Drill, blast, load and haul optimisation of the overburden removal at LO Trimouns

Master thesis written by

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Abstract

Trimouns (Rio Tinto Minerals) in France the fragmentation of waste material via drilling, blasting, loading and hauling is necessary to allow the extraction of the talc ore. Via on-site measurements, calculations and their evaluation assisted by observation, questions concerning drilling and blasting (pattern, blast size, documentation), loading and hauling (load ability, number of haulage trucks), and auxiliary equipment (use of dozer) are answered. This results in proposals for process optimization like geology dependent drill and blast pattern with vertical holes, increased blast size, reduced number of haulage trucks and the minimized use of the dozer besides general improvements for influencing activities.

Kurzfassung

Am Standort *Trimouns* (Rio Tinto Minerals) in Frankreich ist die Zerkleinerung von Abraum mittels Bohren und Sprengen und anschließendem Laden und Transportieren nötig, um die Extraktion von Talk zu ermöglichen. Mittels Auswertung von Feldmessungen und Berechnungen konnten Fragestellungen bezüglich Bohren und Sprengen (Geometrie und Volumen pro Sprengung, Dokumentation), Laden und Transportieren (Ladbarkeit, Anzahl der Ladeeinheiten) und dem Einsatz von Hilfsgeräten (Einsatz von Schürfraupen) beantwortet werden. Auf deren Basis konnten Vorschläge zur Prozessoptimierung – von Geologie abhängige Bohr- und Sprenggeometrie, ausschließliche Verwendung vertikaler Bohrlöcher, erhöhtes Volumen pro Sprengungen, reduzierte Anzahl eingesetzter Ladeeinheiten und verminderter Verwendung von Bulldozern – und deren Hilfsprozesse gemacht werden.

Declaration of authorship

Hereby the author of this work affirms that the present thesis was prepared independently without any inadmissible help by a third party. Texts, illustrations and / or ideas taken directly or indirectly from other sources (including electronic resources), quoted verbatim or paraphrased, have without exception been acknowledged and have been referenced in accord.

Leoben, 01. May 2010

Place and date

Signed

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1 Introduction and objectives

Waste removal – especially the fragmentation via drilling and blasting, its loading and hauling and the use of auxiliary equipment – is the main mining process besides talc extraction at the Luzenac Operation *Trimouns* in France. Main target of this master thesis is on the one hand the evaluation of actual work practice and on the other measurements and calculations for process optimisation and improvement.

Following topics are determined via evaluation of actual data provided by the company and measured on-site and comparison of their advantages and disadvantages:

- Change from inclined to vertical blast holes
- Increase of the drill and blast pattern (burden and spacing)
- Increase of the blast size (holes and cubes per blast)
- Introduction of a systematic drill and blast planning approach
- Reduction of the number of haulage trucks in use
- Reduction of dozer use

Measurements were done to estimate the time of main and influencing activities and for documental reasons. Drilling, blasting, loading and hauling are defined as main operations, whereas opening time and water filling of boreholes, charging, pushing can be assigned to additional activities. Test blasts were planned and documented via *BlastMetrix3D*. Calculations include *Excel VB* macros for data evaluation of load and haul measurements, and equations to determine the number of trucks. All activities – directly or indirectly – depend on each other, e. g. poor fragmentation due to insufficient blasting increases the time and effort for loading, and therefore are analysed for themselves before being linked with other results.

2 General information

2.1 Luzenac Operation Trimouns

The Luzenac talc mine "*Trimouns*" of Rio Tinto Minerals is situated in the French Pyrenees in southern France, 120 km south of Toulouse and 12 km north of the village of Luzenac. The existing quarry is about 2 km long and 800 m wide and therefore one of the largest open pit talc mines in the world. The deposit has been known since prehistoric times. In 1840 the first reported mining occurred, in 1888 commercial production began and production steadily increased to the current rate. With an output of 430,000 t a year of talc- and chlorite-bearing material – with about 8 t of overburden removed to extract on 1 t of talc – *Trimouns* is producing 8 % of the world's supplies and a third of the group's output. Due to the altitude (1,700 m) the mine is operated only from April to November. Around 270 people are employed in the mine and processing plant plus about one hundred seasonal workers who join the permanent staff at the quarry. [1]

Production is split into 18 grades during mining which are transported 5.5 km by an aerial cableway from the mine to the processing plant in Luzenac (at 600 m elevation). The plant operates 12 months per year and processes around 1,800 t of talc a day in form of 60 different products. For this optical sorting, grinding, micron sing, dry selection, palletizing and packaging are used. Ore storage capacity at the plant is approximately around 400,000 t ensuring an adequate ore stockpile while the mine is inoperative. A general overview of extraction and processing can been seen in the *Mine Flow Chart and Capacity 2008*. [2]

[1] Calmein, M. et al. 2005, pp. 11, 13; Howsen, M. P. 2000, p. 247; Rio Tinto Minerals n.d.a, pp. 1, 2 of 4; Rio Tinto Minerals n.d.b, p. 1 of 1; rtm_res_audit.pdf, pp. 1-1, 2-1

[2] Howsen, M. P. 2000, pp. 247, 247; Rio Tinto Minerals n.d.a, p. 3 of 4; Rio Tinto Minerals n.d.b, p. 1 of 1; rtm_res_audit.pdf, p. 1-1

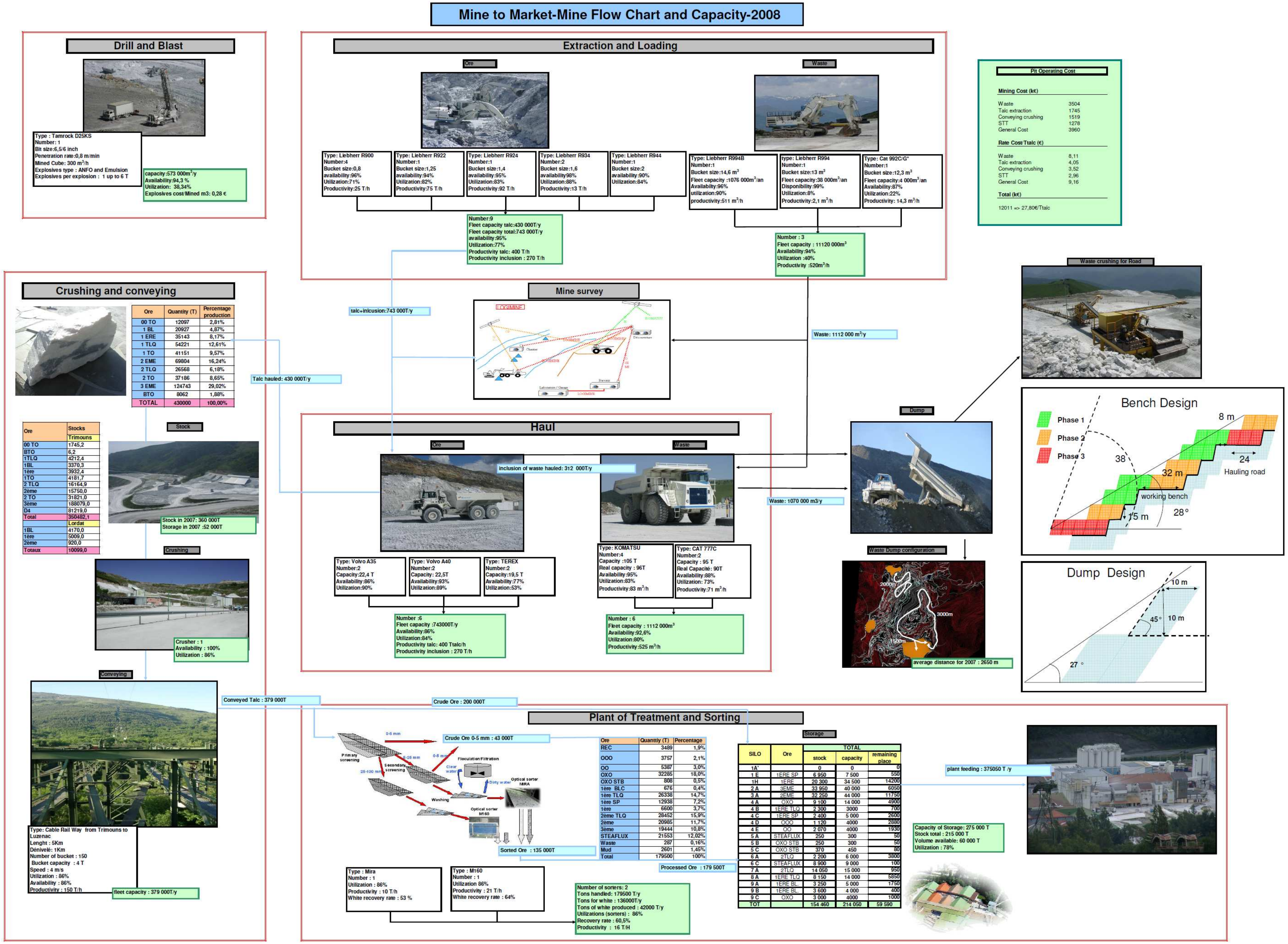


Figure 2.1: Mine flow chart and capacity 2008 [3]

[3] rtm_mine_process_2008.xls

2.2 Geology

The talc-chloritic deposit of *Trimouns* is exposed in the pit for 1.5 km and form an uneven, tabular layer between 20 and 60 m thick and 25 to 75 m wide within the pit. Generally positioned from North to South, the main ore body strikes off to the East at variable angle between 40° (in the North) and 70° (in the South). The orebody is divided into two main veins: a chlorite rich one close to the foot wall, which has roughly the same constant direction of N15-45E, and a talc rich layer located along the hanging wall contact with variable direction. Between these two main veins there is a major mica-schist inclusion of kilometric scale – cutting the topography in the North but disappearing to the South – which causes the upper vein to bend. Other minor chlorite veins belong to the chloritised faults system affecting the foot wall. [4]

The mineralisation occurs along a major fault and is interfoliated between the metamorphic rocks of the foot wall in the West (gneiss and granitic micaschist) and the hanging wall in the East (dolomite, schist and limestone). Talc results from metasomatic reaction between hot brines, migmatites and carbonates along this shear zone. The rock has been crushed by the pressure produced by tectonic movement, resulting in hydrothermal circulation of magnesium and silicates. Carbonates fix in-situ magnesium (as dolomite) reacted with silica to form magnesium silicate or talc, and migmatites (mica-schists) are transformed to chlorite due to the presence of magnesium. Talc found in *Trimouns* can be uniformly white or dark (impurities of pyrite or graphite) and is locally banded with precursor dolomite. The mineral deposit also contains sterile inclusions, large blocks of silica-aluminium close to the footwall or aplitic-pegmatitc near the hanging wall. [5]

[4] Calmein, M. et al. 2005, pp. 11, 13; Howsen, M. P. 2000, pp. 247, 248

[5] Howsen, M. P. 2000, p. 248; rtm_res_audit.pdf, pp. 2-4, 2-5, 2-7

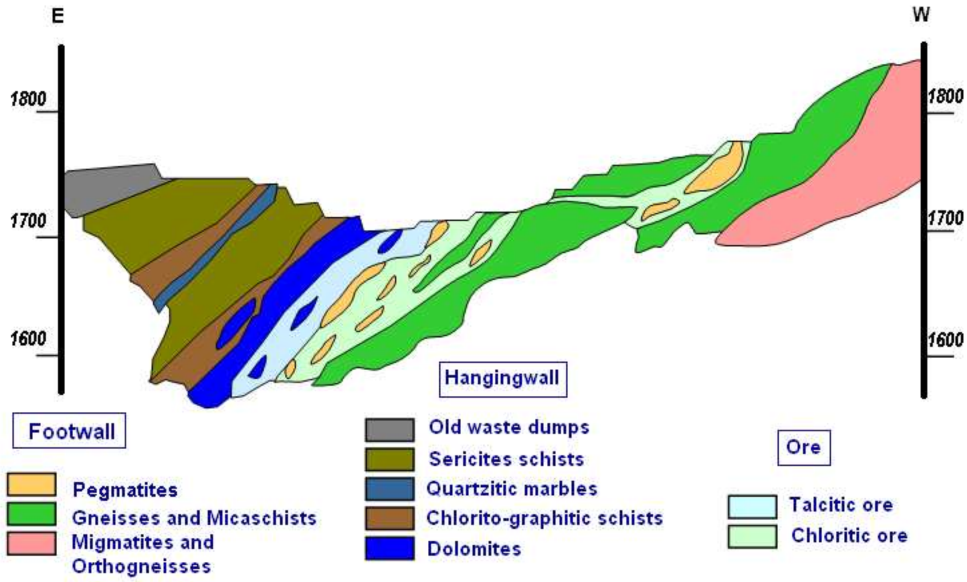


Figure 2.2: Geological profile of *Trimouns*' north part [6]

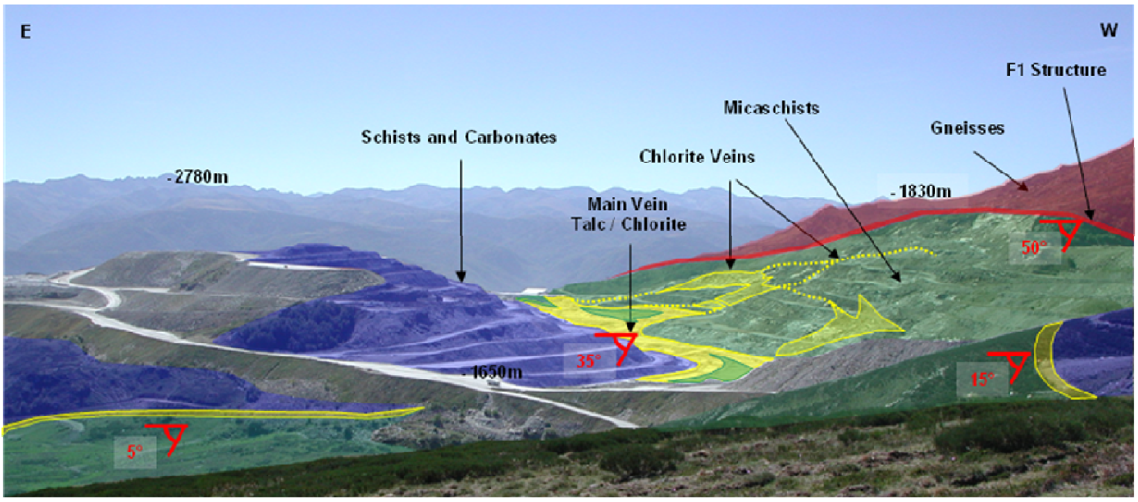


Figure 2.3: Surface geology of *Trimouns*' north part [7]

[6] rtm_trimouns.ppt, p. 10 of 88

[7] rtm_trimouns.ppt, p. 9 of 88

3 Actual work practice

At *Trimouns* two fleets of mixed equipment for the non-selective waste and the selective talc mining are in operation. All mining activities are documented and surveyed via *Logimine*, a Java-based programm.

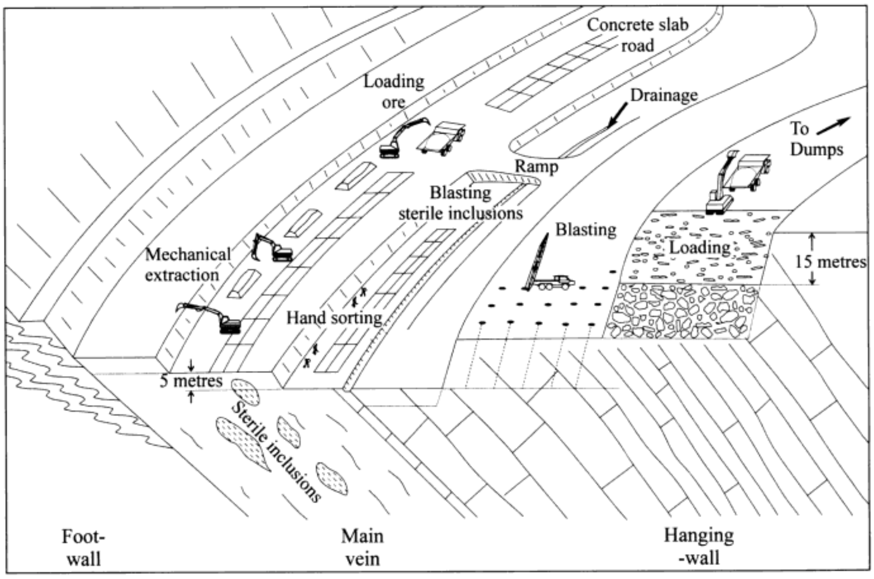


Figure 3.1: Sketch of *Trimouns* mining method [8]

The mining process is responsible for 43 % of general operating costs (waste 29 % and talc extraction 14 %).

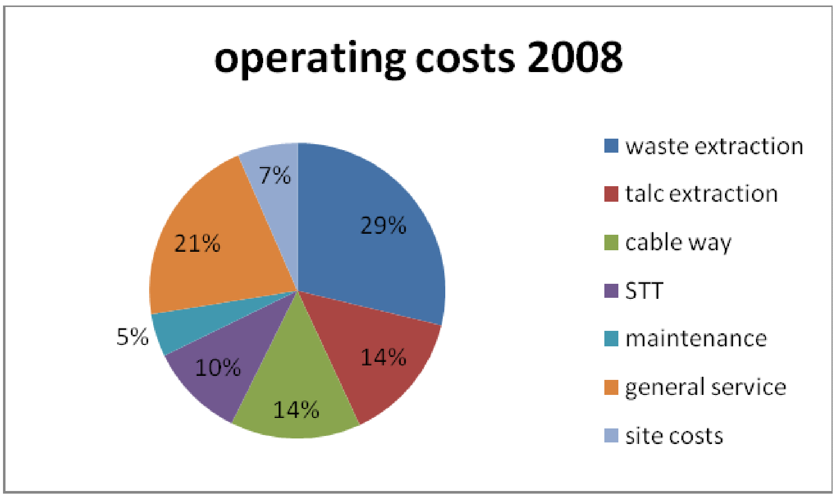


Figure 3.2: Operating costs 2008 and their accounts [9]

[8] Manual sorting has been replaced by selective digging via face shovels; Howsen, M. P. 2000, p. 249

[9] rtm_budget2008.pdf, p. 3 of 5

3.1 Talc extraction

To provide optimal selectivity small hydraulic face shovels are used for mining un-shot talc ore, which is placed in segregated stockpiles. Talc is sampled before being re-handled and hauled by articulated trucks. The material is then stored in bins by material type before being transported to the plant via cable way. Besides, talc blasted sterile intrusions have to be removed to waste dumps. [10]

3.2 Overburden removal

Waste mining includes activities like overburden removal, waste extraction via drilling and blasting, construction and maintenance of the main haul roads, which are all done by the *Découverture* in two 8.5 h shifts. Main goal is to provide access to the talc body over its length and its different qualities while achieving a stripping ratio of 1 : 8. The waste material consists of 8 % sterile inclusions (removed by the talc extraction itself) and 92 % waste material. [11]

The most expensive accounts of the overburden removal are energy (25 %), maintenance (24 %), labour (21 %) and amortisation (18 %). Costs for drilled and blasted material are 0.32 € / m³ and for loaded and hauled 2.99 € / m³. In general, the waste removal charges 8.74 € to extract one ton of talc. [12]

[10] rtm_res_audit.pdf, pp. 1-6, 8-1; rtm_trimouns.ppt, p. 23 of 88

[11] rtm_plan_exploitation_2009.doc, pp. 10, 22 of 50

[12] rtm_budget2008.pdf, p. 3 of 5

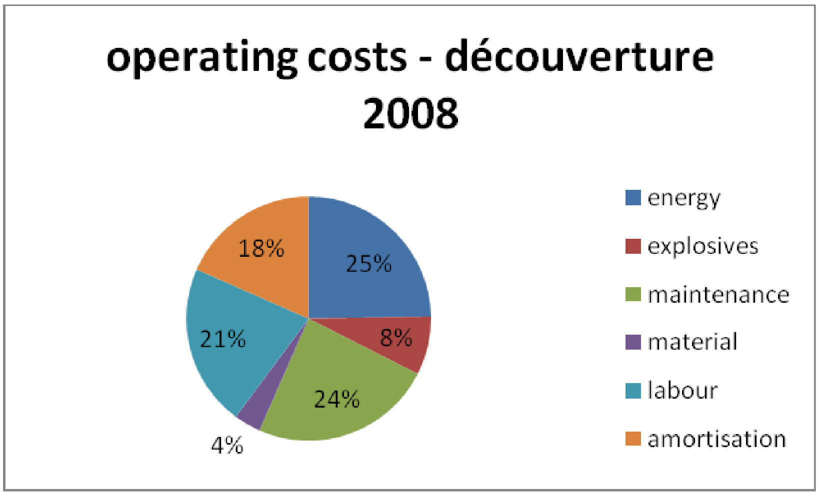


Figure 3.3: Operating costs - découverture 2008 and their accounts [13]

Waste material is mined in *Blocs* (units between 50,000 and 200,000 m³) determined in the *Plan d'Exploitation / Short Term Mine Planning 2009*. For 2009 most of the overburden removal was done in the hanging wall to create a new trench to have access to talc in the north part of the pit. Besides waste extraction creation and maintenance of the haul roads (between 30,000 and 40,000 m³) had to be done as well. [14]

3.2.1 Drilling and blasting

The primary fragmentation of waste material is done via drilling and blasting which is generally one bloc in advance of loading and hauling. Blastholes are drilled using a percussive DTH drill rig, *Tamrock Drilltech D25 KS*. Resisting toe boulders after loading are mostly re-drilled with smaller drill rig, *Ranger HL 600,* which is generally utilized for inclusions in the talc, and afterwards re-blasted. Significantly for drilling and blasting at *Trimouns* is on the one hand the intentionally low explosive in-put which results only in a loosening but not moving of the material or forming of muck-pile and on the other the use of only one drill and blast pattern to cover all different types of material in the hanging wall.

