24e Session d'étude sur les techniques de sautage
Les 1 et 2 novembre 2001

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ABSTRACT

The Bousquet 2 mine is a trackless bulk-mining operation utilizing both transverse and longitudinal stoping techniques. The open stope mining with delayed backfill method is used to take advantage of steeply dipping tabular orebody geometry. In ore widths exceeding four meters, 15 m wide x 30 m high stopes are mined transversely. At lateral fringes of the ore zone, where the ore width narrows, stopes are mined longitudinally with strike lengths ranging from 10 to 15 meters, depending on rock mass quality.

This paper reviews stope design approaches employed at the Bousquet 2 mine operation for the extraction of transverse primary, transverse secondary and longitudinal stopes. Variations in stope and slot design, blast design, blast vibration attenuation and stope recovery are highlighted.

1.0 INTRODUCTION

The Bousquet deposit is a lens of massive sulphide and associated disseminated breccia and stringer sulphides, located approximately 50 km east of Rouyn-Noranda, Québec. The orebody is hosted within a series of volcanic rocks, primarily schists of varying quality. The main massive pyrite lens extends from 180 meters below surface and is open at depth. The orebody strikes east-west and dips steeply (80 degrees) to the south.

2.0 DESCRIPTION OF BOUSQUET OPERATION

The Bousquet property is situated in the Abitibi region of northwest Quebec, in the southern part of the Abitibi Greenstone Belt in the Superior Province of the Canadian Shield. The orebody follows an east-west regional structural trend, dipping steeply south in a tabular form, shown in Figures 1 and 2 and is accessed by a shaft, driven to a depth of 1245 meters on the footwall side of the orebody. Shaft stations located at 122 meter intervals access the main levels of the mine.

Rock mass conditions are controlled extensively by the geological history, with the dominant schistose fabric controlling the behaviour of wall rocks in all underground excavations. The host rock mass is strongly schistose, quartz-mica schist.

The schistocity contains sericite and acts as a dominant low friction angle weakness plane in the rock mass. Schistocity planes form platy blocks up to approximately 50 mm in thickness. The lenticular shaped massive pyrite orebody, ranges in thickness up to 20 meters, and has lateral and vertical dimensions of 300 m and 1500 m respectively.

3.0 MINING METHOD

Bousquet 2 mine is a trackless bulk-mining operation, with production levels connected by an internal ramp. Production from below the 9-1 sublevel, (1200 m depth), is hauled by a 40-ton capacity truck up from the 11-3 level (1380 m depth), to be dumped in ore and waste bins located above the 9-0 level crusher, located at a depth of 1230 m. Mined rock is crushed to minus 150 mm and then transported up to surface in 12 tonne skips. Production rates are approximately 1800 tonnes per day.

An internal ramp connects the main production levels with three sublevels which are developed at 30 meter intervals between the main levels. Footwall haulage drifts, running parallel to the orebody, with 50 meter long drawpoint cross cuts provide direct access for removing the ore bearing rock from the stopes.

The open stope mining with delayed backfill method is used at Bousquet mine to take advantage of steeply dipping tabular orebody geometry, and to optimize production rates and recovery. Primary stopes are mined one lift at a time and backfilled with cemented rockfill. Secondary stopes, mined between two primary stopes when the latter stopes have been mined over two lifts, as indicated in Figure 3, are filled with non-cemented rockfill.

4.0 STOPE DESIGN

In ore widths exceeding four (4) meters, stopes are mined transversely. The main Bousquet ore zone is divided into a grid of 15 m wide stopes with sublevels located at 30 meter vertical intervals. Stope widths may range up to 20 meters. When mined, primary stope strike length are 15 to 17 meters, strike lengths for secondary stopes are 13 to 15. Stope sizes are typically 10,000 to 15,000 tonnes.

At the lateral fringes of the ore zone, where the ore width narrows to less than 4 meters, stopes are mined longitudinally. Strike lengths range from 10 to 15 meters, depending on rock mass quality. Stopes sizes range from 4,000 to 6,000 tonnes.

4.1 Primary Transverse Stopes

In the transverse open stope mining method, an expansion slot is developed by enlarging a 1.07 or 1.3 m diameter slot raise to the width of the stope, using parallel hole blasting. Ore is fragmented in the stope using long parallel (primary stopes) or ring-drilled (secondary stopes), and mucked from a drift, orientated perpendicular to the stope strike, at the base of the stope..

Stope production drilling is performed by Tamrock Data Solo drills. The top sill of the primary transverse stopes are excavated to the full stope strike length to permit drilling of parallel 100 mm diameter blastholes, typically at a 2.5 meter burden and 2.0 meter spacing, with an off-center 1.07 or 1.3 meter diameter raisebore slot. Typical schematic primary stope drilling pattern is shown in Figure 4.

4.2 Secondary Transverse Stopes

Observations from blast vibration monitoring and hanging-wall instrumentation, (Henning and Mitri, 1999), and from modelling of rock mass pre-conditioning (Henning *et al.*, 2001), suggest that mining of secondary transverse stopes occur within a lower stress (stress relieved) environment. Underground, this stress reduction is reflected in a reduced requirement for redrilling of production blastholes and fewer accounts of working ground.

