

# Improved Blasting and Bench Slope Design at the Aitik Mine

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## Abstract

*The Aitik open pit mine, located in northern Sweden, currently faces a substantial production increase, which has necessitated a review and update of design guidelines. This work pointed at the possibility of increasing interramp slope angles a few degrees, which, in turn, requires an improved bench slope design, most notably blasting techniques. Building upon previous development work, new field trials were conducted in conjunction with trimming of the footwall slope to increase ore extraction. Different blast designs were tested and the resulting blast damage was evaluated through measurements and follow-up. These trials quantified the limits in terms of achievable bench face angles and amount of backbreak. The detrimental effects of small-scale structures could be controlled through lighter smooth blasting, whereas the impact of larger, more dominant, structures, prevailed even for very light smooth (for open pit operations) blasting. For the footwall, presplit blasting in inclined (70°) holes over the full double-bench height, proved to be suitable in terms of both achievable bench slope geometry, and production concerns. For the hangingwall, a presplit blast design is used in specific areas to ensure that sufficient ramp width is achieved. The study reported here has effectively provided limits on the maximum achievable bench slope angles at the Aitik mine, as well as an increased confidence in the bench face design currently being implemented at the mine.*

## 1 Introduction

The Aitik open pit mine is situated in northern Sweden, approximately 60 km north of the Arctic Circle and 1200 km north of Stockholm. The mine is owned and operated by the Boliden Mineral AB mining company. The low-grade copper mineralisation (approximately 0.4% Cu) was discovered in the 1930s, but mining did not start until 1968. Annual production started at a modest level of 2 Mton of ore but has since steadily increased. Currently, some 18 Mton of copper ore are produced annually, with an additional 22 Mton of waste rock being extracted. The current deepest mining level (pit bottom) is the 390 m level (actual maximum slope height is about 350 m). Future mining plans call for an increase in annual production up to a total of 36 Mton of ore, and a final pit depth of approximately 600 m.

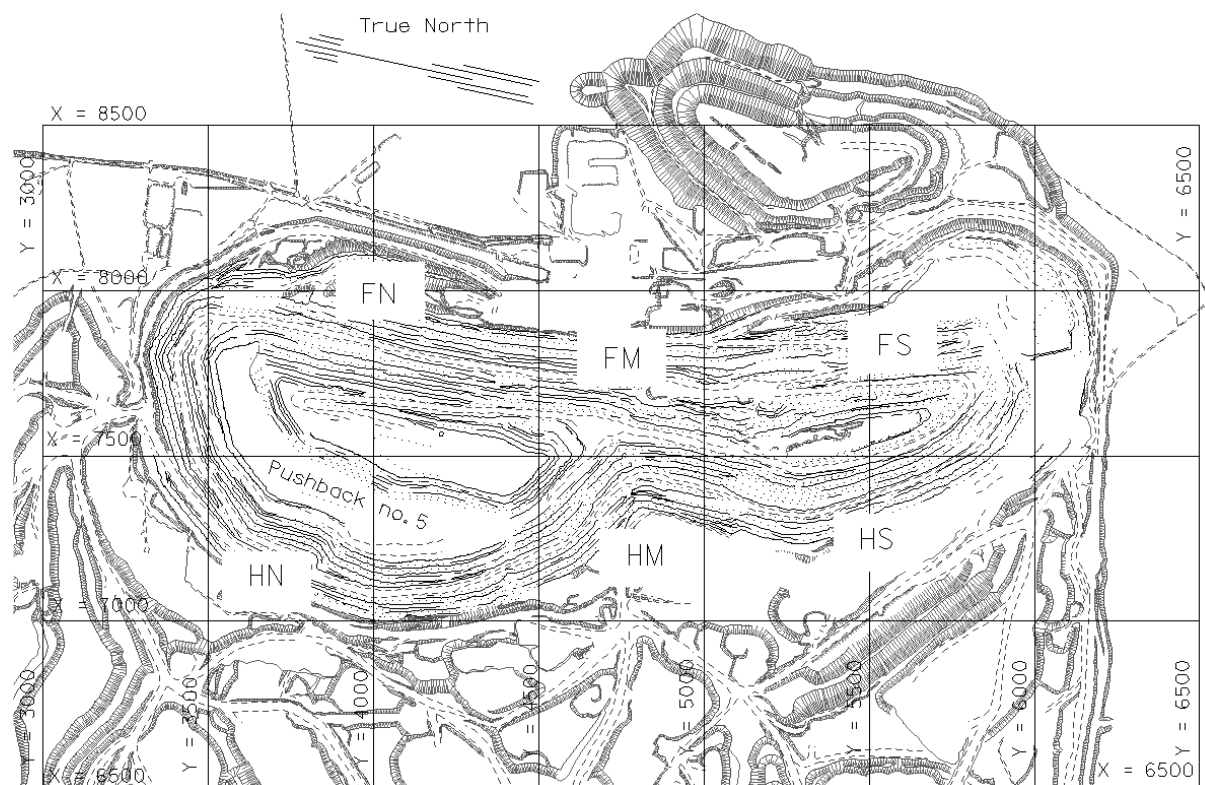
The copper mineralisation at Aitik occurs as disseminated thin veinlets of chalcopyrite, along with minor contents of silver and gold. Ore grade mineralisation is hosted in the main ore zone gneiss, with minor mineralisation in the footwall gneiss. The orebody strikes approximately N20°W and dips about 45° to the west. It is approximately 2000 m long and some 300 m wide, and known to be more than 800 m in depth.

The dominant rock types at Aitik are metamorphic igneous rocks with various mica contents. The majority of the rocks exhibit a well-defined foliation (preferred alignment of mica minerals). In the footwall, diorite rock with only weakly developed foliation is also present. The dominant jointing are foliation joint planes, striking subparallel to the orebody (footwall) and dipping between 45° and 70° towards the west (steeper towards the footwall). Cross-joints and subvertical joint sets also exist. The majority of the pre-existing joints are short, with maximum lengths generally less than 10 m. The hangingwall contact towards the ore zone is an old thrust fault. Aside from this, no large-scale structures have been verified. Intact rock strengths show some variation, which can be correlated primarily to mica content and degree of metamorphosis. Mean compressive strengths as determined from point load tests on drill cores vary from around 60 to 130 MPa.

Mining at Aitik is done using traditional open pit mining with pushbacks. Production blasting is carried out with bulk explosives in 311 mm blastholes, with smooth blasting near final benches. Ore is transported by

trucks to an in-pit crusher. The crushed rock is transported to the plant by conveyor belt through an underground drift in the southern footwall. Waste rock is trucked to dumps located to the west and north of the pit. The pit currently measures approximately 2500 m in length and 750 m in width (Figure 1).

The planned expansion at Aitik will result in a deeper final pit. This necessitated a review and update of the design guidelines for the mine. A historic review is presented below, starting from the first major design work in the 1970's. The concurrent development of blasting techniques for optimum bench slopes is also described. This work also formed the basis for a set of new field trials aiming at achieving safe, stable and economic bench slopes for future mining at Aitik. This paper further describes the methodology used, and the results achieved from this work.