[13] rtm_budget2008.pdf, p. 4 of 5

[14] rtm_plan_exploitation_2009.doc, pp. 10, 11 of 50

The actual used drill and blast pattern has its origin in the *Langefors* formula [15] and has been modified by experience since then. For 15 m high benches boreholes are drilled with a diameter of 165 mm, a burden of 5.4 m and spacing of 7.0 m (37.8 m²). To provide a better fragmentation of the toe a subdrilling of 1.0 m leads to a borehole depth of 16.0 m. The 1st row is inclined with 10 and the 2nd with 5 degrees – subsequent rows are drilled vertically. The borehole is filled with one cartridge of emulsion (*EMULSTAR*) in the bottom and a column charge of 155 kg ammonium-nitrate (*ANFOTITE*) followed by a 5.0 m stemming of crushed dolomite. This results in a specific charge of around 0.290 kg / m³. Non-electrical caps inserted into an emulsion cartridge in the bottom of the borehole are used to initiate the main charge and its cord is fastened to the primer which is then lowered into the hole. The *NONEL*-detonator has a delay of 17 or 25 ms in a line and 42 ms in row.

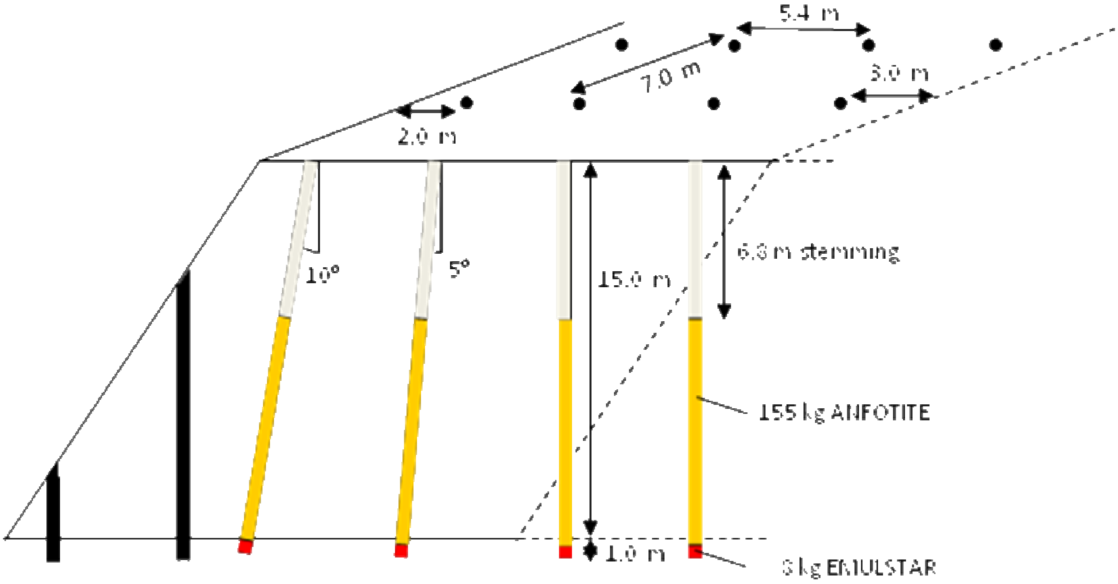


Figure 3.4: Sketch of the actual drill and blast pattern at *Trimouns*

[15] *Abattage par gradins à l'explosif* by R. Bétourné, Transfor, 1980

During the campaign of 2009, 133 blasts were shot in total. Of 64 bigger production blasts on different blocs, 48 occurred in the hanging wall, which had in average 22 holes and fragmented around 12,700 m³. Most of the explosions took place either in dolomite (48 %) or schist (44 %) and fewer in marble (8 %). [16]

3.2.2 Loading and hauling

Generally loading of blasted material is done via *Liebherr R 994 B* backhoe excavator with a planned hourly production of around 600 m³. The wheel loader *Caterpillar C 992* G is used to remove loose overburden, prepare new haul roads and is a stand-by equipment in case of a breakdown of the *R 994 B*. Material is transported via 6 waste trucks – 4 *Komatsu HD 985-5* and 2 *Caterpillar 777 D* – to either *Vers Sud* or in case of pure dolomite, situated in the North of the pit, to a separate stockpile close to the crusher at the *Découverture's* office. This material is later used for haul road maintenance and stemming of blast boreholes.

3.2.3 Auxiliary equipment

On both, load site for the excavator and dump site, a dozer (owned *Komatsu D275 A2* and leased *Liebherr PR 764*) operates for preparation and maintenance. The tractors are also used to create access ramps and maintain existing haul roads. Other auxiliary machines assisting the overburden removal are a grader, a compacter, a gas truck and some water trucks.

[16] calc_blast_exp_09.xls

The bulldozer *D 275 A2* is mainly used for the preparation of haul roads and assistance at the load site. It is operating on the charge site two to four times per day for a short period (15 to 30 min) to optimise the truck's driving conditions. Furthermore the dozer is used to prepare possible boulders to be drilled by *Ranger HL 600* and then re-blasted. The second bulldozer *PR 764* operates at the waste dump – primarily pushing of discharged material over the edge.

Equipment of the *Découverture* is refuelled every morning before shift start and again during the morning break because the machine's tanks are not able to provide enough gas for two whole shifts and personal for the tank truck is only available until midday. Generally the tank truck drives around in the pit searching all machines, starting with the primary loading unit (*R 994 B* or *C 992G*) on-site, continuing with empty dump trucks (*HD 985-5* and *C 777 C*) at the break's parking area and finishing with all other equipment like drill rig, bulldozers, water trucks, etc. While fuel is pumped with an average rate of 3 to 4 l / s, the operator of the gas truck inspects every vehicle for maintenance reason.

4 Technical specifications

This chapter provides a general overview about technical specifications of later observed machinery or means of production including main parameters evaluated from measurements and calculations.

4.1 Drilling & blasting

4.1.1 Tamrock Drilltech D25 KS

Name		Drilltech *D25 KS*
Producer		Tamrock
Description		Percussive DTH drill rig
Function		Drilling of blast holes for waste extraction
Active since		1993
Operating weight	[t]	28
Drill power	[HP]	430
	[kW]	321
Rated drill speed	[RPM]	1.8
Maximum travel speed	[km / h]	2.9
Fuel tank	[l]	758
Fuel consumption	[l / h]	69
Drill bit diameter	[mm]	154, 165
Availability	[%]	98.9
Hourly production	[dm / h]	29
Time per drilled meter	[min]	0.7 – 1.6
Time per drilled borehole (16 m + additional activities)	[min]	18.9 – 28.5
Costs (with amortisation)	[€ / h]	141

Table 4.1: Technical description of *D 25 KS* [17]

[17] calc_activity_FOD_09.xls, calc_costs_09.xls, calc_drill_09.xls, calc_factors_09.xls; rtm_D25KS.pdf, pp. 2 and 5 of 8; **Tab. 12.1**

4.1.2 ANFOTITE and *EMULSTAR*

Name		*ANFOTITE N°1*	*EMULSTAR 5000*
Producer		Titanite S.A.	Nobel Explosifs France
Description		Ammonium-nitrate, bulk explosive	Emulsion in cartridges
Function		Column charge	Bottom (normal) or main (wet holes) charge
Diameter	[mm]	-	130
Storage and handling unit	[-]	Bags of 25 kg	Cartridge of 8 kg
Length per unit	[m]	1.5	0.6
Density	[g / cm³]	0.83	1.28
Velocity of detonation	[m / s]	2,900	5,500
Specific gas volume	[l / kg]	963	863
Specific energy	[MJ / kg]	3.78	3.89
Water resistance	[-]	None	Good
Charge time per unit	[min]	0.3	1.2
Costs	[€ / kg]	0.77	2.21

Table 4.2: Technical description of *ANFOTITE* and *EMULSTAR* [18]

[18] calc_time_charge.xls; rtm_anfo.pdf, p. 2 of 2; rtm_emul.pdf, p. 2 of 2; mail_contrat_exp_2009.pdf, p. 1 of 2

4.2 Loading and hauling

4.2.1 *Liebherr R 994 B* and *Caterpillar C 997 G*

Name		*R 994 B*	*C 997 G*
Producer		Liebherr	Caterpillar
Description		Hydraulic backhoe excavator	Wheel loader
Function		Loading of overburden material	Stand-by for *R 994 B*, loose overburden and roads
Active since		2001	2008
Operating weight	[t]	297	95
Engine output (SAE)	[HP]	1,500	590
	[kW]	1,120	791
Rated capacity (SAE)	[m³]	15.3	11.5
Stuck capacity (SAE)	[m³]	12.5 [1] / 14.6 [2]	9.5
Maximum Speed	[km / h]	3	24
Fuel tank	[l]	5,350	1,562
Fuel consumption	[l / h]	197	76
Maximum Breakout force	[kN]	1,020	600
Availability	[%]	98.4	99.6
Hourly production	[m³ / h]	749	415
Buckets per truck load	[1]	4	6
Time per bucket	[s]	30	63
Time per load cycle	[min]	2.9	7.9
Costs (with amortisation)	[€ / h]	352	167

[1] value used for company-internal calculations
[2] value given in the technical description

Table 4.3: Technical description of *R 994 B* and *C 997 G* [19]

[19] calc_activity_FOD_09.xls, calc_bloc11_load.xls, calc_costs_09.xls, calc_factors_09.xls, calc_prod_bloc _09_01.xls, calc_prod_bloc_09_02.xls, calc_sum_load_haul.xls; Technical description – C 997 G n.d., pp. 1, 16, 17, 20 of 24; Technical description – R 994 B n.d., pp. 1 – 3, 5 of 10

4.2.2 *Komatsu HD 985-5* and Caterpillar *C 777 C*

Name		*HD 985-5*	*C 777 C* [1]
Producer		Komatsu	Caterpillar
Description		Dump truck with payload control unit	
Function		Transport of waste and overburden material	
Body		Dual slope with V-bottom	
No. of units	[1]	4	2
Active since		2004 (*HD 1* & *2*)	2000 (*C 1* & *C 2*)
		2005 (*HD 3*)	
		2006 (*HD 4*)	
Operating weight	[t]	179	164
Engine output (SAE)	[HP]	1,010	938
	[kW]	753	699
Rated capacity (SAE)	[m³]	64	60
Stuck capacity (SAE)	[m³]	37 [2] / 45 [3]	37 [2] / 42 [3]
Maximum speed	[km / h]	45	60
Average speed	[km / h]	21	21
Fuel tank	[l]	1,250	1,137
Fuel consumption	[l / h]	61	55
Availability	[%]	96.1	89.4
Hourly production	[m³ / h]	160	134
Costs (with amortisation)	[€ / h]	112	59

[1] technical specification of *C 777 D* - no info about *C 777 C* available
[2] value used for company-internal calculations
[3] value given in the technical description

Table 4.4: Technical description of *HD 985-5* and *C 777 C* [20]

[20] calc_activity_FOD_09.xls, calc_costs_09.xls, calc_factors_09.xls, calc_prod_bloc_09_01.xls, calc_prod_bloc_09_02.xls;
Technical description – C 777 D n.d., pp. 1, 16, 17, 20 of 24;
Technical description – HD 985-5 n.d., pp. 1, 5 of 8

4.3 Auxiliary equipment

4.3.1 *Komatsu D275 A2* and *Liebherr PR 764*

Name		*D 275 A2*	*PR 764*
Producer		Komatsu	Liebherr
Description		Crawler tractor with semi-U blade in the front and rear single-shank ripper	
Function		Preparation of load site & road maintenance	Preparation of dump site & road maintenance
Active since		2003	2009 (leased)
Operating weight	[t]	51	51
Engine output (SAE)	[HP]	405	422
	[kW]	301	310
Blade capacity (ISO)	[m³]	12.8 [1]	14.0 [2]
Fuel tank	[l]	840	905
Fuel consumption	[l / h]	47	30
Availability	[%]	96.8	96.5
Costs (with amortisation)	[€ / h]	141	Not specified

[1] SAE [2] ISO

Table 4.5: Technical description of *D 275 A2* and *PR 764* [21]

[21] calc_activity_FOD_09.xls, calc_costs_09.xls, calc_factors_09.xls;
Technical description – D 275 A2 n.d., pp. 1 – 3 of 4;
Technical description – PR 764 n.d., pp. 1, 2, 8, 12, 13, 16 of 12

5 Definitions

The following definitions are used in further in measurements and calculations. Some letters or abbreviations have a different meaning or different units due to their specific use, e.g. *speed* as transport speed of a truck given in km / h for load and haul measurements or speed of the drill head in RPM for drill time measurements

5.1 Drilling and blasting

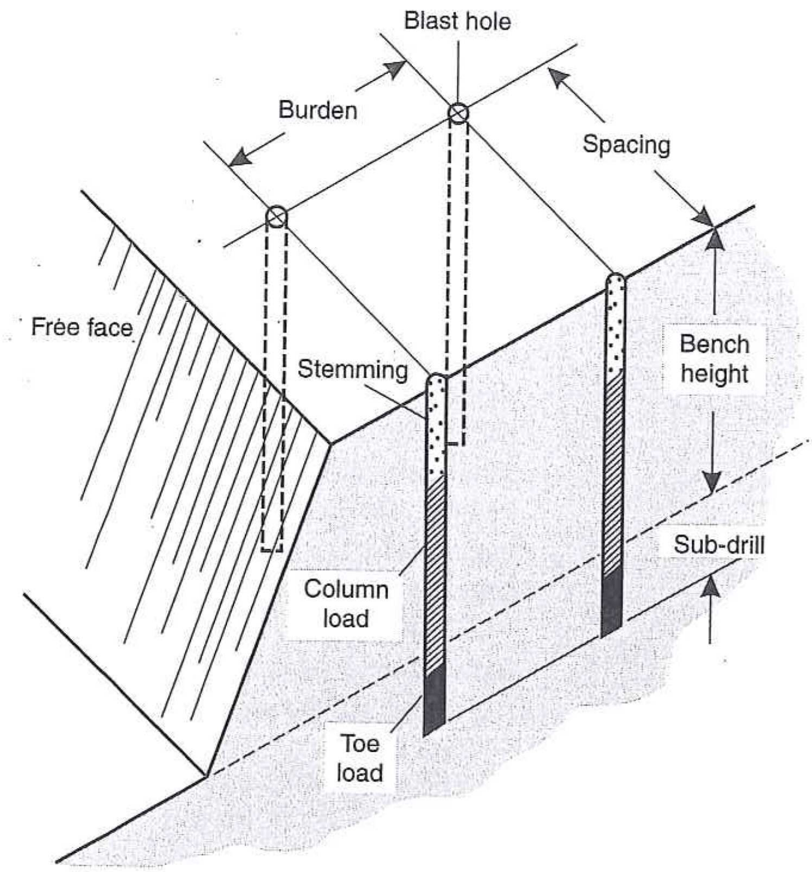


Figure 5.1: Definitions of blasting terms [22]

[22] Wyllie, C. W. & Mah, C. W. 2007, p. 248

Abbr.	Unit	Definition
bench height	[m]	Distance between floor and bottom level
bottom charge	[kg]	High energy toe load (here *EMULSTAR*)
burden (B)	[m]	Distance from a blast hole to the nearest free face
burden to space	[m]	Minimum distance of the 1st row to the free face
column charge	[kg]	Main load (here *ANFOTITE*)
cubes	[m³]	Blasted volume, multiplication of burden, spacing and bench height
depth	[m]	Drilled length of blast holes
diameter	[mm]	Blast hole diameter
inclination	[°]	Angle of a borehole measured from t he vertical
powder factor	[kg / m³]	Specific charge or the weight of explosives required to break a unit volume of rock Sum of column and bottom charge divides by the multiplication of burden, spacing and bench height
spacing (S)	[m]	Distance between blast holes perpendicular to the burden
stemming	[m]	Inert material packed above the charge
sub-drill	[m]	Drilling to a depth underneath the floor level

Table 5.1: General drill and blast parameters

5.1.1 Drill time

Abbr.	Unit	Definition
16 m	[s]	Drilling a 16.0 m deep borehole
1st rod	[s]	De-connection of 1st drill rod
2nd rod	[s]	De- / connection of 2nd drill rod
continuity	[s]	Checking for the hole's consistency once before connecting the 2nd drill rod and once after drilling
drive	[s]	Moving to a new drilling position
rec.	[bar]	Receptor pressure
rig down / up	[s]	Swing of rig into vertical drilling / horizontal driving position
rot.	[bar]	Rotary pressure
speed	[RPM]	Head speed
thr.	[bar]	Thrust pressure
un- / park	[s]	Readjustment / horizontal adjustment of drill rig via hydraulic support
wo.	[bar]	Working pressure

Table 5.2: Parameters of drill time measurements [23]

5.1.2 Charge time

Abbr.	Unit	Definition
1st cartridge + detonator	[s] or [min]	Connecting the detonator with the bottom charge and lowering down
2nd cartridge	[s] or [min]	Lowering of a 2nd cartridge
anfo	[s] or [min]	Pouring of *ANFOTITE*
measure of depth	[s] or [min]	Control of continuity, determination of depth or water filling via rope and weight
positioning of truck	[s] or [min]	Time to park the explosives truck ready to be unloaded
stemming	[s] or [min]	Filling the borehole with crushed dolomite after charging
unload bags	[s] or [min]	Discharging explosives and detonators

Table 5.3: Parameters of charge time measurements [24]

[23] calc_time_drill.xls

[24] calc_time_charge.xls

5.2 Loading and hauling

5.2.1 Load and haul measurements

The following definitions are used in load and haul measurements done at load site, on truck and at dump site for *bloc 3, bloc 11* and waste dumps. [25]

