

The Goldex Mine mining method

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Abstract

Goldex Mine is an underground gold mine located in the prolific Val-d'Or mining district of northwestern Québec, Canada. Because of its shape and low grade, the Goldex deposit is mined using a hybrid mining method between block caving, long hole stoping and shrinkage. Commercial production began in the middle 2008 at a rate of 7,000 tonnes per day. This paper will explain the mining method and will focus on results of the first mining stages compared to what was expected in the feasibility study.

1 Introduction

The Goldex Mine is a gold mine located near the town of Val-d'Or in the Abitibi region of northwestern Quebec. It began commercial production in mid 2008 at a daily rate of 7,000 t. Because of its relative low grade (2.05 g/t), orebody shape and proximity to the town, a new mining method had to be developed to economically mine the orebody. As with anything new, many uncertainties were left with this new method. This paper will detail how the new mining method was developed and will focus on comparing the results of the first mining stages against what was expected in the feasibility study.

2 Geology

The Goldex Mine is a gold mine located 4 km west of the town of Val-d'Or in the Abitibi region of northwest Quebec (Figure 1). It is part the southeastern portion of the Abitibi Sub province, a typical granite-greenstone terrane and part of the Superior Province of the Canadian Shield. The Abitibi belt is the largest greenstone belt of the world spanning over 85,000 km² (Card, 1990) and also one of the richest mining areas.

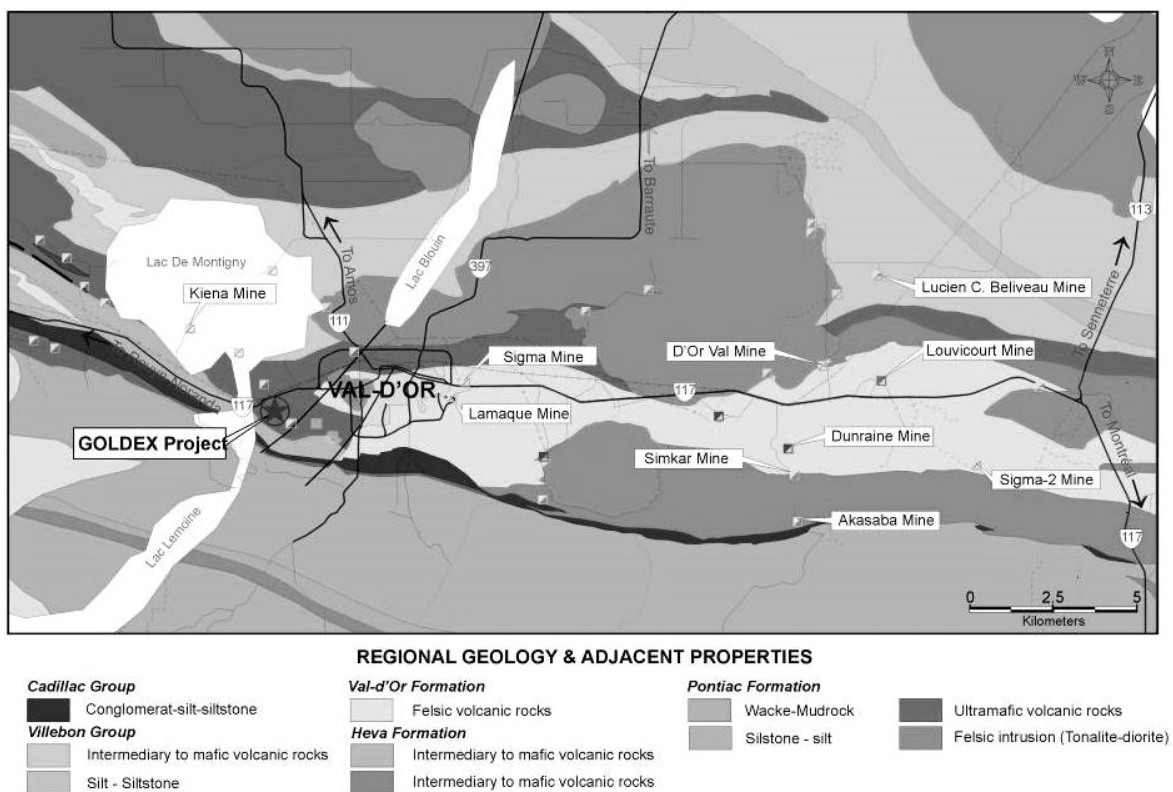


Figure 1 Location of the Goldex Mine and regional geology

The Goldex deposit is hosted within a quartz diorite sill located in a package of mafic to ultramafic volcanic rocks. The geology is oriented generally N280 and dips 75–85° to the north. Plan and section views of the local geology in the vicinity of the mine can be seen in Figures 2 and 3 respectively. The major geological domains are granodiorite, basalt (mafic), mylonite, komatiite (ultramafic) and diabase dykes. The granodiorite hosts the quartz-tourmaline gold bearing veins. The basalt is located both on the north and south of the granodiorite; mylonite and komatiite shears are also located both north and south of the granodiorite and small diabase dykes cut the orebody at an almost perpendicular angle.

