Analysis for a Potential Enhancement of Blasting Performance for Ensuring Ground Stability in the Oil Shale Mine

by

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ORIGINALITY STATEMENT

I, Chingiz Abdykarimov, hereby declare that this submission is my own work and to the best of my knowledge it contains no materials previously published or written by another person, or substantial proportions of material which have been accepted for the award of any other degree or diploma at Nazarbayev University or any other educational institution, except where due acknowledgement is made in the thesis.

Any contribution made to the research by others, with whom I have worked at NU or elsewhere is explicitly acknowledged in the thesis.

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ABSTRACT

This thesis investigates how simulation-based blast design enhancements can improve blasting performance. The work builds on the field findings reported by Sabanov et al. (2023). The author created a complete method that mixes advanced models with an uncertainty analysis.

The study uses JKSimBlast software to simulate three dimensional blasts, it includes mapping how explosive energy spreads, which relies on Kleine field theory. Monte Carlo simulations with 10,000 trials determine how much uncertainty there is in when a blast starts, how strong the rock is, and how much charge is present. This process predicts blast outcomes and includes the probability of those outcomes. A regression study of sensitivities, shown with tornado charts, points out the most important factors that control how well a blast works - these tools help design the best blast pattern for the mine's complex layers of rock. The pattern expands on previous designs based on experience to handle harder situations.

The study's results show the new contributions of this simulation work. The radius of blast damage, which is how much the rock breaks, acts as a random factor - this radius follows a log normal spread, it averages about 0.29 m, which means that cracks do not spread in a fixed, set way, but they vary. The amount of explosive needed per volume of rock changes based on the orientation of the blast hole. Holes in the roof need about 2.02 g/dm^3. Floor holes use about 2.12 g/dm^3, and wall holes use about 2.49 g/dm^3 on average. These differences show how geometric confinement and available free surfaces affect how well blasting works. The sensitivity study shows that the rock's uniaxial compressive strength mainly controls both the blast damage radius also the amount of explosive needed. Borehole diameter comes next. The properties of the explosive, such as its density and velocity of detonation, and factors based on experience, only have small effects. By measuring how much each factor influences the outcome, the study tells practitioners where to focus their efforts to control measures.

In conclusion, this research extends the Sabanov's field results by offering a simulation-based pre-validation framework for blast design. The study shows that a well-planned charge distribution, checked through simulations, can break rock evenly as well as keep the ground stable in different conditions. Engineers can fine tune blast patterns with confidence limits and safety factors. That makes sure that even the worst-case situations stay within safe design limits, this forward-looking, model-based approach improves mine design. It makes blasting operations safer and more effective before full-scale implementation.

ACKNOWLEDGMENT

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1. INTRODUCTION

Blasting is an essential and technically demanding component of underground mining operations, significantly influencing productivity, economic viability, and workplace safety (Persson et al., 1993). The primary purpose of underground blasting is to achieve efficient rock fragmentation and excavation progress while maintaining the structural integrity of essential mine elements such as pillars and roofs (Hustrulid, 1999). The advancements in blasting technology and practices has not yet solved all the existing problems, especially regarding optimal blast pattern design and the effective use of explosives.

The success of blasting depends heavily on blast pattern design through parameters like hole spacing, burden, charge distribution, and initiation timing and sequencing. The incorrect planning of blast patterns results in poor fragmentation, excessive ground vibrations, and unplanned damage to adjacent structures as well as elevated operational costs (Iverson et al., 2013). The undesirable results decrease mining efficiency and create safety risks due to the increased chances of roof collapses and pillar failures. Underground mine blast pattern optimization becomes complicated as a result of geological variations which include uneven rock characteristics, natural fractures, stress differences, and water content (Bieniawski, 1989; Whittaker et al., 1992). Blast pattern design requires site-specific modifications which are controlled by detailed geological assessment and advanced monitoring techniques to achieve reliable outcomes.

The selection of explosives imposes significant complexities for blasting operations. The adoption of emulsion explosives became widespread in underground mining due to their handling properties together with high detonation speeds and water resistance making them safer than traditional explosives (Maranda et al., 2025; Sabanov et al., 2023). Emulsion explosives have gained popularity in difficult underground settings because of their flexibility across different geological and operational conditions. The utilization of emulsion explosives in practice faces significant difficulties despite their multiple benefits. Changes in explosive density and improper handling practices along with inconsistent charging techniques lead to incomplete blasts with reduced efficiency and unpredictable fragmentation results (Mertuszka et al., 2022). These irregularities result in elevated dilution levels which increases operational expenses while decreasing excavation productivity and, most importantly, they weaken roof and support pillar stability.

Additionally, underground blasting operations face severe challenges due to the critical complexity of explosive energy distribution. The division of explosive force between useful rock fragmentation and unwanted effects like excessive vibration or air blast impacts mining operations efficiency (Calnan, 2015). Optimal energy distribution requires special attention to the selection of explosives and their charge configuration as well as blast pattern design and rock confinement conditions according to Esen (2006). When these factors are not properly managed, it results in ineffective blasting procedures alongside environmental issues and considerable damage to essential underground support structures.

Mining professionals address these problems by using sophisticated predictive modeling tools and technologies to achieve better blasting results. Mining engineers can optimize blast designs and choose the best explosives through techniques that predict rock fragmentation and blast damage while cutting down operational uncertainties. Advancements in numerical modeling and simulation methods together with progress in real-time monitoring systems like Measurement While Drilling (MWD) create new possibilities to increase blasting performance while solving typical issues related to underground blasting operations (Eldert et al., 2020).

This study examines critical blasting pattern optimization issues and emulsion explosives applications within the context of their essential role in successful underground mining operations. The research employs a thorough examination of existing blasting techniques along with case study observations and advanced modeling methodologies to develop a detailed understanding and practical guidelines that enhance blasting efficiency while optimizing fragmentation and maintaining roof stability in complex underground mining operations.

The application of emulsion explosives in blasting operations sometimes results in poor fragmentation together with substantial adjacent rock structure damage and roof support destabilization. Emulsion explosives provide enhanced safety yet exhibit wide-ranging performance outcomes. Variability arises from changes in explosive density together with detonation velocity inconsistencies and storage condition effects. The performance inconsistencies in blasting operations using emulsion explosives result in energy usage that can be either excessive or deficient which greatly impacts excavation productivity.

The research analyzes how explosive factors like detonation velocity and density affect both blast damage radius and the distribution of specific charges. The research seeks to establish and confirm an improved blasting pattern through systematic analysis of essential parameters

using controlled field tests and modeling techniques. The objective of this approach is to enhance blasting outcomes while ensuring stability for mine roof structures and support pillars. The results provide actionable instructions for properly using emulsion explosives in underground oil shale mining blasts.

1.1 Problem Definition

Even though technology for blasting and explosives advanced, especially with emulsion explosives, underground mines still deal with problems in getting good blast results - these problems come from the many connected factors that affect how people plan and do blasts in small, underground areas.

A main problem involves designing blast patterns correctly. A good blast plan needs a careful look at things like hole distance, burden, borehole size, where the charge sits, plus when it starts. Poor design often causes the rock to break into wrong sizes, the face to move unevenly, too much shaking along with accidental damage to important mine parts like the roof and side walls. These flaws not only lower how well the mine works and raise expenses, but they also lessen stability underground also put miners in danger.

Another big problem that no one has fixed is using explosive energy poorly. In underground blasting, the goal is to get as much energy as possible to break rock, while keeping bad effects like overbreak, air blasts as well as shaking low. But getting this balance stays hard. Many things affect it, such as how the explosive acts, the blast's shape, differences in the ground, and how much it is closed in. Without strong ways to predict and standard rules for design, the problem gets worse. This makes it hard to always get dependable as well as controlled blast results.

The current theoretical models, like the well-used Kuz-Ram model, only offer broad predictions, they often do not show how complex underground places are. In these places, conditions such as rock with many cracks, different strengths, water coming in in addition to uneven pressure fields often occur; these limits show that today's methods do not help with exact blasting in underground areas.

This project will work on these important missing parts - it will improve how people understand and model damage from blasts in underground mines. The project will focus on making the use of emulsion explosives better. It will also improve how exact blast designs are. To do this, the project will combine advanced simulations, looking at data from tests, and checking things in the field. The main purpose is to make models that are more dependable plus fit the situation. These models will allow for better blast performance. They will also lessen structural damage and make underground operations safer.

1.2 Objectives of the Thesis

1.2.1 Main Objectives

The primary objective of this study is to produce an analysis for improving blasting efficiency and ensure the stability of the immediate roof and pillars in the underground mine by optimizing the blasting pattern.

1.2.2 Specific Objectives

- Test work data analysis
- Blasting pattern modelling under different conditions for improving roof, floor and wall stability
- Simulation of detonation, distribution of explosive energy, near field Peak Particle Velocity prediction
- Evaluation of explosive density, velocity of detonation, Uniaxial Compressive Strength, borehole radius, the empirical field factor impact on Blasting Damage Radius

1.3 Hypotheses

Optimized charge redistribution in blasting patterns through careful control of specific charge, PPV and blast damage radius can significantly enhance blasting efficiency. This improvement will manifest as advanced rock fragmentation, consistent entry advance, and reduced damage to the roof and walls.

1.4 Justification of the R&D

The study in this paper starts with the need to make underground oil shale mining safer and to get more done. This also helps education sector, industry sector and the public.

From an industry view, the work deals with serious dangers from bad blasting - this bad work can cause the roof to fall, pillars to break as well as the mine structure to suffer harm. Such dangers put mine workers at risk plus stop output when not planned. The work tries to lower structural weakness and raise the safety also steady nature of underground work by finding the best blast forms and by managing how far blast harm goes. At the same time, better management of blast harm can break rock better, which means less need for a second blast, as well as it lowers costs from using too much explosive and from fixing things after the blast. That makes mining cheaper, makes more product in addition to stays good for a long time.

As for the natural world, the work helps lessen the harm underground blasting does to the earth. Better management of energy spread during a blast can lessen waste rock, dust in the air next to harmful gas - these things cause big worries in closed underground places; this agrees with wider goals for lasting good plus helps industry meet rules about the environment and what the public wants.

For educational purposes, this work fills a gap in what people know now - it makes clear how blast harm happens in difficult rock forms, especially in oil shale. It builds on also tests what old models can do, and it offers a more detailed way. This way brings together computer tools, checks of small changes, as well as data from the field. What comes from this work can be a base for more studies in how rock moves, how explosives are made, and how underground mining systems work - it also helps in making better books plus lessons for people who work in mining.

Finally, this work does not just move science and industry work ahead. It also helps public safety by giving safer work conditions also by getting natural resources out with more care for the earth.

1.5 Scope of Work

The following steps were done during the research:

1. Conducting the literature review on the thesis topic,

2. Analysis of dataset that was collected on mine site,
3. Produce an optimized model in JKSimBlast software,
4. Run several detonation and blasting simulations,
5. Analyze outcomes with the help of Monte Carlo simulations using @Risk software,
6. Acquiring expected outcomes and compare them.

2. LITERATURE REVIEW

2.1. Blasting in underground mines

Recognized as one of the most critical operations in underground mining, blasting has a direct impact on entry advance rates, rock fragmentation, roof and pillar stability, and worker safety that are all mining key performance indicators. Efficient blasting practices enhance overall productivity in mining operations. For instance, according to Hustrulid and Bullock (2001), optimized blasting procedures contribute markedly to improved advance rates, reducing cycle times and operational costs in underground development projects. Appropriately executed blast designs not only increase excavation rates but also significantly affect subsequent loading, hauling, and processing operations through improved fragmentation quality.

The quality of rock fragmentation directly determines the efficiency of downstream processes in mining. Poor fragmentation, characterized by overly coarse or excessively fine particle sizes, can result in increased operational expenses, delays, and higher mechanical stress on excavation machinery. The research conducted by Dejan and Goran (2018) highlights the need to choose the best drilling and blasting design to obtain a suitable fragment size distribution, while also maintaining the stability of underground galleries, which is a key issue in underground copper mines.

Another important factor affected by blasting operations is the stability of pillars and the immediate roof, which is essential for safe mining conditions. Konicek et al. (2013) highlight the significance of controlled blasting methods for maintaining pillar integrity. They argue that uncontrolled blast-induced vibrations and damage can greatly weaken pillar structures, which may lead to serious failures. The study by Duzgun and Einstein (2004) shows that measures like effective blast energy control and roof support planning can significantly reduce the probability of roof falls and minimize the related consequences.

The blast operations also affect the workers safety in underground mine conditions. Dick et al. (1983) underscore the importance of safe blasting practices in underground mines, noting that controlling factors such as vibration, overbreak is critical to reducing accidents. Investigations by Ali et al. (2024) on the harmful vapors produced by ANFO explosions in confined underground conditions, highlight not only the detonation performance but also the risk it poses to safety of workers

2.2. Factors Influencing Blasting Efficiency in Underground Mining

Blasting in underground mines works well or poorly because of certain operational settings that people control and geological features that they do not control. How well rock breaks, if the fragments are the same size, and if the blast causes little damage depend on how clearly people know these things plus put them into the blast plan. An overall look at both kinds of features helps people make good and flexible blasting plans.

2.2.1. Controllable parameters

Controllable settings mean the design factors that engineers can change to get better blasting results, which include the distance between holes, the diameter of the blast holes, how the charge spreads, the kind of explosive, how fast the explosion moves, and the order of ignition.

Among these, the speed of the explosion has received much thought because it connects strongly to how well rock breaks also how well the blast energy works. Chiappetta (1998) said that faster explosions usually mean smaller rock pieces, better energy movement, and fewer bad effects like cracks behind the blast, extra rock breaking along with weak roof conditions. Then again, slow moving explosives often cause larger rock pieces, which means more blasting or machine work later. This adds to the cost of operations and the time needed to finish a cycle.

Further study by Mertuszka et al. (2018) showed that the explosion's speed is not a fixed trait. The charge diameter changes it a lot. Their findings showed that even with the same liquid explosive, different hole sizes could greatly change the explosion's speed and, as a result, the blast's quality - this shows the need to control the shape of the blast hole exactly in underground places where drilling too much or too little happens often.

The charge's thickness plays a main part in how energy spreads. Kulula et al. (2017) stated that denser rock tends to take in more explosive energy before it breaks. That can lower how well the blast works. Changing the charge's thickness or using more powerful explosives in such places may make up for lost energy. People must balance this against the danger of extra breaking as well as safety worries.

Dotto and Pourrahimian (2024) helped people understand this more by showing that not only the explosion's speed but also the explosion's pressure changes the size and reach of broken areas in hard rock. Their study proved that higher pressure causes more even breaking and

deeper energy spread, which makes choosing both the explosive type plus how it loads more important.

2.2.2. Uncontrollable parameters

Uncontrollable settings mean the specific geology and rock conditions of a place. People cannot change these, but they must consider them in the blast plan, which include the rock's strength, breaks in the rock's structure (like cracks, layers as well as faults), the stress in the rock, the amount of water in addition to if the rock's properties differ in different directions.

The rock's features greatly affect how energy moves and how breaks form during blasting. Breaks such as cracks, splits next to layers can bounce, bend, or take in stress waves; this changes the way energy spreads also how well it works. This often leads to breaking patterns that cannot be guessed and areas of uneven damage. For example, in rock with many cracks, explosive energy may scatter along natural weak spots instead of making new breaks, which leads to poor breaking and uneven relief of the burden.

While general models like the Kuz-Ram formula are widely used, they often assume the rock is the same throughout; they cannot fully show how complex natural geology varies - this problem is especially important in underground mining, where the effects of being enclosed as well as complex stress make the influence of geological features worse.

A very important part of making blasting work better involves describing these uncontrollable settings better and putting them into blast planning. That may include adjusting models for a specific place, making detailed geological maps, and using prediction tools that consider the rock's directional differences plus how energy gets lost.

2.3. Characteristics and Performance of Explosives in Underground Mining

As was discussed before, the most important to explosive performance is velocity of detonation (VOD), which widely affects the quality of fragmentation and the subsequent efficiency of blasting activities. Explosives with higher VOD typically produce finer fragmentation, reducing the need for additional mechanical breakage or secondary blasting (Dick et al., 1983; Persson et al., 1993).

The comparisons of ANFO and emulsion explosives in underground environment can show the differences in blasting outcomes. The study by Widodo et al. (2019) notes that explosives based on emulsions that are used in tunnel blasting may provide more effective overbreak and underbreak control than ANFO explosives. While Heryadi et al. (2024) report that, if blasting patterns are appropriately designed, then bulk emulsion explosives can achieve fragmentation comparable to ANFO at similar overall expenditures. Though the powder factor and fragmentation size distribution differed, emulsions explosives produced better face advance in several narrow-reef gold mines according to Chosi et al. (2024). Finally, the choice between ANFO and emulsion explosive rely on factors such as detonation velocity, detonation pressure, and the volume of gases released (Vieira & Braga, 2023).

Emulsion explosives are also more preferred for the safety reasons. According to new study by Maranda et al. (2025) the water-in-oil matrix of emulsion explosives shows rather low sensitivity to shock, friction, and impact, which greatly lowers handling and transport hazards when compared to traditional cartridge-based products. This reduced hazard level leads to less explosive-related incidents and enhances the overall safety of mining operations

2.4. Fragmentation Prediction and Modeling in Underground Mining

Among all the available predictive methods, the Kuz-Ram model remains widely known and used because of its simplicity and practical applicability. Originally introduced by Cunningham (1983), the Kuz-Ram approach integrates Kuznetsov's empirical formula for mean fragment size with the Rosin-Rammler particle size distribution function. Using parameters such as explosive energy, rock factor, blast geometry (burden, spacing, and hole diameter), and explosive type, the model provides a convenient way to estimate fragmentation across diverse blasting scenarios (Cunningham, 1983). This model is based on simplifying assumptions and does not fully account for complex underground conditions, but it remains a common reference for initial blast design and fragmentation analysis.

The Kuznetsov equation for estimating the mean fragment size is:

$$\bar{x} = A * \left(\frac{V}{Q}\right)^{0.8} * \left(\frac{115}{\sqrt{RWS}}\right) \quad (1)$$

Where: $\bar{x}$ = Mean fragment size (cm); A = Rock factor (depends on rock mass properties); V = Volume of the blasted rock (m^3); Q = Mass of explosive per blast (kg); RWS = Relative Weight Strength of the explosive (unitless).

The Rosin–Rammler distribution is then used to describe the cumulative size distribution of the fragmented rock:

$$P(x) = 1 - e^{\left(\frac{x}{\bar{x}}\right)^n} \quad (2)$$

Where: P(x) = Percentage of material passing through a sieve of size; x = Fragment size (cm); $\bar{x}$ = Mean fragment size (cm), obtained from the Kuznetsov equation; n = Uniformity index, typically ranging between 0.8–1.5 for blasting (higher values indicate more uniform fragmentation).