Abbr.	Unit	Definition
activity	[-]	Operation (W, R, L1, L, LL, U)
area	[-]	Zone of actual activity (LS, DS, H, R)
cleaning [26]	[-]	All operations to create a clear loading site, road and / or face
condition	[-]	Quality of the haul road
distance	[m]	One-way haul distance from load to dump site
fill factor	[1]	Filling degree (1, 2 or 3) including all buckets
fill factor per bucket (without last)	[1]	Filling degree (1, 2 or 3) excluding all last buckets with a fill factor of 1 (non-optimum filling due to reached truck capacity)
haul	[-]	Material transport from load to dump site
normal loading [27]	[-]	General loading process, no difficulties, fluent bucket filling
return	[-]	Driving back from the dump to the load site
ripping [28]	[-]	Interrupted bucket movement and / or more than one digging process
slope	[°]	Inclination of the (partial) haul road section
speed	[km / h]	Transport speed
speed limit	[km / h]	Maximum allowed transport speed
stones	[-]	Handling of boulders (see **Fig. 5.2**)
tons	[t]	Weight showed by the truck's payload control unit
weight per bucket	[t]	Average weight per bucket shown by the truck's payload control unit

Table 5.4: Parameters of load and haul measurements

[25] calc_bloc3_load.xls, calc_bloc3_truck, calc_bloc11_load.xls, calc_sum_load_haul.xls, calc_trench_dump.xls

[26] video_load_b3s4_2608_04.wmv

[27] video_load_b11s4_0709_01.wmv, video_load_b3s3_1208_01.wmv

[28] video_load_b3s4_2608_02.wmv

Abbr.	Unit	Definition
1	[1]	Fill factor of 1, less than the bucket's stuck capacity (see **Fig. 5.3**)
2	[1]	Fill factor of 2, equal to the bucket's stuck capacity (see **Fig. 5.3**)
3	[1]	Fill factor of 3, equal to the bucket's heaped capacity (see **Fig. 5.3**)
A, B, C, D, E	[m]	Road sections (see **Fig. 5.6**)
av. C	[-]	Average *C 777 D* truck
av. H	[-]	Average *HD 985-5* truck
C	[-]	Cleaning
C1, C2	[-]	Used *C 777 D* truck
DS	[-]	Dump site (area)
H	[-]	Hauling (area)
H1, H2, H3, H4	[-]	Used *HD 985-5* truck
L	[-]	Receive a bucket (activity)
L1	[-]	Receive 1st bucket (activity)
LL	[-]	Receive last bucket, which has a fill factor of 1 (activity)
LS	[-]	Load site (area)
N	[-]	Normal loading
P	[-]	Change of the excavator's position
R	[-]	Returning (area) or Reversing (activity)
R	[-]	Ripping
S	[-]	Loading of stones
U	[-]	Dumping (area)
W	[-]	Waiting (activity)

Table 5.5: Input parameters of load and haul measurements

Figure 5.2: Loading of a boulder

Figure 5.3: Example for used fill factor (fill factor 1 left, 2 middle and 3 right picture)

Abbr.	Unit	Definition
(no. of) buckets (buc.) per load	[1]	Theoretical number of buckets necessary to fill one truck
hang time (ex. 1^{st} bucket)	[s] or [min]	Unoccupied time for loader between truck change, incl. time for positioning, material and load site preparation, waiting for arrival of truck $_{(n + 1)}$ From last bucket of truck $_{(n)}$ to 1^{st} bucket of truck $_{(n + 1)}$ diminished by the time to prepare 1^{st} bucket
haul & return cycles per hour	[1]	Theoretical number of haul and return cycles per hour
load cycles per hour	[1]	Theoretical number of load cycles per hour
no. of buckets per hour	[1]	Theoretical number of buckets per hour
queue time upon arrival	[s] or [min]	Waiting time for truck $_{(n + 1)}$ due to loading of truck $_{(n)}$ From truck $_{(n + 1)}$ stops to truck $_{(n)}$ leaves load site (25 s after receiving last bucket)
queue time upon loading	[s] or [min]	Waiting time for truck $_{(n + 1)}$ after truck $_{(n)}$ has left due to auxiliary equipment or loading difficulties From truck $_{(n)}$ leaves load site (25 s after last bucket) to reverse upon loading of truck $_{(n + 1)}$
reverse time at dump site	[s] or [min]	Time to position a truck for discharging
reverse time at load site	[s] or [min]	Time to position a truck for charging Sum of reverse time upon arrival and upon loading
reverse time upon arrival	[s] or [min]	Reversing of truck $_{(n + 1)}$ before truck $_{(n)}$ leaves load site
reverse time upon loading	[s] or [min]	Positioning of truck $_{(n + 1)}$ From truck $_{(n)}$ leaves load site (25 s after receiving last bucket) to 1^{st} bucket of truck $_{(n + 1)}$
time per bucket	[s]	Time to fill a bucket and swing it into discharge position From receiving bucket $_{(n)}$ to bucket $_{(n + 1)}$,
time per dumping	[s] or [min]	Time to discharge a truck load

Table 5.6: Results of load and haul measurements (1 / 2)

Abbr.	Unit	Definition
time per haul & return cycle (see **Fig. 5.4**)	[s] or [min]	Time for truck to complete a cycle of load-, haul-, dump- and returning From 1st bucket of truck load $_{(n)}$ to 1st bucket of truck load $_{(n + 1)}$ Sum of time per load cycle, total waiting (incl. queue) and total reverse time, furthermore time for haul, dump and return
time per load cycle (see **Fig. 5.5**)	[s] or [min]	Time for loader to complete a cycle of hang- and loading From 1st bucket of truck $_{(n)}$ to 1st bucket of truck $_{(n + 1)}$ Sum of total hang time and time per load cycle
total hang time	[s] or [min]	Unoccupied time for loader between truck change used for positioning, material and load site preparation and filling of the 1st bucket From last bucket of truck $_{(n)}$ to 1st bucket of truck $_{(n + 1)}$ Sum of hang time (ex. 1st bucket) and (time per bucket) * 1
total queue time	[s] or [min]	Total waiting time for loader at load site Sum of queue time upon arrival and upon loading
total reverse time	[s] or [min]	Total time for positioning a truck Sum of reverse time on load site (upon arrival and upon loading) and on dump site
total time at dump site	[s] or [min]	Total time for truck at dump site From truck's arrival at load site over reversing and dumping
total time at load site (see **Fig. 5.5**)	[s] or [min]	Total time for truck at load site From truck's arrival at load site over queuing, reversing, loading [(time per bucket) * (no. of buckets – 1)] and leaving (25 s after receiving last bucket)
total time at load site (excl. leaving)	[s] or [min]	From truck's arrival at load site until receiving its last bucket
total time while haul	[s] or [min]	Total time to drive from load to dump site incl. moving and waiting
total time while return	[s] or [min]	Total time to drive back from dump to load site incl. moving and waiting
waiting time while haul / return	[s] or [min]	Interruption of driving, e. g. due to narrow road conditions or use of auxiliary equipment

Table 5.7: Results of load and haul measurements (2 / 2)

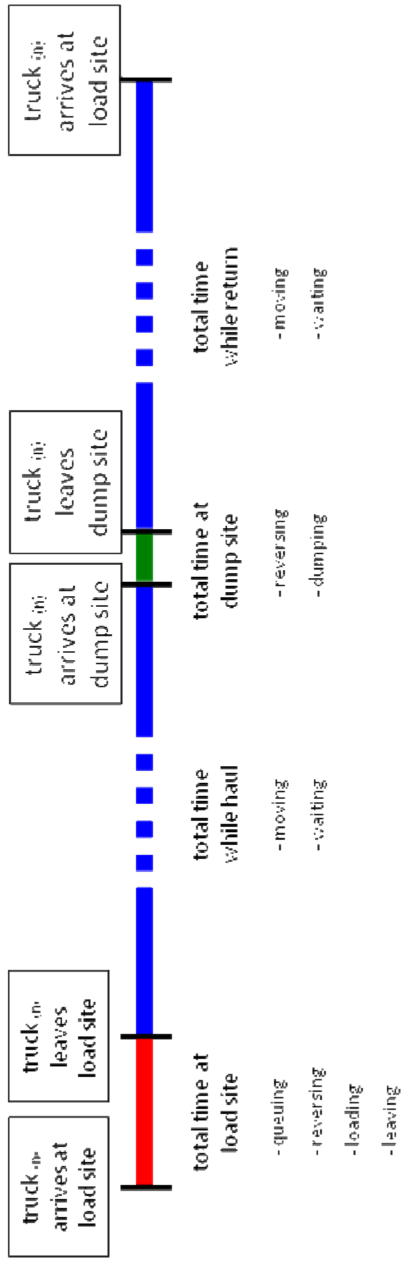


Figure 5.4: Visualisation of a truck’s haul and return cycle

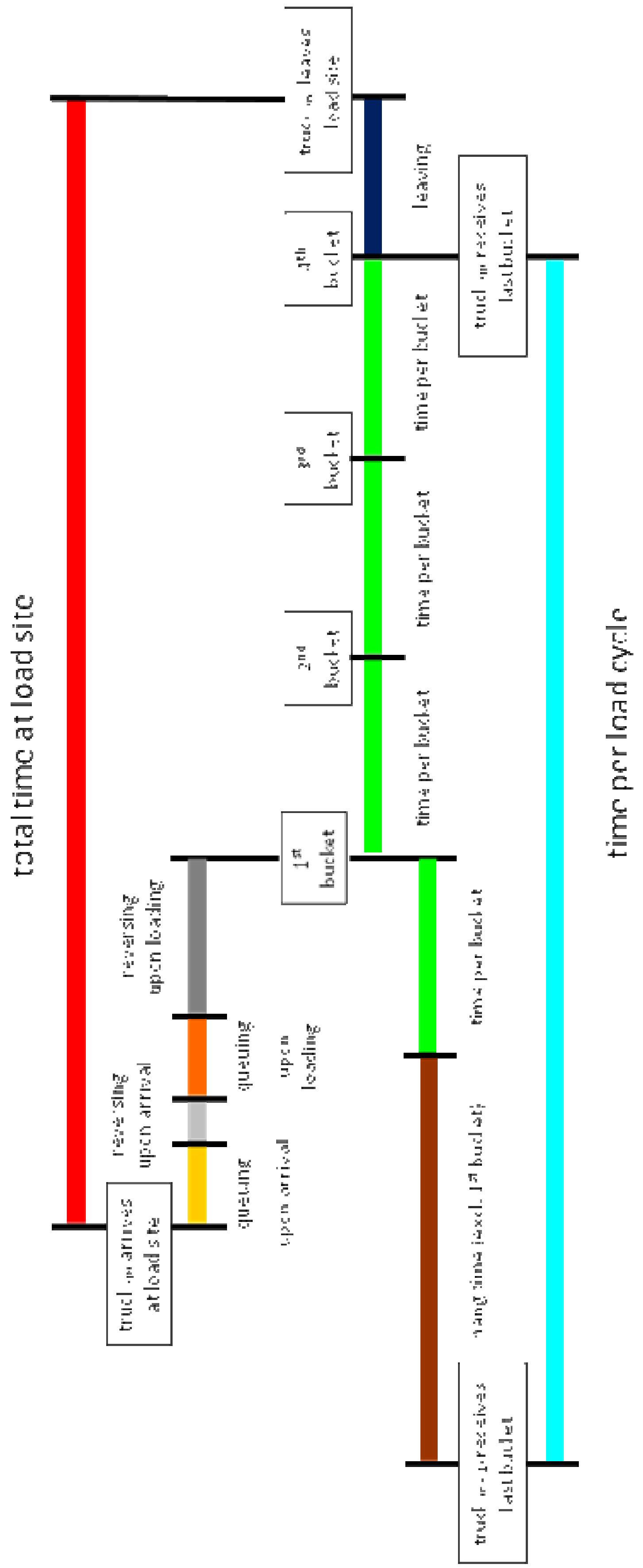


Figure 5.5: Visualisation of a truck's total time at load site and the loader's time per load cycle

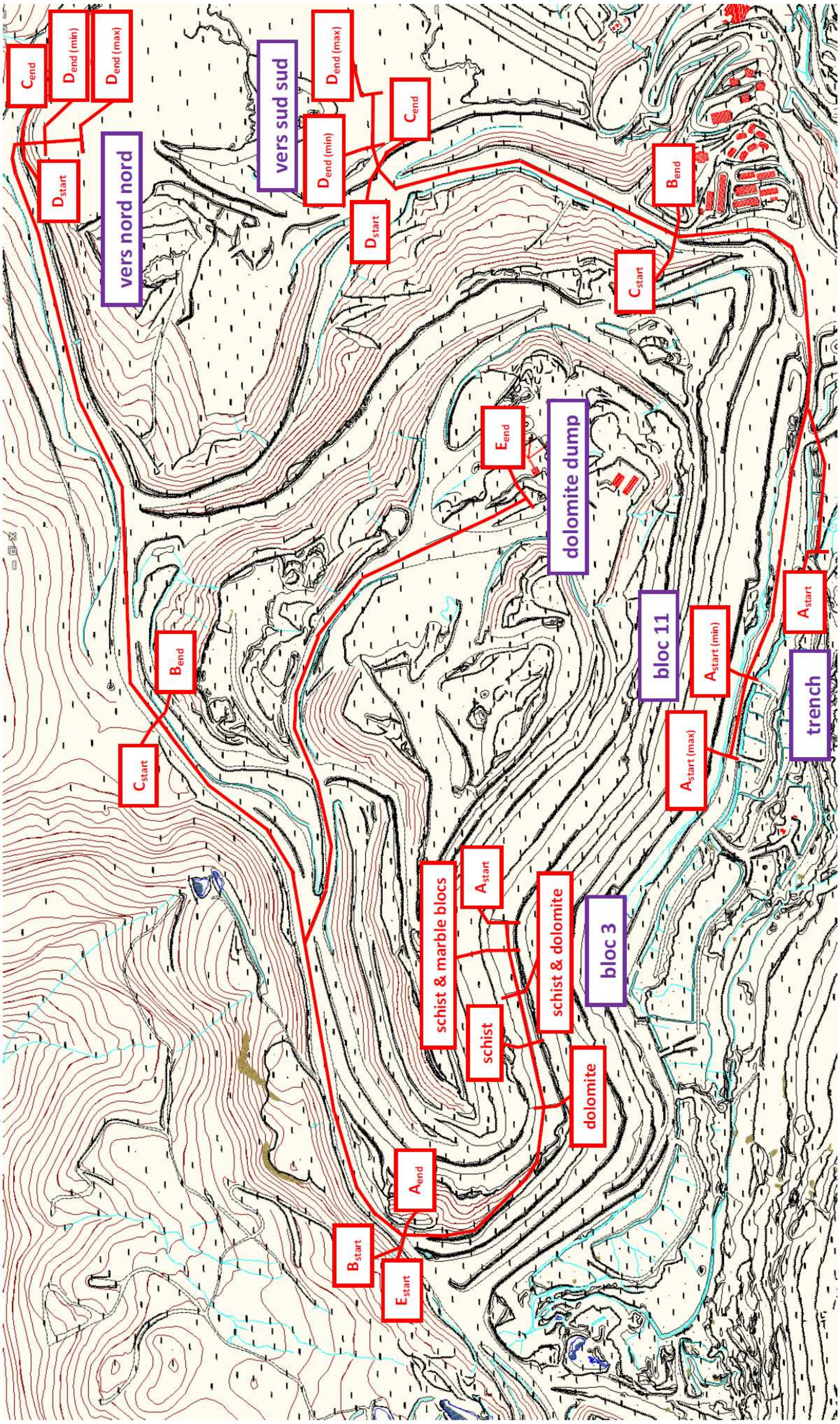


Figure 5.6: Main working areas and road sections of loading and hauling [29]

[29] rtm_map_01.dwg

5.2.2 Number of trucks

Abbr.	Unit	Definition
cap	[m³]	Capacity
cap_L	[m³]	Capacity of the loading unit's bucket
cap_T	[m³]	Capacity of the truck's body, which is equal to one truck load
cyc	[min]	Time per cycle
cyc_L	[min]	Load cycle per bucket, time includes filling of one bucket and its part on hang time (preparation of material, positioning of truck)
cyc_T	[min]	Haul and return cycle for a truck, time includes loading, dumping, haul, return, reverse on the load and dump site
d	[km]	One way distance from load to dump site
D	[min]	Time to dump material
n_T	[1]	Number of trucks suitable for specific loading conditions
prod	[m³ / h]	Hourly production
$prod_L$	[m³ / h]	Hourly production of the loading unit
$prod_T$	[m³ / h]	Hourly production of a truck
rev	[min]	Reverse time on the load and dump site
v	[km / h]	Average speed of a truck

Table 5.8: Parameters of truck number calculations [30]

[30] calc_no_trucks.xls

6 Observations

6.1 Drilling and blasting

While watching drilling, charging and blasting these characteristics occurred:

Observation	Consequences	Proposal
Changes of the bloc limit / geometry and occurrence of uneven crest.	Due to difficulties concerning manual marking of boreholes changes the orginally perpendicular planned drill pattern is changed to a staggered one. Easer holes are needed to decrease extensive burden (esp. *bloc 4*). (see **Fig. 6.1**)	GPS assistance on the drill rig would help to realise planned drill patterns.
Boreholes drilled in geological unstable settings like schist are wider than those in dolomite. (see **Fig. 6.2**)	Unplanned bigger hole diameters increase the stemming length and therefore lead to loss of energy in the top part of the borehole. This could result in boulders and in excessive but unnecessary fragmentation of the bottom area due to higher explosive concentration.	The use of a smaller borehole diameter (e. g. 154 instead of 163 mm) in the affected areas could improve the explosives' distribution.
Non-perpendicular positioning (to the face) of the drill rig *D 25 KS* if there are narrow working areas and / or an uneven crest. (see **Fig. 6.3**)	Deviating borehole direction when drilling inclined. This could lead to greater burden and penetration into former boreholes (1^{st} row), which could – in the worst case – imply residual, non-detonated explosives from former blasts.	If possible the use of inclined boreholes and therefore the need for perpendicular, more time-consuming positioning should be minimised.
Measurements of vertical boreholes via torch and tape have shown no significant deviation in direction.	Vertical boreholes have a more precise and wished development than inclined boreholes. Less deviation from the planned drill and blast pattern, esp. burden, provide a better fragmentation and blast result.	The exclusive use of vertical boreholes should be taken into consideration, as long as later loading performance is not influ-enced negatively.
Esp. at the end of a campaign, the drill rig suffers from more break downs. [31]	Main reason for these machine failures is, according to operators, the minimum preventive maintenance due to a lack of personal. The increase of loss and repair time negatively affects drill performance and costs.	To guarantee constant machine availability and its planned lifetime, a proper maintenance schedule has to be realized.
Only minimum drill documentation (borehole depth, presence of water or soils) is provided before and considered while charging (e. g. no use of intermediate stemming).	Attention is paid to parameters like borehole depth and the presence of water or soils, but not e. g. change of geology. Without the use of inter-mediate stemming, energy could be lost through weaker geological for-mation leaving the surrounding harder rock not fragmented.	Additional use of the drillers' ability to differ rock types by noise and head speed to identify geological properties. These should have more influence on the charge adaption on-site to provide the optimum interaction between explosives' energy and rock.
Drill and blast documentation does not include precise geological in-formation according to the boreholes' position. 32	Eventually occurring boulders (see Tab. 6.7), due to a change of geology and therefore different need for explosives' energy, cannot be linked to their origin.	Introduction of a more informative and detailed drill and blast documentation. Blast results could be compared to their individ-ual blast conditions and help to avoid the need for re-blasting.

