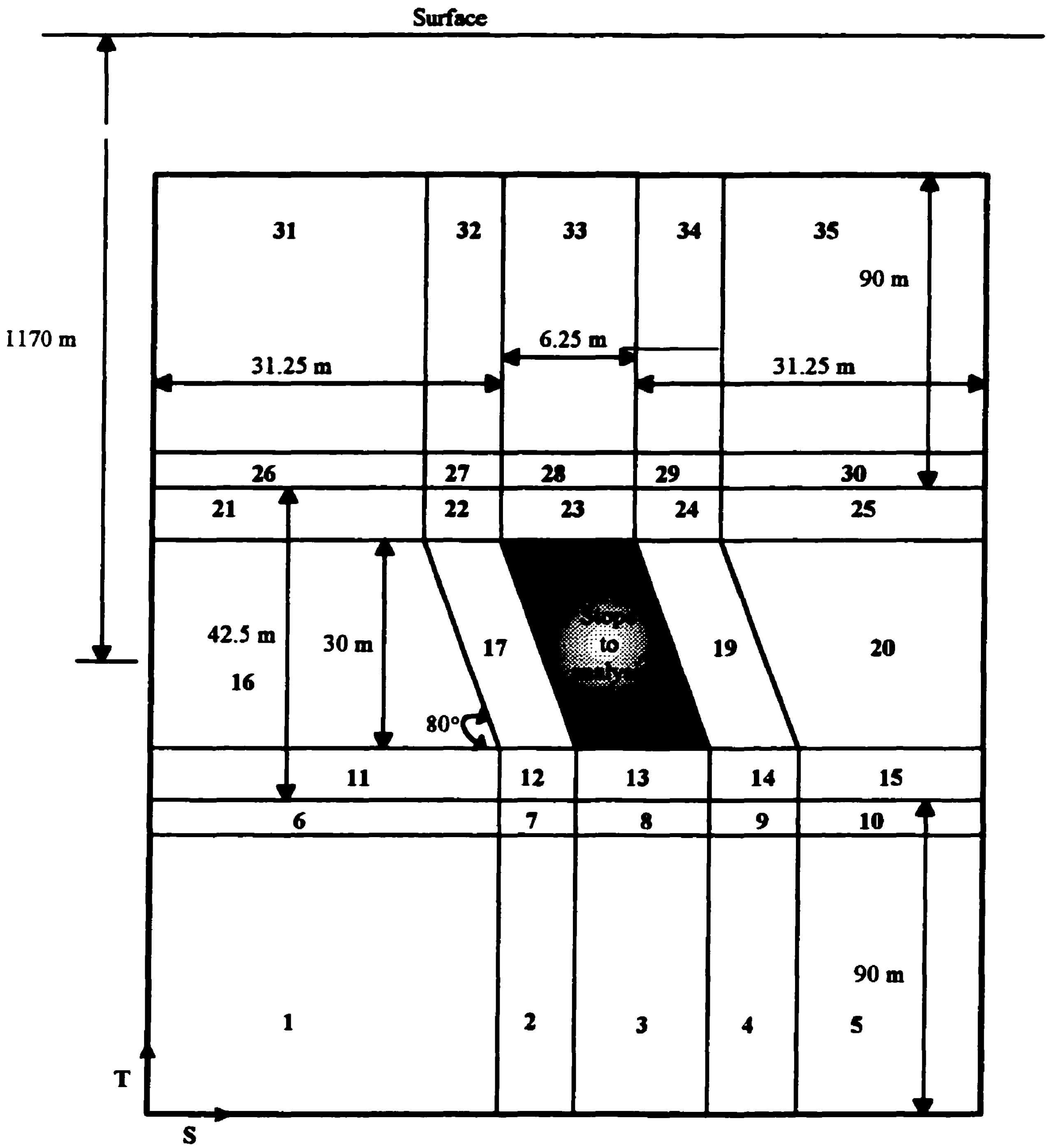


Figure 4.1: Final sketch of zone model for e-z tools modelling

4.2 Geomechanical data

The numerical analysis methods are mainly based, in addition of in situ stress, on rock mass quality and properties provided by geomechanical data. In the others words, any numerical analysis is not possible without a earlier assessment of rock mass classification of the concerned field. Thus, the rock mass classifications have been assess to evaluated its quality and its behavioural characteristics as a whole under various stress conditions. The strength and failure (displacement and deformation) characteristics of the rock is governed by the characteristics of the naturally occurring fracture systems that occur in the rock mass. The rock quality is assessed from its containing in discontinuity (RQD) and associated parameters such as compressive strength, joint spacing, joint condition and orientation, and ground water conditions (RMR and "Q System"). Rock properties are assessed from laboratory tests providing rock strength values and deformation properties. The rock mass classification studies at an area like Bousquet involves firstly, investigation at two levels or scales: The identification and mapping of geological structures that can affect stope wall behaviour. Therefore, the starting point for the development of an engineering understanding of the rock mass structure is a study of the general regional and mine geology as determined during exploration. The information founded (such as bedding, foliation, folds etc.) constitute an essential background of geomechanical study of the mine. The second level is further studies involving underground exposures, and logging core of boreholes (RQD and RMR) drilled for this purpose. This gives precise locale ground quality. The investigation for such study had been made in the level 7-0 at Bousquet 2 mine, the result is located in the Appendix A. The rock mass rating (RMR) and the Q-system show a fair wall condition (see table 4.1 below), whereas the orebody is relatively good quality.

	Hangingwall		Massive ore		Footwall	
Bieniawski	RMR	GSI	RMR	GSI	RMR	GSI
1976	50	50	65	65	55	55
1989	50	45	66	50	59	50
Modified Q'	1.00	44	1.00	44	1.00	44

Table 4.1: Rock mass quality modefied Hoek & Brown criteria, Henning et al., (1996)

The knowledge of pre-mining stress nature and their orientation is fundamental to coherent prediction of the response of the rock mass to mining activity. For dilution problem, the attentions are focused on the displacement field generated in the boundary of the stope and eventually in the stope previously backfilled. At Bousquet the pre-mining ground stresses have been determined at 900 meters depth. This had been done at Elliot Lake Laboratory by B. Arjang (Canmet 1988). The results have indicated that the maximum and the intermediate principal compressive stresses are horizontal and are oriented in northeast and southeast directions. The minimum principal compressive is oriented vertically. The orientation of the principal stresses relatively to the existence of the particular fabric element of the rock mass such as foliation or schistosity, can also help to understand the direction and the mode of deformation under stress conditions. The orientation of the principal compressive stresses and the overall trend of the ore zones are illustrated in the Figure 4.2. The average horizontal compressive gradient amounted to 0.0505 MPa/m, with a ratio of $\sigma_{Hmax}/\sigma_{Hmin}$ in the range of 1.5. In relation to the ore zones, the σ_{Hmax} acts perpendicular while σ_{Hmin} is oriented parallel to the ore zones strike and the main structural features. As the in-situ stresses are function of depth, they can be determined as following:

$\sigma_{Hmax} = 0.061 * D$

$\sigma_{Hmin} = 0.041 * D$

$\sigma_V = 0.027 * D$

where σ_V is the vertical stress; and D is the depth below surface (m). The stresses are expressed in MPa.

Rock mass properties required for input parameters used in numerical modelling are determined from laboratory and in situ measurement. These values are supplied by mine Engineers. Note that fractures may be dominant influence on the strength of the rock mass. Table 4.2 gives the elastic and Hoek & Brown parameters of Bousquet rock mass. Uniaxial compressive strength (UCS), young's modulus and poisson's ratio are obtained from uniaxial testing of BQ size core. Under uniaxial testing almost all the ore samples underwent brittle failure. This is characterized by a sudden release of energy in an explosive manner. The results of these tests are adequately supported by previous research by Wawersk and Brace (1971), Hoek and Brown (1980), Jaeger (1960).

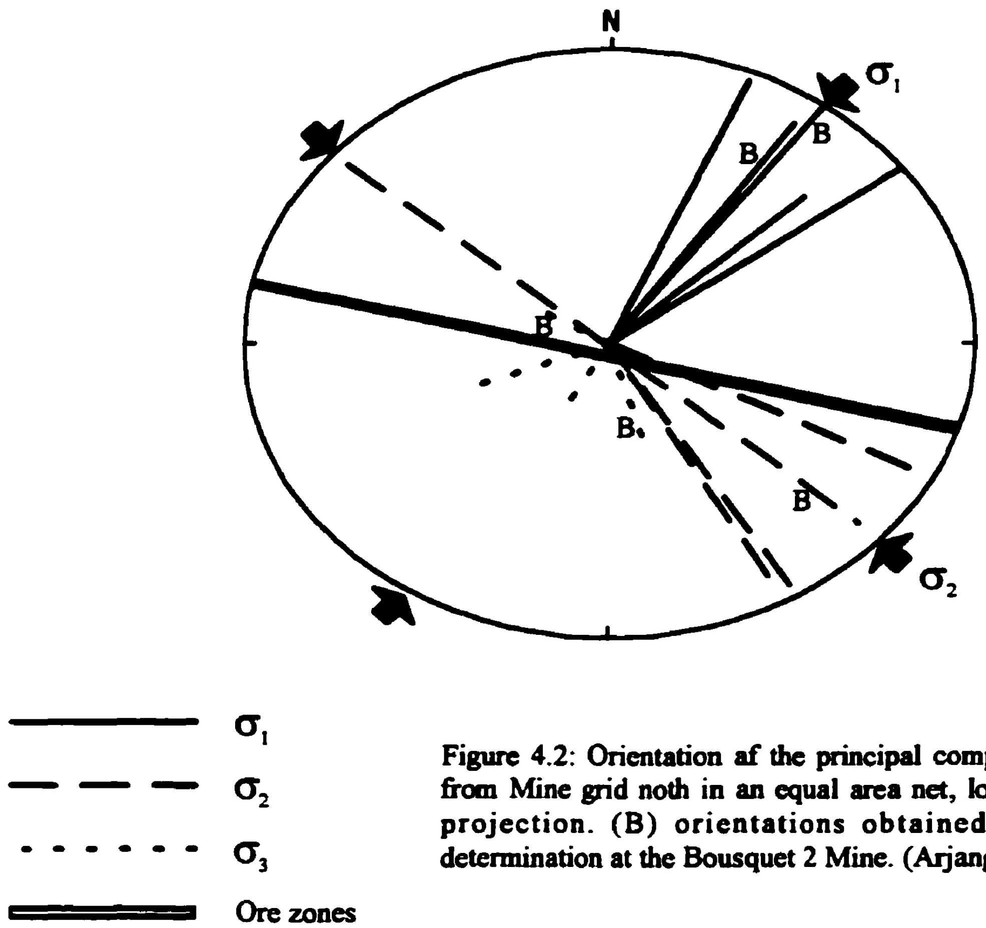


Figure 4.2: Orientation af the principal compressive stresses from Mine grid noth in an equal area net, lower hemisphere projection. (B) orientations obtained from stress determination at the Bousquet 2 Mine. (Arjang, 1988).

4.3 Statement of Problem

The adequate approach of efficient modelling involves two requirements: 1) A better choice of numerical modelling techniques which can provide high resolution of ground behaviour around the stope and near field rock, in accordance with stope design and the structural elements of the rock mass. Sometimes to avoid a choice, several numerical methods could be used in conjunction, allowing for the advantage of each method and providing a comprehensive analysis from different perspectives. 2) A better choice of a stope which represents the design and rock nature of any stope in the mine. The respect of these recommendations are fundamental for getting results that could help to define the type of behaviours which are responsable of failures and caving. Numerical modelling is carried on a typical stope mined at Bousquet 2 mine. This stope is located about 1155 m under surface. Design parameters are shown in the sketch zone model (Figure 4.1). Data concerning rock mass characteristics as well as rock mass properties used in modelling, are described in geological conditions section. The dip of the orebody measured from plan is about 75° west. The in situ geomechanical data are shown in table 4.2; while the horizontal-to-vertical stress ratio (K_x = 2.26, K_z = 1.52) are obtained from previous publication. Datafile permitting running of the program of numerical modelling (e-z tools) is available in annexe page.

Parameter	Hangingwall	Footwall	Sulphide Ore
Young'modulus	70 000 MPa	70 000 MPa	80 000MPa
Poisson's ratio	0.25	0.25	0.25
Shear modulus	28 GPa	28G Pa	32 GPa
Bulk modulus	43 GPa	43 GPa	53 GPa
Density	2 750 kg/m^3	2 750 kg/m^3	3 000 kg/m^3
Compressive strength	80 MPa	80 MPa	120 MPa
H/B m	2.4	2.4	3.4
H/B S	0.007	0.007	0.02
Joint friction	20 degrees	20 degrees	20 degrees
Joint cohesion	1 MPa	1 MPa	1 MPa
Joint tensile strength	4 MPa	4 MPa	5MPa

Table 4.2: Rock mass properties at Bousquet mine, resulting from uniaxial and triaxial tests.

4.4 Analysis of results

The analysis of the results is the main reason of the modelling. Many information are produced from such analysis. Then, data received are used to explore on stress conditions around the stope, and understand specific ground failures conditions, therefore to assess relevant resolution. The procedure used for solving the problem is to divide the concerned failure zones into smaller components in order to get numerical value for whole behaviour. The parameters permitting the examination of rock mass behaviours are evaluated in term of stres redistribution and stress displacements vector. For stope design purpose, the existence of both stress concentration induced in rock mass and the failures around the stope is closely linked to the stope geometry and size. For our dilution study, the location of stress relaxation zones are taken as failures, therefore, are more important since these zones are submitted to shear stresses which can provoke block sliding or local caving (tension failure). Otherwise, the zones of induced stress concentration area can be affected by crushing in major case by bursting (compressive failure),. The presence of other close openings and the effect of progressive mining sequence should be taken account for the both cases.

To have good analysis, 64 nodes have been necessary to create 49zones for the need of modelling. Among these 49 zones, 4 are void zones that consist of a longwall stope and 2 drifts of production and 1 cablebolt drift. Mesh can be seen in figure 4.3. This mesh is dense near the drifts as well as the hangingwall and the footwall, to have much measurements in these critical parts. The principal stress (Figure 4.4) is concentrated at both roof and bottom of the opening. The open stope mining method used at Bousquet 2 implies a redistribution of the in situ stress field in two mains zones: a relaxation zones in which an increase of strain energy is stored; and fields where the stress are concentrated. The maximum principal stress, located at corners, is about 243 MPa. The figure 4.5 and 4.6 show vertical and horizontal stress respectively. Like the principal stress, maximum values occur at the roof and the bottom, in the corners. However, in these figures it possible to identify the tensional zones along the hanging wall and the footwall. The maximum vertical stress is about 127 MPa while tensile force is about 14 MPa. Maximum horizontal stress is about 192 MPa.

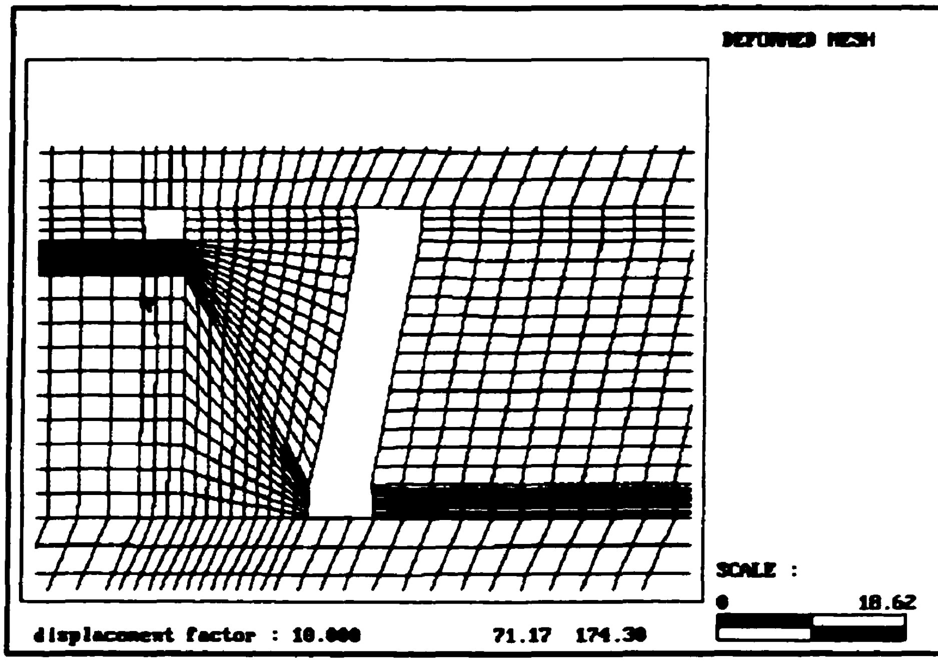


Figure 4.3: Deformed mesh for the purpose of modelling.

While there are relaxation of stress along hangingwall and footwall; then after, as the distance increases from horizontal and vertical axis of the stope, compressive stresses decrease. Displacement stress vectors multiplied to factor 15 is view in Figure 4.7. Hangingwall and footwall stresses are oriented toward the void. this is due to the greater mass of the wall materials. As the distance increase from the stope, the displacement stress vector decrease. The numerical modelling results include also material safety factor (Figure 6.2), that determines the condition of the material under its state of stress and relaxation. It can be seen that near the opening hangingwal and footwall are considerably unsafe.

4.5 Interpretation: Stope wall behaviour

Extraction by longhole open stope method like in Bousquet mine, causes instantaneous and continuous stress redistribution in the surrounding strata. The ability to deal with structural stability problems in bulk mining and associated dilution and risks to mine personnel is highly dependent on understanding stress redistribution during different stages of mining. As dilution is provoked by slip and caving of waste rock, it can be possible to create links between unexpected loss of grade in ore mucked and stress distribution and rock mass quality.

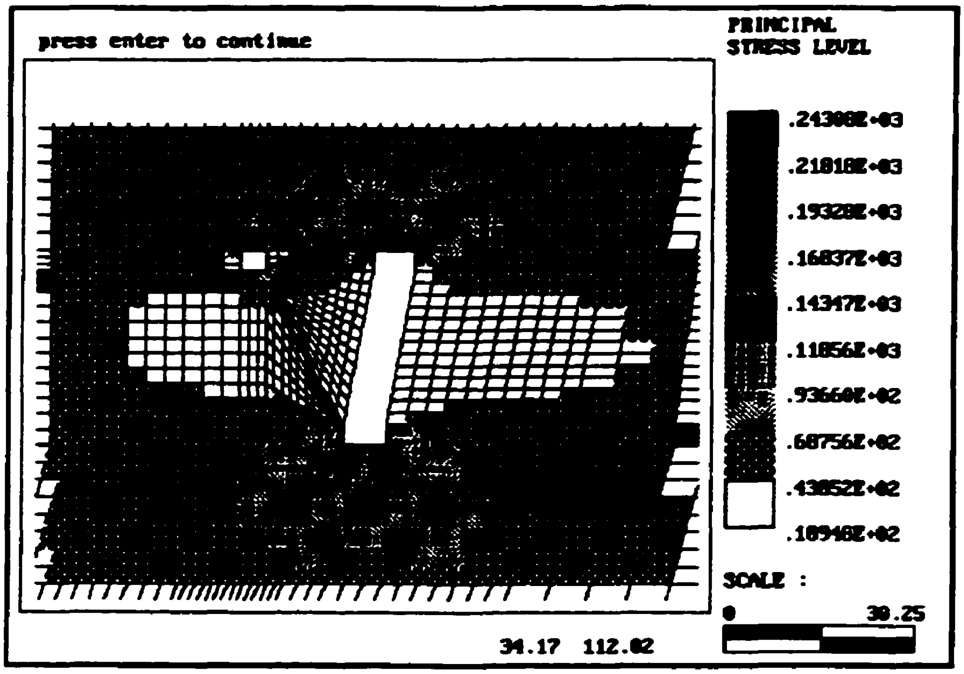


Figure 4.4: Principal stress distribution around the stope.