The calculation procedure of Kuz-Ram model is following:

- Determine the blasting geometry and the charge parameters. Figure out the volume of the rock that will blast. It can be found by multiplying the burden by the spacing and the bench height or the round length. Find the total mass of the explosive.
- Assign a rock factor A. This factor comes from empirical values for different rock types. For example, hard rock has a value between 8 and 20, which depends on its strength, jointing along with brittleness.
- Calculate the mean fragment size x with the Kuznetsov equation.
- After that, pick a suitable uniformity index n. This index relies on blast design experience or on calibration with field data.
- Use the Rosin - Rammler function to estimate the percentage that passes at different fragment sizes - this function helps create a fragmentation curve, and it also helps you evaluate how much of the blasted rock fits inside the desired size range. That allows for easier handling, loading, or crushing.

Despite its widespread use, the Kuz-Ram model incorporates idealized assumptions that can reduce accuracy when rock conditions are more varied or complex (Balakrishnan et al., 2020).

Some issues such as complex joints, inconsistent rock mass strength, and field-specific variations in explosive behavior may depart from the original assumptions, creating gaps between the model's predictions and real fragmentation outcomes. However, reliable fragmentation forecasts are vital due to their improved breakage that makes a rise in loading and hauling efficiency while also reduces the costs related to secondary blasting and crushing. Excessively optimistic or pessimistic fragmentation estimates can lead to operational and financial loses (Ouchterlony, 2005).

Recent studies tried to improve fragmentation prediction by putting field data into models. For example, Dotto and others (2022) built a fragmentation model that uses measured detonation velocities to predict fragmentation - it changes the prediction for local rock conditions plus how the explosive acts. Their method showed that the predicted fragment sizes matched the actual sizes better in rock that had cracks and was tightly held. Their study changed a Rosin - Rammler function. The uniformity index also the average fragment size in this function changed with detonation velocity and rock vibration levels. This change showed how energy moved around in the rock better.

Researchers also look at more complex numerical as well as hybrid models. Balakrishnan and others (2020) compared how well standard models worked against numerical simulations; they used ANSYS Autodyn in addition to LS-DYNA, which are computer programs that copy detonation waves, material breaks along with rock fragmentation. Their results showed that numerical models could capture how stress waves worked with rock breaks, blast hole spacing as well as how the rock held together better than standard models. They also looked at ways to combine standard plus numerical methods. One example is connecting the Kuz-Ram model with tools like JKSimBlast or FRAGLYTICS. This allows researchers to adjust standard estimates with actual fragmentation data and stress spread simulations.

Some researchers also used Discrete Element Method (DEM) also Finite Difference Method (FDM) programs, such as UDEC besides FLAC3D. They used these to look at how blast creates cracks in rock with different layers - these tools can model crack networks, rock piece movement in addition to how energy moves. The Kuz-Ram or other standard models do not consider these factors.

These advanced tools help with accuracy plus how well they show reality. Using them often needs good input data. That includes full rock characterization, stress fields next to detonation

properties - they also need a lot of computer power. Understanding the results needs special knowledge, this limits how often people use them in many working underground mines.

2.5. Optimization of Blasting Patterns with prediction models

To plan underground blast designs, a person needs to grasp what kind of work is happening - development or production blasting. Each kind has different goals and uses different shapes and rules for blasting.

Development blasting cuts tunnels, drifts along with raises - it uses small blasts to move the rock face forward while keeping the ground steady. The holes for development blasting are set up in specific ways plus include cut holes, stoping holes, contour holes as well as lifter holes. Each type of hole has a distinct use to start and control the blast. Figure 2.1 shows a common development round layout. The cut holes make the first empty space for the rock to expand. Stoping holes break the main part of the rock, contour holes protect the edges, and lifter holes break the floor properly.

Production blasting, on the other hand, digs out ore. It uses larger, repeated blast rounds that break the ore well for loading also moving. Production blasts often use deeper holes with more explosives; they aim to break the ore into small pieces in certain areas. The distance between holes, the burden, the type of explosive, and the timing all play a part in controlling how the rock breaks, how much waste mixes in, and how stable the walls remain. Figure 2.2. shows coal production free surface blast layout.

Knowing the ground beneath the surface is important in both situations. Geophysical methods, such as electrical resistivity imaging, seismic tomography in addition to ground penetrating radar, help map rock features like faults, weak areas, or layers - these features directly affect where holes go, the burden next to how much explosive to use (Singh et al., 2015). When engineers understand this variability, they can adjust the design to keep peak particle velocity (PPV) and the frequency of blast induced waves within acceptable limits. This helps reduce breaking too much rock and avoids harming nearby structures.

The pattern of the blast directly changes how well blasting works, including how evenly the rock breaks, how much rock moves forward per round, plus how much vibration occurs. For example, V-cut and burn cut shapes in development headings allow controlled breaking. They provide relief and guide where the energy goes. Chai et al. (2019) saw that better cutting hole

patterns made the rock break more evenly also lessened too much ground vibration during deep drift development in hard rock mines. Yılmaz (2023) showed that placing holes and timing them well in burn cut layouts improved the stability of the rock face and meant less scaling was needed.

New advances have brought in AI-based models to improve blast designs. These models look at past blasting data as well as learn patterns that lead to better fragmentation. People have used Artificial Neural Networks (ANNs) and Adaptive Neuro-Fuzzy Inference Systems (ANFIS) to connect input numbers, like borehole diameter, burden, spacing, VOD, charge type along with delay timing, with output numbers, like mean fragment size, PPV, airblast as well as backbreak (Zou et al., 2021). These models learn from many sets of data from prior blasts - they can offer real time suggestions to change explosive loads and timing plans to fit different rock conditions.

For example, in a copper mine study by Zou et al. (2021), an ANN model that learned from over 200 blast rounds could predict how rock would break with an error of less than 10 %. This helped engineers reduce large pieces of rock by 15 % plus lessened the need for more blasting. Another study by Urošević et al. (2021) combined Multi-Criteria Decision Analysis (MCDA) with fuzzy logic to improve production blast patterns. They balanced how well the rock broke, the cost in addition to the safety of the structure - this system let people decide between competing goals, such as having less waste while getting more work done, based on specific rules of the operation.

Even though they show promise, AI-based systems do not work completely on their own; they need good data, proper setup next to human understanding. They offer a powerful tool to help make decisions. That can make blasting results more consistent, especially in underground places where uncertain ground conditions and limited access make old design methods harder.

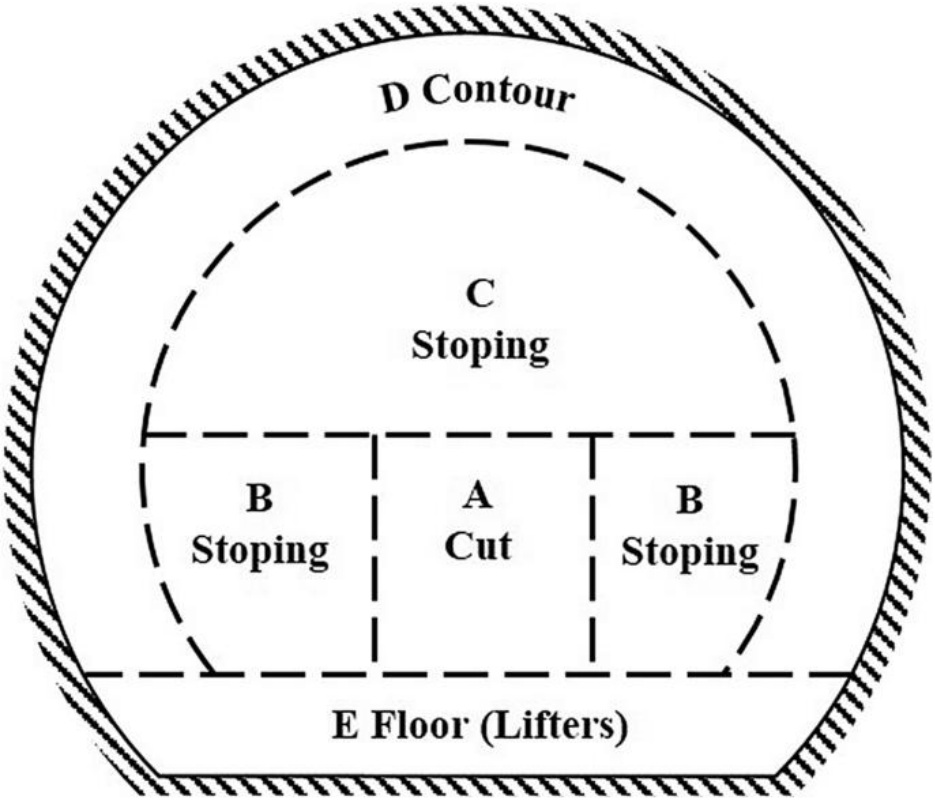

Figure 1. Typical development blasting layout

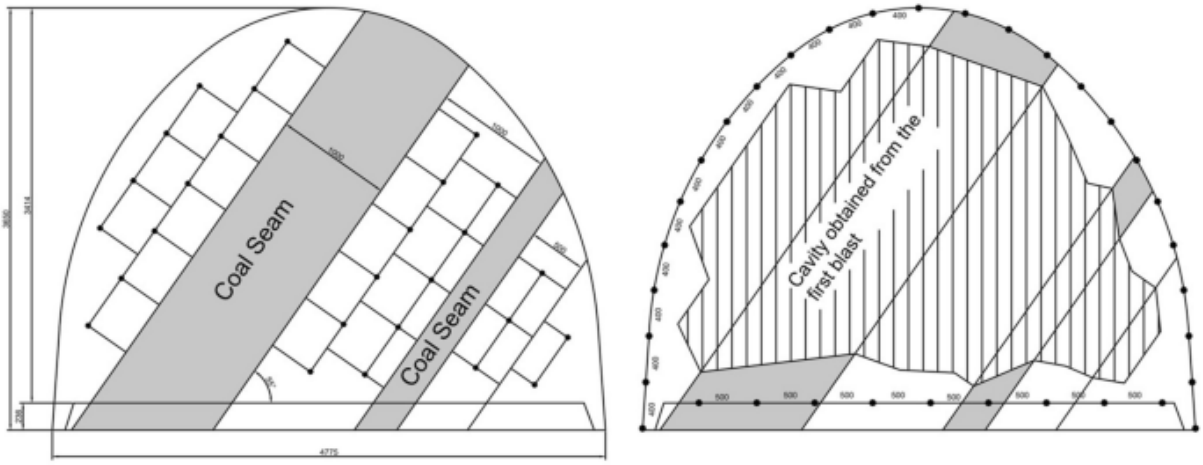

Figure 2. Production blasting layout

2.6. Impact of Geological Conditions on Blasting efficiency

The results of blasting are significantly influenced by the existence of faults, bedding planes, and other discontinuities in the rock mass. The structural features control the propagation of detonation waves, the fragmentation of rock, and the distribution of explosive energy (Persson et al., 1993). Rock masses with many discontinuities or weak planes can be broken apart

unevenly, resulting in a greater number of fines or larger blocks (Ouchterlony, 2005). This irregular fragmentation raises handling costs and may require more rounds of blasting or mechanical crushing.

Yahyaoui et al. (2018) explained the connection between the orientations of discontinuities and blasting results, showing that joint distributions can cause uneven energy propagation and different fragmentation outcomes. Wang et al. (2023) examined how differences in joint angles between blast holes influence crack propagation and stress changes in rock masses. Their findings highlight that even minor geological variations can significantly affect blast-induced damage. high levels of vibrations.

In a layered sedimentary stratum, blasts often take advantage of bedding planes, which can lead to overbreak or produce an abundance of small fragments (Sanchidrián et al., 2008). Modifying blasthole layouts, burdens, and delay timings to fit these structural patterns can enhance fragmentation uniformity and decrease issues like bridging in crushers (Ouchterlony, 2005).

2.7. Blast-Induced Vibrations in Underground Mining

Typically measured via peak particle velocity (PPV), these seismic waves can pose risks to pillars, roofs, and support systems. Deb and Jha (2010) have emphasized the critical role of PPV measurements in anticipating damage to underground openings caused by nearby surface blasting, noting that phenomena such as pillar spalling, roof collapse, and junction failure can result from excessive vibrations. Their empirical findings propose that PPV levels of around 100 mm/s generally do not cause significant harm in underground coal mine workings, providing an initial guideline for safe blasting practices.

The study conducted by Ramulu and Sitharam (2010) demonstrated that fracture initiation and extension triggered by in-situ stress redistribution and dynamic loading can weaken underground support structures. Blast damage not only forms new fractures but also propagates existing discontinuities, as shown in recent damage criteria research (Jha, 2020).

Computational modeling methods, such as artificial intelligence techniques, are being more frequently included in predictive frameworks, enhancing the accuracy of forecasts related to underground vibration effects (Tao et al., 2024). These models consider the size of the blast

and the shape of the hole, while also including the damping effects that are naturally present in underground rock masses.

Vibrations caused by blasts may also indirectly affect production efficiency. Mohseni et al. (2019) discovered that unplanned dilution may happen when fractures reopen, which can destabilize important areas of underground metal mines. In a similar study, Singh et al. (2015) demonstrated that blasts from opencast mines can negatively impact nearby underground structures. This highlights the importance of coordinated blasting strategies to manage both surface and subsurface effects

2.8. Rock mass classification and pillar stability in underground mining

Among the most frequently used systems are Bieniawski's (1989) Rock Mass Rating (RMR) and Barton et al.'s (1974) Q-system. Bieniawski's RMR employs variables such as rock strength, joint spacing, joint condition, and groundwater conditions, providing straightforward guidelines for pillar design and excavation support. Research by Hoek and Brown (1997) highlights the importance of including rock mass classification in the engineering process, emphasizing its ability to accurately gauge pillar stability, particularly after blasts alter the rock mass integrity.

Barton's Q-system is effective for evaluating pillar performance by including factors such as joint roughness, alteration, and stress reduction. Palmstrom (2005) indicates that these classification tools help engineers better predict rock mass behavior and adjust blasting parameters to reduce damage to important structures like pillars, thereby ensuring overall stability.

Effectively managing pillar stability requires combining blasting techniques with strong evaluations of rock mechanics and environmental analyses. Kaiser et al. (2000) emphasize the need to integrate rock mass characterization, numerical modeling, and empirical evaluations to tackle the complex factors that affect pillar design and blasting optimization. This combined perspective ensures that blasting practices consider geotechnical and environmental effects, promoting stability and sustainability in underground mining environments.

Potvin and Hadjigeorgiou (2008) show that interdisciplinary coordination improves the reliability of assessments of post-blast pillar conditions. This approach to decision-making based on data results in improved choices about pillar dimensions, blasting strategies, and excavation scheduling. This collaborative approach enhances safety results and minimizes environmental impacts, highlighting the importance of cooperation among geological, geotechnical, and environmental fields.

3. MATERIAL AND METHODOLOGY

3.1. Case study area and geological conditions

The study was conducted at Estonian oil shale mine, that utilize room-and-pillar mining method in complex geological setting. Oil shale is a layered sedimentary rock that contains a significant amount of organic material, consisting of 15–46% kerogen, 26–57% carbonates, and 18–42% clastic materials. The stratigraphy of oil shale bed consists of eight layers identified from bottom to top as A, A1, B, C, D, E, F1, F2 and F3. There are also seven interbedded limestone layers labeled as A/A1, A1/B, B/C, C/D, D/E, F1/F2 and F2/F3. The dimensions of the working face are approximately 7 meters in width and 3.8 meters in height. Table 1 presents the stratigraphic column indexes for oil shale and limestone, along with the compressive strengths and density of the layers.

The mechanical properties of oil shale and limestone layers differ significantly. The compressive strength of oil shale ranges from 18 to 40 MPa, while limestone has a compressive strength of 65 to 82 MPa. The volume density of the rocks ranges from 1200 to 1700 kg/m^3 and from 2100 to 2500 kg/m^3, respectively, while their strength increases towards the south. The layers are shown in Figure 3.

Table 1. Oil shale and limestone data (Sabanov et al., 2023)

Layer index	Thickness, m	UCS, MPa	Density, kg/m^3
F3	0.37	25	1.73
F2/F3	0.11	67	2.12
F2	0.17	24	1.72
F1/F2	0.18	65	2.10
F1	0.42	24	1.51
E	0.58	18	1.28
D/E	0.07	67	2.10
D	0.06	29	1.59
C/D	0.29	82	2.45

C	0.41	26	1.38
B/C	0.12	75	2.10
B	0.38	40	1.22
A1/B	0.18	65	2.25
A1	0.09	26	1.42
A/A1	0.06	32	2.10
A	0.12	32	1..37

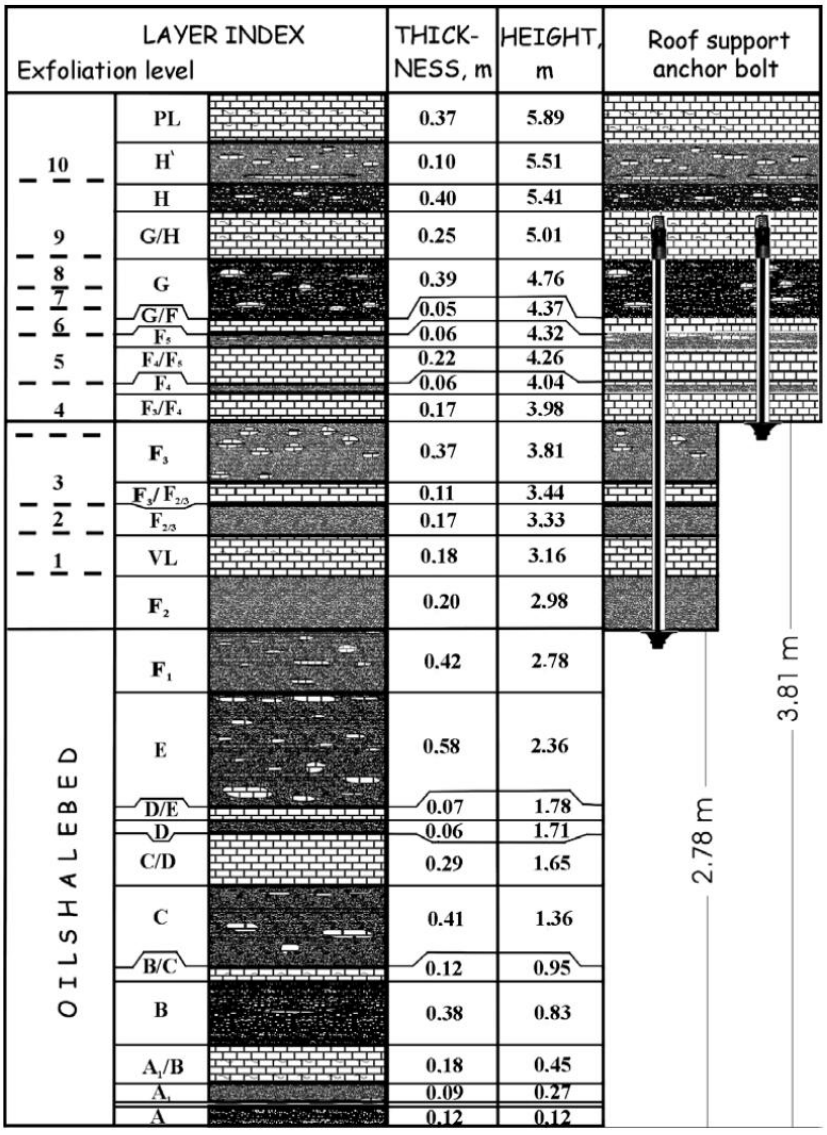

Figure 3. Structural cross section with determined IR exfoliation level (Sabanov et al., 2023)

3.2. Explosives and blasting system

The mine employs an underground bulk emulsion explosives system, where waterproof emulsion explosives are injected by charging equipment into 38-mm-diameter blastholes at a depth of 4.0 m. These waterproof emulsion explosives are designed for underground blasting activities in oil shale mines, which present risks of dust explosions. These explosives have low sensitivity to mechanical impact, and provide a high velocity of detonation. Emulsion explosives offer advantages such as improved fragmentation, reduced consumption, and suitability in wet conditions (Sabanov et al., 2023). The emulsion explosives' technical properties are shown in Table 2.

