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Finn Ouchterlony
Yoneji Kanoh²

Hideo Minamide¹
Koji Nakagawa³

1: Sato Kogyo Co Ltd, Osaka, Japan

2: Sato Kogyo Co Ltd, Tokyo, Japan

3: Dept of Civil Engineering, Yamaguchi University, Ube, Japan

FÖRORD

SveBeFo:s forskningsprogram spänner över bergbyggandets olika stadier, från undermarksplanering och förundersökningar till bergguttag, förstärkning, tätning och långtidsbeständighet hos anläggningar i drift. Forskningen bedrivs till stor del med externa forskare vid högskolor och företag, medan specialområdet sprängningsteknik och sprängämnesforskning huvudsakligen genomförs med egna forskare, anställda vid SveBeFo.

Inom sprängtekniken är en huvuduppgift att åstadkomma effektivare ort- och tunnel-drivning. Tyngdpunkten har legat på att utveckla såväl öppnings- som kontursprängningsteknik för långa salvor, upp till 7,5 m, och under programperioden 1993-96 särskilt på att använda ett förborrat grovhål (Ø 250-300 mm) som öppning i kilen.

Svenskt berg tål oftast långa salvor utan att det rasar innan man hinner förstärka det. I Japan drivs ca 250 km bergtunnel per år, varav en mindre del med borrar-sprängning. Berget är ofta relativt svagt varför salvdjupen normalt begränsas till högst 3 m. Då används plogkilar och andra öppningar med vinklade hål.

Det finns även i Japan en strävan att öka på salvlängderna där förhållandena tillåter. Arbetet i denna rapport hade som mål att ge ett underlag för detta med hjälp av svensk erfarenhet och teknik, dvs användning av parallella hål, fyrsektionskil, prillad ANFO och Nonelupptändning. Det utfördes under 1992 när undertecknad var professor vid institutionen för väg- och vattenbyggnad vid Yamaguchiuniversitet i Ube i västra Japan. Entreprenadföretaget Sato Kogyo har under en följd av år prövat "svensk teknik". Därför kom de att stå som huvudansvariga för projektet.

Undertecknad var med om att lägga upp försöksplanen för projektet, deltog i projektmötena och har sedan utfört huvuddelen av analysen, i gott samarbete med professor Koji Nakagawa samt Hideo Minamide och Yoneji Kanoh från Sato Kogyo.

Det som samarbetet gett, och som denna rapport kan bidra till att ge är;

- kontakter med japanska forskare och sprängtekniker,
- erfarenheter från sprängning i svagare utländskt berg,
- tillgång till stora experimentella resurser och
- en bekräftelse på att Langefors och Kihlströms dimensioneringsformel gäller även för andra bergförhållanden än under vilka den togs fram.

Jag anser att ett fortsatt forskningsutbyte bör ha mycket att ge.

Stockholm i januari 1997

Finn Ouchterlony

SAMMANFATTNING

Sprängförsök med parallellhålskilar gjordes 1992 i Buko kalkbrott i Japan i ett samarbete mellan entreprenadföretaget Sato Kogyo Co och Yamaguchiuniversitet i Ube. Målen var att bekräfta tidigare japanska erfarenheter från användning av emulsionssprängämnen i parallellhålskilar, att fastställa sambandet mellan erforderlig laddning och försättning och att erhålla basdata för användning av ANFO i 4 m salvor.

Totalt sköts 49 salvor, de flesta enbart öppningsdelar med 2 spränghål (Ø 43-48 mm). Här redovisas 32 st sådana öppningsdelar av fyrsektionskilar med 2 grova tomhål (oftast Ø 102 mm). Två sprängämnen användes, prillad ANFO och emulsionssprängämne i form av rörladdningar (Ø 25 och 30 mm). Flera andra faktorer varierades i försöken, spränghålens försättning, tomhålens avstånd, tändartyp och tändfördröjning.

Försöken följdes noga. Sprängresultatet klassades främst som god, partiell eller dålig indrift. Med god indrift förstås över 90% av borrar djup och att minst ett av spränghålen brutit ut mot båda grovhålen. De huvudsakliga slutsatserna från resultatanalysen är:

- Borrhålsavvikelserna var större än vad som antas i beräkningsunderlagen. Spridningen i påhugg och hålrikt var ungefär 5 cm resp 2-2,5 cm/m. Totalfelet i borrhålsbotten var ofta 20-30 cm vilket är mycket jämfört med de försättningar som använts, 20-40 cm.
- Ett kvalitativt mått på borrhålskvaliteten (DQR) har utarbetats med 5 klasser från mycket dålig till mycket bra. Det bygger främst på kilens form och storlek. Vi fann att en borrhålskvalité som är dålig eller bättre inte påverkar sprängningsresultaten. En mycket dålig borrhålskvalité kan påverka dem, men gör det inte nödvändigtvis.
- En fyrsektionskil har redundant brytkapacitet, dvs att endast det ena av de två första spränghålen måste bryta ut mot båda grovhålen för att indriften i salvan skall bli god.
- Två slags felfunktion hos salvorna observerades vilka påverkar sprängresultaten; detonationsavbrott eller nedsatt initieringsfunktion och bösskott. Bösskott inträffar när en laddning inte bryter ut sin försättning. Ibland orsakades detta av avsiktligt för stora försättningar, ibland av stora borrhålsavvikelser.
- Detonationsavbrott med dålig indrift som resultat inträffade av följande skäl; a) icke detonerande sprängkapslar, b) felplacerad sprängkapsel, c) för mycket eller för lite ANFO i spränghålen och d) ett spränghål med ANFO som blåste ut mot ett tomhål i förtid. Utblåsning sker när botten på spränghålet hamnar för nära tomhålet. Detta

skedde också för ett emulsionsladdat hål men då utan detonationsavbrott, troligen därför att emulsionen är mindre beroende av mothåll för att detonera än ANFO är.

- En tändfördröjning mellan de två spränghålen på 75-100 ms bedöms vara tillräcklig för att få god indrift och utblåsning av krossat berg. 50 ms kan räcka men ej 25 ms.
- Data för de kilar som ej haft funktionsfel eller mycket dålig borraringskvalité har plottats i ett konstruktionsdiagram med erforderlig laddningskoncentration som funktion av försättningen för det först initierade spränghålet. Analysen visar att:
 - a) Langefors och Kihlströms försättningsgränsvärde för bränning av berget, som i Bukoförsöken är ca 30 cm, kan överskridas utan att indriften blir dålig.
 - b) För tomhålsavståndet finns en gräns på ca 30 cm utanför vilken tomhålen inte samverkar i sönderbrytningsprocessen.
 - c) Om Langefors och Kihlströms ursprungliga beräkningsformel för erforderlig laddningskoncentration kompletteras med bergkonstanten och viktstyrkan så ger den tillfredsställande överensstämmelse med resultaten från Bukoförsöken.
 - d) Skillnaden mellan sprängämnenas brytförmåga förklaras tillräckligt väl av skillnaden i laddningskoncentration, utan att kopplingsgraden beaktas.
- Storleken på det sönderbrutna berget som blivit kvar i eller nära öppningen har ett starkt samband med dess färg. En svart finandel hade den största medelstorleken, ca 20 mm, en gråvit finandel den minsta, ca 3,3 mm. En mörkare färg kan indikera en icke-ideal detonation med syrebrist, nedsatt detonationshastighet och lägre energiomsättning. Sambandet mellan stenstorlek och laddningskoncentration är emellertid för svagt för att vara signifikant, jämfört med spridningen i mätdata.
- Den maximala kastlängden för stenstycken uppmättes till 8-55 m och kastet från ANFO-kilarna var i allmänhet störst. Mynningshastigheten vid kilöppningen uppmättes till 38-65 m/s. En enkel ballistisk uppskattning med en största kilhöjd över syltan på 3,5 m ger ett maximalt kast på 32-55 m vilket stämmer väl med försöksdata.

Projektmålen har nåtts; basdata för användning av ANFO i 4 m salvor med fyrsektionskil har tagits fram och en kompletterad beräkningsformel från Langefors och Kihlström har visat sig användbar både för ANFO och frikopplade emulsionsladdningar.

Försökserfarenheterna har sammanställts i en lista över vad som krävs för att erhålla god indrift i fyrsektionskilen. För att erhålla ett bättre konstruktionsunderlag för parallellhålskilar skulle inverkan av följande faktorer behöva undersökas bättre; främst samverkan mellan flera tomhål och frikopplingens betydelse samt i någon mån tändfördröjningen och felbörningen.

SUMMARY

Cut blasting tests were performed in the Buko limestone mine in Japan in 1992 by the Sato Kogyo construction company and the department of civil engineering of the Yamaguchi University. The goals were to confirm earlier experience in blasting parallel hole cuts with emulsion explosives, to obtain the relation between charge concentration and burden and to obtain basic data for the use of ANFO in rounds with 4 m advance.

A total of 49 blasts were fired, mostly opening parts of four-section cuts with two blast-holes ($\varnothing$ 43-48 mm) and 2 relief holes (mostly $\varnothing$ 102 mm). 32 such openings are reported here. Two explosives were tested, bulk ANFO and a decoupled cartridge emulsion. Several other factors were changed as well, the burdens of the blast-holes, the spacing between the relief holes, the type of detonator and the initiation delay between the blast-holes. The blasts were closely monitored.

The blasting results were judged primarily in terms of good, partial or bad breakage. Good breakage means over 90% advance and at least one blast-hole breaking towards both relief holes. The main conclusions of the results analysis may be summarized as:

- The drilling precision was worse than assumed in design formulas. The scatter in the collaring position and alignment angle distributions are about 5 cm and 2-2.5 cm/m. The total error at the bottom of a drilled cut was often 20-30 cm which is large in comparison with design burdens in the range of 20-40 cm.
- A qualitative rating of the drilling quality (DQR) with 5 classes from very bad to very good was developed. The shape and size of the drilled cut are the main factors. It was found that a drilling quality which is bad or better, doesn't influence the breakage result directly. A very bad drilling may do so, but not necessarily.
- A parallel hole cut has a redundant breaking capacity since only one of the two initial blast-holes need to break to both relief holes for the opening to function well.
- Two kinds of malfunctioning of the cut blasts that correlate with the breakage were identified; detonation failure or impaired initiation function, and rifling. Rifling occurs when the charge fails to break and remove the burden. Sometimes this was caused by intentionally large burdens, sometimes by poor drilling.
- Detonation failures causing bad breakage occurred because of a) detonator failure, b) a poorly placed detonator, c) extreme apparent densities of ANFO in blast-holes and d) a blast-hole with ANFO venting prematurely to a relief hole. Venting occurred

when a blast-hole bottom came too close to a relief hole. It also happened for a blast-hole with emulsion but then didn't cause a detonation failure. The reason is probably that the emulsion needs less confinement than ANFO to detonate properly.

- A delay of 75-100 ms is judged to be quite sufficient to obtain good breakage and fines removal in the center of the cut. 50 ms is probably sufficient but not 25 ms.
- The cuts with neither malfunction nor very bad drilling quality have been plotted in a design diagram with the linear charge concentration versus the burden of the 1st blast-hole. An analysis shows the following:
 - a) The plastic deformation or burning burden limit of Langefors and Kihlström, which for most cuts was 30 cm, can be exceeded without a loss of good breakage.
 - b) There is an upper limit of about 30 cm for the spacing of the relief holes, above which they may not cooperate in the breakage process.
 - c) Langefors and Kihlströms' original prediction equation for the charge concentration needed to break out a given burden gives a satisfactory agreement with the Buko results, when updated with the rock constant and weight strength values.
 - d) The results are reasonably well explained by the difference in charge concentration, without including the difference in the decoupling.
- The size of the fine rock fragments remaining in or at the cut cavity correlates strongly with the color of the fines. Black fines were the largest with an average size of 20 mm and grey-white fines were the smallest, 3.3 mm. A darker color might be an indication of oxygen deficiency and a loss of energy or VOD due to an imperfect detonation. The correlation with charge concentration was however too weak to be significant, compared to the scatter in the data.
- The maximum throw of the rock fragments lies in the range 8-55 m. That of the ANFO cuts was significantly longer than that of the emulsion cuts. An initial fragment velocity of 38-65 m/s was measured. A ballistic estimate with a 3,5 m cut height gives a maximum throw of 32-55 m, in good agreement with the measured results.

The goals of the project have been reached; basic data for the use of ANFO in 4 m long rounds have been obtained and a design formula for parallel hole cuts which is useful both for ANFO and decoupled emulsion explosives has been found.

Our experience of the requirements for obtaining good breakage in a four-section cut have been listed. In order to obtain a better design basis for parallel hole cuts, the effects of the following factors should be investigated: primarily the cooperation of multiple relief holes and decoupling, and to some extent initiation delay and bad drilling.

Key words: Rock blasting, parallel hole cut, ANFO, emulsion explosive, limestone, drilling precision, detonation failures, initiation delay, charge prediction equation, fines, throw.

Nyckelord: Bergsprängning, parallellhålskil, ANFO, emulsionssprängämne, kalksten, borrhålsavvikelse, detonationsavbrott, tändfördröjning, laddningsberäkning, finandel, kast.

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1. INTRODUCTION

Even though the use of TBM machines in tunnelling has increased lately, blasting is still the most widely used method for excavating tunnels in hard rock. It is however necessary to increase the productivity of blasting for it to remain a competitive method in the future. Recent work in Sweden shows that it is possible to increase the advance per round from 4-5 m to at least 7 m on a long term basis when the rock conditions are good. See Ouchterlony (1992) and Niklasson and Keisu (1993a).

In Japan, due to the more complex geological conditions and regulations, advances per round seldom exceed 3 m. For such short rounds, cuts with angled holes like the V-cut are commonly used. To achieve longer advances using parallel hole cuts has long been the goal of Sato Kogyo Co. e.g., Kanoh and Ohtsuka (1983).

Drilling and mucking is generally made with increasingly large scale mechanized equipment. So is charging in many places. In Japan however, cartridged explosives are usually used in the whole round even when the tunnel area is large and the charging is made manually from a platform on the drill jumbo. One avenue for speeding up the cycle time is to use ANFO in combination with pneumatic charging. Sato Kogyo has combined this together with parallel hole cuts and Nonel initiation of the tunnelling round in what is called the "Swedish method", in order to improve the productivity of tunnel blasting in Japan at e.g. the Fujiyoshida Tunnel (Nakamura and Kawano, 1991).

In general ANFO is however not used very often because firstly the ventilation is often too slow in removing odors and fumes and secondly regulations limit the permitted amount of ANFO stored on site.

The basic Swedish parallel hole cut designs are based on the work by Langefors and Kihlström (1963, 1978) who give the charge necessary for break out towards a parallel empty hole (relief hole) with LFB dynamite in Ø 32 mm blast-holes,

$$I_a = 0.55 \cdot (B - \emptyset/2) / \sin \alpha^{1.5} = 1.56 \cdot (B - \emptyset/2) \cdot (B/\emptyset)^{1.5}. \quad (1)$$

Here I_a is the linear charge concentration in kg/m, B the distance between hole centers and Ø the diameter of the empty or relief hole, both given in m, and 2α the breakage angle. See Figure 1 and equation 7:4 and figure 7:5 in their monograph. The transformation is achieved by using that $\sin \alpha = 0.5\emptyset/B$.

Later Holmberg (1982) generalized this work, accounting for different explosives and rock conditions. The explosive was accounted for by the traditional Swedish weight strength

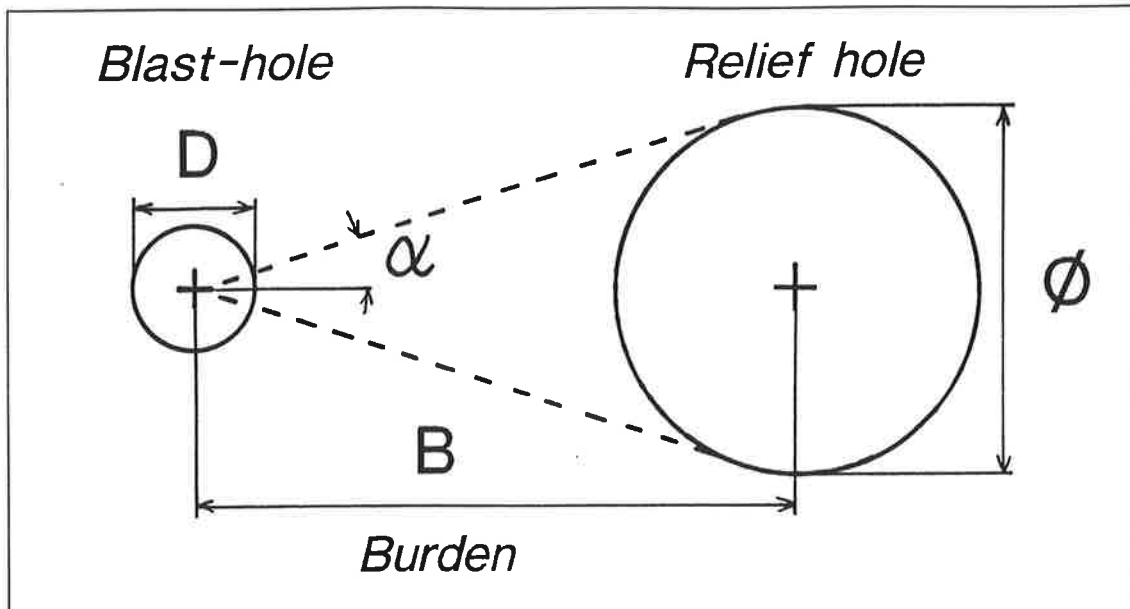


Figure 1: Blasting towards narrow circular opening.

concept and renormalization with ANFO as a reference through the factor $(1.19/s_{ANFO})$. The rock conditions were accounted for by the rock constant through the factor $(c/0.4)$. He also introduced a breakage scaling which is proportional to the diameter D of the fully charged blast-hole through the factor $(D/0.032)$. With D in m the result becomes

$$I_a = 55D \cdot (B - \emptyset/2) \cdot (B/\emptyset)^{1.5} \cdot (c/0.4)/s_{ANFO}. \quad (2)$$

This formula is also given by Persson et al. (1994) in their monograph.

When several relief holes are placed closely enough they are treated as one in the charge calculations, using the relation $\emptyset_{eff} = \emptyset \cdot \sqrt{n}$ where $n = 2$ in our case. Persson et al., (1994) don't tell how close the spacing has to be however. Ideally both blast-holes need to connect with both relief holes during the blasting process.

In the approach of Langefors and Kihlström (1963, 1978) there are essentially two elements. The first one is the minimum charge concentration needed to cause breakage for a certain geometry and which with a sufficient practical margin is given by equation 2 above. The second one is a geometrical restriction which says that, given that equation 2 is satisfied, different degrees of breakage will occur. When

$B \geq 2.1 \cdot \emptyset$	plastic deformation or burning occurs,
$2.1 \cdot \emptyset > B \geq 1.5 \cdot \emptyset$	merely breakage occurs,
$1.5 \cdot \emptyset > B \geq 0.5 \cdot (\emptyset + D)$	the desired volume is clean blasted and
$0.5 \cdot (\emptyset + D) > B$	the blast-hole is drilled into the relief one.

If the latter occurs anywhere along the length of the hole it causes premature venting of the explosion products which in turn may result in a detonation failure.

Persson et al. (1994) state the restriction that the maximum burden $B_{\max} < 1.7 \cdot \varnothing$ for the first quadrangle in the cut to work satisfactorily when the drilling deviation is less than 1% . When the drilling precision is known, the practical design burden is given as

$$B = [1.7 - (\alpha_1 + \alpha_2 \cdot H)] \cdot \varnothing \quad (3)$$

in order to avoid a breakage failure due to an excessive burden. Here H is the drilled depth in the cut, α_1 the collaring deviation in m and α_2 the angular deviation in m/m. In practice the value $B = 1.5 \cdot \varnothing$ is used, see also Olofsson (1991), and Persson et al. (1994) agree. Olofsson uses the practical design interval $\varnothing < B < 2 \cdot \varnothing$.

The extended charge equation, 2 above, is however not substantiated by any new experiments and it doesn't take decoupling of the charges into account, which is the rule when cartridged products are used in Japan. In order to have design data relevant for other rock conditions and explosives, it was thus considered necessary to test its validity by making experiments. Besides working in Japanese rock conditions, an objective of these tests was to compare the breaking efficiency of ANFO versus cartridged emulsion explosives. This was considered to be an important basis for the use of the Swedish method under Japanese conditions.

Here we report parallel hole cut blasting tests with ANFO and a cartridged emulsion explosive in a limestone quarry. The work has been made as a cooperation between the Sato Kogyo Co and the department of civil engineering at the Yamaguchi University in Ube. The testing was made between July 18 and August 14, 1992. The results were presented at the JSCE rock mechanics symposium (Minamide et al., 1993). A preliminary report in Swedish has also been made by Ouchterlony (1993).

The cuts used in these tests were 4 m deep and most of them had 2 large relief holes with only 2 charged holes around, see Figure 2. The main experimental parameter was the burden, i.e. the distance between the charged and the relief holes. The burden was increased blast by blast until no breakage occurred. This sequence was repeated for the different explosives in 3 different charged hole diameters $D = 43, 45$ and 48 mm. Some other tests were also made.

The specific goals of these field tests were to:

- Confirm earlier experience in blasting parallel hole cuts with emulsion explosives.
- Obtain the relation between charge concentration and cut burden for the explosives.
- Obtain basic data for the use of pneumatically loaded ANFO and a step up in advance to at least 4 m.

The participating Sato Kogyo Co personnel on site were:

· Hideo Minamide, project leader

Hitoshi Suzuki, test engineer

Toshitaka Tanbo, test engineer

Katsuya Yoshida, test engineer.