**Figure 1** Overview plan showing the Aitik mine geometry as of 2007, together with design sectors

## 2 Blasting and slope design development — a historic review

### 2.1 Development of slope design at Aitik

The earliest design criteria for the Aitik mine was developed by the consulting firm Call & Nicholas, Inc., through two major studies in 1976 and 1985. These criteria included recommended effective bench face angles, catch bench widths, and interramp slope angles. They were largely based on mapped pre-existing joints and the probability of structurally controlled failures in bench-scale. The mine was divided into six design sectors (Figure 1), in which geological conditions and pit wall orientations were judged similar. The developed design criteria, for the double-bench configuration, are shown in Table 1. The governing criterion for catch bench widths was that the width should be 11 m (or more) for 90% of the benches (i.e. over the total length of the bench slopes). This criterion is based on providing enough catchment for falling rocks. The design criteria in Table 1 were used, principally without exceptions, until 1999. The overall slope angle for the footwall and hangingwall slopes, including ramps, came out as approximately 45° for this design.

The double bench geometry used at Aitik involves two 15 m high benches between each catch bench, as shown in Figure 2. A small drilling offset is left between the upper and lower single bench. For the footwall, this offset was chosen to 5.5 m, so that approximately the same foliation plane would cut through the toe of both single benches. This was based on a foliation dip angle of 69° in the north and middle footwall. The effective bench face angle is flatter than this angle, due to backbreak from blasting at the crest (see Table 1).

**Table 1** Original design criteria for Aitik (see Figure 1)

Parameter	Design Sector					
	FN	FM	FS	HN	HM	HS
Interramp slope angle (°)	47	42 <sup>*)</sup>	49	51	56	53
Bench face angle – median (°)	74.0	70.0	74.3	80.0	84.0	80.0
Bench face angle – 90% > (°)	60.5	53.5	63.3	66.1	72.9	68.8
Catch bench width – median (m)	19.4	22.4	17.5	19.0	17.1	17.3
Catch bench width – 90% > (m)	11.0	11.0	11.0	11.0	11.0	11.0

FN = Footwall North

FM = Footwall Middle

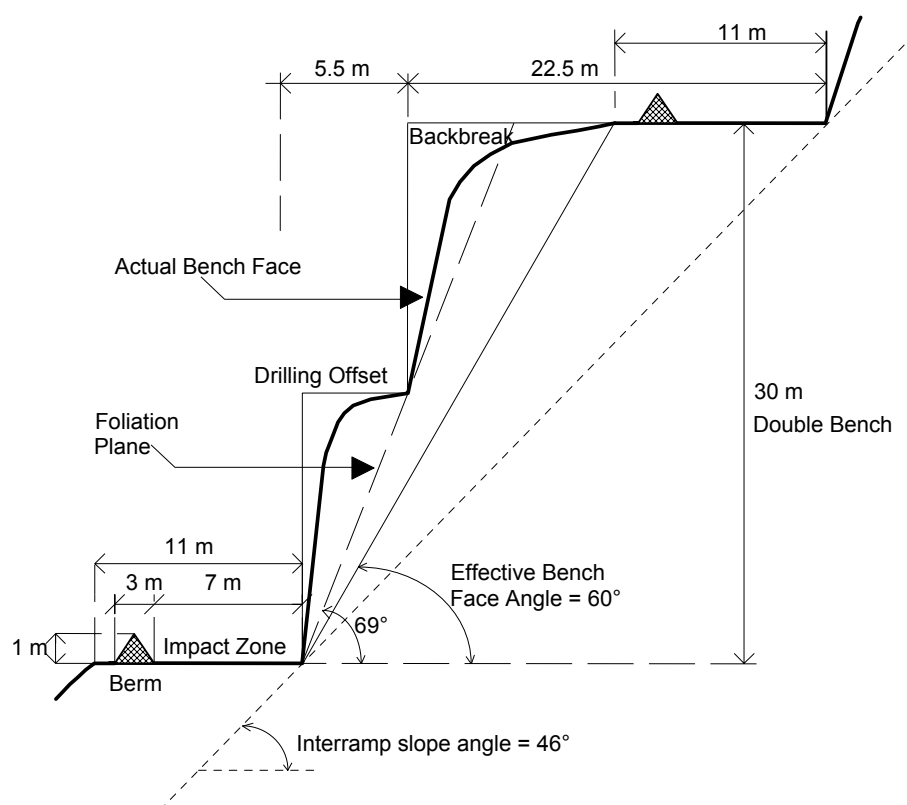
FS = Footwall South

HN = Hangingwall North

HM = Hangingwall Middle

HS = Hangingwall South

<sup>\*)</sup> A steeper alternative with 46° interramp angle was recommended if no large-scale structures could be verified in this design sector; this angle has been used in practice for mining the middle footwall portion.

**Figure 2** Original double-bench and catch bench design for the northern footwall sector at the Aitik mine (figure not to scale)

These initial design criteria were developed with the notion that the risk for large-scale failure was small. Detailed analysis of the large-scale slope stability showed that a steepening of both the interramp and the overall slope angles was possible (Sjöberg, 1999; Sjöberg and Norström, 2001), under the assumption that the slopes were drained. These results led to the decision of mining the northern hangingwall (HN) with 56° interramp slope angles, i.e. a 5° increase. This was implemented starting with the fourth pushback in 2000.

Continued (and more recent) analyses for various proposed future pit configurations showed that mining even to depths of 750 m can be done with stable overall slope angles of 45° for the footwall and 50° for the hangingwall. In practice, these results imply that the interramp slope angles can be increased also on the footwall, without endangering the overall stability of the pit slopes. However, an increase of the interramp slope angles also requires changes in the bench slope design.

To assess the practically achievable slope angles, follow-up of actual slope geometry has been carried out routinely since 1986. The actual slope geometry has been controlled through aerial photography every second year. In the early studies, it was found that actual interramp slope angles were in general slightly flatter than the design criteria in Table 1. This could be correlated to excessive blast damage, as well as poor quality routines. As a consequence, catch benches were too narrow and the design criterion for these was not fulfilled. These findings motivated a more extensive project on smooth blasting, as described below.

## 2.2 Blasting trials

An extensive project on smooth blasting was conducted in the period of 1991 to 1996. The project was divided into four phases. The overall objective of the work was to develop smooth blasting techniques (for final pit walls) to improve bench slope geometry and thus fulfil the stipulated design criteria. Ultimately, this would also result in a more economical ore extraction through reduced waste rock handling, as well as improved safety. The project was a joint research effort between Boliden Mineral AB and SveBeFo (Swedish Rock Engineering Research).

Phase 1 of the blasting project showed that vertical blast damage was a major factor causing crest damage of the benches. Differences in blasting results could also be correlated to variations and imprecision in staking of the borehole positions, which prompted increased quality control concerning mine surveying and drilling.