Table 2. Specifications of explosives (Sabanov et al., 2023)

Parameter	Value
Blasting energy, kJ/kg	2900-2950
Gas volume, l/kg	900-990
Detonation velocity, km/s	3.8-4.5
Density, g/cm^3	0.8-1.0
Oxygen balance, %	-0.9-0.0

Blast initiation uses Dynadet short-delay detonators with 0.25-second intervals over an amount of 26 stages, leading to a total blast duration of 6.5 seconds. Six large-diameter (280 mm) undercutting holes (reamer holes) are drilled into the central seam (C layer) to enhance detonation efficiency and fragmentation. The specific charge and blast damage radius were carefully calculated to ensure safe distances from blastholes to the final contours of the roof, walls, and floor (Sabanov et al., 2023).

The initial blasting pattern, which employed a total charge of 75.0 kg, occasionally resulted in a reduced entry advance of just over 3.3 meters. This corresponded to an average weight-to-volume ratio of 0.85 kg/m³, which proved insufficient to consistently achieve the desired advance of 4.0 meters (Sabanov et al., 2023).

To address this issue, a blast simulation and Monte Carlo simulation were conducted using JKSimBlast 2DFace software, enabling the quantification and visualization of explosive energy distribution and to assess the burden relief within the pattern. So, according to the Figure 12, the initial detonation simulation with Monte Carlo analysis shows many holes with a very low success rate (red ones) meaning these blastholes have inadequate burden relief. The rock surrounding these holes stays tightly consolidated during detonation, which greatly limits the effectiveness of fragmentation and raises the chances of excessive vibration or back-break.

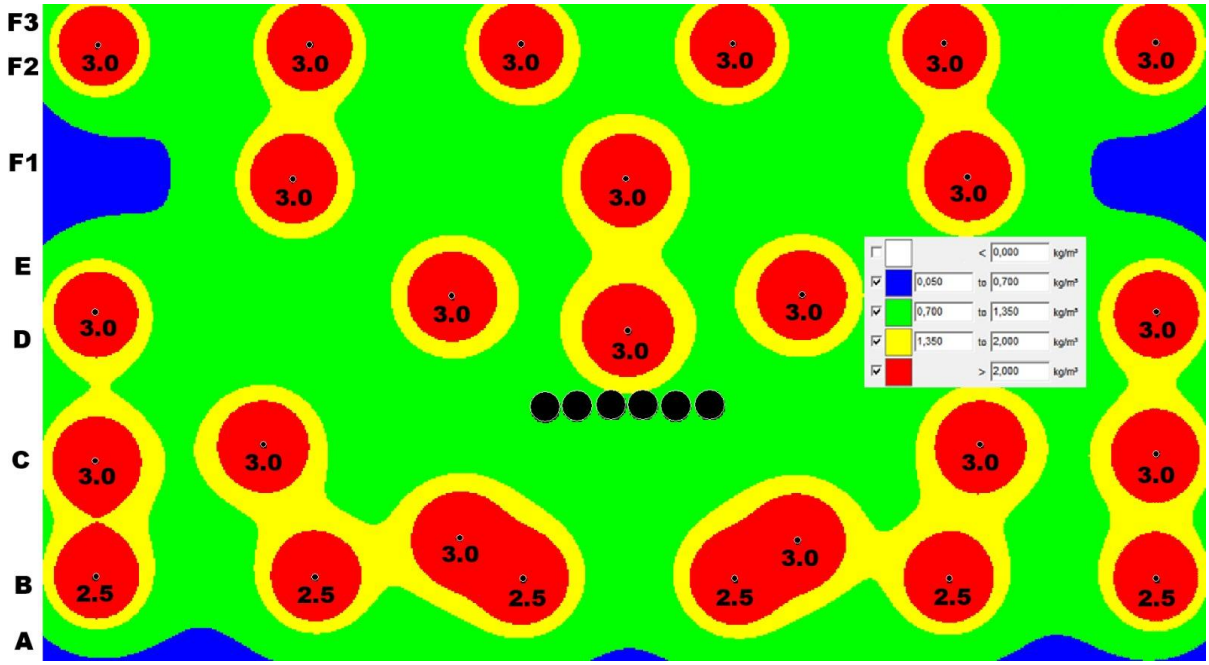

Figure 4. Explosive energy concentration of initial blasting pattern (Sabanov et al., 2023)

As shown in Figure 4, the blast simulation revealed zones of high energy concentration indicated as red, particularly in the F3 layer, where the energy intersected the immediate roof. This concentration of explosive energy contributed to roof instability, including exfoliation and localized roof falls observed during field trials. Conversely, the A layer displayed blue zones, indicating low energy input that was inadequate to create a level floor. Also, these findings, when cross-referenced with field measurements, served as the basis for refining and optimizing the original blasting design (Sabanov et al., 2023).

3.3. Blasthole modelling

When it explodes, the explosive sends out a forceful shock wave. This wave moves fast through the rock. The moving energy causes cracks to appear around the blasthole, it does this by creating pressure and pulling forces that break the rock. Right after this, the still energy from the expanding gases, which come out at high pressure plus heat, puts more force on the wall of the borehole, this gas pressure opens the cracks that are there. It also shoves the rock outward and causes more cracks to spread also the rock to move - these actions work together and are how blasting breaks up rock well. Consequently, a narrow zone of intense rock failure forms around the blasthole, where the material is extensively damaged through mechanisms such as granular disintegration, microcracking, particle compaction, and plastic deformation within the rock matrix (Whittaker et al., 1992). As Jia et al. (2024) explain, the detonation pressure can reach values between 1 and 14 GPa that is far surpassing the rock's mechanical resistance which results in nonlinear deformation and the formation of a crushing zone typically extending 1.5 to 4 times the radius of the explosive. Similarly, Torbica and Lapčević (2018) describe this immediate post-detonation region as the zone where the applied pressure exceeds the rock's strength, leading to dense radial cracking and shearing in the near-borehole area, commonly referred to as the crushed zone. Figure 5 represents the blasthole and the events that are happening around it.

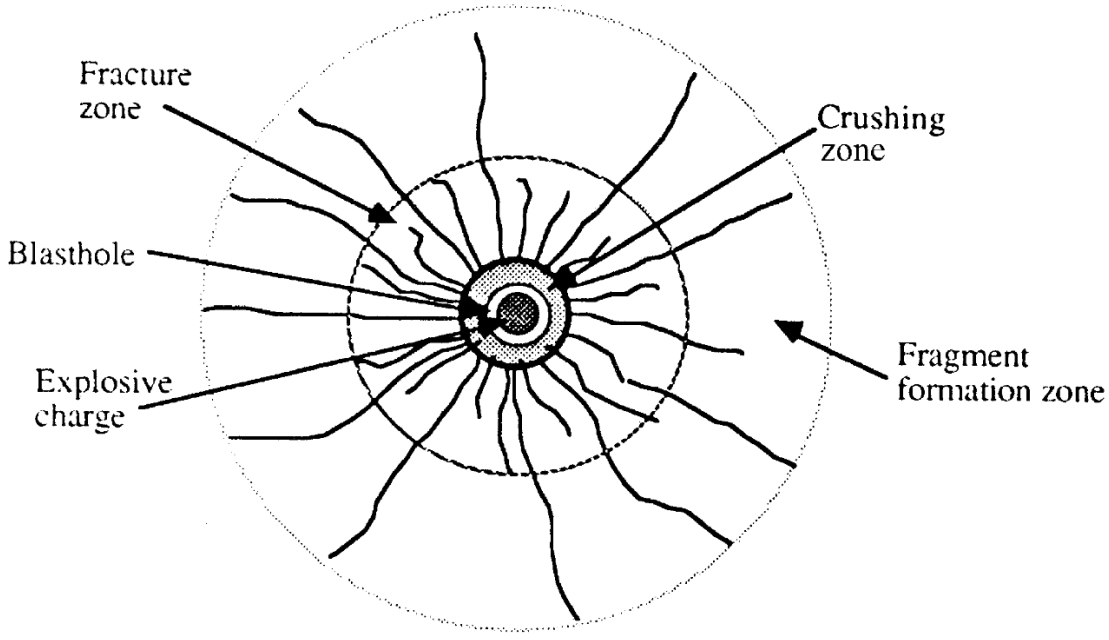

Figure 5. Illustration of events in the rock mass around blast (Whittaker et al., 1992)

The production of fine material in blasting is typically attributed to a cylindrical crushed zone that forms around the blasthole. In this zone, intense compressive and shear stresses cause the rock to fail, resulting in the generation of fine particles through crushing (Kanchibotla et al., 1999). The extent of this crushed zone is generally defined as the distance from the blasthole center to the point where radial compressive stresses surpass the uniaxial compressive strength (UCS) of the rock. It can be calculated using the next equation:

$$r_c = r\sqrt{\frac{P_d}{\sigma_c}} \tag{3}$$

Where r_c is the radius of the crushed zone, r is the radius of borehole (m), σ_c is the uniaxial compressive strength of the rock (Pa) and P_d is the detonation pressure. Detonation pressure can be calculated by the next formula:

$$P_d = \frac{\rho_e {C_d}^2}{\gamma + 1} \tag{4}$$

Where ρ_e = explosive density (g/cm³), C_d is the velocity of detonation (m/s) and γ represents the ratio of the specific heat of the gases produced by detonation, with a value of 3.

Cunningham (1983) highlighted that the ratio of specific heats (γ), typically varies depending on the type of explosive being used. Specifically, γ is approximately 2.6 for Ammonium Nitrate Fuel Oil (ANFO) explosives, while it rises to about 3.2 for emulsion-based explosives. This ratio is important in determining the explosive's detonation characteristics, as it directly influences the detonation pressure and the overall energy distribution during blasting operations. Any reduction in the VOD significantly affects explosive performance, leading to decreases in both detonation pressure (Pd) and shock energy levels delivered to the surrounding rock.

By applying Equations (3) and (4) to the specific conditions of this study, the blast damage radius (Rd) can be estimated using the following expression (Sabanov et al., 2023):

$$R_d = 1.65r\sqrt{\frac{\rho_e {C_d}^2}{4\sigma_c}} \tag{5}$$

Where r is the borehole radius (mm), ρ_e is the explosive density (g/cm³), C_d is the velocity of detonation (m/s), σ_c is the uniaxial compressive strength of the rock (Pa) and 1.65 is the empirical factor that was validated in field tests.

This equation was used to ensure that the blast damage radius did not intersect the final contours of the roof, walls, and floor, thereby maintaining stable mining conditions. Figure 6 illustrates a scheme for the blasthole, crushing zone, fracture zone and blast damage radius.

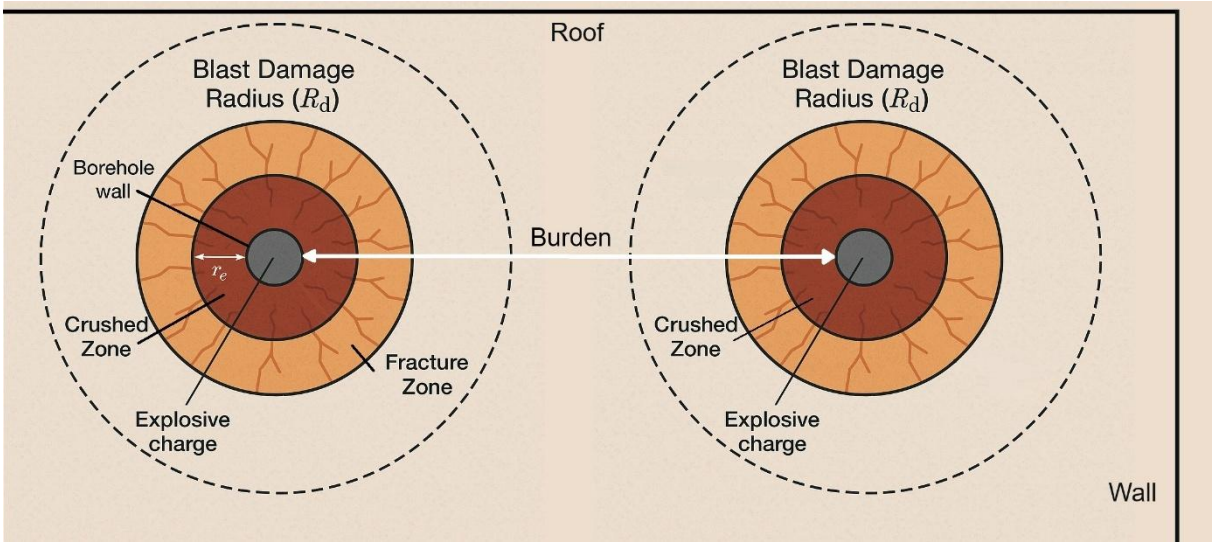

Figure 6. Scheme for Blast damage radius (Sabanov et al., 2023)

3.4. Blast simulation with JKSimBlast software

The study used JKSimBlast 2DFace software to simulate the detonation process, explosive energy distribution, and blast damage radius. The 2DFace module is created for the purpose of developing and analyzing development round blasts in an underground environment. The layout is designed to focus on the face, allowing the user to specify drives and create polygons that represent the complete cross-section of a development face. The simulation of blast considers factors such as:

- Blasthole arrangement: The arrangement, spacing, and dimensions of drill holes, including hole diameter, length.
- Explosive type and loading: Specific explosive characterized by density and energy content.
- Rock mass properties: The density and strength of the oil shale and limestone layers.
- Detonation Timing: The sequence and timing of detonation for each blasthole.

3.4.1. Explosive energy distribution

The explosive energy distribution in JKSimBlast 2DFace is calculated using two approaches: static (3D) and dynamic (4D). The static (3D) approach does not consider timing, assuming all explosive charges detonate simultaneously. Thus, it represents the maximum potential energy distribution. In contrast, the dynamic (4D) approach integrates timing into the calculation, accounting for the sequence and delay intervals between detonations.

Explosive energy distribution can be represented in various units, including kg/t, kg/m³, MJ/t, MJ/m³. These units correspond closely to the standard powder factor calculation, expressing the explosive quantity per unit of rock mass or volume.

Equation (5) for the blast damage radius is closely related to the "Kleine Field," which provides a more comprehensive understanding of energy distribution. Rather than considering the explosive as a whole, it is divided into very small segments, with each segment providing a portion of energy or "explosive concentration" at a specific point P. This integration is the primary reason this method can more accurately connect “charge distribution” to the observed damage boundary, effectively addressing real-world complexities such as deck loading, explosive variations, and local rock conditions.

Three-dimensional explosive energy distribution in 2DFace is calculated without considering the timing of detonations, following the methodology developed by Kleine et al. (1993). The method uses the traditional powder factor concept, it breaks the explosive charge into smaller parts. The process calculates energy concentration at a certain point "P" by adding the effects from all segments. It uses spherical energy fields that begin at each segment, as Figure 7 shows - this method shows how explosive energy spreads in the rock. The general form of the equation representing this calculation is provided below (Kleine et al., 1993):

$$P = \int_{L1}^{L2} \frac{1000 \cdot \rho_e \cdot \pi \left(\frac{D}{2}\right)^2}{\rho_r \frac{4}{3}\pi (h^2 + l^2)^{\frac{2}{3}}} dl \tag{6}$$

This equation can be integrated and rewritten as (Kleine et al., 1993):

$$P = 187.5 \frac{\rho_e}{\rho_r} D^2 \frac{1}{h^2} \left(\frac{L_2}{r_2} - \frac{L_1}{r_1}\right) \tag{7}$$

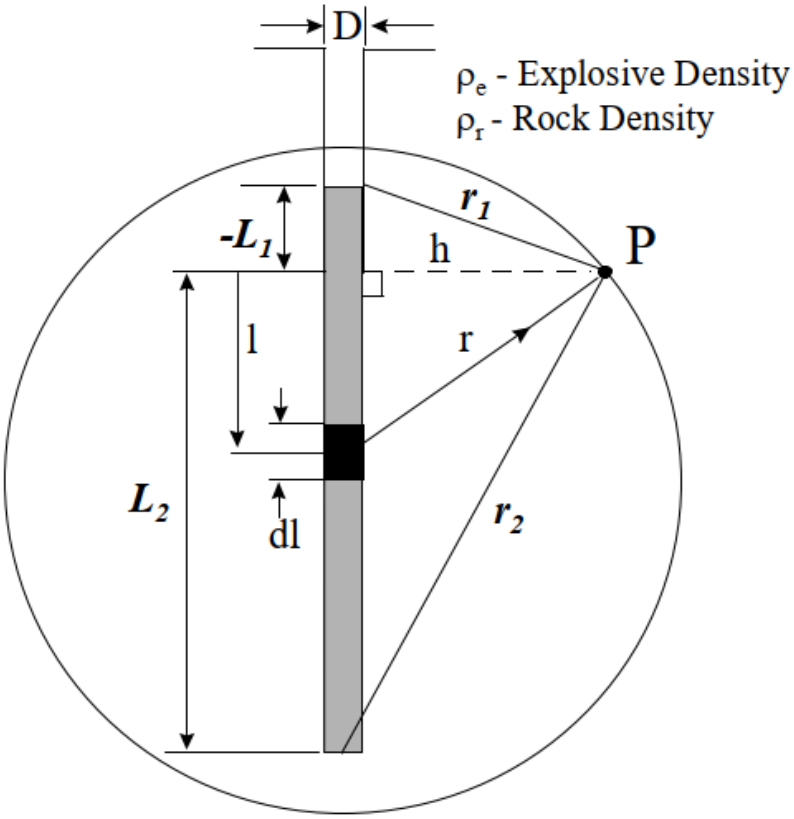

Figure 7. 3D explosive energy concentration at point P (Kleine et al., 1993)

The four-dimensional (4D) explosive energy distribution calculation in 2DFace builds upon the previously discussed three-dimensional (3D) methodology, but includes an additional timing component. This timing component, known as "cooperation time," refers to the duration during which adjacent explosive charges can jointly contribute energy to a particular region of the rock mass. Within the Dynamic (4D) Explosive Distribution settings, the cooperation time serves as a weighting factor, influencing the amount of explosive energy considered from each deck according to its detonation sequence. An initial estimate for cooperation time typically corresponds to the time required for rock movement or fragmentation, analogous to the burden movement time observed in surface blasting practices.

In practice, the cooperation time in a 4D dynamic energy-distribution analysis should align with the moment when the rock in front of a blasting deck starts to move, thus stopping to "cooperate" with other decks. For short underground faces (on the order of a 3–4 m round) in medium-to-hard rock, a typical starting point is often 20–30 ms. All the parameters can be observed in Figure 8.