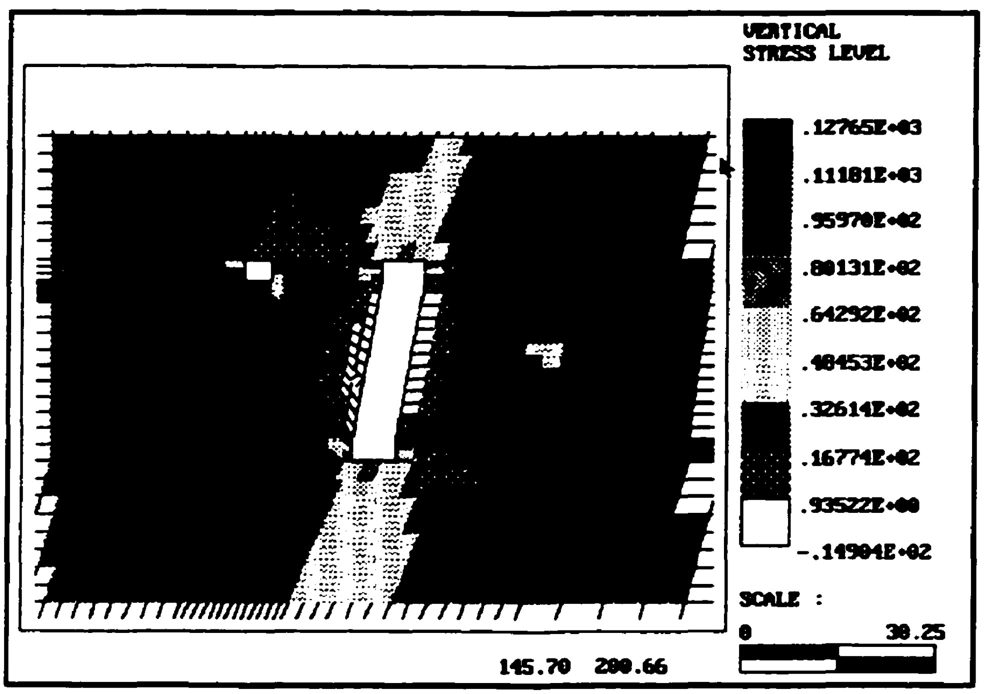


Figure 4.5: Vertical stress distribution around the stope.

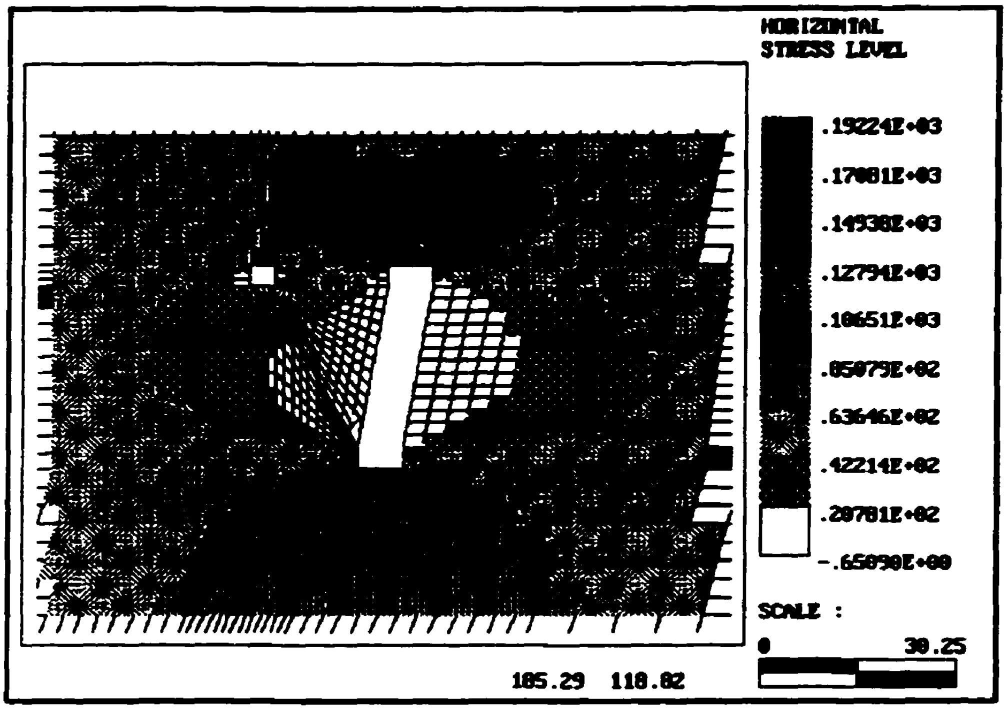


Figure 4.6:Horizontal stress distribution around the stope.

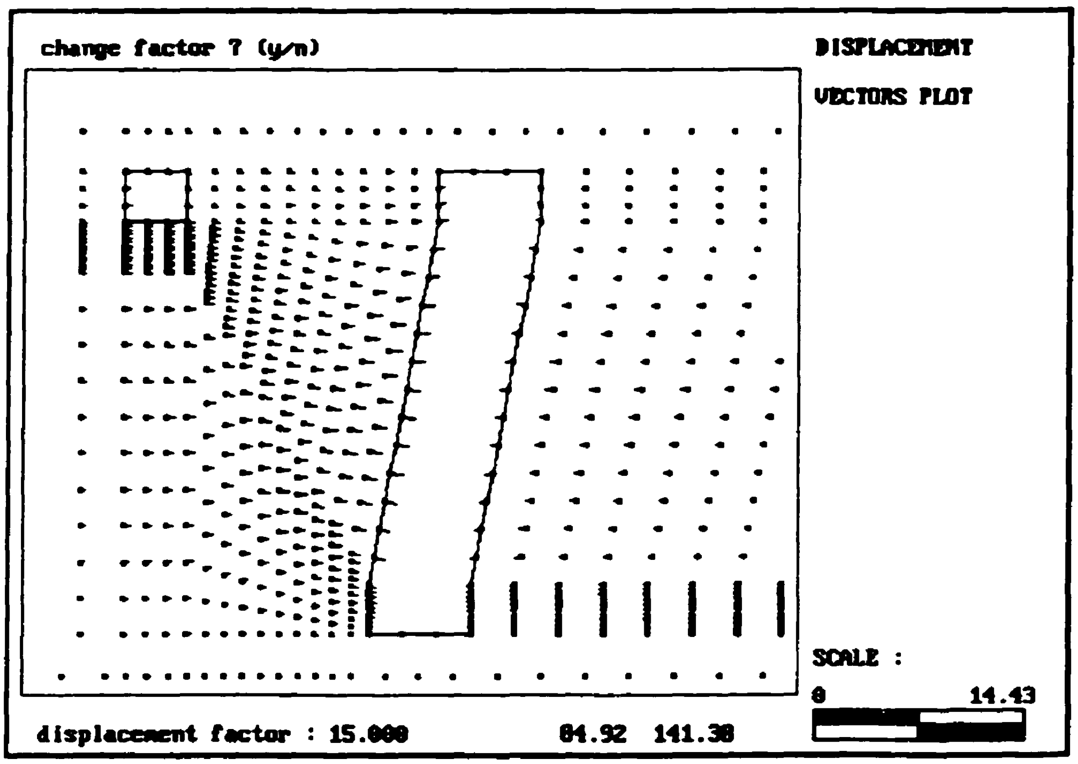


Figure 4.7: Displacement vectors multiplied by factor 15.

Independently to the mining method and operation, poor ground conditions and stress/shear are most likely to induce instability leading immediate caving. In this manner many factors directly contribute to unexpected dilution. These are confined high stresses which are concentrated in the corners of the stope, high shear stresses which are extended along the walls, and geological discontinuities. combination of these three factors are necessary for understanding of geomechanical source of dilution at Bousquet mine. At the boundary of the stope, the zone of relaxation along the hangingwall and footwall (Figure 4.5 and 4.6), potential rockburst can occur during blasting. As seen in table 4.1, the mafic tuff of hangingwall is relatively weak quality, in act, relaxation along this can provide much waste material in ore mucked. the three factors mentioned above allowed to say that there are two modes of rock mass behaviour leading to rock mass instability. One mode involves crushing of rock occurring close to stope boundaries. The second mode involves slip on natural or mining-induced planes of weakness. In fact, instability is provoked both to structural anomalies and induced stress; near surface excavations are significantly influenced by structural feature, while instability of deep stopes depend more on the response of the rock mass to the induced stress. A current notion is that unstable discontinuity slip occurs because the dynamic coefficient of friction for the discontinuity (joint, fracture) is less than the static value (Brace and Byertee, 1966). Thus, excess shear stress on a joint or a foliation plan may be used an indicator of the potential for unstable slip. Having identified state of stress in the host rock mass favourable to unstable discontinuity slip, the issue then is the measures to be implemented to prevent or reduce the moment of the event. The response of rock joint to shear loading in situ depend to a large extent on its surface properties as well as the boundary conditions that are applied across the joint surfaces. The boundary conditions can be represented by assuming that the the rockburst of the rock mass surrounding the joint is modelled by a spring with stiffness $k = d\delta_n / dv$ where $d\delta_n$ and dv are the changes in joint normal stress and displacement, respectively (Seb and Amadei). Also fractures can form from slip along pre-existing joints. As slip increases along these pre-existing fractures, secondary fractures form and link the slipping fractures to form major fractures. A high density of fracturing develops adjacent to the slip surfaces and increases unstable behaviour of the wall.

The mechanism of stope ore dilution closely results from the fundamental behaviour of wall rocks. This mechanism of instability provoked by a discontinuity had been largely explain by Brady and Brown in their rock mechanics book (1985). Then, the presence of discontinuities within the stress relaxation areas (situated along the middle part of the wall) involve shearing and splitting, and in the worst case the caving of wall waste rock inside the stope. The mechanism of unstable slip on a single discontinuity plane has been firstly considered by Rice (1983) and re-explain by Brady and Brown (1985). Otherwise, in strong and intact rock the dilution problem remains relatively minimum. But this situation still rare in Abitibian area. However, intact rock under high confined stress conditions, can be subject to crushing behaviour. In the both cases, the mechanism governing the behaviour of the stope faces stills the opening and sliding of the existing and/or induced discontinuities along and close the boundaries of the stope. Then, the behaviours of stope during mining means to assume the behaviours of discontinuities in the analysis of large scale opening. In this instance, Yoshida and Horii (1997) has developed a constitutive model for analysis of discontinuity behaviours in rock mass, called micromechanics-based continuum (MBC). Initially proposed for civil engineering (tunnelling), this theoretical analysis could be adapted in underground stope mining, specially in jointed rock area like in Abitibi mining district; because it attempt to asses relationship equations between average stress and average strain taken account a representative volume of element which contains a lot of microstructures.

The numerical results give the whole behaviour of rock mass during mining process, and are expressed in displacement in rock mass (provoked by the effect of dominant joints), joint deformation opening displacement. This later could be the reason of waste sliding in the stope, because it is initiated by the stress relaxation and stress redistribution due to stope opening.

4.6 Conclusion

Numerical modelling has demonstrated that Bousquet mine stopes are confronted with instability problems. These are due both to manner of stress redistribution and poor quality of wall rocks. Redistribution is expressed by confinement of high stresses in roof and bottom corners of the stope; while relaxation and shear stresses occur along the hangingwall and the footwall. A major consequence of this induced instability is a high risk of unexpected dilution of ore mucked. From combination of geomechanical data and structural anomalies, it has been identified that more waste material can slide from the hangingwall.

The mechanical behaviour of the stope wall is difficult to accurately predict. It is governed by induced stress redistribution (relaxation, confinement), rock mass properties (Young's modulus, Poison's ratio) and rock mass quality and nature (rock mass stiffness, joint sets orientation and distribution, joint effective length and quality). However, it possible to adopt a numerical method to analysis the effect of some parameters quoted on this behaviour. An other possibility is to adjusted the rock mass parameters in function of a proposed numerical analysis from the field failure observations; the inverse in difficult to assess but much relatively tangible for the control of stability.

Chapter 5: Measurement and Control of Ore Dilution

The estimation and measurement of ore dilution takes place after the stope blasting operation. Two methods of measurement of ore dilution can be used by mine operators: the theoretical calculation and the direct estimation by the cavity monitoring system. The purpose of this chapter is to define an optimal strategy of stope extraction, integrating both the geological, geometrical, technical and economical aspects of stope operations.

5.1 Theoretical calculation

The use of the theoretical approach to estimate ore dilution is based on the data provided by geochemical grade analysis of both the ore and the barren rock at the contact with the orebody, or the knowledge of the parameters of the designed stope and the real stope. The approach of the desirable result on the performance of the particular stope is mainly qualitative; that means the evaluation of the grade of ore after mining operation. Other criteria being the recovery and the selectivity of the mining method used. There are many theoretical formulas which can be used for dilution estimation; we propose here some of these. Laval University (Bourgoin, et al., 1991) have made investigations on dilution equation formulation in the setting of its search and development project. The use of one or another formula depends on the availability of parameters and its adaptability to the mine stope design. In effect, dilution can be estimated at different steps of the mining process by the following equation:

$$d_i = \frac{g_{pi} - g_{fi}}{g_{pi}} \times 100 \qquad (5.1)$$

where: $g_{fi} = g_{pi} \times (1 - 0.01 d_i)$

d_i: dilution of ore at the i^{th} phase (%)

g_{pi}: grade of the ore at the entry of i^{th} phase (% or g/t)

g_{fi}: grade of the ore at end of the i^{th} phase (% or g/t).

The knowledge of ore dilution at different phases of the mining process is important. However, the solution of this problem requires knowledge of the percentage of dilution at each step of mining process; and to take the measures to reduce it at this step. The final ore dilution which considerably influences the actual profitability of the mine operation, can be determined by two ways:

i) Using orebody reserve grade and the final grade of ore:

$$d_f = \frac{g_r - g_f}{g_r} \times 100 \qquad (5.2)$$

where d_f: final dilution of mucked ore at the entry of concentrating process (%);

g_r: grade of orebody reserve (% or g/t);

g_f: final grade of the mucked ore at the entry of concentrating process (% or g/t).

ii) Using the values of "n" different sources of ore dilution (Bourgoin, et al., 1991):

$$df = \frac{10^{2n} - (100-d_1)(100-d_2)\dots(100-d_i)\dots(100-d_n)}{10^{2n-2}} \qquad (5.3)$$

where d_1, d_2, d_i, dn are different sources of ore dilution;

n: number of sources of final ore dilution.

There are usually two sources of final ore dilution (n = 2): planned and unexpected ore dilution. Therefore, the previous equation becomes:

$$d_f = d_1 + d_2 - 0.01 d_1 d_2 \qquad (5.4)$$

When the situation is perfect i. e. there are the addition of only the planned dilution (which is far from being realistic in vein stoping), n = 1, the final ore dilution is equal to the planned ore dilution.

$$d_f = d_1$$

The uncertainty of the detailed shape of the orebody and difficulty to delineate the ore in place (for some mines) make even the planned dilution difficult to estimate. However, it is not acceptable to extract the material that you don't need. All the parameters of the material extracted must be absolutely known for each individual stope of the mine. These parameters are the tonnage and the final grade of the ore. This latter must be known and controlled. There are two graphs (Figure 5.1 and 5.2) established by Bourgoin, et al., (1991) which can help mine operator to not be disagreeably surprised. As it can be note there is an inverse correlation between final dilution and the grade of ore mucked. A small percentage of dilution considerably reduces the final grade of the ore. If the grade of the orebody is, say, 2g/t, at 25% of dilution reduces the final grade to 1.5g/t. Also it is possible to calculate the dilution having planned and unexpected dilution by following equation:

$$d_f = d_p + d_u - 0.01 \times d_u \times d_p \qquad (5.5)$$

where

d_f: final dilution,

d_p: planned dilution,

d_u: unexpected dilution.

In most cases the final dilution is greater than the planned dilution (that can be easily observed by the change of ore qualitative and quantitative parameters). That means that the final dilution hides the unexpected ore dilution which is mainly associated to the stability of the wall rock. Even when the unexpected dilution (additional dilution) can be estimated beforehand, it is possible to calculate the parameters (tonnage and grade) associated with the material added to the ore.

$$g_f = \frac{(t_r \times g_r) + (t_a \times g_a)}{t_r + t_a} \qquad (5.6)$$

where: g_f: final grade of ore extracted;

t_a and g_a are respectively the tonnage and the grade of material added;

t_r and g_r are respectively the tonnage and the grade of orebody reserve.

Generally, the waste rock has no desirable metal content ($g_a = 0$). Thus it become easy to find the tonnage of material added.

$$t_a = \frac{(t_r \times g_r)}{g_f} - t_r \qquad (5.7)$$

Sometimes, the rock added to the ore has little quantity of desirable metal ($g_a \geq 0$). In this case its tonnage is estimated and its grade is calculated.

$$g_a = \frac{(t_r + t_a) \times g_f - (t_r \times g_r)}{t_a} \qquad (5.8)$$

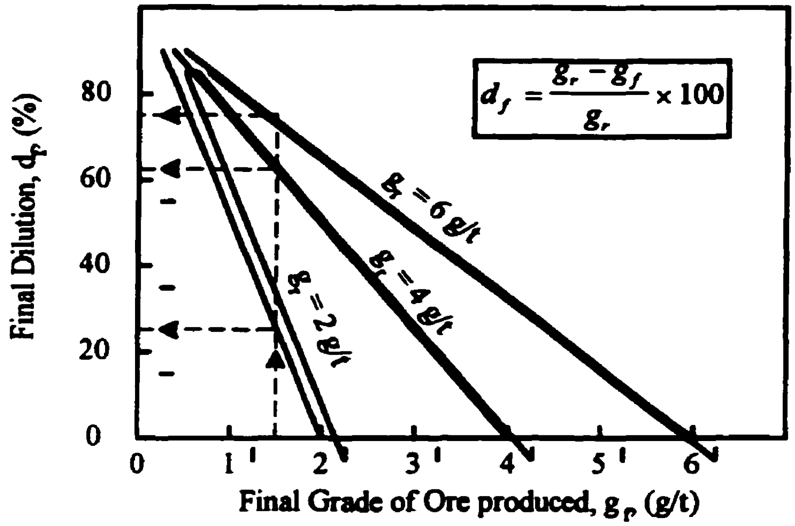


Figure 5.1: Influence of Dilution on Final Grade of Ore extracted (Bourgoin et al., 1991).

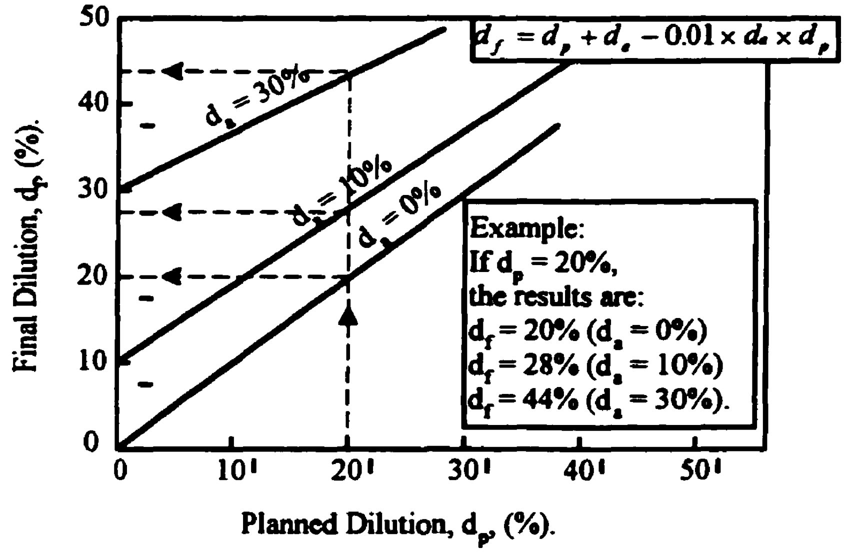


Figure 5.2: Relationship between Planned Dilution (d_p), Additional Dilution (d_a) and Final Dilution (d_f) (Bourgoin et al., 1991).

The addition of no waste rock reduces effect on the final grade of the ore extracted. Therefore its addition in the ore can be tolerated rather some proportions ore/waste established by mine operators. For that, the operational and economic impacts of planned dilution must be evaluated. From the values such as planned dilution, as well as orebody and waste rock grades (g_r and g_a), it becomes possible to evaluate:

- The tonnage of waste rock provoking dilution by mineable orebody tonnes. This is called the factor of rock provoking dilution (FR).

$$FR = \frac{0.01d_p \times g_r}{g_r(1-0.01d_p)-g_a} \qquad (5.9)$$

when the rock provoking dilution is waste for desirable metals (ga = 0), this factor become:

$$FR = \frac{0.01d_p}{1-0.01d_p}$$

- The whole tonnage (ore + waste) by mineable orebody tonnes. That is known as factor of planned dilution (FP):

$$FP = 1 + FR \qquad (5.10)$$

Consequently, the mineable tonnage must be determined in adjusting the tonnage of orebody reserve according to both the ore recovery (OR) and the factor of planned dilution (FP). Then the adjusted reserve tonnage (including planned dilution) is:

$$Ta = 0.01 \times OR \times t_r \qquad (5.11)$$

- The tonnage of waste rock causing ore dilution is

$$ta = Ta \times FR$$

- The tonnage of ore mucked To(ore + waste rock causing dilution) is:

$$To = Ta \times FP$$

- The grade of ore (g_o) after planned dilution is:

$$g_o = g_r(1-0.01d_p) \text{ (\% or g/t).}$$

On the other hand, as seen in the Figure 5.2, the final dilution is directly correlated to the additional ore dilution. That means the unexpected addition of waste rock is completely independent from the planned dilution, since the latter is only associated with the width of planned stope.

