Waste Rock Dilution in Stoping of Steeply Dipping Narrow-Vein Deposits

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Abstract

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Abstract

On the basis of literature reviews, monitoring operational parameters and stope performance, the origins of and the main influences on unplanned waste rock dilution in narrow-vein stoping at the Jokisivu mine were allocated. Stope scanning with a cavity monitoring system (CMS) indicated that most of the unplanned dilution originates from the hangingwall side of the stopes, but also stope footwalls show levels of sloughage. Combining visual observations with scanned stope models, knowledge of past experiences and existing geological structures, revealed that there are two main factors governing the level of unplanned dilution at the Jokisivu operation. The first group of factors are the local geological setting, geological structures and rock-mechanical properties of the host rock. The second factor is the inaccuracy in marking drillhole collar locations. Large deviations in marking drillhole collar locations resulted in designed stope boundaries not being met or being exceeded. This in turn caused additional dilution and ore loss. After monitoring operational parameters and stope performance, several methods have been proposed to minimize unplanned waste rock dilution in future practices of the Jokisivu operation. Prominent measures being the implementation of an alternative method of marking drillhole collar location, reinforcing stope hangingwall support and the continuous monitoring of stopes via the use of a CMS.

Keywords Dilution, Narrow-Vein Stoping, Underground Mining, Electronic Detonators, Orogenic Gold Deposit, Cavity Monitoring System (CMS), Drillhole Deviation, Stope Support, Blasting Induced Vibrations.

Preface

I would like to thank my thesis supervisors Prof. Mikael Rinne and Tuomo Hänninen MSc. for their guidance and support throughout the duration of my thesis project.

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Otaniemi, 26.11.2018

Dennis Louwerse

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Symbols and Abbreviations

®	Registered Trademark Symbol
⌀	Diameter
θ	Drillhole Angle Deviation
3D	Three Dimensional
ANFO	Ammonium Nitrate – Fuel Oil
ASX	Australia Stock Exchange
AUD	Australian Dollar
CMS	Cavity Monitoring System
Cos	Cosine
d	Drillhole Collar Deviation
D&B	Drilling and Blasting
e	Drillhole Trajectory Eccentricity
e.g.	Exempli Gratia (For Example)
ELOS	Equivalent Linear Overbreak or Sloughage
Et al.	Et Alia (And Others)
FW	Footwall
g	Gram
Ga	Giga-Annum (Billion Years)
HKEx	Hong Kong Stock Exchange
HW	Hangingwall
Hz	Hertz
k	Kilo (Thousand)
m	Metre
m^2	Squared Metres
m^3	Cubic Metres
mm	Millimetres
ms	Milliseconds
NONEL	Non-Electric
Oz	Ounce
PPV	Peak Particle Velocity
Psi	Pound-Force per Square Inch
s	Second

s^2	Seconds Squared
t	Metric Tonne
USD	United States Dollar
y	Year

1. Introduction

1.1. Introduction

Narrow-vein mining methods are generally practiced in the extraction of thin tabular orebodies. Within narrow vein mining, waste rock dilution and ore loss are considered to be significant problems. Excessive waste rock dilution and ore loss are unwanted in a mining operation, due to their influences on especially the efficiency and profit of the operation. Within this thesis, dilution refers to unplanned dilution and can be defined as the additional non-ore material (below cutoff grade) which is derived from rock or backfill outside the stope boundaries (Scoble & Moss, 1994). Dilution directly leads to a considerable increase in production costs. Therefore, it is important to understand and control the factors that are influencing dilution.

1.2. Objective

Dilution in narrow-vein deposits can be allocated to numerous factors. Geological and rock-mechanical properties define the setting and condition of a deposit. Also, the design of a stope along with the amount of time the stope is open (exposure time) is important in controlling dilution. Likewise, stope support and drilling and blasting parameters have major influences on dilution. Furthermore, human error also plays its role in various ways when looking into the causes of dilution.

The focus of this thesis, will primarily be kept on some of the drilling and blasting parameters and the influence of rock support on unplanned dilution in steeply dipping narrow-vein deposits. The other aspects which are named above, will be discussed only briefly. The fundamental reasons for delimitating the focus of the thesis to these factors are restrictions of time and resources and the fact that taking too many variable parameters into account will produce results that are difficult to interpret. Hence, allocating results to a root cause is unreasonable and most likely incorrect when testing multiple parameters that are closely related.

This study aims to allocate the origin of and the main influences on unplanned waste rock dilution in the Jokisivu gold mine and to find out, after analysing, what can be done to minimize waste rock dilution in future practices during the operation.

This objective ought to be achieved by meeting the following goals:

1. Conducting a literature review on dilution and factors influencing dilution;
2. Gathering knowledge and experience on current practices of the operation;
3. Acquiring data by monitoring and measuring operational parameters and stope performance.

2. Dilution

At the Jokisivu mine, gold is being produced using sublevel longhole stoping. Open or sublevel longhole stoping has been generally accepted as one of the most efficient and steady mining methods for underground metalliferous mining (Austrade, 2013). Even though this method is highly efficient, many mines using this production method suffer from unplanned dilution and ore loss. Henning & Mitri (2007) proclaimed that roughly 40 % of all longhole stoping operations are dealing with 10 % to 20 % of waste rock dilution. This is also the case for the operation at Jokisivu, which is currently dealing with 15 % to 30 % of dilution, dependent on local stope conditions (ASX, 2018). Dilution is one of the most prevalent reasons for economical mine failure (Miller et al. 1992; Scoble & Moss, 1994) and therefore must be considered as a critical issue in longhole stoping operations.

Within this section, the concept and significance of managing unplanned dilution in underground stoping will be discussed. This will be done by means of a literature review of previous studies on dilution and factors which are proved to be of influence on the concept of dilution in underground longhole stoping. Increased consideration will be given to previous studies related to (steeply dipping) narrow-vein longhole stoping methods, as this is the current method of operation in the Jokisivu gold mine.

2.1. Defining Dilution

Dilution and recovery of a stope are generally considered as a measure of the quality of the stope design and the mining performance (Clark, 1998). A satisfying design is one that maximizes recovery and minimizes dilution. The ideal situation would be to have a recovery of 100% while having 0% dilution during the mining of a stope, however this is in practice essentially impossible due to geological uncertainty, operational limitations, rock mechanical properties and human error (Elbrond 1994).

Wright (1983) describes dilution as the contamination of ore by non-ore material during the mining process. Henning & Mitri (2008) portray the term dilution as any waste material within the mining block, including barren and subgrade rock and backfill and distinguish between ‘unplanned’ and ‘planned’ dilution. This definition comes from Scoble & Moss (1994), which is depicted in Figure 2.1. and defined as:

Planned Dilution is the non-ore material (below cutoff grade) that lies within the designed stope boundaries.

Unplanned Dilution is the additional non-ore material (below cutoff grade) which is derived from rock or backfill outside the stope boundaries.

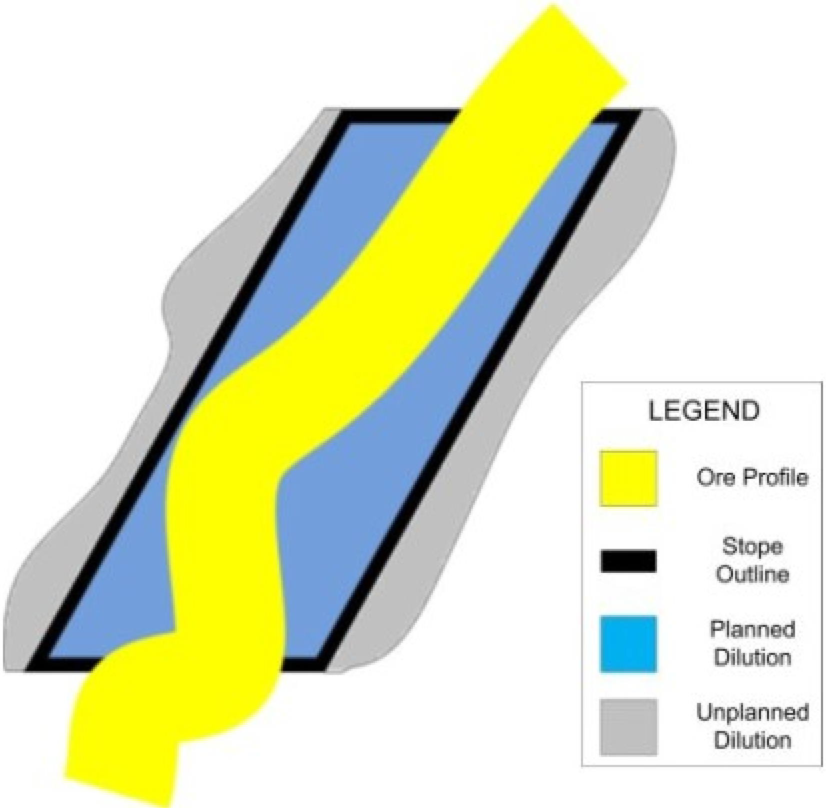

Figure 2.1: Stope cross-section showing planned and unplanned dilution (Mitri et al., 2010)

The origins of planned dilution can be assigned to the limitations of mining method, when mining irregular, narrow-vein deposits, or when unsuitably sized equipment has been selected for the operation (Trevor, 1991).

Unplanned dilution, which is the main subject of interest of this study, describes additional non-ore material which is derived from rock or backfill outside the stope boundaries due to blast induced overbreak, sloughage of unstable hangingwall and footwall rock, or sloughing of backfill (Scoble and Moss, 1994).

2.2. Quantifying Dilution

2.2.1. Percentage

There are multiple ways to compass dilution in a formula. Pakalnis (1986), distinguished ten different definitions of dilution during his survey of 22 mine operations. However, According to Scoble and Moss (1994), the two most common methods used for calculating dilution are based on tonnages and are as follows:

$$Dilution = \frac{Tonnes\ of\ Waste\ Mined}{Tonnes\ of\ Ore\ Mined} \tag{1}$$

$$Dilution = \frac{Tonnes\ of\ Waste\ Mined}{(Tonnes\ of\ Waste\ Mined + Tonnes\ of\ Ore\ Mined)} \tag{2}$$

Out of these two equations, Equation (1) was recommended as a standard measure of dilution by (Pakalnis et al., 1995b), as this measure is more sensitive to wall sloughage. Henning & Mitri (2008) elaborate on this with an illustration. For example, a 2:1 sloughage-to-ore ratio produces a 66 % dilution factor according to Equation (2), while Equation (1) produces a dilution factor of 200 %.

In the case of narrow-vein mining, where almost all unplanned dilution originates from the hangingwall and footwall and geometries are consistent along strike, Martin et al. (1997) suggests that unplanned dilution in narrow-vein stoping can best be determined through Equation (3).

$$Dilution = \frac{Meters\ of\ Footwall\ Waste + Meters\ of\ Hangingwall\ Waste}{Planned\ Mining\ Width} \tag{3}$$

2.2.2. Equivalent Linear Overbreak (ELOS)

One can argue that calculating dilution as a percentage of planned tonnages or planned mining width is appropriate for economic analysis and evaluation. However, it is not satisfactory for objective empirical analysis of all different parameters affecting dilution (Stewart, 2005), which in the end, is the goal of this thesis.

Clark and Pakalnis (1997) came up with the idea to calculate dilution in terms of average metres unplanned material that had sloughed off the stope walls per square metre of wall (m^3/m^2) rather than a percentage. This method is called the equivalent linear overbreak or slough (ELOS) and is depicted by Figure 2.2. and defined in Equation (4).

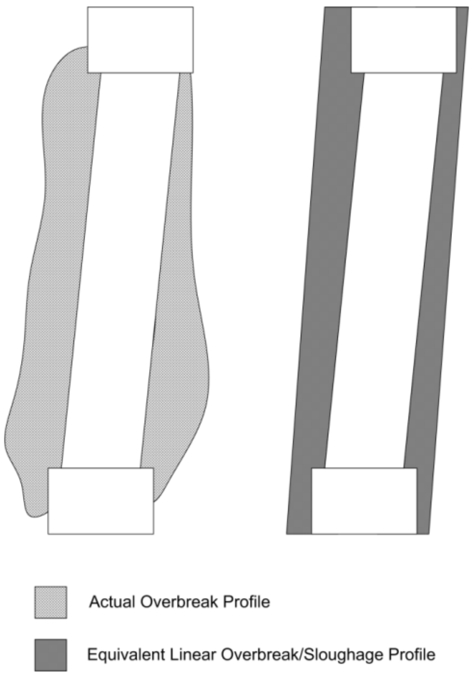

Figure 2.2: Equivalent Linear Overbreak or Sloughage (Hughes, 2011)

The convenience in the method of analysing linear overbreak as a variable is that it is independent of mining width, making it a useful tool in analysing and comparing dilution in different mining widths without introducing a bias, making the ELOS approach more objective. This is particularly of the essence in the case of narrow-vein mining where percentage dilution is very sensitive to mining width. Therefore, during this research, dilution will be expressed as both the equivalent linear overbreak or slough (ELOS) and a percentage.

$$ELOS = \frac{Volume\ of\ Overbreak\ (Slough) From\ Stope\ Surface}{Stope\ Height * Wall\ Strike\ Length} \quad (4)$$

2.3. Importance of Dilution Management

Improper management of dilution is one of the most prevalent reasons for economical mine failure, not only within narrow-vein mining but within underground stoping in general. Economical mine failure due to poor dilution management is a potential threat to a mining operation because unplanned dilution and ore loss directly influence the productivity of underground stopes and therefore the profitability of the entire mining operations (Jang, 2014).

The importance of unplanned dilution and ore loss management is underlined in various studies. Tatman (2001) states that minimizing unplanned dilution is the most effective method of increasing mine profits. The impact of unplanned dilution on the productivity of mining operation has also been emphasized by Pakalnis et al. (1995a) and Henning & Mitri (2008). Their studies demonstrated the serious negative economic impact of unplanned dilution and the opportunity costs which arose from additional mucking, hauling, crushing, hoisting, milling and processing of the superfluous mined waste. Stewart & Trueman (2008) took a closer look on typical narrow vein mines and quantified the costs of unplanned dilution to be 25 AUD per tonne, which is significantly higher than typical mucking and haulage costs of 7 AUD per tonne. Suglo & Opoku (2012) analysed the economic loss due to unplanned dilution at the Kazansi mine in South Africa and concluded that the economic loss due to unplanned dilution from 1997 to 2006 was a soaring 45.95 million USD. Another example is the Konkola Mine in Zambia, which spent 11.30 million USD in 2002 alone to manage unplanned dilution in their operation (Mubita, 2005).

One thing at the Jokisivu operation that instantly strikes the attention considering the financial aspect of dilution is the fact that there is no processing plant on site, meaning that the mined materials must be transported some 40 kilometres away, to the Vammala processing plant. Consequence of having an elevated level of waste rock dilution is the fact that more material has to be transported to the processing plant, which implies more costs and thus reducing the profitability of the operation. The costs involved in handling excessive waste rock in the Jokisivu operation are dependent on stope location of which the material originates, but on average, the total costs of handling one tonne of waste rock are estimated at 30 euros (Ridaskoski, 2018). On top of that, the mill has a certain processing capacity. Processing additional waste rock instead of gold therefore also reduces the efficiency of the operation and lowers the net gold production per hour. It is for these reasons essential to keep dilution to a level as low as practicable.

3. Factors Influencing Dilution

Within this section, the fundamental reasons for unplanned dilution in steeply dipping narrow-vein deposits and in longhole stoping in general will be covered. Unplanned dilution and ore loss are amongst the most complex phenomena in underground stoping operations (Jang, 2014) and respectively, a large amount of known as well as unknown factors contribute to these phenomena during stoping operations. Additional to the fact that there are a vast number of factors influencing unplanned dilution and ore loss, a lot of these factors are dependent on each other and a change in one factor is likely to affect another factor. Therefore, it has been proven difficult to analyse the underlying structures of unplanned dilution on a single causative factor (Jang, 2014). Because of that, Jang (2014) divided the factors influencing unplanned dilution and ore loss into four encompassing groups, being:

- Geological factors;
- Stope design factors;
- Drilling and Blasting factors;
- Human error and others.

Other researchers allocate the same groups of factors as major influences on unplanned dilution and ore loss. Additional to these factors, Clark (1988) emphasizes on the efficiency and time related to mucking and backfilling the stope. Wang (2004) and Stewart (2005) describe this as the stope expose time and designate this, along with stope undercutting as two other crucial factors of influence. As mentioned in the research objective, the focus of this thesis will primarily be kept on some of the drilling and blasting parameters. In addition to these factors, the influence of rock support on unplanned dilution in steeply dipping narrow-vein deposits is being reviewed. The other aspects which are named above, will be discussed only briefly.

3.1. Geology and Rock Mechanics

Several geological factors that have been proven to be of influence on unplanned dilution and ore loss in (steeply dipping narrow-vein) stoping operations are: levels of in situ and induced stresses in the rock mass, as well as joint conditions, rock type and properties, the underground water conditions, block sizes and even other minor components such as thermal, chemical and biological effects could play a role on the geological setting of the stope and therefore the unplanned dilution associated with the production of the stope (Potvin, 1988; Villaescusa, 1998; Clark, 1998; Tatman, 2001; Mubita, 2005; Stewart, 2005). According to Lappalainen & Pitkajarvi (1996), the dilution due to the geology of an orebody may comprise up to one third of the total dilution in an underground mining operation, depending upon orebody complexity.

3.2. Human Error

Clark (1998) stresses the importance of human factors influencing dilution and recovery. He indicates that the only factors that are not within human control are the orebody characteristics, rock structure and quality and to some extent, the stress conditions in the stope. These factors of influence would best be placed in the geological factors group, described by Jang (2014). Assuming Clark (1988) is correct, the mining engineer should have a proficient and adequate understanding of the geotechnical and geological conditions on site; to design the development drifts, stopes and their sequencing accordingly.

However, human error does not only play a part in the design of the mining operation, mistakes can take place at any phase of the operation, especially at production level. A frequent practice where human error regularly occurs is the drilling of blastholes in (narrow) stoping operations (Clark, 1998; Villaescusa, 1998; Wang, 2004; Hughes, 2011; Jang, 2014). Human errors in drillhole deviation can be allocated to inaccurate surveying, alignment errors of equipment and poor hole collaring. Clark (1988) and Villaescusa (1998) also address the lack of supervision and communication to be a cause for human error.

Another factor which does not essentially causes unplanned dilution, but rather does not help preventing or reducing it, is the lack of stope performance review (Villaescusa, 1998). After a stope has been blasted and mucked, the excavated space can be monitored using a cavity monitoring system (CMS) (Miller & Jacob, 1993). A CMS is a three-dimensional laser scanning apparatus, particularly designed for the surveying of underground cavities. The CMS measurements provides a data set from which the excavated cavity can be presented in a three-dimensional model. Overlaying the designed stope profile with the obtained three-dimensional CMS model can give rather helpful insights on the accuracy of the blast, the levels of overbreak, unplanned dilution and therefore human errors in design and production (Clark, 1998; Stewart, 2005; Hughes, 2011). More about the use of cavity monitoring systems will be covered in section 5.

3.3. Drilling and Blasting

As described before, dilution usually occurs as overbreak during blasting and as wall sloughage during the existence of the stope. There are numerous factors influencing unplanned dilution which are associated with blasting. These factors of influence are predominantly operational parameters. Correctly aligning these parameters will result in a successful blast, meaning: achieving the desired fragmentation of ore while limiting the blast to the designed stope boundaries, as exceeding these boundaries will cause overbreak, resulting in unplanned dilution. The goals of achieving sufficient fragmentation and creating no blasting induced damage to hangingwall and footwall (overbreak) are conflicting and can generally not be met (Wang, 2004). However, optimizing these blasting parameters reduces overbreak significantly.

As geological and geotechnical setting and operating conditions differ from site to site and rock breakage by blasting is often highly unpredictable, there is a strong demand for blasting research. According to Wang (2004), quantifying the influence of blasting parameters on blasting induced damage is rather difficult. He describes the following parameters as being the predominant parameters that influence blasting induced damage:

- Rock mass properties;
- Drillhole design;
- Drillhole accuracy;
- Explosive type;
- Wall control methods;
- Explosive distribution;

- Explosive per delay;
- Initiation sequence;
- Free face availability.

Incorporating all these parameters in this research will produce results which are difficult to interpret and allocating results to a specific root causes may result in unreliable conclusions. Therefore, only a few of these parameters will be investigated more closely during this thesis. More information on how and which parameters will be analysed to ultimately optimize the blast design for the Jokisivu operation can be found in section 5.

3.3.1. Drillhole Design

The term drillhole design is a major factor in blast design. It regulates the distribution of explosives in the rock mass and encloses a lot of geometrical parameters, such as drillhole diameter, length, spacing, burden and orientation. The design of drillholes is mostly based on equipment limitations of the machinery that the mine is working with, common practices, past experiences, benchmarking and rule of thumb. During the lifetime of an operation the values of these parameters are regularly updated depending on local rock mass conditions, stope dimensions and whether the result of the previously designed blast is satisfactory or not. These drillhole design parameters are closely related in longhole (narrow-vein) blasting (Wang, 2004).

Reviewing drillhole length and drillhole spacing, one finds the relation that the longer the drillholes, the larger the diameter of the holes should be to reduce hole wander or deviation. A larger hole diameter results in more explosives being loaded per hole. However, to maintain the desired explosive consumption per tonne of rock, for a larger drilled hole diameter, larger spacing and burden must be retained. Drillhole spacing is the distance between drillholes in the direction parallel to the available free surface and burden is the distance between drillholes in the direction perpendicular to the available free surface. When the burden or the hole spacing is too small, the blast will not optimally use all explosive energy and create flyrock, which could damage the stope, or equipment. Contrarily, having too large of a hole spacing or burden will cause insufficient breakage of the ore, which provokes problems in ore handling and when entirely gone wrong, the need for a secondary blast.

Considering the orientation of the drillholes in longhole blast design, Clark (1988) states that whenever possible, the drilling horizon should be designed such that drillholes can be drilled parallel to the orebody contacts, controlling the blasting techniques and minimize wall damage and therefore unplanned dilution. Drillholes that are not drilled in the same direction as the desired stope wall, will generate disproportionate damage, making controlled blasting techniques impossible (Morrison, 1995).

3.3.2. Drillhole Accuracy

Drillhole accuracy or drillhole deviation, is a factor that is very susceptible human error and can be expressed in terms of percentage, as can be seen in Equation (5). Nonetheless, human error is not the only cause of improper drillhole accuracy. Forsyth et al. (1994) illustrate that drillhole deviation originates from both internal and external deviation. Internal deviation refers to causes like drilling equipment, its operation and the complexity of geology. External deviation can be allocated to inaccurate surveying, errors in drill set up and poor hole collaring.

$$Drillhole\ deviation\ (\%) = \frac{Distance\ between\ actual\ \&\ planned\ hole\ toe}{True\ length\ of\ planned\ hole} * 100 \tag{5}$$

Typically, there are three ways in which a drillhole can deviate:

- The starting point of the hole is not in the desired location (d).
- The angle of the hole deviates from the design (θ).
- The drill trajectory changes eccentrically (e).

From these deviation parameters, the eccentricity of a drillhole is the parameter which can be influenced the least. Eccentricity of drillhole trajectory usually originates from drill bit interaction when the drill bit encounters anisotropic bedrock conditions like joints, faults, weathered or altered rock masses and foliation or bedding planes (Scarpato, 2016). Figure 3.1. on the next page, depicts the three possible ways in which a drillhole may deviate.

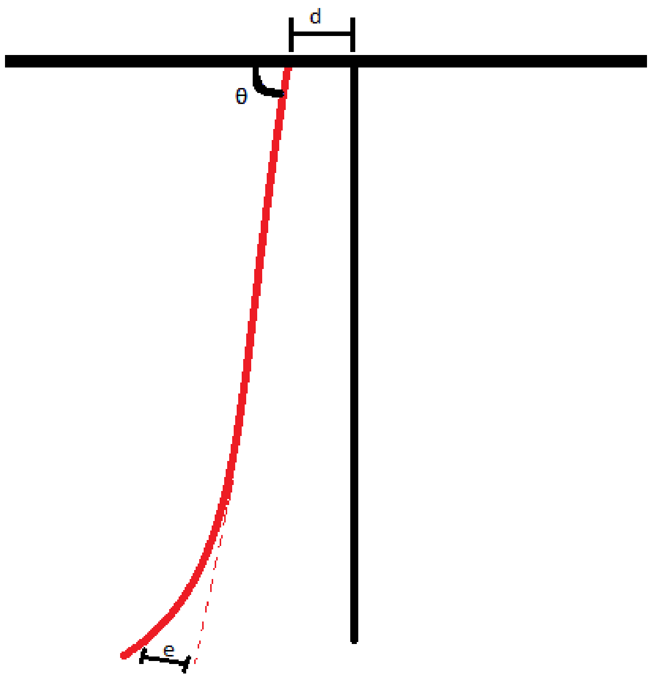

Figure 3.1: Three ways of drillhole deviation.

A drillhole trajectory can divert in two ways from its original dip, being towards or away from the stope hangingwall. Deviation in drillhole trajectory could cause undesirably high explosive concentrations in some area, which could cause damage to the stope walls and therefore unplanned dilution on one hand, and a lack of explosive in another area, resulting in insufficient fragmentation of ore (Wang, 2004).

3.3.3. Explosive Distribution

Another factor which influences the fragmentation of rock and the level of stope wall damage and therefore the unplanned dilution from a blast is the explosive distribution. The distribution of explosives is linked with the mentioned parameters above, such as drillhole deviation, drillhole design and rock mass characteristics. Whenever drilling and charging has been done right, the explosive distribution can be reliably expressed with the term 'powder factor'. The powder factor is a relationship between how much rock is fragmented and how much explosives are used to break it. It can be useful to know the powder factor as it can serve as an indicator of the associated rock mass strength or as an indicator for the costs associated with the quantity of explosives needed and is defined according to Equation (6) (Mindat.org, 2018).

$$Powder\ Factor\ \left(\frac{kg}{t}\right) = \frac{Amount\ of\ Explosives\ used\ in\ Blast}{Tonnes\ of\ rock\ broken} \tag{6}$$

Equation (6) shows that the higher the powder factor, the more explosives are being used and therefore typically the harder the rock mass. A side note to this however, is the desired fragmentation size. A smaller fragmentation means a higher powder factor and vice versa. Therefore, one must not look at powder factor alone and preferably use it as a first indicator rather than a hard rule.

3.3.4. Explosive Per Delay, Initiation Sequence & Free Face Availability

Alternative methods which have been proved working for reducing stope wall damage, are optimizing the initiation sequence and reducing the explosive per delay. Hagan (1996) found that the result of a multi-hole blast is highly determined by interaction between blastholes. The time interval between detonations as well as the sequence in which the blastholes are initiated have a big influence on blast performance. A blast can only be successful if charges are detonating in a regulated sequence at convenient spaced time intervals. Clark (1998) and Villaescusa (1998) suggest that blasting sequences should be designed to make sure there is enough free face available and that the number of holes per delay period should be kept as low as possible to guarantee a successful blast. They also stress the importance of periodically checking the blast performance using blast monitoring techniques. This can be done by measuring the blasting induced vibrations, drillhole deviation, drillhole angles and the distance of the holes to exposed stope walls.

3.4. Stope Design

Designing a stable stope is considered to be a rather complex process, as one has to take into account numerous factors (e.g. mining method, stope dimensions, level spacing, support and fill requirements etc.). Incorrect stope design can cause exorbitant levels of unplanned dilution and ore loss. Therefore, the preliminary geological, geotechnical and rock mechanical analyses should be performed with extreme care (Jang, 2014).

Numerous studies have investigated the relationship between stope design and unplanned dilution. Stewart (2005) found that when designing a stope, attention should not only be paid to mining costs and production rates, but also to stope stability and dilution potential, as dilution and potential equipment damage may entail a lot of opportunity costs. Pakalnis et al. (1995) concluded that the relative amount of dilution (percentage) increases when a stope becomes narrower. Connecting the findings of Stewart (2005) and Pakalnis et al. (1995) really stresses the importance of unplanned dilution control in narrow-vein stopes.

Hughes et al. (2010) performed a case study on unplanned dilution at the Lapa Mine in Canada and found that unplanned dilution in narrow-vein mining can be reduced significantly when the length of the stope strike is being decreased. Diederichs & Kaiser (1999) studied the effect of undercutting and overcutting stopes. They found that undercutting and overcutting significantly increases the stress relaxation zone of a stope, which results in instability of the stope. Stope instability can in its turn cause enormous amounts of dilution. Figure 3.2. displays the increase in stress relaxation zone at different levels of under and overcutting. Situation 1 in Figure 3.2. depicts the normal stress state when a stope has been excavated. However, whenever a stope is being undercut or overcut the stress relaxation zone increases significantly as being shown in situation 2 and 3.

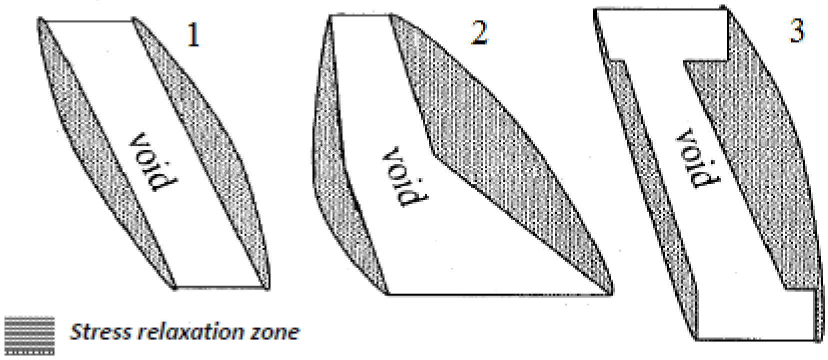

Figure 3.2: Under and Overcutting and its Effects on Stope Stress Relaxation Zones after Hutchinson & Diederichs (1996).

Other studies underlined the effect of stope height and dip on unplanned dilution. Perron (1999) concluded from a field study that the unplanned dilution of a stope is very sensitive to stope height and succeeded in reducing unplanned dilution by developing an extra sub-level which halved the stope height. This resulted in a lower production rate, but the goal of reducing dilution was achieved. Considering the relation between the dip of a stope and unplanned dilution, the general trend is that stope overbreak decreases when the dip in the hangingwall increases (Yao et al., 1999). The reason for this is that when the hangingwall dip reduces, it becomes more difficult for vertical stresses to pass around a stope and therefore create zones of relaxation. Larger relaxation zones may result in caving and therefore making shallow dipping stopes more sensitive to the effects of gravity (Hughes, 2011).

The influence of factors like stope geometry and inclination amongst others described in this section, have to be taken into account during the designing of a stope. Potvin (1988) incorporated these factors into an empirical stability graph, initially designed by Mathews et al. (1981) which he modified. The origins and applications of the Mathews Stability Graph and Potvin's Modified Stability Graph will not be discussed in this literature review as they will not be used in this thesis. However, other studies based on these stability graphs will be.

One of these studies is from Clark & Pakalnis (1997), who quantified dilution as equivalent linear overbreak or slough (ELOS). They have used the Mathews Stability Graph and the Modified Stability Graph and coupled this with their principle of ELOS and incorporated this into an ELOS design graph which indicates the following:

- ELOS < 0.5 metres is considered to be blast damage only;
- 0.5 < ELOS < 1.0 metres is considered as minor sloughing;
- 1.0 < ELOS < 2.0 metres is considered as moderate sloughing;
- ELOS > 2.0 metres is considered as severe sloughing or possible wall collapse.

Suorineni et al. (2001) elaborated on the study of Clark & Pakalnis (1997) and combined the ELOS design graph with Potvin's (1988) Modified Stability Graph to quantitatively define whether a stope is stable, unstable or caved. According to this definition:

- A stope is stable if ELOS ≤ 0.5 metres;
- A stope is unstable when 0.5 < ELOS < 5 metres;
- A stope is caved when ELOS > 5 metres.

Suorineni (1998) found that the three zones of the stability graph have transitional boundaries with considerable amounts of overlap, meaning that a stope which plots in the stable zone could still be unstable or even cave. Given explanation for this is that not all factors which affect stope stability are covered in the stability graph method. Therefore, the stability graph alone is not a reliable source of stope design and other factors should be considered as well.

3.5. Stope Support

When geomechanical conditions of a rock mass are unfavourable, it is likely that an underground excavation such as a stope might not be bear the stresses induced by excavating the stope and keep its structural integrity. In this case, it is crucial to provide some sort of support to the stope to prevent sloughage, spalling or even caving of the stope. Diederichs et al. (1999) define the principle of stope wall support as reinforcing the rock mass to prevent unravelling and allow the stope wall to be able to form a self-supporting beam to keep its structural integrity. Depending on what phenomena one would like to prevent from happening, different suitable ways of supporting a stope a possible. Within this thesis, the effects of meshing and cablebolting on unplanned waste rock dilution will be investigated and therefore the literature study on stope supporting methods will be delineated to these two methods of support.

3.5.1. Cablebolting

One method of supporting a stope is to employ the use of cablebolts. Cablebolts are long flexible tendons which are made from multiple steel strands that are grouted into drilled holes. They aim to reinforce and maintain the integrity of a rock mass, allowing it to form its own load bearing structure (Hutchinson & Diederichs, 1996). Cablebolts can be used to pre-strengthen the hangingwall (Chen & McKinnon, 2012) and have been proven highly successful in controlling dilution (Anderson & Grebenc, 1995) and improving (narrow) stope stability (Kaiser et al., 2001; de Vries et al., 2003). Cablebolts have been proven to work really well for rocks with a strength over 4000 to 5000 psi, especially if the rock blocks are relatively big and the major structure is parallel to the stope strike (Chen & McKinnon, 2012).

