Underground Stope Drill and Blast Designs Optimisation Program

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ABSTRACT

The easiest way to mine valuable orebodies is through the use of explosives and well-planned blasting layouts that produce fragmentation profiles designed specifically for conveyance equipment in underground mines. The design of blasting patterns is specific to not only the explosives being used but also the rock or ore type being fragmented. Many blasting designs are based solely on the experience of those developed for similar orebodies in terms of properties and orientation or individuals that have some blasting experience.

The primary technical objective of this paper is to outline a unique methodology for determining underground blasting methodologies (developing specific blasting parameters) commensurate with explosive energies and rock/ore equations of state for a narrow-vein gold property located in Northern Quebec and for underground mines in general.

The procedures developed are to define the desired fragmentation specification so that a specific thermodynamic break is generated, taking into account powder factor, energy factor, tonnages, explosive energies and distribution along with blasthole diameter and orebody orientation for a specific set of explosive and dynamic rock/ore properties. Determining dynamic modulus values for the ore using seismic sensors to measure P- and S-wave velocities, the aforementioned parameters will be utilised to set the design constraints to maximise recovery and minimise overbreak and dilution.

New underground blasting software (AEGIS) will use thermodynamic break in conjunction with defined fragmentation profiles to create an array of blasting parameters used to design and plan ring layouts. This software will also allow mining planners, prior to drilling and loading a production blast, to appraise future intended blasting patterns to minimise the potential of diluting ore with host rock and the likelihood of damaging mine support structures.

INTRODUCTION

A summary of the main observations and measurements that were made over the last several months are presented. These efforts will help to define the value of the blast optimisation program when using such refined analysis techniques in an underground environment.

BACKGROUND

The easiest way to mine orebodies is through the use of explosives and well-planned blasting layouts that produce fragmentation adequate for further material handling processes. The design of blasting patterns is specific to not only the explosives being used but also the rock or ore type being fragmented. Many blasting designs are based solely on the experience of those developed for similar orebodies in terms of properties, geometry/orientation or individuals that have some blasting experience.

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The procedures developed are to define the desired fragmentation specification so that a specific thermodynamic break is generated, taking into account powder factor, energy factor, tonnages, explosive energies and distribution along with blasthole diameter and orebody orientation for a specific set of explosive and rock/ore properties. Determining dynamic modulus values for the ore using seismic sensors to measure P- and S-wave velocities, the aforementioned parameters will be utilised to set the design constraints to maximise recovery and minimise overbreak and dilution.

New underground blasting software (AEGIS) will use thermodynamic break in conjunction with defined fragmentation predictions to create an array of blasting parameters used to design and plan ring layouts. This software will also allow mine planners, prior to drilling and loading a production blast, to appraise future intended blasting patterns to minimise the potential of diluting ore with host rock and the likelihood of damaging mine support structures.

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This design development process was proposed to the customer in order to follow-up on the discussions that took place while presenting the observations on some of their first large stope blasts. The advantages associated with vibration amplitude measurements while using different electronic timing sequencing during the subsequent trials were basically the main elements that led to the idea of conducting a few other trials, this time in a much more complex stope configuration.

The mine operational group must protect its mine pillars and, by doing so, control its percentage of overbreak/dilution. In addition, shortly before the following test began, the mine experienced an important amount of damage on a level where ejection of broken rocks was surprisingly observed after the firing of a final stope blast. At that time, the mine engineering department started to focus more on blast sequencing and vibration monitoring and ways to reduce it. It realised that when dealing with non-regular orebody shapes and variable stope geometry, it is even more important to allow sufficient time for the broken muck to be displaced.

The table was set to determine how to optimise the loading configuration using the software capabilities in order to achieve an excellent fragmentation, while wall stability and recovery remained as planned. All this would only provide an even more efficient and productive mucking process.

Figure 1 shows the type of double-dip drilling that was required in some non-regular profiles of the stope 0440-05050-357 near its geological limits.

Refer to Figure 2 for the Aegis Designer software representation of SUBTEK bulk emulsion charge in 100 mm blastholes in a narrow-vein stope.

FIG 1 – Double-dip fan drilling of 100 mm blastholes in stope 0440-05050-357.

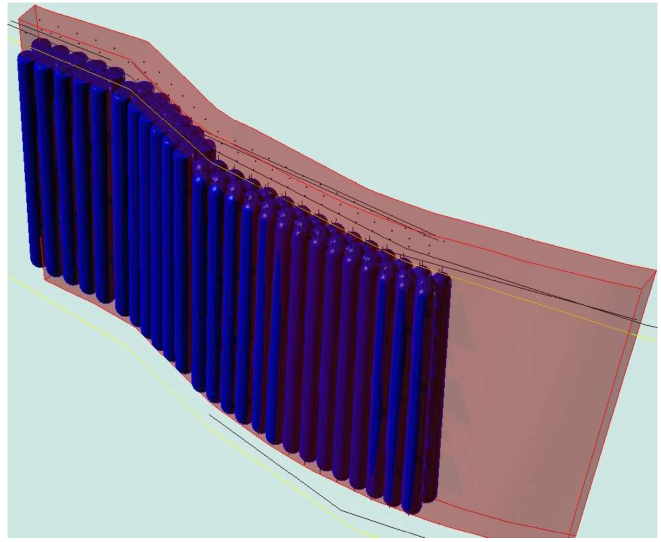

FIG 2 – Aegis Designer software representation of bulk emulsion breaks in holes of 100 mm.

General description

Goldcorp's Eleonore gold mine exploitation started in 2014. The first production stopes were planned for April 2014. Daily tonnage of 3000–4200 t is planned until 2017, when the mine should reach its planned peak production of approximately 7000 t/d in the first half of 2018.

Preliminary evaluations were done on stopes of larger dimensions. This report was one of the first initiatives taken on stopes that do not exceed 3 m in width. As the blasthole dip was almost vertical, it facilitated the drilling process, which was simpler to set-up and resulted in good drilling accuracy. Verification measurements were conducted in that regard as brand new equipment had just been brought in that was the same for the upcoming operators. For the portion located in proximity to the slot raise (0.76 m reamed diameter hole), the dip was towards the east, between 80° and 90°. Once past 'Section 5', the dip was west, sensibly of the same degree.

The need to use final wall control techniques will be defined once the results from a few other stopes become available. Initially, the positioning of the production holes along the hanging wall is planned to be at 0.6 m from the contact to waste rock.

Geological characteristics and dominant structures

The laboratory testing report provided by the mining company offers a description of the geological characteristics of the rock structures in place. It became the foundation of the designs included in this study.

