Mathematical Modelling of Engineering Problems
Vol. 12, No. 10, October, 2025, pp. 3607-3626
Journal homepage: http://iieta.org/journals/mmep

Optimizing Blasting in Narrow Veins: Using DECKs to Reduce Dilution and Improve Recovery in Underground Mining

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https://doi.org/10.18280/mmep.121025

Received: 17 April 2025
Revised: 3 July 2025
Accepted: 11 July 2025
Available online: 31 October 2025

Keywords:
blasting, DECKs, underground mining, dilution, stability, optimization, efficiency

ABSTRACT

This study addresses the optimization of blasting in underground mining, specifically in narrow veins, through the implementation of DECKs, a technique that consists of inserting inert material between explosive charges. The objective was to evaluate the effectiveness of this technique in reducing ore dilution and improving rock mass stability compared to traditional methods. To this end, simulations and field tests were carried out, calculating the operating loads and designing drilling meshes, which allowed the analysis of geomechanical and operational parameters. The results obtained showed a significant improvement in operational efficiency, as ore dilution was reduced from 57.14% to 5.26%, while the Power Factor (PF) decreased from 0.6 kg/MT to 0.4 kg/MT. These findings highlight that the use of DECKs not only optimizes the use of explosives but also reduces operating costs and improves ore recovery. In conclusion, the implementation of DECKs has proven to be an effective solution to improve rock mass stability, to optimize blasting and to make underground mining more efficient and sustainable, underlining the importance of adopting innovative technologies in mining processes.

1. INTRODUCTION

Mining is one of the main economic activities worldwide and a fundamental pillar in the development of many countries [1, 2]. According to the World Bank, mining accounts for around 10% of global Gross Domestic Product (GDP) [3, 4] and is responsible for the provision of essential raw materials for various industries, such as energy, construction and technology [5-7]. In Latin America, mining is even more important, accounting for approximately 20% of the region's GDP, with countries such as Chile, Peru and Mexico standing out as major producers of copper, gold and silver [8-10]. In the specific case of Peru, mining is a key sector for the national economy, representing more than 10% of the GDP and being the main generator of exports, with minerals such as copper, gold and zinc as its main products [11-13]. The stability and sustainability of mining are essential not only for the economic development of the region, but also to ensure a constant supply of mineral resources globally [14].

Underground mining is one of the most complex forms of mineral extraction due to the geological and operational conditions it faces [15, 16]. Unlike open-pit mining, where ore layers are more directly accessible, underground operations requires specialized techniques to work in confined spaces and with high geotechnical pressure [17, 18]. In this type of mining, narrow veins, which are low-thickness mineralized formations, represent one of the greatest challenges [19]. These veins, by their nature, require precise mining methods to extract the mineral without affecting the stability of the rock mass that surrounds them [20-22]. Their complex geometry and high geomechanical variability make them difficult to access and recover efficiently, often resulting in a higher risk of ore dilution. Dilution occurs when, during the blasting or extraction process, waste material is mixed with the valuable ore, reducing the grade of the mined ore and negatively affecting the profitability of the operation [23, 24].

One of the main causes of ore dilution in underground mining is the use of excessive operating charges during blasting [25, 26]. This phenomenon can cause additional damage to the rock mass, compromising its stability and increasing the fracturing of the surrounding rock [27]. Excessive fracturing not only increases dilution but also results in more non-valuable material that must be processed, which increases operating costs and reduces the efficiency of the extraction process [26]. To mitigate this problem, several alternatives have been implemented in recent decades, such as reducing the number of explosives used, controlling sequencing of blasting, and optimizing drilling designs [25]. However, these approaches have not achieved significant results in terms of reducing dilution or improving rock mass stability. Explosive reduction, for example, has proven effective only in certain contexts, but does not always translate into a substantial decrease in fracturing or improved mineral recovery. Similarly, blast sequencing and adjustment of drilling designs sometimes fail to adequately control the effects of explosive energy on stope stability.

In this context, the optimization of the operating charge in underground blasting has become a critical aspect for mining operations, especially those exploiting narrow veins. The proper selection and control of the operating charge not only minimizes rock mass fracturing but also reduces ore dilution and improves operational efficiency. This, in turn, translates into greater mineral resource recovery and a significant reduction in the costs associated with extraction and processing.

Therefore, this study aims to evaluate the use of DECKs as an alternative for the optimization of the operating charge in underground mining. The use of DECKs, which consists of inserting inert material between the explosive charges, has shown potential to better control the energy released during blasting, which in turn minimizes unwanted fracturing of the rock mass. Unlike other solutions that have been implemented in the past, such as reducing the number of explosives or sequencing blasts, the use of DECKs offers a more precise approach to controlling the explosive charge, allowing a more efficient use of the released energy. This method not only reduces dilution by avoiding over-setting of the rock but also improves the stability of the rock mass by reducing the unwanted effects of blasting on the structure of the pit. In addition, DECKs provide a significant benefit in terms of sustainability, as they use inert materials instead of additional explosives, which contributes to reducing the environmental impact associated with blasting.

Regarding the proposed solution, the use of DECKs in underground mining has proven to be an effective technique to optimize blasting processes and minimize ore dilution. By using DECKs, which consist of inserting inert material between the explosive charges, the energy released during detonation is better controlled, which allows reducing unwanted fracturing of the rock mass. This technique is particularly relevant in the exploitation of narrow veins, where precise control of the operating charge is crucial to avoid over-fracturing and the mixing of waste material with the valuable mineral. In addition, the use of DECKs improves the stability of rock mass, since the controlled distribution of explosive energy reduces adverse effects on the structure of the pit, which, in turn, increases operational safety and efficiency.

This study has a high potential in underground mining, especially in the Peruvian context, since, although there are several studies that address blast optimization and dilution reduction, none have focused specifically on the application of DECKs as an effective solution in narrow vein mining. Research of this type is particularly relevant, as it allows filling a gap in the literature on the practical application of this technique in specific geomechanical conditions, which could have a significant impact on the mining industry globally. Thus, Meng et al. [28] addressed the reduction of dilution in narrow vein mining, a critical challenge in underground mining. The research proposes a sequencing of operational strategies to improve drilling and blasting efficiency, comparing conventional methods such as cut and fill (CAF) with more productive methods such as longhole stoping. In this context, decoupled loading techniques were implemented in the blasting process, resulting in a significant reduction of dilution, from 40% to 8%, and a decrease in the Power Factor (PF) (kg/tn) by 25%. Furthermore, this approach allowed a reduction in operating costs of up to 31%. The results demonstrated that the application of advanced drilling and blasting techniques not only optimizes ore recovery, but also offers a more cost-efficient alternative, even in narrow veins with widths less than 1 meter. The study highlights the feasibility of improving productivity and reducing dilution by utilizing mechanized methods and a more controlled approach in the use of explosives.

On the other hand, the aim of this study [29] was to improve the prediction of blast-induced maximum particle velocity (PPV) in mining, using an optimized random forest model. This model seeks to more accurately predict particle velocity during blasting, a crucial factor in minimizing the undesired effects of explosions on and around mining operations. To achieve this goal, metaheuristic techniques, such as the Whale Optimization Algorithm (WOA), the Gray Wolf Optimization Algorithm (GWO), and the Tunicate Swarm Optimization Algorithm (TSA), were employed to improve the accuracy of the random forest model. These techniques were used to adjust the weights of the decision trees in the model, optimizing blasting parameters such as maximum delay charge and dust factor, which directly influence PPV. The results obtained showed a significant improvement in the accuracy of the model predictions, especially when the optimization algorithms were used, compared to traditional methods. This approach allows us to improve the reliability of PPV predictions, which is essential for the planning and control of blasting in underground mining, contributing to the safety of operations and the reduction of environmental impacts. The study demonstrated that the combination of prediction models with metaheuristic techniques can considerably improve the estimation of the velocity of particles induced by explosions in mining, providing a more accurate and efficient tool to optimize blasting operations in the mining sector.

Likewise, the aim of this study was to identify and characterize the geological and mining factors that determine internal dilution (ID) in lateritic nickel and cobalt deposits in Cuba, to optimize their separation during the extraction process [30]. The study analyzed geological and geochemical data, considering the geological complexity and the mining methods used, to improve the quality of the extracted ore. The methodology used included the observation of the mining fronts and drill holes or boreholes, and the interpretation of the data obtained. In addition, advanced mining technologies were used to evaluate low-grade interlayers and their impact on internal dilution. The analysis detailed the physical, geochemical and geometric properties of non-industrial interlayers, allowing the identification and separation of interlayers immediately before the extracted ore enters the industrial process. The results showed that internal dilution was strongly influenced by geological factors, such as the strength and extent of the interlayers, as well as by field supervision, which plays an essential role in improving mining selectivity. Finally, the study concluded that, with improved knowledge of geological models and the application of more effective control during extraction, the impact of internal dilution can be significantly reduced, thereby improving the efficiency and profitability of nickel and cobalt mining in Cuba.

On the other hand, Delentas et al. [31] proposed to combine empirical approaches and numerical simulations to analyze stability conditions and extracted ore dilution in underground mining by using open stope methods. The work aims to provide easy-to-use mathematical tools to estimate the ore dilution rate and to correlate the mine stability conditions with the characteristics of the extracted ore. To do so, they employed a parametric analysis using RS2 finite element software to model different geomechanically and design conditions of the stope. In their methodology, the stability of the stope, over breaks and the external dilution rate that could arise from failures in the walls of the stope were analyzed. The results of the numerical models were compared with empirical approximations of the stability graphs and mathematical equations were provided to estimate the external dilution rate as a function of basic parameters such as the stability number (N) and the hydraulic radius (HR). The results demonstrated that dilution rates calculated from the finite element model were useful for preliminary stope design, allowing engineers to identify potential dilution issues prior to exploitation. The research also established a relationship between the geotechnical design of the stopes and the stability of the surfaces, with special emphasis on the areas of the sidewalls and the top of the stope. This study offers a combined approach that integrates empirical analysis and numerical simulations to optimize the design of open stopes, reducing ore dilution and increasing the profitability of underground mining operations. The tools developed in the article can be applied for mining project planning, improving operational efficiency and the quality of the extracted ore.

In addition, according to the study conducted by Câmara and Peroni [32], a methodology was developed to quantify the dilution caused by operational efficiency in open-pit mining, specifically in relation to adjacent ore blocks that are not mined correctly due to deficiencies in the operation. The study seeks to improve the accuracy in mine planning by identifying and calculating the dilution resulting from the inability of the equipment to perfectly separate the blocks during extraction. The methodology proposed in the article allows to estimate the dilution caused by operational inefficiency by considering several factors, such as the equipment's ability to remove the blocks, the geometry of the deposits, and the interaction between the ore blocks and the waste blocks. In the process, it is determined which blocks are in contact with the blocks planned to be mined, thus calculating the dilution based on operational deficiencies, such as the lack of selectivity of the equipment or the skill of the operator. This calculation is fundamental to improve the estimates of tonnage and ore grade during the mine planning process. The results obtained demonstrate that dilution can be significant, even when the planning and execution reconciliation process is relatively accurate. The authors concluded that the application of this methodology allows us to identify the causes of dilution and provides a more systematic way of dealing with this problem, which can improve the accuracy of ore resources and reserve calculations, as well as optimize dilution control practices and improve the profitability of the mining operation.

The objective of this research is to explore the impact of using DECKs (inert material placed between the explosive charges) in optimizing the operating charge, to significantly reduce ore dilution and improve rock mass stability. This approach not only promises to improve the structural stability of narrow veins but also represents an efficient and cost-effective solution compared to traditional blasting methods.

The main innovation of this study lies in the use of DECKs instead of conventional solutions such as explosive reduction or blast sequencing. The use of DECKs allows for more precise control of the explosive charge, resulting in a reduction of unwanted fracturing and therefore less dilution. Through simulations and field tests, the impact of this technique on improving safety factors, reducing unwanted fractures and optimizing ore recovery will be assessed. This study will provide new insights into how operating charge optimization with DECKs can transform efficiency in underground mining operations, especially in narrow veins.