In Phase 2, several different blasting (drill and charge) patterns were tested and evaluated. The follow-up included measurements of VOD (velocity of detonation), *PPV* (peak particle velocity) and gas pressure, visual observations and photography of bench face conditions, measurement of backbreak, and measurements of heave. Part of this work has been reported by Ouchterlony et al. (1997). It was clearly shown that the blast charges did not force any significant amount of high-pressure blast fumes into the rock and are thus not contributing to blast damage. Consequently, the blast damage assessment was focused on the *PPV* levels. A blast damage model was developed, based on measurement of backbreak and swelling/heaving, theoretical considerations, and a notion that damage levels must be easy to measure in practice. Blast damage was related to calculated *PPV* levels, and involved four different degrees, see Figure 3 and Table 2. The conducted *PPV* measurements enabled a site scaling law to be determined, as follows:

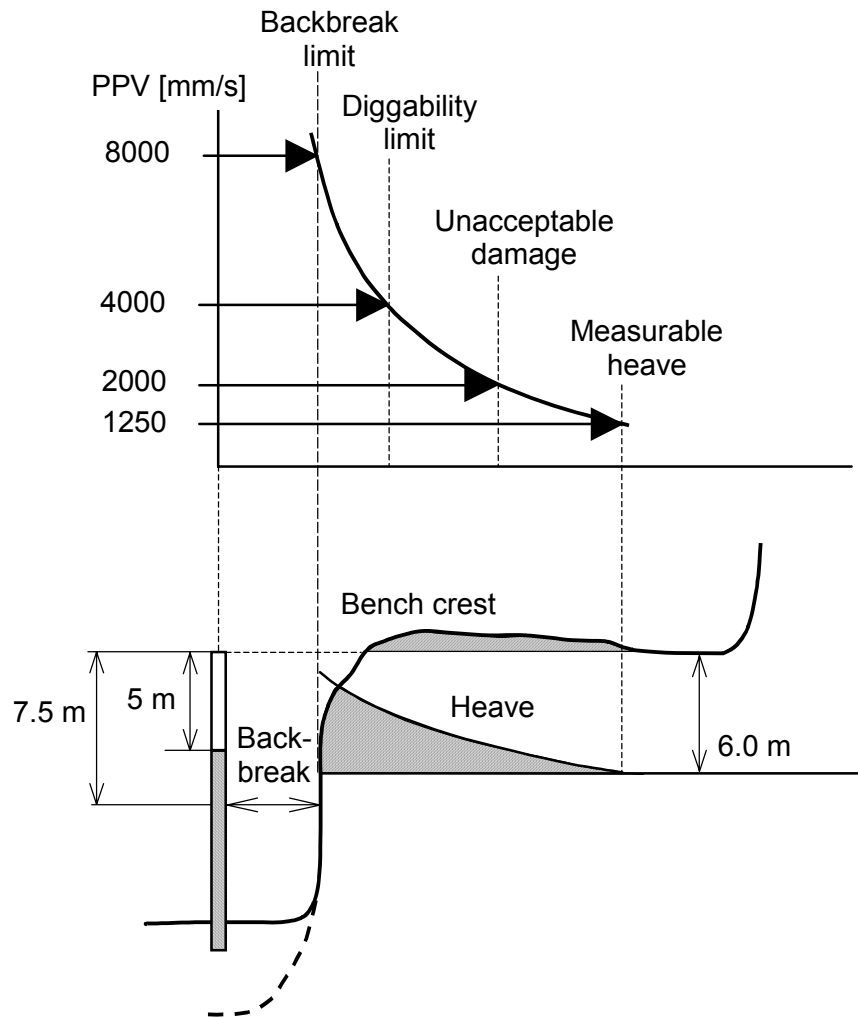
$$PPV = 650 \left( \frac{\sqrt{fQ}}{R} \right)^{1.42} \quad (1)$$

$$f = \frac{\arctan(H/2R)}{(H/2R)} \quad (2)$$

In Equations 1 and 2, *PPV* is the peak particle velocity (mm/s), *Q* is the charge size (kg), *R* is the distance from the charge (m), and *H* is the charge length (m).

Phase 3 involved further practical testing of the most promising smooth blasting patterns from phase 2. One commonality of these patterns was that one to three rows of small-diameter holes were required to achieve acceptable results. These tests were conducted in biotite schist, which is considerably weaker than the diorite rock in which the phase 2 tests were performed. Hence, significantly larger blast damage was found and adjustment of the drill and charge pattern was required for these weaker rocks. The *PPV* levels for various degrees of blast damage were also revised, see Table 2. A lighter smooth blasting pattern was tested in the final phase 4 of the project. For several reasons, tests could not be conducted in the weaker rocks; hence, the practical outcome from a lighter smooth blasting pattern in weaker rocks could not be tested. The results showed, however, that a further development with several slimhole rows and decoupled charges in the contour holes, should be a suitable alternative to regular smooth blasting in biotite schist.

The results of phases 1 through 4 have been applied, to various extents, in production blasting (two to three slimhole rows, but no decoupled charges). More recent studies from aerial photos taken in 1994 and 1998 showed that bench face angles had, indeed, increased. Perhaps the most important product from the blasting project was the blast damage criteria of Table 2 and the site scaling law (Equations 1 and 2). Together, these form a tool for designing smooth blasting patterns, including adjustments to local rock conditions.



**Figure 3** Blast damage model for the Aitik mine with four different degrees of horizontal damage

**Table 2** Blast damage criteria for bench blasting at Aitik (see Figure 3)

Damage Type	Description	<i>PPV</i> in Strong Rock (Diorite, Biotite Gneiss) (mm/s)	<i>PPV</i> in Weak Rock (Biotite Schist) (mm/s)
Backbreak	Crushing of rock and fall-outs	$\geq 8000$	$\geq 8000$
Diggability	Possibility to dig the blasted rock without larger efforts	$\geq 4000$	$\geq 4000$
Unacceptable damage	Not measurable; judgement call (acceptable damage beyond this zone)	$\geq 2000$	$\geq 1250$
Heave	Limit of measurable heave (at 6 m)	$\geq 1250$	$\geq 725$

### 3 Blasting and slope design development — new requirements and trials

#### 3.1 Introduction

The results from the studies of the aerial photographs showed that the design criteria have been fulfilled to an increasing extent. This is directly attributed to the knowledge gained and the implementation of better routines. The studies have, however, also indicated the need for even better control of the backbreak. In the spring of 2003 this was further pronounced when the mining of a lower part of the footwall ramp as well as a

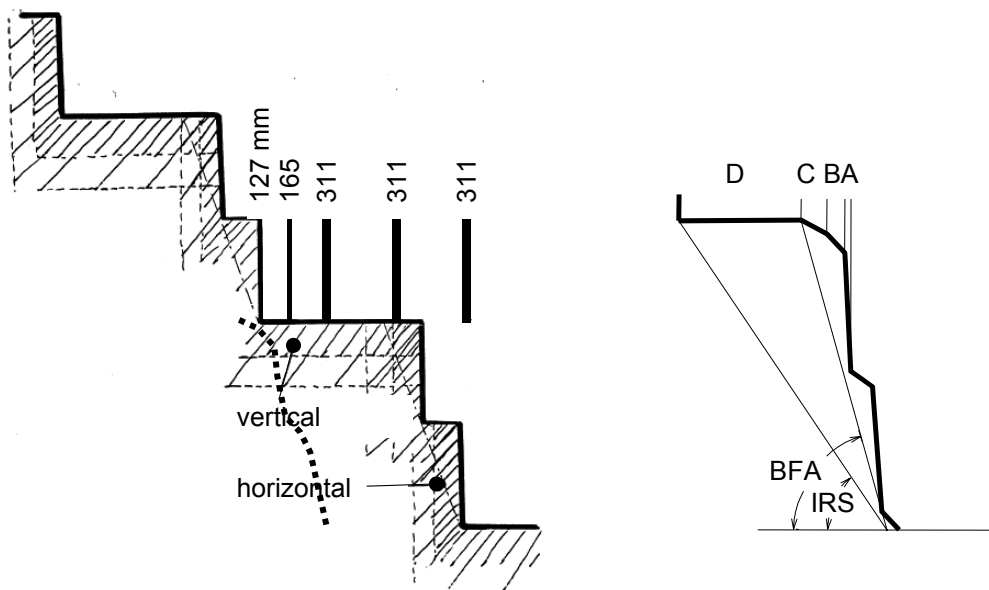
new ramp design for the hangingwall was suggested. In the old ramp design the full width of the bench was reached at a height of 7.5 m and in the new design the bench reaches full width at the full bench height (15 m). Furthermore, a part of the space required for the ramp would be gained from decreased backbreak on the benches above and on the outside of the ramp. Thus, the new ramp design required better control of the backbreak while the mining of the footwall ramp would not be possible without a significant reduction of the blast damage because of the limited width to be mined.

