Optimisation of the drill and blast work in the open pit Rabenwald mine of Rio Tinto Minerals

Author: Florian Bauer Leoben, February 2007

STATEMENT OF ORIGINALITY

„I hereby declare that this report is my own work and that it contains, to the best knowledge and belief, no material previously published or written by another person nor material which to a substantial extent has been submitted for another course, except where due acknowledgement is made in the report.”

SUMMARY

I have been assigned by the Naintsch Mineralwerke to evaluate the As-Is State at the Rabenwald mine and the State of the Art in drilling and blasting. Therefore all aspects of drilling and blasting were examined, from the planning process at the beginning, a wall surveying program, the drilling process, the charging of the holes up to the documentation and measurement of ground vibrations and blasting results.

It was found that in many cases to achieve the aim of being State of the Art a better utilisation of already existing resources is recommended. Nevertheless some aspects need a bigger investment, like a new automatic drilling and GPS-guided surface crawler.

What is common for all aspects is that they need a thorough planning process that works in the background of all of them. This is also the objective for the long term, to build up a management system for the whole drilling and blasting process as a central guidance and planning tool to make the process more economic and safer for all involved people.

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1. INTRODUCTION

The Naintsch Mineralwerke, part of the Rio Tinto and Luzenac group extract at the Rabenwald mine about 100 000 t talc a year. To do that more than 2 million tonnes of overburden have to be removed. This is done by drilling and blasting. Talc is mined mechanically using a hydraulic digger.

In order to realise excellence programs, the Naintsch Mineralwerke are working to bring the drilling and blasting work to the highest possible level. In particular the drilling and blasting work should get more economic and safer.

Moreover Rio Tinto wants to create a reference model for surface drilling and blasting and even use the mine as a training centre for drilling and blasting work for the whole company group.

To achieve this target all necessary data to describe the state of the art in drilling and blasting were evaluated and a gap-analysis was conducted to find the economic and safety benefits of a change to the newest standards and techniques.

1.1. THE DEPOSIT

The Rabenwald mine is located on a crest about 2.5 km in the south east of the Rabenwaldkogel at about 1100 m above sea level. It lies right on the boarder between the two political districts Weiz and Hartberg in eastern styria

The talc deposit is part of the sub-eastern-crystalline at the eastern edge of the Alps in a gneiss basement. It is spacious bonded to tectonic overfolded areas that are part of a big faulted zone. This tabular talc faulted zone dips with 5 to 7 degrees into a south and south-western direction and therefore most times parallel to the hills edge (Figure 1).

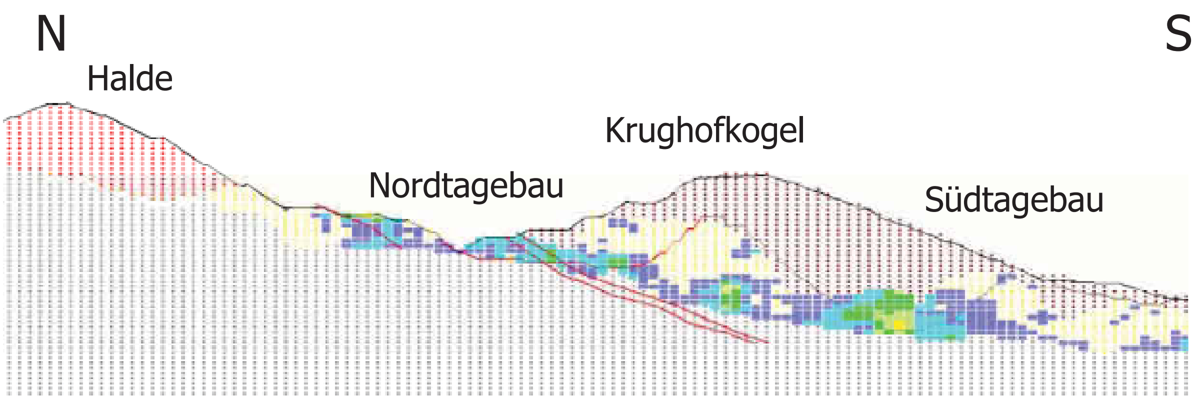


Figure 1: Longitudinal section through the talc deposit, Source [25]

The whole deposit covers an area of about 7.5 km². The talc zone is between 20 and 50 m thick due to different thicknesses of the waste rock strata. The overburden is between a few meters thick where it reaches the surface and 100 to 150 m thick under elevations like the Krughofkogel (see Figure 1).

From the lithology the deposit consists of talc-chlorite-schist, leucophylite and a little bit of dolomite and magnesite. The waste rock stratas are paragneiss, othogneiss and granite-mica schist.

The surrounding rock is in large parts competent rock, where it is untouched. Exceptions are the axial and cross running faulted zones and the pre-damaged areas of the old underground mining operations.

2. Project Overview

The following will explain the goals of this thesis, the approach and the way it was realised.

The thesis is divided into three parts:

The first part is the evaluation of the actual drill and blast work. These comprise:

- Planning of the drilling and blasting work
- Surveying of the wall
- Drilling
- Quality of the boreholes
- Detonators
- Way of initiation
- Blasting agent / column design
- Way of borehole charging
- Documentation of the blast and the blasting quality in terms of fragmentation
- Noise and vibrations

A precise explanation how the data was gathered is shown section 4.

The second part consists of the description of the latest technology in drilling and blasting. This was based on a literature survey.

Further on, in a third part, a gap analysis between the latest technology drilling and blasting and the actual work it is done. This Gap-Analysis comprised also an evaluation of the economical consequences and the safety risks of the gap found.

3. As-Is State

3.1. Layout of the blasting work

At the moment the layout for a blast is defined by the driller and the pit deputy. They visit the blasting site together and define the blasting area. The driller decides about the actual position for each hole, using a wooden stick with self-made marks on it and sprays the position for each hole on the ground. The position of the boreholes is not surveyed nor are they marked in a map.

3.2. Surveying of the Wall

In summer 2006 the Rabenwald mine bought the wall surveying system of the 3G Company. This system allows a photogrammetric surveying of the wall by making two pictures of the wall in a small distance from each other. A reference stick on the picture makes it possible to match the photo later with a scale. The provided computer software compares the two pictures and creates a 3D-image. Further more it is possible to insert boreholes to the 3D-picture, with previously defined burden and spacing, get the total blasted volume in cubic meter or tonnes, and read out the actual burden over the full length of each borehole.

During the data gathering, from August 2006 till the beginning of October 2006 the 3G - system has not been used at all nor was the wall surveyed with a theodolite. It was never recognised that the height of the wall was measured before the drilling process.

3.3. Drilling

In advance of the drilling process material that lies on the edge of the wall for safety reasons as a barricade is pushed down using a bulldozer. Then drilling is conducted with the tophammer drilling machine Atlas Copco F9. The driller drives from each marked top position for each hole to the next and drills the boreholes. Until the middle of September 2006 the holes where drilled with an inclination of 80 degrees then it was changed to 75 degrees. The actual length of each hole is defined by the driller from his experience. The orientation of the drill boom is done with the inclinometers mounted on the drill rig. The azimuth of the drill holes is visually estimated. Therefore the driller tries, if possible, to position the drill rig parallel to the wall and drills the holes.

3.4. Quality of the Boreholes

At the moment the Rabenwald mine does not measure the quality of the boreholes at all. The only thing that is done is that the driller lowers a plumb (piece of metal) to check if the holes are not blocked. Sometimes also a torch or a mirror is used.

3.5. Detonators

The Rabenwald mine is using electric detonators with pyrotechnic delays for blasting. These Polex-detonators are highly insensitive (inner resistance: 0,09 Ohm). They are available in 21 steps including the momentum detonator (delay 0 ms) with 20 ms between each step. This means the delay times are: 0 ms, 20 ms, 40 ms, 60 ms…...400 ms.

3.6. Way of Initiation

The blast is initiated from the top of the borehole using a detonating cord on which the detonator is attached on the sureface (Figure 2).

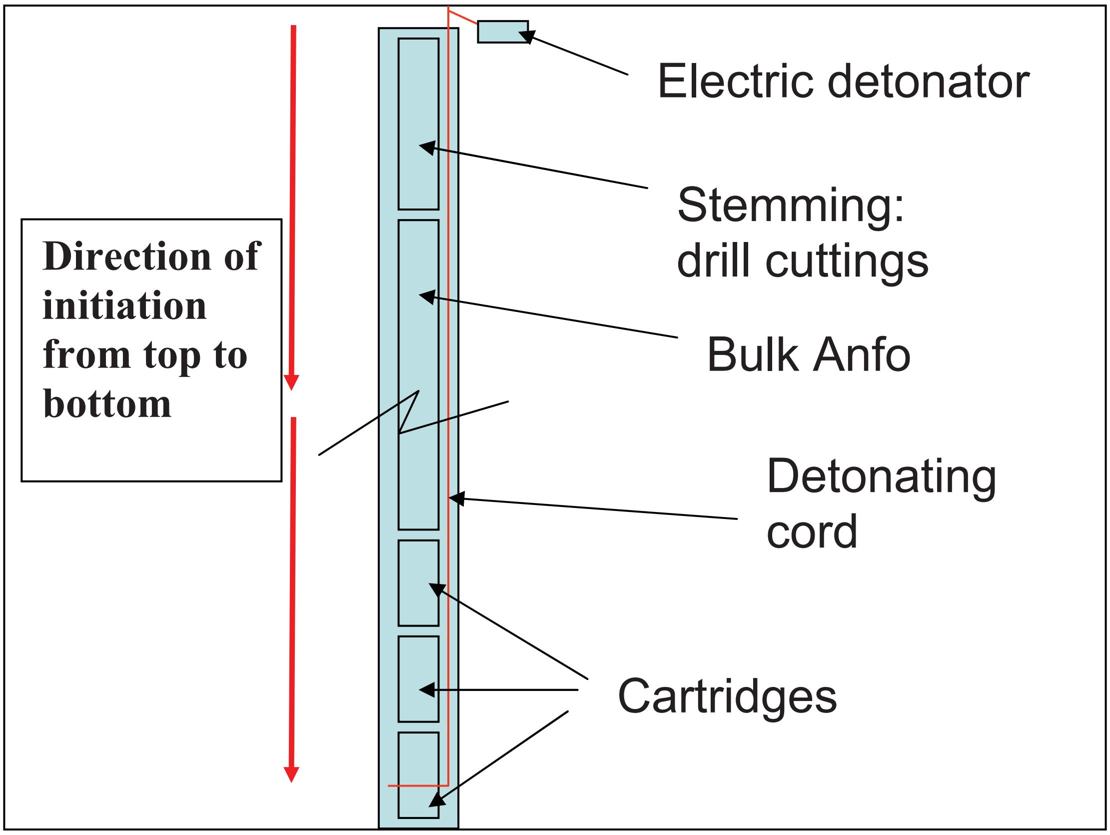


Figure 2: Direction of initiation of a blasthole

3.7. Blastin Agent / Column Design

The way the column is build currently is shown in Figure 2.

Firstly a detonating cord (20 g/m) is fixed on a cartridge, either gelatine or emulsion, and lowered into the borehole. After that another two cartridges follow, depending if the borehole is wet (emulsion) or dry (gelatine). The exact amount is not fixed and depends also on the availability of the explosive. If the hole is filled with water to the top, the whole column is charged with emulsion cartridges.

In dry boreholes, after three cartridges, the hole is charged with Anfo up to a height of 3.5 m from the collar. The rest of the hole is filled with stemming, typically drill cuttings.

On the outside of the hole the detonator is mounted on the detonating cord, using a plastic wrapping. If there is enough time, detonating cords too long are cut off and the part of the detonating cord outside the borehole is covered with drill cuttings.

All explosives are delivered by the Alpspreng-company. The exact specifications of the used explosives are:

Table 1: Specifications of used explosives

Kind of explosive	Name	Length [mm]	Density [g/cm³]	Diameter [mm]	Weight of one package [kg]
Gelatine (cartridged)	Supergel 30	600	1,4	65	2,5/cartridge
Emulsion (cartridged)	Emulgit 82GP	600	1,2	65	2,5/cartridge
Anfo (bulk)	Prillex 1	-	0,82	-	25/bag

3.8. LOADING THE HOLES

By now the workmen drive with a truck to the blasting location. They unload their truck using

Figure 3: The blasters truck with the crane mounted to it

a crane that is mounted to the truck (Figure 3) and distribute the explosives to each borehole manually. This means carrying the 2.5 kg cartridges of gelatine and emulsion explosives and the 25 kg bags of Anfo explosives.

3.9. DOCUMENTATION OF THE BLAST

The blast is documented according to the "Sprengarbeitenverordnung", which is the Austrian regulatory framework for blasting operations. This means the operator of the drilling machine has to record how many holes he has drilled and to which depth and the shot-firer has to draw a plan containing the position of each hole and the amount of explosive it is charged with. This documentation can be seen in Figure 4 (record of the drill operator) and Figure 5 & 6 (record of the shot-firer). All the other records of the blasts analysed in the frame of this study are attached in Appendix A. There is no shot-firer's plan from the 6th and 7th blast. From the 10th blast no data is available at all.

Naintsch Mineralwerke
Ges.m.b.H. Werk
Rabenwald A - 8184
Anger/Stmk

30.08.06

Atlas Copco F9
Bohrprotokoll für die Sprengung Nr. 1

					Betriebsstunden	Fahrz./min.
Datum:	29.8.06	Dieselstart/Uhrzeit	06^{30}	Schichtbeginn	5990	30 min
Revier:	Reithofer	Dieselstopp/Uhrzeit	14^{00}	Schichtende	5997	
Etage:	1090	Bohrkrone Nr. / Ø:	102			
		Bohrist:	Ablasser			

Bohrgeometrie:
29.8.06 CH.6
A 16^{30} — 5997
E 20^{30} — 6001

Wandhöhe H:	10 m	Lochabstand:	3,7
Bohrlochneigung α :	80°	Vorgabe:	3,5
Bohrlochtiefe T: (inkl. Unterbohren)	10 m		

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
29.8.06	1-2				8 m	mittel - Hart
	3-20	3,5	3,7		10 m	— " —
29.8.06	21-35	— " —	— " —		— " —	— " —
Bohrmeter nachbohren		2X mit 5 m oder Troll				
		2X 9 m oder Troll				
Summe:						

(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 4: Drill operator's record of the 1st blast, Source [27]

Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: ..Reithofen........
Ø Wandhöhe : ......10m......
Bohrlochneigung: ....80°.......
Sprengung: am 30.08.2006
Uhrzeit:.......13⁴⁵...........

Gebirge: ....Mittel-Hart......
Bohrlochdurchmesser: ..102......mm
Verantwortlicher Leiter bzw.
Sprengbefugter: ..Hinterbrunner Christian

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 1-18 Stufe
1	3,7m	3,5m	8m					
2			ü 2Bl.	Prillex:	544. →	1350 kg		
3								
4			10m	Supergel:	8 u. →	200 kg		
5			ü 33 Bl.					
6								
7								
8				L = 88 kg ;	R = 220 m		bis Reithofen	
9								
10								
11				Vmax =	4,35 mm/s ist		zu erwarten	
12								
13								
14				UB =	1,86			
15								
16								
17								
18								
19								
20								
Sa.								
Ø				—	—	—		

..... Sohllöcher: Tiefe: ....m
..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff - Zusammenstellung:

Gel.Don.1 (Rowolit)	60/700	................kg
	45/350	................kg
	30/350	................kg
	30/130	................kg
Lambrit (Rowolan)	lose	................kg
Summe:		................kg

ANC - Anteil:

Kopflöcher	......................%
Sonstige-Löcher	.................%
insgesamt	.......................%

Zündmittel:
Det. Zündschnur .....400......m
Zünder(Type) .U....35...........St.
Haufwerk: .4481...m³ = 11202.t
Spez. Sprengstoffverbrauch= .138....g/t
Bemerkungen:

Datum: ....

Figure 5: Shot-firer's record of the 1st blast, Source [27]

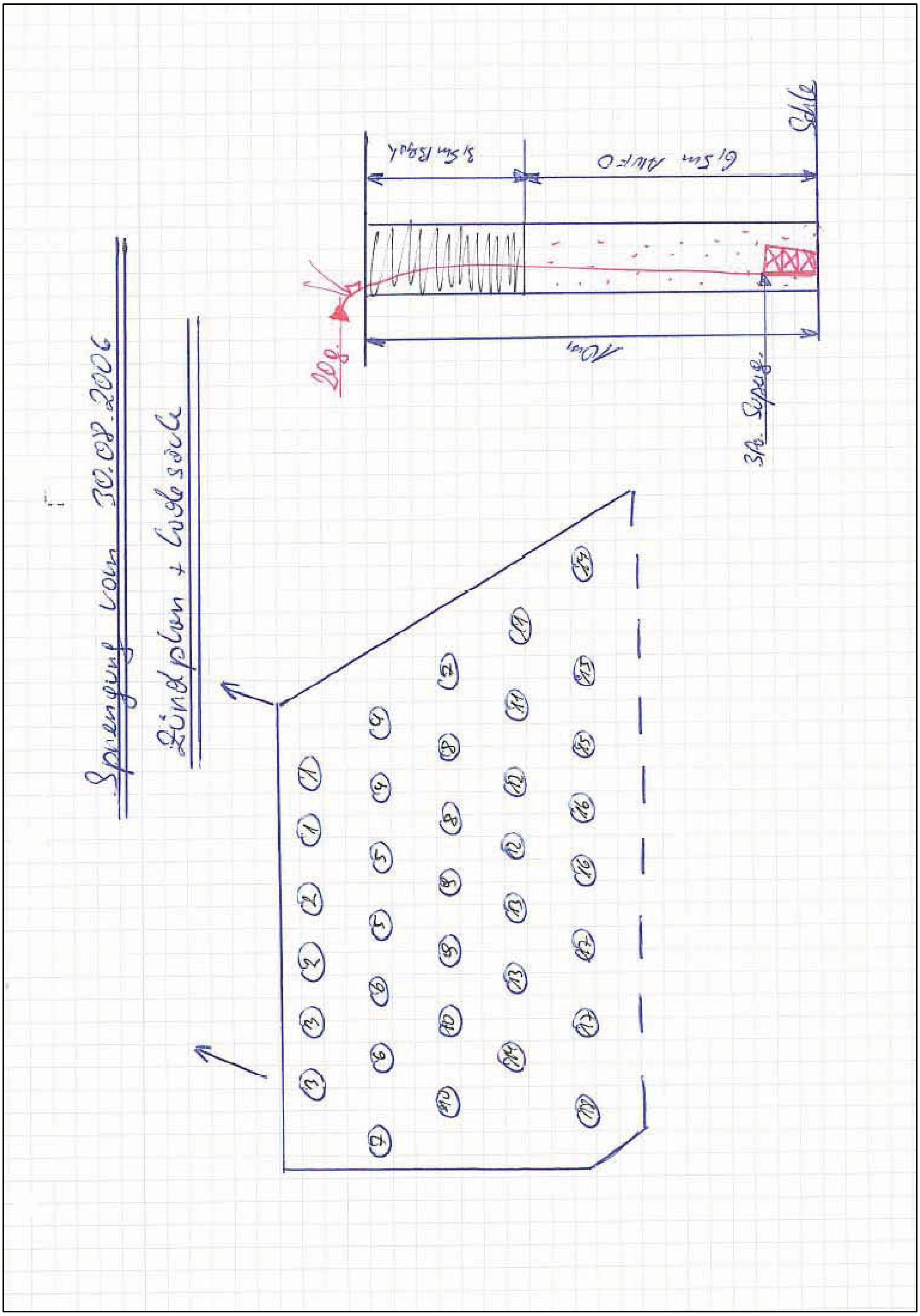


Figure 6: Shot-firer's plan of the 1st blast, Source [27]

3.10. Documentation / Measuring the Blasting Result

At the moment there is no written documentation of the blasting result nor is it measured. The shot-firer just walks back to the blasting area and has a look at the result to make sure all holes detonated successfully.

3.11. Noise and Vibration – Neighbours

The company owns two vibration measurement systems (the Minimate Plus W/ext.Geo and the S3 Std Triaxial-Geophone, both from the Instantel company) which are regularly in use. An example of a vibration measurement event report is shown is Figure 7. All the other event reports are shown in Appendix B. There are no such reports from the 2^{nd}, 3^{rd}, 6^{th}, 9^{th} and 10^{th} blast. To my extent of knowledge noise is not measured.

One neighbour who is within the 300 m safety radius of the blasts most times has to be contacted by telephone and fax about the actual blasting time. All other neighbours know that the official blasting times are between ten o'clock am and two o'clock pm every day.

3.12. Clearing of the Blasting Area

The Austrian law says that every blasting area has to be cleared within a radius of 300 m. During a blast all streets to the pit are secured by a truck, the pit deputy or workmen, who are all equipped with walkie talkies, to be able to stop the blast in case of an unexpected situation. The positions for all people who close streets in case of blasts are always the same; the exact positions of the 300 m safety radius are not read out anew every blast nor are they marked in a plan.

Event Report

Date/Time	Vert at 13:43:48 August 30, 2006
Trigger Source	Geo: 0.510 mm/s Mic: 110 dB(A)
Range	Geo :254 mm/s
Record Time	1.75 sec (Auto=1Sec) at 1024 sps
Job Number:	1

Serial Number	BE9094 V 7.01-4.35 MiniMate Plus
Battery Level	6.1 Volts
Calibration	October 24, 2003 by Instantel Inc.
File Name	K094BAZ2.500

Notes

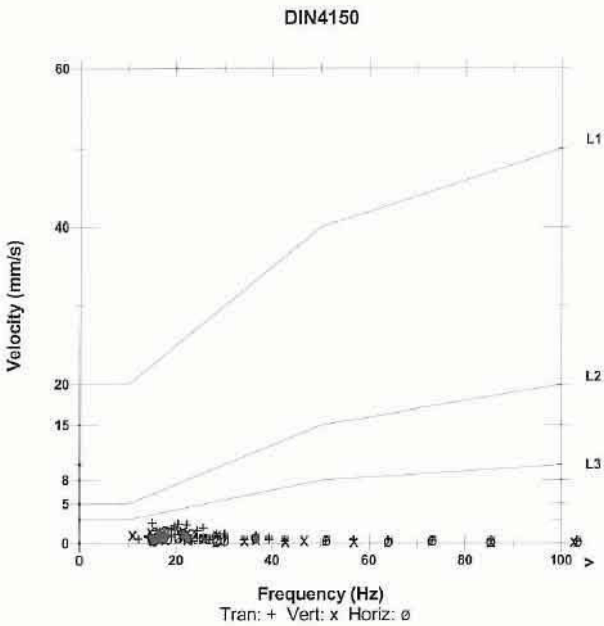
Standort:	Wohnhaus Reithofer
Firma:	RT- Rabenwald
Name:	Abla er Alois
Allgemein:	Ortbauer/NO/E 1096

Bemerkungen

Post Event Notes

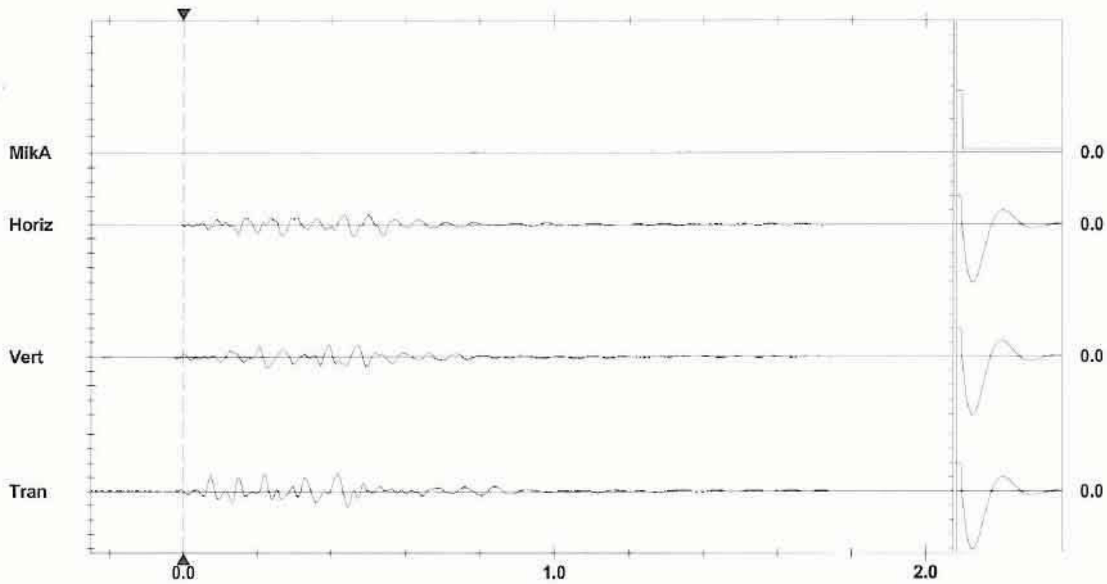
Microphone	'A' Weight
PSPL	<50 dB(A) at 0.028 sec
ZC Freq	N/A
Channel Test	Check (Freq = 0.0 Hz Amp = 0 mv)

	Tran	Vert	Horiz	
PPV	2.54	1.65	1.65	mm/s
ZC Freq	15	21	18	Hz
Time (Rel. to Trig)	0.418	0.392	0.462	sec
Peak Acceleration	0.0398	0.0265	0.0265	g
Peak Displacement	0.0253	0.0184	0.0147	mm
Sensorcheck	Passed	Passed	Passed	
Frequency	7.3	7.6	7.3	Hz
Overswing Ratio	4.0	3.6	4.0	



Peak Vector Sum 2.73 mm/s at 0.224 sec

N/A: Not Applicable



Time Scale: 0.20 sec/div **Amplitude Scale:** Geo: 2.00 mm/s/div Mic: 1.000 dB(A)/div Sensorcheck

Trigger = ▶ ◀

Printed: September 8, 2006 (V 7.01 - 7.01) Format Copyrighted 1996-2003 Instantel Inc.

Figure 7: Blast vibrations event report of the 1st blast, Source [27]

4. Taken Measurements and Data Gathered

Table 2 shows an overview of blasts analysed in the frame of this study, of data gathered and when it was measured.

Table 2: Overview of the gathered data

		parameters registered						
	date of blast	3G wall surveying	Surveying of the holes with theodolite	borehole deviation measurement	charging documentation	vibration measurement	noise measurement	photos of blast
blast 1	30.08.2006			(x)	x	x	x	x
blast 2	01.09.2006			(x)	x	x	x	x
blast 3	05.09.2006		x	(x)	x	x	x	x
blast 4	07.09.2006	x	x	(x)	x	X value from company	x	x
blast 5	09.09.2006	x	x	(x)	x	X value from company	x	
blast 6	15.09.2006		x	(x)	x	x	x	x
blast 7	19.09.2006	x	x	(x)	x		x	x
blast 8	26.09.2006		x	x	x	x	x	
blast 9	28.09.2006	x	x	x	x	x	x	
blast 10	05.10.2006		x	x	x		x	

(x)….values were measured but can not be used because of wrong calibrated measurement device

4.1. Surveying with the Theodolite

The following two figures (Figure 8 & 9) show a typical example of a surveyed blasting plan. All other plans of surveyed holes are attached in Appendix C. The lines between the red cycles show the edge of the wall, the green cycles are the drilled boreholes.

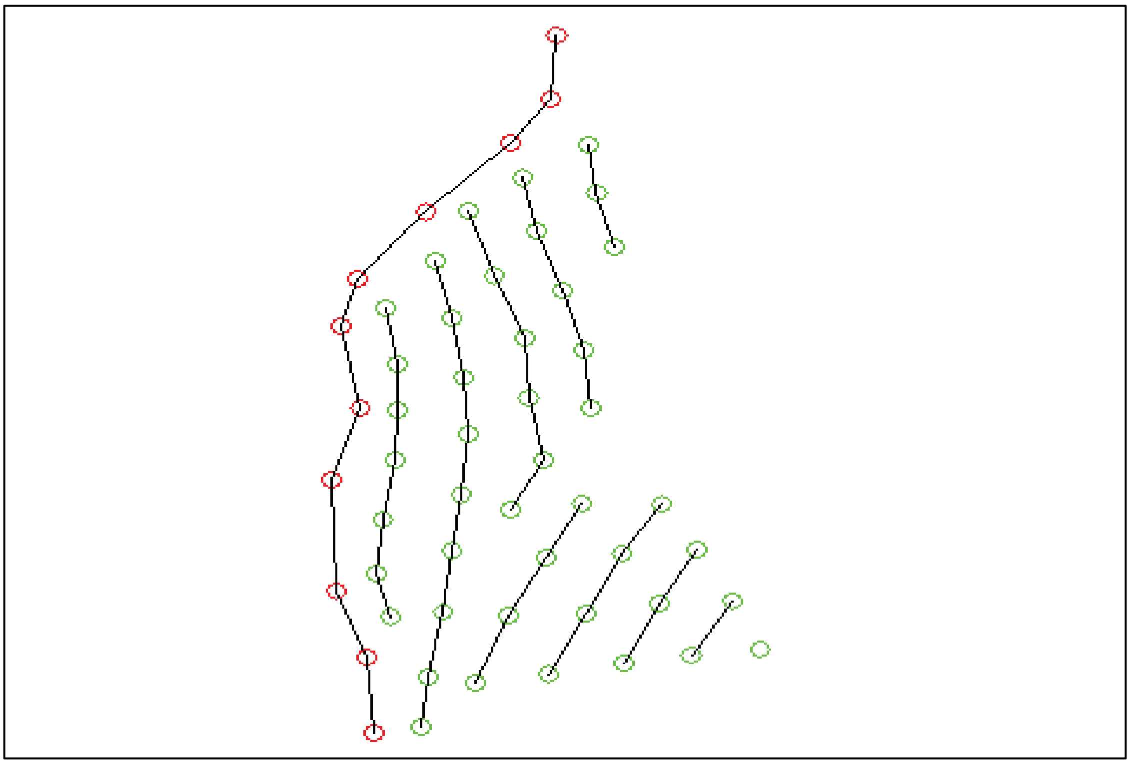


Figure 8: Surveyed shot plan of the borehole starting points from the 10th blast

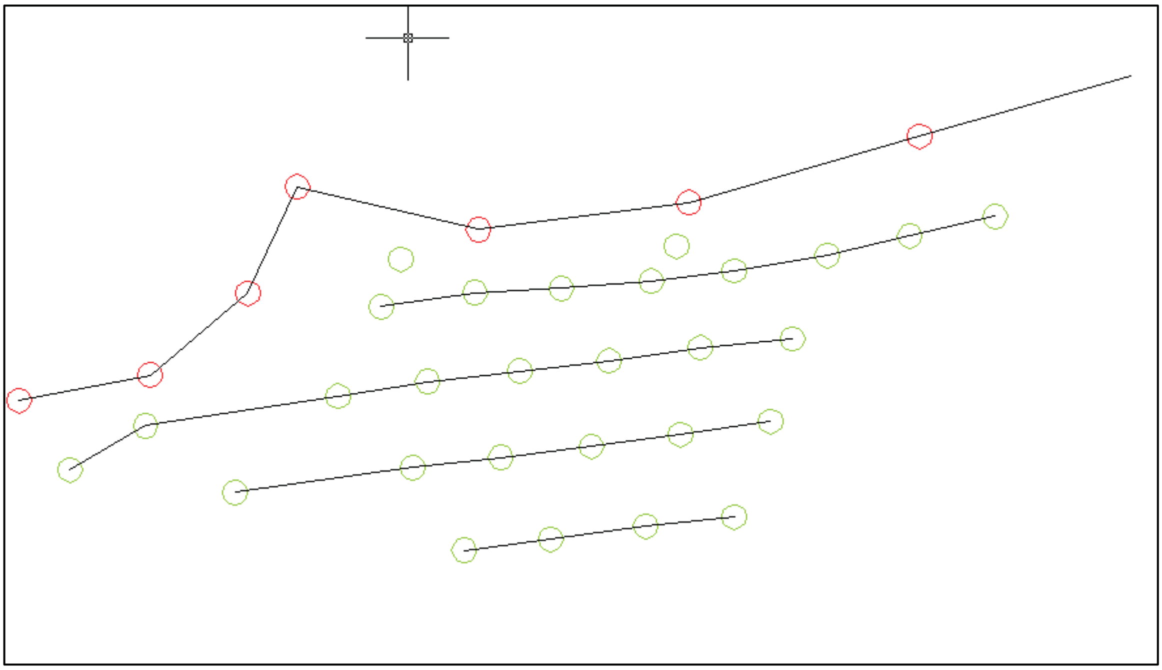


Figure 9: Surveyed shot plan of the borehole staring points from the 9th blast

In order to measure the borehole starting points the theodolite was positioned on a place with known coordinates. Then a mirror was positioned over each hole and the distance to the mirror as well as the vertical and horizontal angle measured. From that the position for each hole could be calculated.

4.2. Surveying of the Wall with the 3G-System

During the data gathering for the 3G-system two problems occurred, which made it often impossible to get a good picture of the blasted walls. Sometimes the wall or parts of it were not visible because of pushed off muck (Figure 12) on the toe of the wall or the wall was not scaled off at all. Another time is was just not possible to see the wall because of too big distances between the wall and the possible photo position or there was no accessible bench in front of the wall to make a photo. Even though some photos created useful results. These can be seen in Figure 10 to 13. All the other pictures of surveyed walls are shown in Appendix D.

Figure 10: 3G-front-picture from the 4th blast

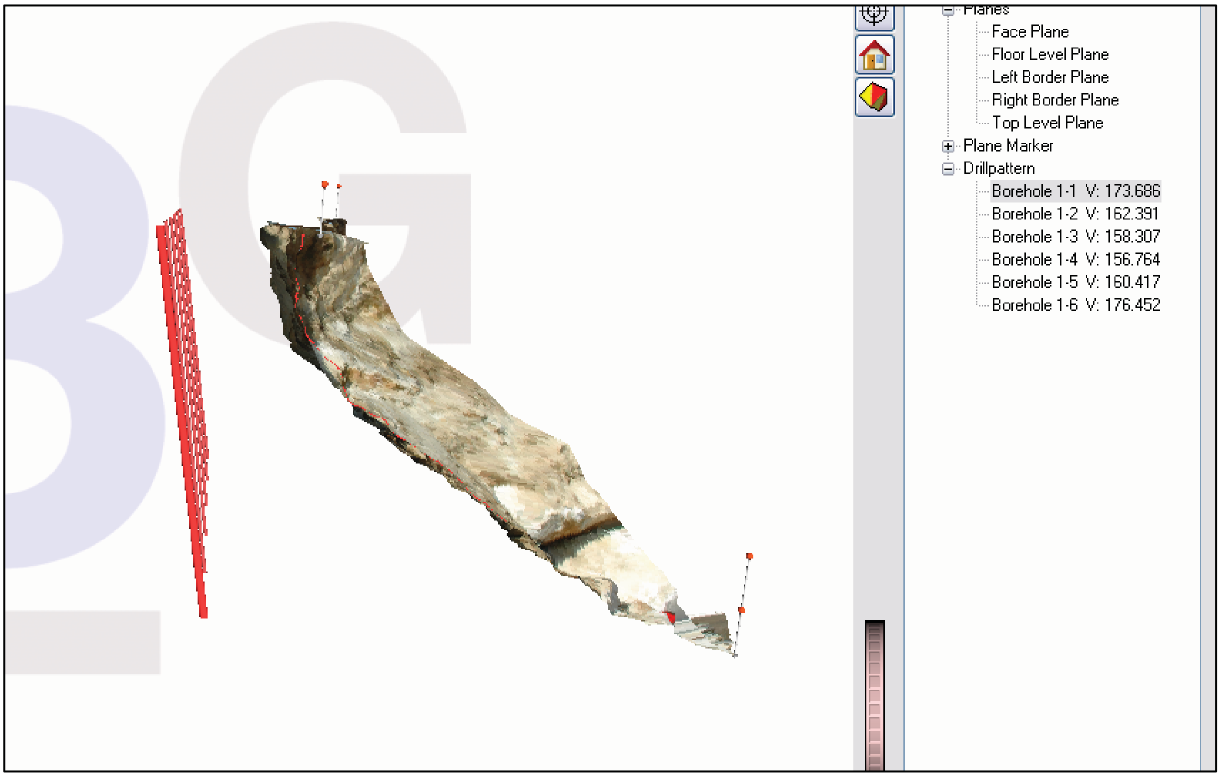


Figure 11: 3G-side-picture from the 4th blast

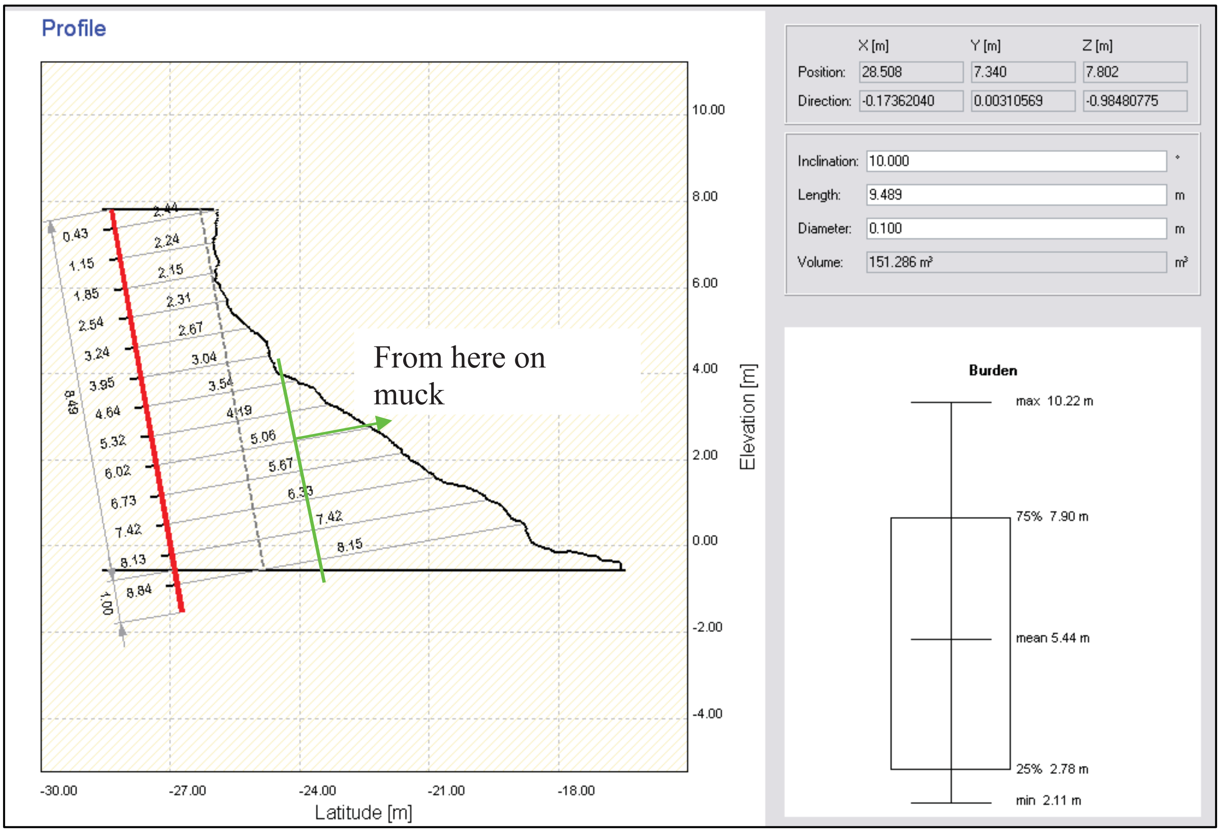


Figure 12: Picture of the first borehole with burdens from the 4th blast

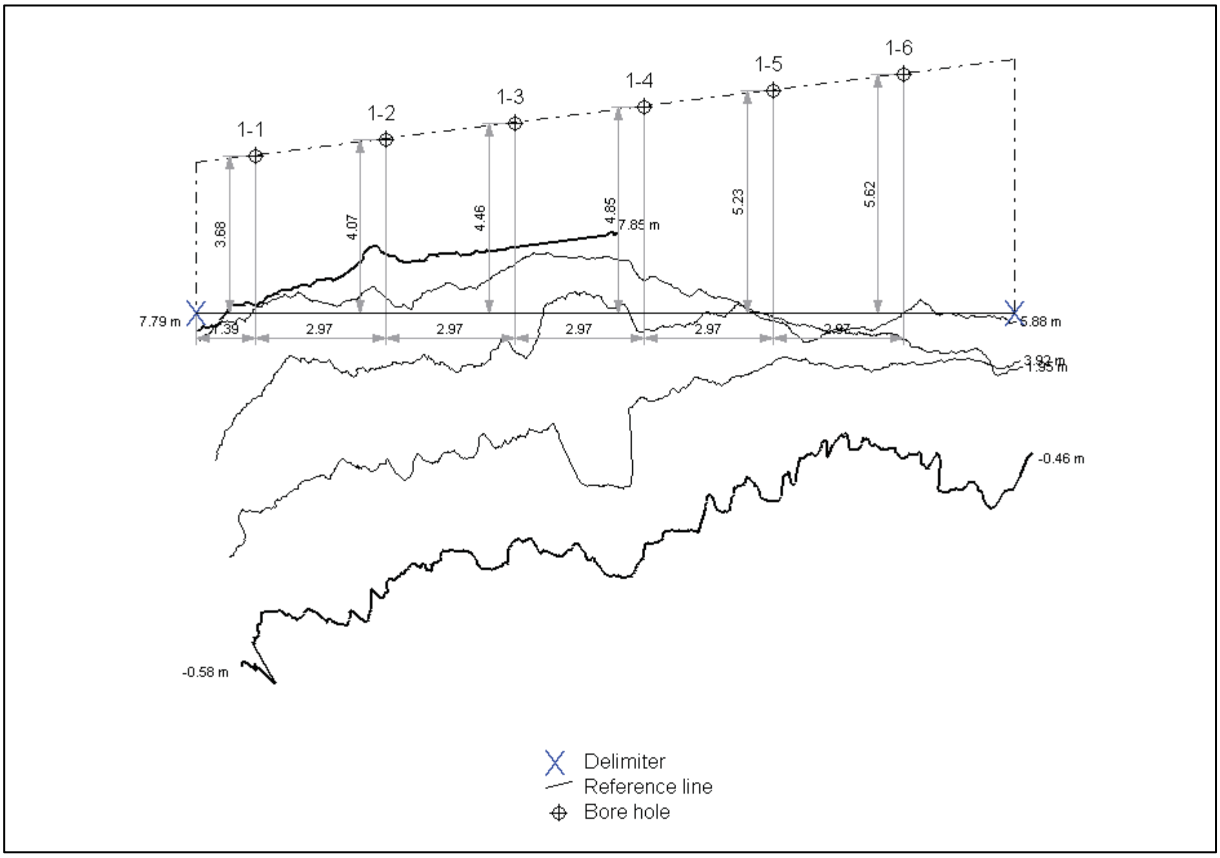


Figure 13: Plan view of the wall from the 4th blast

4.3. Quality of the Boreholes

For measuring the deviation of the boreholes the Boretrack system of MDL was used. Therefore a measurement rod was lowered in the holes and re-raised in 1 m increments. Each meter the deviation from the vertical is measured and the value saved. Software then connects these points and displays the actual run of the hole and the position compared to the other holes as well as the intended direction.