The implementation of DECKs not only represents an innovative approach to optimizing operating charge in underground mining but also opens new possibilities for research and application of advanced techniques in blast design. This research seeks to demonstrate that the use of DECKs can significantly improve rock mass stability and recovery, enhancing the profitability and sustainability of underground mining operations.

2. MATERIALS AND METHODS

Validation of the operating charge optimization technique using DECKs (inert material placed between the explosive charges) is crucial due to the geomechanically complexity of narrow veins and the need to improve the efficiency and safety of mining operations. A thorough understanding of rock mass properties and assessing the effectiveness of DECKs in reducing dilution and improving vein stability is essential to develop more efficient blasting strategies.

To address this issue, advanced geomechanically characterization techniques and rigorous experimental design were implemented. These methods, supported by a multidisciplinary approach, allowed for an accurate and detailed assessment of rock mass stability in narrow vein mining operations. In addition, laboratory and field-testing procedures were applied to measure rock mass mechanical properties and the impact of DECKs on rock mass cohesion and shear strength.

The combination of these advanced techniques with rigorous statistical analysis aims to obtain a comprehensive understanding of the influence of DECKs on vein stability. This approach not only ensures the validity and reliability of the results obtained but also provides a solid basis for decision-making in geotechnical engineering and mining operations management, both in the Ayacucho region and in other areas with similar geomechanically conditions.

2.1 Geomechanical parameters

Geomechanical parameters are fundamental for drilling and blasting design in underground mining, especially in narrow veins. Rock mass stability and ore extraction efficiency depend largely on a detailed understanding of the material's geomechanical properties. In this regard, it is crucial to evaluate rock mass characteristics, such as its compressive strength, internal cohesion, friction, and the presence of fractures or discontinuities, which can directly affect slope behavior and blast quality.

Knowledge of these parameters allows determining the viability of drilling and blasting, as well as the proper design of drilling patterns and explosives to be used. For narrow veins, which often present a more complex geological structure, geomechanical characterization is essential to minimize unwanted fracturing and reduce ore dilution. The procedures for obtaining geomechanical parameters are detailed below:

- **Determination of relative density ($\boldsymbol{\rho_r}$)**

The relative density of rock (ρ_r) is a fundamental parameter that is obtained through a volumetric weight test. For this test, a representative rock sample is taken, and its volume is measured using the water displacement method. The sample is weighed in its dry state and the density is calculated using the formula:

$$\rho_r = \frac{sample\ mass}{sample\ volume} \quad (1)$$

This parameter is crucial for assessing the load-bearing capacity and stability of structures to be built on the ground. The technical standard that regulates this test is ASTM D7263, which establishes the standard procedure for determining the density of rock cores.

- **Geomechanical classification Rock Mass Rating (RMR)**

The RMR is one of the most important indices in geomechanics, as it provides a general classification of the quality of a rock mass. This parameter is calculated using an empirical system that evaluates several characteristics of the rock mass, such as the uniaxial compressive strength of intact rock (σ_c), the rock quality index (RQD), the spacing and persistence of fractures, the alteration and orientation of discontinuities. It is performed by compression tests on rock samples, as well as visual observations of the fracturing characteristics. The result of the RMR is classified on a scale of 0 to 100, where higher values indicate a higher quality of the rock. The RMR is fundamental for the design of tunnels, excavations and mine planning.

- **Geomechanical Index Geological Strength Index (GSI)**

The GSI is determined by detailed observation of the terrain and its geological characteristics, such as rock type, degree of fracturing, and rock mass alteration. Unlike the RMR, the GSI is based on a visual classification of rock quality, making it useful in situations where core samples are not available. The index is obtained by observing the degree of fracturing and rock texture and is classified into values ranging from 0 to 100. This parameter is useful for estimating rock mass strength under different stress conditions and for support planning.

- **Rock Quality Index (RQD)**

RQD is used to describe the quality of intact rock based on the percentage of the length of the drill cores that are sound and free of significant fractures. This parameter is obtained by drilling holes around interest, taking rock core samples and measuring the percentage of intact material. An RQD greater than 75% is considered excellent, while lower values indicate greater rock fracturing. This parameter is crucial for assessing the stability of underground excavations and is used in tunnel and mine planning.

- **Uniaxial compression strength (σ_c)**

Uniaxial compressive strength (σ_c) is one of the most critical properties of intact rock, as it reflects the rock's ability to withstand loads without fracturing. This test involves subjecting a cylindrical rock sample to an axial load in a compression press until the sample fails. The maximum load supported by the sample is the uniaxial compressive strength. This value is used to determine the rock mass's ability to resist mechanical stresses during mining and construction activities. The technical standards governing this test include ASTM D7012, which provides procedures for performing compression tests on intact rock samples, and ISRM, which describes standard methods for evaluating this property.

- **Geomechanical index Q**

The Q index is an empirical formula that assesses the overall quality of a rock mass based on several geomechanical parameters, such as the RQD, the number of fractures, and the type of discontinuities. This index is used to calculate the stability of underground excavations and determine support requirements. A high Q value suggests a stable rock mass, while a low value indicates a higher risk of instability. The Q index is widely used in mining and civil engineering for planning the excavation and reinforcement of tunnels, mines, and other underground structures. The Q formula is standardized by ISRM.

2.2 Slot drilling mesh design (Free face)

Slot drill pattern design on the free face of a vein is a critical aspect for ensuring efficient drilling and blasting operations, especially in narrow veins. Correct hole placement along the free face plays a key role in ore fragmentation and rock mass stability.

Pattern design involves careful planning of hole spacing, drilling angle, and orientation. These parameters, when properly optimized, allow for controlled ore release, minimizing waste inclusions and maximizing ore recovery.

In narrow veins, it is crucial to ensure that the hole spacing is not too large, as this could result in the inclusion of unwanted material within the drill and blast area. Accurate hole design in these conditions also helps avoid mixing ore with waste material. Tables 1 and 2 are based on rock mass classification, which plays a key role in hole placement to optimize drilling efficiency.

Table 1. Distance between drills

Rock Hardness	Distance Between Drill Holes (m)
Tough	0.50–0.55
Intermediate	0.60–0.65
Reliable	0.70–0.75

Table 2. Rock coefficient

Rock Hardness	Rock Coefficient (m)
Tough	2
Intermediate	1.5
Reliable	1

Table 1 presents the relationship between rock hardness and the recommended spacing between drill holes. Tougher rocks require shorter spacing between drill holes to ensure efficient fragmentation, while more brittle rocks can tolerate longer spacings without compromising fragmentation quality.

Table 2 indicates the rock coefficients associated with rock hardness. This coefficient is important for calculating the number of drill holes required, which will depend on the hardness and characteristics of the rock mass.

Furthermore, the choice of drill pattern directly influences the distribution of explosive energy and its impact on the rock mass. A well-designed drill pattern will allow for a homogeneous distribution of the explosive charge, facilitating mineral fragmentation without generating unwanted fractures at the bottom of the vein. The use of long drill holes combined with proper drill pattern design helps reduce the number of drill holes required, thus improving operational efficiency.

The number of drill holes is essential for determining the appropriate drill pattern. The formula used to calculate the number of drill holes required based on various parameters is presented below:

$$N° \, Tal = \frac{4 \times \sqrt{S}}{dt} + C \times S \quad (2)$$

where,

N° Tal: Number of drill holes required.

S: Area to be drilled (in m²).

dt: Distance between drill holes (in meters).

C: Rock coefficient.

The number of drill holes varies depending on the area to be drilled, the spacing between drill holes, and the rock coefficient. As rock hardness increases (which implies a higher coefficient), the number of drill holes required also increases to ensure effective fragmentation. The equation highlights the importance of adjusting the spacing between drill holes and the rock coefficient according to the characteristics of the rock mass, which is essential to optimize the drilling operation and ensure better mineral recovery.

2.3 Calculation of operating load

Calculating the maximum operating charge (Q'max) is a crucial aspect of blast design to optimize mineral fragmentation and rock mass stability in underground mining, especially in narrow veins. The operating charge determines the quantity of explosives to be used in the blast, and its correct adjustment is essential to achieve efficient fragmentation, reduce mineral dilution, and minimize the risk of slope instability.

To calculate the operating charge, several geomechanical and operational parameters must be considered, such as the shear strength, cohesion, and internal friction of the rock mass, as well as mineral characteristics such as hardness and compaction. In addition, the geometry of the vein must be considered, since, in narrow veins, the distribution of the explosive charge must be more precise to avoid excessive fracturing of the walls and reduce the amount of non-valuable material in the payload. Calculating the operating load involves a detailed assessment of the critical velocity (CV), which is the maximum velocity of the particles generated by the explosion without causing damage to nearby structures. This parameter is key for adjusting the number of explosives, preventing excessively strong vibrations, which could cause unwanted fractures in the rock mass. The critical velocity also makes it possible to determine the most suitable type of explosive for each rock type and the distance between drilling holes.

Once the critical velocity has been determined and the rock mass and vein characteristics have been considered, the explosive load is calculated using standard formulas based on the relationship between drill hole diameter, drilling depth, and explosive type. This calculation aims to ensure a homogeneous distribution of explosive energy to achieve efficient mineral fragmentation without compromising the stability of the slope walls.

The safety factor is also an important parameter when calculating the operating load. The operating load setting must ensure that the forces generated by the explosion do not exceed the strength of the rock mass and that the slope is not compromised. For this purpose, methods such as the limit equilibrium method under the Mohr-Coulomb criterion are used, which allows the stability conditions of the slope walls to be calculated under the stresses generated by the blast. In addition, the results of vibration and fragmentation tests are considered, allowing the operating charge to be adjusted to optimize blasting efficiency, minimizing the negative effects of vibrations and ensuring that the ore is adequately fragmented for processing without further dilution.

In practice, calculating the operating charge also involves simulating different blasting scenarios using software such as JK Simblast and DESWIK, which allow modeling of explosive charge distribution, rock mass behavior, and ore fragmentation. These programs provide valuable data for adjusting blast parameters and optimizing the operating charge according to the specific mine conditions and narrow veins.

2.4 Simulations and models

Simulations and models are fundamental tools for optimizing blasting in underground mining, especially when dealing with narrow veins, where operating margins are narrower and geomechanical conditions are more complex. The use of simulations allows modeling different blasting scenarios and predicting rock mass behavior under variations in drilling, blasting, and operating load parameters, facilitating more informed decision-making and optimizing operational results.

In this context, simulations are based on the use of specialized software such as JK Simblast and DESWIK.UGDB, which allow for the creation of detailed blast models, simulating mineral fragmentation behavior, and evaluating the effects of vibrations and fractures generated by the blast. These programs allow for the incorporation of geomechanical data obtained from laboratory tests and field studies, such as shear strength, cohesion, friction, and vein geometry. Through these models, critical parameters such as the number of explosives, hole distribution, and detonation pattern can be adjusted, helping to achieve more controlled fragmentation, reduce dilution, and optimize rock mass stability.

One of the most significant advantages of simulations is the ability to perform multiple blasting scenarios without the need for costly field testing. Simulations allow for evaluating how different combinations of operating charge, explosive types, and hole arrangement affect fragmentation and slope stability. This also helps predict blasting behavior in specific geomechanical conditions, such as those found in narrow veins, where rock variability is high, and conditions are more challenging.

Simulation models also allow for calculating the critical velocity, which is key to determining the number of explosives to use without compromising mine stability and ensuring that the generated seismic waves do not affect nearby structures. These models provide detailed information on seismic wave propagation, allowing explosive charges to be adjusted to avoid unwanted fractures and improve mineral fragmentation.

Regarding fracturing models, simulations also make it possible to predict how fractures generated by blasting will affect the integrity of the rock mass. In the case of narrow veins, it is crucial to avoid fractures that could compromise slope stability or dilute the extracted mineral. Fracturing models allow the simulation of fracture expansion and propagation, helping to adjust the blast energy so that it is sufficient for the desired fragmentation, but without causing excessive damage to the rock mass.

Statistical analyses performed in simulations also play an important role, as they allow for the assessment of the variability and uncertainty associated with blasting results. Using techniques such as Monte Carlo analysis, it is possible to estimate the range of possible outcomes and adjust blasting parameters to maximize operational efficiency. This information is essential for planning blasts more efficiently and with greater certainty, reducing the risk of undesired events and improving operational safety.