Figure 8. Explosive energy distribution settings (Produced with JKSimBlast software 2025)

3.4.2. Detonation simulation and time contouring

The simulation of the blast detonation sequence enables visualization and details of the detonation sequence. It involves running a Monte Carlo simulation to account for timing uncertainties. During this process, the software evaluates whether each explosive deck achieves sufficient burden relief at the time of detonation. Burden relief is related to the displacement or removal of rock in front of the blasthole, so the next charges can detonate into free face instead of confined rock. It is important for achieving optimal fragmentation, less vibration rates, and it prevents structural damage like overbreak. Monte Carlo simulation is done to evaluate burden relief by including realistic changes in detonation timing and explosive performance. Specifically, this analysis assesses deck performance by repeatedly calculating success or failure outcomes. These outcomes are determined based on three main criteria: the number of other charges detonating within a defined proximity and several milliseconds before the current deck; the proximity distance, typically represented by the largest spacing between

holes in the blasting pattern; and the estimated burden movement time, which is the duration required for the rock mass to move (Figure 9).

Figure 10 shows the 26-hole development pattern that JKSimBlast 2DFace simulated. Six cut holes in the center fire first, they go off in 250 ms steps, from 0 to 1250 ms, to start the initial fractured area. Twelve stoping or lifter holes follow from 1500 to 4250 ms - these holes spread the crack network. Eight contour holes, which have a square symbol, finish the sequence by 6250 ms.

Dynadet detonators set this 250 ms delay scheme. The plan lets dynamic shock waves begin in the middle. Static gas pressure then moves outside. This improves how the rock breaks - it also holds back overbreak and vibrations. Overlaid contours show when shock waves arrive and how energy moves from the reamer holes to the perimeter.

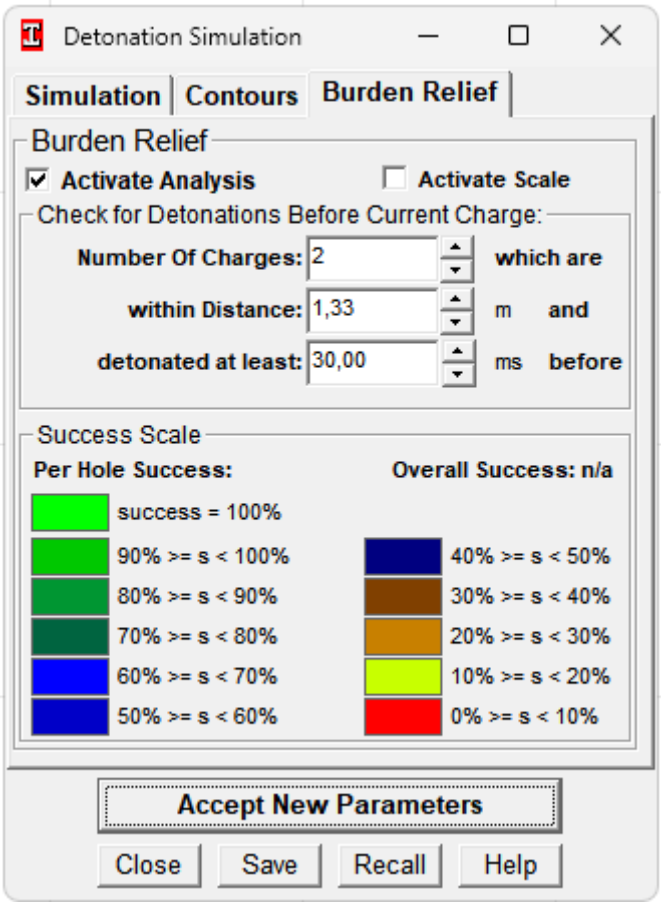

Figure 9. Monte Carlo burden relief simulation (Produced with JKSimBlast software 2025)

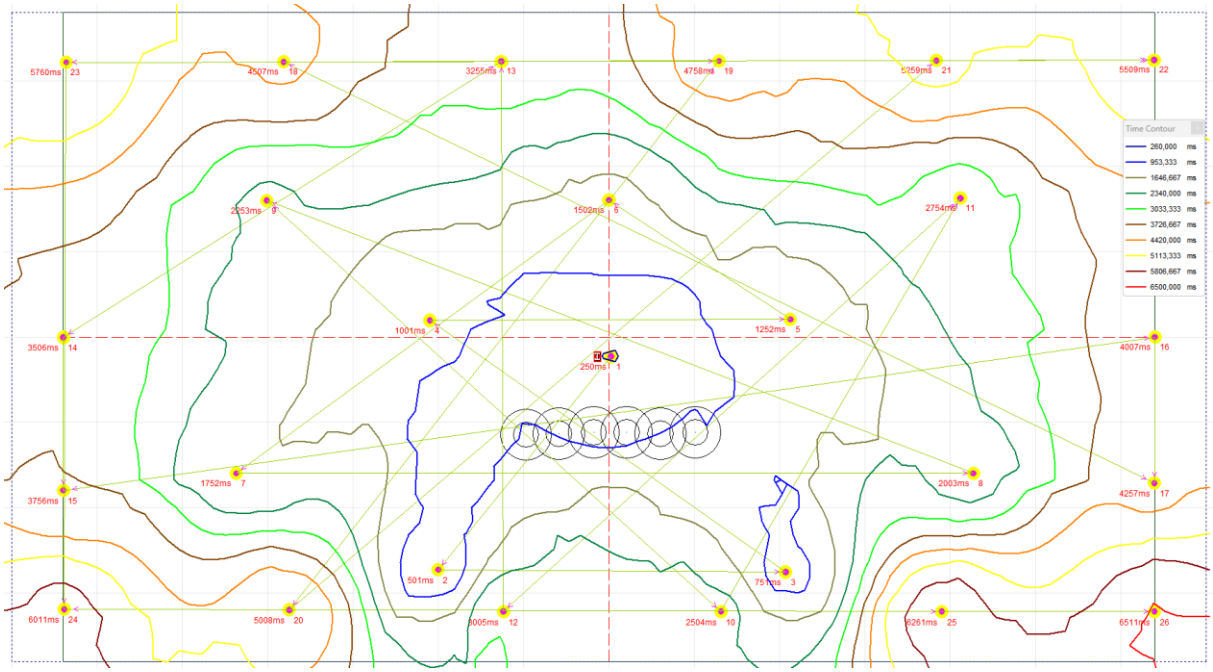

Figure 10. Blasting pattern and detonation sequence (Sabanov et al., 2023)

3.4.3. Near field PPV Prediction

Another useful tool in 2DFace app is the near field peak particle velocity prediction. The Holmberg-Persson model is commonly used in blasting engineering to estimate near-field PPV and evaluate possible rock mass damage from explosive charges. The model created by Holmberg and Persson (1980) links the induced strain in rock masses to PPV, based on the assumption of elastic medium behavior. This model has been shown to be relevant in real-world situations by several researchers, including McKenzie et al. (1995).

The model by Holmberg and Persson (1980) suggests that in the area near a charge, lasting damage happens when a certain level of induced strain is achieved. They proposed that in an elastic medium, the induced strain (e) can be roughly related to PPV using the following equation:

$$\varepsilon = \frac{PPV}{c} \tag{8}$$

where c is the velocity of stress wave. To predict PPV, the model employs a charge-weight scaling law:

$$\vartheta = K\frac{W^{\alpha}}{R^{\beta}} \tag{9}$$

where: W is the charge weight (kg), R is the distance from the charge (m), K, α, β are site-specific attenuation constants derived empirically.

The Holmberg-Persson model operates under several assumptions:

- Infinite VOD: Timing effects between charge segments are neglected.
- Elastic Medium: Rock is treated as a homogeneous, isotropic elastic material, disregarding fractures or heterogeneity.
- Additivity: PPV contributions from individual charge segments are linearly summed.
- No Free Face Effects: Boundary conditions, such as free surfaces, are not considered.

For a cylindrical charge (Figure 11), the general equation for PPV at a point P in 3D space integrates contributions from infinitesimal charge segments along the borehole:

$$PPV = K\left[l\int_{x_3}^{x_3+H}\frac{dx}{[r_0^2+(x-x_0)^2]^{\frac{\beta}{2\alpha}}}\right]^{\alpha} \quad (10)$$

where H is the charge length, ℓ is the linear charge concentration (kg/m), and r_o is the radial distance from the charge axis. Under the condition β=2α, the equation simplifies to:

$$PPV = K\left[\left(\frac{l}{r_0}\right)\left(\tan^{-1}\left(\frac{H+x_3-x_0}{r_0}\right)+\tan^{-1}\left(\frac{x_0-x_3}{r_0}\right)\right)\right]^{\alpha} \quad (11)$$

This relationship indicates that the peak particle velocity at a specific point in three-dimensional space is determined by the position of that point relative to the charge, the type of explosive used, and the geometry of the hole, which is characterized by the linear charge concentration "l" (kg/m). Additionally, it is significantly influenced by the attenuation characteristics of the rock mass, represented by the site-specific constants K and α. According to study by McKenzie et al. (1995) the K and α parameters for strong shale are 175 and 1.25, critical value for PPV is 350 mm/s and PPV breakage point is 1400 mm/s. Figure 12 shows these and parameters for Holmberg/Persson model.

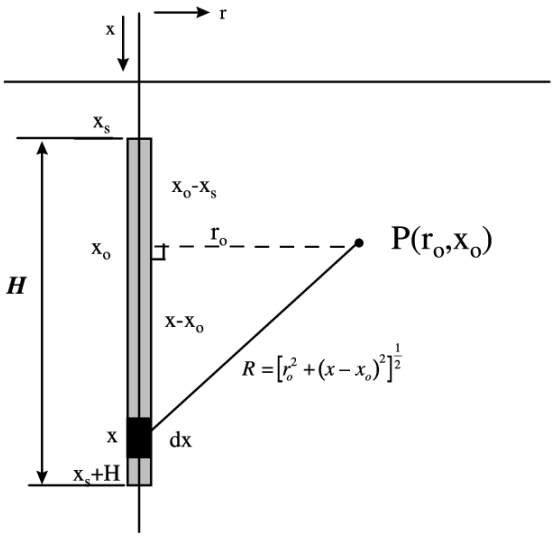

Figure 11. PPV at point P (Holmberg and Persson, 1980)

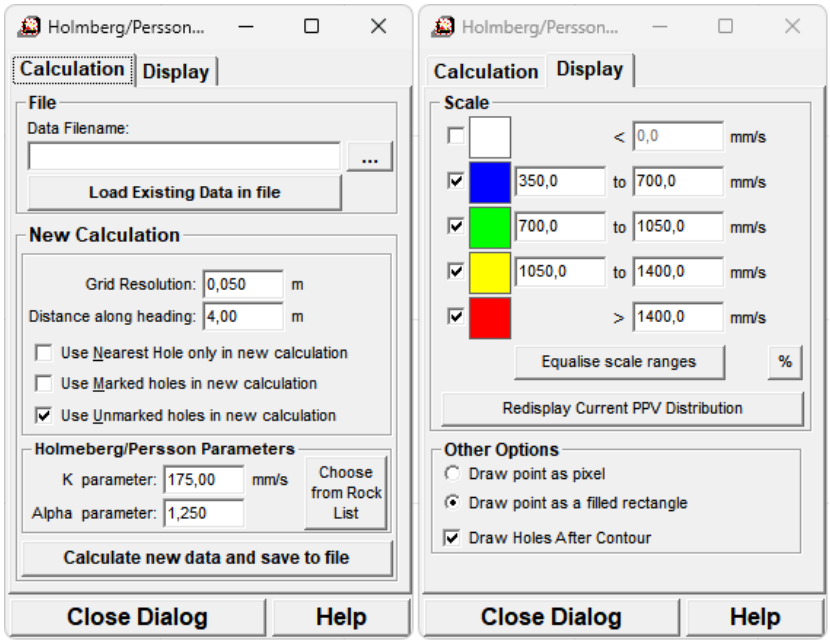

Figure 12. Holmberg/Persson PPV parameters

3.5. Proposed analysis methodology

Sensitivity analysis was performed with @RISK (Lumivero LLC) to find out which drilling and blasting settings affect the main performance indicators most, primarily the blast damage radius (Rd) and the specific charge (SC). This add-in for Excel changes the model from deterministic one into a stochastic by substituting uncertain inputs with probability distributions. After that, it performs a Monte Carlo simulation that involves sampling each distribution, recalculating the spreadsheet, and storing the outputs thousands of times. It helps to estimate the full probability distribution of each outcome.

The deterministic spreadsheet model (Figure 13) was first put together from the physical and ground variables that direct it, which include the borehole radius (r), the density of the explosive (ρ_e), the velocity of detonation (Cd), the uniaxial compressive strength (σc), and an empirical field factor - dimensionless coefficient used to adjust theoretical results based on observed field conditions. Such parameters directly manage how much explosive energy stays contained, how waves move along with how the rock mass reacts mechanically. The tracked outputs were the blast damage radius, which is the radial reach of rock that the explosion weakens, also the specific charge.

After the inputs became @RISK distributions, and the program ran the 10,000-trial simulation, four other sensitivity methods appeared. For this study, the standardized regression method was picked - it creates one equation that shows the output as a weighted sum of the inputs. That equation ranks each parameter from - 1 to 1 based on how much and in what way it affects the output. For instance, one trial with r = 0.016 m, ρ_e = 940 kg m^{-3} as well as UCS = 92 MPa predicted a damage radius of about 0.18 m. It shows how small changes in how much energy stays contained or how strong the rock is can change Rd.

The resulting sensitivity charts as well as tables display that borehole radius and explosive density control the outcome. VOD, UCS in addition to the empirical field factor follow in order of decreasing influence. Such understanding helps engineers direct design work to the most important parameters, so they can select appropriate explosives, adjust hole diameters, or fit charges to local rock types to get the needed fragmentation while holding back over break and unwanted damage.

	A	B	C	D	E	F	G	H	I	J
1										
2		Borehole radius (r), m	Explosive density (ρe), kg/m3	VOD (Cd), m/s	UCS (σc), Mpa	Empirical field factor			Blast damage radius, (m)	
3		0,016	800	3800	25	1,4			0,184627844	
4		0,017	820	3850	67	1,5				
5		0,018	840	3900	24	1,6				
6		0,019	860	3950	65	1,7				
7		0,02	880	4000	24	1,8			$R_d = 1.65r\sqrt{\frac{r_e C_d^{\,2}}{4\sigma_c}}$	
8		0,021	900	4050	18					
9		0,022	920	4100	67					
10			940	4150	29					
11			960	4200	92					
12			980	4250	26					
13			1000	4300	75					
14			1020	4350	40					
15			1040	4400	65					
16			1060	4450	26					
17			1080	4500	32					
18										
19										
20	Average value	0,019	940	4150	45	1,6				
21	Min value	0,016	800	3800	18	1,4				
22	Max value	0,022	1080	4500	92	1,8				
23									Blast Damage Radius (Rd), m	
24		0,019	940	4150	48	1,6			0,278184058	
25										
26										
27										
28						Specific charge for Roof (SC), g/dm3			2,052541721	
29						Specific charge for Floor (SC), g/dm3			2,151629942	
30						Specific charge for Wall (SC), g/dm3			2,524390393	

Figure 13. Excel model for sensitivity analysis

Table 3 shows the initial input parameters that were used for calculations in Excel. The average values were used to calculate the deterministic model. To calculate the stochastic model and assess the impact of certain fixed input parameters, such as borehole radius and empirical field factor, additional values were included. The uniform distribution function was applied to the borehole radius since all values are equally probable. The three-point estimate method was applied to other parameters. This approach is simple and takes into account the minimum, most probable, and maximum values for variables. This approach commonly uses PERT or triangular distributions. In this simulation, PERT distribution was used because this method allows for the transformation of the three-point estimate into a more continuous curve by incorporating a Normal distribution. When the distribution is not symmetrical, the PERT distribution provides a more appropriate shape than the Triangular distribution. The primary limitation of the triangular shape is that it is not natural and rarely accurately reflects a natural process. In comparison to the PERT distribution, it typically overestimates the tails and underestimates the shoulders.

Table 3. Input parameters for sensitivity analysis

Input parameters	The range of values	Average value
Borehole radius (r), mm	16 – 22	19
Explosive density (ρe), kg/m3	800 - 1080	940
VOD (Cd), km/s	3.8 - 4.5	4.15
UCS (αc), MPa	18 - 92	45
Field factor	1.4 - 1.8	1.6

4. Results

4.1. Initial blast design

According to the Figure 14, the initial detonation simulation with Monte Carlo analysis shows many holes with a very low success rate (red ones) meaning these blastholes have inadequate burden relief. The rock surrounding these holes stays tightly consolidated during detonation, which greatly limits the effectiveness of fragmentation and raises the chances of excessive vibration or back-break.

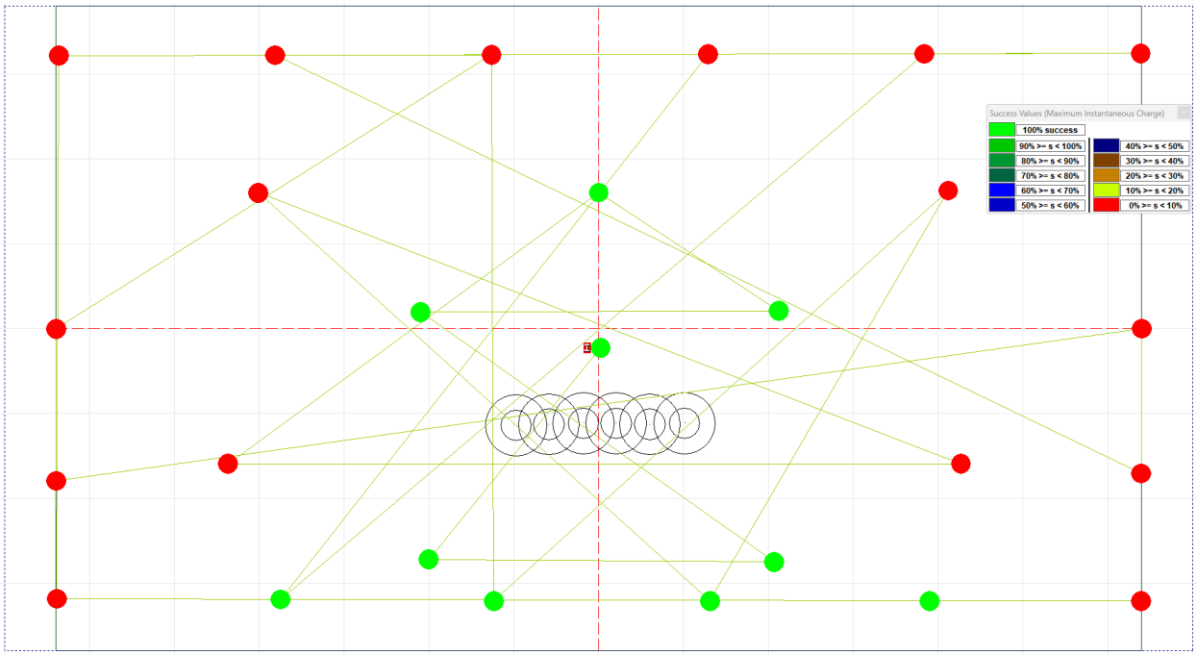

Figure 14. Monte Carlo simulation of the detonation sequence for the initial blasting pattern

4.2. Enhanced blasting pattern and efficiency

Drawing on the outcomes of the simulation and field test data, the blasting pattern was modified by reallocating the distribution of explosive charges to address the observed issues related to roof stability, floor flatness, and insufficient entry advance. The detonation sequence remained unchanged during this optimization. In the optimized blasting pattern, the charge concentration at the F3 layer was reduced by 3.6 kg to mitigate excessive energy near the roof, thereby improving its stability. Similarly, a slight reduction was applied at C layer to protect the integrity of the pillar walls. Conversely, to enhance face advance and floor formation, the charge concentrations were increased by an average of 0.6 kg in the F1, D, and B layers, and

by 1.8 kg at A layer to ensure the formation of a level floor. The revised charge allocations are summarized in Table 4.