The work on site was also monitored by graduate student Masayuki Hatayama from the Yamaguchi University.

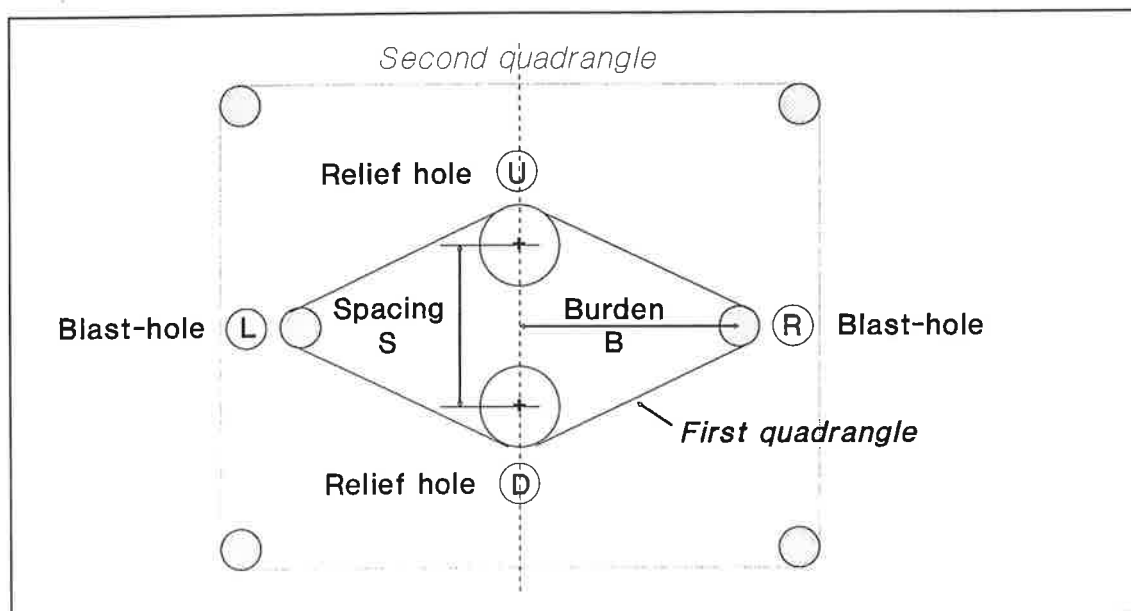


Figure 2: Definition of geometry in Buko cut blasting tests.

2. TEST SITE DESCRIPTION

2.1 Site Description

The test site was the Buko Mine near Chichibu in the Saitama prefecture west of Tokyo. The mine is situated on the Buko mountain, half of which consists of a roughly 100 million years old limestone of paleozoic age with occasional oolites (schalstein). The mountain is 1305 m high and rises high above the surrounding valleys below at between 300-400 m.

There are 5 different quarries and the Buko quarry lies in the middle with its present working level at 1130 m, see Figure 3. Otaka (1994) gives a description of the cooperative development made to meet environmental requirements and natural resources conservation at Mt Buko.

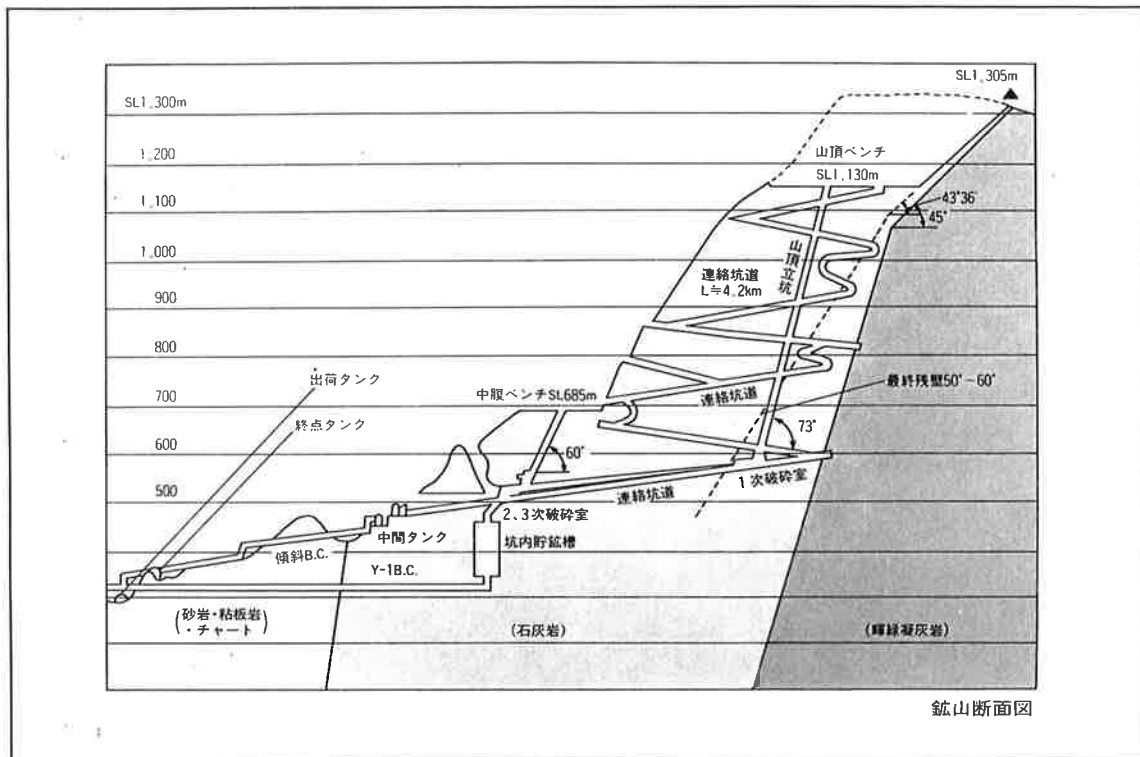


Figure 3: Vertical section through the Buko mine.

The Buko quarry is owned by the Buko Mining Co which was established in 1970 by Nihon Cement Co and Chichibu Lime Industry Co. The whole operation, including the mine, crushing plant, transportation, marketing etc. employed 75 people plus 16 people from contractors in 1993.

The quarrying is mainly done in single 10 m benches using 12 m long, Ø 115 mm holes angled at 70° and with the burden of 4.2 m and spacing of 4.5 m. Each hole is charged with an 8 m column of about 40-50 kg of explosive. One blast is fired each day and it consists of several rounds of 1 to 2 rows of holes at different places in the quarry. All rounds are connected to the same Hercudet trunkline and fired in a close sequence. The specific charge is about 0.25 kg/m³.

The limestone has an in situ density of 2590-2690 kg/m³ and contains about 55% CaO and 0,5% MgO. It is loaded on dump trucks and transported to a chute which leads down to the crusher on the 350-450 m level. The total production was 4.66 Mtons in 1993 out of which 2.25 was cement, 0.61 lime and 1.80 limestone aggregate.

The final wall of the quarry has an overall slope angle of 43°. See Figure 4. Presplitting with Ø 75 mm blast-holes at a c-c of 0.9 m, 30 holes in each delay, is used to produce the final wall. The single benches are drilled with inclined holes, 70°, and the catchment berm at the wall is roughly 7 m wide. The parallel hole cut blasting tests were carried out in the face of a bench about 40-50 m from the final wall, see Figure 5.

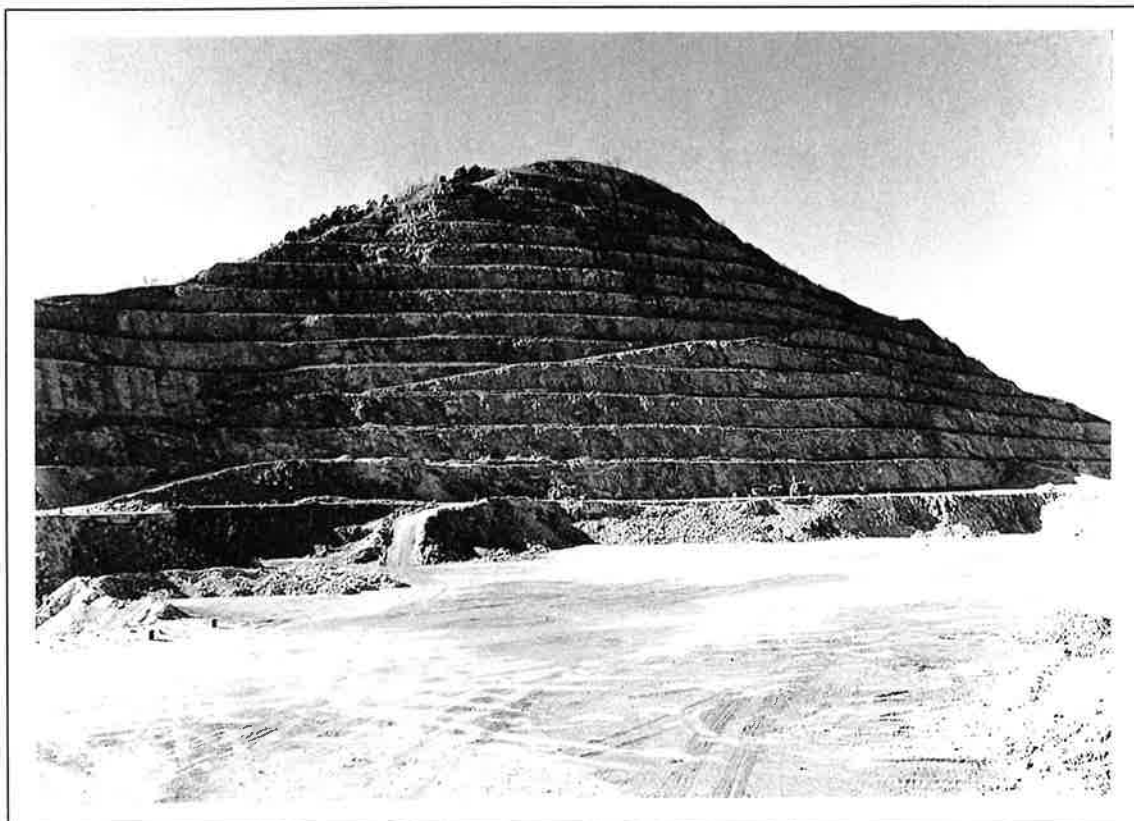


Figure 4: Photo of final slope on Mt Buko.

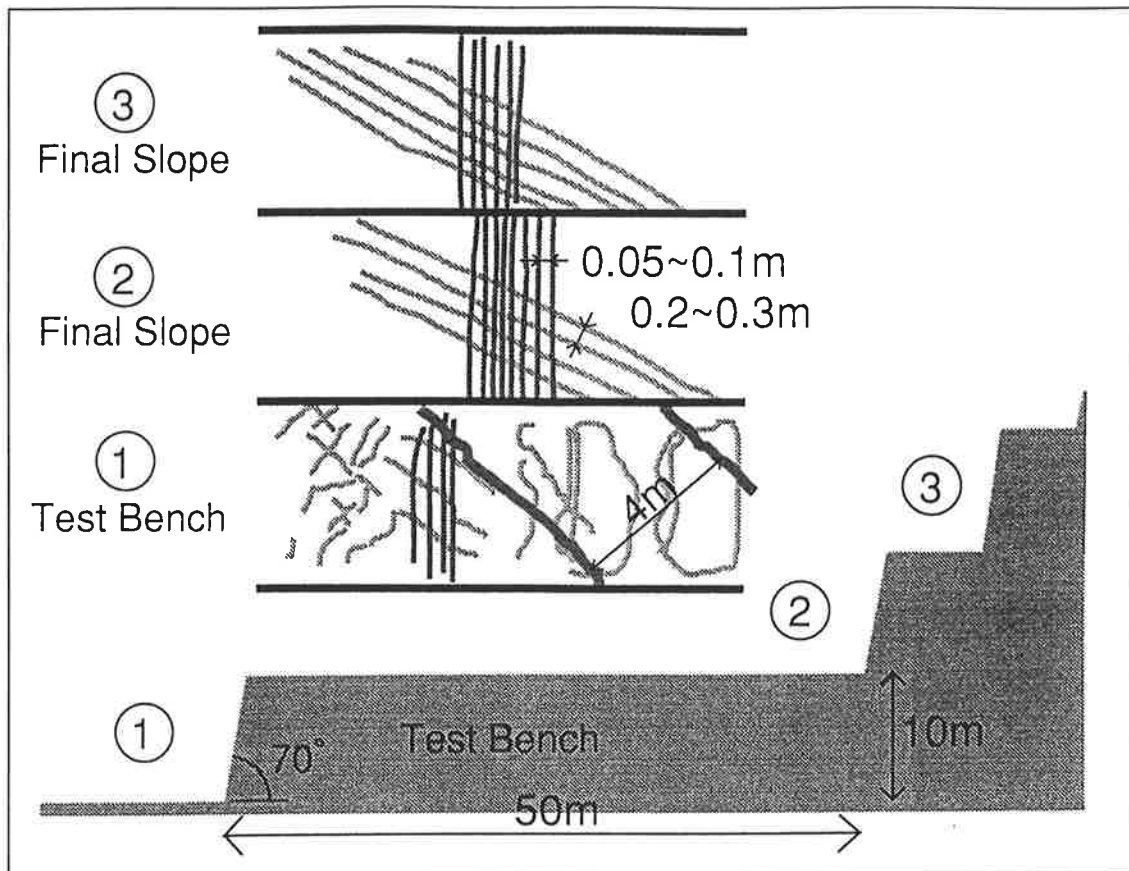


Figure 5: Vertical section through test bench and fracture mapping.

2.2 Rock Conditions

A rough structural mapping was made of the face of the test bench and 2 faces of the final slope above it. This showed that there are at least 4 joint sets. Directing a temporary North into the face, there are firstly 2 sets of weathered joints with dip/dip directions of roughly 45/135 and 45/225. See Figure 5. The characteristic distance between these joints was about 2-4 m and the joint infill consisted of schalstein. They were clearly visible in the test bench and were expected to influence the blasting results in an uncontrollable fashion.

The other two joint sets were intact and had much smaller spacings. The third set had a characteristic spacing of 0.2-0.3 m and was oriented roughly 30/90 and the fourth one had a spacing of 0.05-0.1 m and the orientation 90/90. The latter set was not visible everywhere and the rock between the joints was sometimes of sugar cube character. These two joint sets were expected to govern the average breakage behavior of the rock.

Blast damage was clearly visible in the face of the test bench. Therefore this face was cleaned by a back hoe before the tests began.

The mine had some rock mechanics testing data. It uses the following rock classification based on P-wave speeds;

- | | |
|------------------------|----------------------------|
| - good limestone, | $V_p = 2500-4000$ m/s, |
| - poor limestone, | $V_p = 1000-2500$ m/s, and |
| - weathered schalstein | $V_p = 1000-1800$ m/s. |

The point load tensile strength index varies from 0.49 to 15.7 MPa, or more typically within 5.3-7.9 MPa with an average of 5.8 MPa. The uniaxial compressive strength typically lies in the range $\sigma_c = 120-125$ MPa and the Shore hardness within 46-55.

The P-wave velocity of the rock in the test bench was found to be $V_p \approx 2300$ m/s so the limestone is relatively poor. Based on the above information the rock was judged to belong to the C_M class, see Appendix A, which is frequently encountered in Japanese tunnelling and in which the prescribed drilled depth of a round is 1.5-2.0 m.

2.3 Rock Constant or Rock Blastability

The blastability of the rock is given by the rock constant c in Langefors and Kihlström (1963, 1978). It is, with a technical margin for satisfactory breakage $c = 1.2c_o$, the limit amount of bottom charge required to break loose 1 m³ of rock. The explosives consumption at Buko is given by the specific charge 0.22-0.25 kg/m³ which is a lower limit to c .

As a preparation for the cut blasting tests, four shots with Hamamite emulsion explosive to determine c were made in Ø 32 mm diameter holes a low bench, see Figure 6. The results are shown in Table 1. If a throw ≤ 2 m is taken as indicative of a limit charge, then $c_o \approx 0.3$ kg/m³ and the rock constant $c \approx 0.35$ kg/m³ with the desired margin. This corresponds very well with the range of c -values usually used in Japan, 0.30-0.35 kg/m³. The corresponding values used in Sweden are 0.40-0.45 kg/m³ which means that Swedish rock usually is harder to blast than Japanese rock.

If there is any doubt about the blastability of the rock in a tunnelling project, this kind of evaluation can easily be made, e.g. in the bottom bench with Ø 32 mm blast-holes (Persson et al., 1994). We consider that the blastability value $c \approx 0.35$ kg/m³ is an appropriate starting point for our tests.

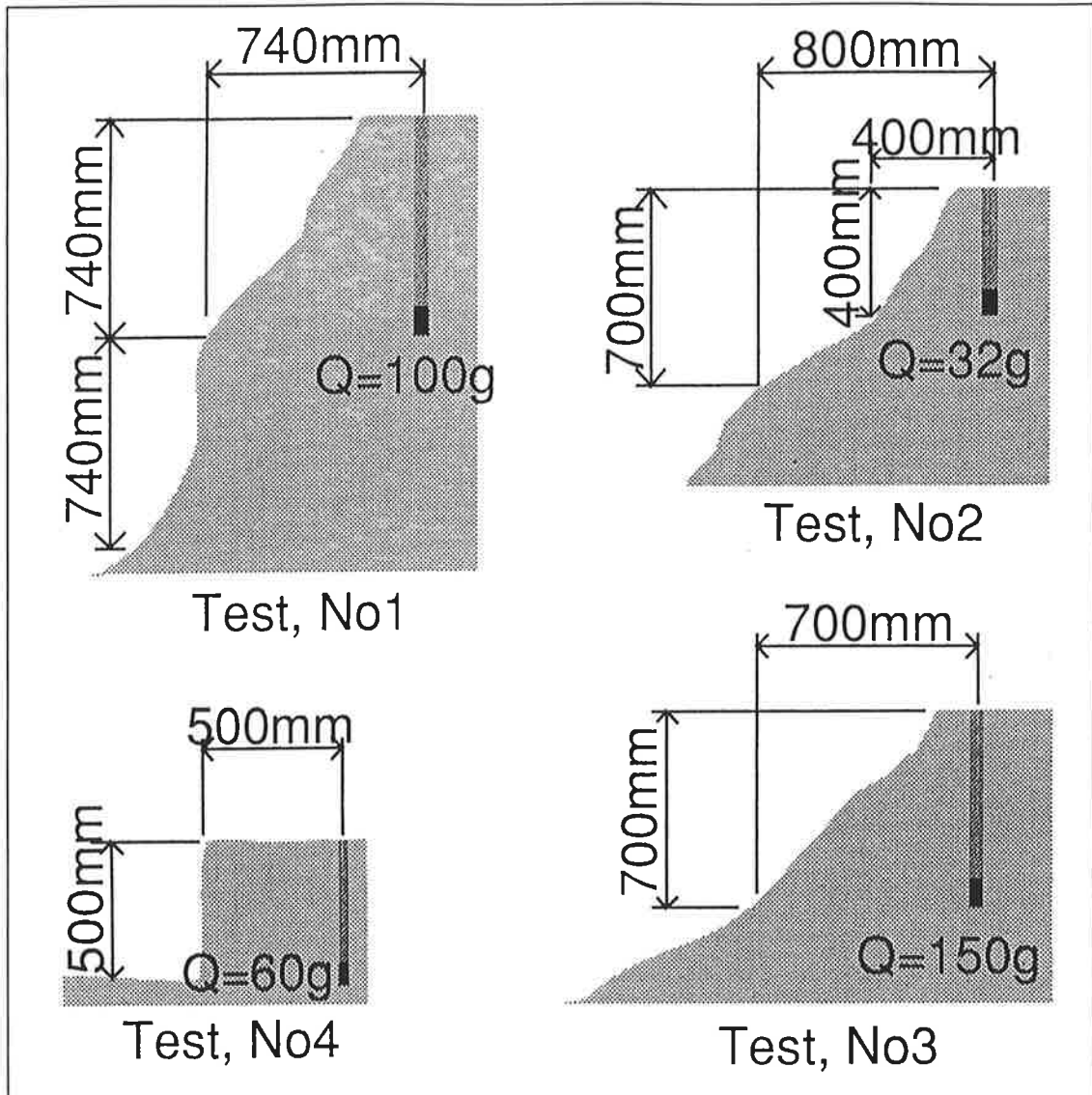


Figure 6: Geometry of test shots for rock constant determination.

Table 1: Data for rock constant determination.

Shot no	Bottom	Burden m	Charge g	Throw m	Rock constant
1	free	0.74	100	0	0.22
2	fixed	0.40	32	2	0.30
3	fixed	0.70	150	>2	0.35
4	fixed	0.50	60	2	0.30

3. CUT BLASTING TESTS

3.1 Explosives and Initiating System

Two explosives were used in our tests, ANFO and the emulsion explosive Hamamite. The ANFO was manufactured by Nippon Carlite Co or AN prills were supplied by Mitsubishi Chemical Co and Sumitomo Chemical Industry Co. The density of the ANFO in the hole was estimated at 800-900 kg/m³ prior to the tests. The expected VOD of the ANFO is 3000 m/s in a Ø 35 mm steel pipe. A 200 g paper cartridge of Hamamite was used to ensure proper initiation of the ANFO.

Hamamite is a cap sensitive emulsion manufactured by the Nippon Carlite Co in Yokohama. Its matrix consists of an AN plus SN (sodium nitrate) solution with emulsifier and wax. It is sensitized by glass micro balloons and has a density which lies in the range 1050-1150 kg/cm³. Its VOD for unconfined Ø_{exp} = 25-30 mm charges lies in the range 5000-5600 m/s. Its explosion energy is 3.64 MJ/kg and the NTP gas volume 0.94 m³/kg so the corresponding Swedish weight strength value is s = 0.94 kg ANFO per kg Hamamite (Persson et al., 1994). In this case it was not necessary to use a primer.