The effects of unplanned addition of waste rock is expressed in terms of decrease of ore quality. Using the results of previous investigations and of other mines, it should be possible to estimate the rate of dilution. However, because of uncertainty knowledge of all the local associated parameters and their implication it becomes impossible to predict the rate of final dilution. The theoretical approach nevertheless, should help in a preliminary estimation of ore dilution; and usually is adopted before the extraction process begins.

5.2 Practical measurement

Mine operators opt to manage their own production system. In effect, it will mean the implication of parameters that can be controlled in place. As the main source of ore dilution is linked to the geomechanical and design characteristics of the stope, it should be reasonable to use these characteristics in dilution estimation.

5.2.1 Background

The design characteristics of the stope can be used to determine the planned ore dilution. First, it is expressed by following quantitative relationship:

$$d_p = \frac{g_r - g_o}{g_r} \times 100 \qquad (5.12)$$

where g_o is the grade of ore produced (% or g/t). This grade can be calculated using the dimensions of the stope (Figure 5.3)

$$g_o = \frac{g_r \times w_b + g_w (w_s - w_b)}{w_s} \qquad (5.13)$$

where

g_w is the grade in metal of the waste rock causing dilution (% or g/t);

w_s is the width of the stope opened (m);

w_b is the width of orebody (m).

To express the dilution as function of stope design parameters, we can introduce above equation in the expression of planned ore dilution.

$$d_p = \left[\ 1 - \frac{w_s}{w_b} - \frac{g_w (w_s - w_b)}{g_r \times w_s}\ \right] \times 100 \qquad (5.14)$$

In most case the grade of the waste rock is nil or negligible (gw =0). Then the planned dilution is:

$$d_p = 1 - \frac{w_b}{w_s} \times 100$$

As can be noticed, the ratio of stope width/opening of the orebody (w_b/w_s) is significant in the assessment of planned dilution. The production management requires a strict minimum planned dilution. That means the smallest possible opening. For that, the narrow orebody should be mined using hand-held air leg drills, scrapers, and/or rail-mounted loaders. These methods are manpower intensive and while productive enough to support a small operation are not sufficiently flexible or cost effective to support a larger operation. But if the ore has high grade or is concentrated in sub-products (Au, Ag, Cu Zn etc.), it should be economical to use specialised equipment. In effect, now there are small scale equipment for orebodies under 3 meters wide. However, for large scale mining (mechanized method), the drift sizes must exceed 4.5 m width. But large scale mining for narrow vein (width at least that 3 meters) means the acceptance of high dilution ratios. Such methods are typically used on even modest deposits while gold price is high. The ideal performance is found when the vein width is so large that it is not necessary to include barren rock in the design of the stope: $\frac{w_b}{w_s} \geq 1$.

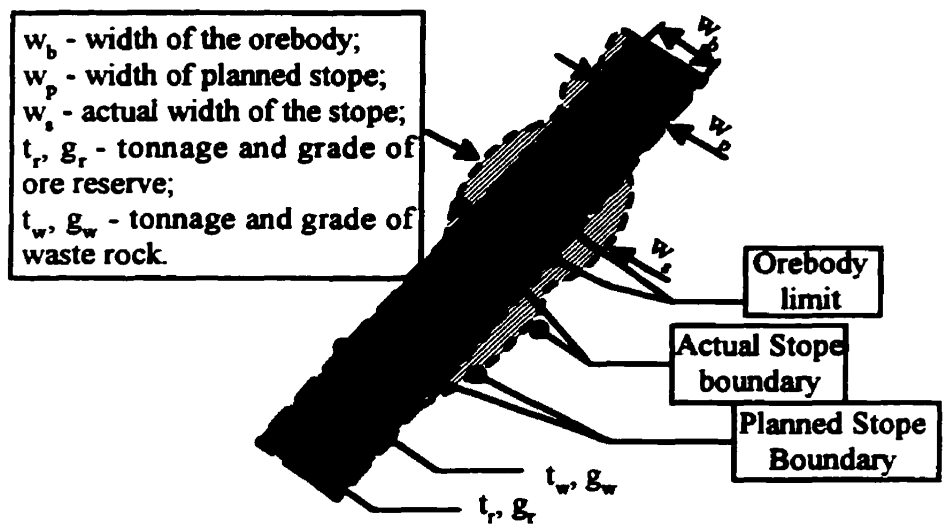


Figure 5.3: Illustration of parameters used for ore dilution estimation.

5.2.2: The Cavity Monitoring System and measurement of open stope

The laser light sectioning measurement technique was first applied in 1969 to mine tunnels by Chrzanowski and El Masri. Later it was used in civil engineering tunnels to detect tunnel deformations over the time. The technique permits to capture the image on videotape and then to enhance and evaluate it on a microcomputer. Now the light measurement technique is improved and thus has become much more suitable for the measurement of stope overbreak. In Europe, the prototype used is called Rig originally constructed by Scott, now improved by Camborne School of Mines (Wetherelt and Beer, 1996). It consists of a camera fixed to a tube at the end of this section. The rods to rotate the light source are mounted on the upper surface of the boom. The second section is a 2 m length of boom with a rotation rod mounted on the upper side. In the cavity (or the stope), the light source is rotated slowly using the handle at the end of the rotating rods and a long exposure picture is taken. Then the light section profiles are taken at various intervals depending primarily on the stope advance or where there is overbreak or underbreak that requires quantification. In Canada, the Noranda Technology Centre and

Optech System Inc. have created a prototype called Cavity Monitoring System (CMS) which is efficient in stope measurement. This new tool provides to operators significant solution against dilution problem in open stope. The CMS consists of a laser scanning device that can be inserted into any cavity or stope and rotated in three planes to accurately survey the entire stope. The first laser system ("Optech G150 Laser Rangefinder") was successfully tested at Gaspe Mines in 1990. This prototype is actually used by several mines in Canada which are confronted with ore dilution problem; the better example study of using the CMS is done at Golden Giant Mine by mine operators (Anderson and Grebenc, 1995). After blasting, the opening area remains risky and must be splitted and carefully tested against rock caving. However, in sublevel mining the access for men to the stope is strictly forbidden; thus measurement device (CMS) is used, after ore mucking, from the upper level of the stope. The Optech laser is mounted on a set of aluminium rods that could be inserted inside the stope where it must be oriented downwards the opening.

The measurements are automatically taken every four or five degrees and the device itself is equipped with a core processor for storing the measurement data. After the measurement operation, the CMS device is brought outside the mine to be connected to a computer which has the Optech software. From this computer the output of the CMS is produced as 3D AutoCad files (DXF or DWG). From the Optech program wire-frame mesh files are generated which can be directly overlaid on the planned stope outline in similar file format. In this way waste overbreak, backfill sloughage and ore left unbroken can be easily quantified by reference to the designed stope shape. Appendix D shows the detailed results of stope 7-2-5B at Bousquet 2 mine. As can be seen, the efficiency of Optech program is that it becomes possible to obtain automatically on the same plan, the planned design and the actual boundaries of the stope. Also, we can observe several vertical levels and longitudinal sections of the stope. The final result of the investigation is very significant since it provides the performance parameters. It includes the real tonnage of ore broken, the tonnage of wall overbreak, the tonnage of backfill sloughage; and finally (when the density and orebody grade is available) the percentages of ore dilution and the ore recovery. At Bousquet 2 mine all the stopes are surveyed in this fashion, and must have an operational report which contains both the CMS results and a detailed description of the stope: drawpoint, behaviour of openings (deformations, sketch of observation).

As it has been demonstrated above, in sublevel bulk mining conditions the accurate stope measurement is an important practice for the efficient running of the mine. There are then three main purposes of use of CMS:

1) To establish the performance of stope extracted;
2) The understanding of the various components of ore dilution in the stope;
3) The understanding of ground movements (instability problem).

In the past it has been difficult or even impossible to establish any of theses parameters as there was no satisfactory method of accurately determining the dimensions of the empty stope. Now the opportunity offered by this technique can be ascertained with high degree of accuracy and stope designs can accordingly be modified to reduce ore dilution and overbreak while increasing ore recovery.

5.3 Control Against Ore Dilution

The practical approach of dilution evaluation and measurement involves a tangible help for the control of ore dilution problem. However, even if we can evaluate the planned dilution, it remains difficult, even impossible to predict the actual final dilution due to additional dilution. This additional dilution is proved (by numerical modelling and CMS) as the direct product of both blasting effects and structural failures of the stope boundaries. Consequently, the only way to handle or reduce this unexpected dilution in sublevel mining methods is the use of cable bolt.

5.3.1 Cable Bolting in sublevel mining

5.3.1.1 General

Basically, cable bolts may be installed everywhere in target zones in the walls, backs and pillars of proposed stopes from lateral access drives as shown in Bousquet 2 stope plane (Figure 5.4). Unfortunately, installing cable bolts in radial fans from drives outside of proposed stopes means that reinforcement density is fairly low, especially in the larger spans. An alternative method for narrow orebodies is to install radial fans of cables from development sub-levels within the planned stope in Bousquet 2 radial cable bolt arrays. This creates bands of heavily reinforcement rock that effectively reduce the rock caving. Careful scheduling of stope extraction and filling is required to minimize the span open along strike and to allow the complete orebody to be mined with minimum ore dilution.

5.3. 1.2 Cable bolting Techniques in Sublevel mining

The advent of cable insertion equipment and cable bolting machines means that completely mechanized hole drilling, cable insertion and grouting is now possible. With these machines stiffer grouts are used and the cable bolts are usually "plunged" into a pre-grouted hole. The technology of grouting has also improved from a simple cement-water mix delivered by a single stage pump to high early strength cements delivered by two-stage, continuous pumps. The practice of pre-tensioning cable bolts is abandoned when it is realized that tension quickly develops in untensioned pre-reinforcement in response to rock mass movement induced by blasting or stress redistribution effects during or after excavation. Variation on pre-tensioning now includes placing the cable bolt in the hole, grouting an anchorage at the far end, fitting face restraint, pre-stressing the cable bolt and then grouting the balance of the hole. Many operations replace this two-stage grouting process by pre-stressing with a special expansion shell anchor and then grouting that has been fitted. Some operations simply install a short debonding tube near the collar, grout the full length of the hole, attach the restraint system and then pre-tension. When a pre-tension is applied to a fully grouted strand to hold the face restraint only a small force remains on the system (Thompson 1992).

5.3.1.3 Cable Bolt Design Procedures

The availability of geomechanical data is the first stage of the procedure. Figure 5.5 (C. Windsor, 1992) schematically illustrates the choice of an appropriate cable bolt relative to work man condition. When deciding on the arrangement of the cable bolt array may be made. There are four assessment methods that may be useful in a cable bolt design procedure (Windsor, 1992):

1. Empirical methods.
2. Analytical methods.
3. Experimental simulation methods.
4. Numerical simulation methods.

The empirical methods and the analytical methods are useful in helping to propose a preliminary design. They are used to determine the overall stability of the stope and hence the need of cable. The experimental simulation techniques and numerical simulation techniques are best suited to checking designs. Empirical and analytical approaches can be combined to suggest an initial design. Therefore empirical and analytical methods should be able to assess rock mass stability, predict and identify the type of instability, and the component mechanisms of instability. The numerical simulation backed up by experimental investigation is the ideal approach for the design of cable bolts. Ideally, cable instrumentation is used to help validate th numerical simulation and improve the accuracy of future modelling studies of subsequent stopes.

5.3.2 Factors controlling cable Bolt performance

This section presents some tools which mine operator can use to base operational decision and to achieve optimal cable bolt design. Some key factors may influence, directly or indirectly, cable bolt performance. These factors can be divided in two main categories (see Figure 5.5): uncontrollable and controllable factors.

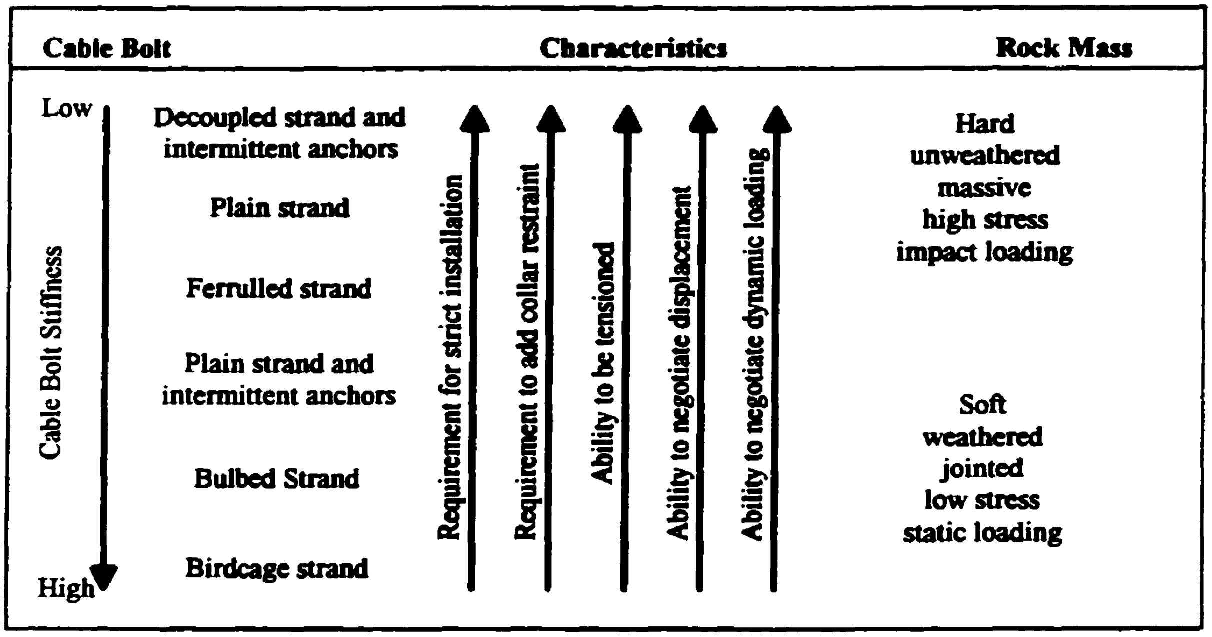


Figure 5.4: The choice of an appropriate cable bolt type relative to rock mass conditions; (Windsor, 1992).

5.3.2.1 Uncontrollable factors

- *Mining induced effects:* Mine development and the extraction of orebody itself allow the redistribution of stress field, rock mass quality and influence cable bolt capacity by shifting the design point on Figure 5.6. This influence can occur in two ways. Firstly mining induced stress increase, if high enough, may cause stress fracturing or slabbing in stope backs which reduces the effective fracture spacing and hence the effective cable bolt capacity. This tends to shift initial design points such as A vertically down to lower capacity such as B. Secondly, mining induced stress decrease (relaxation in walls), causes a reduction in confinement, and hence in cable capacity. This shifts initial design points such as A to the left to some lower capacity such as C. These effects are not yet well understood and further reseach in the area of stress control on cable bolt capacity is needed.

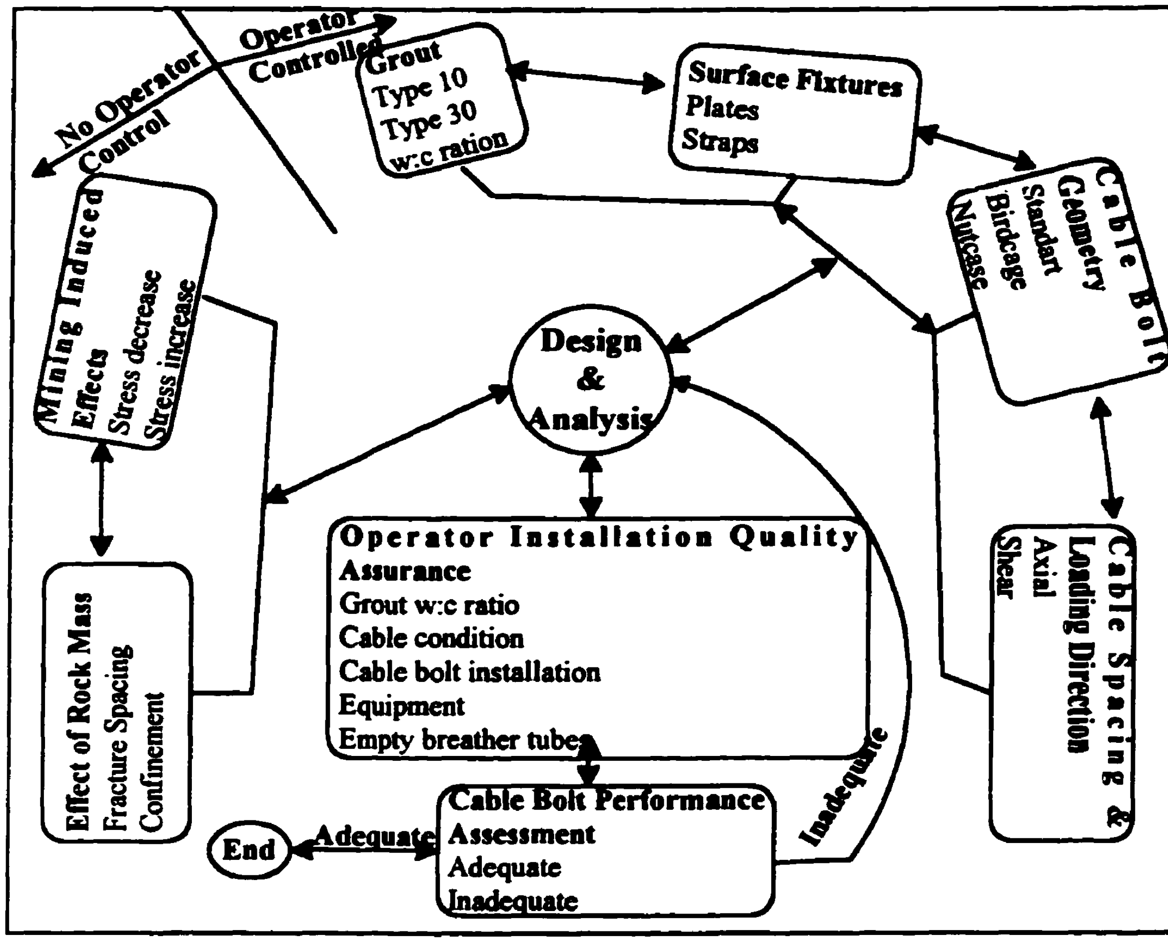


Figure 5.5: Cable Bolt performance assessment, (Bawden, et al, 1992).

- *Rock mass effects:* Cable bolt capacity and behaviour are linked to rock mass quality and properties. Three factors strongly influencing these are: 1) natural fracture spacing; 2) confinement as related to rock mass quality; and 3) confinement as related to in-situ stress levels.

5.3.2.2 Controllable Factors

Grout: The performance of grouted cable bolt support is greatly linked to the type of grout or the w:c ratio. Hence, depending of the test results, mine operator has the choice between the proposed types or make his own w:c mixture. The grout water:cement ratio factor is largely dependent on the pumping/mixing system used in the mine. Nature of the cement and how cement is stored and handled on the site may influence final grout characteristics. Type 10 and type 30 are two commonly used cements; type 30 is less attractive in practical mining than type 10 because of its increased hygroscopicity, its higher cost and its lower uniaxial compressive strength.