In the design of the footwall the interramp slope angle (IRS) was  $47^\circ$ . To be able to mine as much as possible of the ore beneath the ramp and to have a reasonable bench width to work on, the desired IRS was  $56^\circ$ . However, the old interramp slope angle was governed by a steeply dipping foliation (see Figure 2) and it was decided to aim for  $52^\circ$  but to test the potential for a steeper IRS. In the weaker rock on the footwall, the decreased backbreak was believed to be achieved by limiting the extent of heave to 3–6 m from the contour holes, using the lower *PPV* damage limit ( $PPV=725$  mm/s) according to the earlier studies. On the hangingwall the aim was to decrease the backbreak to 2 m. The key to both of these problems is to control blast damage, i.e. to: (i) quantify it, (ii) determine what governs it, and (iii) find techniques that make it possible to choose the acceptable level of blast damage.

### 3.2 Blast damage and alternative solutions

In open pit mining, blast damage determines the resulting effective bench face angle (BFA), see Figure 4. For the different portions of backbreak (Figure 4b), A is mainly due to horizontal blast damage and structures, B to both vertical and horizontal blast damage and structures, and C mainly to vertical blast damage and cleaning of the crest (D is the resulting bench width). In the earlier studies, the horizontal blast damage was quantified and design criteria for blasting were developed. The vertical blast damage caused by the holes in the upper bench was however not studied, even though its significance for the resulting geometry of the bench was realized.

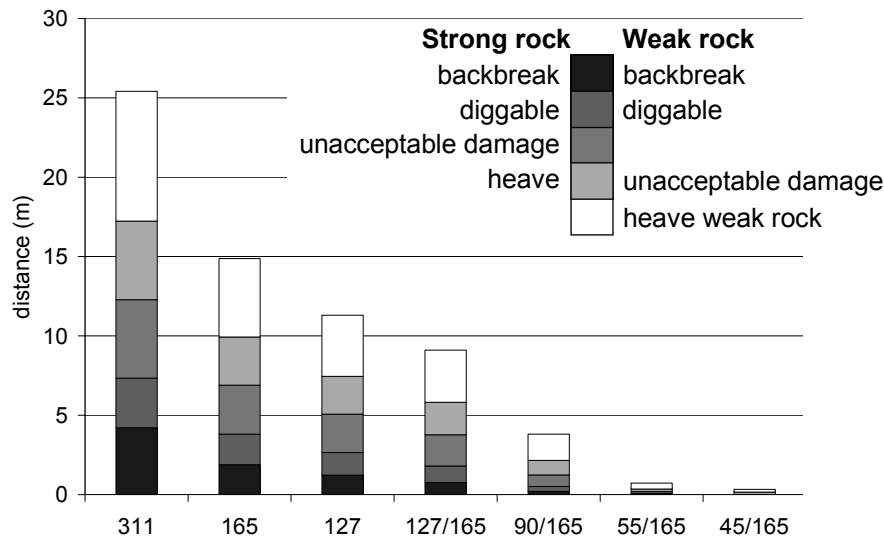
Using the site scaling law (Equations 1 and 2) and Table 2, the horizontal blast damage for different charges were assessed (see Figure 5) in order to compare different blast designs. Figure 6 shows the range of horizontal blast damage for a standard blast at Aitik and a tentative blast design where the contour hole fulfils the requirement of a heave range limited to 3–6 m for  $PPV=725$  mm/s. However, the blast damage from the 2nd and the 3rd rows, and even the 311 mm hole in the 5th row, exceed those caused by the contour hole, which means that even more decoupled and fully charged 165 mm holes would have to be added. When comparing this to the presplitting in the Ekati Mine, described by Todd (2001) and Mathis and Todd (2002), the presplitting seemed to be a very attractive alternative.



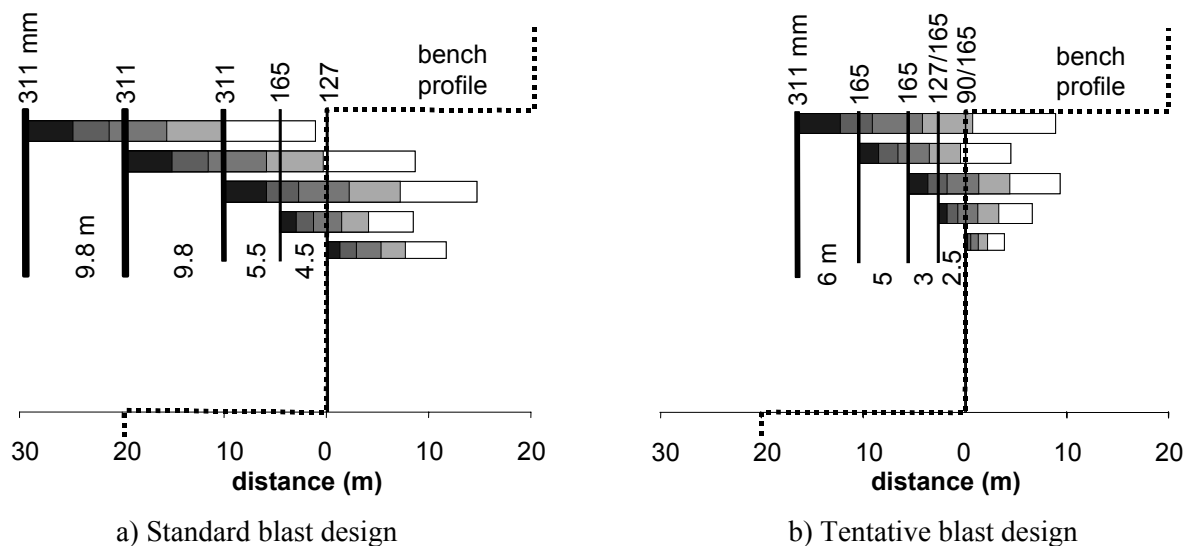
a) Vertical and horizontal blast damage

b) The different parts of backbreak

**Figure 4** Vertical and horizontal blast damage in bench blasting (a) and backbreak (b)



**Figure 5** Range of horizontal blast damage levels for different charges in 15 m holes (diameters in mm) based on Table 2 (x/y means decoupled charges)



**Figure 6** Assessment of horizontal blast damage for; a) standard blast at Aitik; and b) alternative blast design using de-coupled charges in the two rows closest to the bench face (bench profiles are drawn before backbreak)