Unfortunately only 3 of the 10 blasts can be used, because the compass of the measurement rod was not calibrated correctly. Nevertheless it can be said that the holes from the first blasts were really bad. One was not even able to see to the bottom of the holes with a torch that was lowered into every hole. For 75 % of the holes the light vanished before the depth of 7 m, which is already an information about the poor quality of the holes. This circumstance changed after the first presentation on gathered data at the company and the holes got quite better. This can be seen from the comparison of blast 6 to blast 7 in Table 3.

Table 3: Comparison of vanishing lights from the 6th and 7th blast

6th blast		7th blast	
Number of Borehole	Light vanishing [m]	Number of Borehole	Light vanishing [m]
14	3	38	7,3
39	3,4	7	7,8
13	3,7	21	8,4
40	4	39	8,4
36	4,2	32	9,3
5	4,5	10	10,3
12	4,6	33	10,6
8	5	36	10,7
19	5,1	8	10,8
21	5,1	9	10,8
34	5,3	29	10,9
42	5,4	37	11
4	5,5	41	11,4
41	5,5	5	11,5
3	5,6	27	11,5
30	6,3	6	11,6
17	6,4	14	11,6
22	6,4	22	11,6
35	6,4	40	11,6
38	6,5	1	11,7
25	6,6	18	11,7
33	6,6	24	11,7
2	6,7	30	11,7
31	6,7	13	11,8
32	6,7	17	11,9
24	6,8	42	11,9
37	6,8	3	12
1	6,9	11	12

6	6,9	16	12
23	7	25	12
27	7	15	12,1
44	7	26	12,2
43	7,1	20	12,3
10	7,2	2	visible till bottom
16	7,2	4	visible till bottom
20	7,3	12	visible till bottom
7	7,4	19	visible till bottom
11	7,6	23	visible till bottom
9	7,7	28	visible till bottom
15	7,7	31	visible till bottom
28	7,7	34	visible till bottom
26	8,2	35	visible till bottom
29	8,5	43	visible till bottom
18	9,3		

After that the quality of the holes remained better, which was underlined by the Boretrack measurements. The lowering of the torch was not done again because of lack of time.

The evaluated borehole deviations are shown in Figure 14 to 17. The red lines show the intended direction of the holes, the green lines are the actual location.

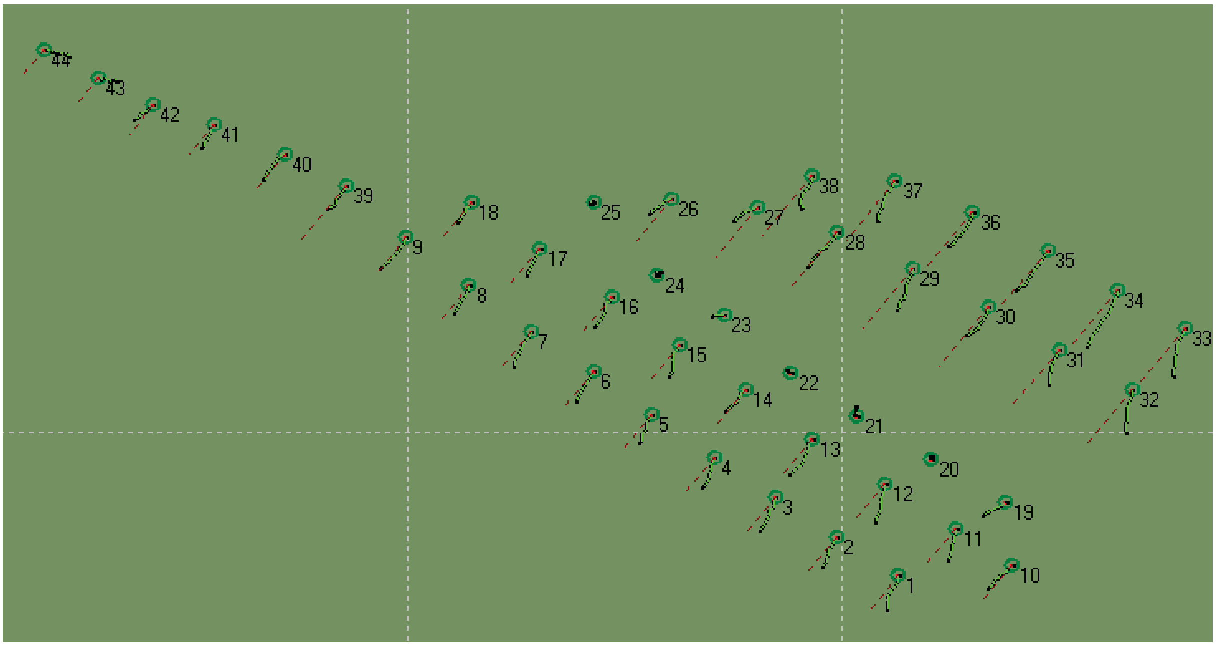


Figure 14: Borehole deviation display from the 8th blast

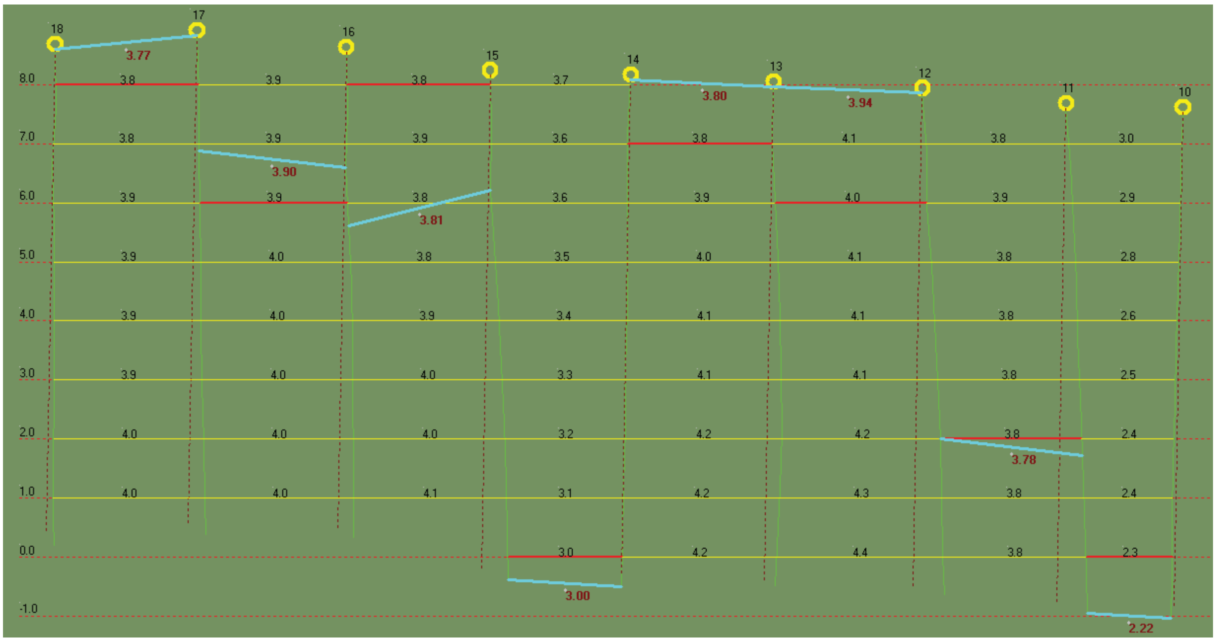


Figure 15: Front view of the boreholes from the 8th blast

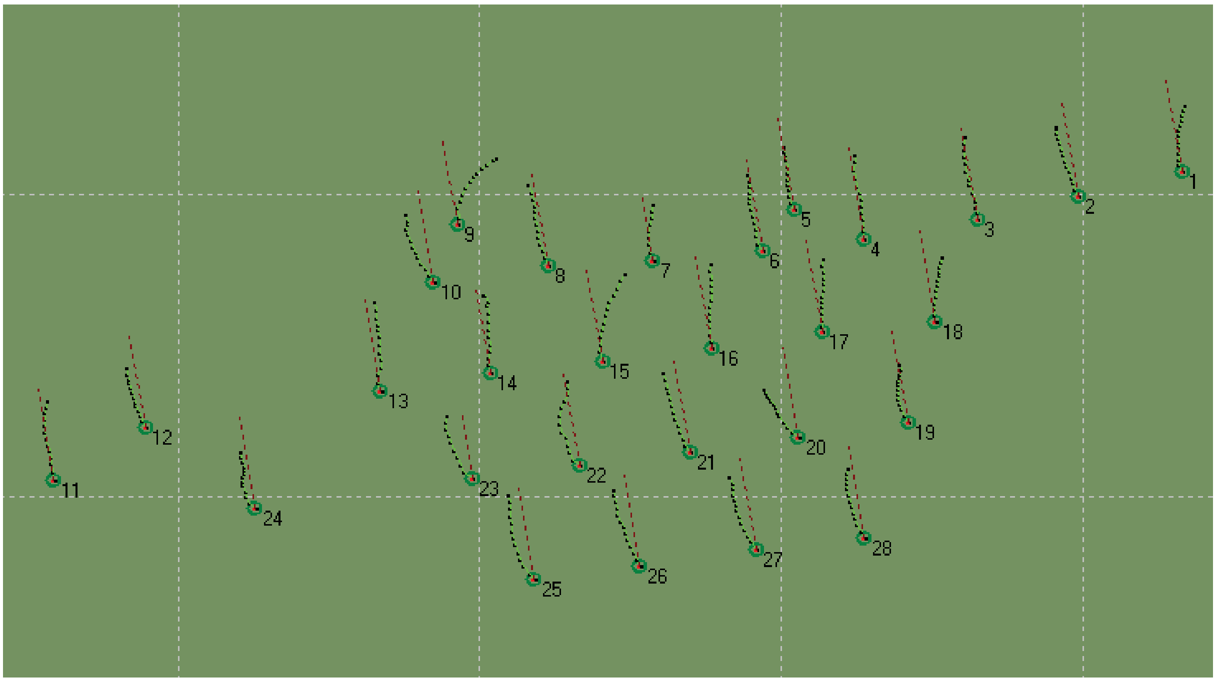


Figure 16: Borehole deviation display for the 9th blast

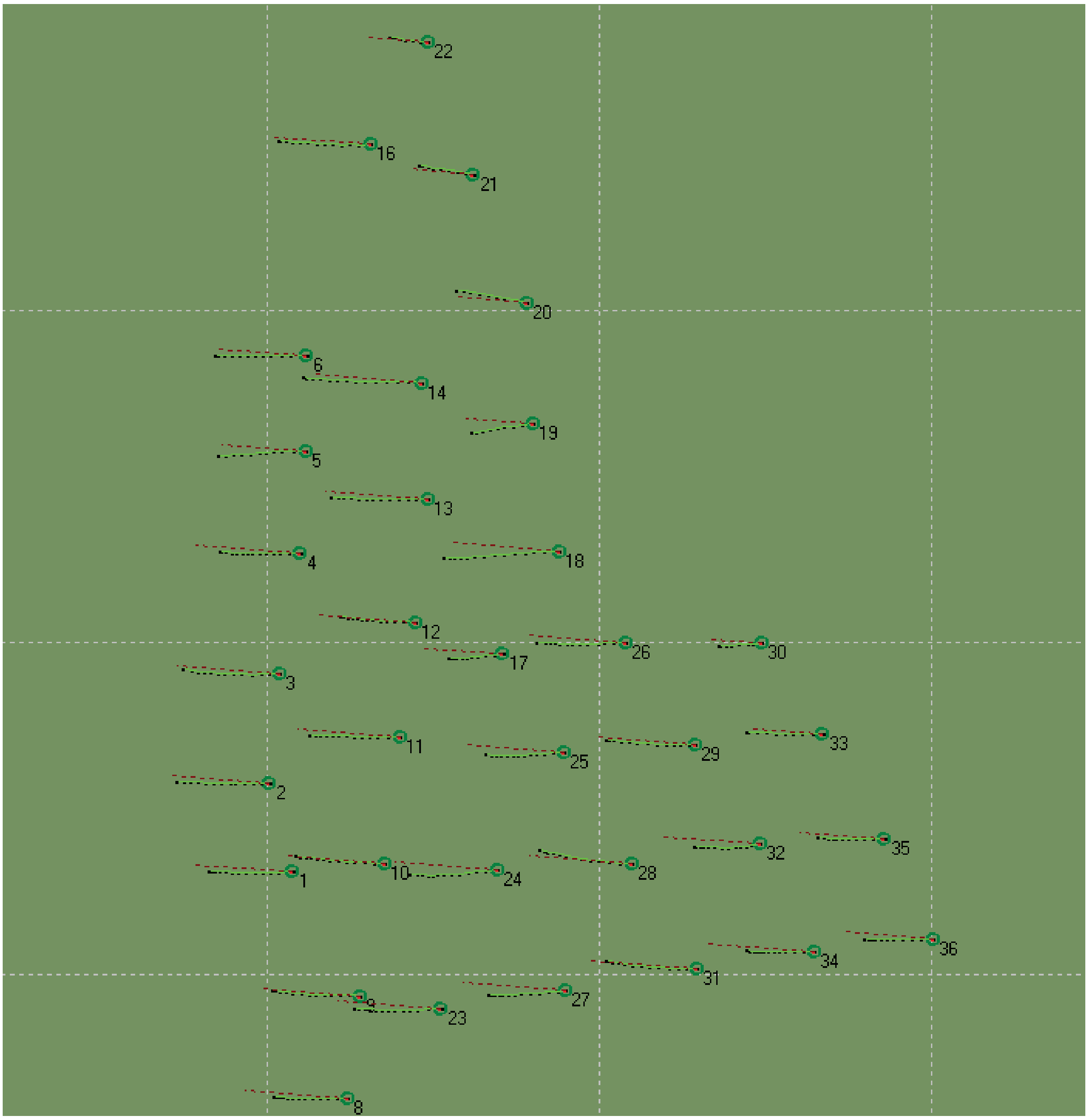


Figure 17: Borehole deviation display from the 10th blast

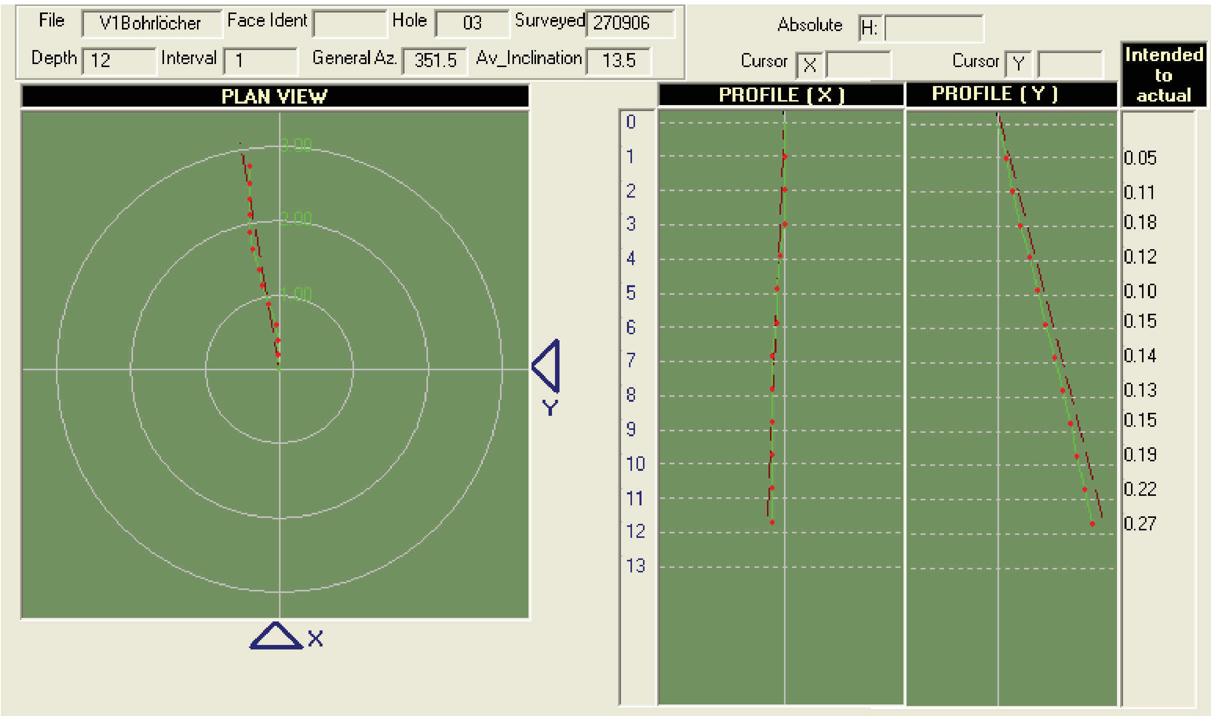


Figure 18: Detailed display of hole number 3 from the 9th blast

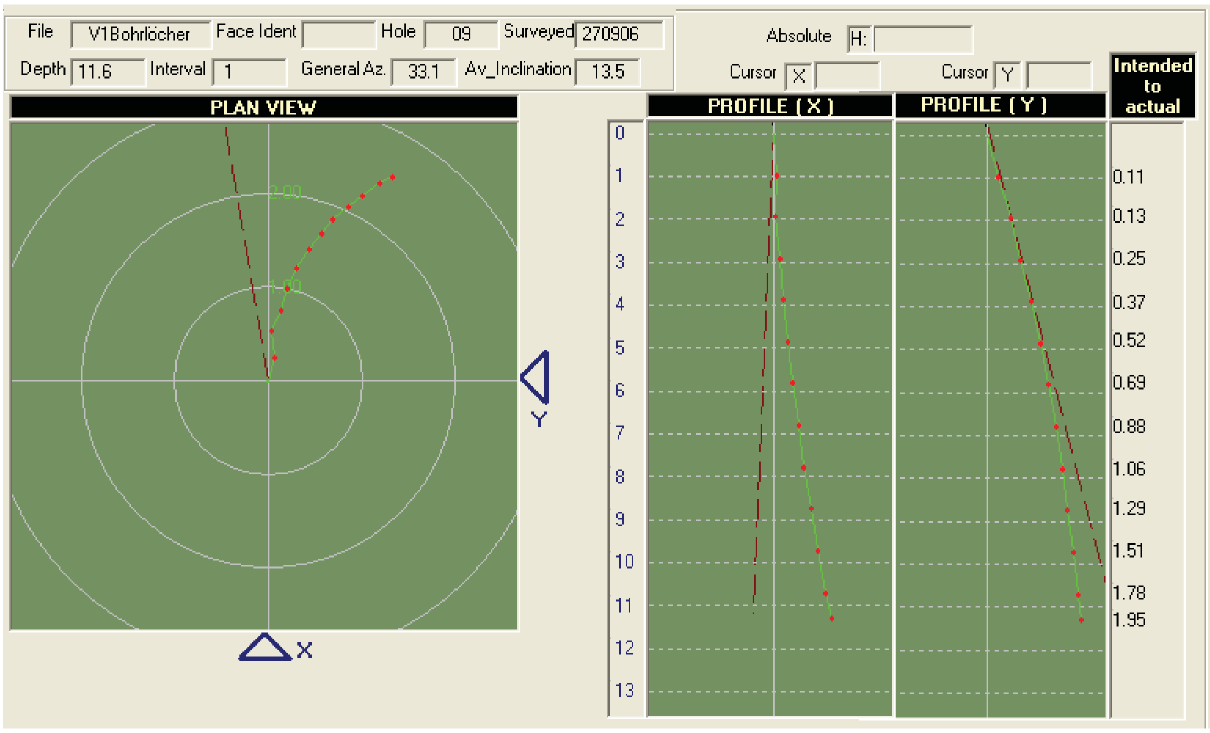


Figure 19: Detailed display of hole number 9 from the 9th blast

Figure 18 and 19 also show the differences of borehole quality within one drilled blast, whereas hole three in Figure 18 is nearly ideal, hole 9 in Figure 19 deviates more than 1 m from it's planned position. Table 4 shows and overview of the measured borehole deviation in relation to the planed end position.

Table 4: Overview of borehole deviations

Blast 8		Blast 9		Blast 10	
Hole number	Deviation [m]	Hole number	Deviation [m]	Hole number	Deviation [m]
1	0,89	1	1	1	0,61
2	0,85	2	0,79	2	0,74
3	0,79	3	0,27	3	0,35
4	0,95	4	0,32	4	0,83
5	0,95	5	0,97	5	0,54
6	0,68	6	0,47	6	0,41
7	0,61	7	0,43	7	not measured
8	0,84	8	0,32	8	0,96
9	0,32	9	1,95	9	0,48
10	0,74	10	0,85	10	0,41
11	1,11	11	0,46	11	0,5
12	0,98	12	1,08	12	0,76
13	0,4	13	0,33	13	0,42
14	0,92	14	0,32	14	0,55
15	1,06	15	1,29	15	not measured
16	0,7	16	0,54	16	0,33
17	0,95	17	0,8	17	0,87
18	1,15	18	1,18	18	0,76
19	1,17	19	1,13	19	0,61
20	0,22	20	1,51	20	0,11
21	0,51	21	0,47	21	0,31
22	0,16	22	0,24	22	0,71
23	0,53	23	0,42	23	0,63
24	0,31	24	1,14	24	0,74
25	0,1	25	0,37	25	0,84
26	1,44	26	0,57	26	0,56
27	1,95	27	0,67	27	0,92
28	1,34	28	0,67	28	0,27
29	1,85			29	0,39
30	1,96			30	0,43
31	2,19			31	0,5
32	2,17			32	1,03
33	1,84			33	0,34
34	0,88			34	1,32
35	1,24			35	0,62
36	1,65			36	0,65
37	1,74				
38	2,25				
39	2,14				
40	0,95				
41	0,73				
42	0,8				
43	2,31				
44	2,42				

4.4. Documentation of Charging

The charging process of the holes was documented completely. Table 5 shows an example of documentation. All other documentation sheets are in Appendix E.

Table 5: Data sheet of the 1st blast

Blast date 30.08.2006 (13:45) Bench: 1060							
		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Hight	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	3	1,5		1,5	3,5	drill cuttings	
2	3	1		1,75	3,5	drill cuttings	
3	2	3		2	3,5	drill cuttings	
4	2	3		1,5	3,5	drill cuttings	
5	1	3		2	3,5	drill cuttings	
6	1	3		3,25	3,5	drill cuttings	
7	7	2		0,5	3,3	drill cuttings	
8	6	2		0,5	4	drill cuttings	
9	6	1,5		1,25	3,5	drill cuttings	
10	5	3		1,75	3,5	drill cuttings	
11	5	3		2	3,5	drill cuttings	
12	4	3		1,75	3,5	drill cuttings	
13	4	3		2	3,5	drill cuttings	
14	10	2		1	3	drill cuttings	
15	10	1		0,5	3,5	drill cuttings	
16	9	3		3	3,5	drill cuttings	
17	9	3		2	3,5	drill cuttings	
18	8	3		1,5	3,5	drill cuttings	
19	8	3		2	3,5	drill cuttings	
20	7	3		1,75	3,5	drill cuttings	
21	14	2		0,75	3,5	drill cuttings	
22	13	3		1,75	3,5	drill cuttings	
23	13	2		0,75	3,5	drill cuttings	
24	12	3		1,5	3,5	drill cuttings	
25	12	3		1,75	3,5	drill cuttings	
26	11	3		1,5	3,5	drill cuttings	
27	11	3		1,5	3,5	drill cuttings	
28	18	2		1,75	3,5	drill cuttings	
29	17	2		1,5	3,5	drill cuttings	
30	17	2		0,75	3,5	drill cuttings	
31	16	1		0,75	3	drill cuttings	
32	16	3		1,5	3,5	drill cuttings	
33	15	3		1,5	3,5	drill cuttings	
34	15	3		1,5	3,5	drill cuttings	
35	14	3		1,25	3,5	drill cuttings	

4.5. VIBRATION MEASUREMENT

Vibrations were measured using the VIBRAS-system of Walesch Elektronik together with up to four geophones. Measurements were conducted once at the house of Mr. Stoppacher, about 500 m away from the blast and all the other times on the next neighbour's property, around 200 m away from the blast. The first measurement at Mr. Stoppacher's house gave no results at the trigger level of 1 mm/s. All other measurements at Mr. Reithofer's house were undertaken at a trigger level of 0.1 mm/s.

Figure 20 shows a map of the mine and the houses of Mr. Stoppacher and Mr. Reithofer.

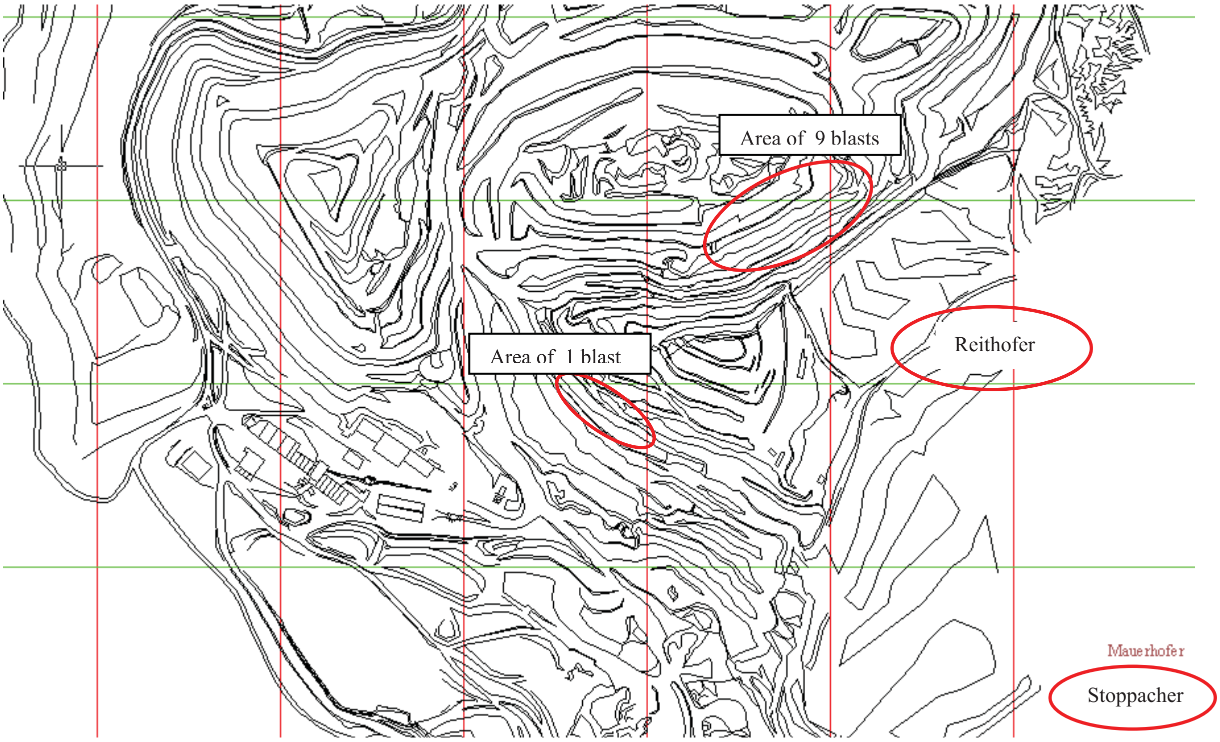


Figure 20: Map of the Rabenwald mine showing the blasting areas and the neighbours Reithofer and Stoppacher, Source [26]

Table 6 shows a summary of the measured vibration velocities.

Table 6: Measured vibration velocities

Distance [m]	Vibration velocities [mm/s]
370	2,54
330	6,07
280	4,7
240	2,22
230	9,41
220	8,94
220	2,35
220	4,23
220	4,31
220	4,11
200	4,69
180	16,05
180	11,56
180	6,47
130	6,07
80	30,58
80	8,03

The vibration velocity and the frequency that where measured right at the next neighbour's house are shown in Figure 21.

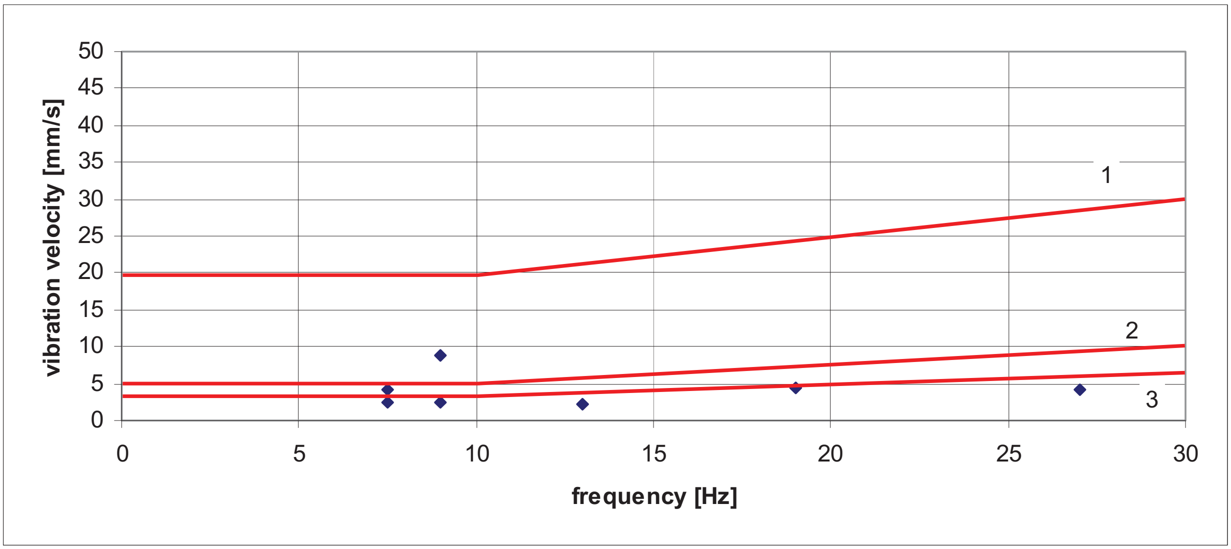


Figure 21: A DIN 4150 diagram showing the vibration velocity versus frequency

The DIN 4150 diagram is divided into 3 classifications of buildings (line 1 to 3).Values below line 1 represent industrial buildings, values below line 2 represent residential buildings and values below line 3 represent buildings that are listed.

The company's goal is to stay below line 3 whenever possible and this goal is reached for 71 % of the blasts.

4.6. NOISE MEASUREMENTS

Noise measurements were conducted with the Norsonic Sound Level Meter. The results are shown in Figure 22.

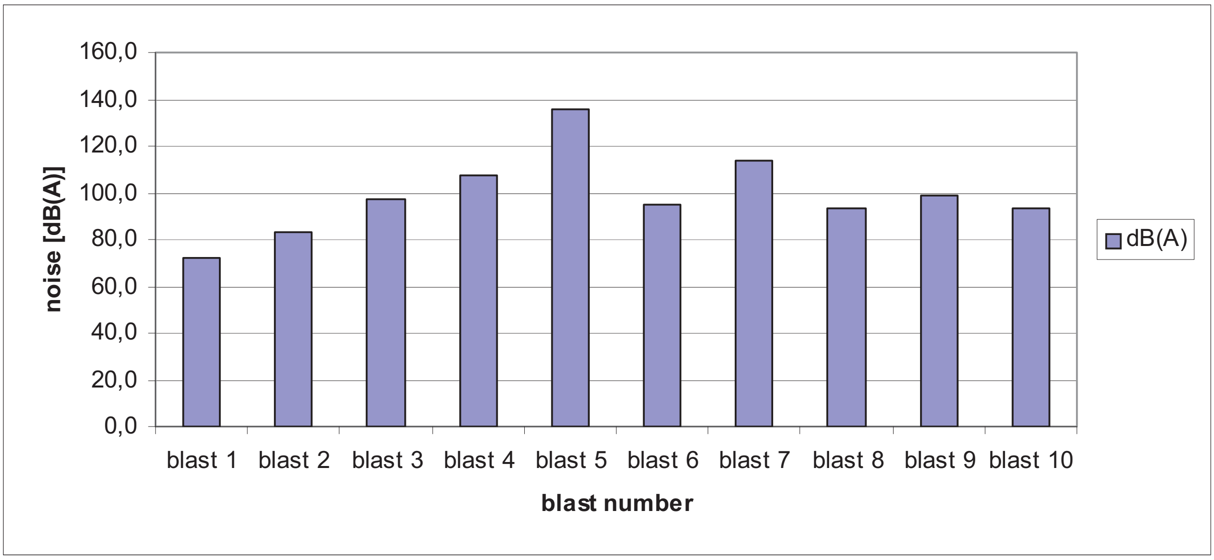


Figure 22: Results from noise measurements

The value from the 1st blast was taken at Mr. Stoppacher's house about 500 m from the blast. The values from the 5th and 7th blast were taken inside the mine about 150 m away from the blast with detonating chords hanging out of boulders that were also fired.

5. ANALYSIS OF DATA MEASURED

5.1. ANALYSIS OF BLAST GEOMETRY

5.1.1. BURDEN AND SPACING

From the measured positions for all boreholes the actual drilled burdens and spacings were calculated for the borehole staring points. The spacing is the distance from each borehole to the next hole; the burden is the distance from the borehole to the next drillhole row. This is shown in Figure 23.

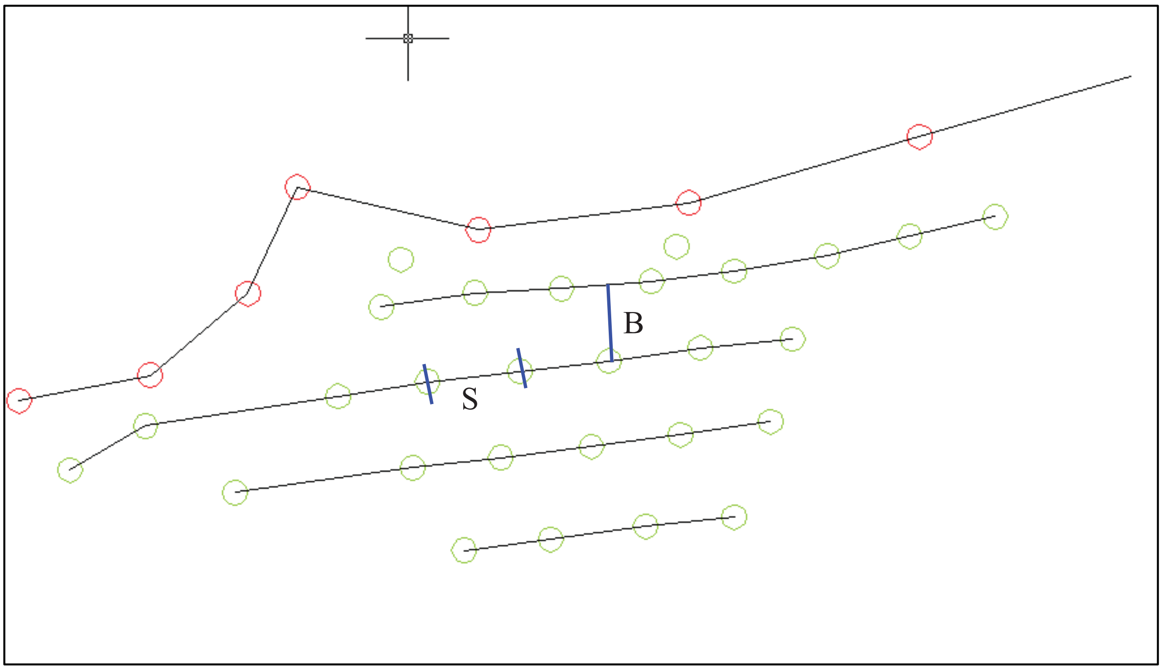


Figure 23: Explanation of spacing S and burden B

From the documented ten blasts eight were surveyed with the theodolite. Then burdens and spacings were calculated by summarizing all values and building an average. The calculated average burdens and spacings are shown in Table 7 and Figure 24.

Table 7: Summary of average burdens and spacings

	spacing	burden
blast 3	3,28	3,55
blast 4	3,19	3,35
blast 5	2,73	3,57
blast 6	3,44	3,40
blast 7	3,39	2,55
blast 8	3,60	3,24
blast 9	4,94	3,15
blast 10	3,69	3,04

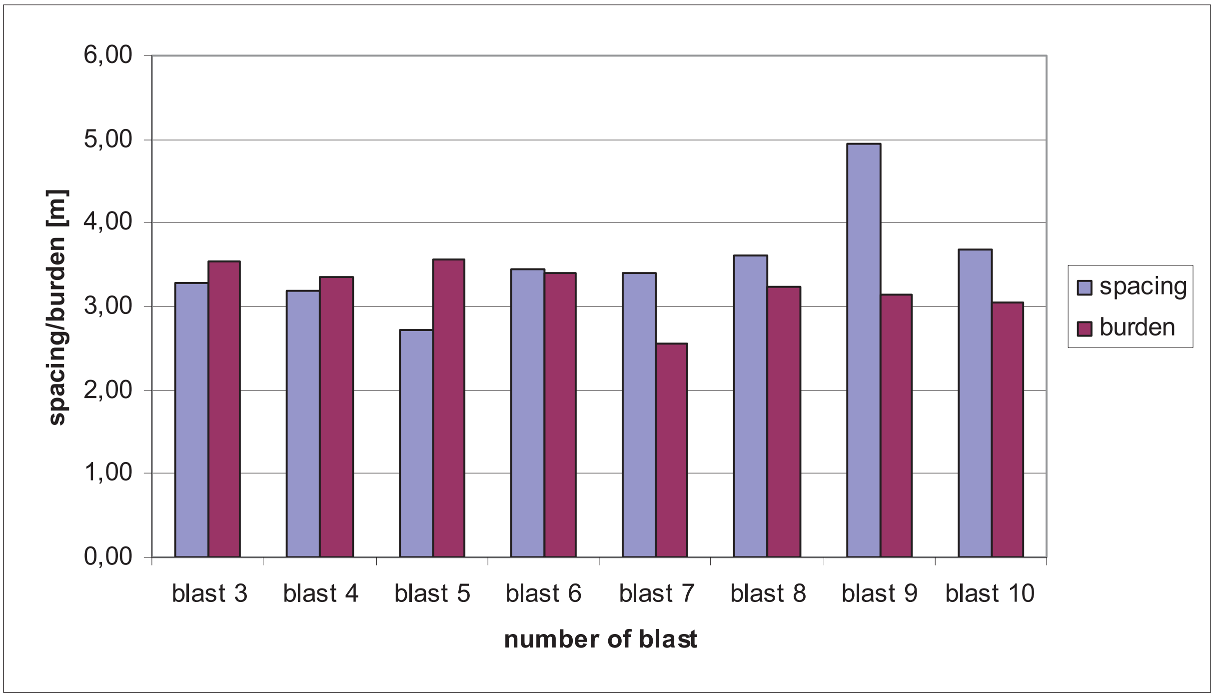


Figure 24: Average burdens and spacings from the eight surveyed blasts

As can be seen the values vary a lot between the different blasts, spacings from 2.7 m to 4.9 m and burdens from 2.5 m to 3.6 m. Moreover there is no regularity (e.g.: the burden is always bigger than the spacing) visible in the blast geometry. In order to demonstrate these variations even better Figure 25 shows the variation between the biggest and the smallest burdens and spacings for eight surveyed blasts.

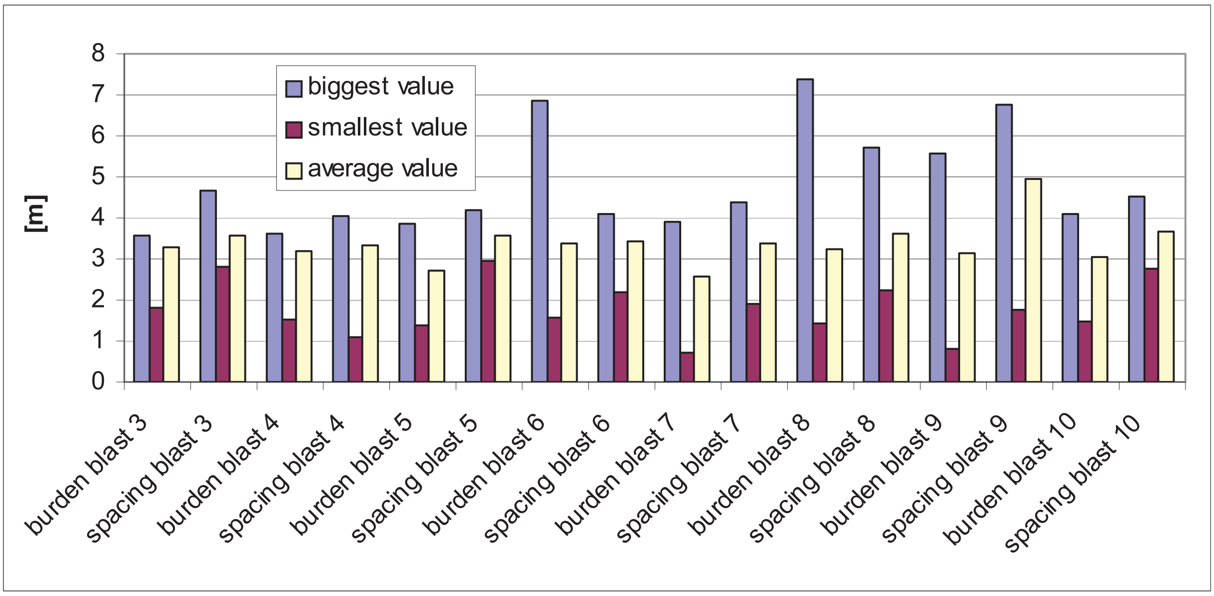


Figure 25: Difference between smallest and biggest burdens and spacings for the eight surveyed blasts

5.1.2. Powder Factor

The powder factor is defined by the amount of explosives per cubic meter or tonne blast material. The allocated amount of material is derived from multiplying the average burden and spacing with the hole length (derived from borehole deviation measurement). The amount of explosives was documented during the charging of the holes.

The average powder factors are displayed in Table 8 and Figure 26.