Within stoping, support is primarily installed in the stope hangingwall. Fuller et al. (1983) illustrated three main approaches to install hangingwall cablebolt support. These methods are shown in Figure 3.3. However, in the case of the Jokisivu mine, where the geologic setting describes a steeply dipping narrow-vein deposit, options (a) and (b) are seldom applied due to the fact that there rarely is development parallel to the ore drift where the stope is to be situated. Therefore, the point anchor method is the most used cablebolting pattern in narrow stopes. Despite the point anchor method being a successful method in stabilizing hangingwalls in narrow-vein stopes, the installation might become uneconomical when dealing with tabular narrow-vein deposits (Stewart, 2005).

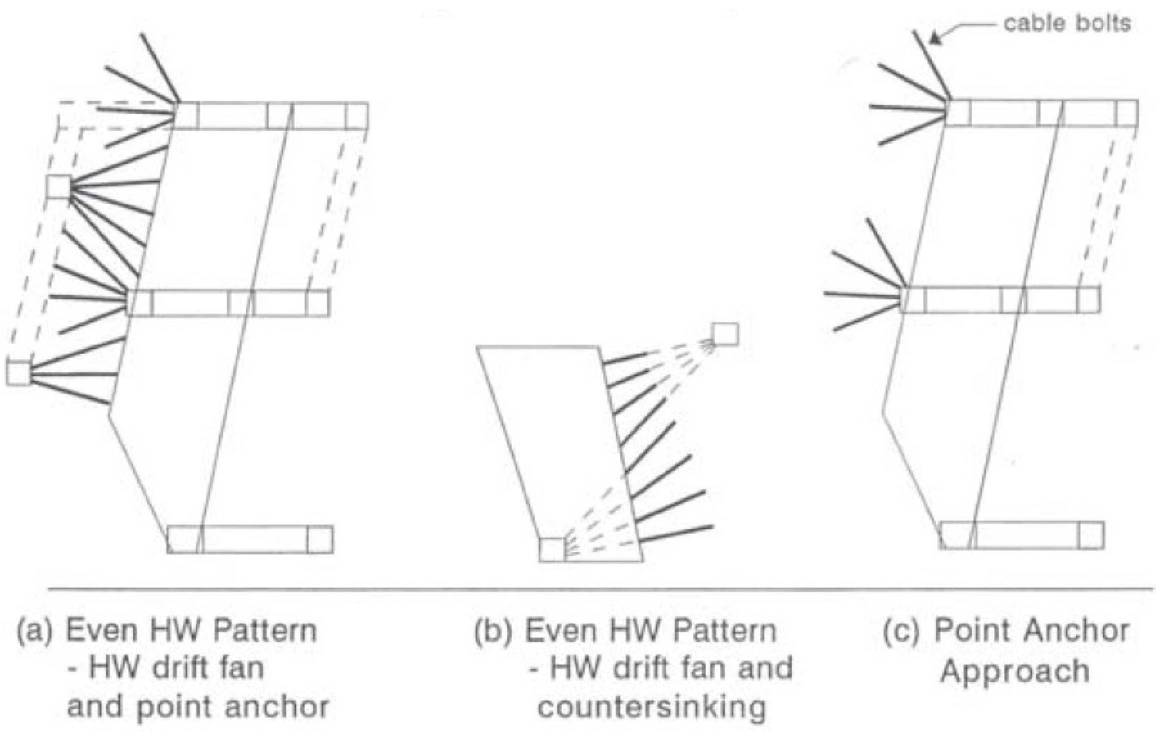

Figure 3.3: Three dominant methods of cablebolt hangingwall support (Fuller et al., 1983)

An important element in designing a cablebolting support plan, is that the cablebolts have to be long enough to anchor the unsupported rock mass into place. Hutchinson & Diederichs (1996) propose that cablebolt lengths have to be installed in such a way that there is at least an anchorage of two metres beyond the maximum depth of relaxation. Additionally, if it is not possible to determine the extent of the relaxation zone, Hutchinson & Diederichs (1996) recommend the cablebolt length to be equal to roughly 1.0 to 1.5 times the hydraulic radius of the surface to be supported, which is in most cases, the stope hangingwall.

Within narrow-vein stoping, cablebolts must be preinstalled to support the stope hangingwall. Mining of the stope can negatively affect the cablebolt strength and thereby the level of stope support (Hyett et al., 1992). Rock mass deformability can reduce the bond strength between a cablebolt and the grout which initially keeps the cablebolt in place. Failure of this bond can be observed after mining of the stope by cablebolts hanging loose from the stope hangingwall. Another factor which can reduce the performance of cablebolts is the decrease in radial stiffness due to mining induced stress changes. Radial stiffness can be defined as the ratio of load to the elastic deformation, also often referred to as deflection (SKF Group, 2018). Hyett et al. (1992) studied the relationship between rock mass properties and the radial stiffness of drillholes thoroughly. During this research, they found that the reduction in mining induced stress decreases radial stiffness and with that, the cablebolt bond strength. Hutchinson & Diederichs (1996) stated that these problems, related to stress relaxation and decrease in stress can mostly be avoided by plating the cablebolts and using bulbed cables instead of plain ones.

Cablebolting will, in most cases, not completely eliminate dilution in narrow-vein stoping. Due to the fact that some parts of a stope are not accessible for bolting or cablebolting machines have a certain operational limitation, a guaranteed amount of dilution due to wall sloughage is unavoidable (Heslop & Dight, 1993). On top of that, cablebolts do not provide sufficient support to prevent failure caused by larger structures such as faults. In these cases, it is required to leave pillars in place which stabilize the rock mass and prevent stope failure. Small pillars have been proven to be beneficial in the reduction of dilution in numerous mines. However, leaving these pillars in a high stress environment may result in a violent collapse and potential danger and damage (Heslop & Dight, 1993). If the hosting rock mass has low strength, cablebolts will not be as effective. Chen & McKinnon (2012) propose reduced stope length as an alternative. Shortening stope length results in a smaller exposed stope area and reduces the stope exposure time during mucking and when applicable, backfilling.

3.5.2. Meshing

Where cablebolting is considered to be an active rock reinforcement method, meshing is designed to be a passive method of support (Windsor, 1999). The practice of meshing is a vital aspect of providing a safe working environment in underground excavations. Where cablebolts or rockbolts in general are used to control the excavation stability, mesh is principally installed to prevent small pieces of loose rock from collapsing (Villaescusa, 1999). Mesh is not designed to bear substantial amounts of broken rock. Loosening of small rock pieces predominantly occurs in rock masses which consist of smaller blocks or rock masses that undergo mining induced stress changes. Hoek & Wood (1987) report that mesh is used to successfully keep small blocks in place between cablebolt faceplates. More information on the research concerning the effects of meshing on waste rock dilution in the Jokisivu operation can be found in section 5.

4. The Jokisivu Operation

The research of this thesis has been conducted at the Jokisivu gold mine. To grasp the essence and the purpose of this research, it is key to have a general understanding of the regional geological and mineralogical setting and a basic knowhow of the practiced mining method.

The Jokisivu gold mine is situated in Finland, in the municipality of Huittinen, 7 kilometres south-south-west of Huittinen town and 85 kilometres north-east of Turku (Figure 4.1.). It is owned by Dragon Mining Limited., an HKEx listed, Australia-based company with its main operations in Finland and Sweden. As of per 30 September 2017, the Jokisivu total reserves are estimated at 1,013 kt, grading a 2.9 g/t average for 95.2 kozs of gold. The cut-off grade of the operation is dependent on location and origin but generally, the cut-off grade for productional stopes ranges between 2.0 and 2.4 g/t. The operation currently runs at 240 kt/y with a varying grade between 3 g/t and 4 g/t.

Figure 4.1: Location of the Jokisivu gold mine (from: Google Maps, 2018).

4.1. Geological and Mineralogical Setting

The Jokisivu operation comprises two deposits. The ‘Kujankallio’ deposit and the ‘Arpola’ deposit, which are approximately 200 metres apart. Figure 4.2. gives an overview of the dimensions of both the deposits as well as the geometries and the development. The Jokisivu deposits are mesothermal orogenic gold deposits which are hosted within the Paleoproterozoic Vammala Migmatite Belt (Dragon Mining Ltd., 2018a).

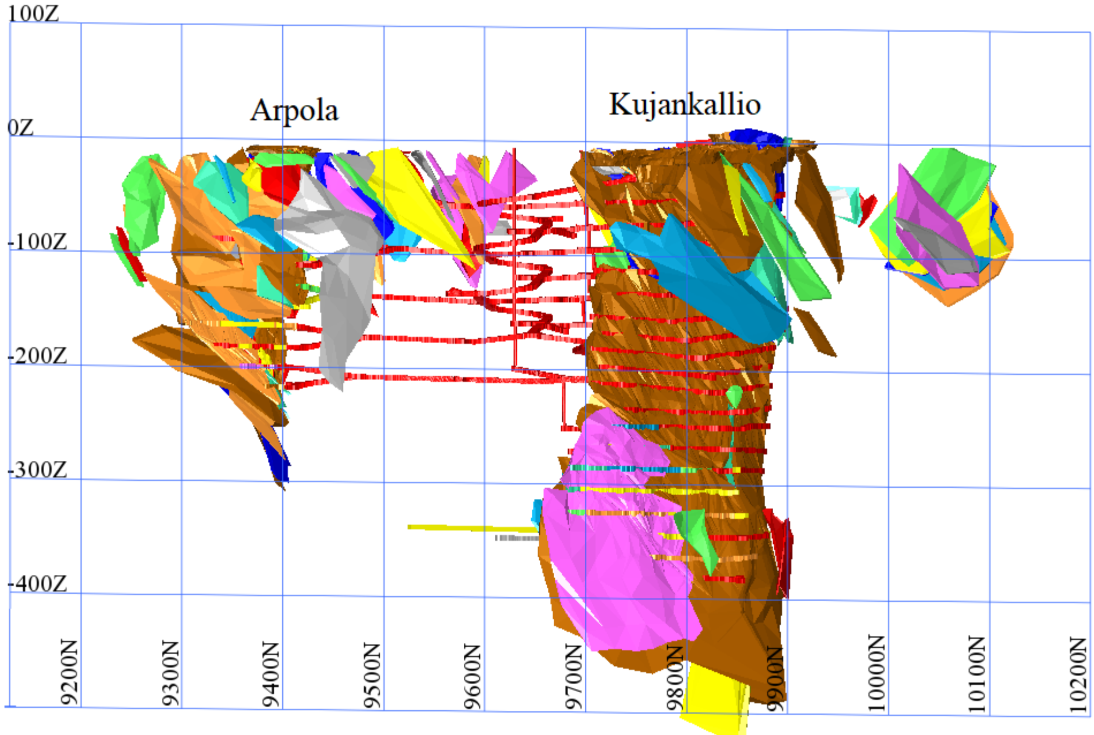

Figure 4.2: Resource model and development of the Jokisivu Operation as of July 2018. (from: GEOVIA Surpac, 2018)

The host rock is Diorite (1.88 Ga) surrounded by mica gneisses, volcanogenic and arenitic metasedimentary gneisses and intermediate metavolcanics rocks with a tonalitic to granodioritic composition. The host rock contains sets of pegmatite veins, which both pre and postdate the mineralization. The mineralization (1.8 Ga) primarily consist of free gold particles in quartz veins, which are locally related to arsenopyrite, loellingite, pyrrhotite and scheelite. The grain size of the gold ranges from a few micrometres to a few millimetres in diameter and some gold particles can also be found in altered intrusive host rock (Dragon Mining Ltd., 2018b).

The mineralization zone is strongly controlled by faulting, folding and a shearing related boudinage, which means that the structure of the ore zone is controlled by a 1 to 5 metres wide shear zone that is characterised by laminated pinching and swelling quartz veins. These ore zones are generally narrow (approximately 2 metres average) made up by separate subparallel and crosscutting quartz veins, dipping 45° to 70°. Figure 4.3. displays a picture of a drifting cross-cut which gives an overview of the general geological and mineralogical setting.

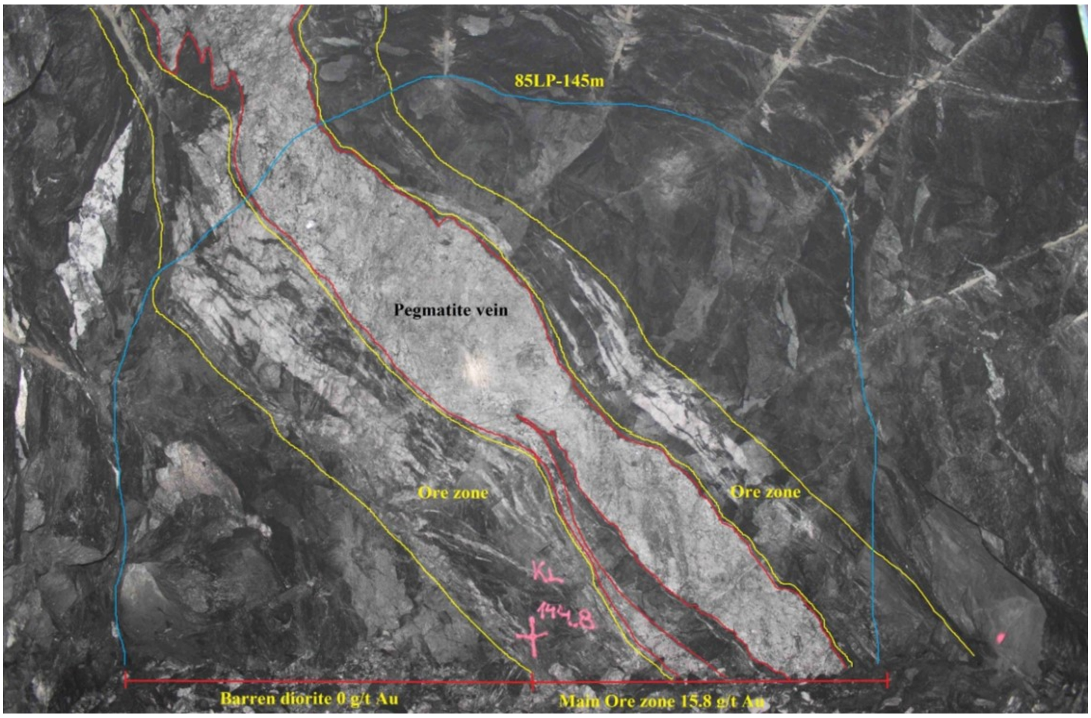

Figure 4.3: Drifting cross-cut with a typical geo-mineralogical setting (from: Dragon Mining Ltd., 2018b).

4.2. Mining Method and Production

The Kujankallio deposit has been into production since 2009 by open pit mining methods. Consequently, the mine progressed, and underground development commenced in 2010. Likewise, the Arpola deposit started producing as an open pit in 2011 and switched to underground mining in 2014. As can be seen in Figure 4.2., underground access to both deposits is realized by the decline, which is located at the eastern most end of the Kujankallio open pit at a depth of 35 metres (Dragon Mining Ltd., 2018a). The decline is under continuous development and currently has its lowest point at 420 metres below the surface.

The underground mining method which is being used for extraction of the ore is narrow-vein (longhole) stoping. In general, stope dimensions and geometries are predominantly dictated by the geological structure of the ore-bearing zone, rock-mechanical properties and operational parameters. More on these matters can be found in section 5.1.

5. Research Methods and Data Acquisition

It has been decided that for this research, data sets will be obtained from two stopes. Both stopes were designed from scratch by the researcher. Designing the research stopes and obtaining the data sets rather than working with existing data sets has two major benefits. The first benefit lies in the matter that the researcher has a much clearer understanding of all the assets of the stopes and its conditions and therefore, the origin of the data. The second advantage is the fact that the researcher is able to design, influence, monitor, map and report throughout the entire life of the stope, from the designing phase until the end of backfilling the stope. With this in mind, it is assumed that results will be more reliable and enhance the research.

Both research stopes are both situated at the Jokisivu mine, but in different deposits, namely the Kujankallio deposit and the Arpola deposit. These deposits are located at different depths and have different geometries (Figure 4.2.). Therefore, these stopes face other conditions and challenges, both in their design and during production. Decisions made in the designing of the stopes are predominantly based on previous experiences and expertise of the people working at the operation. The decision to rely on previous experiences is due to the complexity of the operation and its geological and rock mechanical conditions. However, there are some aspects in the designing which have not been tested at the operation before. These will be covered more in depth in their corresponding sections further on in the report.

5.1. Stope Design

Other than the factors mentioned before, a crucial factor which dictates stope design is the direction and dimensioning of drifting. While drifting, the aim is to keep the ore-bearing mineralization intersected in the middle of the drift face as drifting progresses. This directly influences the ease of operating the longhole drill rig and the quality of the resulted stope. In the Jokisivu operation, a Sandvik DL 421-7C drill rig is being used for stope drilling. This drill rig is relatively large, making positioning of the boom difficult if the drift does not follow the ore-bearing mineralization. Geology mainly dictates stope design in width and dip. Jokisivu has steeply-dipping narrow vein gold deposits, therefore stopes are designed accordingly. In-situ rock conditions dictate the height and the span of the stope. Joints, faults, foliations and fractures dictate the span of the stope. The general rule is that the more disturbances are present in the host rock, the smaller the stope span will be. This urges the need of (half-) pillars inside or in between stopes.

5.1.1. 320K14 Stope Design

The first stope that has been designed is the stope situated in the Kujankallio deposit, at a depth of 320 to 300 metres (drifts 320Lp3 and 300Lp3) and will be referred to as stope 320K14. As can be seen from Figure 5.1. and from the more detailed plans that have been handed out to the contractor (Appendix A.), the 320K14 stope has two opening raises. The need for this arises from the fact that a half-pillar has been designed to support the stope midway, between rings 10 and 12.

A half-pillar is a pillar of which the bottom part will be blasted. It is being used to provide support in stopes where a normal full pillar would provide excessive support and one does not want to change the pillar width. It also minimizes additional ore loss compared to leaving a full pillar in place and speeds up production, because a new opening raise only needs to be made throughout half the height of the stope. A half-pillar provides an alternative stress path around the excavation and a mean to relieve some mining induced stresses in the stope. Drill holes are drilled through the pillar, but only the bottom half will be blasted.

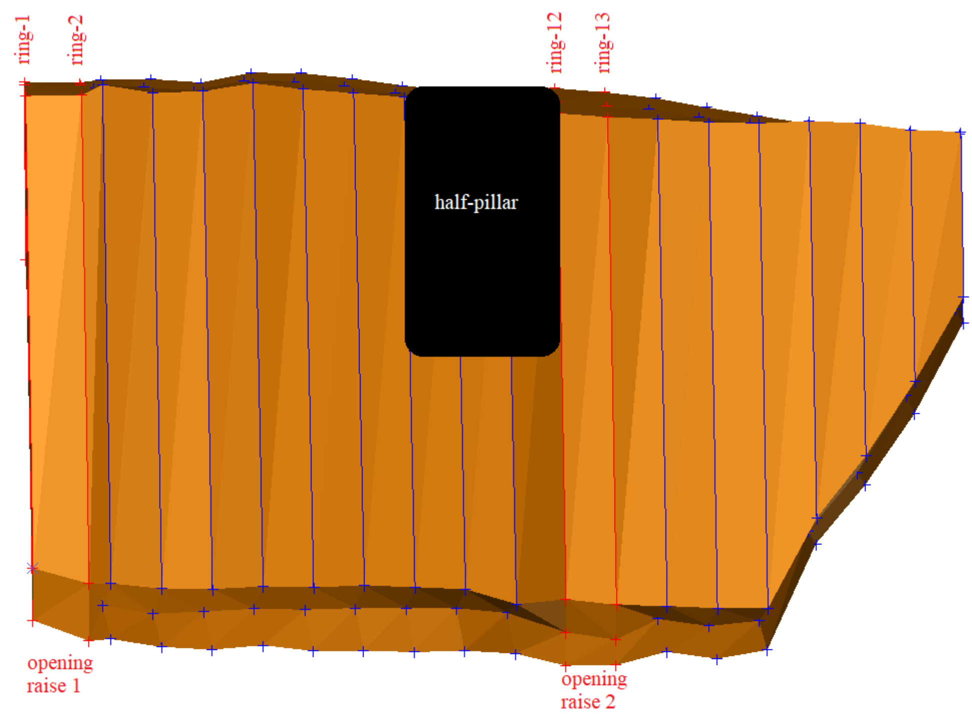

Figure 5.1: 320K14 Designed stope outline.

The reasoning behind the urge for this half-pillar is that the first part of the stope (rings 1-12) is partly hosted in mica-gneiss. Mica-gneiss has been known in the Jokisivu mine as being a weak and (highly) fractured host rock formation.

Another thing that catches the eye in the design of the 320K14 stope is the fact that the rings start shortening from ring 17 onwards. This has been done purely because of the plunge of the high-grade ore-bearing material. Drilling and blasting rings 17-20 in the same fashion as previous rings would result in unnecessarily mining additional waste rock material.

The stope has been designed to contain 3966 in-situ tonnes of ore with an estimated grade of 3 g/t. The designed half-pillar that has to be left in place contains 466 in-situ tonnes of ore, also at a grade of 3 g/t. Expected is that due to poor rock conditions in the mica-gneiss at the beginning of the stope, unplanned dilution levels will be rather high, ranging from 30 % to 60 %. This would result in an expected diluted stope tonnage between 5155 and 6346 tonnes.

Converting this into terms of ELOS requires the stope height and stope hangingwall strike length as well as the expected volume of overbreak. Tonnages of rock can be converted to volumes of rock by dividing the tonnages of rock by the density of the rock which is set at 2.8 t/m^3. Stope height in Equation (4) refers to vertical stopes. However, the shallow dip angle (Figure 5.2.) increases the height of the 320K14 stope. The stope wall height averages 20 metres and the wall strike length is about 35 metres. Using Equation (4) provided in the literature review, results in an ELOS estimation between 0.6 m and 1.2 m, which according to Suorineni et al. (2001) means the stope is defined as unstable.

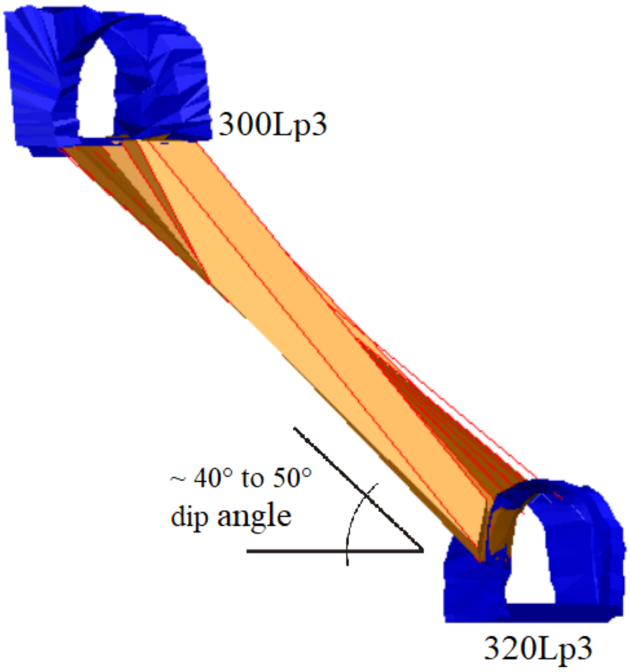

Figure 5.2: 320K14 Shallow dip angle.

5.1.2. 100A22 Stope Design

The second stope that has been designed is situated in the Arpola deposit, at a depth of 100 to 80 metres (drifts 100LpA22 and 80LpA22) and will be referred to as stope 100A22. As can be seen from the plans that have been handed out to the contractor (Appendix A.), the 100A22 stope is designed to be drilled both uphole (first 15 rings) and downhole (last 8 rings). The separation between uphole and downhole had to be made because it would have been impossible to position the drill rig inside a drift in such a way that the entire stope could be drilled from one level. The inclination of the upholes originates from the habit of designing uphole stopes in this way. This is usually being done when there is no drift situated above a proposed stope which the holes can penetrate. This 15° angle together with adding overdrill to the uphole holes serves the purpose of preventing ore loss. However, since the 100A22 stope has a drift situated above the stope, the inclination only serves the additional purpose of increasing the ease of throw of the blasted rock and offers more resistance against leaking explosive emulsion due to gravity.

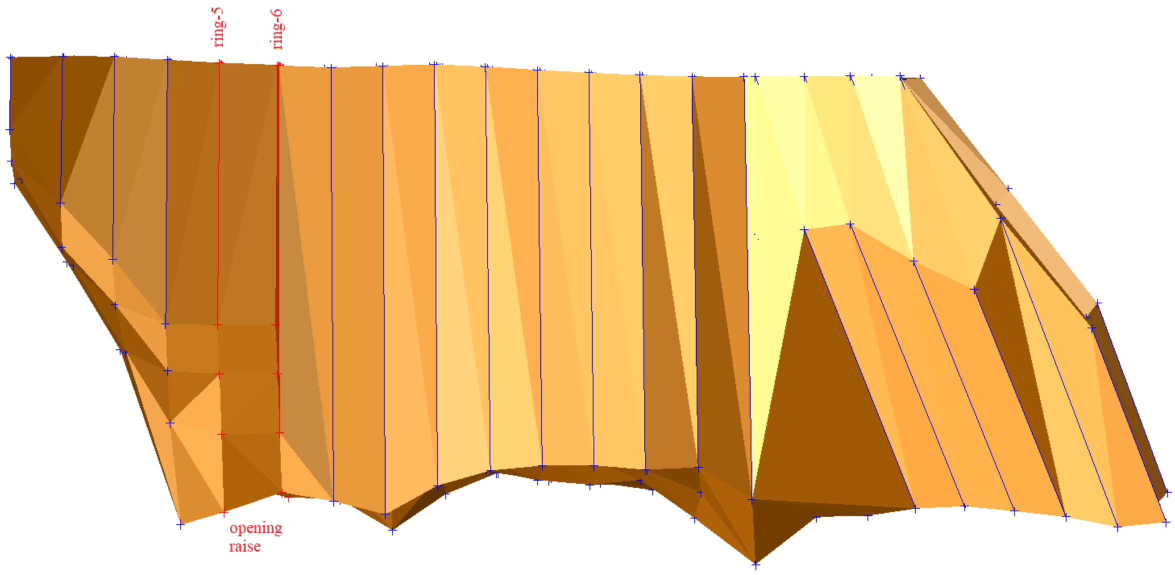

Figure 5.3: 100A22 Designed stope outline.

It can be seen from Figure 5.3. that the opening raise of the stope is located between rings 5 and 6. The reason behind this is that rings 1 to 4 do not contain ore along the whole height of the stope as the gold mineralization zone follows a certain plunge. Locating the raise in these rings would result in unnecessary waste rock. Therefore, rings 1 to 4 will be pre-charged and blasted after the raise has been opened and enough room has been created for the rock of rings 1 to 4 to throw into. Just like rings 1 to 4, rings 21 to 23 also do not contain ore throughout the entire height of the stope due to the plunge of the orebody. Therefore, the rings are shortening towards the end of the stope.

The 100A22 stope has been designed to contain 6003 in-situ tonnes of ore with an estimated grade of 3 g/t. For the largest part, the 100A22 stope is dipping relatively steeply (70° to 80°). Also, the stope is not hosted in the weak mica-gneiss. These two factors lead to the assumption that the unplanned dilution levels will be significantly lower than the 30% to 60% which have been planned for the 320K14 stope. The unplanned dilution levels for the 100A22 stope are estimated at 15% to 45%. this would result in an expected diluted tonnage between 6904 and 8705 tonnes. (more details can be found in Appendix B.)

The stope wall height approximates 14 metres on average and the wall strike length is about 38 metres. Estimating the ELOS with Equation (4) provided in the literature review, results in an estimated ELOS of 0.6 m to 1.8 m. These ELOS estimates exceed the ones of the 320K14 stope, which are 0.6 m and 1.2 m respectively. This is due to the fact that the 100A22 stope wall height is relatively small (only 14 metres, compared to the 20 metres of the 320K14 stope).

5.2. Drilling and Blasting (D&B) Design

Drill and blast parameters are closely related in longhole narrow-vein stoping. Therefore, it has been decided that for drill and blast design, past experiences will be governing. This implies that in general, three holes will be drilled per ring with a maximal burden of 1.8 metres (excluding the opening raise) and the spacing between rings is set at 1.8 metres. Within a ring, the middle hole will be made with an offset of 0.3 metres towards the previous ring to help improve the throw of the blasted rock. The holes will be drilled with a diameter of 76 mm. Figure 5.4. displays the basics of this typical design that has been used in the Jokisivu gold mine. The width of a stope may vary within a stope itself. The thickness of the ore-bearing zone is governing in this. In cases where the ore-bearing zone is wider (> 4,5 m – 5 m), an extra hole will be added in the same manner as described above and depicted in Figure 5.4.

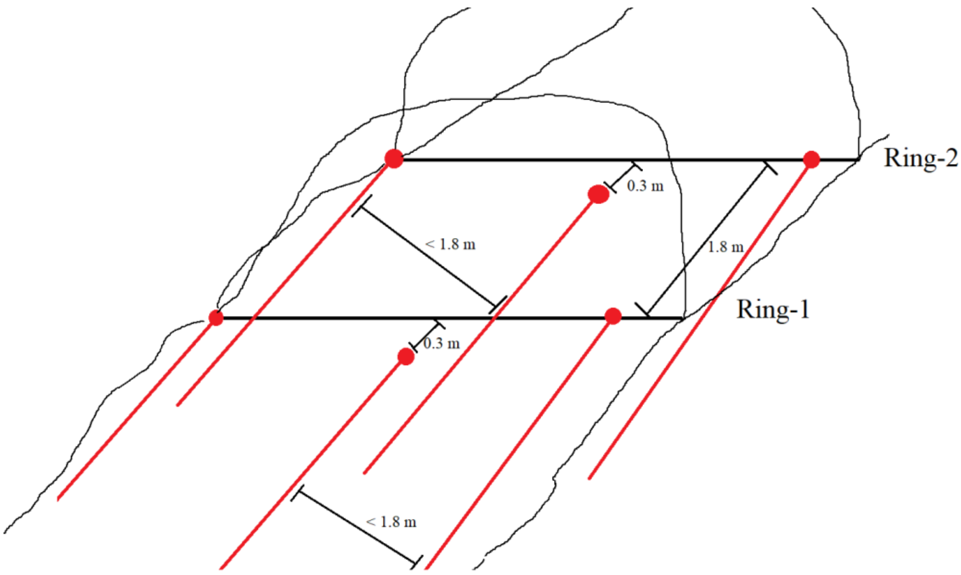

Figure 5.4: Schematic overview of a typical drillhole design in the Jokisivu mine.

Kemiitti 810 pumped emulsion (1.0 kg/dm^3) explosive in combination with Kemix A (1.2 kg/dm^3) emulsion cartridges (⌀ 50 or 40 mm) will be used to break the rock during a blast. The Kemix A emulsion cartridges (Figure 5.5.) are used at the bottom of the hole to prevent the pumped emulsion to leak out. One cartridge is used at the bottom of the hole to stem it and to host a detonator. A second detonator is placed in a cartridge in the middle of the hole, to guarantee the blast to be successful if the first detonator happens to fail or when energy escapes through fractured rock and stops the detonation.

Figure 5.5: Kemix A emulsion cartridges (⌀50mm) used during production blasts.

The aim of this thesis is to not only allocate causes of waste rock dilution, but also to minimize it. Therefore, in addition to designing drill and blast parameters according to past experiences, two additional measures have been undertaken to pursue this goal. The first measure has been designed to improve the condition of the hangingwall of the stope. To achieve this, an additional hole is being drilled in between each ring on the hangingwall side of the stope. This hole is drilled with average length and angle of the hole on the hangingwall in the preceding and following ring. This hole will remain uncharged but will be a guiding hole for the crack-formation and propagation at the hanging wall during the blast (Figure 5.6.).

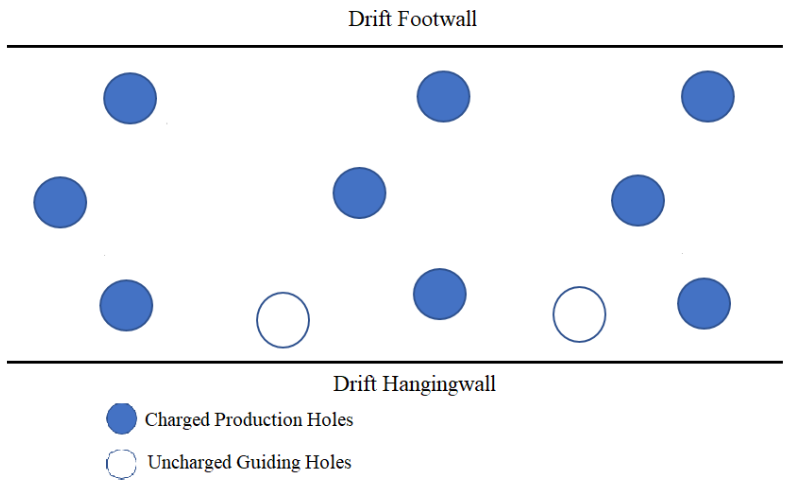

Figure 5.6: Illustration of the use of guiding holes.

The strive of this hole is to attain a better wall control which will result in a smoother hangingwall surface. Doing so, minimizes stope wall damage from the blast, creating a smoother stope wall. In its turn, this ideally results in a higher stope stability, which reduces sloughage and therefore waste rock dilution.

The second parameter that will be added in this research is the use of electronic detonators (Figure 5.7.) rather than the previously used, shock tube detonators. The electronic detonators being used are the Daveytronic® OP detonators. The use of electronic detonators will control the time delay between initiation of rings more precisely than before and therefore ideally, optimize the initiation sequence. Generally, optimizing the initiation sequence has been proven to be an effective measure in reducing stope wall damage and therefore waste rock dilution.