In the secondary stopes, 100 mm diameter blastholes are usually drilled in a fan-pattern from a narrow, (5 m wide), top sill access, with a central 1.2 m diameter raisebore slot. Stope production drilling is performed by Tamrock Data Solo drills. Typical secondary stope drilling pattern is shown in Figure 5.

4.3 Longitudinal Stopes

For Bousquet, longitudinal stoping has following positive benefits: (1) Improved wall stability and dilution control. Strike length can be reduced to compensate for low quality hanging-wall or footwall conditions, (2) Reduced pre-production development, and (3) Selective mining - by re-slotting, zones of subgrade material are left in place.

Disadvantages of longitudinal mining include: (1) Reduced productivity, due to long haulage distances and the smaller stope size, and (2) Reduced mining rate, since pillarless longitudinal stoping allows fewer active stope blocks than transverse open stoping.

With longitudinal mining at Bousquet, normally only the first stope in the longitudinal stoping sequence requires a 1.07m diameter raise bore slot. In a variation of common longitudinal, stoping practices, subsequent stope slots are generated from slot blasting against a Styrofoam core suspended against the previous stope end wall prior to backfilling with cemented rockfill. Using this technique, referred to locally as the Eureka mining method, (Trahan, 1995), the Styrofoam provides the void for slot blasting. See Figure 6.

Stope production drilling is performed by Tamrock Data Solo drills. The top sills of the longitudinal stopes are excavated to the full stope strike width to permit drilling of parallel 100 mm diameter blastholes, typically at a staggered 2 meter burden and 2 meter spacing pattern, as illustrated in Figure 7 and 8.

5.0 BLASTING PRACTICES

Production blasting of the transverse and longitudinal stopes is performed with AN-FO (AMEX) explosives in mid-stope blastholes. Lower density AN-FO explosives (AMEX K40) are used in the footwall blastholes. Low energy cartridge explosives (Powersplit) are used in the blastholes located closest to the hanging-wall to minimize hanging-wall blast vibration damage, (Henning *et al.*, 1997).

An example of transverse stope loading practice is illustrated in Figure 9. Blast design parameters for slot and production blasting of both transverse and longitudinal stopes are listed in Table 1. Loading procedures for both AN-FO (AMEX and AMEX K40) and the cartridge explosives are illustrated schematically in Figure 10.

5.1 Transverse Stopes

Transverse stopes are normally mined in three blasts, as illustrated in Figure 11. In primary stopes, the first two blasts widen the slot area to the full stope thickness in 10 to 14 meter lifts. For the stope final blast, the remaining blastholes are loaded full column, to a maximum charge per delay of 175 kg, and fired into the open slot. Typically, the first two 'slot' blasts represent approximately 5% and 20%, respectively, of the total stope volume. The remaining 75% is broken in the third and final blast.

With the secondary stopes, the lower half in the stope excavated by the first two blasts. For the final blast, the remaining blasthole rings are fired inwards, towards the central slot. The combination of a fan-drilling pattern with a lower stress environment, permit the blasting of larger volumes at the initial stages of secondary stope extraction. Usually, the first two 'slot' blasts represent approximately 20% and 38%, respectively, of the total stope volume. The remaining 42% is broken in the third and final blast.

5.2 Longitudinal Stopes

Longitudinal stopes are mined in three or four blasts, depending on the type of slot used, as indicated in Figure 12. A stope with a drilled slot is mined in three blasts, in a manner similar to the primary transverse stopes. The first two blasts widen the slot area to the full stope thickness in 14 meter lifts. For the stope final blast, the remaining blastholes are loaded full column, to a maximum charge per delay of 160 kg, and fired into the open slot. Typically, the first two 'slot' blasts represent approximately 15% and 20%, respectively, of the total stope volume. The remaining 65% is broken in the third and final blast.

Longitudinal stopes utilizing a Styrofoam slot, ('Eureka' mining method), are mined in four blasts. A small initial blast, representing roughly 4% of the stope volume creates a narrow, 10 meter-high excavation along the cavity against consolidated backfill of the previous stope. The second and third blasts, representing approximately 12% and 24%, respectively, of the stope volume, complete the 'slot' blasting. The remaining 60% of the stope volume is broken in the fourth blast.

5.3 Blast Vibration

The severity of production blast vibrations within the transverse primary and secondary stope hanging-walls was monitored using triaxial geophones installed onto a solid, competent wall surface, Henning et al. (1997).

Hanging-wall vibration data was compiled using the peak vector sum velocities of individual blast holes. Non-distinct or overlapping waveforms were omitted from the database, as were blast vibration values likely influenced by air gaps between the blasthole and the geophone. Hanging-wall blast vibration data was statistically analyzed using scaled distance relationships (Atlas, 1987), to determine the Site Factors "K" and "a", used in the following equation:

$$PPV = K\left(R / W^{0.5} \right)^{-a} \qquad (1)$$

Where: PPV = peak particle velocity (mm / second);
R = radial distance from blast center (meters); and
W = explosive charge per delay (kg).

The Site Factors "K" and "a" are functions of the effect of local rock characteristics on ground motion. Factor "K" applies to amplitude whereas "a" indicates vibration attenuation. Calculated Site Factors are listed in Table 2. The lower Site Factors for the secondary stope indicates that a lower amplitude vibration reached the geophones, due to increased hanging-wall vibration attenuation from the blast source.