2.4.1 Characterization of the inert material used as DECKs

Inert materials used as DECKs—quartz sand and crushed andesitic gravel—were characterized for their role in energy modulation during blasting. Selected for their inert nature, low water absorption, and mechanical stability, these materials were modeled in simulations (JK Simblast and DESWIK.UGDB) to assess their impact on fragmentation, seismic wave attenuation, and blast damage reduction. The DECKs were positioned between explosive charges with an average thickness of 0.30 m per segment, adjusted to the blast geometry, allowing for predictive analysis of variables such as critical velocity, PF and fragmentation uniformity.

To ensure the validity of the comparative analysis, the control blasts (without DECKs) were conducted under the same geotechnical and operational conditions as those with DECKs. Parameters such as explosive type, borehole diameter, hole spacing, detonation timing, and rock mass characteristics were kept constant in both cases. This methodological consistency guarantees that the differences observed in fragmentation, stability, and dilution can be attributed exclusively to the use of DECKs.

2.4.2 Experimental setup for reproducibility

To ensure that the study can be reliably reproduced under similar conditions, a standardized experimental setup was carefully established. This procedure aimed to maintain consistent geomechanical and operational variables, allowing the observed effects to be attributed exclusively to the implementation of DECKs. The main technical parameters applied were as follows:

- Equipment used: MUKI LHBP 2R jumbo, specialized for narrow vein operations.
- Drill hole length: 12 meters per hole.

Drill diameter:

- Production holes: 64 mm.
- Relief holes: 127 mm.
- Drilling pattern: Zigzag configuration with 0.4 to 0.5 m spacing and 1.2 m burden, adjusted to vein thicknesses ranging from 0.6 to 1.0 m.
- Explosives: ANFO as the main charge, combined with Emulnor 3000 (1½" × 12") primers and 25 ms delays using MS fanels.

DECK configuration:

- Three segments per hole.
- Material: Dry quartz-based sand or andesitic gravel.
- Segment thickness: 0.30 m, placed between explosive charges.
- Controlled variables: Rock type, loading pattern, delay timing, drill diameter, and geomechanical conditions were kept constant across all tests to ensure comparability.

3. RESULTS

3.1 Results of geomechanical parameters of drilling and blasting

Geomechanical analysis is a fundamental component of rock mass characterization, as it provides the basis for understanding its behavior under the loads and operating conditions under which mining activity will take place. In this study, the Pampeñita Vein, one of the main geological structures in the Sierra Antapite mining unit, presents parameters including relative density (ρ_r), rock classification using RMR, the GSI index, the rock quality index (RQD), compressive strength (σ_c), and the geomechanical index (Q). These parameters were evaluated for various sections of the vein, including the Caja Piso, Caja Techo, and Mineral areas, with the aim of providing a comprehensive view of the geomechanical variations throughout the structure. The results obtained indicate that the Pampeñita Vein has a fair to good geomechanical quality classification, implying that, although the rock is suitable for mining operations, there are certain areas that may require additional reinforcement due to its degree of fracturing and fragmentation. These values provide key information for the design of support systems and exploitation strategies, aiming to optimize safety and efficiency in drilling and extraction activities.

Detailed results of the geomechanical parameters are presented in Tables 3 and 4.

The geomechanical classification of the Pampeñita Vein reveals variable rock mass quality, with some areas showing high fragmentation and low cohesion that may compromise stability. This variability highlights the need to adapt support design, with DECKs being an effective solution to improve stability and safety in the most critical zones.

Table 4 shows that the rock mass has generally favorable conditions, but moderate fracturing requires proper support design to ensure stability and safety in mining operations.

3.2 Slot drilling mesh design (free face) and production results

3.2.1 Slot perforation mesh (free face)

The drill pattern designed for the Pampeñita Vein aimed to efficiently generate a slot (free face) to initiate ore extraction. Practical and perimeter-based methods were applied to determine the optimal number of drill holes, considering the geomechanical conditions of the rock mass. The final design, with a 1.50 m × 1.50 m section, included 12-meter-longhole stoping drilled directly into the quartz vein hosting the gold oreization. Production holes were 64 mm in diameter, with reaming at 127 mm. This configuration ensured drilling stability and improved blasting efficiency. The setup is illustrated in Figure 1.

Application of the Holmberg and Pearson formula:

- **Initial parameters**

$\emptyset_{prod}$= 0.064 m, Production drill diameter

$\emptyset_{rimado}$= 0.127 m, Diameter of reamed holes

Table 3. Geomechanical classification of the Pampeñita Vein

Vein	Range/Quality	CP	Cashier Floor	Mineral	CT	Roof Box
		Remote	Immediate		Immediate	Remote
Pampeñita	RMR	56–70	52–65	54–58	45–55	56–70
	Quality (type)	IIIA-II	IIIA	IIIA	IIIB-IIIA	IIIA-II

Table 4. Geomechanical parameters of the Pampeñita Vein

Parameter	Value	Analysis
ρ_r (Relative density)	2.6	Relative density of the rock, representing a moderately dense material. Important for load and bearing capacity assessments.
RMR (Geomechanical Classification)	56	An RMR rating of 56 indicates a rock mass of fair quality (III-A), suggesting that the ground has moderate resistance to fracturing.
GSI (Geomechanical Index)	51	A GSI of 51 suggests a rock mass with significant fracturing. The quality of the rock mass could be affected by a high degree of fracturing.
RQD (Rock Quality Index)	60%	An RQD of 60% indicates a rock with moderate fragmentation, which may pose a stability challenge in some areas of the mine.
σ_c (Compressive Strength)	143	A compressive strength of 143 MPa suggests a high-strength material, suitable for supporting heavy structures without significant deformation.
Q (Geomechanical Index)	3.79	A Q value of 3.79 suggests favorable conditions for stability, but reinforcement may be required in more fractured areas.

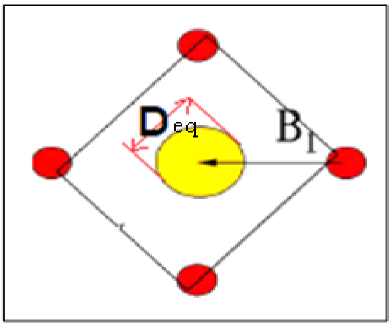

Figure 1. Equivalent diameter starter design

From Figure 1, the starting design is cylindrical, where D represents the diameter of the equivalent (reamed) drill, yellow, and B represents the burden, distance from the loaded drill (red) to the reamed drill.

The equivalent diameter (D_{eq}) must be large in relation to the depth of the drill, allowing at least 95% progress in each shot.

3.2.2 Design coefficients

- Burden coefficient for relief drills (K_b)

When analyzing the stability and performance of relief drill holes in drilling projects, one of the key factors to consider is the burden coefficient (K_b), which varies depending on the hardness of the rock being drilled. This coefficient is essential for determining the efficiency and effectiveness of the drilling process, as it directly influences the selection of equipment and techniques, and impacts operational safety.

The classification of burden coefficients (K_b) according to the type of rock and its hardness is presented in Table 2, which facilitates the identification of the appropriate values for each specific situation in the drilling of relief holes.

Table 5 presents the classification of burden coefficients (K_b) by rock type and hardness, which facilitates the identification of appropriate values for each specific relief drilling situation.

From Table 5, for medium hardness rocks the value of K_b is adjusted from (3 to 4) for narrow veins.

Table 5. Burden coefficients K_b

Rock Type	Hardness	K_b Value
Very soft rock	Low	11–12
Soft rock	Medium-Low	12–15
Medium-hard rock	Medium	15–20
Hard rock	Medium-High	20–25
Very hard rock	High	25–30

- Spacing coefficient K_S

In the context of drilling, another key parameter to consider is the spacing coefficient (K_S), which is used to determine the appropriate spacing between drill holes. This coefficient varies according to rock hardness, since the strength and fracturability of the material directly influence stress distribution during drilling. Table 6 shows the recommended values for the spacing coefficient (K_S), classified according to rock type and hardness.

Table 6. Spacing coefficient K_S

Rock Type	Hardness	K_S Value
Very soft rock	Low	0.8–1.0
Soft rock	Medium-Low	1.0–1.1
Medium-hard rock	Medium	1.1–1.2
Hard rock	Medium-High	1.2–1.4
Very hard rock	High	1.4–1.6

Table 6 indicates that as rock hardness increases, greater spacing between drill holes is required (higher K_S), while in softer rocks, closer spacing is allowed to optimize drilling. K_S = 1.1, typical range for narrow vein control.

For the design, the requested slot must have dimensions of 1.5 m × 1.5 m, and the equivalent diameter (D_{eq}) must be determined to obtain homogeneous values in the application of burden and spacing.

$$D_{eq} = D_{rhymed} \times \sqrt{N} \tag{3}$$

where,

D_{eq}: Equivalent diameter

D_{rhymed}: Reamed drill diameter

N: Total number of reamed drills

The EXSA manual tells us that to generate an optimal free face for a 10 or 12-meter slot, three or four reamed holes should be used. In this sense, four holes are considered.

$$D_{eq} = 0.127 \times \sqrt{4} = 0.254 \text{ m}$$

- Calculating the burden

$$B = K_b \times D_{eq} \tag{4}$$

$$B = 3 \times 0.254 = 0.76 \text{ m}$$

- Spacing calculation

Using Holmberg's formula for spacing and a coefficient of 1.1, common for regular hardness rock.

$$S = K_s * B \tag{5}$$

$$S = 1.1 \times 0.76 = 0.84\ \text{m}$$

- Calculation of the number of relief holes (N)

To cover the area of the slot, it is necessary to calculate the number of relief holes.

$$N = \frac{Total\ Area}{B \times S} \tag{6}$$

$$Total\ Area = 1.5\ \text{m}\ \times\ 1.5\ \text{m} = 2.25\ \text{m}^2$$

$$N = \frac{2.25\ \text{m}^2}{0.76\ \text{m} \times 0.84\ \text{m}} 3.5 \cong 4\ \text{drills}$$

Rounded to 4, the value of 4 relief holes is within the acceptable range. The value given by EXSA and the Holmberg and Pearson method are checked, and they coincide.

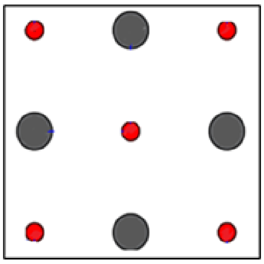

Figure 2. Slot START design

Figure 2 shows the start with a breaker, which is the center drill, i.e., the initiator for generating free faces. This step is key in mining drilling planning, as it defines the initial geometry of the excavation and establishes the pattern for subsequent work progress. Furthermore, by designing the drill pattern, as illustrated in Figure 3, it is possible to optimize the performance of the MUKI LHBP 2R Jumbo rig, ensuring proper hole distribution, which improves material fragmentation and operational safety.

Figure 3 shows a 1.5 m × 1.5 m cross-section pattern for creating the free face. Four relief holes with a diameter of 127 mm and 13 production holes with a diameter of 64 mm are shown.

A well-designed drilling pattern also facilitates the reduction of operating costs, as it maximizes the efficiency of resource and equipment utilization, minimizing possible failures or deviations during the drilling process.

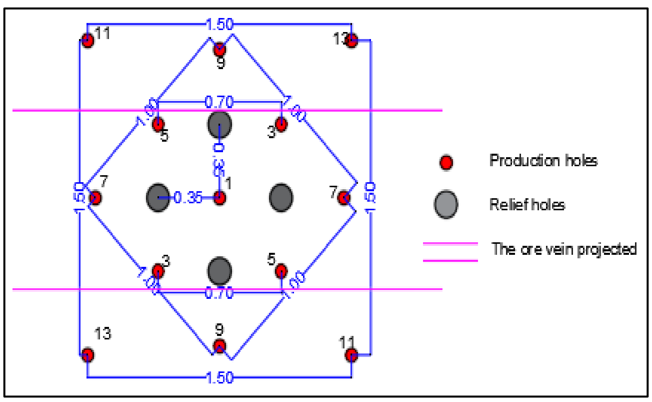

Figure 3. Slot perforation mesh 1.5 m × 1.5 m

3.2.3 Production drilling mesh

The Pampeñita Vein has an average thickness of 0.60 m, with a minimum of 0.40 m and a maximum of 1.0 m. Once the free face has been generated, the pit is exploited; for this, the burden and spacing are calculated.