Figure 5.7: Daveytronic® OP electronic detonators used for this research

5.2.1. 320K14 D&B Design

The 320K14 stope consists of 20 rings. These rings have been drilled with 3 holes each, adding up to 60 holes drilled plus the extra holes required for both opening raises. The total metres drilled within this stope sums up to 1227 metres and all holes have been drilled before any blasting has been done.

The average dip of the designed drillholes and therefore the stope is 44°, which is very shallow dipping for the Jokisivu operation. This shallow dip is the cause that the average hole length of the stope is 20.5 metres. When excluding the last 4 shorter rings of the stope, the average hole length even totals 22.4 metres. Designing longer holes gives more room for drillhole deviation and other D&B errors. The shallow dip of the stope also increases the hangingwall area and vertical stresses induced in the stope, making it more prone to the effects of gravity (see section 3.4.). Expected is that the shallow dip of the stope, along with the weak mica-gneiss host rock, contribute to higher levels of dilution than normal.

The initiation sequence in the 320K14 stope has been designed in such a way that first the middle hole, designed with an offset of 0.3 m will detonate, followed by the hangingwall hole and the footwall hole respectively. Believed is that this initiation sequence keeps blast damage of the hangingwall as low as possible. The delay between holes has been set at 77 ms, which is an increase in delay time by 15% compared to the previously used delay of 67 ms. It is believed that increasing the delay between holes gives the rock more freedom to throw into the open space of previously blasted rings and therefore reduce damage to stope walls. The 'back-up' detonator located in the centre of the hole will be programmed at a delay of + 10 ms with respect to the detonator situated at the bottom of the hole. The number of rings per blast has been designed in such a way that there should always be enough available free space for the blasted rock to throw into and can be seen from Appendix E.

5.2.2. 100A22 D&B Design

Stope 100A22 consists of 23 rings. Contrary to the 320K14 stope, the 100A22 stope does contain more than 3 holes per ring in general. This is due to the fact that in some parts of the stope the ore-bearing zone is wider than in the 320K14 stope. The 100A22 has been designed with 108 holes of which 30 uphole and 78 downhole, totalling 1295 metres. The average hole length of the 100A22 stope is 12.0 metres. Excluding the shorter holes (rings 1 to 3 and 22-23), the average hole length is 13.9 metres. The 100A22 holes are considerably shorter on average than the respectively 20.5 metres average hole length and 22 metres average hole length excluding the shorter holes of the 320K14 stope. The shorter length of holes in the 100A22 stope originate from the steep dip of the stope (70° to 80°) and result in the expectation that the 100A22 stope will be less prone to drillhole deviation than the 320K14 stope.

Rings 1 to 7 will be drilled and blasted before the rest of the stope will be drilled. This is because after the opening raise (rings 5-6) and ring 7 have been drilled and blasted, the charging rig needs to access the 80LpA22 level to charge rings 1 to 4, behind the raise. If the rest of the stope has been pre-drilled, the charging rig cannot manoeuvre inside the drift without driving on top of the polyethylene tubes which are installed at the drillhole collars to prevent the holes from getting blocked by small rocks or clogged by sludge (Figure 5.8.).

Figure 5.8: Polyethylene tubes installed at drillhole collars.

Because rings 1 to 7 will be blasted before the rest of the stope is drilled, drillhole deviation measurements are only carried out between rings 8 to 23. Expected is that only measuring rings 8 to 23 should still provide adequate information to get a good overview of the drilling performance in the 100A22 stope.

The initiation sequence of the 100A22 stope is a little bit more complicated than in the 320K14 stope where there are only 3 holes per ring. For the 100A22 stope, the first hole that will detonate is the hole designed with the 0.3 m offset. This hole dictates the direction the stope progresses and is preferably kept in the middle of the ring to be blasted. The hole to be detonated second is the hole closest to the 0.3 m offset hole, followed by the hole on the other side of the 0.3 m offset hole. This pattern repeats until the entire ring has been blasted and repeats for the rings following up. Similar to the 320K14 stope, the delay times set for the electronic detonators are 77 ms between holes and 10 ms for the ‘back-up’ detonator, situated in the centre of the hole and the blasting schedule can be found from Appendix E.

5.3. Rock Support Design

The third and final factor which will be considered during this thesis, is the influence of rock support on waste rock dilution. The extent and methods of rock support will be predominantly dependent on local in-situ rock conditions and drift and stope geometry. Since the two proposed research stopes are situated in different deposits, at different depth and have different geometries, the support plan for these stopes will be different.

5.3.1. 320K14 Rock Support Design

As can be seen from Figure 5.9., the 320K14 stope has no parallel drift from which cablebolts can be drilled, making it more difficult to design an efficient and effective support plan. To achieve at least a certain level of support for the stope hangingwall, the point anchor approach (Figure 3.3c.) has been applied for the cablebolting design. The cablebolt fans in the upper and lower drift of the 320K14 stope have been designed with a spacing of 2.0 metres. Care should be taken while drilling the stope, to not intercept cablebolts at the lower level. The upper level (300Lp3) of the stope has been bolted rather densely and wire-mesh has been installed (Figure 5.10.). The installation of wire-mesh has been done to limit sloughage of small to medium sized rocks from the upper drift walls and roof, to prevent them from collapsing into the stope. Expected is that especially in the mica-gneiss section of the stope the wire-mesh could have a positive effect, as the many cracks and joints in the mica-gneiss host rock will cause the rock mass to fracture into small to medium sized fragments.

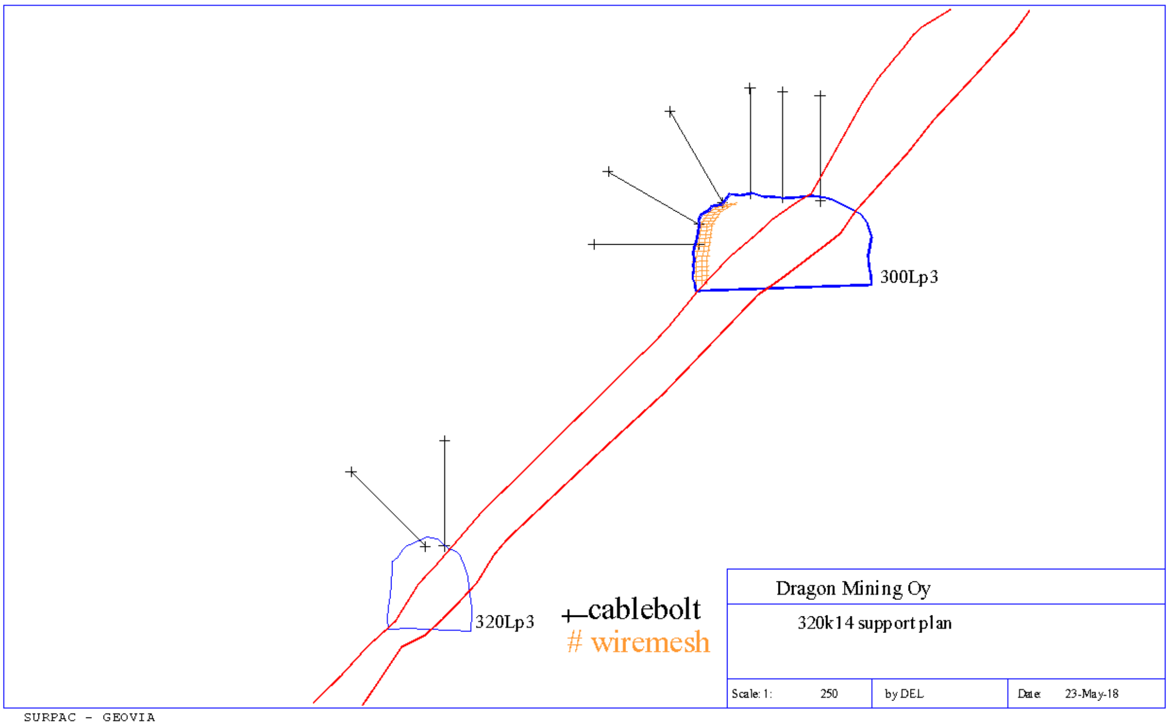

Figure 5.9: 320K14 Stope support cross section.

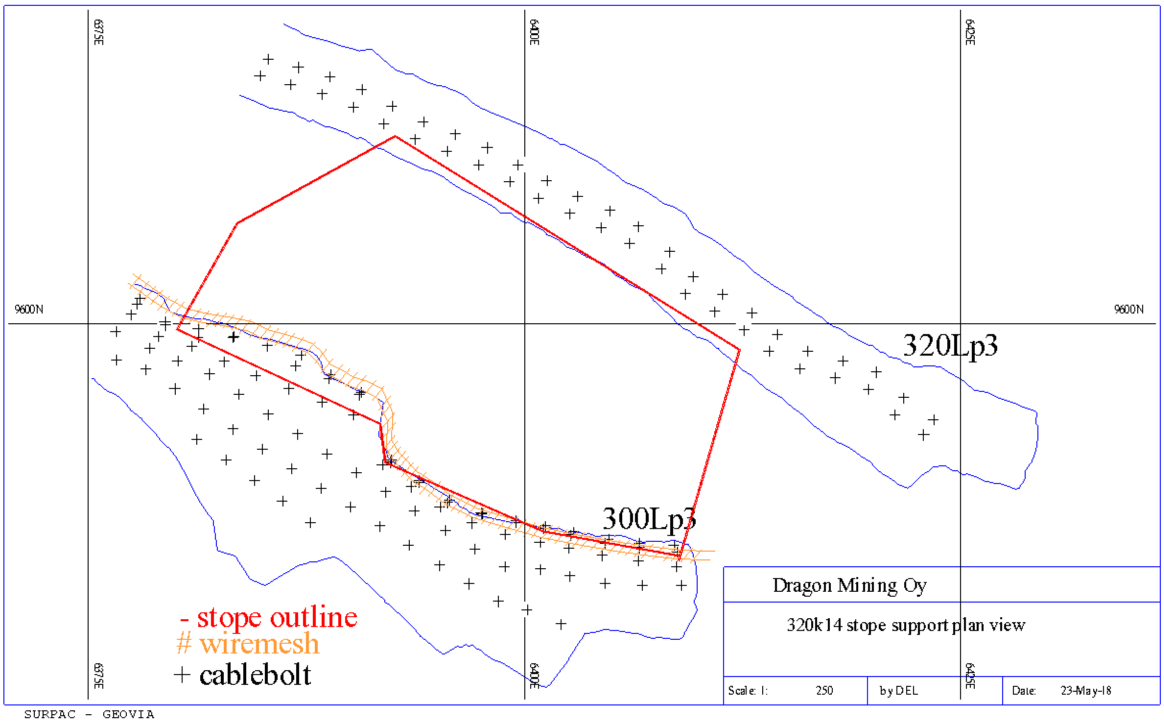

Figure 5.10: 320K14 Stope support plan view.

5.3.2. 100A22 Rock Support Design

Contrary to the 320K14 stope, the 100A22 stope has a parallel drift situated next to the lower level of the stope, separated by a pillar. This parallel drift offers the convenience to drill cablebolt support perpendicular to the dip of the stope to provide anchorage and maximal support to the rock mass in the stope hangingwall. The pillar which separates the two drifts is supported by wire-mesh to mitigate the ravelling to some extent. In addition to the two bolts per fan that are drilled in the parallel drift, there is another cablebolt installed in the ore drift, to provide support to the lower part of the stope hangingwall. The cablebolt fans in the 100A22 stope are designed with a spacing of 1.8 metres. This spacing is equal to the spacing of the drillholes of the DL 421-7C production rig. This will make it easier for the operators to not intercept the installed cablebolts while drilling the stope. Figures 5.11. and 5.12. give an overview of the designed and installed rock support in the 100A22 stope.

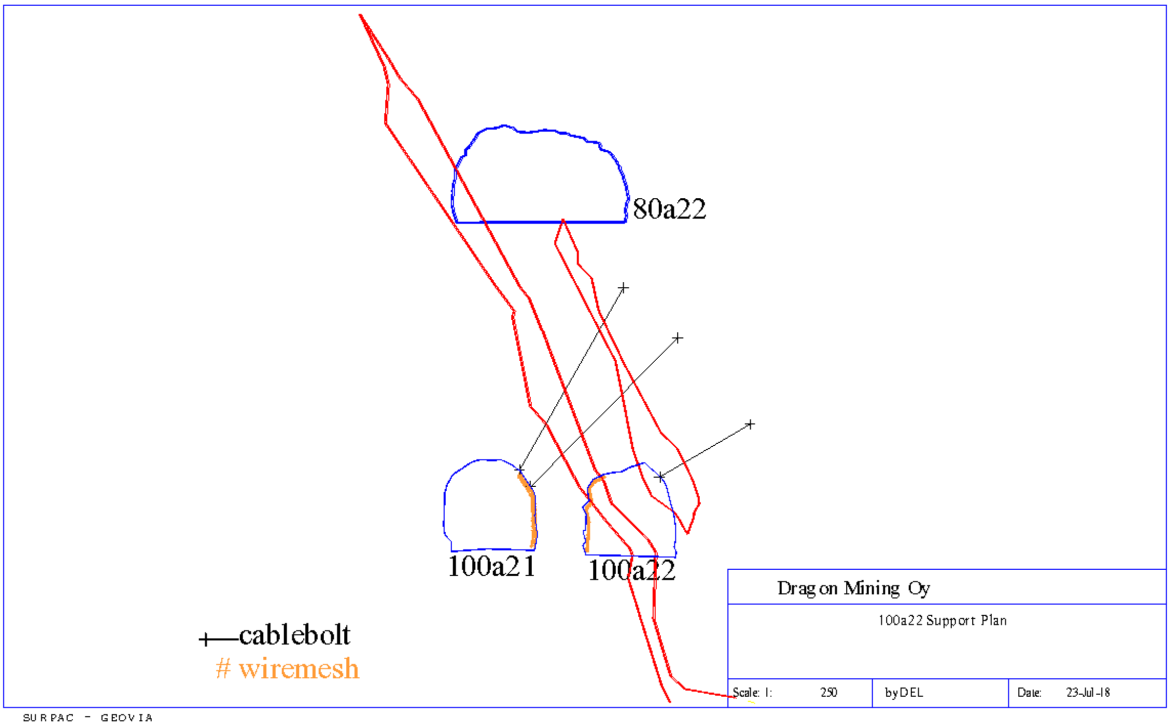

Figure 5.11: 100A22 Stope support cross section.

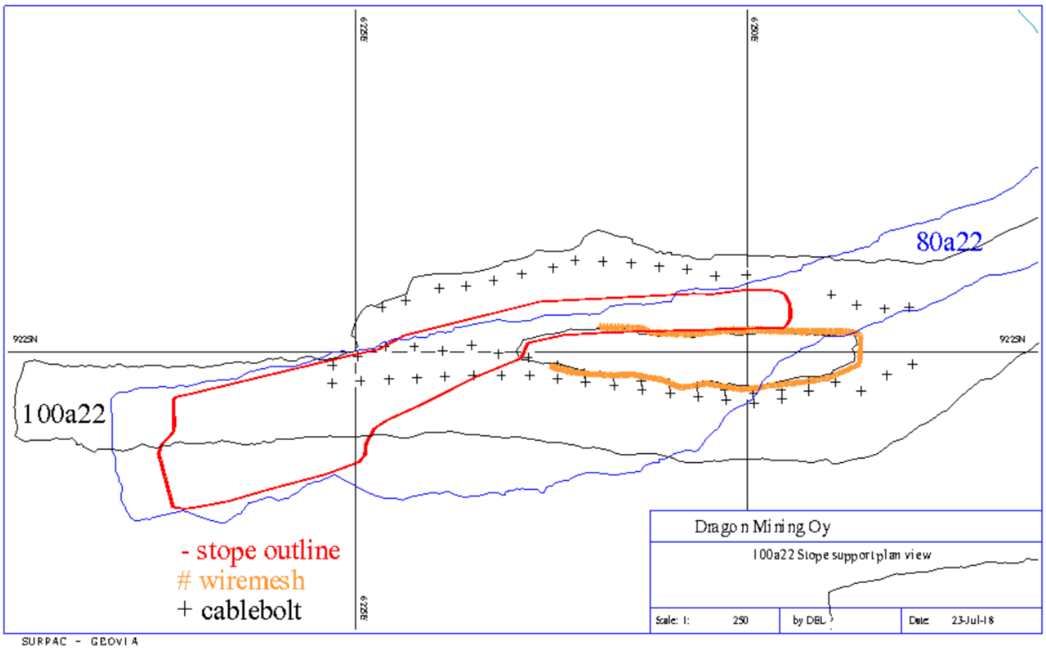

Figure 5.12: 100A22 Stope support plan view.

5.4. Stope Performance

After the stopes have been designed, the support is in place and the drillholes have been drilled and charged, the production will commence. During production as well as after production has ended, the stope performance will be continuously monitored.

5.4.1. Visual Inspection

The first and most evident way of checking whether the stope performance is satisfactory and up to expectations, is to carry out a visual inspection. During visual inspections on the performance of a stope, attention is given to certain factors.

The first thing that will be checked is whether the stope support is still in place. Good indicators of failed support are the absence of or damage to cablebolts, base plates and wire-mesh. After that, stope hangingwall and footwall will be inspected for dilution. Expected is that most dilution will originate from the hangingwall. Stope dimensions such as height, width and deviations in hangingwall and footwall will be measured with a laser range meter, so that they can be compared with the designed stope outline. In addition to that, attention will be given to any structures that could subsequently pose a threat to the condition of the stope.

5.4.2. Tonnage Report

The contractor which is operating at the Jokisivu mine keeps a report on how much tonnes have been mucked and hauled from the stope. Comparing these reported tonnages with the tonnage that has been designed to be mined from the stope is another way of checking stope performance. Doing so is an easy indicator whether the stope has suffered from dilution or ore loss. If the contractor’s tonnage reports indicate larger values than the designed tonnages, the stope suffers from dilution. If it indicates lower values, the stope is dealing with ore loss.

However, there are two major drawbacks to this method. The first one being that ore loss compensates for dilution in the contractor’s tonnage reports. Both ore loss and dilution are undesired as they will negatively affect the resulted grade of the ore. Therefore, sometimes these tonnage reports could be misleading. The second drawback is the fact that tonnage reports allocate the origin of dilution or ore loss rather poorly as blasts will be performed a couple of rings at the time. This means that whenever comparing the reported tonnages to the designed tonnages, one can only give an educated guess about the location of dilution and ore loss.

5.4.3. Cavity Monitoring System (CMS)

The method that solves the two problems stated in the previous section, is the utilization of a cavity monitoring system (CMS). The CMS is based on reflectorless laser technology. The CMS uses this technology to delineate the contour of an underground excavation or cavity. The CMS technology was developed by the Noranda Technology Centre and Optech Systems (Miller et al, 1992). The instrument scans the cavity with a laser scanning unit and converts this information into a three-dimensional mesh image which can be imported to be analysed with Surpac, the programme which is also used for designing the stope.

Overlaying the CMS image with the designed stope shows exactly where dilution and ore loss originated from or whether a stope has been undercut or overcut. On top of that, a tonnage report can also be extracted from Surpac for the resulted CMS stope outline. This does not only provide the possibility of comparing the designed and resulted tonnages, but also offers a tool to check the contractor’s performance.

The scans performed with a CMS ideally should be done when the stope has been mucked out, so that the full cavity can be scanned. Leftover blasted ore in the lower drift result in the CMS not being able to fully scan the cavity. This might pose problems with reliability in the tonnage estimates and comparison afterwards. Therefore, the CMS scans will only be carried out after the blasted material has been completely mucked out of the stope. Scanning more often requires the drift to be mucked out continuously, resulting in less efficient production. Therefore, the number of stope scans will be scheduled to obtain an accurate model of the stope, while at the same time avoiding hampering the production rate.

5.5. Drilling and Blasting Performance

Throughout the process of stoping, the drilling and blasting performance will be monitored. As portrayed in the literature review, the factors in drilling and blasting that influence dilution are predominantly operational parameters. Therefore, the focus of the research on drilling and blasting performance is to draw conclusions from the review of these operational parameters. In addition to that, the research methods set up for the stope performance review are utilized to substantiate drawn conclusions.

5.5.1. Visual Inspection

Visually inspecting the stope after a blast is also the first way of reviewing drilling and blasting performance. Just like with checking stope performance, attention is given to the presence and condition of stope support, blast damage, dilution on hangingwall and footwall from sloughage and any present structures that could pose a threat to the condition of the stope.

Focussing more specifically on drilling and blasting performance, the fragmentation of the ore inside the stope is being inspected. A successful blast is characterized by a uniform fragmentation size. Another indicator of a satisfactory blast is the (partial) visibility of drillholes on the stope walls. Drillholes are designed such that they indicate the stope boundary. If holes are visible, this means that the stope boundary has been closely matched.

5.5.2. Tonnage Report

Just like with monitoring stope performance, the contractor's tonnage reports are solid tools for checking the drilling and blasting performance. They offer insights on whether stopes have been undercut or overcut, and whether ore loss or dilution has occurred. Having a tonnage report closely matching the designed stope tonnage is an indicator for a successful drilling and blasting design.

5.5.3. Cavity Monitoring System (CMS)

Since the CMS is being used to check stope performance, the data can also be used to check the blasting performance. In addition to the contractor's tonnage report, a 3D model obtained from a CMS scan can be useful to check the level of wall control. This can be done by checking the smoothness of the stope walls obtained from the 3D stope model and comparing it with the designed stope outline. Smooth stope walls indicate proper wall control and therefore a decent drilling and blasting performance.

5.5.4. Drillhole Deviation Measurements

Drillholes have been designed to blast the stope as optimal as possible. Deviating from this design may result in unplanned dilution and ore loss. Therefore, drillhole deviation measurements will be conducted. The drillhole deviation measurements will be carried out by an expert from OY FORCIT AB, a Finnish explosives manufacturer and mining consultant. OY FORCIT AB will provide a dataset with which will be worked.

5.5.5. Vibration Measurements

An additional measure of checking whether a blast has been performed successfully is the monitoring of the vibrations induced by the blasting. Two tri-axial geophones have been installed per stope to continuously monitor the production blasts of the designed stopes. The geophones will measure the four vibration parameters, being:

- Peak particle velocity (PPV – mm/s);
- Acceleration (m/s^2);
- Displacement (mm).

- Frequency (Hz)

Analysing these parameters is believed to be helpful in determining whether a blast has been successful or not. The data measured by the geophones will be stored in a monitor and when the monitoring device is taken to the surface, data is sent to the general server. This server can be accessed through the internet. From the server, various reports can be requested, including a waveform of any blast. These waveforms plot the peak particle velocity against time for all three axes (longitudinal, transverse and vertical).

The main application of these waveforms is to interpret whether all holes have detonated correctly. Each hole that detonates correctly provides a 'peak' value in the waveform graph. The amount of 'peaks' should correspond to the number of holes that have been charged prior to the blast. If these values correspond, then all holes have detonated correctly.

For the 320K14 stope, it has been decided that one geophone (JOK 2), will be situated in the 300Lp3 drift, around 50 metres away from the stope. The second geophone (JOK 1) will be installed one level above, some 70 metres away, in the 285Lp3 drift (Figure 5.13.).

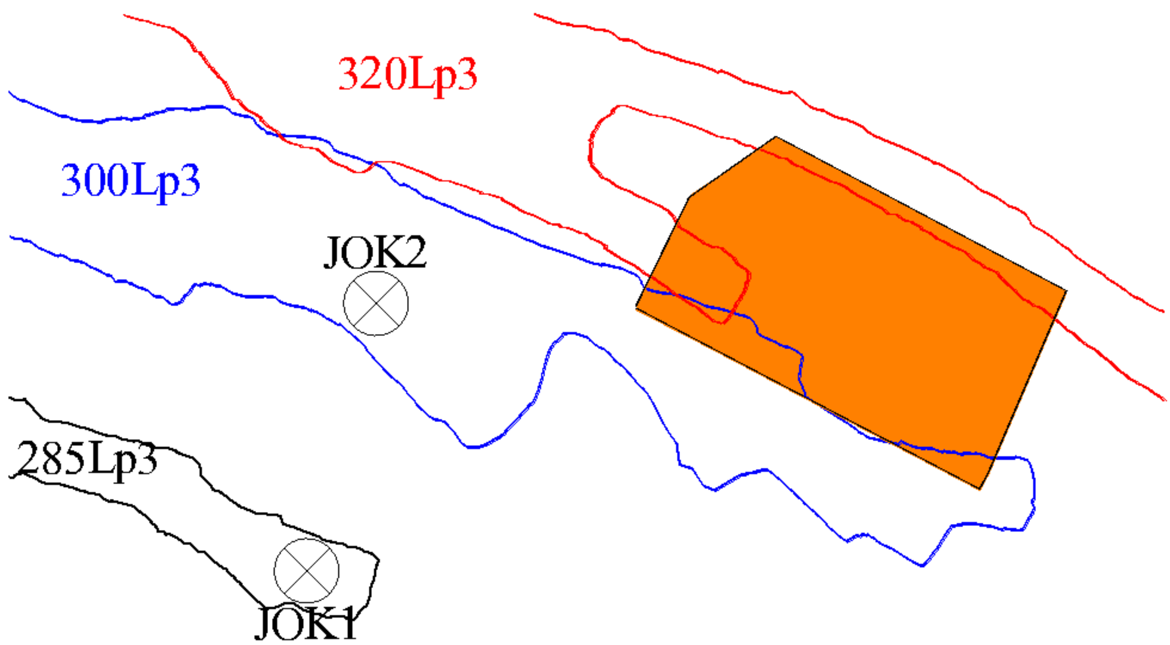

Figure 5.13: Locations of vibration monitoring systems for the 320K14 stope.

For the 100A22 stope, the first geophone (JOK 3) has been installed right above (15 metres) the stope in the 65LpA22 drift. The second geophone (JOK 4) is installed on the 80 level, 50 metres away from the 100A22 stope, just around the corner of the 80LpA22 drift. (Figure 5.14.)

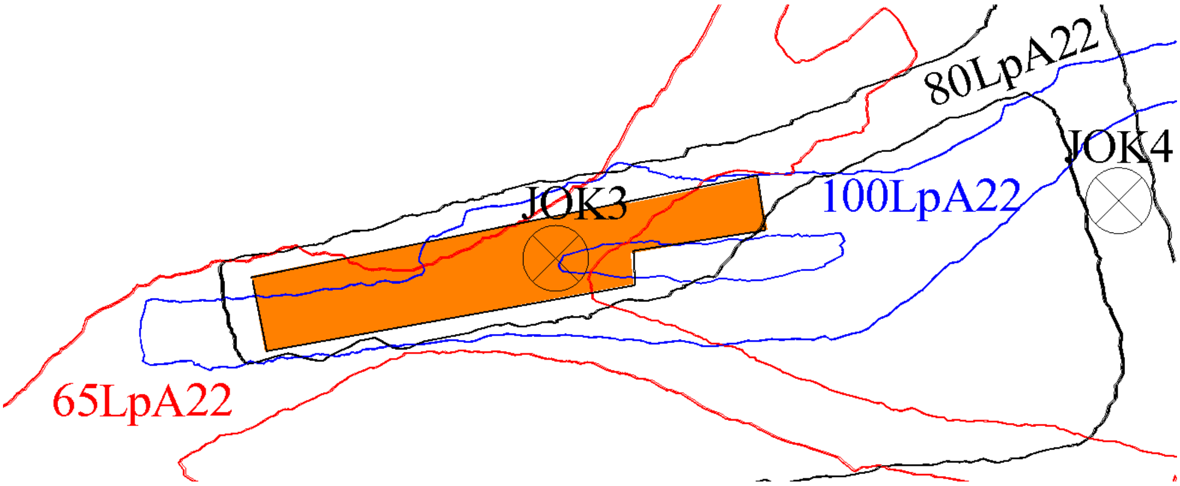

Figure 5.14: Locations of vibration monitoring systems for the 100A22 stope.

6. Results

The results of researching both the 320K14 stope as well as the 100A22 stope will be reviewed individually. A discussion on and comparison of the results obtained can be found in section 7. Appendix G. contains additional images to support the understanding of the described phenomena in this section.

6.1. 320K14 Stope and Rock Support Performance

As reported in section 5.1.1., the 320K14 stope was partly hosted in weak mica-gneiss and expected was that the 320K14 stope would suffer from relatively elevated levels of unplanned dilution. On top of that, these poor rock conditions urged the need to leave a half-pillar inside the stope for extra support.

During the production, the reality exceeded these expectations. At the 320 level, perpendicular carbonate fields, joints and bedding planes were visible between rings 2 to 5, creating massive weakness zones. Expected was that these weakness zones would continue throughout the rest of the hangingwall and cause, to some extent, partial caving on the hangingwall side of the stope. As can be seen in Figure 6.1., these weakness zones indeed caused the collapse of an approximately 1 metre thick slab throughout the span of the entire hangingwall from rings 1 to 9. The cablebolts that have been designed on the hangingwall side in the 300Lp3 drift (Figures 5.9. and 5.10.) prevented the caving on the hangingwall to extend even further than displayed in Figure 6.1.

This hangingwall collapse resulted in the decision to redesign the planned half-pillar at rings 10-12 into a full pillar, leaving behind 809 tonnes of ore instead of the previously designed 466 tonnes. This resulted in an extra ore loss of 343 tonnes or 7.5 %. Behind the pillar at rings 12-20, where the mica-gneiss hosted section of the stope ended, no more complications arose related to the stope stability and stope support.

The mesh that had been installed on the hangingwall side of the 300Lp3 drift, was severely damaged by the numerous blasts and was deemed useless, as it did not provide any additional support.

Figure 6.1: 320K14 Exposed surface (black) after hangingwall collapse rings 1-7.

6.2. 320K14 Drilling Performance

Checking the drilling performance of the 320K14 stope was done in three ways. The first method, was the visual inspection of the drillholes. Checked before blasting was whether all drillholes that initially were planned have also been drilled. On top of that, directly after the blasting, some drillholes were still partially visible in the hangingwall of the stope, which indicate that locally, the designed stope boundary had been closely matched.

The second manner of checking drilling performance were the drillhole deviation measurements. The data obtained from these measurements were, together with the drillhole collar points measured by the mine surveyor and the initially designed drillholes, compiled into a model. The results are depicted in Figures 6.2. and 6.3., where the numbers indicate the ring number of the 320K14 stope.

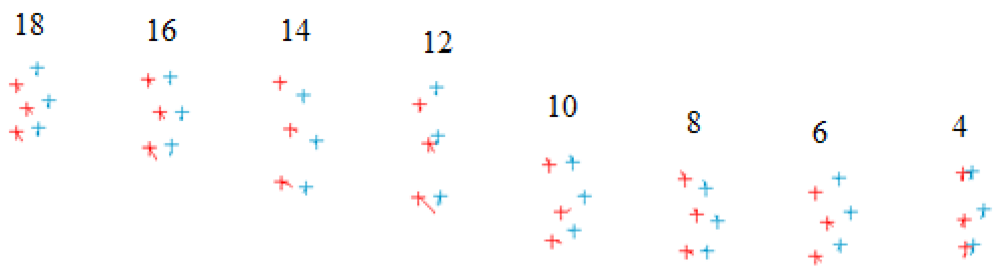

Figure 6.2: 320K14 Designed drillhole collar location (blue) vs. actual drillhole collar location (red).

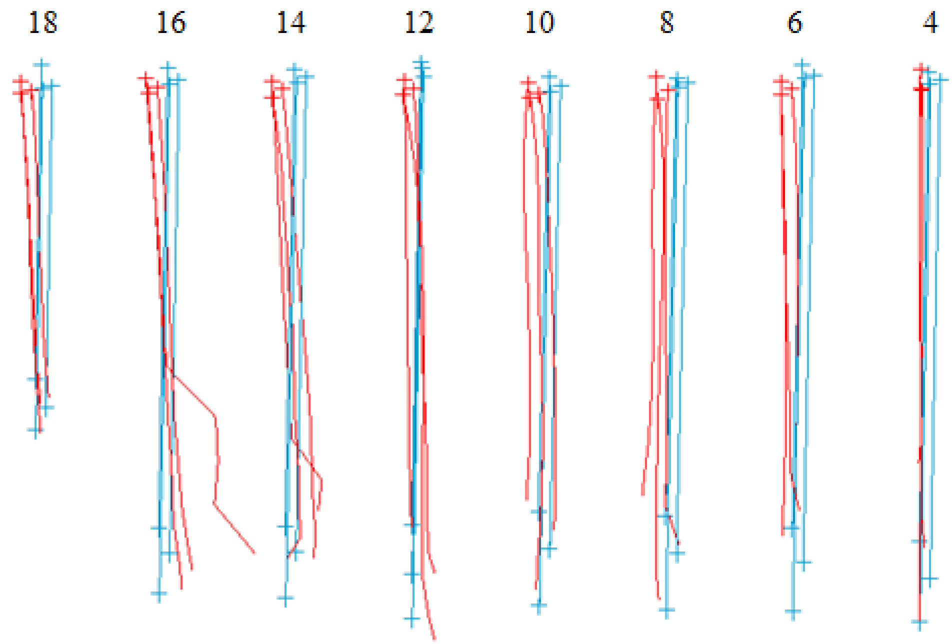

Figure 6.3: 320K14 Designed drillhole trajectories (blue) vs. actual drillhole trajectories (red).

From this model, hole collar and toe differences can be measured together with the actual hole length and a clear image of the drillhole deviation can be generated. From Figures 6.2. and 6.3. it becomes evident that drillhole deviation occurs in all three forms described in section 3.3.2. It can be seen from Figure 6.3. that some measurements provide unrealistic drillhole trajectories (one hole in ring 14 and one hole in ring 16). This data has been excluded from drillhole deviation calculations. Table 6.1. displays both the minimum and maximum drillhole toe and collar deviation as well as the average values. The complete drillhole deviation database can be seen in Appendix C.

Table 6.1: 320K14 Drillhole collar and toe deviation.