To predict the impact of individual blastholes on the hanging-wall, explosive-specific Site Factors were calculated from the vibrations generated by individual explosive types. AN-FO loaded 100 mm diameter blastholes, representing 48% and 40% of the total blast populations. Vibration attenuation plots for a typical blasthole, located at 2.5 m from the stope / hanging-wall boundary, and loaded with 100 kg AN-FO, provided in Figure 13, show estimated hanging-wall vibration levels within five meters of the stope boundary.

A blast vibration level of 600 mm/second was selected from published damage criteria, CANMET (1993), as a typical threshold limit for vibration induced blast damage. Using this vibration threshold, Figure 12 indicates that existing blasting techniques negatively influence the hanging-walls of both primary and secondary transverse stopes, to a distance of approximately one meter from the stope/hanging-wall contact. The severity of blast vibration damage diminishes away from the stope boundary. Mid-stope blastholes, typically located at a distance of 2.5 meters from the hanging-wall contact, generated significant peak vibrations into the hanging-wall. Lower vibration levels recorded during mining of the secondary stope were associated with observed de-lamination of schistocity parallel to the stope wall.

6.0 STOPE RECOVERY

6.1 Dilution

Dilution has a direct and large influence on the cost of a stope, and ultimately on the profitability of a mining operation. A review of techniques to quantify the cost of dilution by Pakalnis et al. (1995) has shown that there are several definitions of dilution.

A common definition for recording dilution in use at many mines, including the Bousquet mine is:

$$\% \text{ Dilution} = \frac{\text{Tonnes waste rock milled}}{\text{Tonnes ore milled}} \quad (2)$$

Where: Waste = Wall rock outside of the planned stope boundary;
Ore = Rock planned, drilled and blasted within the stope boundary.

The measurement of mined stope profile has traditionally been difficult, due to the non-entry nature of the open stope mining method. In recent years, accurate surveying of excavated stope surfaces has been made possible with the application of automated non-contact laser rangefinders, such as the Cavity Monitoring System, (CMS), described by Miller et al. (1992). Surveys conducted on each mined stope provide a detailed picture of the stope boundary, from which dilution values are determined.

A CMS has been employed at Bousquet on a systematic basis since 1995. Historically, extraction of the secondary stope panels has resulted in lower levels of dilution and hanging-wall sloughage than that encountered with the primary stopes. Stoping results, summarized from a limited database in a study by Golders (1999), and presented in Table 3, indicated that compared to primary stope results, measured stope dilution was approximately 13% lower in secondary stopes.

The higher percent dilution of the longitudinal stopes reflects the influence of the narrower stope width on dilution calculation. For example, 1 metre of sloughage off the walls of a 15 metre-wide stope results in 7% dilution, whereas 1 metre of slough in a 5 meter-wide stope becomes 20% dilution.

To calculate dilution independently of stope width, the Equivalent Linear Overbreak / Slough (ELOS) approach, as described by Clark and Pakalnis (1997) was used. ELOS values, representing average metres of sloughage per square metre of stope hanging-wall and footwall are included in Table 3.

6.2 Dilution Source Estimation

The dilution source varies between the three stope types. While a complete analysis of the data has not yet been completed, trends of the origin of hanging-wall and footwall dilution were derived from a sampling of stope recovery records. The results of the analysis are summarized in Table 4.

Intrinsic differences between the sources of hanging-all and footwall dilution in the primary and secondary transverse stopes are mainly associated with a reduction in radial confining stress on the secondary stopes, (Henning, *et al.*, 2001).

The source of the longitudinal dilution is a combination of factors including: (1) a generally poor rock mass quality on both stope walls, and (2) elevated stress conditions associated with both local and regional (abutment) mining.

7.0 CONCLUSIONS

The type of blasthole stoping technique employed at the Bousquet mine varies according to stope sequence and ore zone width. Within this range of stopes, blasting design practices have been standardized in terms of drilled hole diameter, range of powder factor, and the type and pattern of explosives used.

Although blast design plays a role, observed differences in blast vibration attenuation and stope recovery are also associated with external factors, such as rock mass quality, stope reinforcement and induced stress.

8.0 ACKNOWLEDGEMENTS

The support of Barrick Gold Corporation - Complexe Bousquet for permission to publish is gratefully acknowledged.