- Calculation of burden and spacing with the Torbica-Lapcevic and Konya Models.

We will compare two models for calculating the load: the modern model and the classic model.

Modern model

$$B = 0.17 \times PD_e \times (\frac{\emptyset_{tal}}{2 \times K \times \sigma_\tau}) \tag{7}$$

$$K = \frac{1 - v}{(1 + v)(1 - 2v)} \tag{8}$$

where,

$$B = Burden\ (\text{m})$$
$$PD_e = 51\ \text{KBar} = 5100\ \text{MPa (Anfo)}$$
$$\emptyset_{tal} = 51\ \text{mm} = 0.051\ \text{m};\ \sigma_\tau = 15\ \text{MPa}$$
$$\sigma_c = 150\ \text{MPa};\ v = 0.17$$

So

$$K = \frac{1 - 0.17}{(1 + 0.17)(1 - 2 \times 0.17)} = 1.075$$

$$B = 0.17 \times 5100\ \text{MPa} \times (\frac{0.051\ \text{m}}{2 \times 1.075 \times 15\ \text{MPa}}) = 1.30$$

Konya model

$$B = 0.012 \times \left(2 \times \frac{\rho_e}{\rho_r} + 1.5\right) \times \emptyset_{tal} \tag{9}$$

where,

$$B = Burden\ (\text{m});\ \rho_e = 0.8\left(\frac{g}{cc}\right)$$
$$\rho_r = 2.6\left(\frac{g}{cc}\right);\ \emptyset_{tal} = 51\ (\text{mm})$$

So

$$B = 0.0.12 \times \left(2 \times \frac{0.8}{2.6} + 1.5\right) \times 51 = 1.20\ \text{m}$$

In burden's calculation the following is observed:

$B_{max} = 1.30$

$B_{min} = 1.20$

For the mesh design it is considered:

Burden (Theoretical) = 1.20 m;

Burden (Practical) = 0.60 m;

Spacing = 0.40 m – 0.50 m

A 12 cm deviation will be considered on a 12 m bench (Muki LHBP 2R rig, worn).

Figure 4 shows the drill pattern design for the production holes. This design is optimized for veins less than 1.0 m thick, where a zigzag pattern will be used, and the relief holes will be oriented toward the roof box. This configuration ensures greater blasting efficiency and control, maximizing operational safety and performance. This design not only improves extraction efficiency but also contributes to process safety by minimizing potential ground failures during blasting.

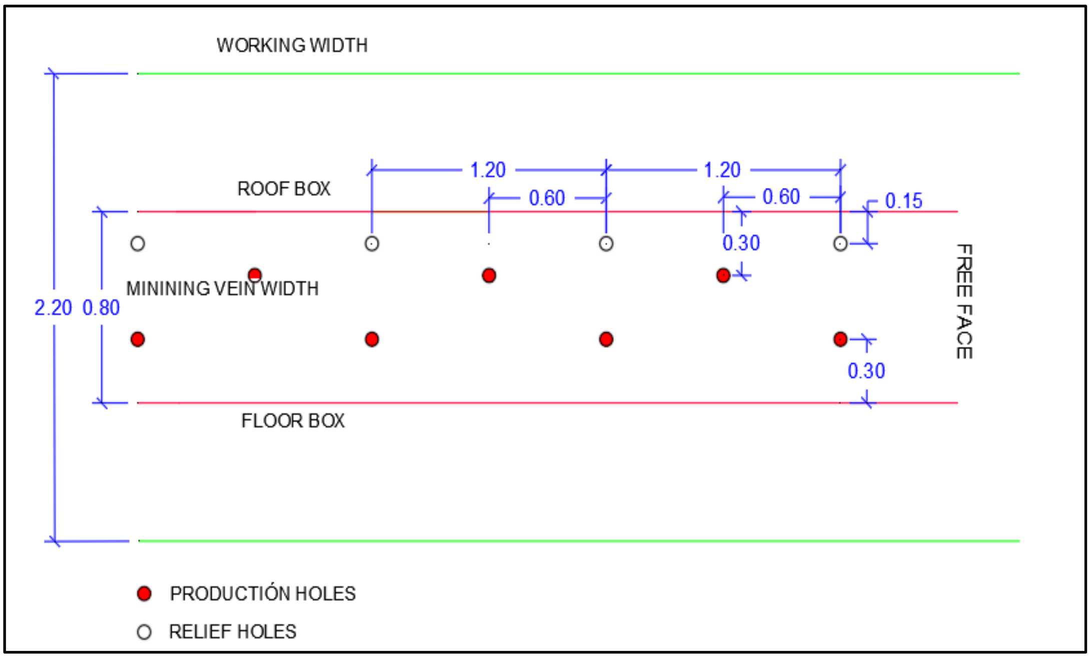

Figure 4. Grid design-production drills

Figure 4 shows a detailed configuration of the drilling pattern, specifically designed for production holes in a working space with a working width of 2.20 m. The average vein thickness is 0.8 m, with variations ranging from 0.4 m to 1.2 m. This variability is crucial for adapting the drilling to the deposit's characteristics, allowing for optimized material fragmentation and explosive consumption.

Production holes are drilled at 0.30 m from the casings, allowing for an efficient blasting pattern, while relief holes, located 0.15 m from the casing roof, help reduce stress and improve blast control. Zigzag drilling, specifically designed for veins less than 1.0 m thick, ensures a more homogeneous distribution of explosive charges, reducing operational risks and increasing fragmentation accuracy.

3.3 Results of the operating load and blast design

This section presents the results derived from the blast design for Pit 431, aimed at optimizing rock fragmentation and ensuring safety during mining operations. A series of key geomechanical parameters, such as RMR, Young's modulus, and critical velocity, among others, have been considered to determine the required operating load and the maximum operating load that should not be exceeded, all with the aim of preventing damage to the rock mass and minimizing the risk of excessive vibrations. In addition, DEKs have been used to optimize blasting in narrow veins and reduce dilution and improve recovery. Through detailed calculations and simulation in Python, the most appropriate operating load is established for the geomechanical conditions of the pit, ensuring efficient and controlled blast execution.

3.3.1 Calculation of variables of the maximum operating load according to damage criteria

- Geomechanical parameters of Tj. 431 The following data are available:

$$RMR = 56\ (Regular\ III - A)$$
$$\sigma_c = 143\ \text{MPa}$$
$$\sigma_\tau = 0.1 * \sigma_c$$
$$\sigma_\tau = 0.1 * 143\ \text{MPa}$$
$$\sigma_\tau = 14.3\ \text{MPa}$$

$$Q(Bartón) = e^{\frac{RMR-44}{9}} \tag{10}$$

$$Q = e^{\frac{56-44}{9}} = 3.79$$

$$GSI = RMR - 5 \tag{11}$$

$$GSI = 56 - 5 = 51$$

- Calculation of the propagation velocity of the p wave (V_p)

$$V_p = 3500 + 1000 \times \log{(Q)} \tag{12}$$

$$V_p = 3500 + 1000 \times \log{(3.79)} = 4079\ \text{m/seg}$$

- Calculation of Young's Modulus in situ

To calculate Young's Modulus, Hoek and Diederich's estimates were used, which are based on Damage and GSI.

Based on Hoek and Diederich's estimates, the Young's modulus of the rock mass is calculated.

$$E_{mr} = 100000 \times \left[\frac{1 - \frac{D}{2}}{1 + e^{\left(\frac{75+25\times D-GSI}{11}\right)}}\right] \tag{13}$$

Considering that there will be no harm (D = 0)

$$E_{mr} = 100000 \times \left[\frac{1 - \frac{0}{2}}{1 + e^{\left(\frac{75+25\times 0-51}{11}\right)}}\right] = 10139.5\ \text{MPa}$$

Finally, the in-situ deformation is calculated.

$$\frac{E_{mr}}{E_i} = \left[0.02 + \frac{1-\frac{0}{2}}{1+e^{\left(\frac{60+15*0-51}{11}\right)}}\right]$$

$$E_i = \frac{E_{mr}}{\left[0.02 + \frac{1-\frac{D}{2}}{1+e^{\left(\frac{60+15\times D-GSI}{11}\right)}}\right]}$$

$$E_i = \frac{9367.82}{\left[0.02 + \frac{1-\frac{0}{2}}{1+e^{\left(\frac{60+15\times 0-51}{11}\right)}}\right]} = 28721 \text{ MPa}$$

$$\boldsymbol{E_i = 28.72 \text{ GPa}}$$

- Calculation of the critical velocity (V_{crit}) in rock.

$$V_{crit} = \left(\frac{\sigma_\tau \times V_p}{E_i}\right) \tag{14}$$

$$V_{crit} = \left(\frac{14.3 \text{ MPa} \times 4079 \text{ m/seg}}{28.72 \text{ GPa}}\right) = 2030.978 \text{ mm/seg}$$

$$\boldsymbol{V_{crit} = 2030 \text{ mm/seg}}$$

This speed (V_{crit} = 2030 mm/sec) is what resists the rock in vibration.

3.3.2 Calculation of operating load (Q')

The calculation of the operating load in mining is essential to determine the amount of explosives needed to fragment the rock safely and efficiently. Figure 5 is presented below:

Drill diameter (mm)	51
K	357
Alfa	-2,07
VPPcrit (mm/s)	2030

Dist. (m)	Q (kg)
0,01	1E-05
0,02	0,0001
0,04	0,0008
0,06	0,0027
0,08	0,0064
0,10	0,01
0,12	0,02
0,14	0,03
0,16	0,05
0,18	0,07
0,20	0,10
0,30	0,34
0,40	0,79
0,50	1,55
0,60	2,68
0,70	4,26

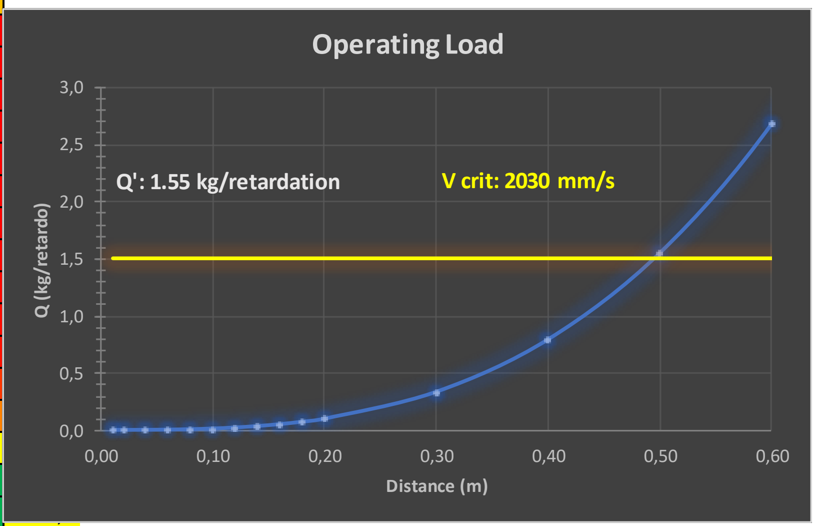

Figure 5. Operating load calculation

Figure 5 shows the operating load of 0.50 meters, which is 1.55 kg/delay. The critical speed is 2030 mm/s.

To detail the calculations, it is considered: d = 0.50 m, from the vein to the contact zone:

$$K_{prom} = 357;\ \propto_{prom} = -2.07$$

$$VPP = k \times \left(\frac{d}{Q'^{\frac{1}{3}}}\right)^{-\propto} \tag{15}$$

Equalizing

$$VPP = V_{crit};\ V_{crit} = k \times \left(\frac{d}{Q'^{\frac{1}{3}}}\right)^{-\propto}$$

Clearing the Q' (Operant Load)

$$Q' = \left(\frac{V_{crit} \times d^{-\propto}}{k}\right)^{3/-\propto} \tag{16}$$

$$Q' = \left(\frac{2030 \text{ mm/seg} \times (0.5)^{-(-2.07)}}{357}\right)^{3/-(-2.07)} = 1.55 \text{ kg}$$

- Calculation of the maximum operating load (Q'max).