Table 8: Average powder factors

	Powderfactor	
	g/m³	g/t
blast 3	384,1	147,7
blast 4	401,9	154,6
blast 5	495,0	190,4
blast 6	395,4	152,1
blast 7	492,9	189,6
blast 8	341,7	131,4
blast 9	376,3	144,7
blast 10	374,8	144,2

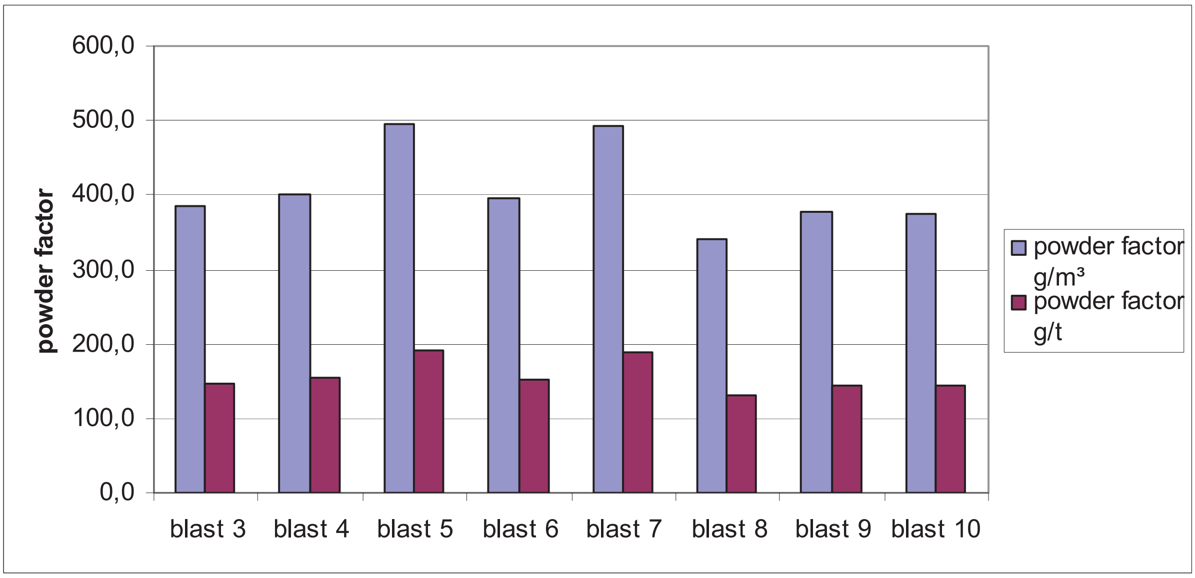


Figure 26: Average powder factors for blast 3 to blast 10

Besides blast five and blast seven, powder factors range around 150 g/t, calculated with an average rock density of 2.6 g/cm³. On closer inspection of the powder factors per hole, one might find that the variation between holes is quite big, which is a consequence of the variation of burdens and spacings. This is shown is Figure 27 and 28.

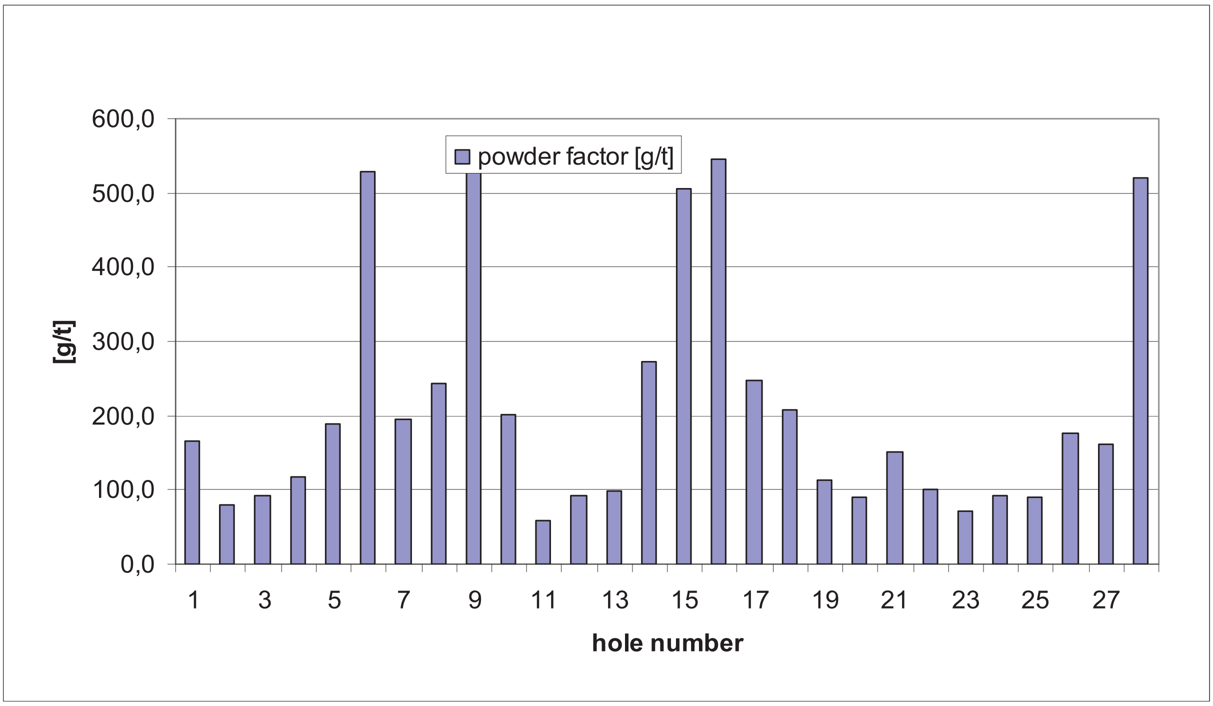


Figure 27: Changing powder factors per hole for blast 9

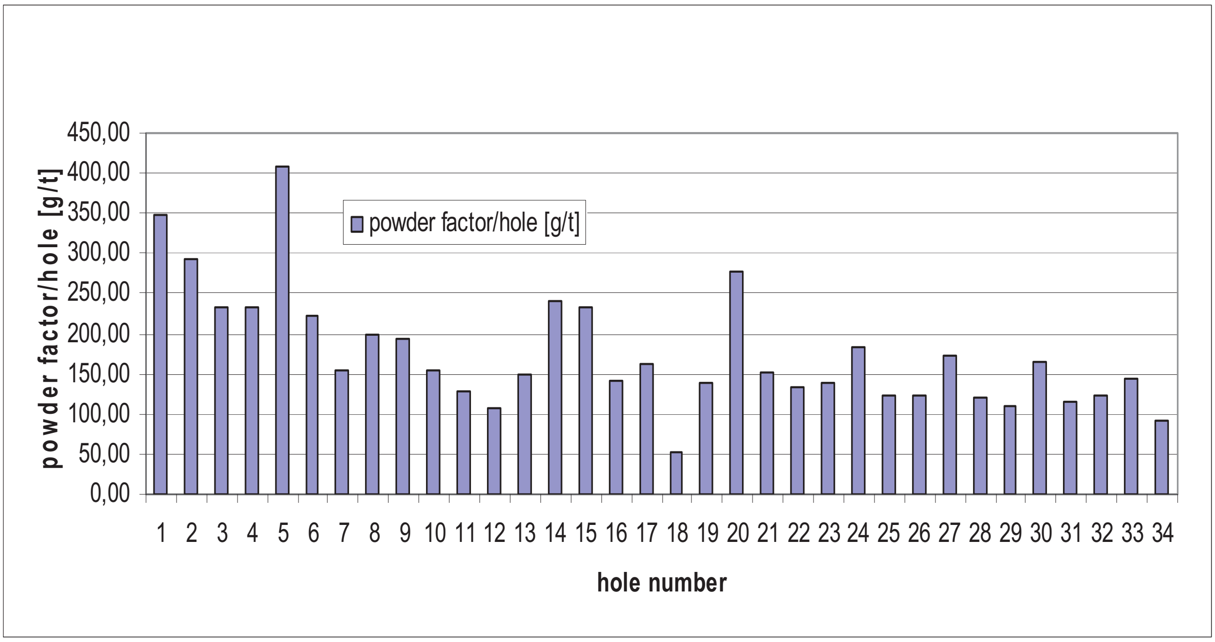


Figure 28: Changing powder factors per hole for blast 10

5.2. POSITIONING OF THE DRILL RIG

As was stated before the orientation of the drill boom is done with the inclinometers mounted on the drill rig. The azimuth of the drillholes is visually estimated. Therefore the drill operator tries if possible to position the drill rig parallel to the wall. This may have the effect that not all holes have the same direction of dipping. A slight turn of the drillrig for only a few degrees may therefore result in big deviations of the end position of the hole. This effect is displayed in Figure 29.

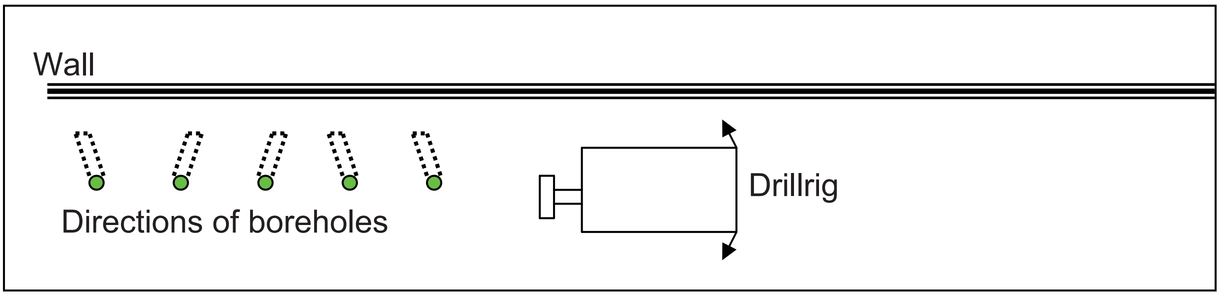


Figure 29: Borehole direction resulting from slight turns of drillrig

5.3. WALL SURVEYING WITH 3G SYSTEM

The wall surveying with the 3G system shows beside the problems of using the system which were already discussed in chapter 4.2, that the walls are quite different. A summary of derived data from the 3G-system is shown in Table 9.

Table 9: Summary of derived data from 3G-system

	Calculated dipping	Dipping	Lenght of holes [m]	
Blast number	(includes material in front of the wall)	(without material in front of the wall)	calculated with 3G software	average drilled length
4	45,92°	70°- 75°	8,90	10
5	23,16°	75°	10,9-11,7	12
7 (1)*	54,36°	60°- 65°	8,7-10	12,2
7 (2)*	41,13°	45°	9,9-10,2	12,2
9	26,46°	30°	10,5-8,6	11,7
10	60,32°	65°- 70°	9,2-10	11,2

*…Wall was surveyed from two sides

The material that lies on the toe of the wall results in very big burdens and therefore a high confinement of the hole in the bottom. That's why the calculated dipping is always very flat and it would be much steeper without the material. Nevertheless even the dipping without the material is not always the same as the borehole dipping, just because most times the digger defines the wall angle and not the inclination of the blast holes from the previous blast. In blast 7 (2) and blast 9 the dipping without the material could not be derived because the material was lying from the toe up to the edge of the wall.

Another aspect is that the boreholes are drilled too long. This is shown in Figure 30.

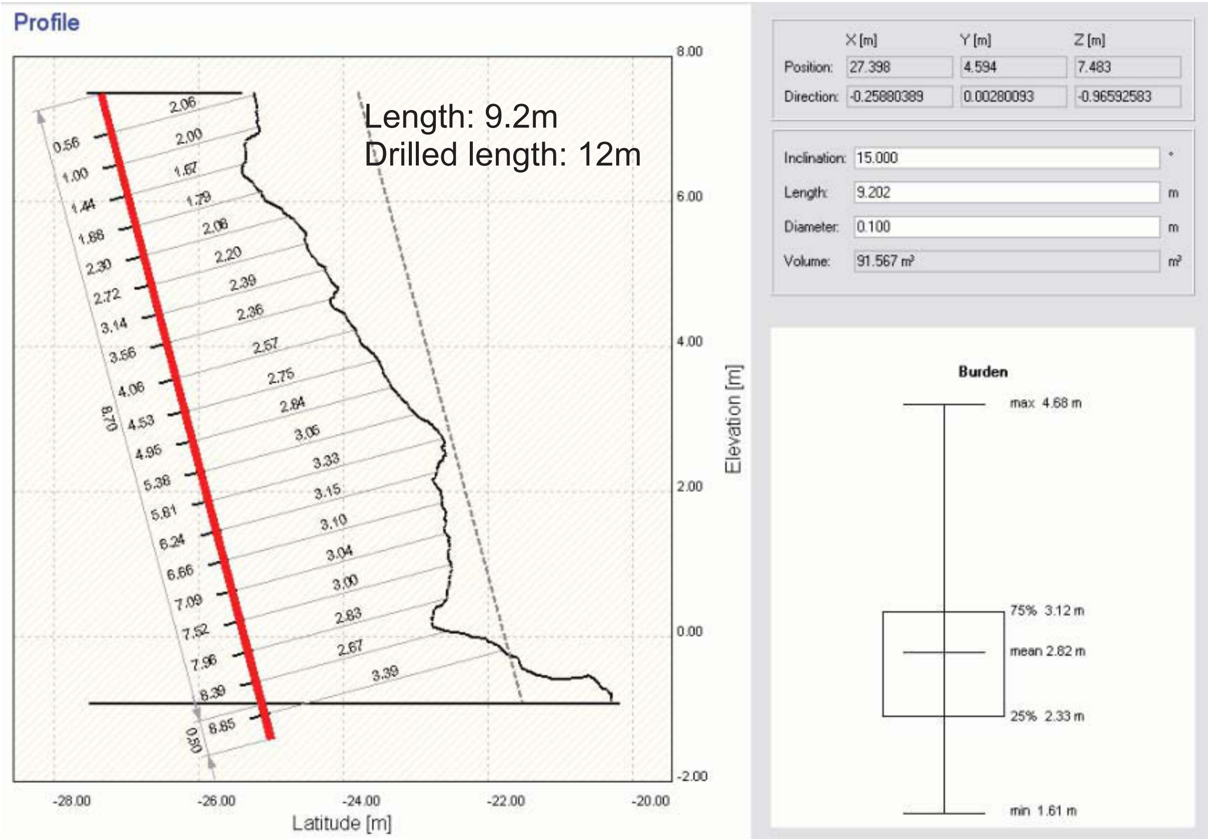


Figure 30: Example of a too long drilled borehole from the 10th blast

The 3G-software that comes with the 3G-system is also able to calculate volumes and tonnes that have to be blasted by each borehole. Unfortunately this information can not be used here because of the irregular blast patterns and material that is lying on the toe of the wall. Until now one is not able to put blastholes on free chosen positions. Moreover the blastholes can only be lined up in a row between the two marks that define the limits of the picture and further more only complete rows (it is not possible to delete individual boreholes) can be realised. As can be seen from the surveyed boreholes these ideal situations never existed.

5.4. QUALITY OF THE BOREHOLE

Although the values from the first seven deviation measurements can not be used because the compass of the rod was calibrated wrong, it can be said that the quality of the boreholes was really bad. This is underlined by the calculated values from the experiment with the torch that was lowered into the boreholes. In the 6th blast the light of the torch vanished in 72 % of the holes before the depth of 7 meters and none of them could be seen till the bottom.

It was already stated that this effect suddenly changed after the 6th blast. In the 7th blast already 23 % of the holes could be seen till the bottom and none vanished before a depth of 7 meters.

The deviation measurement with the Boretrack system shows that the deviations of the drilled holes are quite big. A summary of the measured deviations is shown is Table 10.

It should be noted that only 30 % of the holes are less than 0,5 m away from their planned position. More than 43 % deviate between 0,5 m and 1 m and even 27 % deviate between 1 m and 2,5 m.

Table 10: Summary of measured borehole deviations

Deviation	Percentage
< 0,5 m	30,20%
0,5 - 1 m	43,40%
1 - 1,5 m	13,20%
1,5 - 2 m	7,50%
2 - 2,5 m	5,70%

5.5. Analysis of Vibration Measurements

The measured ground vibrations are at a quite acceptable level. Even the predictions for ground vibrations match very well with the real measured values.

To calculate these predicted ground vibrations the prediction formula of Lüdeling/Hinzen (1986) for sedimentary rock (which is also used by the company) was used:

$$v_{max} = 969 * L^{0.6} * D^{-1.5}$$

where v_{max} = the maximum oscillation vibration (mm/s),

L = maximum explosive charge fired instantaneously (kg) and

D = distance from the blast (m)

The values for "the maximum explosive charge fired instantaneously" were taken from the borehole charging documentation. The measured and predicted values for all blasts are displayed in Figure 31.

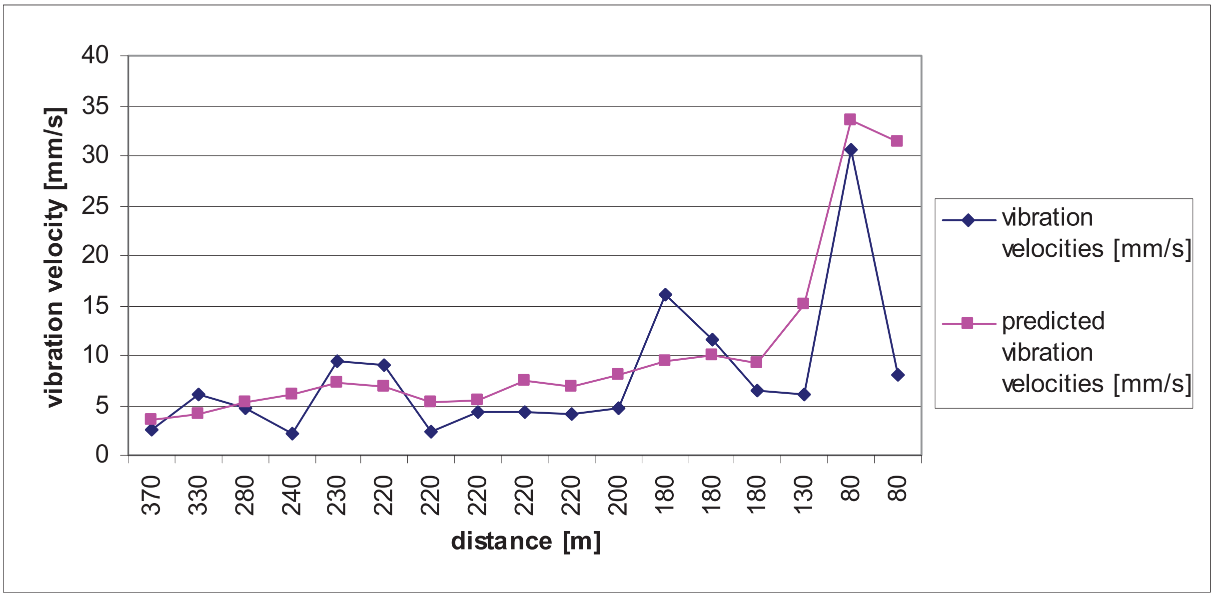


Figure 31: Measured and predicted vibration velocities

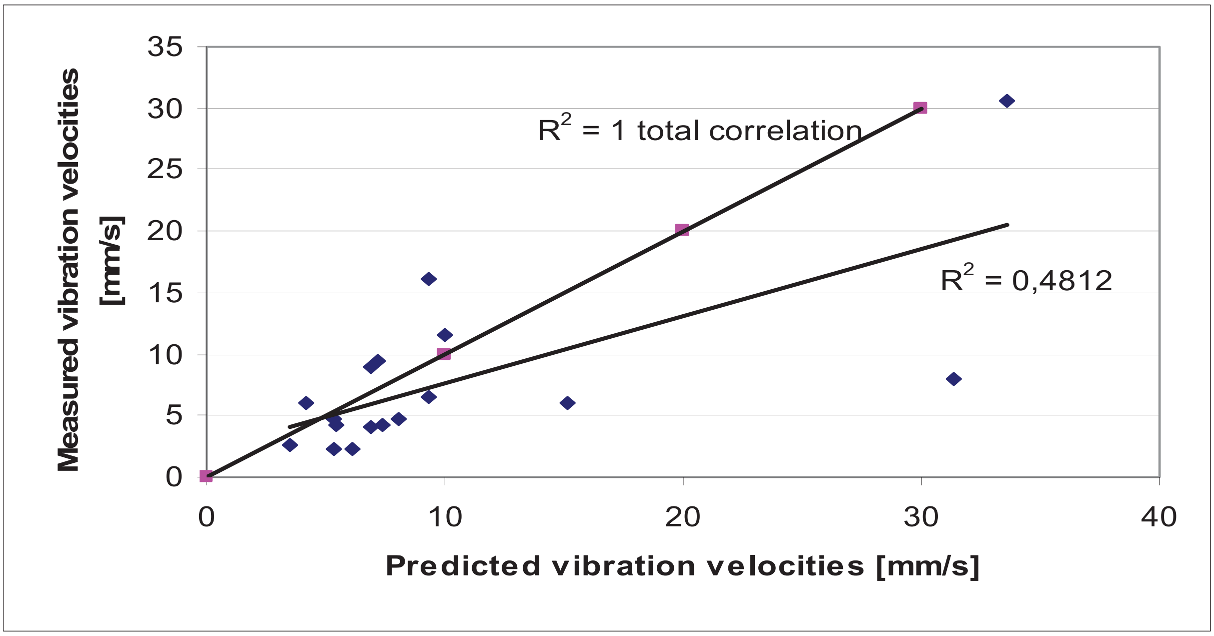


Figure 32: Correlation of predicted and measured vibration velocities

As can be seen the measured and predicted values correlate quite well, the differences can be explained with irregularities of the rock mass that can not be foreseen.

To be able to predict the vibrations better a “log maximum velocities/log scaled distance” diagram was prepared (Figure 33).

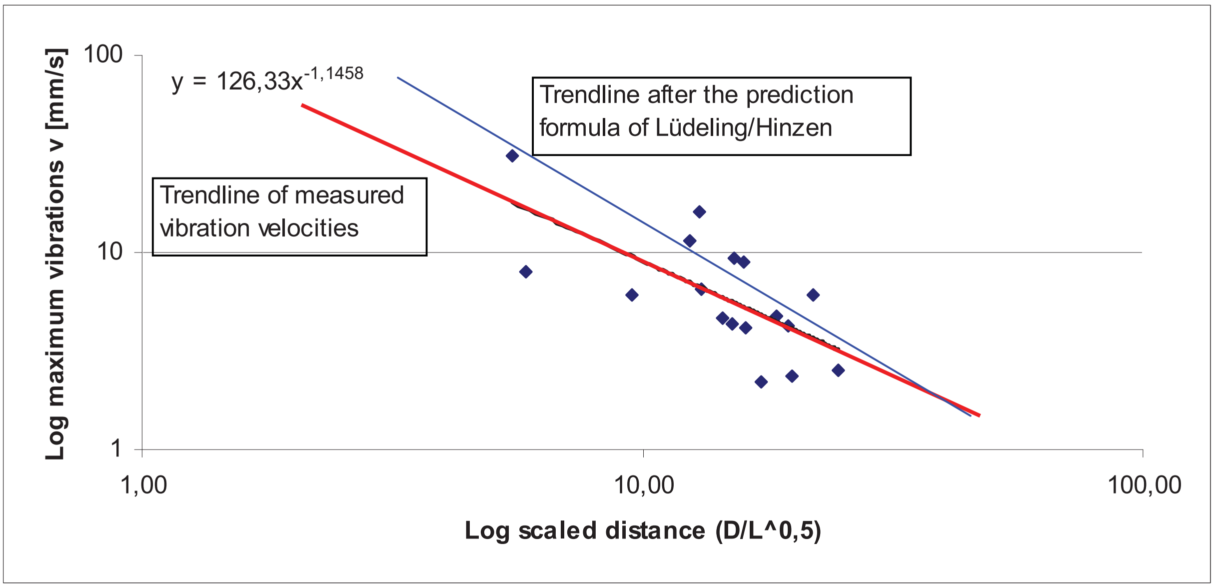


Figure 33: Log maximum vibrations versus log scaled distance diagram

The red line shows the trendline of measured data points, the blue line shows the trendline after the prediction formula of Lüdeling/Hinzen. This diagram is a good instrument to predict vibrations if it is fed with enough data points, because it reflects the unique geology of the mine, that no prediction formula can capture.

5.6. Analysis of Noise Measurements

Beside the values of blast 5 and 7 which were taken inside the mine just 150 m away from the blast where detonating cords were hanging out of boulders, all values stay constant below 100 dB(A), which is an acceptable magnitude for noise from a blast. The threshold of noise pain is about 120 dB(A).

6. DEVIATIONS BETWEEN PLANNED AND MONITORED DATA

This part of the thesis shows a comparison between documented data by the company and actually monitored data. The documented data from the company was taken from the drill operator's records and the shot-firer's records as well as from the vibration velocities event report.

6.1. BOREHOLE SPECIFICATIONS

Table 11 shows a comparison of measured, calculated and from the drill operator recorded borehole specifications.

Table 11: Comparison of measured, calculated and drill operators records of borehole specifications

	Borehole inclination	Length of holes				Drill operators record	
Blast number	[degrees]	minimum drilled length [m]	maximum drilled length [m]	average drilled length [m]	calculated with 3G software [m]	minimum drilled length [m]	maximum drilled length [m]
blast 1	80°	5	11	9,1	-	5	10
blast 2	80°	9,5	12,8	11,8	-	3	12
blast 3	80°	8	14	11,96	-	8	12
blast 4	80°	9,2	10,5	10	8,90	10,5	10,5
blast 5	80°	9,9	13,5	12	10,9-11,7	12	12
blast 6	75°	10,2	13,7	11,7	-	11	12
blast 7	75°	11,3	12,8	12,2	7,7-10,2	12	12
blast 8	75°	6,1	14	10,3	-	6	12
blast 9	75°	8,2	12,3	11,7	8,6-10,5	12	12
blast 10	75°	6,1	12,8	11,2	9,2-10	no records available	no records available

Values between measured, from the 3G-software calculated and by the drill operator recorded borehole lengths vary a lot. The reasons for shorter measured than actually drilled holes may be that material fell into the hole after the drilling process. Longer holes measured than recorded may have the reason that the drill operator wanted to make sure that the hole is long enough , so he drilled a little bit more. Nevertheless at this point it should be stated again that it is very important to know the desired borehole length, because in all blasts where the 3G-software was used the holes were drilled too long.

6.2. BLASTING PATTERN

The differences between recorded and actually drilled burdens and spacings are shown in Table 12 and Figure 34 and 35.

Table 12: Comparison of recorded and measured burdens and spacing

	planned value (company)		monitored values	
blast number	burden	spacing	burden	spacing
blast 1	3,5	3,7	not measured	not measured
blast 2	3,5	4	not measured	not measured
blast 3	3,5	3,8	3,28	3,55
blast 4	3,5	3,8	3,19	3,35
blast 5	3,5	3,8	2,73	3,57
blast 6	3,5	3,8	3,44	3,40
blast 7	3,2	3,4	3,39	2,55
blast 8	3,4	3,8	3,60	3,24
blast 9	3,4	3,6	4,94	3,15
blast 10	not measured	not measured	3,69	3,04

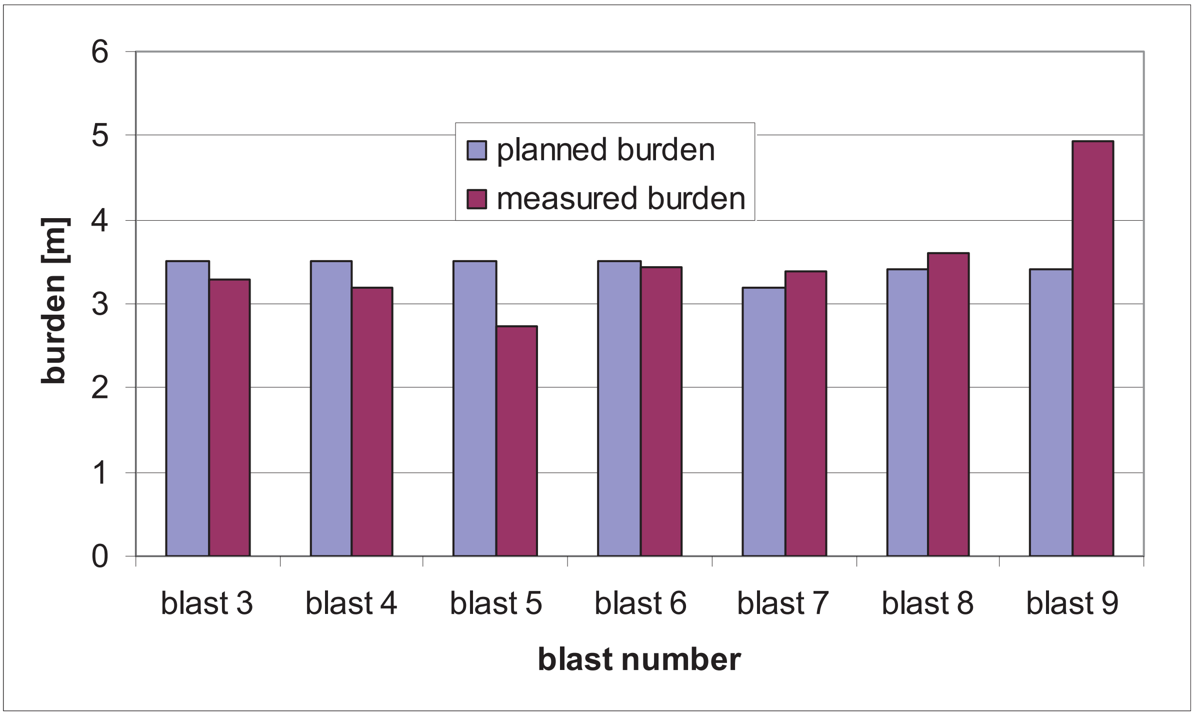


Figure 34: Comparison of recorded and measured burdens

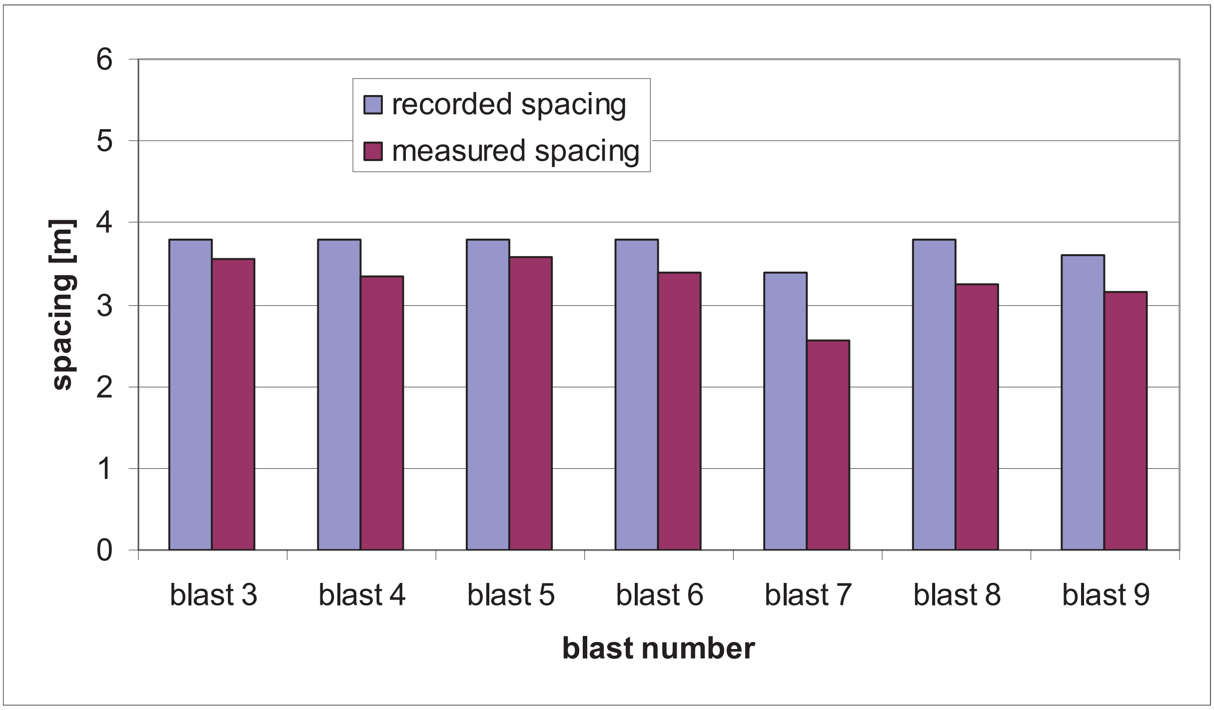


Figure 35: Comparison of recorded and measured spacings

Beside the burdens in blast 7, 8 and 9 the recorded values are higher than the actually drilled values. This effect can also be seen in the blasted area that belongs to each borehole, which is calculated by multiplying the burden and the spacing for every blast (Table 13 and Figure 36).

Table 13: Comparison of recorded and monitored average blasted area per hole

	planned value (company)		monitored values	
blast number	burden	spacing	burden	spacing
blast 1	3,5	3,7	not measured	not measured
blast 2	3,5	4	not measured	not measured
blast 3	3,5	3,8	3,28	3,55
blast 4	3,5	3,8	3,19	3,35
blast 5	3,5	3,8	2,73	3,57
blast 6	3,5	3,8	3,44	3,40
blast 7	3,2	3,4	3,39	2,55
blast 8	3,4	3,8	3,60	3,24
blast 9	3,4	3,6	4,94	3,15
blast 10	not measured	not measured	3,69	3,04

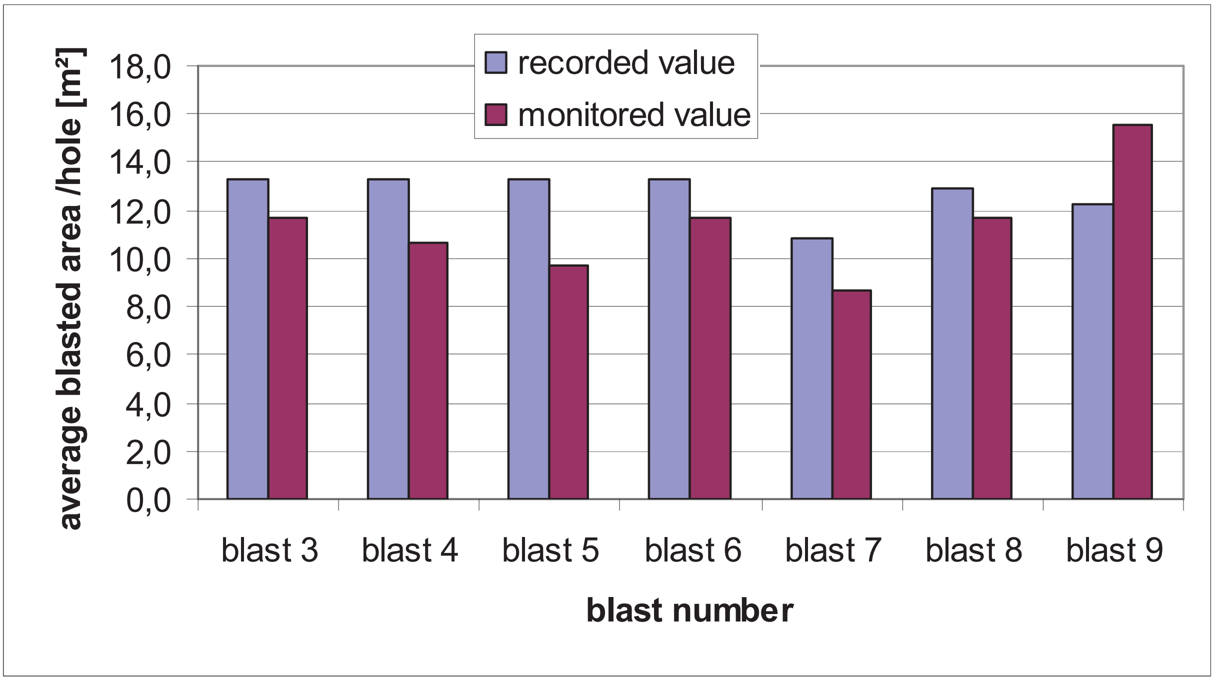


Figure 36: Comparison of recorded and monitored average blasted area per hole

Only in blast 9 the monitored value is actually bigger than the recorded value. In all other blasts the difference between the recorded and actually monitored area lies between 1,3 m² (blast 8) and 3,6 m² (blast 5) per hole.

These values can be explained by many factors:

- Because of irregularities of the wall, additional holes have to be drilled in front of the first row, to make sure the material gets blasted, which reduces burden for this special area;
- If suddenly the driller reaches a talc-zone, these holes don't get charged in order to avoid mixing talc and waste material, which of course increases the spacing for the neighbouring holes;
- In five of the ten documented blasts, the holes were drilled parallel to the wall, whereas the shot-firer turned the direction of blast with the delay time of the detonators by 45°, which of course changes the actual effective burdens and spacings. Figure 37 and 38 show this effect for better understanding.

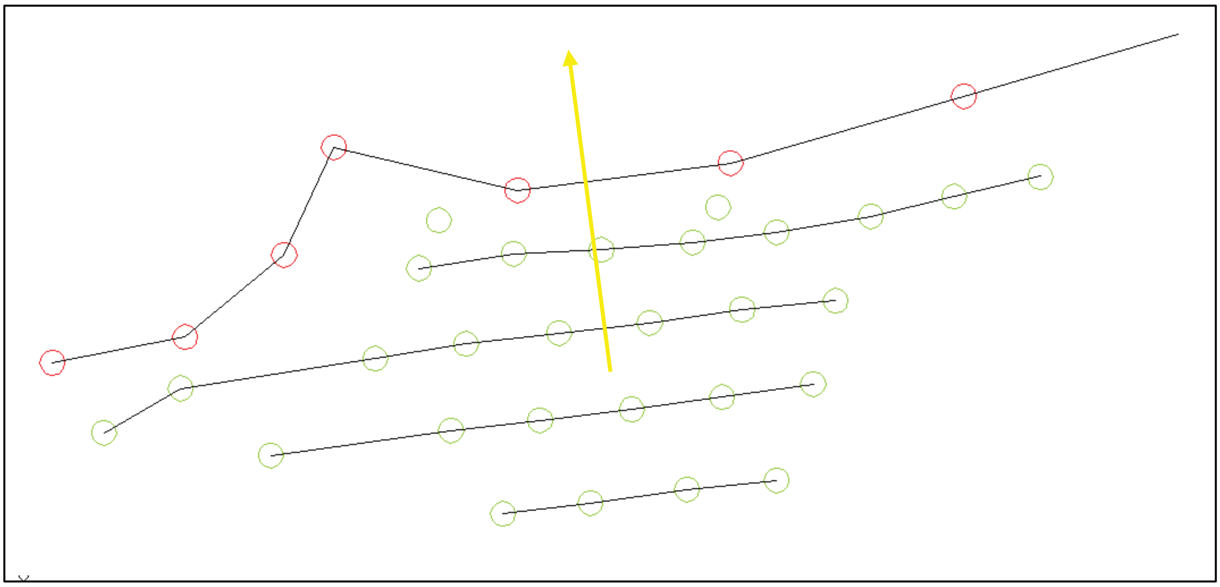


Figure 37: Intended direction where muck should be thrown

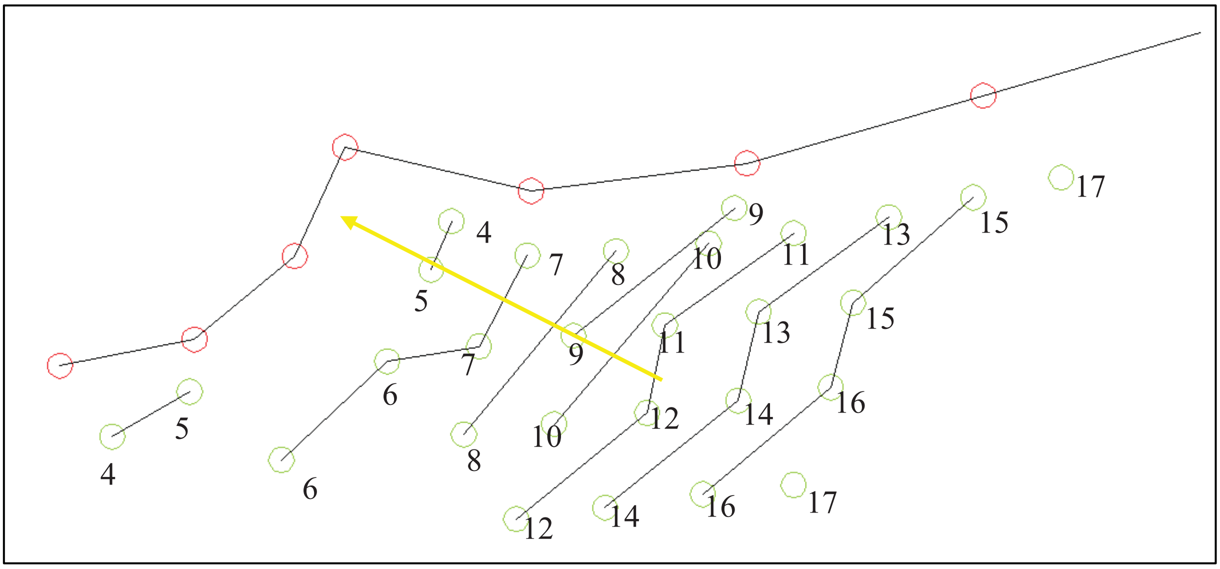


Figure 38: Direction where the muck was actually thrown by changing the ignition sequence (delay numbers shown)

An effect that results from this change of ignition sequence is that the powder factor and the energy per borehole sometimes differs significantly, as the side spacings get very big and the burdens very small (Figure 39 & 40).

