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Improvement of blasting practices to minimise wall damage at Bozshakol copper mine

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Abstract

Wall damage in large open pit mines is often induced by adjacent blasting activities as a consequence of improper blast design considerations on unforeseen geotechnical conditions. Post-blast damage may vary from slight to severe levels, which are represented by the presence of tension cracks to significant back-break causing crest deterioration or, even worse, loss of the berm itself. Wall control blasting practice is essential in minimising the effect of wall damage so as to reduce geotechnical risks related to rockfall potential, single to multi-bench stability and, less frequently, overall pit wall stability. On some occasions, associated risks may have the potential to impact on critical mine infrastructure such as long-term haulage ramps. Significant economic costs are associated with remediation works required to mitigate rockfall risks, which may range from support measures for rockfall containment to mine design changes and deviations from the mine schedule that typically result in either additional waste removal and ore deferral or sterilisation.

This paper discusses the improvement of wall control blasting practices at Bozshakol open pit mine, located in Kazakhstan, Central Asia. It particularly focuses on trim blasting and pre-splitting to minimise wall damage by considering the effects of vibration and blast energy factors within various geotechnical domains. Input of geotechnical information including rock mass characteristics and geological structures also plays an important role for proper wall control blast design considerations. Vibration monitoring is also carried out to assess ground vibration levels expressed by peak particle velocity and establishing site-specific correlations for recommending maximum charge weight per delay.

Despite prioritising for wall protection purposes, blasting also needs to achieve a level of fragmentation that supports efficient and productive mining rates. With collaborative interdisciplinary efforts, it is expected that fragmentation needs will be met while achieving final wall design within the tolerance of toe-crest position, bench face angle and berm width while minimising wall disturbance so as to provide a more reliable mine plan.

Keywords: *wall control, blasting, blast design, blast vibration, vibration monitoring*

1 Introduction

Bozshakol copper mine is located approximately 250 km to the east of Astana in North Kazakhstan as shown in Figure 1. It has an annual ore processing design capacity of 30 million tonnes and a remaining mine life of around 40 years at an average copper grade of 0.33%. The project is being developed as an open pit mine with an onsite concentrator and a clay plant.

In the mining production cycle blasting is the most economical and efficient method to fracture and move the ground (Silva et al. 2016). Blasting also adversely affects the in situ rock mass in the slope behind the blast limits, however, the blast damaged zone is typically less than a distance of 2.5 times the blasting bench height, even for large, confined production blasts (Hoek & Karzulovic 2000). As such, it is commonly accepted that blasting has less influence on larger scale slope stability than geological conditions and groundwater (pore pressure). Due to the nature of blasting, it is inevitable that the rock mass behind the slope face in an open pit operation will be damaged to some degree. Along with rock mass damage, ground vibrations are another undesirable consequence of blasting.

Negative impacts from blasting in large open pit mines may include:

- Bench scale hazards, instability and failures.
- Rockfall risk resulting from repeat bench scale instability and the loss of berm reliability.
- Larger scale slope stability issues on sensitivity slopes (e.g. causing undercutting of major geological structures or disrupting rock bridges between the slope and non-daylighting faults).

The latter two points may be particularly troublesome from an economic perspective by impacting on essential mine infrastructure such as haulage ramps or, in cases where inter-ramp slope designs are governed, by bench geometry requirements (Alejano et al. 2007; Haile et al. 2020). Where geological conditions permit the design of steep inter-ramp and overall slopes, achieving them requires proactive rockfall risk management. This requires consistent bench geometry conformance during the excavation of steep, smooth bench faces which reduce the likelihood of 'launch features' that may induce significant horizontal rockfall trajectories (Bar et al. 2016). In such cases, blast design has increased relevance as a variable to consider as failure to do so may result in significant additional expenditure to manage hazards that may otherwise be avoidable. By way of example, such risks may need to be managed by additional ground support measures for rockfall protection or design changes that would economically impact short-term or life of mine plans (e.g. step-ins or additional pushbacks).

Figure 1 Location of Bozshakol copper mine

Safety aspects of bench scale hazards, including brittle failures, are becoming evermore manageable through a combination of safe operating practices that minimise personnel exposure to the line-of-fire of potential rockfalls near the slopes, and detection through real-time monitoring and response protocols (Bar et al. 2022).

Meeting fragmentation needs while maintaining wall control blasting is a critical aspect for both short- and long-term mine productivity and the overall profitability of a mine. Steep slope design for reducing waste material movement is only possible with effective mining practices, which begin with effective blasting.

2 Geology and geotechnical conditions

Bozshakol copper mine is located on the western part of the Central Asian Orogenic Belt (Shen et al. 2015). It comprises a Cambrian-Ordovician, high-level intrusive complex emplaced into the east-northeast-trending Bozshakol anticline (Figure 2). Major geological units in the area include Lower-Middle Cambrian sandstones and intermediate-mafic volcanic rocks (lavas and tuffs), unconformably overlain by Upper Cambrian to Lower Ordovician age mainly clastic sediments. The Lower and Middle Cambrian formations are intruded by granodiorite to tonalite porphyry and porphyritic dykes, which are spatially associated with mineralisation (AMC 2010).