Figure 6 represents the damage criteria. At 0.50 meters, the critical velocity is 2842 mm/s. This is colored yellow, indicating a damage threshold, with the creation of new fractures.

Damage criterion: (V_{crit}*1.4) new fractures are generated:

$$Vcrit. = 1.4 \times 2030 = 2842 \frac{\text{mm}}{\text{seg}}$$

$$Q'max = \left(\frac{2842 \text{ mm/seg} \times (0.5)^{-(-2.07)}}{357}\right)^{3/-(-2.07)} = 2.53 \text{ Kg}$$

Maximum Operant Load. Load that should not be exceeded according to the vibration limits, the maximum operating load is shown in Figure 7.

DIAM. HOLE	51 mm
K	357,00
Alfa	-2,07
Q (kg)	2,53
Dist. (m)	VPP (mm/s)
0,01	9350210
0,02	2226841
0,04	530343
0,06	229112
0,08	126306
0,10	79583
0,12	54565
0,14	39659
0,16	30081
0,18	23573
0,20	18953
0,30	8188
0,40	4514
0,50	2844
0,60	1950
0,70	1417
0,80	1075
0,90	842
1,00	677
1,10	556

TIPO DE DAÑO	CRITERIO DE DAÑO (Vcrit.)	Vpp critico
Intense fracturing	> (4 * PPVcritico)	11368
Creation of new fractures	>= (1-1.4 * PPVcritico)	2842
Mild propagation of pre-existing fractures	< (1 / 4 * PPVcritico)	711

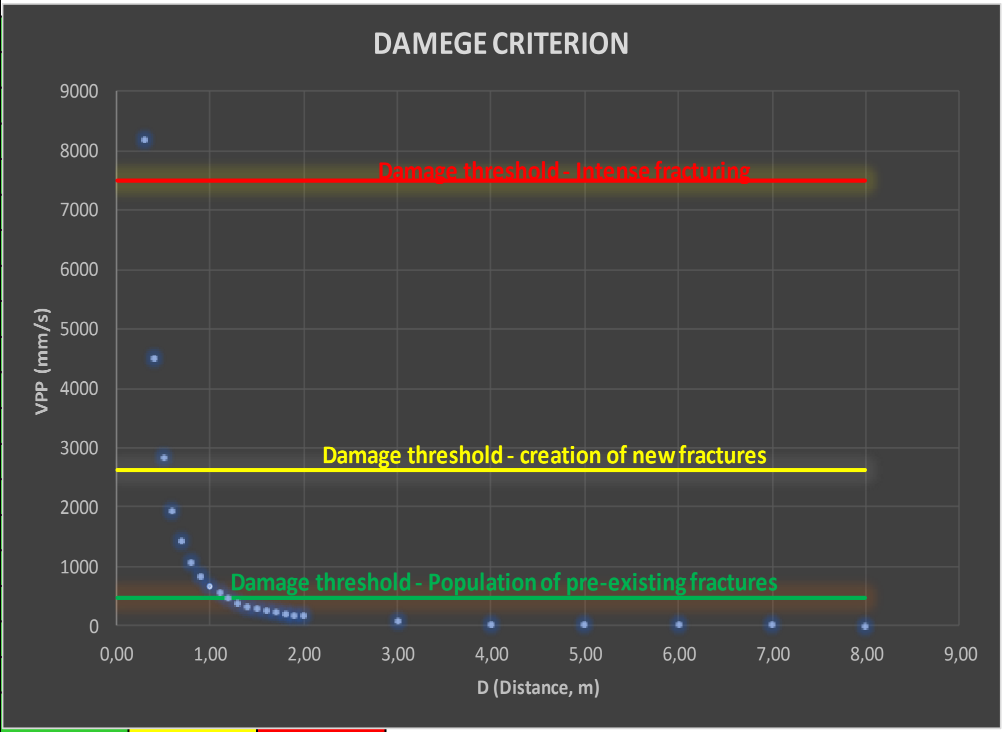

Figure 6. Maximum operating load calculation

Diam. Tal. (mm)	51
K	357
Alfa	-2,07
VPPcrit (mm/s)	2842

Dist. (m)	Q' max (kg)
0,01	2E-05
0,02	0,0002
0,04	0,0013
0,06	0,0044
0,08	0,0104
0,10	0,02
0,12	0,03
0,14	0,06
0,16	0,08
0,18	0,12
0,20	0,16
0,30	0,55
0,40	1,29
0,50	2,53
0,60	4,37
0,70	6,93
0,80	10,35
0,90	14,74
1,00	20,22

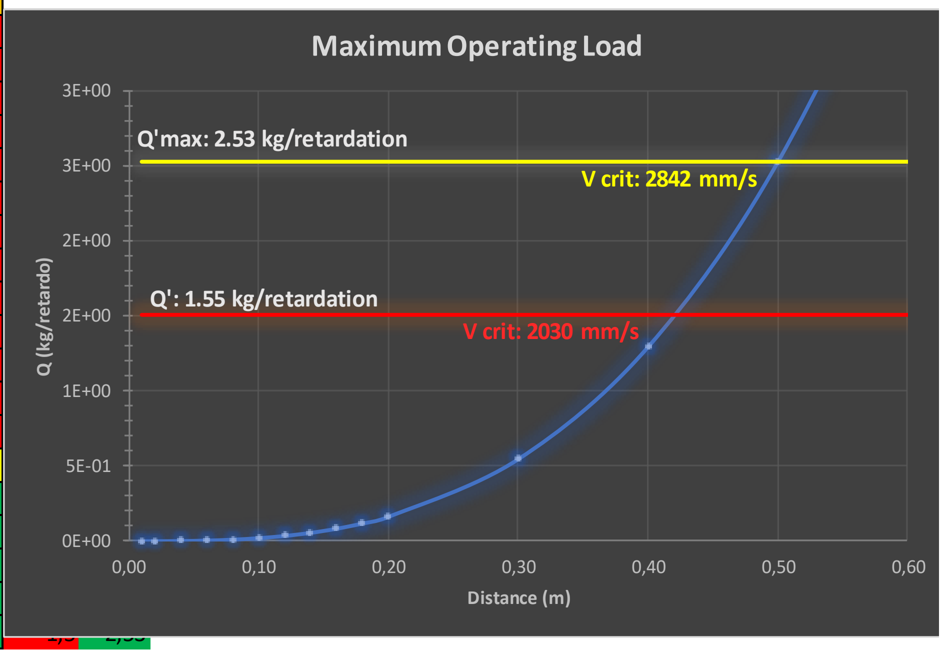

Figure 7. Maximum operating load

Figure 7 shows the relationship between distance and maximum operating load. At 0.50 meters, the maximum operating load is 2.53 kg/delay. This is shown in yellow.

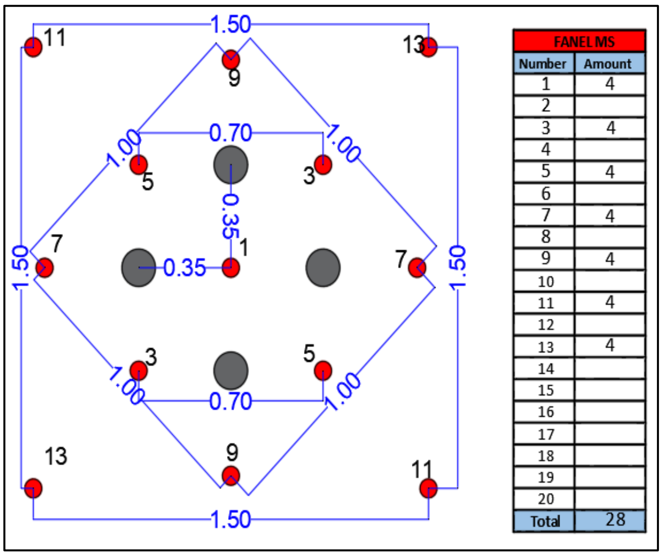

Figure 8. Maximum operating load

3.4 Simulation results and models

3.4.1 Slot or free face blasting design

Slot blasting design in the exploitation of narrow veins with longhole stoping is a fundamental technique for initiating rock fragmentation and creating a space for subsequent hole charges to expand. Blasting is conventional, using short-period (MS) delays with 25 ms intervals. Adequate delays between holes are used to ensure controlled fragmentation and that the rock has room to move toward the free face. The maximum operating load design is shown in Figure 8.

Figure 8 shows the exit sequence from the inside out, starting with number 1, known as the mouth breaker, to generate the free face with the relief holes. Four fanels are used per hole, i.e., four primers.

For a total of 28 fanels, the explosive loading design for the free face is shown in Figure 9.

From Figure 9, the explosive charge design for a 12 m length consists of four primers per hole with the same delay number, starting the exit through the mouth breaker to generate the free face with the reamed holes. The figure was developed by the company using DESWIK.UGDB software.

Figure 10 shows the simulation of damage in the slot blast.

3.4.2 Blasting design of production drills

A drilling pattern was designed to optimize fragmentation and minimize damage to the rock mass. Appropriate spacing between drill holes was used to ensure efficient blast coverage, as shown in Figure 11.

The use of DECKs made of dry quartz sand and crushed andesitic gravel was evaluated for their mechanical stability and chemical inertness. Inserted between explosive segments, they enabled controlled energy distribution, reduced unwanted fracturing, improved blast selectivity, and enhanced ore recovery. Both materials were physically and chemically characterized to confirm their effectiveness as energy-dissipating elements. Table 7 presents their key properties.

Figure 11 shows the explosive charge design for production drill holes in narrow veins with a length of 12 meters. Different delays are used, starting from the free face, and DECKs (spacers) are used to fragment the explosive charge.

Figure 12 shows the charge design and output sequence using JK Simblast 2D Ring software, which allows for design, simulation, and testing before blasting.

Figure 12 also shows the loading design in the drill hole facing the floor box, and an empty drill hole facing the roof box, which is where the greatest control is required. In the loaded drill hole, the ANFO charge is colored yellow, the spacer deck is colored lead, and the primer, which is differentiated by the exit time, is colored red.

Figure 13 shows the blasting simulation.

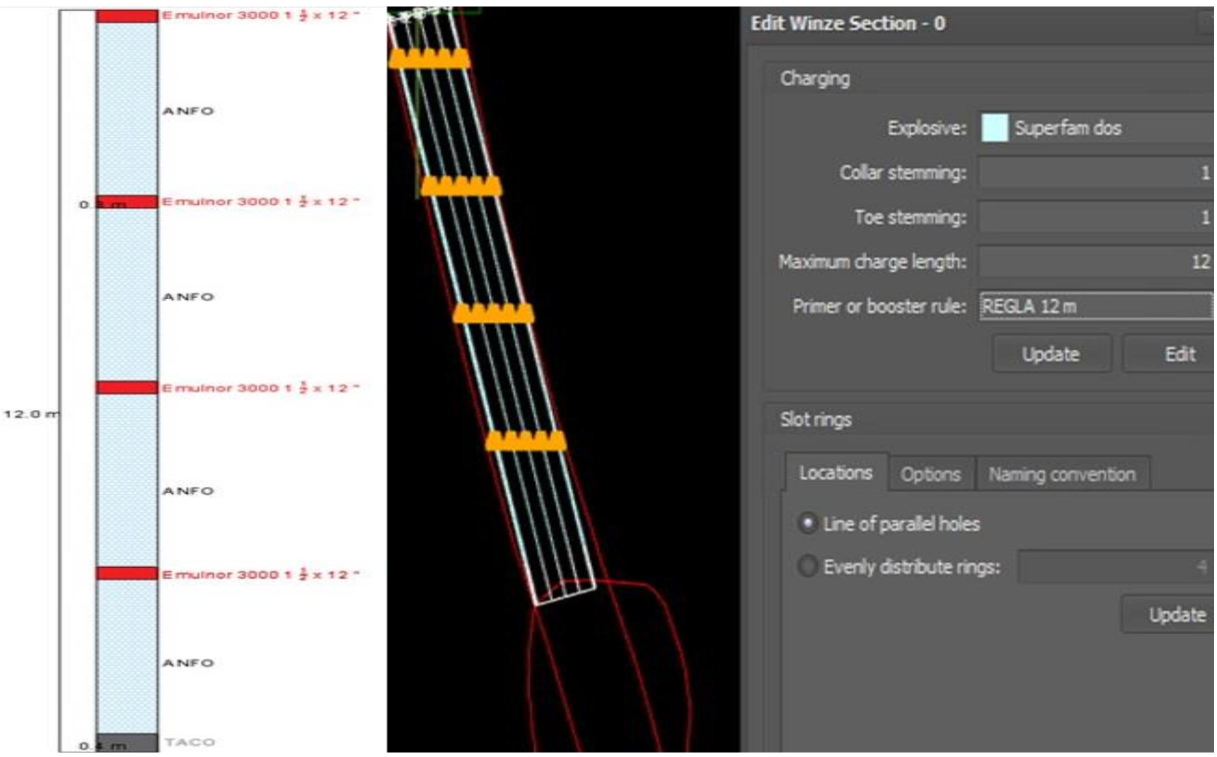

Figure 9. Design of explosive charge for the free or slot face with DESWIK software