A revision of the supplied documents was conducted to offer some baseline drill and blast loading scenarios, which were used for comparison. The ultimate objective was to produce an excellent ore fragmentation while protecting the final wall structures from high vibration amplitudes in order to control dilution (Table 1).

The geomechanical parameters used in the break calculations for both the ore and waste are summarised in Tables 2 and 3.

Explosives design break calculation

Over the years, information was collected on hundreds of commercial explosive products that are available to the mining industry and a large database was built. It contains most of the energy values required to conduct thermodynamic evaluation

TABLE 1
Rock properties.

Type of rock	$R_{compressive}$ (MPa)	$R_{tensile}$ (MPa)	Young's modulus (GPa)	Poisson ratio	Density (mt/m^3)
Mineralised wacke (ore)	122.0	17.0a	42.63	0.16	2.74
Wacke (waste)	162.0	15.0a	39.05	0.14	2.75

Note: these values $R_{tensile}$ were used in the estimation of the design break of each explosives types.

TABLE 2
Ore rock parameters.

General rock classification – 1		**General structure and *in situ* geology – 3**	
Rock name	Ore Eleonore	Join sets	Three joint sets: 50
Type	Metamorphic (ID: 215)	Geology	1000
Location	NW Quebec	Q (10–100)	80
Source	Goldcorp Eleonore Mine	Joint roughness coefficient (10–100)	80
Global properties – 2		Rock mass rating (10–100)	Low: 60; high: 80
Young's modulus (GPa)	42.63	**Local properties – 4**	
Poisson's ratio	0.16	Static compressive strength (MPa)	121.97
Rock density (kg/m^3)	2739.63	Dynamic compressive strength (MPa)	152.46
Fracture index (%)	25.3	Static tensile strength (MPa)	6.16
P-wave velocity (m/s)	4098.70	Dynamic tensile strength (MPa)	7.7
S-wave velocity (m/s)	2607.63	*In situ* compressive rating (MPa)	121.97
Shear modulus (GPa)	18.38	*In situ* tensile rating (MPa)	6.16
Bulk modulus (GPa)	20.89		
Crack velocity (m/s)	1206.07		

TABLE 3
Waste rock parameters.

General rock classification – 1		**General structure and *in situ* geology – 3**	
Rock name	Wacke W Eleonore	Join sets	Three joint sets: 50
Type	Metamorphic (ID: 214)	Geology	1000
Location	NW Quebec Eleonore Mine	Q (10–100)	80
Source	Goldcorp Eleonore Mine	Joint roughness coefficient (10–100)	80
Global properties – 2		Rock mass rating (10–100)	Low: 60; high: 80
Young's modulus (GPa)	39.05	**Local properties – 4**	
Poisson's ratio	0.14	Static compressive strength (MPa)	161.96
Rock density (kg/m^3)	2749.71	Dynamic compressive strength (MPa)	202.45
Fracture index (%)	25.078	Static tensile strength (MPa)	9.04
P-wave velocity (m/s)	3883.96	Dynamic tensile strength (MPa)	11.30
S-wave velocity (m/s)	2512.91	*In situ* compressive rating (MPa)	161.96
Shear modulus (GPa)	17.12	*In situ* tensile rating (MPa)	9.04
Bulk modulus (GPa)	18.07		
Crack velocity (m/s)	1321.88		

of the associated amount of energy available for rock breakage. The 'break calculation routine' allows the break radius of any explosive charge to be determined in any rock type that has been entered in the rock properties database.

Table 4 provides a summary of the break calculation results and applicable drilling patterns for the three explosives products that had been considered during the stope exploitation. Different percentages of break overlaps between the charges are listed, allowing the mine engineers to select the degree of intensity of cracks required to produce the desired fragmentation.

Table 5 represents a typical results window obtained from the break calculator once all the necessary information about the explosive and specific rock type has been entered. The minimum and maximum breaks are displayed over a spectrum of ±30 per cent of the nominal *in situ* tensile and compressive strengths rating. Borehole pressure calculations

TABLE 4
Gold Corp Eleonore mine – test bloc 1, break radius simulations per charge types.

Blastholes of 0.1 m over 30 m	Blastholes of 0.09 m over 30 m		Design break envelop diameter used
1. SUBTEK charge 0.1 m (fully coupled)	**1. SUBTEK charge 0.09 m (fully coupled)**		
0% overlap:	0% overlap:		
Rmin: 3.13 m	Rmin: 2.87		
Rmax: 6.76 m	Rmax: 7.46 m		
Pattern: 4.38 × 4.99	Pattern: 3.98 × 4.68 m		
25% overlap:	15% overlap:		
Rmin: 2.52	Rmin: 2.52		
Rmax: 5.98 m	Rmax: 6.34 m		
Pattern: 2.96 × 3.37 m	Pattern: 3.19 × 3.74 m		
35% overlap:	25% overlap:		
Rmin: 2.31	Rmin: 2.31		
Rmax: 3.96 m	Rmax: 5.59 m		
Pattern: 2.46 × 2.81 m	Pattern: 2.7 × 3.17 m		
45% overlap:	35% overlap:	45% overlap:	42% overlap:
Rmin: 2.13 m	Rmin: 2.13	Rmin: 1.96	Rmin: 1.99
Rmax: 4.38 m	Rmax: 4.85 m	Rmax: 4.1 m	Rmax: 4.25 m
Pattern: 2.02 × 2.36 m (closest to mine pattern of 1.95 × 2.29 m)	Pattern: 2.25 × 2.64 m	Pattern: 1.85 × 2.17 m	Pattern: 1.95 × 2.29 m
2. E-113 0.075 m (air dcpld)	**2. E-113 0.065 m (air dcpld)**		
0% overlap:	0% overlap:		
Rmin: 2.37	Rmin: 2.09		
Rmax: 5.18 m	Rmax: 4.48 m		
Pattern: 3.47 × 4.07 m	Pattern: 3.17 × 3.72 m		
15% overlap:	15% overlap:		
Rmin: 2.09	Rmin: 1.85		
Rmax: 4.40 m	Rmax: 3.80 m		
Pattern: 2.78 × 2.81 m	Pattern: 2.55 × 2.99 m		
25% overlap:	25% overlap:		
Rmin: 1.92	Rmin: 1.7 m		
Rmax: 3.88 m	Rmax: 3.38 m		
Pattern: 2.36 × 2.77 m	Pattern: 2.17 × 2.55 m		
35% overlap:	35% overlap:		
Rmin: 1.77	Rmin: 1.57		
Rmax: 3.36 m	Rmax: 2.93 m		
Pattern: 1.99 × 2.33 (closest to the mine pattern)	Pattern: 1.83 × 2.15 m		
45% overlap:	45% overlap:	30% overlap:	
Rmin: 1.65 m	Rmin: 1.46 m	Rmin: 1.64	
Rmax: 2.84	Rmax: 2.48	Rmax: 3.16 m	
Pattern: 1.64 × 1.93	Pattern: 1.52 × 1.79 m	Pattern: 2.00 × 2.35 m	
3. DYNOSPLIT C 0.032 m in 0.1 m (air dcpld)	**3. DYNOSPLIT C 0.032 m in 0.090 m (air dcpld)**		
Rmin: 0.9 m	Rmin: 0.94 m		
4. DYNOSPLIT C 0.022 m in 0.1 m (air dcpld)	**4. DYNOSPLIT C 0.022 m in 0.1 m (air dcpld)**		
Rmin: 0.61m	Rmin: 0.70 m		

are indicated for each scenario as are the recommended pattern burden and spacing and the associated amount of tons broken with the associated powder factor.