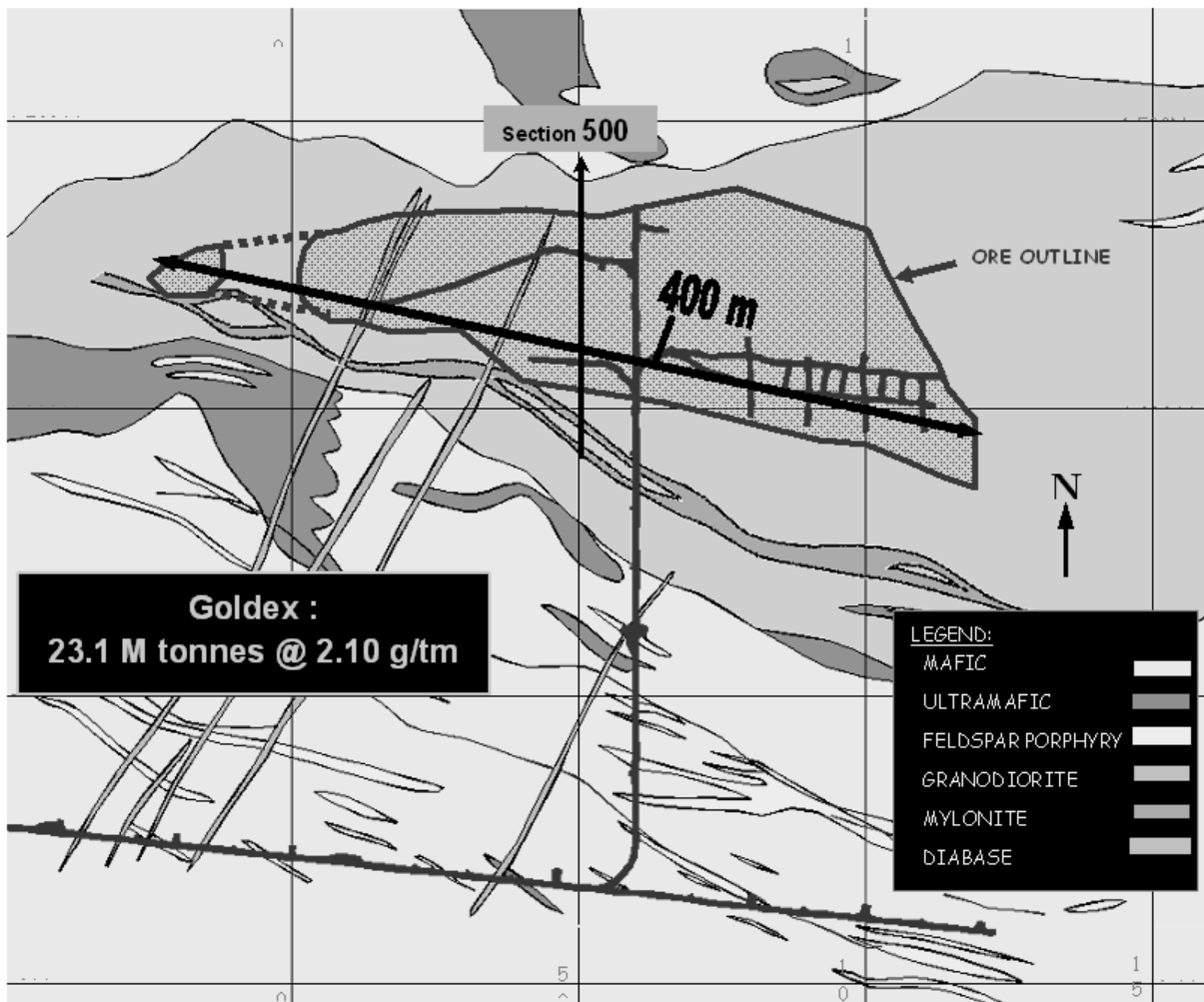


Figure 2 Plan view of geology of the Goldex deposit on level 73

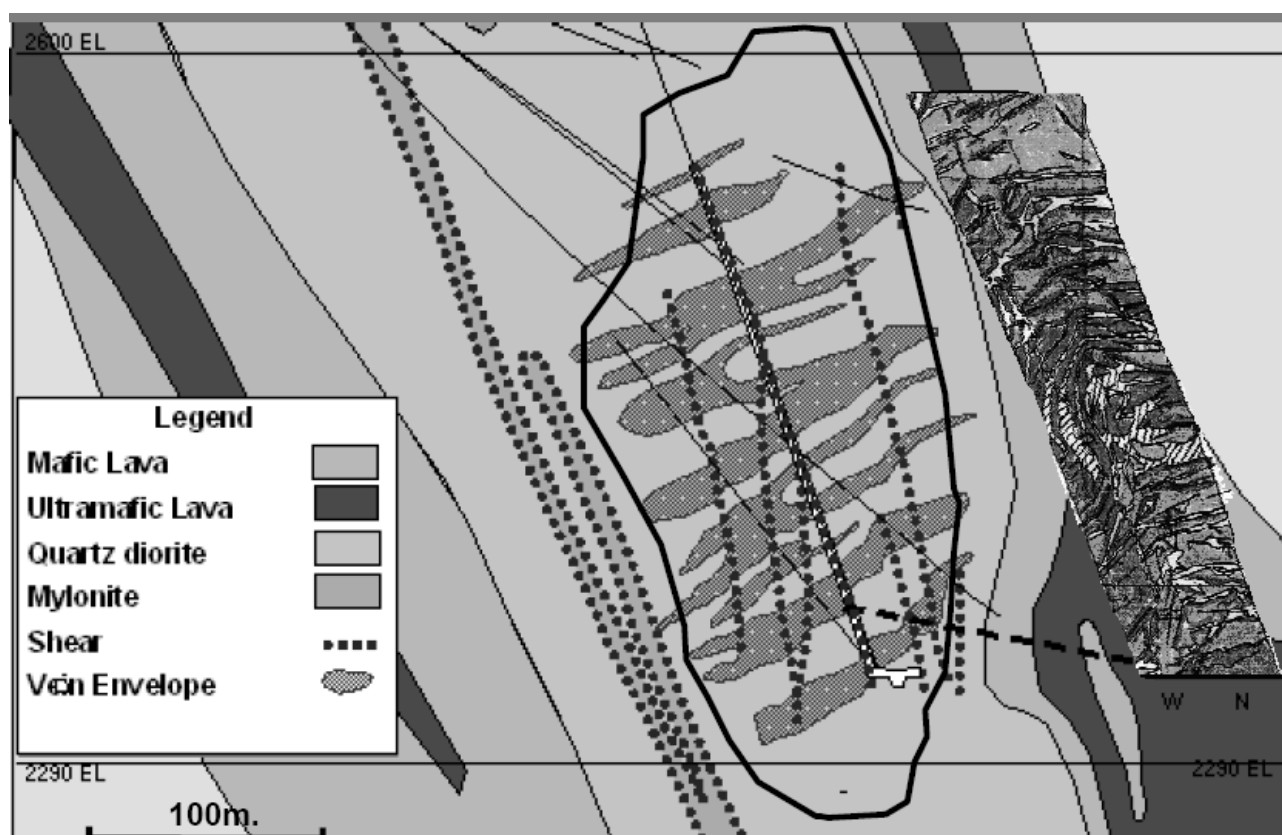


Figure 3 Section 500E of the Goldex deposit looking west

3 Exploration

The property has been staked since the early 1930s with several exploration phases occurring since the 1960s, but because of low grade and high nugget effect which resulted in lower metallurgical recoveries than expected, the orebody was never put into production until 2008 when a new suitable mining method was designed.

The first major phase of exploration occurred between 1963 and 1968 when 30,000 m of surface diamond drilling help define what is now known as the Main Zone. Between 1972 and 1975, 730 m ramp and lateral development was excavated and a 31,000 t bulk sample using conventional mining method was mined. Metallurgical results were deceiving which prompted to stop mining. Sparse exploration work continued until 1985 when a 457 m shaft was sunk and three levels were developed. Another bulk sample of 34,000 t was mined. Once again, following disappointing milling results, mining activities were stopped.

Diamond drilling continued and in 1989, the Goldex Extension Zone (GEZ) was discovered deeper and more to the west of the Main Zone. In 1994, the shaft was deepened to 790 m and over 800 m of lateral development on level 73 enabled a 90,000 t bulk sample. For the first time, sampling and milling grade were similar.

A first prefeasibility study made in 1998 showed the mine could be mined using a bulk method at an approximate cost of \$335 per oz at a time when gold was below \$300 per oz. The project was put on hold. In 2002, with rising gold prices, an update on the prefeasibility study was ordered. A new bulk sample consisting of three alimak raises 250 m high each was undertaken in 2004. These raises enabled a better understanding of the vertical distribution and continuity of veins in the orebody. A positive feasibility study was published in 2005 (Agnico-Eagle Mines Limited, 2005), following the results of the latest bulk sample and the conception of a new mining method. Figure 4 outlines the different exploration phases.

As of 31 December 2008, the orebody is estimated at 25 Mt, grading 2.05 g/t giving a 10 year mine life. Several smaller lenses are also located on the property and could be exploited during the mine life.