	Drillhole Collar Deviation (m)	Drillhole Toe Deviation (m)
Average	0.66	0.52
Minimum	0.22	0.07
Maximum	1.23	1.23

The third and final way of checking drilling performance within this thesis, is the calculation of drillhole deviation in percentages according to Equation (5), discussed in section 3.3.2. In order to do so, the true length of the planned hole and the drillhole toe deviation has to be known. These values can be extracted from the Surpac model and are also included in Appendix C. Table 6.2. displays the minimum, maximum and average drillhole deviation in percentages, according to Equation (5).

Table 6.2: 320K14 Drillhole deviation calculated according to Equation (5).

	Drillhole Deviation (%)
Average	2.5
Minimum	0.3
Maximum	5.1

6.3. 320K14 Blasting Performance

During the production of the 320K14 stope, the majority of predicament that emerged was related to the blasting of the 320K14 stope. Numerous unsuccessful blasts were performed in the 320K14 stope. The most common fail during the production blasts of the 320K14 stope was that a rock column of approximately 1-2 metres thick did not break, leaving a 'lid' on top of the stope. This phenomenon occurred four times and is displayed in Figure 6.4. These lids were blasted with shock tube detonators during so called 'repair' blasts.

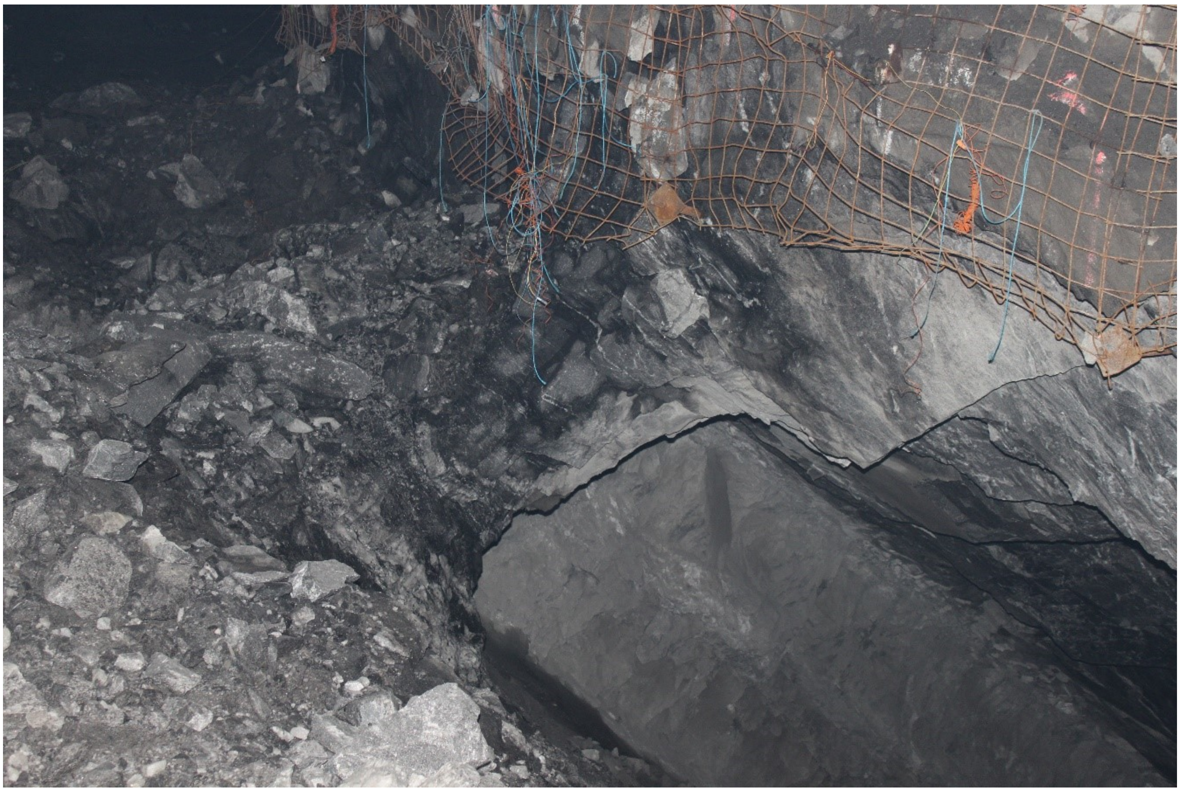

Figure 6.4: Unsuccesfull blast resulting in a 1-2 metres thick 'lid' on top of a stope (rings 7-9).

Another anomaly that occurred was that the blasting of the second opening raise did not go according to plan. One of the holes on the footwall side did not blast the top 10 metre column of the opening raise.

The final complication observed related to the drilling and blasting performance of the 320K14 stope was the failed detonation of blastholes altogether. This happened only once, for the footwall hole in ring 15.

The guiding holes on the hangingwall side of the 320K14 stope, described in section 5.2. did not show any direct results. However, its potential effects will be discussed in section 7.

For the production of the 320K14 stope a total of 2243 kg of explosives has been used to blast a rock mass of 3623 t. The resulting powder factor is 0.60 kg/t. The vibration monitors JOK 1, situated at the 285 level and JOK 2, situated at the 300 level recorded all the blasts successfully. The resulting waveforms, along with their responsive values have been attached in Appendix D. These waveforms do back-up the observations made from inspecting the 320K14 blasting performance and will be discussed more elaborately in section 7.

The initial delay times for the electronic detonators used to blast the 320K14 stope were designed at 77 ms between holes and 10 ms for the back-up detonator inside the hole. The first electronic detonator blast (rings 4-5) was unsuccessful to an extent that a lid stayed intact on top of the stope. This made the contractor decide to change delay times to 50 ms between holes and 5 ms for the back-up detonator inside the hole. Yet again, with these smaller delay times, another lid was left on top of the stope at rings 7-9. Believed by the contractor was that blasting with smaller delay times would give a larger impact and more power for the rock mass to move.

After analysing the waveforms produced by the vibration monitors JOK 1 and JOK 2, it was concluded that there was nothing out of the ordinary with the detonation of the holes and thus decided was that an even further reduction in delay times could do no harm. The delay times were set to 25 ms delay between holes, leaving the delay times for the back-up detonator unchanged. Nevertheless, the phenomenon of having a lid on top of the stope after a blast occurred two more times with these delay settings. The peculiarity of the lids on top of the stope after blasting will be covered more elaborately in section 7.

6.4. 320K14 Dilution

Estimated before production started was that unplanned dilution levels for the 320K14 stope would end up between 30 % and 60 % with an ELOS between 0.6 m and 1.2 m due to bad host rock conditions. The decision of leaving a full pillar behind rather than a half-pillar positively affected dilution levels for the 320K14 stope. The tonnage report provided by the contractor stated that in total 4485 tonnes have been mucked from the 320K14 stope, which would mean a dilution of 24 % and an ELOS of 0.44 m. The stope scans that have been carried out with the CMS provided a model (Figures 6.5. - 6.7.) of the empty stope. This model was estimated to contain 4360 tonnes of material, which in turn represents a dilution of 20 % and an ELOS of 0.38 m.

Both the dilution figures from the contractor's tonnage report and from the CMS model provide dilution levels just below the average dilution level for stopes in the Kujankallio main zone, in which the stope is situated. The average dilution level for stopes in the Kujankallio main zone is 25 % and has been calculated based on 19 previously mined stopes.

As can be seen from Figures 6.5. - 6.7. and Appendix F., the CMS model confirms what has been noticed during visual inspections of the 320K14 stope: the majority of the 320K14 stope its dilution originates from the hangingwall side of the stope. However, a thing that was not immediately apparent from visually inspecting the stope is that the footwall side of the stope has some regions in which the CMS model does not closely match the designed stope outline. This indicates that presumably some ore loss can be allocated to the footwall side of the stope. In appendix F., the dilution has been depicted more detailed by comparing the designed stope outline in a cross section, with the resulting stope outline, obtained from scanning the stope. Section 7 elaborates on potential causes for ore loss and dilution.

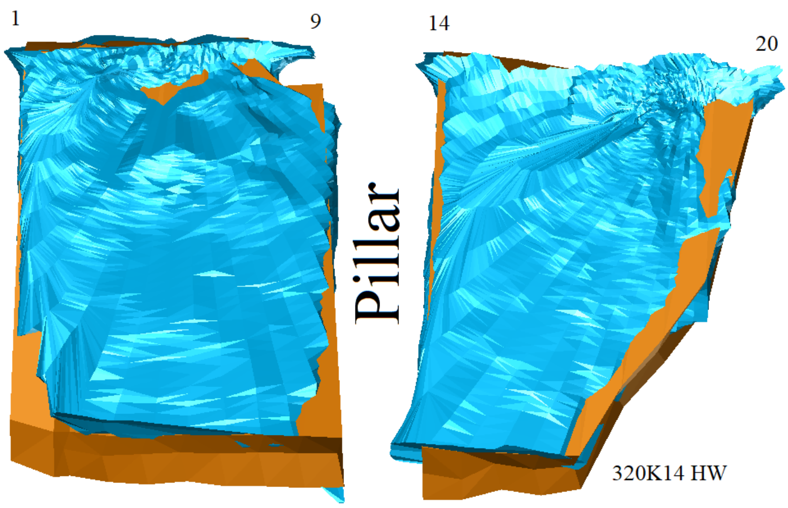

Figure 6.5: Hangingwall of 320K14 designed stope (orange) vs. CMS model (blue).

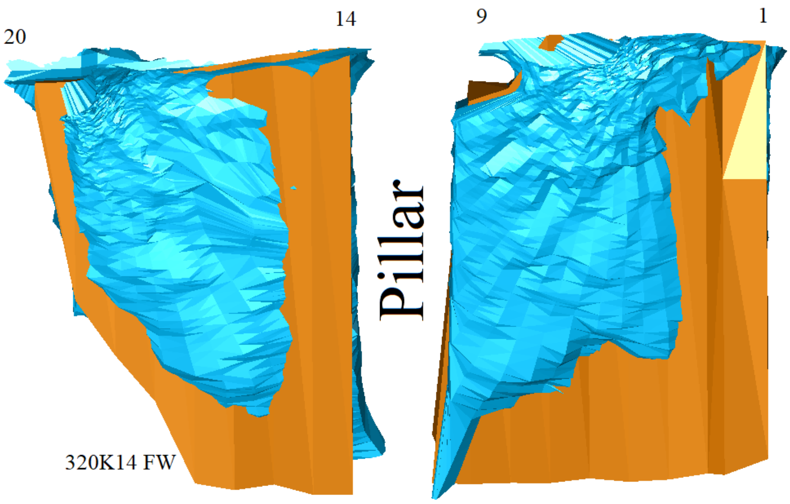

Figure 6.6: Footwall of 320K14 designed stope (orange) vs. CMS model (blue).

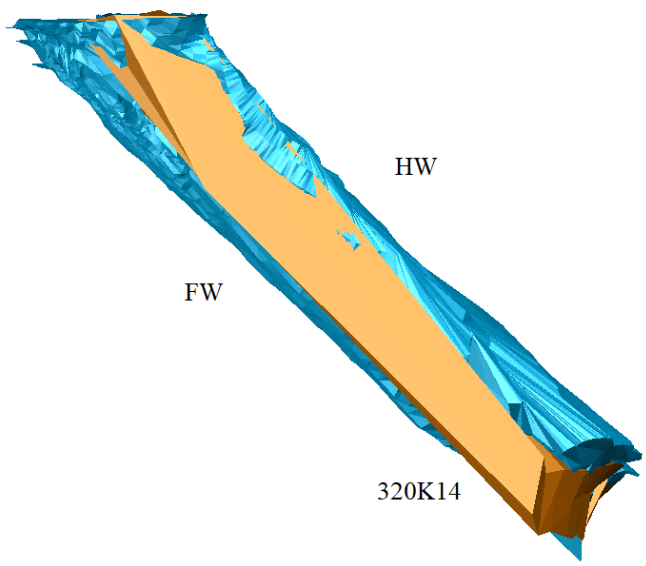

Figure 6.7: 320K14 Stope side view of designed stope (orange) vs. CMS model (blue). Taken from ring 1.

6.5. 100A22 Stope and Rock Support Performance

The production of the 100A22 stope started with some complications. The production of the shortening rings (1-4) behind the opening raise did not meet the desired designed stope outline. After blasting and scanning of the stope, it was noted that 500 t ore loss occurred from rings 1-6. Beforehand, it was known that the design of these rings was a risky one, due to the challenging orebody geometry. However, it was anticipated that this design should have worked. The suggested reasons behind this ore loss has been covered into more detail in section 7.

Before production started, a set of parallel joints and a small fault plane was observed on the hangingwall side within weak pegmatite rock. These structures were expected to continue throughout the full extent of the stope. This joint set became visible on the hangingwall during production, between rings 8 to 18. These joints resulted in a heavily fractured hangingwall and local caving of 1.5-2 metres thick slabs. Figure 6.8. depicts these present structures before production and Figure 6.9. illustrates the caving plane, photographed from the 100 level.

Figure 6.8: 100A22 Hangingwall present fault plane and joint structure before production commenced.

Figure 6.9: 100A22 Caving Hangingwall photographed after stope blast of rings 11-15.

The rock support that has been designed and installed for the 100A22 stope performed exceptionally well. The installed cablebolts and the mesh around the pillar on the 100 level kept the majority of the pillar intact, while allowing the gold mineralization at the pillar contact to be mined succesfully (Figure 6.10.). The cablebolts penetrating the ore-bearing zone also anchored a large part of the jointed rock slabs on the hangingwall side. Together with the steep dip angle, the cablebolts prevented extensive caving of the hangingwall.

Figure 6.10: 100A22 Pillar after the stope has been mined out.

The footwall side of the 100A22 stope showed a faultplane between rings 11-15 which also caused considerable levels of dilution. The stope had been designed in such a way that this fault plane would affect dilution levels as little as possible, as the presence of this fault plane was known before production started. The footwall fault plane can be seen in Appendix G., and its effect on dilution levels can be seen in Appendix F.

6.6. 100A22 Drilling Performance

Visual inspection of the 100A22 stope drilled holes before production showed that there was one hole missing in ring 18. Also, one hole in ring 5 has been drilled with the designed angle of a hole in ring 4. As far as known, during production this resulted in no complications. However, something that did affect production was the fact that the holes of rings 22 and 23 were drilled longer than planned. This error caused some ore material (approximately 200 tonnes) below cut-off grade to dilute the 100A22 stope. Reasons for these errors in drilling performance are miscommunication between drill rig operators and misinterpretation of drilling plans. In Appendix F., the visualization of these longer drillholes in rings 22 and 23 are depicted along with the result these longer holes had on the resulting stope.

Drillhole deviation measurements were carried out for a part of the 100A22 stope. The results are depicted in Figures 6.11. and 6.12., where the numbers indicate the ring number of the 100A22 stope.

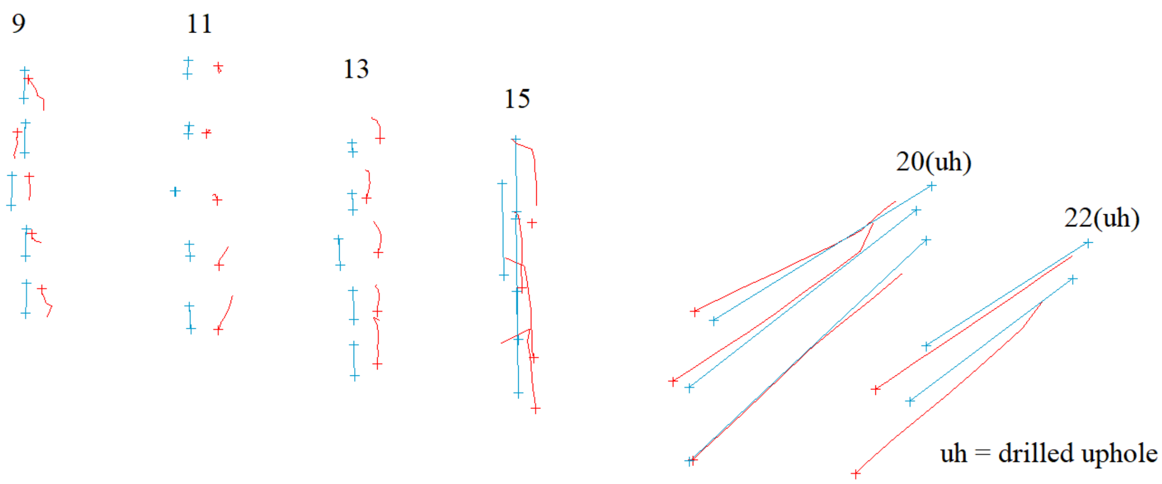

Figure 6.11: 100A22 Designed drillhole collar location (blue) vs. actual drillhole collar location (red).

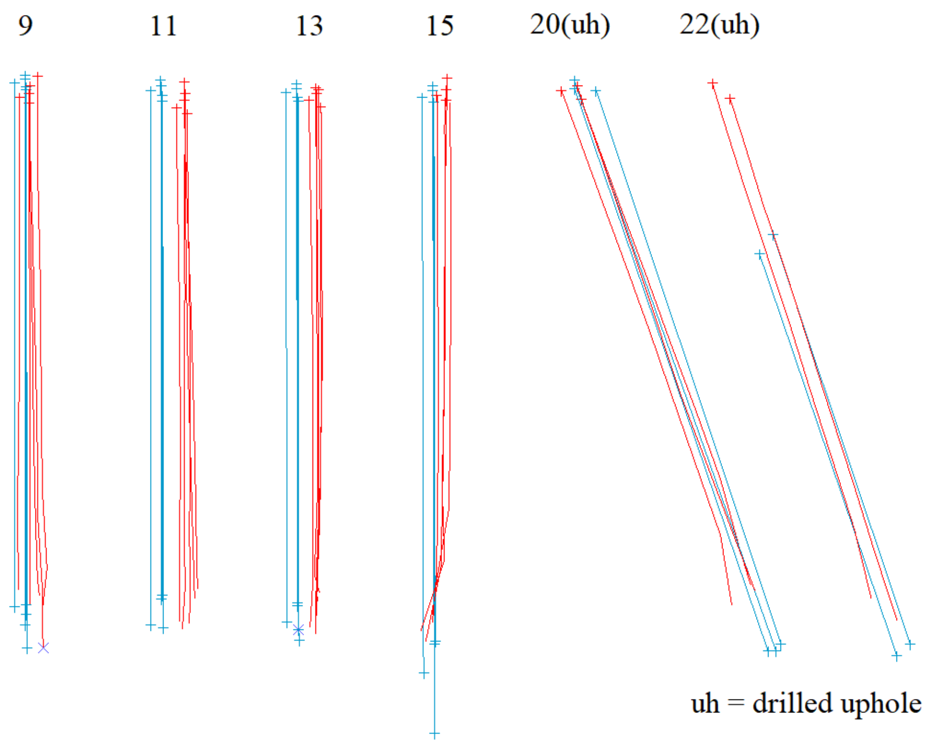

Figure 6.12: 100A22 Designed drillhole trajectories (blue) vs. actual drillhole trajectories (red).

Figures 6.11. and 6.12. show that the starting point of the holes have an offset, the angle of some holes deviates from the design and the drill trajectory changes eccentrically. Table 6.3. displays both the minimum and maximum drillhole toe and collar deviation as well as the average values. The entirety of the drillhole deviation dataset for the 100A22 stope is displayed in Appendix C.

Table 6.3: 100A22 Drillhole collar and toe deviation.

	Drillhole Collar Deviation (m)	Drillhole Toe Deviation (m)
Average	0.47	0.57
Minimum	0.14	0.09
Maximum	0.92	1.11

Table 6.4. displays the minimum, maximum and average drillhole deviation in percentages, according to Equation (5).

Table 6.4: 100A22 Drillhole deviation calculated according to Equation (5).

	Drillhole Deviation (%)
Average	4.1
Minimum	0.6
Maximum	7.4

6.7. 100A22 Blasting Performance

Opposite to the 320K14 stope, the blasting of the 100A22 stope encountered only little complications. The blasting of the opening raise (rings 5-6) and ring 7 was executed. Hereafter, rings 1 to 4, behind the raise were blasted. Because the expansion factor of the rock had not been fully considered during the blast design, the opening (rings 4-7) did not offer enough space to host the blasted rock in. Consequently, this resulted into some rock being thrown in the 80LpA22 drift (Figure 6.13.) as well as parts of the charged column not breaking. This, in combination with a risky design resulted in approximately 500 tonnes ore loss. In Appendix F., the ore loss is visualized by comparing the designed stope outline in a cross section, with the resulting stope outline, obtained from scanning the stope.

Figure 6.13: 80LpA22 Drift after blasting of rings 1-6.

The production of the 100A22 stope required 3067 kg of explosives to blast the designed 6003 t stope. This results in a powder factor of 0.51 kg/t. Vibration monitors JOK 3 and JOK 4 recorded all but one blast successfully. The recorded values and waveforms are displayed in Appendix D. The failed recording at both monitors cannot directly be explained, as batteries were changed, and the recording unit had been turned on.

In general, the waveforms depict successful blasts and confirm conclusions drawn from visual inspections of the 100A22 stope that were carried out after each individual blast. During the blasting of rings 11-15, at the set delay time 771 ms, vibration monitors only recorded minor vibrations (Figure 6.14.). At this set time, the ignition of two holes at the footwall side in ring 13 were programmed.

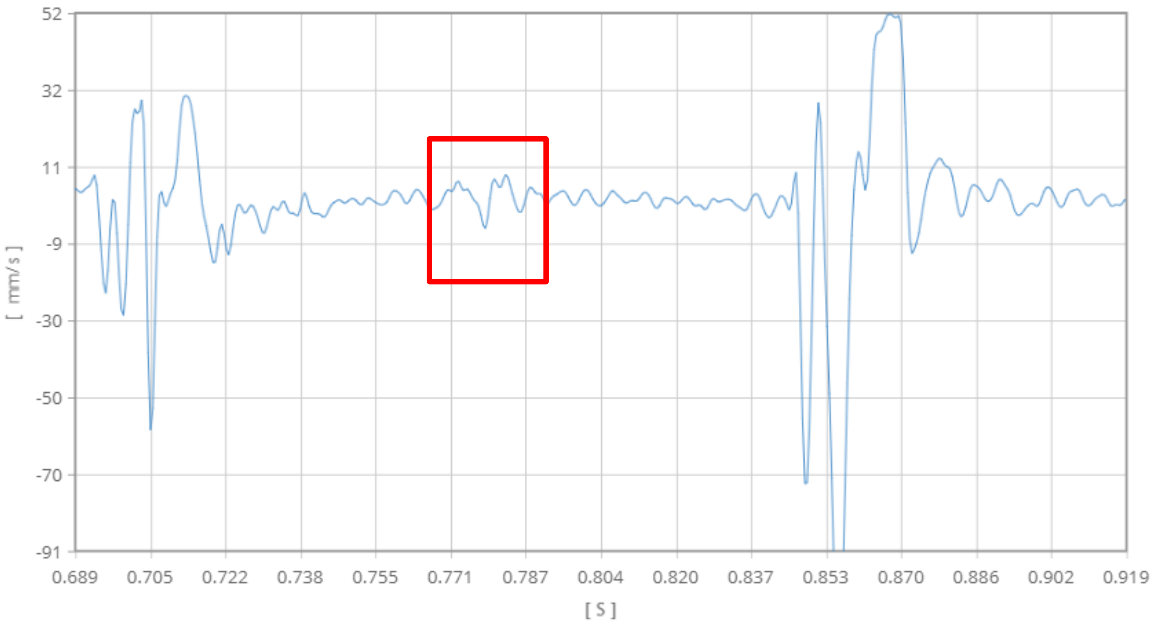

Figure 6.14:Vibration response JOK4 at 771 ms for 100A22 stope blast rings 11-15.

Another blast which showed minor vibrations at a two set time interval is the blast of rings 16-18. As can be seen from figure 6.15., minor vibrations can be detected at -154 ms and -77 ms. These two responses belong to the holes set to detonate at 1 ms and 78 ms.

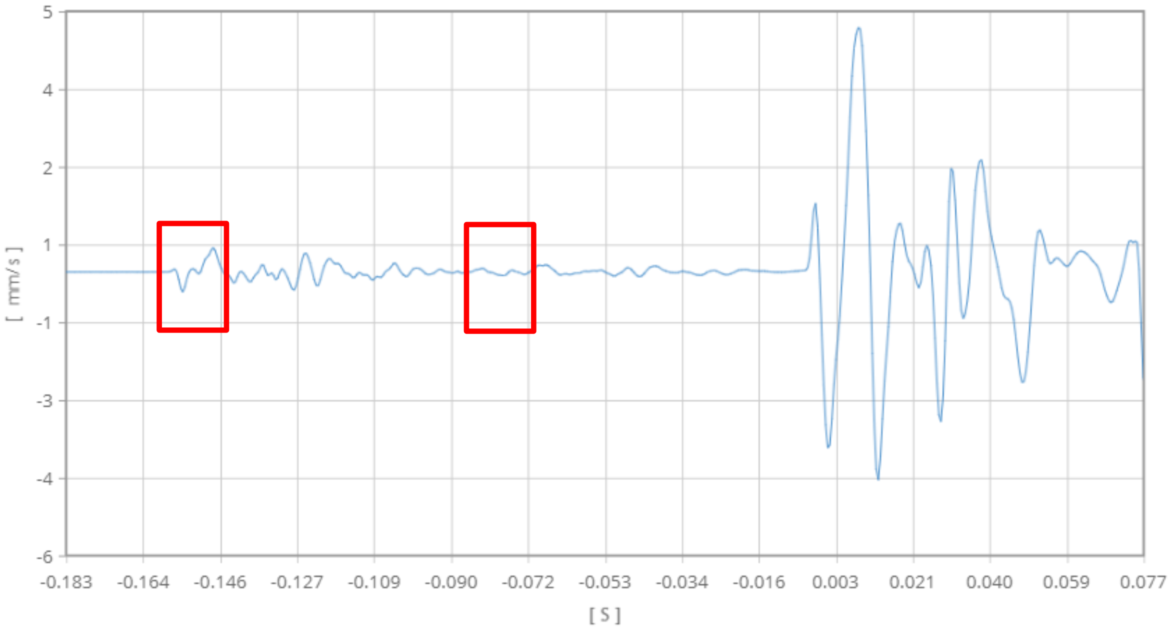

Figure 6.15: Vibration response JOK3 at < 0 ms for 100A22 stope blast rings 16-18.

6.8. 100A22 Dilution

Beforehand, it was estimated that the dilution level of the 100A22 stope would end up between 15 % and 45 % with an ELOS of 0.6 m to 1.8 m. The 500 t ore loss on which has been reported in sections 6.5. to 6.7. has been excluded from the planned stope tonnages, to get a more accurate estimation on the dilution levels of the 100A22 stope.

The contractor's tonnage report showed that in total 7673 tonnes have been mucked from the 100A22 stope, which suggests a dilution of 41 % and an ELOS of 1.49 m. The stope scans that have been carried out with the CMS provided the model, displayed in Figures 6.16. to 6.18. This model was estimated to contain 7065 tonnes of material, which in its turn represents a dilution of 30 % and an ELOS of 1.08 m.

Especially the data obtained from the contractor's mucked tonnage report show elevated levels of dilution and ELOS. The 41 % dilution and 1.49 m ELOS still fall within expectation but are on the high end. The minimum observed dilution for previously mined Arpola stopes is 6 % whilst the maximum value is 60%. The average dilution for stopes situated in the Arpola deposit is 31 %. These values are based on 21 previously mined stopes but cannot offer reliable data for comparison as each location is heavily dependent on local rock conditions and geology.

Figures 6.16. to 6.18. and Appendix F. display that dilution originates from both the hangingwall and the footwall of the stope. These stope scans confirm that indeed the joint set in the hangingwall and fault planes in both hangingwall and footwall caused sloughage of the weak pegmatite rock and caving of 1-2 m thick slabs.

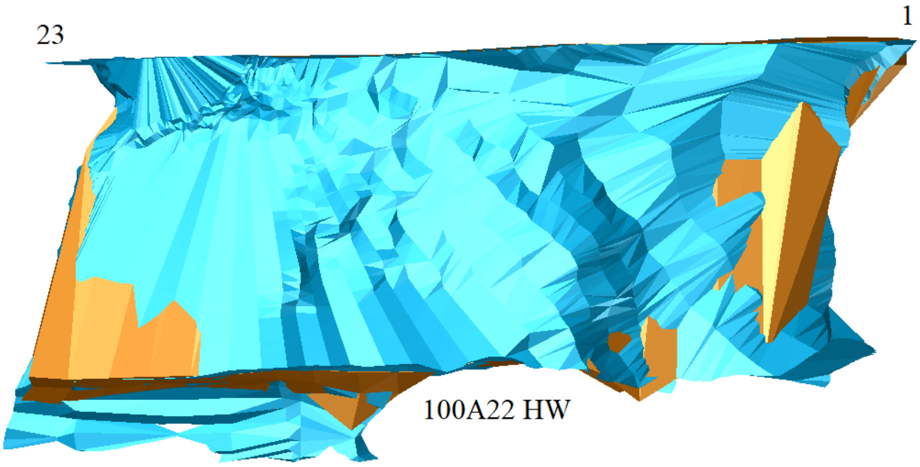

Figure 6.16: Hangingwall of 100A22 designed stope (orange) vs. CMS model (blue).

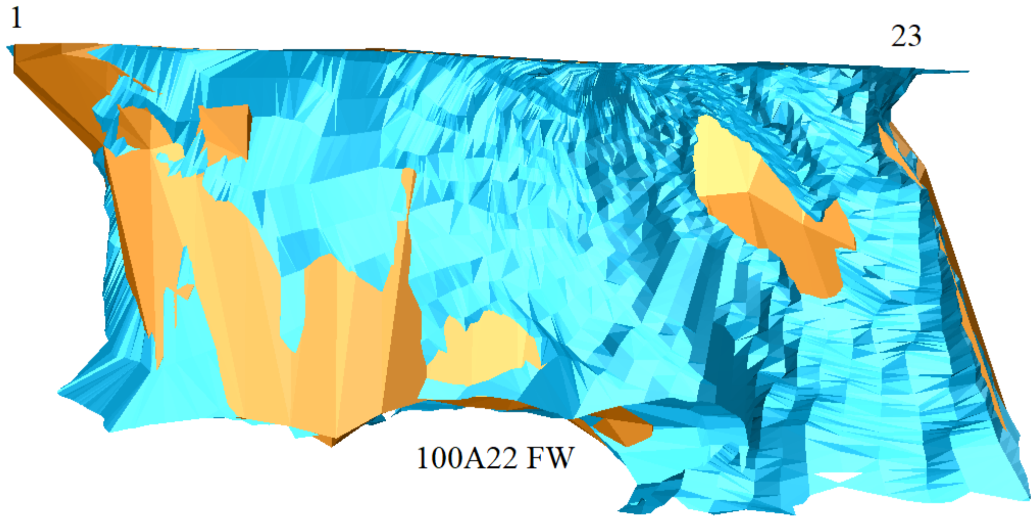

Figure 6.17: Footwall of 100A22 designed stope (orange) vs. CMS model (blue).

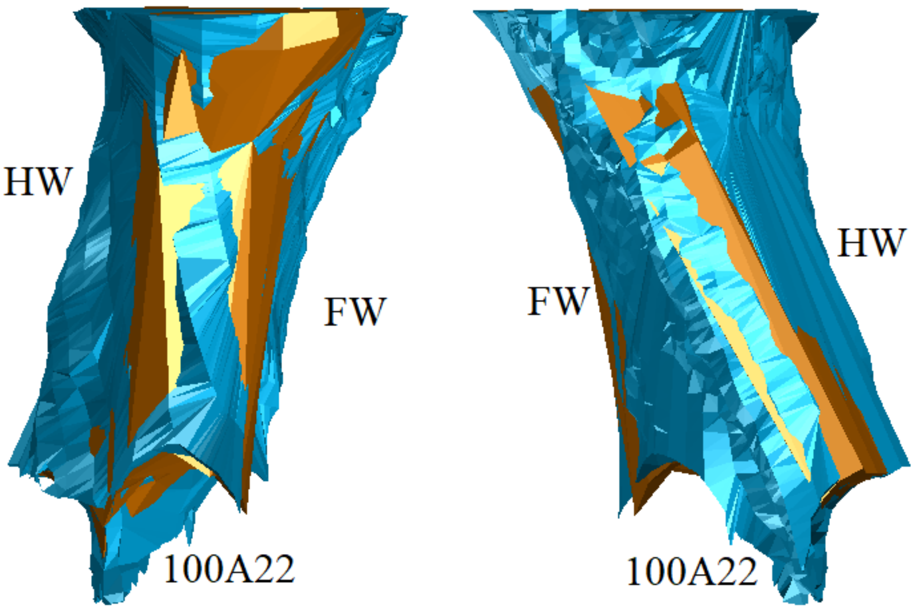

Figure 6.18: 100A22 Stope side view of designed stope (orange) vs. CMS model (blue). From ring 1: left. From ring 23: right.

7. Discussion

In this section the results found in this study are discussed. This is done by comparing the gathered data of each research method with the literature on that topic and the observations made during research. This discussion is made to find possible explanations for observed phenomena and obtained results. The discussion will be grouped into four sections. The first section will dig into stope performance, rock support and what role geology plays in affecting dilution. Secondly, the drilling and blasting performance of this research will be discussed. Finally, the obtained levels of dilution and their estimation methods will be discussed. Along with the discussion on the obtained results, the uncertainties of the used research methods will also be discussed in their corresponding sections.

7.1. Geological and Rock-Mechanical Influences on Stope Performance

A problem that presented itself multiple times during the production of the 320K14 stope was the issue of having 1-2 metres thick lids remaining on top of the stope after blasting. The occurrence of these lids cannot directly be explained. A potential cause for this could have been the lack of energy during blasts to break the rock mass in the top column of the stope. This lack of energy could be explained by the possibility of energy escaping through fractures and the non-charged guiding holes on the hanging wall side of the stope.