Loading options per blasts and rings

Multiple loading scenarios were developed and were displayed and compared in terms of explosives charge

TABLE 5

Subtek bulk emulsion 100 mm break design.

Standard break parameters					
Dynamic values sensitivity on break	-30%	-15%	Mean	15%	30%
In situ tensile rating (MPa)	4.31	5.23	6.16	7.08	8.01
In situ compressive rating (MPa)	85.37	103.67	121.97	140.26	158.56
Borehole pressure (MPa)	5036.26	5196.28	5361.38	5531.72	5707.48
Minimum break (m)	2.33	2.21	2.12	2.05	2.00
Maximum break (m)	5.08	4.68	4.38	4.15	3.97
	BKO	45.000	**SBR**	1.140	
J: pattern spacing (m)	2.45	2.36	2.30	2.24	2.20
K: pattern burden (m)	2.15	2.07	2.02	1.97	1.93
Tonnes broken (metric tonne)	213.9	198.91	187.86	179.32	172.53
Powder factor A for a full explosives column (kg/t metric)	1.13	1.21	1.28	1.35	1.40

distribution, associated break, percentage of break overlap selected by the mine engineer and dilution estimation based on ore contours.

In a first step, the simulations were conducted with a single explosive type: the bulk emulsion in 90 mm and 100 mm holes. After that, packaged products were introduced and compared along a few diameter combinations.

Optimal loading scenarios were then presented to the mine at the start of the production stope blasting process.

Regular scenarios (Table 6)

Drilling diameter of 100 mm:

- Scenario 1 – bulk emulsion in all holes
- Scenario 2 – 75 mm chubs of E-113 in all holes.

Drilling diameter of 90 mm:

- Scenario 3 – bulk emulsion in all holes
- Scenario 4 – 65 mm chubs of E-113 in all holes.

Optimised scenarios (Table 7)

- Scenario 5 – bulk emulsion centre holes (2.13 m), E-113 HW and FW holes (75 mm – 1.65 m)
- Scenario 6 – bulk emulsion centre holes (2.13 m), E-113 HW (65 mm – 1.46 m) and E-113 FW holes (75 mm – 1.65 m)
- Scenario 7 – bulk emulsion centre holes (2.13 m), Dynosplit HW (32 mm – 0.9 m) and E-113 FW holes (75 mm – 1.65 m).

Optimal scenario 5 (holes of 100 mm)

Refer to Table 8 and Figures 3a and 3b.

Optimal scenario 7 (holes of 100 mm)

Refer to Table 9 and Figure 3c.

Vibration monitoring program (Stope 440-05050-354)

The vibration monitoring program implemented during 2014 is now starting to provide key information. As most of the mine's large mass blasts are being recorded, lots of knowledge is gained on the damping of the vibration signals that are produced by typical charge weights. Attenuation curves are being compiled and will be useful in future modelling.

The monitoring results obtained in the stope 440-05050-354 in December 2014 marked a crucial step forward in the comprehension of the impact of blast sequencing and its related vibration generation, especially in complex stope geometry. Such combinations of different drilling panel orientations produced a dual axis stope geometry. The delay sequencing of the multiple rings was done using a timing model developed in-house, which helped to avoid redundancy and ending up having multiple charges firing on the same delay, although they were located in opposite directions.

TABLE 6

Regular scenarios.

ID	Ore volume (ft^3)	Host rock volume (ft^3)	Ore tonnage (t)	Host rock tonnage (t)
Default scenario	0	0	0	0
Blast 1 – Subtek – 2.13 m (100 mm)	136 283.00	166 426.00	10 222.83	16 561.99
Blast 2 – E113 – 1.65 m (100 mm)	134 201.00	113 507.00	10 066.66	11 295.72
Blast 3 – Subtek – 1.98 m (90 mm)	135 498.00	150 557.00	10 163.95	14 982.78
Blast 4 – E113 – 1.46 m (90 mm)	133 135.00	93 110.00	9986.69	9265.90

TABLE 7

Optimised scenarios.

ID	Ore volume (ft^3)	Ore tonnage (t)	Ore volume (m^3)	Ore tonnage (t)	Host rock volume (ft^3)	Host rock tonnage (t)	Host rock volume (m^3)	Host rock tonnage (t)
Original	0	0	0	0	0	0	0	0
Scenario 5 – Subtek centre holes (100 mm – 2.13 m), E-113 HW and FW holes (75 mm – 1.65 m)	138 757.00	10 408.41	3929.16	9442.36	123 319.00	12 272.17	3492.01	11 133.14
Scenario 6 – Subtek centre holes (100 mm – 2.13 m), E-113 HW (65 mm 1.46 m) and E-113 FW holes (75 mm – 1.65 m)	139 740.00	10 482.15	3957.00	9509.26	123 540.00	12 294.16	3498.26	11 153.09
Scenario 7 – Subtek centre holes (100 mm – 2.13 m), Dynosplit HW (32 mm 0.91 m) and E-113 FW holes (75 mm – 1.65 m)	131 563.00	9868.78	3725.45	8952.81	82 065.00	8166.75	2323.82	7408.76