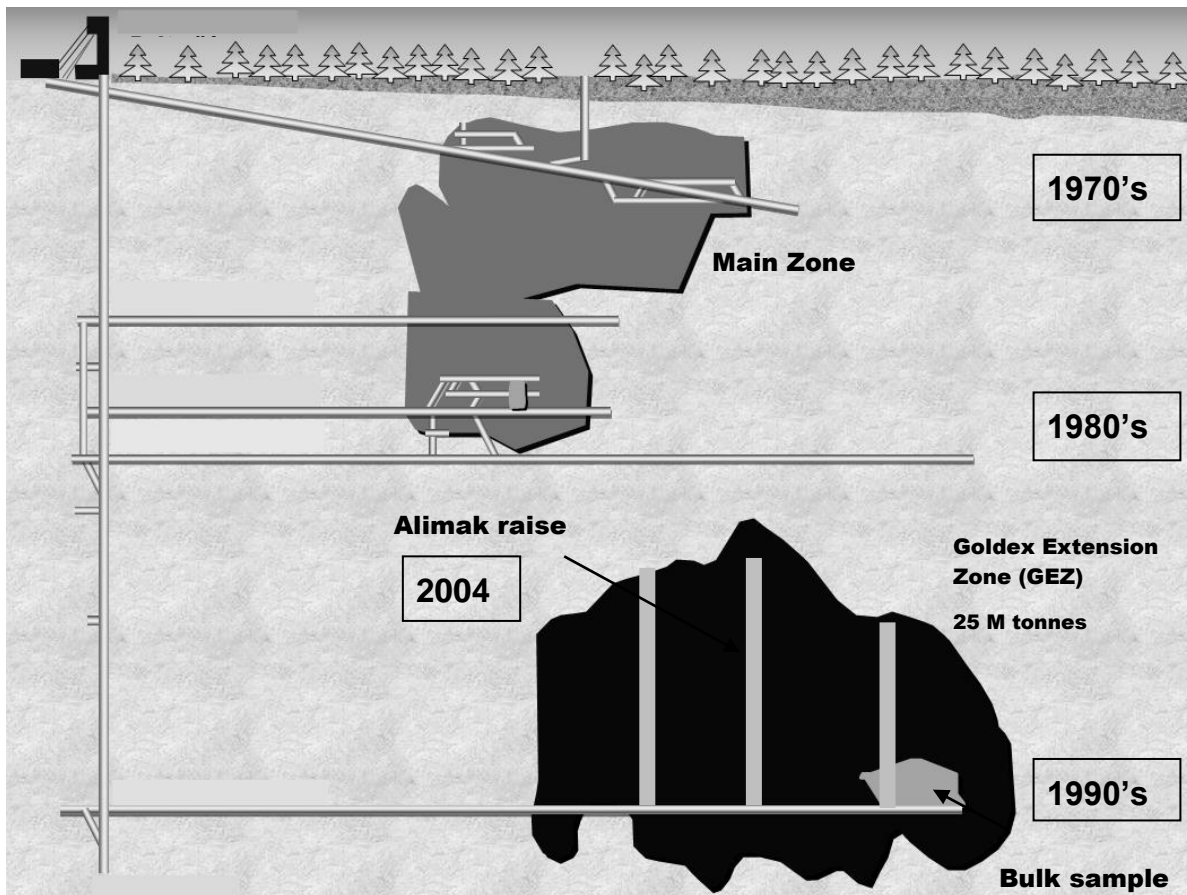


Figure 4 Exploration stages

4 Mining method

4.1 Steps toward the new mining method

The uniqueness of the Goldex Mine is its mining method. To be able to mine such a low grade and discontinuous orebody, only a bulk mining method could be used. A couple of options were looked at over the years.

4.1.1 Overhand transverse long hole stoping

Bharti Engineering Associates came up with the first option in 1996 which consisted of transverse long hole stoping. As shown in Figure 5, the mine design consisted in two drilling levels and one mucking horizon. A total of seven primary stopes 24 m wide by 213 m high were designed to be taken in two blasts with no backfilling. The same number of secondary stopes 30 m wide was to be taken in the same way once the primaries were finished mucking. This method showed many possible problems such as poor recovery mostly related to the 7:1 height to width ratio of the secondary pillars, high dilution and air blasts. It was, however, a good starting point for minimising drilling and production level development.

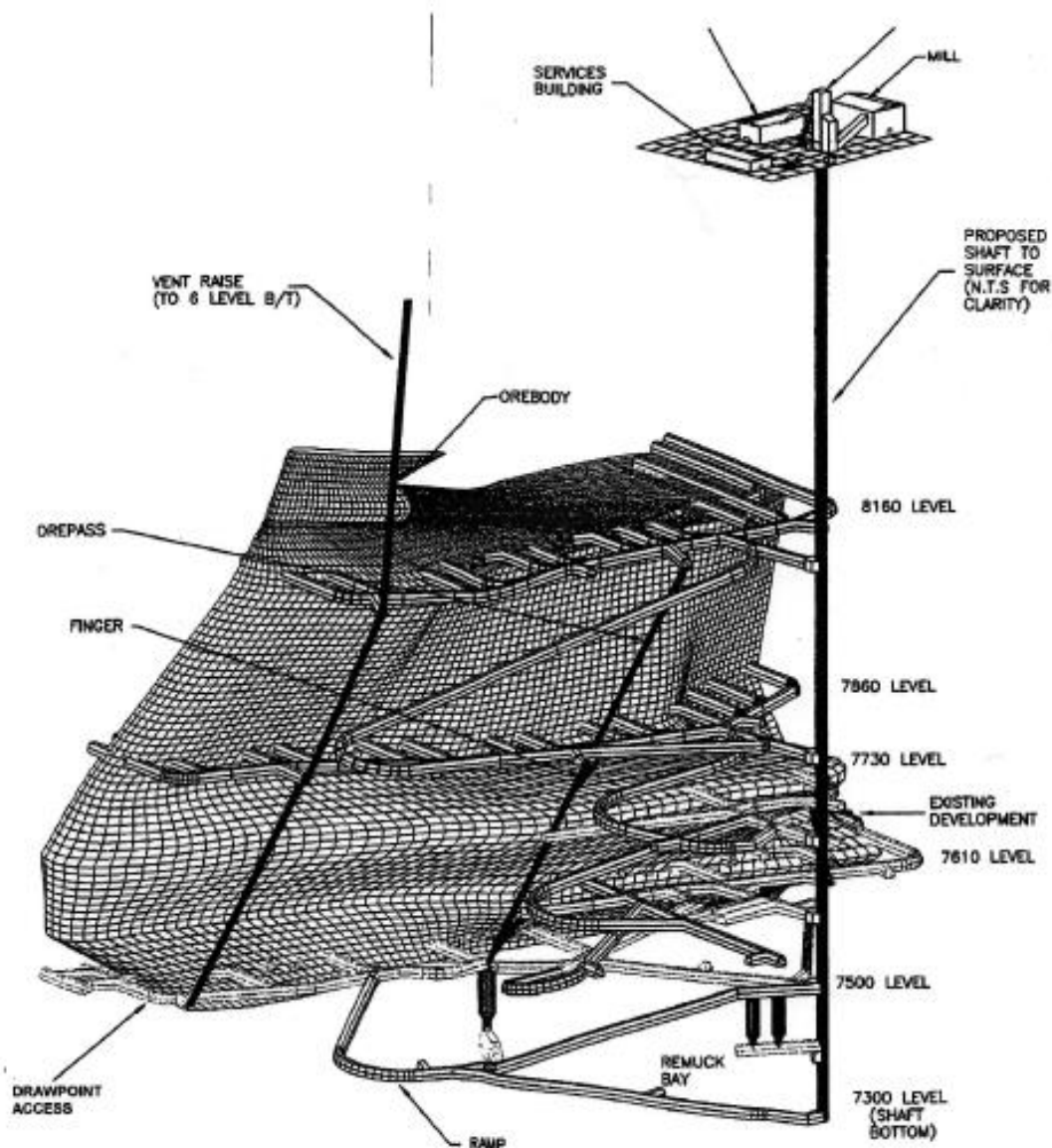


Figure 5 Mining plan layout for the overhand transverse long hole stoping method (Bharti, 1996)

