DOI: https://doi.org/10.68732/f4306585770e6e9f

ISBN: 978-9989-2053-7-8

Design of a Controlled Blasting Operation for the Removal of an Unstable Rock Mass from a Rock Slope

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Abstract

Explosives remain a highly effective technique for rock-face remediation (slope clearing and stabilisation) when safety, geometric flexibility, and certainty of detachment are critical. Explosive techniques allow remote or rope-assisted operations where direct access is unsafe, typically requiring lower specific explosive consumption than production blasting due to variable free-surface conditions. The design approach emphasises precise drill-hole positioning, static criteria for sizing detachment charge, and controlled confinement to achieve clean, damage-free niches that prevent subsequent instability. Simultaneous initiation is often preferred in irregular or non-repetitive layouts to reduce the risk of charge exposure or incomplete detonation. Key safety considerations include ensuring effectiveness on the first attempt, maintaining a minimum safety clearance of 500 m in unfavourable topography, and preventing blast-induced overbreak. This paper presents design principles and operational guidelines for controlled blasting in remediation works, with a focus on confined-charge configurations, timing strategies, and safety optimisation. A case study on rock-slope remediation is presented, in which the proposed controlled blasting methodology was applied. The case shows the method's effectiveness in achieving stability, minimising overbreak, and ensuring operational safety under challenging field conditions. The paper contributes practical recommendations to enhance the reliability and safety of blasting-based remediation in complex rock slopes.

Keywords

Explosive, Controlled blasting, Tensile strength, Shear strength, Static Criterion.

1 Introduction

Stabilisation blasting, often referred to as scaling by blasting, encompasses specialised operations to remove unstable or hazardous portions of a rock slope while preserving the integrity and geometry of the remaining wall. The objective is controlled detachment along existing discontinuities rather than extensive fragmentation. For this purpose, confined charges are typically used, enabling safer remote or rope-assisted operations in areas where direct access is difficult, while requiring lower specific explosive consumption than conventional excavation. The design of detachment charges relies on static analysis criteria (Hoek and Brown 2019; Persson et al. 1994), ensuring a sound final niche free of significant blast-induced damage. Simultaneous initiation is often preferred in irregular patterns to ensure reliable detachment.

Strict compliance with safety requirements is essential (Singh et al. 2016; Langefors and Khilström 1978; Dowding 1985). The blast must be effective on the first attempt, as secondary interventions on unstable slopes are significantly more hazardous, while avoiding excessive charging. Additional considerations include control of fly rock, vibration effects, and energy confinement, particularly where affected areas are located below the blast face or where open fractures may favour uncontrolled propagation (Kutter 2010; Ouchterlony 2005; Cardu et al. 2004). These operations increasingly benefit from numerical modelling and monitoring techniques to improve reliability and control.

2 Case Study Description

The targeted rock mass covers approximately 615 m² with a thickness of 10–14 m, totalling a volume of roughly 6,000 m³. The base is defined by a schistosity plane that serves as the sliding surface; its dip angle varies from 30° to 35° on the north side to 60° to 65° on the south side. As shown in Fig. 1, this basal plane is open in its overhanging section, and the volume is further bounded by three free surfaces and several open fractures.

While techniques like presplitting (Jimeno et al. 1995; Chiappetta 1996; Goris et al. 1994) or smooth blasting were considered to achieve fragmentation, they were deemed unsuitable. Both methods pose significant safety risks because they require drilling in unstable zones. Furthermore, presplitting raised concerns about mid-air fragmentation, whereas smooth blasting would have compromised the resistant surface required for detachment. Since fragmentation is not critical, given a 10,000 m³ capacity drop area at the base, the final design utilises static criteria for a controlled detachment blast, as detailed in the following sections.