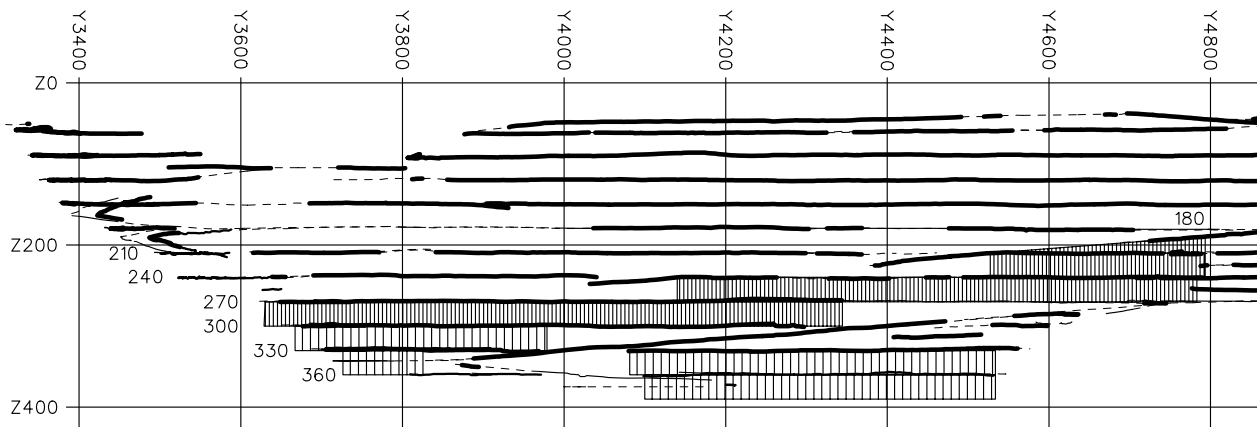
### 3.3 Full-scale blasting test

A project was started to investigate the governing parameters and develop the techniques needed to fulfil the requirements for the hangingwall and the footwall. The blast designs in Figure 6 indicated that decoupled charges would be required in conventional cautious blasting and techniques for loading decoupled charges with bulk explosives would have to be developed. This, and the large amount of drilling of 165 mm holes, led to the decision to choose presplitting as the main alternative for the full-scale test, while searching for techniques for bulk loading of decoupled charges.

On the footwall a full-scale test was carried out from the 180 to 270 m level, see Figure 7. The first presplit round was made over a limited bench height between the 182 and 210 m levels, using maximum 22 m long holes. On the following levels presplitting was performed on 30 m bench height and along the ramps with drill depths ranging from 2–30 m. On the hangingwall the tests consisted of 15 m high presplits, with and

without a reduction of vertical blast damage through an increased number of 165 mm holes in the upper bench, and a standard contour blasting with, and without, reduction of the vertical blast damage.

The follow-up activities included heave measurements, measurement of backbreak, assessments of diggability, measurement of fragmentation, comparison to geological conditions, assessment of the vertical blast damage from the production (311 mm) holes, and hole deviations and their effect on the result. The most important results will be presented and discussed in the following section.



**Figure 7** Longitudinal view of the northern footwall showing the full height sections of the test area (dense hatching) and full height sections of presplit areas after the test (light hatching)

### 3.4 Results

#### 3.4.1 Blast design

The development of the blast design for the presplitting of the footwall has resulted in the plan presented in Figure 8. The 30 m high presplitting holes are drilled at an inclination of 70°. In the upper part of the double bench the first vertical row consists of the production holes. In the lower part the number of rows with 165 mm holes is increased to reduce the vertical blast damage on the crest of the next bench. The presplitting was usually made with 45 mm diameter Dynopre, watergel cartridges on a 6 g/m downline, doubled up the last 1 m. When blasted on its own, 10 holes per interval were initiated with 17 ms between intervals.

On the footwall the presplitting is subparallel to the foliation while on the hangingwall the presplitting holes are vertical and cutting through the foliation. This has required reduced hole distances both within the presplitting row and between the presplit and the first row. On the footwall, only 165 mm holes were used while mining the ramp between the 180 and 270 m levels. From the 300 m level and downwards 311 mm holes were used. On the hangingwall the presplitting is vertical, drilled with c/c 1.8–2.0 m, and the first buffer row is drilled 1.8–2.0 m from the contour. The presplitting is also limited to 15 m height as this is enough to fulfil the specific requirements for the hangingwall.

#### 3.4.2 Vertical blast damage

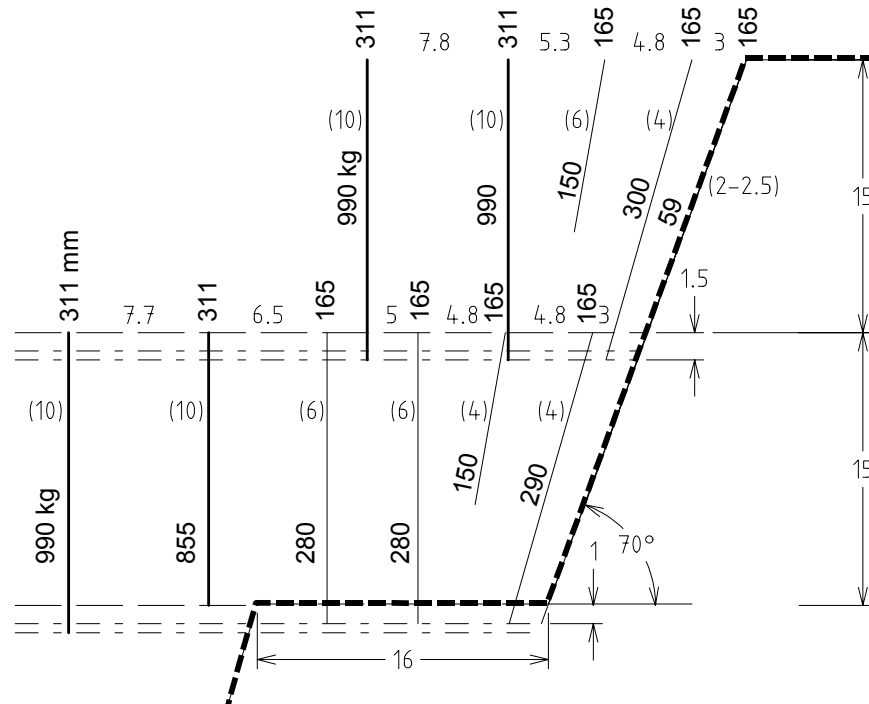
The vertical blast damage in 21 holes in two standard rounds were assessed using MWD-data from the drilling rigs, see Figure 9a). The depth of what was interpreted as debris, i.e. very finely fragmented rock, was  $1.4 \pm 0.6$  m (mean  $\pm$  standard deviation) below the bottom of the holes from the blast above. When comparing data from locations directly below the above lying holes with data between the holes, no significant difference was found. Assuming a normal distribution, 90% of the blast damage would reach 2.2 m below the bottom of the holes in the above lying round. This is equal to about half of the horizontal distance to the *PPV*-level for backbreak (4.0 m, see Figure 5) and corresponds quite well with the *PPV*-attenuation at the end of a linear charge, as in Hustrulid (1999). Using 1.5 m of subdrilling, this means that the vertical blast damage from the production holes is 3.7 m below the planned bench level.