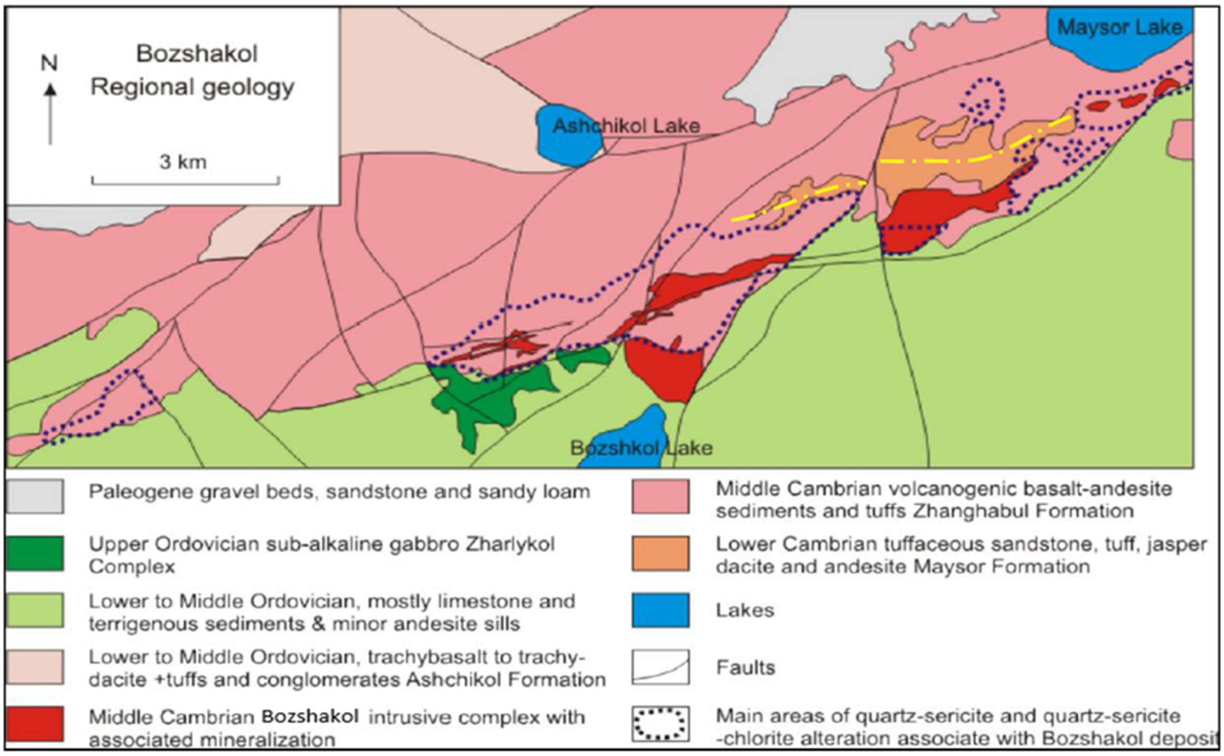
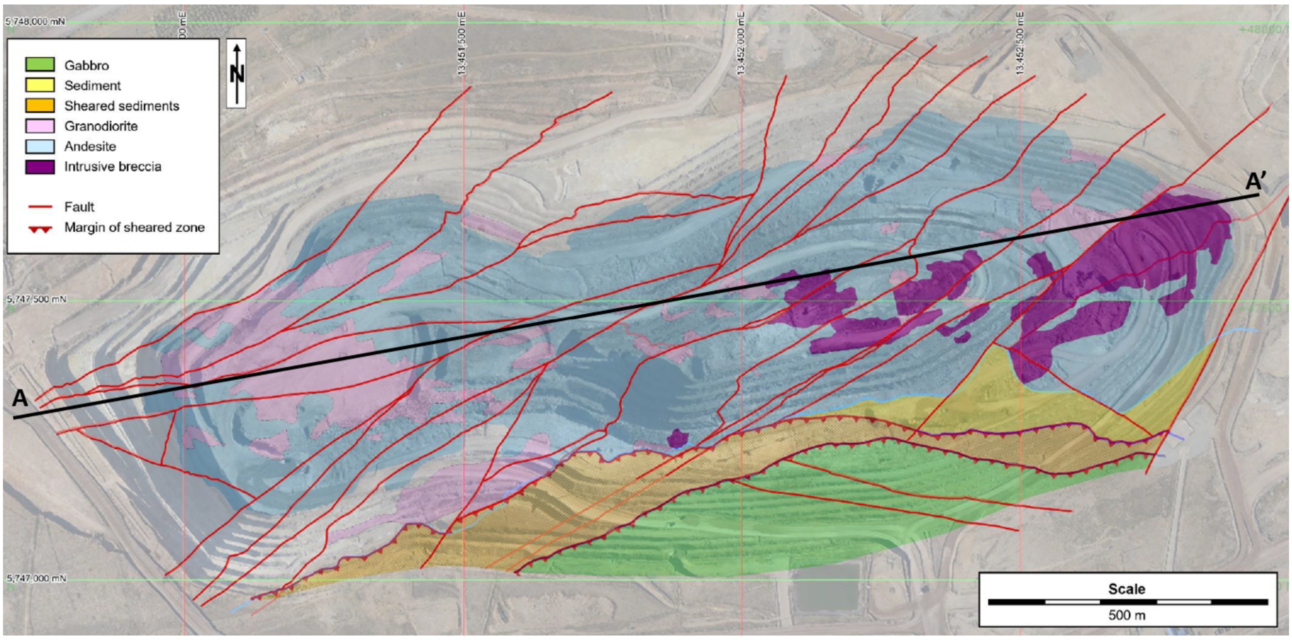
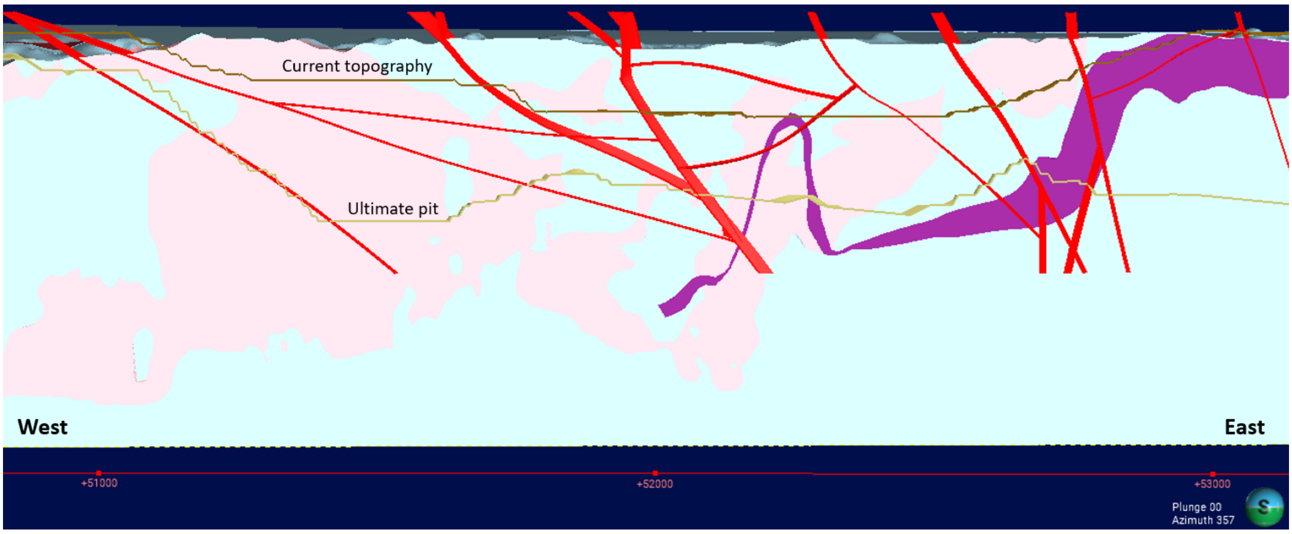


Figure 2 Local geology of the Bozshakol porphyry copper deposit. Yellow dashed line indicates the approximate location of the core of the Bozshakol anticline (Kudryavtsev 1996)

Bozshakol geological models simplify lithology units into eight main groups consisting of sediments, gabbro, andesite, granodiorite, breccia, diorite, saprolite and clay. The dominant fault systems strike in the direction of east-northeast–west-southwest and northeast–southwest, and are sub-vertical to steeply dipping faults that have a general tendency to dip towards the southeast (Tektonik Consulting Limited 2022) as shown in Figure 3.



(a)



(b)

Figure 3 Surface summary map showing principal features of the Bozshakol Central Pit in (a) Plan view; (b) Cross-section

Geotechnical domains adopt simplified lithology units and the weathering profile as the basis. These domains are further divided into eight different geographical sectors which also represent different jointing patterns (minor structure) to form slope design sectors or domains. The mine plan is guided by slope design sectors for implementation of various slope geometries.

Determination of slope design sectors in Figure 4 considers the pit geometry with respect to the location and orientation of major faults, similarities of geotechnical characteristics and rock mass strength properties. The presence of unfavourably orientated faults could potentially affect the stability of pit walls in both fresh and weathered rock masses. Additionally, in weak and weathered rock masses such as oxides, saprolites and clays, rock mass strength can affect the stability of the pit walls. Minor structure including jointing, and less-pervasive faults and shears, form the basis of bench design geometry.

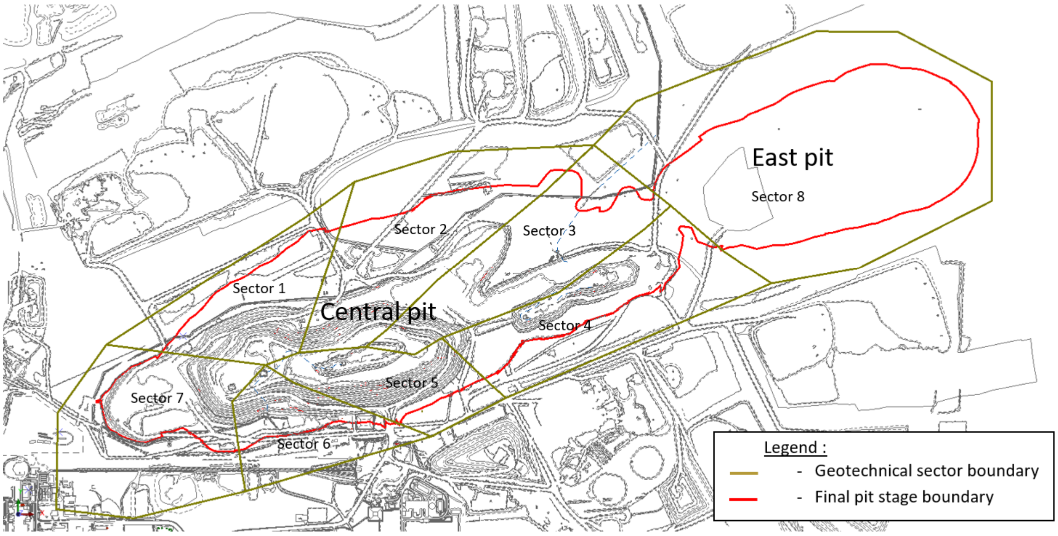


Figure 4 Slope design sectors' overlay to pit topography and the boundary of the final pit stage

Geotechnical strength and stiffness parameters are shown in Table 1 and have been derived from several site investigation campaigns including diamond drilling, face mapping, laboratory testing and slope failure back-analyses.

Table 1 Geotechnical strength and stiffness parameters

Domain		Class	Sector	γ kN/m³	c' kPa	ϕ' °	σ_c MPa	GSI	m_i	D	E_i Gpa	ν	σ_t Mpa
Clay		1	1, 6, 7	17	22	33							
		2	2, 3, 4, 5	15	25	31							
Saprolite		1	1, 6, 7	17	22	39							
		2	2, 3, 4, 5	22			3	20	8	0–0.7	0.42	0.31	0.1
Andesite			All	29			48	60	25	0–0.7	41	0.25	9
Breccia			All	28			43	55	19	0–0.7	38	0.30	6
Diorite			All	28			72	65	25	0–0.7	38	0.28	9
Gabbro			All	27			14	45	15	0–0.7	14	0.24	4
Granodiorite			All	28			50	65	29	0–0.7	26	0.28	5
Sedimentary	Rock mass	1	6	20	25	34							
	Bedding	1	4, 5	16	5	16							
Sedimentary	Rock mass	2	6	28			48	30	22	0–0.7	20	0.26	6
	Bedding	2	4, 5	16	5	16							
Faults			All		1	22							

3 Current wall control blasting practices

Current blasting practices are given sufficient attention to maintain wall stability. Trim blasts are implemented to all final and interim slopes within all geotechnical domains with the exception of clay and weathered rocks in which free dig is feasible.