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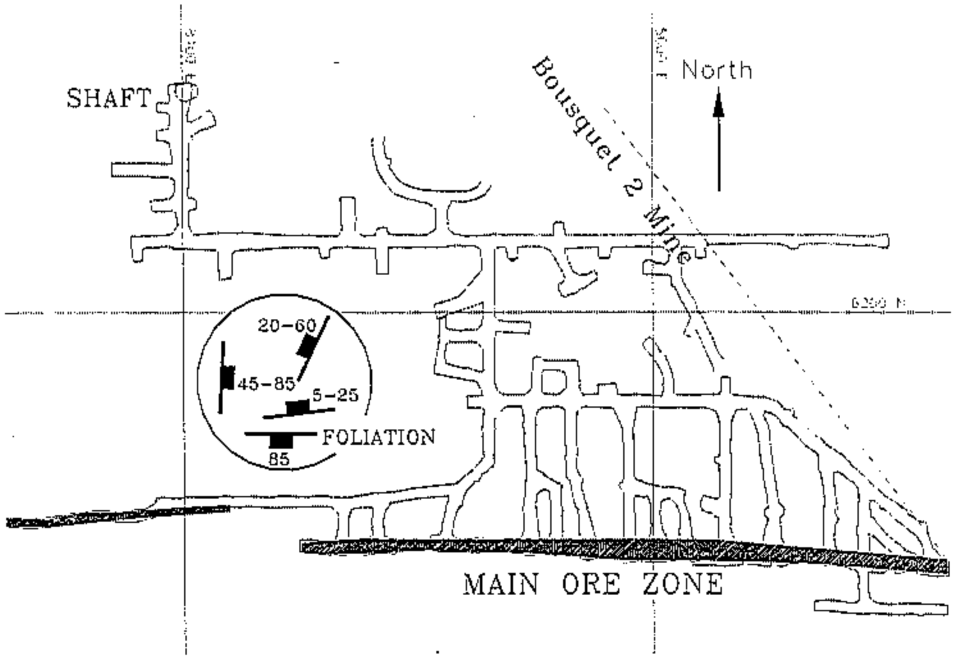


Figure 1. Plan view showing typical mine infrastructure.

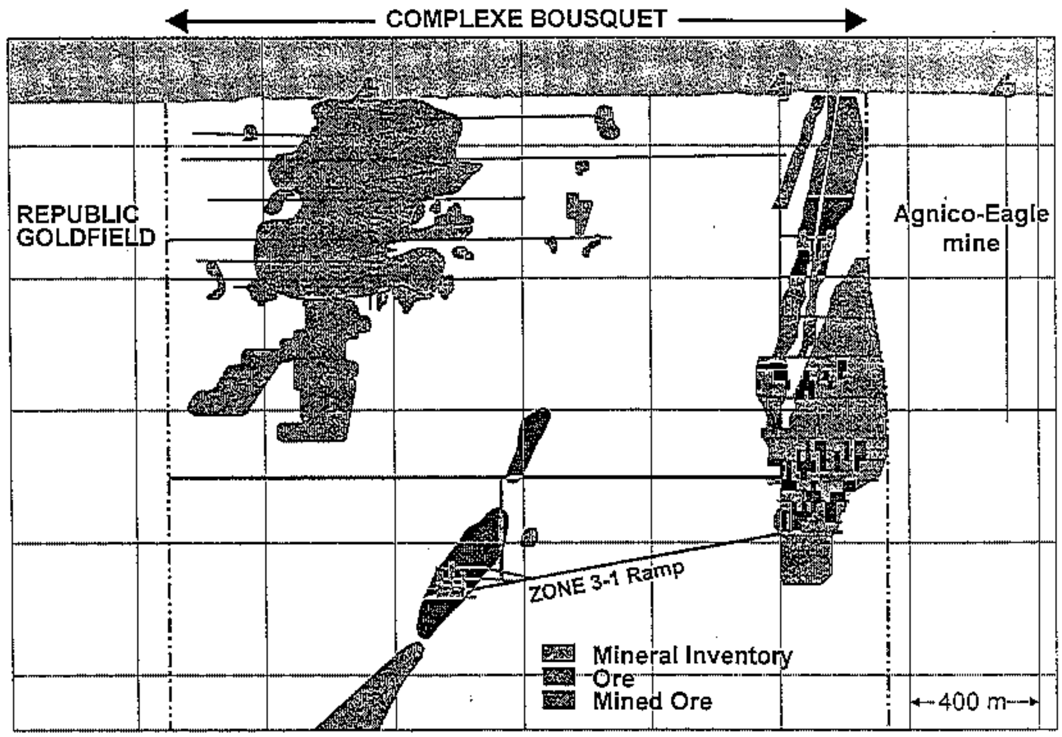


Figure 2. Longitudinal section of Bousquet orebody

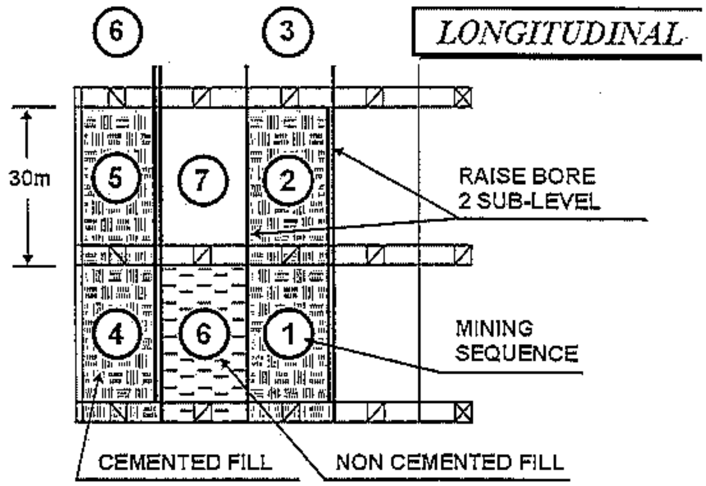


Figure 3. Longitudinal sketch of transverse primary and secondary stoping sequence.