PARAMETERS

Explosive	SUPERFAM DOS	
Diameter	51,00	mm
Carga	1,2	mm
k	357,00	
Alfa	-2,07	
Long. Initial Taco	0,50	m
Long. Taco Final	0,00	m
Largo. Carg.	11,50	m
Long. Drill	12,00	m
Linear charge density	1,60	kg/m
Explosive W	18,40	kg
Explosive density	0,80	g/cc
Rock density	2,6	g/cc

DAMAGE CRITERION

DAÑO		VPP	d (m)
Intense fracturing	(4 x VPPc)	11368	0,50
Creation of new fractures	(1-1.4 x VPPc)	2842	0,90
Extender fracturas preexistentes	(1/4 x VPPc)	711	1,30

VPPcritical = σt x Vp / Ei — Ec. Critical particle velocity of the rock

Donde

VPPcritical	Critical particle velocity - maximum	2842
σt	Tensile strength	14
Vp	Wave velocity P	4079
Ei	Young's modulus: Modulus of elasticity of the intact rock (GPa)	28720

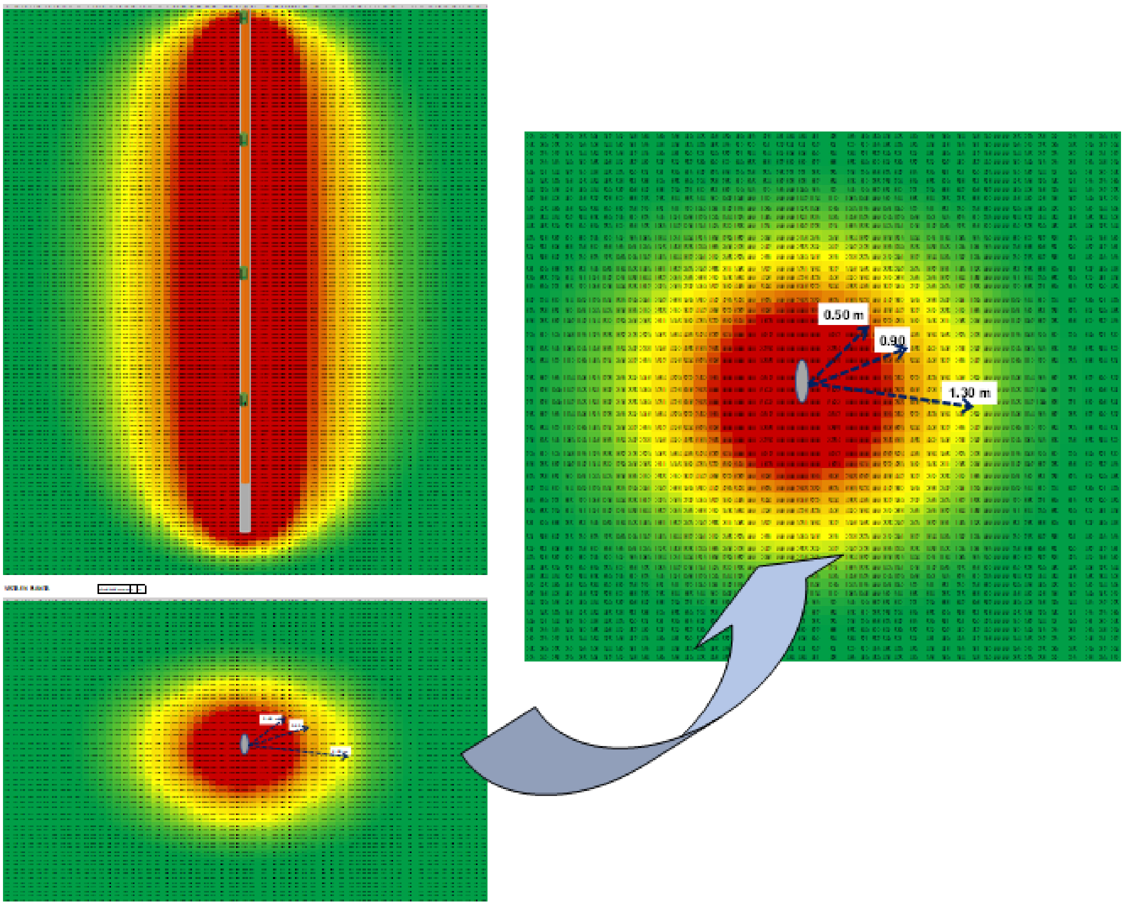

Figure 10. Slot damage analysis and simulation (Excel)

Table 7. Physical and chemical properties of the inert materials used as DECKs

Property	Quartz-Based Sand	Andesitic Gravel
Particle size	0.3–0.6 mm	6–12 mm
Density (g/cm³)	2.65	2.7
Main composition	>95% silicon dioxide (SiO_2)	Silica and andesite
Water absorption (%)	< 0.5	0.5–1.5
Chemical reactivity	Inert	Inert
Mechanical behavior	Stable and incompressible	High resistance and stability

Figure 13 shows the influence of the energy of the explosive as a function of the Load Factor in kg/ton, that is, based on the amount of explosive used for 1 (one) ton of ore. The rock density (quartz vein) is 2.6 g/cc. The simulation shows different colors that indicate the energy levels, the most critical being red, yellow for controlled damage, green where the damage is minimal, and blue for the area where there is no damage.

On the other hand, Figure 14 is presented, which responds to the Simulation of the Influence of the energy of the explosive with JK Simblast.

Figure 14 shows the detonation and vibration levels produced by the simulated blast. The sequential detonation of the primers can be seen, starting from the bottom, with the lowest number, 25 milliseconds, followed by the delays spaced 25 ms apart. The colors are damaged indicators: red indicates the crushed zone, yellow indicates the area where new fractures are generated, green indicates areas where pre-existing fractures are present, and blue indicates the unaltered zone.

3.4.3 Comparison before and after the implementation of DECKs

Initially, longhole stoping were fully loaded. After observing the negative results, the use of desks was proposed to reduce the operating load and thus obtain better results.

- Comparison of load design before and after using DECKs.

Figure 15 shows the difference in loading design. The first involves fully loaded holes, that is, to their entire length and with the same number of holes (delays). This generates excess energy during blasting, which directly affects the stability of the rock mass and consequently increases dilution, resulting in ore loss in the pit. The second drill uses DECK technologies, thus reducing the explosive charge, which was previously calculated using the critical velocity. For better control, delays are used with a difference of 25 milliseconds (MS), which reduces the energy generated by the detonation.

Figure 15 shows the difference in the charge design, where Emulnor 3000 1 ½ × 12" primers are used, and ANFO as the explosive agent. The charge design on the left side is a traditional charge design, having the same number of delays (25 MS), the primers detonate at the same time, which generates excess energy. For better control of this excess, DECKs are used, their charge design is the figure on the right, when using DECKs, and delays with different numbers, the energy decreases considerably.

- Comparison of results before and after using DECKs.

Figure 16 clearly contrasts the results. The image on the left is the result of a traditional blast, showing a slump, where mineral loss is due to the inability to recover the mineral buried in waste rock banks caused by the excess energy of the explosive. The image on the right is the result of blasting using DECKs, showing good control of the slumps, which favors mineral recovery.