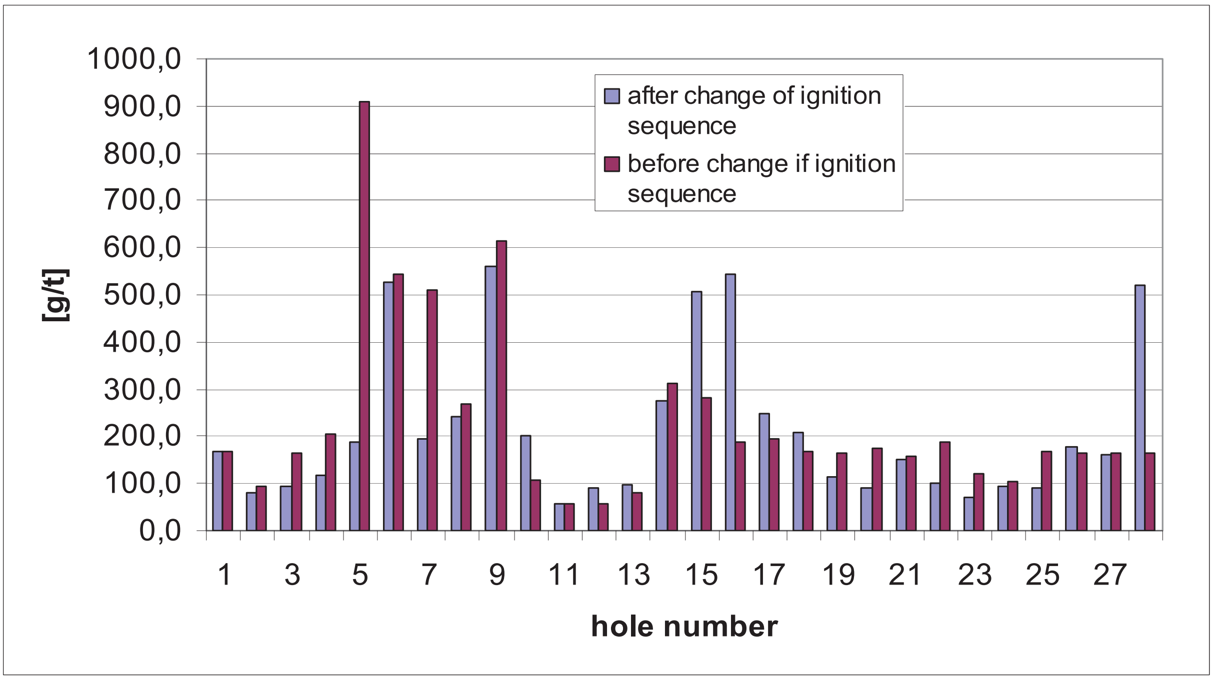


Figure 39: Different powder factors before and after the change of ignition sequence for blast 9

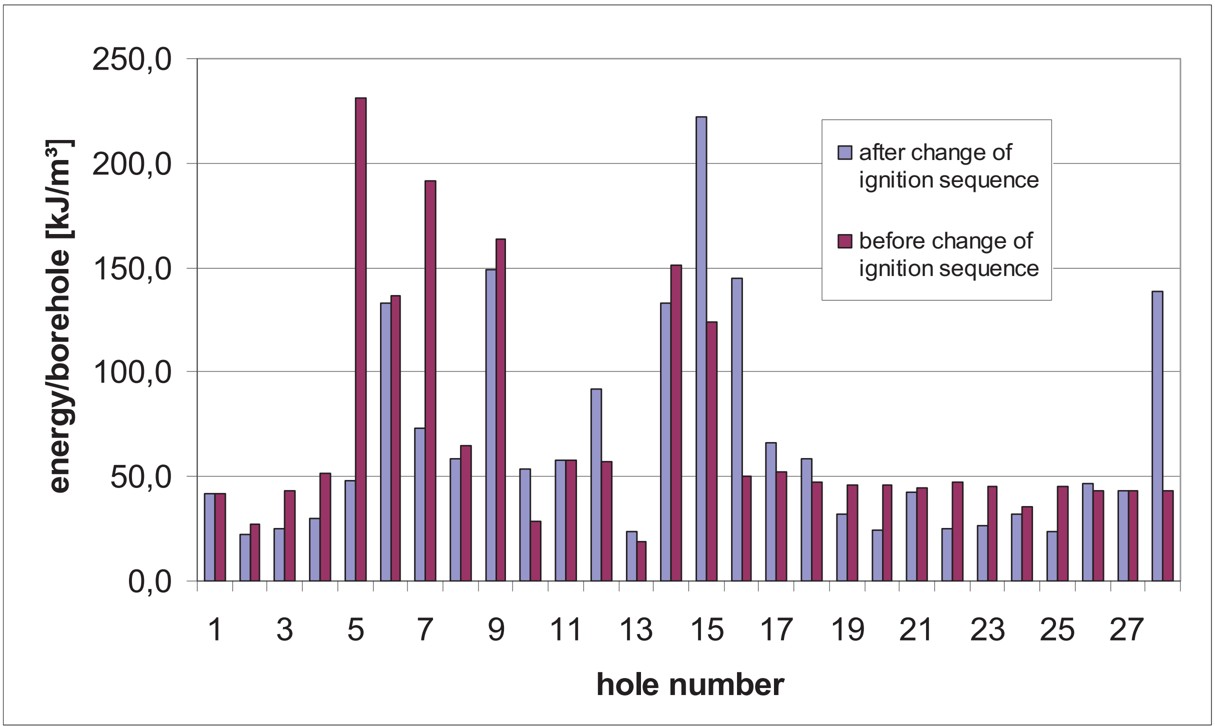


Figure 40: Different energy/borehole before and after the change of ignition sequence for blast 9

Nevertheless the change of ignition sequence had probably no effect on the blast result, because the explosive anyway was not distributed more regularly on the holes before the change of ignition sequence. If the blasting pattern was drilled more precise, this change could have influenced the blast strongly, even in disadvantageous way, by producing a lot of fines on one hand and a lot of boulders on the other hand.

6.3. Powder Factor

The analysis of the shot-firer records concerning the powder factor showed that he calculated the powder factor with an average rock density of 2,5 g/cm³. Therefore in the following Table 14 and Figure 41 the planned powder factor [g/t] is also calculated with 2,6 g/t like in all other calculations of the thesis.

Table 14: Comparison from the shot-firer's and the monitored powder factor

	planned value	monitored value
blast number	[g/t]	[g/t]
blast 3	not calculated	147,7
blast 4	127,9	154,6
blast 5	138,5	190,4
blast 6	not calculated	152,1
blast 7	147,1	189,6
blast 8	130,8	131,4
blast 9	not calculated	144,7
blast 10	not calculated	144,2

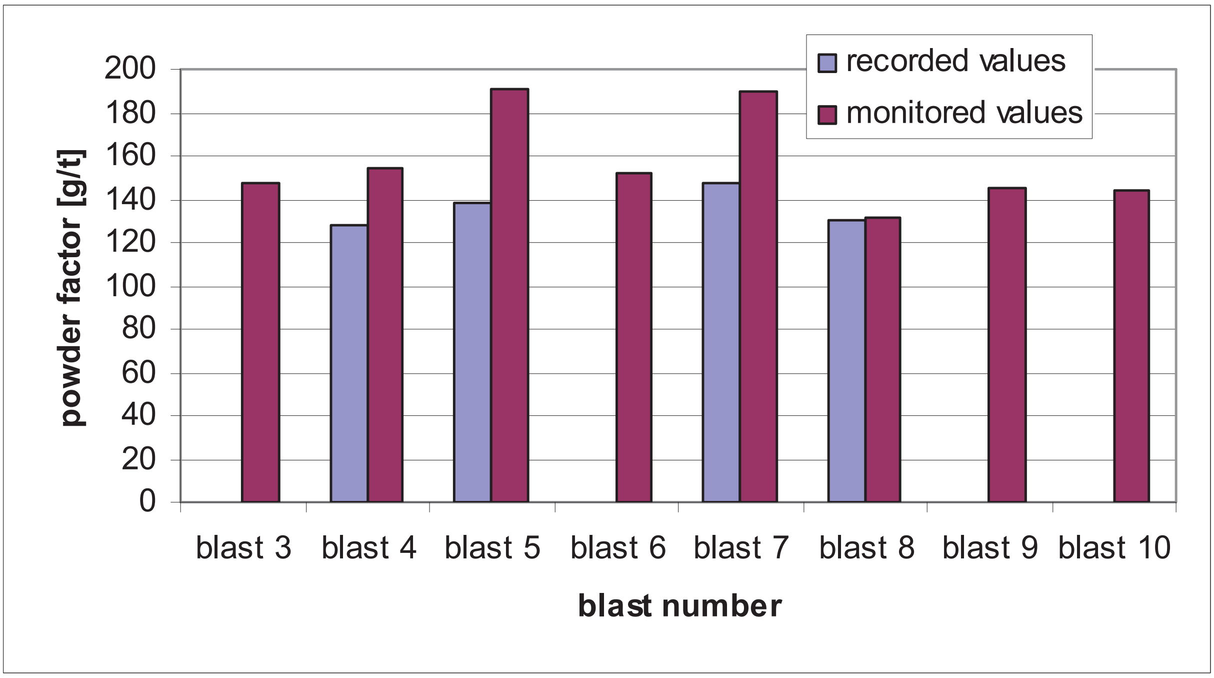


Figure 41: Comparison of recorded and planned powder factors

For blast 3, blast 6, blast 9 and blast 10 no records are available.

Only 2 values of the monitored powder factors are quite high, ranging around 190 g/t, all others are around 150 g/t which is the desired value from the company. The shot-firer's record show a lower powder factor, probably because he calculated with the documented burdens and spacings from the drill operator, which, as was already seen, do not always correspond with reality.

6.4. Mass of Explosives

It was also tried to compare the recorded column design as the shot-firer sketched it on his drawing of the blast to the actual monitored one. This was done for blast 8 and 9. In blast 8 the shot-firer's record says the holes were charged with 4 cartridges of gelatine explosives, 3,7 stemming and the rest anfo. The amount of explosive that would fit into such a hole is compared to the actual charging in Figure 42.

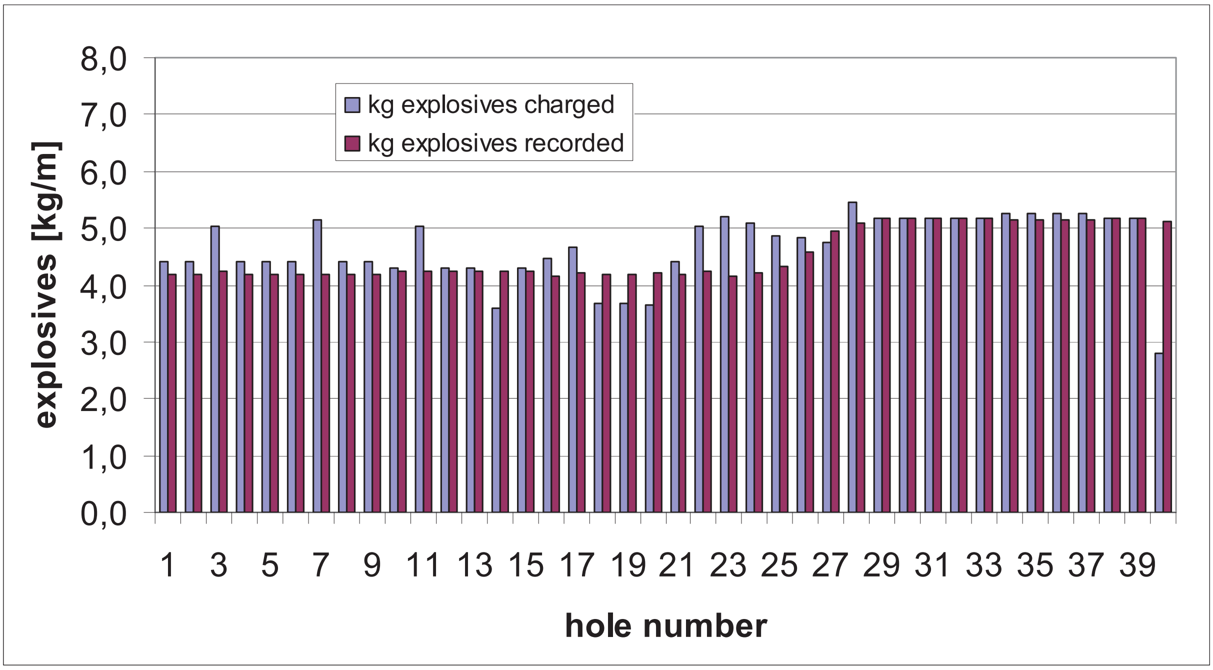


Figure 42: Comparison of mass explosive recorded and charged for blast 8

The same was done for blast 9, where the shot-firer's record says the holes were charged with 4 cartridges emulsion explosives, 3,6 m stemming and the rest with anfo. In Figure 43 the

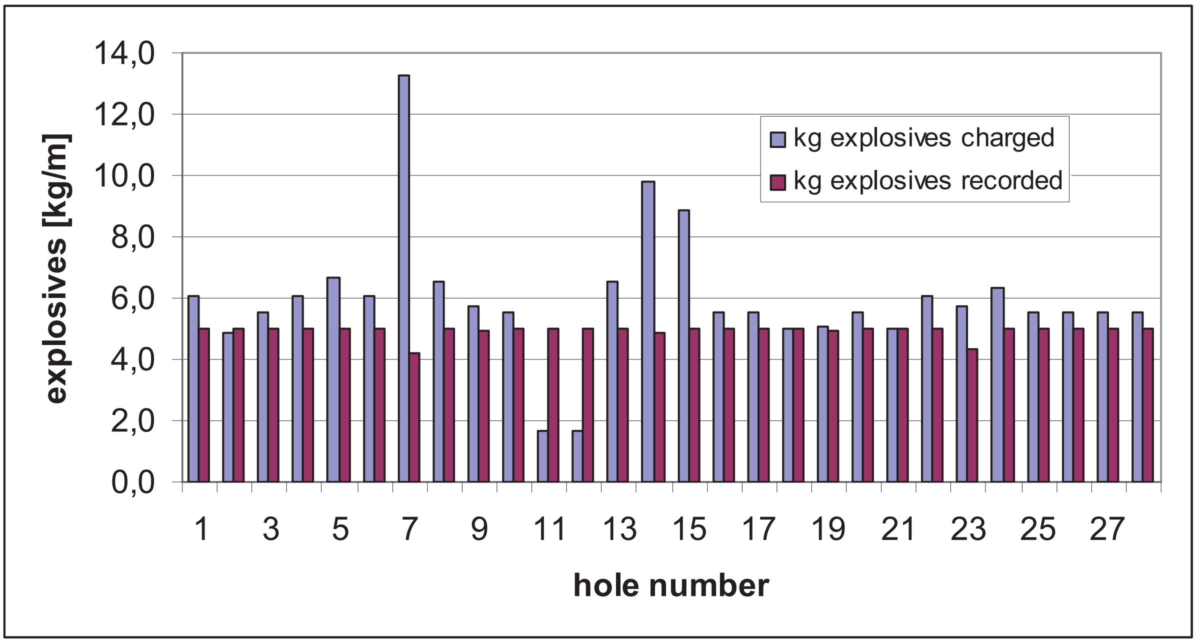


Figure 43: Comparison of mass explosive recorded and charged for blast 9

amount of explosive that would fit in such a hole is again compared to the actual charging. To be able to compare this two situations better special holes in blast 9 that were very short, were taken out.

Most times the mass of explosives charged is higher than the mass of explosives recorded by the shot-firer. One reason for that is that the amount of explosive per hole is regulated from the shot-firer's experience. The sketch of the charge the shot-firer is drawing is only a very imprecise one, that's why his values of mass of explosives per meter do not vary that much. His sketch just shows the charge of one hole to represent between 30 and 40 holes.

6.5. VIBRATIONS

Table 15 and Figure 44 show a comparison of the company's precalculated vibration values, my own precalculated values and the actual measured ones.

Table 15: Comparison of vibration values

	precalculated values (company)	precalculated values	measured values
blast number	[mm/s]	[mm/s]	[mm/s]
blast 1	4,35*	1,73*	<1*
blast 2	4,10	6,09	2,22
blast 3	4,90	6,93	8,94
blast 4	2,30	5,32	2,35
blast 5	3,74	5,44	4,23
blast 6	4,25	7,37	4,41
blast 7	7,00	7,24	2,1
blast 8	6,35	3,52	2,54
blast 9	3,77	6,86	4,11
blast 10	not available	6,36	-

* values for different measuring locations, therefore no comparison possible

** values from company's measuring device

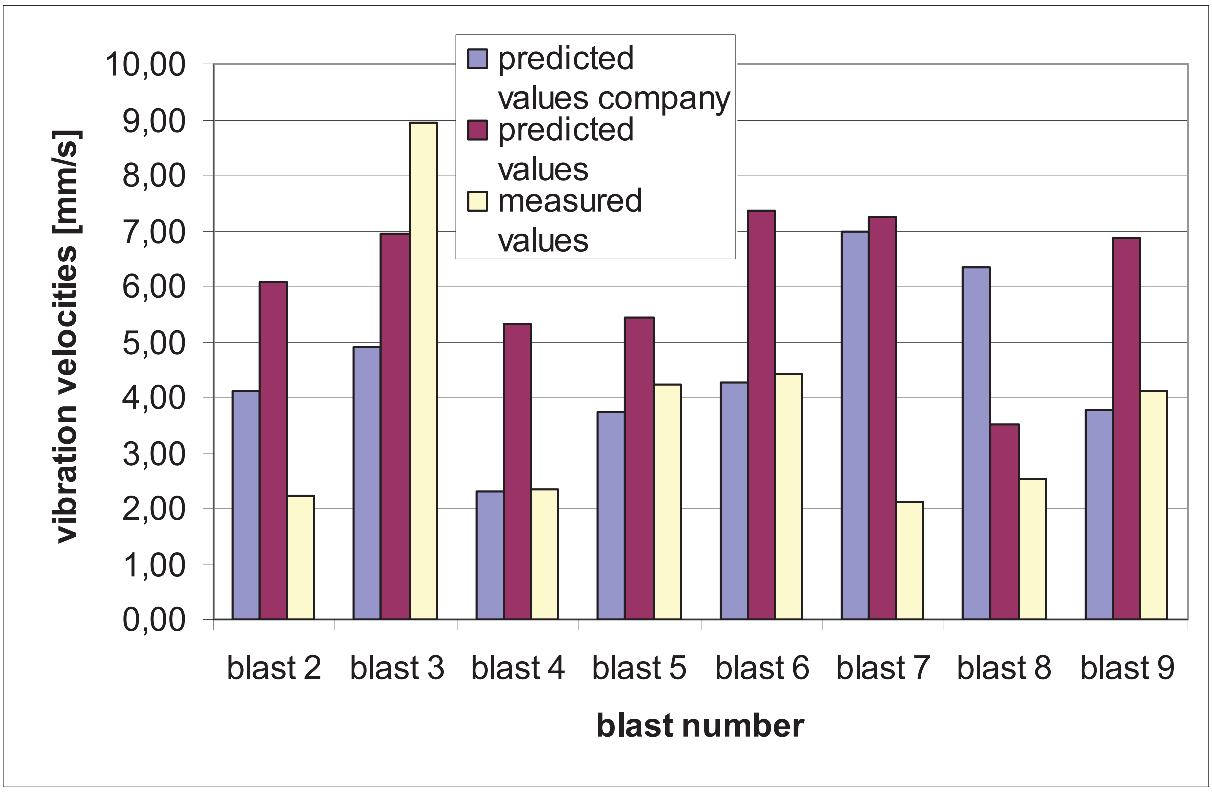


Figure 44: Comparison of vibration values

Except blast 8 the company's precalculated vibration values are always lower than my values. This is because the company precalculates the vibration values with an average amount of explosives per delay time, whereas I precalculated the values for the maximum instantaneous charge per delay, because I knew from my documentation exactly the amount of explosive for every hole, whereas the shot-firer just divides the total amount of explosive by the number of holes blasted that got the same delay.

Nevertheless all my precalculated values except one are higher than the actual measured ones, which is an advantage, because it always creates a safety pillow.

7. The State of the Art in Drilling and Blasting

7.1. Planning a Blast

The modern way to design a blast is a computerized blast design. But it should not be forgotten, that for the state of the art of planning a blast much more than a computer-software is needed to do the work properly. Quite a number of aspects like geology, burden and spacing, used explosive, initiation-system, initiation sequence and many more have to be taken care of. Therefore it is advisable to build up an own management system for the process “drilling and blasting”. Here all experiences and collected data can be stored and new findings can be added. Especially in the drilling and blasting design, which is normally done by young engineers, who’s area of responsibility change often and who might do the job for just a few years, a lot of information is lost through the job fluctuation.

7.1.1. Review on the planning process of the blast

A blast design should be done by a person with sufficient engineering knowledge about blasting. Surely the driller who by now decides where each blast hole is drilled, has collected a lot of knowledge through his working years, but won’t have the same overview as a well trained engineer. No two blasts are the same, so accordingly no two blasts can be drilled in the same way.

It should be also said that someone who designs the blast with a database in his background has access to much more helpful information that can influence the design and where to place the holes. After the planning process the engineer can review the design, maybe get a second opinion on his first plan and then redo the whole process again. The driller who drills his hole will probably not think about the position of the hole again after it is drilled, but will proceed to drill the next hole.

7.2. Surveying of the Wall

Nowadays two systems of surveying the wall are available. The first one is the laser scanner, who creates a very detailed picture of the wall, but has some disadvantages. The investment costs are very high, the scanning of the wall is very time consuming and it creates an enormous amount of data that has to be stored somewhere.

Secondly there is the photogrammetry, like the system from the 3G-company. It is very flexible, fast to use and sufficiently accurate and has the big advantage, compared to the laser scanner, that it shows colours, which is a really helpful instrument for interpreting possible risky parts in the wall.

7.2.1. Review on the Actual Process of Surveying the Wall

The already available 3G-system was still in the introduction phase of the operating procedure at the quarry when the data gathering was conducted and therefore not used as much as possible. A more intensive use would have shown that

- Firstly: at some walls where the toe of the wall was not cleared off, the burden of the lower parts of the hole show an enormous burden of more than eight meters (Figure 12) and
- Secondly: In many cases the drilled length is much higher than actually necessary. An example for this excessive length drilling is shown in Figure 29 where the drilled length is 12 m whereas only 9.2 m are necessary.

The following two calculations in Table 16 show the possible drill length savings for one month:

Table 16: Two examples for a possible drill length saving

Case 1:	holes are on average 2 m too long		
	number of holes		holes are on average 10 m long
Blast 1		35	
Blast 2		45	
Blast 3		31	
Blast 4		45	
Blast 5		37	
Blast 6		44	
Blast 7		47	
Blast 8		64	
Blast 9		28	
Blast 10		44	possible drill meter savings for one month
Sum		420	840
=> one could have drilled another 84 holes more			
Case 2:	holes are on average 1.5 m too long		possible drill meter savings for one month
			630
=> one could have drilled another 63 holes more			

Table 16 shows that in case 1 another 84 hole could have been drilled, which is about the amount of holes for another two blasts. This means a saving of about 20 % in drilling lengths and costs.

7.3. DRILLING

From a technical point of view there are 2 systems available: Tophammer drilling machines and Down the Hole Hammer drilling machines. Which one to use is most times a matter of the length of the drill hole, which is shown in Figure 45.

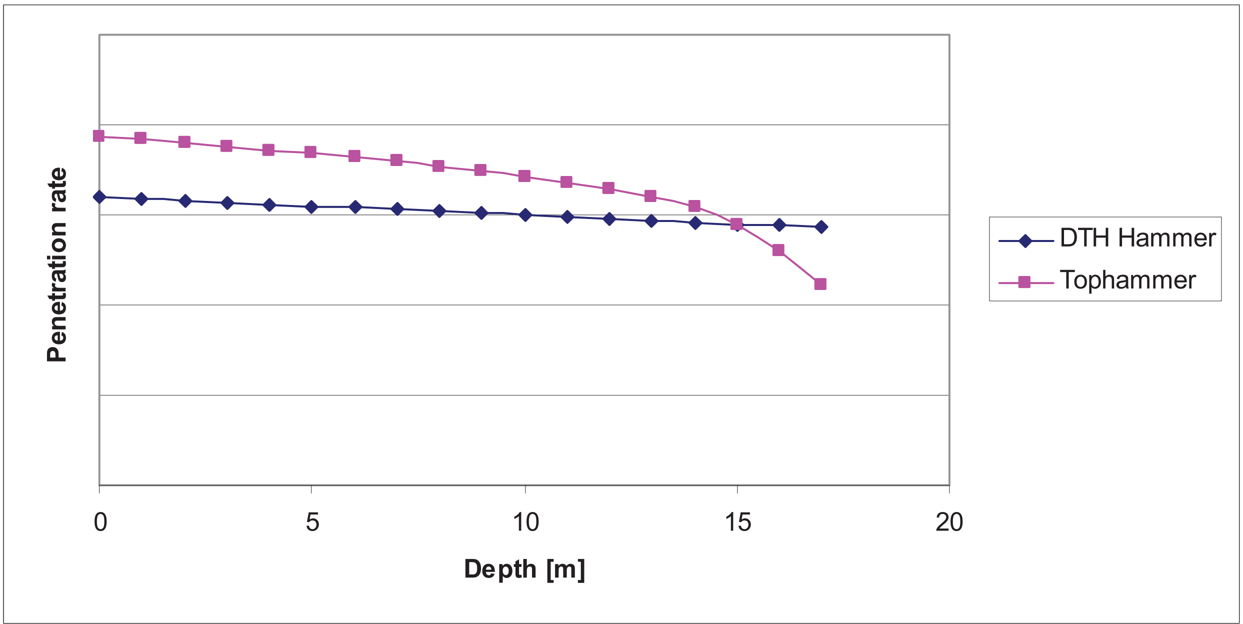


Figure 45: Difference between DTH-hammer and tophammer in relation to the penetration rate

As can be seen tophammer machines are mostly used up to drilling lengths of 15 to 20 meters. For longer holes it is advisable to use Down the Hole (DTH) hammer machines, because of their higher penetration rate. Another aspect distinguishes the two systems. Because of the physical size of the DTH Hammer, hole diameters start here at about 90 mm and go up to 200 mm, whereas Tophammers start at 35 mm and end at 127 mm.

7.3.1. SMART RIG SYSTEM

The newest achievement in automation of surface crawlers is the SmartRig System, developed by Atlas Copco (Figure 47). It is characterised by the following new specifications:

- SmartRig control system
- The AutoRAS (automatic rod adding systems)
- ROC Manager and MWD (Measure While Drilling)
- GPS Navigation
- Silenced Version for urban areas

The SmartRig control system uses electrical signals to control the hydraulic valves. This reduces the number of hydraulic components by 30 % compared to normal Hydraulic Control Systems (HCS) and the noise level for the operator. All control gauges are replaced by a display unit which results in more space in the cab and increases visibility for the operator. This Control System includes an anti jamming function which reduces wear of material and prolongs service intervals. Further more it comes with a laser plane as a reference height to make sure all hole are drilled to the same depth and an automatic feed positioning to set up the boom to predefines angles with just one button.

The AutoRAS drills all holes to predefined depths and allows the operator to take care of other duties as maintenance checks during the drilling process.

The ROC Manager is a PC software to plan drill patterns which then can be transferred to the drillrig using a data card.
The Measure While Drilling System automatically logs parameters, like hole depth, penetration rate, feed, percussion and rotary pressure. Together with the ROC manager these data can be used for analysis of rock properties and can be even displayed graphically in slices through the bench.

Using the GPS navigation there is no need to mark the holes manually anymore. The operator just drives the drillrig to predefined coordinates, which come from the planed blast pattern of the ROC manager. Together with the hole navigation system the GPS makes it possible to drill all holes to the same direction compared to conventional systems where the drillrig is positioned with a landmark in the distance (see Figure 46).

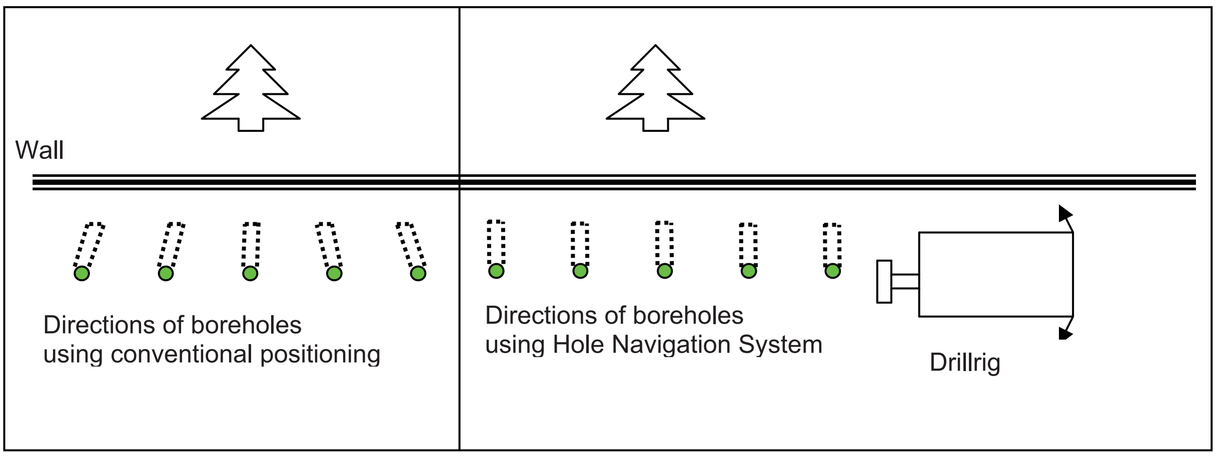


Figure 46: Comparison of conventional positioning and HNS

There is also a silenced version especially for urban areas available. Therefore the boom is enclosed with a frame of lightweight aluminium, which reduces noise levels of the drillrig by 10 dB(A).

Figure 47: Atlas Copco Roc F9 SmartRig Surface Crawler, Source [7]

In the future the system will be even able to drill without an operator and drive from hole to hole automatically.

7.3.2. Review on the Actual Drilling

Referring to the actual bench height of ten meters it should be possible to drill much more accurate with the available surface crawler. Hole deviations therefore should not be bigger than 20 cm. A possibility is to install a guiding rod as the first rod to minimise deviations.

7.4. MEASURING BOREHOLE DEVIATIONS

The only reliable way to make a statement about the quality of the drilled boreholes is a deviation measurement with a system like the Boretrack-System.

There is always a safety risk present if you don't know where exactly your boreholes are drilled. Especially if the geology you have do drill your hole in is as irregular and the rock as jointed as it is in this mine. The possibility of a hole that is drilled with insufficient burden is therefore always a flyrock hazard.

7.5. DETONATORS

Nowadays the trend goes definitely into the direction of electronic detonators, like the ikon-system of Orica for example (Figure 48).

Each detonator contains a programmable computer chip. After attaching the detonator to a wire, the logger that is also connected to the wire, recognises the detonator, tests its functionality and assigns it with a delay time. Up to this time there is no way that the detonator can initiate, because the logger doesn't have enough energy to do so. To initiate the detonators the blaster is needed, which is connected to the logger and who needs a separate key.

Figure 48: The ikon-system of Orica, showing the electronic detonator, the blaster, the logger and the planning software, Source [10]

7.5.1. *REVIEW ON THE ELECTRIC DETONATORS*

The now used electric detonators are not comparable to electronic detonators referring to their accuracy of delays and number of possible different delays. Especially if you have to take special care of vibrations because of neighbouring houses you should not use one delay time for two different holes if at the same time you don't want to reduce the number of holes per blast.

7.6. WAY OF INITIATION

The blast should be initiated from the borehole bottom with a TNT-Booster or a cartridged high energy blasting agent like gelatine explosives. In case of problems with a full detonation of the whole explosives column occur, it is advisable to use redundant initiation with bottom and top detonators. Figure 49 shows an ideal column.

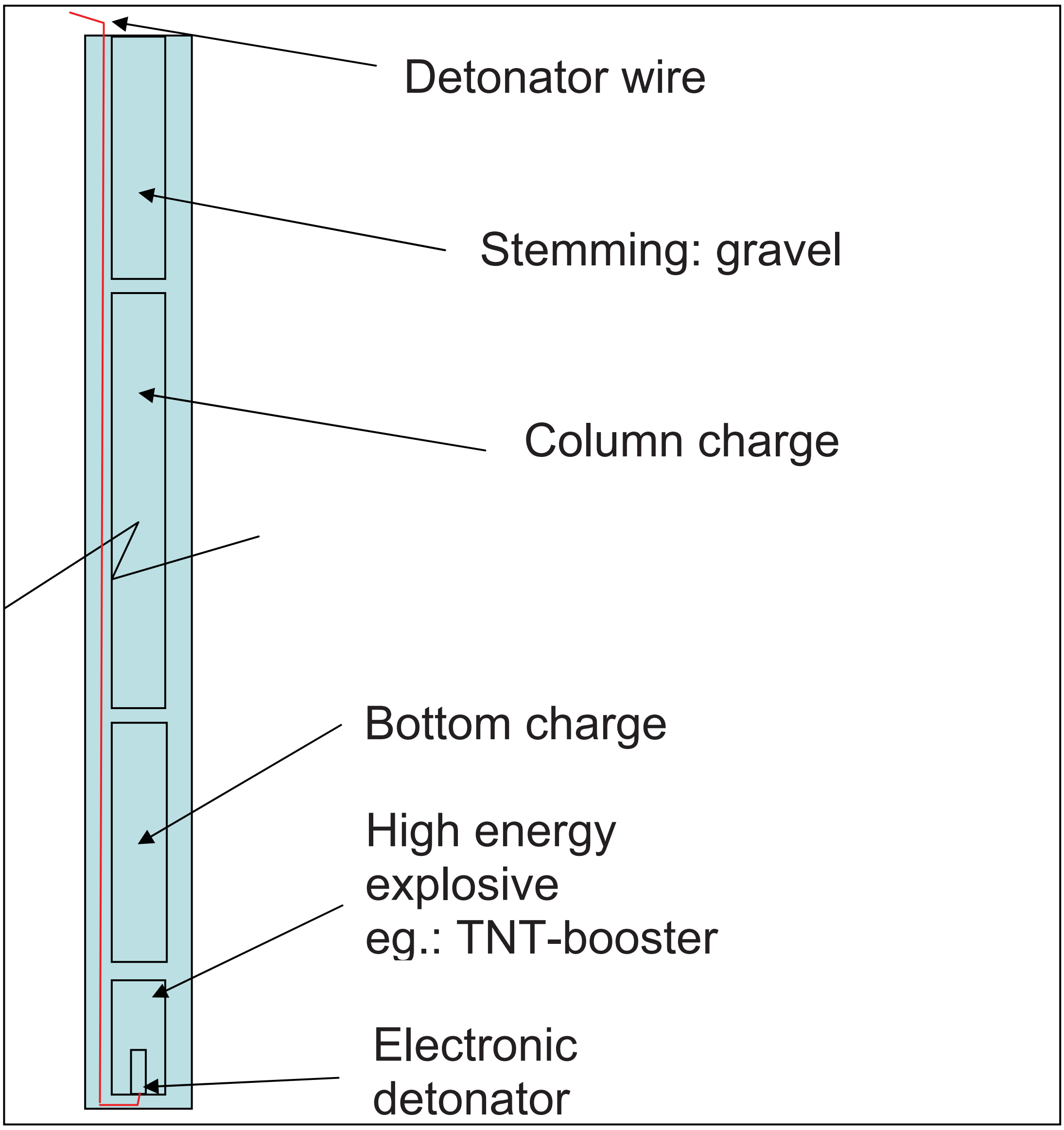


Figure 49: Drawing of an ideal blasting column

7.6.1. REVIEW ON THE ACTUAL WAY OF INITIATION

The initiation of the explosive from the bottom of the hole results in a better energy efficiency. The detonating cord at the top of the hole also ejects the stemming right after the initiation and cannot do what it is assigned for, namely to act like a closure for the hole and keep the gas pressure inside the borehole as long as possible.

7.7. Blasting Agent/Column Design

Concerning the kind of blasting agent there are some possibilities:

- Anfo explosives
- Slurry explosives
- Emulsion explosives or
- Heavy-Anfo explosives

From the described options Heavy Anfo is the newest one, although it can not be said, that this is best one. The explosive should be chosen in a way to best fit the rock. This can only be done by experimental trying.

7.7.1. Review on the column design

There is no clear rule how the column should be loaded and it does not only depend on the rock but also very much on the person that charges the holes. Sometimes the difference between loaded holes is more than one bag (25 kg) of Anfo although the holes got the same length and are just more widened from the drilling process, which results that more space is available for explosives.

7.8. Loading the holes

State of the art in loading the holes is using an adequate tool, depending on the kind of explosive. If you are using bulk emulsion explosives a pump truck is recommended. Explosives manufactures offer such systems also for anfo. It is also possible to use a trailer which could be pulled by the trucks available on site. Such a system is shown is Figure 50. It is equipped with a 3000 liters tank for anfo and two 100 liters tanks for fuel oil. It is able to mix and charge between 25 – 100 kg of explosives per minute. The charging unit also comes with its own power supply, which makes it even independent from the truck.

.

Figure 50: Example of an anfo trailer from DynoNobel, Source [8]

7.8.1. Review on Loading the Holes

With regard to a healthy and in the length of time not harmful working condition, it should be said, that carrying 25 kg bags of anfo is not advised. This is also consistent with Luzenac's occupational health act which says:" Employee health is a top priority for Luzenac and it uses best available health and safety practices appropriate to its operations. …", Source [9]

7.9. Documentation of the Blast

State of the art in documentation of the blast is much more than the "Sprengarbeitenverordnung" regulates. It should be the goal not only to have to document the blast but to be able to benefit from the documentation. Therefore it is meaningful to film or make pictures of the blast.

Without sufficient documentation a lot of information on already fired shots is lost, which would help in the planning process of further blasts.

7.10. Documentation of the Blasting Result

State of the art in documenting the blasting result would be to do this by making written notes and photos. An example for such a written documentation is shown in Table 17.

Table 17: Possible documentation sheet for the blasting result

Documentation of the Blasting Success	
Number of holes:	Date of blast: Time:
First impression	[from 1 (good) to 5 (bad)]
Muck shape	[sketch]
Muck throw	[from 1 (far) to 5 (close)]
Fragmentation of muck	[from 1 (even) to 5 (uneven)]
Number of boulders	
Any backbreak	
Extent of backbreak	[m]
Any visible missfires	
More notes	

If you want to measure the blasting result the best thing to do is to measure the muckpile shape and the achieved fragmentation. This can be done by surveying the muck pile which is very time intensive. Another way to measure the fragmentation is to scan each truck with muck using a camera and post-process the pictures with an appropriate software.

7.10.1. Review on Documentation of the Blasting Result

Unfortunately a lot of knowledge is lost by not documenting a fired blast. Written information can be very helpful in planning future blasts and also assists different shot-firers to get information from blasts they didn't fire.

7.11. Ground Vibrations and Noise Measurements

The state of the art and the regulatory requirements in Austria are to measure ground vibrations. Noise levels should be monitored regularly.

Regular information for the neighbours about the blasting work meets the standards in today's blasting practice. This can be done by:

- Organising regular information meetings with neighbours,
- Sending them information by mail or
- Setting up an information hotline.

7.11.1. Review on the actual state in ground vibrations and noise measurements

The measuring of the ground vibrations is done quite well. Noise levels should be measured from time to time and especially when delicate blasts are fired. To make sure the noise levels keep low, detonating cords should always be cut away and not hang out of holes or boulders and covered with sufficient material. Moreover the initiation of the column from the borehole bottom reduces noise levels because the produced energy that also produces the noise is kept as long as possible in the borehole.

Moreover the company should try to make further use of the ground vibration data, instead of just measuring them and filing them away.

The contact to neighbours could be intensified.

8. Gap Analysis

This part of the thesis should describe what has to be done to come from today's practice in drilling and blasting to the ideal situation that was presented in chapter 7.

8.1. Layout of the Blasting Work

In order to reach the status of the state of the art it is necessary to build up a management system for the complete planning process and the acquisition of all the relevant information. These comprise:

- Surveying of the wall and the drilled boreholes
- Planning and designing of a blast with an appropriate software
- Measuring of the borehole deviation
- Documentation of already blasted sites
- Adequate communication with neighbours
- Noise and Vibration monitoring

8.2. Surveying the wall

There is no further investment necessary. It is only advised to use the photogrammetric system more often in order to reduce holes that are drilled too long and know about the actual burdens.

8.3. Drilling

As the SmartRig-System is one of the big technological developments, the actual drilling machine should be traded in to get a Roc F9 C.

8.4. Quality of the Boreholes

It is advised to measure the hole deviation regularly. This can be done by either purchasing a system similar to the Boretrack-System. Maybe it is then even possible to buy such a system in connection with a drill rig, where no extra time is consumed for the measuring, but measuring is conducted by lifting up the drill rod, after drilling. Or it is also thinkable to conclude a contract for example with the University of Leoben, who already possess such a system, to measure hole deviations regularly.

8.5. Detonators/Way of Initiation

To have the latest technology in initiation a changeover to electronic detonators should be done. This should be accompanied by a complete redesign of the blast (burden, spacing, etc.) with an initiation of the blasting agent from the bottom of the hole with a redundant detonator if necessary.

8.6. Blasting Agent/Column Design

A test procedure should be started to find the explosive that fits the rock best respectively results in the desired fragmentation with fewest costs.

It should be stated that such a new choice in blasting agents should only be done if it goes hand in hand with a complete redesign of the blast, otherwise the positive effects won't come out that clearly.

8.7. Loading the Holes

The purchase of an adequate charging tool is recommended. If a purchase is not wanted it is also possible to make a service contract with one of the available explosives provider.

8.8. Documentation of the Blast

It would be necessary to buy a film camera or use the existing photo camera for a good documentation of the blast. Further on this information should be technical processed and stored to be able to go back and access data whenever needed.

8.9. Documentation of the Blasting Result

It should be started to document the blasting result with written forms (see example in chapter 7.10), making photos of the muck, any back breaks and other useful and visible information one could benefit of later and store these data in well processed form which makes it easy to access and use these data in the future.

8.10. Ground Vibrations and Noise

Ground vibrations should be measured further on. Also noise levels should be measured from time to time which would be possible with the existing ground vibration measurement device.

Further on ground vibration data should be processed more detailed to evaluate a prediction tool.

To intensify the contact to the neighbours it would be possible to send an email to all neighbours that are interested about the actual blasting time simultaneously to the contact to Mr. Reithofer, the neighbour whose house is within the safety radius of nearly all blasts.

9. Economic and Safety Benefits

9.1. Surveying of the Wall

The economic benefits are clear: a more regular use can reduce drill lengths and drill costs up to 20 % (see calculation in chapter 7.2.1)

The real good visualisation in colour of the actual burden for nearly every point of the wall is an important safety aspect that can show possible flyrock areas early and make quick reacting easy.

9.2. Drilling

The advantages of the new drilling technology SmartRig are:

- There is no need to manually mark the drilling starting points, because of the GPS navigation.
- Therefore the set-up time is much higher, which increases the rig utilisation and saves time and money.
- An automatic feed alignment also reduces set-up time and eliminates operator errors, because of predefined drillings angles.
- The automatic rod adding system makes it possible that the operator can carry out maintenance checks or other duties while drilling, because there is no need for him to stay in his cabin for the drilling process.
- The Rig Control System (RCS) adjusts power, feed and penetration rate to the properties of the rock, which results in less bit wear and reduced fuel consumption up to 30 %.
- The Measure While Drilling System logs important data for a rock property analysis, which makes an adjustment of explosives to the rock more easy and results in more efficient blasting and less explosives usage.

9.3. Quality of the Boreholes

On the economic side should be stated that better quality holes lead to a much more regular fragmentation, less wear for the shovel, faster loading and less energy for further comminution. Boreholes without deviations also lead to bigger possible drill patterns, which reduces explosives costs and drilling footage.

From the safety advantages it is clear that better knowledge of the holes position lead to less risk of very low burden and therefore reduces the flyrock hazard. Even the case of too much burden has an unpleasant aspect: blasting energy cannot be used for fragmentation if the burden is too big but will be transformed into vibration energy and may lead to damages on surrounding buildings.