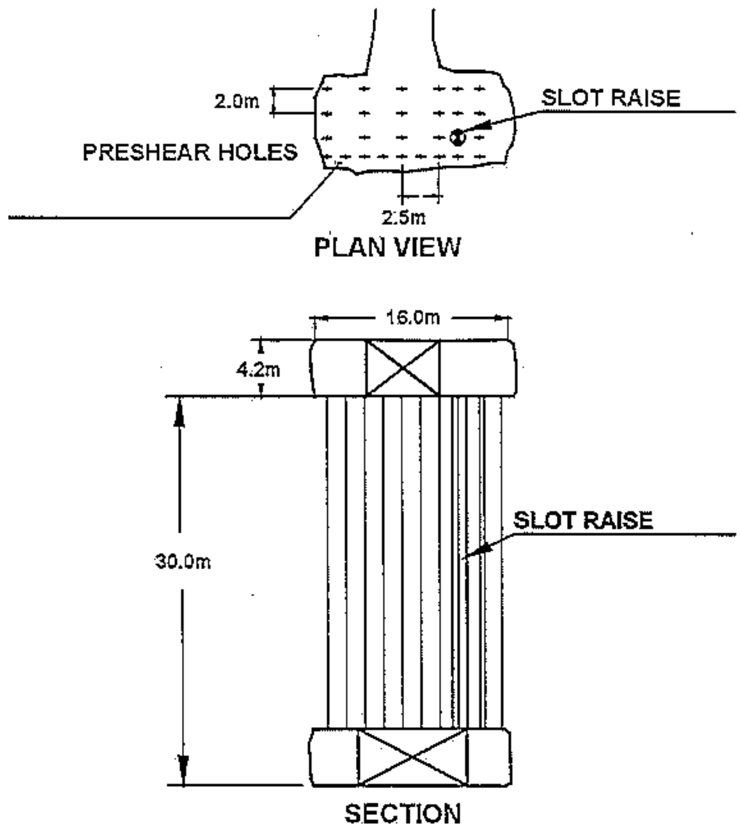


Figure 4. Primary stope schematic drilling pattern and blast sequence.

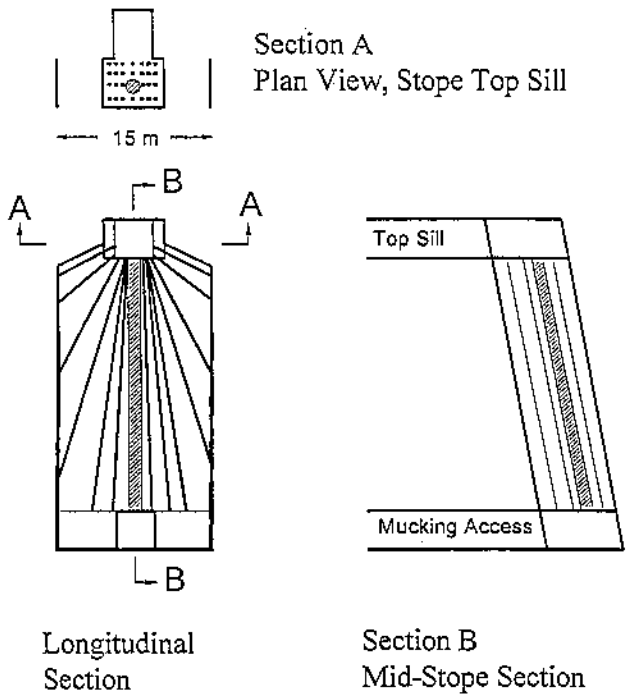


Figure 5. Secondary stope schematic drilling pattern and blast sequence.

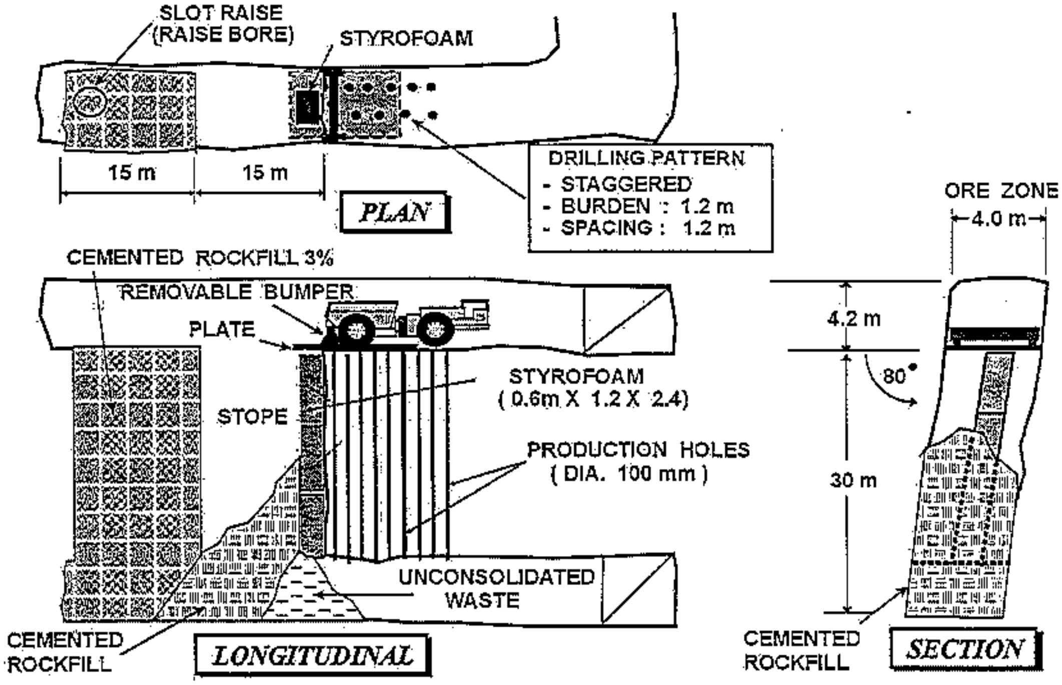


Figure 6. Eureka mining method, (after Trahan, 1995)