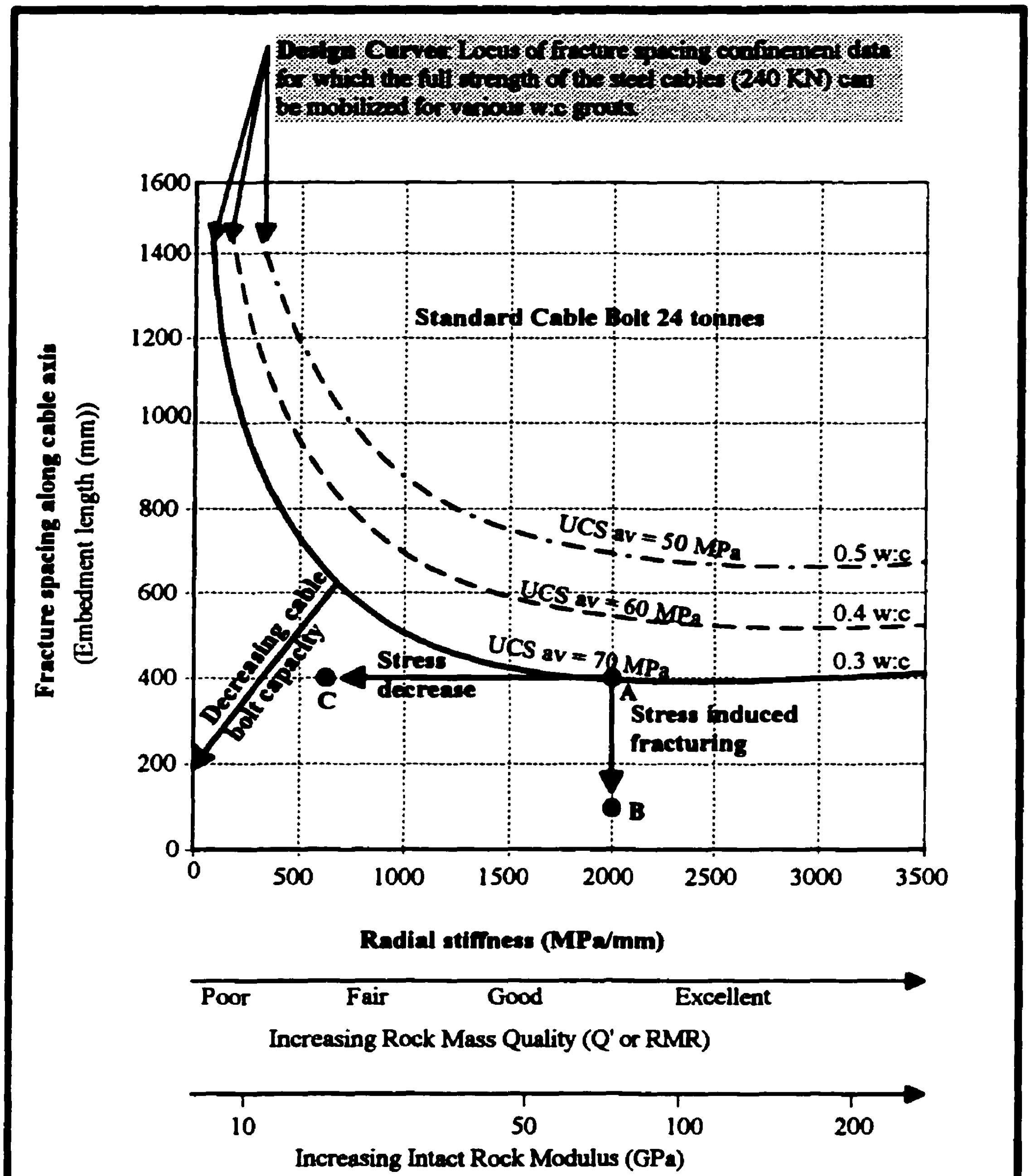


Figure 5.6: Effect of rock mass and mine induced stress change on ultimate cable bolt bond capacity (Bawden, et al., 1992).

- *Surface fixtures:* The use of surface fixtures on the ends of cable bolts is commonly used to secure the ground between cable bolts, it increases the bearing surface at the immediate collar of the hole; and to act, in effect, to reduce the negative effect of short embedment lengths. The use of such fixtures should: 1) reduce the span between cable botls, 2) prevent ground from unravelling up the cable due to short embedment lengths and 3) prevent rotation of the free end of the cable under loading conditions. Plates and straps are used to secure thin slabby ground, effectively creating a higher embedment lengths immediately adjacent to the collar of the hole.
- *Cable bolt configuration:* Geometrically there are many types of cable bolts such as standard, birdcage and nutcase, etc.. For a standard cable the maximum potential extent for geometric mismatch between the cable and cement annulus during cable displacement is small and is equal to the radius of a single strand. For birdcage and nudcase cables the potential extent for geometric mismatch between the cable and the cement annulus, (including the grout trapped within the case structure), is much greater because the cage structure effectively increases the radius of the cable. Theoretically, cable geometries with a higher potential for geometric mismatch should overcome, at least partially, the negative effect of low radial confinement. That is, birdcage and nutcase cables should perform better than standard cables in more highly fractured rock masses. In addition they should be less susceptible to problems created by poor quality grouts. The performance of different cables geometries under different conditions of rock mass confinement and different embedment lengths is an issue requiring further research.
- *Cable bolt spacing and orientation:* Cable bolt pattern 2×2 m is most common, in reality there are no method to reliably determine cable bolt spacing. A theoretical method suggests the weight of unstable local rock mass divided by the cable capacity and then adjust this for some factor of safety. The only justification for reducing cable bolt spacing would be the observation of broken cables in failures or ravelling failures between cables. For cable bolt orientation it is important to assume that if they are installed such that rock mass movements induce shear loading the maximum cable

capacity will be reduced. Then, cable bolts should be installed perpendicular to rock mass discontinuity planes.

Analysis of reinforcement by cable bolt can show how the cable works in terms of its behaviour, load carried, and reduction of local failure. The primary mechanism of load transfer from the rock to the cable is frictional (Stillborg, 1984). The frictional bond resistance and therefore the ultimate bond strength, τ, are directly related to the interface pressure as follows:

$$\tau = \delta \tan (\phi + i)$$

where δ is the nomal stress

ϕ is the bolt-grout friction angle

i the dilation angle.

However, the mechanics of a bonded cable bolt in jointed rock are not so simple. The cable bolt installed in a hole which is already deformed in response to the stresses in the rock, is submitted to load at discrete intervals along its length at intersections with discontinuities. Therefore, the cable bolts act as grouted dowels that provide a passive resistance to any movements in the rock mass. Where these dowels intersect a joint or fracture, the tendency for the rock mass to separate due to tensile stresses or shearing accompanied by dilation is resisted in the immediate area of the intersection. If a failure of the grout bond occurs near a joint, the dowel elongation that then occurs is confined to a short section of its length. If such a joint separation was initiated across a point anchored bolt, the bolt would be tensioned and strained uniformly over its entire length, resulting in less resistance to joint separation. As the role of cable bolt is to reinforce rock mass by holding mainly, near surrounding discontinuities, it is possible for up to six displacements to occur. Three rotations and three translations (Windsor,1992). These may produce complex interactions involving axial, shear, bending and torsion loading of the reinforcing element. As shown in Figure 5.7, even when the displacement is simple, the orientation of the discontinuity in relation to the reinforcement direction means that only under special circumstances are loads simplified to pure tension or pure shear.

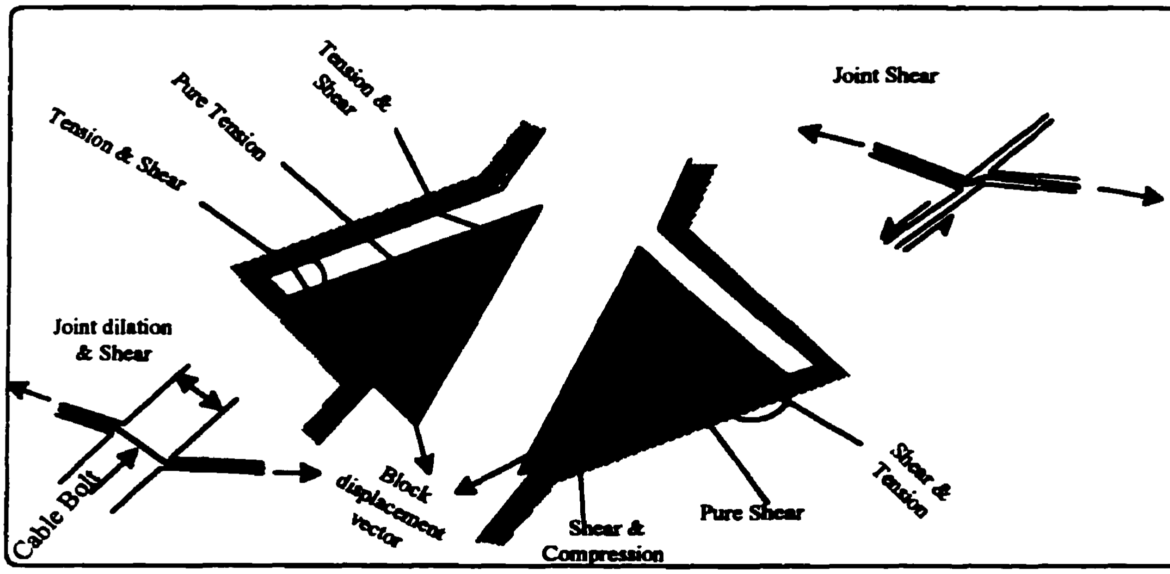


Figure 5.7: Schematic diagram of the modes of reinforcing loading at a discontinuity (after Windsor, 1992).

5.3.3 Cable bolting at Bousquet 2 Mine

In order to minimize the dilution coming from the hanging wall and foot wall, an important measure involved the development of cable bolt drifts for the installation of cable bolts. From cabling drift, about eight rings of cable bolts are installed toward the orebody in advance of stoping. Each comprising a fan configuration of about 9 cable ranging from 18 m to 27 m in length. A single bulge cable with a 12 inch bulb spacing was placed in each 2.5 inches hole, and was grouted from toe to collar using a 0.35 w:c ratio grout (M. Ruest et al.). Figure 5.8. Show the cable bolt layout for the 9-1-15 stope. While Figure 5.9 show the overall vertical section view of patterns of cable bolt reinforcement. The cable bolts used are standard seven-wire cables with steep strap (birdcage) installed every 2 meters; the cross sectional area of the single bolt is 138 mm^2, with a modulus of elasticity of 200 000 MPa. As host rock at Bousquet is very foliated and affected by many orogeny events that provided fractures, faults and joints, the objective of cable bolting is based upon supporting some 2 or 3 m thick zone paralleling the wall. In fact the cables are applied to prevent separation and slip along planes of weakness (foliation and joints) in the rock mass. Induced rock fracturing due to stress distribution is another prevailing reason. Consequently success of these measures against immediate waste caving are observed by reduction of ore dilution and the distribution of tensional force in cables applied. One other reason is the pre-reinforcement of wall rocks while increasing blast energy to get better fragmentation of ore produced. Presently, the main effort of mine engineers is focused on qualitative and quantitative cable bolting data for following objectives:

- The improvement of pre-existing cable bolt design and pattern;
- The understanding of load distribution along single cable bolt; and
- Quality control.

For the purpose of efficiency of cable bolting at Bousquet mine, two experiments were made. The first experiment is made in order to evaluate the use of cable bolts for back reinforcement; while the second experiment is made in order to evaluate the use of cable bolts for hanging wall pre-reinforcement, the benefit of changing conventional cable, the pattern of cabling. The results of these tests confirm the success of the conventional cable (against the birdcage cable and the nutcase cable) which work with only 35% of its capacity while for other cables the rupture occurs at 30 mm of displacement. The results also show that the cable bolts failed by two mechanisms: *i)* Bond failure involving pullout and, in particular, slip at the cable-grout interface; and *ii)* tensile rupture of the steel cable. Consequently the decision taken about the choice of cable bolting depends upon the results from detailed rock mechanics study which includes stability analysis and numerical modelling.

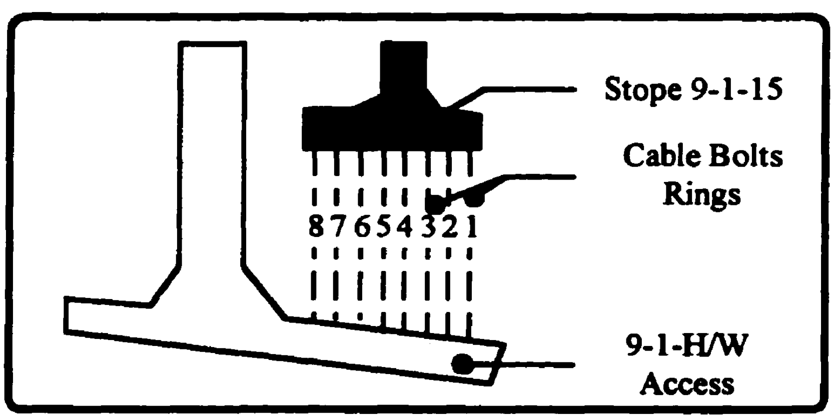


Figure 5.8: The Cable bolt layout for 9-1-15 stope in plan (Hyett et al., 1997).

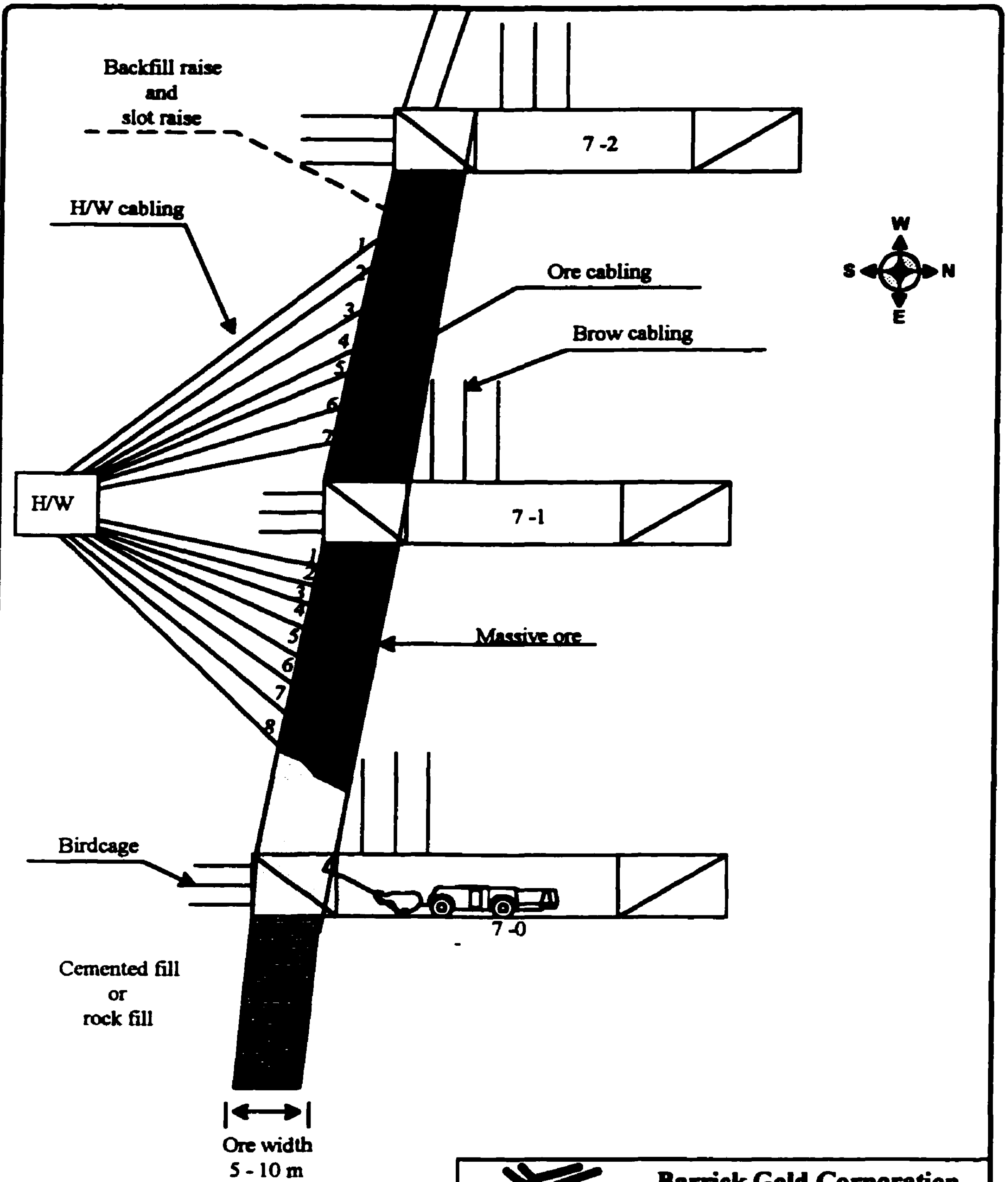


Figure 5.9: Hangingwall cable bolting in transversal stoping (Source: J. Henning, 1997).

The need that the cable works against stope boundary failures, has entailed the use in place of tension measuring device. Hyett et al, (1997), have developed a multi-point extensometer called Stretch Measurement to Assess Reinforcement Tension or SMART cable. The advantage of this technology is that it is possible to install sufficient disposable instrumentation to understand how the patterns of cable bolts are working, not just a single isolated cable bolt. The result of tests made for the stope 9-1-15 with this technology have shown that (Hyett et al):

• The movement of hanging wall; therefore the big part of load supported along the cable is concentrated near the boundary of the stope (from between 4 m and 7m);

• The increase of cable load for each blasting sequence of the stope.

•The cable picks up load immediately after the blast and continues to pick up load. Such behaviour of the stope indicate the readjustment of stress redistribution. The rate of these events decreases with time as stability is progressively attained.

The analysis of cable bolt pattern behaviour is discussed in the numerical modelling section (next chapter). Without that, the data provided by Cavity Monitoring system surveys have proved the performance of cable bolt implementation. Other aspect of rock support in the mine is the intensive use of rockbolts and mesh for reinforcement. Because the rocks in the mine are highly foliated, all the drifts are systematically bolted at about 1.5m distance. In poor ground zones, mesh is combined to the rockbolts. Exceptionally, where drawpoints are hazardous, cable bolts are installed in the areas. Cemented rockfill is also an option used at Bousquet 2 to reduce post mining induced stress. The primary stope (stope type A) should be backfilled before the recovery of the relevant secondary stope (type B).

Chapter 6. Comparison Between CMS Measurements and Numerical Modelling Results

6.1 Comparison

The comparison between the actual stope geometry (CMS results) and the numerical modelling results is attractive since it permits to calibrate/validate the numerical model using the field data. The predicted caving zones given by the modelling are assessed from: 1) the tensional zones in the stope walls and 2) the safety factor using rock failure criteria. Figure 6.1 shows the distribution of principal stresses around the stope. The caving zones are identified as the areas of tensile stress where relaxation dominates. Notice that tensional forces are parallel to the stope wall along the stope wall, while they are nearly perpendicular at the upper and lower part of the walls.

Figure 6.2 shows the safety factor distribution for a primary stope, according to Mohr-Coulomb criterion (figure 6.2a) and Hoek and Brown criterion (figure 6.2b). It can be seen that the zones where the safety factors are below 0.5 closely coincide with the zone of tensional failure zones observed in principal stress trajectories (figure 6.1). The rock strength parameters used in the safety level analysis are indicated on the figures. Figure 6.3 presents a comparison between the CMS measurements and the numerical modelling results. Figure 6.4 and 6.5 present similar type of comparisons while using Morh-Coulomb and Hoek and Brown failure criteria, respectively. Referring to the safety level results, it can be seen that numerical modelling results correspond more closely to the CMS results when the tensional stress zone is considered as a basis for evaluating the caving zones. For the stress analysis of the secondary stope (7-2-5B), it was necessary to reduce the field horizontal stresses since mining of the primary stopes causes the ground to work (or relax). In order to determine the appropriate level of horizontal stress reduction, a sensitivity analysis was carried out with the following scenarios:

Analysis 1: $\sigma_H = 75\ \%$ of σ_H^{o}

Analysis 2: $\sigma_H = 50\ \%$ of σ_H^{o}

where:

σ_H° represents the pre-mining horizontal stress field which was used in the analysis of the primary stope;

σ_H represents the pre-mining horizontal stress field for the secondary stope.

In the following, only the results of the stress trajectory distribution are presented. It has already been established from the primary stope analysis that this method correlates better with the CMS results. Figure 6.6 and 6.7 present results which are superimposed on the CMS measurements. For the two analyses conducted, as can be seen, the first analysis with $\sigma_H = 75\ \%\sigma_H^{\circ}$ correlates better with the CMS results, since at $\sigma_H = 50\ \%\sigma_H^{\circ}$ there are not tension failure in the foot wall whereas a part of this zone is caved.

From the above analyses, the following remarks can be made:

1. The use of Morh-Coulomb and Hoek & Brown criteria does not appear to give reasonable estimate of ore dilution due to wall caving and sloughing, i.e. in low or even tensile stress zones.
2. A simple method based on tracing the zone of tensile stresses in the stope walls, and considering such areas to be potentially caving, appear to give reasonable prediction of ore dilution as it agrees more closely with the CMS results.
3. Horizontal field stresses acting on the secondary stope are lower than those acting on the primary stope. At 25% reduction factor of the original (pre-mining) horizontal stress gave reasonable results.