Pre-split blasts are implemented for walls that are standing for at least three years within the fresh geotechnical domains. In some instances, due to narrow production areas, trim and pre-split blasts are combined with modified production blasts.

Geotechnical input has been integrated into drilling design for trim and pre-split blast but requires ongoing improvements to minimise blast-induced wall damage and achieve an optimal degree of fragmentation. In order to facilitate continuous improvement at Bozshakol, a multidisciplinary approach to blasting is applied and involves mine planners, drill and blast engineers, geotechnical engineers and mine operations following the structured workflow in Figure 5.

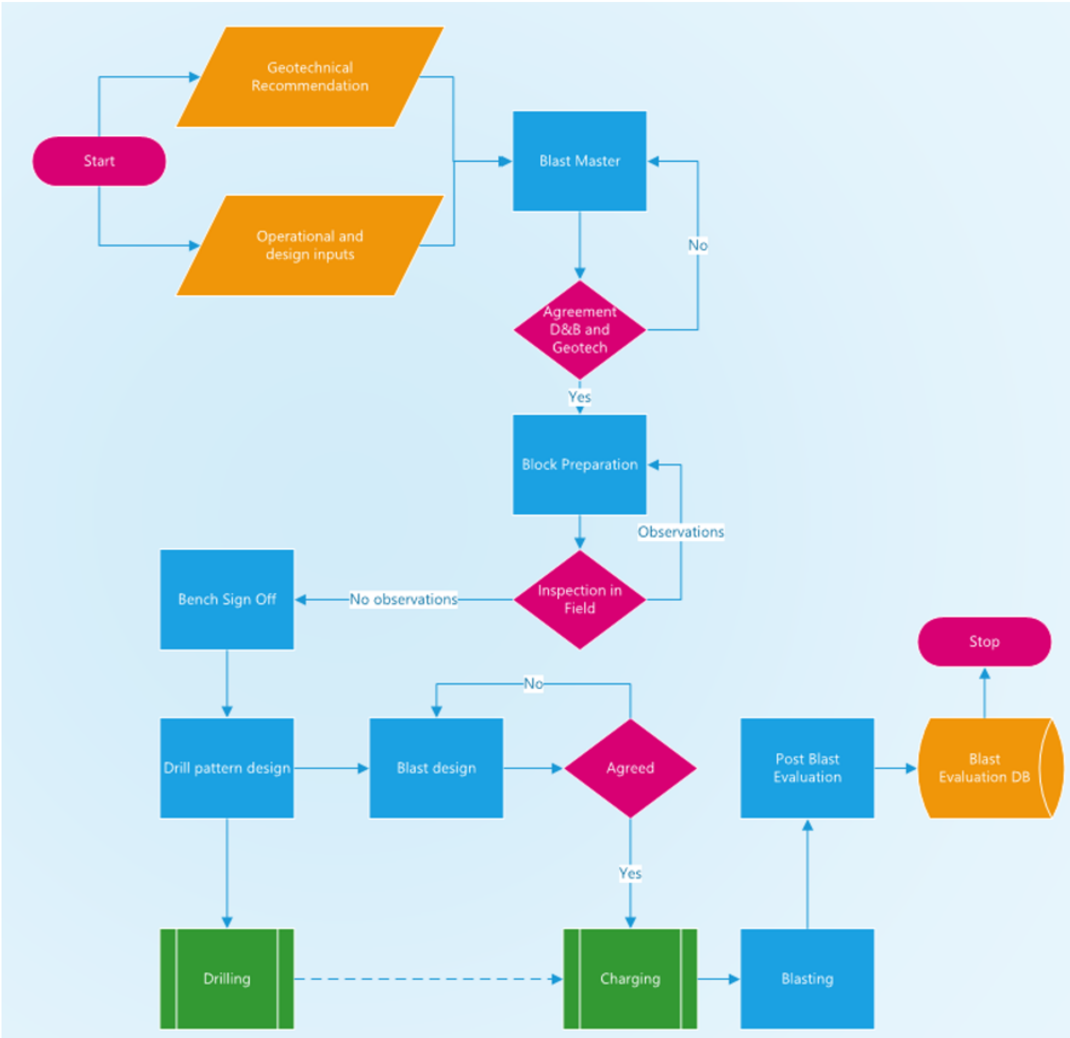


Figure 5 Integrated drill and blast workflow at Bozshakol

The integrated drill and blast workflow ensures that several specific blast techniques are applied to reduce blast-induced damage such as pre-split blasting and trim blasting with a buffer row adjacent to the slope. It also ensures that the concept of the explosive energy efficiency illustrated in Figure 6 is followed, with preferably two free faces for trim patterns to distribute and adequately release confined explosive energy.

Figure 6 Explosive efficiency triangle (Read & Stacey 2009)

3.1 Trim blasting

Trim blasting is the most commonly used controlled blasting technique. It can provide good results in a wide variety of geological conditions as long as the designs are refined in a logical manner. The explosive distribution, confinement and level can be enhanced with the use of air decks, pattern modifications and/or reduced hole diameters (Read & Stacey 2009).

Trim blast design at Bozshakol normally consists of four rows and is shot in the direction of open free faces. Current trim blast practices are carried out on every 10 m bench per flitch, or else there are top and bottom flitches for 20 m double-bench slope geometry.

Figure 7 shows a typical existing trim blast design for fresh rocks. The last row adjacent to the wall is considered as a buffer. The main purpose of the buffer row is to decrease the blast energy in order to minimise crack extension damage propagated to the designed slope. Therefore the buffer row spacing is narrower, and is given less explosive charge.

In some circumstances, air deck is commonly used in the buffer row to reduce blast energy and to increase fragmentation in the top part of the sub-bench. In the same sense, less stemming length will be applied hence energy confinement is lower. A stand-off distance of 0.3 m is implemented at the bottom flitch design for offsetting the toe and avoiding toe undercut. For other types of bench parameters, and depending on the ground condition, different blast designs can be adjusted and efficiently implemented.

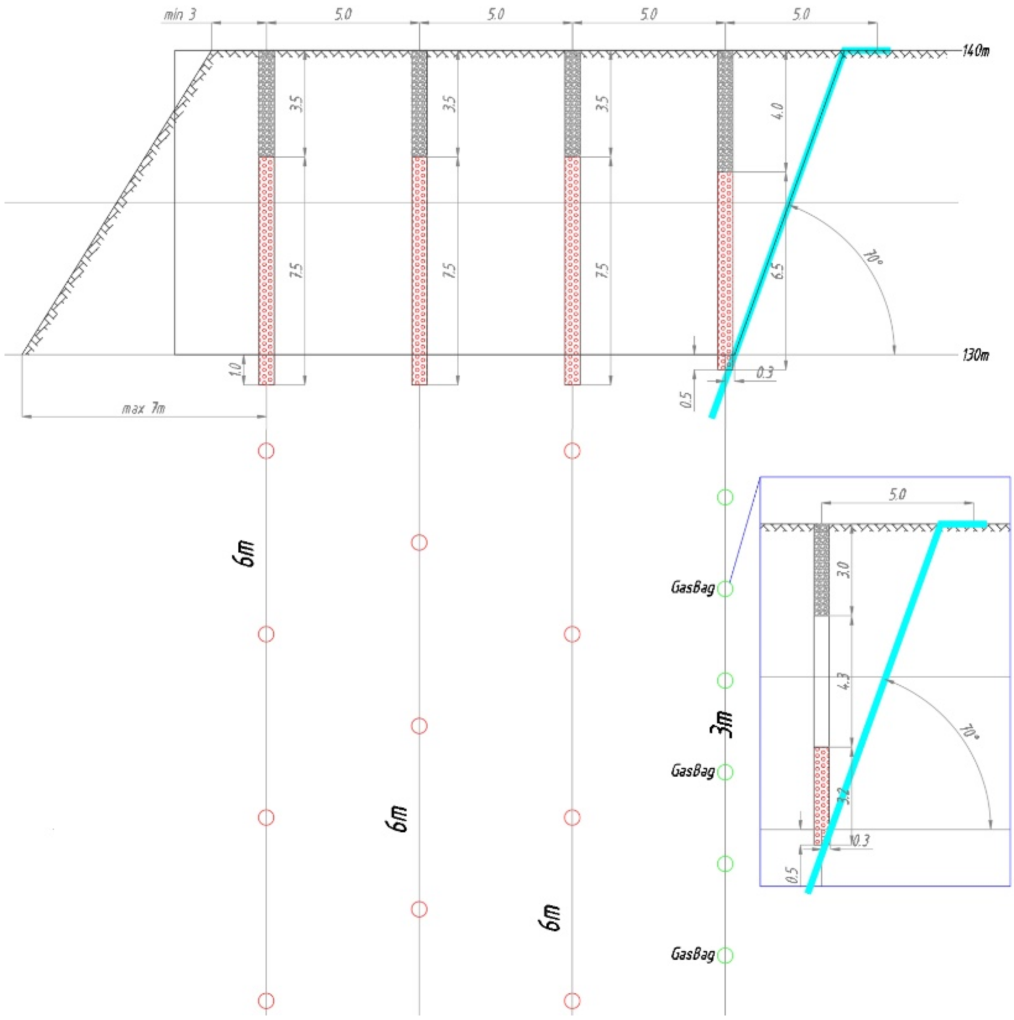


Figure 7 Template of the trim blasting design adopted for fresh rocks

3.2 Pre-split blasting

Pre-split blasting involves drilling a row of closely spaces holes along the designed dig limit. These holes are loaded with decoupled charges to split the gap between holes in tension without causing compressional damage to the slope (Read & Stacey 2009). In current practice at Bozshakol, the pre-splits are shot in advance prior to performing trim blasting for the final wall (or the wall that will stay in place for at least three years as per the mine plan schedule).