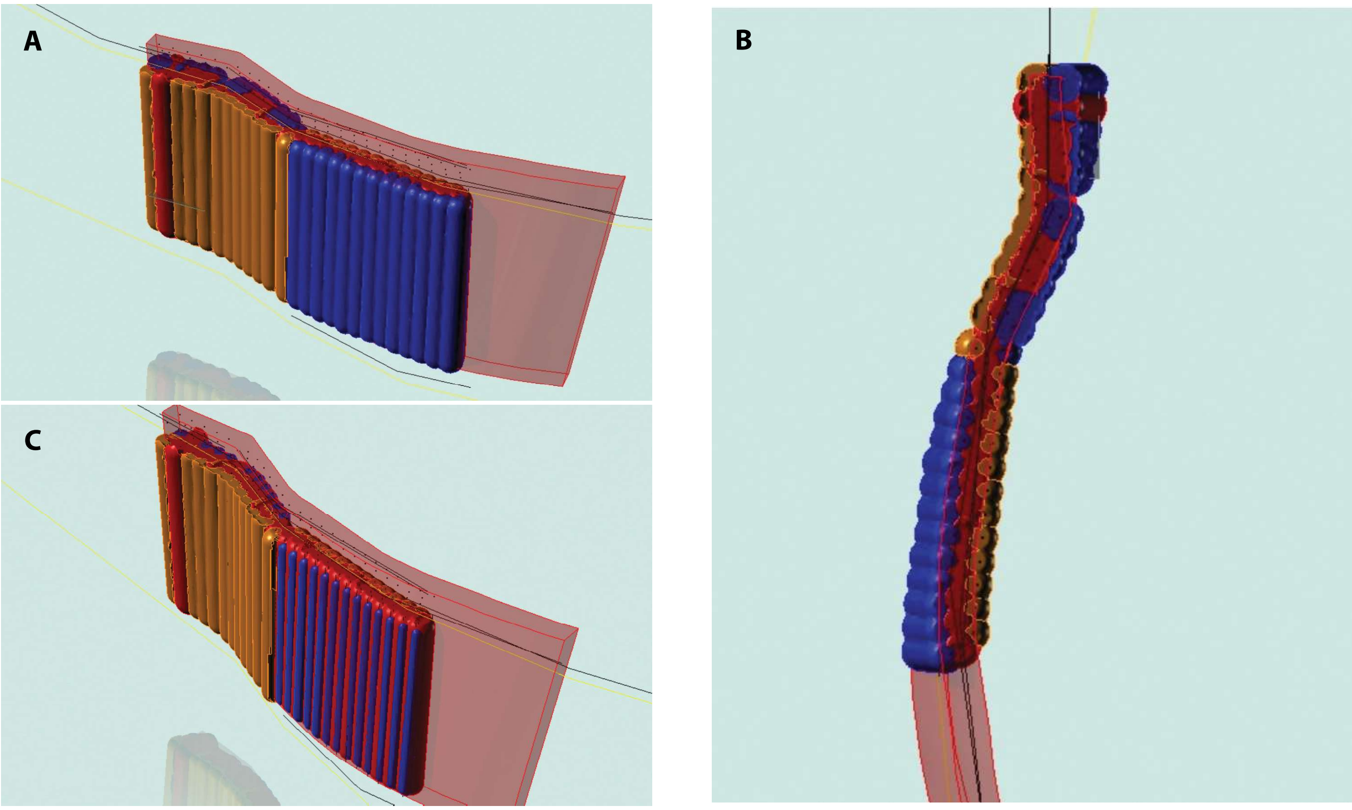

FIG 3 – Optimal scenarios. (A) North-east view; (B) north view; (C) north-east view.

TABLE 8
Optimial scenario 5.

Subtek in centre	E-113 (75 mm) on FW, E-113 (75 mm) on HW

TABLE 9
Optimal scenario 7.

Subtek au centre	E-113 (75 mm) au FW, Dynosplit C 32 mm

It also had to be determined what would be the necessary delays between the multiple longhole rings to maintain the proper blast movement dynamic, considering the amount of void available, some ~30 per cent of the blast volume, while making sure charge confinement does not become detrimental to the generation of good rock fragmentation.

Figure 4 represents the longitudinal view of the stope.

The planned blast sequencing of the final blast number 3 is presented in Figure 5.

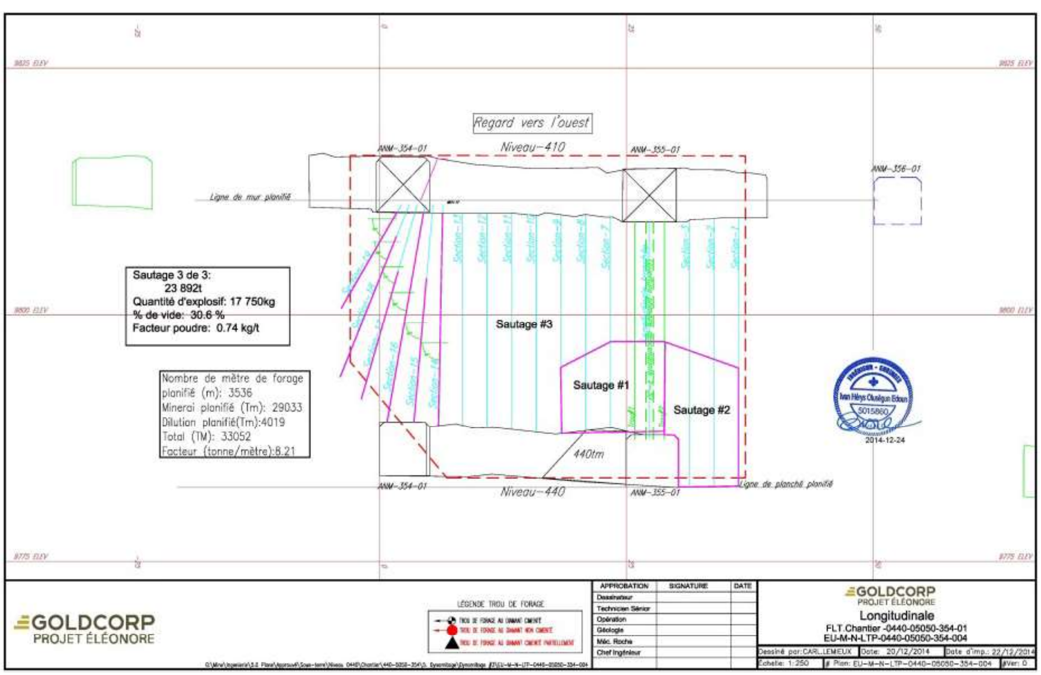

FIG 4 – Longitudinal view of the stope.

The stope configuration included two perpendicular portions, with the trench included in the slot being E–W and the bulk portion of the tonnage along the N–S axis. The initial phase was to open the E–W trench (rings 20 to 27) surrounding the slot raise containing a 0.76 m reamed hole. Once this volume was fired, two groups of sections were attacked, rings 7 to 19 were mainly shooting from south to north and vice versa for sections 1 to 3 on the north tail end. This change of flow of muck movement is the main cause of the very high vibration amplitude that was produced by most of the holes of ring number 7. Near the end of the blast, ring number 14 also showed high vibration peaks, especially from hole number 10, which was closest to the hanging wall, along the high-frequency geophone 'HF-02' (Figures 6a and 6b).