The different parts of the backbreak were measured for 54 double benches between 210 and 330 m levels on the northern hangingwall (HN). This showed that the steepest part had an inclination of  $81^\circ \pm 4^\circ$  and that the

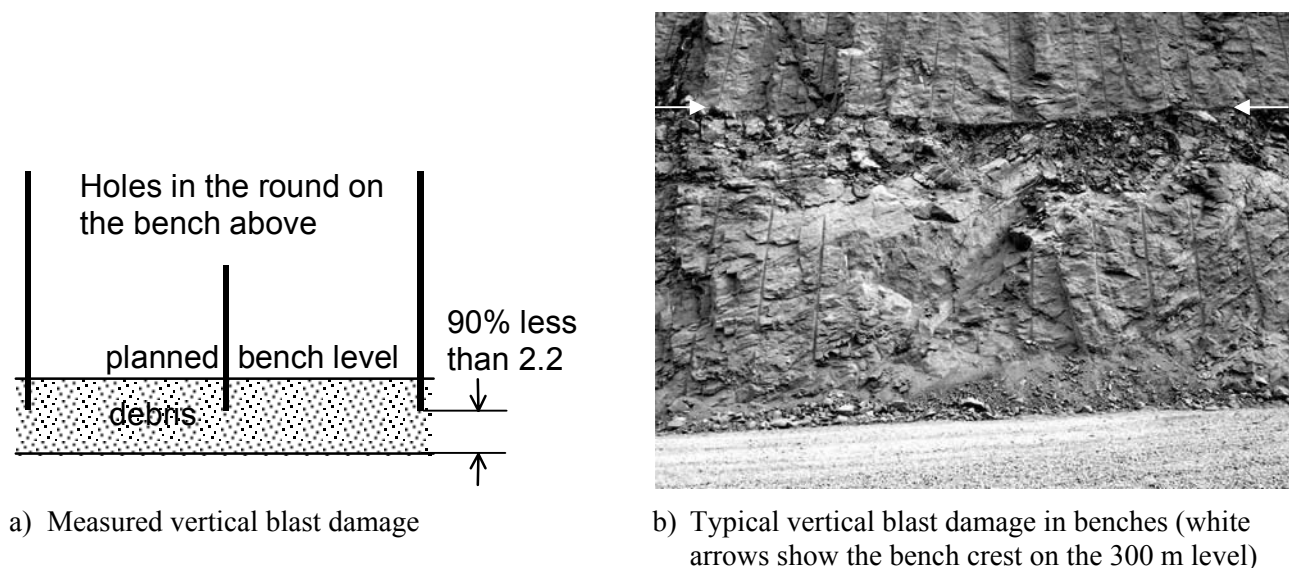


bench crest had a slope in the range of 30–45° over 2–4 m height. The crest is formed within the zone of backbreak, Figure 10a). In Figure 10b) the same geometry is used but now together with a blast design with 4 buffer rows. The backbreak zone has decreased from 3 to 1 m, which should decrease the backbreak of the bench crest with approximately the same amount.

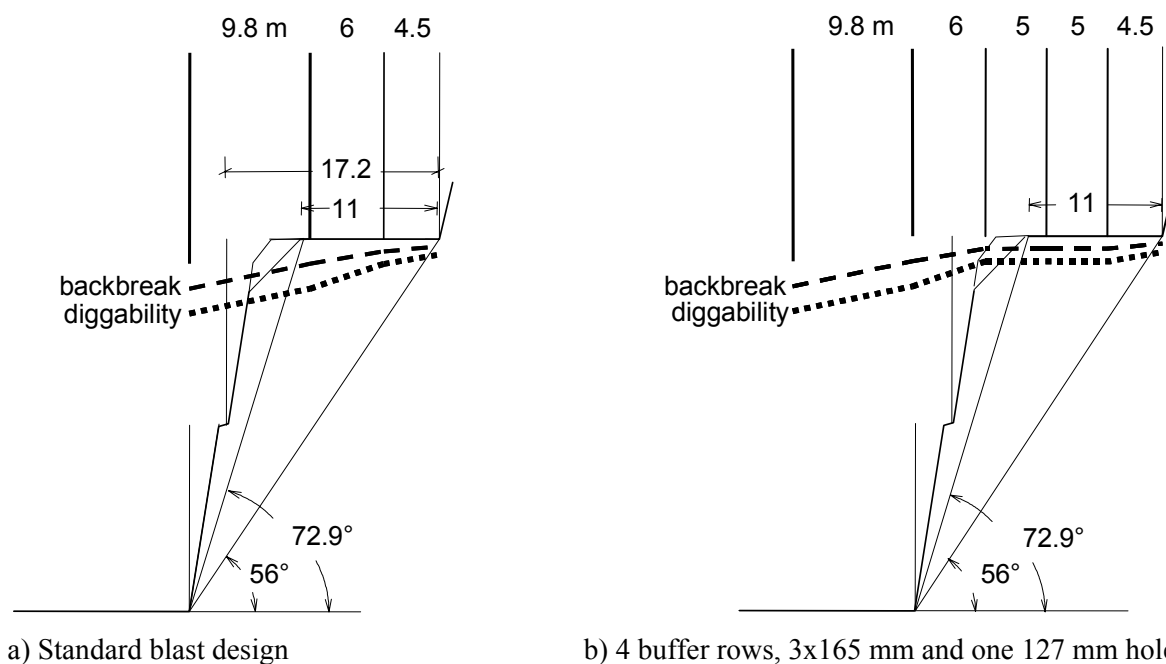
The two blast designs in Figure 10 were tested on the hangingwall on two 100 m long sections next to each other. The standard blast resulted in  $6.3 \pm 0.7$  m of backbreak and the modified design in  $5.3 \pm 1.2$  m or, excluding a geological disturbance,  $4.8 \pm 0.6$  m of backbreak. This indicates a decrease of the backbreak of about 1.5–2 m, which is similar to the assessment based on the damage criteria. In parts of the hangingwall ramp, presplitting has been used to decrease the backbreak, both with and without reduced vertical blast damage from the blast above using limited subdrilling or 165 mm holes. The results are presented in Table 3.



**Figure 8** Vertical section through the resulting design of the presplitting on the footwall showing charged weight in each hole, burdens and, within brackets, c/c-distances between holes perpendicular to the plane of the figure (m)



**Figure 9** Vertical blast damage from production holes (311 mm)



**Figure 10** Bench geometry and vertical blast damage on the hangingwall for standard rounds and for 4 rows of 165 mm buffer holes

**Table 3** Backbreak in presplit areas from 195 to 240 m levels on the hangingwall

Level	Average (m)	Std. Dev. (m)	Median (m)	90% Less Than (m)	Length (m)	Subdrilling (m)
210–240	2.7	1.1	2.9	3.9	324	1.5–3
195	2.4	0.7	2.3	3.2	88	0

### 3.4.3 Horizontal blast damage and backbreak

The backbreak of the presplitting areas on the footwall, from the first test rounds on the 180 m level down to the current pit depth, 390 m level, is presented in Figure 11 and Table 4. The drilling and the charging of the presplitting and the buffer holes are more or less the same from the 240 m level and downwards, except for the introduction of 311 mm holes from the 270 m level, and results in a backbreak of  $3.5 \pm 1.6$  m, where 90% is less than 5.3 m. This allows for an IRS of  $48^\circ$  when the presplitting is drilled with an inclination of  $70^\circ$ .