[31] e. g. failing of the greasing system (06/10/2009), break of the oil cooling unit (07/10/2009)

[32] see drill and blast documentation by R. Sarda, e. g. rtm_b3_57_2707.xls, rtm_b4_95_0109.xls, rtm_b11_77_1108

Observation	Consequences	Proposal
Drill and blast documentation provided by R. Sarda (incl. main parameters and a sketch) and *Logimine* (daily reports for each blast or extraction of all data for specific period) show some inconsistency (evaluated for *bloc 3, 4* and *11*). (see **Tab. 11.1** and **11.2**, **Fig. 11.1 – 11.3**)	15 % of all values for *bloc 3*, 18 % for *bloc 11* and 10 % for *bloc 4* have a deviation according to the comparison of the main parameters [33] occurring in all sources. Esp. the number of holes (43 – 60 % of total deviation) per blast and the stemming height (9 – 29 %) per hole are varying.	The amount of occurring deviations advise an improvement of the actual drill and blast documentation to achieve consistent data for further calculations and interpretations.
Production blasting is done every day (on average 20 boreholes in the hanging wall). [34]	Daily blasting increases unproductive time for all unit operations, e. g. inefficient charging, survey of the charged site, less drill time due the rig's removal from the blast site, evacuation of all working personal. Furthermore, premature escape of gases through existing cracks while blasting could lead to poorer fragmentation.	Blast less often but more volume. The optimum number of holes is depends on limiting factors like maximum amount of stocked explosives, opening time and water filling of boreholes, charging performance (which should be increased with the use of the explosives truck).
Single-row blasts due to geometry occur at the beginning and at the end of a *bloc*. (see **Fig. 6.4** and [35])	Inadequate fragmentation which results in difficult loading conditions (loading on the same level, more ripping) and the need for re-blasting.	Avoid single-row blasts and eventually create blocs departing from the general used sickle geometry.
If there is water in the hole it is either blown out with the drill rig *D 25 KS* shortly before charging [36] or emulsion cartridges are used to rise above standing water.	Blowing out before charging reduces the number of necessary *EMULSTAR* cartridges and allows the use of cheaper *ANFOTITE* in top part of the hole. If the removal of the water is not possible (e. g. source) the whole length is filled with expensive *EMULSTAR*.	Actual work practice seems to be adequate for handling water in holes. If bigger blasts are realised, the increase of water over time should be taken into consideration (see **8.1.2**).
When using two cartridges the 2nd one is immediately lowered after the first one with a rope and "fish-hook". The last few meters the cartridge falls free. [37]	Free fall of cartridges is always a safety risk and should be avoided under any circumstances.	The use of a longer rope would guarantee a smooth lowering of the emulsion.
If it is not possible to pour all *ANFOTITE* into a hole, the spare explosives are distributed to the surrounding holes, but not necessarily mentioned in the drill and blast documentation.	It is not possible to compare changes of the explosives length, actual geology or blast results due to lack of documentation.	Add eventual changes of explosives' amount concern*ing ANFOTITE* to the drill and blast documentation, like it is actually done for *EMUSTAR*.
Due to French blasting regulations, bags, boxes and plastics used for explosive storage are burnt close to the blast side immediately after charging, when there is still no stemming material added.	Any fire close to explosives is dangerous.	A detailed investigation of how the French blasting regulations can be interpreted to provide more safety should be done.
The big wheel loader C 992 G transports stemming from the dolomite crusher close to the Découverture's office to the blast site. The crushed dolomite is then brought into the borehole by using a second, smaller wheel loader, which drives to each borehole where the material is shovelled of its bucket. (see Fig. 6.5)	Neither the C 992 G nor the smaller wheel loader workings under optimum and intended conditions.	To avoid the use of the C 992 G for the transport of stemming material, the material could be hauled via a small articulated truck from the talc during break time. The stemming material could be distributed via hopper. 38

[33] zone, terrain, no. of holes, cubes, diameter, burden, spacing, depth, sub-drill, stemm height, anfo, emulsion

[34] calc_blast_exp_09.xls

[35] rtm_b3_57_2707.xls, rtm_b3_63b_3007.xls, rtm_b3_74_0708.xls, rtm_b4_122_2409.xls

[36] video_b11_85_2508_03.wmv, video_b11_85_2508_05.wmv

[37] video_b11_85_2508_02.wmv

[38] mail_hopper.doc.

Observation	Consequences	Proposal
In general, no ejection [39] of stemming or only of one [40] or two holes [41] per blast occured during the test blast firing.	The ejaculation of stemming sometimes created a crater. [42]	If more ejected boreholes occur, a greater stemming length could minimise the loss of energy through the top of the hole.
Back break occurred after the removal of blasted material, esp. in weaker geology (*bloc 3* [43] and *11* [44]). Some cracks way beyond the *bloc* limit occured before loading as well (*bloc 4* [45]). (see **Fig. 6.6**)	Back break and therefore weakness and instability is the main security risk when the wheel loader *C 992 G* is preparing the site for drill and blast. Furthermore is drilling of front row holes and their charging more dangerous due to the zone's instability.	Smooth blasting methods, the use of unloaded drill holes to prefer wanted cracks or an inclined last row should be tested for suitability.
During the run of test blasts boulders occurred once at *bloc 3* (South part, schist with marble blocs) and some at *bloc 11* (North and South, see **Fig. 6.7**) – both have been re-blasted (documentation of this blasts is not separately done by the company).	Boulders appeared despite using a smaller or wider grid. This could be the result of a weak geological zone, e. g. *bloc 11* had some talc layers at the bottom, or a too big pattern in case of compact intrusions. Boulders lead to additional preparation, drill and blast work.	To avoid boulders smaller distances between boreholes, a staggered pattern with overlapping fragmentation or high energy explosives could be used. [46]

[39] video_b11_83_1408.wmv, video_b3_61_2907.wmv, video_b3_63_3007.wmv, video_b4_101_0409.wmv, video_b4_105_0809.wmv, video_b4_109_109.wmv

[40] video_b11_85_2508_01.wmv, video_b3_59_2807.wmv, video_b3_61_2907.wmv

[41] video_b4_107_0909.wmv

[42] video_b11_85_2508_01.wmv, video_b4_107_0909.wmv

[43] little, generally closed fractures, reaching 1 – 2 m into bench, see Fig. 11.4, 11.6 and 11.7

[44] big cracks, open up to 30 cm, lasted up to 8 m into bench; continuous rock movement favours the elongation of existing and the creation of new cracks over time, see Fig. 11.5, 11.9 and 11.10

[45] mail_effet_arriere.pdf, see Fig. 11.4 and 11.8

[46] mail_marble_block.pdf

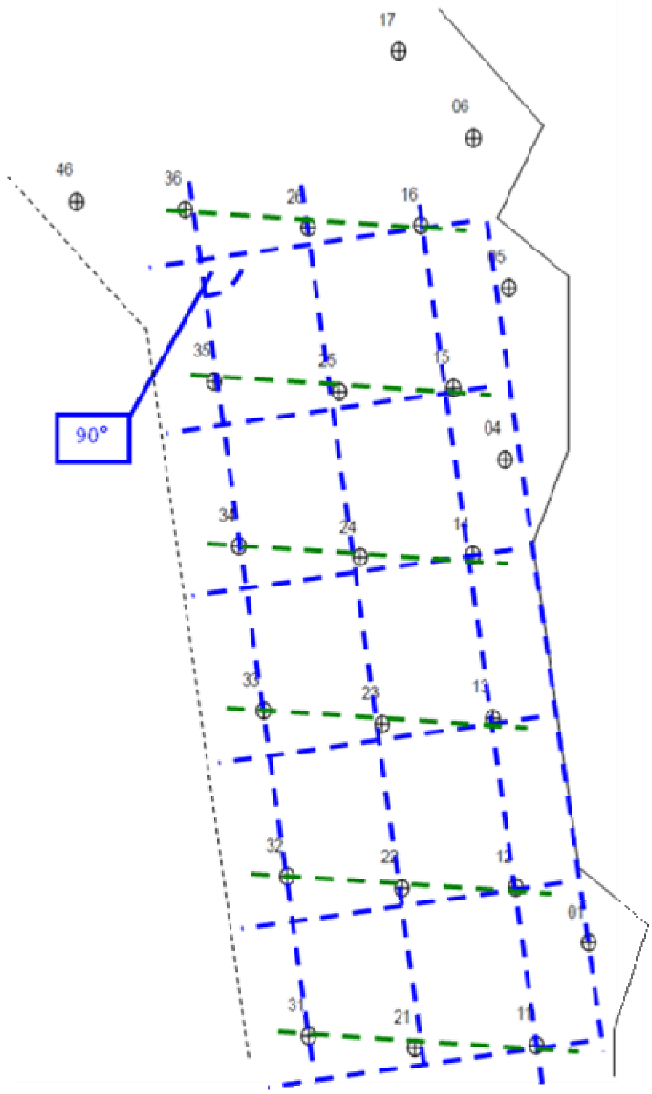


Figure 6.1: Planned perpendicular and real staggered pattern, incl. easer holes [47]

Figure 6.2: Borehole with increased diameter in schist (left) and with straight width in dolomite (right)

[47] bm_bloc3_02_blast59

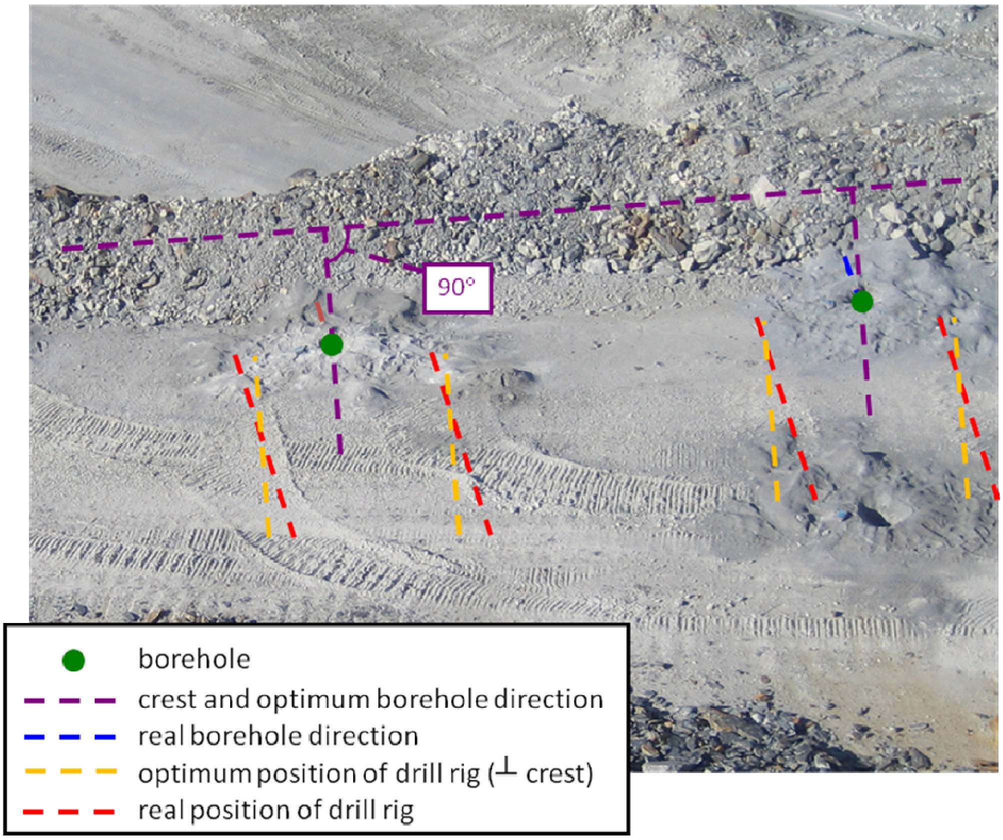


Figure 6.3: Non-perpendicular placement of drill rig *D 25 KS* while drilling inclined holes (*bloc 3*, South)

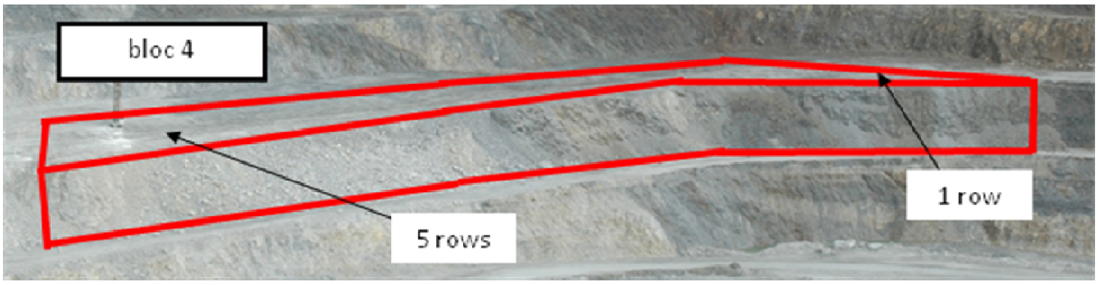


Figure 6.4: Single-row blasts at the end of *bloc 4*, South

Figure 6.5: Stemming of boreholes after charging via wheel loader's assistance

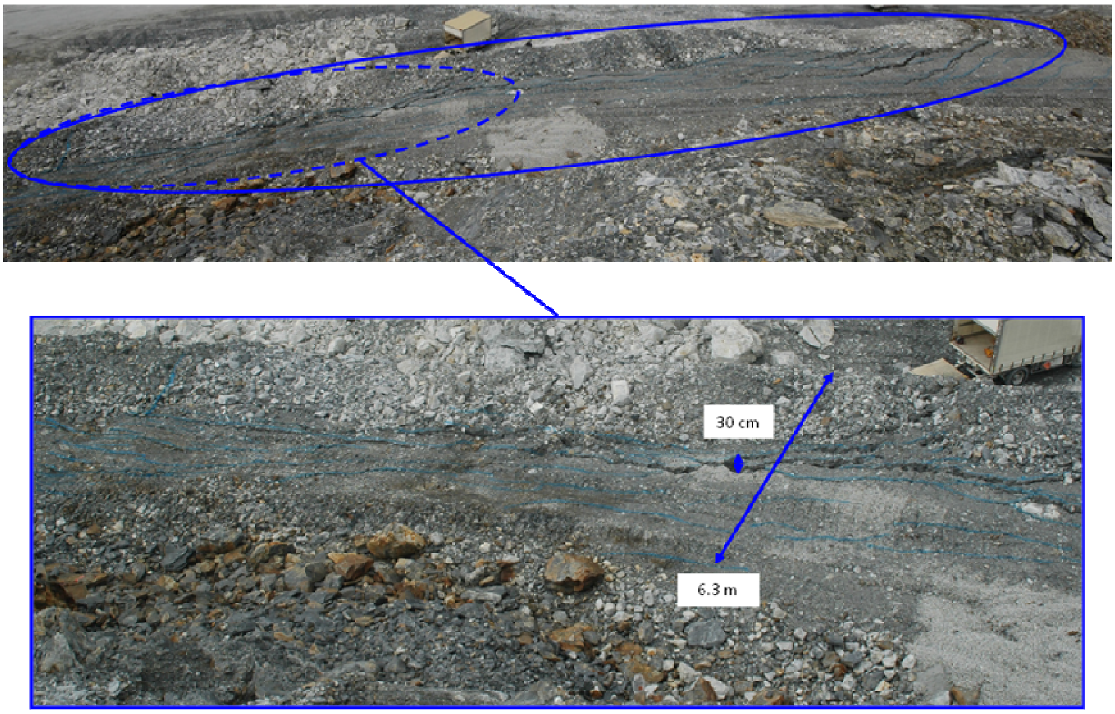


Figure 6.6: Back break at *bloc 11*, North

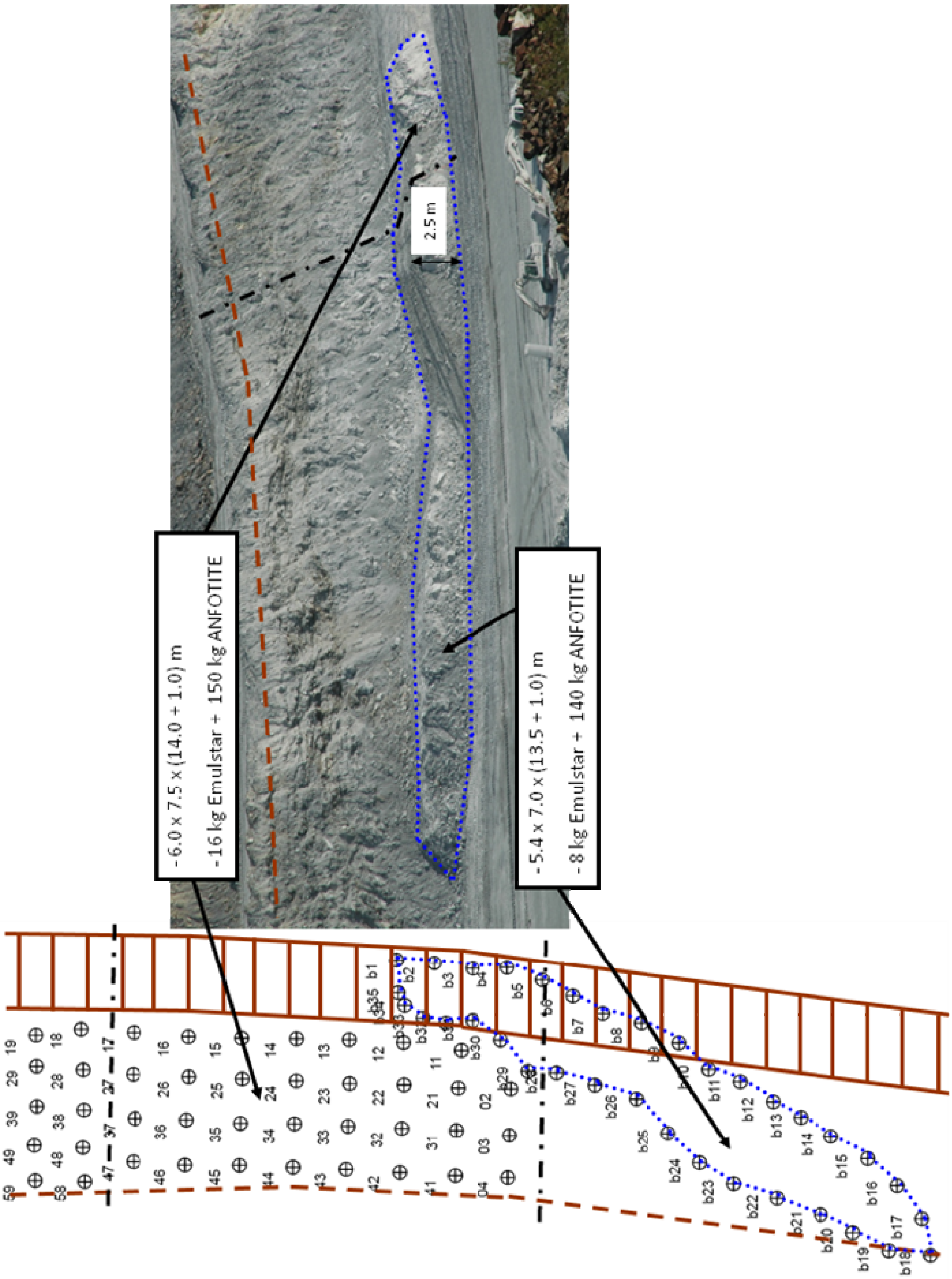


Figure 6.7: Dolomitic boulder on *bloc 11*, North [48]

[48] bm_bloc11_holes_boulder

6.2 Loading and hauling

General observations applying the load and haul process are following:

Observation	Consequences	Proposal
More ripping appears in the last rows of a blast, but the toe burden of the 1st row is well fragmented and easy to load even when using vertical boreholes. [49]	Despite the appearance of back break (see **6.1**), the loading of a blast's last rows encounter more loading resistance. Loading of the 1st row is as moderate as middle rows because already blasted material covers the face which does not need to fragment anymore.	A decrease of the last row's burden with an adaption of the explosives amount could improve the loading conditions. Due to good fragmentation and already blasted material it is not necessary to drill inclined boreholes.
There are zones of instability with continuous rock fall after loading has been finished. (see **Fig. 6.8** and [50])	The zones of instability are mainly situated in weak rock formations and can be a result of leaving blasted material behind.	Documentation and eventually loading of material disturbing the traffic on the bench should be enough. Working directly under it should be avoided under any circumstances.
R 994 B is sometimes overloading the trucks. (see **Fig. 6.9**)	Overloading of trucks minimizes the truck's and tires' life. Additionally there is the risk of losing material while driving and endangering trucks and other vehicles.	The loader should prefer loading less than too much on a truck and all traffic participants should pay attention for eventually lost material. Lost material positioned on the haul road should be immediately reported and removed as quickly as possible.
Very often the break is longer than 30 min (35 – 40 min), esp. of the morning shift.	Longer breaks minimise the haul and load performance (50 min relates to a weekly loss of 500 m³).	Additional break time can be avoided via communication and sensibilisation.
Generally after the break all trucks start from the same point at the same time (see **Fig. 6.10**) and have to wait before being loaded. If refuelling during shift / work, it is possible that there is no truck for available loading.	Bad timing after the break leads to additional queuing [51] and hanging This decreases the load and haul performance and interrupt the working cycle.	Proper planning of activities and modified behaviour can decrease hanging and queuing time occuring after breaks. [52]
In general, one or more arriving trucks are waiting at load site because another truck is still charged by the R 994 B. (see **Fig. 6.11**)	Because there is always a truck ready to be loaded, the R 994 B has no possibility to prepare material and can increase the time per bucket due to difficult loading condition. Furthermore proper cleaning of the load site is minimised and could increase the dozer use. Arriving trucks tend to drive in low speed to avoid queuing 53 which makes it difficult to accurately identify waiting time.	Queuing of trucks should be minimized if not eliminated. This could be achieved by reducing the number of trucks.