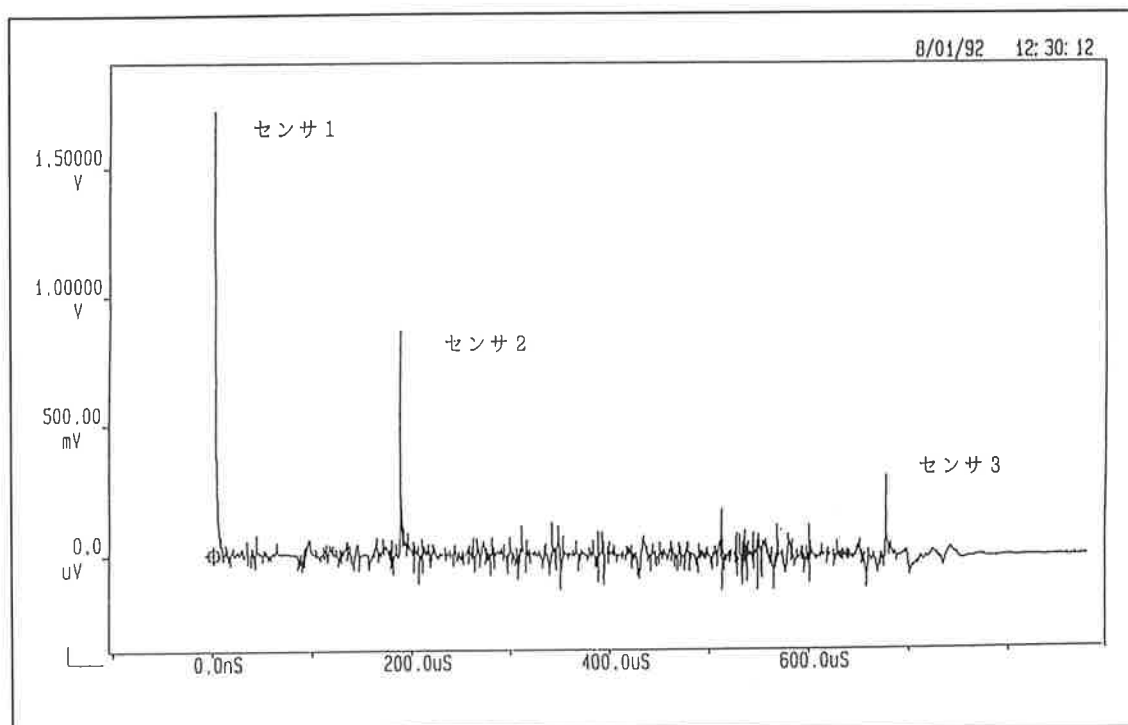


Figure 7: VOD registration from the three sensors in cut E-2.

During our tests, the VOD was measured in 2 emulsion blast-holes and in 3 ANFO holes, using twinned wire ionization probes that are short circuited by the reaction front.

Three probes were used in each hole at 1 m intervals, starting at the bottom. The measurement results were rather poor. A registration from cut blasting test A-10 (new E-2, see Table 2 below and Appendix B) is shown in Figure 7. The first interval corresponds to a VOD of 5350 m/s and the second one to 2045 m/s. The latter value is much too low and we don't know whether it is a sign of unreliable measurements, of the channel effect or of something else. There was no post blast sign of malfunctioning however since the breakage results were good. The other emulsion blast gave a VOD of 5035 m/s over the first interval.

In the ANFO blast-holes, the first probe was positioned in the primer, the two others fixed to the downline shock tube by vinyl tape. The VOD results still weren't good. One of the blasts, 115 (new A-14), gave an average VOD of 4215 m/s over the first interval. The other two, A-20 (new A-4) and A-36 (new A-5) gave values in the range 1650-2800 m/s, which are too low. The breakage was satisfactory in all 3 cases however.

The mine was using a Hercudet trunkline for its blasting work and we had to hook up our test blasts to it in order not to disturb the production. Hence it was natural to use the Hercudet connectors, downline tubing and detonators or caps for our tests too, to begin with. To achieve the delay 100 ms between the initiation of the two charged holes, L (left hand) and R (right hand) in Figure 2, Hercudet cap pairs with numbers 4 - 6, 5 - 7 or 6 - 8 were used.

After some failures where the explosive didn't detonate we let a Hercudet cap initiate two Nonel GT/T tunnel series downlines with caps number 0 and 1 instead. In this case the nominal delay is 75 ms. In the cases where we tested a delay of 25 or 50 ms between holes this was achieved by delaying the initiation of the no. 1 downline with extra coils of Hercudet tubing.

The primary explosive in the Hercudet caps was DDNP (diazodinitrophenol) and in the Nonel caps lead azide.

3.2 Testing Plan, Actual Test Series, and Measurements

The cut layout is shown in Figure 2. The intended drilling depth was 4 m and the basic cut design had two relief holes. In a majority of the blasts only two blast-holes were drilled and fired. As explained above, the nominal delay between them was 75 or 100 ms in most cases. The testing plan was devised to cover practical aspects while at the same time finding the maximum burdens for our cut geometry and explosives combinations.

The primary variables in the test series were thus

- Explosive; emulsion ($\varnothing_{\text{exp}} = 25$ or 30 mm cartridges) or ANFO.
- Blast-hole diameter; $D = 43, 45$ or 48 mm.
- Burden B of the first two blast-holes.

Five variables of lesser importance were:

- Distance or spacing S between relief holes.
- Diameter of relief holes; $\varnothing = 102$ mm and in a few cases 45, 48 or 114 mm.
- Detonator; Hercudet or Nonel.
- Initiation delay between blast-holes, 100, 75, 50, 25 or 0 ms.
- Type of cut; a single large relief hole or a slit type of cut.

A second series of 6 blasts, each one a single reaming shot breaking into the cavity remaining from the first series, was also planned as was a series of 6 complete cuts to be filmed using high speed video cameras. Due to problems, the testing plan was revised many times. A major reason was that many blasts virtually pulled down the whole bench face which again had to be cleaned with the back hoe before further blasts could be made.

The actual number of blasts made during the testing was as follows

- 9 cuts with 2 large relief holes, using Hamamite,
- 21 cuts with 2 large relief holes, using ANFO,
- 3 cuts with 2 small relief holes, using ANFO,
- 4 cuts with 1 large relief hole, using ANFO,
- 5 cuts of slit type with only 48 mm holes, using either Hamamite or ANFO,
- 3 reblasts of cuts, and
- 4 complete cuts that were recorded by high speed video.

Thus in all 49 test blasts were made. This report covers only the 33 first cuts. One of them vanished completely because the bench face fell in. Data for the remaining 32 cuts are given in Table 2. The slit cut had been used in the Sofia project, see Ouchterlony (1992), but at Buko the slit cut blasts were all unsuccessful. The four parallel hole cuts with one large relief hole were left out because of their small number.

The 100 g emulsion cartridges have lengths of 190 and 140 mm respectively and corresponding linear charge concentrations of 0.53 and 0.71 kg/m. The planned charging was either 20 sticks of $\varnothing_{\text{exp}} = 25$ mm Hamamite, leaving 0.2 m of the 4 m cut uncharged or 26 sticks of $\varnothing_{\text{exp}} = 30$ mm Hamamite, leaving 0.35 m uncharged. The charging of the Hamamite was done manually. The emulsion holes were stemmed.

Table 2: Basic data for the parallel hole cuts with two relief holes tested at Buko.

Cut test	Date 1992	Explosive brand	Detonator brand	Delay time	Blast holes	Charge diam	Relief holes	Relief spacing	Blast-hole burden
no	day-mo	name	name	ms	mm	mm	mm	mm	mm
A-18	22-07	ANFO	Hercudet	100	45	45	114	200	200
A-26	23-07	ANFO	Hercudet	100	48	48	114	200	200
A-18'	23-07	ANFO	Hercudet	100	45	45	45	200	250
A-21	23-07	ANFO	Hercudet	100	45	45	102	300	300
A-2'	24-07	Hamamite	Hercudet	100	45	30	102	200	200
A-3	24-07	Hamamite	Hercudet	100	45	30	102	200	300
A-28	25-07	ANFO	Hercudet	100	48	48	102	300	200
A-19	28-07	ANFO	Nonel	75	45	45	102	200	300
A-18"	28-07	ANFO	Nonel	75	45	45	102	200	250
A-27	29-07	ANFO	Nonel	75	48	48	102	200	300
A-34	29-07	ANFO	Nonel	75	43	43	102	200	200
A-4	30-07	Hamamite	Nonel	75	45	30	102	300	200
A-5	30-07	Hamamite	Nonel	75	45	30	102	300	300
A-20	31-07	ANFO	Nonel	75	45	45	102	300	200
B-5	31-07	ANFO	Nonel	75	48	48	48	150	200
A-29	31-07	ANFO	Nonel	75	48	48	102	300	300
A-10	01-08	Hamamite	Nonel	75	45	25	102	200	200
A-11	01-08	Hamamite	Nonel	75	45	25	102	200	250
A-12	03-08	Hamamite	Nonel	75	45	25	102	250	200
A-13	03-08	Hamamite	Nonel	75	45	25	102	250	250
B-6	05-08	ANFO	Nonel	75	48	48	48	200	250
A-35	05-08	ANFO	Nonel	75	43	43	102	200	300
A-36	05-08	ANFO	Nonel	75	43	43	102	300	200
A-37	05-08	ANFO	Nonel	75	43	43	102	300	300
105	11-08	ANFO	Nonel	75	48	48	102	200	400
106	11-08	ANFO	Nonel	75	45	45	102	200	400
108	11-08	Hamamite	Nonel	75	45	30	102	300	350
115	12-08	ANFO	Nonel	50	48	48	102	250	250
110	13-08	ANFO	Nonel	75	43	43	102	200	350
113	14-08	ANFO	Nonel	0	48	48	102	250	250
114	14-08	ANFO	Nonel	25	48	48	102	250	250
105'	14-08	ANFO	Nonel	75	48	48	102	200	400

For the ANFO a design degree of packing of 800 kg/m^3 was used, giving linear charge concentrations of about 1.3, 1.5 and 1.7 kg/m respectively. An uncharged hole length of 0.25-0.35 m was planned. The charging was done with a pneumatic loader. The ANFO holes were not stemmed. The actual charge concentrations are reported in section 3.5.

The final testing matrix of these 33 cuts is shown in Figure 8, as charge concentration versus planned blast-hole burden. The three curves of required charge concentrations were calculated from equation 2 using $c = 0.35 \text{ kg/m}^3$ and $s_{\text{ANFO}} = 0.94$ for the Hamamite but with different geometrical conditions. The upper one represents a 45 mm blast-hole breaking towards one 102 mm relief hole, the middle one a 45 mm blast-hole breaking towards two 102 mm holes etc. The three vertical ticks on the latter delineate the different types of breakage given by the geometrical restrictions for that case, i.e. clean blasting, breakage and plastic deformation or burning. See section 1.

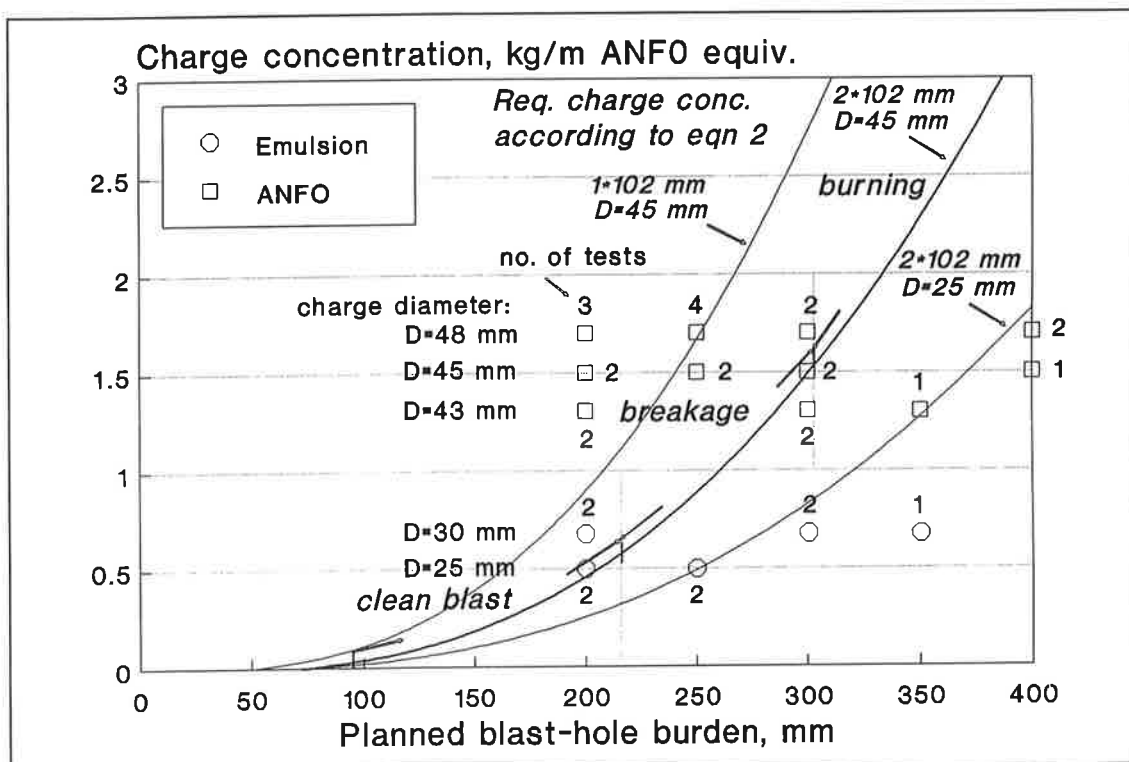


Figure 8: Cut test series plotted as charge concentration versus burden.

The fact that many ANFO blasts lie above the curve in the breakage region implies that these holes are overcharged for the burdens chosen, if the charge concentration curves are correct. On the other hand possible drilling deviations prevent the design burden from being too large and it is an often made comment that breakage in the cut will improve if the charge concentration were decreased, see also Sofia project results (Niklasson and Keisu, 1993b).

Hence one would expect that malfunction due to plastic deformation and burning could be a problem for the ANFO blasts. Since most blasts were repeated with different distances between the relief holes and they cover the upper charge concentration curve one would expect to find how good the predicted necessary charge concentrations are.

The lower charge concentration curve was calculated using a 25 mm charge breaking towards 2 cooperating relief holes. For it to be valid, the blast-hole diameter should probably be 25 mm too. As a first approximation we assume however that the actual blast-hole diameter is filled with the same amount of explosive which thus has a much lower density but about the same weight strength. Thus the region between the charge concentration curves might cover the decoupling effect in 45 mm blast-holes.

Since the emulsion blasts cover this region and lie within the breakage and clean blasting regions, one could expect to find out what influence the decoupling of the charges has on the breakage and perhaps also how close the relief holes need to be for them to cooperate.

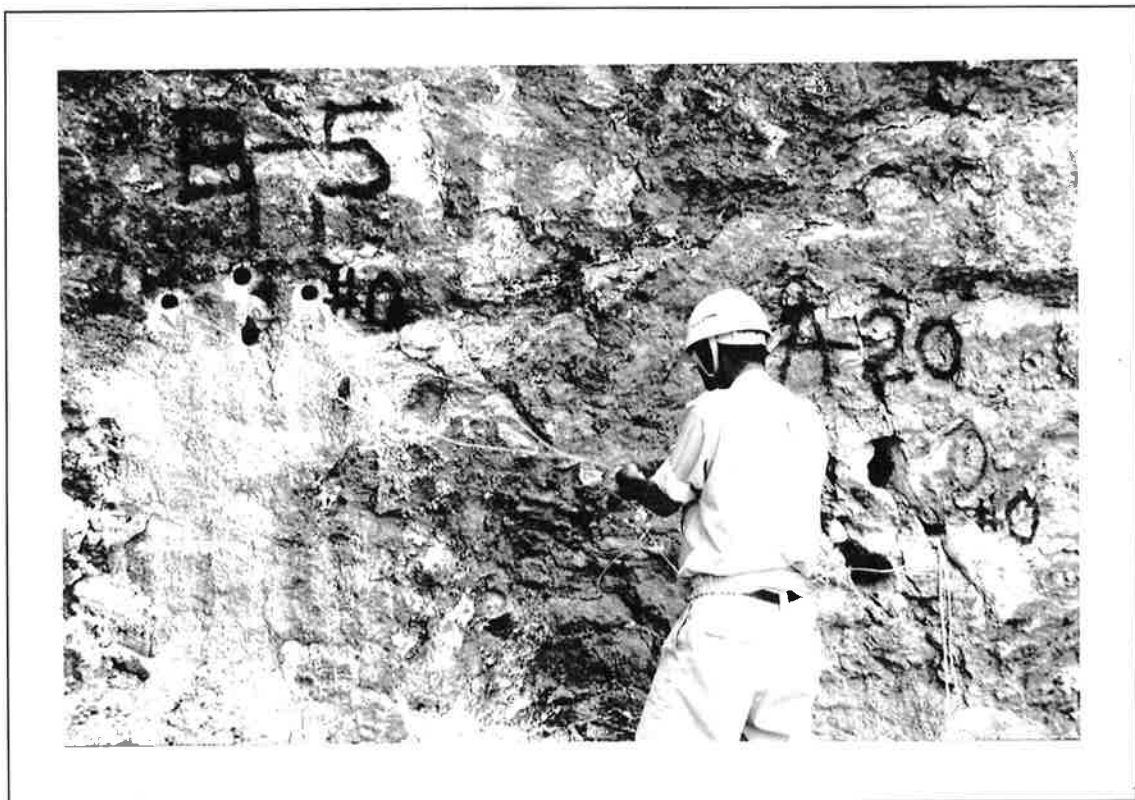


Figure 9: Photo of bench face before blasting with cuts B-5 (new A-8) and A-20 (new A-4) marked out.

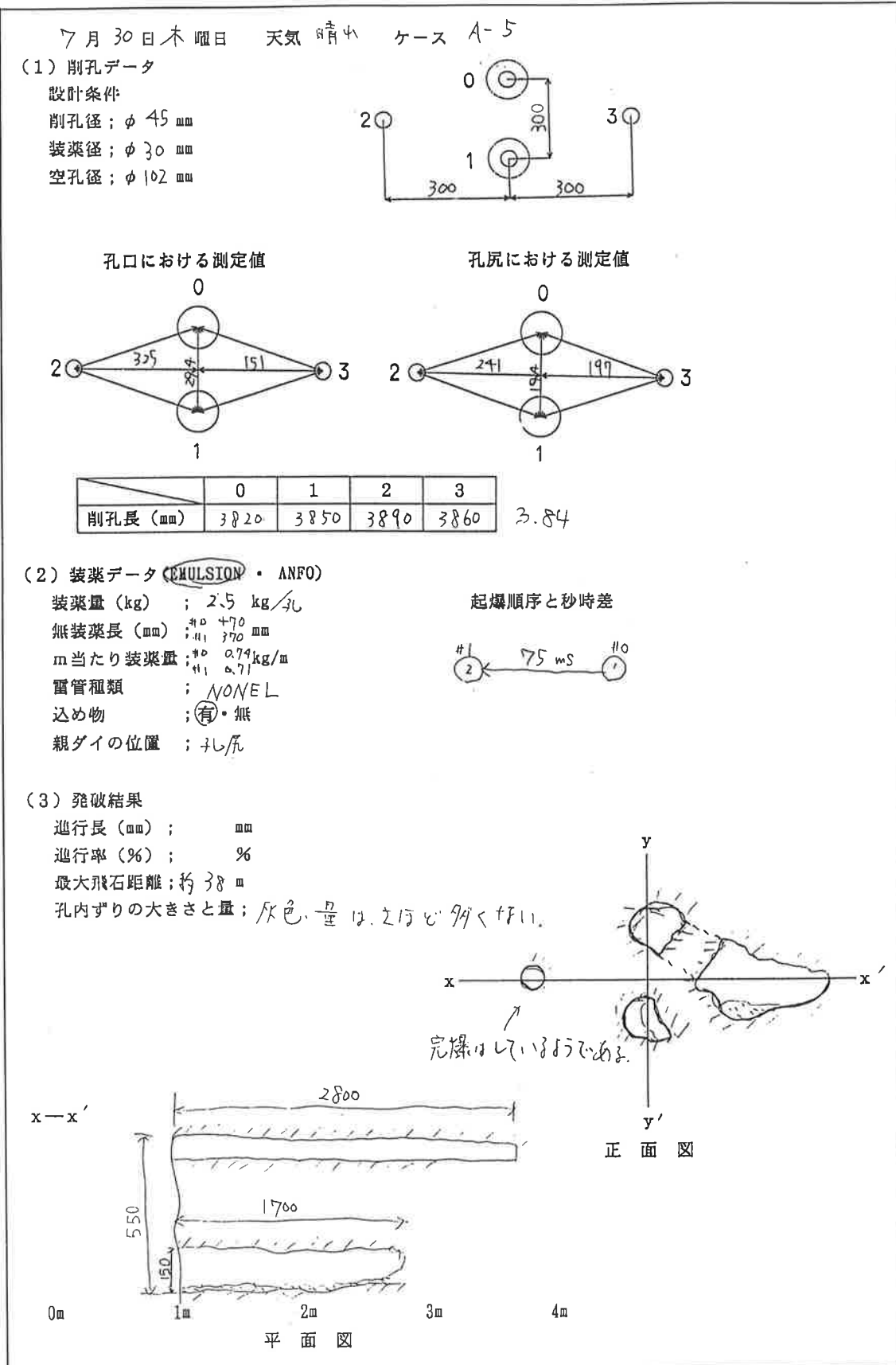


Figure 10: Parts of filled in test form for cut A-5 (new E-7). Hole no 3 detonated first.