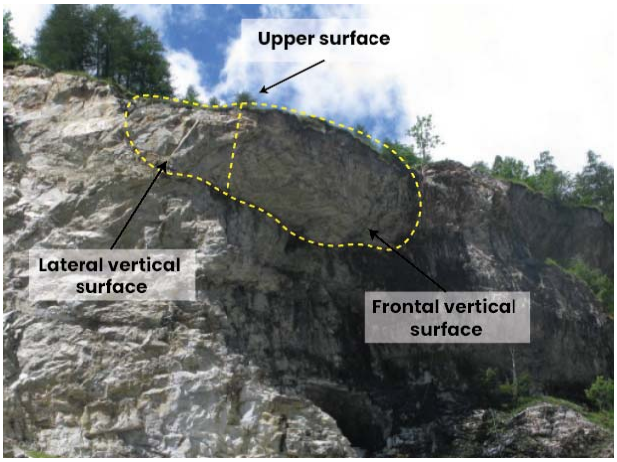


Fig. 1 Side view of the rock spur subject to the intervention. Three distinct free surfaces are visible: the top surface, the lateral surface, and the front face.

3 Blast Design According to the Static Criterion

The explosive action is modelled as a force acting perpendicularly to the blast-hole plane, aimed at detaching the target volume. The rock mass is treated as a prismatic block held by tensile forces along the charged surfaces and shear forces along the remaining boundaries.

Detachment occurs when the force generated by the explosive exceeds the total resisting force (F_r), obtained by summing the tensile contributions of surfaces *S1a*, *S1b*, and *S1c*, and the shear contributions of the basal (*S2*) and lateral (*S3*) surfaces. Each surface is multiplied by its corresponding strength parameter, and all contributions are summed to obtain F_r (Fig. 2).

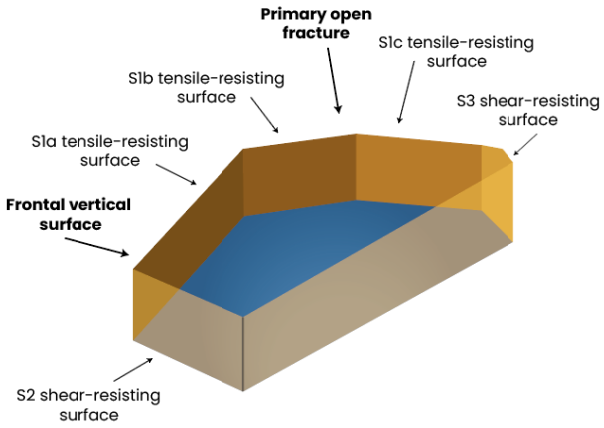


Fig. 2 Three-dimensional schematic of the intervention, showing the surfaces referenced for the design of the blast.

The required active surface (*Sa*) is obtained by dividing the resisting force by the explosion pressure P_e:

$$S_a = \frac{F_r}{P_e} \tag{1}$$

This surface corresponds to the product of the borehole diameter and total charged length (L_c):

$$L_c = \frac{S_a}{\phi} \tag{2}$$

The total explosive quantity (Q) is then calculated from the charged volume and the in-hole density (δ):

$$Q = 0.785 \cdot \phi^2 \cdot \delta \cdot L_c \tag{3}$$

The total hole length (L_t) includes both the charged length and the stemming.
Hole spacing is determined by geometric considerations and is typically set to 15-20 times the hole diameter to ensure regular detachment.

To ensure the safe, strictly simultaneous initiation of all charges and to prevent misfires, the use of detonating cord is mandatory. Rock strength parameters were obtained from laboratory testing, yielding:

UCS (C_0): 140 MPa
Tensile strength (from Brazilian test, σ_t): 9 MPa
Cohesion (c): 17 MPa
Friction angle of intact rock (φ): 63°
Shear strength (τ_R): 60 MPa

The detonation pressure (Pd), acting on the borehole wall, is derived from detonation parameters:

$$P_d = 0.25 \cdot \rho_e \cdot VD^2 \tag{4}$$

where ρ_e is the density of the explosive, expressed in kg/m³, and *VD* is the detonation velocity, expressed in m/s.
Due to imperfect coupling, the effective pressure is assumed to be approximately half of the theoretical detonation pressure:

$$P_e = \frac{P_d}{2} \tag{5}$$

The selected explosive (Table 1) yields an explosion pressure of approximately 3,750 MPa, which is used in the design calculations.