Figure 7 illustrates the vector sum analysis that was conducted on final blast number 3. The impact on the vibration amplitudes of the change of direction in the muck throw movement can be clearly seen. Many holes of ring number 7 generated high amplitudes, and this kept increasing until ring number 14, after which a certain relief was felt.

These interpretations led us to realise that the muck displacement velocity of 15–25 m/s we had been assuming was not always achieved, especially when such change in muck flow direction was taking place. An additional, much more important delay needs to be programmed between the initial trench blasting phase and the rest of the mass ring blasting.

Equivalent break and stress contour determination

In order to accurately determine the correlation between vibration amplitude and stress level encountered in proximity to the stope, it was preceded with a set of two high-frequency geophones collected on the same recording instrument (eight-channel Instantel Minimate Plus unit). The geophones were positioned on a straight axis so that the P-wave velocity could be calculated based on the difference in wave arrival time (2.9 ms) to both geophones and the known variation in true distance of the two geophones from the single charge (18.6 m).

Figure 8 shows the underground set-up used to collect these readings, while Figures 9 and 10 show the trace recorded and a zoomed view on longitudinal channels 2 and 6.

After determining the exact coordinates and distances separating the geophones, P-wave velocity was determined to be 6.414 m/s. This *in situ* value measured using a dynamic methodology is significantly higher than the one we had in our database (4.069 m/s) for the initial design break simulation phase.

Applying the following equation allowed us to transfer peak particle velocity (PPV) into stress level:

$$PPV = T_{dyn}/density \times P_v$$

From readings obtained at the Éléonore mine, it became evident that the recorded peak vector sum of 1096 mm/s exceeded the limit that the rock mass could tolerate, which explained some of the damage observed in the cross-cuts a few days after the blast took place.

Working with the same equation, it can be determined that a maximum level of amplitude should not be exceeded in future blasts so as to avoid additional damage:

- rock density: 2.74 kg/m^3
- P-wave: 6.414 m/s
- dynamic tensile resistance: 7.70 MPa (1117 psi) (refer to rock properties sheet number 215)
- PPV: 428.2 mm/s.

If the dynamic tensile strength goes up and down from -30 per cent to +30 per cent around the mean value, these become the maximum levels of amplitude not to be exceeded to avoid rock failure (see Table 10).

This is to show the range of values of stress for a corresponding range of values for PPV that will precipitate damage to rock/ore structures. Table 11 shows that the PPV measured corresponds to an extensive rock breakage level.

The break design calculation was used to create a view representing the break envelope obtained for sections 7 (in blue) and 14 (in brown) of blast number 3, which were the ones where the highest amplitude readings were recorded. Figures 11a and 11b are representations of the charge breaks in Aegis 3D.

The addition of the mineralised ore contour is visible in Figure 11b.

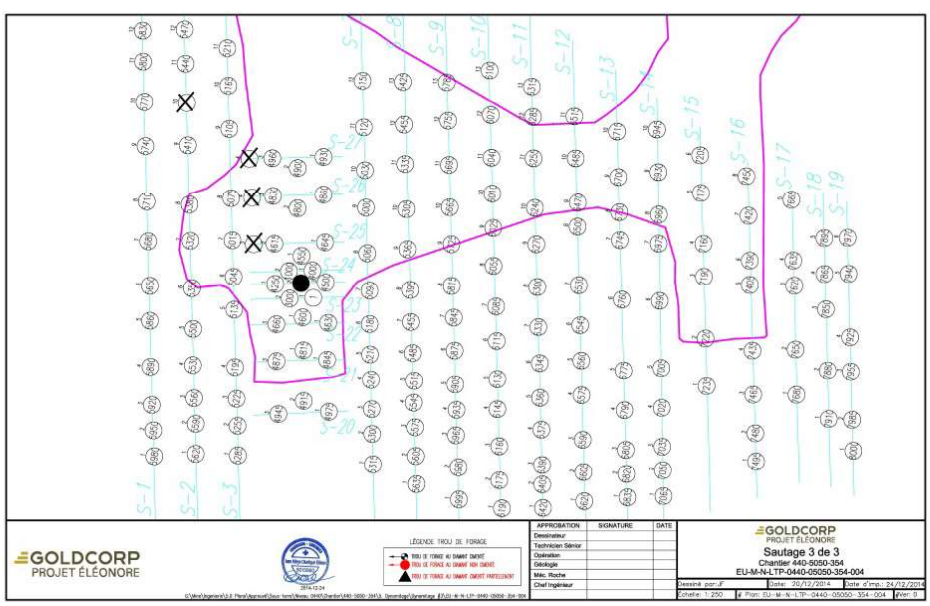

FIG 5 – Planned blast sequencing for final blast number 3.