Another, more likely explanation for the lack of energy, is the shallow dip angle of the 320K14 stope. Because the stope is shallow dipping (40° to 50°) and the drillhole spacing has been designed parallelly, the holes do intersect the drifts further apart, as can be seen in Figure 7.1. This results in leaving a larger rock mass to be blasted at the top of the stope. A factor which might have contributed to this phenomenon is the pressure, caused by induced stress that the hangingwall exerts on the stope and the footwall which results in clamping of the top column of the stope.

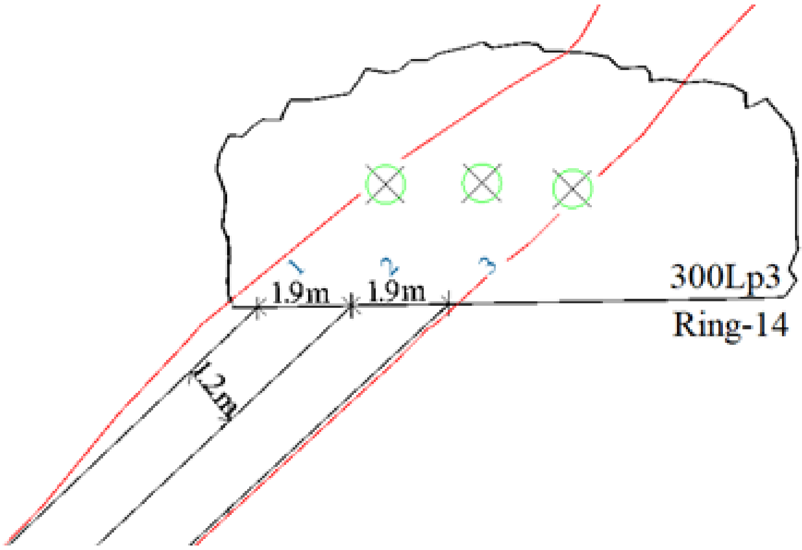

Figure 7.1: 320K14 Shallow dip and hole spacing illustration.

The 320K14 shallow stope dip did not only affect the blasting results but also the levels of dilution, as in general shallow dipping stopes suffer more from the effects of gravity than steeply dipping stopes. The stope dip did not negatively affect dilution levels of the 100A22 stope, as it had a relatively steep dip of 70° to 80°.

As Lappalainen & Pitkajarvi (1996) stated, the geological conditions of an orebody and therefore a stope can account for up to one third of the total dilution, depending on the complexity of the geology. Both the 320K14 and the 100A22 stope were dealing with geological and rock mechanical issues. The 320K14 stope was partly hosted in weak mica-gneiss, which can be confirmed from the stope scans, accounted for a large part of the dilution originating from the hangingwall side. The 100A22 stope suffered from high levels of dilution, mainly due to numerous amounts of joints and two fault planes. Regardless of other factors (e.g. drilling and blasting performance) playing a role, it is safe to say that for at least the 100A22 stope, a major factor governing levels of dilution are the geological and rock-mechanical setting.

The meshing of the 300Lp3 drift on the hangingwall side of the 320K14 stope was the only aspect of stope support which did not meet the desired results. The initial idea was to install this mesh to prevent sloughage of small and medium sized rocks from the upper drift hangingwall. However, the heat generated by detonation of the explosives and the flyrock from the stope blast caused the mesh to bend and break. In hindsight, an alternative solution, such as reinforced fibre shotcrete could have been a better alternative.

7.2. Drilling and Blasting

From the drillhole deviation measurement results, it can be noticed that for the 320K14 stope, the drillhole collar deviation is larger than the drillhole toe deviation. This suggests that either: the starting point of the holes had a relative large offset and the angle of the holes deviate towards the planned hole toe; the drill trajectory changes eccentrically towards the designed hole toe; or a combination of these. From Figures 6.2., 6.3. and the 3D model can be concluded that the drillhole collars have an offset to one direction, while the drill angle and drillhole eccentricity deviate towards the opposite direction, being the hole toe. This peculiarity resulted into the fact that the calculated drillhole deviations in Table 6.2. do not accurately represent the level of deviation, as the calculated drillhole deviation in Equation (5) is dependent on the distance between the planned and the actual toe location. As the eccentricity of a drillhole trajectory is largely dependent on drill bit interaction when encountering anisotropic bedrock conditions (section 3.3.2.), it cannot be influenced. Therefore, if the hole trajectory had changed towards any other direction, the drillhole deviation in Table 6.2. would have been significantly larger.

From the drillhole deviation model of both the 320K14 and the 100A22 stope it can be seen that, on both the hangingwall as well as on the footwall the designed stope boundaries have not been met (Appendix F.). These errors in drilling result in dilution on one side of the stope while it causes ore loss on the other side of the stope. Reason for these drillhole deviations (drill angle and eccentricity of trajectory) are partially due to operational limitations and the aforementioned drill bit interaction.

However, the most likely reason for this drillhole deviation, is the way the drillhole collars are being marked. Currently, the designed stope plans are handed out on paper to the contractor. These paper plans indicate the location of the proposed drill hole collar location in the drift. The contractor measures the distance between the drift wall and the proposed collar location on this paper and then measures out the distance in the actual drift with measuring tape, followed by marking the spot with a spray can. Even though, the plans are scaled on paper and measured out with a scale ruler, this method offers a lot of room for error and is not completely accurate. A likely explanation for this is that during drifting, the drift profile does get measured irregularly. These profile measurements are at least taken every cut (4 metres) and more often in special locations, where visually the drift differentiates from the previous profile measurements. In between these profile sections, the drift model is completed by triangulating between measurement points. This triangulation results in the drift model not being 100% accurate. A recommended solution for this is given in section 9.

The actual (true) length of a drillhole that is being used in Equation (5) to determine the drillhole deviation, has been obtained by the total metres of cable that enters the hole, until the probe reaches the toe of hole. The cable is marked with a label each metre (Figure 7.2.) and does not have any markings in between metres. This resulted in the true length of the drillhole being reported within an accuracy of only 0.5 metres.

Figure 7.2: Drillhole deviation measurement probe and cable.

It is difficult to conclude anything from the use of the uncharged guiding holes (section 5.2.) which have been drilled between rings in the 320K14 stope. It could be argued that rather than helping the crack-formation and propagation during the blast and in turn attaining a better wall control, it helped the blasting energy escape more easily at the top column of the stope and therefore causing the lids on top. Regardless of the shallow dip of the 320K14 stope being a more likely reason for the occurrence of these lids, the guiding holes did not show notable improvements in wall control for the 320K14 stope and therefore have not been drilled in the 100A22 stope, nor discussed elaborately in this research.

The blast of one hole in the raise of rings 12-13 and the failed detonation of one hole in ring 15 of stope 320K14, can be explained due to the shallow dip of the stope. Just like the problem of having the lids on top of the stope, the reason for these unsuccessful detonations can be allocated to the shallow dip of the stope creating larger burdens to be blasted at the top of the stope.

During the blasting of rings 11-15 of the 100A22 stope, minimal PPV was noticed at a delay time of 771 ms (Figure 6.14.). By studying the detonation plan (Appendix E.) and the stoping plans, an explanation was found for this phenomenon. This minimal PPV can be allocated to the ignition order of the holes in rings 11-15. The set ignition pattern caused the rock around these holes to already have been (partially) blasted. This resulted in little or no remaining burden for the charged holes at 771 ms delay to be blasted. Therefore, when there is no burden to be blasted, there are no resulting vibrations measured by the monitoring system.

The blast of stope 100A22 rings 16-18 also showed minor PPV and a time shift of the recorded values (Figure 6.15.). The time shift of this recording is a result of the lower threshold value of the geophone not being met by the detonation of these holes. However, due to the recorder its buffer time of 1 second, these values have still been recorded. The cause for not meeting the threshold value is likely to be the small and fractured burden of ring 16.

The small PPV in the blasting of rings 11-15 and rings 16-18 show that when a rock mass is highly fractured, explosive energy can escape during the detonation and blasting smaller burdens allow for less vibrations than blasting larger burdens.

7.3. Dilution

From evaluating the reported dilution levels in sections 6.4. and 6.8., it can be seen that there is a significant difference between the dilution figures obtained from the stope scans and the dilution figures retrieved from the contractor’s mucked tonnage report. For the 320K14 stope the difference is only 4% while for the 100A22 stope the difference is 10%. In both stopes, the dilution obtained from the stope scans is lower than the one from the tonnage report. The explanation for this is a combination of uncertainties and inaccuracies of both dilution estimation methods.

The method of calculating dilution from the contractor's mucked tonnage report has only a small share when it comes to uncertainties and inaccuracies. the mucked tonnes are being weighted and an automatic report is generated. Small errors in calibration of the scale is possible yet unlikely, as the scale is being calibrated each year. However, during mucking some of the backfill on the floor of the bottom drift can be loaded together with the stope ore. This is not likely to influence the dilution more than 1%. Also, when comparing the amount of tonnes that are being mucked from the stope, to the tonnes that are being backfilled, noted was that these are nearly equal, which favours the idea that the majority of the dilution estimation errors originate from the stope scanning.

The method of retrieving dilution from the CMS model is prone to more errors. There are several operational limitations of the CMS, which in turn can contribute to errors in the model and therefore the estimated dilution. Irregular stope geometry can invoke problems with the 'line of sight' of the laser. The excavation method of drilling and blasting is likely to produce rugged stope surfaces and could potentially lead to blind spots which the laser of the CMS cannot reach. Another limitation of the system is that when the laser probe itself is moving during a scan, no account of this is taken into the collected data, potentially causing measured points being slightly in the wrong location.

The final source of inaccuracy in the estimation of dilution from the CMS model comes from modelling the stope cavity out of the obtained data set. The model is created by triangulating between measured data points. Since the measurements are based on a rotation in degrees, the distances between measured points get larger further away from the probe. Triangulating over larger distances creates a less realistic view of the model and could therefore cause errors in estimating dilution.

Before conducting a scan, the probe location, scan density and the pattern (horizontal or vertical) must be configured. The scan density determines the amount of data points. The higher the density, the more data points, but the heavier the model. For the scans, a density of 3° has been used. Meaning, a full 360° scan would take 120 measuring cycles. Narrowing this down to 1° did not improve the accuracy of the model but increased the size of the data set considerably. It had been noticed that also the scan pattern influences the result of the scan. In general, the horizontal scan pattern provided the best results.

After evaluating the limitations of the dilution estimation methods, conceived is that the actual levels of dilution for both stope are most likely to be in between the levels suggested by the stope scan model and the contractor's tonnage report. Regardless, an uncertainty of 5% to 10% must be considered.

Comparing the dilution of 320K14 to the average dilution in the Kujankallio main zone, one notes that calculated dilution of both the scans and the contractor's tonnage report (20% and 24% respectively), are lower than the average of 25%. This indicates good stope performance. This is confirmed by the resulting ELOS from both the CMS model and the contractor's tonnage report, as they fall in Clark & Pakalnis' (1997) category of < 0.5 m ELOS, which is classified as blast damage only. According to Suorineni et al. (2001) the stope is defined as stable, as ELOS ≤ 0.5 m.

The 100A22 stope has a higher dilution than the 31% average of the previously mined Arpola stopes. According to Clark & Pakalnis (1997) the 100A22 stope falls in the category of 1 < ELOS < 2 m and is classified as moderate sloughing. The study of Surorineni et al. (2001) defines the 100A22 stope as unstable (0.5 < ELOS < 5 m). Most likely, the reason for these elevated levels of dilution are the joints and faults described and depicted in section 6.5. and in Appendix G.

8. Conclusions

Along with conducting a literature review, operational parameters and stope performance were monitored to allocate the origin of and the main influences on unplanned waste rock dilution in the Jokisivu mine. In addition to that, measures were designed to attempt to minimize waste rock dilution during narrow-vein stoping. Gathered data and results were interpreted to determine what can be done to minimize waste rock dilution in future practices during narrow-vein stoping at the Jokisivu operation.

Two stopes were researched, both in different geological and rock-mechanical setting. Resulted dilution levels did not indicate sizable improvements relative to previous stopes, regardless of successful implementation of stope support and the use of electronic detonators. This phenomenon can be allocated to two main factors governing dilution levels at the Jokisivu operation.

The first group of factors are the local geological structures and rock conditions around a stoping area. The use of a Cavity Monitoring System (CMS) substantiates this claim, indicating higher levels of dilution at locations where rock-mechanical conditions are poor due to anisotropic bedrock conditions (e.g. joint sets and fault planes) and weak host rock (mica-gneiss). CMS models indicated both dilution on hangingwall and footwall side of the stope. However, the most dilution originated from the hangingwall.

The second factor is the inaccuracy in marking drillhole collar locations. Drillhole deviation measurements pointed out that maximum drillhole collars deviations were as large as 1.23 metres in the 320K14 stope and 0.92 metres in the 100A22 stope. From the measured drillhole trajectories it was seen that, even though drillhole angles did not deviate that much, both the designed footwall and hangingwall stope boundaries were locally not met or exceeded. This resulted in additional dilution and ore loss. Section 9. covers recommendations on improving this in future practices.

9. Recommendations

Due to the geological complexity, the narrow-vein mineralization allows for only one stope in a certain location. Getting to know the geological and rock-mechanical conditions in one location does not immediately provide answers for the next stoping location. This proclaims that minimizing waste rock dilution regarding geological and rock-mechanical conditions is rather difficult. Recommended is to focus on other practices to minimize dilution.

As the stope scans indicated, the largest share of unplanned dilution originated from the hangingwall side of the stopes. Therefore, it is recommended to provide extra support to the hangingwall side of the stope when designing stope support. Where possible, this should be done by increasing the density of cablebolting. Recommended is to keep monitoring stope performance by scanning each stope after it has been mucked out in order quantify and interpret the origins of dilution more accurately.

Within this research, electronic detonators have not been tested too elaborately and thus, no significant improvements during regular production blasts were noticed in comparison to the use of shock tube detonators. However, for bigger blasts, just like the blast of rings 11-15 of 100A22, the use of electronic detonators was proven useful. Recommended is to conduct more research into the implementation of electronic detonators for especially these larger blasts. In specific, recommended is to experiment more with the delay times of the electronic detonators. The freedom of setting distinct delay times offers potential for a reduction in overbreak, a better wall control and therefore less dilution and ore loss.

The biggest potential for minimizing waste rock dilution and ore loss lies in improving the drillhole collar marking system. The reasons and explanations for this were covered in sections 7.2 and 8. Recommended are two approaches to tackle this issue. The first one being replacing the current system altogether with a system that automatically reads and marks out the desired locations. However, purchasing such a system comes at additional costs.

The second recommendation to improve the drillhole collar marking system is to perform a scan of the drifts enclosing the desired stope location before designing the stope. In this manner, the exact drift model is known before production. Knowing the exact drift outlines causes errors in drillhole collar marking to be less likely to happen and result into smaller collar deviations. Knowing the exact drift model, does also ease the process of stope designing. Operational limitations in positioning the longhole drill rig and its boom have higher accuracy and thus, drillers are less prone to encounter operational difficulties.

The financial benefit of implementing this measure is best illustrated with a cost estimate for processing additional waste rock and having unnecessary ore loss due to inappropriate drillhole collar markings. From tables 6.1. and 6.3. it can be seen that the average drillhole collar deviation for stopes 320K14 and 100A22 are 0.66 m and 0.47 m respectively. For this example, we take a deviation of 0.5 metres. Approximately the same deviation that has been noted at the footwall hole of 100A22 ring 13. 100A22 Stope height and stope strike length are 14 m and 38 m respectively. Assuming 30 euros processing costs for one tonne of waste rock and a stope grade of 3 g/t with a current gold price of 35 euros per gram and the 0.5 metres deviation being continuous throughout the full length and height of the stope. The 100A22 stope would suffer severe additional waste rock processing costs (Equation (7)) and ore loss (Equation (8)).

$$14\,m * 38\,m * 0.5\,m * 2.8\frac{kg}{m^3} * 30\frac{€}{t} = €\ 22.344 \tag{7}$$

$$14\,m * 38\,m * 0.5\,m * 2.8\frac{kg}{m^3} * 3\frac{g}{t} * 35\frac{€}{g} = €\ 78.204 \tag{8}$$

Naturally, drillhole collar deviation can occurs into all directions and is not necessarily always directed outwards to the designed stope boundaries, like here. Nevertheless, this example shows the possible positive impact implementing these measurements have on the economic viability of the operation.

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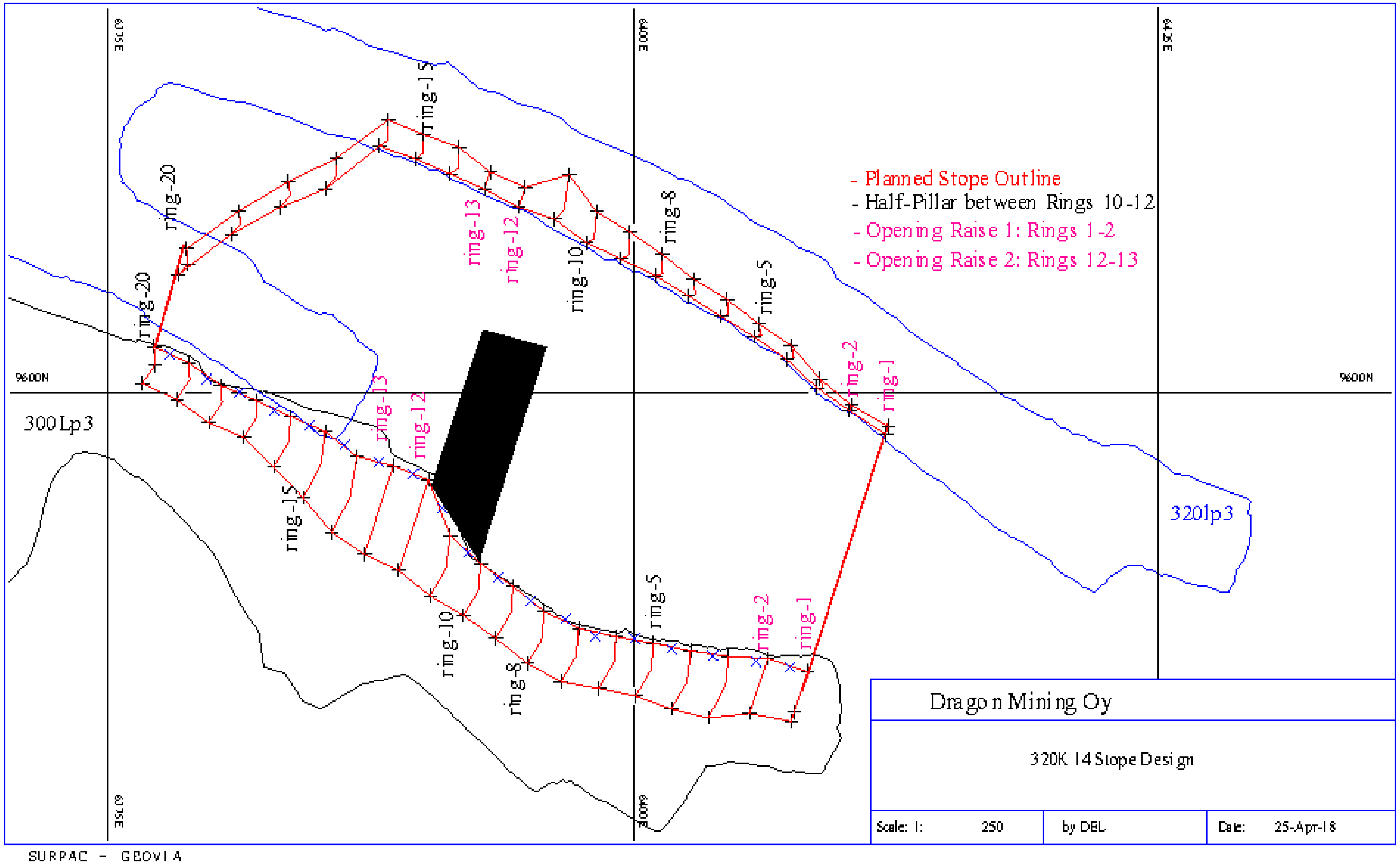
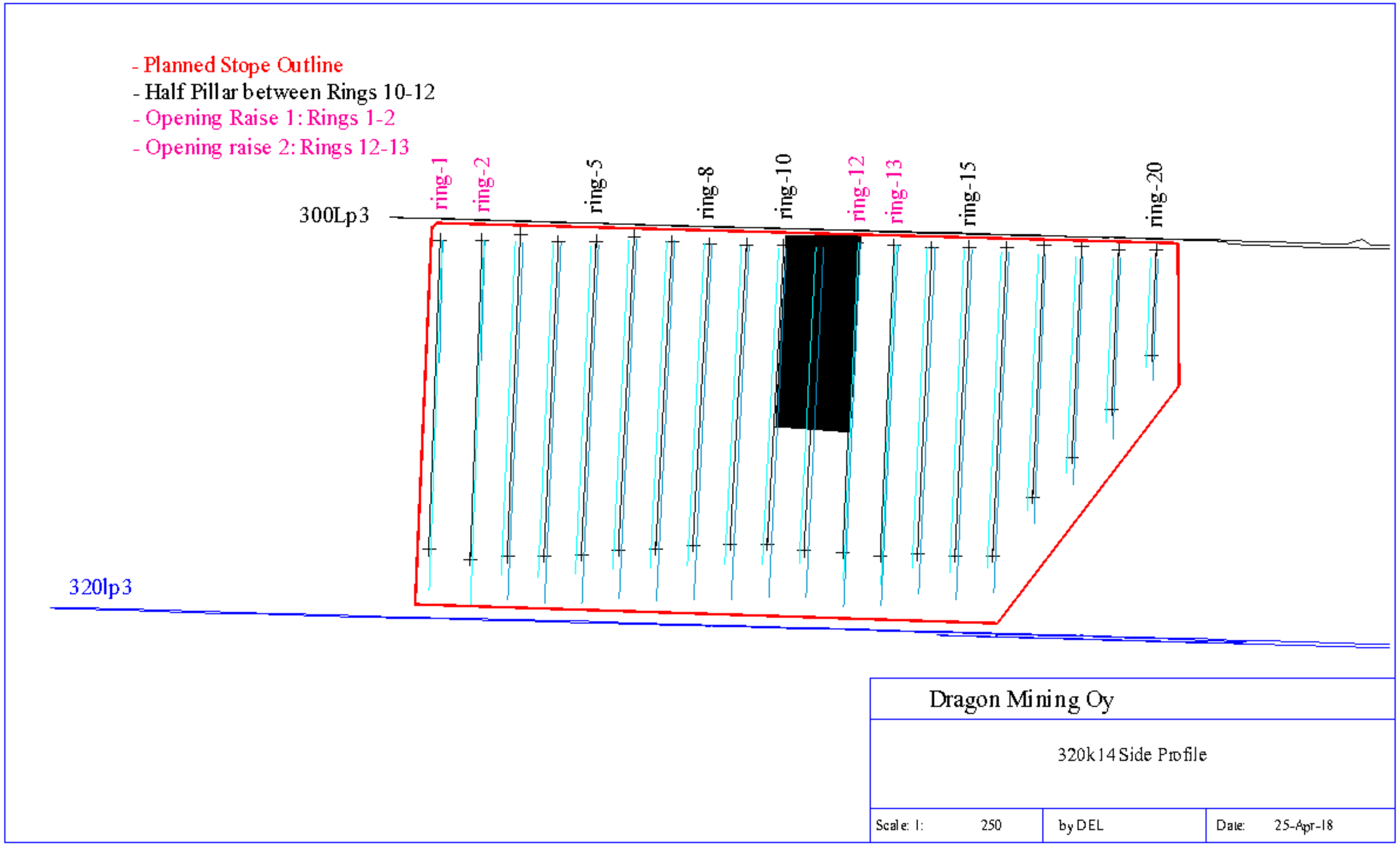
Appendix A – Stope Design Plans

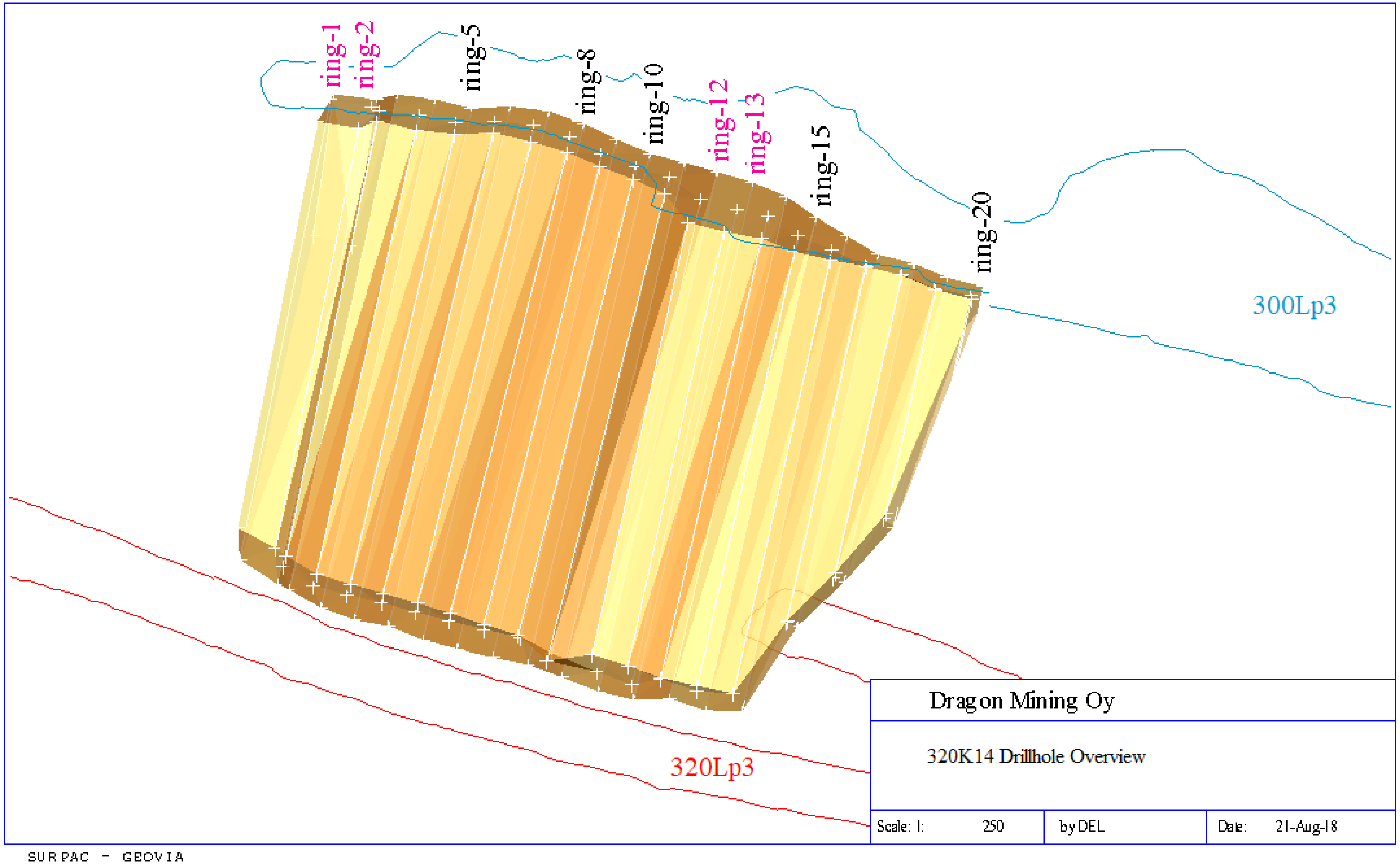
320K14 Stope plans handed out to the contractor – Stope Design (Pohjakuva)



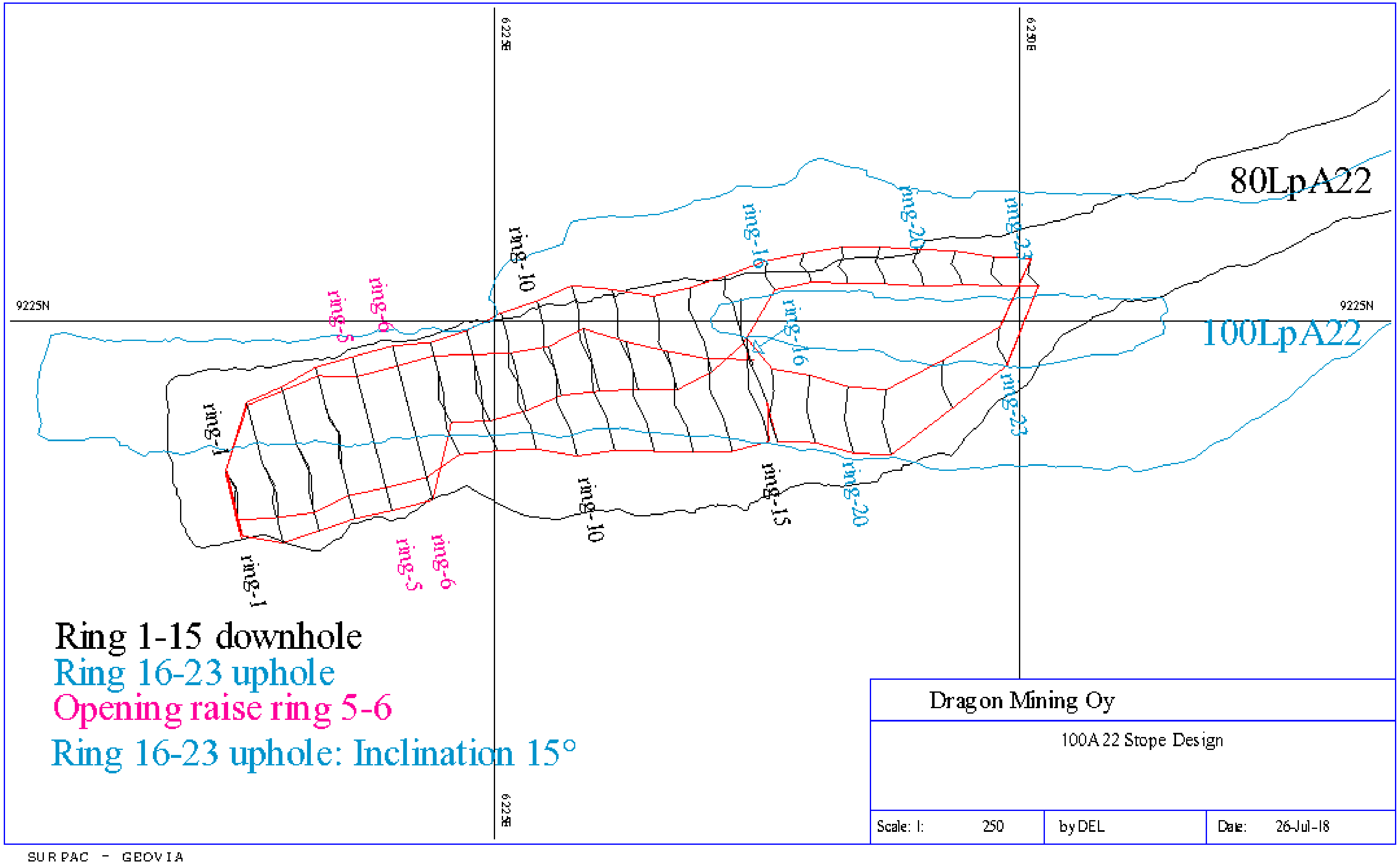
320K14 Stope plans handed out to the contractor – Side Profile (Sivuprofiili)