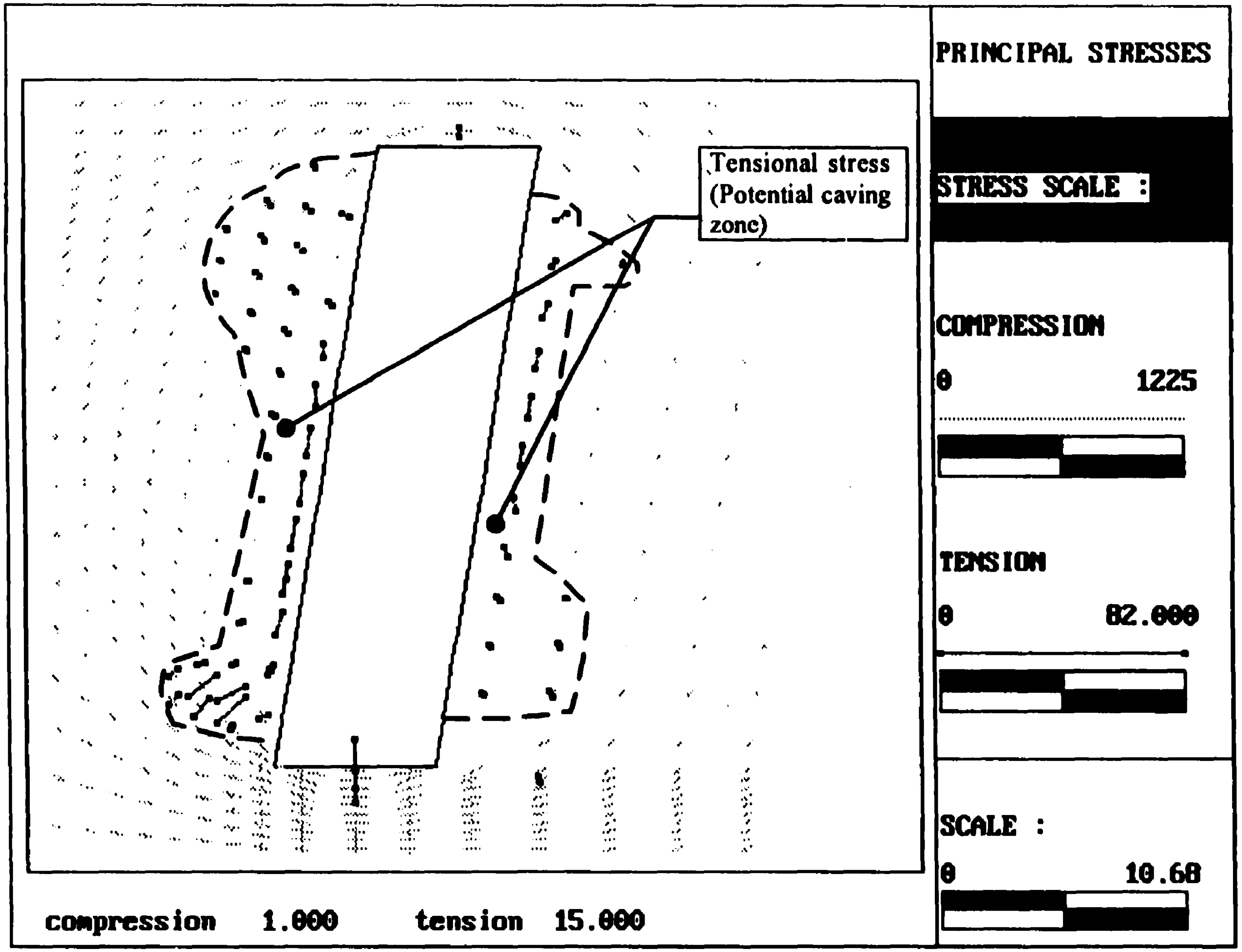


Figure 6.1: Principal stress trajectories around the stope; Primary stope.

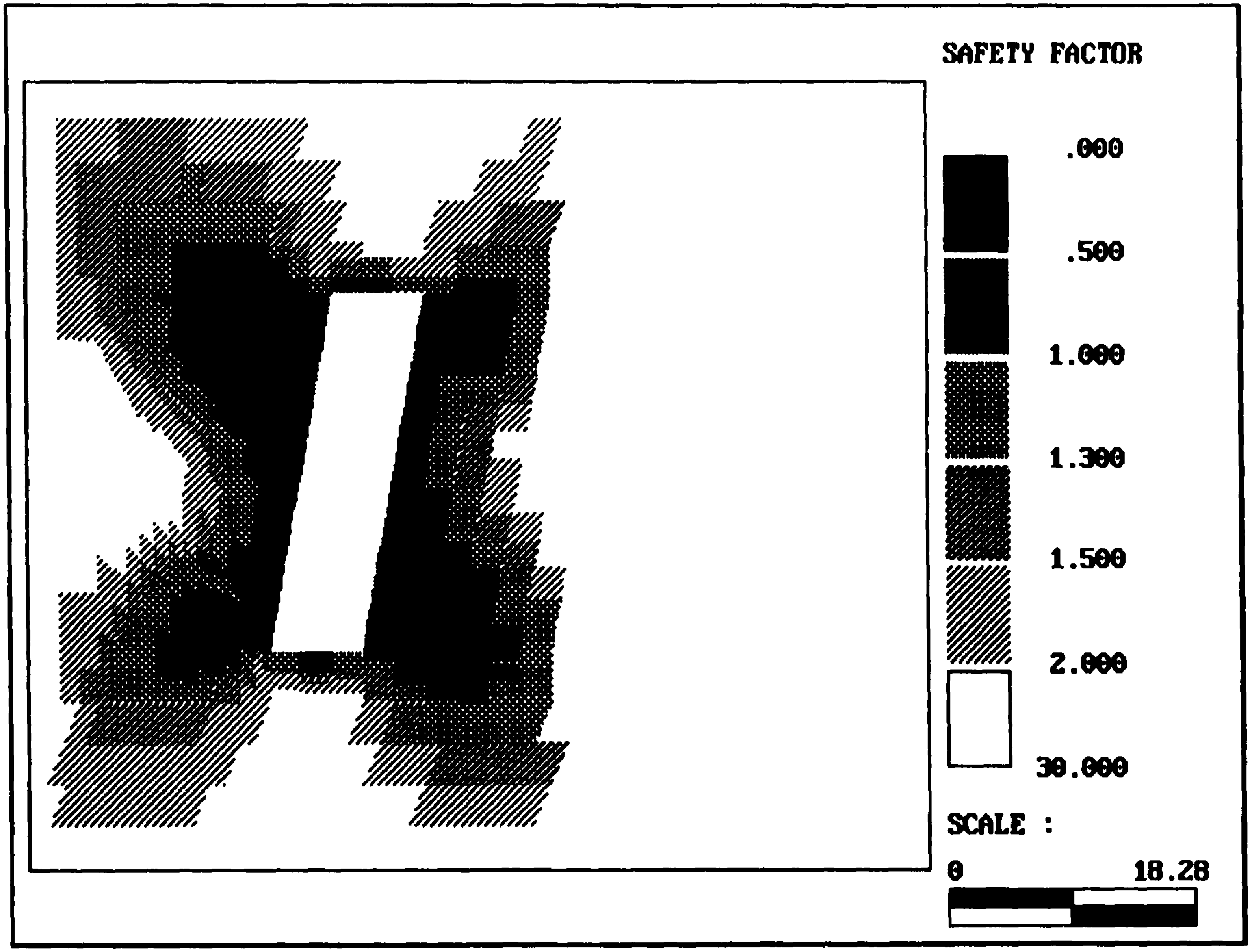


Figure 6.2a: Safety level result using Mohr-Coulomb criterion; primary stope.

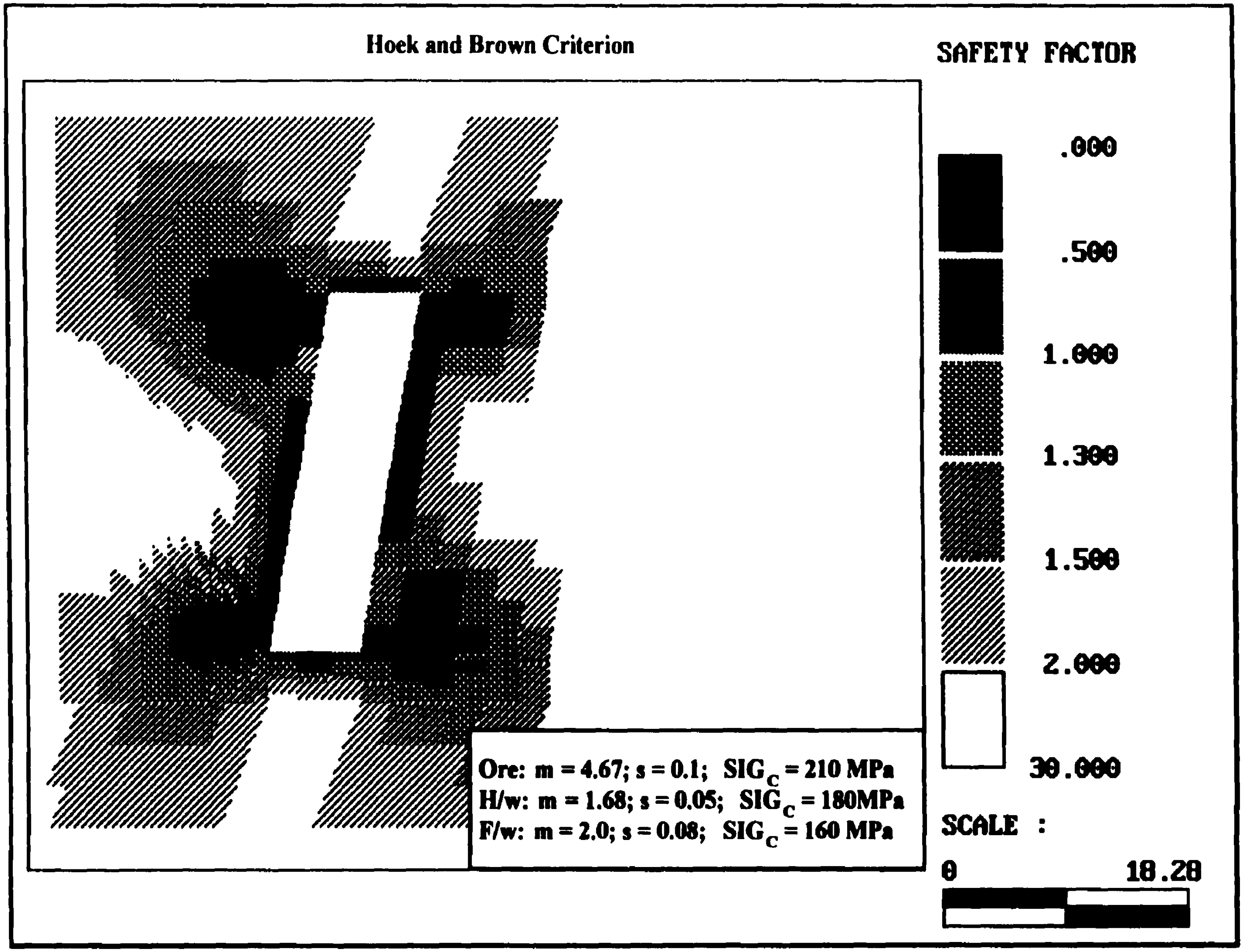


Figure 2b: Safety level result using Hoek and Brown criterion - Primary stope.

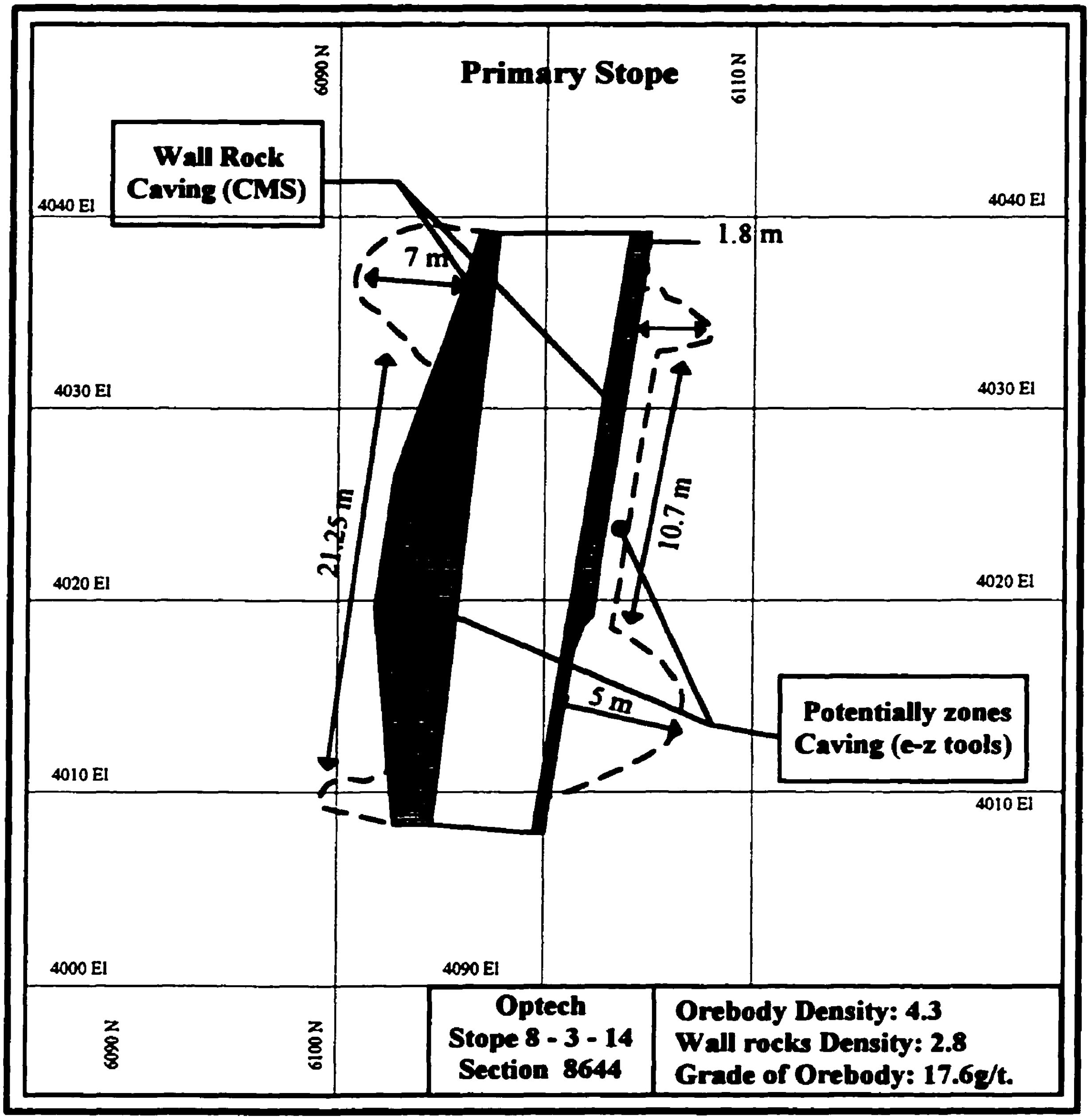


Figure 6.3: Superposition of CMS and numerical modelling, tension zone results - Primary stope.

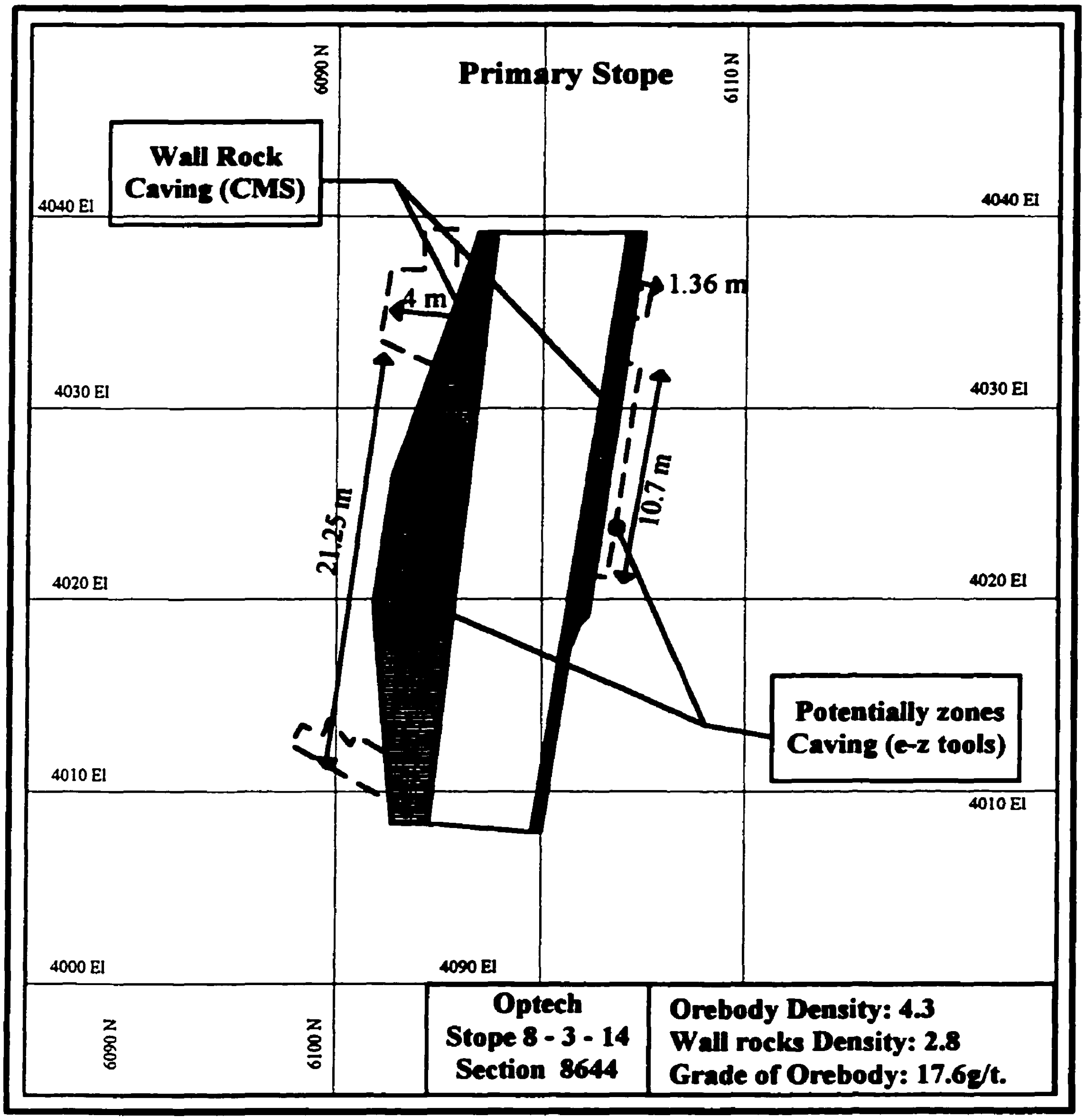


Figure 6.4: Superposition of CMS and numerical modelling, Morh-Coulomb safety level results - Primary stope.

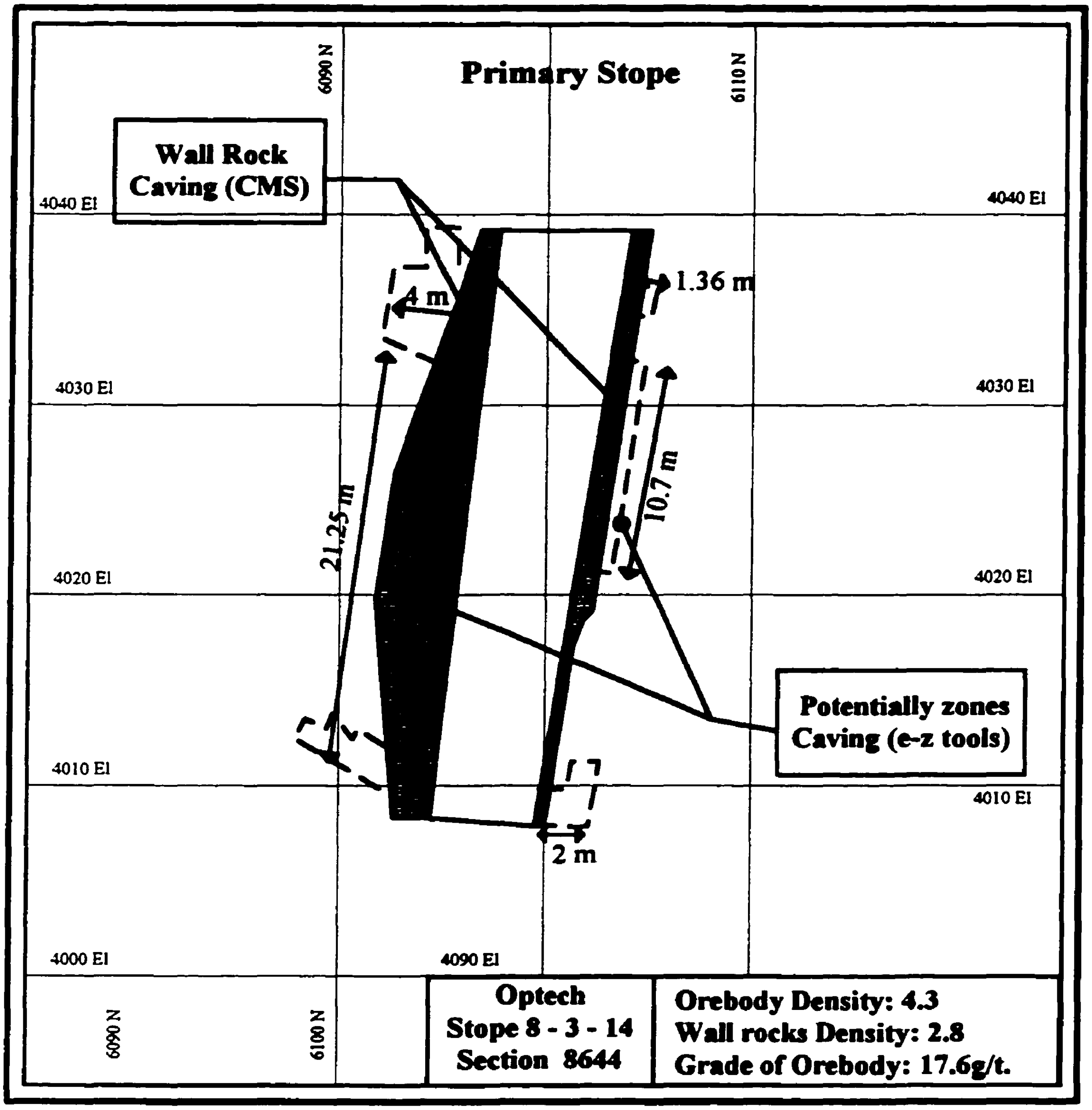


Figure 6.5: Superposition of CMS and numerical modelling, Hoek & Brown safety level results - Primary stope.