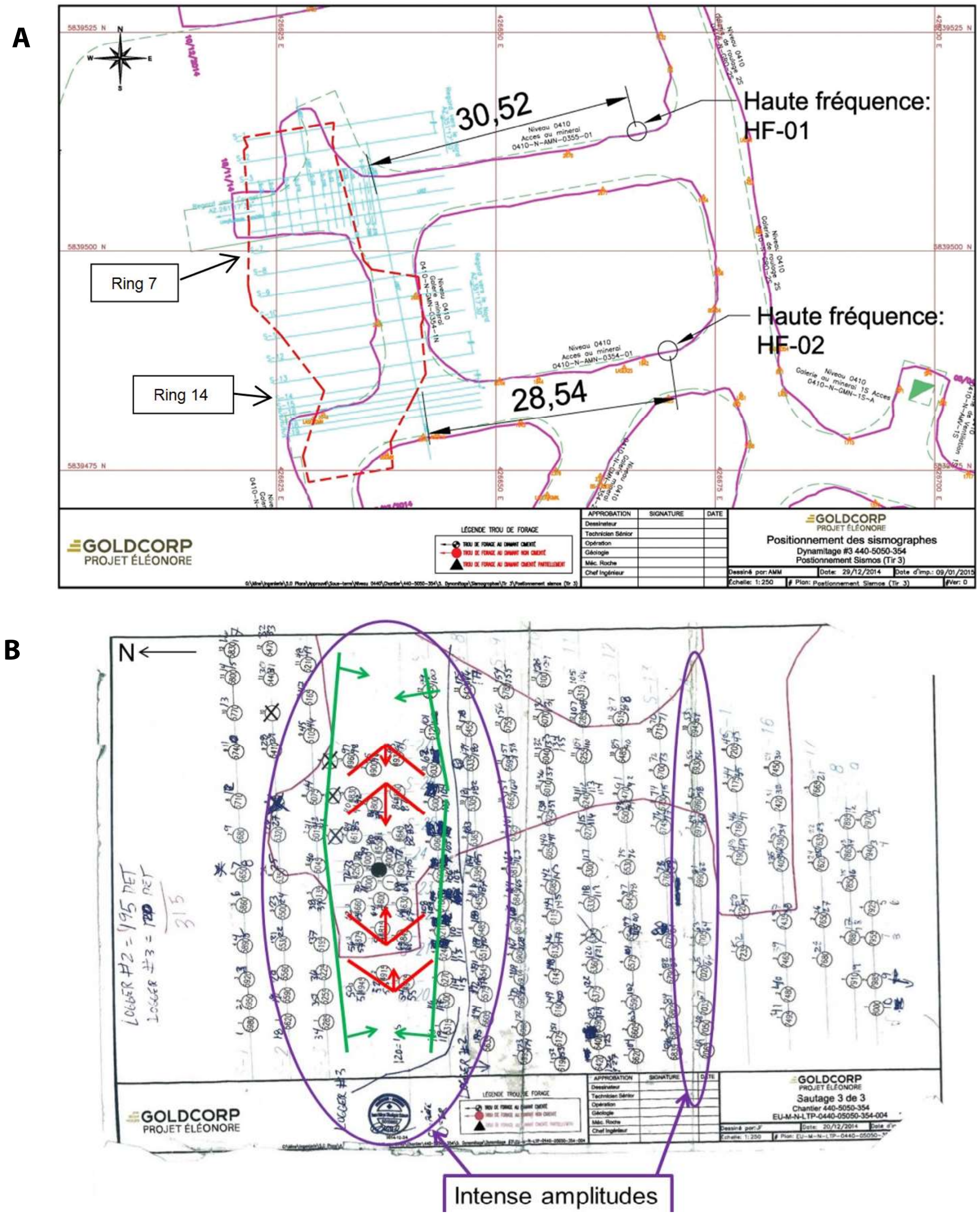

FIG 6 – (A) High frequency geophone locations; (B) vectors showing flow directions of broken muck.

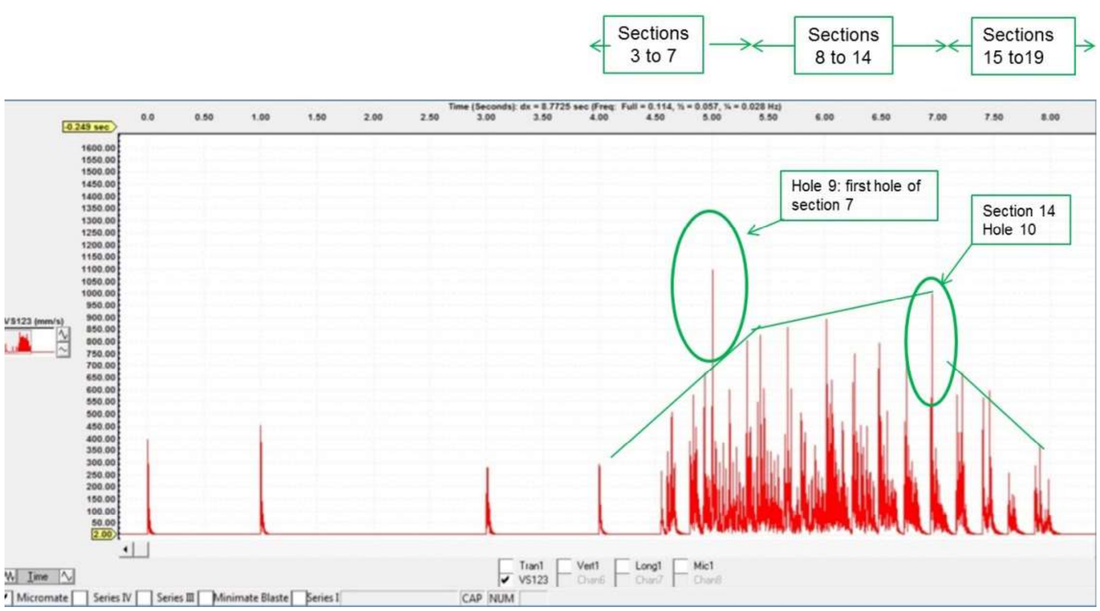

FIG 7 – Peak vector sum and highest peaks of amplitude.

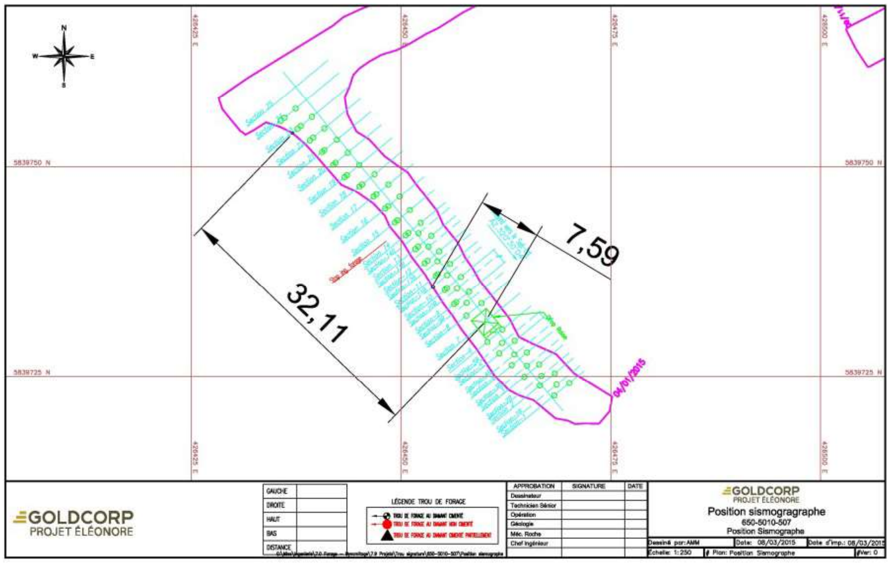

FIG 8 – Underground set-up.