In order to minimise damage produced by tension cracking and crack extension behind the trim pattern, pre-split blasting is implemented with immediate 20 m long holes (double benching) which have been drilled at typical 1.5 m spacing following the proposed bench face angle of between 65 and 75°. A 15–18 kg charge weight of continuous cartridge emulsion explosive allows splitting of the rock mass. The diameter of the pre-split holes is 165 mm, which is slightly smaller than the 177 mm diameter of the trim blastholes.

In adverse geological conditions, decoupled charges provide better energy distribution. As stemming the pre-split holes will cause over-confinement, and following the blast efficiency triangle concept, unstemmed pre-split blasting is applied.

3.3 Pre-blast assessment

Pre-blast assessment includes the following:

- Recommendations for the sequence of mining trim patterns.
- Blast direction.
- Overlay of major structure (faults) and minor structure (stereonets for sectors).
- Plan sections of lithology, hardness class, geotechnical blockiness index, rock quality designation and blastability index from 3D wireframes and a block model.

The purpose of the pre-blast assessment is to provide in advance all the necessary geotechnical input data the drill and blast engineer requires for their controlled blast design. Sequence mining of production blast patterns directly affects which side of free face will be opened for trim patterns to follow the proper trend direction of structures. Since mine production priorities are not always consistent with geotechnical recommendations, the decision to blast towards free faces is often accepted when the future wall has already been pre-split. This practice assumes that all of the geological structures, particularly persistent low-to-moderate dipping joints and faults, are sufficiently cut or disrupted by the pre-split to prevent reactivation or crack propagation from adjacent blast energy.

In most of cases at Bozshakol, the understanding and integration of structural orientation data are critical due the poor quality of some joint sets, which have to be protected from both back-break due to heaving of the muck pile and, more importantly, excessive vibration that can potentially trigger cracks along weak infilled joints dipping into the pit.

3.4 Post-blast assessment

The performance of each controlled blast should be monitored and analysed to make sure the design is refined to meet changing slope conditions. Post-blast assessments consist of:

- Visual observation in association with the blast result or overbreak that may influence the berm-batter integrity.
- Evaluation of bench design achievement based on geometric design compliance and face condition.

Post-blast evaluation is a good retrospective tool to quantify the blast results for different ground conditions, i.e. performance verification of the blast design. This evaluation considers reaching the geometric design toe and crest, an absence of back-breaks on the berm and visible intact rock breakage on the bench face, an absence of overhanging blocks and loose material, and the existence of half barrels (from pre-splitting).

Evaluated parameters are compiled in a table and entered into a chart to review the overall quality of the results; as shown in Figure 8, and adopted from *Guidelines for Open Pit Slope Design* (Read & Stacey 2009) with a slight modification to adjust for Bozshakol site considerations.

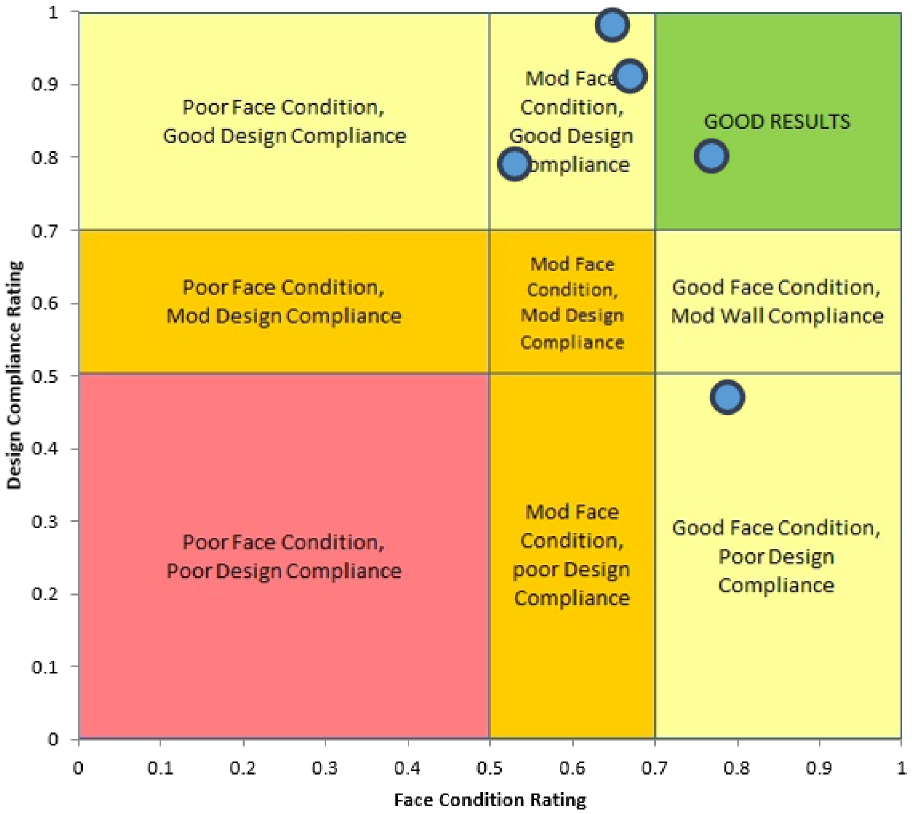


Figure 8 Example of the blast results evaluation chart

4 Blasting challenges

Three principal mechanisms exist in which rock blasting can impact on the stability of nearby rock structures and slopes (Holley et al. 2003). These are:

- Generation of new fractures and cracks in previously intact rock (vibration controlled).
- Dilation of joints by the action of high-pressure explosion gases.
- Promotion of slip along unfavourably orientated joint and fracture surfaces (vibration controlled).

It is important to recognise that the first two mechanisms described above are near-field effects occurring close to the blast zone, whereas the last mechanism may occur at distances of tens of metres from the blast zone.

The main causes of the aforementioned mechanisms impacting the stability of geological structures on Bozshakol are (Cebrian 2022a):

- Blast-induced vibrations by pre-split shooting 50 holes per delay.
- Vibrations induced by buffer and first production/trim charges.
- Vibrations/confinement due to too much burden on the free face.
- Sub-drilling close to the crest underneath.

Figure 9 shows an example of back-break detected after a trim shot in which the consequence was the loss of almost the entire catch berm. The back-break (and subsequent bench failure) propagated along a weak, steep, in-pit dipping fault with gouge infill within the saprolites.

Typical consequences of failures due to back-breaks are:

- Delays to mine production and the schedule, and the requirement for additional geotechnical controls.
- Operational risks associated with the rehabilitation works.
- Additional costs related to the remediation works.
- Need for design changes that may affect the mining sequence and ore feed (downstream effects).
- Repeat back-break and bench loss increases rockfall risk and will impact on inter-ramp and overall slope design requirements (i.e. impair the design and execution of steep slopes).

Figure 9 Significant back-break along a fault at Bozshakol pit

Rehabilitation work for double-bench (20 m high) hazards or failures are often time-consuming and costly where hazard elimination is required. Depending on the specific situation, such a process usually includes:

- Geotechnical assessment (time delays can ensue from technical work being completed).
- Discussion of, and agreement with, the mine plan and mine operation.
- Pad design to access the bench crest.
- Material requirement for constructing the pad (usually to prevent ore dilution).
- Involvement of mining equipment (haul trucks, dozer and loader) for constructing the pad.
- Ramp and pad construction.
- Hazard or failure removal (i.e. elimination).
- Material loading (removal of pad and access ramp).

5 Wall control blasting improvements

Blasting at Bozshakol has a real potential impact on production, both in terms of safety and economic costs. In addition to the main target of blasting for fragmentation, it also damages the adjacent in situ rock mass. This is mostly due to vibrations and the pressure of gases resulting from detonation of explosives.

Both vibration and pressure should be considered in the blast design to minimise the impact on the wall rock mass, which needs to be left as undamaged as possible. This chapter reviews the quantitative characteristics of the vibrations that are most likely to damage the rock masses at Bozshakol. Other aspects related to the confinement of energy will also be considered.