### 3.4.4 Heave

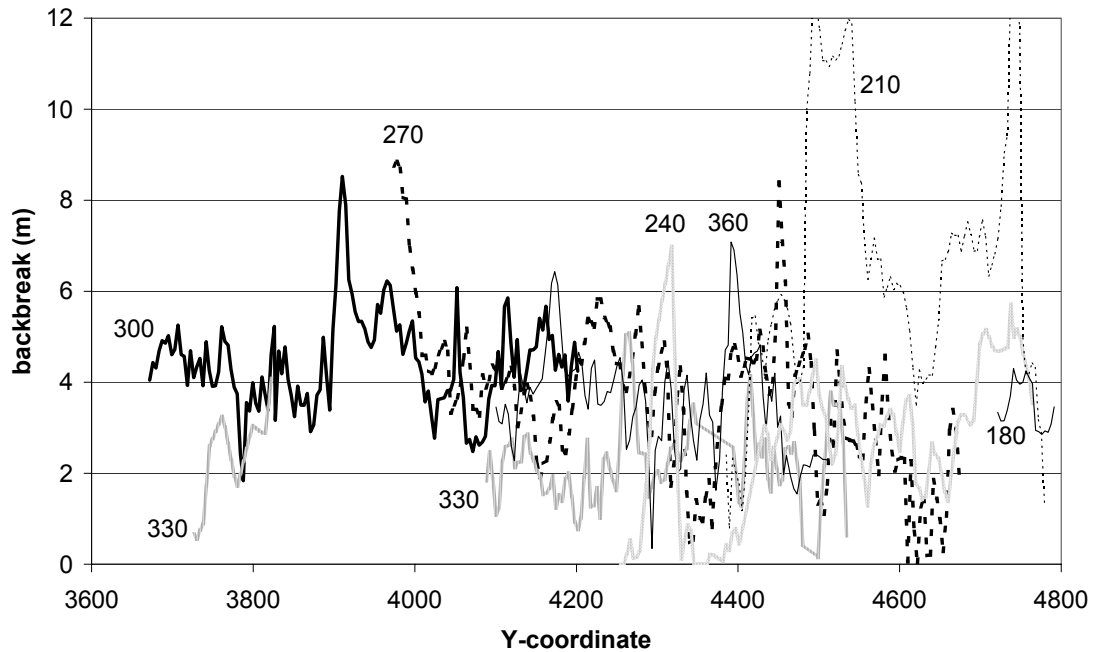
The heave was measured after the presplit and after the production blast in 9 profiles on the 180 through 270 m levels, except for 3 profiles where it was only measured after the production blasting. The results for eight of these profiles are presented in Figure 12. For the presplitting, the extent of the heave was in the range of 2.0–5.5 m, while the limiting distance of heave from the first buffer row (165 mm) was 4–7 m from the contour, which is 7–10 m from the hole on the bench level. This corresponds to the limit of heave for "good rock" according to Figure 5.

### 3.4.5 Comparison to geological conditions

The comparison to geological conditions consisted of two parts, firstly the effect of presplitting on the structures and secondly the variation of the results with rock type. For the comparison to structures five drillholes were logged with respect to the surface conditions of joints and other structures. The structures and joint surfaces were logged as: (1) joint or weathered rock, (2) slickensided with less than 1 mm of filling and (3) slickensided with more than 1 mm of gouge. Structures of type 1 and 2 had very little influence on the

blasting result, except for a somewhat increased backbreak when occurring close to the bench crest, while structures of type 3 affected the geometry of all of the bench faces. The first encounter with such a structure was on the first 30 m high presplit (210 m level) where the hole inclinations were 75–80°. As the structure was dipping 65–70° parallel to the bench, wedge failure occurred while mining the lower part of the double bench. The clean-up delayed the production on the footwall by several weeks.

A comparison between backbreak and rock type showed that on the upper levels the muscovite schist is the rock type where the backbreak is the largest, see Table 5 (backbreak includes blast damage, failures due to structures, and cleaning of the bench crest). On the lower levels, where no larger wedge failure occurred, the difference is not obvious. The content and degree of schistosity varies, hence the rock types are not clearly defined, but the result indicates that the type and extent of schistosity, within the variations occurring in this study, is not the critical parameter for backbreak.

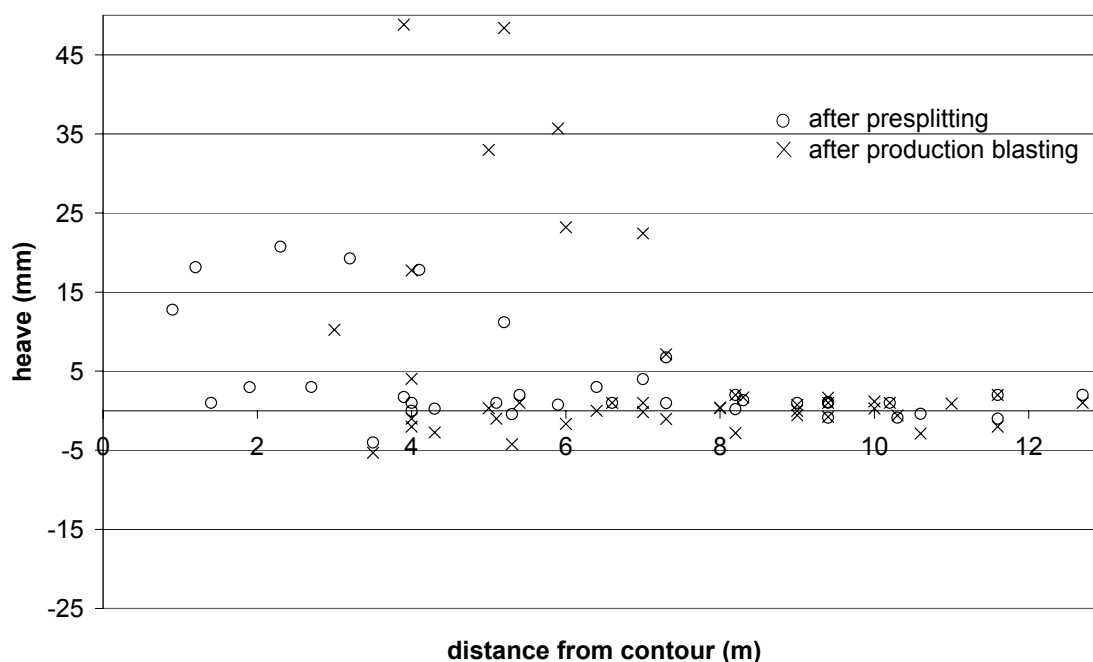


**Figure 11 Backbreak for bench crests from 180 to 360 m levels**

**Table 4 Backbreak and details from 180 to 360 m levels on the footwall**

Level	Average (m)	Std. Dev. (m)	Median (m)	90% Less Than (m)	c/c Presplit (m)	Crest Holes (m)	No. of 165 Rows (m)	Comment
180	3.5	0.5	3.4	4.1	1.5–1.8	165	>4	First test round
210	6.7	3.0	6.3	3.9	1.8–2.2	165	>4	75–80° presplit*
240	2.8	1.7	2.9	4.7	1.8	165	>4	
270	3.3	1.2	3.0	5.1	2	311	2	
270	5.9	1.7	5.8	8.0	2	165	2	
270	3.7	1.7	3.9	5.3	2–2.5	165	>4	
300	4.4	1.0	4.3	5.6	2.5	311	2	
330	2.4	0.9	2.7	3.2	2.5	165	2	
360	3.6	1.2	3.6	4.9	2.5	165+311	2	
240-360	3.5	1.6	3.6	5.3				

\* wedge failure along most of the bench due to a more shallow dipping fault parallel to the bench face