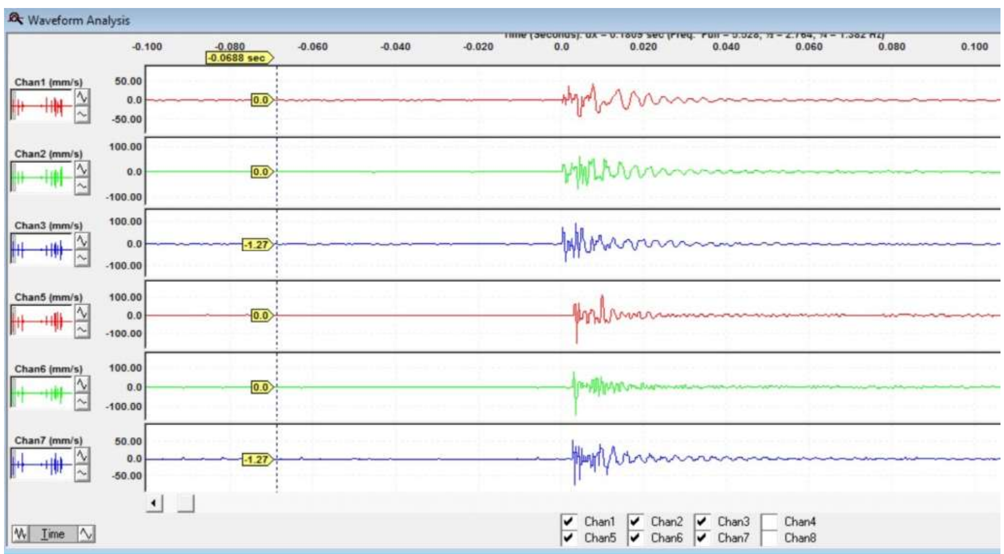

FIG 9 – Wave arrival time on all six channels.

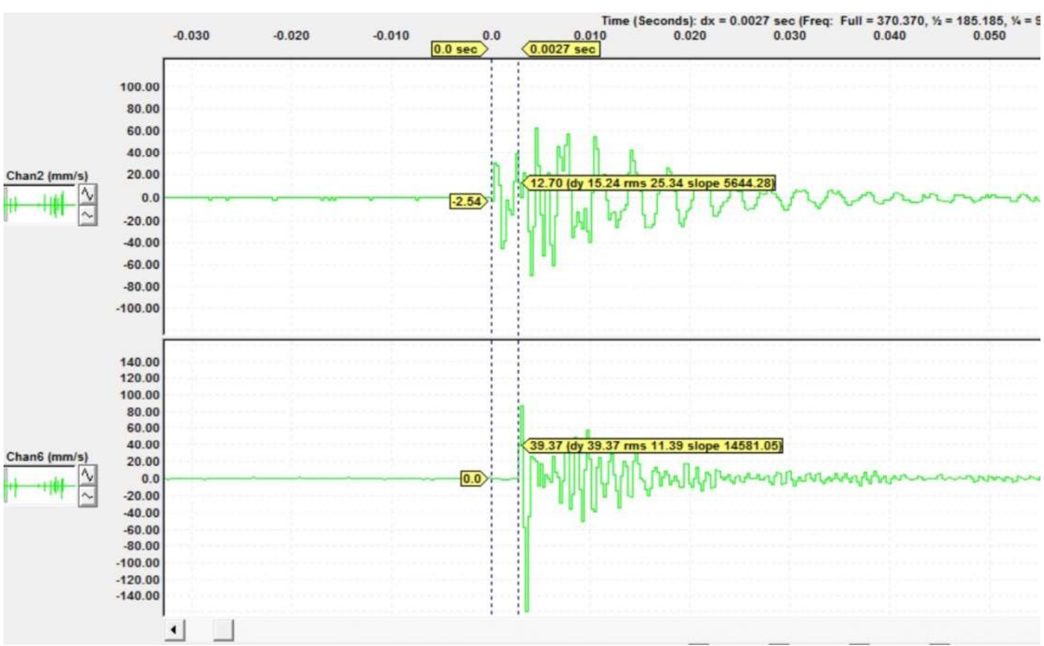

FIG 10 – Channels 2 and 6 arrival time difference of 2.9 ms.

TABLE 10
Maximum vibration amplitude limits.

Dyn T	-30%	-15%	Mean	+15%	+30%
(MPa)	5.39	6.55	7.70	8.86	10.01
Peak particle velocity (mm/s)	306.7	372.7	438.2	504.1	569.6

TABLE 11
Peak particle velocity measurements.

Parameter	Peak particle velocity (mm/s)
Rock breakage	2540
Onset of rock breakage	635
Rockfalls in unlined tunnel	305
Horizontal displacement in loose material	762
Weakening of bulkheads underground	457
Electric motor shaft misalignment	254
Cracked plaster	50.8

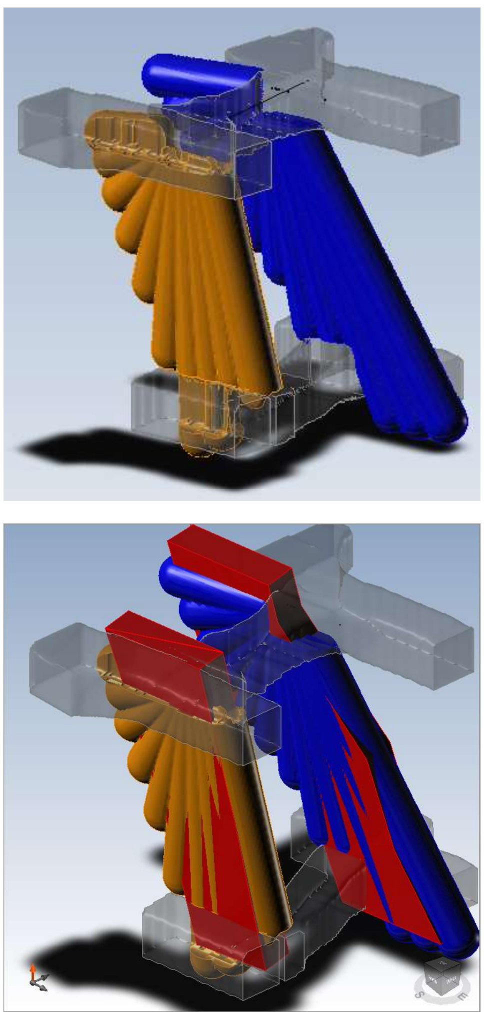

FIG 11 – Charge breaks in Aegis 3D.

SIGNATURE HOLE ANALYSIS

After a number of large and complex stope configurations have been excavated, the mining engineering department realised that it was essential to develop appropriate timing sequences as they would be mining narrower stopes varying from 3 to 7 m in width. In order to maintain good control on the wall damage and allow productive mucking activities, it became necessary to review the approaches by which blast sequencing was established.

It was decided to proceed with the monitoring of a few signature holes, with the objective of capturing wave attenuation signals in the surrounding rock mass and using them to customise the damping effect of geological structures. We installed multiple geophones around a single charge of 75 kg located close to the 0.76 m slot reamed hole and fired it. This exercise gave very clear seismic signals that allowed us to conduct multiple linear superposition simulations.

Figure 12 shows the underground charge and sensors set up for the recording of the signature hole.