4.1.2 Sublevel open stoping

Following a geomechanical review of the previous method, Itasca Consulting Group modified the mining method to try to overcome the stability problems by leaving muck inside the primaries to buttress the secondary pillars, just like backfill would do. Primary stopes would so be mined overhand, but secondaries underhand in a V-shape mining front. This results in a very large shrinkage stope instead of a very large open stope. Using this method, it was estimated that the break even gold price was about \$335 per oz, which was higher than the price of gold at that time. Figure 6 shows the different phases of mining, phase 1 being the mining of the primary stopes and leaving ore in the stope; phase 2 where the upper part of the secondaries is blasted and finally phase 3 when the lower part of the secondaries is blasted and ore is left inside the large stope, just as in shrinkage mining.

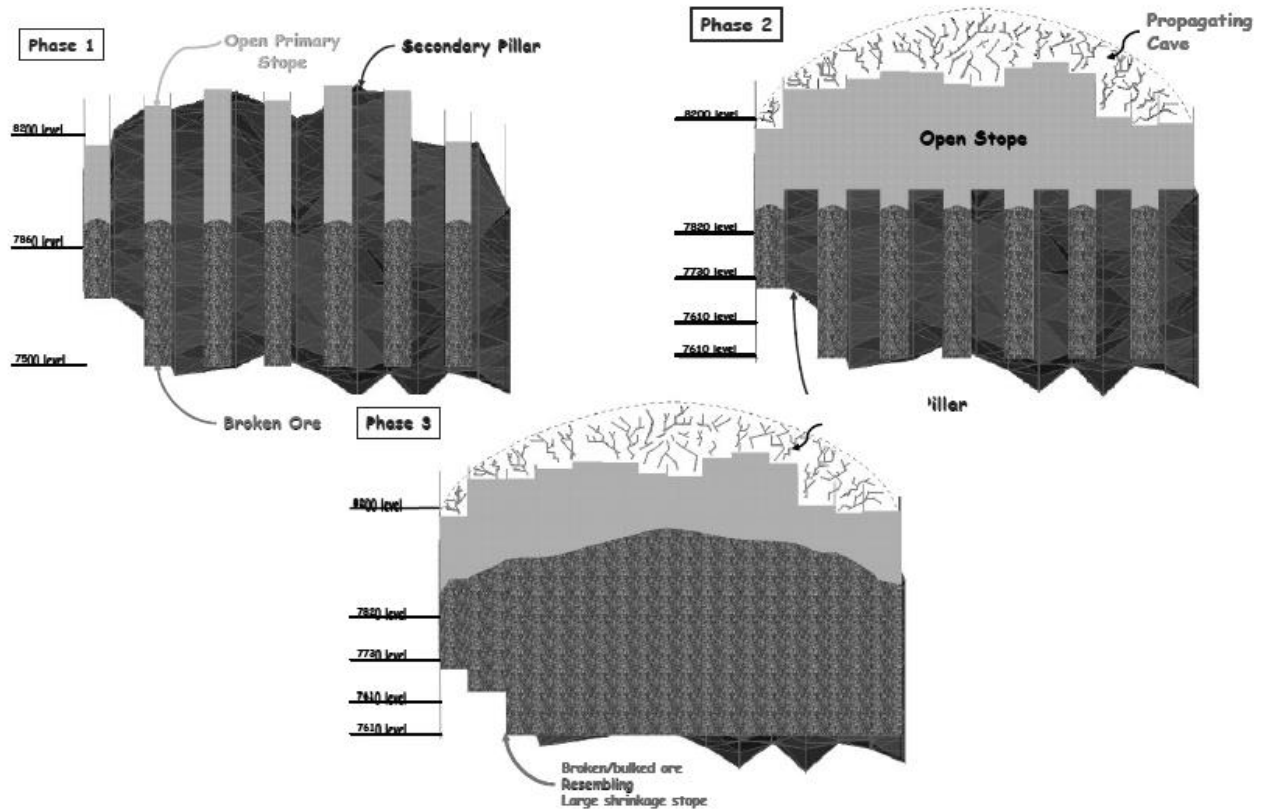


Figure 6 Phases of the hybrid transverse long hole stoping method (Brummer and Board, 1998)

4.1.3 Block and sublevel caving

Both block caving and sublevel caving methods were also briefly looked at in 1998, but had to be dismissed for many reasons. First, if draw control was not carefully implemented, a potential for piping of the cave through to surface was identified as a showstopper. Also, extensive pre-production development was needed for a deposit somewhat relatively small in terms of block caving mines. Finally, the quality of the hangingwall made these methods not well suited.

Block caving was nevertheless reconsidered in 2002. Numerical modelling showed that even using test stope as the undercut, the cave would stall before the orebody was totally recovered as can be seen in Figure 7. The results of this analysis confirmed that a blast assisted mining method was necessary.