9.4. DETONATORS

The fact that all detonators are identical, before a delay time is allocated, simplifies stockkeeping and eliminates the circumstance that a delay time can be out of stock. The high variability of delay times makes well pre-designed blasts with software possible and allows blasts with a higher number of holes without having to fear that vibrations increase. At the same time bigger blasts mean less numbers of blasts, which not only reduces complains by neighbours but is also a safety factor, because we might not forget that blasting is still a highly risky process.
In addition more precise delay times result in better fragmentation and easier rock-handling.

9.5. WAY OF INITIATION

Better energy efficiency causes less explosives consumption and therefore a cost reduction. Moreover an initiation of the blasting column from the bottom without a detonating cord at the top results in less generation of noise and reduces the risk of flyrock from stemming material due to blowouts.

9.6. BLASTING AGENT

The explosive that fits the rock best will probably result in less explosives consumption, hence reduced explosives costs, although it is not said that the outcome of a test procedure to find the best fitting explosive is not the actual used one.

9.7. LOADING THE HOLES

The whole charging process will be done within less time, because the workmen do not need to carry the explosives to every hole. This fact also improves the physical stress on the employees. Furthermore less manpower is needed. Whereas now the holes are charged by two or three men, with the DynoNobel charging unit just one is needed.
A charging unit enables the operator to put exact the same amount of explosive into every hole, which results in a better distribution over all holes, more homogeneous fragmentation, less explosives consumption and therefore reduces explosives costs.

9.8. DOCUMENTATION OF THE BLAST

Filming or making photos of a blast is a very easy and not cost intensive instrument to document a blast.

It could also be a good proof with regard to possible flyrock accusations of neighbours, especially now that the new part of the mine in the south is much nearer to surrounding houses, than it was before in the northern part, where just one neighbour was actually affected.

9.9. DOCUMENTATION OF THE BLASTING RESULT

In order to follow the company's goal of continuous improvements it is necessary to measure what is actually done. Only by doing this it can be assured that any changes result in a reduction of costs, namely explosives costs but also loading time and wear of diggers and crushers.

9.10. NOISE AND VIBRATIONS

The benefits from a more intense use of already measured ground vibrations are to be able to predict vibrations better and find out how the blast vibrations propagate in the various parts of the rock mass stronger than other parts and what the possible reasons for this effect are. It is feasible that differences occur when blasting after heavy rainfall or in dry periods. Even effects from different temperatures in summer and winter are thinkable.

By doing this, areas that are sensitive in particular can be discovered and special care can be taken with maximum amount of explosives fired at once, when blasting there.

Keeping the noise levels down at blasts is also important because neighbours are always firstly frightened because of a sudden bang and the created air overpressure and not of vibrations. A blast that can barely be heard is therefore always a blast that does not create too much attention. This might not be an economic or safety benefit first, but saves a lot of discomfort and therefore time which most times results in cost savings.

Concerning neighbours it can be an advantage when abutting owners know a lot of what is going on especially about blasting, which in their view is always a threat to their property and a lot of troubles don't even arise. A good relationship between company and neighbours is therefore most times a part that can influence the success of such a big project like a mining operation, positively.

10. CONCLUSIONS

10.1. SHORT TERM REALISATION

All issues in this paragraph can be realised already this year, because either the company already owns the necessary equipment or no further investments are needed to realise them. Therefore the existing resources should be used better and more intense. This includes:

- The regular surveying of faces with the system of the 3G-company in order to be able to adapt the borehole depth to the bench height.
- A reduction of borehole deviations.
- The trying out of other explosives and a change of initiation practice together with a new design of the column with detonators at the bottom of the hole and if necessary redundant detonators on the top of the charge.
- A stepwise extension of the existing blast pattern of at least 50 % of the now used area per hole. This means from 12 m² to 18 m² blasted area per hole. This would be even possible without changing the detonators or delay times. Only the drilling accuracy has to be improved to reach this goal.
- The documentation of the blast and the blasting success with the existing photo camera, combined with a written evaluation as it is shown in chapter 7.10.
- A better utilisation of the vibration data, that has to be gathered by law anyway.
- The start of a community information campaign to intensify the contact to all neighbours. Especially to those, who will hear and feel the mine much more, because the main extraction area moves from the northern part of the mine to the southern part in the near future.

10.2. MIDDLE TERM REALISATION

Topics here may already call for investments and reorganisation of working procedure. Therefore the timeframe is about two to four years. These comprise:

- The purchase of a technical tool that helps to make charging the holes easier, even faster and healthier for the workmen, because they would not have to carry 25 kg bags of explosives around anymore.
- The change from the existing electric detonators to electronic detonators, who allow bigger blasts, without increasing ground vibrations. Moreover blasts should be planned from then on with adequate software, which most times come with the electronic detonators.

- The purchase of AC's SmartRig-System, although it is relatively new on the market. Besides the advantages that were discussed in chapter 9.2, the company would be one of the first in the world to work with a new ground-breaking technology in surface drilling.

10.3. Long Term Realisation

The long term's aim has to be to build up a well working management system for the whole drilling and blasting process. All information for the design of each blast could be available there and all gained information from fired blasts is returned to it. This system should be aimed on continuous improvements of existing procedures in order to return not only more profit but also to create and keep a work environment were people work willingly and safely.

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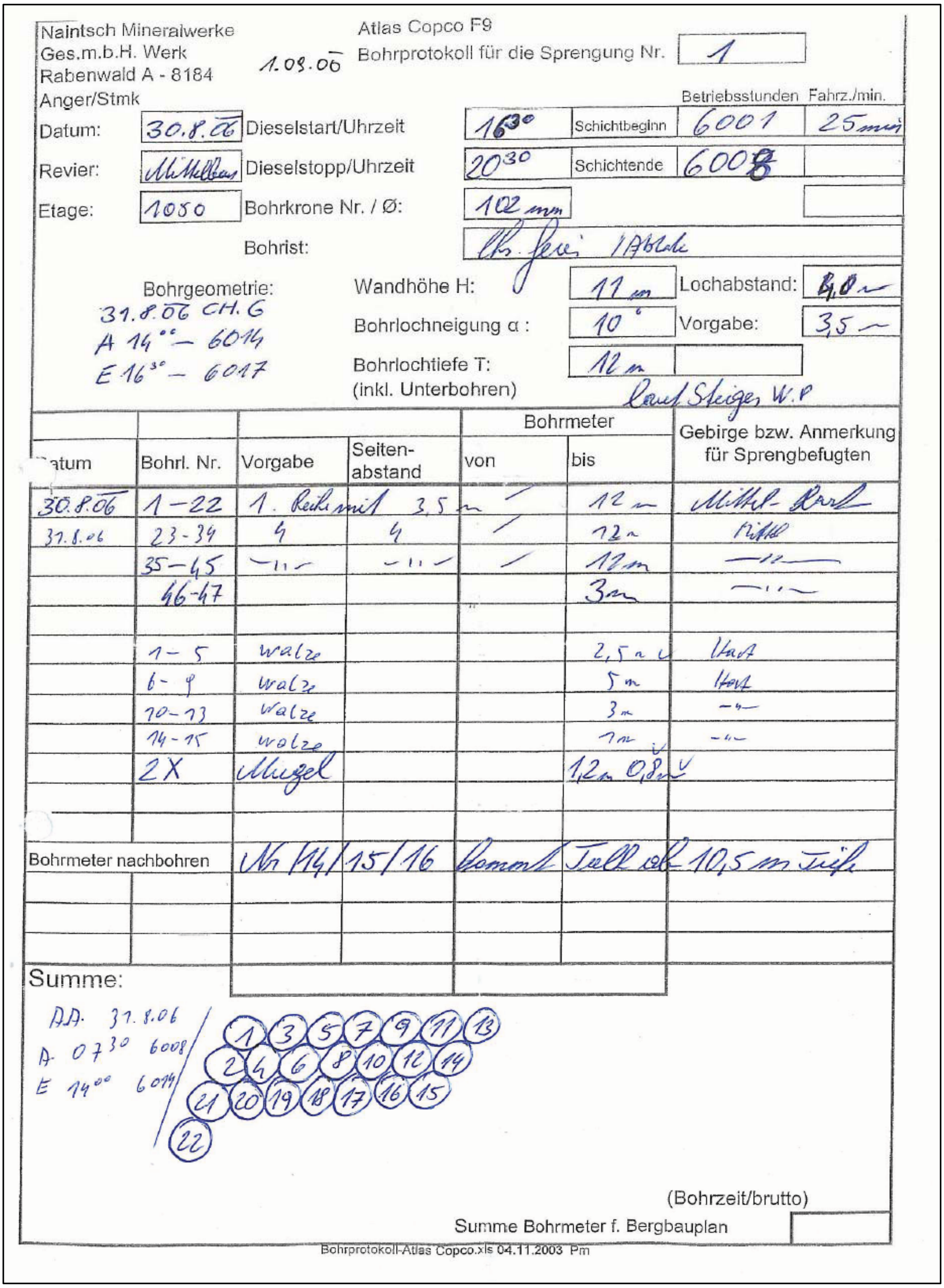
APPENDIX A: DRILL OPERATOR'S AND SHOT-FIRER'S BLAST RECORDS



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1.09.06 Bohrprotokoll für die Sprengung Nr. 1

					Betriebsstunden	Fahrz./min.
Datum:	30.8.06	Dieselstart/Uhrzeit	16^{30}	Schichtbeginn	6001	25 min
Revier:	Mittelbau	Dieselstopp/Uhrzeit	20^{30}	Schichtende	6008	
Etage:	1050	Bohrkrone Nr. / Ø:	102 mm			
		Bohrist:	Chr. Jeri / Ablede			

Bohrgeometrie:
31.8.06 CH. G
A 14^{00} – 6014
E 16^{30} – 6017

Wandhöhe H:	11 m	Lochabstand:	4,0 m
Bohrlochneigung α :	10°	Vorgabe:	3,5 m
Bohrlochtiefe T: (inkl. Unterbohren)	12 m		

Paul Steiger, W.P.

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
30.8.06	1–22	1. Reihe mit	3,5 m	/	12 m	Mittel-Hart
31.8.06	23-34	4	4	/	12 m	Mittel
	35–45	–"–	–"–	/	12 m	–"–
	46-47				3 m	–"–
	1–5	Walze			2,5 m	Hart
	6–9	Walze			5 m	Hart
	10–13	Walze			3 m	–"–
	14–15	Walze			1 m	–"–
	2X	Kugel			1,2 m 0,8 m	
Bohrmeter nachbohren	Nr 14/15/16	kommt	Teil ab	10,5 m Tief		
Summe:						

AA. 31.8.06
A 07^{30} 6008
E 14^{00} 6014

1 3 5 7 9 11 13
2 4 6 8 10 12 14
21 20 19 18 17 16 15
22

(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 51: Drill operator's record of the 2nd blast, Source [27]

Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: ...Mittelbau.........
Ø Wandhöhe : ....11m.........
Bohrlochneigung: ...80°.........
Sprengung: am .....1.08.2006...
Uhrzeit:.......18.15..........

Gebirge: .....Mittel / Hart........
Bohrlochdurchmesser: ...102.......mm
Verantwortlicher Leiter bzw.
Sprengbefugter: ..[illegible]

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 0-20 Stufe
1	4m	3.5m	1m					
2			ℓ 4Bl.					
3			2,5m	Supergel:	17 H →	425 kg		
4			ℓ 5Bl.					
5			3m	Paillex:	92 H →	2300 kg		
6			ℓ 4Bl.					
7			3m	L = 121 kg;	R =	260 m bis Bauernhof!		
8			ℓ 2Bl.					
9				Vmax =	4,1 mm/s	ist zu erwarten		
10	4m	4m	12m					
11			ℓ 45 Bl.					
12								
13				UB = 0,707	1008 ·	2,38 · 07		
14								
15				UB =	1,68			
16								
17								
18								
19								
20								
a.								
Ø				—	—	—		

.....Sohllöcher: Tiefe: ....m
..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff - Zusammenstellung:

Gel.Don.1 (Rowolit)	60/700	................kg
	45/350	................kg
	30/350	................kg
	30/130	................kg
Lambrit (Rowolan)	lose	................kg
Summe:		................kg

Datum: ....

ANC - Anteil:

Kopflöcher.......................%
Sonstige-Löcher..................%
insgesamt........................%

Zündmittel:
Det. Zündschnur ....610.......m
Zünder(Type).HU..62...........St.
Haufwerk:.........m³ =..........t
Spez.Sprengstoffverbrauch=.........g/t
Bemerkungen:

Figure 52: Shot-firer's record of the 2nd blast, Source [27]

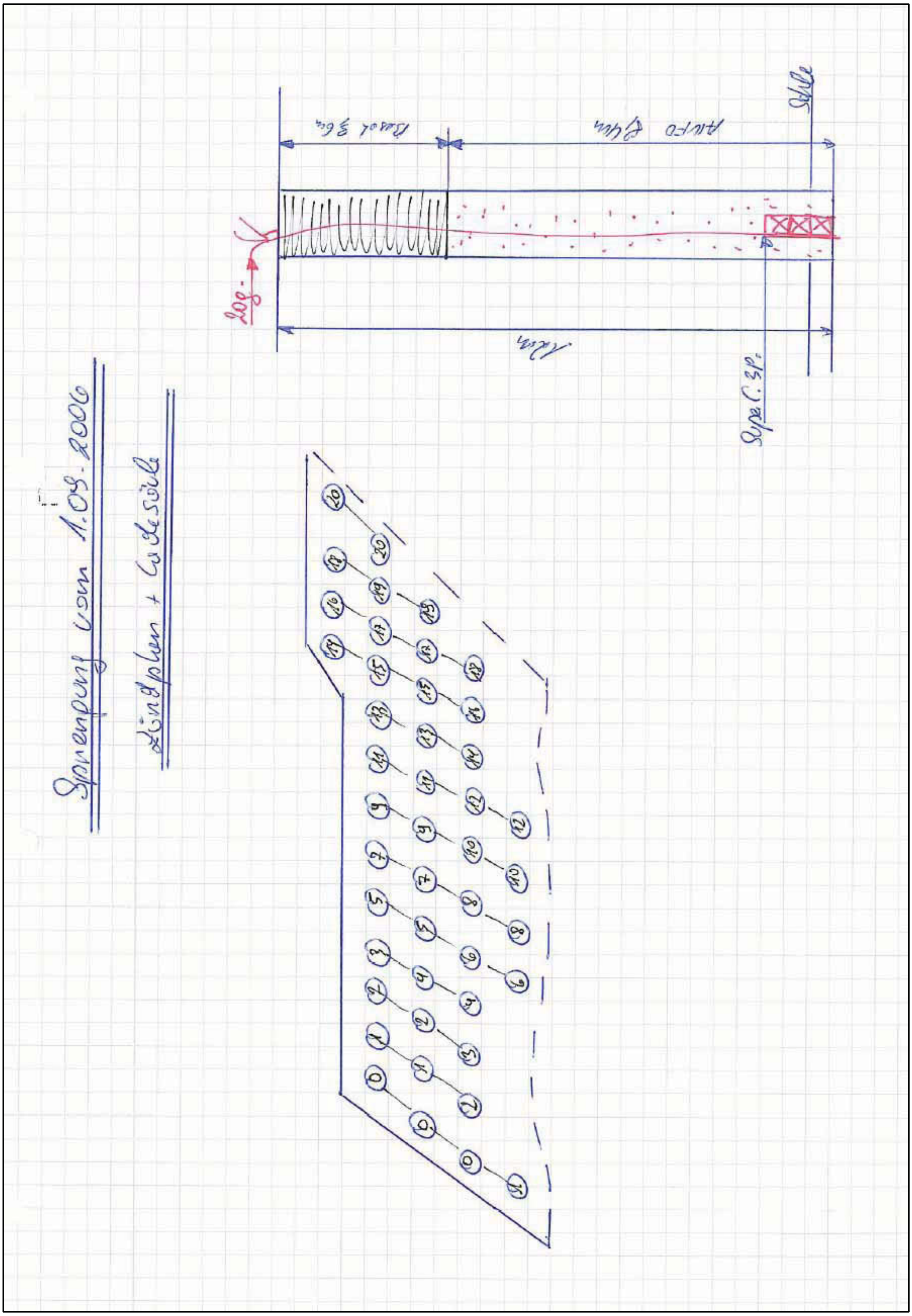

Figure 53: Shot-firer's plan of the 2nd blast, Source [27]

Naintsch Mineralwerke Ges.m.b.H. Werk Rabenwald A - 8184 Anger/Stmk

5.09.06

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Bohrprotokoll für die Sprengung Nr. 1

					Betriebsstunden	Fahrz./min.
Datum:	2.9.06	Dieselstart/Uhrzeit	08°°	Schichtbeginn	6017	15 min
Revier:	Reithofer	Dieselstopp/Uhrzeit	14⁴⁰	Schichtende	6022	+ 20 min
Etage:	1060	Bohrkrone Nr. / Ø:	100			
		Bohrist:	Ablack			

Bohrgeometrie:
4.9.06 CH.6
A 6³⁰ — 6022
E 13³³ — 6028

Wandhöhe H:	14 m	Lochabstand:	3,8
Bohrlochneigung α :	80°	Vorgabe:	3,5
Bohrlochtiefe T: (inkl. Unterbohren)	14 m		

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
2.9.06	1-2	Muggel	Ø 95		1 m	Hart
	3-9	Walze			2 m	Mittel
	10-20	Walze			3 m	Mittel
	21-31	Walze			5 m	Mittel
	32-37	3,5	3,8		12 m	Hart
	38-58				12 m	—"—
	58-59				6 m	—"—
	60-61				8 m	—"—
4.9.06	62-64	Walze			5 m	Mittel
	65-66	Walze			3 m	—"—
	67-69	Walze			1,5 m	—"—
	~~70-72~~					
Bohrmeter nachbohren						
	70-72	Walze	1. Etage tiefer (1030)		1,5 m	Hart
	73	Muggel			1 m	—"—

Summe:

A.A. 4.9.06
A. 14⁴⁵ 6028
E. 16¹⁰ 6030

(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 54: Drill operator's record of the 3rd blast, Source [27]

Firma

T I E F B O H R L O C H S P R E N G U N G Nr.: ....

Sprengort: ...Reithofen........
Ø Wandhöhe : ......11,5m......
Bohrlochneigung: ...80°.........
Sprengung: am ....5.09.2006....
Uhrzeit:......13 45..............

Gebirge: .....Mittel-Hart.........
Bohrlochdurchmesser: ...100.....mm
Verantwortlicher Leiter bzw.
Sprengbefugter: ...Heidenbauer Christian

B O H R-, L A D E- UND Z Ü N D P L A N:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 0-20 Stufe
1	3,8m	3,5m	1m					
2			↓ 3Bl.					
3			1,5m	Supergel:	2U. ->	50kg		
4			↓ 6Bl.					
5			2m	Emulgit:	18U. ->	450kg		
6			↓ 7Bl.					
7			3m	Prilled:	56U. ->	1400 kg		
8			↓ 13Bl.					
9			5m	L = 120kg;	R=	230m	bis Reithofen	
10			↓ 14Bl.					
11			6m					
12			↓ 2Bl.	Vmax =	9,8	mm/s ist	zu erwarten	
13			8m					
14			↓ 2Bl.					
15								
16			12m	UB =	0,707	1068 · 2,84	806	
17			↓ 32Bl.					
18				UB =	2,0			
19								
20								
Sa.								
Ø				——	——	——		

.....Sohllöcher: Tiefe: ....m
..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff – Zusammenstellung:

Gel.Don.1 60/700 ....kg
(Rowolit) 45/350 ....kg
30/350 ....kg
30/130 ....kg
Lambrit lose ....kg
(Rowolan)

Summe: ....kg

ANC – Anteil:

Kopflöcher........................%
Sonstige-Löcher...................%
insgesamt.........................%

Zündmittel:
Det. Zündschnur ...520.........m
Zünder(Type) .HU....56.........St.
Haufwerk:.........m³ =..........t
Spez.Sprengstoffverbrauch=.........g/t
Bemerkungen:

Datum: ....

Figure 55: Shot-firer's record for the 3rd blast, Source [27]

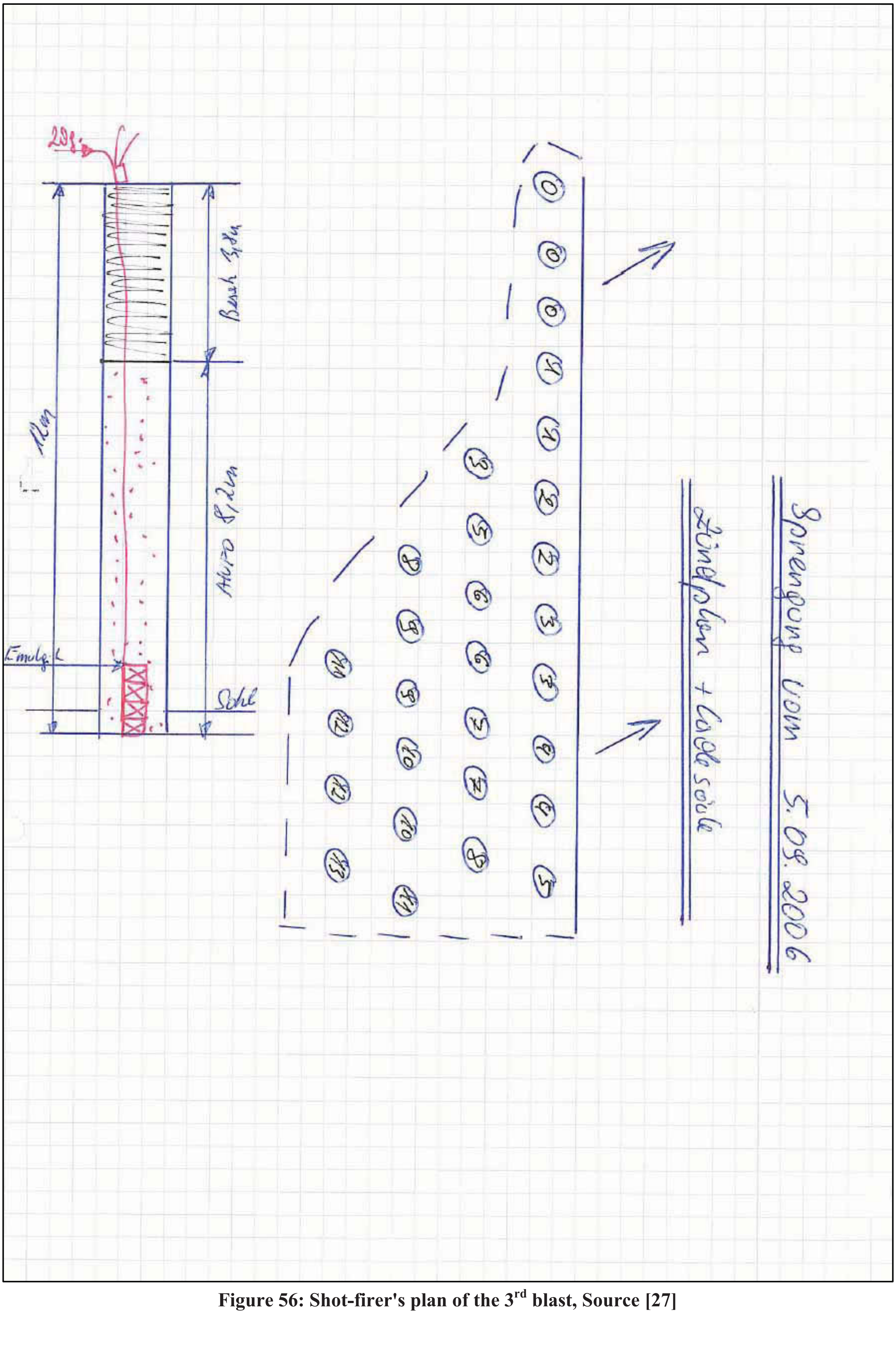
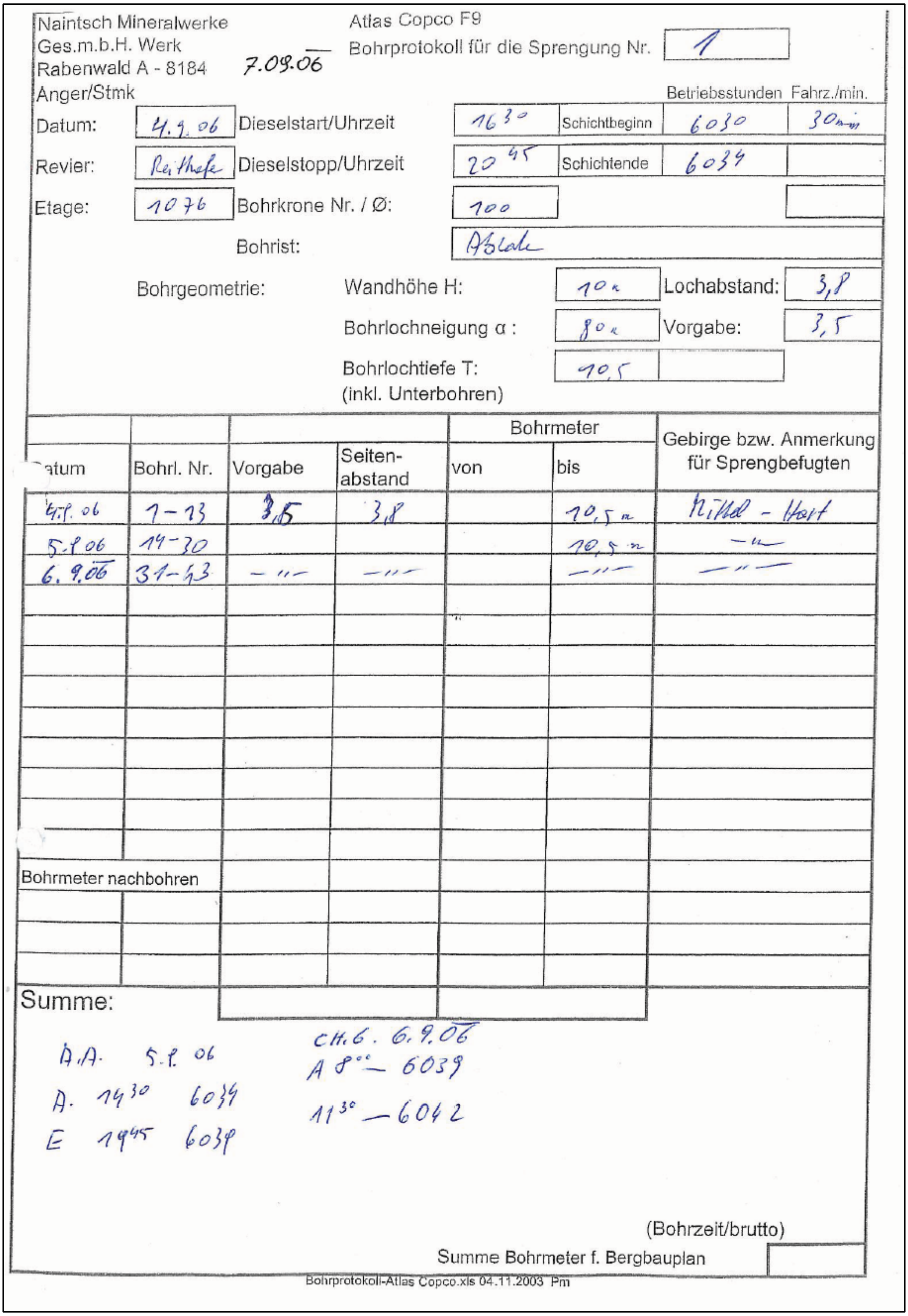


Figure 56: Shot-firer's plan of the 3rd blast, Source [27]



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7.09.06

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Bohrprotokoll für die Sprengung Nr. 1

				Betriebsstunden	Fahrz./min.
Datum: 4.9.06	Dieselstart/Uhrzeit	16^{30}	Schichtbeginn	6030	30 min
Revier: Reithofe	Dieselstopp/Uhrzeit	20^{45}	Schichtende	6034	
Etage: 1076	Bohrkrone Nr. / Ø:	100			
	Bohrist:	Ablake			

Bohrgeometrie:

Wandhöhe H:	10 m	Lochabstand:	3,8
Bohrlochneigung α :	80 °	Vorgabe:	3,5
Bohrlochtiefe T: (inkl. Unterbohren)	10,5		

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
4.9.06	1 – 13	3,5	3,8		10,5 m	Mittel - Hart
5.9.06	14 – 30				10,5 m	– " –
6.9.06	31 – 43	– " –	– " –		– " –	– " –
Bohrmeter nachbohren						
Summe:						

A.A. 5.9.06
A. 14^{30} 6034
E 19^{45} 6039

CH.6. 6.9.06
A 8^{00} — 6039
11^{30} — 6042

(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 57: Drill operator's record of the 4th blast, Source [27]

Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: ..Reithofen........ Gebirge: ......Mittel-Hart........

Ø Wandhöhe : ....10m...... Bohrlochdurchmesser: .100.......mm

Bohrlochneigung: .....80°.......

Sprengung: am ......7.09.2006 Verantwortlicher Leiter bzw.

Uhrzeit:.....13⁴⁵.................. Sprengbefugter: ..Kindenbauer Christian

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 0-20 Stufe
1	3,8m	3,5m	1m					
2			" 1Bl.	Supergel:	6h. →	150kg		
3								
4			19,5m	Priller:	67h →	1675kg		
5			" 43Bl.					
6				Emulgit:	7h →	175kg		
7								
8								
9								
10								
11								
12								
13								
14				Vmax =	2,3mm/s ist	zu ...		
15								
16				UB =	0,707 1068 ·	1,40	23348	
17								
18				UB =	0,97			
19								
20								
Sa.								
Ø				—	—	—		

.....Sohllöcher: Tiefe: ....m

..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg

GD1 ....kg ANC ......kg

Sprengstoff - Zusammenstellung:

Gel.Don.1 (Rowolit)	60/700	................kg
	45/350	................kg
	30/350	................kg
	30/130	................kg
Lambrit	lose	................kg
(Rowolan)		
Summe:		................kg

Datum: ....

ANC - Anteil:

Kopflöcher......................%	
Sonstige-Löcher.................%	
insgesamt.......................%	

Zündmittel:

Det. Zündschnur ......520....m

Zünder(Type)..HU..94..........St.

Haufwerk: 6.018...m³ = 15.046..t

Spez.Sprengstoffverbrauch=133....g/t

Bemerkungen:

Figure 58: Shot-firer's record of the 4th blast, Source [27]

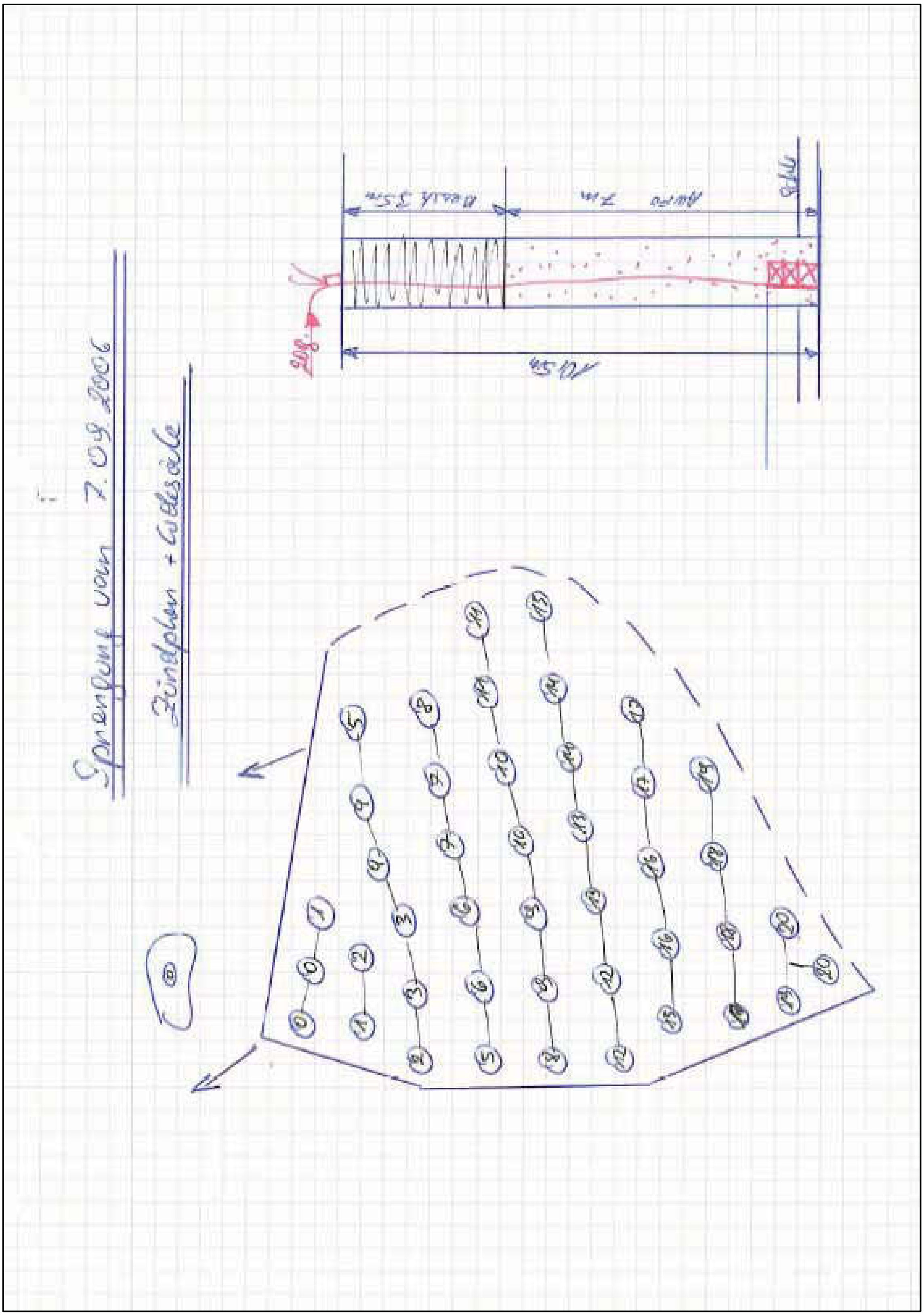
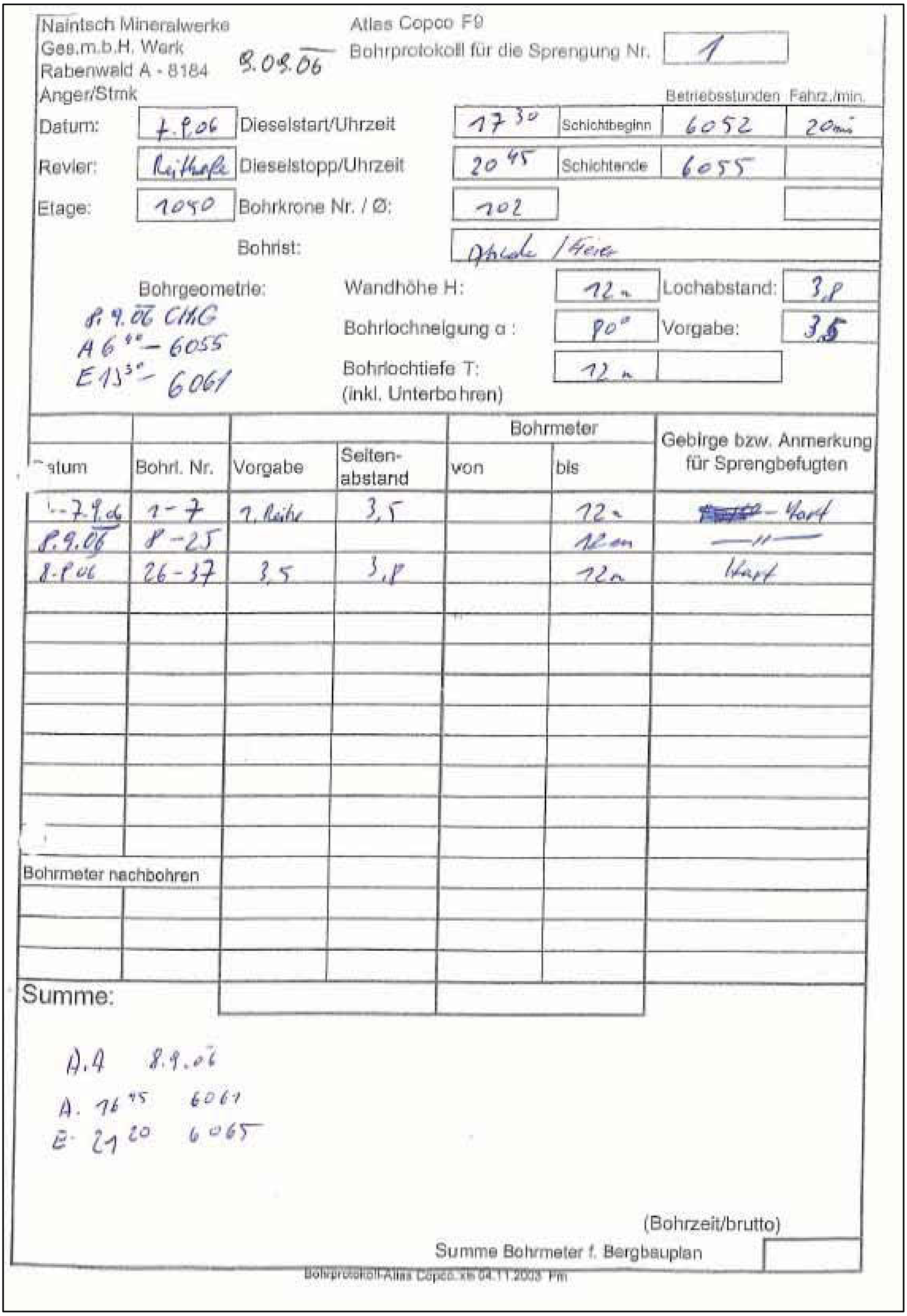


Figure 59: Shot-firer's record of the 4th blast, Source [27]



Naintsch Mineralwerke
Ges.m.b.H. Werk
Rabenwald A - 8184
Anger/Stmk

8.09.06

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Bohrprotokoll für die Sprengung Nr. 1

					Betriebsstunden	Fahrz./min.
Datum:	7.9.06	Dieselstart/Uhrzeit	17^{30}	Schichtbeginn	6052	20min
Revier:	Reithofer	Dieselstopp/Uhrzeit	20^{45}	Schichtende	6055	
Etage:	1080	Bohrkrone Nr. / Ø:	102			
		Bohrist:	Ahrule / Heier			

Bohrgeometrie:
8.9.06 CMG
A 6^{40} – 6055
E 13^{30} – 6061

Wandhöhe H:	12 m	Lochabstand:	3,8
Bohrlochneigung α :	90°	Vorgabe:	3,5
Bohrlochtiefe T: (inkl. Unterbohren)	12 m		

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
7.9.06	1 – 7	1. Reihe	3,5		12 m	~~[illegible]~~ – Hart
8.9.06	8 – 25				12 m	—"—
8.9.06	26 – 37	3,5	3,8		12 m	Hart

Bohrmeter nachbohren

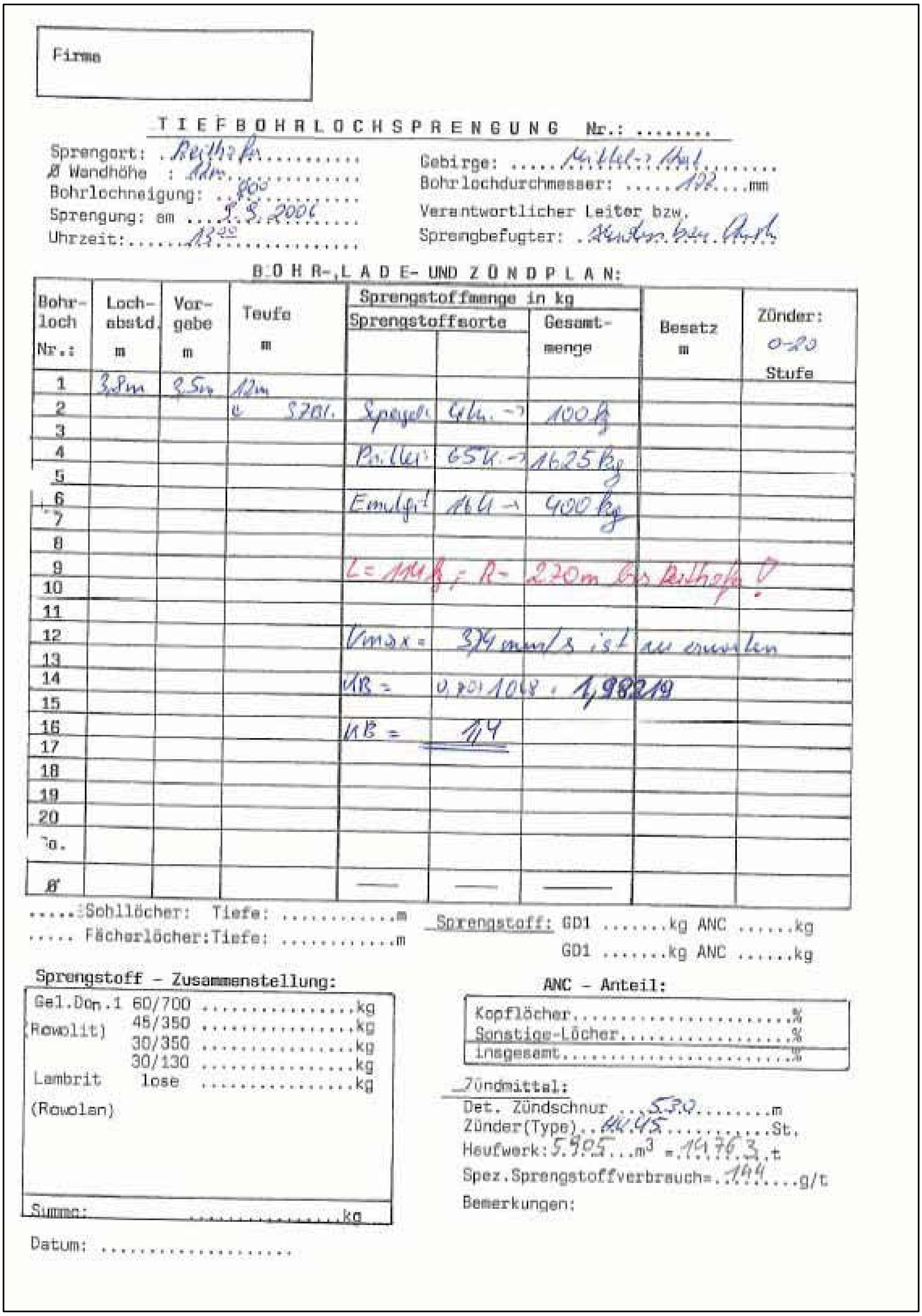
Summe:

A.A 8.9.06
A. 16^{45} 6061
E. 21^{20} 6065

(Bohrzeit/brutto)
Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Fm

Figure 60: Drill operator's record of the 5th blast, Source [27]



Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: Reithofe
Ø Wandhöhe : 12m
Bohrlochneigung: 80°
Sprengung: am 9.5.2006
Uhrzeit: 13[00]

Gebirge: Mittel-> Hart
Bohrlochdurchmesser: 102 mm
Verantwortlicher Leiter bzw. Sprengbefugter: Studer bzw. Ruck

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 0-20 Stufe
1	3,8m	2,5m	12m					
2			ø 5701.	Spreng:	Gh. →	100 kg		
3								
4				Pr. (?):	65 k. →	1625 kg		
5								
6				Emulg.!	16 k →	400 kg		
7								
8								
9				L = 114 kg; R =		220 m bis Reithofe!		
10								
11								
12				Vmax =	3,4 mm/s ist	zu erwarten		
13								
14				KB =	0,701 1068 =	1,98219		
15								
16				KB =	1,4			
17								
18								
19								
20								
..o.								
ø				—	—	—		

..... Sohllöcher: Tiefe: ....m
..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff – Zusammenstellung:

Gel.Don.1 60/700 ....kg
(Rowolit) 45/350 ....kg
30/350 ....kg
30/130 ....kg
Lambrit lose ....kg
(Rowolan)

Summe: ....kg

Datum: ....