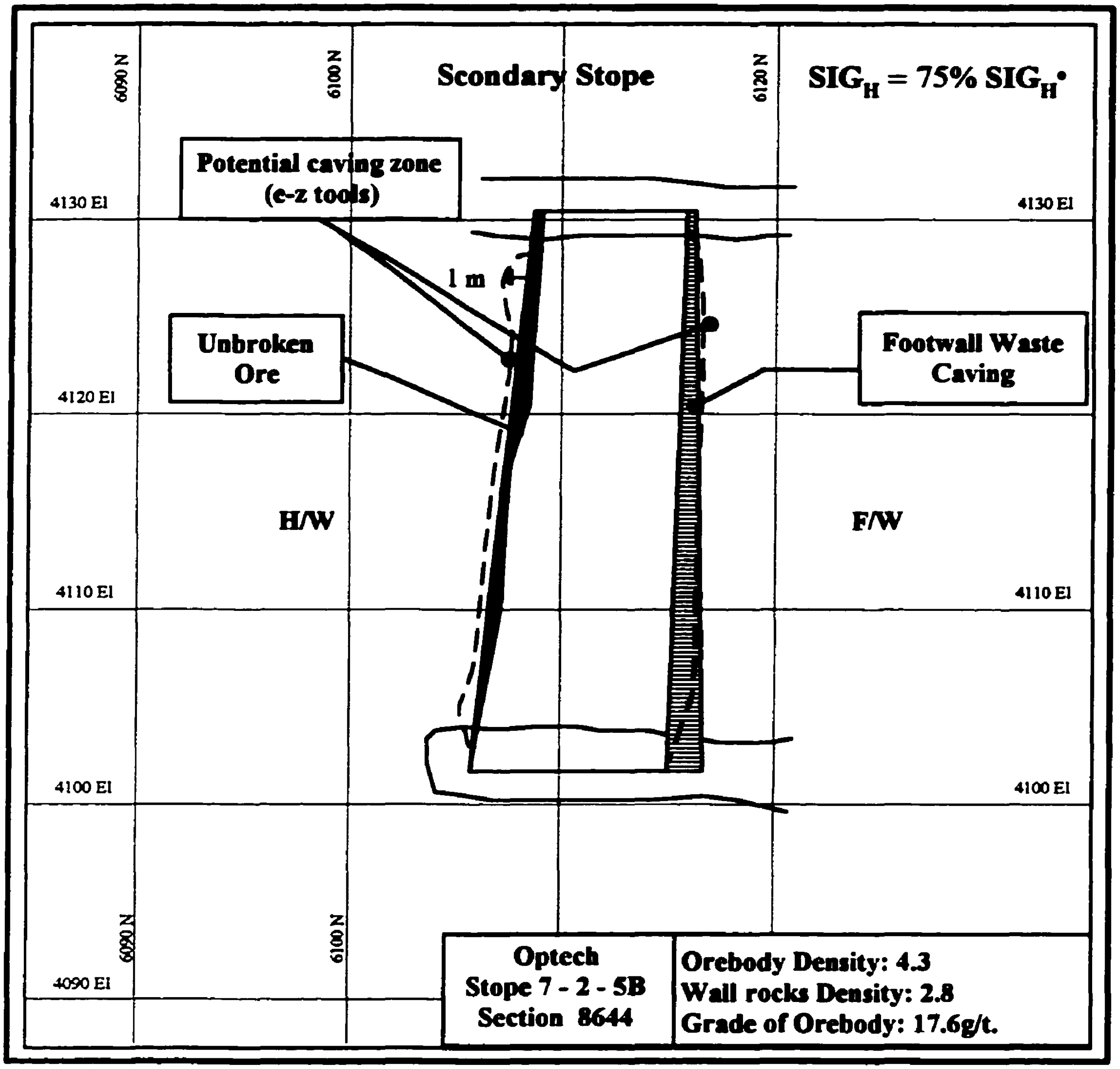


Figure 6.6: Superposition of CMS and numerical modelling tension zone results - Secondary stope - Analysis 1.

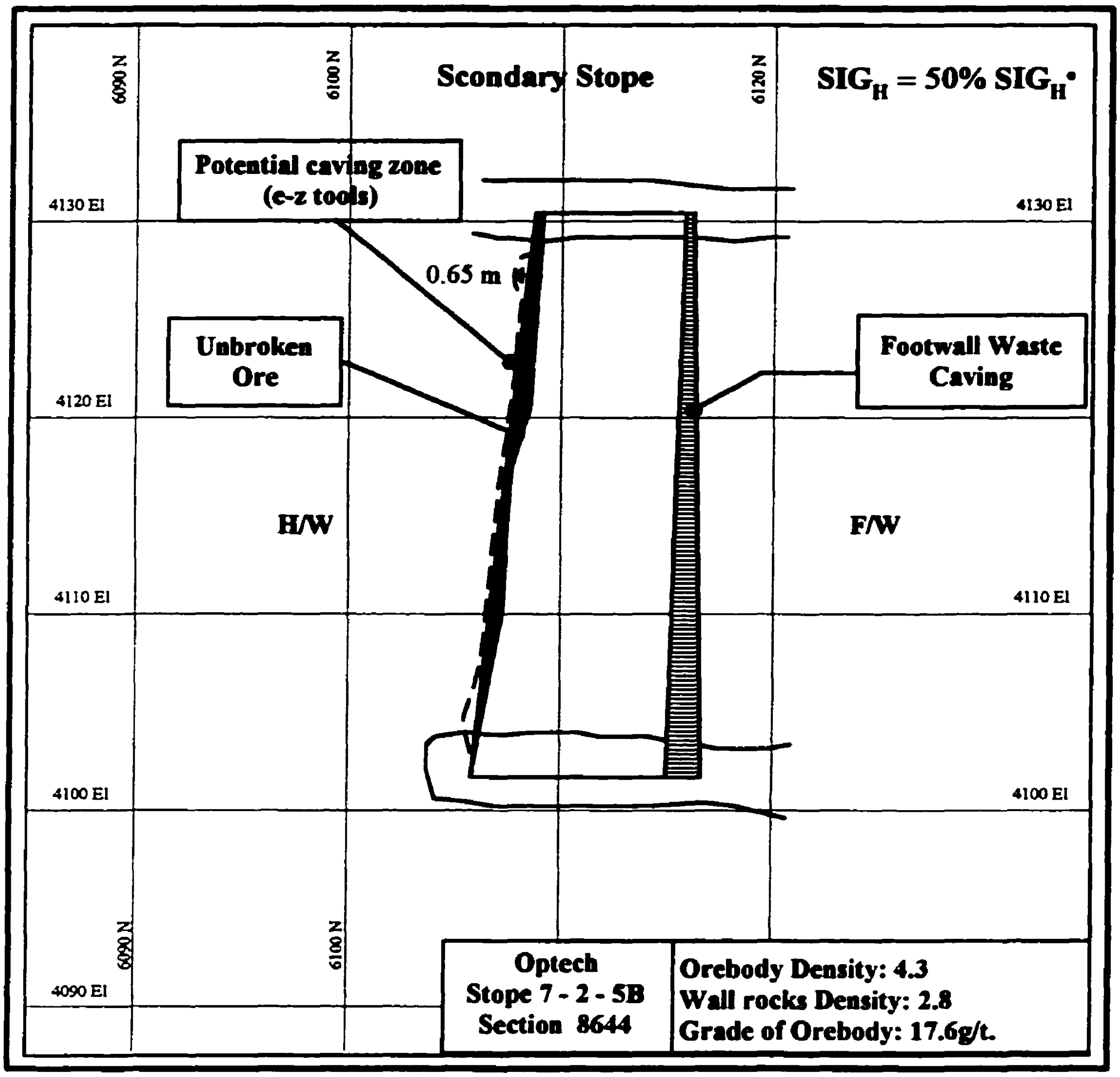


Figure 6.7: Superposition of CMS and numerical modelling tension zone results - Secondary stope - Analysis 2.

Notice: Dilution is relatively highest in this stope. Usually for open stope mining acceptable ore dilution rank between 5 to 15 %. The problem of this stope could be due to particular local ground conditions (Highly foliated and/or jointed); however, cable-bolt have been efficient in the upper part of the hanging wall since this zone is submitted to much tensional failure force. Also the level of induced stresses is relatively highest in primary stopes, that can affect the stability of those stopes.

6.2 Discussion

The expressions "narrow vein" and "ore dilution" are a natural couplet in fair to poor rock mass quality area. The decision to mine faster and deeper such orebody still presents two inevitable options: to appreciate the lower production cost in high dilution mining, or to opt the higher production cost in reduced dilution mining. For choice between these options, the grade or the value of ore is the governing factor. Highest grade orebody should be extracted with possible measure against dilution, if necessary by a relatively high cost method, and total recovery must be required, in the other hand, for lowest grade orebody, however, higher dilution risk can be considered for lower cost. Another governing factor is entirely geomechanical therefore associated with the applied stress and quality and the nature of the rock mass itself. For the understanding of dilution problem, we have emphasised on the stope face behaviours; this undergoes the assumption that the larger part of dilution results from the consequence of ground instability conditions. However the data given by CMS shows the stope configuration resulting from the combination of overbreak and rock caving in the stope. Therefore, it remains complicated to identify the relative proportion of the effect of each of these unexpected components. Absolute better alliance of these two factors (appropriate blasting pattern and efficient cable bolt support) is the solution for excellent performance in stope mining. However, to attain these dextrous combination remains laborious; and it no rare that some mines expire their active life without to achieve this objective. The efficient management of mining operation in such condition infers precarious responsibilities. In fact, the continual need of efficiency performance at lower cost remains of paramount importance to all concerned in stoping operation. For the correction of ore dilution problem, field studies and investigations for the assessment of recovering stope extraction as well as their working time require much energy and cost. The application of their recommendations also impose additional expensive development work; therefore postpone any local active operations. The cumulative consequences of these decisions make timorous mine operational executives. However, if some measures are not taken it could evenly destroy the profitability of the operation. Thus. It is no rational to still without imply the adequate means to enhance or reduce this operational ailment. It is also important to report that many operations have closed because of uncontrolled ore dilution, and in serious cases before they even completed their first year of production. Even if it has been mentioned in previous sections, it remains significant to signal further that local geological occurrence thoroughly stays the main source of ore dilution. The occurrence of orebody in weak host rocks (jointed rocks, foliated rocks or soil rocks) are usually associated with high ore dilution. Independently of the characteristical parameters of the concerned deposit, the mining method employed and the delicacy of mining operations proportionally induce or control dilution problem. Without forgetting the structural nature of the orebody itself (irregular ore/waste contacts, indistinct ore/waste contacts) which are also the potential sources of dilution.

For narrow vein mining in Abitibi area the effect of magnitude of stress/stability have critical incidence of the dilution. The significant engineering judgement is to emphasize that it can bepossible for stability of the stope. This remains crucial big challenge in jointed and/or foliated rocks as at Bousquet 2 mine. Because there still two particular restraints in this types of orebody mining. One is that the better ore recovery make uncertain the acceptable stope stability. It is known the stability of the stope as well as the factor of safety of the whole mine still closely linked to number and the size of the sill pillars. These pillars are in many cases the ore leaving in place. While an ideal situation should imposes maximum ore recovery. The second restraining factor imply the effect of depth on stope stability. Mining in deeper conditions involve the combination of the stress magnitude and discontinuities. Also the scale of tress literally increase with the depth; with crucial grow of stress, the resulting induce failure will be proportional. In the other hand the existence of the combination (shear stress-discontinuities) may be the main cause of dilution in open stope. The mechanism of stope ore dilution closely results from the fundamental behaviour of wall rocks. This mechanism of instability provoked by shear stress and discontinuity is explained in stope wall behaviour section.

Chapter 7: Conclusions and recommendations

7.1 Conclusions

Sublevel open stoping remains an improved mining method for steeply dipping orebody in Abitibi archean area. The presence of several close openings, in jointed volcanic rock, required by this mining method provokes the instability along the stope walls. In such conditions, the mechanisms causing ore dilution can be summarized by the combination of several factors such as ground condition, blasting damage, state of stresses around the stope, and stope design. Thus, the evaluation of stope walls behaviour by numerical modelling and CMS has permitted to understand the various components provoking ore dilution. It has been proved at Bousquet 2 mine that dilution comes from both wall rock overbreak (due to explosive energy) and hangingwall caving and footwall sloughing (due to stress redistribution and ground conditions). However, it is still complicated to identify the relative proportion of overbreak and caving (or sloughing) because of the implication of various parameters such as presence of joints and geological inhomogeneity. From a ground condition point of view, dilution inside the stope is provoked by unfavourable rock properties and the occurrence of discontinuities. Stope instability associated with stresses is linked to post-mining stress instantaneous and continuous redistribution in two zones: stress concentration zones with maximum principal stress about 240 MPa (in the corner of the stope) and stress relaxation zones; these zones occurring along the hangingwall and the footwall are submitted to tensile stresses (about 0.65 MPa in horizontal direction, and 14 MPa in vertical direction) provoking the opening as well as the sliding of induced failures. In relaxed field, tensile stresses are proved to be the main cause of dilution; that has permitted to calibrate the numerical model. A sensitivity analysis has proved that horizontal field stress acting on the stope is reduced by 25% for the secondary stope; this can be a cause of lower dilution in secondary stope. Otherwise, stope overbreak caused by blasting effects is due to the inelastic behaviour of the rock mass under the explosive energy; and the proportion of overbreak is allied to the Peak Particle Velocity generated by the interaction of explosive material-geological material. Controlled blasting has been demonstrated as a good option for reducing dilution. It has been demonstrated by CMS observations and the numerical modelling results, as well as the results of blasting effects that the ore of the primary stopes is much more diluted than this of the secondary stopes. This is due to the higher level of induced stress around the primary stopes and the higher vibration of wall during blasting. Larger part of dilution comes from caving of the hanging wall (poor quality of this rock and effect of gravity). Thus, primary stopes must be carefully prepared against dilution. To control ore dilution, cable bolting of the hanging wall from parallel drift is still the satisfactory way. The use of cable bolts as a pre- reinforcement method of ground control has proved effective in minimizing ore dilution from stope wall rocks. The result of modelling has demonstrated the change in the stress field after cable bolt installation. This is accompanied by a reduction of ore dilution. Consequently, the study has shown the effectiveness of cable bolt pre-support of the fair quality wall in Bousquet sublevel stoping.

7.2 Suggestions for Further Research

The control of open stope wall stability must be done for efficient and economical ore recovery. In spite of several studies, ore dilution remains a problem implying various complex parameters, difficult to evaluate. These parameters should be clearly identified and understood. Even though we have emphasized our study on some items and data, our ability to quantify field stress is probably limited, in part because stress measurements are difficult and expensive to be carried out, and because the stresses vary to an unknown degree from point to point within the rock mass. Other future studies which can be considered are:

- The use of several numerical methods in conjunction, allowing for the advantage of each method and then providing a good analysis from different perspective.
- In a particular mine study, it should be interesting to compare dilution values obtained from CMS survey with the corresponding stability number N. The values N and R (hydraulic radius) can be plotted in modified stability graph (proposed by Mathew). The resultant graph could help to create a relationship between ore dilution and stope stability.
- Further studies involving underground borehole drillings to determine the location of discontinuities as faults joints etc. and their effect on stope stability.

- Further investigation must be performed on the understanding of failure mechanisms and criteria to determine rock mass ability to sustain confined and shear stresses before stope design.
- Assessment of various cable bolting designs and patterns and then to choose the most effective one for the stope design.

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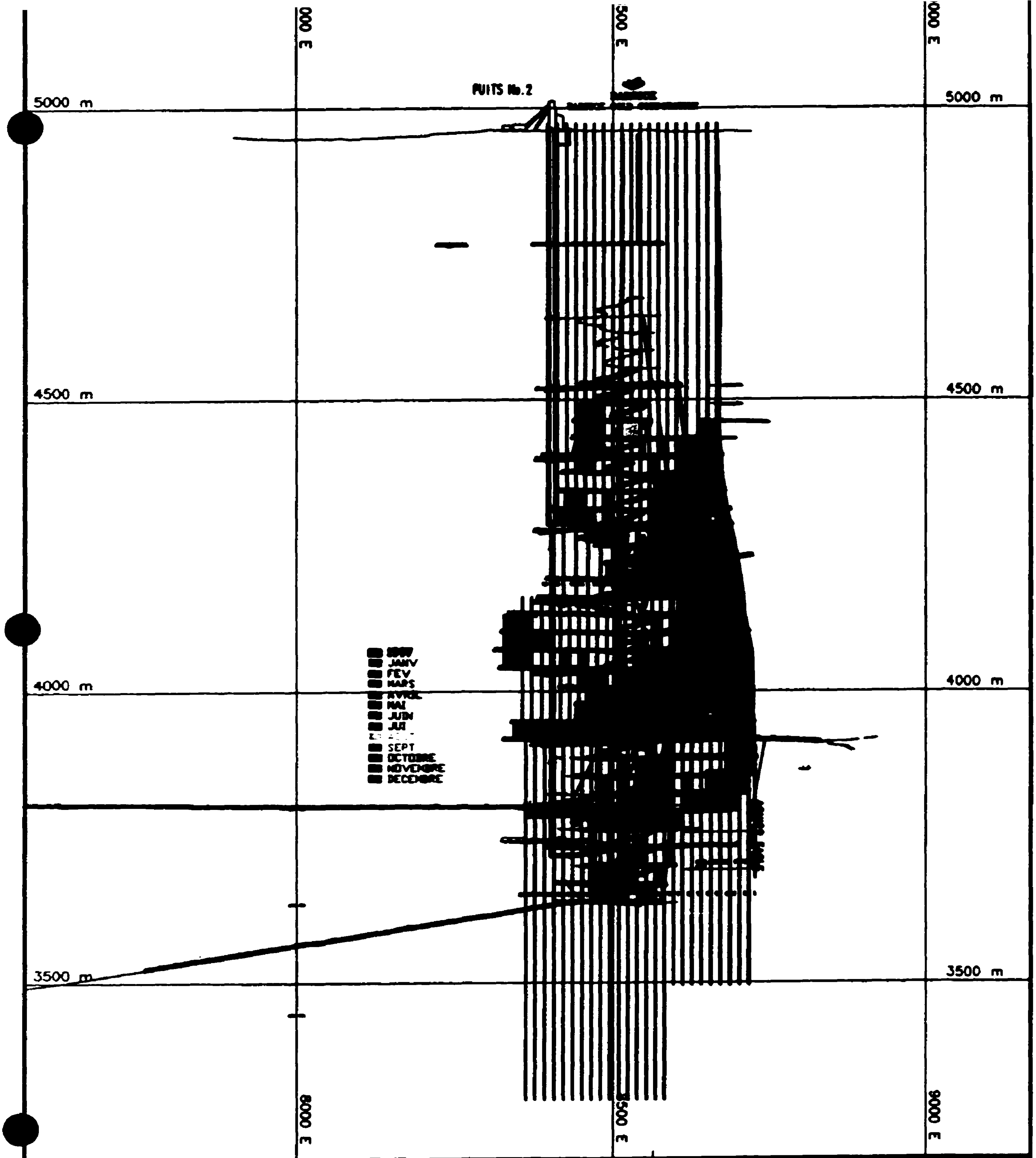
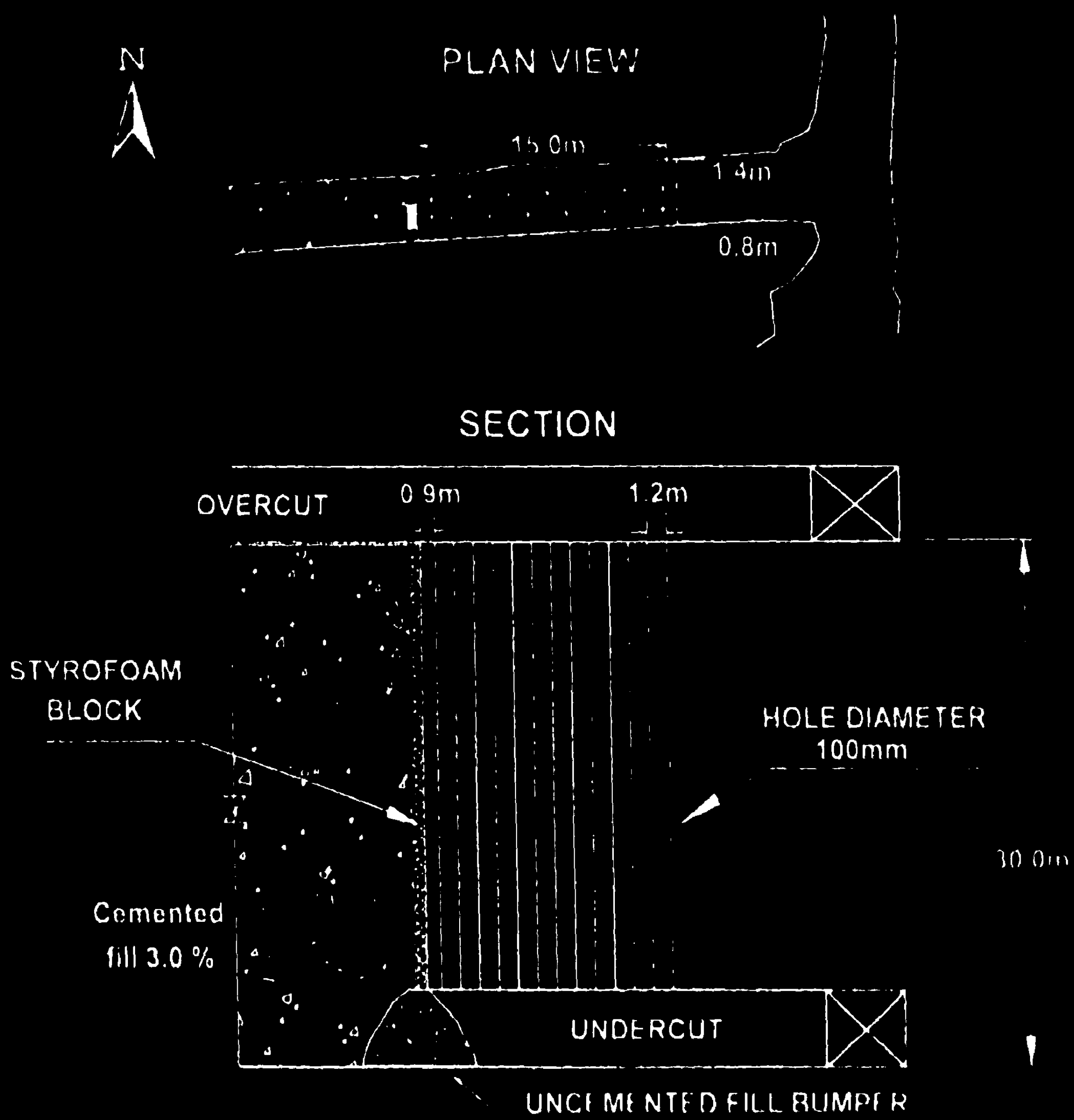
Appendix A: Bousquet 2 Mine Planes