Table 4. Comparison of initial and enhanced blast patterns

Layer index	Initial conditions UCS, MPa	Initial scheme of explosives, kg	Complicated conditions UCS, MPa	Optimized scheme of explosives, kg	Difference, kg
F3	25	18	28	14.4	-3.6
F2	24	0.0	26	0.0	0.0
F1	24	9.0	26	9.6	0.6
E	18	6.0	19	6.4	0.4
D	29	9.0	31	9.6	0.6
C	26	12	27	11.8	-0.2
B	40	6.0	42	6.4	0.4
A	32	15	37	16.8	1.8
Total		75		75	0.0

To assess the effectiveness of the optimized blasting pattern, a Monte Carlo simulation was conducted to validate the detonation sequence by evaluating the burden relief at the moment each hole detonates. Using the detonation and time contouring tool, the simulation determined the likelihood of successful detonation for each blasthole based on specific criteria: the number of adjacent charges detonating within a proximity distance of 1.62 meters and a time interval of 30 milliseconds before the hole in question. The simulation results identified holes with a 100% probability of successful detonation, they are indicated as green circles in Figure 15 demonstrating effective burden relief across the enhanced design. The selected proximity distance of 1.62 meters corresponds to standard underground blasting (rule of thumb) with close spacing between holes.

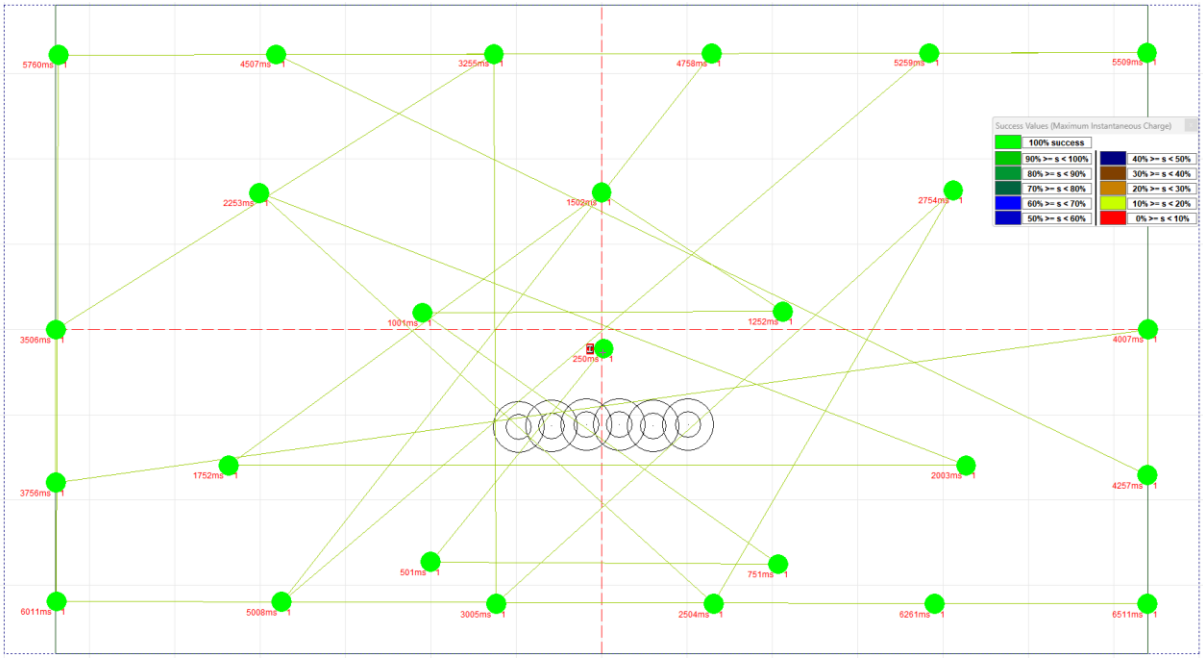

Figure 15. Monte Carlo simulation of the detonation sequence for the optimized blasting pattern

The simulation results presented in Figure 16 indicate a more uniform and controlled distribution of explosive energy for the improved blasting pattern. Notably, the areas of high energy concentration, depicted in red, no longer intersect with the roof or wall boundaries, reducing the risk of structural overbreak. These simulation outcomes were confirmed by field test data, which demonstrated stable conditions in the immediate roof and a consistently level floor. As a result, the optimized blasting pattern successfully increased the entry advance to 4.0 meters and achieved a more efficient average weight-to-volume ratio of 0.70 kg/m³, in contrast to the 0.85 kg/m³ observed in the initial design (Sabanov et al., 2023).

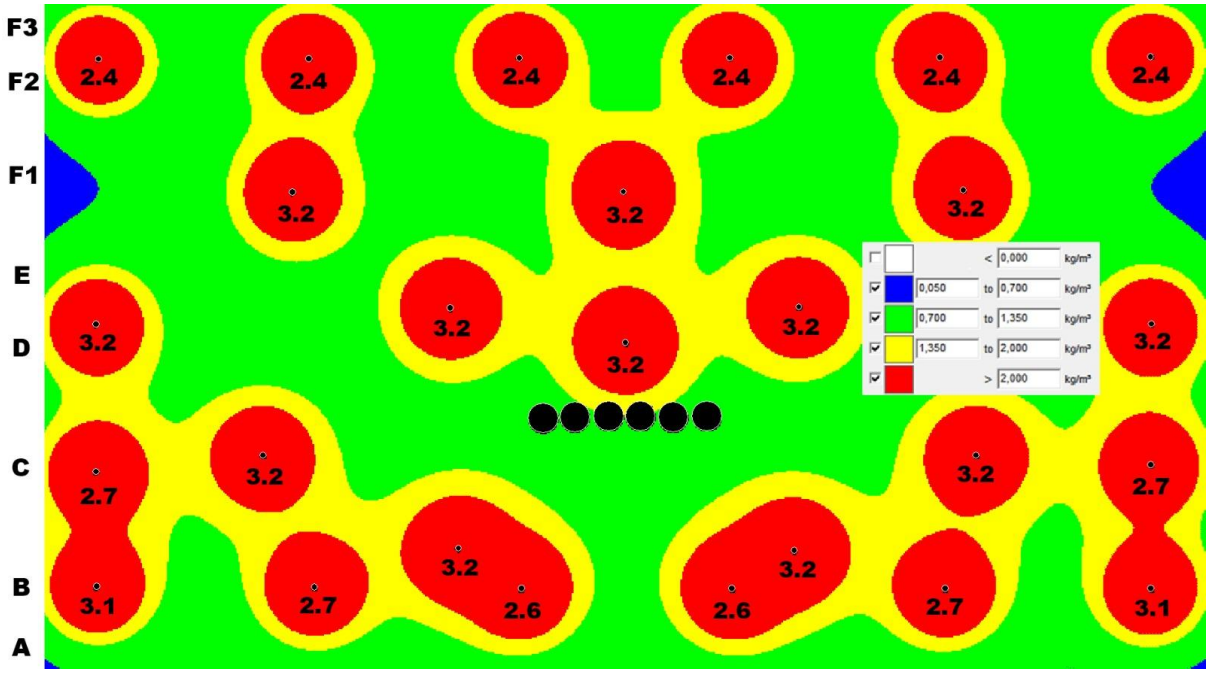

Figure 16. Explosive energy concentration for the optimized blasting pattern

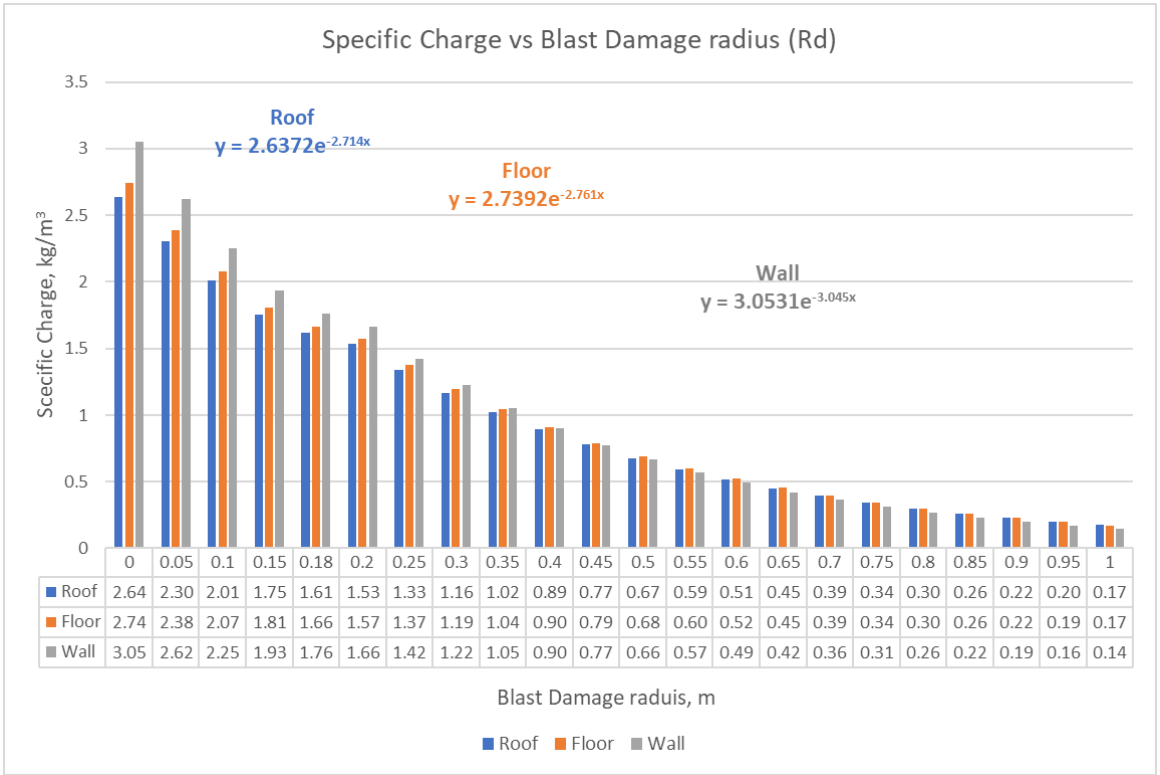

Figure 17. Specific charge vs Blast damage radius

The derived relationships shown in Figure 17 can be integrated with Equation (5), which calculates the blast damage radius (Rd), to estimate the required specific charge (SC). For

instance, the specific charge for blastholes positioned near the immediate roof can be determined using the exponential expression:

$$SC = 2.64e^{-2.7Rd} \quad (12)$$

Applying this to the improved blasting pattern, where the boreholes of the roof are located approximately 0.3 meters from the final contour, the specific charge should not exceed 1.2 kg/m³ in order to avoid excessive damage. Similar calculations can be carried out using the exponential trendlines from Figure 16 for the floor and wall holes. These correlations between specific charge and damage radius are effective tools for estimating the extent of the crushing and fracturing zones around blastholes.

4.3. Near Field PPV

Figure 18 shows PPV contours calculated by JKSimBlast 2DFace using the Holmberg–Persson model. According to study by McKenzie et al. (1995) the K and α parameters for strong shale are 175 and 1.25, critical value for PPV is 350 mm/s and PPV breakage point is 1400 mm/s. The contours show the distribution and strength of vibration intensity created by the blasting design. The blue zones show PPV values between 350 and 700 mm/s, indicating areas with lower vibration intensities. It typically experiences minimal structural impacts and incipient damage. The green (700–1050 mm/s) and yellow zones (1050–1400 mm/s) indicate areas with moderate to significant vibration levels. The green zone is probably the best for fragmentation efficiency while the yellow has a possibility of greater rock damage and induced fractures. The central red areas, which show PPV values over 1400 mm/s, indicate that rock experiencing the highest levels of stress from blasts. This relates to the proximity of blast holes and leads to the crushing of the rock mass.

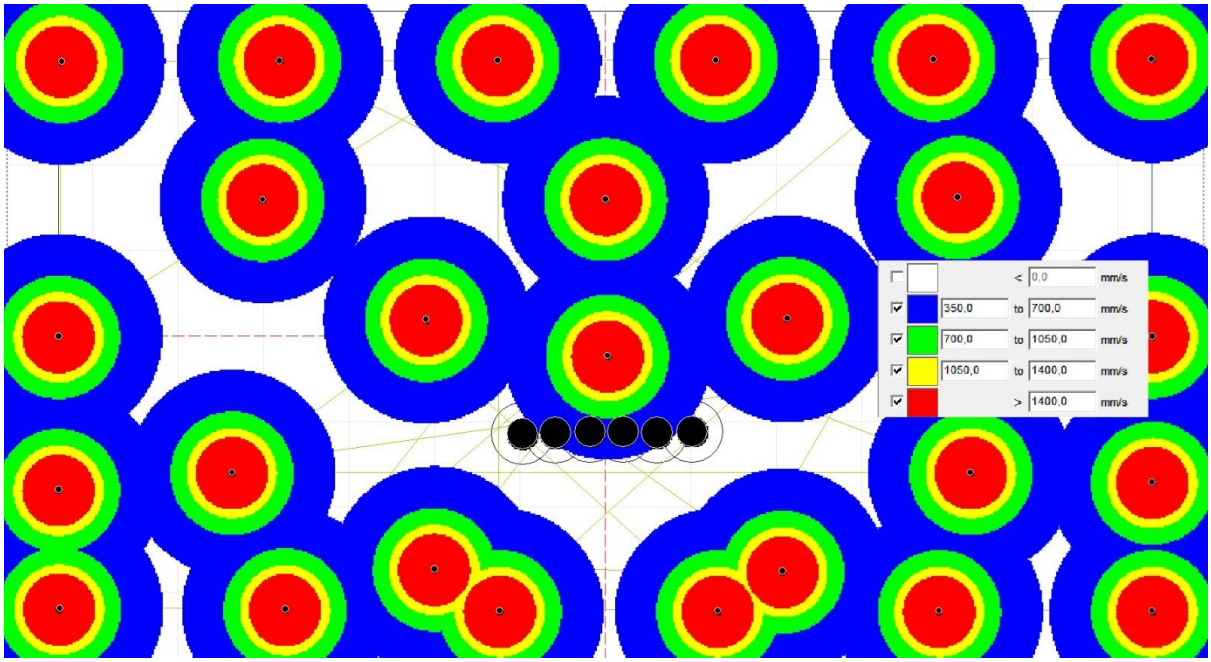

Figure 18. Holmberg-Persson PPV contours

4.4. Distribution and sensitivity analysis results

Figure 19 displays the empirical distribution of probability density for Blast damage radius (Rd) after 10,000 Monte Carlo trials of equation (5) using @Risk software. Blue bars show the histogram of simulated Rd values. The values go from about 0.163 meters to 0.523 meters. The average value is 0.287 meters, and the standard deviation is 0.055 meters.

The red curve on the chart is a lognormal fit with the central peak near 0.27 meters. It also shows the rate at which probabilities decrease at the ends of the distribution. Based on this fit, the average stays at 0.287 meters, and its standard deviation is 0.055. About 4.9 percent of the distribution sits below the fifth percentile cutoff, which is 0.210 meters. 5.1 percent sits above the 95th percentile cutoff, which is 0.388 meters.

A green curve shows the PERT distribution that limits itself by the lowest and highest possible values, which are 0.163 meters and 0.524 meters. It yields a slightly higher average of 0.291 meters. It also shows a wider spread, with a standard deviation of 0.065 meters.

This comparison shows that the lognormal model more accurately represents the simulated behavior of the explosion damage radius than the PERT model.

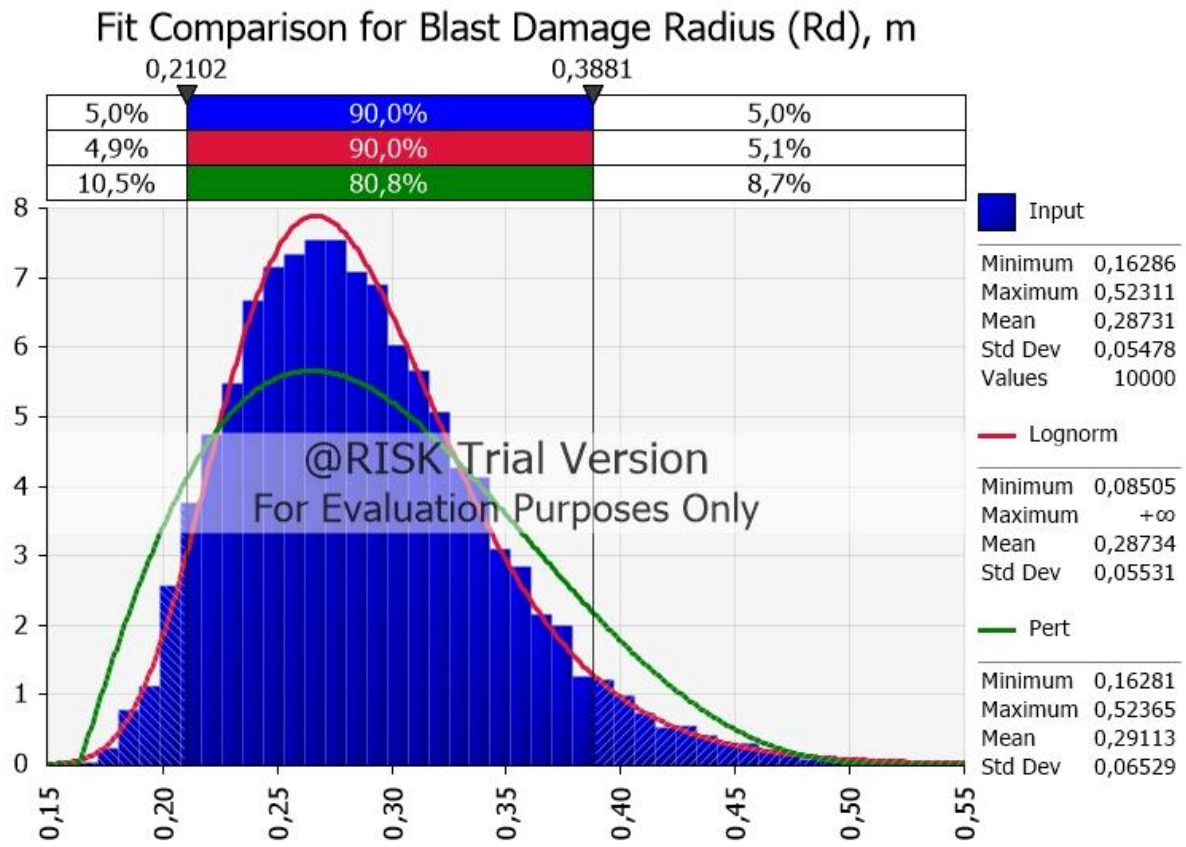

Figure 19. Probability density graph of Rd

Figure 20 shows how specific charge (SC) distributes for roof, floor along with wall settings. The red histogram displays the SC for the roof. This value ranges from about 0.99 g/dm³ to 2.83 g/dm³. It averages 2.02 g/dm³ and has a standard deviation of 0.28 g/dm³.