5.1 Vibration monitoring

Blast vibration monitoring started in 2022 and continues to date. During this time, the blast effects of more than 40 patterns have been measured using Minimate Pro4TM seismographs. Monitoring was performed during production and trim pattern blasting, as well as during the pre-splits. The blasting of each pattern was measured using three vibration monitors. Therefore, over 100 records of vibrations in the near-field along with in the far-field were obtained. Most of the measurements were performed at distances of 20–50, 100 and 150 m from the blasting patterns contour, as shown in Figure 10. During planning of the geophone installations, the blast direction, possible fly rock trajectory and existing cracks on the berms were taken into account. Vibration monitoring configurations were adjusted according to the required blast recording time.

Blast vibration monitoring is performed in the various geotechnical domains with a focus on the available exposed pit area. For blast assessment purposes and to obtain representative data with respect to current conditions, the rock types in each of the geotechnical domains were simplified to the three main rock groups:

1. Fresh rocks (andesite, breccia, diorite, gabbro, granodiorite).
2. Saprolites (saprolite class-1 and class-2).
3. Sedimentary rocks (sediments class-1 and class-2).

The study excludes clay as it is generally considered free dig and usually does not require blasting, except under some conditions involving freezing during the winter (when improved mining productivity is required).

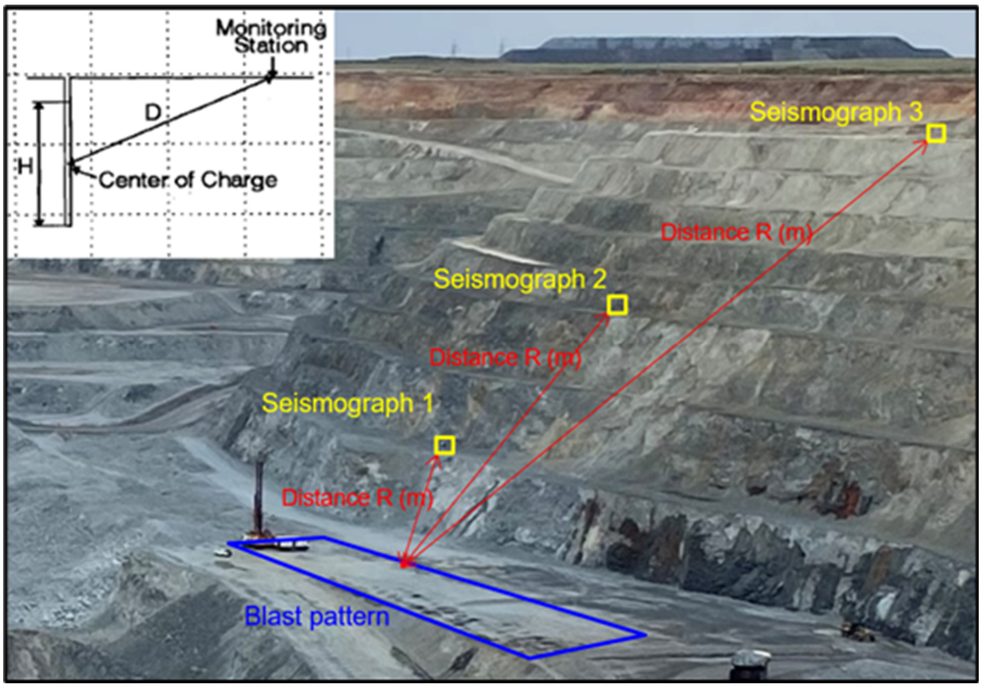


Figure 10 Schematic of typical seismographs installation

5.1.1 Blast damage criteria

During the detonation of an explosive charge, two major energy forms develop to produce the overall damage to the surrounding rock mass. The initial form of energy is a high-intensity, short-duration shock wave, followed by a high-pressure gaseous reaction. The strain produced within a rock mass during an explosive detonation is proportional to the particle velocity generated (Paul Singh & Lamond 1994).

Most of the existing blast damage criteria relate to the ground vibrations resulting from the dynamic stresses induced by the blasting process.

The scaled distance (SD) concept is used to predict the maximum peak particle velocity (PPV), V_{max}, from an explosive charge, Q, at a known distance, R (Duvall & Fogelson 1962). This relation is expressed in the following equation:

$$V_{max} = K(\frac{R}{\sqrt{Q}})^{-\beta} \tag{1}$$

where:

K = site vibration modifying factor (such as confinement, rock strength, explosive coupling etc.).

β = constant related to the vibration attenuation characteristics of the site, always a negative number (typically around -1.2 to -1.8).

The SD equation for a common cylindrical charge is expressed as:

$$V_{max} = K(SD)^{-\beta} \qquad where\ SD = \frac{R}{\sqrt{Q}} \tag{2}$$

For regression analysis, the most reliable seismograph vibration monitoring data for recorded PPV and associated SD were selected from blast patterns in Sectors 2, 3 and 5 as shown in Figure 11.

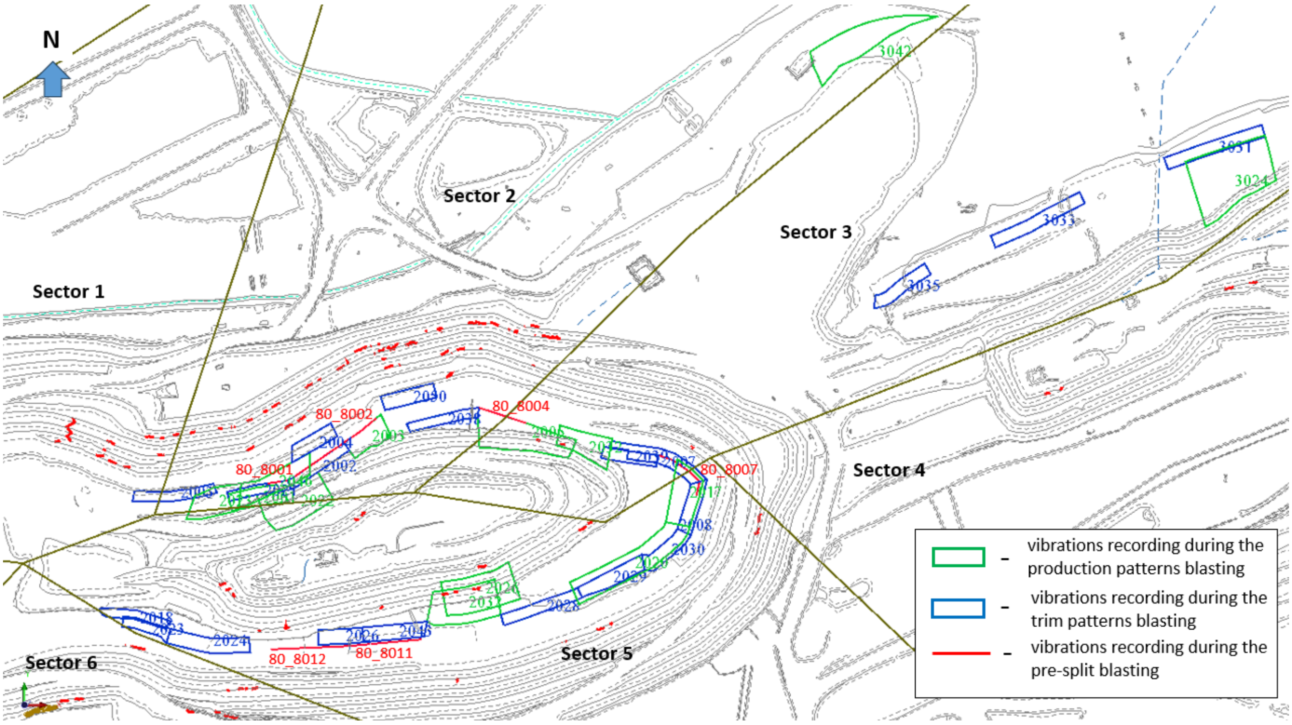


Figure 11 Selected blast patterns for vibration monitoring

Based on regression analysis results of the vibration monitoring, data with a 95% upper bound confidence level was used to refine site-based K and β parameters for predicting the maximum PPV (V_{max}) in relation to SD and explosive charge. The function graphs of the selected rock groups are plotted in Figure 12, and the resulting parameters associated with the vibration attenuation characteristics of the site and vibration modifying factors are presented in Table 2.

The 'goodness of fit' of the data which is equivalent to R^2 is ranging from 78 to 89% (equal to R^2 ranging from 0.78 to 0.89), which is considered a good correspondence (above 70%).