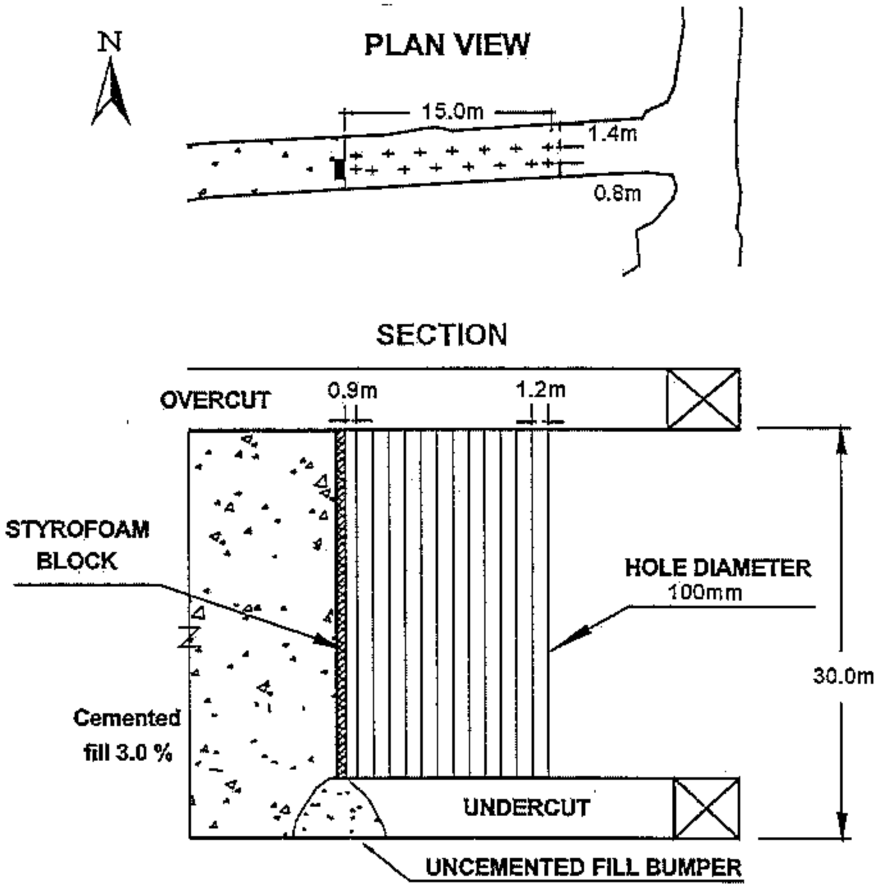


Figure 7. Longitudinal (Eureka) drilling pattern

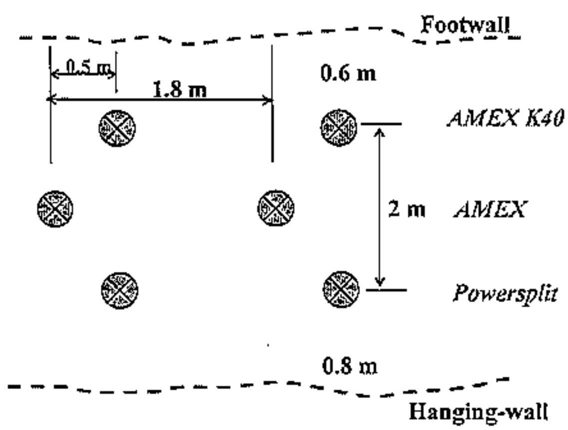


Figure 8. Schematic detail plan of longitudinal stope drilling and loading practice.