Two vertical lines at 1.523 g/dm³ plus 2.463 g/dm³ indicate the 5th and 95th percentiles for the roof. Five percent of simulated values fall below the lower line. Another five percent rise above the upper line - this leaves 90 percent of values within these bounds.

The blue overlay shows the floor distribution - it stretches from roughly 1.04 g/dm³ to 2.97 g/dm³. The floor distribution has a mean of 2.12 g/dm³ also a standard deviation of 0.30 g/dm³. Only 84.5 percent of floor values rest between the same percentile cutoffs. Three percent fall below, and 12.5 percent go above.

The green histogram depicts the wall distribution. It moves more to the right, from a minimum of 1.22 g/dm³ to a maximum of 3.48 g/dm³. The wall distribution also exhibits the largest spread, with a mean of 2.49 g/dm³ as well as a standard deviation of 0.35 g/dm³. Only 44.6

percent of wall values land between the roof's 5th and 95th percentiles. About 0.4 percent lie below, plus 55.1 percent stay above.

A review of the three overlaid distributions shows that the most probable specific charge becomes greater, and its spread increases, when a person moves from roof to floor to wall holes. In daily use, when a person uses one central charge estimate for all directions, it often leads to undercharging, especially in wall uses. In our information, over half of the wall hole tests went past the 95th-percentile range that the roof hole example set.

This behavior, which depends on direction, is well recorded. Jimeno, Jimeno & Carcedo, 1995 show that blastholes drilled into side walls undergo greater confinement in the rock and have less relief from a free face. Under such conditions, a bigger part of the explosive power gets used just to get past the rock's hold, instead of spreading cracks. A bigger charge amount is needed to get the same damage size. In contrast, roof holes get help from gravity that helps cracks grow, and they have a wide free face. This lets cracks grow with less charge. Floor holes sit between these two points - they have some hold and some help from gravity and the floor's surface.

Because of this, blast design should adjust the specific charge for each direction. Roof holes can be planned around the 90 % confidence range, which is about 1.52-2.46 g/dm³. Floor holes need a small increase to cover their larger spread. Wall holes require a charge near the top part of the distribution, above about 2.87 g/dm³, to keep a constant blast damage size of 0.3 m, which means side wall holes generally need about 15-20 % more explosive per cubic decimeter than roof or floor holes - this extra amount makes up for the added confinement.

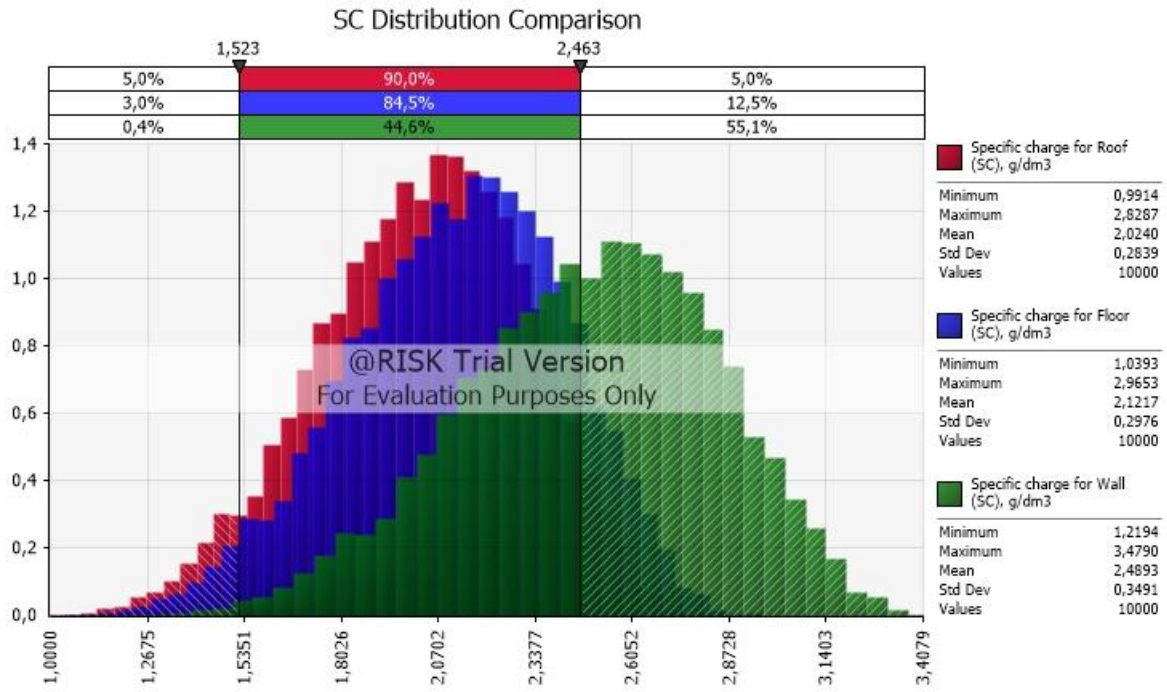

Figure 20. Probability density distribution graph of SC for Roof, Wall and Floor

The next results would show the outcomes for the sensitivity analysis based on regression coefficients that were obtained from plots of simulation results in @Risk software. A positive bar means the high input – high output, a negative bar the opposite.

Figure 21 displays the results from a multiple regression analysis. The analysis measures how each input variable changes the Blast damage radius (Rd). The bars represent standardized coefficients that come from Risk's regression function.

Uniaxial compressive strength (UCS, σc) has the largest coefficient, which is -0.78, which means that when rock strength goes up, the predicted damage radius gets much smaller. Borehole radius (r) comes next, with a positive coefficient of 0.48. This confirms that bigger hole sizes cause a larger damage zone. The empirical field factor holds a moderate positive coefficient of 0.25. This shows its job in adjusting theoretical numbers to what people see in the field. VOD presents a smaller positive effect (0.17), while explosive density (ρ_e) adds the least (0.15).

The results from the multiple regression analysis on the blast damage radius match prior studies in rock blasting. A large negative coefficient for uniaxial compressive strength shows that more rock strength lowers damage. He et al. discussed this, and they looked at how seismic waves spread based on rock features and blast conditions (He et al., 2022).

This study found that bigger borehole sizes raise the damage radius. This agrees with Villaescusa et al., who pointed out how borehole design helps lower blast damage in stope work (Villaescusa et al., 2004). The moderate effect of the empirical field factor confirms the model by Sućeska et al. Their model advises using real data to change theoretical forecasts of how explosives behave (Sućeska et al., 2024).

In addition, the roles of velocity of detonation plus explosive density agree with Wu in addition to Chen's study of how detonation waves move; they showed how these factors affect fragmentation and shock wave creation during blasting (Wu & Chen, 2023). These connections show how carefully managing UCS and drilling parameters improves the ability to predict blast damage.

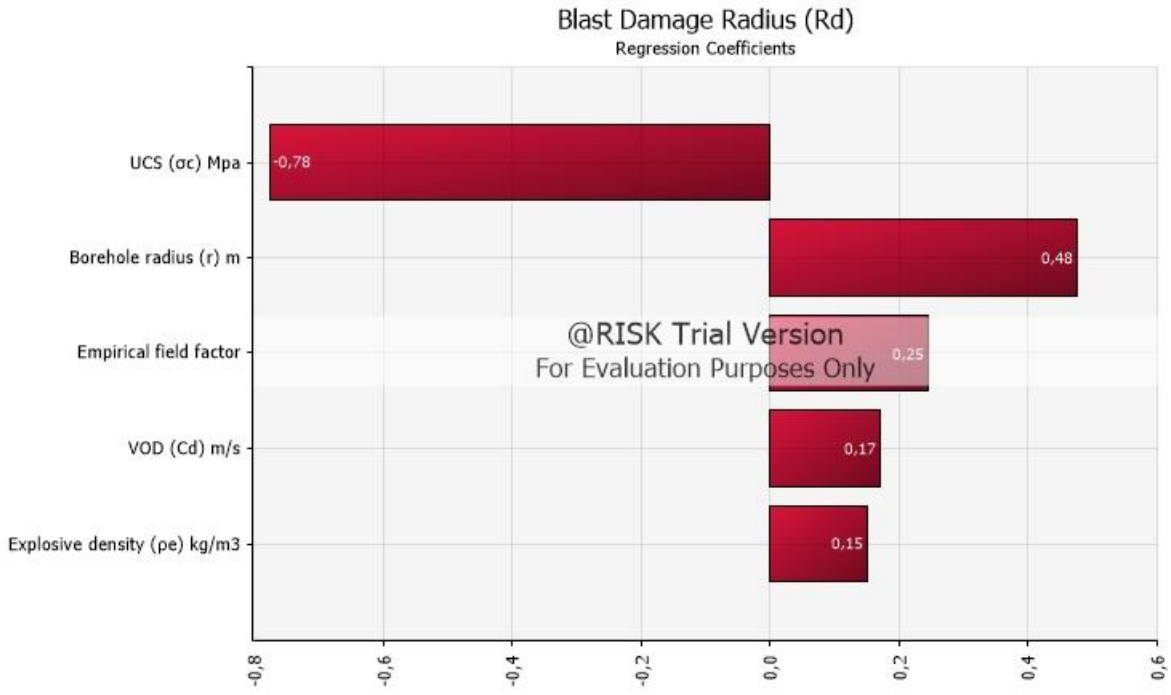

Figure 21. Regression Coefficients for Rd

Figure 22 shows the standardized regression coefficients for specific charge across roof, floor along with wall conditions. The three colored bars represent the roof with red, the floor with blue as well as the wall with green; they all rank the inputs the same and have the same sizes. UCS holds the top spot at +0.79. Borehole radius comes next at - 0.49, and then the empirical field factor at - 0.26. The velocity of detonation has a value of - 0.17, and explosive density has a value of - 0.15.

This means that rock strength and hole diameter control the amount of explosive mass per unit volume needed. This applies no matter if the hole is in the roof, the floor, or the wall. The other inputs change demand by only a third as much.

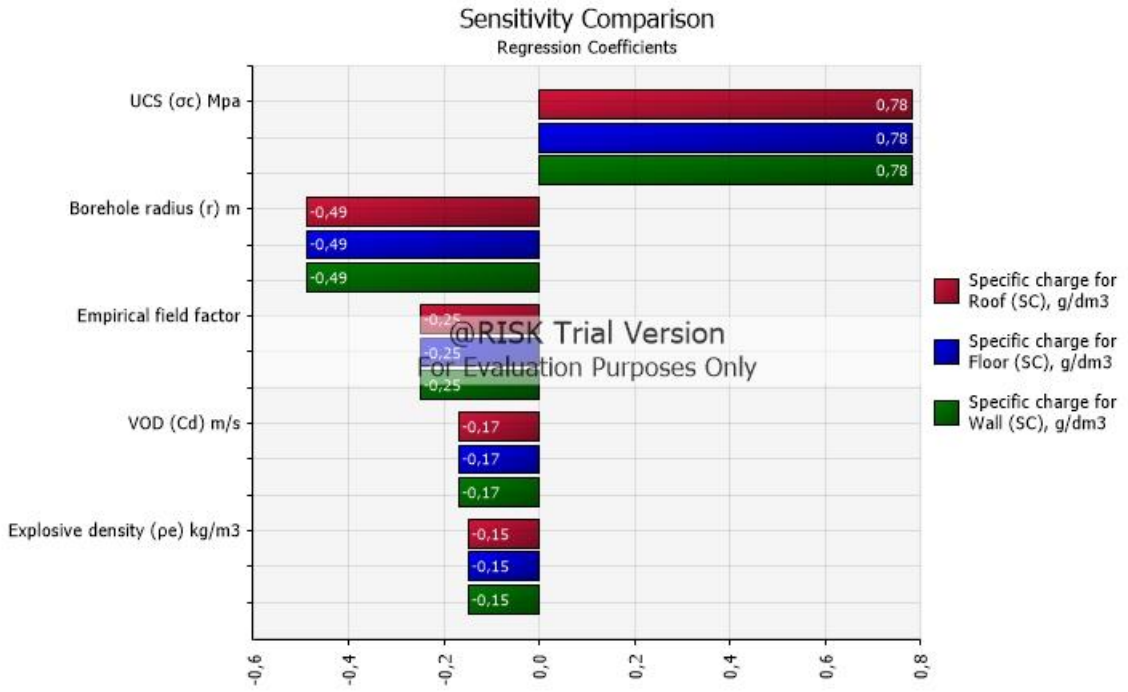

Figure 22. Regression Coefficients for SC Roof, Floor and Wall

Figure 23 illustrates the tornado chart that arranges each input by how much it affects the average blast damage radius. UCS has the largest effect - it moves the average radius from around 0.226 meters at the high strength end to 0.376 meters at the low strength end, a change of about 0.15 meters.

The borehole radius changes the average by and or minus 0.03 meters as the hole size moves within its normal limits. The empirical field factor, detonation velocity along with explosive density each move the average by smaller amounts, somewhere between 0.04 to 0.06 meters.

When one considers these factors together, this shows that reducing error in rock strength readings and keeping close watch on drilling sizes will cause the best changes in telling how far damage goes. For a usable suggestion, the design process should first center on exact UCS tests and careful hole setup. One can adjust predictions by setting the field factor and picking explosive traits, such as density in addition to VOD, to finely tune the blast damage radius.

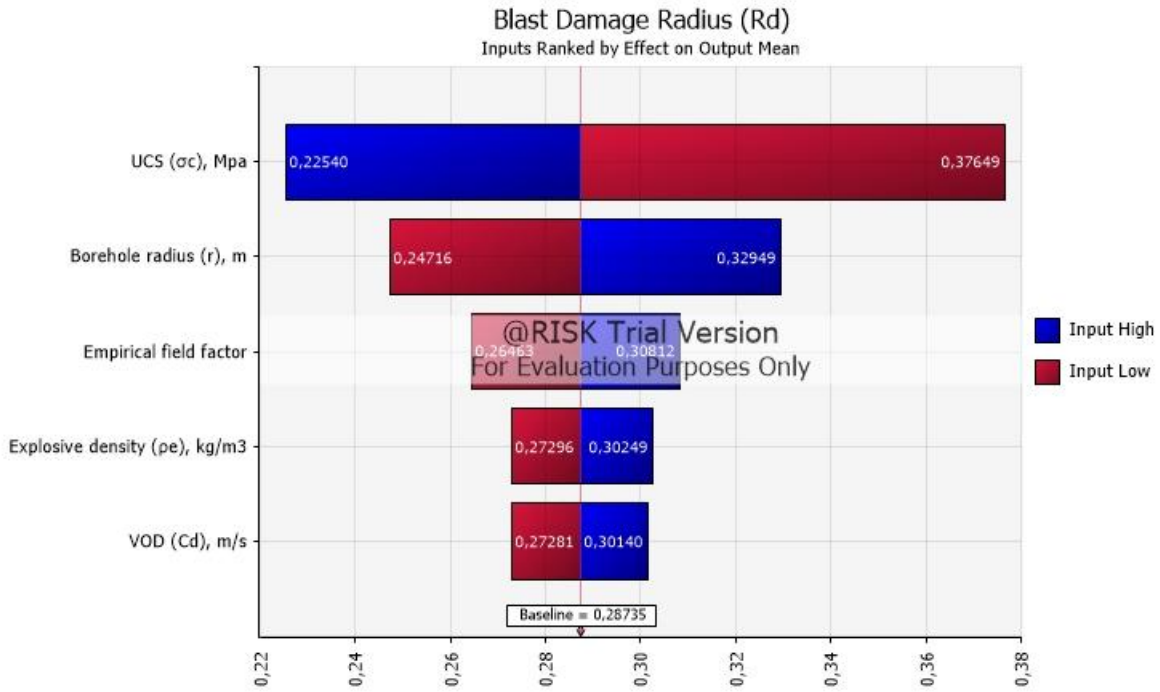

Figure 23. Tornado chart for Blast damage radius (Rd)

Figure 24 shows the spider chart of the mean blast damage radius. Each input parameter changes across its 0 - 100th percentile range. The red UCS curve goes down sharply from about 0.28 m at the 0th percentile to around 0.23 m at the 100th percentile. This confirms that rock strength and damage extent have a clear inverse relationship. The blue borehole radius line goes up from about 0.25 m to 0.33 m - this shows that hole size has a positive effect on radius, which fits the results from Jessu et al. (2018). They showed that wider boreholes cause a bigger damage radius, which backs up their work on how blast damage spreads in hard rock.

The empirical field factor, which is green, causes a moderate rise from roughly 0.26 m to 0.31 m. But VOD, which is purple along with explosive density, which is yellow, stays relatively flat. Each of these shifts the mean radius by only about 0.02 m across their full ranges.

All five lines meet near the 40 - 50th percentiles at the baseline mean of about 0.29 m. The model's central prediction responds most to UCS and borehole radius. Field-factor effects are secondary, plus VOD and density are largely small. That suggests these have lesser effects, as studies by Jia et al. (2024) pointed out, they noted these factors only slightly affect blasting results. The largest effects on damage radius come mostly from exact UCS measurements and borehole setup that matches the advice in studies by Singh (2018), which push for strict control of these factors to better predict blast damage.