Before the test blasts, qualitative structural mapping of the rock surface was made and the cut marked out with red paint before it was photographed, see Figure 9. The drill hole positions and hole angles were measured on the surface as were the hole depths. According to the original plans, the ground vibrations were to have been measured during the blasts but the more or less simultaneous initiation of the production blasts in the quarry made this impossible. During 5 test blasts VOD measurements were made, see section 3.1 above.

Test forms for all relevant data were filled in, parts of one is shown in Figure 10. A comprehensive table of the test data, including comments, is given in Appendix B. Note that in the sequel a new numbering for the cuts is used (Appendix B) where E denotes the Hamamite emulsion explosive and A denotes ANFO. The ordering E-1, E-2 etc is based on the measured burden value of the first blast-hole at the bottom of the cut.

3.3 Drilling Precision

A correct drilling is vital for obtaining the planned advance in tunnelling with parallel hole cuts. The drilling precision achieved in practice is determined by three main factors, the performance of the drilling equipment, the skill of the operator and the rock conditions. The drilling in our tests was made with a 2 boom Furukawa JCH 2-75 jumbo without special equipment for the positioning and the angling of the feed beams. The drilling was made with 4.5 m long 38 mm rods (code EH3ZR38R32-4500) and bits manufactured by Mitsubishi Material Co. One reaming bit was manufactured by Atlas Copco Co.

The operator was a skilled man with long drilling experience. The cut holes were all drilled to blast-hole dimensions at first and the relief holes then reamed to either Ø 102 or 114 mm, except in three cases; A-3, A-8 and A-13. The reaming was a comparatively inefficient and time consuming operation.

The drill hole depths were measured using wooden sticks with tape markings. The drill hole collaring positions and their angles were measured with a MONMOS model NET2, 3 dimensional digital surveying equipment. A tamping rod with two reflecting targets was inserted into the holes until the innermost target was flush with the face. The outermost target was then 1 m out. Of the 33 cuts in the test series, data from 1 cut were partially lost so complete data were obtained only from 32 cuts.

From the recorded data, the positions of each drill hole could then be calculated at the actual drilled depth, assuming that the in-hole deviations are negligible. The latter is

reasonable for 4 m deep holes in limestone. The resulting geometries for the individual cuts are shown in Appendix C.

The statistics of the hole depths for the 32 cuts given as mean $\pm$ standard deviation are given in the Table 3:

Table 3: Statistics for drilled hole depths in the Buko cuts, given in m.

Upper relief hole	Lower relief hole	1st blast hole	2nd blast hole	Overall average
3.88 ± 0.20	3.90 ± 0.20	3.83 ± 0.15	3.82 ± 0.21	3.86 ± 0.15

Thus the blast-holes were on average somewhat shorter than the relief holes. The drilled depth was defined as the average depths of the 2 relief holes and the 2 blast-holes. It ranged between 3.45 and 4.11 m. The statistics are given by 3.86 ± 0.15 m which means that the cuts were on average somewhat shorter than planned.

The values of the drilling precision, relative to the upper (U) relief hole in Figure 2, are plotted in Figure 11. The figure shows the relative collaring errors of the three other holes, their projected alignment errors at the average drilled depth and their sums.

Except for an extreme cut, the collaring error of the downmost (D) relief hole was less than 10 cm and on average 6 cm. The collaring errors for the blast-holes are larger. Except for an outlier at 20 cm, all values are less than 15 cm with averages of 7 cm for both the left (L) and right (R) blast-holes respectively. The center of gravity of each cluster is displaced somewhat towards the center of each cut. This means that in practice the cuts became somewhat narrower than planned. This is not uncommon, Ouchterlony et al. (1991).

The projected alignment errors are larger than the collaring errors. Except for a few outliers, the values are less than 15-20 cm or 4-5 cm/m. The averages are 10, 11 and 13 cm for the D, L and R holes respectively. When the two components are added to give the total drilling deviation or error, the results become slightly worse. For the lower relief hole and the left blast-hole, the errors still generally fall within 20 cm and, apart from the outliers, there seems to be only a small systematic component. For the right blast-hole the errors are larger, in many cases between 20 and 30 cm. There seems to be a systematic component as well which puts the average right blast-hole bottom 5-10 cm closer to the upper relief hole than planned.

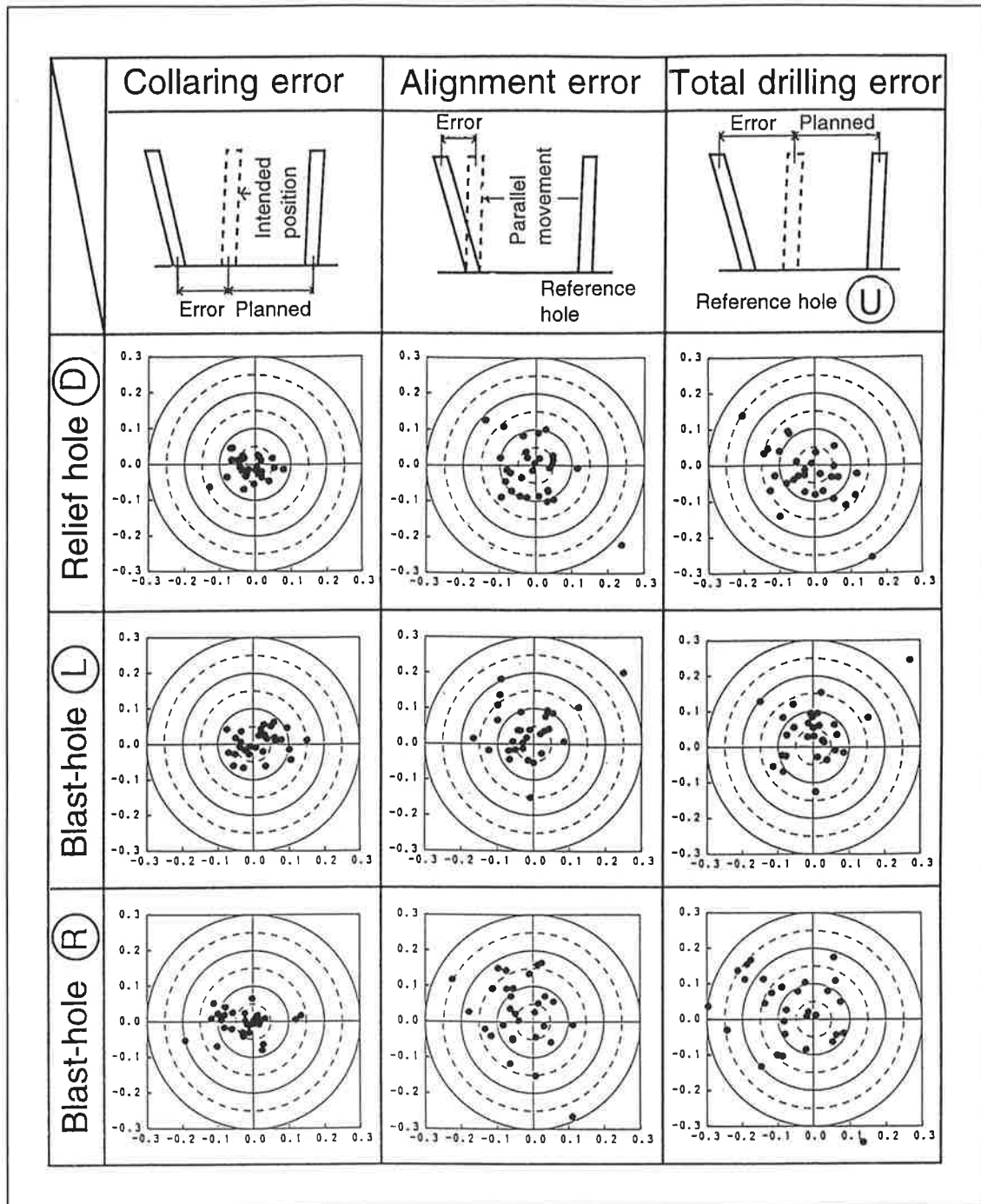


Figure 11: Scatter diagrams of collaring, alignment and total drilling errors.

The errors in the collaring and the alignment, about 10 cm and 4-5 cm/m, are large, much larger than the values 2 cm and 1 cm/m used by Persson et al. (1994) in their sample calculations. However, if we take half of these amplitudes as estimates of the standard deviations of the error distributions then the values 5 cm and 2-2.5 cm/m agree exactly with the values obtained in the Storsjö tunnel in shale (Ouchterlony et al., 1991).

This shows the importance of accurate drilling if one is to avoid malfunctions in the parallel hole cuts due to either an impaired initiation function from blast-hole interference or an impaired breakage function from an excessive burden or a narrow break out angle.

3.4 Practical Burden, Order of Initiation and Drilling Quality Rating

One can relate the statistics of the drilling errors to the calculated risks for hole interference during initiation and for breakage failure due to excessive burdens, see (Ouchterlony, 1992). We have instead chosen to calculate the actual burdens for both blast-holes. Their breakage geometry is defined by a triangle with corners at the hole centers and its base at the relief holes. The usual definition of the burden is the shortest, i.e. perpendicular, distance from the blast-hole center to that base. See Figure 12.

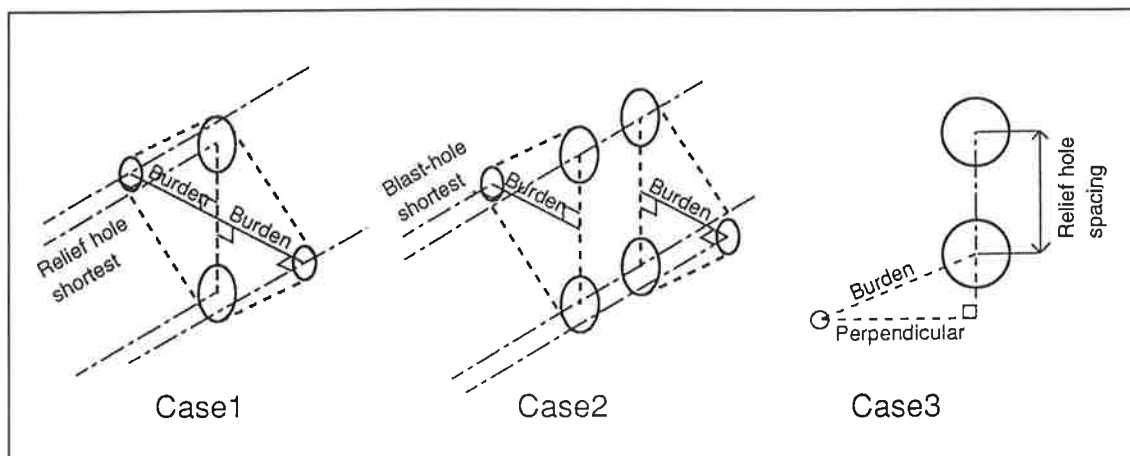


Figure 12: Definition of burden at bottom of cut, depending on actual hole depths.

The burdens have been calculated both at the face and at depth. The depth at which the burden has been calculated is given by the shortest drill hole in the breakage triangle. When one of the relief holes is the shortest we have case 1, when one of the blast-holes is shortest case 2. Sometimes the drilling error is so large that the perpendicular burden falls outside the base, case 3. Then the burden is defined as the c-c distance between the blast-hole and the nearest relief hole. These burden calculations were part of the evaluation procedure made after the test blasts.

The spacing values between the relief holes have also been calculated at the face and at the same depth as the corresponding burden values.

Appendix B summarizes the essential test variables for the 33 cuts in this test series. The cuts are listed with the new numbering, i.e. in order of increasing burden at the bottom of the first blast-hole. This is not the same order as that of the testing. Appendix B gives a translation table from the new test case numbers to the original ones.

The order of initiation of the blast-holes was decided on site after the drilling precision had been measured. That hole which seemed to have the easiest breakage, i.e. which of L and R in Figure 2 had the smallest burden on the bench face, was usually initiated first in the subsequent blast, see Figure 13. As can be seen from Appendix B, the burden values at the bottom usually follow the same order, but not always.

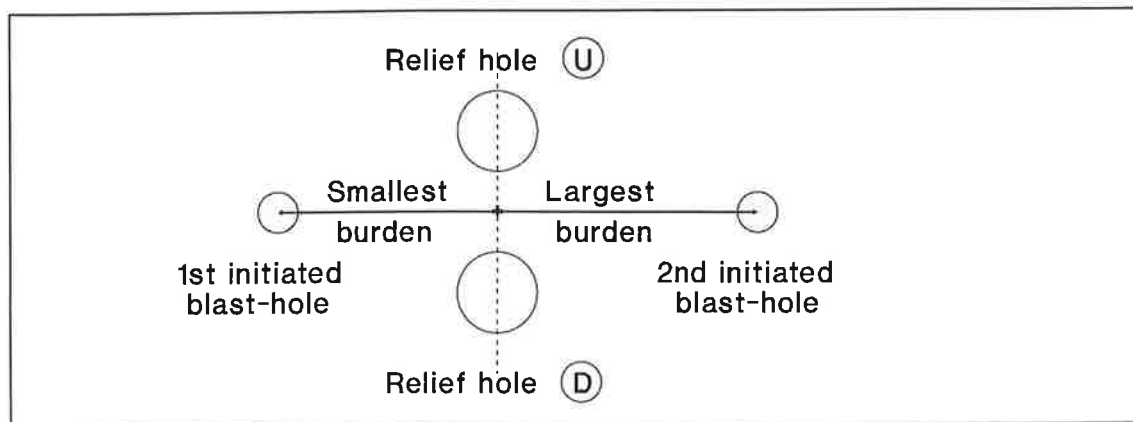


Figure 13: Definition of usual initiation order in cut, first and second blast-holes.

Appendix B gives a complete set of burden and spacing values. Composite drawings of most cut geometries at the bench face and at the bottom are given in Appendix C together with the deviations from planned spacing and burden values. The statistics, mean $\pm$ std deviation, for these deviations are summarized in Table 4.

In the light of Table 4 and the underlying data it can be said that;

- 1 The average spacing and burden values lie within ± 2.0 cm of the intended values at the face and, as a whole, within ± 2.5 cm at the bottom. Thus the systematic part of the error is small.
- 2 The cut area at the bottom tends to be the same as at the face since a slight spreading of the relief holes is compensated by a slight decrease in burden. The average triangular cut area is 0.064 m^2 and the statistics for the area difference between face and bottom are $0.003 \pm 0.023 \text{ m}^2$.
- 3 The scatter in the spacing and burden deviations hide individual values which may be quite large. The range of spacing error values is -4 to 8 cm at the face and -10 to 18 cm at the bottom, for the average burden error the ranges are -15 to 13 cm

at the face and -18 to 16 cm at the bottom. These errors are large compared to intended blast-hole burdens in the range 20 to 40 cm and sometimes a blast-hole has been drilled into a relief hole.

- 4 The scatter in the spacing and burden errors at the face is mostly about half of the scatter at the bottom, 3.5 versus 7.0 cm. This implies that the geometry of the cuts is more disordered at the bottom. This difference may be converted to a scatter in the alignment (Ouchterlony, 1992; equation 6) of 1-2 cm/m.

Table 4: Statistics for spacing and burden deviations in the Buko cuts.

Test cases	Position	Spacing mm	1st burden mm	2nd burden mm	Both burdens mm
E1-9	face	4±37	-12±74	8±42	-2±59
	bottom	14±80	-47±72	24±62	-14±75
A1-23	face	17±35	-12±35	-5± 6	-8±32
	bottom	21±86	-48±74	-1±71	-25±75
All	face	13±30	-12±50	-2±30	-7±41
	bottom	19±57	-48±70	5±70	-22±75

From a blast malfunction point of view it is the extreme cases which interest us. From the drawings and the burden and spacing data in Appendix C, a drilling quality rating (DQR) has been designed. The quality rating has 5 classes; very bad (VB), bad (B), acceptable (A), good (G) and very good (VG). It is based on the following factors

- 1a Cut shape factor 1 - distortion and twist of cut with depth; qualitative judgement based on drawings in Appendix C and based on same class division as the DQR, VB to VG
- 1b Burden case factor; if case 3 then DQR is not better than A.
- 2 Cut shape factor 2 - cut area at bottom versus area at face; if the cut opens up towards the face then the flow of debris is easier than if the cut narrows down.
- 3a Distance between blast-hole and relief hole at the bottom factor; if ≤ 15 cm at the bottom, risk of venting to relief hole, then DQR is not better than A.
- 3b Cut size factor; if burden ≥ 30 cm there is a risk of an impaired breakage function.

As shown in section 1 above, Langefors and Kihlström (1963, 1978) state that plastic deformation or burning will impair the breakage if $B \geq 2.1 \cdot \varnothing$ even if the minimum charge requirement of equation 1 is met. Since for most relief holes $\varnothing = 102$ mm, the equivalent relief hole diameter is $\sqrt{2} \cdot \varnothing = 144$ mm and the critical distance is $B \approx 30$ cm.

Note also that in our testing, the burden was intended to be larger than 30 cm in the following cuts; E-9, A-11, A-15, A-16 and A-23. The DQR value as applied here doesn't distinguish between these cuts and those where the oversized burden was a result of poor drilling precision. The DQR values are given in Appendix C and Table 6. For cut E-4 no value could be obtained because of the loss of data for the right blast-hole.

3.5 The Charging of the Explosive

The measured charge concentrations and the charged and uncharged hole lengths are given in Appendix B. From these values and the charge diameter (emulsion cuts) or the blast-hole diameter (ANFO cuts), the apparent explosive density in the blast-hole can be calculated. These individual values are also given in Appendix B. Their statistics, range plus mean $\pm$ std deviation, are given in Table 5 below.

Table 5: Statistics of charge concentration and uncharged hole lengths.

Quantity	Uncharged length m	Charged length m	Charge concentration kg/m	Apparent density kg/m ³
E: planned	0.20-0.35	3.65-3.80	0.53-0.71	1050
actual	0.17-0.94	2.93-3.70	0.44-0.85	904-1207
actual	0.37 $\pm$ 0.18	3.42 $\pm$ 0.23	0.65 $\pm$ 0.11	1070 $\pm$ 80
A: planned	0.25-0.35	3.65-3.75	1.38-1.72	800-1000
actual	0.00-0.60	2.93-4.00	0.85-2.74	469-1874
actual	0.21 $\pm$ 0.17	3.63 $\pm$ 0.26	1.61 $\pm$ 0.36	970 $\pm$ 220

As usual in blasting tests, the actual charged and uncharged length values stray considerably outside their planned ranges. The charge concentration for the emulsion stays reasonably well within the planned range, as it should for a packaged explosive. The charge concentration for ANFO though strays well outside its planned range, even if the average value lies within. This is not unexpected for a compressible bulk explosive.

When the charge concentration values are converted to apparent density it easier to judge if they are reasonable. A histogram for density data is shown in Figure 14. The actual emulsion density, 1070 $\pm$ 80 kg/m³ agrees so well with the value given by the manufacturer that one could attribute the scatter to random measurement errors.

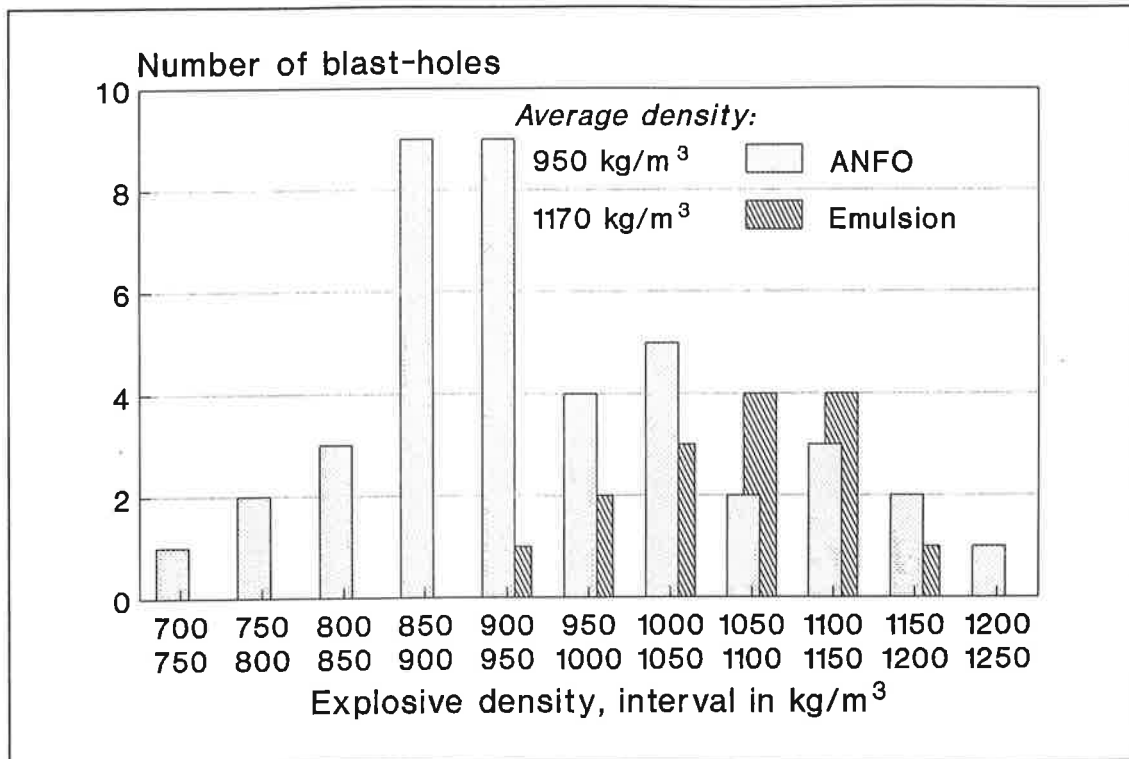


Figure 14: Histogram of apparent densities of explosives in blast-holes.