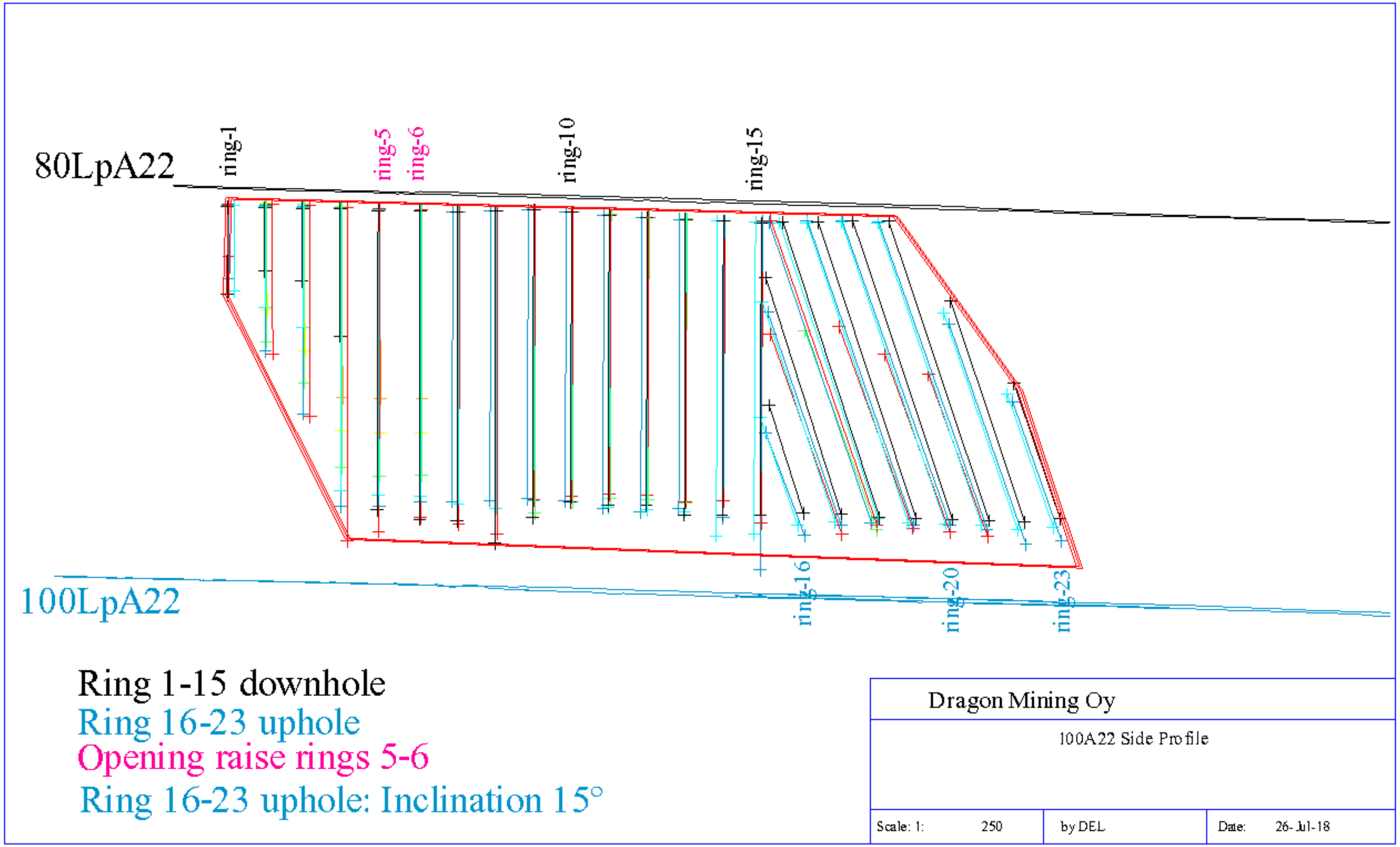
320K14 Drillhole Design Overview



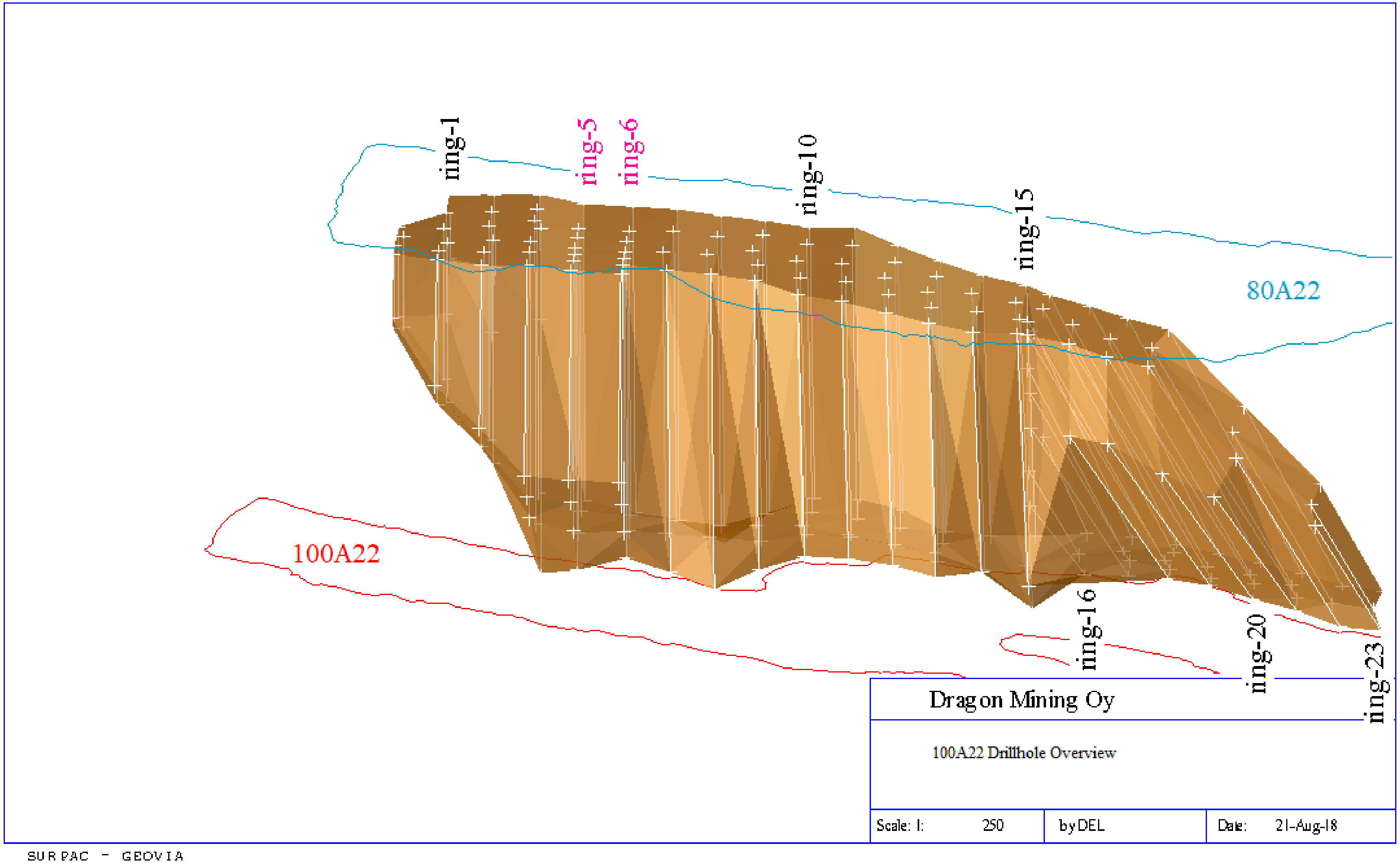
100A22 Stope plans handed out to the contractor – Stope Design (Pohjakuva)



100A22 Stope plans handed out to the contractor – Side Profile (Sivuprofiili)



100A22 Drillhole Overview



Appendix B – Stope Reports

Planned Stope Report 320K14

	07/06/2018							
	By	**Stope**	**Stope #**					
	DEL	320k14	104					
	Ring (#)	**Volume (m3)**	**Tonnage (t)**	**Grade (g/t)**	**Drilled Meters (m)**			
	1 (raise)	x	x		47.8			
	2 (raise)	100	280		49.5			
	3	39	109.2		66.6			
	4	80	224		69			
	5	89	249.2		69.9			
	6	84	235.2		71.7			
	7	85	238		72.9			
	8	90	252		73.2			
	9	98	274.4		72			
	10 (pillar)	101	282.8		71.2			**Pillar**
	11 (pillar)	106	296.8		71.2			809.2
	12 (raise)	126	352.8		69			289
	13 (raise)	114	319.2		69.2			
	14	102	285.6		68.6			
	15	94	263.2		66.9			
	16	87	243.6		65.5			
	17	67	187.6		52.1			
	18	52	145.6		44.5			
	19	40	112		33.8			
	20	29	81.2		22.4			
Total Actual	x	1583	4432.4	3	1227			
Total Excluding Pillar	x	1294	3623.2					
			Min	**Max**				
Dilution (%)			0.3	0.6				
Total Including Dilution (t)			4710.16	5797.12				

Min and Max dilution figures for both 100A22 and 320K14 are lower and higher threshold boundary estimates which are assumptions made in advance based on knowledge of the stope location. more detailed explanation can be found from section 5.1.

Planned Stope Report 100A22

26/07/2018						
By	**Stope**	**Stope #**				
DEL	100a22	108				
	Ring (#)	**Volume (m3)**	**Tonnage (t)**	**Grade (g/t)**	**Drilled Meters (m)**	**Holes**
	1	x	x		14.5	4
	2	41	114.8		33.7	6
	3	79	221.2		44.7	6
	4	121	338.8		81.7	7
	5 (raise)	154	431.2		89.9	7
	6 (raise)	172	481.6		88.9	7
	7	137	383.6		57.4	4
	8	117	327.6		59.3	4
	9	124	347.2		70.9	5
	10	120	336		69.8	5
	11	130	364		69.3	5
	12	132	369.6		69.9	5
	13	120	336		69.9	5
	14	111	310.8		58.3	4
	15	108	302.4		62.5	4
	16+17+18*	166	464.8		139.6	3+4+5
	19	85	238		60.2	4
	20	84	235.2		59.7	4
	21	81	226.8		59	4
	22	62	173.6		35.7	3
	23	34	95.2		22.5	3
Total Planned	x	2144	6003.2	3	1294.9	108
			Min	**Max**		
Dilution (%)			0.15	0.45		
Total Including Dilution (t)			6903.68	8704.64		
* rings 16 and 17 were too small and individual triangulation resulted in wrong estimations. therefore combined						

*This is the initial planned stope report for the 100A22 stope. Because of error in contractors drilling performance and ore loss, the updated planned ore tonnage is **5454.4 tonnes** (after remodelling the stope according to stope scans and drillhole deviation measurements) see section 6.6 for explanation.

320K14 Stope Tonnage and Dilution Figures

Stope		
320K14		
In-Situ (t)	4432.4	
Pillar (t)	809.2	
Planned (t)	3623.2	
	Mucked (t)	Stope Scan (t)
rings 1-9	2262.4	2466.8
rings 14-20	2222.25	1892.8
Total	4484.65	4359.6

Dilution Planned vs. Mucked (%)	Elos Planned vs. Mucked (m)
24	0.44
Dilution Planned vs. Stope Scan (%)	**Elos Planned vs. Stope Scan (m)**
20	0.38
Average Dilution Kujankallio mainzone (%)	
25	

100A22 Stope Tonnage and Dilution Figures

Stope		
100A22		
In-Situ (t)	5454.4	
Planned (t)	5454.4	
	Mucked (t)	Stope Scan (t)
	7672.6	7064.4

Dilution Planned vs. Mucked (%)	Elos Planned vs. Mucked (m)
41	1.49
Dilution Planned vs. Stope Scan (%)	**Elos Planned vs. Stope Scan (m)**
30	1.08
Average Dilution Arpola stopes (%)	
31	

Appendix C – Drillhole Deviation Measurements

320K14 Drillhole Deviation Measurements

	Hole # (numbered from HW to FW)			
	Hole collar difference	1	2	3
Ring #				
1-3				
4	Difference (m)	0.265	0.553	0.224
5				
6	Difference (m)	0.63	0.634	0.671
7				
8	Difference (m)	0.703	0.799	0.714
9				
10	Difference (m)	0.618	0.741	0.63
11				
12	Difference (m)	0.491	0.45	0.888
13				
14	Difference (m)	FALSE	1.227	1.073
15				
16	Difference (m)	FALSE	0.688	0.638
17				
18	Difference (m)	0.599	0.612	0.615
19				
20	Difference (m)	x	x	x
	Average (m)	0.657		
	Min (m)	0.224		
	Max (m)	1.227		

	Hole # (numbered from HW to FW)			
	Hole toe difference	1	2	3
Ring #				
1-3				
4	Difference (m)	0.076	0.217	FALSE
5				
6	Difference (m)	0.475	0.596	0.63
7				
8	Difference (m)	0.711	0.07	0.642
9				
10	Difference (m)	0.363	0.647	0.342
11				
12	Difference (m)	0.946	0.661	1.234
13				
14	Difference (m)	FALSE	0.546	0.198
15				
16	Difference (m)	FALSE	0.598	0.914
17				
18	Difference (m)	0.726	0.085	0.177
19				
20	Difference (m)	x	x	x
	Average (m)	0.517		
	Min (m)	0.07		
	Max (m)	1.234		

	Hole # (numbered from HW to FW)			
	actual hole length	1	2	3
Ring #				
1-3				
4	Length (m)	18	21.401	FALSE
5				
6	Length (m)	21	22	20.8
7				
8	Length (m)	22	23	24
9				
10	Length (m)	22	22	24
11				
12	Length (m)	20	22	24
13				
14	Length (m)	FALSE	22	22
15				
16	Length (m)	FALSE	21.7	22
17				
18	Length (m)	14.5	14.2	15.3
19				
20	Length (m)	x	x	x
	Average (m)	20.85		
	Min (m)	14.2		
	Max (m)	24		

	Hole # (numbered from HW to FW)			
	Drillhole Deviation From Literature	1	2	3
Ring #				
1-3				
4	Deviation (%)	0.42	1.01	FALSE
5				
6	Deviation (%)	2.26	2.71	3.03
7				
8	Deviation (%)	3.23	0.30	2.68
9				
10	Deviation (%)	1.65	2.94	1.43
11				
12	Deviation (%)	4.73	3.00	5.14
13				
14	Deviation (%)	FALSE	2.48	0.90
15				
16	Deviation (%)	FALSE	2.76	4.15
17				
18	Deviation (%)	5.01	0.60	1.16
19				
20	Deviation (%)	x	x	x

Drillhole Deviation From Literature	
Average (%)	2.46
Min (%)	0.30
Max (%)	5.14

100A22 Drillhole Deviation Measurements

		Hole # (numbered from HW to FW)				
	Hole collar difference	**1**	**2**	**3**	**4**	**5**
Ring #						
1-8						
9	Difference (m)	0.141	0.226	0.378	0.141	0.348
10						
11	Difference (m)	0.654	0.391	0.918	0.653	0.612
12						
13	Difference (m)	0.683	0.395	0.887	0.555	0.561
14						
15	Difference (m)	0.411	0.492	0.535	0.535	
16						
17						
18						
19						
20	Difference (m)	0.262	0.74	0.221		
21						
22	Difference (m)	0.322	0.165			
23						
	Average (m)	0.468				
	Min (m)	0.141				
	Max (m)	0.918				

		Hole # (numbered from HW to FW)				
	Hole toe difference	**1**	**2**	**3**	**4**	**5**
Ring #						
1-8						
9	Difference (m)	0.496	0.284	0.416	0.476	0.475
10						
11	Difference (m)	0.682	0.451	0.855	0.864	0.974
12						
13	Difference (m)	0.681	0.572	0.891	0.541	0.82
14						
15	Difference (m)	0.086	0.46	0.532	1.11	
16						
17						
18						
19						
20	Difference (m)	0.471	0.383	0.097		
21						
22	Difference (m)	FALSE	FALSE			
23						

*False because designed drillholes were not supposed to penetrate.

Average (m)	0.574
Min (m)	0.086
Max (m)	1.11

		Hole # (numbered from HW to FW)				
	Actual hole length	**1**	**2**	**3**	**4**	**5**
Ring #						
1-8						
9	Length (m)	14.4	13	13.5	13.5	14
10						
11	Length (m)	13.5	13.6	14	13.4	13.5
12						
13	Length (m)	14	14	14	13.5	13.5
14						
15	Length (m)	14	14	14	15	
16						
17						
18						
19						
20	Length (m)	14	15	15		
21						
22	Length (m)	15	15			
23						
	Average (m)	14.0				
	Min (m)	13				
	Max (m)	15				

		Hole # (numbered from HW to FW)				
	Drlilhole Deviation From Literature	**1**	**2**	**3**	**4**	**5**
Ring #						
1-8						
9	Deviation (%)	3.44	2.18	3.08	3.53	3.39
10						
11	Deviation (%)	5.05	3.32	6.11	6.45	7.21
12						
13	Deviation (%)	4.86	4.09	6.36	4.01	6.07
14						
15	Deviation (%)	0.61	3.29	3.80	7.40	
16						
17						
18						
19						
20	Deviation (%)	3.36	2.55	0.65		
21						
22	Deviation (%)	FALSE	FALSE			
23						

Drillhole Deviation From Literature	
Average (m)	4.129
Min (m)	0.614
Max (m)	7.4

Appendix D – Vibration Monitoring

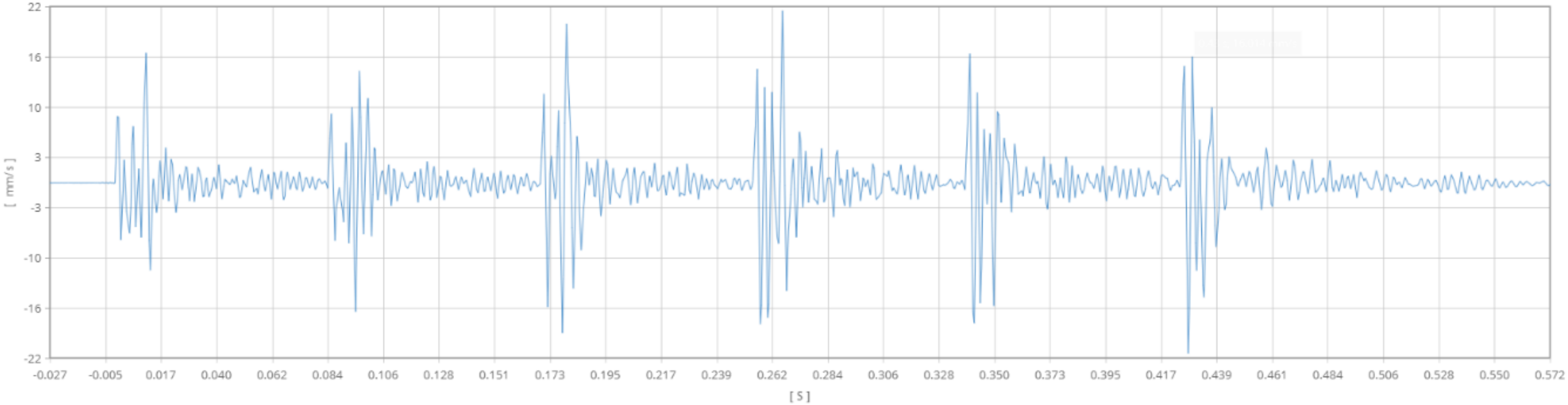
320K14 Vibration Responses

Stope	Vibration Monitor	Installed level						
320k14	JOK1	285						
					Vertical Node			
Date	**Time**	**Rings**	**Holes**	**Peaks**	**PPV (mm/s)**	**Frequency (hz)**	**Acceleration (m/s2)**	**Displacement (mm)**
not measured	not measured	1-2	-	-	-	-	-	-
not measured	not measured	3	3	-	-	-	-	-
31/05/2018	15:50:22	4-5	6	6	21.9	256	38.7	0.02
05/06/2018	16:00:29	6	3	3	24.5	208	35.9	0.033
06/06/2018	16:37:11	7-8-9	6	6	41.6	169	41.8	0.044
-	-	10-11	-	-	-	-	-	-
08/06/2018 - 12/06/2018	-	12-13	-	-	-	-	-	-
14/06/2018	16:04:00	14	4	4	38.7	124	39.4	0.0862
19/06/2018	16:10:07	15	4	3	37	120	42.7	0.0806
20/06/2018	04:12:19	16	3	3	56.7	131	47	0.109
20/06/2018	15:59:05	17-18-19-20	13	13	33.9	85.6	42.5	0.0935

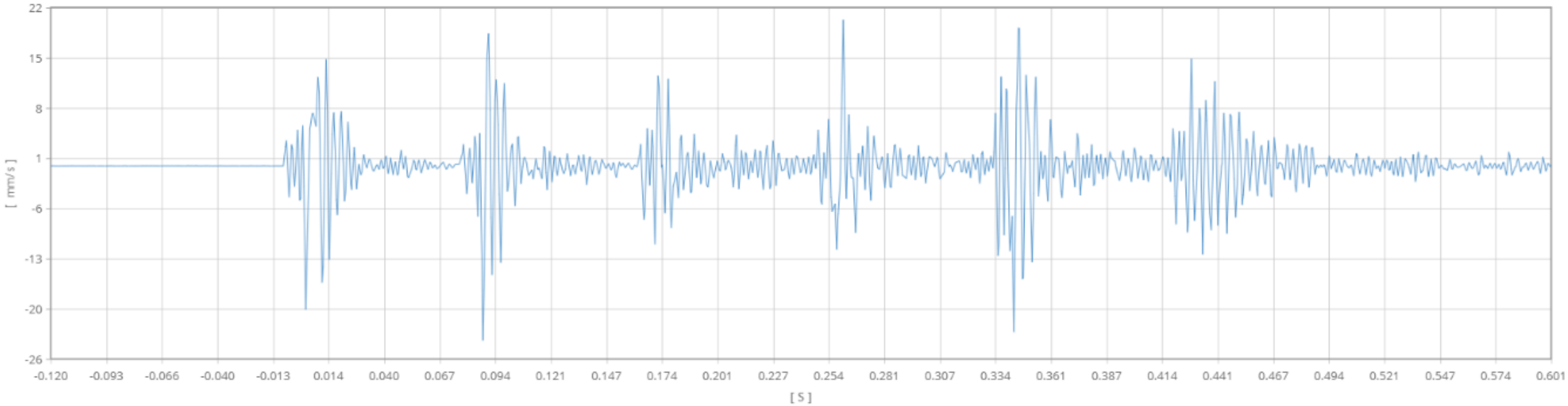
Stope	Vibration Monitor	Installed level						
320k14	JOK2	300						
					Vertical Node			
Date	**Time**	**Rings**	**Holes**	**Peaks**	**PPV (mm/s)**	**Frequency (hz)**	**Acceleration (m/s2)**	**Displacement (mm)**
not measured	not measured	1-2	-	-	-	-	-	-
not measured	not measured	3	3	-	-	-	-	-
31/05/2018	15:50:27	4-5	6	6	24	250	35.6	0.031
05/06/2018	16:00:39	6	3	3	24.1	216	40	0.025
06/06/2018	16:37:12	7-8-9	6	6	28.8	235	42.8	0.028
-	-	10-11	-	-	-	-	-	-
08/06/2018 - 12/06/2018	-	12-13	-	-	-	-	-	-
14/06/2018	16:04:02	14	4	4	48.2	249	68.2	0.0463
19/06/2018	16:10:13	15	4	3	37.8	214	45.9	0.0444
20/06/2018	04:12:25	16	3	3	34	224	50.2	0.0489
20/06/2018	15:59:12	17-18-19-20	13	13	27.7	313	49	0.0264