ANC – Anteil:

Kopflöcher........................%
Sonstige-Löcher..................%
insgesamt........................%

Zündmittel:
Det. Zündschnur ...530........m
Zünder(Type)..M.45...........St.
Haufwerk: 5.905...m³ = 14763 t
Spez.Sprengstoffverbrauch= 144 ....g/t
Bemerkungen:

Figure 61: Shot-firer's record of the 5th blast, Source [27]

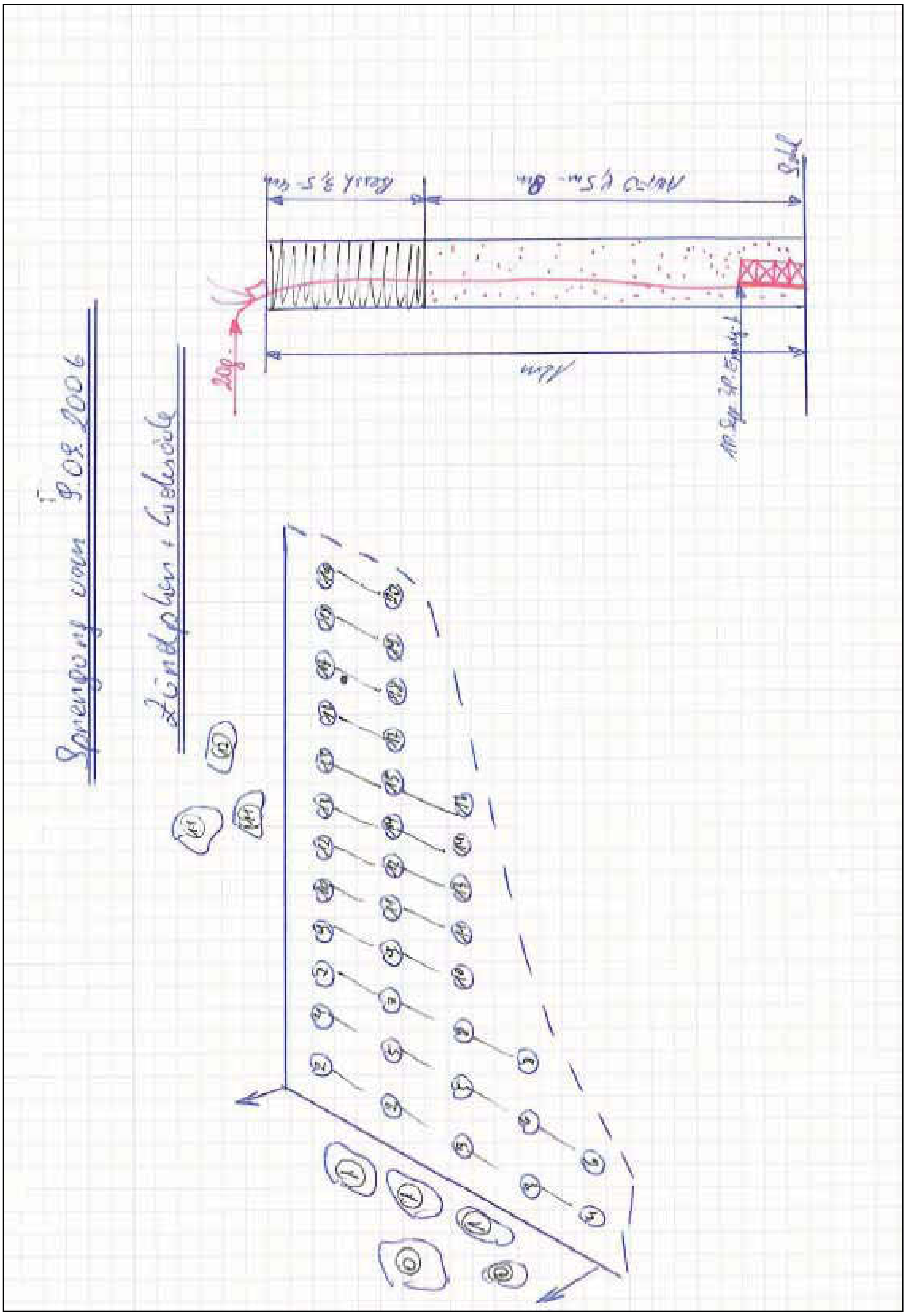


Figure 62: Shot-firer's plan of the 5th blast, Source [27]

Naintsch Mineralwerke Ges.m.b.H. Werk Rabenwald A - 8184 Anger/Stmk

15.09.06

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Bohrprotokoll für die Sprengung Nr. 1

Datum:	13.9.06	Dieselstart/Uhrzeit	14^{30}	Schichtbeginn	6101
Revier:	Reithofer	Dieselstopp/Uhrzeit	21^{30}	Schichtende	6107
Etage:	1060	Bohrkrone Nr. / Ø:	102 m		
		Bohrist:	Ch. Gross / Ablak		

Betriebsstunden: Schichtbeginn 6101, Schichtende 6107; Fahrz./min.: 15 min

Bohrgeometrie:

14. 9. 06 CH/G
A 14^{10} – 6112
E 20^{0} – 6117

Wandhöhe H:	11 m	Lochabstand:	3,5
Bohrlochneigung α :	15°	Vorgabe:	3,5
Bohrlochtiefe T: (inkl. Unterbohren)	12 m		

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
13.9.06	1 – 20			11 –	12 m ✓	Hart
	1 – 16				2 m	– " –
14.9.06	1 – 9	3,5	3,8		11 m ✓	Mittel – Hart
	10 – 16	3,5	3,8		11,5 m ✓	– " –
	17				12 m	– " –
14.9.06	18 – 24	– "	– " –		12 m	– " –
Bohrmeter nachbohren		10X Nachgebohrt				
		vorne sehr Bruchig				
Summe:						

A.A. 14.9.06
A. 08^{30} 6107
E. 14^{00} 6112

(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 63: Drill operator's record of the 6th blast, Source [27]

Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: Reithofer
Ø Wandhöhe: 11m
Bohrlochneigung: 75°
Sprengung: am 15.09.2006
Uhrzeit: 12⁰⁰

Gebirge: Mittel - Hart
Bohrlochdurchmesser: 101 mm
Verantwortlicher Leiter bzw.
Sprengbefugter: Heidenbauer Christian

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 0-17 Stufe
1	3,8m	3,5m	2m					
2			↓ 16Bl.					
3				Supergel	5k. ->	125kg		
4			11m					
5			↓ 9Bl.	Emulgit	13k. ->	325kg		
6			11,5m					
7			↓ 7Bl.	ANFO	77k ->	1925kg		
8			12m					
9			↓ 28Bl.					
10								
11								
12				Vmax = 4,25 mm/s ist zu erwarten				
13								
14								
15				UB = 0,7071068 · 2,59127				
16								
17				UB = 1,83				
18								
19								
20								
Ge.								
Ø				——	——	——		

..... Sohllöcher: Tiefe: ....m
..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff - Zusammenstellung:

Gel.Don.1 (Rowolit)	60/700	................kg
	45/350	................kg
	30/350	................kg
	30/130	................kg
Lambrit (Rowolan)	lose	................kg
Summe:		................kg

ANC - Anteil:

Kopflöcher	........................%
Sonstige-Löcher	..................%
insgesamt	........................%

Zündmittel:
Det. Zündschnur 580 m
Zünder (Type) HU 51 St.
Haufwerk: ....m^3 = ...t
Spez. Sprengstoffverbrauch = ...g/t
Bemerkungen:

Datum: ....

Figure 64: Shot-firer's record of the 6th blast, Source [27]

Naintsch Mineralwerke
Ges.m.b.H. Werk
Rabenwald A - 8184 19.09.06
Anger/Stmk

Atlas Copco F9
Bohrprotokoll für die Sprengung Nr.

					Betriebsstunden	Fahrz./min.
Datum:	16.9.06	Dieselstart/Uhrzeit	06^{45}	Schichtbeginn	6119	35 min
Revier:	Reithofer	Dieselstopp/Uhrzeit	14^{40}	Schichtende	6127	+ 20 min
Etage:	1086	Bohrkrone Nr. / Ø:	102			
		Bohrist:	Ablaler / Tödling			

TG.
A: 6127 $\cancel{14}$
E: 6133 14^{05}

Bohrgeometrie:

Wandhöhe H:	11,5 m	Lochabstand:	3,4
Bohrlochneigung α :	~~85~~ 75 °	Vorgabe:	3,2
Bohrlochtiefe T: (inkl. Unterbohren)	12 m		

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
16.9.	1 - 18	3,2	3,4		12 m	Hart
18.9.	19 - 26	3,2	3,4		12 m	Hart
18.9.	27 - 47				12 m	—"—
Bohrmeter nachbohren						
Summe:						

A.A. 18.9.06
A. 14^{00} 6133
E. 21^{25} 6140

(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 65: Drill operator's record of the 7th blast, Source [27]

Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: ..Reithofer.........
Ø Wandhöhe : ..11,5..........
Bohrlochneigung: ..75°..........
Sprengung: am ..19.09.2006..
Uhrzeit:...12⁰⁰................

Gebirge: ...Hart..................
Bohrlochdurchmesser: ...102.....mm
Verantwortlicher Leiter bzw.
Sprengbefugter: ..Maierhofer........

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 0–17 Stufe
1	3,4	3,2	12 m					
2			↓ 48 BL					
3								
4				Supergel:	22	K. →	550 kg	
5								
6				Emulgit:	7	K. →	175 kg	
7								
8				ANFO:	67	S. → 1	675 kg	
9								
10				L = 15	3,20 kg	; R = 2	00 m bis	Reithofer!
11								
12				Vmax =	7,00 mm/s	zu	erwarten	
13								
14				UB = 0,	7071068 ·	5,96	608	
15								
16				UB =	3,86			
17								
18								
19								
20								
Ga.								
Ø				——	——	——		

.....Sohllöcher: Tiefe: ....m
..... Fächerlöcher:Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff – Zusammenstellung:

Gel.Don.1 (Rowolit)	60/700	................kg
	45/350	................kg
	30/350	................kg
	30/130	................kg
Lambrit (Rowolan)	lose	................kg
Summe:		................kg

ANC – Anteil:

Kopflöcher......................%
Sonstige-Löcher.................%
insgesamt.......................%

Zündmittel:
Det. Zündschnur .....520.......m
Zünder(Type)..H.U....48.........St.
Haufwerk:.6267...m³ = 15667..t
Spez.Sprengstoffverbrauch=..153...g/t
Bemerkungen:

Datum: ....

Figure 66: Shot-firer's record of the 7th blast, Source [27]

Naintsch Mineralwerke Ges.m.b.H. Werk Rabenwald A - 8184 Anger/Stmk

Atlas Copco F9

Bohrprotokoll für die Sprengung Nr.

26.09.06

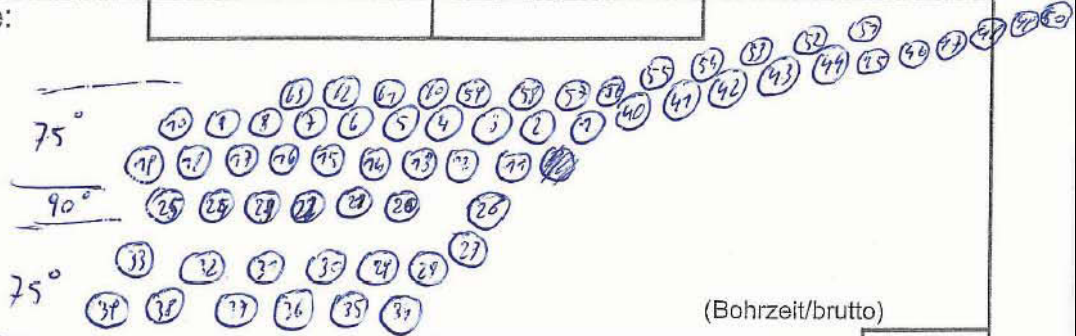
				Betriebsstunden	Fahrz./min.
Datum:	25.9.06	Dieselstart/Uhrzeit	07^15	Schichtbeginn 6760	15 min
Revier:	Kruphof	Dieselstopp/Uhrzeit	17^30	Schichtende 6771	
Etage:	1068	Bohrkrone Nr. / Ø:	100		
		Bohrist:	[illegible]		

Bohrgeometrie:

A.A. 26.9.06
A. 06^30 6771
E 10^00 6779

Wandhöhe H:	8 m	Lochabstand:	3,8
Bohrlochneigung α :	75°	Vorgabe:	3,4
Bohrlochtiefe T: (inkl. Unterbohren)	8,5 m		

				Bohrmeter		
Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	von	bis	Gebirge bzw. Anmerkung für Sprengbefugten
25.9.06	1 - 25	3,4	3,8		8,5 m	Mittel - hart
	26				10 m	- " -
	27				12 m	- " -
	28 - 39	3,6	3,8		14 m	- " -
	40 - 44				7 m	- " -
26.9.06	45 - 55				6 m	- " -
	56 - 63				9,5 m	- " -
Bohrmeter nachbohren						
Summe:						



(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 67: Drill operator's record of the 8th blast, Source [27]

Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: ..Krughof..........
Ø Wandhöhe : ..8–12m........
Bohrlochneigung: ..75°........
Sprengung: am ..26.9.2006.......
Uhrzeit:....13:45...............

Gebirge: ..Mittel-Hart..........
Bohrlochdurchmesser: 100.......mm
Verantwortlicher Leiter bzw.
Sprengbefugter: ..SCHWARZ.......

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Sprengstoffmenge in kg: Gesamtmenge	Besatz m	Zünder: 0–17 Stufe
1	3,8	3,4	6 m					
2			↓ 11 BL	Supergel	16k 5P →	412,5 kg		
3			7 m					
4			↓ 5 BL	Prillex	86S	– 2150 kg		
5			8,5 m					
6			↓ 33 BL					
7			10 m	R =	240 m	Lges – 205 kg		
8			↓ 1 BL					
9			12 m					
10	3,8	3,6	↓ 1 BL					
11			14 m		Vmax:	6,35 mm/sec		
12			↓ 12 BL					
13					KB =	2,35		
14								
15								
16								
17								
18								
19								
20								
Sa.								
Ø				——	——	——		

..... Sohllöcher: Tiefe: ....m
..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff – Zusammenstellung:

Gel.Don.1 (Rowolit)	60/700	...............kg
	45/350	...............kg
	30/350	...............kg
	30/130	...............kg
Lambrit (Rowolan)	lose	...............kg
Summe:		...............kg

Datum: ....

ANC – Anteil:

Kopflöcher......................	%
Sonstige-Löcher.................	%
insgesamt.......................	%

Zündmittel:
Det. Zündschnur ..640.........m
Zünder(Type)...HU – 63.......St.
Haufwerk:..7511..m³ =..18779.t
Spez.Sprengstoffverbrauch=..136...g/t
Bemerkungen:

Figure 68: Shot-firer's record of the 8th blast, Source [27]

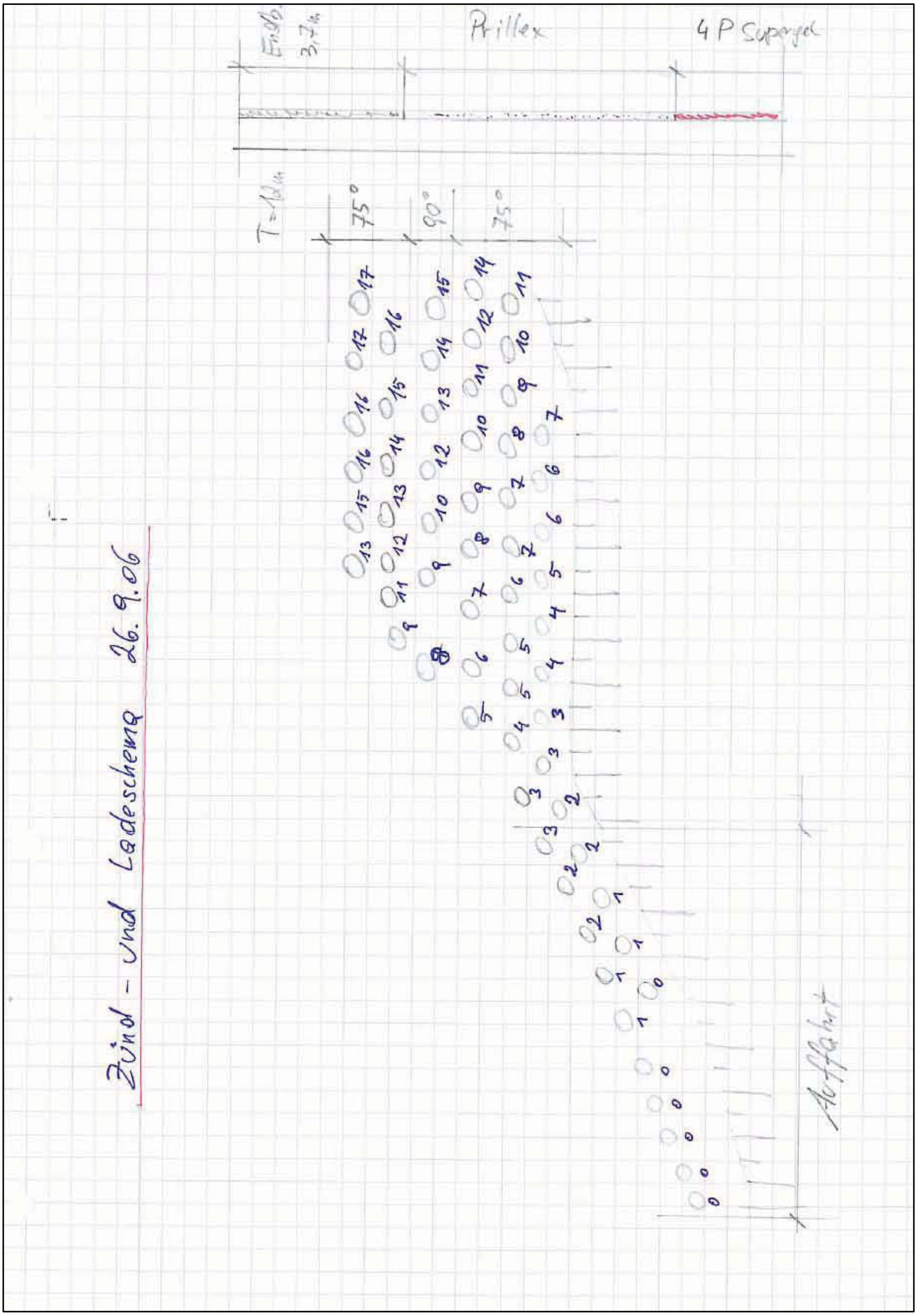
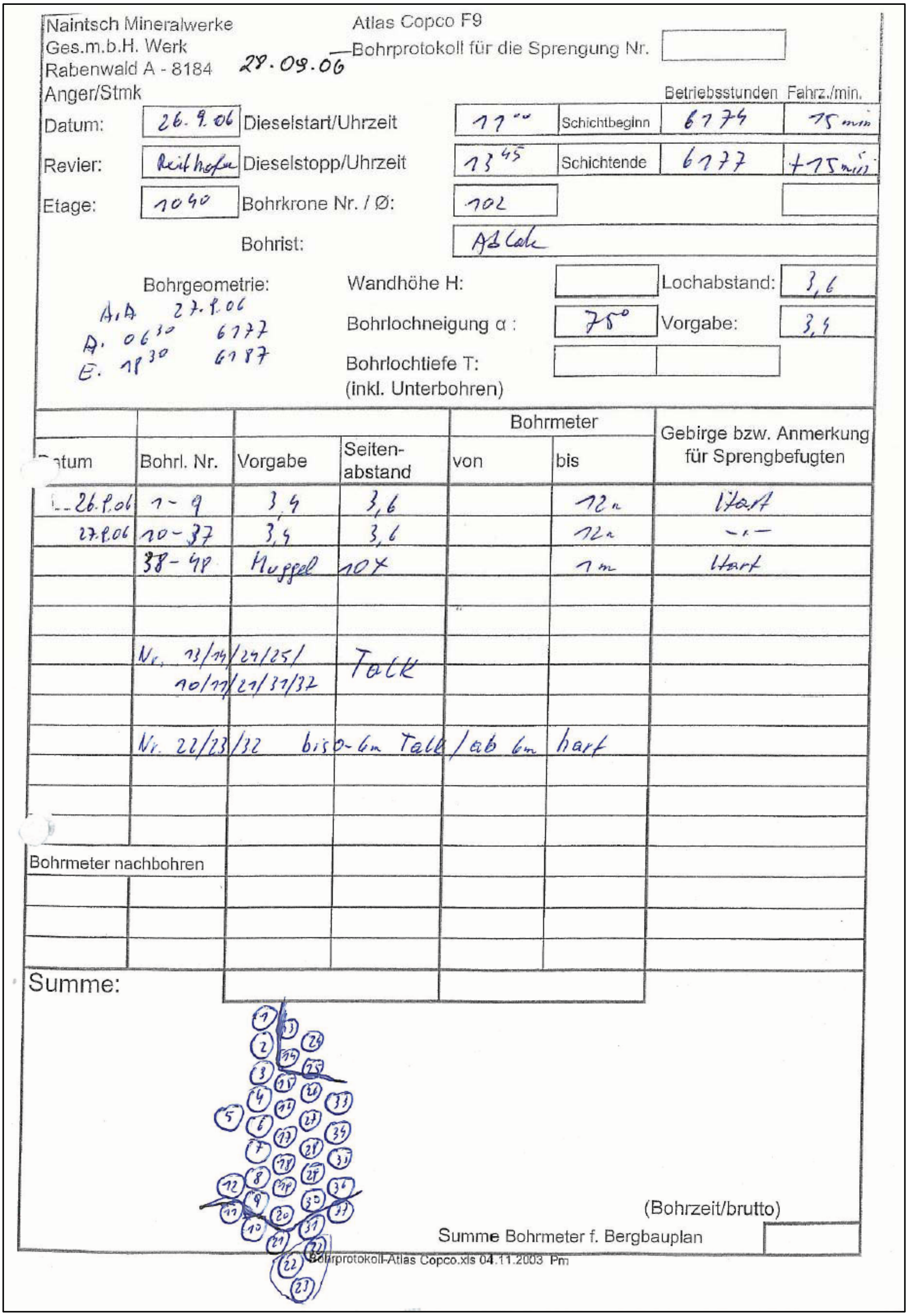


Figure 69: Shot-firer's plan of the 8th blast, Source [27]



Naintsch Mineralwerke Ges.m.b.H. Werk Rabenwald A - 8184 Anger/Stmk 28.09.06

Atlas Copco F9

Bohrprotokoll für die Sprengung Nr.

					Betriebsstunden	Fahrz./min.
Datum:	26.9.06	Dieselstart/Uhrzeit	11⁰⁰	Schichtbeginn	6774	15 min
Revier:	Reithofer	Dieselstopp/Uhrzeit	13⁴⁵	Schichtende	6777	+15 min
Etage:	1040	Bohrkrone Nr. / Ø:	102			
		Bohrist:	Ab Cale			

Bohrgeometrie:

A.A 27.9.06
A. 06³⁰ 6777
E. 18³⁰ 6787

Wandhöhe H:		Lochabstand:	3,6
Bohrlochneigung α :	75°	Vorgabe:	3,4
Bohrlochtiefe T: (inkl. Unterbohren)			

Datum	Bohrl. Nr.	Vorgabe	Seiten-abstand	Bohrmeter von	Bohrmeter bis	Gebirge bzw. Anmerkung für Sprengbefugten
26.9.06	1 - 9	3,4	3,6		12 m	Hart
27.9.06	10 - 37	3,4	3,6		12 m	– " –
	38 - 48	Muggel	10 x		1 m	Hart
	Nr. 13/14/24/25/ 10/11/21/31/32		Talk			
	Nr. 22/23/32		bis 0-6 m Talk	/ ab 6 m	hart	

Bohrmeter nachbohren

Summe:

(Bohrzeit/brutto)

Summe Bohrmeter f. Bergbauplan

Bohrprotokoll-Atlas Copco.xls 04.11.2003 Pm

Figure 70: Drill operator's record of the 9th blast, Source [27]

Firma

TIEFBOHRLOCHSPRENGUNG Nr.: ....

Sprengort: ..Reithofer..........
Ø Wandhöhe : ..11m..........
Bohrlochneigung: 75°..........
Sprengung: am ..28.9.06.....
Uhrzeit:....13⁴⁵..............

Gebirge: ..Hart (Talk)..........
Bohrlochdurchmesser: ..102......mm
Verantwortlicher Leiter bzw.
Sprengbefugter: ...Schwarz......

BOHR-, LADE- UND ZÜNDPLAN:

Bohrloch Nr.:	Lochabstd. m	Vorgabe m	Teufe m	Sprengstoffmenge in kg: Sprengstoffsorte		Gesamtmenge	Besatz m	Zünder: 9–20 Stufe
1	–	–	1 m	Supergel :	13KMP -	3275 kg		
2			↓ 13 BL					
3	3,6	3,4	12 m	Pillex :	585	1450 kg		
4			↓ 37 BL					
5				Emulgit	3k =	75 kg		
6								
7								
8				R = 300m		Lges: 150 kg		
9								
10								
11								
12				Vmax :	3,77			
13								
14								
15				KB =	1,4			
16								
17								
18								
19								
20								
ʼa.								
Ø				——	——	——		

.....Sohllöcher: Tiefe: ....m
..... Fächerlöcher: Tiefe: ....m

Sprengstoff: GD1 ....kg ANC ......kg
GD1 ....kg ANC ......kg

Sprengstoff – Zusammenstellung:

Gel.Don.1 (Rowolit)	60/700	................kg
	45/350	................kg
	30/350	................kg
	30/130	................kg
Lambrit (Rowolan)	lose	................kg
Summe:		................kg

Datum: ....

ANC – Anteil:

Kopflöcher......................%
Sonstige-Löcher..................%
insgesamt.......................%

Zündmittel:
Det. Zündschnur ...360...........m
Zünder(Type)....Hu....41......St.
Haufwerk:.........m^3 =..........t
Spez.Sprengstoffverbrauch=.........g/t
Bemerkungen:

Figure 71: Shot-firer's record of the 9th blast, Source [27]

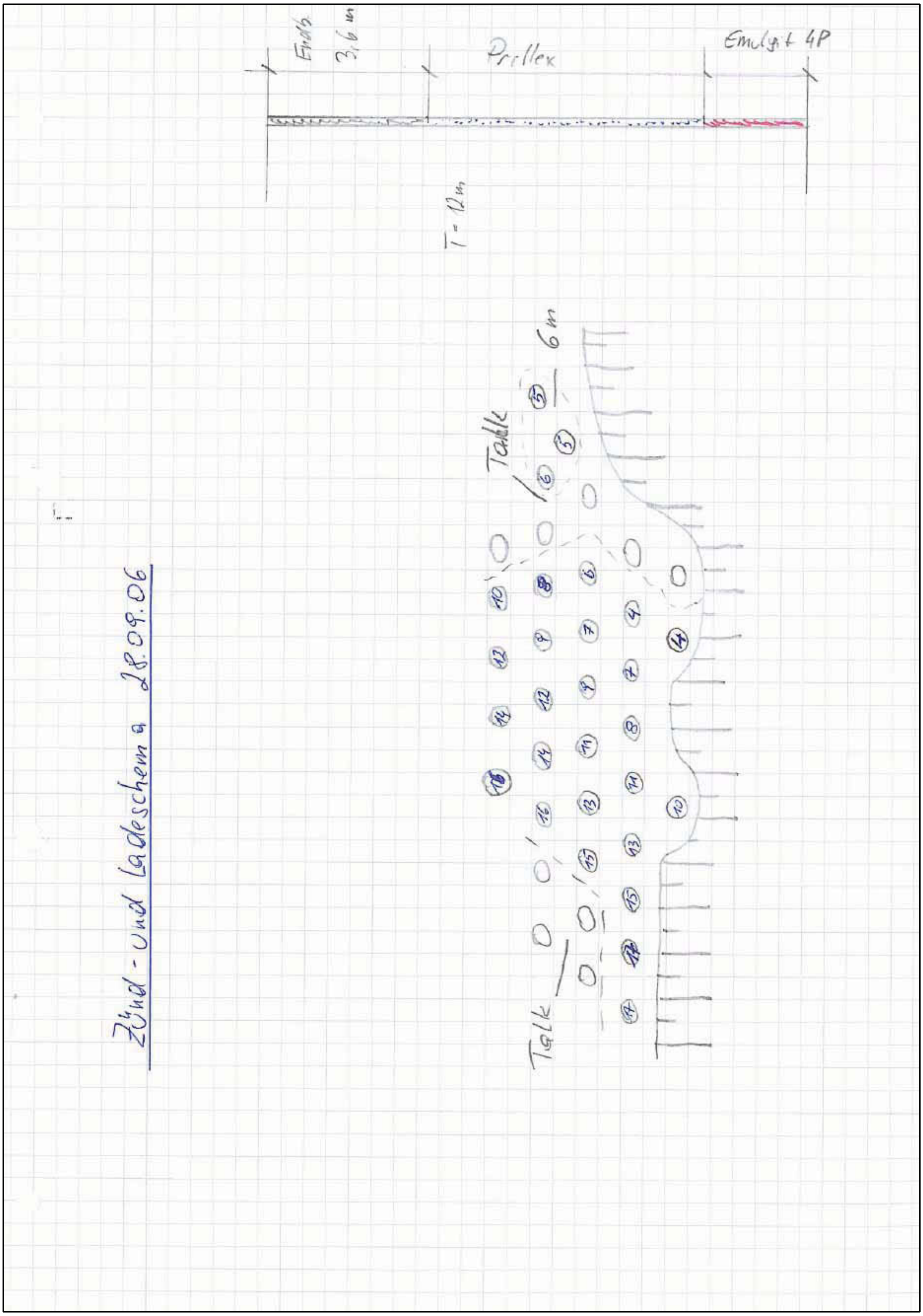
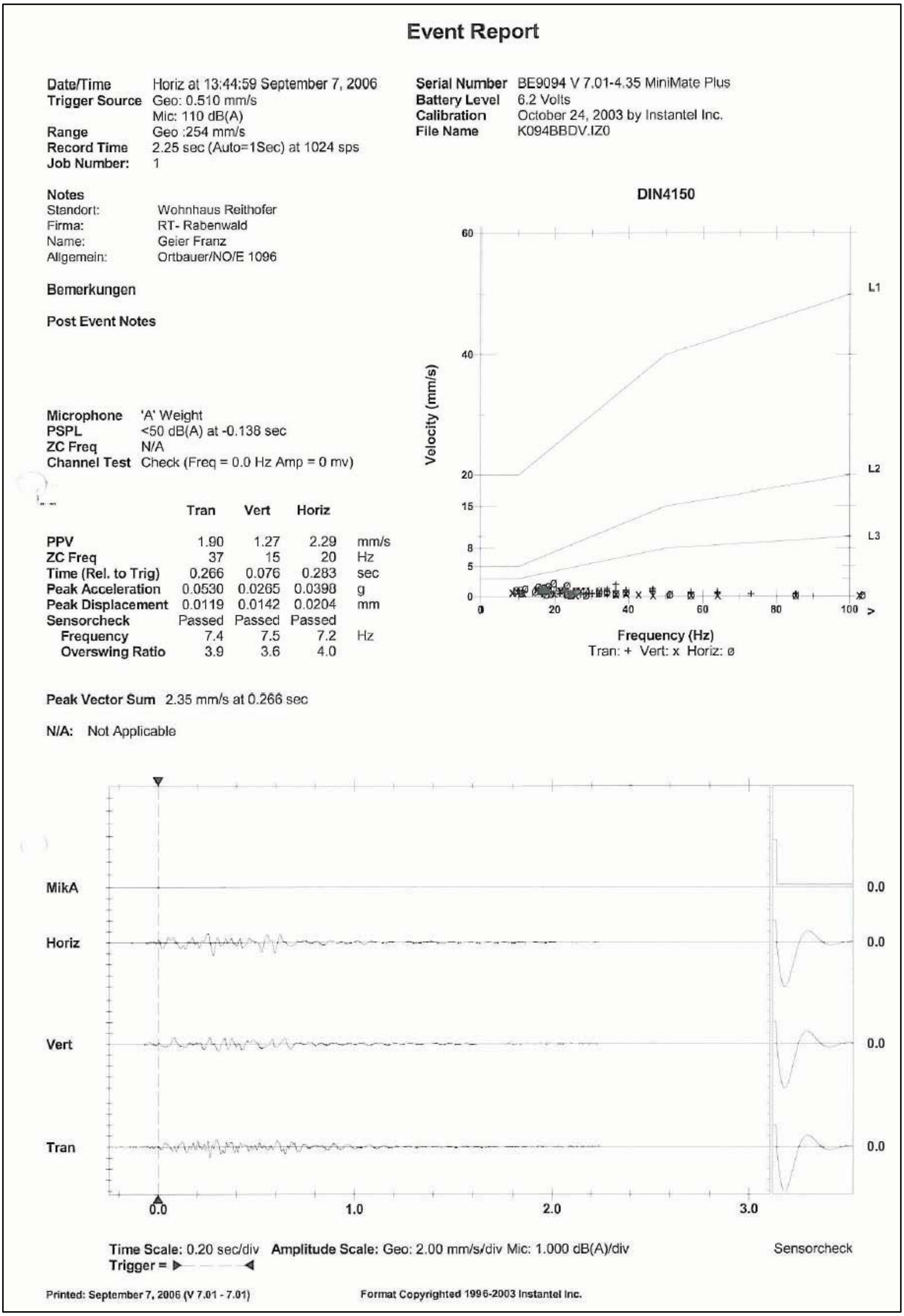


Figure 72: Shot-firer's plan of the 9th blast, Source [27]

APPENDIX B: VIBRATION EVENT REPORTS



Event Report

Date/Time	Horiz at 13:44:59 September 7, 2006
Trigger Source	Geo: 0.510 mm/s Mic: 110 dB(A)
Range	Geo :254 mm/s
Record Time	2.25 sec (Auto=1Sec) at 1024 sps
Job Number:	1

Serial Number	BE9094 V 7.01-4.35 MiniMate Plus
Battery Level	6.2 Volts
Calibration	October 24, 2003 by Instantel Inc.
File Name	K094BBDV.IZ0

Notes

Standort:	Wohnhaus Reithofer
Firma:	RT- Rabenwald
Name:	Geier Franz
Allgemein:	Ortbauer/NO/E 1096

Bemerkungen

Post Event Notes

Microphone	'A' Weight
PSPL	<50 dB(A) at -0.138 sec
ZC Freq	N/A
Channel Test	Check (Freq = 0.0 Hz Amp = 0 mv)

	Tran	Vert	Horiz	
PPV	1.90	1.27	2.29	mm/s
ZC Freq	37	15	20	Hz
Time (Rel. to Trig)	0.266	0.076	0.283	sec
Peak Acceleration	0.0530	0.0265	0.0398	g
Peak Displacement	0.0119	0.0142	0.0204	mm
Sensorcheck	Passed	Passed	Passed	
Frequency	7.4	7.5	7.2	Hz
Overswing Ratio	3.9	3.6	4.0	

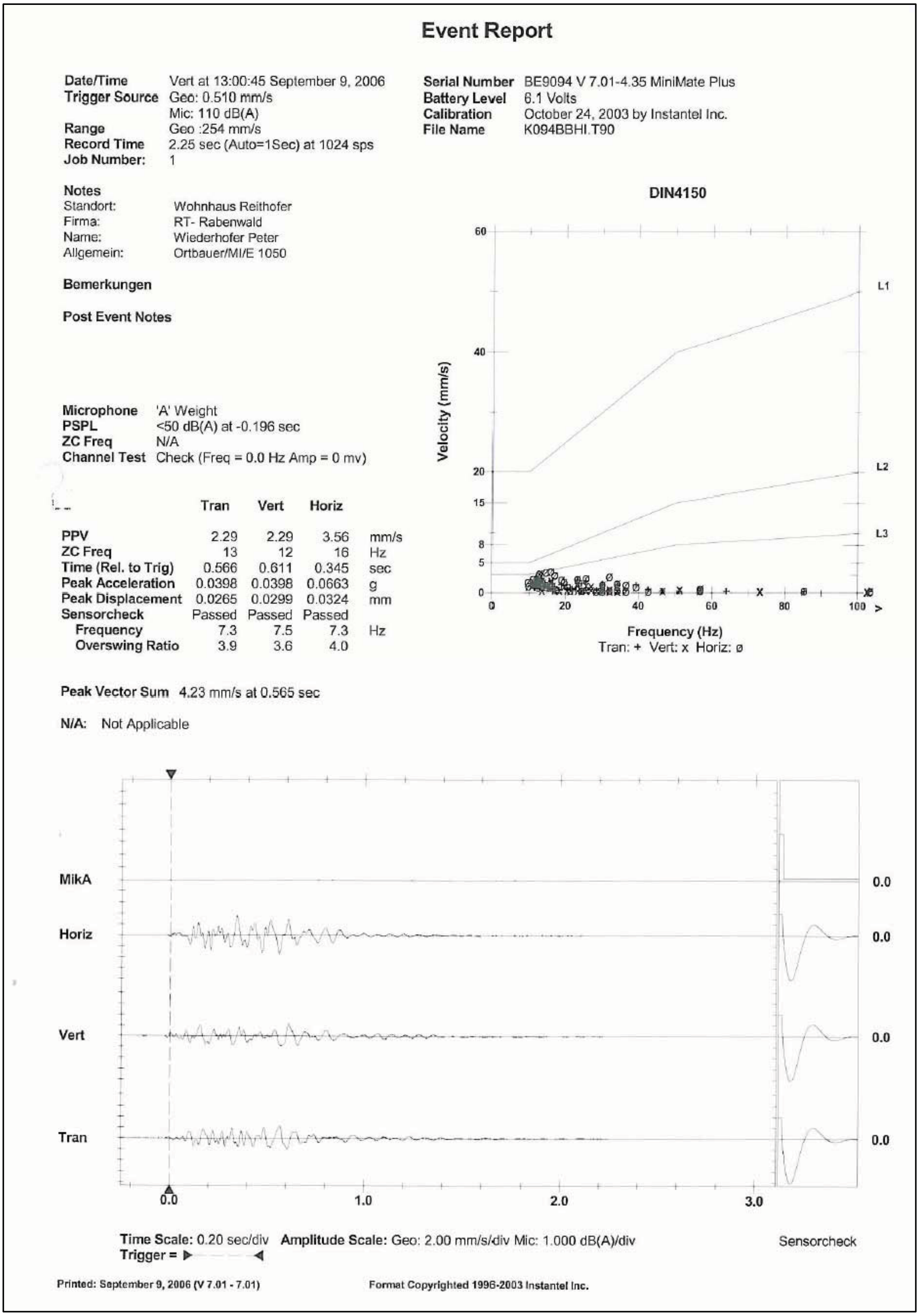
Peak Vector Sum 2.35 mm/s at 0.266 sec

N/A: Not Applicable

Time Scale: 0.20 sec/div **Amplitude Scale:** Geo: 2.00 mm/s/div Mic: 1.000 dB(A)/div
Trigger = ▶———◀

Printed: September 7, 2006 (V 7.01 - 7.01) Format Copyrighted 1996-2003 Instantel Inc.