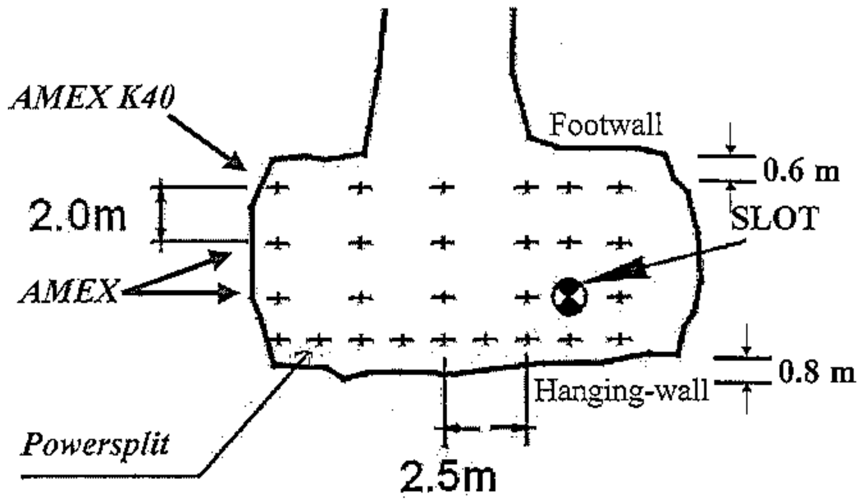
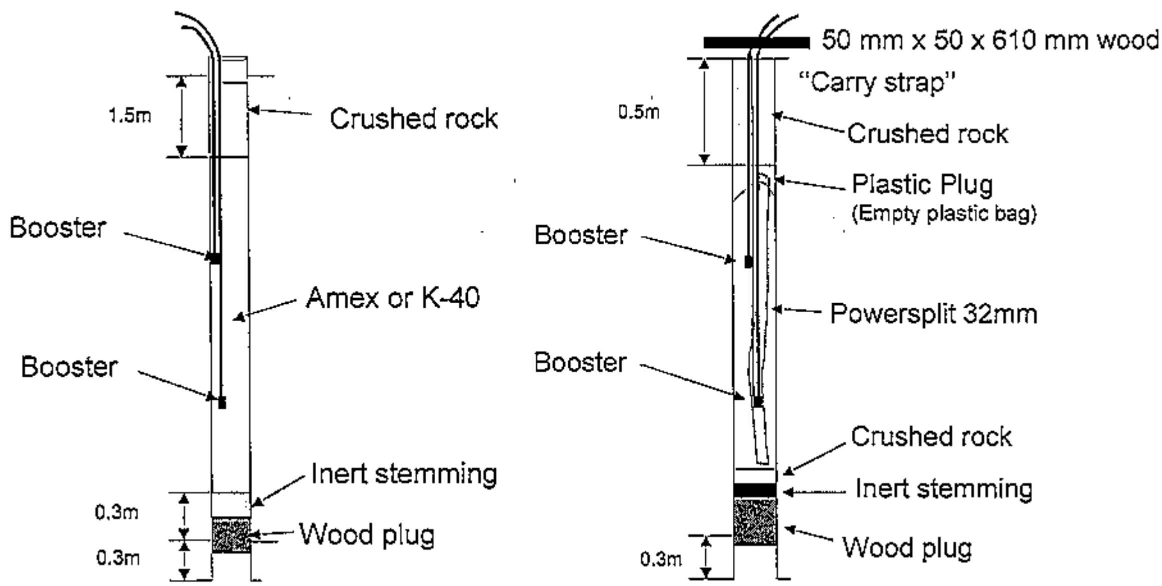


Figure 9. Plan of transverse (primary) loading practice.



AN-FO (AMEX and AMEX K40)   Cartridge explosive (Powersplit)