320K14 Waveforms

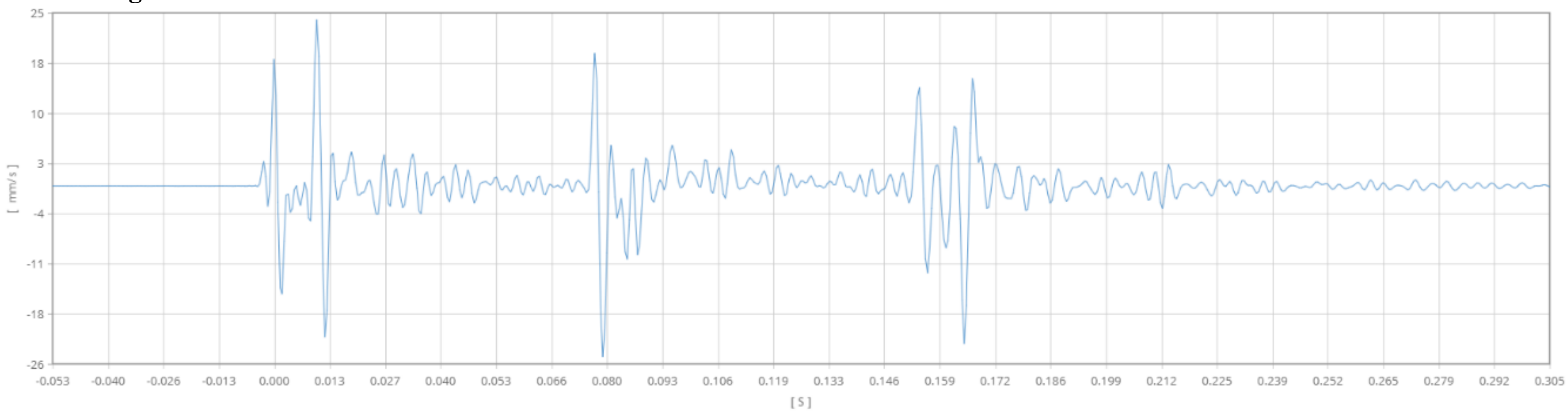
JOK 1: Rings 4-5



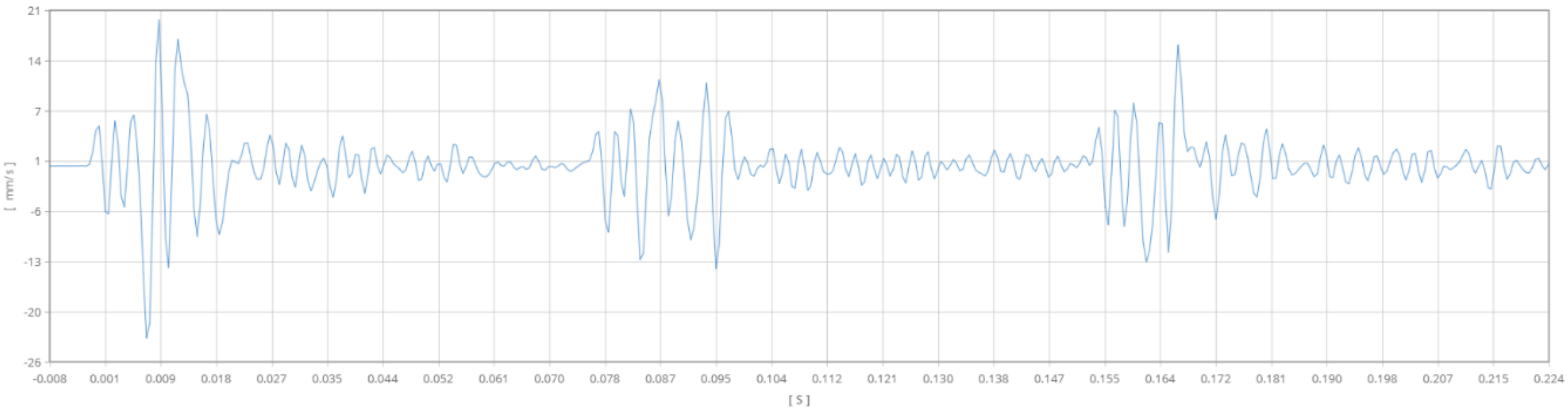
JOK 2: Rings 4-5



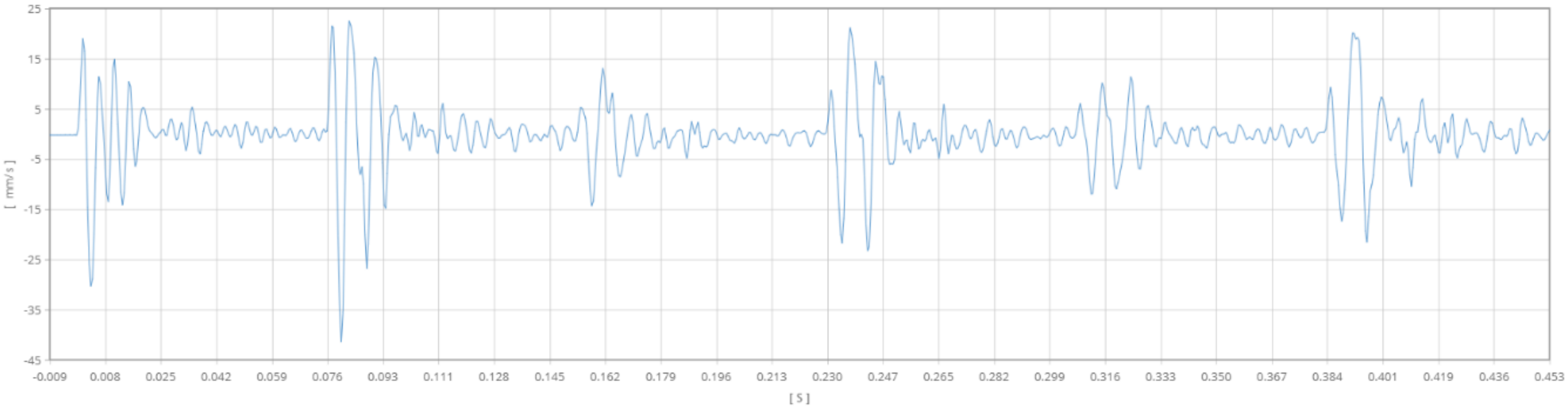
JOK 1: Ring 6



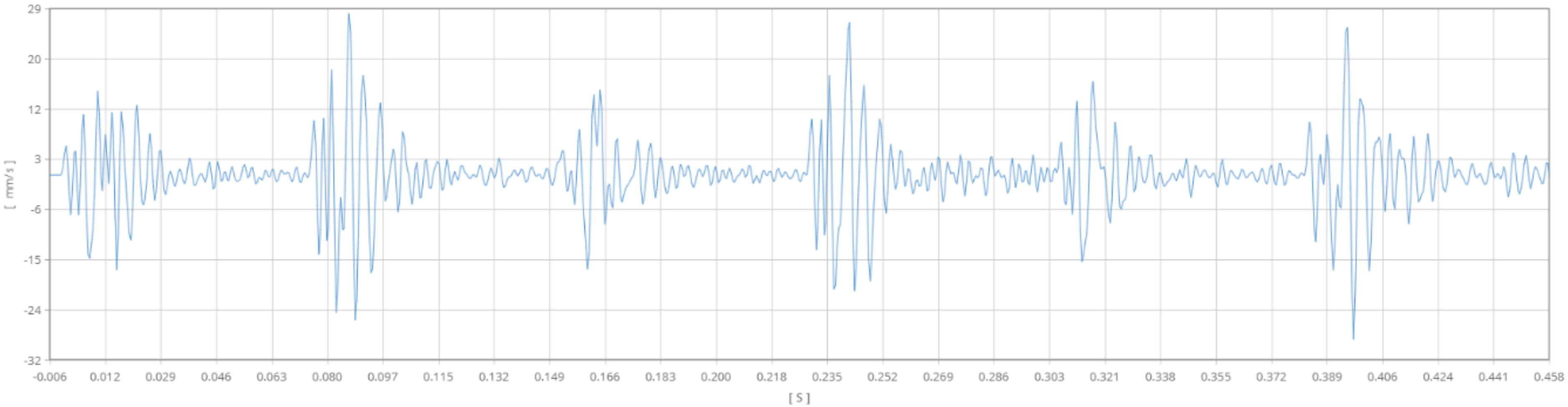
JOK 2: Ring 6



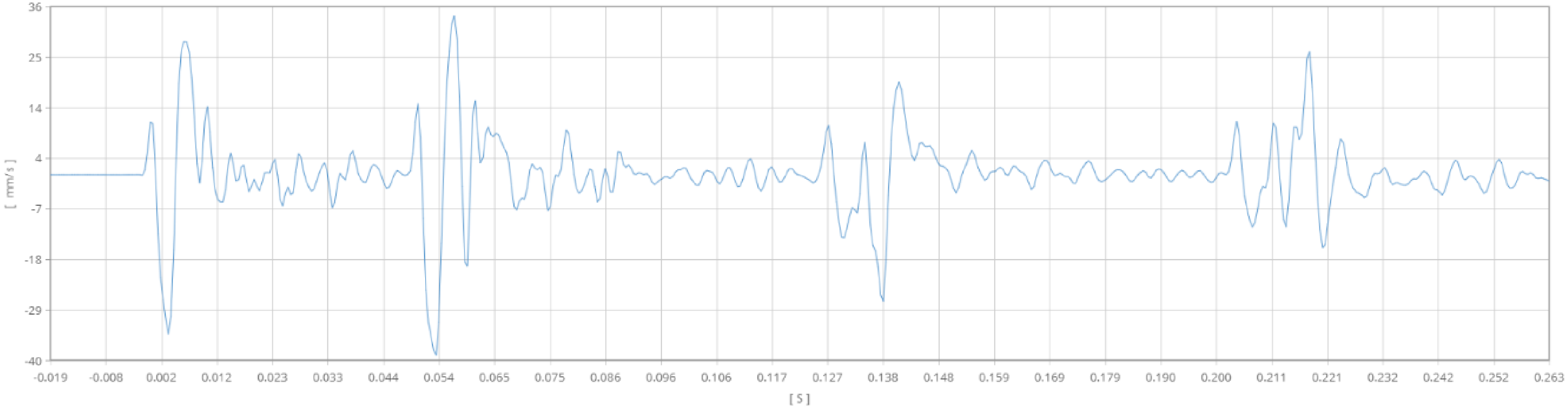
JOK 1: Rings 7-9



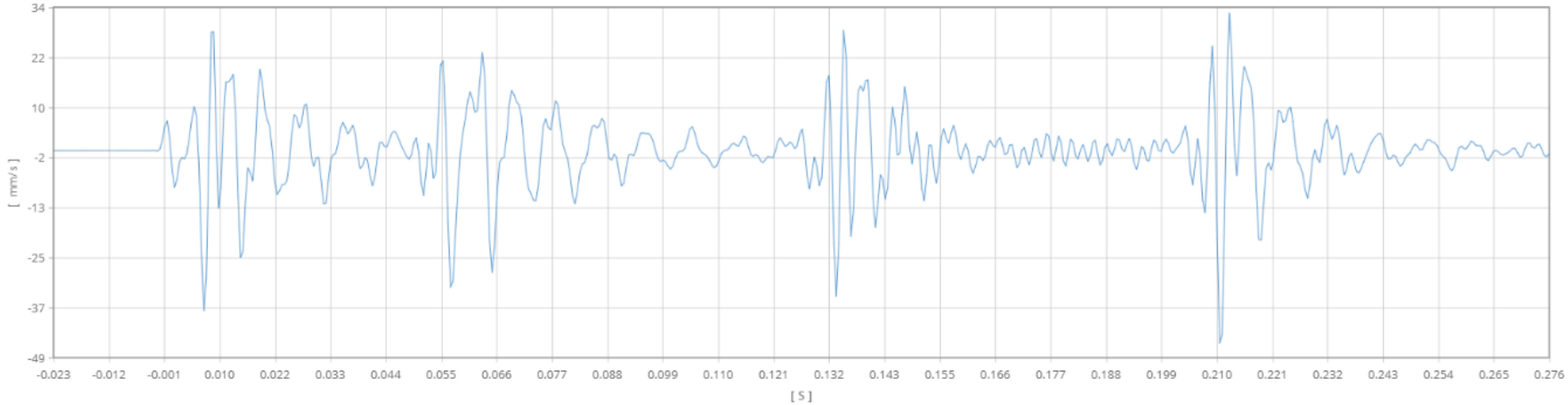
JOK 2: Rings 7-9



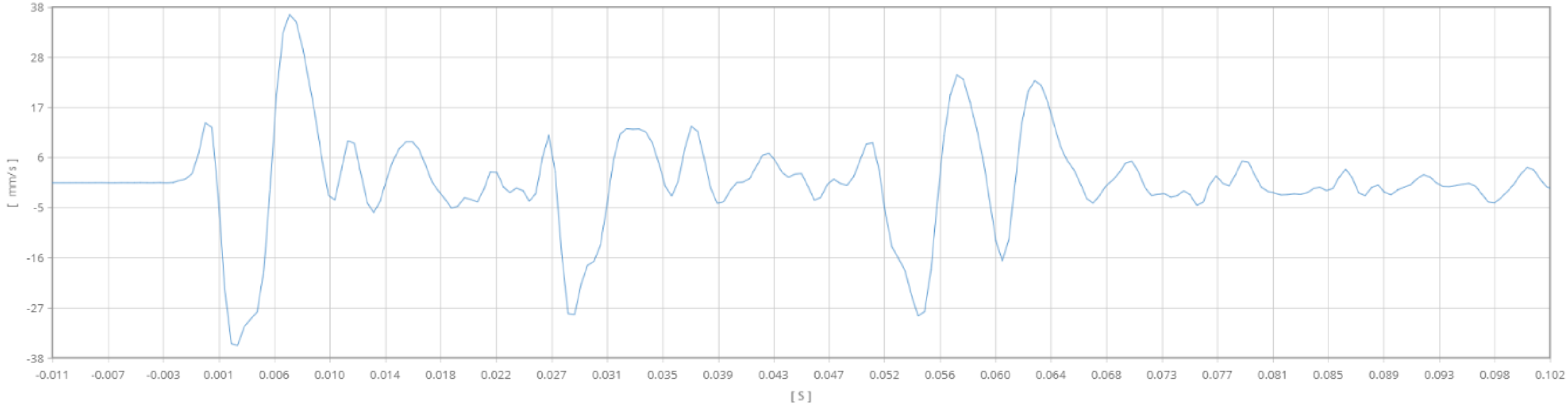
JOK 1: Ring 14



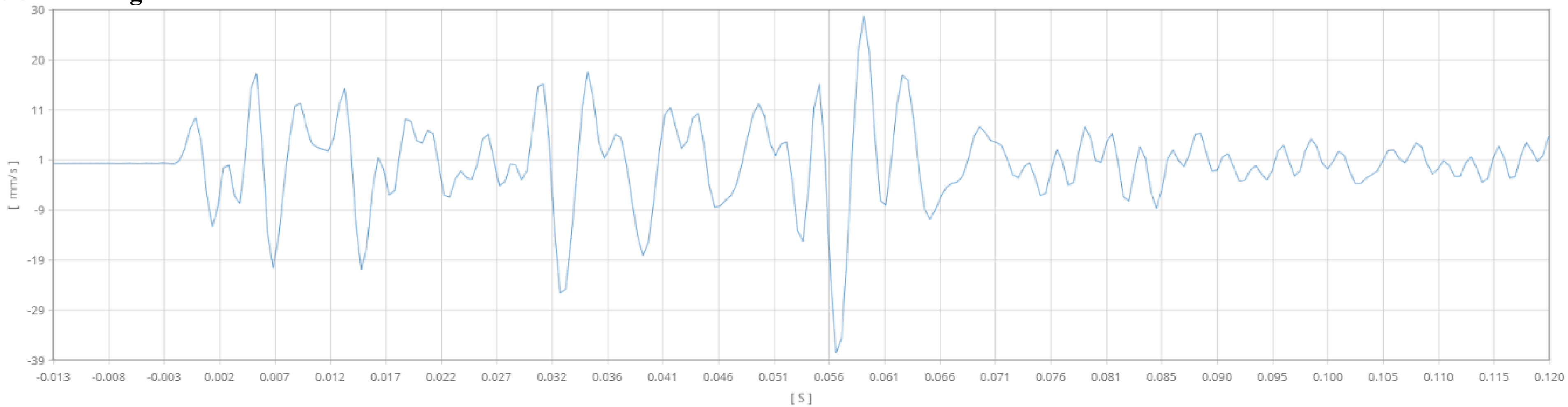
JOK 2: Ring 14



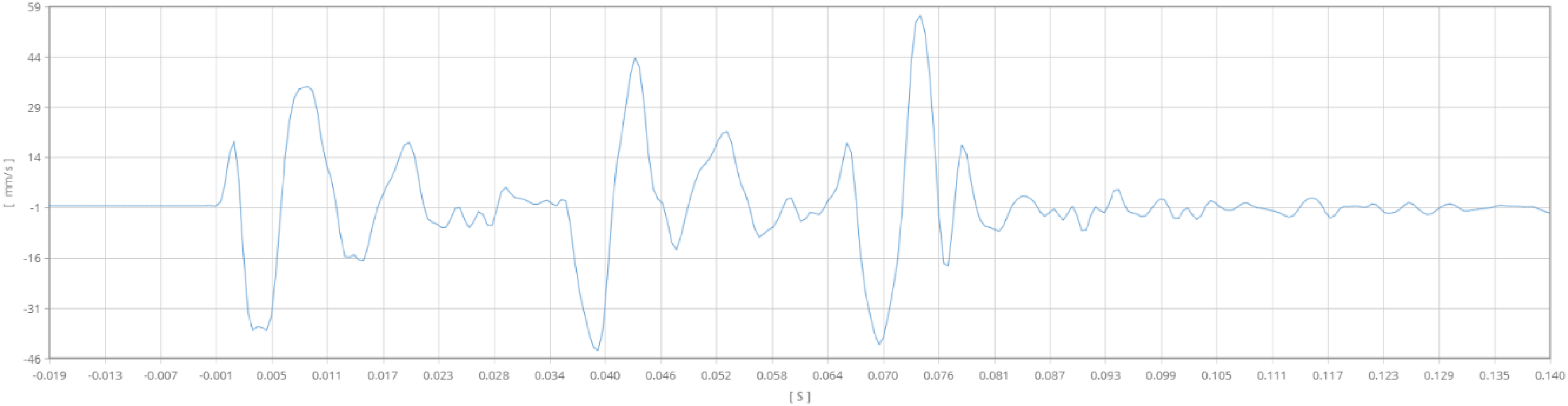
JOK 1: Ring 15



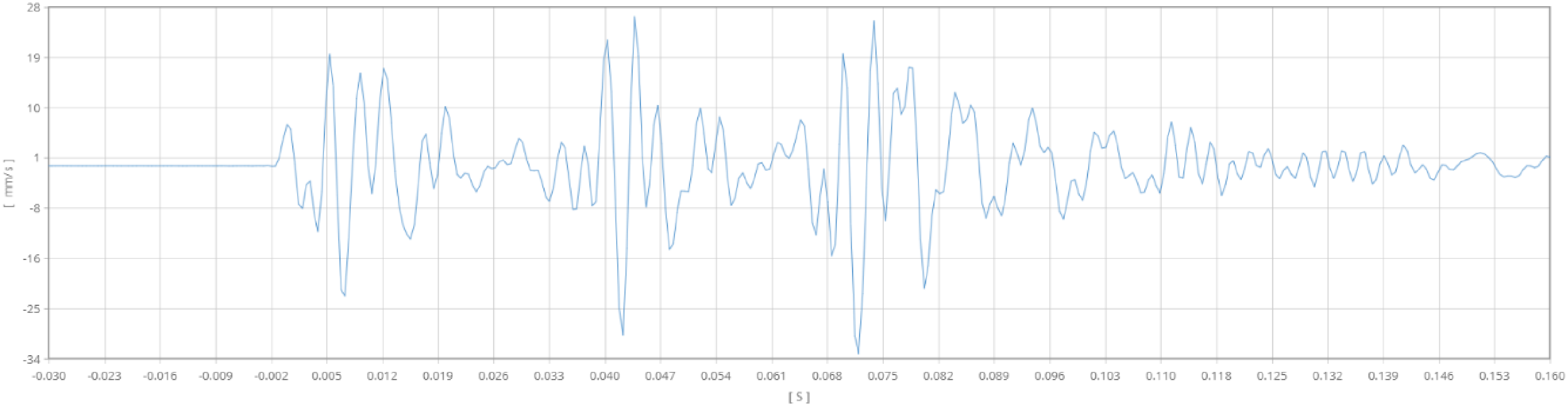
JOK 2: Ring 15



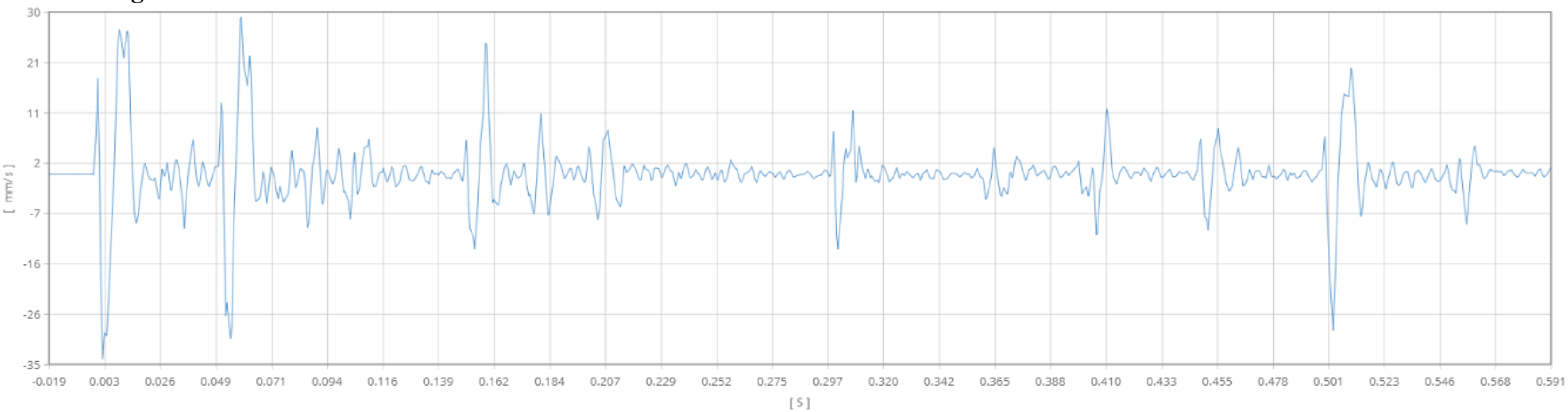
JOK 1: Ring 16



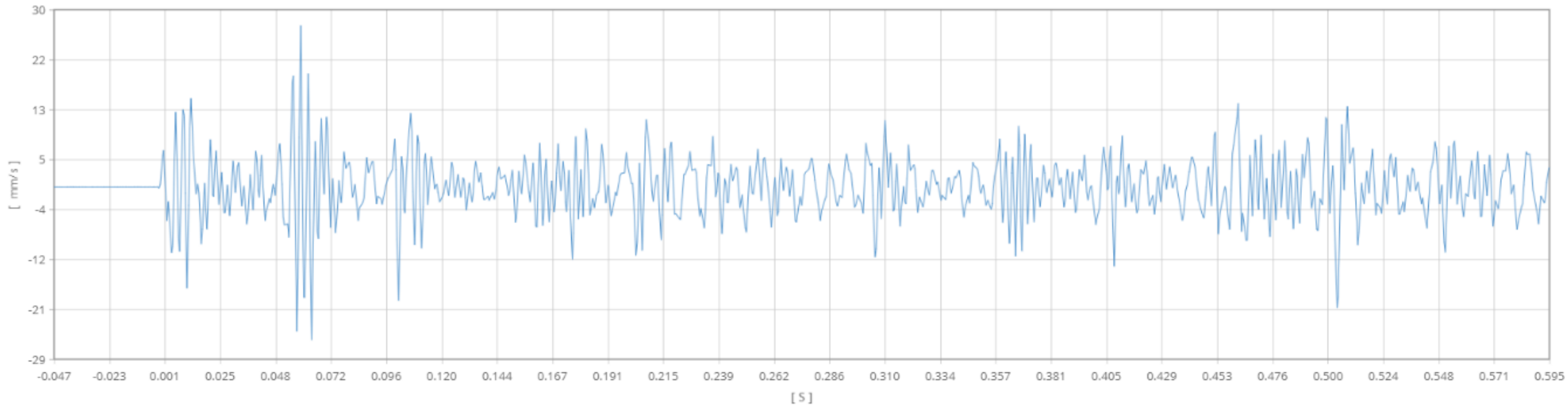
JOK 2: Ring 16



JOK 1: Rings 17-20



JOK 2: Rings 17-20



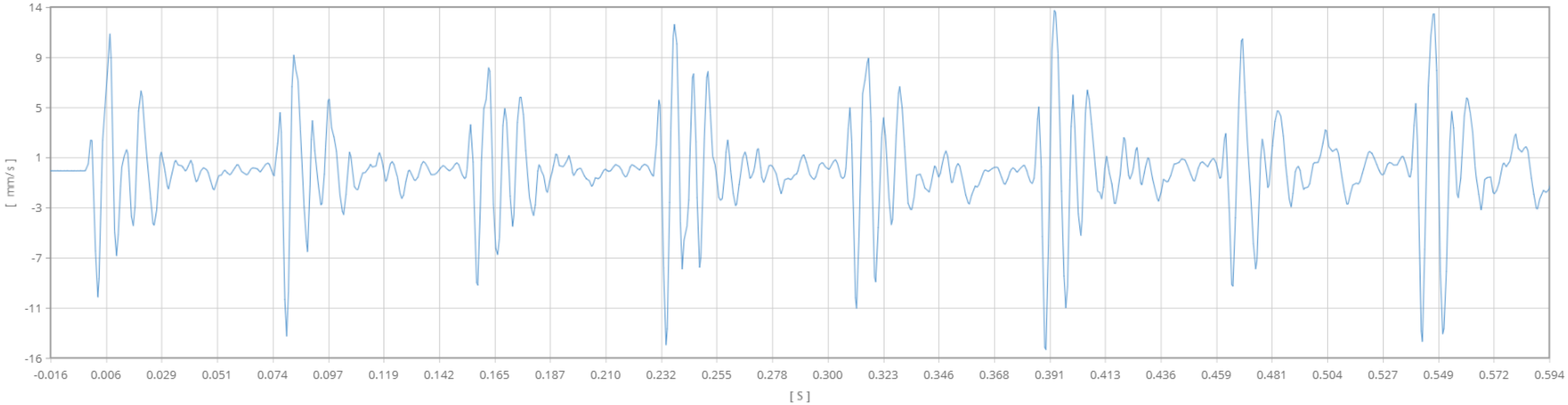
100A22 Vibration Responses

Stope	Vibration Monitor	Installed level						
100a22	JOK3	65						
					Vertical Node			
Date	Time	Rings	Holes	Peaks	PPV (mm/s)	Frequency (hz)	Acceleration (m/s2)	Displacement (mm)
not measured	not measured	1-7	-	-	-	-	-	-
06/09/2018	12:06:00	8-10	14	8	15.2	147	12.9	0.0388
13/09/2018	12:04:26	11-15	23	12	13.2	141	12.3	0.0391
20/08/2018	08:44:49	16-18	11	8	5.52	55	3.6	0.016
27/09/2018	09:01:25	19-21	9	6	9.91	151	10.7	0.018
not recorded	not recorded	22-23	6	-	-	-	-	-

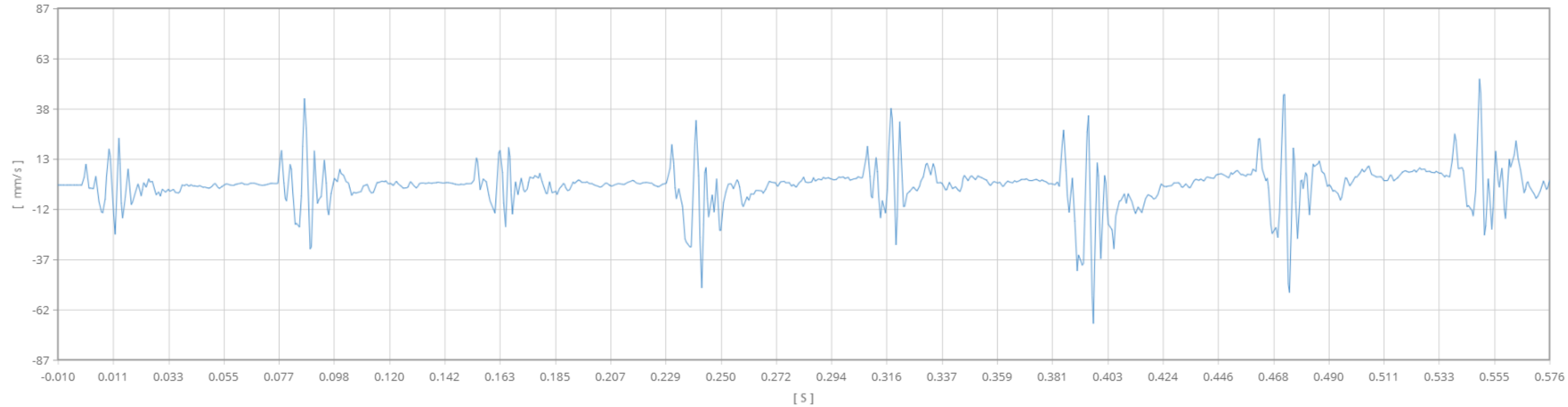
Stope	Vibration Monitor	Installed level						
320k14	JOK4	80						
					Vertical Node			
Date	Time	Rings	Holes	Peaks	PPV (mm/s)	Frequency (hz)	Acceleration (m/s2)	Displacement (mm)
not measured	not measured	1-7	-	-	-	-	-	-
06/09/2018	12:06:12	8-10	14	8	68.5	214	82	0.214
13/09/2018	12:04:40	11-15	23	12	123	77.5	69.6	0.489
20/09/2018	08:44:59	16-18	11	8	86	13.2	29.8	0.72
27/09/2018	09:01:31	19-21	9	6	146	13	84.9	1.17
not recorded	not recorded	22-23	6	-	-	-	-	-

100A22 Waveforms

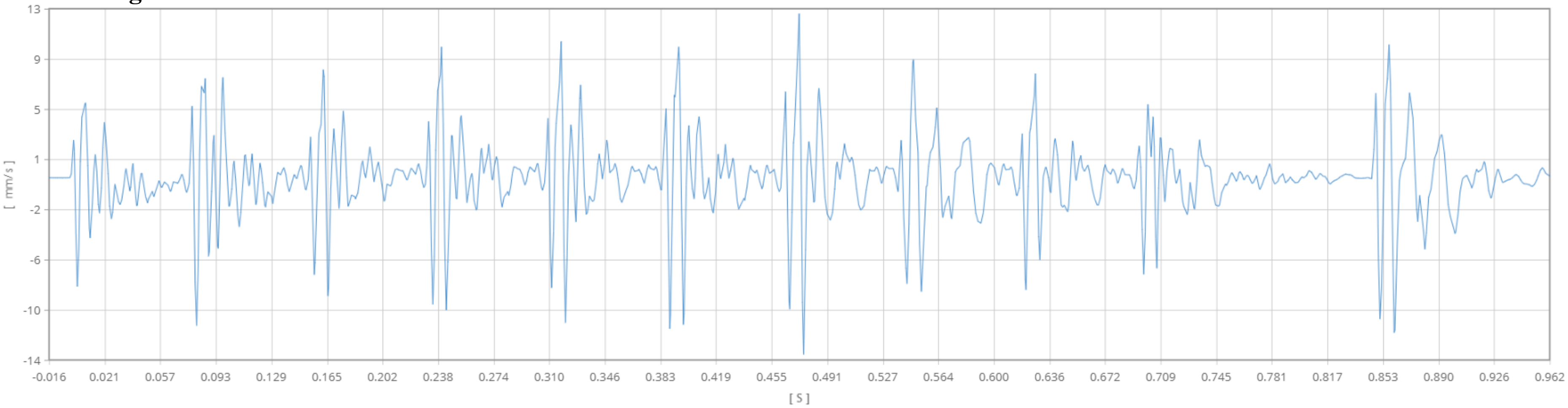
JOK 3: Rings 8-10



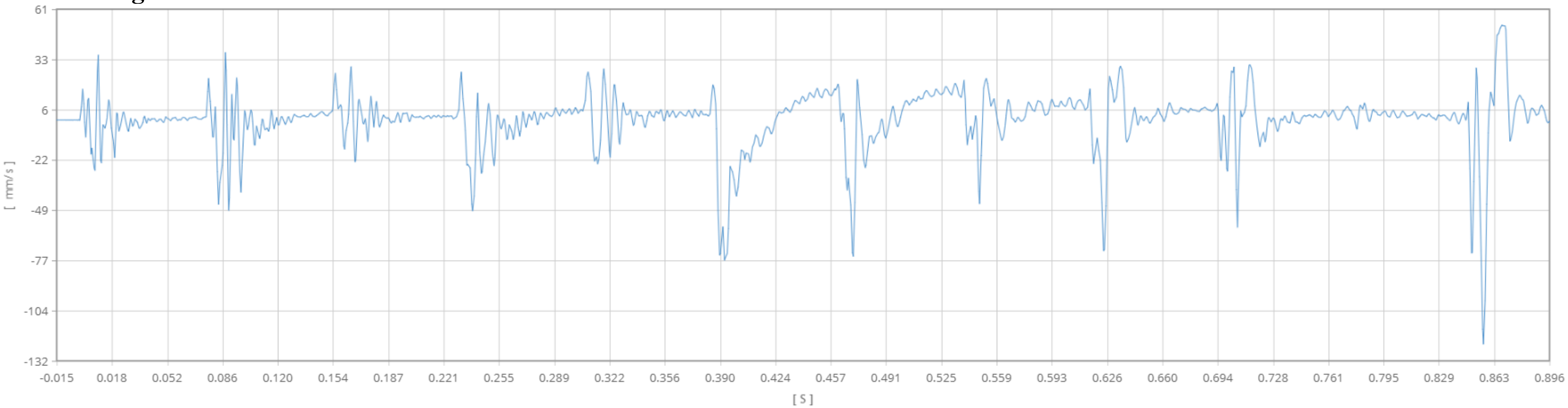
JOK 4: Rings 8-10



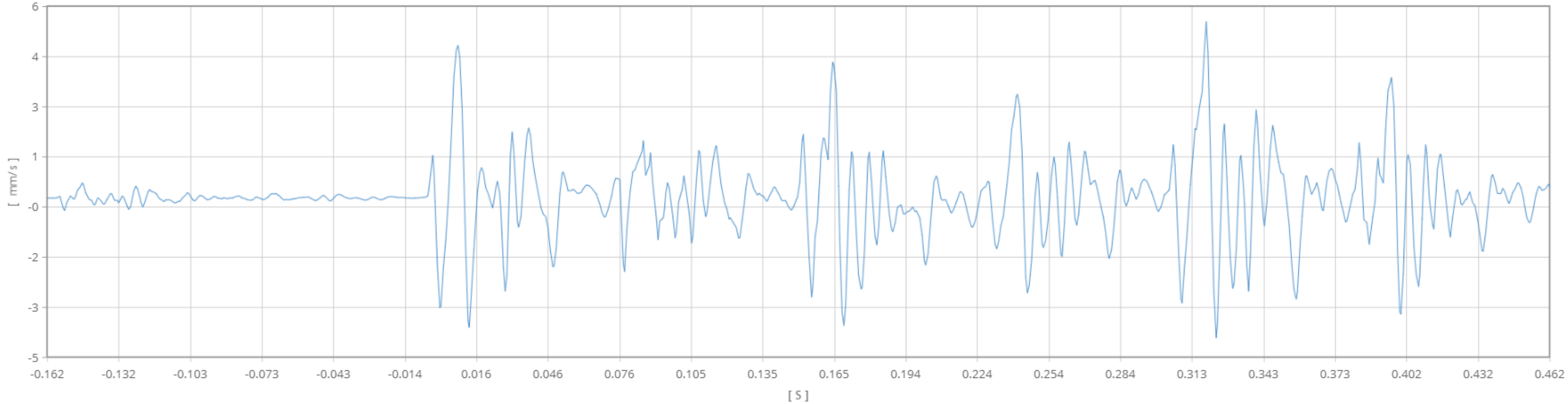
JOK 3: Rings 11-15



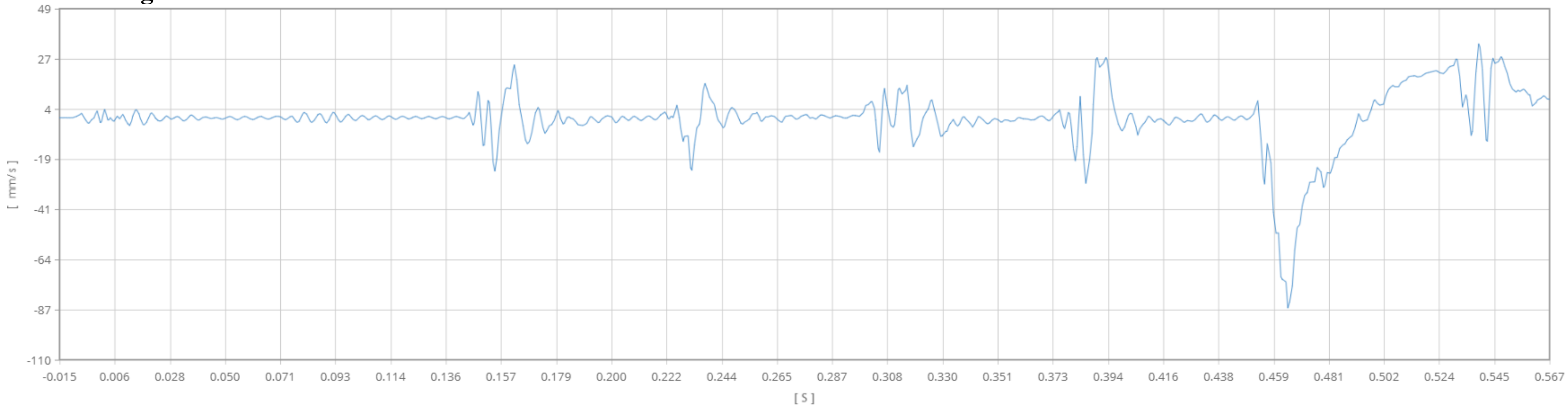
JOK 4: Rings 11-15



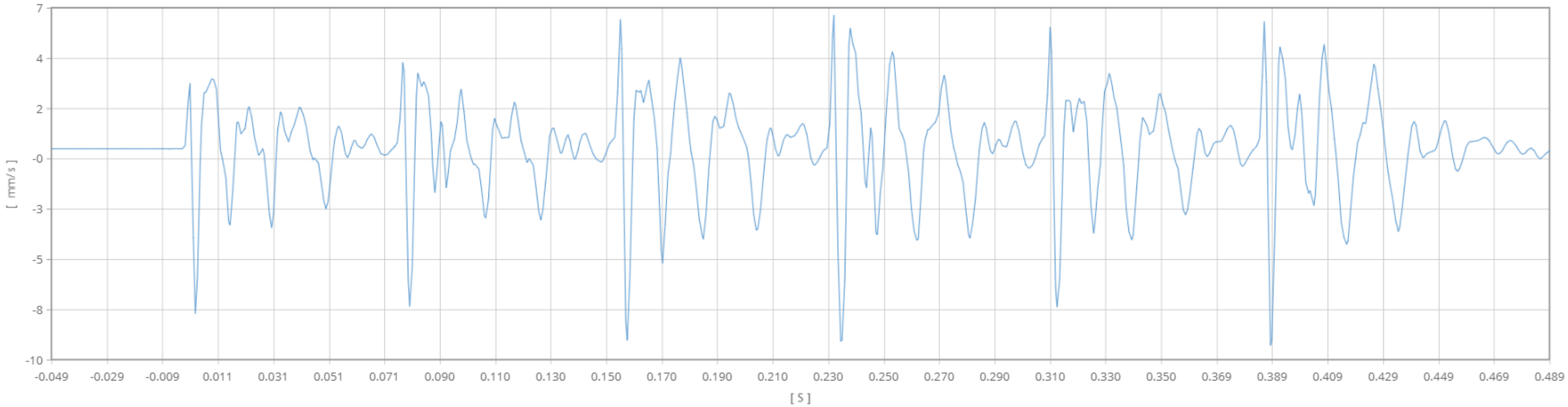
JOK 3: Rings 16-18



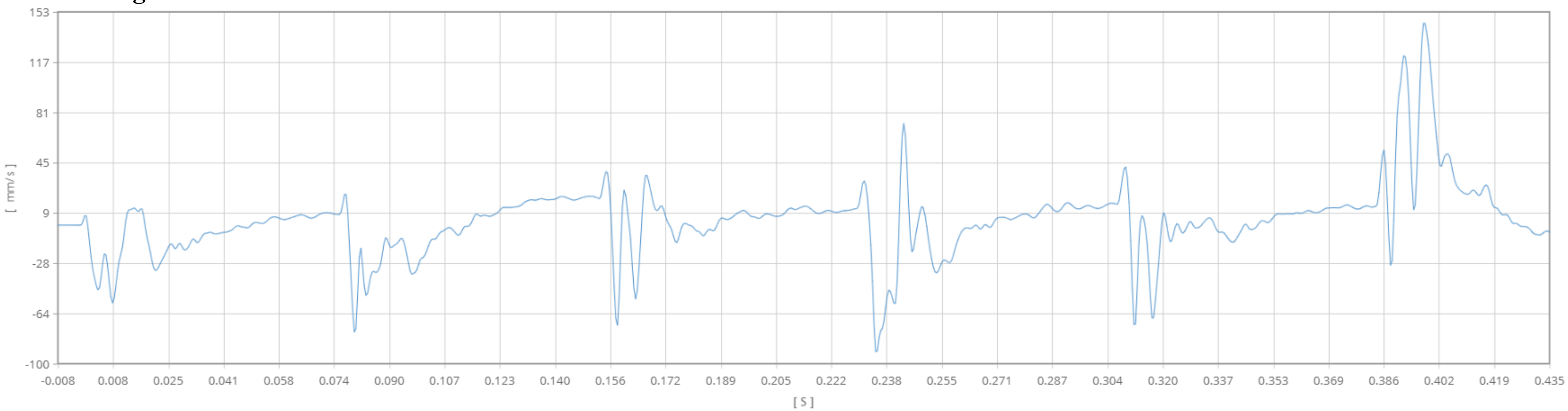
JOK 4: Rings 16-18



JOK 3: Rings 19-21



JOK 4: Rings 19-21



Appendix E – Blasting Specifics

Blasting Schedule Stope 320K14

Stope: 320K14		
Blast #	**Rings**	**Recorded**
1	1-2 (raise)	No
2	1-2 (raise)	No
3	3	No
4	4-5	Yes
5	6	Yes
6	7-9	Yes
7	12-13 (raise)	No
8	12-13 (raise)	No
9	14	Yes
10	15	Yes
11	16	Yes
12	17-20	Yes

Blasting Schedule Stope 100A22

Stope: 100A22		
Blast #	**Rings**	**Recorded**
1	5-6 (raise)	No
2	5-6 (raise)	No
3	1-4 (pre-raise)	No
4	7	No
5	8-10	Yes
6	11-15	Yes
7	16-18	Yes
8	19-21	Yes
9	22-23	Failed Recording

Explosive Quantity Used: Stope 320K14

Stope 320k14	Hole # (numbered from HW to FW)				
Ring #		1	2	3	Total/Blast
1-3	Opening Raise				
4	Explosive Quantity (kg)	56.2	57	67.5	
5	Explosive Quantity (kg)	61.4	61.4	77.8	381.3
6	Explosive Quantity (kg)	64	62.9	69	195.9
7	Explosive Quantity (kg)	76.4	67.5	61.8	
8	Explosive Quantity (kg)	68.9	67	48.1	
9	Explosive Quantity (kg)	72	65.5	61	588.2
Pillar					
12-13	Opening Raise				
14	Explosive Quantity (kg)	66.4	63.6	70.8	200.8
15	Explosive Quantity (kg)	62	58.6	50.1	170.7
16	Explosive Quantity (kg)	61	58	51	170
17	Explosive Quantity (kg)	58	62	51	
18	Explosive Quantity (kg)	39	35	38	
19	Explosive Quantity (kg)	35	30	28	
20	Explosive Quantity (kg)	27	22	24	449
			KG Explosives used		2155.9
			in-situ tonnes		3623.2
			PF (kg/ton)		0.60

Explosive Quantity Used: Stope 100A22

Stope 100A22	Hole # (numbered from FW to HW)						
Ring #		1	2	3	4	5	Total/Blast
1-7	**Opening Raise**						
8	Explosive Quantity (kg)	29.7	39.5	41.7	35		
9	Explosive Quantity (kg)	38.2	40.8	39.1	41.7	43.9	
10	Explosive Quantity (kg)	47.1	39	37.5	46.5	45.3	565
11	Explosive Quantity (kg)	58.2	51.1	44	45.2	53	
12	Explosive Quantity (kg)	56	56.4	54	58.3	54	
13	Explosive Quantity (kg)	57.5	57.8	58.7	57.4	57	
14	Explosive Quantity (kg)	63.8	57.6	66	53.3		
15	Explosive Quantity (kg)	50	52.7	66.5	61.4		1289.9
16	Explosive Quantity (kg)	10	13.8	15			
17	Explosive Quantity (kg)	43.9	34	39.9	45.7		
18	Explosive Quantity (kg)	52	53	56	53		416.3
19	Explosive Quantity (kg)	48	58	60			
20	Explosive Quantity (kg)	60	66	65			
21	Explosive Quantity (kg)	79	72	69			577
22	Explosive Quantity (kg)	43	39	40			
23	Explosive Quantity (kg)	34	31	32			219
					KG Explosives used		3067.2
					Tonnes Rock blasted		6003.2
					PF (kg/ton)		0.51