[49] video_load_b3s4_2608_01.wmv, video_load_b3s4_2608_03.wmv, video_load_b3s4_2608_04.wmv

[50] video_load_b3s3_1408_01.wmv

[51] video_haul_b11s3_0409_01.wmv, video_load_b3s3_1208_02.wmv

[52] e. g. the 1st truck arriving at break site is the 1st one to leave after 30 min even if official break time is not over yet; divison into more than one break points along the haul road to gurantee a proper coverage; less loaded trucks could stop without being discharged; more in **6.3**

[53] video_load_b3s3_1408_04.wmv

Observation	Consequences	Proposal
Narrow road conditions do not allow two trucks passing at the same time. (see **Fig. 6.11**)	Bottlenecks interrupt constant driving cycles and operators have to pay more attention to passing trucks. Additional waiting increases the necessary time for a haul and return cycle and therefore decreases the haul performance. Waiting for a clear road and multi-stage reversing at load site increases queue and hang time.	Appropriate road widths which allow two trucks to pass could avoid any interruption of hauling and decrease waiting time of any kind.
Narrow curves before waste dumps (Vers Sud) decrease the truck's haul speed (from 25 to 5 – 10 km/h). (see **Fig. 6.11**)	The abrupt drop of speed stresses equipment, decreases the machine's lifetime and can lead to more falling material of the truck's hoist. There is a danger of pulling of the curve if speed is too high.	Narrow curves before waste dump should be made wider. In general, this can be done easily and with low effort (see yellow line on the right, **Fig. 6.11**). This would allow the trucks to decrease their speed more smoothly before dumping.
Due to the company's rule, reversing is done as far as possible from the loading unit. (see **Fig. 6.12**)	Unnecessary back driving could make it difficult to position the truck and increases the reveres time and therefore hang time, when the *R 994 B* is waiting with a filled bucket.	A truck should reverse as closely as possible to the loading unit without risking any accidents and excessive back driving.
The operators of the *R 994 B* prefer low slice heights for loading. This resulted in 5 slices on *bloc 11* (15.0 m high before blasting) and in 4 slices on *bloc 3* (13.0 m). (see **Fig. 6.13**)	Despite regarding a volume increase after blasting, the actual working height is less than the optimum of 4.5 m. Such working conditions cause the low fill factor and the danger of undermining or touching the tracks with the bucket. Furthermore a lot of effort creating a new haul road, preparation of the load site and loading itself cost additional working time and increase the costs.	The operators of the *R 994 B* should be trained to operate within the optimum working height and maybe assisted via additional measurements to confirm the actual working level.
The limit of loading is marked via sticks and is around half the burden away from the last blast line. The operator of the *R 994* B is creates a bench face angle of 55 ± 10°. [54]	Due to the working experience of *R 994 B* operator the face angle achieved after loading is close to the planned value.	No proposals can be made concerning this matter.
Loads have been discharged closer than 5.0 m to crest even in direct tire contact to the safety berm though no person was there waving in.	Discharging close to the crest is a safety risk because the safety berm is not designed to hold off a truck.	Truck drivers should be sensitized for possible risks due to this behaviour.
Loading of normal blasted material via the wheel loader *C 992 G* does not only result in wheel spin but also in the lifting of the rear suspension even if the machine operates with an angle. Furthermore stone fall is occurrs [55] .	Loading difficulties of the wheel loader *C 992 G* minimize the machines' life expectancy. Stone fall while loading can load to an uncontrollable collapse.	Designed blasts which provide a better fragmentation and promote the creation of an easy load-able stock pile should be used in order have optimum working conditions.
Most of the load and haul data in *Logime* are from manual input due to technical problems. [56]	Unfortunately a direct comparison between load and haul measurements and actual data cannot be done in detail.	With the proper adjustment of Logimine and the used equipment done in autumn 2009 manual input should be minimised.
Values like distribution of truck's speed, bulldozer's or drill rig's activity like driving or working are not separately measured in *Logimine*.	The whole process is only roughly documented and therefore some information which can lead to improvement is not visible.	An adapting or including of more values concerning operating equipment would make it easier to monitor activities without being on site.
Load and haul activities are documented in different files, which show some inconstancies as well.	E. g. for *bloc 3* variations up until 1.4 % for cubes and 14.5 % for operating hours occur.	Deviations show that improvement for load and haul documentation is necessary to achieve consistent data and provide optimum data for further calculations.

[54] bm_bloc3, bm_bloc11

[55] video_load_b5_2508_02.wmv

[56] log_b3_HD4.pdf, log_b11_HD4.pdf

Figure 6.8: Rock fall after finishing loading and hauling, *bloc 3*

Figure 6.9: Normal (top) and overloaded truck (bottom)

Figure 6.10: Collective parking during break time

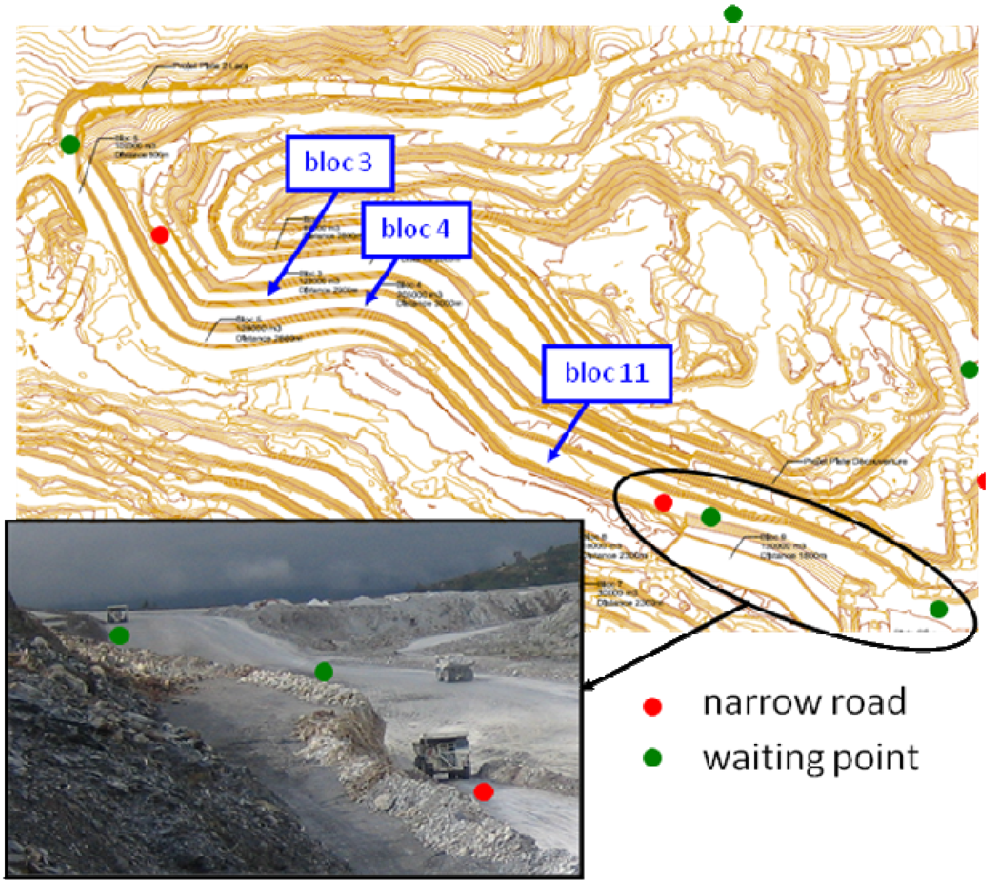


Figure 6.11: Location map of narrow road conditions and waiting points and resulting queuing at load site [57]

[57] rtm_trimouns.ppt, p. 23 of 88

Figure 6.12: Narrow curves before loading on *Vers Sud Nord* (left) and *Vers Sud Sud* (right)

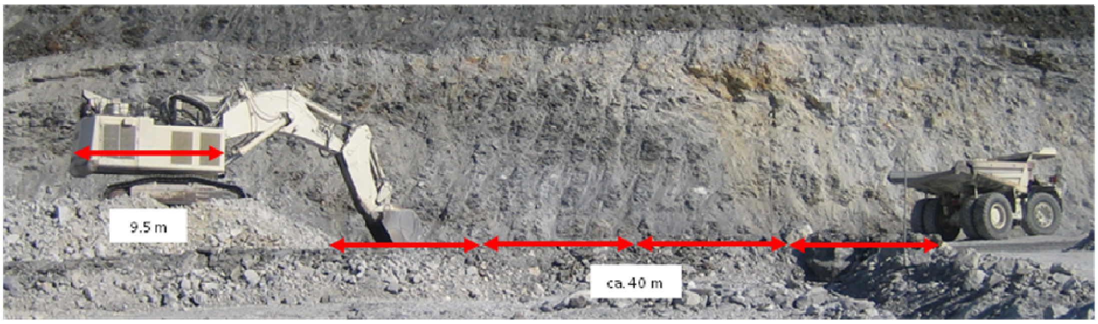


Figure 6.13: Reversing far away from the truck (*bloc 11*, slice 5)

Figure 6.14: Low slice height on *bloc 11*, slice 5

Figure 6.15: Discharging directly at the safety berm without waving in

6.3 Auxiliary equipment

Main observation points of the use of auxiliary equipment are shortly described in the subsequent paragraphs:

Observation	Consequences	Proposal
Despite dozing, driving conditions at the load [58] and dump site [59] are bad, resulting in percussion while driving and decrease of speed from 25 to 10 km / h. (see **Fig. 6.16** and **6.17**)	Bad road conditions do not only diminish haul and return speed but also increase equipment wear, esp. tires and suspension, and make driving uncomfortable.	Procedures should be evaluated to improve the road conditions at load and dump site. The use of the bulldozer *D 275 A2* cannot be reduced due to the loader's inability and / or time to prepare the load site. Dozing should be done if there is enough space available that does not disturb the actual load and haul process. In case of narrow work benches dozing should be done as quickly as possible or during break time. (bad example see **Fig. 6.18**)
Most of the time the *PR 764* is positioned perpendicularly to the dump's crest and waits with running engine until a truck dumps its load and then the dozer pushes it over the edge. [60]	The dozer stays at position, even if this time could be used to improve road conditions at load site.	It should be defined if either waiting besides the discharge position and pushing of single loads but that makes it possible to dump close to the edge or the dozing of multiple piles and other use of the dozer in the meantime is more important.
Neither the *D 275 A2* nor the *PR 764* is observed regularly operating with the ripping unit and all work is done by using the front blade.	Both dozers seem to be equipped with the ripping unit although this is not necessary.	Because the ripping unit is not part of the general dozing process, at least one of the two track dozers could be replaced by a more flexible wheel dozer.
The tank truck drives around in the pit searching for equipment to be refuelled without any information on the machine's need for gas or its position.	Driving around without knowledge of the machines' position, technical problems while refuelling or a long distance between the loader's working place and parking area of the dump trucks decrease the time available for refuelling and interrupt the load and haul process.	The driver of the tank truck should have information on every machine's need to be refuelled and on its location. Furthermore it should be taken into consideration to change the procedure to tank up if it is not possible to finish refuelling the loading unit and a minimum number of dump trucks during break time to avoid additional hang time. [61]

[58] video_load_b3s3_1408_04.wmv, video_load_b11s4_0709_01.wmv

[59] video_cycle_b3s3_1208_03.wmv

[60] video_haul_b11_0209_01.wmv, video_dump_b3s3_2608_01.wmv

[61] a possible solution would be the splitting up one location of parking and filling up the dump trucks to two or three smaller units; refuelling of low loaded dump trucks where there is no danger of stone fall while inspecting the vehicle, e.g. last two trucks before the break are charged less and park loaded; the use of a stand-by truck to bypass diminished haul capacities due to refuelling and to guarantee a minimum number of operating trucks; changing of refuelling time, e.g. some trucks should be filled up prior or after the break; if dump trucks are tanked up during active working time, the gas tank should be a on a fixed location along the haul and return route;

Figure 6.16: Bad road conditions at dump site

Figure 6.17: Dozer obstruct load and haul process due to limited space at load site

7 Measurements

Measurements are documented via sketches, pictures or videos. Besides measured values significant information like date, time, location (e. g. bloc, slice, rock type), used machinery, general comment, sketch and eventually drill and blast pattern have been collected. Results and analysis of done measurements will be provided in **chapter 9**.

The main measurements and observations during summer 2009 were done on *bloc 3, 4* and *11* and the waste dumps (*Vers Sud Sud*, *Vers Sud Nord* and *dolomite dump*) and following figures and paragraphs should summarize their main characteristics.

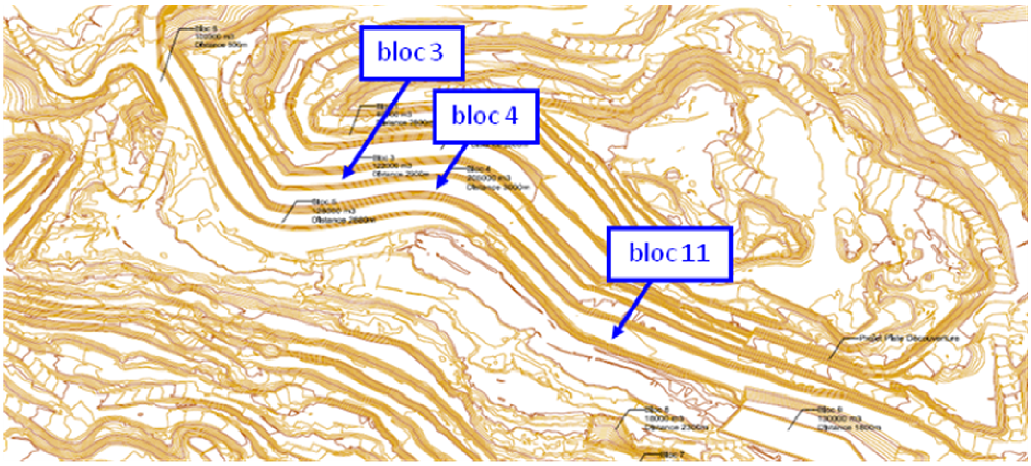


Figure 7.1: Location of investigated *blocs 3, 4* and *11* in the hanging wall (1 of 2) [62]

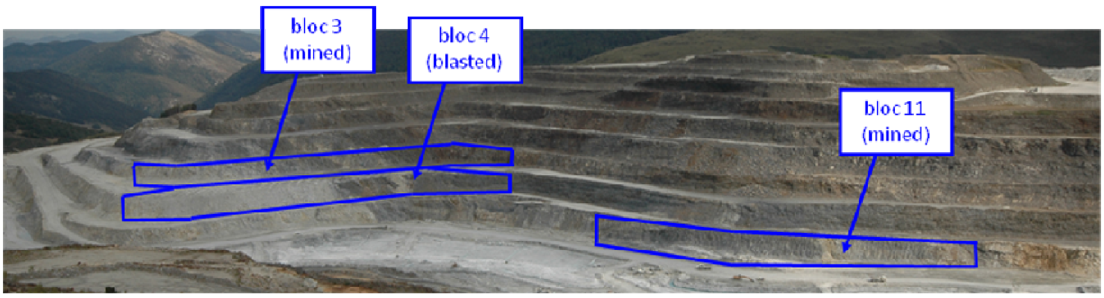


Figure 7.2: Location of investigated *blocs 3, 4* and *11* in the hanging wall (2 of 2)

[62] rtm_plan_exploitation_2009.ppt, pp. 23 of 70

Bloc 3 (1,684 to 1,669 m above sea level) was situated in the North part of the hanging wall. Drilling and blasting were done in the end of July and beginning of August, loading and hauling in August 2009. *Bloc 3* consisted mainly of dolomite or schist both pure and mixed. In the South some schist with marble bloc intrusions occurred which resulted in a re-blast. The bloc's width and therefore the number of rows blasted decreased from North to South. Most of the material was transported to *Vers Sud Nord* and some of the dolomite to a stockpile close to the dolomite crusher. The mined volume of *bloc 3* was estimated with around 122,400 m³, but actually 134,500 m³ were transported. [63]

Bloc 4 (1,669 to 1,655 m above sea level) was drilled and blasted in August 2009. Despite planned excavation of *bloc 4* in September, mining was not possible because of a breakdown of the excavator *R 994 B* and extra mining activities on *bloc 11*. Only some of the dolomite – former, finer blasted material from tests for the production of granulates – was removed via wheel loader *C 997 G* from this bloc. Due to its location directly beneath *bloc 3, bloc 4* appears to have a similar geology but without any marble blocs.