Table 1 Characteristics of the explosive (cartridges)adopted, supplied by Maxam (https://www.maxamcorp.com).

Density [kg/m³]	Specific Energy [kJ/kg]	Detonation Velocity [m/s]	Length [mm]	Diameter [mm]	Mass [kg]
1,200	4,460	5,000	380	60	1.316

4 Blast Implementation Procedure

The design procedure described in the previous section was directly applied to the case study geometry. In particular, the total resisting force (Fr) calculated from the surfaces S1a, S1b, S1c, S2 and S3 was used to determine the required active surface (Sa) through Eq. (1). This value was then translated into total charged length (Lc) using Eq. (2), which directly defined the borehole configuration reported in Tables 2 and 3.

The corresponding explosive quantity (Q), derived from Eq. (3), was subsequently adjusted to discrete cartridge values for field implementation. In this way, all operational parameters (spacing, charge distribution, and hole geometry) are directly derived from the theoretical static criterion.

The blast design was carried out using the procedure previously described. In particular, the calculations yielded a spacing of 1.5 m between boreholes for surface S1a and 1 m for surfaces S1b and S1c.

The data presented in Table 2 refer to the charge calculations for surface S1a, with boreholes inclined at 30° to the vertical, while those in Table 3 refer to the charge calculations for surfaces S1b and S1c, with vertical boreholes.At the junction between surfaces S1a and S1b, an additional uncharged hole was drilled to ensure better fracture guidance at the point where the slope changes between the two surfaces.

A summary table of the total quantities of explosives required to carry out the blast is provided in Table 4.

Table 2 Main data relating to the blast (surface *S1a*).

Parameter	**Unit**	**Symbol**	**Value**
Borehole diameter	[m]	$ø_f$	0.076
Total charged length	[m]	L_c	36.8
Total charge (theoretical)	[kg]	Q	133.5
Total drilled length	[m]	L_t	70
Average spacing	[m]	E	1.5
Number of holes	[-]	N	10
Charge/hole (theoretical)	[kg]	Q_{th}	13.35
Number of cartridges (theoretical)	[-]	N_c	10.15
Number of cartridges (actual)	[-]	N_{ce}	11
Charge per hole (actual)	[kg]	Q_{actf}	14.48
Total charge (actual)	[kg]	Q_{totact}	144.8

Table 3 Main data relating to the blast (Surfaces *S1b* and *S1c*).

Parameter	**Unit**	**Symbol**	**Value**
Borehole diameter	[m]	$ø_f$	0.076
Total charged length	[m]	L_c	51.13
Total charge (Theoretical)	[kg]	Q	186.3
Total drilled length	[m]	L_t	119
Average spacing	[m]	E	1.0
Number of holes	[-]	N	17
Carica/foro (Teorica)	[kg]	Q_{th}	10.96
Number of cartridges (theoretical)	[-]	N_c	8.33
Number of cartridges (actual)	[-]	N_{ce}	9
Charge per hole (actual)	[kg]	Q_{actf}	11.84
Total charge (actual)	[kg]	Q_{totact}	201.3

The initiation system uses an HU-type electric detonator connected to a 15 g/m main detonating cord, which triggers the branch cords (15 g/m) for each blast hole.

As shown in Fig. 3, the cord is tied to the bottom cartridge to ensure full column detonation. Branch connections must be secured with strong adhesive tape in accordance with the detonation direction. Charges are continuous for holes on surface $S_{1a}$and segmented for those on S_{1b} and S_{1c}(Fig. 4). The drilling geometry for the intervention area is detailed in the plan view in Fig. 5.

Table 4 Summary of blast data.