Detonation Plan: Stope 320K14

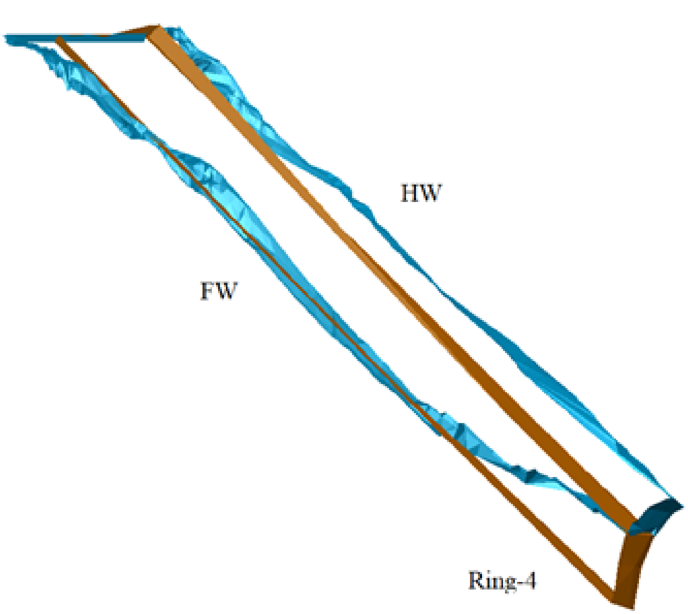
Stope 320k14	Hole # (numbered from HW to FW)				
Ring #		1	2	3	Notes/Remarks
1-3	Blasted with nonel detonators				
4	top (ms)	86	8	163	
	bottom (ms)	78	1	155	
5	top (ms)	317	240	394	
	bottom (ms)	309	232	386	77 ms delay
6	top (ms)	185	108	262	100 ms because FW hole
	bottom (ms)	177	100	254	repairblast from ring 5 first
7	top (ms)	86	8	86	
	bottom (ms)	78	1	78	
8	top (ms)	240	163	240	
	bottom (ms)	232	155	232	
9	top (ms)	394	317	394	
	bottom (ms)	386	309	386	77 ms delay
Pillar					
12-13	Blasted with nonel detonators				
14	top (ms)	61	138	215	extra hole in HW blasted first. Top: 11 ms. Bottom: 1 ms.
	bottom (ms)	51	128	205	One hole has 50 ms delay in hopes of better blast.
15	top (ms)	36	11	61	Extra hole between rings 14&15 blasted second
	bottom (ms)	26	1	51	because of large burden: Delay: 2ms. 25 ms delay.
16	top (ms)	36	11	61	
	bottom (ms)	26	1	51	25 ms delay
	top (ms)	61	11	111	Extra hole between rings 17 and 18
17	bottom (ms)	51	1	101	because of large burden
	top (ms)	211	186	261	top: 161ms bottom: 151ms
18	bottom (ms)	201	176	251	
	top (ms)	361	311	411	
19	bottom (ms)	351	301	401	
	top (ms)	511	461	561	
20	bottom (ms)	501	451	551	50 ms delay

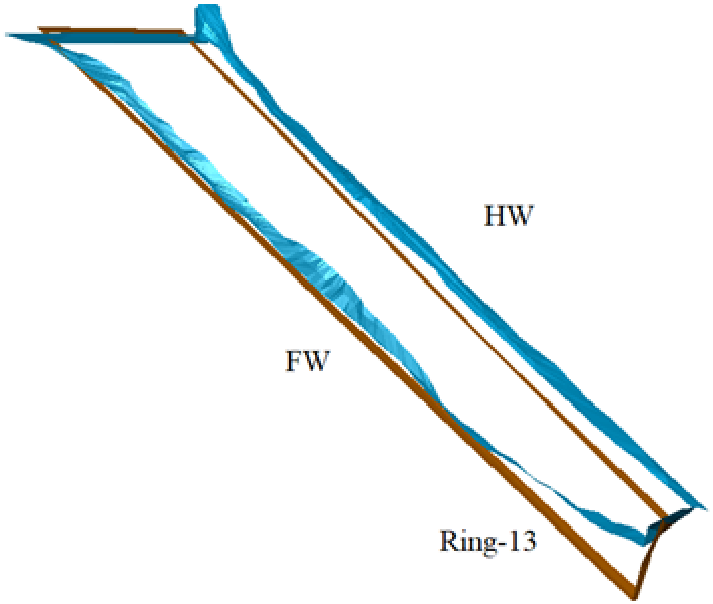
Detonation Plan: Stope 100A22

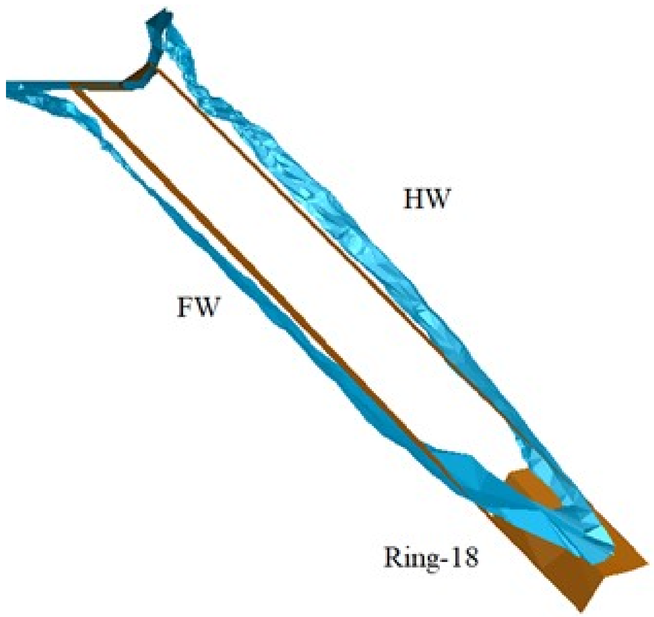
Stope 100A22	Hole # (numbered from FW to HW)					
Ring #	Delay times	1	2	3	4	5
1-7	Blasted with Nonel detonators					
8	top (ms)		83		83	
	bottom (ms)	463	78	1	78	
9	top (ms)	545	468	237	160	237
	bottom (ms)	540	463	232	155	232
10	top (ms)	545	468	391	314	391
	bottom (ms)	540	463	386	309	386
11	top (ms)	545	545	83	6	83
	bottom (ms)	540	540	78	1	78
12	top (ms)	699	699	237	160	237
	bottom (ms)	694	694	232	155	232
13	top (ms)	776	776	391	314	391
	bottom (ms)	771	771	386	309	386
14	top (ms)	853	545	468	545	
	bottom (ms)	848	540	463	540	
15	top (ms)	835	699	622	699	
	bottom (ms)	848	694	617	694	
16	top (ms)	-	-	-		
	bottom (ms)	78	1	78		
17	top (ms)	-	-	-	-	
	bottom (ms)	232	155	232	309	
18	top (ms)	-	-	-	-	
	bottom (ms)	463	386	463	540	
19	top (ms)	83	6	83		
	bottom (ms)	78	1	78		
20	top (ms)	237	160	237		
	bottom (ms)	232	155	232		
21	top (ms)	391	314	391		
	bottom (ms)	386	309	386		
22	top (ms)	83	5	83		
	bottom (ms)	78	1	78		
23	top (ms)	237	160	237		
	bottom (ms)	232	155	232		

Appendix F – Stope Scan Cross-Sections and Drillhole Deviations

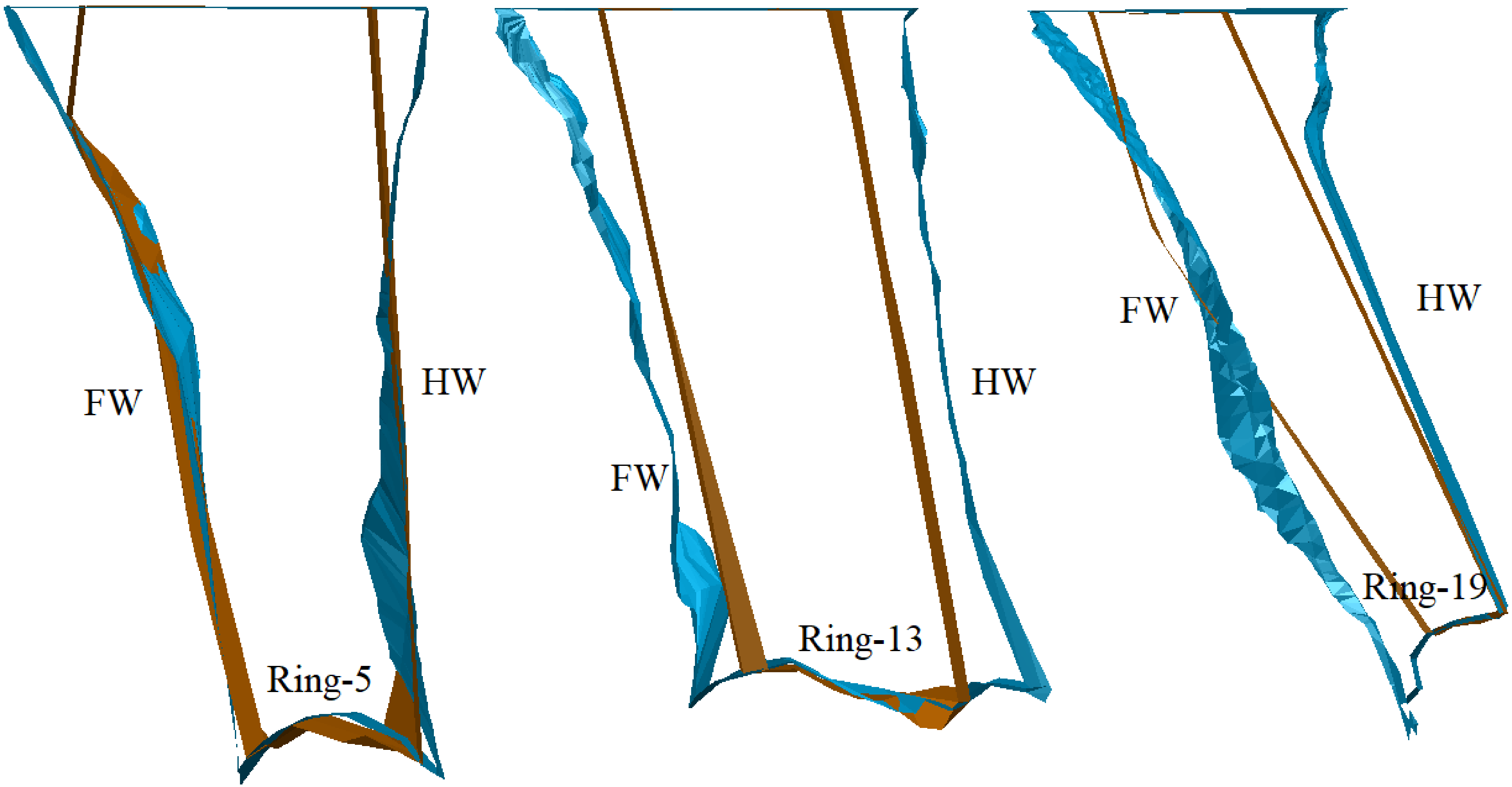
Cross sections: Stope 320K14



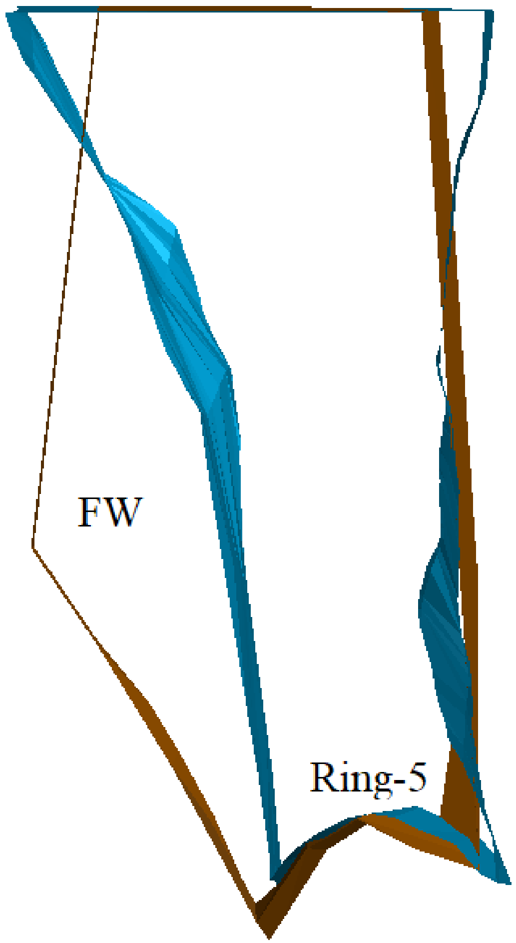


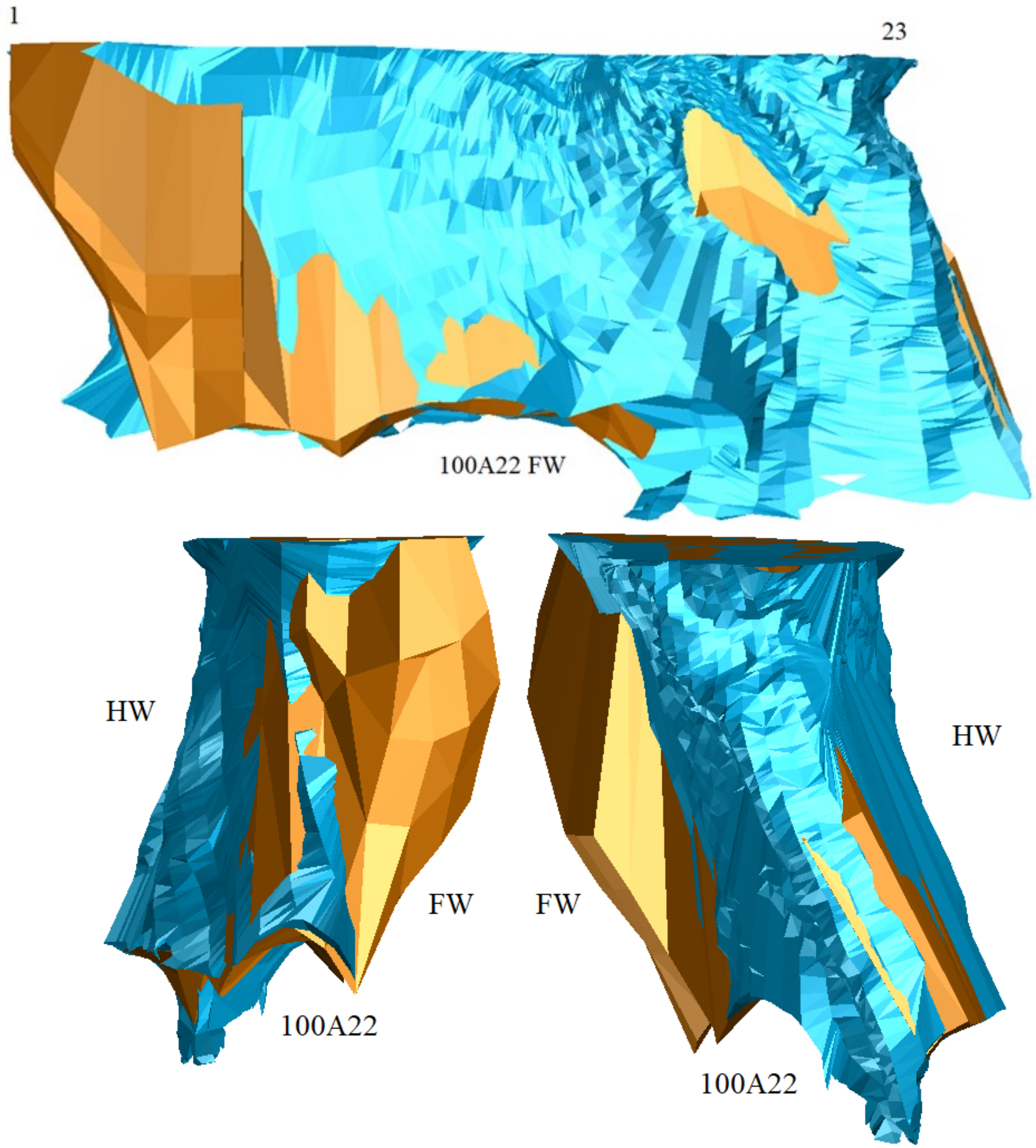


Cross sections: Stope 100A22

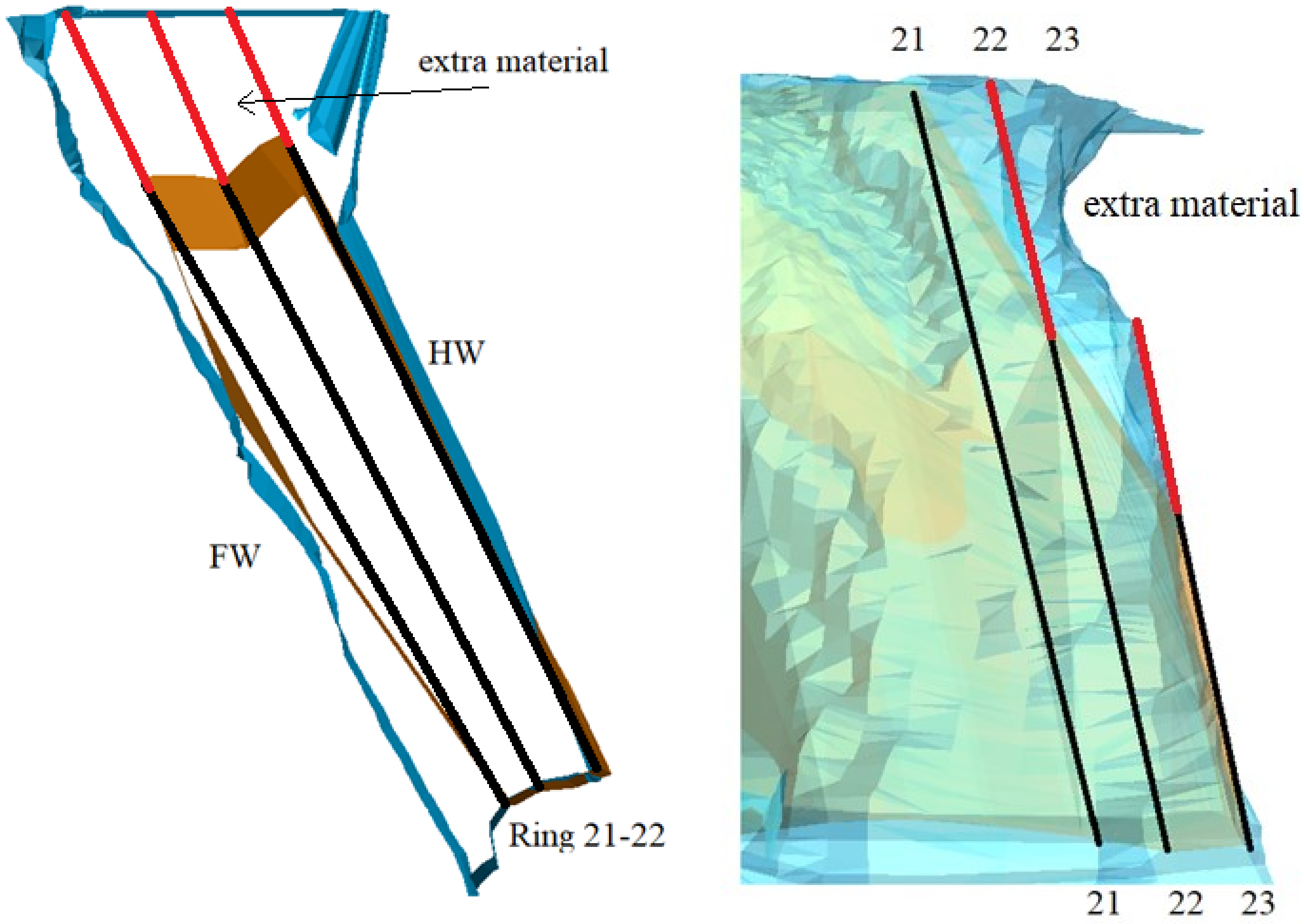


100A22: Visualisation of ore loss rings 1-6

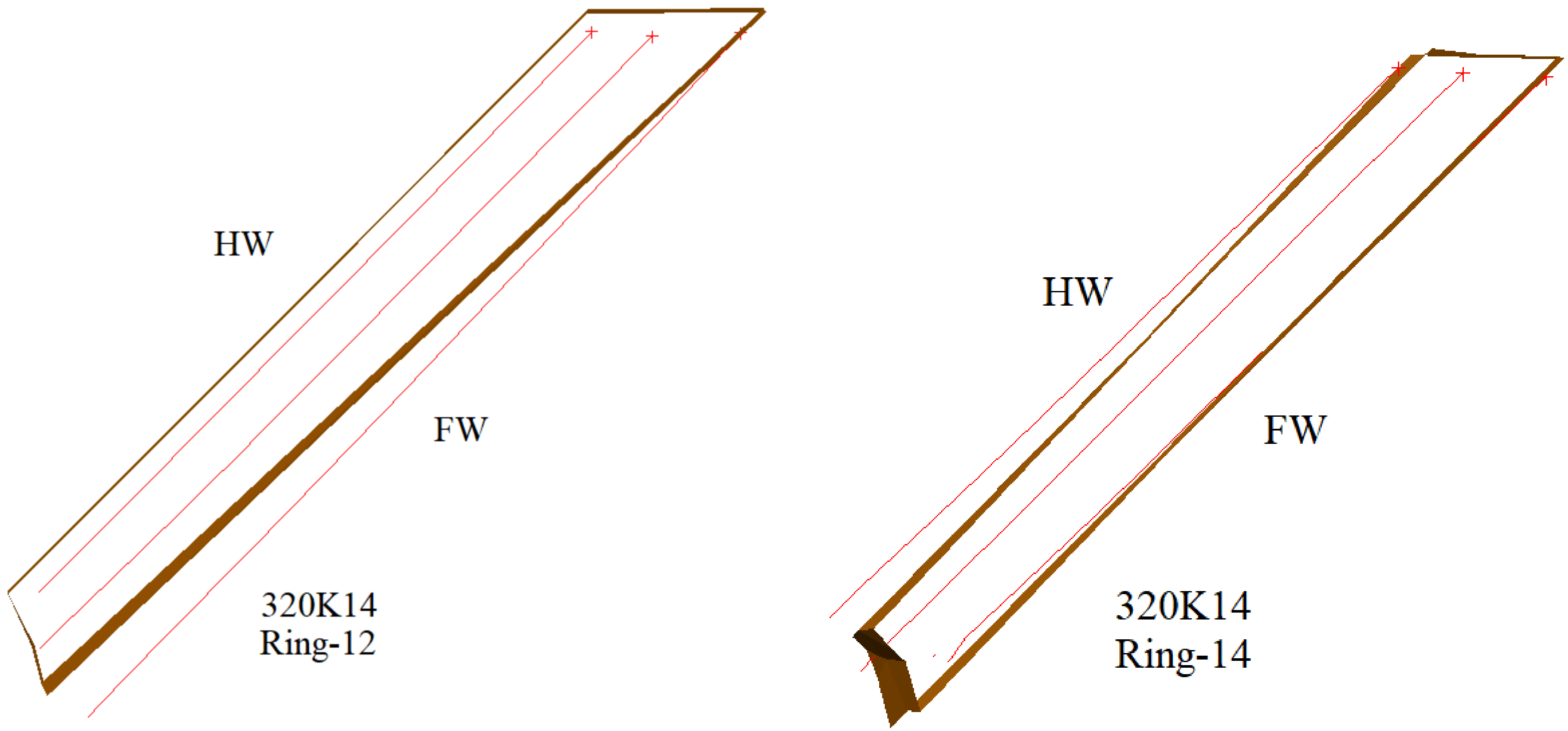




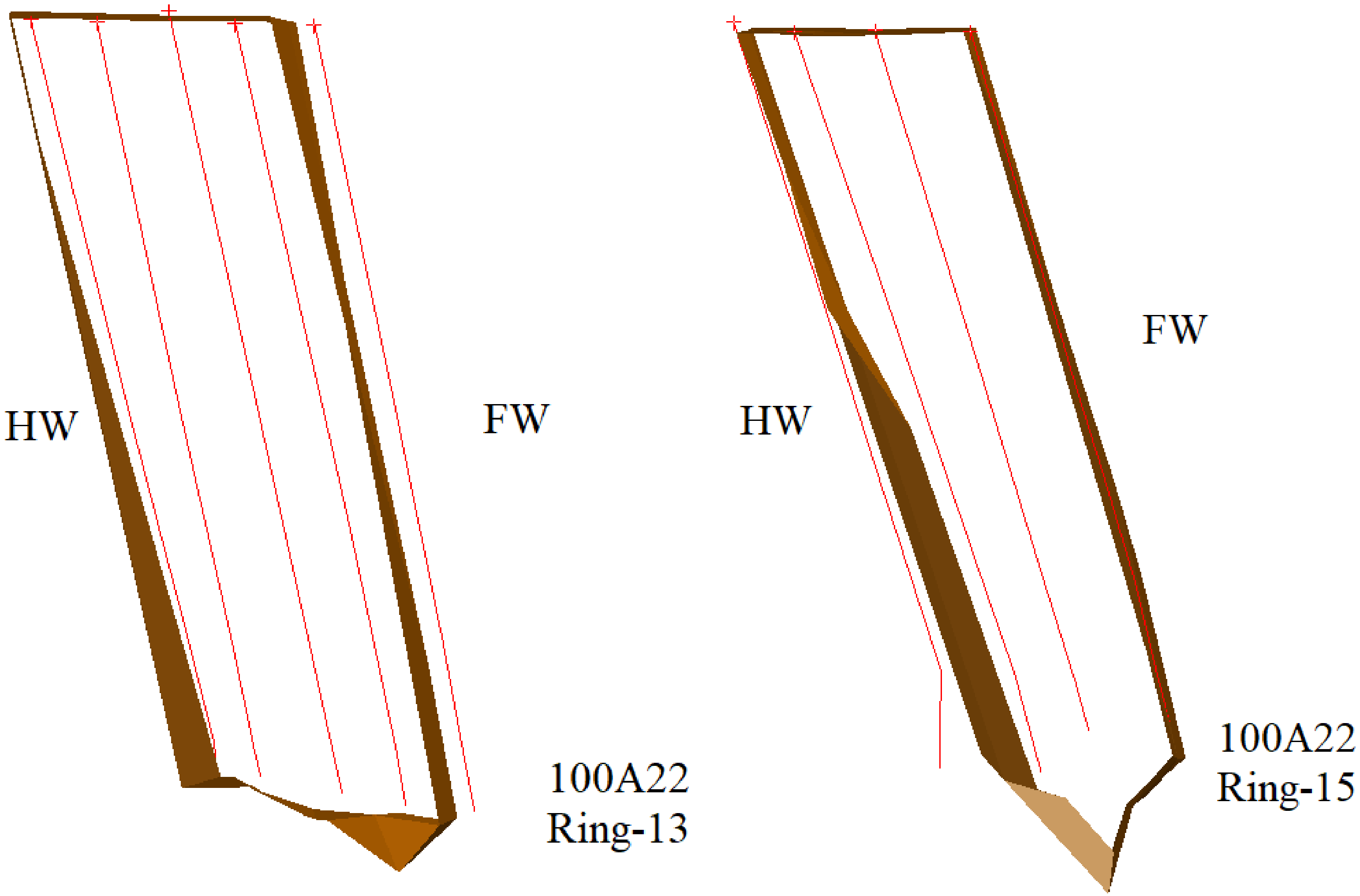
100A22: Visualisation of drilling errors rings 21-23



320K14: Drillhole deviation (red) versus designed stope outline (brown)



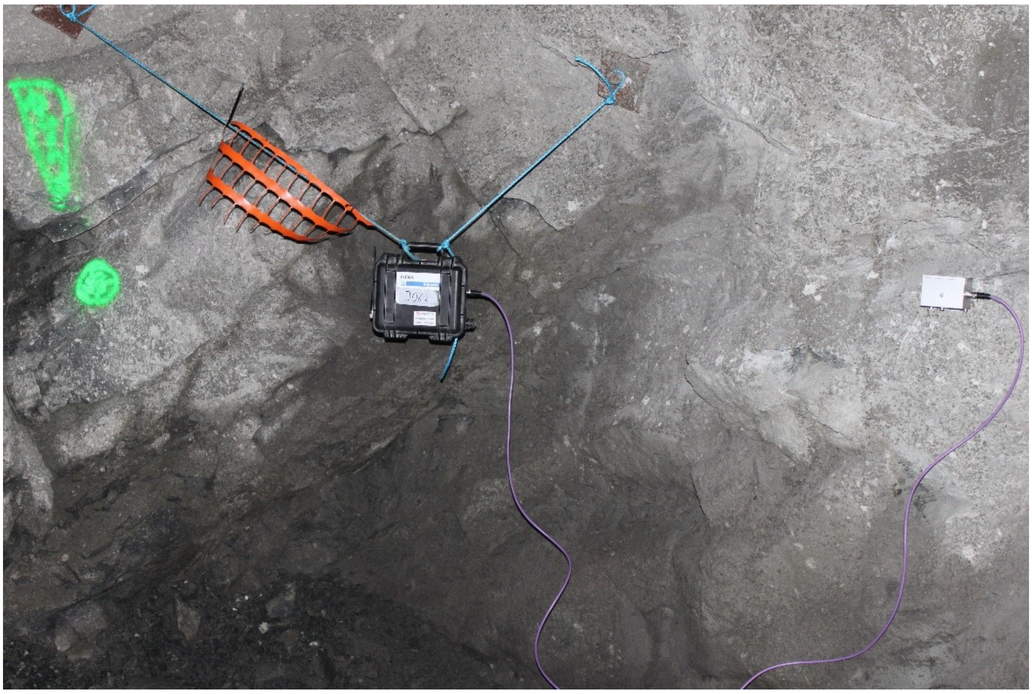
100A22: Drillhole deviation (red) versus designed stope outline (brown)



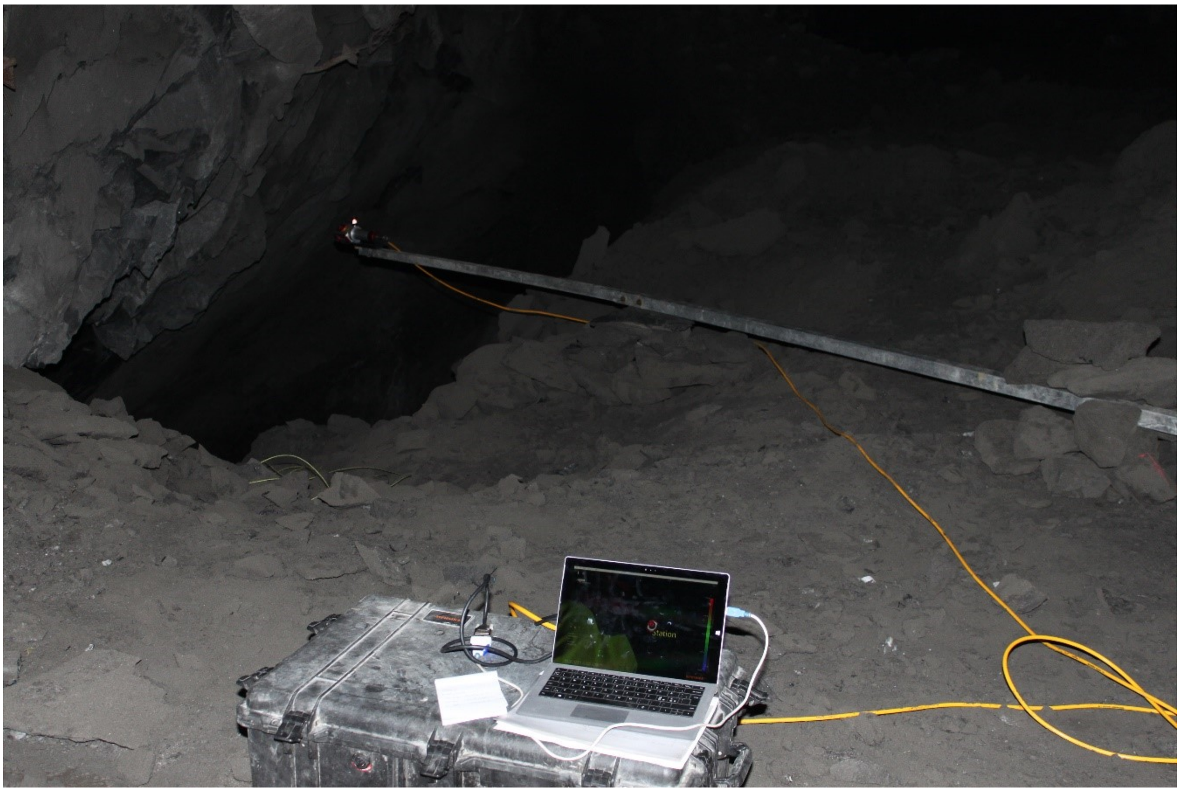
Appendix G –Stope Photos

320K14 Photos

Failed detonation of footwall hole in 320K14 ring 15 as described in section 6.3.



Installation of Vibration Monitoring set-up JOK 2 in the 300 level



320K14 Stope scanning rings 14-20 post pillar.

320K14 Stope rings 4-5 charged holes with programmed electronic detonators in place.

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100A22 Photos

100A22 Footwall fault plane between rings 11-15 (on the right of the picture) next to the pillar (left) as described in section 6.5

100A22 Fractured Hangingwall (right) photographed during stope scanning of rings 1-21.

100A22 left hand side of pillar. Surprisingly intact, as part of it was designed to be mined, due to ore contact.

Large overview of 100A22 pillar left hand side. Installed cablebolts keeping the pillar in place.