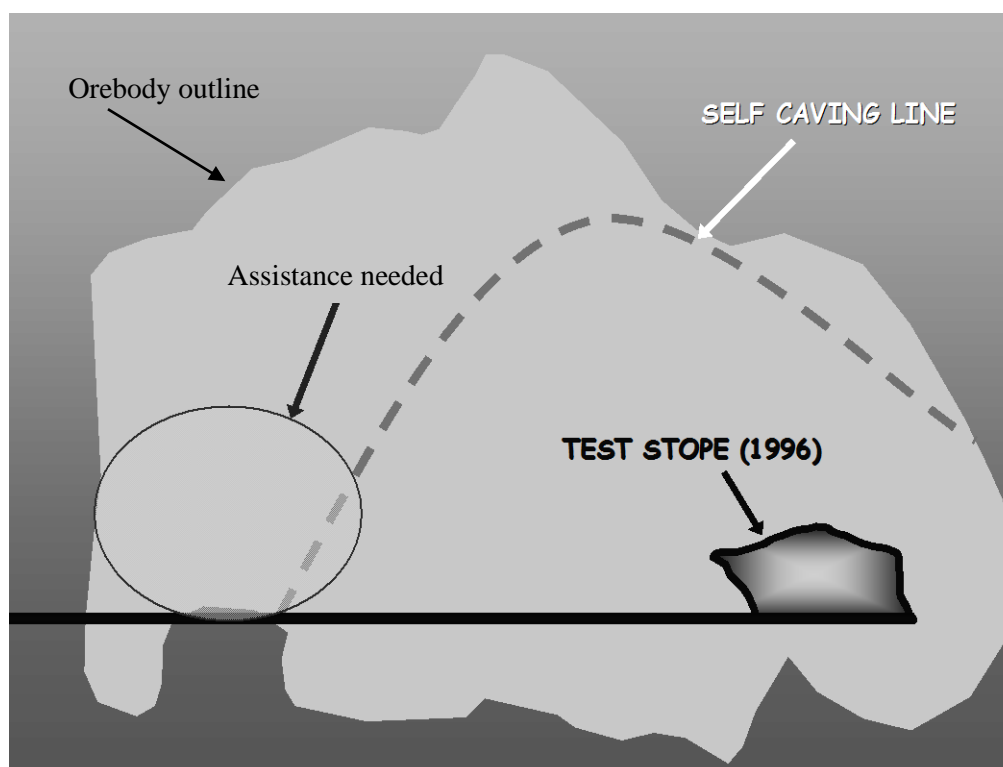


Figure 7 Expected results of block caving

4.1.4 Hybrid VCR blast-assisted caving

In 2003, Itasca came up with a different approach which used Vertical Crater Retreat VCR blasting to initiate and control caving. Two different alternatives were proposed:

- primary/secondary panel caving
- large block caving.

Both methods would require large drilling horizons where very long ITH VCR holes would be drilled. The problem of both methods is the length of the holes required, 125 m for the primary/secondary panel cave and up to 260 m for the large block cave. It was not sure at the time that it could be done effectively and accurately enough. Figure 8 and 9 illustrate both methods proposed by Itasca.

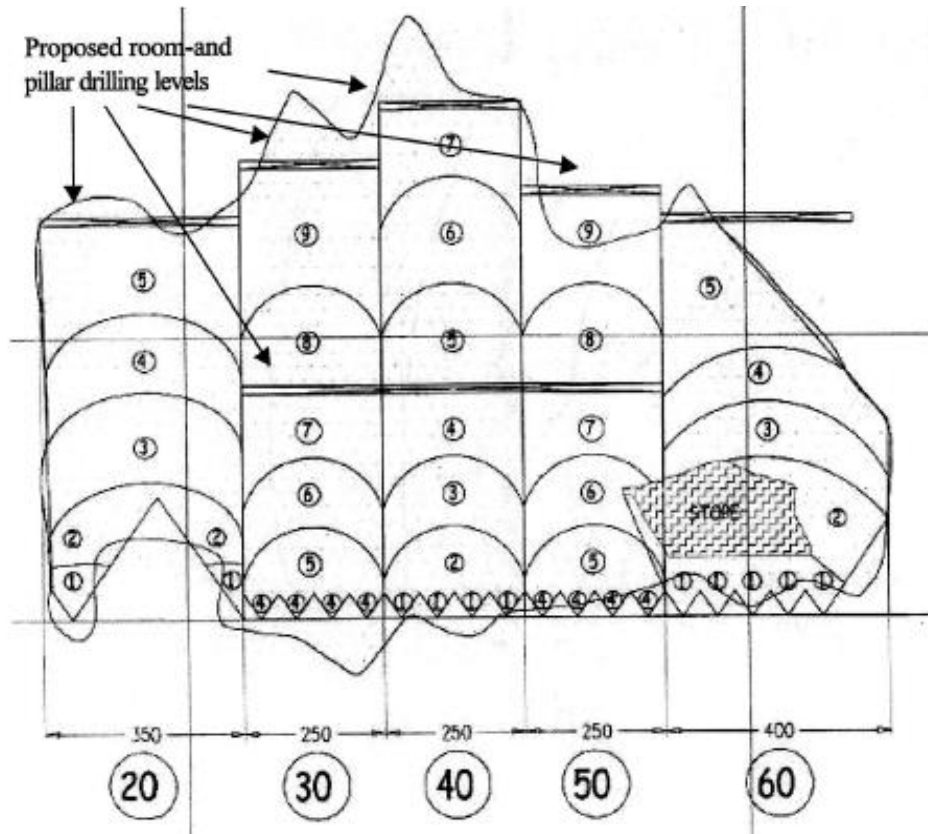


Figure 8 Proposed primary/secondary panel caving with blasting sequence (Brummer and O'Connor, 2003)

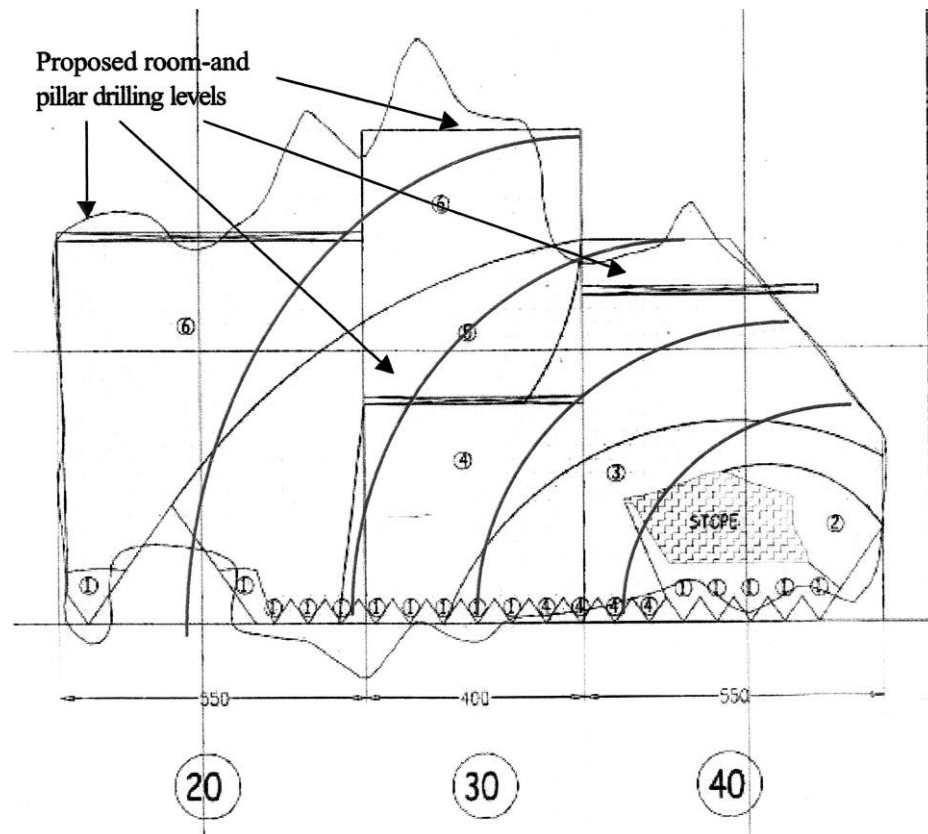


Figure 9 Proposed large block caving with blasting sequence (Brummer and O'Connor, 2003)

4.1.5 *Primary/secondary long hole blasting with three stopes (long hole shrinkage)*

Finally, in 2004, a method deemed effective and feasible was developed using bits and parts of the previous attempts. In this method, the orebody is subdivided in two primary stopes and one secondary. Drilling sublevels are spaced approximately 80 m apart. Drilling is achieved by four ITH drills with 2.8 MPa screw compressors to help flush cuttings in the longer holes and improve both penetration and accuracy. Large blast as much as 4 Mt are necessary in order to keep safe horizontal pillar thickness between the drilling level and the top of the stope. As much ore as possible is left in the stope to act as support for the walls. In fact, drilling and blasting of the 25 Mt will be achieved in the first four years while mucking will be going on for 10 years. Only one mucking horizon is necessary with ore being drawn from trenches at the bottom of the orebody. In order to extract 7,000 tpd, the haulage level contains 46 drawpoints located in nine different trenches. Production is achieved with only two 11.5 m³ scoop trams having 20 t loads, a third one being used as a spare.