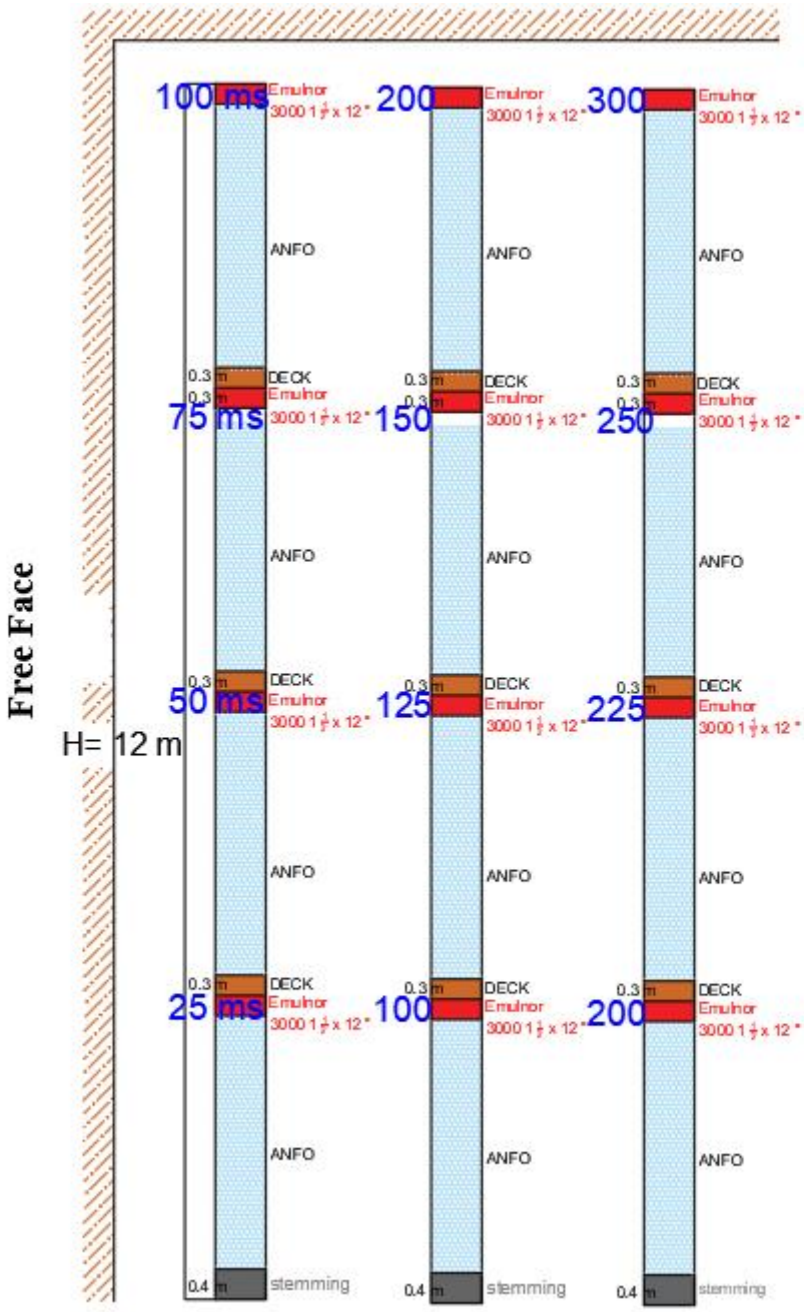

Figure 11. Explosive charge design – production drills

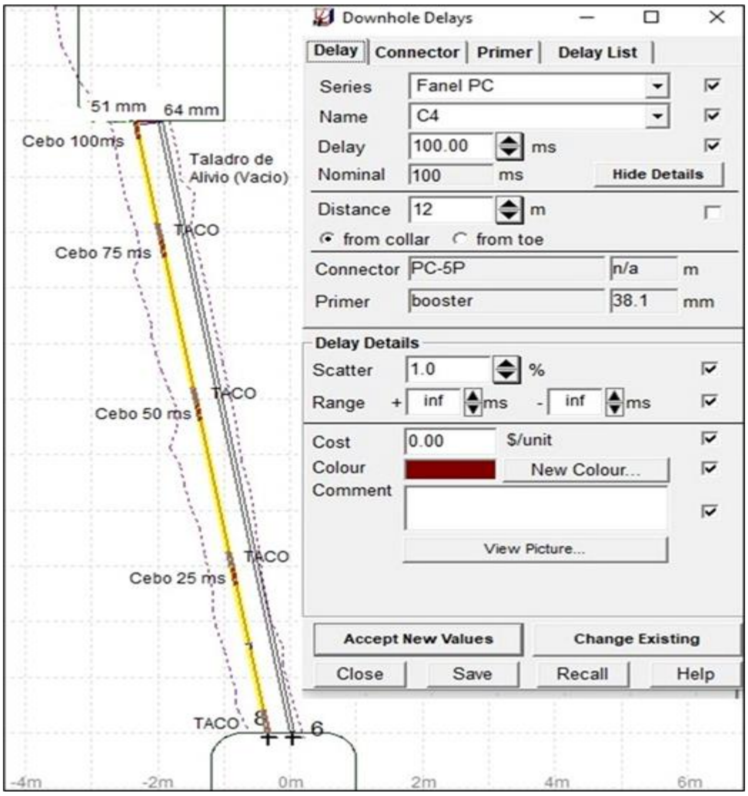

Figure 12. Load design and output sequence with JK Simblast 2D Ring Software

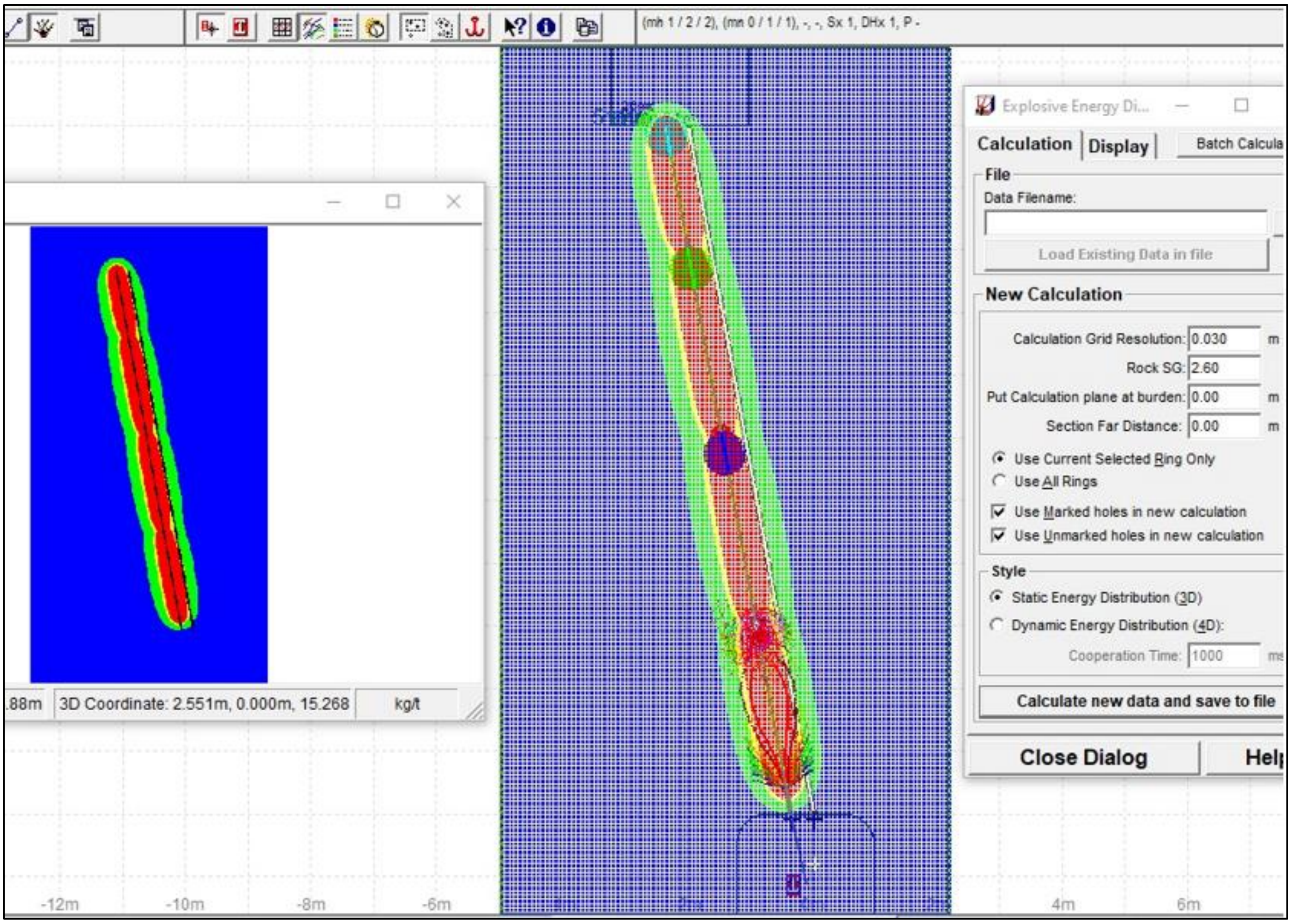

Figure 13. Simulation of the influence of explosive energy with JK Simblast

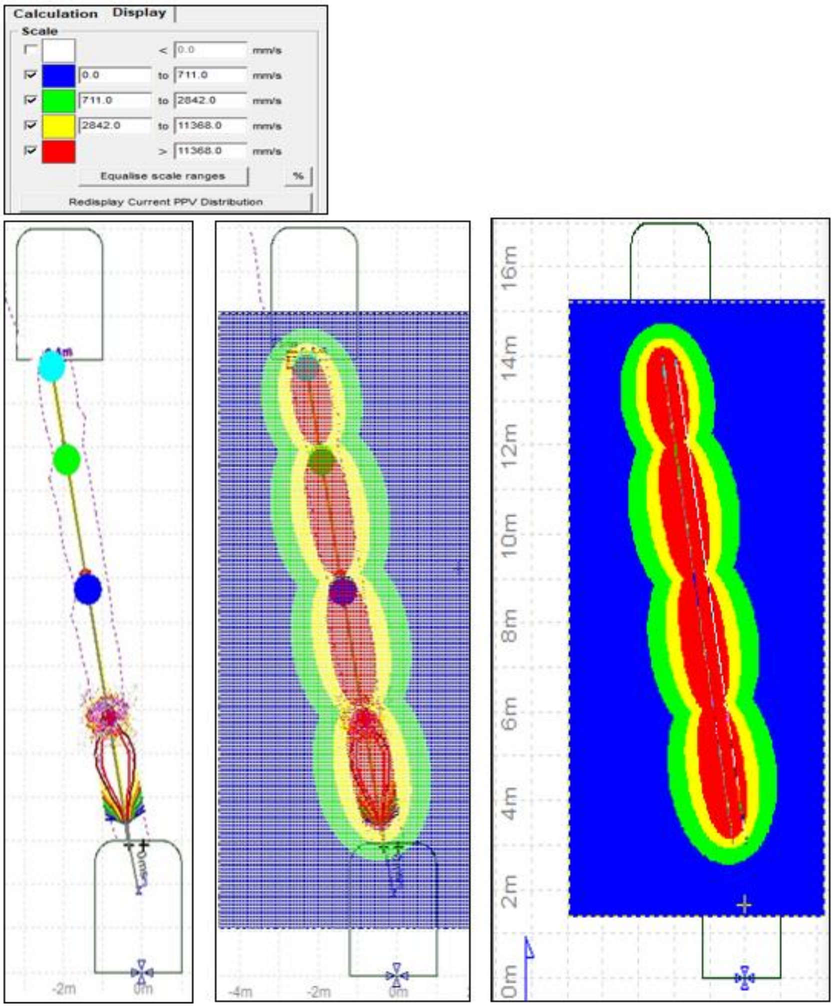

Figure 14. Simulation of the influence of explosive energy with JK Simblast

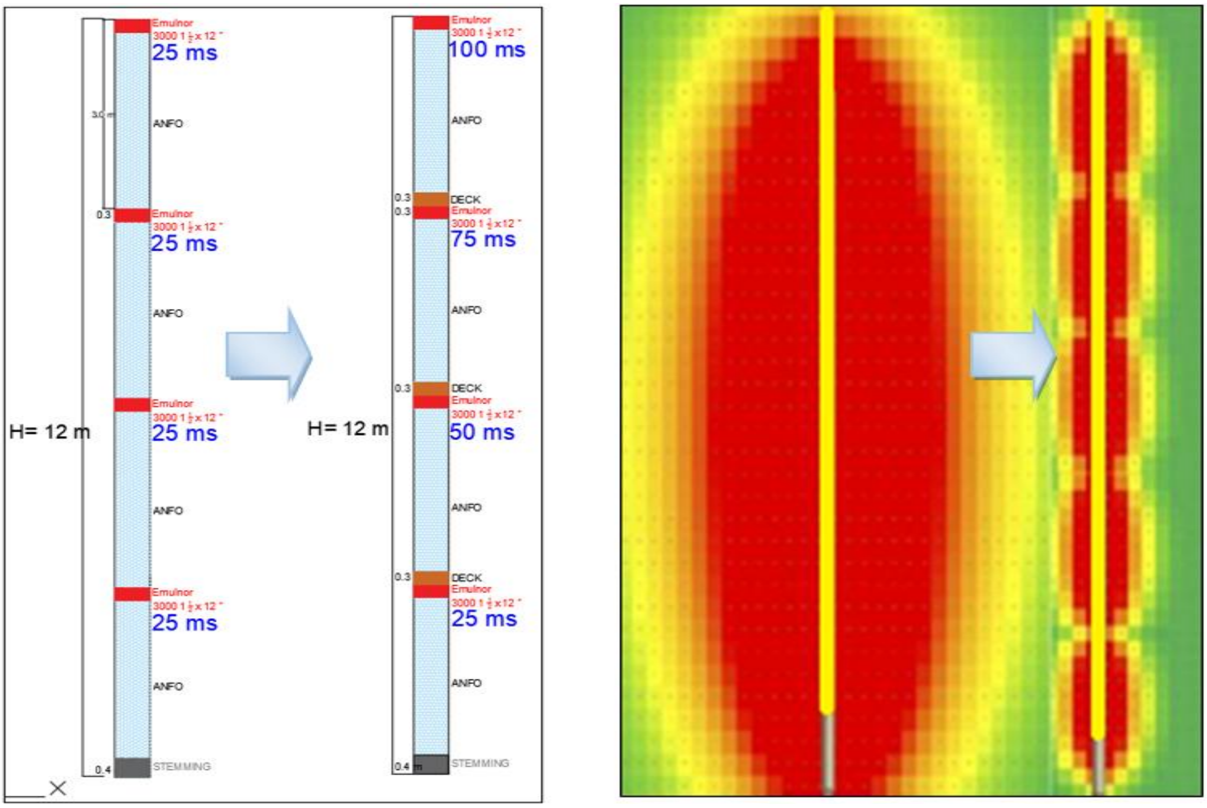

Figure 15. Load design comparison before and after using DECKs

Figure 16. Results before and after implementation of DECKs

- Design and loading parameters before and after using DECKs.

To clearly visualize the design and loading parameters of longhole stoping in narrow veins, Tables 8 and 9 show all the differences between traditional blasting and another using DECKs.