Figure 10 Schematic diagrams of blasthole charging procedure

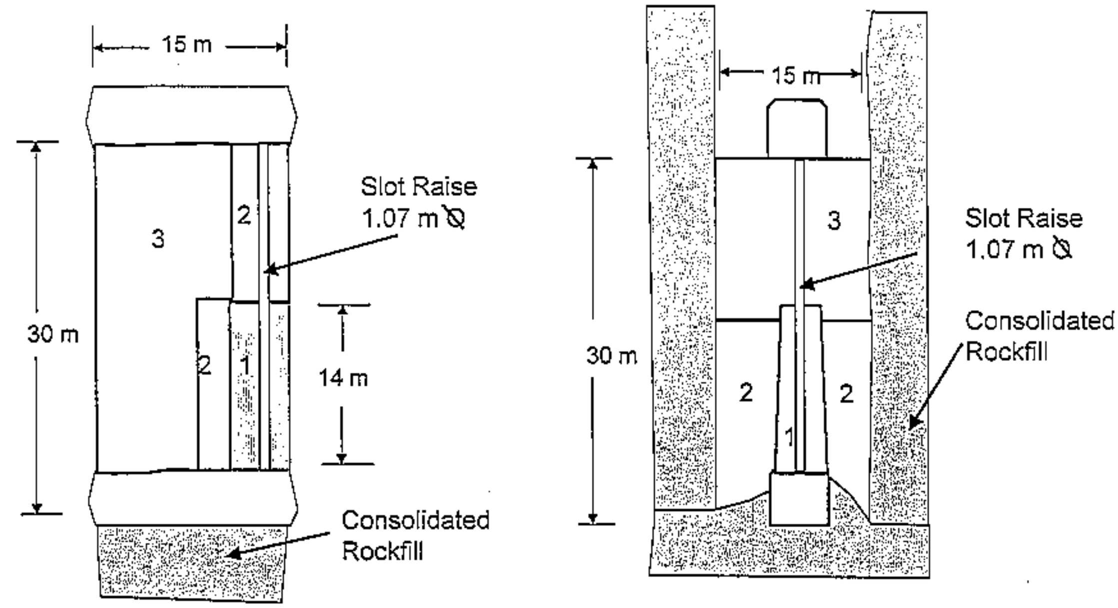


Figure 11(a) - Primary stope

Figure 11(b) - Secondary stope

Figure 11 Example blast sequence for primary and secondary transverse stopes

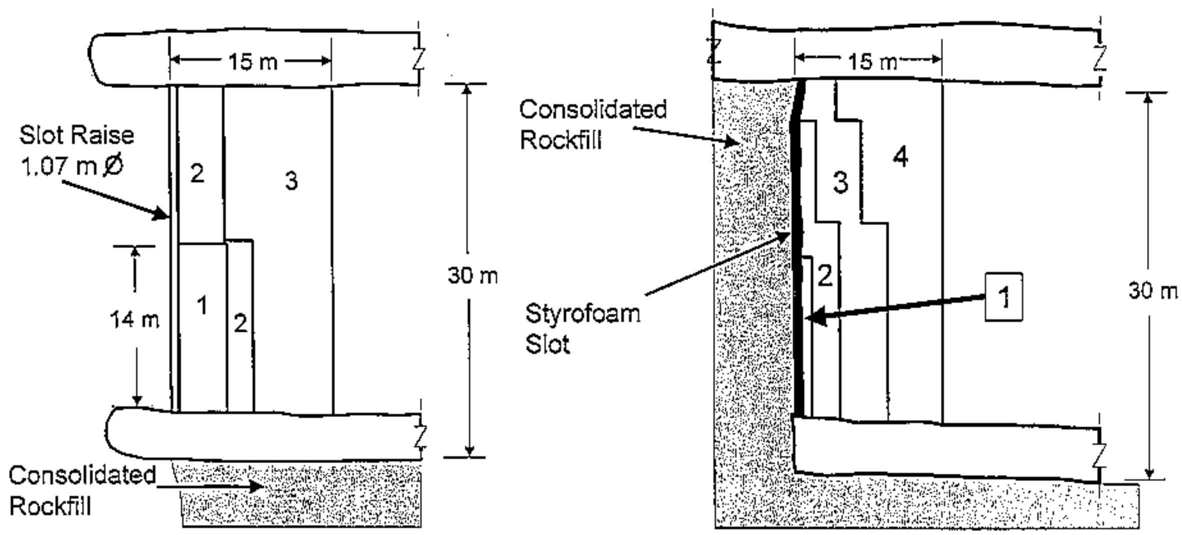


Figure 12(a) Sequence with slot raise

Figure 12(b) Sequence with Styrofoam raise

Figure 12 Example blast sequence for longitudinal stopes

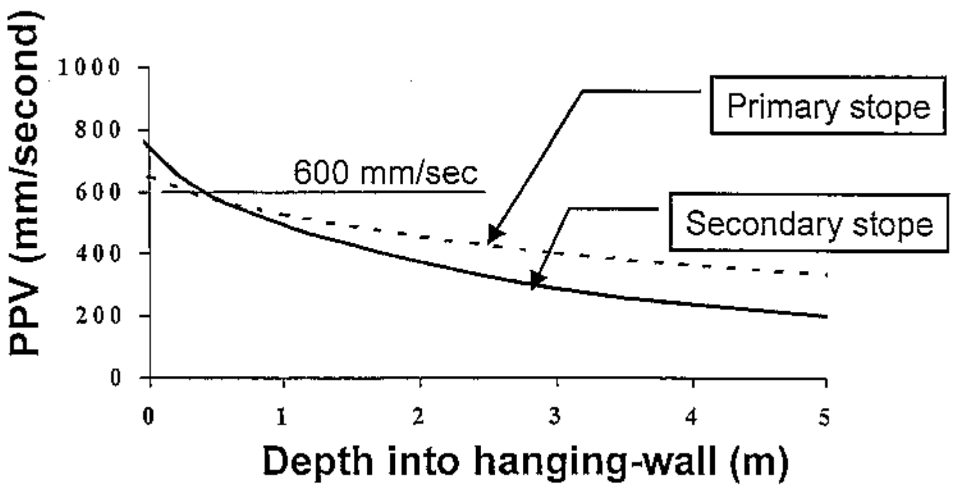


Figure 13 Hanging-wall blast vibration attenuation. AN-FO explosive population: 100 Kg charge, located in blastholes 2.5 meters from hanging-wall contact.

TABLE 1 STOPE BLAST PARAMETERS

	Transverse (Primary & Secondary)		Longitudinal	
	Slot	Square	Slot	Square
Pattern (m)	0.8	2 x 2.5	0.9	2.2
Powder Factor (Kg/Tm)	0.85	0.4 – 0.7	1.1	0.7
Powder Detonated / Delay (kg/delay)	120	175	85	120 - 175

TABLE 2. CALCULATED BLAST VIBRATION SITE FACTORS.

	Primary Transverse Stope		Secondary Transverse Stope	
	K	a	K	a
Total Blast Population	498	1.19	126	0.70
AN-FO Blast Population	283	0.58	141	1.20

TABLE 3 SUMMARY OF SURVEYED STOPE DILUTION AND RECOVERY

Stope Type	Total Dilution (CMS survey)	Recovery	Total ELOS
Transverse Primary	25 %	87 %	2.3 m/m^2
Transverse Secondary	22 %	93 %	1.9 m/m^2
Longitudinal	29 %	89 %	1.9 m/m^2

TABLE 4 SOURCES OF STOPE DILUTION

Primary Transverse stope		Secondary Transverse stope		Longitudinal stope	
HW source	FW source	HW source	FW source	HW source	FW source
85 %	15 %	55 %	45 %	40 %	60 %