This method allowed to satisfy the four criterion imposed for an economical exploitation:

- minimise development
- minimise wall sloughing
- achieve productive fragmentation
- have recoverable and stable pillars.

At the same time, this method had many advantages:

- large size operation as block caving
- walls stability as shrinkage
- fragmentation of long hole
- flexibility of blasting and mucking as VCR.

Figure 10 details the sequence of development, drilling, blasting and mucking involved in this method, Figure 11 details the long section and Figure 12 gives an overview of the haulage level. The main mining equipments, 11.5 m³ scoop trams and ITH long hole drills are shown in Figures 13 and 14 respectively.

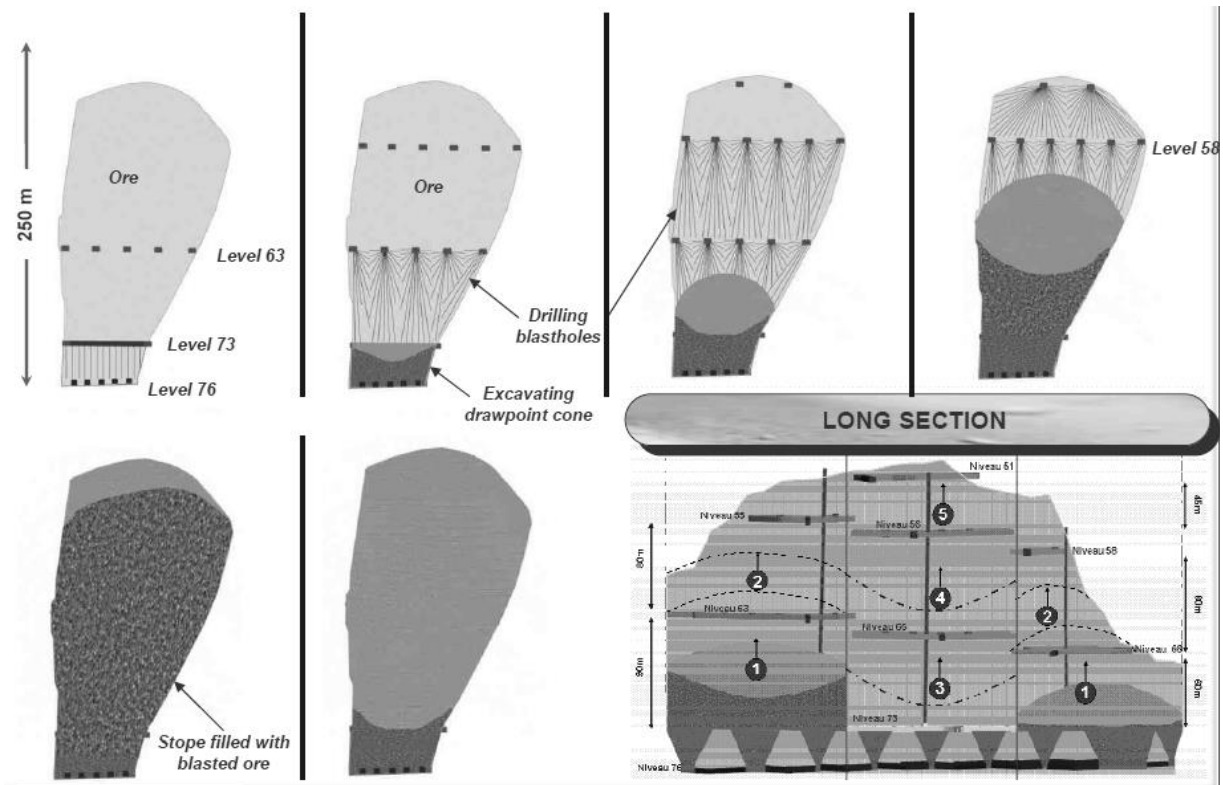


Figure 10 Schematic representation of the long hole shrinkage mining method

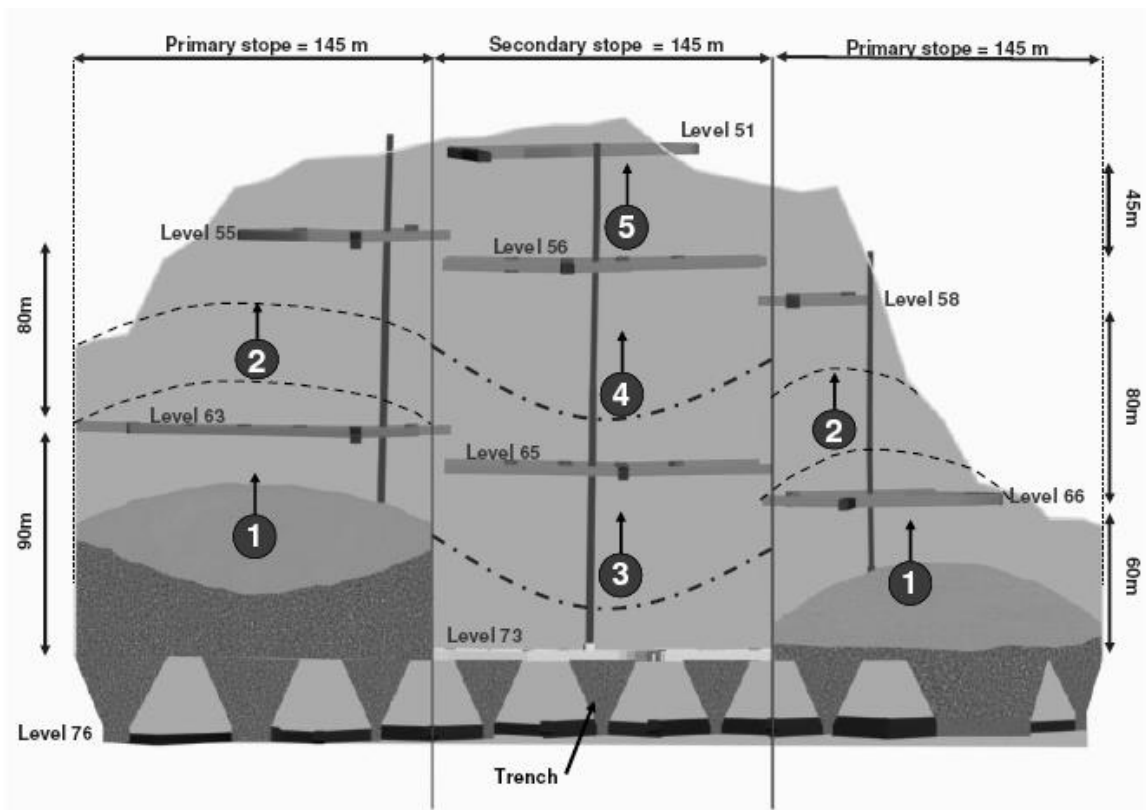


Figure 11 Long section showing the blast sequence of the long hole shrinkage mining method