Parameter	Symbol	Unit	Value
Total amount of charge	Q_{tot}	[kg]	346
Total drilled length	L_{tot}	[m]	203
Required detonating cord	L_{dc}	[m]	250
Electric detonator (HU type)	-	[-]	1

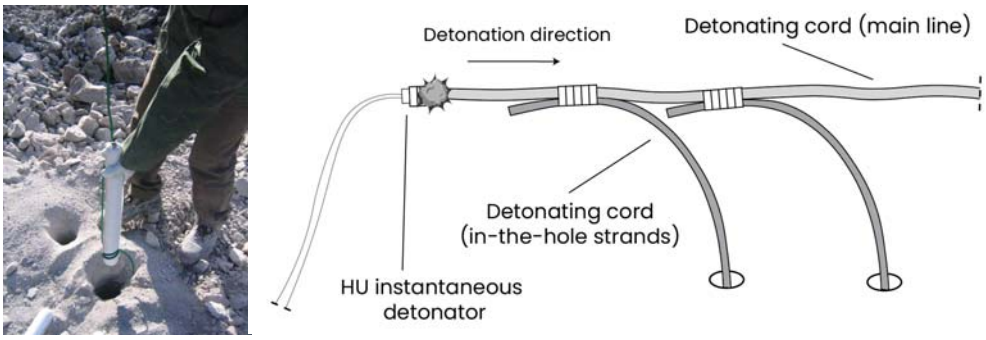


Fig. 3 Left: An example of the connection between the branch detonating cord and the bottom charge cartridge. Right: Example of the connection between the main detonating cord and the branch cord coming out from the blast holes.

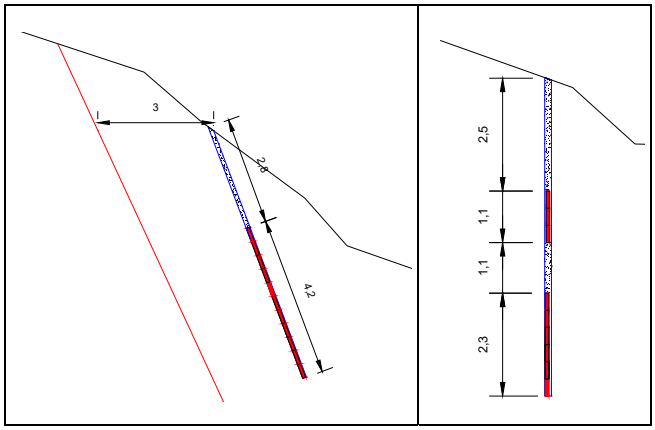


Fig. 4 Left: Schematic longitudinal section of a loaded borehole (continuous charge consisting of 11 emulsion cartridges). The diagram shows the drilling and charging configuration to be used for surface S1a. Right: Schematic longitudinal section of a loaded borehole (segmented charge consisting of 6 emulsion cartridges at the toe and 3 cartridges in the column, separated by an intermediate stemming of 1.1 m). The diagram shows the drilling and charging configuration to be used for surfaces S1b and S1c.

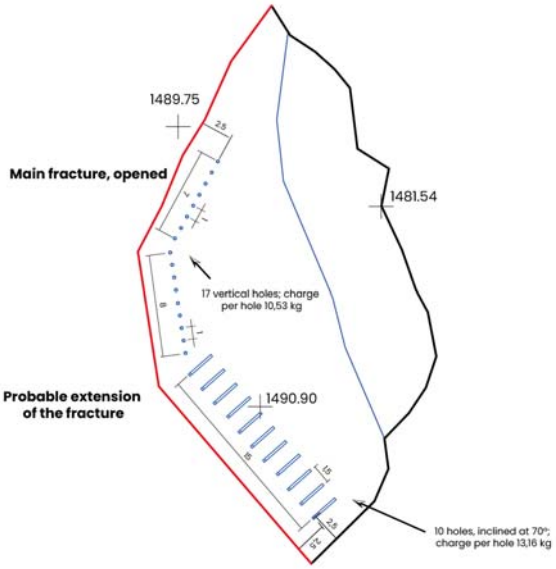


Fig. 5 Plan view of the work area showing the layout of the blast holes.