An optimal delay of 15 ms per hole on a single ring (Figure 13) was selected, while an optimal delay of 127 ms was chosen between each of the rings.

This result cannot be applied without considering the percentage of void available and the amount of rings to be fired within a single mass blast (Figure 14).

The known coordinates and distances separating the two high-frequency geophones were used to determine the P-wave velocity as arrival time to both units could be accurately determined. This *in situ* value provided reliable data for our upcoming vibration control program.

CONCLUSION

For the last 12 months, the Éléonore mine engineering department and several external experts have been trying to define optimal blasting practices for the continuously changing mining conditions. The challenges remain to define a customised stope design that will deliver the mineral value expected and allow full recovery. Several blasting software tools were used to assist with the interpretation and visualisation of the different types of monitoring that were performed. Different tools/components, such as the 'break design calculator', 'Aegis/I-Ring 3D' for blast design, stress interpretation and dilution control, 'I-BLAST' for vibration and control and Signature Hole Analysis were combined to supply the mine engineering department with innovative blast design methodology and accurate analysis techniques that allowed them to achieve their production targets for tonnage and dilution ratios.

The challenge that the constantly changing precious metals market represents is obligating the blast designers to search for innovative approaches to implement optimal blasting applications. The introduction of electronic initiation systems has opened up a large spectrum of opportunities. The science, knowledge and high level of engineering required to determine what should be the optimal delay per hole and per ring in a given rock mass condition and blast configuration is being continually developed as new software tools and monitoring equipment become available and help to optimise accuracy.

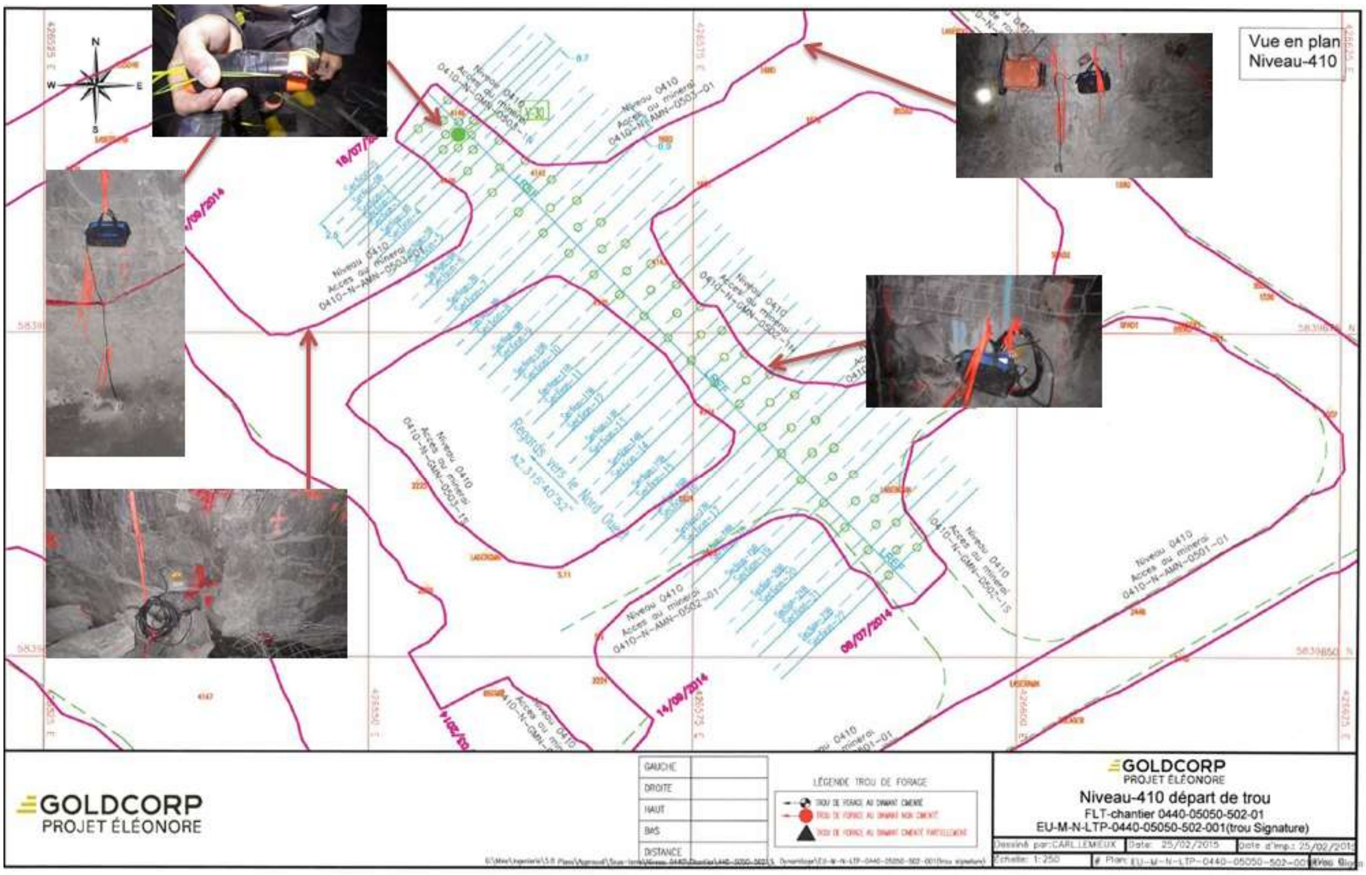

FIG 12 – Underground charge and sensors set up for the recording of the signature hole.

FIG 13 – Optimal delay per hole on a ring.

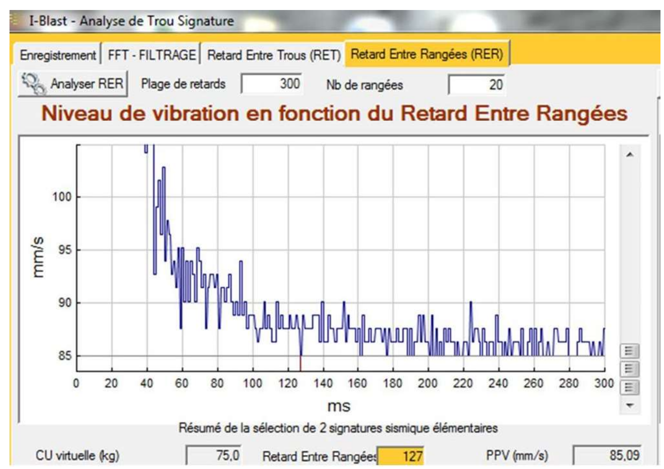

FIG 14 – Optimal delay per ring.