Table 2 Summarised results of blast damage criteria calculations

Rock group	K factor	β constant
Fresh rocks	454.64	-1.58
Saprolites	309.14	-1.39
Sedimentary rocks	302.84	-1.27

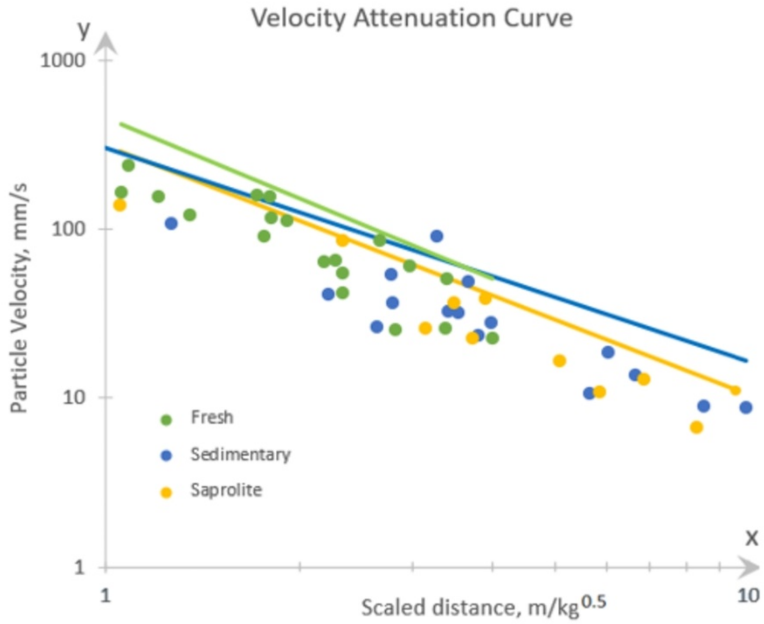


Figure 12 Relationship between particle velocity and scaled distance expressed by the velocity attenuation curve

5.1.2 Rock damage velocity prediction

During the detonation of an explosive charge, the magnitude of the dynamic stresses that develop around the blasthole will be large enough to induce primary cracking in the rock mass. The critical particle velocity V_d at which the damage occurs is calculated with the following equation:

$$V_d = \left(\frac{V_p T_s}{E}\right) \tag{3}$$

where:

V_p = longitudinal wave velocity of rocks (m/s).

T_s = tensile strength of the rock (also termed σ_t).

E = Young's modulus (for intact rock, also termed E_i).

Longitudinal wave velocity (V_p) of rocks is determined by the elastic and density properties (Rzhevsky & Novik 1984):

$$V_p = \sqrt{\frac{E(1-\upsilon)}{\rho(1+\upsilon)(1-2\upsilon)}} \tag{4}$$

where:

E = elastic/Young's modulus.

υ = Poisson's ratio.

ρ = density of rocks.

According to the specified parameters, ProWall, a Blast Dynamics Inc. software, calculates prediction values of critical velocity at which the extension of existing cracks, the development of new cracks and the crushing of the rock mass occurs. The calculation results are shown in Table 3.

Table 3 Calculation results of the PPV prediction values

Rock group	Longitudinal wave velocity, (m/s)	Extension of existing cracks, (mm/s)	Predicted PPV (mm/s) Development of new cracks (critical velocity)	Crushing rock mass, (mm/s)
Andesite	4,084	224	896	3,586
Saprolite – class1	274	69	274	1,096
Saprolite – class2	503	30	120	479
Breccia	4,237	167	669	2,676
Diorite	4,129	244	978	3,912
Gabbro	2,428	173	694	2,775
Granodiorite	3,415	164	657	2,627
Sediments – class 1	563	7	28	113
Sediments – class 2	927	70	278	1,112

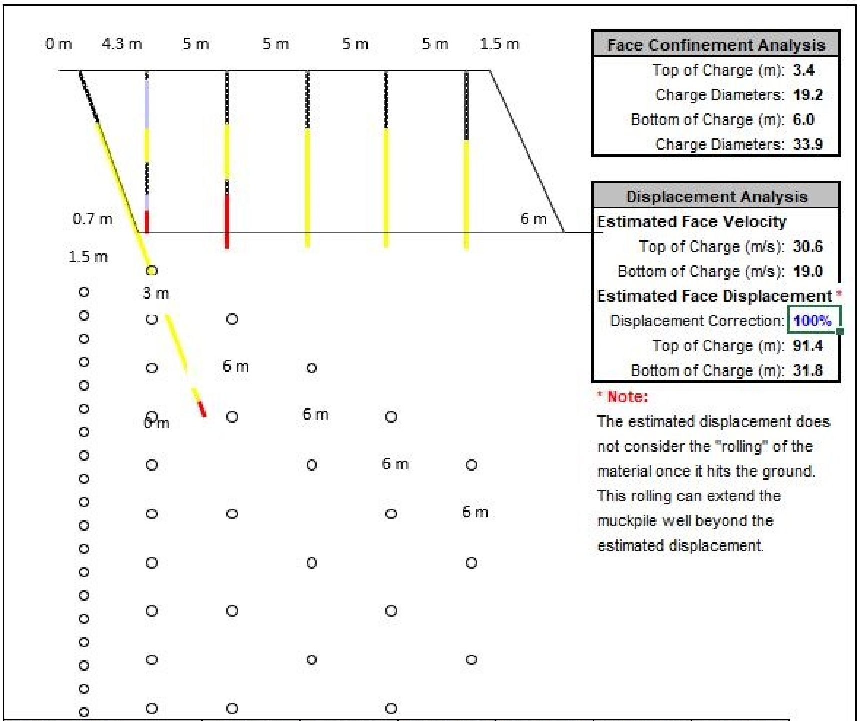
5.2 Blast design adjustments

5.2.1 Adjustment of trim blasting design

Current blasting practice is charging only a single charge into the blastholes. One buffer row is adopted for all trim blasting independent of the lithology and rock mass conditions. Results of critical PPV calculations confirming historical failures analysis showed that an initiation of charge weighing more than 60 kg in the first buffer row would cause extension in the existing structures to crack. Moreover, more than 200 kg charge blasting in the normal production row next to the buffer will also damage the structures of all rock types except fresh rock. Therefore, the following changes have been developed and should be introduced into the current trim blast designs:

- Implementation of two buffer rows.
- Decked blasting for both buffer rows using more than 10 ms delay between charges.
- Increased stand-off distance/toe offset from 0.3 to 0.7 m in blast designs with critical/weak structures (all saprolite, any disintegrated or disturbed sedimentary). When double benching, the toe offset of the buffer row (0.3 m of the upper bench) is too close to the final wall, inducing damage and excessive vibration to structures (wedges/planes).

Figure 13 shows an example of a new trim blast design for the top flitch, including the above changes.



Design Data	Presplit Row	1st Buffer	2nd Buffer	Production Row	Production Row	Crest Row
Hole Diameter (mm):	165	177	177	177	177	177
Hole Depth (m):	22.9	10.0	11.0	11.0	11.0	11.0
Hole Angle (deg):	20					
Burden (m):	4.3	5.0	5.0	5.0	5.0	3.8
Spacing (m):	1.5	3.0	6.0	6.0	6.0	6.0
Stemming (m):	3.4	0.5	3.2	3.5	3.5	4.2
Airdeck (m):		3				
Subdrill or Standoff (m):	11.5		1.0	1.0	1.0	1.0
Offset from Slope (m):		0.7	5.7	10.7	15.7	20.7
Main Charge Type:	Powersplit	Fortis Extra 7	Fortis Extra 7	Fortis Extra 7	Fortis Extra 7	Fortis Extra 70
Main Charge Weight (kg):	17	60	100	220	220	200
Crest Scaled Distance:	6.43	2.05	2.56	2.54	3.22	4.14
Mid Inert or Air Deck (m):		3.0	1.0			
Bottom Charge Type:	Powersplit	Fortis Extra 7	Fortis Extra 7	ortis Extra 10		ortis Extra 100
Bottom Charge Weight (kg):	4	44	100			
Crest Scaled Distance:	25.19	3.64	3.11			
Powder Factor (kg/tonne):	0.12	0.26	0.25	0.27	0.27	0.33
Energy Factor (kjoule/tonne):	422	873	840	923	923	1119
Comments:						

Figure 13 Example of refined trim blasting design

5.2.2 Adjustment of pre-split timing

A special case for near-field damage is the detonation of pre-split charges by the wall. Pre-splitting is supposed to protect the final bench by creating a crack that defines the bench face angle with a surface between half-barrel holes drilled closely together. This crack also creates a filter that reduces vibrations created by the trim or production blasts by 40–50% PPV. But, even if the charge per hole is uncoupled (usually an 80–90% ratio), the timing of groups of holes can create a high vibration pulse into the rock mass (Cebrian 2022b).

Current pre-split blasting shots usually comprise 50 holes blown simultaneously with the same time delay. With different configurations of charge (pre-split explosive or detonating cord and cartridges) up to 50 shots of 18 kg/hole (i.e. 900 kg) are made instantaneously, right by the wall, with no vibration filter. A field test of five holes delayed 25 ms from 24 holes was performed and monitored in the upper wall with seismographs. Figure 14 shows the quantitative monitoring results to understand the vibration values generated by the pre-split. Results at 45 m distance in the upper bench from the pre-split show that shot every five holes create 32 mm/s while every 24 holes induce 61 mm/s. It is a clear indication of the high vibration levels induced by the pre-split. It is expected that 50 shots will have even higher vibrations, which have more potential to initiate a failure mechanism that can be further exacerbated with the subsequent trim blasting. Thus it was determined to blast no more than 10 holes per delay, except in critical locations where a maximum of five holes was stipulated.