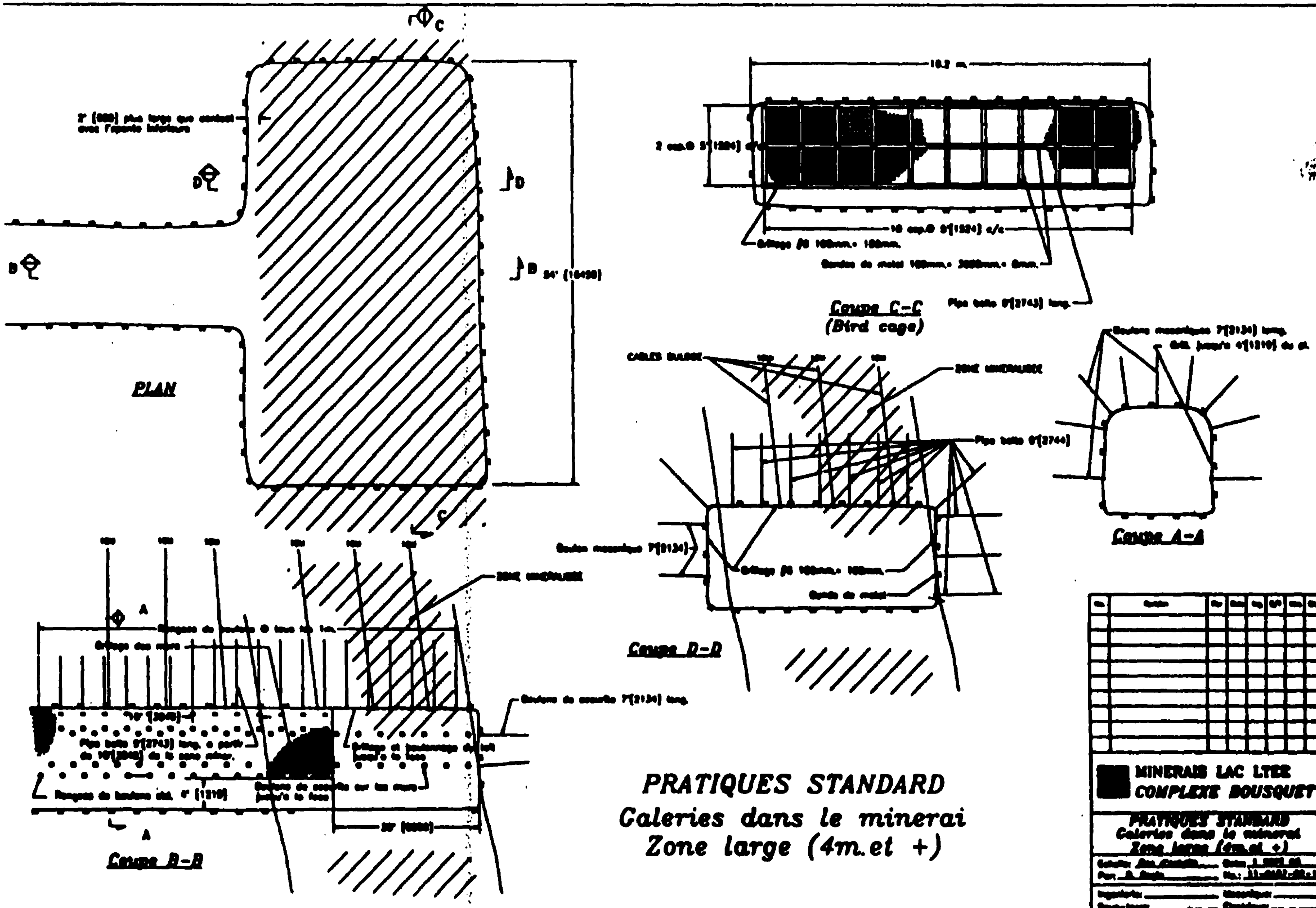


Complexe Bousquet	PUITS #2		ECHELLE : 1:8000	DES.:	DATE 97/08/22	TITRE: COMPLEXE BOUSQUET

TYPICAL DRILLING PATTERN IN LONGITUDINAL MINING



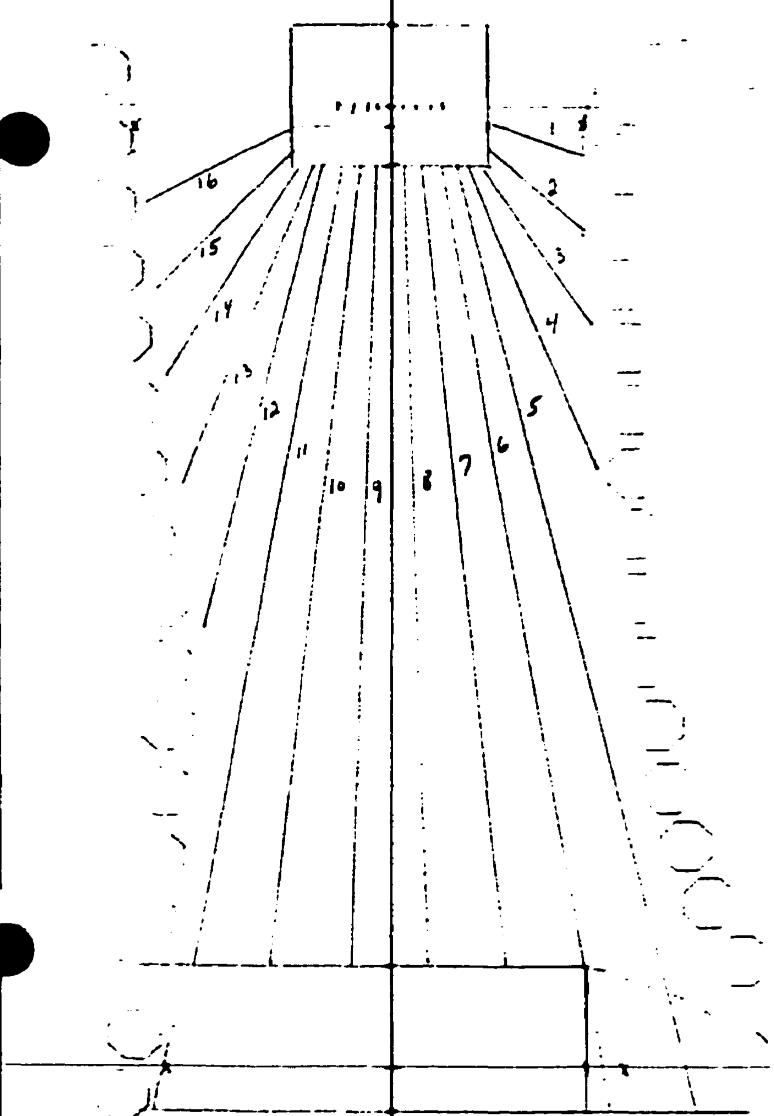
BARRICK


PLAN
10.2 m.
Coupe C-C
(Bird cage)
ZONE MINERALISEE
Coupe A-A
Coupe D-D
Coupe B-B
PRATIQUES STANDARD
Galeries dans le minerai
Zone large (4m.et +)
MINERAIS LAC LTEE
COMPLEXE BOUSQUET

Appendix B: Blasthole Pattern & Blasting Sequences

EVENTAIL: 11

EVENTAIL:



N° DU TROU	12-16	11	10	9	8	7	6	1-5
MET.	1.6	1.1	0.7	0.4	0.4	0.8	1.3	1.6
ANG.								

N° DU TROU

MET.

ANG.

R — 1

N° DU TROU	DEG	LONG MET		LONG FOREE	
1	339°	3.5	ciment	3.5	c
2	316°	4.5	ciment	4.5	c
3	302°	6.5	ciment	6.5	c
4	291°	11.0	ciment	11.0	c
5	284°	32.0	ciment	28.0	cim ✓
6	280°	31.7	ciment	29.0	✓ ?
7	276°	26.5	BT	26.5	✓ c
8	271°	26.4	BT	26.4	✓
9	269°	26.5	BT	26.5	✓
10	264°	26.5	BT	26.5	✓

VUE Sud

7-1 CHANTIER #8

13.6

toutes les longueurs sont mesurees du collet

13.6

13.6 date: MAI 97

13.6 échelle: 1:250

R —

N° DU TROU	DEG	LONG MET		LONG FOREE		
11	260°	26.8	BT	29.0	BT ✓	
12	256°	16.0	ciment	16.0	✓	E
13	249°	11.5	ciment	12.2	✓	c
14	239°	8.0	ciment	9.0	✓	c
15	227°	6.5	ciment	7.6	✓	c
16	208°	5.5	ciment	4.0	✓	c
7						
8						
9						
10		269.4				

COMPLEXE BOUSQUET

Endroit de travail : 7-1 CHANTIER #8

DATE : 30-5-97

Par : [signature]

No. EVENT	No. TROU	LONG. TOT.	G.T	BOUCHON (MÈTRE)	COLLET (MÈTRE)	EXPLOSIFS	No DÉLAI	QTÉ CHARGÉE PAR DÉLAI (KG)	[redacted]	LONGUEUR CHARGÉE	EXPLOSIFS UTILISÉS	REMARQUES
R-1	1	3.5	N	3.5	1.0	TRIMRITO 2C	90	3				
	2	4.5	N	4.5	1.0	TRIMRITO 2C	90	3				
	3	6.5	N	6.5	1.0	LOMEX II	90	13				
	4	11.0	N	11.0	1.0	TRIMRITO 2C	80	10				CASING
	5	13.0	O	12.7	2.0	TRIMRITO 2C	72	10				CASING
	6	13.0	O	12.7	1.0	LOMEX II	64	28				CASING
	7	13.0	O	12.7	1.0	TRIMRITO 2C	28	11				CASING
	8	13.0	O	12.7	1.0	TRIMRITO 2C	28	11				CASING
	9	13.0	O	12.7	1.0	TRIMRITO 2C	28	11				CASING
	10	13.0	O	12.7	1.0	TRIMRITO 2C	28	11				CASING
	11	13.0	O	12.7	1.0	LOMEX II	36	28				CASING
	12	16.0	N	16.0	3.0	TRIMRITO 2C	40	12				CASING
	13	12.2	N	12.2	1.0	TRIMRITO 2C	48	11				CASING
	14	9.0	N	9.0	1.0	LOMEX II	56	19				CASING
	15	7.6	N	7.6	1.0	TRIMRITO 2C	56	6				CASING
	16	4.0	N	4.0	1.0	TRIMRITO 2C	56	3				CASING
R-2	1	3.5	N	3.5	1.0	APEX ULTRA	90	21				
	2	4.0	N	4.0	1.5	APEX ULTRA	90	22				
	3	6.0	N	6.0	1.5	APEX ULTRA	90	39				
	4	11.0	N	11.0	1.5	AMEX	80	65				CASING
	5	13.0	O	12.7	3.0	AMEX	72	66				
	6	10.6	O	—	NE	PAS	CHARGER	—				
	7	10.5	O	10.2	1.5	APEX ULTRA	18	75				
	8	12.0	O	—	NE	PAS	CHARGER	—				
	9	10.5	O	10.2	0.5	APEX ULTRA	16	84				
	10	10.5	O	—	NE	PAS	CHARGER.	—				
	11	10.5	O	10.2	1.5	AMEX	36	59				
	12	12.0	O	11.7	3.0	AMEX	40	59				CASING
	13	10.5	N	10.5	1.5	APEX ULTRA	48	48				CASING
	14	9.1	N	9.1	2.0	APEX ULTRA	56	38				CASING
	15	7.3	N	7.3	1.5	APEX ULTRA	56	50				
	16	5.0	N	5.0	1.0	APEX ULTRA	56	35				

SCEAU

QUÉBEC 108426

2/06/97

EXPLOSIFS TOTAL (KG) ________ ________ Kg/ T.M

TONNES DYNAMITÉES (T/M) ________ ________ Kg / M³

MATERIEL DE DYNAMITAGE

ENDROIT: 7-1 **CHANTIER #** 8

SEQUENCE: 1 | 2 | 3 | 4

 # POSTE: 21 999 3154 708 999

25m/s

EXEL CONSTADET	5M	25M
# 1		2
# 2		
# 3		
# 4		
# 5		2
# 6		
# 7		
# 8		
# 9		2
# 10		
# 11		

EXEL CONSTADET	5M	25M
# 12		
# 13		2
# 14		
# 15		2
# 16		
# 17		2
# 18		
# 19		2
# 20		2
# 22		2
# 24		2

EXEL CONSTADET	5M	25M
# 26		2
# 28		2
# 32		2
# 36		2
# 40		2
# 48		2
# 56		2
# 64		2
# 72		2
# 80		2
# 90		2

EXPLOSIFS

		CODE
50	POCHES AMEX II	1
	POCHES LOMEX II	2
	POCHES LOMEX III (BLEU)	3
	POCHES AMEX WR	4
3	CAISSES TRIMRITE 2C 32X27 metres	5
25	CAISSES APEX ULTRA 65X400	6
	CAISSES SUPER FRAC 32X300 (BAZOOKA)	7
		8

CONNECTEUR DE SURFACE (MS)

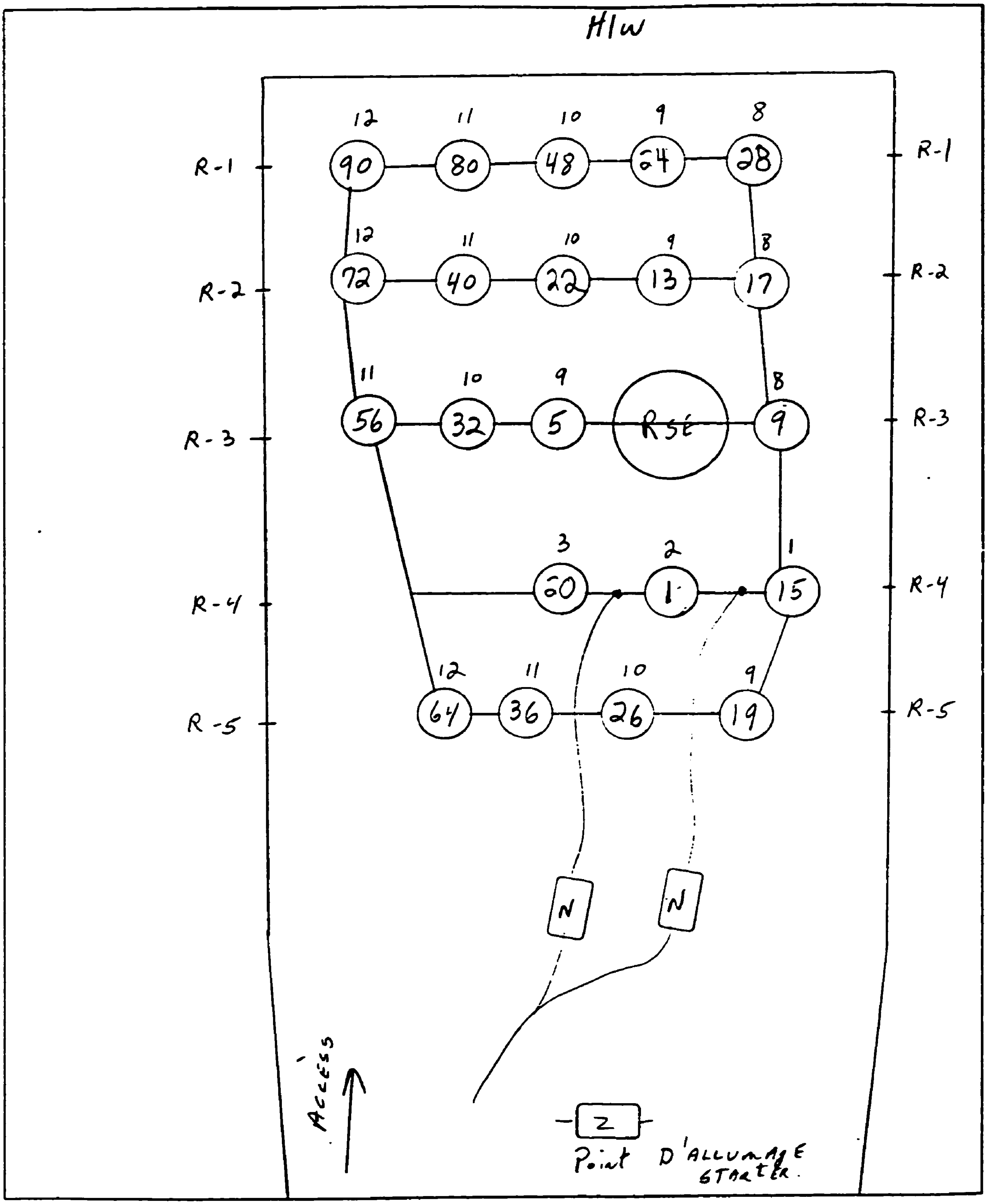
	9
	17
	25
	35
	50
	75
	100
	TOMBEAU

REMARQUES: Blastronic sur ce dynamitage. (C.P)