Bloc 11 (1,655 to 1,645 meters above sea level) was not mentioned in the *Plan d'Exploitation / Short Term Mine Planning 2009* and therefore no value for planned extracted volume was available but the removal of around 122,400 m³ was necessary to provide better access to the talc in 2010. *Bloc 11* consisted mainly of schist (top) and dolomite (bottom) and some marble at the South. All material was transported to *Vers Sud-Sud*. Due to close contact with talc in the toe area no sub-drilling was used while blasting. [64]

[63] calc_prod_bloc_09_01.xls, calc_prod_bloc_09_ 02.xls; rtm_plan_exploitation_2009.doc, p. 10 of 50

[64] calc_prod_bloc_09_01.xls

7.1 Drilling and blasting

7.1.1 Drill time

Drill time was measured for boreholes drilled in the hanging wall's dolomite, schist and marble at the beginning of October 2009. Boreholes were drilled inclined with 0, 5 and 10 ° and had a depth of 16.0 m. Side spacing between boreholes was 7.0 m, the 1st row was drilled as close as possible to the crest without removing the safety berm (2.0 – 3.0 m) and the 2nd row with a burden of 5.4 m. Besides the time for drilling, additional activities before, while and after boring were documented. Furthermore average values for pressure (receptor, working, rotary and thrust) and speed of the rotary head were recorded. [65]

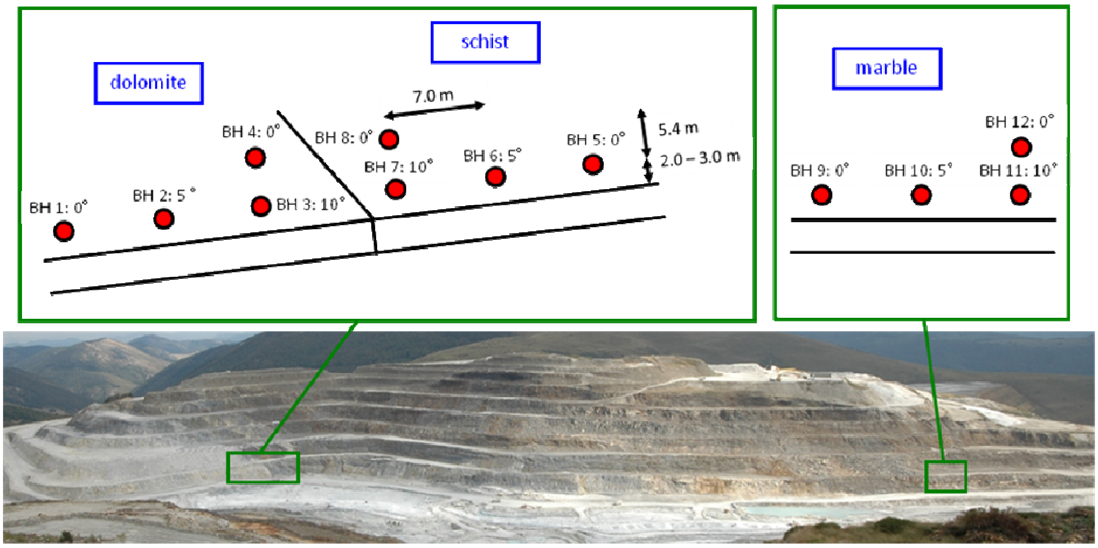


Figure 7.3: Location and position of the test holes for drill time and opening time and water filling measurements

[65] calc_time_drill.xls

7.1.2 Opening time and water filling of boreholes

In order to drill in advance or to make bigger blasts it is necessary to evaluate the boreholes' ability to stay open without significant changes of depth and a minimum rate of water filling. The boreholes drilled to measure specific drill time (see **7.1.1**) were object of further measurements concerning their opening time and water filling. These boreholes were measured via tape and torch to document changes in depth and eventually water level over a period of two rainless weeks at the beginning of October 2009. To avoid drill cuttings falling into the boreholes, each hole was shovelled free after drilling. [66]

7.1.3 Charge time

The time to prepare and actually fill a borehole with explosives including additional activities was estimated at the end of September 2009 for a special one-row blast designed to provide a good fragmentation easy loadable by the wheel loader *C 992 G* and then converted to traditional drilling and blasting. The charging process started with the positioning of the truck and the unloading of bags, cartridges and detonators, followed by continuity, depth and water measurement. Afterwards the detonator was connected with the bottom cartridge of *EMULSTAR* and lowered down the hole followed by another cartridge. Then *ANFOTITE* was poured including a last depth measurement before stemming the hole. [67]

[66] calc_survey_holes.xls

[67] calc_time_charge.xls

7.1.4 Test blasts

Main goal of these experiments was to increase the distances for burden and spacing intentionally risking a re-blast and therefore find the optimum drill and blast pattern according to geology. Furthermore the exclusive use of vertical boreholes even in the front rows and their negative effect on later loadability should be examined. Test blasts with varying blast parameter (geometry and amount of explosives) were done on *bloc 3, 4* and *11*, which were later object of load and haul measurements (see **7.2.1**). Prior to drilling and blasting the geology was roughly estimated – difficulties due to face covered with loose rocks from removal of the safety berm – and the pattern adapted to it. Following figure gives an overview about the geology and the planned drill and blast parameter for *bloc 3, 4* and *11*.

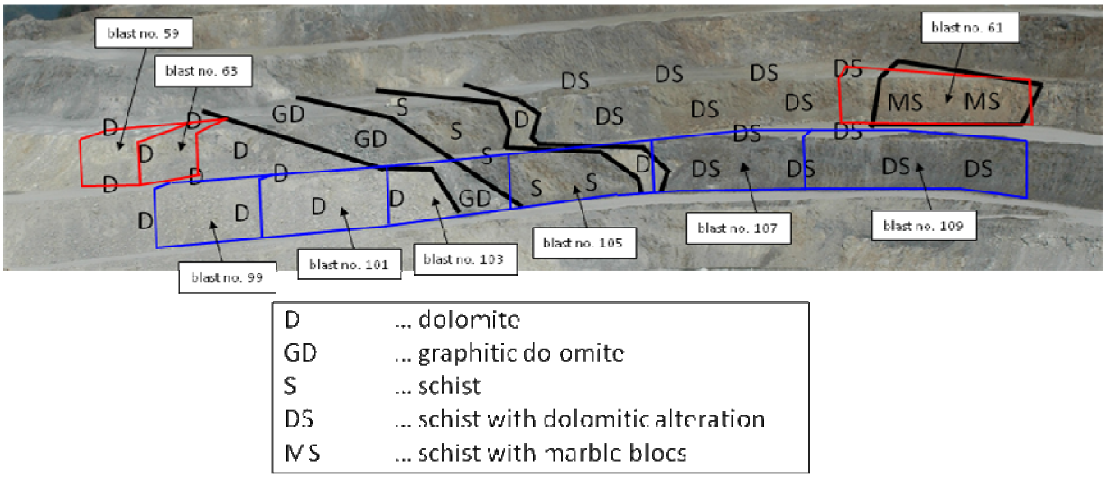
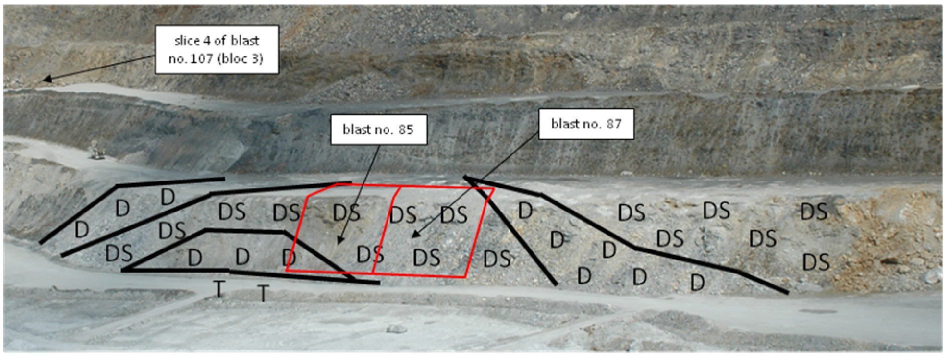


Figure 7.4: Test blasts on *bloc 3* (blast no. 59, 61 and 63) and *bloc 4* (blast no. 99, 101, 103, 105, 107 and 109)



D	... dolomite
DS	... schist with dolomitic alteration
T	... talc

Figure 7.5: Test blasts on *bloc 11* (blast no. 85 and 87)

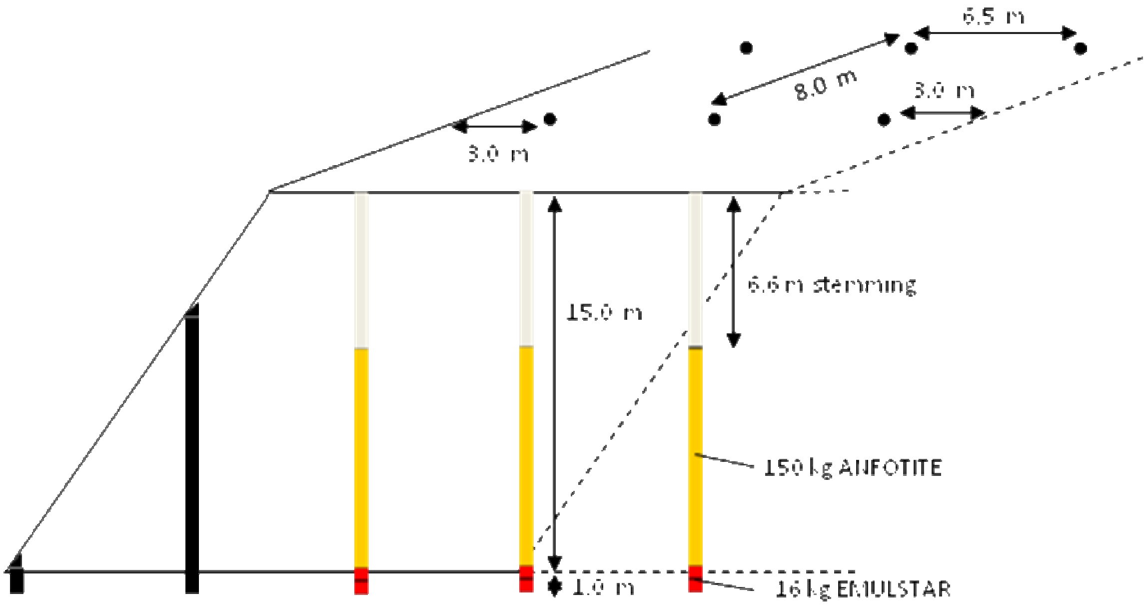


Figure 7.6: Use of vertical boreholes and a maximally increased burden and spacing

title: summary of planned drill and blast parameter on bloc 3, 4 and 11

bloc		3		11		4		
blast no.		57, 63b, 65, 67, 69, 71	59, 61, 63a	77, 81, 83	85, 87	95, 97	99, 101, 103	105, 107, 109
blast scheme		RS	JT	RS	JT	RS	JT	
material		dolomite, schist	dolomite (& schist), schist & marble	schist (top) & dol. (bottom)	schist (top) & dol. (bottom)	dolomite	dolomite	(dolomite &) schist
diameter	[mm]	165		165		165		
burden to face	[m]	2.0	3.0	2.0	3.0	2.0	3.0	
burden	[m]	5.4	6.5	5.4	6.0	5.4	6.5	6.0
spacing	[m]	7.0	8.0	7.0	7.5	7.0	8.0	7.5
inclination [1]	[°]	10 / 5 / 0	0	10 / 5 / 0	0	10 / 5 / 0	0	
bench height	[m]	12.6 - 12.9	12.8 - 12.9	13.5	14.0	15.0	14.6 - 15.0	14.3 - 14.8
sub-drill	[m]	1.0		1.0			1.0	
ANFOTITE	[kg]	130 - 133	150	140	150	155	150	150 - 163
EMULSTAR	[kg]	8	16	8	16	8	16	8 - 16
stemming	[m]	5.4 - 5.5	3.8 - 3.9	5.7	5.0	6	5.6 - 6.0	5.2 - 5.8
powder factor	[kg / m³]	0.289 - 0.291	0.249 - 0.247	0.290	0.263	0.287	0.246 - 0.253	0.249 - 0.258
costs [2]	[€ / m³]	0.26	0.24	0.26	0.25 - 0.26	0.25	0.23	0.22 - 0.24

[1] 10 / 5 / 0: 1st row 10°, 2nd 5° and ≥ 3rd 0°

[2] price per unit: 0.77 € / kg ANFOTITE, 2.21 € / kg EMULSTAR, 6.81 € / NONEL-detonator

Figure 7.7: Summary of planned drill and blast parameter on *bloc 3, 4* and *11* [68]

[68] mail_contrat_exp_2009.pdf, all drill and blast documentation by R. Sarda (rtm_b11_77_1108.xls – rtm_b4_109_1009.xls)

All blasts were documented and later analysed. To measure the geometry of bench faces *BlastMetriX3D* by 3G Software & Measurement GmbH was used. It is a system based on metric 3D imaging system (marked stereo photogrammetry plus computer vision) for contact-free measuring and a software for planning a blast (drill plan). The imaging equipment consisted of a calibrated digital SLR (single lens effect) camera, two so-called delimiters and two range poles for providing scale. The delimiters were arranged near the edge of the bench face and the range poles at the bottom level to indicate the area to blast. With the camera two pictures of the bench face to survey are taken from different standpoints (stereoscopic image pair) which compute a three-dimensional image. *BlastMetriX3D* models and boreholes (occasionally measured via torch and tape) were referenced via total station and evaluated (real burden, sub-drilling and specific charge) for documental reasons. [69]

BlastMetrix3D is originally designed to plan and not to document a blast as it is the case here. Furthermore the pictures should be taken in a moderate distance from the bench face to provide a good image section which due to the short bench width was only possible for *bloc 11*. Two solutions with non optimum distances were: pictures from the floor level (maximum distance 10 m) and from the other site of the bench (200 m). In the first case models had to be merged together to picture the whole blast site (*bloc 3*). The second solution allowed the rough documentation of a whole bloc (*bloc 4*) if the processing of pictures was possible.

[69] BlastMetriX3D 2007, pp. 3-5, 10 of 15

7.2 Loading and hauling

7.2.1 Load and haul measurements

To determine specific times and actions of the load and haul process intense studies were done on-site in summer 2009. Measurements via stop watch were taken at load site, from the inside of a truck or at the waste dump. The observation done at load site was either from the bench above or directly from the level of loading if enough space was available. Data were achieved especially for loading activities on *bloc 3* and *11* for areas which were drilled and blasted in the actual and a modified pattern with main focus on the *R 994 B*. Summaries of the via the *Excel VB* macros (detailed description in **8.1.1**) processed data can be found in **12.2.1**. – raw data and the calculation of each observation on the attached CD [70].

Each measurement was evaluated independently and then compared per bloc (and measuring point) before estimating average values for each parameter. Only values for optimum loading and hauling, excluding activities at shift change, before and after breaks or machine break down, are presented.

[70] calc_bloc3_load.xls, calc_bloc3_truck, calc_bloc11_load.xls, calc_sum_load_haul.xls, calc_trench_dump.xls

7.3 Auxiliary equipment

7.3.1 Push time

To evaluate the possible time deviation for pushing piles consisting of one or more stockpile, trucks created three different stockpiles consisting of 1, 5 (in a row) and 10 (2 times 5 in row) loads of non-blasted, unclean talc from the trench on the waste dump *Vers Sud Sud*. Then the time was measured for the stockpiles to be pushed over the crest by the bulldozer *PR 764*. The whole process was repeated three times. [71]

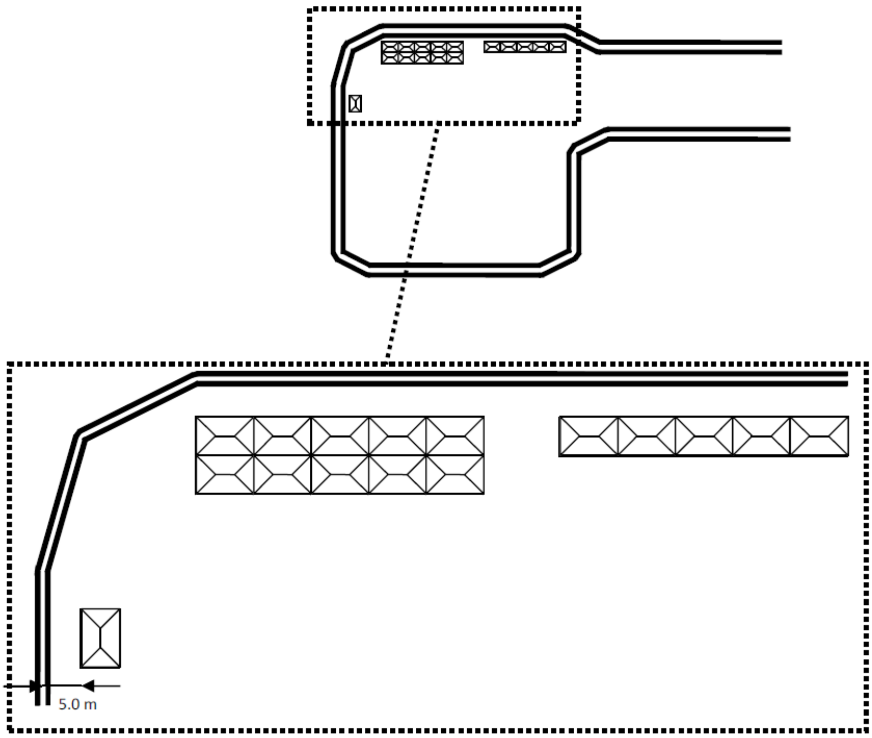


Figure 7.8: Sketch of push tests at dump site

[71] calc_push.xls

8 Calculations

This chapter provides information about more complex mathematical evaluations.

8.1 Loading and hauling

8.1.1 Load and haul measurements

Due to the complexity of the data from load and haul measurements it was necessary to write *VB* macros in *Excel*. Depending on the location where the data has been achieved different information is available and can made calculation with. All three macros including additional commentary can be found on the attached CD [72] – module 1 *load_site* consists of 1,523, module 2 *on_truck* of 543 and module 3 *waste_dump* of 956 lines.

In general all values were calculated as described in **5.1.2**. Because of truck sequence changes (e.g. breakdown, break or shift change) it was necessary to exclude load and haul cycles which were not representative for further calculations and so there therefore a lower limit of 500 s and an upper of 1,500 s were introduced.

8.1.2 Number of trucks

The necessary number of trucks to haul overburden material from the load to the dump site is primary depends on the loader's and truck's capacity and their cycle time. Following equations for the time per cycle (cyc) and the number of trucks (n_T) will be used in further calculations, both in the actual and also the modified approach [73]:

[72] calc_macro.xls

[73] calc_no_trucks.xls

$$cyc = \frac{60}{prod} \cdot cap$$

$$n_T = \frac{cyc_T}{cyc_L} \cdot \frac{cap_L}{cap_T}$$

The main differences between the actual and the modified calculations are the diverse fixed values and another determination of the haul and return time.

For the actual calculation fixed values besides capacities (cap_L, cap_T) is the planned hourly production of the loading unit ($prod_L$ = 600 m³ / h). The haul and return cycle of a truck (cyc_T) is calculated in the *Plan d'Exploitation / Short Term Mine Planning* with this equation:

$$cyc_T = cyc_L \cdot \frac{cap_T}{cap_L} + \frac{2 \cdot d}{v} \cdot \frac{60}{1.000} + rev$$

For this modified scheme of calculation is the average value of the following two equations the load time per bucket (cyc_L) [74] :

$$cyc_L = \frac{1}{60} \cdot \bar{x} \left(\frac{time\ per\ load\ cycle}{number\ of\ buckets} \middle| \left(time\ per\ bucket + \frac{hang\ time\ (excl.\ 1^{st}\ bucket)}{number\ of\ buckets} \right) \right)$$

Via the load time per bucket (cyc_L) the theoretical load performance ($prod_L$) which is minimised by additional breaks of the loader operator, use of dozer or water truck on the load site, longer break due to refuelling or maintenance, building of access ramps, etc. is determined. A more detailed calculation of a total cycle based on actual load and haul time studies has been developed by the author.

$$cyc_T = cyc_L \cdot \frac{cap_T}{cap_L} + \frac{2 \cdot d}{v} \cdot \frac{60}{1.000} + rev + D$$

[74] terms originate from load and haul time studies, calc_sum_load_haul.xls

9 Analysis and proposals

This chapter provides analyses and proposals concerning done measurements and calculations.