For ANFO some density values are obviously either much too low or much too high. For cut A-12 the apparent density values are 469 and 480 kg/m³ for the two blast-holes respectively. For cuts A-1 and A-7 the apparent densities are 1874 and 1516 kg/m³ for the first blast-holes respectively. The remaining densities are plotted in the histogram in Figure 14. It shows that the ANFO distribution has several peaks and thus that the scatter may have other causes than random measurement errors.

For pneumatically charged ANFO, a nominal in-hole density of 800-950 kg/m³ is to be expected and this corresponds well with the mode of the distribution in Figure 14 which lies between 850-950 kg/m³. The crystalline density of AN is 1730 kg/m³. ANFO can be made to detonate at 1350 kg/m³ (Nie et al., 1993), a practical limit is about 1200 kg/m³ however (Persson, 1975). So 1250 kg/m³ is a reasonable cut off value in the histogram.

The extreme values 1874 and 1516 kg/m³ plus the skewness towards higher values in the histogram could be explained by the tendency of a bulk explosive to penetrate into cracks and other cavities in the rock. Thus the drawing of cut A-1 in Figures 16 and 18 below e.g. show that the first blast-hole was drilled almost into the upper relief hole. The c-c distance is 12 cm, leaving less than 5 cm of rock between. It is easy to envisage a crack connection between the holes which would divert some of the ANFO from the blast-hole.

The density values for cut A-12, 469 and 480 kg/m³ indicate that the blast-holes were roughly half filled with ANFO which means that the ANFO was decoupled. The most plausible reason for this is some temporary malfunction of the charging equipment. Decoupled ANFO is known to detonate stably at least down to a 40% degree of filling if the ANFO is evenly distributed (Ouchterlony, 1992).

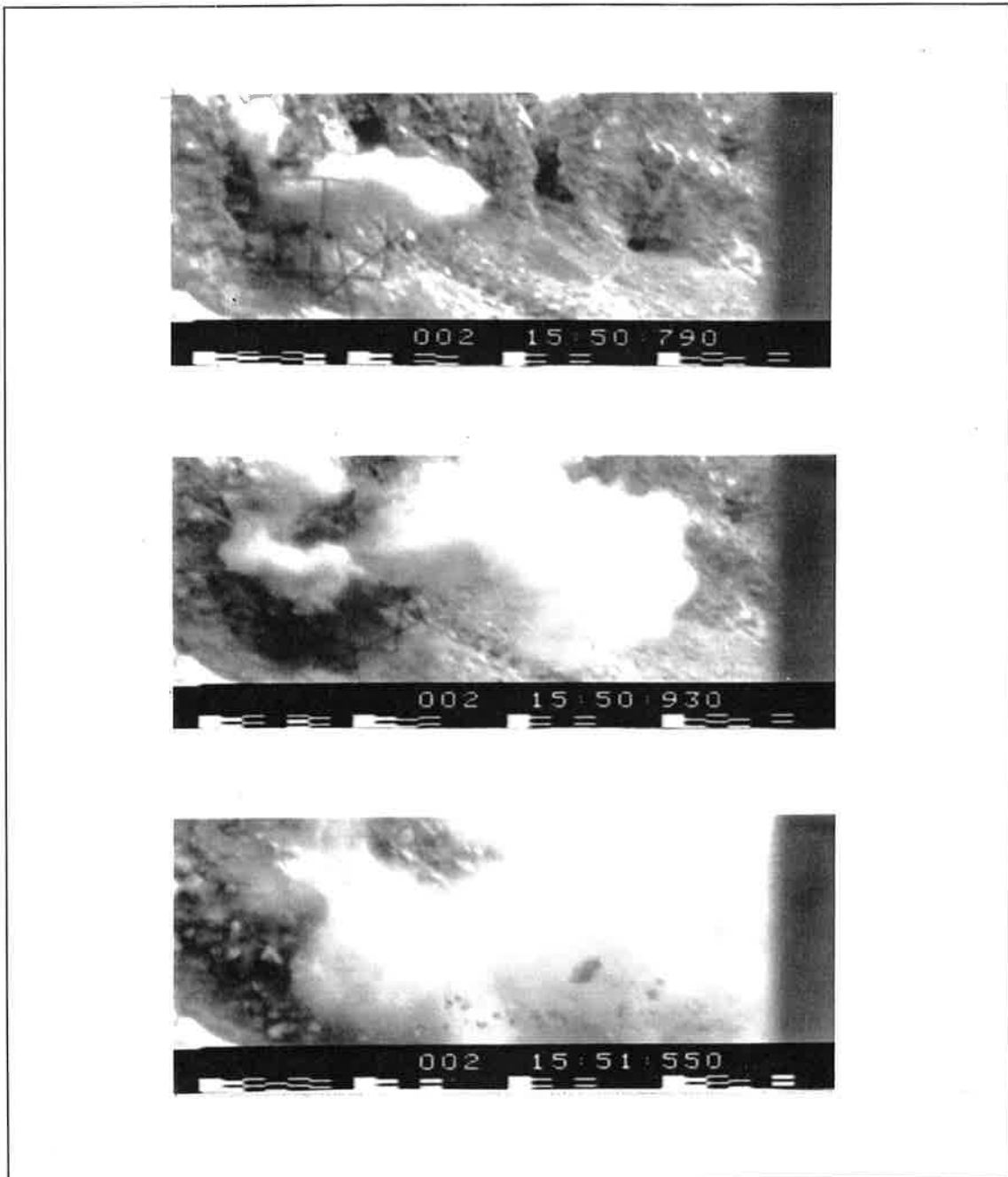


Figure 15: High speed video recording of a parallel hole cut during the blast.

3.6 The Blasting and Post Blast Measurements

Each blast resulted in a jet of blasting fumes and crushed material rushing out from the cut, see Figure 15. Very often parts of the bench face fell down as well, leaving only parts of the cut intact after the blast. After the blasts, the rock face was again cleaned by the back hoe which on average resulted in a total removal of about 2 m of the cut or half of its planned depth, see Figure 10 e.g. Then the cuts were photographed and the cavities investigated. In one case, cut A-23, the whole cut was obliterated by a bench face which caved when the back hoe was scaling it so no cavity remained to investigate.

Measurements of the cavity size and depths were made and related to the preblast hole depth measurements so that the advance could be calculated. The results were sketched on the test forms, see Figure 10. The size and appearance of the crushed rock in the cut cavity was noted and sometimes the throw was registered as well, see Appendix B.

The success or failure of breakage was judged according to the following criteria:

- | | |
|--------------------------|--|
| <i>Good breakage:</i> | A medium to large size opening with over 90% advance. |
| <i>Partial breakage:</i> | A medium to large size opening with under 90% advance. |
| <i>Bad breakage:</i> | A small opening and, by definition, no advance. |

By advance is meant the depth to which at least one of the 2 blast-holes have connected with the two relief holes and removed all the rock in between. This is the maximum depth to which the rest of a tunnel round may be expected to break out over the entire section area. The advance is also given relative to the average drilled depth of the four cut holes, two blast-holes plus two relief holes. See section 3.3 and Figure 2.

By a small opening is meant one where none of the blast-holes have connected with the relief holes, so by the definition above there has been no advance. The breakage function of the cut and the subsequent of the round is then completely absent. The malfunction reasons for this and their occurrence is discussed in sections 4.1 to 4.3.

By a large size opening one is meant one where both blast-holes have connected with the two relief holes and by a medium size opening where only one blast-hole has connected with both relief holes.

A deep medium size opening is classified as good breakage since the next pair of blast-holes in the round could be expected to clean out the remaining damaged burdens if they are not placed too far away. This gives the cut a safety margin against bad overall breakage.

4. TEST RESULTS AND DISCUSSION OF RESULTS

4.1 General Presentation of Test Results

Detailed test results were presented in Appendix B, including specific comments. The most relevant data are summarized in Table 6 and Figure 16 shows an overall plot of the remaining openings in the space of charge concentration versus burden of the first blast-hole. Figure 17 shows post blast photographs of 4 different cuts.

In Table 6 three general kinds of malfunction which have occurred are marked:

1. *Detonation failure* or impaired initiation function, either because of a non-detonating explosive or cap.
2. *Rifling* or impaired breakage function of one or both of the blast-holes, i.e. their failure to connect with the relief holes.
3. *Burning* or impaired fines removal, i.e. compacted or cemented crushed material remaining in the cut cavity.

Figure 16 is a general lay-out of the sketches from the test forms. The forms have been arranged as charge concentration versus burden for the firstly initiated blast-hole, which was the hole with the smallest burden on the rock face. This hole is marked out for each case. It can be seen that when the backhoe has cleaned the bench face, on average half of the drilled cut depth remains. The contour line of each sketch tells whether there was a malfunction during the blasting or not.

Figure 16 has been divided into three broad areas, successful blasts, unsuccessful blasts and an area with intermediate results. In rock blasting one never has the ideal situation, which is a distinct border between successful blasts giving good breakage in the sense of section 3.6 and unsuccessful ones giving bad breakage.

The main reasons for the unclear borders are the variability in rock conditions, in drilling precision, in charging and initiation procedures etc. which lead to the different kinds of malfunction observed.

To the successful blasts area belong cuts A-1 to A-6, A-8 plus E-1 and E-2. They all have large openings and good breakage except A-1 which achieved partial breakage due to a slightly lower advance than the acceptance limit, 87 versus 90%, and E-1 which gave bad breakage. The partial breakage of the border line case A-7 with only 83% advance may be due to the non-existent delay, 0 ms, between the two blast-holes.

Table 6: Summary of data relating to cut blasting results.

New cut no	Relief spac. mm	L-R dist. mm	DQR value	Charge density kg/m3	Initiat. funct.	Type of malfunction			Rel. advan. %	Cut cavity size	Break. result
						Det.fail	Rifling	Burn.			
E-1	200	195	B		H-cap	2			---	small	bad
E-2	211		B						103	large	good
E-3	330		B						101	medium	good
E-4	248		---						46	large	bad
E-5	434	553	B		N-cap	2			---	small	bad
E-6	197		A				1+2		---	small	bad
E-7	197		VB				1+2		---	small	bad
E-8	170		A					1	108	medium	good
E-9	341	780	B				1+2		---	small	bad
A-1	282	240	B	1874		2			87	large	partial
A-2	383		A						95	large	good
A-3	145		B						83	large	partial
A-4	331		G						101	large	good
A-5	258		G						104	large	good
A-6	264		G						96	large	good
A-7	205		VB	1516	0 ms				83	large	partial
A-8	190		G						102	large	good
A-9	89		B		1,25 m		1+2		80	small	bad
A-10	217		A	1201		1			98	medium	good
A-11	270	768	B				1		---	medium	good
A-12	242	379	VG	469	H-cap	2			---	small	bad
A-13	138		VB						104	large	good
A-14	241		G		50 ms				100	large	good
A-15	195		B						101	large	good
A-16	245	665	B	786			1		103	medium	good
A-17	374	645	B	1191					94	medium	bad
A-18	327		G						96	medium	good
A-19	324		A	1200	25 ms				79	large	partial
A-20	258		A						90	large	good
A-21	291		B					1	103	large	good
A-22	262		VB	726				1	103	medium	good
A-23	160		---						---	---	---

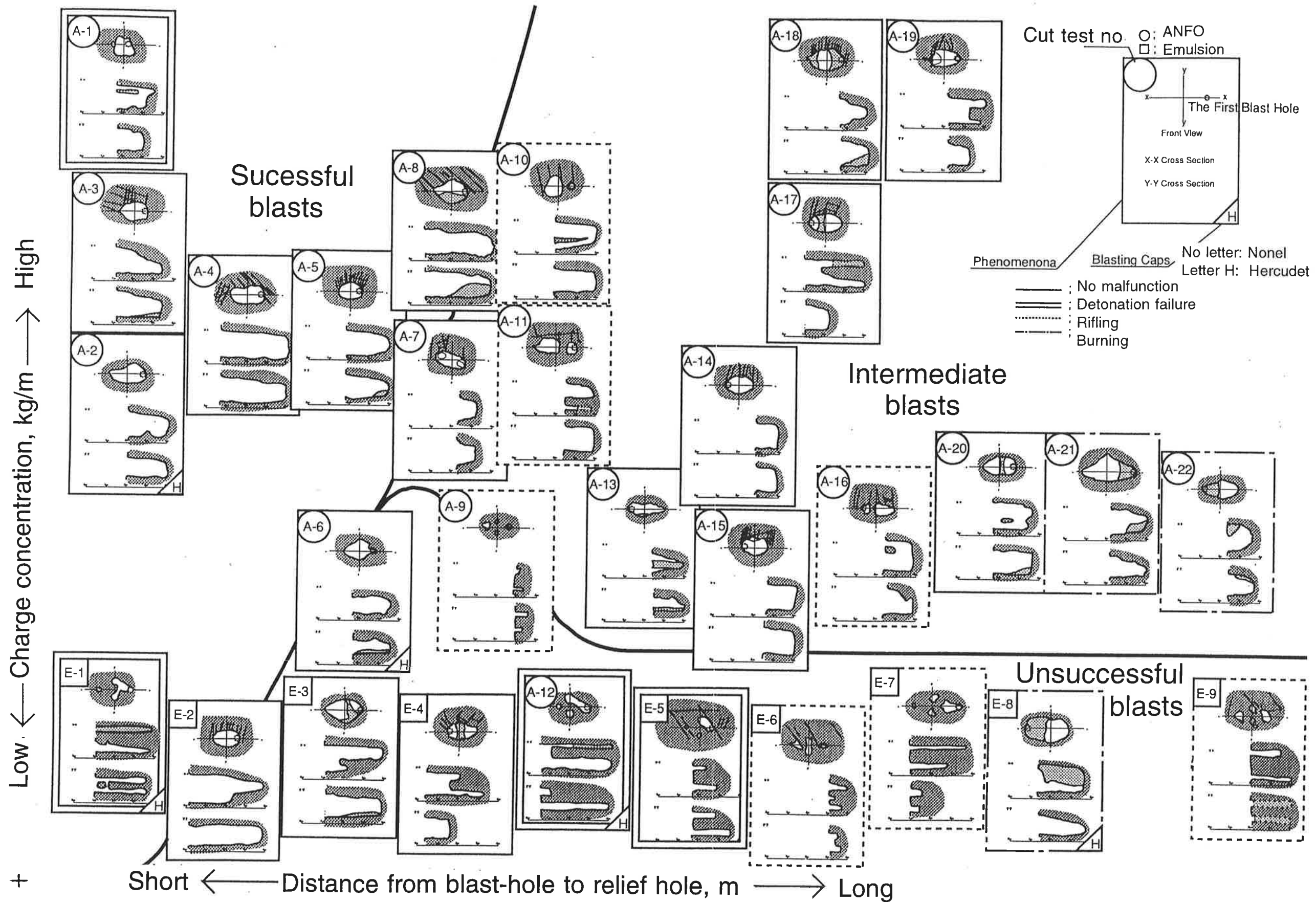


Figure 16: Spread sheet of remaining cut openings after blasting.



Figure 17: Post-blast photos of cuts; top row A-20 (new A-4) & A-18" (new A-20), bottom row A-34 (new A-1) & A-12 (new E-6).

To the unsuccessful blasts area belong all the remaining emulsion cuts, E-3 to E-9 plus one ANFO cut, A-12. Most of these emulsion cuts gave bad breakage, with the exception of E-3 and E-8. They have had low charge concentrations compared to the ANFO cuts. A-12 is a combination of low charge concentration and a non-detonation of the second hole. The latter was due to a non-functioning Hercudet detonator which left 2 m of undetonated ANFO in the hole.

As the burden in the cut increases, the success of the ANFO blasts diminishes. Cuts A-9 to A-11 are examples of different kinds of malfunction. In cut A-9 a clay-like schalstein seam made it impossible to put the primer deeper in the first blast-hole than 1.25 m from the face and it became a rifle shot. In cut A-10 the first blast-hole seems to have vented into one of the relief holes at the bottom. This has probably lead to detonation failure rather than a rifle shot. The same mechanism has probably been active in cut A-1. Cut A-11 is a rifle shot, i.e. a case of the first blast-hole detonating, as substantiated by VOD measurements, but failing to break out to the relief holes.

For some of the remaining ANFO cuts with large burdens we observed either burning or the formation of bridges in the cut. Many should however reasonably have been classified as successful blasts in the sense that the breakage was judged to be good.

This broad classification into successful blasts area, unsuccessful blasts area and blasts with intermediate results area is relatively superficial and must be supplemented by a detailed discussion of the drilling and charging results and of the different types of malfunction which have occurred.

4.2 Discussion of Influence of Drilling Quality Rating

A drilling quality rating (DQR) was introduced in section 3.4 and the values are given in Table 6. A simple correlation diagram between the breakage results and the DQR values is given in Table 7. Cut E-4 which has no DQR-value is given in parenthesis.

A closer inspection of Table 7 shows that the breakage being less than good usually is not correlated directly with the drilling quality but with an impaired initiation function. Some reasons for an impaired initiation function are:

- | | |
|---|-------------------|
| • Initiation failure in detonator | E-1, E-5 and A-12 |
| • Detonator position at clay seam | A-9 |
| • Extreme apparent density of ANFO in holes | A-1, A-7 and A-12 |
| • Blast-hole venting to relief hole | A-1. |

Table 7: Correlation diagram between drilling quality rating and breakage.

Breakage	Drilling Quality Rating				
	VB	B	A	G	VG
Good	A-13, A-22	E-2, E-3, A-11, A-15, A-16, A-21	E-8, A-2, A-10, A-20	A-4, A-5, A-6, A-8, A-14, A-18	
Partial		A-1, A-3	A-19		
Bad (E-4)	E-7, A-7	E-1, E-5, E-9, A-9, A-17	E-6		A-12

For other cuts the breakage function may have been impaired by:

- Planned large spacings and/or burdens E-9, A-7 and A-17.

The fines removal may, in turn, have been impaired by:

- Short inter-hole delay, ≤ 25 ms A-7, A-19
- Burning E-8, A-21 and A-22.

It is obvious that burning is not an indication of bad breakage in the Buko tests.

For cuts E-4, E-6 and E-7 there are no associated malfunctions which can explain the bad breakage. For E-7 the very bad drilling may be the direct explanation.

An important conclusion is that an ordinary drilling quality (DQR = B-VG) doesn't influence the breakage result in Buko limestone directly. A very bad drilling may influence it, but in the case of cuts A-13 and A-22 it has not. By very bad (DQR = VB) drilling is meant a badly twisted and distorted breakage geometry as for E-7, A-7, A-13 and A-22, which possibly also narrows down towards the face as for A-7, A-17 and A-22.

However, bad drilling may be the cause of the observed detonation failure in cut A-1 e.g., see below, and thus an indirect cause of less than good breakage.

4.3 Discussion of Malfunction Mechanisms

Different malfunctions occurred in 14 out of the 32 blasts reported here. As pointed out above, one cause may be bad drilling. Figure 18 shows the face and bottom geometries

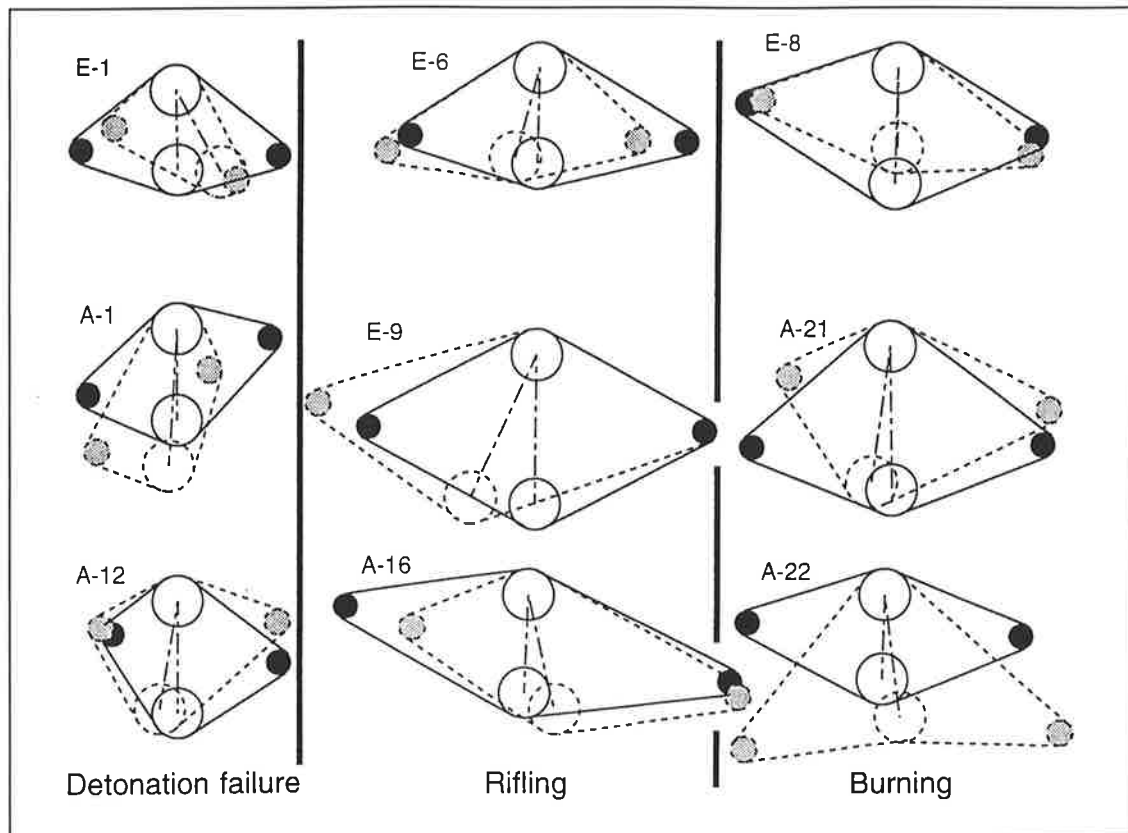


Figure 18: Cut geometry at bench face and bottom (dashed lines) for selected cuts.

of 3 cuts in each category. The solid lines and filled circles depict the face and the dashed lines and dotted blast-holes depict the bottom. The upper relief hole has been used as a reference. The geometries of all cuts are shown in Appendix C, but there solid lines depict the bottom. Table 6 is a correlation table between the breakage results, the drilling quality, other deviations from normal conditions and the observed malfunctions.