M:\ING\WK4\DANIELP\MATERIEL.WK4 22/01/97

COMPLEXE BOUSQUET

Endroit de travail : 7-1 CHANTIER #8



Appendix C: Datafiles for Numerical Modelling

Datafile for Numerical Modelling: .dat file

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3 2 18.1 0.0
4 2 22.3 0.0
5 2 34.85 0.0
6 2 41.85 0.0
7 2 71.85 0.0
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10 0 33.3 120.0
11 0 58.8 120.0
12 0 63.0 120.0
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14 0 82.55 120.0
15 0 112.55 120.0
16 1 178.55 120.0
17 1 0.0 150.0
18 0 41.0 150.0
19 0 71.0 150.0
20 0 75.2 150.0
21 0 87.75 150.0
22 0 94.75 150.0
23 0 124.75 150.0
24 1 178.55 150.0
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26 0 41.0 176.4
27 0 71.0 176.4
28 0 75.2 176.4
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30 0 94.75 153.6
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32 1 178.55 153.6
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37 0 92.4 180.0
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39 0 129.4 180.0
```

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47 0 129.4 183.6
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54 0 111.6 213.6
55 0 137.1 213.6
56 1 178.55 213.6
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58 2 41.0 333.6
59 2 71.0 333.6
60 2 75.2 333.6
61 2 145.3 333.6
62 2 152.3 333.6
63 2 161.3 333.6
64 3 178.55 333.6
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2 2 3 11 10 0 0 0 0 2 3 0
3 3 4 12 11 0 0 0 0 2 3 0
4 4 5 13 12 0 0 0 0 2 3 0
5 5 6 14 13 0 0 0 0 1 3 0
6 6 7 15 14 0 0 0 0 3 3 0
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16 18 19 27 26 0 0 0 0 2 3 0
17 19 20 28 27 0 0 0 0 2 3 0
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20 22 23 31 30 0 0 0 0 3 3 0
21 23 24 32 31 0 0 0 0 3 3 0

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37 42 43 51 50 0 0 0 0 2 3 0
38 43 44 52 51 0 0 0 0 2 3 0
39 44 45 53 52 0 0 0 0 2 3 0
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41 46 47 55 54 0 0 0 0 3 3 0
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0 1 1 1 1 0 1 0 1. 5 1 1. 0
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115000. .10
2 .027 1
31000. .21
3 .027 1
49000. .15
4 0.025 1
3100. .4
5 0.025 1
4900. .4
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Datafile for Numerical Modelling: .cab file

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's' 20 20 20 20 20 20 20 20 20 20 20
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'n' 2161 2162 2163 2164 2165 2166 2167 2168 2169 2170 2171
's' 20 20 20 20 20 20 20 20 20 20 20
'3' 200000 .000138 11
'n' 2111 2112 2113 2114 2115 2116 2117 2118 2119 2120 2121
's' 20 20 20 20 20 20 20 20 20 20 20
'4' 200000 .000138 11
'n' 2061 2062 2063 2064 2065 2066 2067 2068 2069 2070 2071
's' 20 20 20 20 20 20 20 20 20 20 20
'5' 200000 .000138 11
'n' 2011 2012 2013 2014 2015 2016 2017 2018 2019 2020 2021
's' 20 20 20 20 20 20 20 20 20 20 20
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'n' 1961 1962 1963 1964 1965 1966 1967 1968 1969 1970 1971
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'n' 1861 1862 1863 1864 1865 1866 1867 1868 1869 1870 1871
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's' 20 20 20 20 20 20 20 20 20 20 20
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's' 20 20 20 20 20 20 20 20 20 20 20
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's' 20 20 20 20 20 20 20 20 20 20 20
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's' 20 20 20 20 20 20 20 20 20 20 20
END
```

Appendix D: Results of Stope Light Measurement by CMS

02-déc-97

BARRICK BOUSQUET

COMPLEXE BOUSQUET

OPTECH

CHANTIER : 7-2-5B

	TENEUR RÉSERVE (g/tm) :	17.6
	TENEUR CÉDULÉ (g/tm) :	17.6
	TENEUR MUCK (g/tm) :	21.3
	TENEUR USINÉ (g/tm) :	N.D.
0	DENSITÉ REMBLAI :	2.2
1	DENSITÉ STÉRILE :	2.8
2	DENSITÉ MINERAI :	4.3

Item	Type	Calculs	Épais. moy.	%	Tonnes
6	MINERAI LAISSÉ DANS LE CHANTIER :				0 tm
7	STÉRILE LAISSÉ DANS LE CHANTIER :				0 tm
8	STÉRILE TRIÉ :				0 tm
9	MINERAI A L'INTÉRIEUR DES LIMITES DE SAUTAGE (14% Dév. enlevé) :	(2x4)			10 706 tm
10	MINERAI RÉEL CASSÉ : (-14% Dév. enlevé):	(2x5)			10 030 tm
11	MINERAI USINÉ :	(10-6)			10 030 tm
12	STÉRILE + REMBLAI EFFONDRÉ :	(12.1+12.2+12.3)			1 819 tm
	12.1 REMBLAI EFFONDRÉ :			28%	515 tm
	12.2 STÉRILE EFFONDRÉ H/W :		0.0 m	1%	26 tm
	12.3 STÉRILE EFFONDRÉ F/W :		1.0 m	70%	1 278 tm
13	STÉRILE USINÉ :	(12-(7+8))			1 819 tm
14	MINERAI EFFONDRÉ CÔTÉ EST :				0 tm
15	MINERAI EFFONDRÉ CÔTÉ OUEST :				0 tm

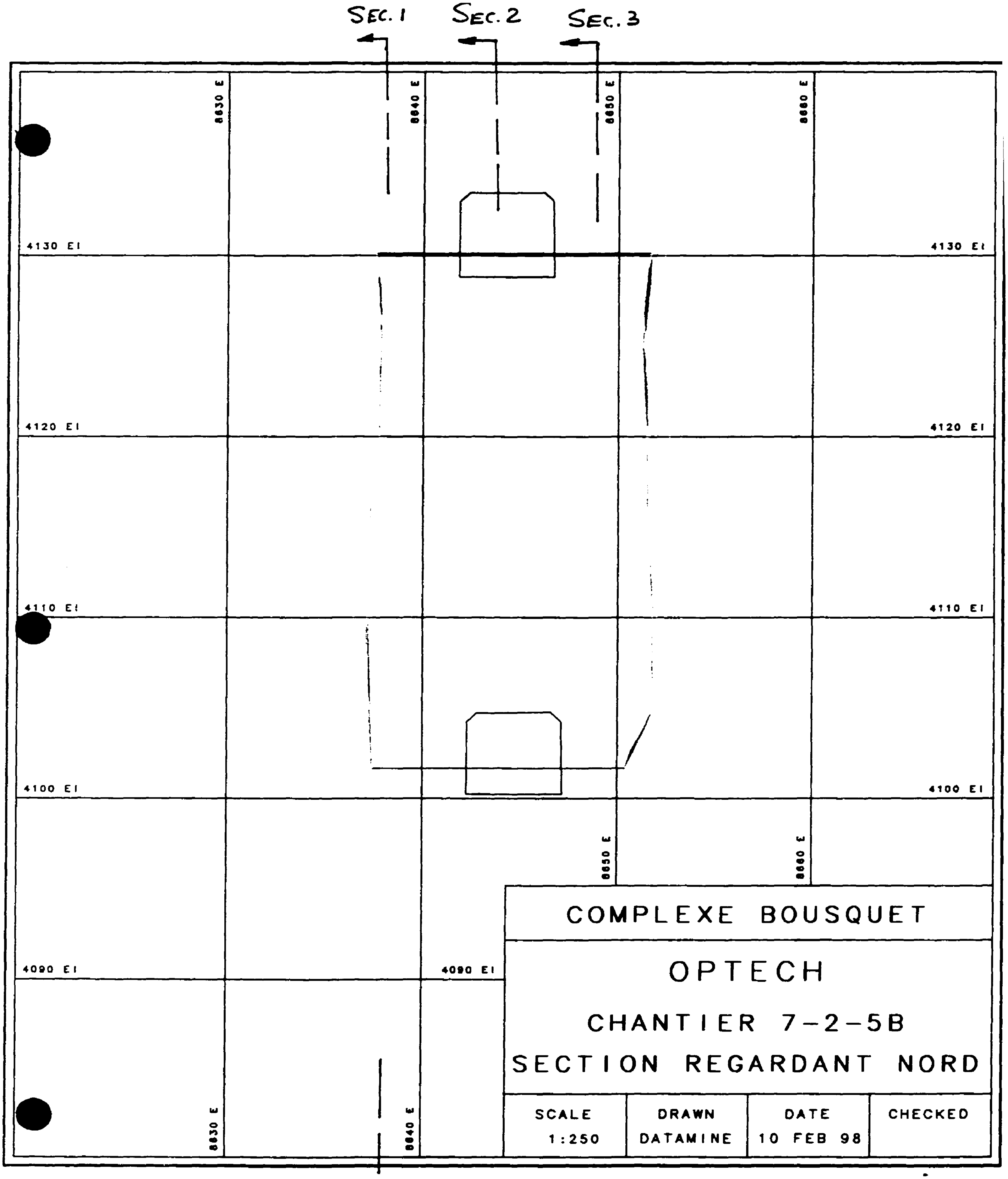
Item	Type	Calculs	Pourcentage
16	DILUTION À L'USINE :	(13/(11+14+15))	18.1%
17	EFFONDREMENT STÉRILE TOTAL :	(12/(10+14+15))	18.1%
18	RÉCUPÉRATION À L'USINE :	(11/9)	93.7%
19	MINERAI CASSÉ LAISSÉ :	(6/9)	0.0%

VOLUME À REMBLAYER : 3412 m³

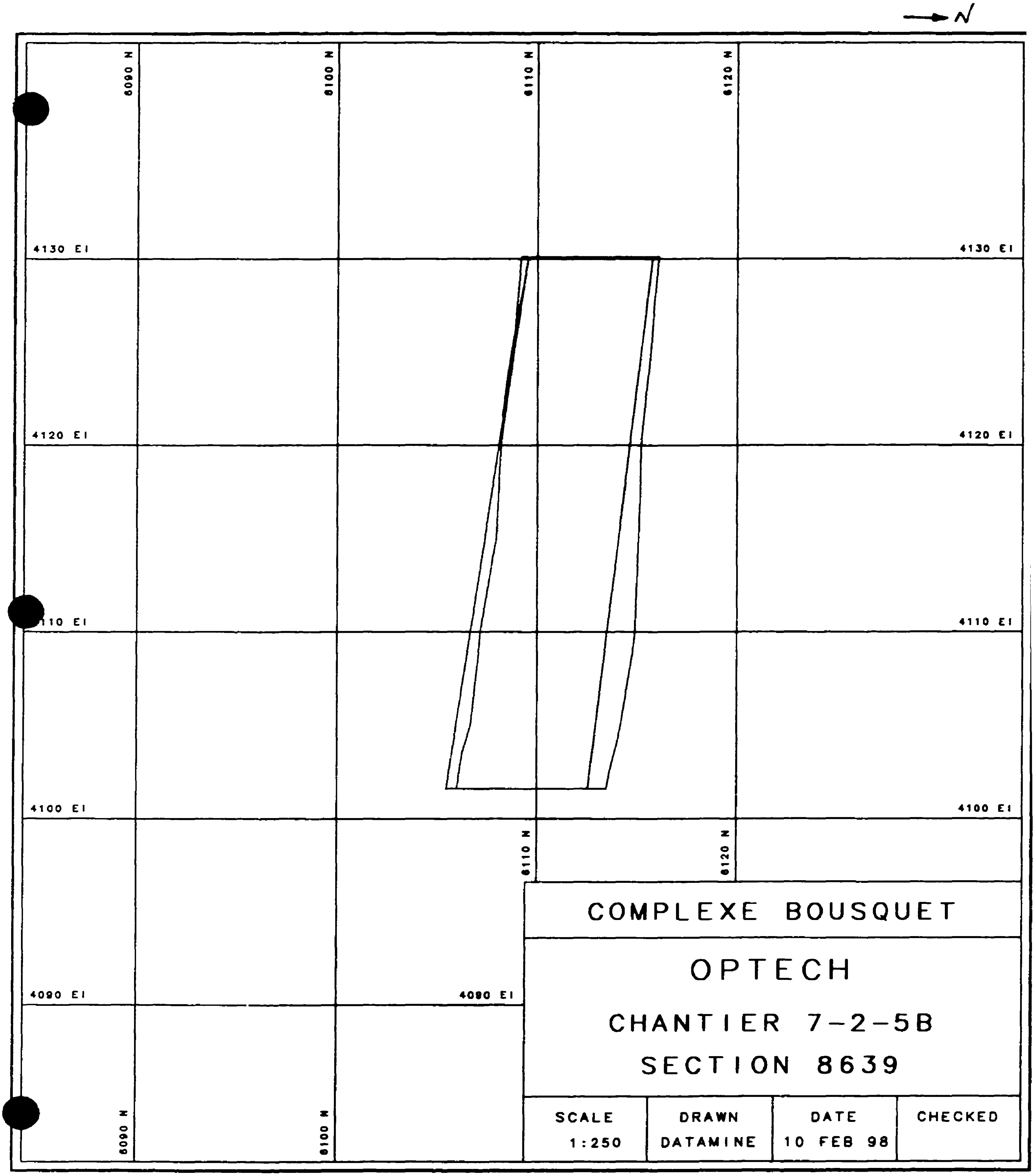
NOTE : Chantier vide lors de l'arpentage. Plan élévation 4101.6 interprété

DATE ARP. : 27-nov-97
PAR: D.G., R.C. & G. Labbé

Par: R. Gaulin
Tech. Prod.



LONGITUDINAL SECTION



COMPLEXE BOUSQUET

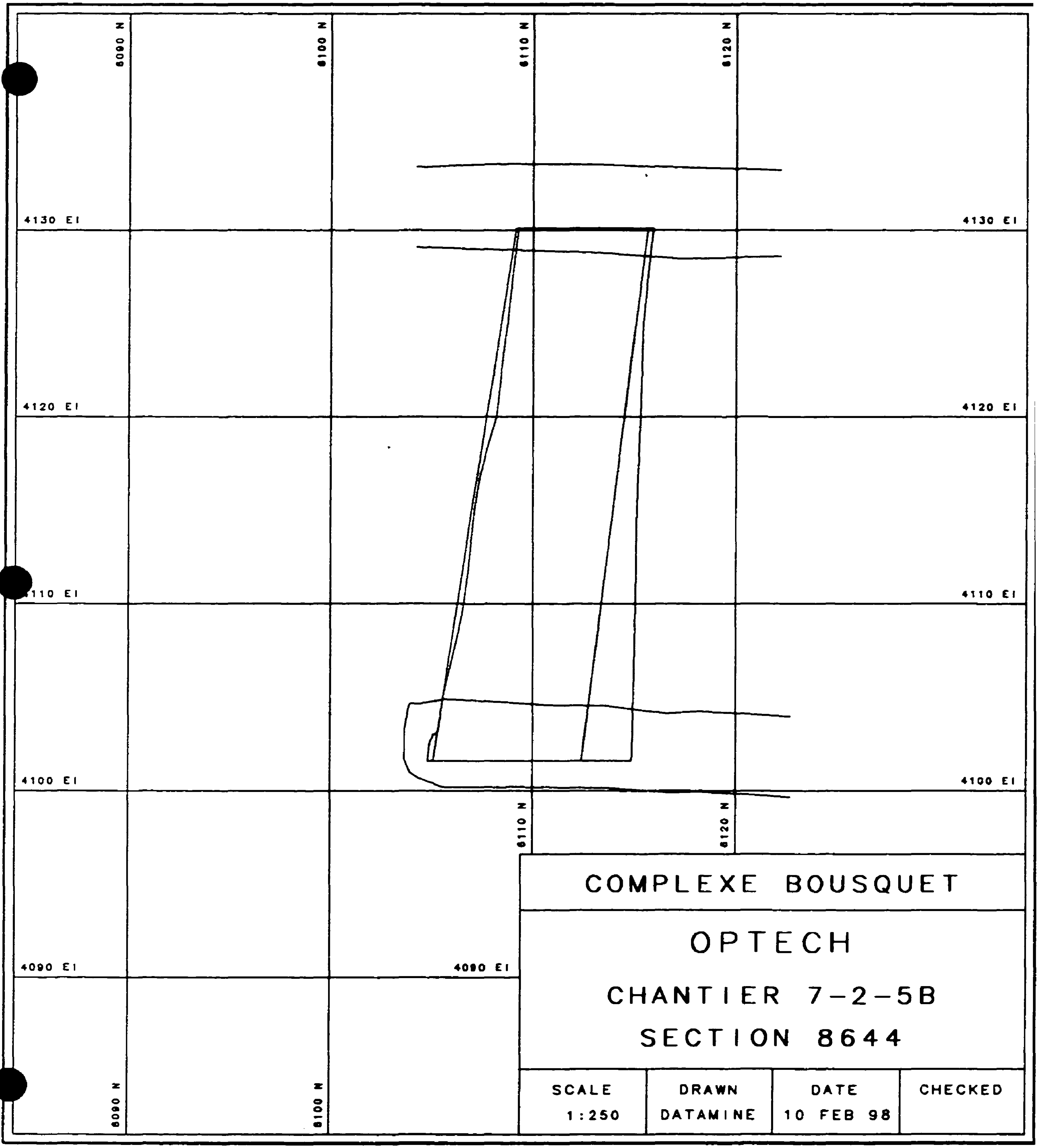
OPTECH

CHANTIER 7-2-5B

SECTION 8639

SCALE	DRAWN	DATE	CHECKED
1:250	DATAMINE	10 FEB 98	

SEC. 1



COMPLEXE BOUSQUET

OPTECH

CHANTIER 7-2-5B

SECTION 8644

SCALE	DRAWN	DATE	CHECKED
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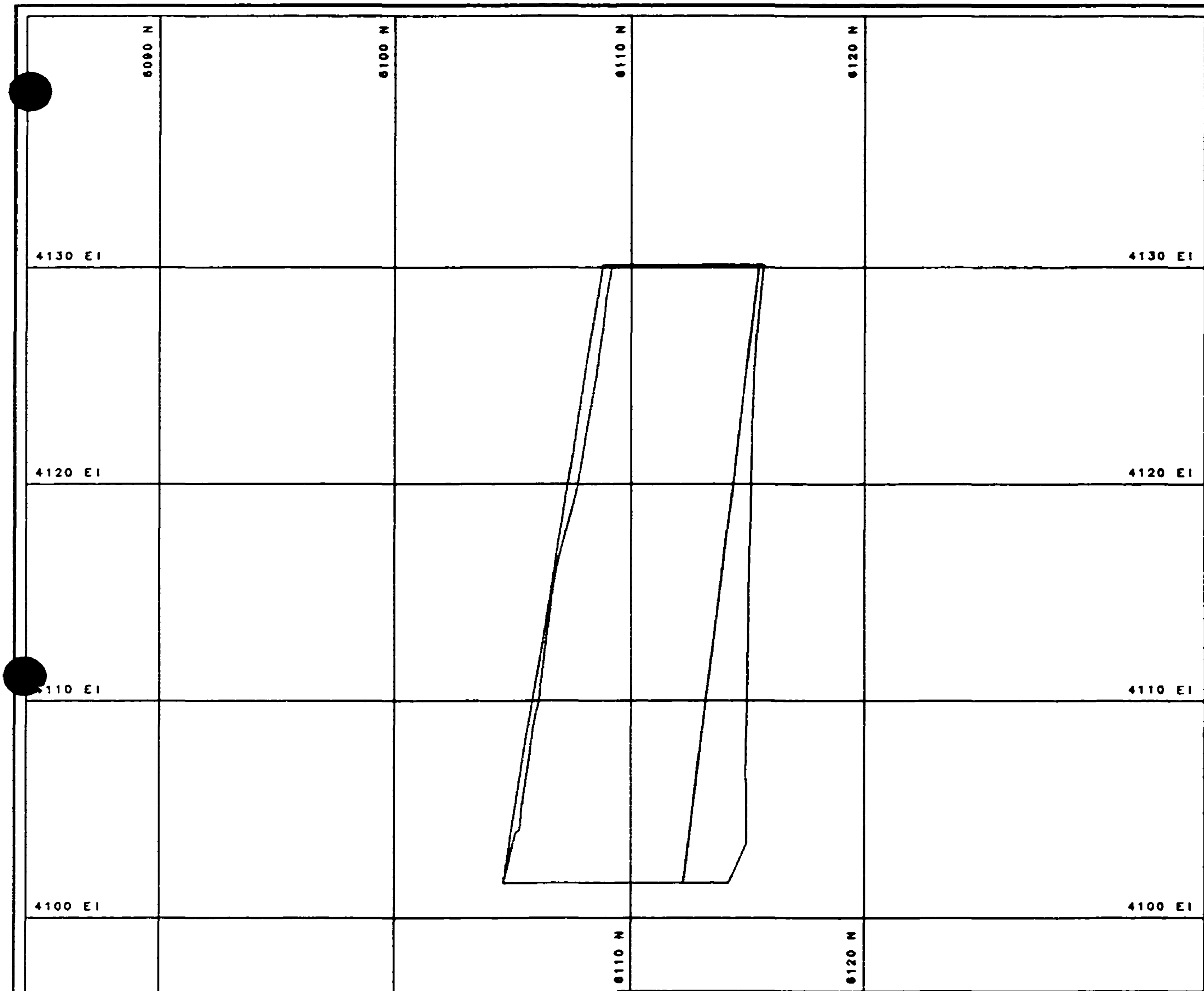

COMPLEXE BOUSQUET

OPTECH

CHANTIER 7-2-5B

SECTION 8649

SCALE	DRAWN	DATE	CHECKED
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