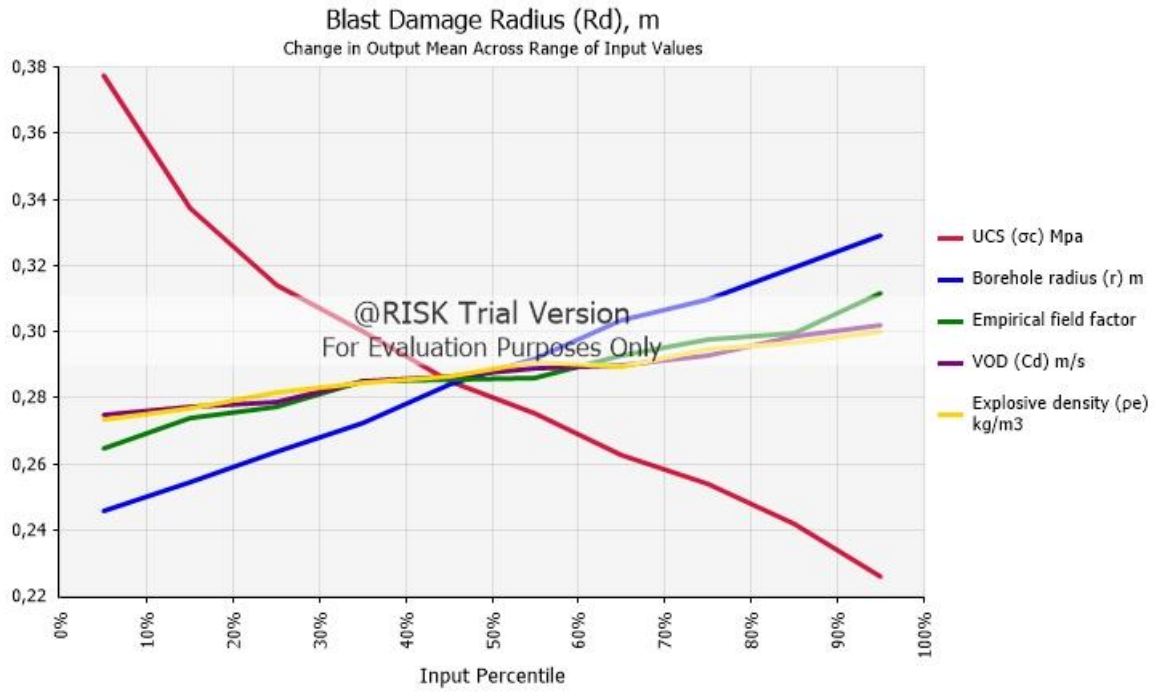

Figure 24. Spider chart for Blast damage radius

Figure 25 shows a scatter plot that displays the damage radius from a simulated blast against UCS. A red cross stands for each trial, which shows a clear inverse pattern. When UCS rises from about 10 MPa to 90 MPa, the damage radius generally drops from around 0.55 m to 0.15 m.

A vertical line and a horizontal line show the average UCS of 48.33 MPa plus the average radius of 0.287 m - these lines split the plot into four parts. The upper left part has a low UCS and a high radius (38.4%). In contrast, only 6.9 percent of trials are in the upper right part, which has high UCS also a high radius.

For the lower left part, with low UCS and a low radius, 13.6 percent of points appear there. 41.1 percent fall into the lower right part, which has high UCS as well as a low radius. The Pearson correlation is - 0.78, and the rank correlation is - 0.79. This measures the negative link between the two variables. This figure validates that differences in rock strength primarily drive the reach of blast damage.

Furthermore, the distribution of trial results showing a significant proportion of high UCS values paired with low damage radii merits additional scrutiny. He et al. reported a general trend of lower damage radii corresponding to increased rock strength, supporting the observational data (He et al., 2022).

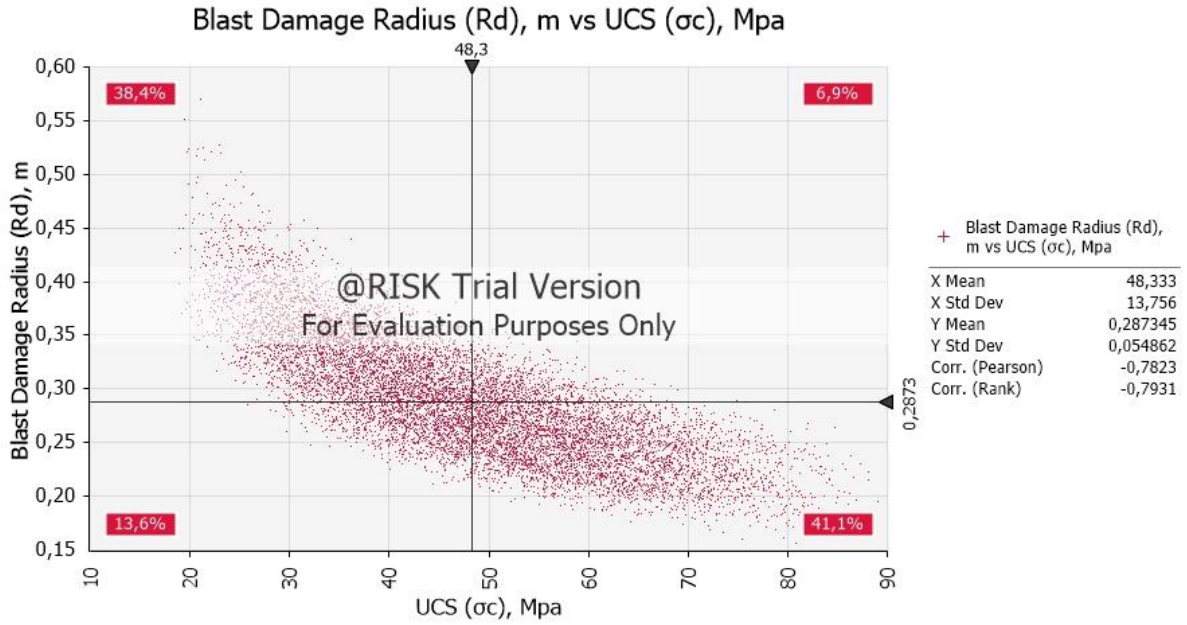

Figure 25. Scatter plot of Rd vs UCS

Figure 26 displays simulated blast damage radius plotted against VOD. The chart shows a mean of 4,150 meters per second and a mean radius of 0.287 m, which the crosshairs denote. The VOD points shape a wide, almost flat band. This differs from the narrow, downward sloping pattern in the UCS plot (Figure E). The VOD points show a very weak positive relationship, which the Pearson r of about +0.16 indicates.

Roughly 20 percent of trials settle in the high-VOD/high-radius quadrant, and 25 percent settle in the high-VOD/low-radius quadrant - these figures nearly match the 29.7 percent plus 25 percent in the low-VOD quadrants - this pattern shows that changes in detonation velocity add little to radius uncertainty. The small effect of VOD contrasts with the substantial - 0.78 correlation for UCS. This confirms that refining rock strength measurements and borehole sizing, which earlier figures show, will bring about much better predictive accuracy than additional exactness in detonation speed.

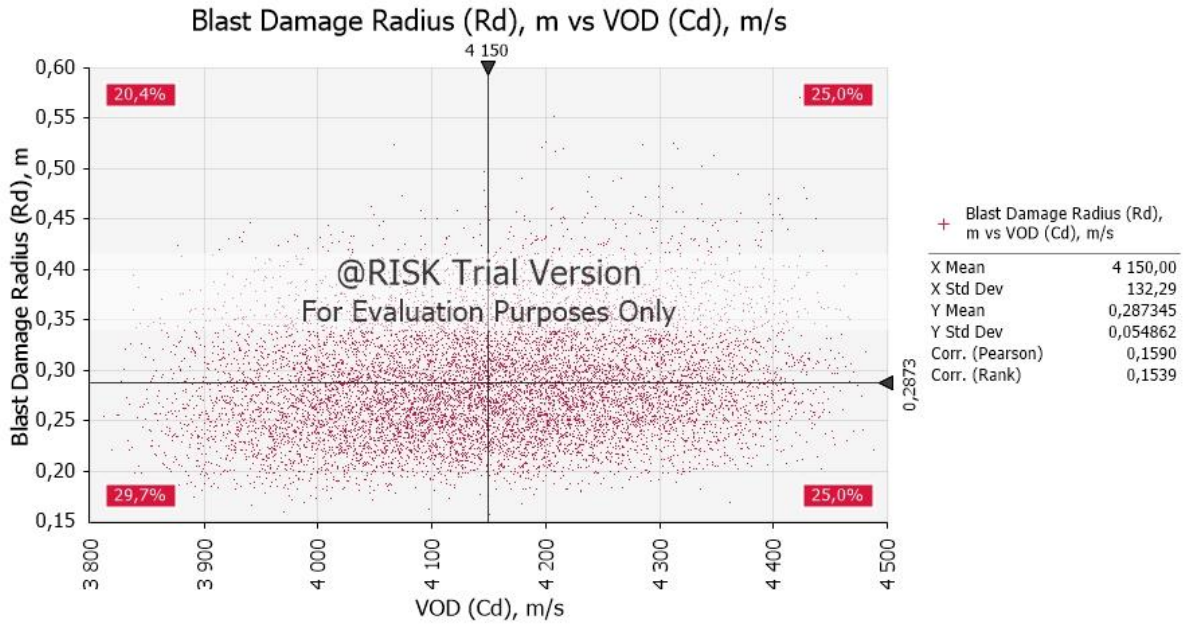

Figure 26. Scatter plot of Rd vs VOD

Figure 27 displays blast damage radius plotted against explosive density. The chart shows a mean density of 940 kg/m³ and a mean radius of 0.287 m. Crosshairs indicate these averages. Like the VOD scatter in Figure 25, the points create a broad horizontal band. This band uncovers a very slight positive link, which the Pearson r of about +0.16 supports. About 20% of trials land in the high-density/high-radius section. Another 25.3% fall into the high-density/low-radius section - this nearly mirrors the amounts at low density.

The small slope indicates that density changes move the average radius by only about ±0.02 m across the whole range. When compared side by side, both density in addition to VOD put out small effects on damage radius. That is especially true against the more considerable effects of UCS and borehole radius. Previous figures show these influences. In practice, this proves that adjusting explosive density or detonation speed will only bring small gains in prediction accuracy. Directing work toward exact rock strength measurement and hole size control continues to be much more important.

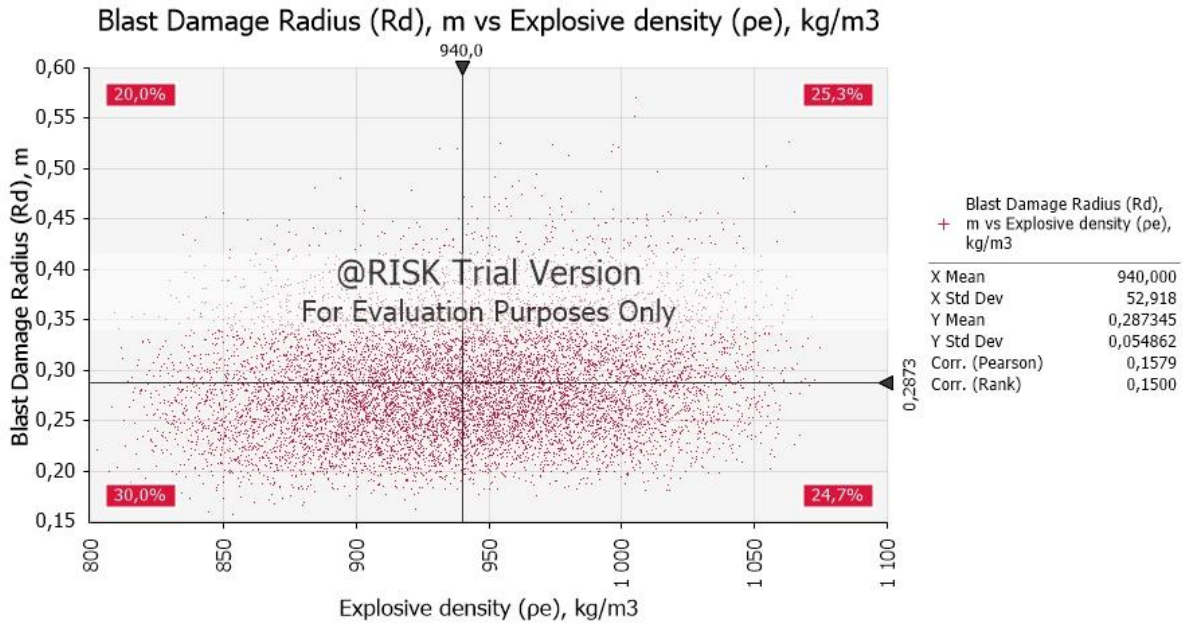

Figure 27. Scatter plot of Rd vs Explosive density

Figure 28 shows the spread of blast damage radius against its borehole radius. Crosshairs show the average radius, which is 0.287 m along with the average hole size, which is 0.019 m. VOD and density created nearly flat bands, but the points on this chart show a clear upward pattern. When the borehole radius grows from 0.016 m to 0.022 m, the damage radius goes up from about 0.22 m to 0.53 m.

A Pearson correlation of +0.48 and a rank correlation of +0.49 show a moderate positive link. About 31.5% of the trials are in the high-radius/high-radius section, which is above both average values. Only 13.8% of the trials are in the low-radius/high-radius section. 36.2% of the runs are in the low-radius/low-radius section, and 18.5% are in the high-radius/low-radius section.

The inverse effect of UCS is much stronger, with an r value of approximately - 0.78. Borehole radius appears as the second most important factor - it explains about one third of the change in damage size, which means that after a person describes rock strength, controlling drilling sizes closely is the next best way to guess and handle the blast damage radius.

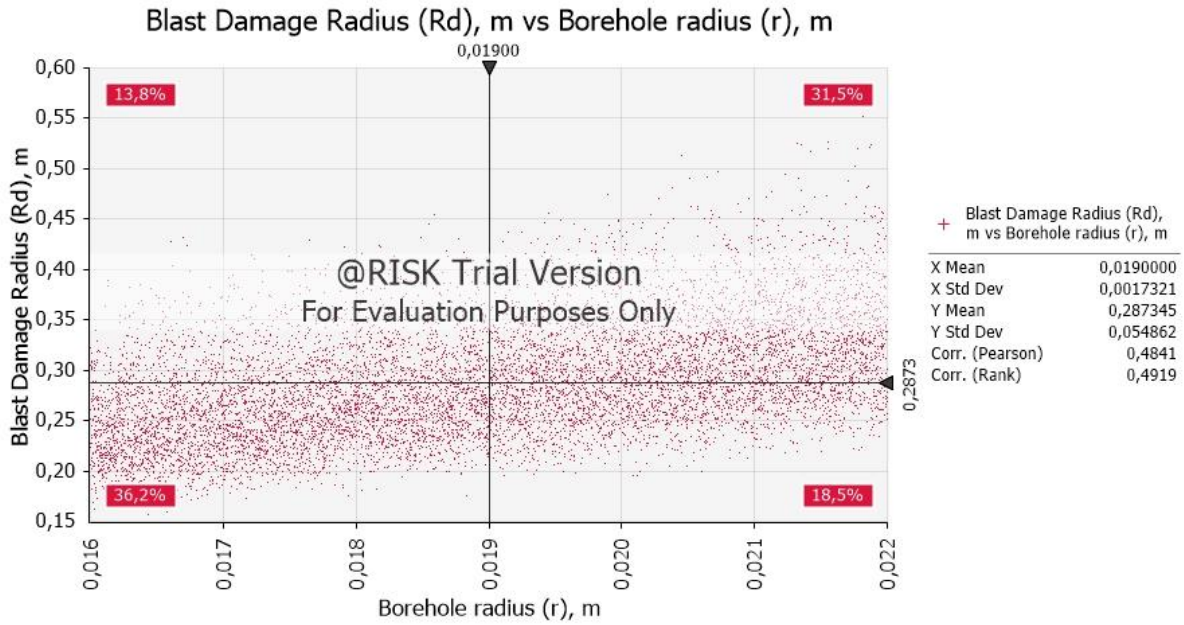

Figure 28. Scatter plot of Rd vs Borehole radius

Figure 29 shows the spread of blast damage radius against the empirical field factor. The average field factor is 1.60, and the average radius is 0.287 m.

The points form a band that slopes up a little. This shows a Pearson correlation of about +0.25. This connection is less powerful than the effect of UCS, which measures - 0.78. It is also less powerful than the effect of borehole radius, which measures +0.48. It is a bit more powerful than the effect of VOD plus density, both of which measure +0.16.

About 26.7 percent of the tests fall in the area with a high factor and a high radius. In contrast, 18.6 percent of the tests fall in the area with a high factor also a low radius. About 31.4 percent of the tests fall in the area with a low factor and a low radius. The remaining 23.3 percent fall in the area with a low factor as well as a high radius.

This confirms that setting the empirical field factor does change the predictions of damage radius. But its effect stays less important than getting good results from rock strength tests and measuring the hole size exactly.

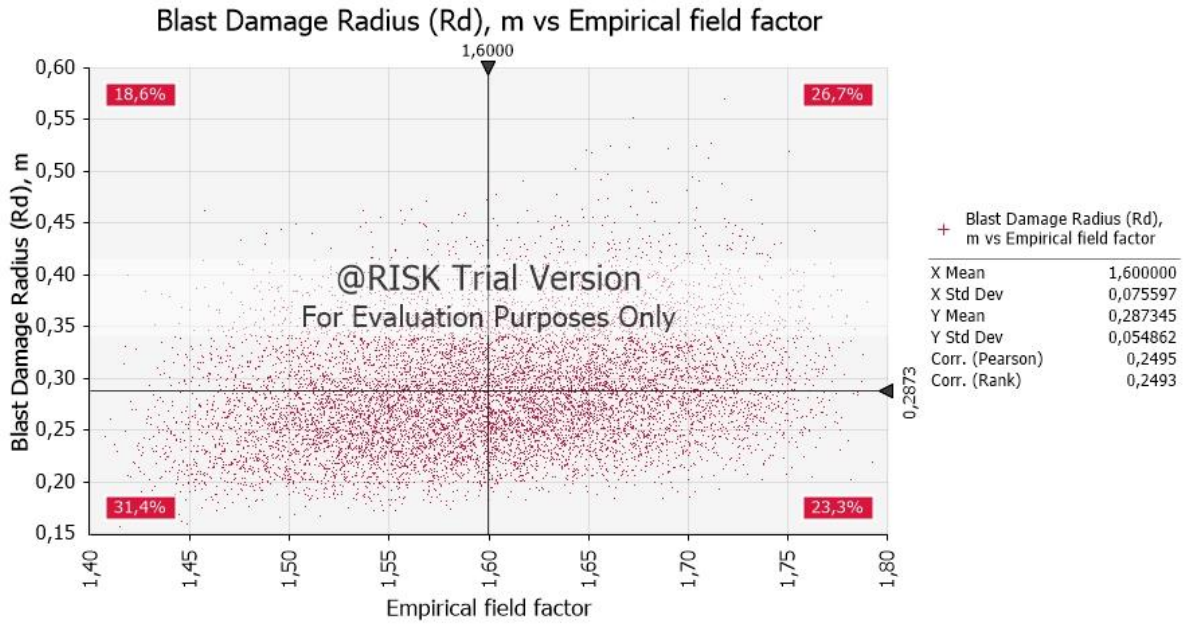

Figure 29. Scatter plot of Rd vs Field factor

5. Discussion

5.1. Validation of the Hypothesis

The initial blasting pattern in the mine produced irregular face advances, about 3.3 meters per blast, and unwanted overbreak in the roof rock layers. The optimized plan achieved a more even entry advance of about 4.0 meters per blast. It was achieved by moving explosive charges, so they reduced the charge in layers that broke too much, such as the weaker F3 roof layer, plus put more explosives in harder or less broken layers, like F1, D along with B (Sabanov et al., 2023). This outcome shows that moving the charge improves the balance between breaking rock well and keeping mine openings safe also stable. Spreading the explosive load more evenly, and away from sensitive roof areas, caused better breaking of tough rock - it also lessened damage to important support structures.