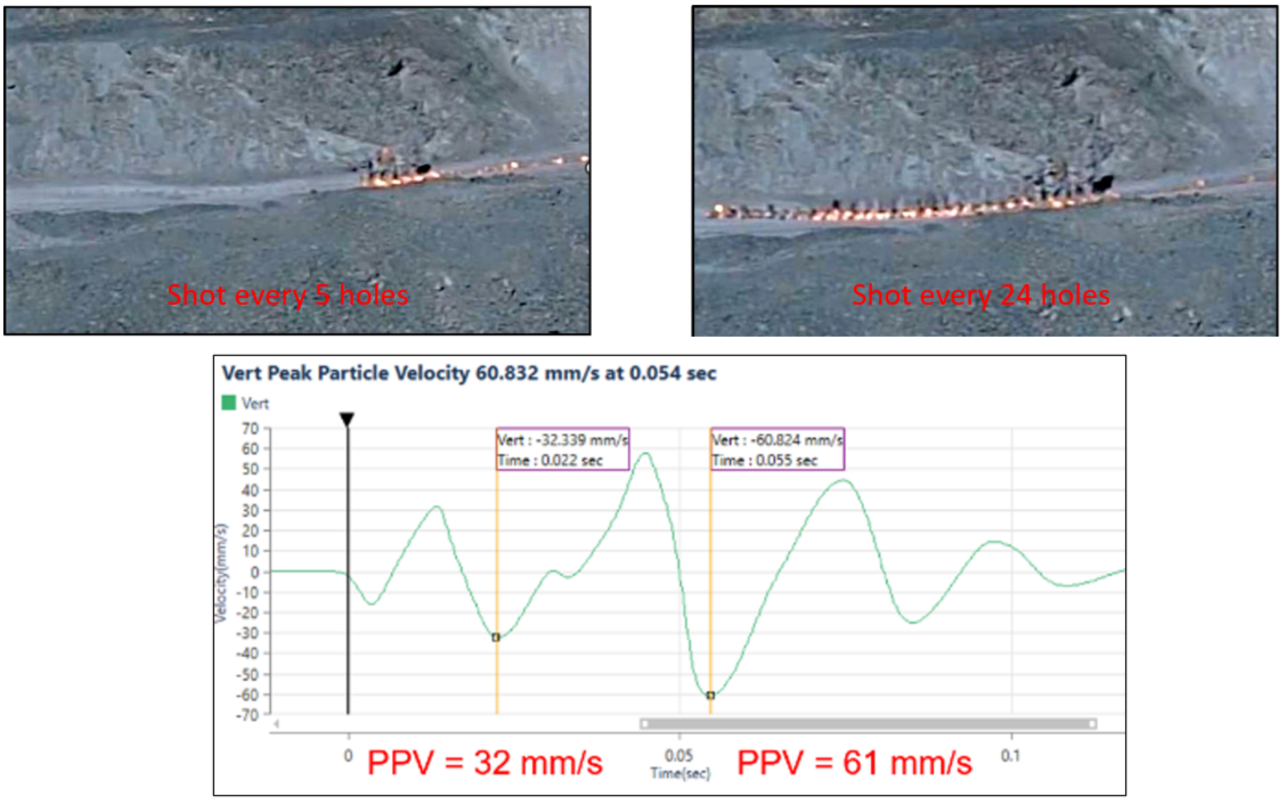


Figure 14 Pre-split blasting field test

5.2.3 Adjustment of blast master boundary

Typically, the trim pattern boundary at Bozshakol is in the shape of an elongated rectangle. In rectangular form, the energy is not sufficiently distributed in the opposite angle from the initiation point due to limited releasing space, which may potentially move backward and affect the wall. In addition, the obtuse angle at the initiation point has a favourable effect for rock mass ejection/release in the early stage of the blasting sequences. The ideal option seems to be an obtuse angle of 135° degrees at the initiation point, but in practice it is often impossible to perform as the rig needs more space to drill at the adjacent sharp corner. So it is more probable that the maximum angle at the initiation point will be less than 110–115°. To reduce these risks, trim patterns have been adjusted to parallelogram shapes in the blast master boundaries for a more effective control of blast energy.

5.2.4 Consideration of major structure orientation

Reactivation of weak structures due to blast energy propagation is one of the main factors initiating bench scale instability at Bozshakol. In some instances this leads to the development of geotechnical hazards and a need for exclusion zones, or even design deviations if instability occurs on several benches. Directing the blast parallel to the structures is a decent approach but somehow it did not account for the dip direction. This is key for stability if the blast initiation starts opposite the dip direction of a weak structure: the blast energy will shear/move the rock mass upward along the structure since movement along that trajectory toward the free face is obviously easier. Figure 15 illustrates how the dip and dip directions of key geological structures are considered in the blast design and the proper blast direction choice.

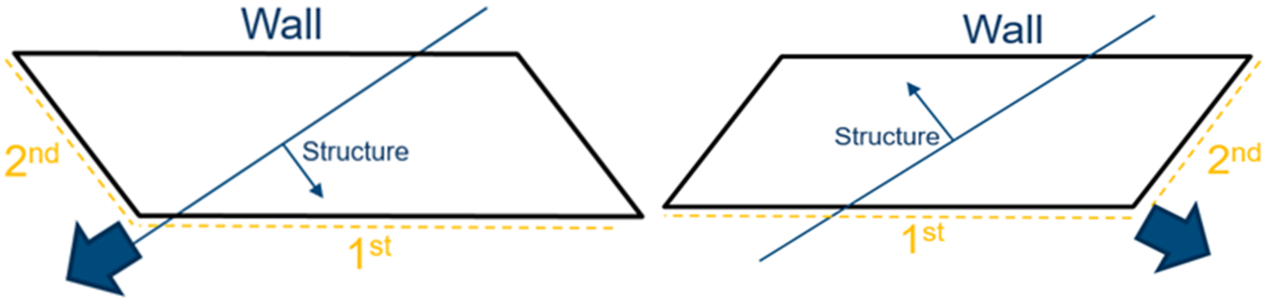


Figure 15 Trim blast direction considering the dip direction of the structure

5.2.5 Crest damage induced by sub-drilling

The main factor for crest damage in small to medium blasthole diameter blasting is not so much related to the mechanism of over-confinement (especially when air decks are used in one or two buffer rows) but instead by the sub-drilling from the blast pattern above the design bench crest. Although the blast design does not show production holes drilled immediately on top of crests, the sub-drilling of these production holes in previous benches can potentially create damage if the design is changed. In this situation, the blasted sub-drills become relatively close to the new crest position.

The real and significant damage to crests identified at Bozshakol occurs when berm widths vary as they follow the pit limit shapes. When this happens, drilling of holes is done right on top of the crests, causing the further failure of 2 m or more of crest despite applying pre-splitting techniques. Examples of crest damage are illustrated in Figure 16.

Figure 16 Crest damage due to improper sub-drilling in the proximity of the designed crest

In order to prevent crest damage by sub-drilling it is important to consider such simple things as:

- Comprehensive QA/QC processes in the field to ensure blastholes are drilled at the correct locations and to the correct depths.
- Mine plan design changes being overlaid to the actual drilling design.
- On lower flitch benches, plotting the pit design crest underneath it.
- When holes fall within 1 m from the crest, move individual holes outside that exclusion zone' for holes.

5.2.6 *Consideration of geotechnical domain*

Figure 17 gives an example of particle velocity prediction value calculations from ProWall, Blast Dynamics Inc. software, for which fresh rock damage will occur. The predicted PPV values in the 'yellow' zone (in Figure 17) are considered sufficient to extend the existing cracks associated mostly with natural structures. Thus there is a close parallel between natural structures in the rock mass and existing cracks on the berms. Therefore, to achieve 'optimal'/'good' blast results, it is necessary to consider the values of critical velocity at which the extension of existing cracks occur as these limits require more careful blasting compared with the critical PPV on the development of new cracks. This requires a good understanding of the structural settings which are causing potential slope failure and associated hazards. The targeted PPV values in the 'white' zone (in Figure 17) guide the drill and blast engineer to design the charge weight with related distance from the slope accordingly. The iterative process with the cycle of design, blast, vibration monitoring and PPV prediction is established.