Figure 73: Vibrations event report from the 4th blast, Source [27]



Event Report

Date/Time	Vert at 13:00:45 September 9, 2006
Trigger Source	Geo: 0.510 mm/s
	Mic: 110 dB(A)
Range	Geo :254 mm/s
Record Time	2.25 sec (Auto=1Sec) at 1024 sps
Job Number:	1

Serial Number	BE9094 V 7.01-4.35 MiniMate Plus
Battery Level	6.1 Volts
Calibration	October 24, 2003 by Instantel Inc.
File Name	K094BBHI.T90

Notes

Standort:	Wohnhaus Reithofer
Firma:	RT- Rabenwald
Name:	Wiederhofer Peter
Allgemein:	Ortbauer/MI/E 1050

Bemerkungen

Post Event Notes

Microphone	'A' Weight
PSPL	<50 dB(A) at -0.196 sec
ZC Freq	N/A
Channel Test	Check (Freq = 0.0 Hz Amp = 0 mv)

	Tran	Vert	Horiz	
PPV	2.29	2.29	3.56	mm/s
ZC Freq	13	12	16	Hz
Time (Rel. to Trig)	0.566	0.611	0.345	sec
Peak Acceleration	0.0398	0.0398	0.0663	g
Peak Displacement	0.0265	0.0299	0.0324	mm
Sensorcheck	Passed	Passed	Passed	
Frequency	7.3	7.5	7.3	Hz
Overswing Ratio	3.9	3.6	4.0	

Peak Vector Sum 4.23 mm/s at 0.565 sec

N/A: Not Applicable

Time Scale: 0.20 sec/div **Amplitude Scale:** Geo: 2.00 mm/s/div Mic: 1.000 dB(A)/div

Trigger = ▶———◀

Sensorcheck

Printed: September 9, 2006 (V 7.01 - 7.01)

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Figure 74: Vibrations event report from the 5th blast, Source [27]

Event Report

Date/Time	Tran at 12:00:56 September 19, 2006	**Serial Number**	BE9094 V 7.01-4.35 MiniMate Plus
Trigger Source	Geo: 0.510 mm/s	**Battery Level**	6.1 Volts
	Mic: 110 dB(A)	**Calibration**	October 24, 2003 by Instantel Inc.
Range	Geo :254 mm/s	**File Name**	K094BBZY.PK0
Record Time	1.25 sec (Auto=1Sec) at 1024 sps		
Job Number:	1		

Notes

Standort:	Wohnhaus Reithofer
Firma:	RT- Rabenwald
Name:	Rossegger Karl
Allgemein:	Ortbauer1085

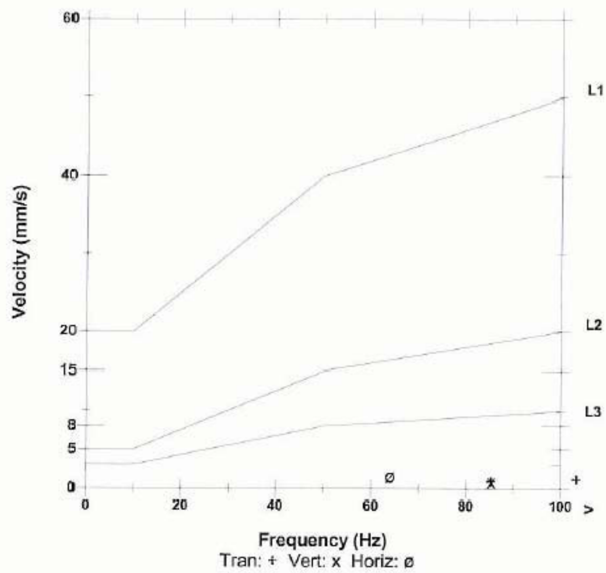
Bemerkungen

Post Event Notes

Microphone	'A' Weight
PSPL	<50 dB(A) at 0.000 sec
ZC Freq	N/A
Channel Test	Check (Freq = 0.0 Hz Amp = 0 mv)

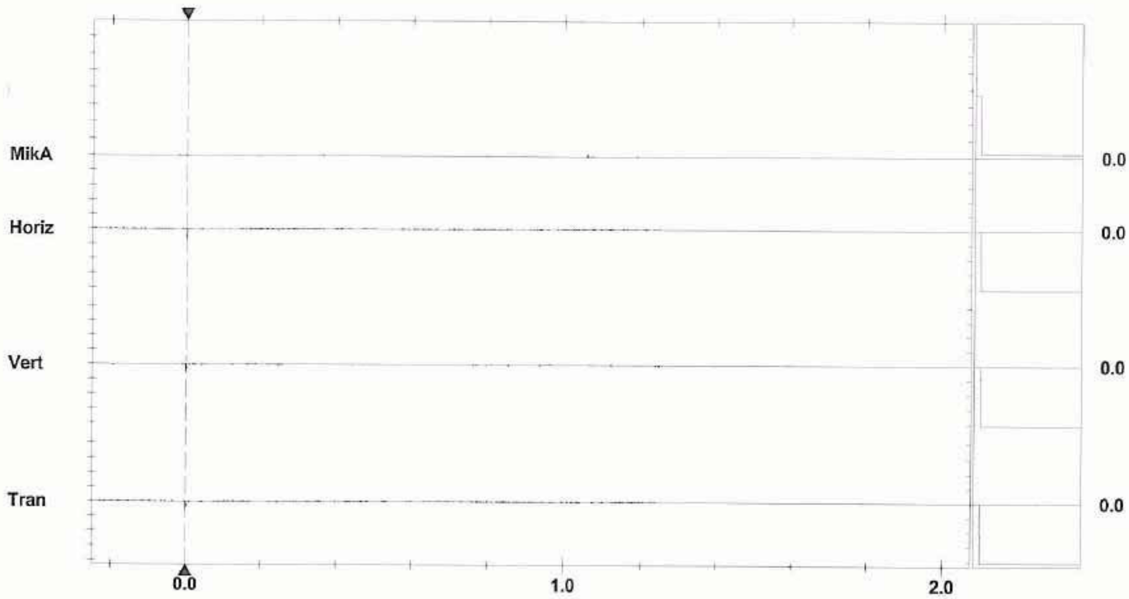
	Tran	Vert	Horiz	
PPV	1.14	0.889	1.52	mm/s
ZC Freq	>100	85	64	Hz
Time (Rel. to Trig)	0.000	0.000	0.000	sec
Peak Acceleration	0.119	0.0795	0.146	g
Peak Displacement	0.00118	0.00130	0.00229	mm
Sensorcheck	Check	Check	Check	
Frequency	1024.0	1024.0	1024.0	Hz
Overswing Ratio	0.0	0.0	0.0	

DIN4150



Peak Vector Sum 2.10 mm/s at 0.000 sec

N/A: Not Applicable



Time Scale: 0.20 sec/div **Amplitude Scale:** Geo: 2.00 mm/s/div Mic: 1.000 dB(A)/div Sensorcheck

Trigger =

Printed: September 20, 2006 (V 7.01 - 7.01) Format Copyrighted 1996-2003 Instantel Inc.

Figure 75: Vibrations event report from the 7th blast, Source [27]

Event Report

Date/Time	Horiz at 13:43:42 September 26, 2006
Trigger Source	Geo: 0.510 mm/s
	Mic: 110 dB(A)
Range	Geo :254 mm/s
Record Time	2.25 sec (Auto=1Sec) at 1024 sps
Job Number:	1

Serial Number	BE9094 V 7.01-4.35 MiniMate Plus
Battery Level	6.2 Volts
Calibration	October 24, 2003 by Instantel Inc.
File Name	K094BCD2.4U0

Notes

Standort:	Wohnhaus Reithofer
Firma:	RT- Rabenwald
Name:	Scheer Christian
Allgemein:	KH E-1016

Bemerkungen

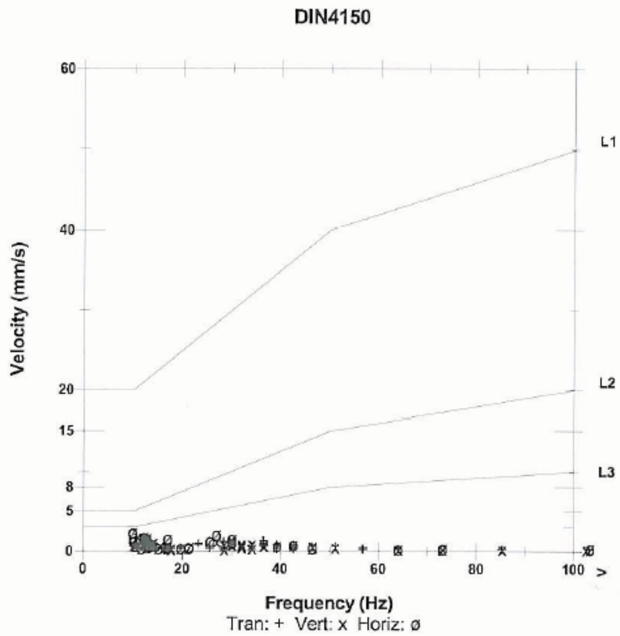
Post Event Notes

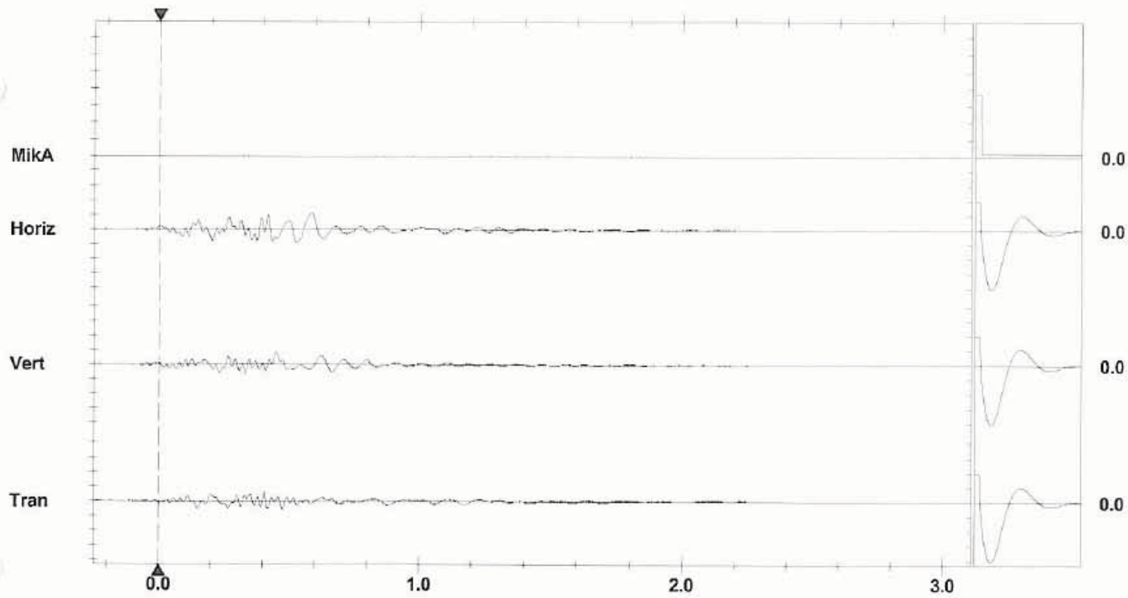
Microphone	'A' Weight
PSPL	<50 dB(A) at 0.001 sec
ZC Freq	N/A
Channel Test	Check (Freq = 0.0 Hz Amp = 0 mv)

	Tran	Vert	Horiz	
PPV	1.27	1.78	2.29	mm/s
ZC Freq	37	11	9.8	Hz
Time (Rel. to Trig)	0.407	0.451	0.581	sec
Peak Acceleration	0.0265	0.0265	0.0398	g
Peak Displacement	0.0120	0.0193	0.0376	mm
Sensorcheck	Passed	Passed	Passed	
Frequency	7.4	7.5	7.3	Hz
Overswing Ratio	3.9	3.6	4.0	

Peak Vector Sum 2.39 mm/s at 0.582 sec

N/A: Not Applicable





Time Scale: 0.20 sec/div **Amplitude Scale:** Geo: 2.00 mm/s/div Mic: 1.000 dB(A)/div Sensorcheck

Trigger = ▶— — — ◀

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Figure 76: Vibrations event report from the 8th blast, Source [27]

APPENDIX C: SURVEYED SHOT PLANS

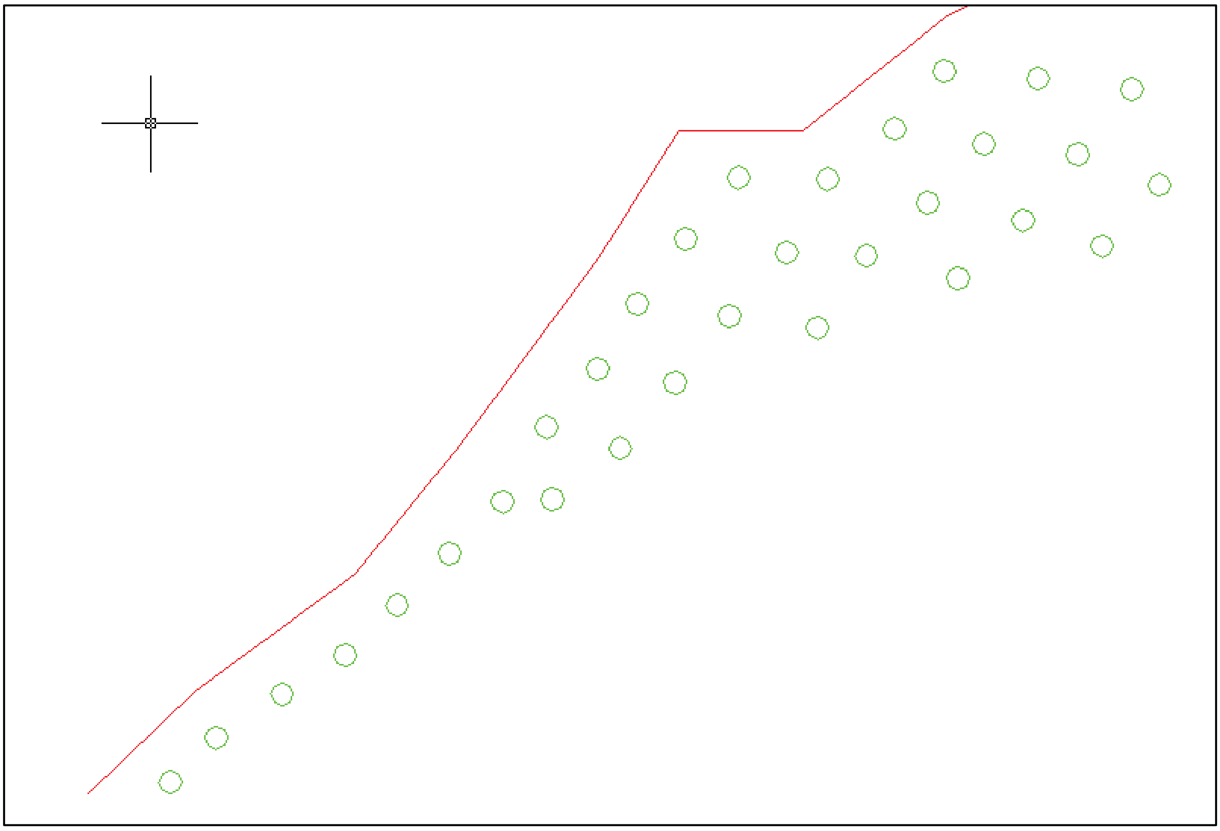

Figure 77: Surveyed shot plan of the borehole starting points from the 3rd blast

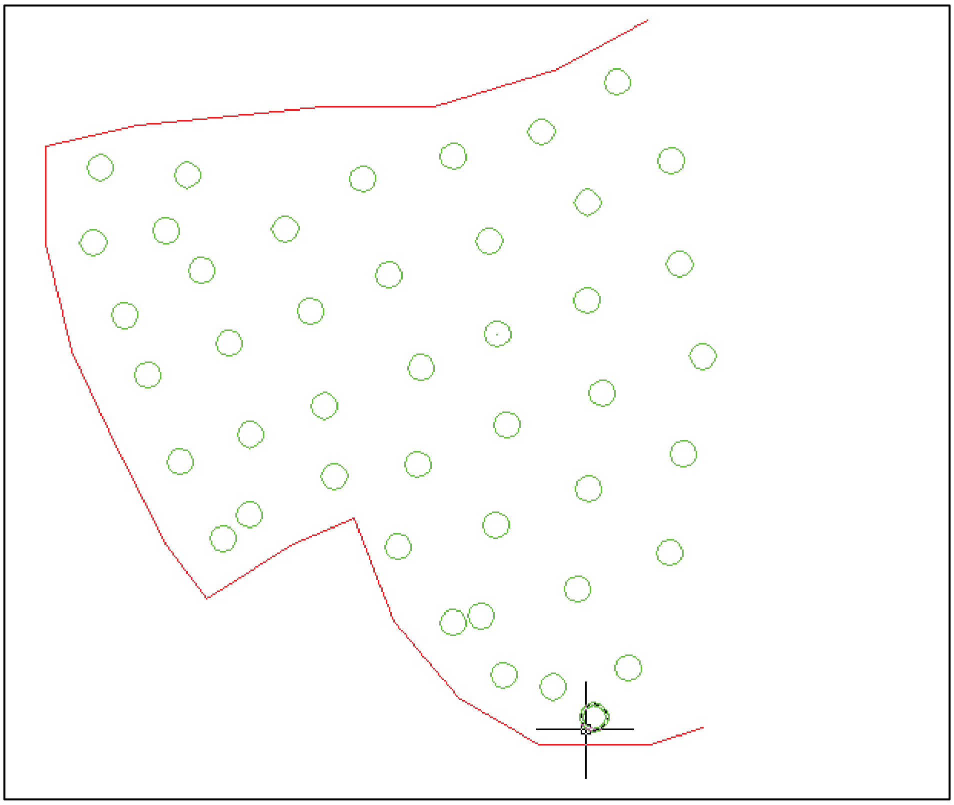

Figure 78: Surveyed shot plan for the borehole starting points from the 4th blast

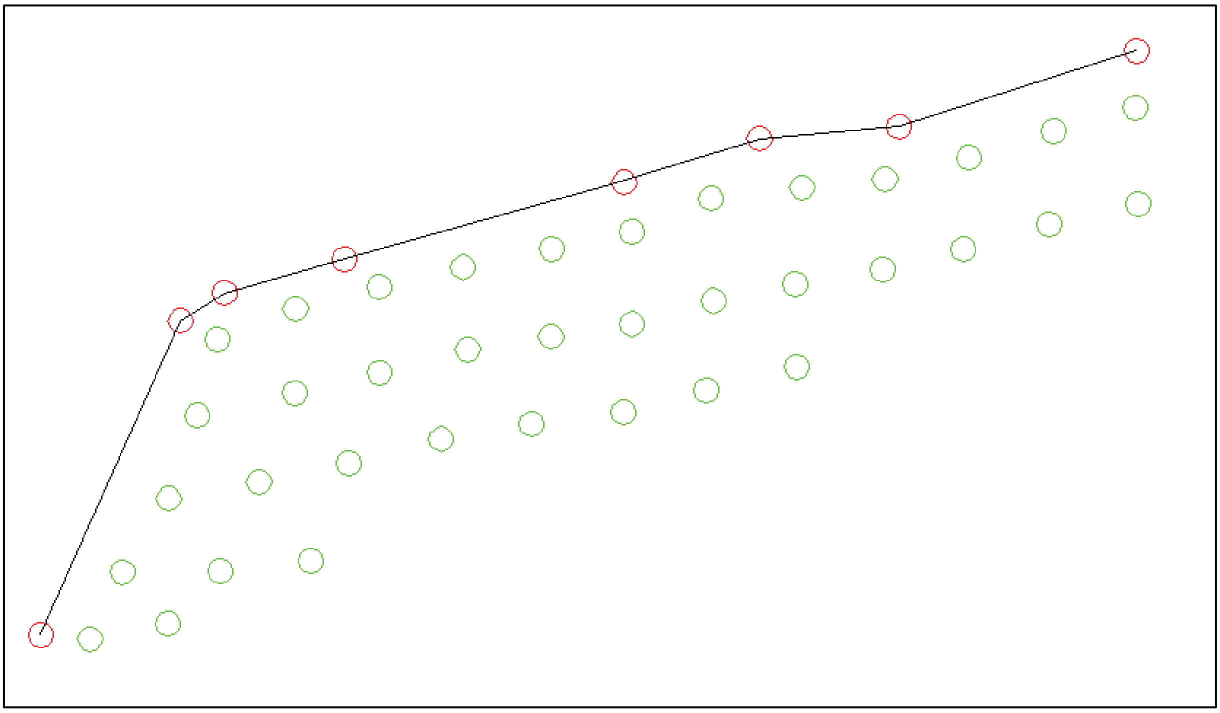

Figure 79: Surveyed shot plan for the borehole starting points from the 5th blast

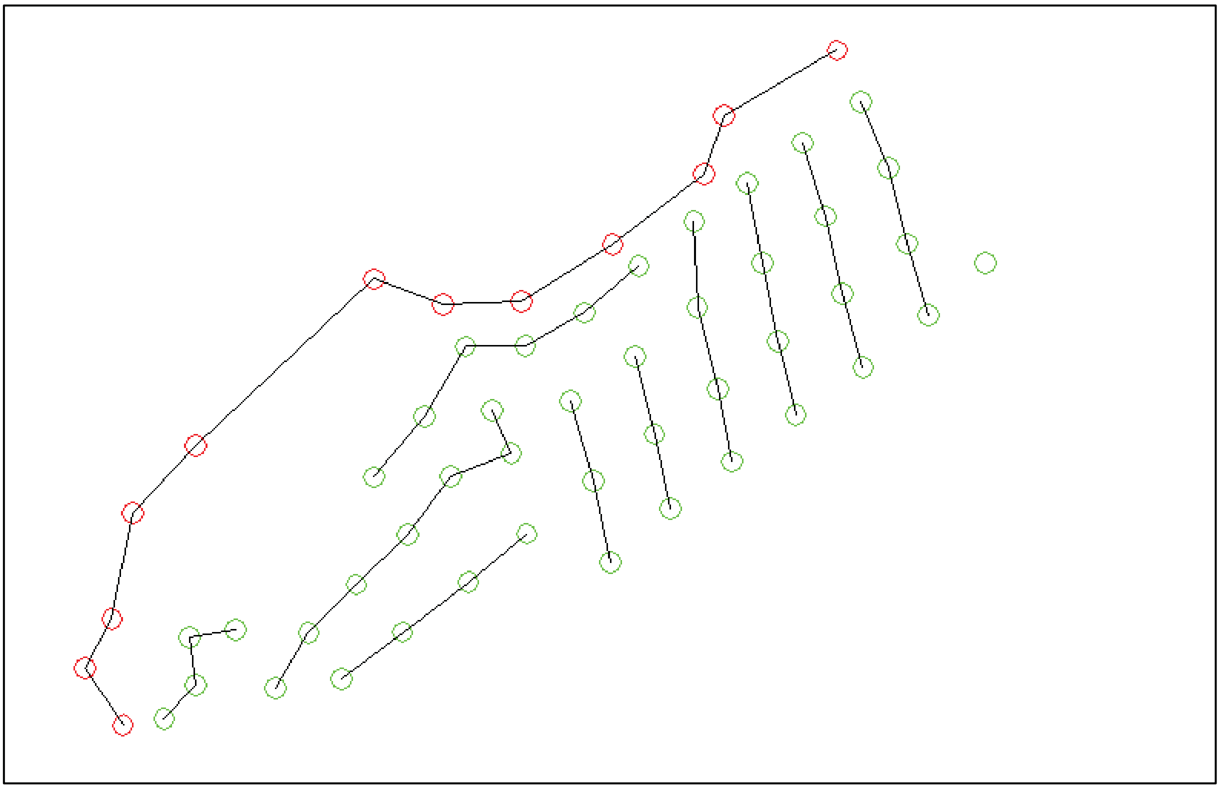

Figure 80: Surveyed shot plan for the borehole starting points from the 6th blast

Figure 81: Surveyed shot plan for the borehole starting points from the 7th blast

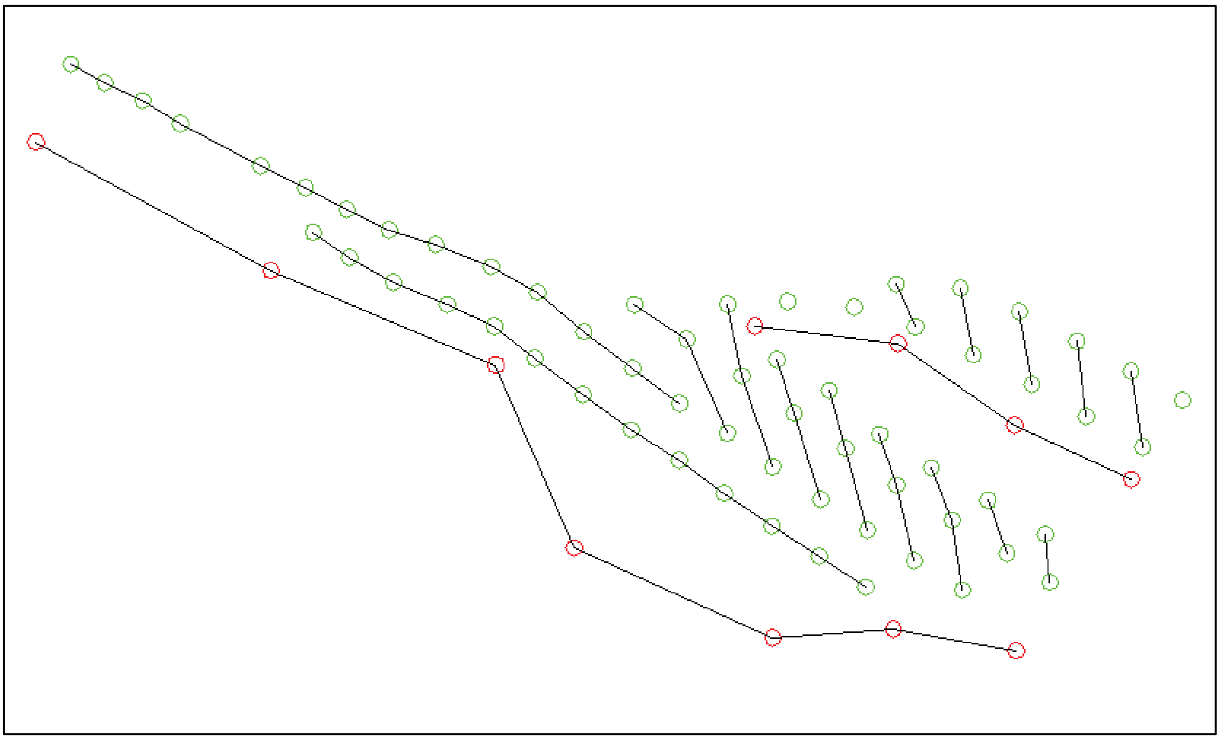

Figure 82: Surveyed shot plan for the borehole starting points from the 8th blast

APPENDIX D: PICTURES OF THE SURVEYED WALLS WITH THE 3G-SOFTWARE

Figure 83: 3G-front-picture from the 5th blast

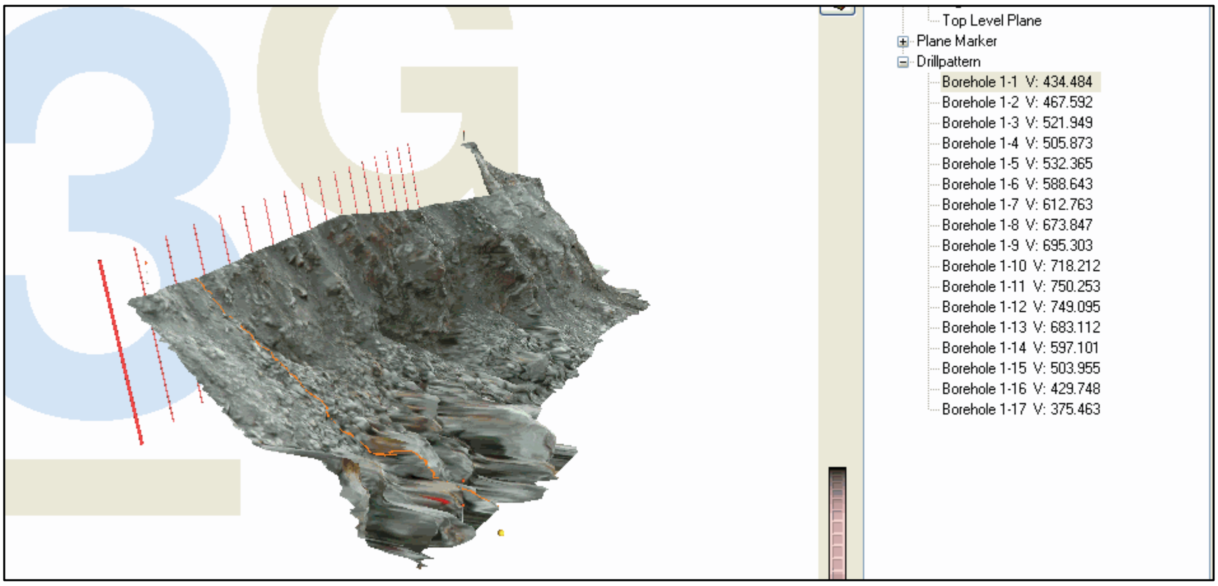

Figure 84: 3G-side-picture left from the 5th blast

Figure 85: 3G-side-picture right from the 5th blast

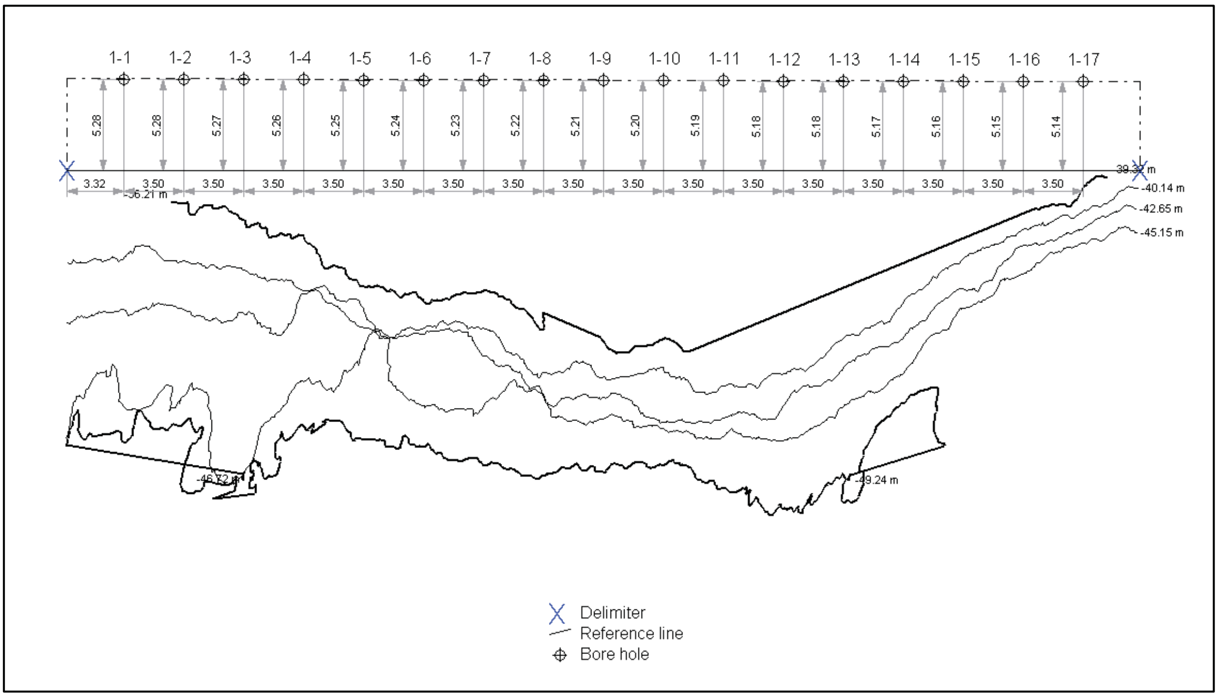

Figure 86: Plan view of the wall from the 5th blast

Figure 87: 3G-front-picture of the 7th blast (left side)

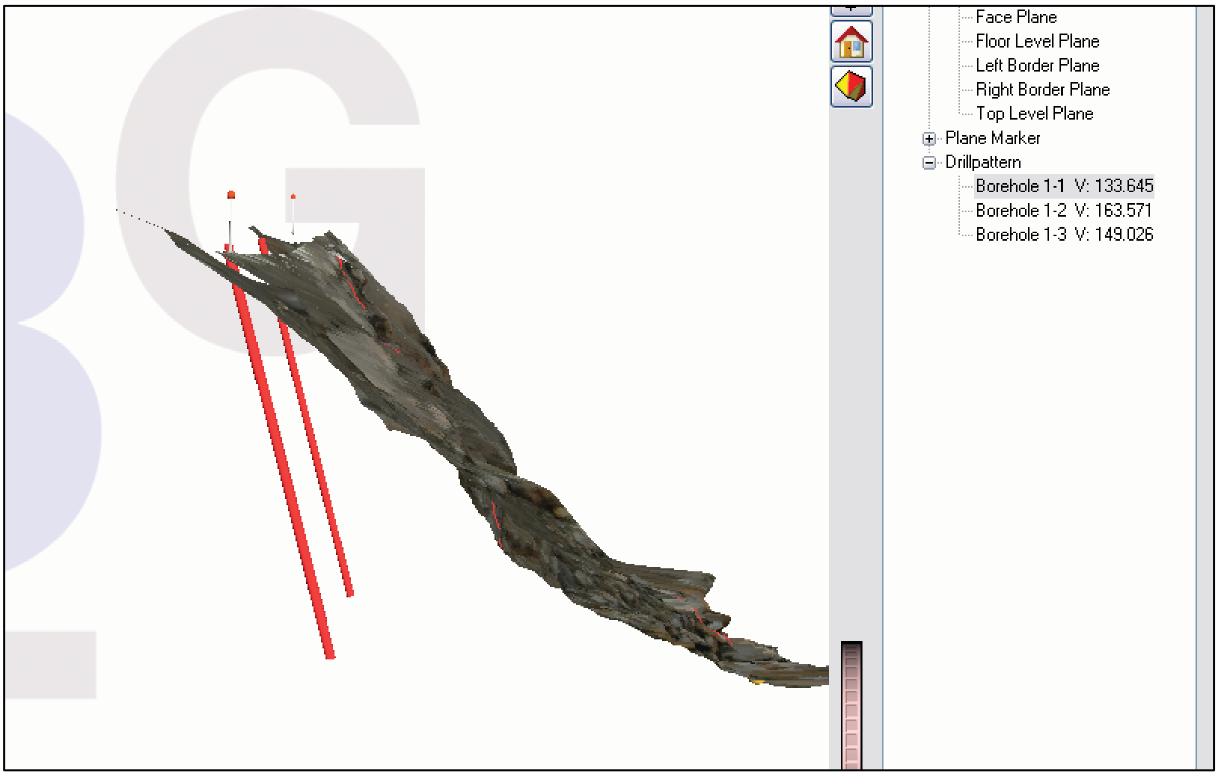

Figure 88: 3G-side-picture of the 7th blast (left side)

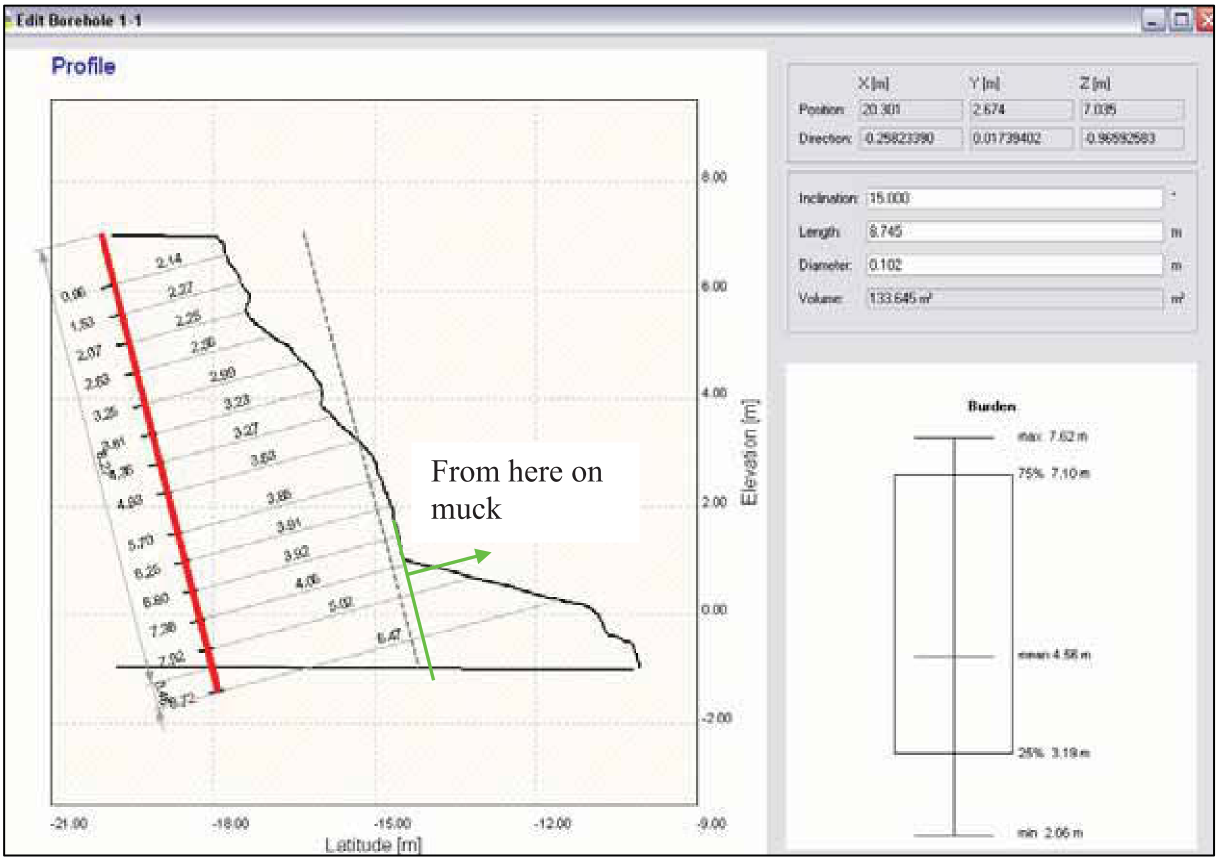

Figure 89: Picture of the first borehole with burdens from the 7th blast (left side)

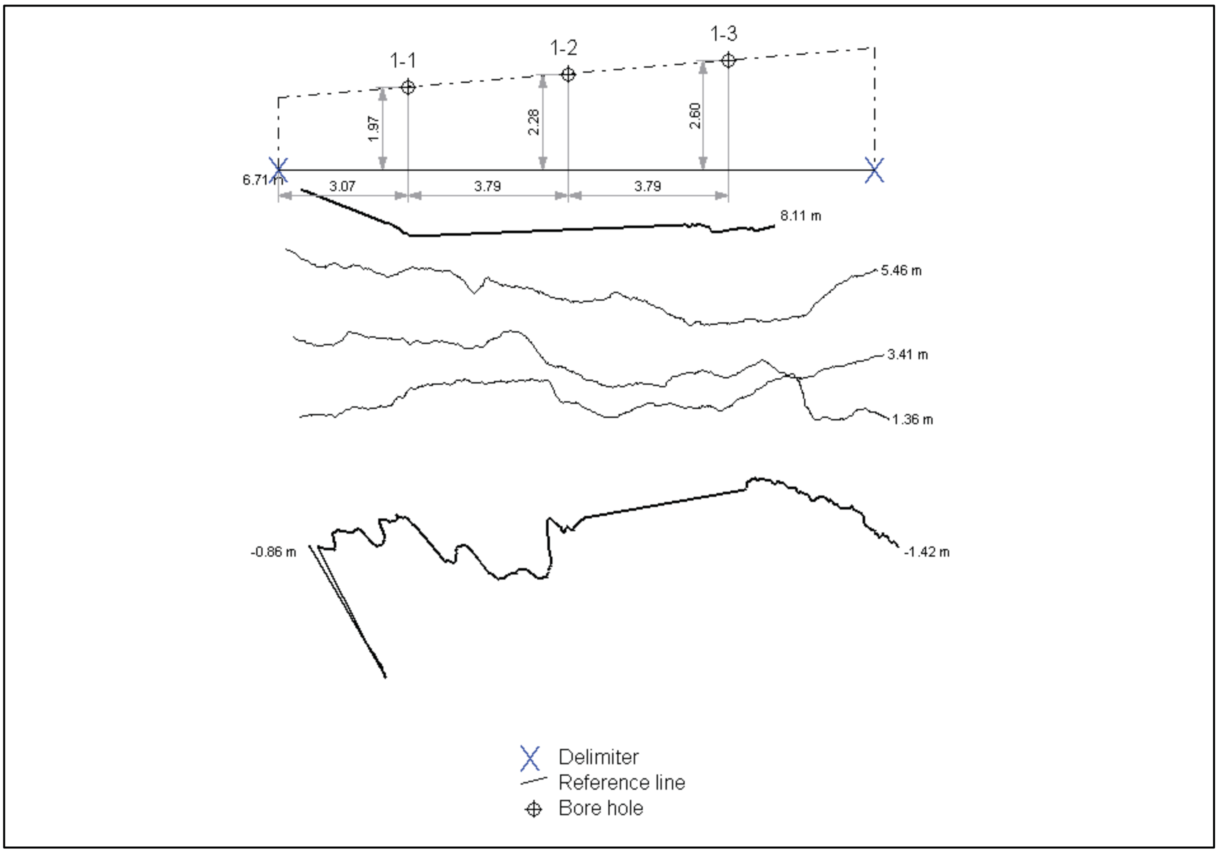

Figure 90: Plan view of the 7th blast (left side)

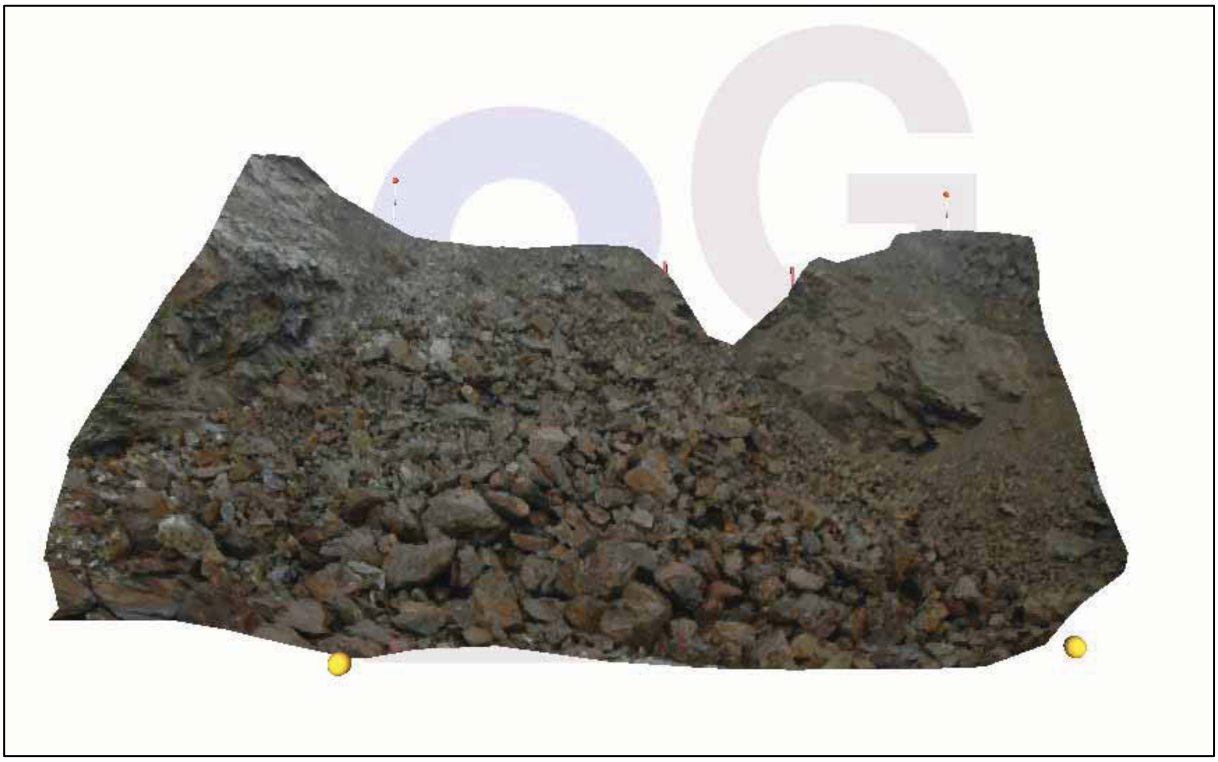

Figure 91: 3G-front-picture of the 7th blast (right side)