9.1 Drilling and blasting

9.1.1 Drill time

All 12 holes were bored under ideal conditions. They have been drilled without any disturbances like collapsing of boreholes and therefore re-drilling was never necessary. In general it takes around 25.5 min to complete a borehole from driving and positioning of the drill rig until removing the rods and unparking it. Most time consuming was the drilling of 16 m itself with around 17.6 min. [75]

time [min] for drilling & additional activities		
before drilling	drive	0.9
	park	0.5
	rig down	0.5
drilling	16 m	17.6
	continuity	0.9
	2^{nd} rod	1.4
after drilling	continuity [2]	1.3
	2^{nd} rod	1.0
	1^{st} rod	0.7
	rig up	0.4
	unpark	0.4
total [1]		25.5

[1] (drilling & add. activities)
[2] not really tested while drilling in marble → not included in average values

Figure 9.1: Time to complete a 16 m boreholes, incl. drilling and additional activities

[75] More information about each boreholes and not mentioned but also observed parameters can be found in calc_time_drill.xls and a summary in **Tab. 12.1**

As expected net time per borehole was the highest in hard marble with 25 min for drilling a 16 m deep hole and the shortest for schist with 12 min. To assign the period for additional activities around 7.5 min have to be added to the net drill time. The big time deviation between different rock types results from the low data available to be compared.

drill time [min] per geology					
rock type		dol.	schist	mar.	av.
drilling	per 1 m	1.0	0.7	1.6	1.1
	per 16 m	15.9	11.9	25.2	17.6
total (incl. add. activities)		27.8	18.9	28.5	25.1

Figure 9.2: Drill time according to geology [76]

There are big difference between the measured drilled meters per hour – 35 for dolomite, 51 for schist and 34 for marble – and the actual values provided by the company [77] – 29 for dolomite, 28 for schist and 29 for marble. This variation can be explained on by the optimum drilling conditions while measuring and the low number of observed boreholes. Furthermore the actual drilled meters per hour include values from the hanging wall where drilling difficulties can be expected – increasing the drill time up until 2 h per hole according to the drill rig's operators.

[76] dol. stands for dolomite, mar. for marble and av. for average

[77] calc_drill_09.xls

9.1.2 Opening time and water filling of boreholes

According to the drill rig each borehole was drilled up to a depth of 16.0 m and measurements showed a standard deviation of 0.1 m shortly after drilling. Boreholes situated in dolomite rock showed minimal changes in the borehole depth over a period of 2 weeks and no water could be observed. Boreholes in schist or schistose marble tend to lose about 0.3 m depth. Around 75 % of all boreholes in schist and every borehole in marble showed a presence of water, which is much higher than the observed 20 – 25 % at an average blast side in these areas. If there was water, it occurred on the 1st or 2nd day after drilling and after fast rise between the time of drilling and its measurement (from 0.0 up to 1.5 m) the water level stayed constant in schist and increased over time at a lower rate in marble (plus 0.3 m per day). In **12.2** more details about each borehole are provided.

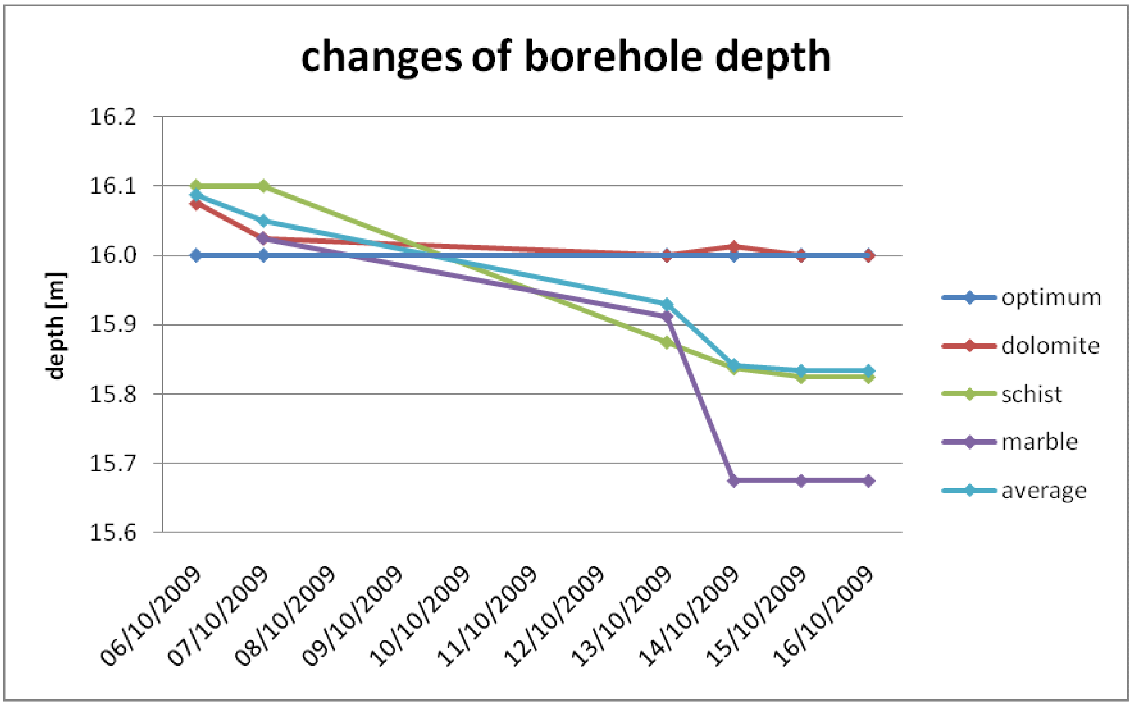


Figure 9.3: Measured changes and potential development of the borehole depth per geology [78]

[78] calc_survey_holes.xls

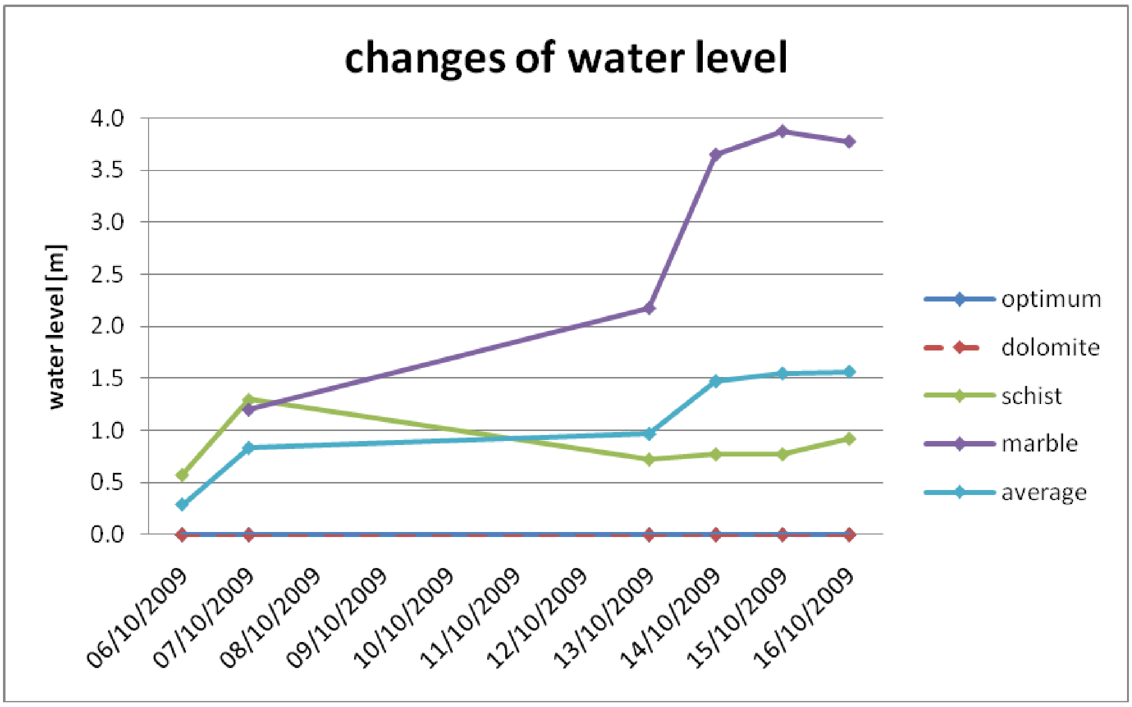


Figure 9.4: Measured changes and potential development of the water level per geology [79]

In order to drill in advance or to make bigger blasts it is necessary to evaluate the boreholes' ability to stay open without significant changes of depth and a minimum rate of water filling. Survey of boreholes over time showed, that there was not really a decrease of borehole depth or its continuity, but a possible rise of the water level (esp. in marble). Additional water would increase the time of blowing water out via the drill rig or the use of more expensive emulsion cartridges. In areas where water is normally not an issue like dolomite more drilling in advance and bigger blasts are not limited. For zones with possible water occurrence like schist or marble, smaller units for blasting seem to be more reasonable.

[79] calc_survey_holes.xls

9.1.3 Charge time

To charge a 16 m deep borehole with 155 kg *ANFOTITE*, one 8 kg cartridge of emulsion and 6 m of stemming 14 min are needed. This time decreases around 3 min if only 150 kg *ANFOTITE* but therefore 16 kg of *EMULSTAR* are used. The most time consuming activity besides unloading and measurements (both 3 min) is stemming of the boreholes (4 min) whereas filling the boreholes with explosives is faster (1 min for lowering a cartridge and 2 min for pouring anfo). Thus two persons working on-site are theoretically able to finish 20 holes in 3.7 h and 40 holes in 7.5 h. **Tab. 12.2** shows results of these hypothetical calculations. Physical effort due to the manual handling of the explosives' weight and fuel vapours will increase the time over the amount of boreholes filled.

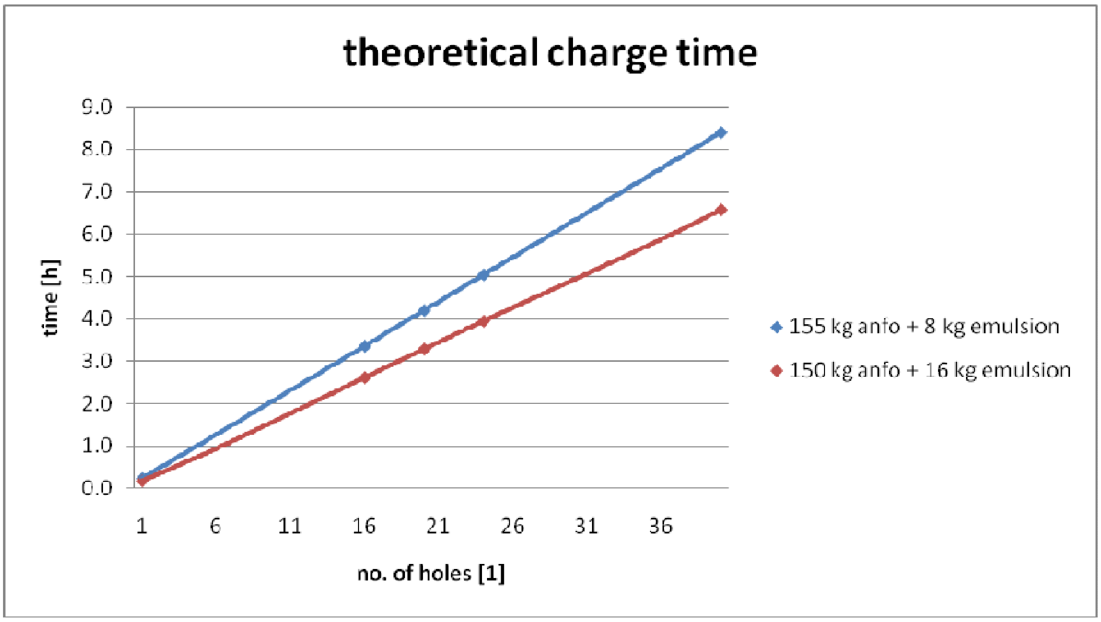


Figure 9.5: Theoretical charge time according to the number of holes [80]

Charging was done with a team consisting of two persons. If the daily drill target was finished midday the driller is helped to fill the boreholes as well. In order to fill more holes with explosives to make bigger blasts it is necessary to have more people available. With the arrival of the explosives truck the manual effort and therefore the charging time per hole should decrease.

[80] calc_time_charge.xls

9.1.4 Test blasts

title:			summary of drill & blast documentation										
bloc			3			11		4					
blast no.			59	61	63a	85	87	99	101	103	105	107	109
no. of holes			24	16	13	22	19	24					27
material			dolomite	schist & marble	dolomite & schist	schist (top) & dolomite (bottom)		dolomite			dolomite & schist	schist	
diameter	real	[mm]	165			165		165					
burden to face	real	[m]	3.0			3.0		3.0					
	plan	[m]	2.9	4.6	4.8	3.2	3.2	3.5	3.3	4.3	4.1	2.8	1.7
burden	real	[m]	6.5			6.0		6.5			6.0		
	plan	[m]	6.4	6.8	6.0	5.6	5.9	6.5	6.7	6.4	6.1	5.8	5.8
spacing	plan	[m]	8.0			7.5		8.0			7.5		
inclination	plan	[°]	0	0	0	0	0	0	0	0	0	0	0
	real	[°]	0	0	0	1	0	1	0	0	0	0	2
bench height	real	[m]	12.9	12.6	12.9	14.0	14.0	15.0	14.6	14.6	14.8	14.7	14.3
	plan	[m]	13.0	12.6	13.0	15.5	15.5	14.7	14.7	14.7	14.7	14.7	14.7
sub-drill	real	[m]	1.0			1.0		1.0					
	plan	[m]	1.1	0.7	0.8	-0.3	-0.5	0.6	0.3	0.9	1.0	1.1	0.7
ANFOTITE	plan	[kg]	150			150		150				163	
	real	[kg]	150			150		150				163	157
EMULSTAR	plan	[kg]	16			16		16				8	
	real	[kg]	16			13	16	16				8	
stemming	plan	[m]	3.9	3.6	3.9	5.2	5.0	6.1	5.7	5.6	5.9	5.6	
	real	[m]	4.5			5.5	5.6	6.1					
powder factor	real	kg / m³	0.250	0.243	0.267	0.252	0.242	0.216	0.210	0.220	0.248	0.266	0.259
costs [1]	real	[€ / m³]	0.24	0.24	0.24	0.24	0.25	0.20	0.21	0.21	0.24	0.23	0.23

[1] price per unit: 0.77 € / kg ANFOTITE, 2.21 € / kg EMULSTAR, 6.81 € / NONEL-detonator

Figure 9.6: Summary of drill and blast documentation for *bloc 3, 4* and *11* [81]

[81] calc_bm_sum.xls

The comparison of planned and real values received from measurements showed only little deviation of around ± 0.5 m due to adaption of the planned pattern on-site, e. g. use of inclined boreholes to break massive toe burden of the front row (blast no. 109), mixture of actual and modified drill and blast geometry (blast no. 59), easer boreholes in the front and last row (esp. blasts on *bloc 4*) or use of more *EMULSTAR* due to water presence in the borehole (blasts no. 63a and 103) (see **Fig. 12.1**). Powder factor and explosive costs per cube were recalculated with real values, like burden, bench height, sub-drill and stemming. The average powder factor used for the test blasts was 0.247 ± 0.017 kg / m³ for the test blast pattern (see **Fig. 12.2**) and varied between different rock types due to the use of either 1 cartridge of *EMULSTAR* and therefore approximately 12.5 kg additional *ANFOTITE* in weaker formations or 2 cartridges and less *ANFOTITE* in harder ones. Despite increased burden and spacing the real powder factor was lower than the one achieved via actual drilling and blasting (0.287 kg / m³). Explosive costs per cube were around 0.24 € for the test and 0.26 € for the actual blasts (see **Fig. 12.3**). Calculations, data extracted from *BlastMetriX3D* and some additional tables can be found in calc_bm_sum.xls.

9.2 Loading and hauling

9.2.1 Load and haul measurements

Results of load and haul measurements are summarized first and then split up into location and measurement. The following table provides a general overview of the main results achieved from all done load and haul measurements which were later used to modify the calculation scheme for number of necessary trucks (**9.2.2**). Values for *bloc 3* originate from observations done at load site and on truck, for *bloc 11* at load site, for the trench at dump site. Due to different loading conditions on the trench (e. g. un-blasted and wet material, load site partially under water, fewer trucks in use because of time extensive loading) mainly time concerning specific activities and not the whole haul and return process should be taken into consideration.Main results and diagrams of these load and haul measurements can be found in **12.2** and raw data and calculations of each observation at the attached CD [82].

In general, 4.1 buckets à 30 s were necessary to complete a truck load. The average time for the *R 994 B* to complete a cycle of hanging and loading or time per load cycle was 2.9 min. A truck spent around 3.0 min at load site, for reversing, queuing, being loaded and leaving and 1.1 min at dump site for reversing and discharging. Most time consuming was moving – leaving, haul and return – with 73 % of the total haul and return time, followed by loading (10 %). Around 6 % of the total haul and return cycle is waiting which is mainly queuing upon arrival at load site (71 %)

[82] calc_bloc3_load.xls, calc_bloc3_truck, calc_bloc11_load.xls, calc_sum_load_haul.xls, calc_trench_dump.xls

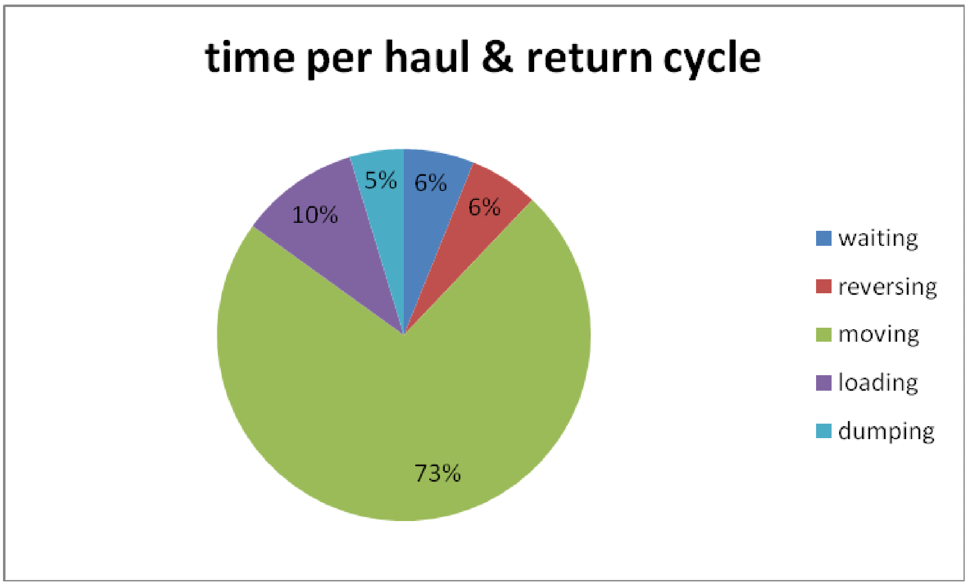


Figure 9.7: Time per haul and return cycle

Theoretical loading and haul performances were calculated based on the number of haul and return cycles because those have been the most reliable ones. E. g., if the theoretical loading performance be evaluated using the number of load cycles per hour, buckets per truck and loader capacity, the value would have been extremely high (ca. 1,050 m³ / h) and not realistic. Possible reasons for that are that not all and especially the last bucket could not achieve 12.5 m³, generally the time per load cycle does not include delays due to use of auxiliary equipment or breakdown of used load and haul equipment. Comparing the theoretical loading and haul performances to the actual production rate [83] a good correlation and only little deviation can be seen.

There were no signifiant variations in time due the use of a different drill and blast pattern, the geology, or the progress of loading on a bloc. It was not possible to establish trends, e. g. on *bloc 3* the top slice was the most time consuming to load and on *bloc 11* it was the bottom slice. This leads to the conclusion that neither the changed drill and blast pattern or material changes or to the postion of loading (top or bottom slices) resulted in a de- or increase of partial loading times.