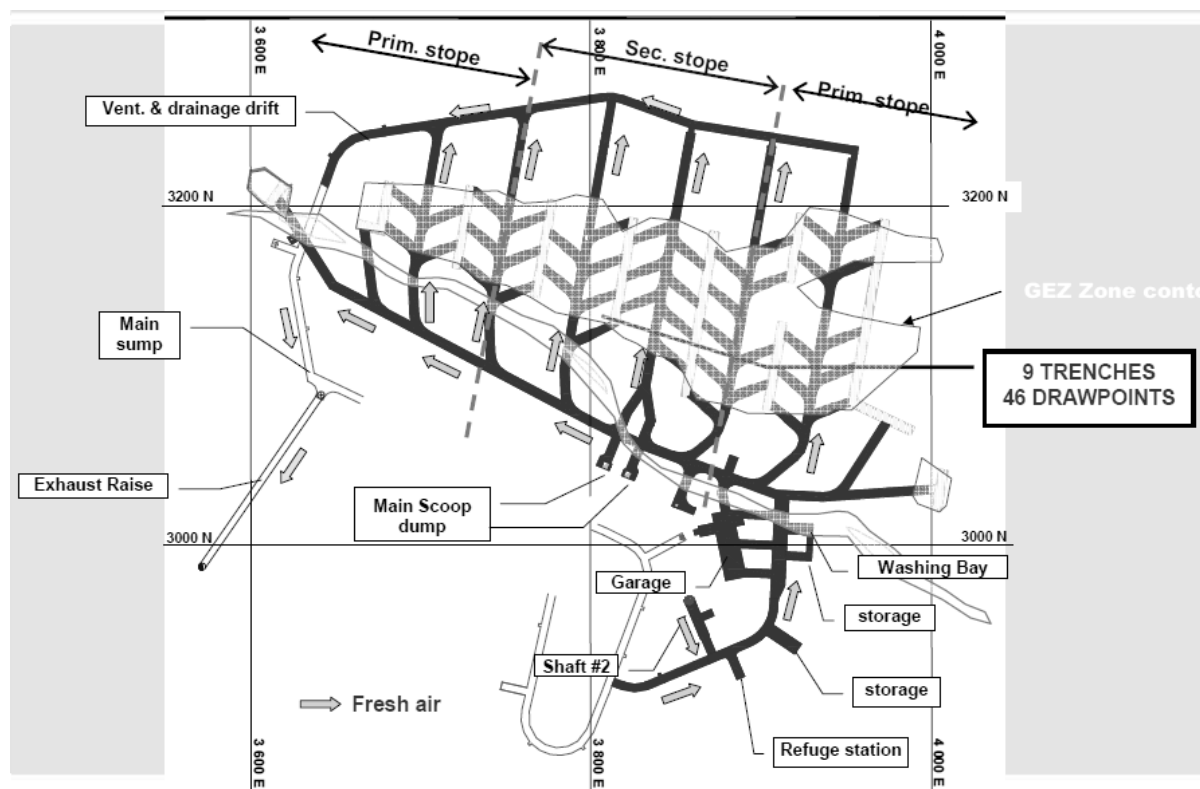


Figure 12 Haulage level layout



Figure 13 Caterpillar 2900G XTRA scoop trams used for mucking (Frenette, 2008)



Figure 14 Cubex 6200 HH Megamatic ITH drill (Frenette, 2008)

5 Comparison between reality and theory

The biggest question when the Goldex Mine was started was the stability of the stopes. Numerical modelling showed that when mining flat back, a 30–40% height to span arch ratio was to be expected as shown in Figure 15. As shown in Figure 16, the mining of the first trenches proved this estimate to be correct, ratio ranging from 30–42% with one exception when a dyke was present where the ratio was at 66%. Blasting of the full width stope showed a higher ratio of about 50% as shown in Figure 17. Interestingly, seismicity after blasting of the stopes showed instantaneous caving of the arch profile and stabilisation within a couple of hours. Once stability has been achieved, minimal sloughing or caving is encountered in the following weeks. A laser survey tool lowered in the stope through production holes is used periodically to monitor the shape of the back and walls along with ore levels. This tool proves very helpful in correlating seismic activity and the location of the arch.

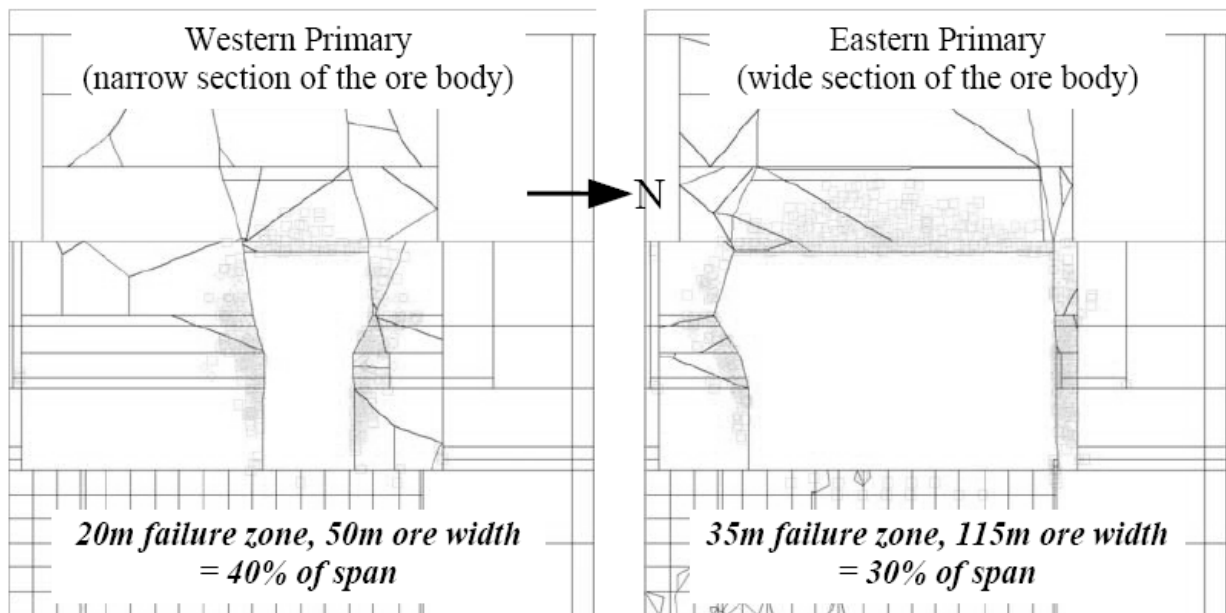


Figure 15 Expected arch effect from numerical modelling (Brummer et al., 2005)