**Figure 12** Heave measurements on the footwall after presplitting and after production blasting

**Table 5** Backbreak and details from 180 to 360 m levels on the footwall

Level	Rock Type	Average (m)	Std. Dev. (m)	Length (m)
240	Biotite gneiss	2.4	1.6	404
	Muscovite schist	4.7	0.6	80
270	Biotite gneiss	3.7	1.4	804
	Biotite schist	3.4	1.4	164
	Muscovite schist	7.9	0.8	80
300	Biotite schist	5.0	1.3	168
	Muscovite schist	4.2	0.8	380
330	Biotite gneiss	2.5	1.2	132
	Biotite schist	2.4	0.7	132
	Muscovite schist	1.9	0.6	140
360	Biotite gneiss	4.1	0.5	24
	Biotite schist	3.6	1.3	336
	Muscovite schist	3.5	1.0	76

### 3.4.6 Hole deviations

By measuring three points on half-barrels on the bench faces after presplitting, the hole deviations and their influence on the result was investigated. For 22 holes in a round between the 240 and 270 m levels, the total deviation at the hole bottom was  $1.1 \pm 0.6$  m. An inspection of the bench face would still classify the result of the presplit as "good", however with some rock left at the toe and 5 m up within the area. This is probably caused by the steepest holes, ending more than 1 m deeper into the toe than planned.

Inspections of the bench faces showed that it is the variation of inclination between adjacent holes that is critical. As several holes are initiated at the same time, the hole deviation along the bench is not as critical as long as the average hole distance is kept within the plan. However, even smaller deviations in the inclination results in rough bench faces, increasing the risk of loose rock and the need for adjustment blasting of rock at the bench toe.

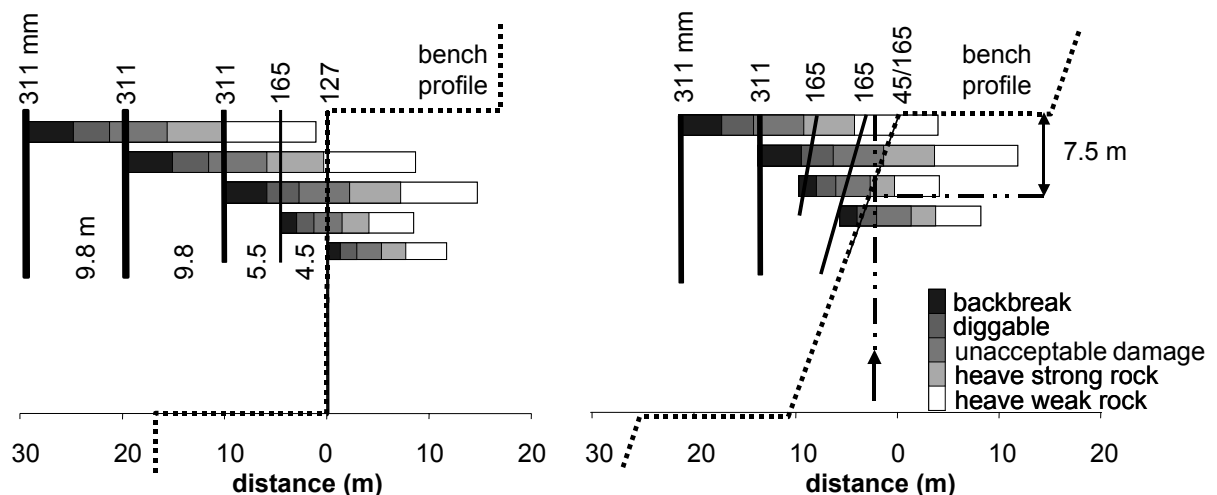
## 4 Discussion

In Figure 13, a comparison of the horizontal blast damage for the standard round in Aitik and the developed blast design for the footwall is compared. For the developed blast design, the extent of the damage is shown for a location at mid height of the upper 15 m of the double bench, 7.5 m below the planned bench crest. The first production hole (311 mm) is 0.5 m farther away from the contour holes and the first buffer row is 2 m closer than in the standard round. The largest difference for the backbreak and diggability is caused by the elimination of the 127 mm hole as a contour hole. It is also possible that the presplit crack to some extent isolates the bench from the vibrations from the blast holes in the round (Ouchterlony, 1997), which would lessen the blast damage. For the heave there is no major difference though, which also was indicated by the heave measurements.

Comparing the backbreak for the 300 and 330 m levels (Table 4), the first without reduction of vertical blast damage and the latter with reduced vertical blast damage, the importance of vertical blast damage is obvious as the backbreak is reduced by 2 m on average. This was also indicated for the hangingwall, Table 3, even though the difference was not as pronounced. This is probably due to the angle towards the schistosity, both with regard to horizontal blast damage and to the amount of scaled material.

On the footwall the backbreak has been measured for three different rock types with various degree of foliation with no apparent difference in the results. The backbreak, or fallouts, are governed mainly by the occurrence of large-scale shear zones, subparallel to the bench faces.

As a major part of the backbreak seems to be influenced by the vertical blast damage the question arises whether the presplitting is useful in itself. The presplitting does not reduce the heave but it does reduce the backbreak at the lower parts of the bench face and the extent of the diggability zone. It also reduces the influence of the zone of vertical blast damage on the backbreak. With less loose rock on the bench faces it has also improved working safety. The problem with wedge failures decreased when the inclination of the presplit holes was decreased to 70° and has been manageable since. This increased the IRS in the footwall from 47° to 48°. The new ramp design on the hangingwall was also implemented.



**Figure 13** Comparison of horizontal blast damage for the standard blast design and for the presplit blast design developed in the project (Figure 8), 7.5 m below the planned bench level

## 5 Conclusions

This paper has presented the evolution of bench blasting design at the Aitik open pit mine. The historic work undertaken during the 1990s was instrumental as this provided a basis for further development. In particular, the development of practical blast damage criteria and a site scaling law, made it possible to appraise different drill- and charge patterns with respect to the degree of blast damage and the potential of achieving a certain bench face geometry.

The major development project during the last few years, prompted by trimming of footwall slopes, involved investigating the governing factors of the backbreak. The results showed that by decreasing the vertical blast damage, backbreak was reduced by 2 m on the hangingwall. This translates to large reductions in waste rock mining.

On the footwall a slight increase of the interramp slope angle was possible. A blast design including presplitting and an increased number of 165 mm buffer holes helps control backbreak and results in safer benches. The increase in interramp slope angle may seem small (about 1°), but the confidence in the design has also increased through this work.

Hole inclination proved to be a critical factor in achieving desired blasting results. Even small deviations in inclination may result in the need for adjustment blasting at the toe, as well as an increased risk of rock falls. It was also found that small-scale structures had little influence on the blasting result. However, slickensided structures with thicker gouge had a large effect on bench face geometry, regardless of blasting pattern. Fallouts along these structures cannot be avoided even with very light smooth blasting, and contingency plans for such failures must be in place.

The methodology used for follow-up (including measurements of heave, backbreak, hole deviations, etc.) proved to work well and to yield practical results. This coupled with a routine follow-up of slope geometry through aerial photography (every second year) provides a good statistical database on which design decisions can be based.

Future work in further improving blasting and bench design may involve more work on developing bulk charging of decoupled charges. Such work may enable smooth blasting (without presplitting) to be used as an alternative to the presently implemented design.

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