The general improvement, which involved greater advance with less damage, matches field results by Sabanov et al. (2023). But the rock conditions in this study were more complex. The current simulation-based plan needed to consider several rock layers with different strengths as well as stiffness. The earlier field trials had a more uniform setting. A smaller specific charge, reduced from about 0.85 to about 0.70 kilograms per cubic meter, still resulted in better advance and stability under these layered conditions. This shows that careful charge placement improves blasting results even in difficult rock. Even with the mine's complex layers of rock, adjusting the blast pattern as predicted led to higher output, meaning longer rounds, plus greater safety, which means less overbreak and damage.

5.2. PPV impact (Persson-Holmberg model)

The peak particle velocity, or PPV, shows how much blast vibrations damage rock. The Holmberg-Persson model was used (Holmberg & Persson, 1980) to figure out the near field PPV throughout the rock. This model connects the way explosive charges are set up to the peak vibration speed at a certain spot, assuming the rock responds elastically.

Scientists consider several assumptions when they predict peak particle velocity close to a blast. Persson, Holmberg and Lee (1994) showed that their model's predictions of the damage zone were accurate, they did this by comparing the number of crack frequency before and after a blast. They used either core logging or a borehole camera.

The Persson-Holmberg model predicts the PPV at any place in the rock. The model needs certain information - it needs the point's location compared to the explosive. It also needs the linear charge concentration, "l" (kg/m), which depends on the type of explosive and the hole's geometry. The model also uses the rock's attenuation characteristics, which specific constants K and a define for each location. Table 5 shows some of calculated K and a values for other rock conditions.

Table 5. Characteristics for diffenent rock mass conditions

Rock type	K	a	PPV critical, mm/s	PPV breakage, mm/s
Massive granite (Persson et al. 1994)	700	0.7	1000	>4000
Andesite (McKenzie et al. 1995)	200	0.9	600	>2400
Strong sandstone (McKenzie et al. 1995)	400	0.78	450	>1800
Strong shale (McKenzie et al. 1995)	175	1.25	350	>1400

In hard, solid rocks, like granite or strong shale, studies show a clear link between PPV levels and rock damage. A higher PPV usually means more damage, so the model offers useful damage estimates. For example, a PPV of about 350 mm/s often represents a point below which damage is small. A very high PPV, close to 1400 mm/s, points to severe cracks or rock crushing.

The near field PPV simulation of the blast showed that most of the rock shook at lower speeds. Blue zones had PPV values of about 350 - 700 mm/s, which meant low vibration that caused little structural harm. Green zones, about 700 - 1050 mm/s, and yellow zones, about 1050 - 1400 mm/s, sat around the blast holes - these colors showed moderate to large vibration levels. These middle zones likely help break the rock well, especially the green zones, which often break rock with good results, but they could also cause some cracks. Only a small center area near the charges showed red zones where PPV went above 1400 mm/s - this meant the highest

stress and actual rock crushing near the blast holes. Red zones only appeared right next to the explosive charges. That meant the changed pattern successfully kept extreme vibration, and thus damage, away from the roof and pillars; this fits with the goal of guarding important parts of the mine. Most of the rock around the mine saw PPV below the damage level of 350 mm/s or in a manageable range. This confirmed that the design could limit harmful vibrations close by.

The Holmberg-Persson model has limits, especially in different rock conditions. The model worked well for our situation of strong shale, using site specific settings $K = 175$ and $\alpha = 1.25$, as McKenzie et al. (1995) suggested for oil shale. In such conditions, strain and stress waves mainly cause rock to break, so PPV reliably indicates damage. But in rock with more joints or weaker rock, explosive gases play a larger role in damage by pressing into joints and open spaces. The Holmberg-Persson approach, based on PPV, does not directly account for gas effects, which means that in softer ground, even with moderate PPV, gas pressure and longer blast times could still cause much damage.

How quickly the blast happens and how long it lasts become important. A slower blast or delays that let gases escape could reduce damage, while a fast blast that traps gas could increase it. Our talk about the model means that one must view PPV estimates with care; they help estimate damage in hard rock, where vibration causes most breakage. But in softer or very fractured rock, one must think about other things, such as gas induced damage and longer vibration, not just the peak speed.

5.3. Blast Damage Radius and the Kleine Field

One of the main ideas in this research deals with the blast damage radius (Rd). This radius shows the distance from a blasthole where the blast significantly harms or cracks the rock. This radius helps in designing blasts in underground openings - it guides how near charges can go to important boundaries, such as roofs or pillars, without breaking them.

For our analysis, we used the idea of the Kleine Field (Kleine et al., 1993) to better understand and show Rd. This approach looks at each small piece of an explosive charge separately when it figures out energy release and damage. It does not treat the entire blasthole as one consistent energy source. When put to use, if a blasthole's explosive column gets split into many small

pieces, or if deck charging occurs, each piece adds to the total stress and shaking in the rock nearby.

This divided view is important - it captures how uneven charge placements, for example, decks with air gaps or different charge amounts along the hole, affect rock breakage past the immediate crushed space around the borehole. By using the Kleine Field idea, the model can display "energy contours" or high energy spots around each charge piece - this view proved helpful for finding where these high energy "red zones" might touch the mine's roof or walls.

Instead of a blasthole making a regular circle of damage, the Kleine Field analysis showed that certain parts of the charge could send focused energy to the side or up. For instance, a charge piece located near the roof might send too much energy into the roof if not considered, enlarging the damage radius upward. Our damage radius formula, which came from the Kleine Field theory, helped set rules for placing charges. We made sure that no piece of explosive came so close to a free surface that its damage radius would reach into that surface. That way, the design keeps a space between any high energy spot and the roof or pillar edge.

Using this detailed approach helped improve the blast pattern. It confirmed that spreading out charges and not putting many charges near sensitive edges would keep damage where it belonged, in the rock to be broken, and not where it could put stability at risk.

5.4. Correlation Between Simulation and Field Measurements

The main challenge in this thesis was comparing the outcomes of the simulations to field data. The modified blast pattern was not subjected to full-scale field testing for this investigation. The obtained results were verified indirectly by comparing them to previously collected field data. The simulation findings were checked against the general patterns Sabanov et al. (2023) described, which were tested by similar blast methods in the field. In that published field study, moving the explosive charges (especially putting less near the roof) and changing the timing improved things. The face advanced more, and less rock broke from the roof. We saw these same signs in our simulations.

For example, the computer model showed specific spots, like the F3 roof layer in the mine, where many high energy zones from different charges could harm the roof. When we put less charge in these risky layers in the simulation, the model showed much less damage in the roof. Sabanov et al. (2023) reported that in their tests, when they used less explosive in the top layer

and changed the delay times in the hole, roof overbreak dropped a lot. This kind of match tells us our simulation captured how things really work. In short, we did not gather new field measurements ourselves. The way the model's predictions match what past field tests showed makes the simulation's results believable. It can be noted that the redesigned blast pattern was simulated correctly. The simulation took real world uncertainties into account, and its predicted benefits match what others have seen in practice.

To make the simulation as real as possible, the stochastic methods, like Monte Carlo simulations were used that helped to copy the variety and unsure parts of actual blasting. For instance, in real life situations, delay times, drilling exactness along with rock qualities always vary a bit. By running 10,000 Monte Carlo simulation trials, it was accounted for these changes using statistics. The result was that in most simulated cases, each blasthole in the best pattern still fired under good conditions, which means it had a clear face in front of it, even when random timing changes happened. The simulation's output visually showed "success" holes, often colored green in the JKSimBlast software, when a hole fired into an adequately free face. Seeing many green labeled blastholes across the random tests suggested that the new design works well even with typical timing changes or drilling mistakes. That gives more belief that the design would work reliably in the field.

It is important to note that these results are only from the simulation. They fit with ideas proven in the field and show clear improvements on the computer model, but new field data from this specific mine do not yet confirm them. Therefore, the discussed match is a comparison of the simulation against past field results, rather than direct checking at the mine site. A next logical step would be to do controlled field blasts in the mine with the best pattern. Then it could be measured with actual outcomes, such as how much rock breaks, how much the face advances, vibration as well as damage; this would directly check the simulation's predictions. At this point, it can be concluded that the simulations are very promising and fit with known good practices, but we will not say they are truly successful in the real world until field tests are done.

5.5. Enhanced blasting design efficiency

The improved blasting outcomes connect with what other underground mining studies found, but differences appear in how they approach problems and what results they get. Zhang et al. (2021) showed a similar rise in blast usefulness in a deep copper mine; they used the rock's

stress from being deep underground. By adding a stress correction to how they planned the charge, they cut the specific charge by around 19.8 % and lowered the large rock piece rate from 6.8 % to 2.84 %. Their method kept the wall strong - the blasting edge stayed protected even with less explosive. Their results show a similar specific charge cut, about 18 % lower, from 0.85 to 0.70 kg/m³. This also made the rock break into smaller pieces, with no large blocks along with caused little harm to pillars or roofs. The method is the difference - Zhang et al. used the natural high stress deep underground to break rock better, while we got usefulness by placing the charge and timing the blast well. Both ways show that you can save a lot of explosive and break rock better with good plans. One good thing about Zhang et al.'s plan is that it makes a deep mine's high stress helpful - it uses the rock's wish to break under stress to need less explosive. That method fits only certain situations, working best in deep, high stress places. The design method in this study does not need strange ground conditions - you can change it for different underground mines by looking at weak spots at the site and changing the blast pattern.

Other researchers tried different ways to improve advance rate and reduce damage. Kim et al. (2022) used a long, large hole cut blasting method in a hard rock tunnel, which brought big benefits. They drilled several large empty holes in the tunnel face to act as relief areas - they moved forward about 3.75 times more per round than with a normal burn cut, while using 30 % less explosive. This method made more free faces, which allowed for easier breaking. As a result, the zone of blast damage was about 1.25 times smaller than with regular cuts. The good thing here is a big gain in how far they moved forward, almost four times as well as less shaking and harm because of the lower explosive amount. In comparison, improvement in our case was about a 21% rise in advance rate, to 4.0 m, with around 18 % less explosive, it is smaller but came without special drilling of large empty holes. The LLB method needs extra drilling machines and time to make those relief holes. Our plan worked within the holes they already had by moving charges and changing when they went off. A main good point of our way is how practical it is - mines can do it by changing blast plans and using common machines. But the LLB method's bad point is the extra work involved, drilling big empty holes, even if it works better. We checked the plan with statistical tests, 10,000 Monte Carlo runs, to make sure it stood up to timing problems, which traditional methods often do not address.

It is also good to compare how different aims (fragmentation vs. vibration vs. profile control) change blast designs in underground work. Zhou et al. (2024), for example, improved a tunnel blasting plan mainly to reduce vibration and control the tunnel's shape. They used exact

electronic delay detonators and added empty holes around the edges; they cut the fastest particle speed by about 20 % at nearby structures, from 12.3 cm/s to 9.9 cm/s, while keeping the tunnel shape from breaking too much - this differs from our study, where controlling shaking was important but not the main goal, which was to move forward as much as possible and keep things stable. Zhou's work shows a possible problem if you only try to cut blast energy - using low vibration methods, such as explosives that go off slower or spreading charges out a lot, can hurt how well rock breaks. In fact, a tunnel study found that an explosive with about 60 % of the speed of a normal emulsion did cut shaking, but it broke rock worse, leading to poor breaking if the blast plan was not set right. In our case, we kept a high energy emulsion explosive for good breaking and instead controlled shaking and harm by smart charge placing, for example, putting less charge in the roof holes. This way, the pitfall of insufficient breakage was avoided while still keeping vibrations and overbreak in check. The advantage of our balanced approach is evident: we did not have to trade excavation efficiency for stability – both were achieved by fine-tuning the blast parameters.

5.6. Sensitivity analysis and Stochastic evaluation of Rd and SC

A significant part of this research involved a stochastic sensitivity study for evaluation of how uncertainty in drilling and blasting parameters affects two performance measures - the Blast damage radius (Rd) and the Specific charge (SC) needed to get the desired fragmentation. Ten thousand Monte Carlo simulations in @RISK changed inputs like rock strength plus explosive properties. This produced a range of possible Rd values, not just one fixed number.

The simulated damage radius followed a log normal distribution. It had a mean of about 0.287 m ($\sigma = 0.055$ m), which shows average variability around the center value. A PERT distribution was also checked, but it showed a slightly higher mean ($\approx$0.291 m) and fatter tails - this suggested it might guess too high for extreme Rd events. The log normal fit, which peaked near 0.27 m, better showed that very large damage radii were rare, though they could happen. For practical purposes, most blasts should cause a damage radius near 0.29 m. But design plans must account for low chance situations where Rd approaches 0.5 m. Conservative standoff distances and buffer zones near important structures are advised. Even a 5 % chance of going over 0.38 m (the 95th percentile) could risk safety if the pattern is too close.

The same Monte Carlo setup gave distributions for SC in roof, floor along with wall holes. That showed a clear link to hole direction also confinement. Roof holes showed SC values

around 2.02 g/dm³ ($\sigma = 0.28$). This reflects how a free roof face and gravity help cracks spread. Floor holes needed a bit more explosive - 2.12 g/dm^3 ($\sigma = 0.30$) - because of added ground resistance, even with the eventual free face of the floor. Wall holes, which had the most confinement, showed the highest SC needs (mean = 2.49 g/dm^3, $\sigma = 0.35$) and the most variability, ranging from about 1.2 to 3.5 g/dm^3. Over half of the wall hole situations needed an SC higher than the 95th percentile of roof hole needs. This confirmed that people often load perimeter holes more heavily to get uniform damage radii across the excavation.

These findings support a charging plan that depends on direction. SC changes based on hole orientation, rather than using one amount of powder for everything. For example, roof holes could take between 1.5 - 2.5 g/dm^3 (covering about 90 % of cases). Floor holes would take slightly more, as well as wall holes often need 2.8 - 3.0 g/dm^3 or more to keep an average Rd near 0.30 m. This careful adjustment stops both undercharging in very confined areas and overcharging where there is plenty of room. It gives a more even damage radius around the excavation.

Regression based sensitivity analysis (tornado charts and spider diagrams) found how much each input parameter affects Rd in addition to SC. The rock's uniaxial compressive strength (UCS) came out as the most important factor (standardized $\beta \approx -0.78$ for Rd plus +0.79 for SC). This shows that stronger rock greatly lessens damage radius but demands more explosive energy. Borehole radius followed in importance ($\beta \approx +0.48$ for Rd and -0.49 for SC). This reflects how a larger hole volume affects both the damage amount and energy use. The empirical field factor had a small influence ($\beta \approx +0.25$). But changes in velocity of detonation also explosive density had relatively small effects ($\beta < \pm 0.20$). The high coefficients of determination ($R^2 = 0.82$ - 0.90) showed that these parameters together explain most of the variability in the results.

6. CONCLUSIONS AND RECOMMENDATIONS

This research closely aligns with the findings of Sabanov et al. (2023) and builds upon them, it extends their validated blast design approach with advanced modeling and analysis. In Sabanov's case study, redistributing explosive charges away from the immediate roof improved outcomes. The average face advance grew from 3.3 to 4.0 meters, plus the specific charge fell from 0.85 to 0.70 kg/m³. The thesis adopted these design changes, for example, moving explosive decks lower in the rock mass, as a foundation. It expanded upon them. By using simulation driven methods, the study not only supported the prior field results but also adapted the blasting strategy to more complex geomechanical conditions than those originally examined.

One of the contributions of this work is the use of blast simulations with JKSimBlast. It helps visualize and quantify explosive energy distribution in the rock. The Kleine Field mapping concept served to break each charge into small segments, it computed energy spread in three dimensions, which offered a detailed picture of how the revised pattern disperses energy. This allowed the author to confirm that peak energy zones stayed away from critical supports - this insight matched the supervisor's findings but now had in silico verification. The simulations confirmed that shifting explosive mass out of the roof vicinity prevents high energy flux from intersecting the roof or pillar walls that lessened overbreak and damage risks. The thesis translated the empirical design improvements of Sabanov et al. into a modeled energy field framework, yielding a deeper understanding of why the new blast pattern supports both advance rates also stability.

The integration of Monte Carlo simulations accounts for timing uncertainties and rock property variability when predicting blast outcomes. By stochastically simulating thousands of blast iterations, the study measured the reliability and range of expected performance. The analysis showed that each blasthole in the new design detonates with proper burden relief, which means a free face was available, validating the timing sequence under uncertainty. The blast damage radius emerged as a log normally distributed variable (mean ≈ 0.29 m) rather than a fixed value; this highlighted the inherent variability in how far fractures extend. The thesis characterized the probability distributions of required specific charge for different portions of the round. For example, it found that wall perimeter holes demand a higher specific charge (~2.5 g/dm^3) than roof holes (~2.0 g/dm³). These probabilistic results, not considered in the earlier case study,

provide engineers with confidence bounds. This allows the design to be fine-tuned with safety factors so that even in worst case scenarios the blast remains effective yet within damage limits. The Monte Carlo approach extended the work by adding a statistical validation of the blast design's performance.

The thesis also performed a comprehensive sensitivity analysis. It identified the most influential parameters on blast efficacy, which was another contribution beyond the prior field study. Using regression-based tornado charts as well as correlation diagnostics, it was determined that the rock's UCS is the dominant factor governing both the blast damage radius and the specific charge needed. Even within the optimized pattern, variations in UCS produced the largest swings in outcomes, on the order of ±0.15 m in damage radius. Borehole diameter was the second most critical parameter, whereas explosive properties like density and velocity of detonation, plus the empirical field factor had minor effects - these findings direct attention to geological characterization and drilling precision. The improved blasting efficiency documented can be reliably reproduced only if the rock strength and hole parameters are well controlled. By quantifying each factor's impact, the research offers clear guidance on where future efforts or safety margins should be applied when implementing the blast design in practice - this level of analytical insight was not present in the supervisor's original case study - it marks a distinct methodological advancement by the author.

Leveraging the above tools also insights, this thesis designed and evaluated an improved blast pattern. It tailored this pattern to the mine's complex geology and stress conditions. The refined design retains the improvements from Sabanov et al., for example, avoiding heavy roof charging as well as using appropriate delay sequencing. But it further adapts them to the stratified oil shale - limestone formation and the room-and-pillar layout. Through energy modeling and iterative adjustments, the author optimized charge distribution for each layer. That ensured that even in stronger rock bands or more confined areas, fragmentation is sufficient without compromising nearby pillars or the roof. Simulation results demonstrated that this optimized pattern achieves uniform breakage plus stable excavation profiles under a range of geomechanical scenarios; this extended the prior field validated results to more challenging conditions. The improvements first proven by Sabanov et al. were not only confirmed but were generalized to hold under variability in rock strength, layer thickness along with other in situ complexities. This adaptation shows the strength of the blasting strategy. The thesis showed that with careful modeling, a blast design can be fine-tuned to maintain

performance across different parts of the mine that were beyond the scope of the initial case study.

Based on the study's findings, the following recommendations is given:

- Put simulation tools like JKSimBlast into regular blasting plans. This helps assess blast designs with precision and improve them over time.
- Validate and update the empirical field factors and explosive properties regularly. Ongoing field data collection also analysis assist in this process. Accuracy of models and their ability to predict outcomes get better.
- Set up comprehensive monitoring systems, they assess the condition of roofs as well as pillars right after blasting, this allows for early detection and reduction of structural risks.
- Conduct more research that would investigate the larger effects of better blasting practices on mine finances, safety rules along with environmental well-being. That informs long term choices about strategy.

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