Detonation failure: The cuts in which we judge detonation failure to have occurred are E-1, E-5, A-1, A-10 and A-12. As mentioned above A-1 and A-10 are cases where at least the second and first blast-hole respectively have vented prematurely to a relief hole at the bottom, see Figures 16 and 18. ANFO is believed to be more sensitive to such venting because its critical diameter is much larger than that of packaged emulsion explosives. The venting has probably caused detonation failures in the ANFO.

The first blast-hole in cut E-1 would also seem to have vented, compare Figures 18 and 16. As the post blast sketch shows however it has broken out to both relief holes so the venting didn't lead to a detonation failure. The reason for this behavior is probably that the emulsion explosive is not as sensitive to decoupling or loss of confinement, c.f. the ANFO cases above.

The second blast-hole in cut E-1 failed to detonate however, both the explosive and the primer remained in the hole despite that the Hercudet tubing had visible signs of reaction. The reason might be that the detonator had become dead-pressed by the shock wave from the first blast-hole. The comparatively close distance between the two blastholes at the bottom of the cut, about 20 cm, supports this conjecture.

In cuts A-12 and E-5, the second blast-holes have also failed to detonate. Again explosive remained in the hole afterwards and the Hercudet and Nonel tubings respectively showed visible signs of reaction. Deadpressing of Nonel caps is known to have occurred (Nie et al., 1991) but E-5 is the only case out of 25 where this happened in our tests. The distance between the blast-holes at the bottom of cut E-5 is so large however, 55 cm, that deadpressing seems unlikely.

For the 7 Hercudet initiated cuts, 2 are possible cases of dead-pressed detonators in blast-hole no 2. Two reasons which might explain the relative sensitivity of the Hercudet caps are its more complicated construction (Persson et al., 1994) and the fact that the primary explosive is DDNP, diazodinitrophenol, rather than lead azide.

Rifle shots: The cuts in which we judge rifle shots to have occurred are E-6, E-7, E-9, A-9, A-11 and A-16. As one would expect, Figure 18 shows that the actual burdens of the rifling blast-holes are larger than for the detonation failures. For the emulsion cuts E-6 and E-9 the reason for this malfunction might be too low a charge concentration, see the testing matrix in Figure 8.

For the ANFO cuts the situation may be the same, but since the charge concentration is higher one of the blast-holes in cuts A-11 and A-16 has broken out to both relief holes, giving a medium size opening all the way to the bottom. Thus the breakage has been classified as good since the next pair of blast-holes could be expected to clean out the remaining rock. The same is probably true of cut A-10.

Figures 17, 18 and Appendix B show that the relationship between burden and breakage isn't simple. In cut A-16 the 1st, left hand blast-hole with a smaller burden at the bottom failed to break out whereas the 2nd blast-hole with a larger burden did. The same is true of cuts A-10 and A-11, considering that the right hand blast-holes detonated first. It seems reasonable to assume that the first blast-hole damages the surrounding rock so that this effect is possible.

A similar effect of damaged rock remaining in place and being removed by subsequent blasts is reported by Singh (1995) in the blasting of burn cuts.

In cut A-9 we can not give any other explanation of the failure of the blast-holes to break out than the effect of the clay seam. It caused the primer to be positioned at 1.25 m depth in the first blast-hole and might possibly have caused a pressure drop in the gases by allowing partial venting of them into the seam.

Burning: The cuts in which burning or cementation of rock material occurred are E-8, A-21 and A-22. What they have in common is firstly the large burden, leaving a large opening around the 1st blast-hole. Secondly, the charge concentration in the 1st blast-hole was relatively low and, thirdly, it is material from the 2nd blast-hole which has cemented to the wall behind the 1st hole.

One might conclude that this is an effect of choking, i.e. too many rock fragments having to be pushed out by too small charges so that the fragments from the 1st blast-hole are still cluttering the relief volume when the 2nd blast-hole detonates. It doesn't seem to be a pure timing effect though since the cuts initiated with shorter delays, i.e. A-7 (0 ms), A-19 (25 ms) and A-14 (50 ms), show no signs of burning.

The breakage in cut A-22 was judged to be good even if the opening was judged to be only of medium size. The breakage in cuts E-8 and A-21 were also judged to be good. Thus, as noted in section 4.2, burning doesn't seem to impair the fines removal.

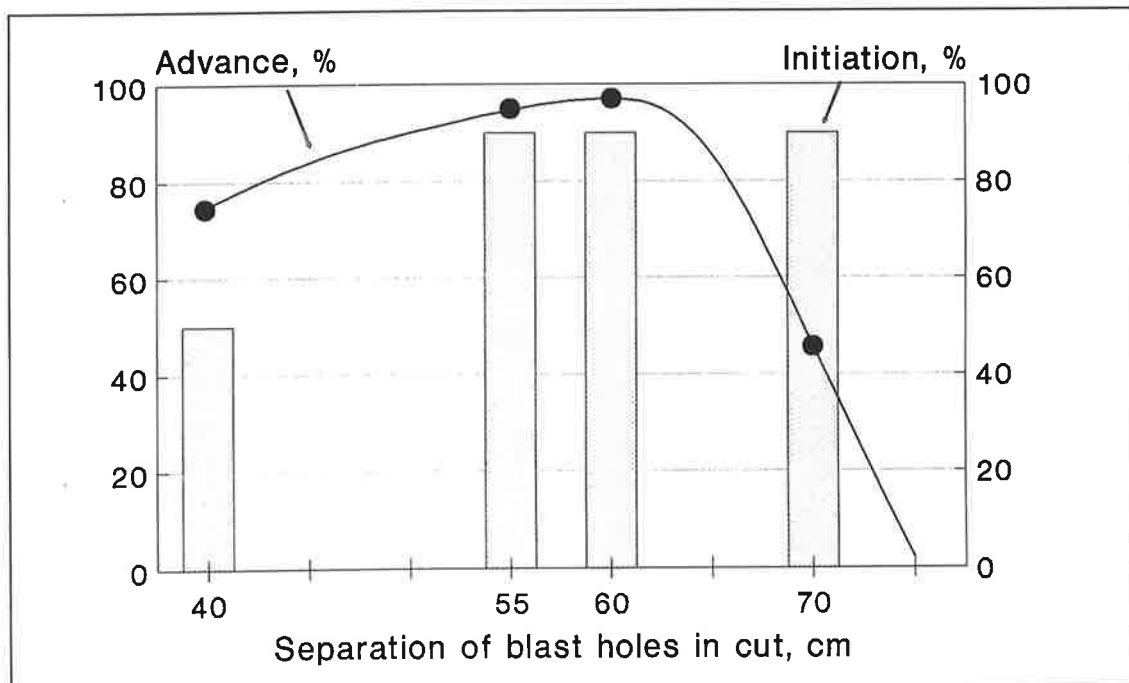


Figure 19: Advance and initiation function of long rounds with parallel hole cuts.

This leaves impaired initiation function and impaired breakage function as the two important threats to good cut breakage and consequently to good advance in a tunnel round. Figure 19 above (Niklasson and Keisu, 1993b) shows a plot of initiation function, as measured by PPV-records, and breakage function, as measured by the relative advance (in 7.5 m long rounds with parallel hole cuts) versus blast hole separation, which equals twice the intended burden in our case.

When the burden is too small the initiation function is impaired and the advance goes down, when the burden is too large the initiation function is good but the breakage function is impaired and the advance goes down. There seems to be an optimum where both functions are high and the resulting advance lies in the 90-100% interval.

4.4 Burden versus Charge Concentration

The function of the large uncharged relief holes in the cut is to serve as free faces towards which the first blast-holes can break out and as channels for the removal of broken rock or fines. It usually is more practical to drill several moderately large relief holes than one very large so it is common for parallel hole cuts to have two relief holes, like the cuts studied in our tests. Recently though, extremely good results have been obtained with parallel hole cuts with a single very large, Ø 250-300 mm, relief hole (Fjellborg and Olsson, 1996).

Figure 20 shows a plot of the linear charge concentration in terms of ANFO equivalents versus the actual burden of the first blast-hole at the bottom of the cut with breakage as an extra parameter. Only those cuts with bad or partial breakage have been included for which malfunction or very bad drilling quality reasons for the results were not found, i.e. E-4, E-6, E-9, A-3 and A-17.

The cuts with large relief holes and good breakage are denoted by open symbols. Those with bad or partial breakage with crosses plus their cut numbers. The two cuts with small relief holes and good breakage, A-8 (Ø 48 mm) and A-13 (Ø 45 mm), are denoted by filled squares.

Compared with the planned situation in Figure 8 it is directly seen that the drilling quality and the variations in apparent density for the ANFO have caused substantial deviations from the planned testing matrix.

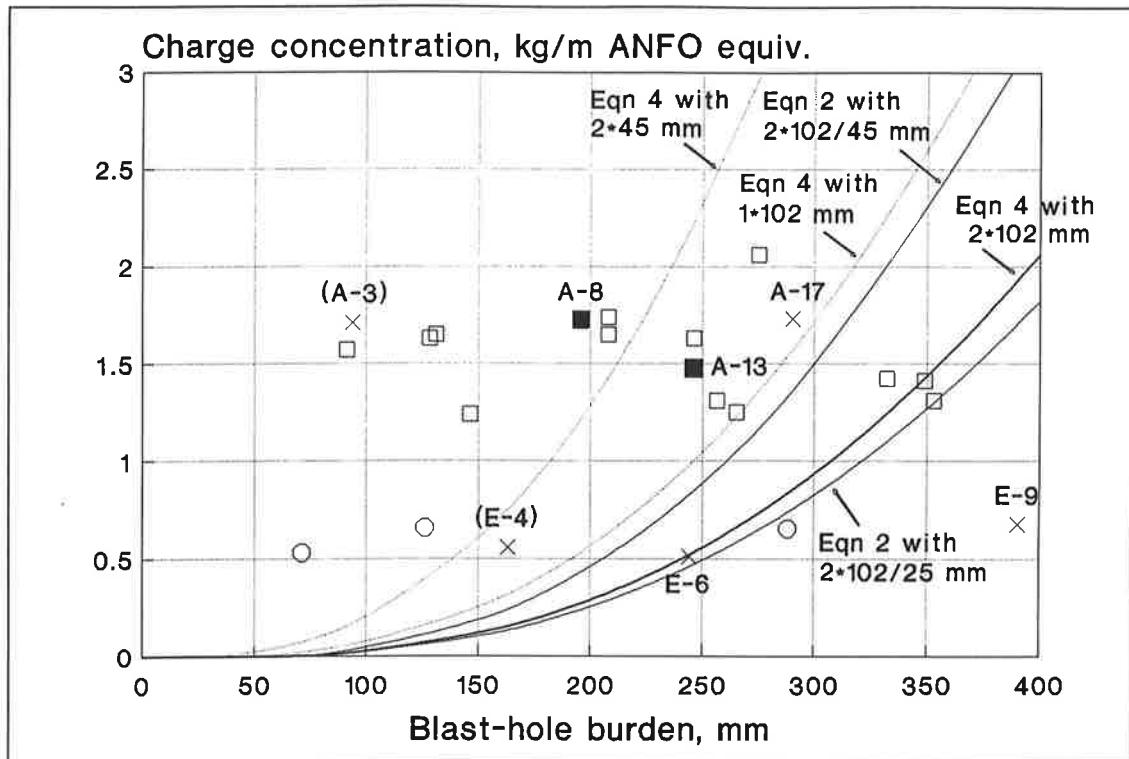


Figure 20: Charge concentration vs burden of 1st blast-hole at bottom of the cut.

According to section 3.4 above, Langefors and Kihlström's (1963, 1978) limits for impaired breakage due to plastic deformation or burning are about 30 cm for the large relief holes and slightly less than 15 cm for the small relief holes. Several cuts with good breakage have burdens exceeding these limits. Hence Langefors and Kihlström's limits do not seem to apply to the Buko limestone despite the fact that a limestone generally deforms in a more plastic manner under confinement than igneous rocks do.

Note also that three blasts with malfunctions for which the first blast-hole failed to break out to the relief holes, A-10, A-11 and A-16, were judged to give good breakage because the second blast-hole did break out to nearly full depth. This redundant breaking capacity is an important advantage of the symmetrical parallel hole cut over cuts like the spiral cut (Koorem, 1992) which try to optimize the volume broken by each blast-hole.

In cuts E-6 and E-9 it is reasonable to assume that a low charge concentration, or large burden, is the main cause for the first blast-hole failing to break out to both relief holes.

The bad breakage in E-4 has no immediate explanation. A partial data loss has prevented a complete evaluation of the drilling quality. The remaining data gives no reason suspect that the DQR would be very bad, but the point is best left open.

Most ANFO cuts in Figure 20 gave good breakage but two didn't, A-3 and A-17. For A-3 the small relief holes, Ø 48 mm, may have played a role. Cut A-13 is an anomaly in that the breakage was good despite the large burden, B = 245 mm, and the very bad drilling.

There are several ANFO cuts with a good breakage that have a larger 1st burden than cut A-17 with bad breakage. No other aspect than the spacing between the relief holes seems to explain as well why cut A-22 gave good breakage when A-17 did not.

Obviously there is a largest spacing when the two relief holes stop cooperating in the breakage even when the drilling quality otherwise would permit it. Examples of this are probably the cuts E-5, E-9 and A-17, see Figure 16 and Appendix C. For all three at least 3 of the blast-hole to relief hole c-c distances exceed 30 cm. The relief holes spacing values are 434, 341 and 374 mm respectively, see Table 6.

For cuts E-3 and A-18 with good breakage and hence probably cooperating relief holes, their spacings were 330 and 327 mm respectively. Thus a reasonably safe upper limit for the spacing is 30 cm which leaves a bridge of rock of slightly less than 20 cm between the relief holes.

As a consequence the blast-holes in cut A-17 in Figure 20 should be judged as breaking towards individual relief holes. The smallest c-c distance is 290 mm between the first blast-hole (left hole) and the lower relief hole and this is the burden used in the figure.

The next question is how well the data in Figure 20 is modelled by the charge concentration equations 1 and 2 in section 1. The lowest curve is given by Holmberg's equation 2 with 2·Ø 102 mm relief holes, D 25 mm blast-holes and $c = 0.35$, as was the case for half of the emulsion cuts. The curve is reasonable dividing line between emulsion cuts with good and bad breakage. The corresponding line from equation 2 for ANFO cuts, 2·Ø 102 mm relief holes and D 45 mm blast-holes, is however not a reasonable dividing line since three cuts with too large burdens gave good breakage.

To Langefors and Kihlströms's original prediction, equation 1, we have added the rock constant and weight strength factors of equation 2 to be able to compare our two different explosives in the Buko limestone. This gives

$$I_a = 1.56 \cdot (B - \emptyset/2) \cdot (B/\emptyset)^{1.5} \cdot (c/0.4) / s_{\text{ANFO}} \quad (4)$$

Using again $c = 0.35$ one obtains the thick solid curve irrespective of blast-hole diameter. It thus replaces the two thinner solid curves from equation 2. An inspection of Figure 20 shows that this line is a quite reasonable dividing line between cuts with good and bad

breakage, for both ANFO and emulsion. To be completely certain we would have liked to have tested a couple of ANFO cuts with burdens that were too large.

It may also be seen that cut A-17, which was judged to break towards individual relief holes, falls very near the prediction line of equation 4 for 1- $\varnothing$ 102 mm relief hole. Further, the prediction line of equation 4 for 2- $\varnothing$ 45 mm relief holes goes right between the filled squares for ANFO-cuts with small relief holes.

Altogether, we conclude that equation 4 is a better prediction for the necessary charge concentration to obtain good breakage than Holmberg's equation 2 is.

There are several drawbacks with this equation however. Firstly it is not dimensionally consistent in that the constant 1.56 must have the physical dimensions kg/m². A proper dimensional analysis of cut blasting would probably set this straight.

Secondly, neither of equations 1, 2 or 4 account for any effect of the decoupling of the explosive from the blast-hole wall. We have compared two test series, one with decoupling and one without, but with different explosives and the results are reasonably well explained in terms of difference in ANFO equivalent charge concentration without including the difference in the decoupling.

Decoupling causes a considerable decrease in the borehole pressure. Persson et al. (1994) estimate the pressure to be proportional to $(\varnothing_{\text{exp}}/D)^3$. For a $\varnothing_{\text{exp}} = 25$ mm charge in a $D = 45$ mm hole this would give a pressure decrease of over 80% and if the breakage capacity is proportional to the borehole pressure, the decrease would be substantial.

It is however known that the breakage capacity, measured as critical burden e.g., in bench blasting with an extended charge of given diameter may increase if the hole diameter is increased (Persson et al., 1969). They tested $\varnothing_{\text{exp}} = 11$ mm charges of a high VOD explosive with VOD = 6670 m/s in blast-holes with $D = 12, 22, 29$ and 45 mm and found that maximum breakage occurred for the 22 mm hole and then decreased again as the hole size increased.

No explanation for this observation is given but the test records show that the hole diameter after the blast has increased considerably for the 12 mm hole but not for the other ones. The resulting hole diameters lay in the range 25-40 mm so the energy wasted in crushing the blast-hole walls may account for some of the loss in breakage capacity. The same effect could be expected in cut blasting.

In order to correctly model the balance between a loss of breakage capacity with increasing decoupling, due to the pressure drop, with an increase of breakage capacity due to less crushing one probably needs accurate hydrodynamic computations with a code which incorporates all the necessary phenomena; explosion dynamics, decoupling, deformation, crushing and fracturing of the rock, gas flow and movement of fines, in three dimensions. Such codes are beginning to appear, Minchinton and Lynch (1996).

The use of these codes to improve the design of cut blasting would be welcome because the effect of parameters like VOD, decoupling, point of initiation, delay time, rock strength etc could be investigated relatively easily. Their predictions need to be backed up by careful experimentation before the codes can be trusted though.

With such a code one could also model the sequence of events in cut blasting. Of special interest would be to find out under what conditions the two relief holes cooperate in the breakage and if the design rule that $\varnothing_{\text{eff}} = \varnothing \cdot \sqrt{2}$ is motivated.

4.5 Fragment Size and Throw

A visual judgement of the size of the rock fragments or fines remaining in or at the cut cavity was made for nearly all cuts. For some, the color of the fines and their range and median values were also estimated. This was done for 5 emulsion cuts, E-2, E-4 to E-6 and E-9, as well as for 14 ANFO cuts, i.e A-3 to A-5, A-7, A-8, A-10, A-11, A-14 to A-19 and A-23. See Appendix B. Some of these cuts had given bad breakage. For cuts A-10 and A-11, different values were observed for the right and left hand blast-holes and they are given separately as 10:1 and 10:2 respectively 11:1 and 11:2 in the shaded area.

The statistics for the median values of the fines, given as mean_{no of data} $\pm$ std deviation or just as a mean, are given in Table 8 below. Note that the number of data points for ANFO is $14+2 = 16$.

Table 8: Statistics for fines median values, by different groupings

Grouping	Emulsion	ANFO	Black fines	Black- grey	Grey	Grey- white
fines, mm	$9.4_5 \pm 6.8$	$7.0_{16} \pm 5.6$	20.0	13.3	8.0	3.3

The size of the fines thus correlates with two factors, color and explosive. The correlation with color is shown in more detail in Figure 21 where the ranges and medians of the observed fines are grouped according to their color.

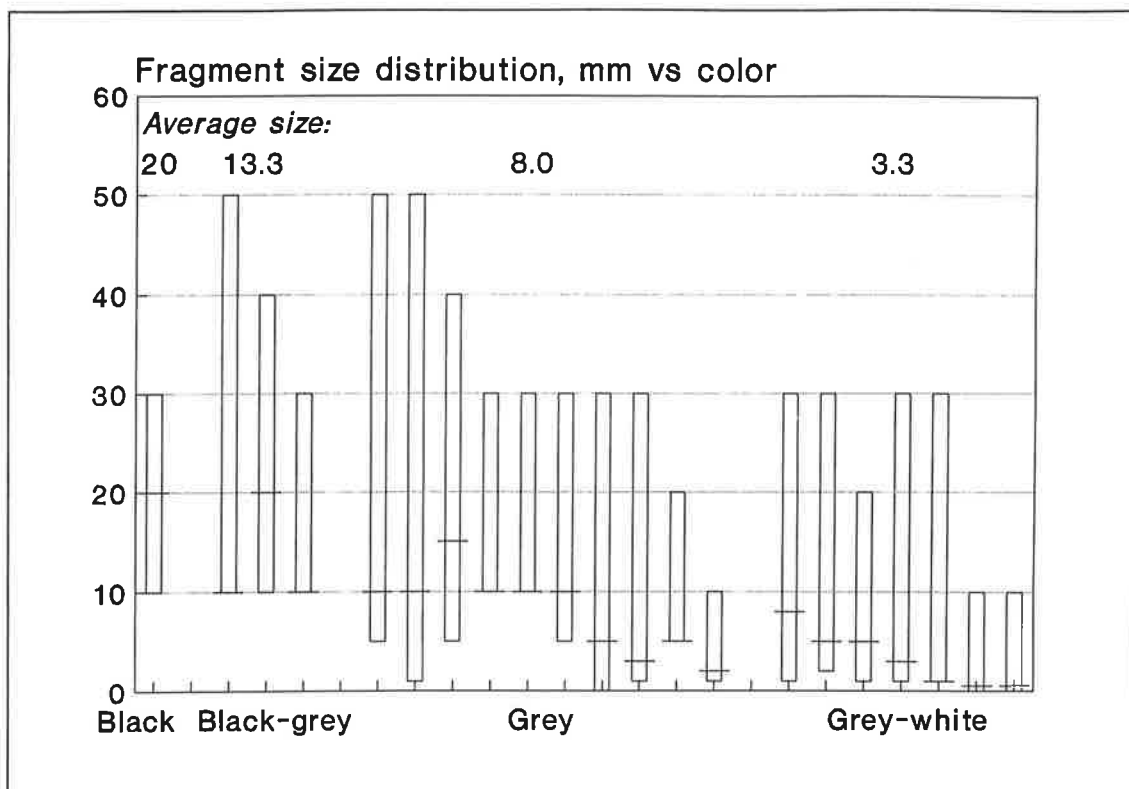


Figure 21: Plot of fragment size distribution, range and median, vs color of fines.