Rock Damage Particle Velocity Prediction (mmps)

		Typical Ranges	
Tensile Strength (MPa):	9	1	20
Young's Modulas (GPa):	41	5	120
Longitudinal Velocity (mps):	4,084	2	8,000
Calculated Critical Velocity (mmps):	896		
Extension of Existing Cracks (mmps):	224	896	
Development of New Cracks (mmps):	896	3,586	
Crushed Rockmass Factor:	4		
Crushed Rockmass (mmps):	3,586		

$$V_c = \left(\frac{V_p \cdot T_s}{E}\right)$$

T_s = tensile strength of the rock
E = Young's modulus

Site Factors	K	α	β
	454	0.8	1.58

Typical Site Factors	K	α	β
Massive Granite	700	0.7	1.5
Jointed Granite	190	0.86	1.72
Andesite	200	0.9	1.8
Strong Sandstone	400	0.78	1.56
Strong Shale	175	1.25	2.5

Chart Variables

Initial Charge Weight (kg):	30	Initial Distance (m):	1
Charge Step Increase (kg):	10	Distance Step Increase (m):	0.5

	Charge Weight (kg)									
Distance (m)	30	40	50	60	70	80	90	100	110	120
1	1,131	1,424	1,702	1,969	2,227	2,479	2,723	2,963	3,198	3,428
1.5	596	750	897	1,038	1,174	1,306	1,435	1,561	1,685	1,807
2	378	476	569	659	745	829	911	991	1,070	1,147
2.5	266	335	400	463	524	583	640	697	752	806
3	199	251	300	347	393	437	480	522	564	604
3.5	156	197	235	272	308	342	376	409	442	474
4	127	159	190	220	249	277	305	331	358	384
4.5	105	132	158	183	207	230	253	275	297	318
5	89	112	134	155	175	195	214	233	251	270
5.5	76	96	115	133	151	168	184	200	216	232
6	67	84	100	116	131	146	161	175	189	202
6.5	59	74	88	102	116	129	141	154	166	178
7	52	66	79	91	103	115	126	137	148	158
7.5	47	59	71	82	92	103	113	123	133	142
8	42	53	64	74	83	93	102	111	120	128
8.5	38	48	58	67	76	84	93	101	109	117
9	35	44	53	61	69	77	85	92	99	107
9.5	32	41	49	56	64	71	78	85	91	98
10	30	37	[illegible]	52	[illegible]	[illegible]	72	78	84	90
10.5	28	35	41	[illegible]	[illegible]	[illegible]	66	[illegible]	78	83
11	26	32	39	45	50	56	62	67	72	78
11.5	24	30	36	42	47	52	57	62	67	72
12	22	28	34	39	44	49	54	58	63	68
12.5	21	26	31	36	41	46	50	55	59	63
13	20	25	30	34	39	43	47	51	56	60
13.5	19	23	28	32	36	41	45	49	52	56
14	17	22	26	30	34	38	42	46	49	53
14.5	17	21	25	29	33	36	40	43	47	50
15	16	20	24	27	31	34	38	41	44	48
15.5	15	19	22	26	29	33	36	39	42	45
16	14	18	21	25	28	31	34	37	40	43
16.5	13	17	20	23	27	30	32	35	38	41
17	13	16	19	22	25	28	31	34	36	39
17.5	12	15	18	21	24	27	30	32	35	37
18	12	15	18	20	23	26	28	31	33	36
18.5	11	14	17	20	22	25	27	29	32	34

PPV target: White zone

- Common minimum distance from closest blast holes to berm contour

- Charge weight per delay does not impact the extension of existing cracks

Figure 17 ProWall software calculation results for PPV prediction

The structures in saprolite which are deeper on the north wall, as well as in the high altered or fractured sedimentary rocks on the south wall, require more accurate blasting than that in fresh rock. Therefore, the main charge value in the first buffer row hole should have a smaller weight and not exceed 44 kg. The rest of the parameters are similar to fresh rock, except when there is a 'critical' joint. A 'critical' joint is defined as a well-known back-break on the berm or any other weak structure with the potential risk to induce to a failure. To avoid damage, it is necessary to reduce the charge with less stemming length in the first buffer row hole. More detailed information is provided in Table 4.

Table 4 Accepted charge design parameters for buffer rows

Rock group	Flitch / sub-bench	1st buffer row								2nd buffer row					
		Distance to crest, m	Offset from slope, m	Bottom charge, kg	Mid air deck, m	Mid inert deck, m	Main charge, kg	Air deck, m	Stemming, m	Distance to crest, m	Bottom charge, kg	Mid inert deck, m	Main charge, kg	Air deck, m	Stemming, m
Saprolite, weak sedimentary	Top	4.3	0.7	44	1.0	2.0	44	3.0	1.0	9.0	100	1.0	100	–	3.2
	Bottom	4.0	0.4	60	2.0	1.0	44	–	3.5	9.0	100	1.0	80	–	2.9
Fresh, hard sedimentary	Top	4.3	0.7	44	1.0	2.0	60	3.0	0.5	9.3	100	1.0	100	–	3.2
	Bottom	4.0	0.4	60	2.0	1.0	44	–	3.5	9.0	100	1.0	50	3	1.4
Critical joint	Top	4.3	0.7	44	–	–	–	–	1.0	9.3	100	1.0	50	3	2.4
	Bottom	3.9	0.3	44	3.0	1.0	44	2.0	1.0	8.9	100	1.0	50	3	1.4

6 Results

Between June 2021 and August 2022, in a 60 m high portion of the interim slope (160 to 100 benches), a series of bench failures occurred on the north wall associated with subparallel faults (refer to Figure 18). Due to the loss of catch berm on 160 and 140 benches, the access ramp below was exposed to higher risk.

As a result, the interim slope design on 120 bench was revised with a step-in to reduce rockfall risk and an adjusted slope strike as a preventive effort against further failures. However, due to damaged similar structures, a 12 m-wide part of the step-in berm splintered off and additional remediation work was carried out within two weeks.

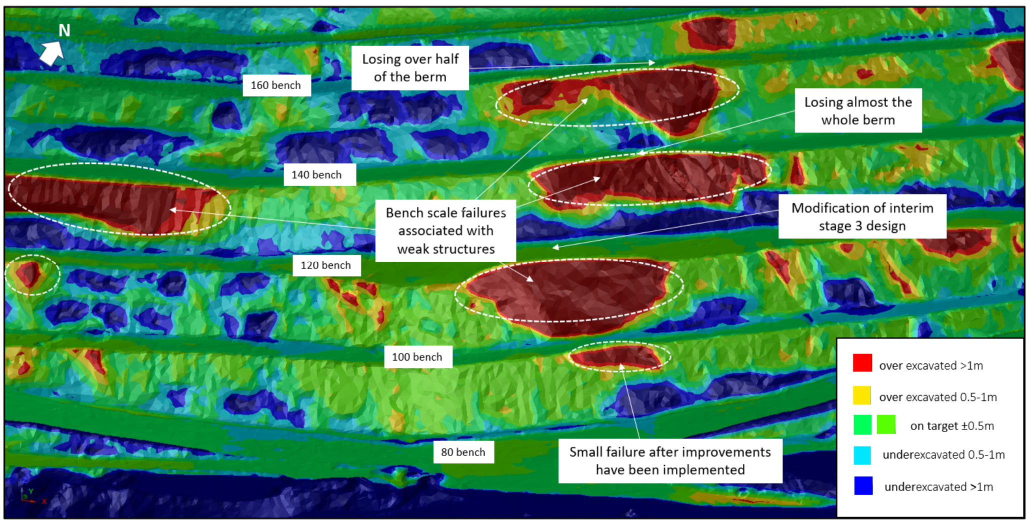


Figure 18 Heat map of the most unstable pit wall area related to weak structures

By applying improved blasting techniques, the 100-80 bench adjacent to the ramp was retained. Changes have been introduced to all components of the pre-split and trim pattern blasts. While complete bench failure avoidance is not considered practicable, the improvement is considered very positive. Figure 19 compares pit wall photographs in similar domains, pre- and post-improvements.

(a)

(b)

Figure 19 Bozshakol pit walls. (a) 2021, before damage minimisation improvements; (b) 2023, after implementing the improvements

7 Conclusion

Bozshakol copper mine has developed over several pushbacks with the experience of instabilities induced by blasting. With improving confidence in the geotechnical understanding, slope design optimisation has become feasible for future pushbacks. In order to design, and safely and reliably execute, steeper slope designs, improving blasting practices to minimise wall damage has become a higher priority.

During the development of the Bozshakol pit there has been an accumulation of historical data on failures, back-breaks, rockfalls, etc. A clear understanding has emerged of the need for a comprehensive approach to improving implementation so as to minimise blast damage affects. This process, which started as simply recording wall damage, has evolved into analysing it, identifying flaws and mistakes, and making adjustments through collaborative discussions and practice.

Calculations of critical velocity for extension of existing cracks were performed for all types of rock. However, for convenience, fresh rocks were combined into one group and andesite properties, which dominate at Bozshakol, were adopted. In the regression analysis, 90% of data input was taken from near-field monitoring to avoid attenuation of vibrations on the way to the geophone location due to large structures/faults and existing cracks on the berms. As expected, PPV values required to damage structures in fresh rocks was much higher in comparison to the saprolite and sedimentary rocks. Extension of weak structures in these rocks would not necessitate significant PPV values and, even when the correct delays and charge reductions are used, the predicted values don't pass beyond the damage zone.

It also makes sense to monitor the vibrations of production blasting. The negative impact is usually due to the simultaneous blasting of large quantities of explosive and, as the calculations show, if a production blast is closer than 30 m to a wall it is essential to control this to maintain a balance between excavatability and wall damage.

Within a single year, significant changes have been implemented to improve blasting and validating impacts on walls as part of a continuous improvement cycle. The multidisciplinary approach in solving this problem has not only improved operational safety but also productivity, with fewer bench failures requiring remediation.

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