Figure 92: 3G-side-picture of the 7th blast (right side)

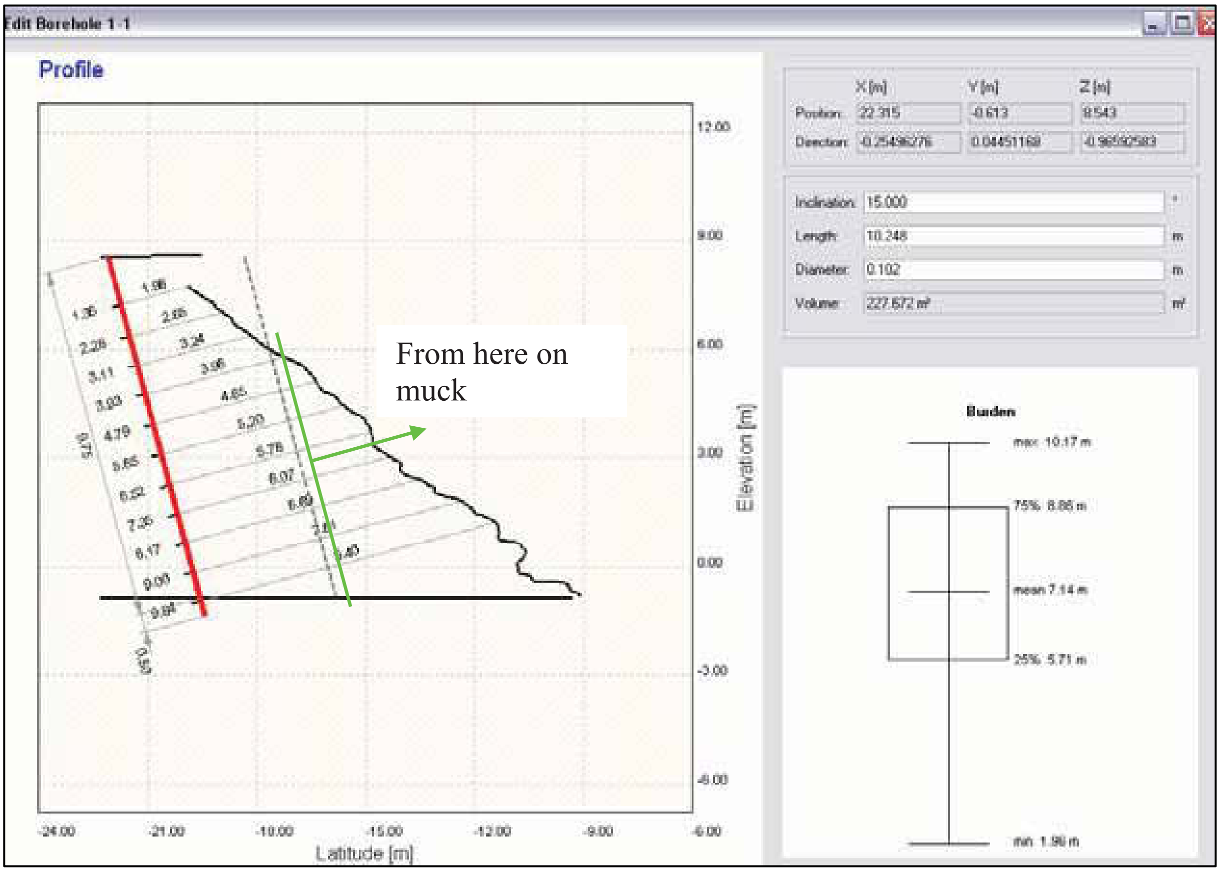

Figure 93: Picture of the first borehole with burdens from the 7th blast (right side)

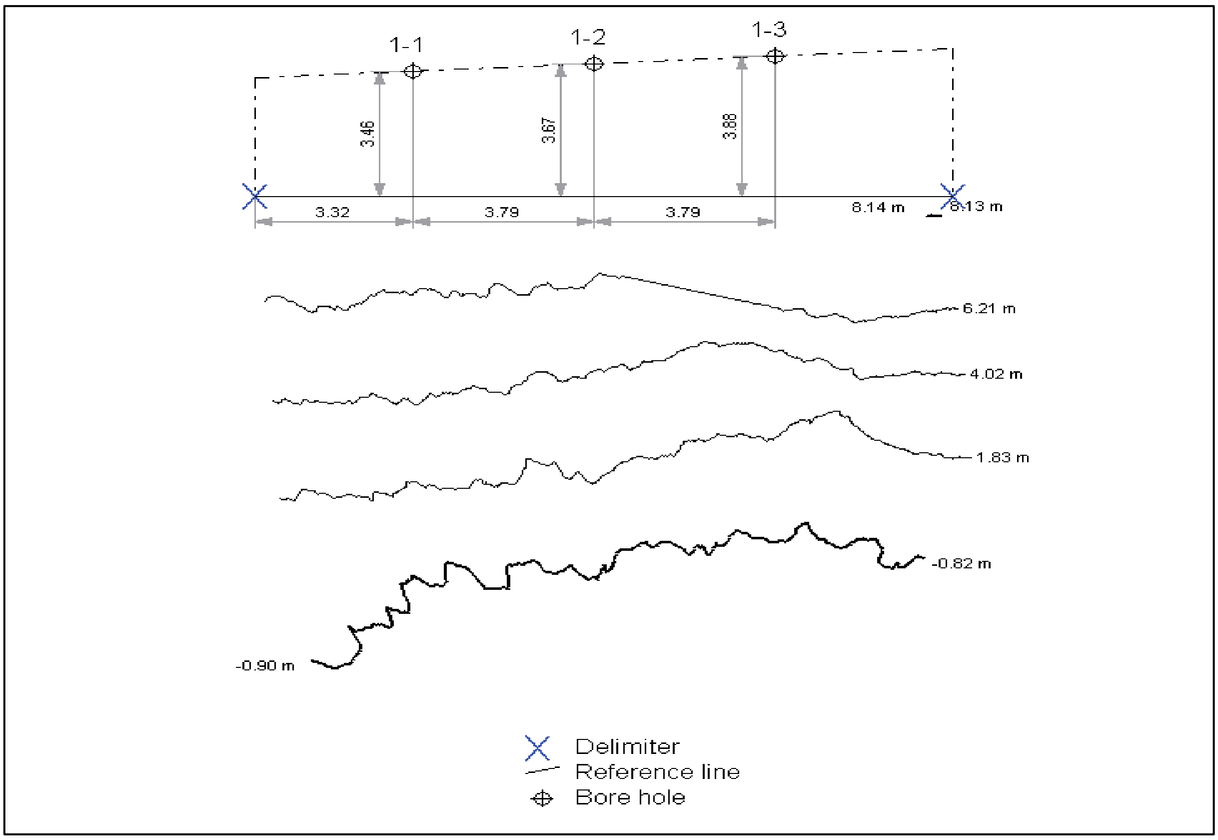

Figure 94: Plan view of the 7th blast (right side)

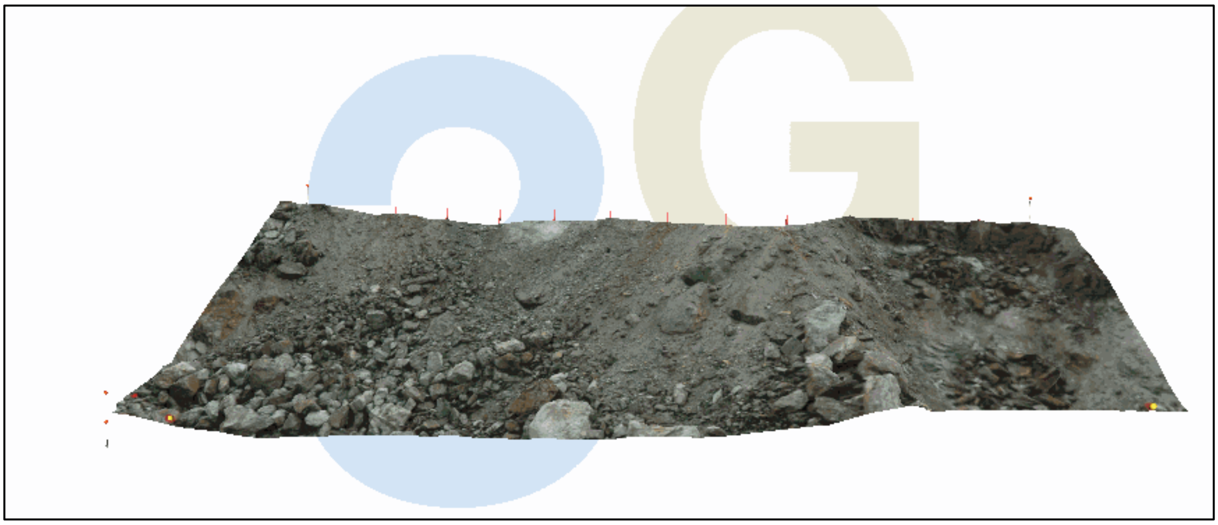

Figure 95: 3G-front-picture of the 9th blast

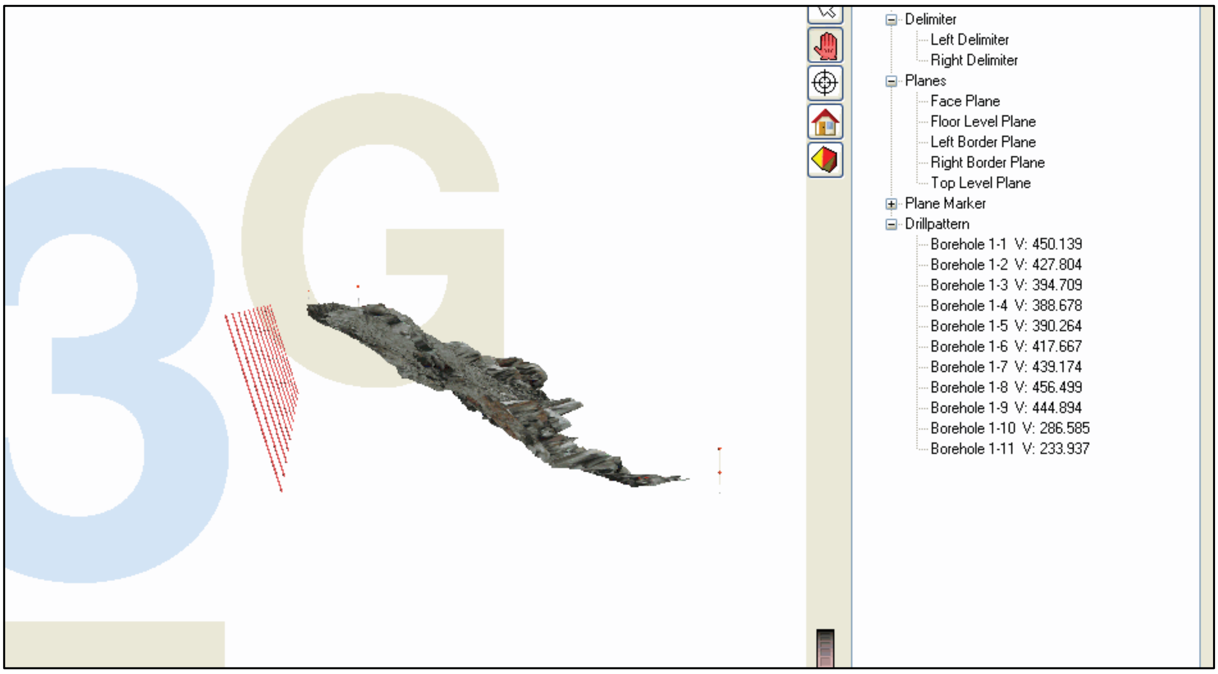

Figure 96: 3G-side-picture from the 9th blast

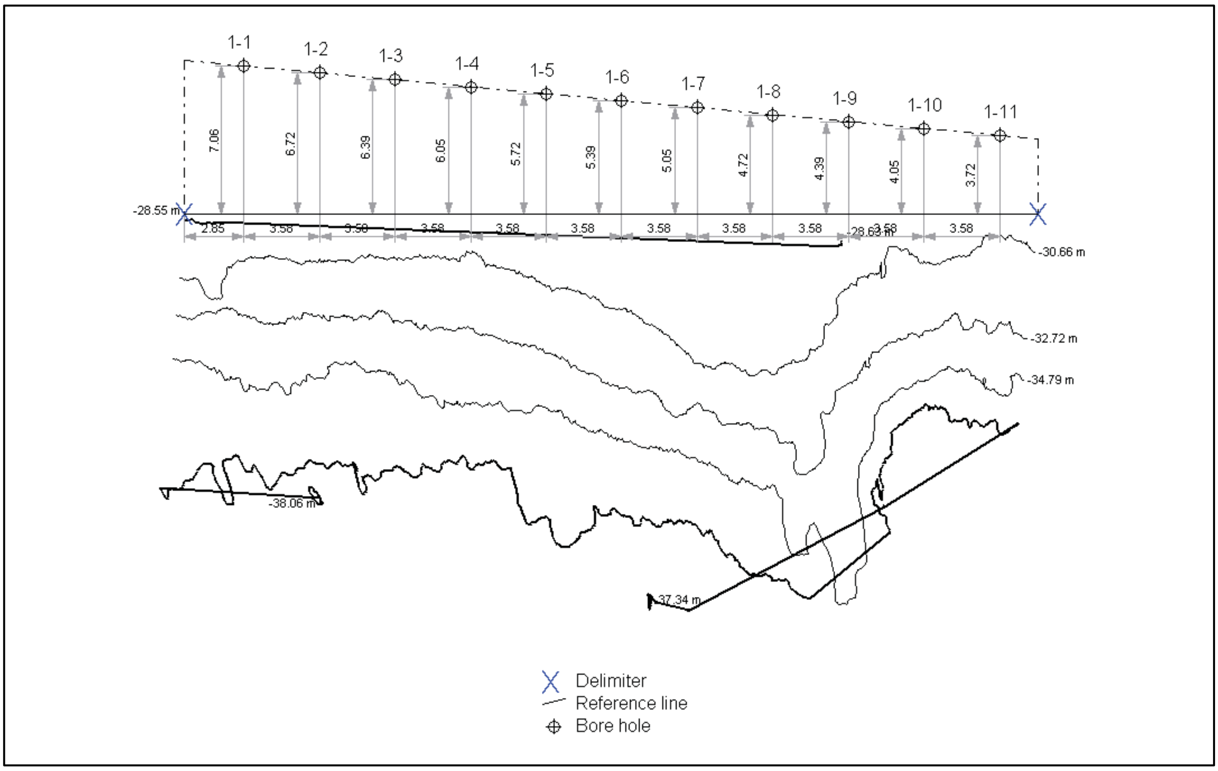

Figure 97: Plan view from the 9th blast

Figure 98: 3G-front-picture of the 10th blast

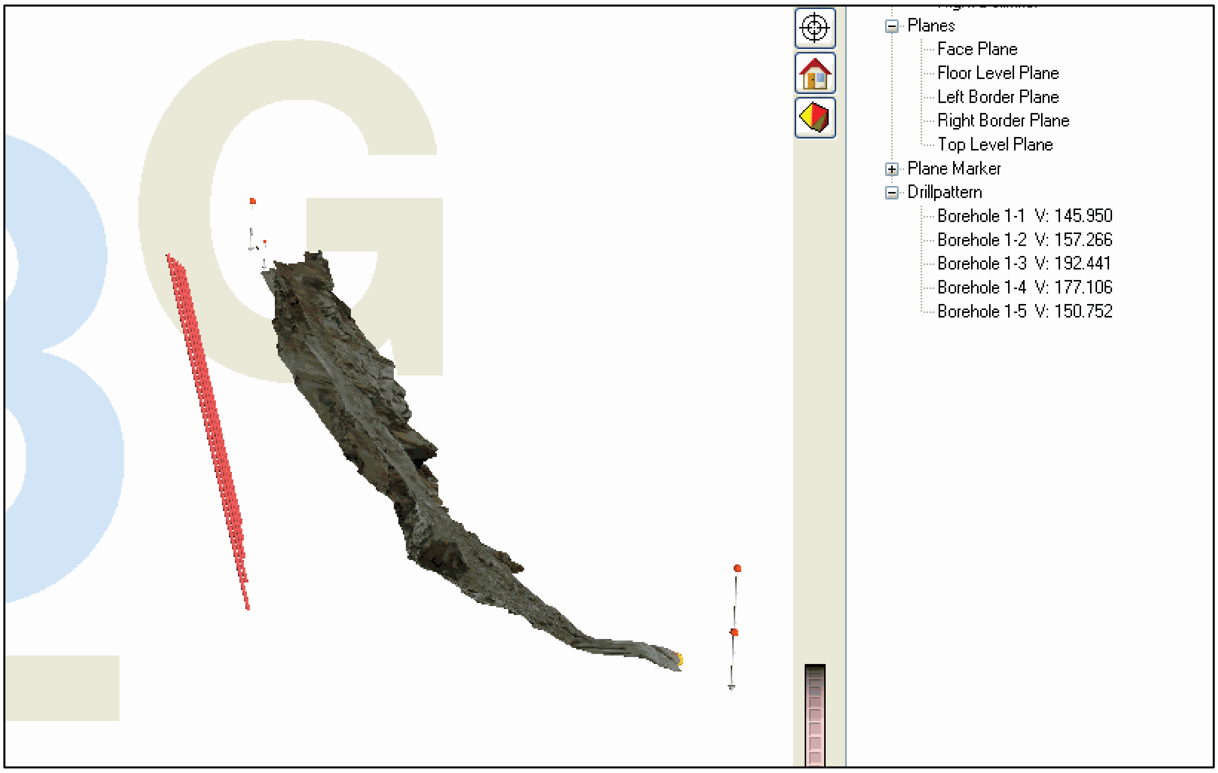

Figure 99: 3G-side-picture of the 10th blast

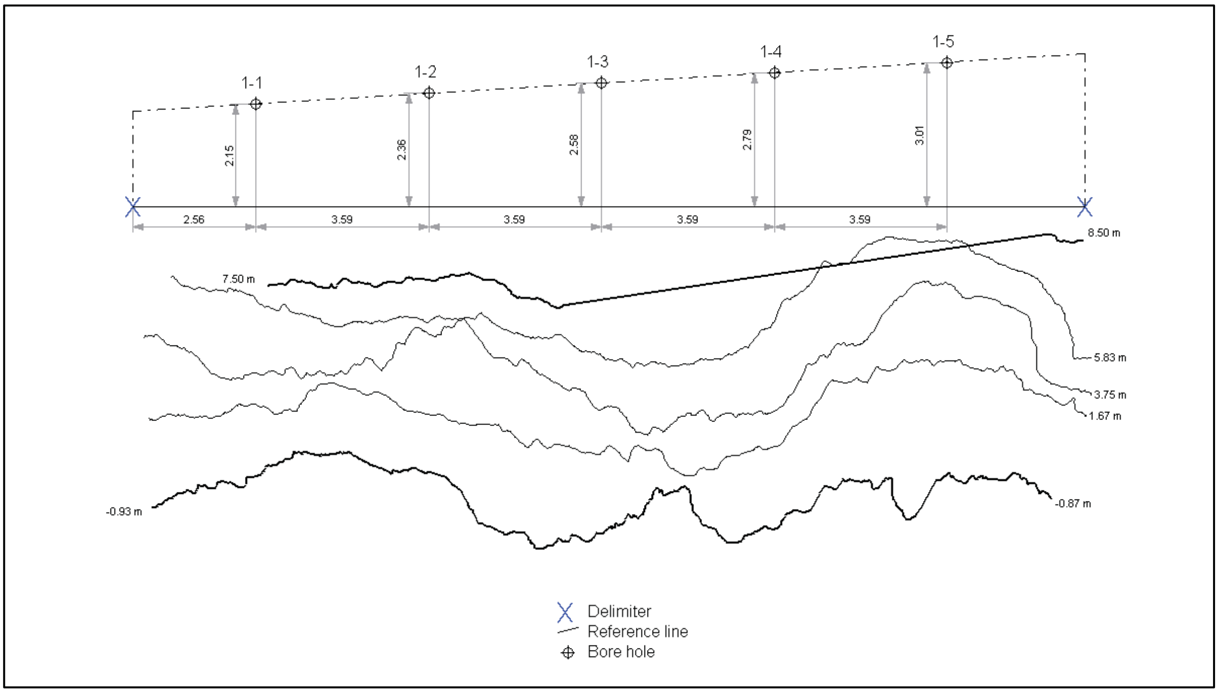

Figure 100: Plan view of the 10th blast

APPENDIX E: DATA SHEETS FROM LOADING THE HOLES

Table 18: Data sheet of the 2nd blast

Blast date 01.09.2006 (12:15)

Bench: 1050

		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	0	3		1,75	4	dill cuttings	
2	1	3		2	4	dill cuttings	
3	2	3		2,25	3,6	dill cuttings	
4	3	3		2	4	dill cuttings	
5	5	3		2,5	3,5	dill cuttings	
6	6	3		2	3,5	dill cuttings	
7	8	3		2	4	dill cuttings	
8	14	4		2,5	3,5	dill cuttings	
9	16	3		2	3,5	dill cuttings	
10	17	3		2	3,5	dill cuttings	
11	19	3		2,25	3,5	dill cuttings	
12	0	3		1,75	3,7	dill cuttings	
13	1	3		2	3,5	dill cuttings	
14	2	3		2	3,5	dill cuttings	
15	4	3		2	3,5	dill cuttings	
16	5	3		2	3,5	dill cuttings	
17	7	3		2	3,5	dill cuttings	water in hole
18	9	3		2	3,5	dill cuttings	water in hole
19	10	3		2,25	3,5	dill cuttings	
20	11	3		1,5	3,5	dill cuttings	
21	12	3		2,25	3,5	dill cuttings	
22	14	3		2,25	3,5	dill cuttings	
23	16	5		1,75	3,5	dill cuttings	
24	18	5		1	3,5	dill cuttings	
25	0	3		2	3,5	dill cuttings	stemming in between
26	2	3		2,5	3,5	dill cuttings	stemming in between
27	3	3		1,75	3,5	dill cuttings	
28	4	3		2	3,5	dill cuttings	stemming in between
29	6	3		2,5	3,5	dill cuttings	stemming in between
30	7	3		2,25	3,5	dill cuttings	stemming in between
31	9	3		1,5	3	dill cuttings	
32	11	3		2,25	3,5	dill cuttings	
33	13	3		2	3,7	dill cuttings	
34	15	3		2,25	4	dill cuttings	
35	17	3		2	3,5	dill cuttings	
36	18	3		1,25	3,5	dill cuttings	
37	1	3		2,25	3,5	dill cuttings	
38	8	3		1,5	3,6	dill cuttings	
39	10	3		1,75	3,5	dill cuttings	
40	12	3		2,25	3,5	dill cuttings	
41	13	3		2	3,5	dill cuttings	
42	15	3		2	3,5	dill cuttings	

43							hole not loaded because blocked
44	19	3		2	3,5	dill cuttings	
45	20	5		2	3,5	dill cuttings	water in hole

Table 19: Data sheet of the 3rd blast

Blast date 5.9.2006 (13:45) Bench:1060							
		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	0		8	1,5	4	drill cuttings	
2	0		8	1,5	4	drill cuttings	
3	0		9,5	1,5	4	drill cuttings	
4	1		4	2,5	3,8	drill cuttings	
5	1		8,5	1,5	3,8	drill cuttings	
6	2		10	1,5	4	drill cuttings	
7	2		9	1,5	3,5	drill cuttings	
8	3		5	2	3,6	drill cuttings	
9	3	1,5	3	2,25	3,8	drill cuttings	
10	4	1,25	4	2	3,8	drill cuttings	
11	4		4	2,5	3,8	drill cuttings	
12	5		4	2,25	3,8	drill cuttings	
13	4		7	1,5	4,5	drill cuttings	
14	5		7	1,75	3,6	drill cuttings	
15	6		4	2,25	4	drill cuttings	
16	6		4	2,25	3,8	drill cuttings	
17	7	0,5			1,7	drill cuttings	
18	7		4	2,25	3,8	drill cuttings	
19	8		4	2,25	3,8	drill cuttings	
20	8		4	2,5	3,8	drill cuttings	
21	9		10		3,8	drill cuttings	
22	9		4	2	3,8	drill cuttings	
23	10		4	2,25	3,8	drill cuttings	
24	10		1	1,5	3,5	drill cuttings	
25	11		4	0,25	3	drill cuttings	
26	11		8	1,25	4	drill cuttings	
27	12		4	2	3,8	drill cuttings	
28	12		3	1,25	3	drill cuttings	
29	13		2	0,5	3	drill cuttings	
30	14		6	1	3,1	drill cuttings	
31	15		3	0,75	4	drill cuttings	

Table 20: Data sheet of the 4th blast

Blast date 7.9.2006 (13:45)

Bench:1095

		Charge			*Stemming*		
No.	**Det. No.**	**Gelatine**	**Emulsion**	**ANFO**	**Height**	**Kind**	**Notes**
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	0	1	2	1,5	3,5	drill cuttings	
2	0	1	2	1,75	3,5	drill cuttings	
3	1	1	2	1,75	3,5	drill cuttings	
4	4	1	2	1,25	3,5	drill cuttings	
5	1	1	2	1,5	3,5	drill cuttings	
6	2	1	2	1,5	3,5	drill cuttings	
7	2	1	2	1,75	3,5	drill cuttings	
8	3	1	2	1,75	3,5	drill cuttings	
9	3	1	2	1,5	3,5	drill cuttings	
10	4	1	2	1,5	3,5	drill cuttings	
11	5	1	2	2,25	3,5	drill cuttings	
12							hole not loaded because of cavity
13	5	1	2	1,5	3,5	drill cuttings	
14	6	2	2	1,25	3,5	drill cuttings	
15	6	1	2	1,75	3,5	drill cuttings	
16	7	1	2	1,25	3,5	drill cuttings	
17	7	1	2	1,75	3,5	drill cuttings	
18	8	3			3,5	drill cuttings	
19							hole not loaded because of cavity
20	8	1	2	1,75	3,5	drill cuttings	
21	9	1	2	1,75	3,5	drill cuttings	
22	9	1	2	1,75	3,3	drill cuttings	
23	10	2	2	1,25	3	drill cuttings	
24	10	4		1,75	3,5	drill cuttings	
25	11	3		1,5	3,5	drill cuttings	
26	11	3		1,75	3,5	drill cuttings	
27	12	1	2	1,5	3,5	drill cuttings	
28	12	1	2	1	>5	drill cuttings	
29	13	1	2	1,5	3,5	drill cuttings	
30	13	1	2	1,5	3,5	drill cuttings	
31	14	3		1,5	3,5	drill cuttings	
32	14	3		1,5	3,5	drill cuttings	
33	15	3		1,75	3,5	drill cuttings	
34	15	1	2	1,5	3,5	drill cuttings	
35	16	1	2	1,5	3,5	drill cuttings	
36	16	1	2	1,5	3,5	drill cuttings	
37	17	1	2	1,5	3	drill cuttings	
38	17	1	2	1,5	3,2	drill cuttings	
39	17	1	2	1	3,5	drill cuttings	
40	18	1	2	1,25	3,5	drill cuttings	
41	18	1	2	1,25	3,5	drill cuttings	
42	19	1	2	1,75	3,5	drill cuttings	

43	19	1	2	1,25	3,5	drill cuttings	
44	20	1	2	1,5	3,5	drill cuttings	
45	20	1	2	1,75	3	drill cuttings	

Table 21: Data sheet of the 5th blast

Blast date 9.9.2006 (13:00)

Bench:1050

		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	2	1	3	1,75	4	drill cuttings	stemming in between
2	4	1	3	2	3,7	drill cuttings	
3	7	1	3	2,25	3,5	drill cuttings	
4	9	1	3	2,5	3,7	drill cuttings	
5	10	1	3	2	3,7	drill cuttings	
6	12	1	3	2	3,5	drill cuttings	
7	13	1	3	1,75	3,5	drill cuttings	
8	15	1	3	2	3,5	drill cuttings	
9	16	1	3	2	4	drill cuttings	
10	17	1	3	1,75	3,3	drill cuttings	
11	18	1	3	2	3,8	drill cuttings	
12	19	1	3	2	3,5	drill cuttings	
13	2	1	3	2,25	3,8	drill cuttings	stemming in between
14	5	1	3	2	3,5	drill cuttings	
15	7	1	3	2	3,5	drill cuttings	
16	9	1	3	1,75	3,5	drill cuttings	
17	11	1	3	2	3,7	drill cuttings	
18	12	1	3	2	3,5	drill cuttings	
19	14	1	3	2	3,5	drill cuttings	
20	15	1	3	2	3,5	drill cuttings	
21	17	1	6	1,5	3,5	drill cuttings	
22	18	1	6	1,5	4	drill cuttings	
23	19	1	5	1,5	4	drill cuttings	
24	20	1	16	0,25	3,5	drill cuttings	
25	3	1	3	1,5	4	drill cuttings	stemming in between
26	5	1	3	2,25	3,5	drill cuttings	
27	8	1	3	2	3,5	drill cuttings	
28	10	1	3	2,25	3,5	drill cuttings	
29	11	1	6	1,5	3,5	drill cuttings	
30	13	1	6	1,5	4	drill cuttings	
31	14	1	6	1,5	3,5	drill cuttings	
32	16	1	15		3,7	drill cuttings	
33	3	1	3	1,75	4	drill cuttings	
34	6	1	7	1,5	3,5	drill cuttings	
35	8	1	6	2	3,8	drill cuttings	
36	4	1	3	1,75	3,5	drill cuttings	
37	6	1	6	1,5	3,5	drill cuttings	

Table 22: Data sheet of the 6th blast

Blast date 15.09.2006 (12:00)

Bench: 1050

		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	0	1	2	1,25	3,5	drill cuttings	
2	0	1	2	1,75	3,5	drill cuttings	
3	0	1	2	1,75	3,5	drill cuttings	
4	1	1	2	1	3,5	drill cuttings	
5	1	1			3,5	drill cuttings	
6	1	2	1		3,5	drill cuttings	
7	2	1	3	1,5	3,5	drill cuttings	
8	1	2	3	1,5	3,5	drill cuttings	
9	2	1	3	2	3,5	drill cuttings	
10	5	1	3	1,75	3,5	drill cuttings	
11	6	1	3	2	3,5	drill cuttings	
12	8	1	3	2	3,5	drill cuttings	
13	10	1	3	2	3,5	drill cuttings	
14	12	1	3	2	3,5	drill cuttings	
15	14	1	3	2	3,5	drill cuttings	
16	4	1	3	2,25	3,5	drill cuttings	
17	3	1	2	2	3,5	drill cuttings	
18	3	1	2	1,25	3,5	drill cuttings	
19	3	1	2	1,75	3,5	drill cuttings	
20	2	1	3	1,75	3,5	drill cuttings	
21	4	1	3	2	3,5	drill cuttings	
22	6	1	3	2	3,5	drill cuttings	
23	7	1	3	2	3,5	drill cuttings	
24	9	1	3	2	3,5	drill cuttings	
25	11	1	3	2	3,5	drill cuttings	
26	13	1	3	2	3,5	drill cuttings	
27	15	1	3	2	3,5	drill cuttings	
28	5	1	3	1,25	3,5	drill cuttings	
29	4	1	3	3	3,5	drill cuttings	
30	5	1	4	2	3,5	drill cuttings	
31	6	1	3	2,25	3,5	drill cuttings	
32	7	1	4	2,25	3,5	drill cuttings	
33	7	1	3	2,25	3,5	drill cuttings	
34	9	1	3	2,5	3,5	drill cuttings	
35	11	1	3	2	3,5	drill cuttings	
36	13	1	4	2	3,5	drill cuttings	
37	15	1	3	2	3,5	drill cuttings	
38	8	1	3	1,5	3	drill cuttings	stemming in between
39	10	1	3	1,75	3	drill cuttings	stemming in between
40	12	1	3	1,5	3	drill cuttings	stemming in between
41	14	1	3	1,5	3	drill cuttings	stemming in between
42	16	1	3	2	3	drill cuttings	stemming in between

43	17	1	3	1,25	3	drill cuttings	stemming in between
44	17	1	3	1,75	3	drill cuttings	stemming in between

Table 23: Data sheet of the 7th blast

Blast date 19.09.2006 (12:00)

Bench: 1085

		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	0	4		2	3,8	drill cuttings	
2	0	4		2	3,5	drill cuttings	
3	1	4		2	3,8	drill cuttings	
4	1	4		1,75	3,5	drill cuttings	
5	3	4	3	1,5	3,8	drill cuttings	water in hole
6	4	4		2	3,8	drill cuttings	
7	5	4	7	0,5	4	drill cuttings	water in hole
8	7	4	2	1,5	3,8	drill cuttings	water in hole
9	8	4	8	0,75	3,8	drill cuttings	water in hole
10	2	4		2	3,8	drill cuttings	
11	4	4		1,5	4,3	drill cuttings	
12	5	4		2,5	4	drill cuttings	
13	6	4		1,5	4	drill cuttings	
14	8	4	6	1	4	drill cuttings	water in hole
15	9	4	2	1,5	3,8	drill cuttings	water in hole
16	2	4	2	1,5	4	drill cuttings	water in hole
17	3	4	4	1,25	4	drill cuttings	water in hole
18	5	4	2	1,25	4,2	drill cuttings	water in hole
19	6	4		2,25	3,8	drill cuttings	
20	7	4	2,25	2,25	3,8	drill cuttings	water in hole
21	9	4	2	1,25	3,8	drill cuttings	water in hole
22	9	4	7	0,5	3,8	drill cuttings	water in hole
23	4	4		1	3,8	drill cuttings	
24	6	4		2	4	drill cuttings	
25	7	4		0,25	2	drill cuttings	hole got blocked after cartridges were loaded
26	8	4		1,5	4,5	drill cuttings	
27	10	4		2	4	drill cuttings	
28	11	4		2	4,2	drill cuttings	
29	12	4		3	5	drill cuttings	
30	13	4	2	2	5	drill cuttings	
31	13	4	6	2	3,5	drill cuttings	
32	14	4		2	4	drill cuttings	
33	15	4	2	2	3,5	drill cuttings	
34	16	4		2	4	drill cuttings	
35	13	4		1,25	4,2	drill cuttings	
36	14	4		1,5	4,2	drill cuttings	
37	15	4		1,5	4	drill cuttings	
38	17	11	2	0,25	4	drill cuttings	
39	10	4		2,25	3,8	drill cuttings	
40	11	4		2	4	drill cuttings	
41	12	10	3		4	drill cuttings	

42	12	10	3		3,5	drill cuttings	
43	14	10	3		4	drill cuttings	
44	10	4	2	1	4	drill cuttings	
45	11	4	2	1,5	4	drill cuttings	
46	11	10	3		4	drill cuttings	
47	15	4	3	0,75	4	drill cuttings	

Table 24: Data sheet of the 8th blast

Blast date 26.09.2006 (13:45)

Bench: 1050 new south part of the mine

		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	13	2,5		1,25	3,5	drill cuttings	
2	12	2,5		1,25	3,5	drill cuttings	
3	9	2,5		1,5	3,5	drill cuttings	
4	8	2,5		1,25	3,5	drill cuttings	
5	7	2,5		1,25	3,5	drill cuttings	
6	6	2,5		1,25	3,5	drill cuttings	
7	5	2,5		1,5	3,5	drill cuttings	
8	4	2,5		1,25	3,5	drill cuttings	
9	4	2,5		1,25	3,5	drill cuttings	
10	3	2,5		1,25	3,5	drill cuttings	
11	14	2,5		1,5	3,5	drill cuttings	
12	13	2,5		1,25	3,5	drill cuttings	
13	12	2,5		1,25	3,5	drill cuttings	
14	10	2,5		1	3,5	drill cuttings	
15	8	2,5		1,25	3,5	drill cuttings	
16	7	2,5		1,25	3,5	drill cuttings	
17	6	3,5		1,25	3,5	drill cuttings	
18	5	2,5		1	3,5	drill cuttings	
19	5	2,5		1	3,5	drill cuttings	
20	15	2,5		1	3,5	drill cuttings	
21	14	2,5		1,25	3,5	drill cuttings	
22	12	2,5		1,5	3,5	drill cuttings	
23	10	2,5		1,5	3,5	drill cuttings	
24	9	2,5		1,5	3,5	drill cuttings	
25	8	2,5		1,5	3,5	drill cuttings	
26	6	2		1,75	3,5	drill cuttings	
27	9	3		2	3,5	drill cuttings	
28	10	4		2,5	3,9	drill cuttings	
29	13	4		2,5	3,9	drill cuttings	
30	14	4		2,5	3,9	drill cuttings	
31	15	4		2,5	3,9	drill cuttings	
32	16	4		2,5	3,9	drill cuttings	
33	16	4		2,5	3,9	drill cuttings	
34	17	4		2,5	3,9	drill cuttings	
35	17	4		2,5	3,9	drill cuttings	
36	16	4		2,5	3,9	drill cuttings	
37	15	4		2,5	3,9	drill cuttings	
38	14	4		2,5	3,9	drill cuttings	
39	13	4		2,5	3,9	drill cuttings	
40	3	2,5		1,25	3,3	drill cuttings	
41	2	2,5		1,25	3	drill cuttings	
42	2	2		1,25	3	drill cuttings	

43	1	2		1	3	drill cuttings	
44	1	2		0,75	3	drill cuttings	
45	1	2		0,75	2,8	drill cuttings	
46	0	1		0,75	2,8	drill cuttings	
47	0	1		0,5	2,8	drill cuttings	
48	0	1		0,5	2,8	drill cuttings	
49	0	1		0,125	2,8	drill cuttings	
50	0	1		0,125	2,5	drill cuttings	
51						drill cuttings	does not exist
52						drill cuttings	hole blocked
53	1	2		1	2,8	drill cuttings	
54	1	2		1	2,8	drill cuttings	
55	2	2		1	3	drill cuttings	
56	2	2,5		1,25	3,5	drill cuttings	
57	2	2,5		1,5	3,3	drill cuttings	
58	3	2,5		1,25	3,5	drill cuttings	
59	3	2,5		1,25	3,5	drill cuttings	
60	4	2,5		1,25	3,5	drill cuttings	
61	4	2,5		1,25	3	drill cuttings	
62	5	2,5		1,25	3	drill cuttings	
63	5	2,5		1,5	3	drill cuttings	
64	7	2,5		1	3,5	drill cuttings	

Table 25: Data sheet of the 9th blast

Blast date 28.09.2006 (13:45)

Bench: 1040

		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	17	4		2,5	3,5	drill cuttings	
2	15	4		2	3,5	drill cuttings	
3	13	4		2,25	3,5	drill cuttings	
4	11	4		2,5	3,5	drill cuttings	
5	9	4,5		2,75	3,5	drill cuttings	
6	10	4		2,5	3,5	drill cuttings	
7	8	11		3,25	3,5	drill cuttings	
8	7	4		2,75	3,5	drill cuttings	
9	4	4		2,25	3,5	drill cuttings	
10	5	4		2,25	3,5	drill cuttings	
11	5	2	6		3,5	drill cuttings	
12	4	2	6		3,5	drill cuttings	
13	6	4		2,75	3,5	drill cuttings	
14	7	14	2	2,75	3,5	drill cuttings	
15	9	7	8	2,75	3,5	drill cuttings	
16	11	4		2,25	3,5	drill cuttings	
17	13	4		2,25	3,5	drill cuttings	
18	15	4		2	3,5	drill cuttings	
19	16	4		2	3,5	drill cuttings	
20	14	4		2,25	3,5	drill cuttings	
21	12	4		2	3,5	drill cuttings	
22	10	4		2,5	3,5	drill cuttings	
23	8	5		1,5	3,5	drill cuttings	cavity, therefore stemming in between
24	6	2	6	2,25	3,5	drill cuttings	
25	12	4		2,25	3,5	drill cuttings	
26	14	4		2,25	3,5	drill cuttings	
27	16	4		2,25	3,5	drill cuttings	
28	17	4		2,25	3,5	drill cuttings	

Table 26: Data sheet of the 10th blast

Blast date 05.10.2006 (13:45)

Bench: 1050

		Charge			Stemming		
No.	Det. No.	Gelatine	Emulsion	ANFO	Height	Kind	Notes
		(Cartridges)	(Cartridges)	(Bag)	(m)		
1	0	2	2	2	3,5	drill cuttings	
2	0	2	2	2	3,5	drill cuttings	
3	0	2	2	2,25	3,5	drill cuttings	
4	1	2	3	2,25	3,5	drill cuttings	
5	1	2	2	2	3,5	drill cuttings	
6	1	2	2	1,5	3,3	drill cuttings	
7							hole blocked, not charged
8	2	2	2	2,25	3,3	drill cuttings	
9	2	2	2	1,75	3,3	drill cuttings	
10	2	2	2	2,25	3,5	drill cuttings	
11	3	2	2	2,25	3,5	drill cuttings	
12	3	2	2	1,75	3,5	drill cuttings	
13	3	2	4	1,25	3,5	drill cuttings	
14	4	2	2	2	3,5	drill cuttings	
15	4	2	2	1	3,5	drill cuttings	
16	4	2	2	2	3,5	drill cuttings	
17	5	2	2	1,75	2,8	drill cuttings	
18	5	2	2	2,5	3,5	drill cuttings	
19	5	2	2	1,25	3	drill cuttings	
20	6	2	4		3,5	drill cuttings	
21	6	2	1	1	3	drill cuttings	
22	6	1	2	1	3	drill cuttings	
23	4	2	2	1,75	3,5	drill cuttings	stemming in between
24	5	2	2	2	3,5	drill cuttings	
25	7	2	4	1,75	3,5	drill cuttings	
26	7	2	2	2,25	3,5	drill cuttings	
27	10	2	2	1,75	3,5	drill cuttings	stemming in between
28	10	2	2	2	3,5	drill cuttings	
29	11	2	2	2,25	3,5	drill cuttings	
30	11	2	2	0,75	3,5	drill cuttings	
31	12	2	2	1,5	3,5	drill cuttings	stemming in between
32	12	2	2	2	3,5	drill cuttings	
33	13	2	2	0,75	2	drill cuttings	
34	13	2	2	1,5	3,5	drill cuttings	stemming in between
35	14	2	2	1,5	3,5	drill cuttings	
36	15	2	9		3	drill cuttings	water in hole
37	7	2	10		3,5	drill cuttings	water in hole
38						drill cuttings	hole blocked, not charged
39	8	2	2	1	3	drill cuttings	
40	8	1	1	1	3	drill cuttings	
41	8	1	1	1,25	3	drill cuttings	
42	9	1		0,5	2,5	drill cuttings	

43	9	1		0,75	2,5	drill cuttings	
44	9	1		0,75	2,5	drill cuttings	