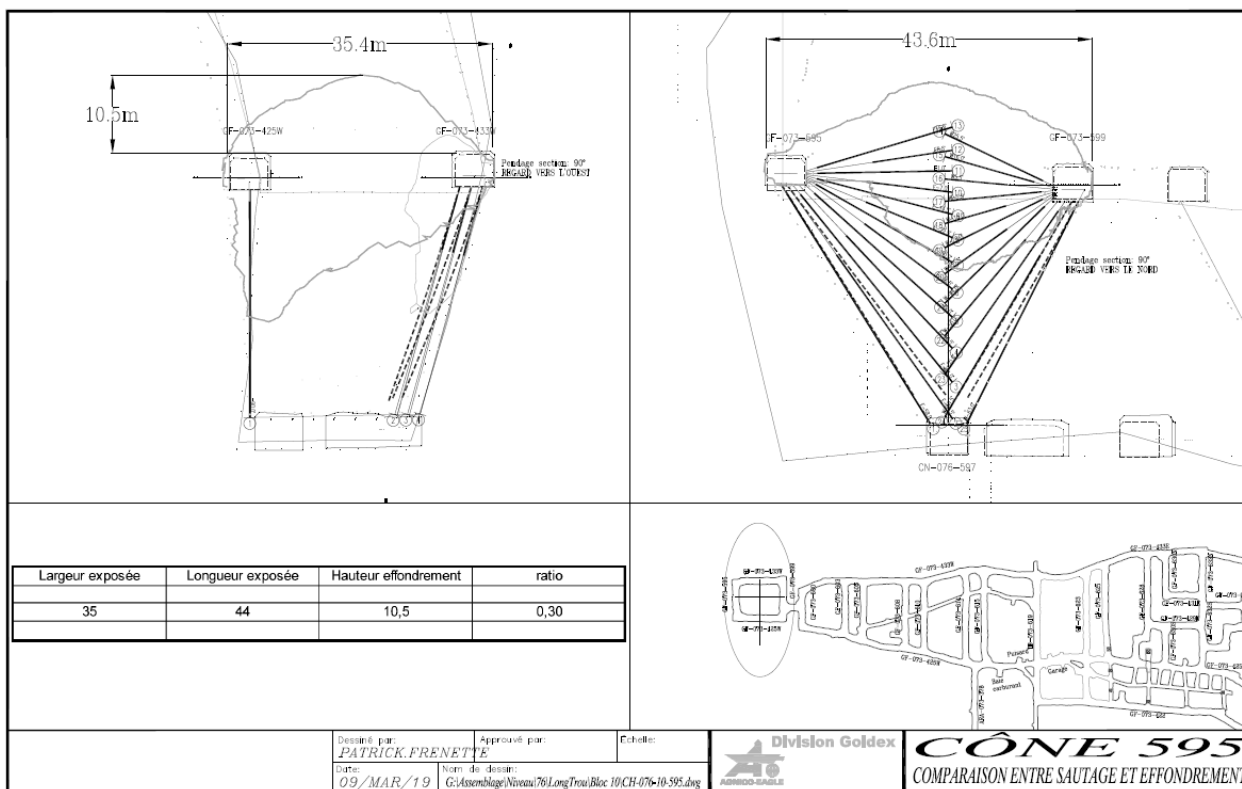


Figure 16 Evolution of the arch ratio in the opening of a trench (Frenette, 2009)

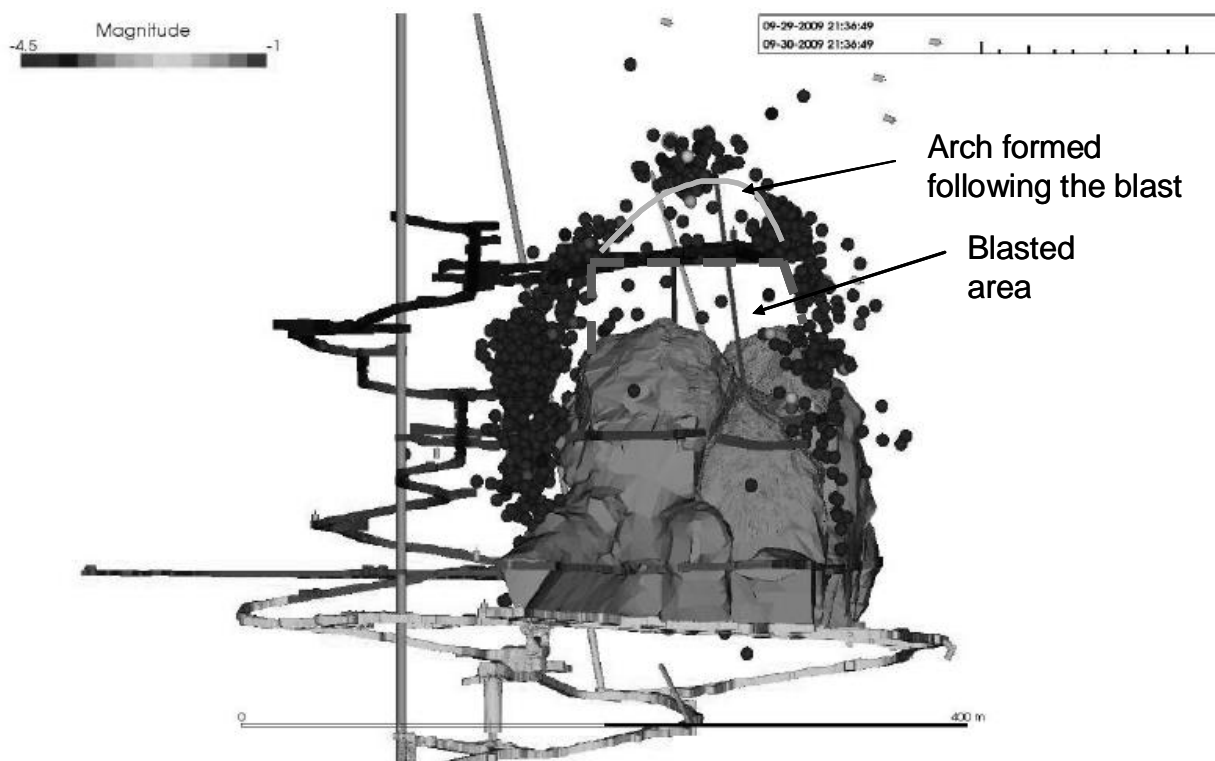


Figure 17 View looking west of seismicity around the Eastern Primary stope following a 1.5 Mt flat back blast showing the arch created above level 58. The arch ratio is about 50% (50 m high for a 105 m span)

All the information collected while mining the first two primary stopes will be used to reassess the shape and sequencing of the secondary stope. Once the orebody is completely blasted, stability still needs to be achieved to avoid a cave that could reach the surface. Remediation plans options such as using tailings as backfill are being developed to enable a swift reaction if such was the case.

Fragmentation was an issue that affected productivity when the trenches were first opened with a flat back. Once enough pre-drilling was available to blast the arch above the trench at the same time as the trench itself, less problem were noticed. The startup of the mine was very fast, achieving commercial production one month after starting blasting the trenches and it took four months to reach the designed output of 7,000 tpd. For 2009, the average output has been around 7,500 tpd with peaks over 11,000 tpd, already 500 tpd more than the designed productivity. Plans to increase the average daily output to over 8,300 tpd are being worked on.

6 Conclusion

The successful startup of the Goldex Mine is the result of perseverance and creative thinking. A long time was needed before a feasible and cost effective method could be put into production. The Goldex success also shows that alternatives to block or sublevel caving can be economical and safe now that the technology exists.

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