5 Results and Discussion

The geomechanical response of the site was controlled by the schistosity plane, which served as the primary sliding surface and defined the detachment geometry. Despite the high intact rock strength (UCS ≈ 140 MPa; tensile strength ≈ 9 MPa), stability was governed by discontinuity configuration, resulting in a kinematically constrained block prone to planar or combined failure. The application of the static design criterion proved appropriate, as detachment required overcoming tensile and shear resistance along predefined surfaces. Numerical comparisons showed that the calculated blast forces provided an adequate safety margin, ensuring that fracture propagation remained confined to the targeted planes without inducing overbreak. Simultaneous initiation promoted uniform stress release and controlled detachment. Post-blast observations confirmed a clean release along the schistosity surface and the integrity of the remaining rock mass, validating the effectiveness of the methodology under structurally anisotropic conditions.

Optimised charge distribution and low specific explosive consumption resulted in reduced vibration levels, air overpressure, and fly rock, allowing safe blasting under geotechnically sensitive and constrained conditions. The controlled confinement of explosive energy ensured that its effects remained limited to the target volume. The initiation system, based on an HU-type electric detonator and a detonating cord network, ensured reliable and complete detonation, eliminating the risk of misfires and enabling successful execution on the first attempt. Post-blast surveys confirmed that the fragmented material reached the designated containment area and required minimal secondary breakage. Monitoring and visual inspections indicated no damage to adjacent structures or stable rock portions, while the residual face maintained its structural integrity.

6 Conclusions

The study demonstrates that controlled blasting based on static criteria is an effective and safe solution for the remediation and stabilisation of unstable rock slopes. A rigorous geomechanical interpretation, combined with precise drilling and charge confinement, enables reliable detachment of rock masses along predefined structural surfaces with limited environmental impact.

Field observations confirmed the absence of unintended cracking or overbreak, indicating that the explosive energy remained confined within the designed detachment limits. The close agreement between predicted and observed behaviour validates the adopted design approach and confirms the robustness of static-force-based modelling.

From a practical perspective, the method provides a reproducible and adaptable solution for similar stabilisation scenarios, ensuring operational safety, efficient use of explosives, and controlled environmental effects.Future developments should focus on integrating the deterministic design approach with real-time monitoring and numerical modelling tools to improve the prediction of stress distribution and vibration effects, further enhancing the reliability of blasting-based remediation.

References

Cardu, M., Oggeri, C., Peila, D., Casale, M. (2004) Bonifiche con esplosivo. Atti Convegno su Bonifica dei versanti rocciosi per la protezione del territorio, Trento. ISBN 88-901342-0-8, pp. 251-277.

Chiappetta RF (1996)Blast Vibration and Airblast Control. Explosives Engineering Handbook.

Dowding CH (1985) Blast Vibration Monitoring and Control. Prentice Hall.

Goris JM et al. (1994)Controlled Blasting Techniques: A Review of Practice and Design. International Journal of Rock Mechanics and Mining Sciences, 31(4), 349–361.

Hoek E, Brown ET (2019) The Hoek–Brown Failure Criterion and GSI – 2018 Edition. Journal of Rock Mechanics and Geotechnical Engineering, 11(3), 445–463.

Jimeno CL, Jimeno EL, Carcedo FJA (1995) Drilling and Blasting of Rocks. CRC Press.

Kutter HK (2010)Controlled Blasting for Slope Stabilization. Proc. EUROCK Conference, ISRM.

Langefors U, Kihlström B (1978) The Modern Technique of Rock Blasting. Wiley.

Ouchterlony F (2005)The Sizing and Fragmentation of Blasted Rock. International Journal of Rock Mechanics and Mining Sciences, 42(6), 965–978.

Persson PA, Holmberg R, Lee J (1994) Rock Blasting and Explosives Engineering. CRC Press.

Singh PK, Xavier P (2016). Rock Blasting Techniques for Ground Control. Taylor & Francis.