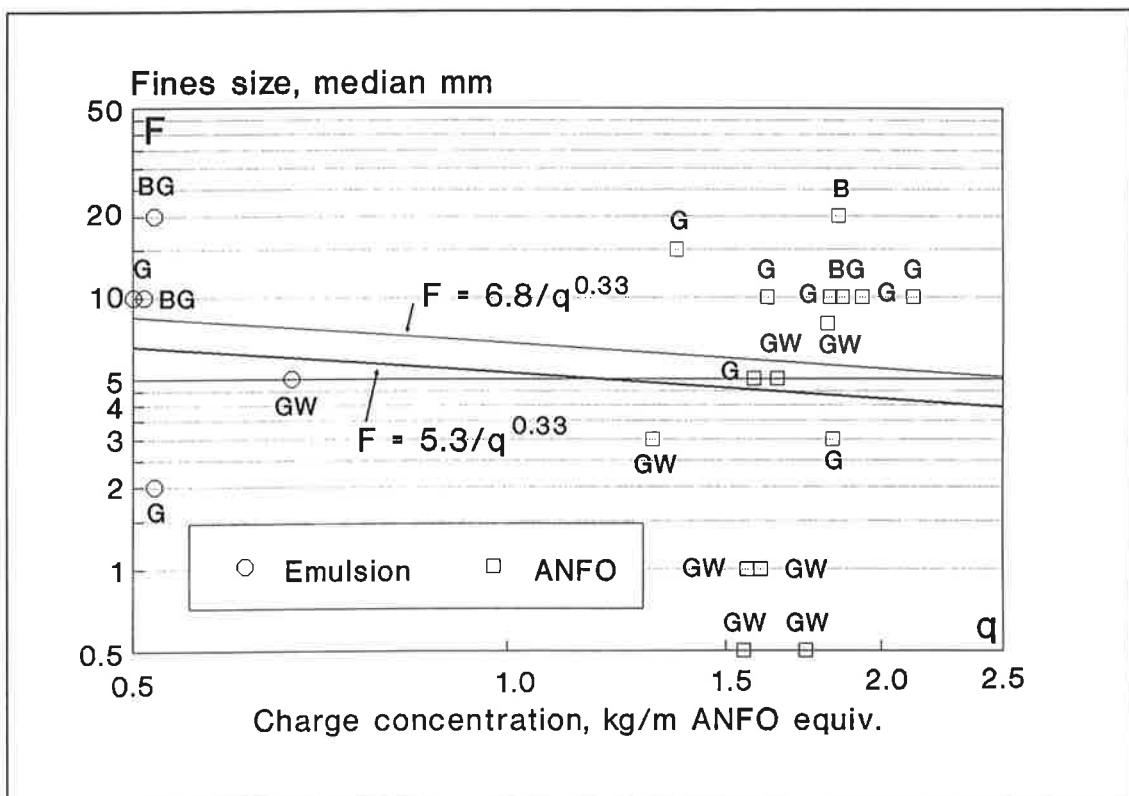


Figure 22: Plot of median fines size versus charge concentration.

A plot of fines size versus charge concentration is given in Figure 22. A linear least squares regression gives the result that

$$F \approx 5.3/q^{0.33}, \quad (5a)$$

where q is given in kg ANFO equivalents per meter and F in mm. This equation is given as the broad line in the figure. Because the basic data are median values, a median line which passes through the emulsion and ANFO averages in Table 8 has also been drawn in the Figure. It's equation is

$$F \approx 6.8/q^{0.33}. \quad (5b)$$

The scatter is large so the significance of the exponent value 0.33 might be doubted. In fact a 95% confidence interval for the difference between the emulsion and ANFO mean values, based on Student's t -distribution with $16+5-2 = 19$ degrees of freedom, becomes

$$\text{Fines(Emulsion)} - \text{fines(ANFO)} = 2.4 \pm 6.3 \text{ mm.}$$

Since the value 0 is contained in the interval there is no significant difference between the Emulsion and ANFO fines sizes. The implication of this is that the difference in charge concentration between the ANFO and emulsion cuts is too small to bring out any difference in fines size which is significant compared to the scatter in the data.

However, trench blasting is also a confined blasting situation. Early Swedish data (Olsson, 1954) has been cast in the following form

$$F = 280/q'^{0.56}, \text{ where} \quad (6)$$

q' is the specific charge in kg/m^3 . A transformation of our data, taking into consideration that the emulsion cuts were on average somewhat smaller than the ANFO cuts at the bottom, 0.058 m^2 versus 0.062 m^2 , would increase our exponent somewhat to roughly 0.4. This is different from 0.56 but not unreasonable considering the scatter in the data.

The correlation between the color of the fines and charge concentration also been illustrated in Figure 22. It could be that the same underlying factor as above, i.e. charge concentration, were operating. However it seems more reasonable to assume that the color of the fines is a measure of the success of the detonation. Blackish fines would then be an indication of an imperfect detonation with insufficient oxygen present.

If this were the case then one would also expect a minor loss of energy, VOD and pressure in the explosive during an imperfect detonation. An indication that the early part of the blast event is responsible is the fact that cuts with bad or partial breakage have median fines sizes that differ very little from the values for cuts with good breakage. This then could explain why the darker fines tend to be larger and consequently color variations would be more or less synonymous with a large scatter in the fines size.

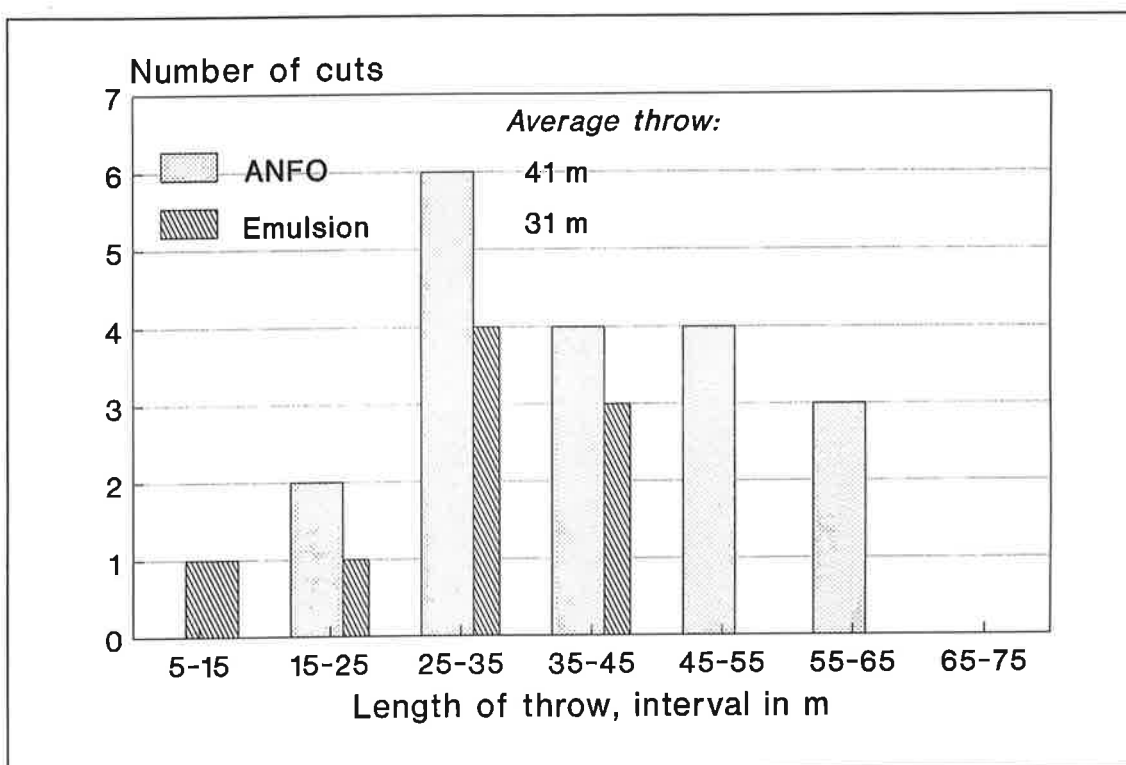


Figure 23: Histograms of throw values from Buko cuts.

The measured values of maximum throw are given in Appendix B. A histogram is shown in Figure 23. Some statistics, i.e mean_{no of data} ± std deviation, are given in Table 9 below.

Table 9: Statistics for throw data.

Grouping	Emulsion	ANFO	All cuts
throw, m	30.9 ₉ ±10.4	40.7 ₁₉ ±12.2	37.6 ₂₈ ±12.4

A 95% confidence interval for the difference between the emulsion and ANFO mean values, based on Student's t-distribution with 19+9-2 = 26 degrees of freedom, becomes

$$\text{Throw(ANFO)} - \text{throw(Emulsion)} = 9.9 \pm 9.7 \text{ m.}$$

Since the value 0 is not contained in the confidence interval, the difference is significant at the 5% level.

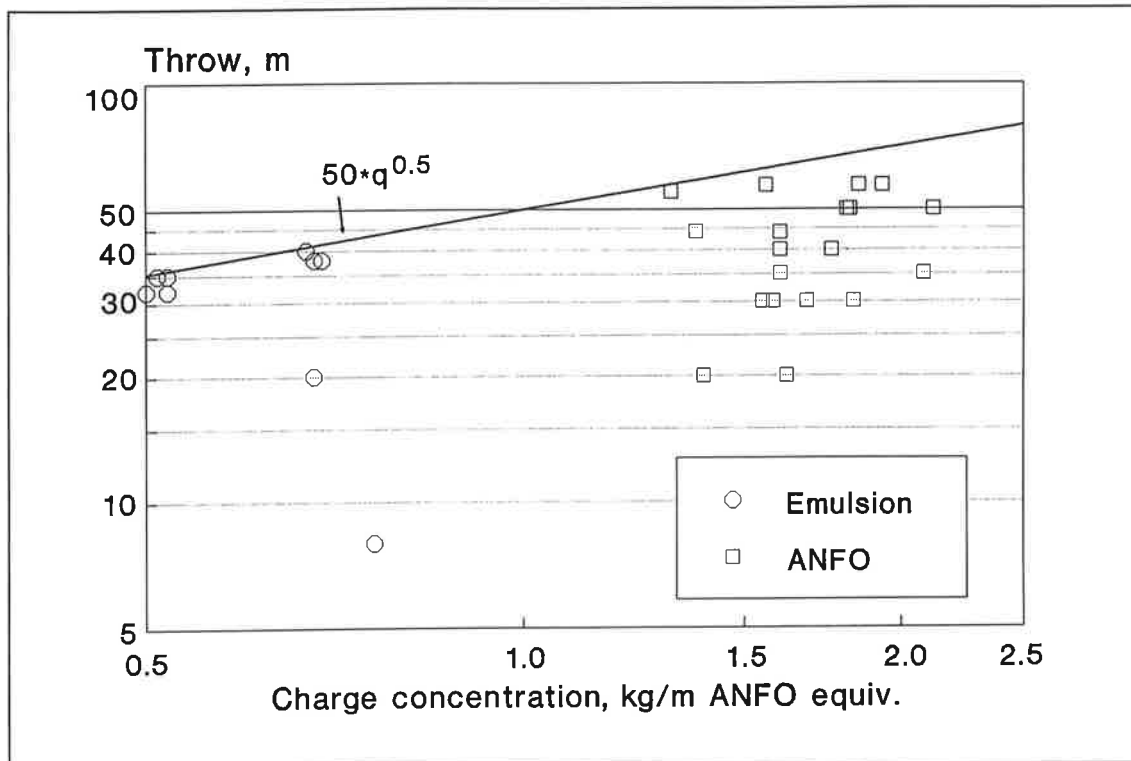


Figure 24: Plot of throw lengths versus charge concentration.

A curve of throw versus charge concentration is plotted in Figure 24. An upper limit to the maximum throw is given by the equation

$$T_{\max} = 50 \cdot q^{0.5}, \quad (7)$$

where q is given in kg ANFO equivalents per meter and $T_{\max}$ in m.

Assume that the throw is generated by a given ejection velocity V_o at the opening of the cut, which is situated in the bench face, h m above the horizontal ground. A simple ballistic estimate, disregarding the air drag and the ejecta rolling along the ground, is then that the throw is given by

$$T = V_o \cdot \sqrt{(2h/g)}, \quad (8)$$

where $g \approx 9.8 \text{ m/s}^2$ is the acceleration of gravity. An evaluation of the high-speed video recordings gave $V_o \approx 38\text{-}65 \text{ m/s}$.

According to equation 8, a maximum cut height of $h = 3.5$ m is consistent with maximum throw lengths of the ejecta of 32-55 m. According to Appendix B, the measured maximum lengths were 40 m for the emulsions cuts, cut E-9, and 57 m for the ANFO cuts, cuts A-4, A-8 and A-18, so the agreement with the simple theory is quite reasonable.

Other recordings of cut blasting (Ouchterlony, 1992) have given $V_o \leq 120$ m/s which would give throw lengths that are at least twice as long. Thus the values obtained in the Buko tests can not be used in establishing safety limits.

5. CONCLUSION

5.1 Conclusions of Buko Cut Blasting Tests

In the part of the Buko cut blasting tests reported here, 32 parallel hole cuts with 2 blast-holes ($D = 43\text{-}48\text{ mm}$) and 2 relief holes (mostly $\varnothing 102\text{ mm}$) at the center were drilled in limestone and blasted. The goal was to confirm earlier experience in blasting parallel hole cuts with emulsion explosives, to obtain the relation between charge concentration and burden and to obtain basic data for the use of ANFO in rounds with 4 m advance.

During the monitoring of the tests, the drill hole positions, their angles and depths were measured. A qualitative drilling quality rating was developed.

Two explosives were tested, bulk ANFO and a decoupled cartridged emulsion ($\varnothing_{\text{exp}} = 25\text{ or }30\text{ mm}$) with the brand name Hamamite. The amount of explosive and the charged depth in each blast-hole was measured. In some cases VOD-measurements were made during the blasts and in a few extra blasts, the blast sequence was filmed with a high speed video camera.

Apart from the explosive, several other factors were changed during the testing, the blast-hole diameter, the burdens of the blast-holes, the spacing between the relief holes, the type of detonator and the initiation delay between the blast-holes.

The post-blast measurements consisted of a careful investigation of the resulting breakage geometry, measuring the advance, the size and color of the fines and the throw of fragments. Different types of malfunction relating to initiation and breakage have been identified and analyzed.

The main analysis has consisted of finding correlations between the different input parameters and the blast results. The verification or rejection of existing formulas for the design of parallel hole cuts was a central part.

The main conclusions of these tests and the results analysis may be summarized as:

1. The drilling and blasting of cuts in a blast-damaged bench face was impaired by the collapse of the rock in a large area around the cut. For many cuts about half of the remaining cavity had collapsed before inspection was possible, in one case the whole cavity. Nevertheless, doing test blasts in free air rather than in a tunnel has significant advantages.

2. Even if these tests were made with dedicated personnel, the drilling precision has not been as good as is assumed in design formulas. The scatter in the collaring position and alignment angle distributions are approximately 5 cm and 2-2.5 cm/m. The compounded error at the bottom of a drilled cut is often 20-30 cm.
3. Due to a certain covariation in the drilling errors, the scatter in the burden and relief hole spacing values are somewhat smaller, about 3.5 cm at the surface and 7 cm at the bottom. The burden errors range from -15 to 13 cm at the face and -18 to 16 cm at the bottom of the cut. They are substantial in comparison with designed burden values which lie in the range 20-40 cm.
4. A qualitative measure of the drilling precision was developed. It is called DQR, the drilling quality rating, and has 5 classes from very bad (VB) to very good (VG). The DQR value is based mainly on the shape and size of the drilled cut. It was found that an ordinary drilling quality, meaning bad or better, doesn't influence the breakage result directly. It was also found that a very bad drilling may influence the breakage, but not necessarily so.
5. The control over the charging work was not so good. The actual charged length lay between 2.93-4.00 m whereas the planned range was 3.65-3.80 m. The apparent density of the cartridge emulsion agreed well with the nominal value 1050 kg/m³. That of the ANFO in several cases did not. Instead of a normal variation between 800-1000 kg/m³, the calculated values lay in the range 469-1874 kg/m³. This indicates that some holes were perhaps only half filled with ANFO and that in other holes, ANFO may have spilled into a connecting relief hole or connecting cracks.
6. The method used to make VOD measurements gave unsatisfactory results with VOD-values ranging from 1650-5350 m/s. More advanced instruments to make such measurements with exist however, see Chiappetta (1993) e.g.
7. Good breakage has been defined as meaning over 90% advance and a medium to large size opening of the cut cavity. An advance is considered to have occurred when at least one of the blast-holes has broken the burdens of and connected with both relief holes, because then the following blast-holes in a real round may be expected to clean out the remaining burdens. In this sense the parallel hole cut has a redundant breaking capacity and a cut like the spiral cut does not.
8. Three kinds of malfunctioning of the cut blasts were identified; detonation failure or impaired initiation function, rifling or impaired breakage function and burning or compacted crushed material remaining in the cut cavity. Observed burning did not

- correlate with bad breakage but both detonation failure and rifling did. Rifling occurs when the charge fails to break and remove the burden. Sometimes these breakage failures were caused by intentionally large burdens, sometimes by poor drilling.
9. Detonation failures with ensuing bad breakage have occurred because of detonator failure, because of a poorly placed detonator, because of extreme apparent densities of ANFO in blast-holes (see point 5 above) and because of a blast-hole with ANFO venting prematurely to a relief hole. Such venting has been observed when a blast-hole bottom gets too close to a relief hole. A premature venting of an emulsion filled blast hole doesn't seem to cause a detonation failure however, probably because the emulsion explosive is not as sensitive to decoupling or loss of confinement as ANFO.
 10. Bad or partial breakage resulted when the initiation delay between blast-holes was 25 ms or less. A delay of 75-100 ms is judged to be quite sufficient not to impair the breakage and the fines removal in the cut. 50 ms is probably sufficient.
 11. The cuts with good breakage, and those with bad or partial breakage for which neither malfunction nor very bad drilling quality reasons were found, have been plotted in diagram showing the linear charge concentration versus the burden of the 1st blast-hole. An analysis shows the following:
 - a The plastic deformation or burning limit of Langefors and Kihlström, which for most Buko cuts is about 30 cm, can be exceeded without a loss of good breakage.
 - b There is an upper limit for the spacing of the relief holes above which they may not cooperate in the breakage process. For the cuts in the Buko tests, with $D = 43-48$ mm blast-holes and $\varnothing 102$ mm relief holes, it is about 30 cm.
 - c Langefors and Kihlströms' original prediction equation for the charge concentration needed to break out a given burden, when updated with the rock constant and weight strength values, gives a better agreement with the Buko results than Holmberg's modified equation. Their equation has the drawback that it is not dimensionally consistent though.
 - d The results are reasonably well explained in terms of difference in ANFO equivalent charge concentration without including the difference in decoupling. Decoupling has counteracting effects on the breakage capacity of an extended charge, the balance of which are poorly understood. Ideally they should be incorporated in a prediction model and suitable hydrodynamic computer codes are beginning to appear.

12. An analysis of the fine rock fragments remaining in or at the cut cavity shows that the fines size correlates both with the explosive and with the color of the fines. It is natural to associate the explosive with the charge concentration in the Buko tests. The scatter in the data is however too large to bring out a difference in fines size which is significant. The correlation with color is stronger, black fines being the largest with an average size of 20 mm and grey-white fines being the smallest with an average size of 3.3 mm. Their color might be taken as an indication of how perfect the detonation was, a darker color indicating oxygen deficiency and a loss of energy, VOD or pressure.
13. The maximum throw of the rock fragments lies in the range 8-55 m. The throw from the ANFO cuts is significantly longer than the throw from the emulsion cuts. An evaluation of the high-speed video recordings gave an initial ejection velocity of the fragments of 38-65 m/s. A simple ballistic estimate with a cut height above ground of 3.5 m then gives a maximum throw of 32-55 m which is in good agreement of the measured values.