Table 8. Design parameters without DECKs

Design Parameters		
Equipment	MUKI LHBP 2R	Resemin
Mineralization	VETA	Qz - Au
RMR	56	Quality
Burden	1.2	meters
Rod Length	4	feet
Drill Diameter	2.5	inches
Cassing Diameter	2	inches
Tail	0.5	meters
Bench Height	12	meters
Width	0.9	meters
Length	1.2	meters
Ore Density	2.6	TM/m^3
Tons Removed	**37.07**	**TM/tal.**
ANFO Density	0.85	g/cc Confined.
Linear Charge Density	1.72	Kg/m
Primers	4	cart./tall.
Emulnor Density	0.39	Kg/cart.
Total, Explosive	**21.34**	**Kg/drill**
Power Factor	**0.6**	**Kg/TM**

Table 8 shows the design and loading parameters for a traditional blast, i.e., one without DECKs. This table shows a PF of 0.6 kg/ton, a very high value due to the high explosive consumption for an ore tonnage of 37 tons/ton. This indicates that the amount of explosive must be reduced, as this excess energy is generating instability, resulting in flaking and loss of ore recovery.

Table 9. Loading parameters without DECKs

Loading Parameters Without DECKS		
Number of drilled holes	5	Drills
Number of drilled holes loaded	5	Drills
SUPERFAM DOS	**98.9**	**Kg**
Emulnor 3000 1 1/2 × 12"	20	Units
Detonating cord	2	meters
Carmex	2	Units
Fast fuse	0.15	meters
Fanel MS	4	Units

Table 9 details the materials required to carry out the blasting drill. The high explosive consumption is notable, at 98.9 kilograms; this amount represents four bags of ANFO. Loading is carried out for all drilled holes; in this case, there are five holes, with four primers per hole. Similarly, there are four fanels, all of which have the same delay.

Table 10. Design Parameters using DECKs

Design Parameters - Anfo		
Equipment	MUKI LHBP 2R	Resemin
Mineralization	VETA	Qz - Au
RMR	56	Calidad
Burden	1.2	metros
Rod Length	4	pies
Drill Diameter	2.5	pulgadas
Cassing Diameter	2	pulgadas
Walk-off	0.4	metros
DECKs (0.3 m)	3	Unid
Bench Height	12	meters
Width	0.9	meters
Length	1.2	meters
Ore Density	2.6	TM/m^3
Tons Removed	**37.07**	**TM/tal.**
contact distance	0.5	m.
K_{prom}	357	
α_{prom}	-2.07	
V_{crit}	2842	mm/seg
$\boldsymbol{Q_{max'}}$	**2.53**	**Kg/retardo**
ANFO Density	0.85	g/cc Confin.
Linear Charge Density	1.2	Kg/m
Baits	4	cart. /tal.
Bait Length	0.3048	m
Emulnor Density	0.39	Kg/cart.
Total Explosive	**15.13**	**Kg/tal.**
Power Factor	**0.4**	**Kg/TM**

Table 10 outlines the design and loading parameters for a controlled blast using DECKs. This table shows a PF of 0.4 kg/ton, an adequate value with low explosive consumption for an ore tonnage of 37 tons/ton. This indicates that the PF is within the parameters. Explosive energy is controlled, minimizing vibration and consequently controlling rock mass stability. Additionally, accessory consumption and total explosive consumption for detonating four production holes are shown.

Table 11. Loading parameters with DECKs

Loading Parameters - Anfo		
Drilled holes	6	Drills
Loaded holes	4	Drills
SUPERFAM DOS	**55.2**	Kg
Emulnor 3000 1 1/2 × 12"	16	Units
Detonating cord	3	meters
Carmex	2	Units
Fast fuse	0.2	meters
Fanel MS	16	Units

Table 11 specifies the materials required to carry out the blasting drill. The explosive consumption is notable, at 55.2 kilograms; this amount represents a little more than two bags of ANFO. Loading is not required for all drilled holes; in this case, six holes will be drilled, but only four holes will be loaded, with four primers per hole and using the DECKs. Similarly, the number of fanels is four, all with the same delay.

3.4.4 Comparison of KPIs before and after using DECKs

When viewing Figure 17, the differences existing when using DECKs in blasting longhole stoping in narrow veins are shown. For a bench of 12 meters on average, with vein powers of 0.8 meters on average, with a burden of 1.2 meters; a high contrast can be seen when performing conventional blasting and controlled blasting with DECKs. The number of tons removed is similar in both cases since it is the same pit. This is 37.1 Metric Tons per hole (MT/Tal.), equivalent to one and a half of 23 MT dump trucks.

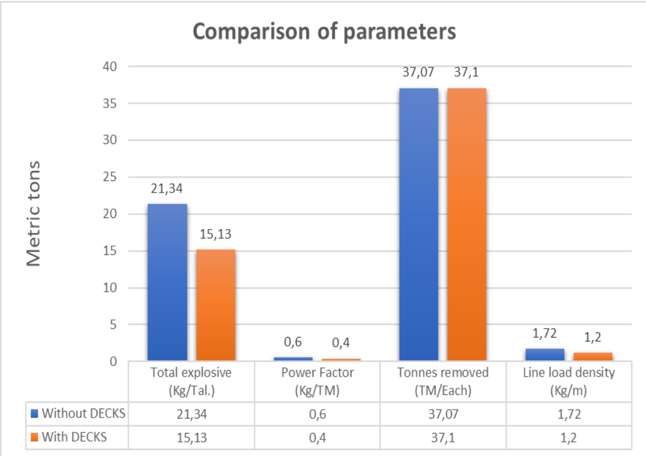

Figure 17. Comparison of KPIs before and after using DECKs

Figure 17 shows the KPI comparison before and after using DECKs. The linear charge density when blasting without DECKs is 1.72 kg/m², and the charge when using DECKs is 1.2 kg/m², presenting a difference of 0.52 kg/m². This excess energy is the cause of the instability of the rock mass in the pit. Explosive consumption when using DECKs is 15.3 kg/m², while when not using DECKs, it is 21.34 kg/m². There is a considerable difference in explosive consumption. Using DECKs minimizes explosive costs, which is beneficial for the company. Finally, the efficiency indicator for blasting without DECKs is high, with a PF of 0.62 kg/m², which is considered high for this method and outside the efficiency parameters. On the contrary, when using DECKs, a PF of 0.4 kg/TM is obtained, which is within the appropriate parameters, indicating good performance of the explosive.

3.4.5 Dilution comparison

Mineral dilution refers to the incorporation of unmineralized (or lower value) material into the mined ore, which can lower the average grade of the processed ore and affect the profitability of the mining project. Several strategies have been adopted to mitigate dilution. These include improving the precise design of ore block boundaries, ongoing personnel training to ensure that operators are well-trained, constantly monitoring, and most importantly, improvements in the drilling and blasting techniques that are the subject of this study. Several methods exist for numerically calculating dilution.

$$Dilution = Ton.Stripping/Ton.ore(*)$$
$$Dilution = T.Stripping/(T.Stripping + T.mineral)(**)$$

The standard of measurement used to measure dilution is the second equation, since it is more sensitive to the increase in stripping.

Figure 18 shows that, with the same pit parameters, the dilution percentage is very high compared to traditional blasting, at 57.14%. This indicates ore loss in the pit due to ore shedding. In contrast, when controlled blasting using DECKs technologies, the dilution is within the planned range of 5.26%, while the planned dilution is 12%. This clearly shows the difference between blasting without DECKs and blasting using DECKs.

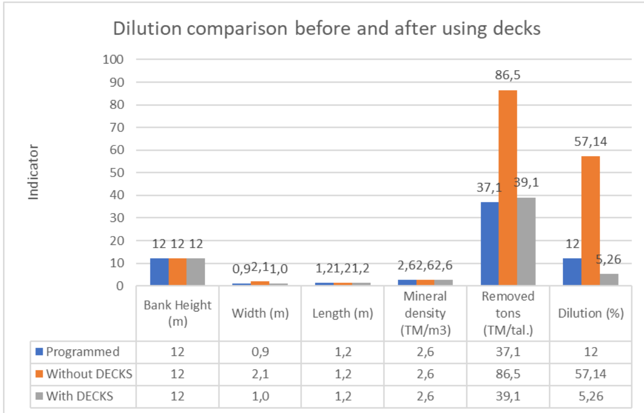

Figure 18. Comparison of dilution before and after using DECKs in blasting

3.4.6 Statistical validation of results

To assess the significance of the observed improvements in ore dilution and rock mass stability after implementing DECKs, statistical analyses were conducted. A two-sample t-test was applied to compare key performance indicators (KPIs), such as ore dilution (%) and PF (kg/TM), between blasts with and without DECKs.

The results show a statistically significant reduction in ore dilution (from 57.14% to 5.26%, $p < 0.01$), indicating that the improvements are not due to random variation. Similarly, the reduction in PF from 0.6 to 0.4 kg/TM yielded a significant difference ($p < 0.05$). These findings confirm that the implementation of DECKs has a measurable and statistically supported impact on reducing dilution and optimizing explosive efficiency.

All statistical tests were performed using standard significance thresholds ($p < 0.05$) and validated with the Shapiro-Wilk test to confirm normal distribution of the data.

4. CONCLUSIONS

- The use of DECKs has proven to be an effective solution for optimizing explosive charges in narrow vein blasting. Their implementation at the mining unit led to a substantial reduction in ore dilution—from 57.14% with conventional blasting to 5.26%—which translates into significant improvements in operational efficiency and ore recovery.
- The application of DECKs also resulted in a notable decrease in explosive consumption. The PF was reduced from 0.6 kg/MT to 0.4 kg/MT, reflecting greater energy efficiency and improved cost-effectiveness in the blasting process.
- The insertion of DECKs—segments of inert material between explosive charges—helps decouple the detonation energy, modifying its transmission and reducing stress wave concentration. This energy attenuation minimizes blast-induced fractures, preserves rock cohesion, and enhances overall stability. Field

observations and simulations confirm that DECKs reduce overbreak and slabbing while improving fragmentation, making underground operations in narrow veins safer and more efficient.

- These improvements also positively impacted environmental performance. By reducing energy release and minimizing unnecessary breakage, DECKs helped lower structural damage and decreased the volume of waste generated, aligning the operation with more sustainable mining practices.
- Beyond operational and economic improvements, the use of DECKs contributes to reducing the environmental footprint of underground mining. By lowering the amount of explosive required per blast, DECKs help reduce the generation of harmful gases and particulate matter, while also minimizing ground vibrations and acoustic disturbances. These effects not only improve working conditions and safety for personnel, but also lessen the environmental impact on surrounding geological structures and ecosystems, reinforcing the role of DECKs as a sustainable blasting innovation.
- Regarding long-term effects, the use of DECKs has not shown evidence of introducing additional geotechnical risks under the evaluated conditions. On the contrary, their ability to reduce overbreak and limit fracture propagation helps preserve the integrity of the rock mass over time. However, continuous monitoring in critical zones is recommended to assess post-blast structural behavior, as rock mass response may vary depending on lithology, degree of fracturing, and the support design implemented.
- The results obtained from tests and numerical simulations confirm that DECKs offer significant advantages in terms of efficiency and sustainability. However, their effective implementation requires accurate blast design and a detailed understanding of the geomechanical conditions of each site. Variability in rock types or structural settings may demand specific adjustments to the drilling and charging patterns.
- The contrast between conventional methods and DECK-based blasting highlights the value of adopting technological innovations in traditionally rigid mining processes. Enhancing blast control not only drives economic benefits but also reduces operational risks and improves mineral resource management—key factors for long-term profitability.
- Although DECKs have shown clear benefits in narrow vein mining, their scalability across different geological contexts requires careful assessment. In highly competent rock masses with low fracturing, the energy decoupling effect may be less relevant and could reduce fragmentation efficiency. Conversely, in highly fractured or poorly consolidated formations, their use might result in underbreak or incomplete fragmentation. Therefore, it is essential to thoroughly analyze the geomechanical conditions of each deposit prior to implementation, as performance depends on the interaction between detonation energy, rock structure, and confinement.
- Based on these findings, it is recommended that the DECK technique be extended to other underground operations in narrow veins, with the caveat that drilling and blasting parameters must be tailored to the specific geological and structural characteristics of each deposit to maximize performance.

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NOMENCLATURE

DECKs	Inert material between explosive charges
FP	Power factor
GSI	Visual geomechanical quality index
V_{cr}	Maximum particle speed without damage
K_b	Coefficient de burden
K_s	Spacing coefficient