5.2 Requirements for Cut Blasting

The preceding analysis and conclusions allow us to write down the following requirements for a well functioning cut in the Buko limestone:

GOOD CUT DESIGN

OK if burden not too large and given by design rule, equation 4
and
if relief hole spacing not larger than 30 cm
and
probably not very sensitive to relief hole size

DRILLING

OK if drilling quality rating is bad or better
or
not OK if drilling quality rating is very bad or
if blast-hole and relief hole too close at bottom then
venting and
if ANFO then detonation failure
but
if emulsion then OK

DRILLING, ctd

or not OK

if blast-holes are too close then

if very close then sympathetic detonation of charges

or

if less close then dead-pressing of explosive and/or detonator

CHARGING

OK independently of stemming

and

if ANFO density 700-1200 kg/m³

else

not OK and

if too small charge concentration then rifling and no breakage

or

if bulk explosive deposited in cracks then disturbance of initiation function

INITIATION & DETONATION

OK if inter hole delay is 50 ms or larger

and

if detonator placed at bottom

else not OK

BREAKAGE

OK independently of burning

and

not OK if drilling quality is very bad

FINES REMOVAL & THROW

OK independently of burning

5.3 Suggestions for Future Work

Carefully conducted and instrumented blasting tests are necessary but difficult to make. In the Buko tests we have shown that Langefors and Kihlströms' equation for cut blasting could be used to design 4 m deep parallel hole cuts in limestone with ANFO and Nonel initiation. Thus we have gained confidence to continue testing the "Swedish method" in other Japanese sites, e.g. the Gorigamine tunnel (Koike et al., 1994), the Kuki tunnel (Sato et al., 1994) and the Shinfukazawa no. 2 tunnel (Okamura et al., 1995).

Some questions regarding the design of parallel hole cuts and how they function are still poorly understood. One of these is the cooperative effect of multiple relief holes. Another is the effect of decoupling. Still another is the effect of bad drilling and initiation delay. We believe that there is a need to develop better computer codes which can model this while at the same time carrying out careful tests to guide the development. The goal of such work should be to develop better design rules for parallel hole cuts.

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Appendix A: The CRIEPI Rock Mass Classification System

CRIEPI (Central Research Institute of Electrical Power Industry) proposed a rock mass classification system in the 1950's to assess the suitability or quality of solid rock masses for use as dam foundations. Since this is significantly affected by weathering, the CRIEPI classification originally focused on the degree of weathering and on comprehensibility and simplicity for practical use at candidate dam sites.

The rock mass quality is determined by primary and secondary factors. The primary one is related to rock type and age of formation as reflected by the degree of consolidation and diagenese. The secondary one concerns the degree of weathering, fracturing and alteration.

The classification system uses the following indices:

- discoloration of rock forming minerals
- hardness, judged from the sound generated by a hammer blow
- properties of fractures (aperture, filling materials).

Rock masses may be classified into four classes, A, B, C and D. Class C has three subgroups, C_H (high, close to class B), C_M (medium) and C_L (low, close to class D):

Class A:

The rock mass is very fresh and the rock forming minerals and grains have undergone neither weathering nor alteration. Joints are extremely tight and their surfaces have no visible signs of weathering. The sound from a hammer blow is clear.

Class B:

The rock mass is solid. There are no open joints and cracks (even with an aperture of 1 mm). The rock forming minerals and grains have undergone slight (a little) weathering and partial alteration. The sound from a hammer blow is clear.

Class C_H :

The rock mass is relatively solid. The rock forming minerals and grains have undergone weathering, except for the quartz. The rock is contaminated by limonite etc. The cohesion of joints and cracks has been slightly weakened and blocks of rock can be separated by a firm hammer blow along the joints. Clay minerals remain on the separated surfaces. The sound from a hammer blow is slightly dull (a little dim).

Appendix A: The CRIEPI Rock Mass Classification System, ctd

Class C_M:

The rock mass is relatively (somewhat) soft. The rock forming minerals and grains have been somewhat softened by weathering, except for the quartz. The cohesion of joints and cracks has been somewhat weakened and blocks of rock can be separated by an ordinary hammer blow along the joints. Clay materials remain on the separated surfaces. The sound from a hammer blow is relatively dull (somewhat dim).

Class C_L:

The rock mass is soft. The rock forming minerals and grains have been softened by weathering. The cohesion of joints and cracks has been weakened and blocks of rock can be separated by a soft hammer blow along the joints. Clay materials remain on the separated surfaces. The sound from a hammer blow is dull (dim).

Class D:

The rock mass is very (remarkably) soft. The rock forming minerals and grains have been softened by weathering. The cohesion of joints and cracks is almost non-existent (absent). The rock mass collapses from a light hammer blow. Clay materials remain on the separated surfaces. The sound from a hammer blow is very dull (remarkably dim).

The CRIEPI rock mass classification system applies mainly to hard rocks such as igneous rocks and consolidated sedimentary rocks. Recently it has been applied not only to dams but also to large underground caverns, tunnels, nuclear power plants, bridges and quarries, i.e. being increasingly adapted to various kinds of structures.

Data on the number of joints and cracks and the degree of weathering are pooled to evaluate the suitability of the rock mass for the structure. For underground caverns e.g., the degree of weathering is usually of little importance.

Appendix B: Comprehensive Tables of Test Data

New cut	Old cut	Delay time	Blast holes	Charge diam	Relief holes	Relief spacing		Blast-hole burden, bottom		
no	no	ms	mm	mm	mm	plan	bottom	plan	hole 1	hole 2
E-1	A-2'	100	45	30	102	200	200	200	35	160
E-2	A-10	75	45	25	102	200	211	200	71	280
E-3	A-4	75	45	30	102	300	330	200	126	250
E-4	A-11	75	45	25	102	200	248	250	163	---
E-5	A-13	75	45	25	102	250	434	250	223	330
E-6	A-12	75	45	25	102	250	197	200	244	280
E-7	A-5	75	45	30	102	300	197	300	286	220
E-8	A-3	100	45	30	102	200	170	300	288	280
E-9	108	75	45	30	102	300	341	350	390	390
A-1	A-34	75	43	43	102	200	282	200	80	160
A-2	A-28	100	48	48	102	300	383	200	91	220
A-3	B-6	75	48	48	48	200	145	250	94	250
A-4	A-20	75	45	45	102	300	331	200	128	270
A-5	A-36	75	43	43	102	300	258	200	131	150
A-6	A-18	100	45	45	114	200	264	200	146	260
A-7	113	0	48	48	102	250	205	250	177	250
A-8	B-5	75	48	48	48	150	190	200	195	250
A-9	A-19	75	45	45	102	200	89	300	202	300
A-10	A-35	75	43	43	102	200	217	300	208	370
A-11	105	75	48	48	102	200	270	400	208	560
A-12	A-26	100	48	48	114	200	242	200	219	160
A-13	A-18'	100	45	45	45	200	138	250	245	190
A-14	115	50	48	48	102	250	241	250	246	190
A-15	110	75	43	43	102	200	195	350	256	170
A-16	106	75	45	45	102	200	245	400	265	400
A-17	A-37	75	43	43	102	300	374	300	275	370
A-18	A-29	75	48	48	102	300	327	300	275	320
A-19	114	25	48	48	102	250	324	250	283	220
A-20	A-18"	75	45	45	102	200	258	250	332	150
A-21	A-21	100	45	45	102	300	291	300	349	210
A-22	A-27	75	48	48	102	200	262	300	353	340
A-23	105'	75	48	48	102	200	160	400	384	460

Appendix B: Comprehensive Tables of Test Data, ctd

New cut	Spacing at face	Burden at face		Depth hole 1	Unchar. hole 1	Charge hole 1	Charge size	Charge conc.	Appar. density
no	mm	mm	mm	m	m	m	kg	kg/m	kg/m3
E-1	178	226	219	3,87	0,94	2,93	2,50	0,85	1207
E-2	179	194	152	3,58	0,35	3,23	1,80	0,56	1135
E-3	263	149	143	3,72	0,17	3,55	2,50	0,70	996
E-4	210	200	250	3,86	0,30	3,56	2,10	0,59	1202
E-5	320	239	288	3,59	0,23	3,36	1,90	0,57	1152
E-6	224	331	276	3,97	0,27	3,70	2,00	0,54	1101
E-7	294	151	325	3,86	0,47	3,39	2,50	0,74	1043
E-8	254	310	290	3,97	0,33	3,64	2,50	0,69	972
E-9	315	341	376	3,89	0,42	3,47	2,50	0,72	1019
A-1	215	202	207	3,49	0,55	2,94	8,00	2,72	1874
A-2	320	192	243	3,95	0,00	3,95	6,20	1,57	867
A-3	221	190	215	4,05	0,30	3,75	6,40	1,71	943
A-4	317	200	210	4,02	0,35	3,67	6,00	1,63	1028
A-5	383	200	190	3,63	0,35	3,28	5,40	1,65	1134
A-6	227	286	154	3,78	0,30	3,48	4,30	1,24	777
A-7	294	255	230	3,79	0,00	3,79	10,40	2,74	1516
A-8	175	190	220	4,03	0,30	3,73	6,40	1,72	948
A-9	261	269	271	3,68	0,30	3,38	4,70	1,39	874
A-10	234	210	327	3,79	0,35	3,44	6,00	1,74	1201
A-11	189	390	375	4,05	0,30	3,75	6,20	1,65	914
A-12	242	195	168	3,95	0,30	3,65	3,10	0,85	469
A-13	192	236	260	3,69	0,30	3,39	5,00	1,47	927
A-14	278	231	243	3,81	0,00	3,81	6,20	1,63	899
A-15	214	370	330	3,96	0,00	3,96	5,20	1,31	904
A-16	211	401	443	3,84	0,00	3,84	4,80	1,25	786
A-17	283	230	289	3,73	0,03	3,70	6,40	1,73	1191
A-18	348	330	300	4,04	0,30	3,74	7,70	2,06	1138
A-19	241	292	247	3,96	0,00	3,96	8,60	2,17	1200
A-20	205	220	235	3,83	0,30	3,53	5,00	1,42	891
A-21	315	313	275	3,84	0,30	3,54	5,00	1,41	888
A-22	188	227	302	3,88	0,30	3,58	4,70	1,31	726
A-23	166	352	399	3,60	0,00	3,60	6,20	1,72	952

Appendix B: Comprehensive Tables of Test Data, ctd

New cut	Depth hole 2	Unchar. hole 2	Charge hole 2	Charge size	Charge conc.	Appar. density	Depth relief D	Depth relief U	Average depth
no	m	m	m	kg	kg/m	kg/m3	m	m	m
E-1	3,83	0,56	3,27	2,50	0,76	1082	3,85	3,80	3,84
E-2	3,90	0,25	3,65	2,00	0,55	1116	3,78	3,94	3,80
E-3	3,74	0,47	3,27	2,50	0,76	1082	3,72	3,99	3,79
E-4	3,13	0,20	2,93	1,30	0,44	904	3,79	3,70	3,62
E-5	3,69	0,19	3,50	1,90	0,54	1106	3,74	3,92	3,74
E-6	3,97	0,24	3,73	2,00	0,54	1092	4,03	3,88	3,96
E-7	3,89	0,37	3,52	2,50	0,71	1005	3,85	3,82	3,86
E-8	3,75	0,47	3,28	2,50	0,76	1078	3,79	3,36	3,72
E-9	4,02	0,46	3,56	2,50	0,70	993	4,18	4,18	4,07
A-1	3,39	0,35	3,04	4,40	1,45	997	3,37	3,55	3,45
A-2	3,82	0,00	3,82	6,20	1,62	897	4,00	4,00	3,94
A-3	4,10	0,16	3,94	7,70	1,95	1080	4,11	3,67	3,98
A-4	3,97	0,35	3,62	5,40	1,49	938	4,16	4,13	4,07
A-5	3,74	0,30	3,44	5,00	1,45	1001	4,03	3,71	3,78
A-6	3,65	0,30	3,35	4,30	1,28	807	3,77	3,65	3,71
A-7	3,69	0,00	3,69	5,50	1,49	824	4,08	3,87	3,86
A-8	4,02	0,29	3,73	7,40	1,98	1096	4,03	4,03	4,03
A-9	3,53	0,60	2,93	5,40	1,84	1159	3,80	4,00	3,75
A-10	4,06	0,35	3,71	6,00	1,62	1115	4,00	4,00	3,96
A-11	3,91	0,20	3,71	6,90	1,86	1028	4,00	4,03	4,00
A-12	3,87	0,30	3,57	3,10	0,87	480	3,94	3,92	3,92
A-13	3,65	0,30	3,35	5,00	1,49	938	3,63	3,68	3,66
A-14	3,93	0,00	3,93	6,20	1,58	872	3,95	3,98	3,92
A-15	3,98	0,00	3,98	5,20	1,31	900	4,12	4,07	4,03
A-16	3,70	0,00	3,70	5,50	1,49	935	3,84	3,70	3,77
A-17	3,99	0,03	3,96	5,70	1,44	991	3,92	4,03	3,92
A-18	4,03	0,31	3,72	6,70	1,80	995	4,18	4,20	4,11
A-19	4,00	0,00	4,00	5,90	1,48	815	4,07	4,13	4,04
A-20	3,87	0,42	3,45	4,70	1,36	857	3,81	3,97	3,87
A-21	3,89	0,30	3,59	5,00	1,39	876	3,83	3,86	3,86
A-22	3,73	0,35	3,38	6,40	1,89	1046	3,71	3,68	3,75
A-23	3,80	0,00	3,80	7,20	1,89	1047	3,70	3,65	3,69

Appendix B: Comprehensive Tables of Test Data, ctd

New cut	Absolute advance	Relative advance	Cut cavity	Cut breakage	Fines color	Fines size		Throw length
no	m	%	size			mm	mm	m
E-1	---	---	small	bad	---	---	---	8
E-2	3,90	102,6	large	good	G	1-10	2	---
E-3	3,83	101,0	medium	good	---	---	---	38
E-4	1,68	46,4	large	bad	G	5-30	10	32
E-5	---	---	small	bad	BG	10-40	20	35
E-6	---	---	small	bad	BG	10-30	10	35
E-7	---	---	small	bad	---	---	---	38
E-8	4,00	107,6	medium	good	---	---	---	20
E-9	---	---	small	bad	WG	1-20	5	40
A-1	3,00	87,0	large	partial	---	---	---	35
A-2	3,76	95,4	large	good	---	---	---	40
A-3	3,30	82,9	large	partial	G	1-30	3	30
A-4	4,10	100,7	large	good	WG	---	1	57
A-5	3,91	103,5	large	good	WG	0-10	0,5	30
A-6	3,55	95,6	large	good	---	---	---	---
A-7	3,20	83,0	large	partial	G	5-50	10	50
A-8	4,10	101,8	large	good	B	10-30	20	57
A-9	3,00	79,9	small	bad	10:1 WG	0-10	0,5	20
A-10	3,90	98,5	medium	good	10:2 G	10-30	10	30
A-11	---	---	medium	good	11:1 WG	2-30	5	40
A-12	---	---	small	bad	11:2 BG	10-50	10	---
A-13	3,80	103,8	large	good	---	---	---	---
A-14	3,90	99,6	large	good	WG	1-30	1	44
A-15	4,06	100,7	large	good	G	1-30	3	55
A-16	3,90	103,4	medium	good	G	5-40	15	44
A-17	3,70	94,4	medium	bad	G	0-30	5	30
A-18	3,95	96,0	medium	good	G	10-30	10	57
A-19	3,20	79,2	large	partial	G	1-50	10	50
A-20	3,50	90,4	large	good	---	---	---	20
A-21	3,97	103,0	large	good	---	---	---	---
A-22	3,85	102,7	medium	good	---	---	---	35
A-23	---	---	---	---	WG	1-30	8	50

Appendix B: Comprehensive Tables of Test Data, comments

- E-1: Bad drilling precision. Upper relief hole and right hand blast-hole were connected at the bottom. Left hand blast-hole failed to detonate. Suspected dead-pressing of detonator.
- E-2: Bad drilling precision. Upper relief hole and right hand blast-hole were too close at the bottom.
- E-3: Bad drilling precision, both blast-holes too close to downmost relief hole at bottom. Large spacing between relief holes.
- E-4: Survey data for right hand blast-hole (no 2) lost during analysis. Vertically sintered bridge at about 1.8 m from face.
- E-5: Very large distance between relief holes and no cooperative effect between them. Some Hamamite left in left hand blast-hole (no 2).
- E-6: Good drilling precision. No apparent reason for bad breakage results.
- E-7: Bad drilling precision. Left hand blast-hole (no 2) rifled and right hand blast-hole broke out only to upper relief hole.
- E-8: Good drilling precision. Large opening at the face but smaller than at the bottom. Cementation around left hand blast-hole (no 1) about 2.5 from bottom and out.
- E-9: Good drilling precision but very large burden. Many visible cracks on face after cleaning by back hoe.
- A-1: 0.8 m of undetonated ANFO left between 1.8-2.6 m in left hand (no 2) blast-hole. Cause believed to be detonation failure due to venting to lower relief hole.
- A-2: Very large distance between relief holes.
- A-3: Burden of left hand blast-hole (no 2) was very large at depth.
- A-4: Good drilling precision.
- A-5: Conical plug shape of cut which doesn't throttle fragment flow. Long throw.
- A-6: Good drilling precision. Conical plug shape of cut which doesn't throttle fragment flow.
- A-7: Very bad drilling precision but both blast-holes connected with relief holes. Intentionally no initiation delay (0 ms) between holes.

Appendix B: Comprehensive Tables of Test Data, comments ctd

- A-8: Good drilling precision. Large cut area at depth.
- A-9: Clay seam at 1,25 m prevented deeper positioning of primer and detonator in right hand blast-hole (no 1).
- A-10: Right hand blast-hole failed to break out. First judged as rifle shot but changed to detonation failure due to venting to relief hole at bottom. Note 2 fines values, given as 10:1 and 10:2 in shaded area for blast-holes 1 and 2 respectively.
- A-11: Very bad drilling precision. Right hand blast-hole (no 1) failed to break out despite smaller burden. Note 2 fines values, given as 11:1 and 11:2 in shaded area.
- A-12: Changed order of initiation compared to other cuts (usually 1st hole had smallest burden on the bench face). Left hand blast-hole (no 2) failed to detonate due to dead-pressed detonator. 2 m of compressed ANFO left.
- A-13: Bad drilling precision and small cut area at depth. Soft rock conditions.
- A-14: Good drilling precision and good balance between 1st and 2nd burden. Large cut area at face. Initiation delay 50 ms.
- A-15: Bad drilling precision. Good balance between 1st and 2nd burden and conical plug shape of cut which doesn't throttle fragment flow.
- A-16: Bridge formed. Clean breakage from right hand blast-hole (no 2) despite large burden.
- A-17: Bad drilling precision. Short throw of fragments because of smaller opening at the face than at the bottom throttled flow.
- A-18: Good drilling precision. Large burden. Minor cementation. 1,2 m of left hand blast-hole didn't break out.
- A-19: Bad drilling precision. Initiation delay 25 ms.
- A-20: Right hand blast-hole (no 1) has a burden which increases with depth. Small bridged formed.
- A-21: Some cementation. Large sized fragments.
- A-22: Bad drilling precision. Some cementation. Large sized fragments.
- A-23: All preblast data exists but blasting results were obliterated by back hoe cleaning of bench face.

Appendix C: Plots of Cut Geometries and Table of Drilling Deviation Data

New cut	Spacing & deviations			Burden plan	1st hole dev.		2nd hole dev.		Cut areas		DQR value
	plan	face	bottom		face	bottom	face	bottom	face	bottom	
no	mm	mm	mm	mm	mm	mm	mm	mm	m2	m2	
E-1	200	-22	0	200	26	-165	19	-40	0,040	0,020	B
E-2	200	-21	11	200	-6	-129	-48	80	0,031	0,037	B
E-3	300	-37	30	200	-51	-74	-57	50	0,038	0,062	B
E-4	200	10	48	250	-50	-87	0	---	---	---	---
E-5	250	70	184	250	-11	-27	38	80	0,084	0,120	B
E-6	250	-26	-53	200	131	44	76	80	0,068	0,052	A
E-7	300	-6	-103	300	-149	-14	25	-80	0,070	0,050	VB
E-8	200	54	-30	300	10	-12	-10	-20	0,076	0,048	A
E-9	300	15	41	350	-9	40	26	40	0,113	0,133	B
A-1	200	15	82	200	2	-120	7	-40	0,044	0,034	B
A-2	300	20	83	200	-8	-109	43	20	0,070	0,060	A
A-3	200	21	-55	250	-60	-156	-35	0	0,045	0,025	B
A-4	300	17	31	200	0	-72	10	70	0,065	0,066	G
A-5	300	83	-42	200	0	-69	-10	-50	0,075	0,036	G
A-6	200	27	64	200	86	-54	-46	60	0,050	0,054	G
A-7	250	44	-45	250	5	-73	-20	0	0,071	0,044	VB
A-8	150	25	40	200	-10	-5	20	50	0,036	0,042	G
A-9	200	61	-111	300	-31	-98	-29	0	0,070	0,022	B
A-10	200	34	17	300	-90	-92	27	70	0,063	0,063	A
A-11	200	-11	70	400	-10	-192	-25	160	0,072	0,104	B
A-12	200	6	42	200	-5	19	-32	-40	0,044	0,046	VG
A-13	200	-8	-62	250	-14	-5	10	-60	0,048	0,030	VB
A-14	250	28	-9	250	-19	-4	-7	-60	0,066	0,053	G
A-15	200	14	-5	350	20	-94	-20	-180	0,075	0,042	B
A-16	200	11	45	400	1	-135	43	0	0,089	0,081	B
A-17	300	-17	74	300	-70	-25	-11	70	0,073	0,121	B
A-18	300	48	27	300	30	-25	0	20	0,110	0,097	G
A-19	250	-9	74	250	42	33	-3	-30	0,065	0,081	A
A-20	200	5	58	250	-30	82	-15	-100	0,047	0,062	A
A-21	300	15	-9	300	13	49	-25	-90	0,093	0,081	B
A-22	200	-12	62	300	-73	53	2	40	0,050	0,091	VB
A-23	200	-34	-40	400	-48	-16	-1	60	0,062	0,068	---

Appendix C: Plots of Cut Geometries and Table of Drilling Deviation Data, ctd