[83] rtm_haul_bloc_0809.xls, rtm_load_bloc_0809.xls, log_b3_sum_exc.pdf, log_b3_sum_truck.pdf, log_b11_sum_exc.pdf, log_b11_sum_truck.pdf

- Load and haul measurements at load site – *bloc 3*

4 buckets à 30 s were necessary to complete a truck load. The time per bucket was the highest at the bottom (33 s) and later nearly constant for all other slices. In general, the bucket's fill factor was around 2.5 (maximum for dolomite and schist and minimum for graphitic dolomite). As expected, normal loading had the highest fill factor of 2.6 and stone removal the lowest with 2.1.

The time per haul and return cycle depends on the haul and return distance and despite changes (± 300 m) due to the bloc's extend and the dump's advance no changes in values were observed – average haul and return time was 14.3 min. Its maximum (17.0 min) was found at slice 1 due to longer time at load site which was a result of more time intense reversing and bucket filling.

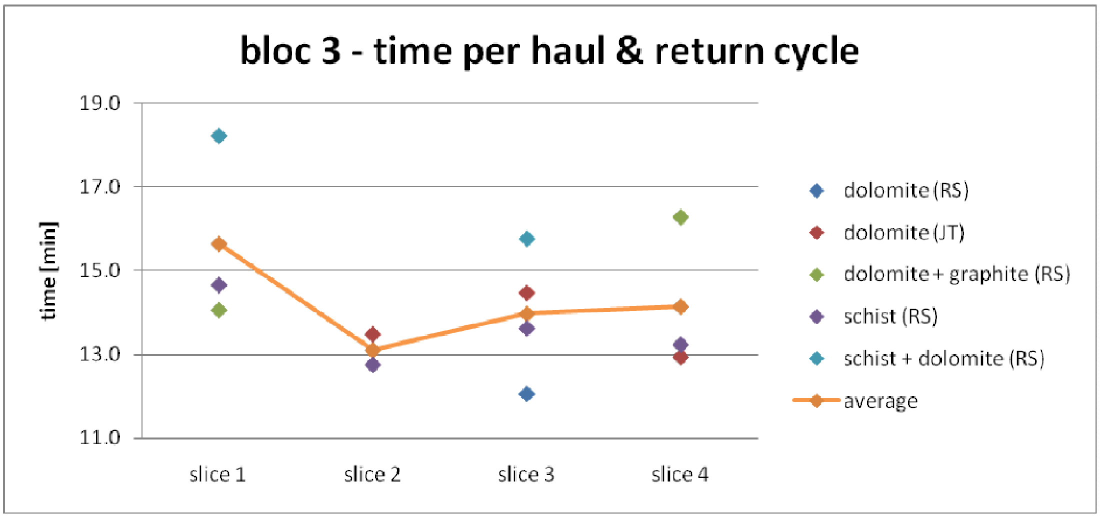


Figure 9.8: Bucket time per material, blast desgin and slice, *bloc 3* [84]

Around 71 % buckets were filled via normal loading, 21 % via ripping, 7 % via cleaning and 1 % via loading of stones. The percentage of normally loaded buckets slightly increaesed from top to bottom and ripping decreaesed. Maximum ripping (50 %) occured in schistose dolomite shortly before entering the dolomitic zone with marble inclusions.

[84] calc_bloc3_load.xls

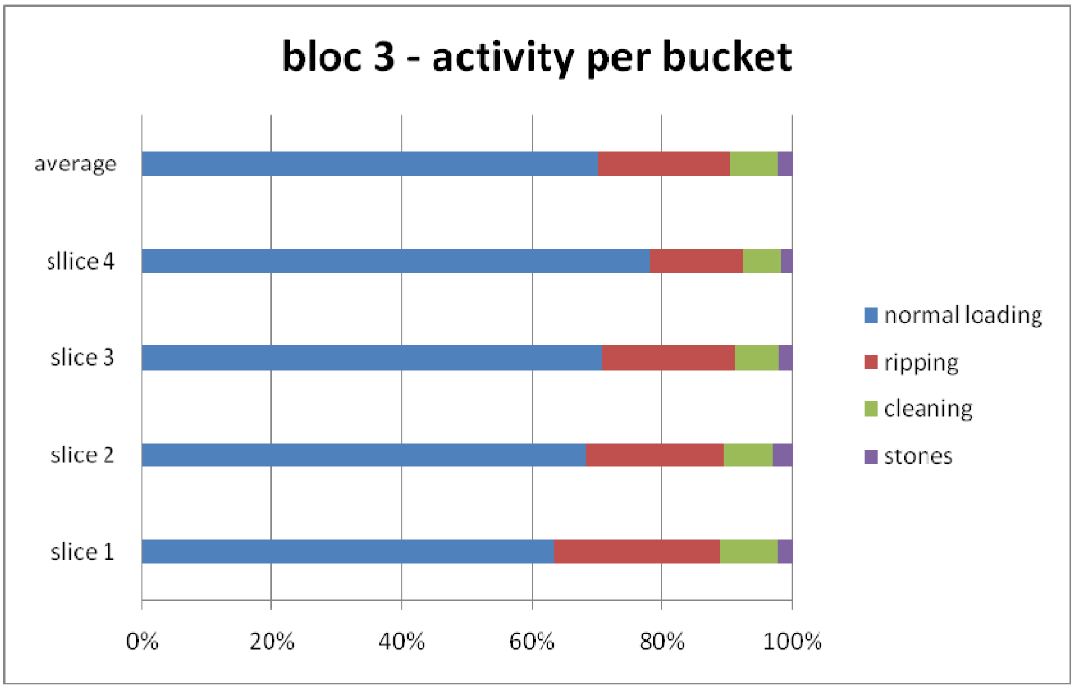


Figure 9.9: Bucket activity per slice, *bloc 3* [85]

The comparison between the *HD 985* and the *C 777D* trucks showed that the *HD 985* needs around 0.1 min longer to complete a load cycle and 0.3 minutes longer to finish a haul and return cycle. Using 4 trucks due to difficult loading conditions and a limited space at load site, the hang time was high and the queue time low – with 5 trucks the opposite occurred. The R 994 B was able to complete 18.2 load cycles per hour with 4 trucks and 2 more with one truck more available.

[85] calc_bloc3_load.xls

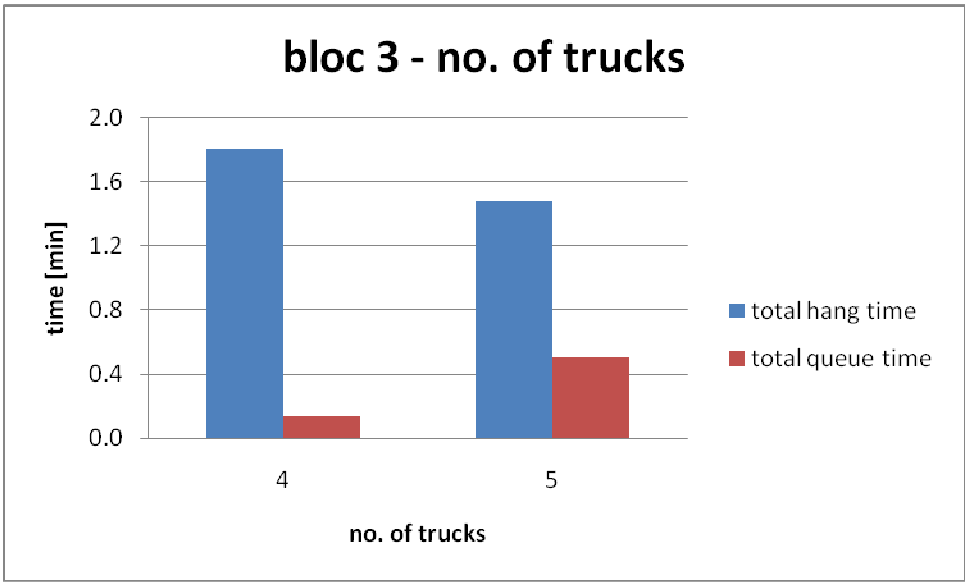


Figure 9.10: Number of trucks and their effect on haul and queue time, *bloc 3* [86]

Dolomite was the only area with the same geological conditions where blasts via actual and a modified drill and blast pattern were done. The average time per bucket differed around 2 s (RS: 28 s, JT: 30 s) and 10 % more ripping occurred in loading the material blasted by JT. Loading and hauling in schist had the highest fill factor (2.7) due to the fine material and easy loading conditions and therefore fewer than 4 buckets were needed to complete a truck load. Working conditions in schist and marble were very difficult – loader on the same level as truck, 180° swing, and narrow loa d site due to one row blast – especially after re-blasting the area. I. g. the time per bucket was 32 s before and 44 s after, the fill factor before 2.6 and after 1.8, and normal loading 49 % before and 68 % after re-blasting. Comparing values of pure and graphitic dolomite with those of pure and dolomitic schist no differences significant occured.

The described values were calculated excluding the results from load and haul in the schist and marble formation due to their big deviation. Tables and diagrams for demonstration can be found in **12.2.2** and calc_bloc3_load.xls.

[86] calc_bloc3_load.xls

- Load and haul measurements at load site – *bloc 11*

On *bloc 11* it took around 28 s to fill one of the 3.9 necessary buckets, which had an average fill factor of 2.7 despite loading in pure or schistose dolomite. A truck needed 11.0 min to complete a haul and return cycle and around a quarter of this time was spent at load site. Of the 2.8 min per load cycle one third was spent hanging and two third loading.

The time per bucket slightly increased from top to bottom and the change from schist and dolomite (27 s) to a pure dolomite formation (31 s). In the top slices the blast configuration by R. Sarda seemed to be a little bit better loadable (2 s less), in the bottom slices it was the opposite (5 s more).

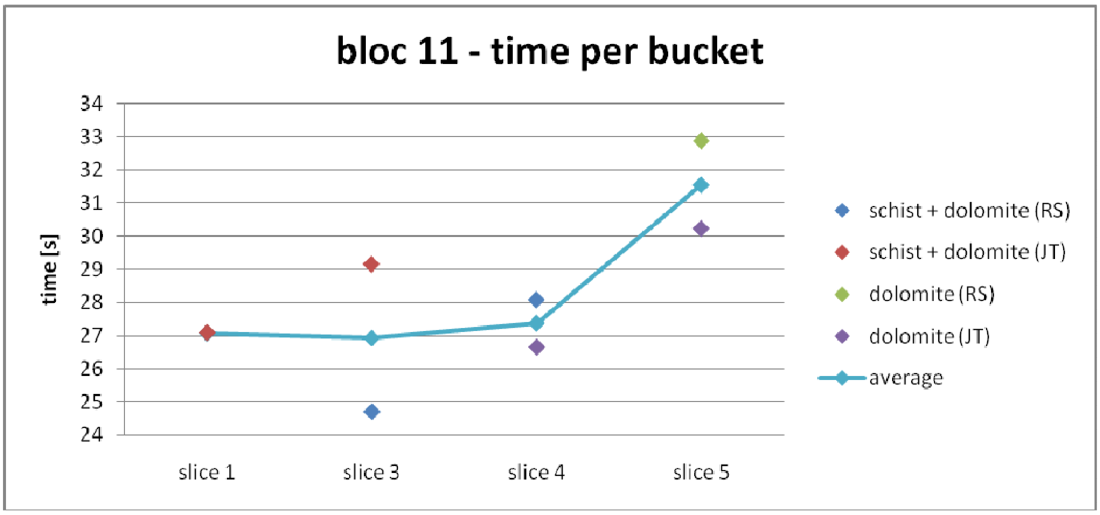


Figure 9.11: Bucket time per material, blast design and slice, *bloc 11* [87]

Minimum values for the time at load site (2.1 min) and time per haul and return cycle (9.7 min) occurred on slice 4 due to a high percentage of quick normal loading (95 %, 27 s per bucket) and a low queuing (0.2 min) and reversing (0.3 min). The loading of slice 5 were most time consuming, mainly according to bad loading conditions due to low working bench height.

[87] calc_bloc11_load.xls

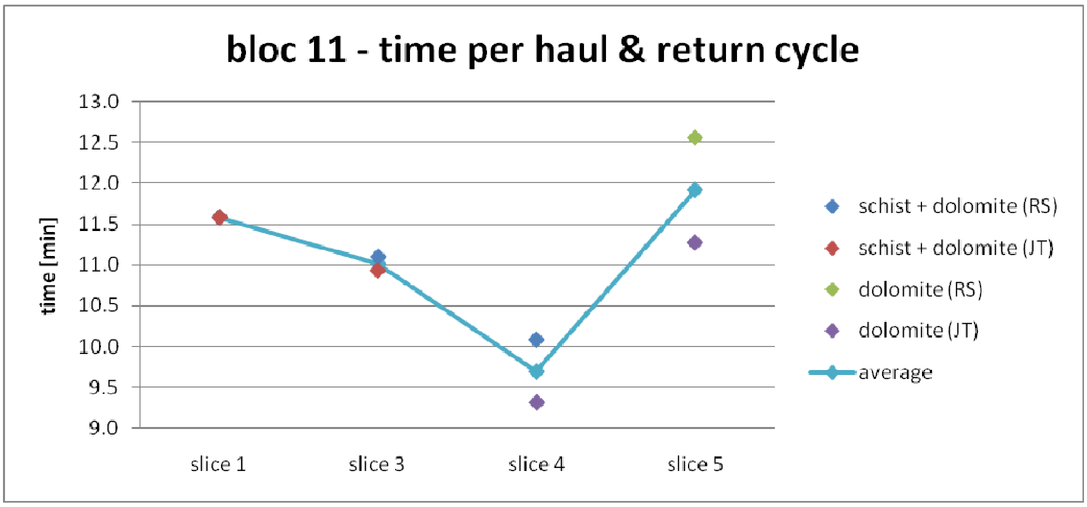


Figure 9.12: Total haul and return cycle time per material, blast design and slice, *bloc 11* [88]

For *bloc 11* most of the buckets were filled via normal loading (87 %) and esp. top material, schistose dolomite, was easy to load (93 %). The worst loading conditions occurred on the fifth and bottom slice with maximum ripping of 11 % and cleaning of 13 %. The appearance of stone loading (2 %) in bottom slices was equivalent with the finding of big boulders, which needed to be re-blasted (see **Fig. 6.7**).

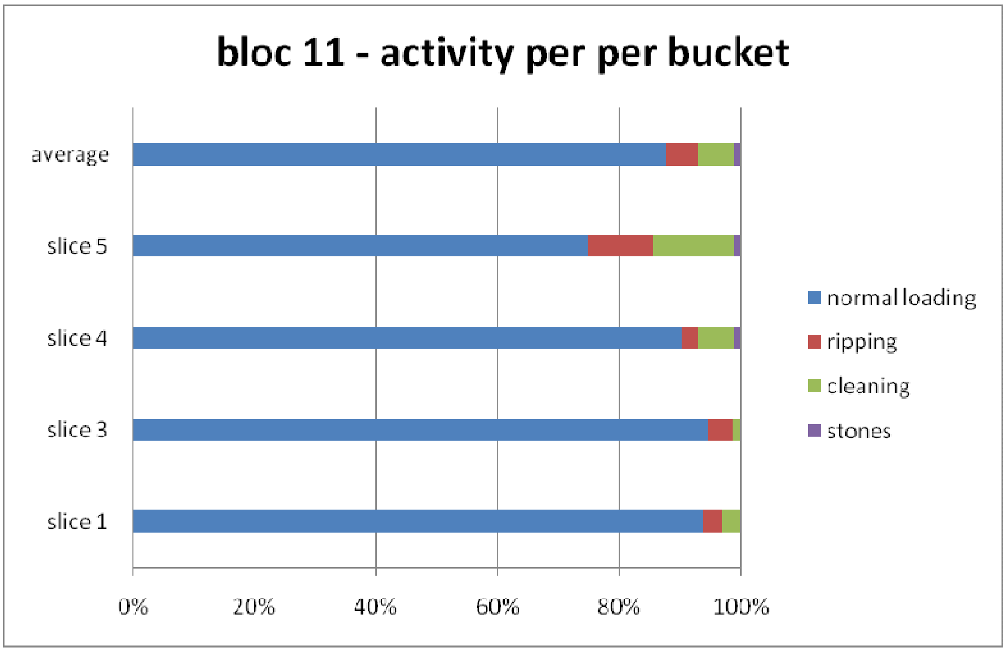


Figure 9.13: Bucket activity per slice, *bloc 11* [89]

[88] calc_bloc3_load.xls

The comparison between the *HD 985* and the *C 777D* trucks shows that the *HD 985* needed around 0.5 min longer to complete a load cycle resp. a haul and return cycle.

The loading performance of the wheel loader *C 997 G* was compared with the excavator *R 994 B*. As expected, loading itself was much more time consuming – 6 buckets à 63 s to fill one truck. Although the material was especially blasted to provide a better fragmentation, loading was difficult (72 % ripping). The low number of trucks in use completely eliminated queuing but increased hanging (3.8 min). To complete a haul and return cycle around twice as much time was necessary.

12.2.2 and calc_bloc11_load.xls provide more information regarding the discussed parameters.

- Load and haul measurements on truck – *bloc* 3

Load and haul measurements on truck confirm the results achieved from measurements at load site, like number and time per bucket, haul and return cycle. Again the values for schist and marble were excluded from determining average values. Additional parameters to complete the haul and return cycle and which were mainly independent of geology or drill and blast pattern were extracted as well, like timer per dumping (0.7 min), reverse time at dump site (0.5 min), average haul (22.4 km / h) and return speed (27.2 km / h).

To fill one truck, 4.2 buckets each 30 s were loaded and after 2.7 min a truck left load site. The time per haul and return cycle (13.6 min) was divided in three parts: time at load site (4 %), at dump site (10 %) and while driving (86 %). As expected, haulage was more time consuming than returning.

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[89] calc_bloc3_load.xls

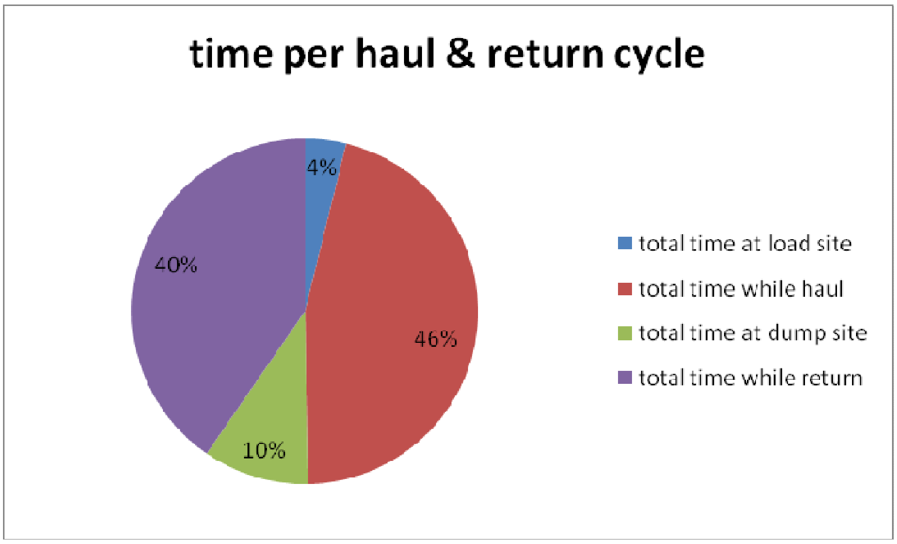


Figure 9.14: Time per haul and return cycle, *bloc 3* [90]

One bucket contained in average 26 t of material according to the truck's payload control unit, which was around 1.8 m³ less than the bucket's capacity of 12.4 m³. Confirmation of the changes from queue and hang time depending on the number of trucks in use could not be done due to minimum available measured opportunities. More information can be found in calc_bloc3_truck.xls and **12.2.3**.

- Load and haul measurements at waste dump – *trench*

These measurements were done to specify and confirm the main results from *bloc 3* and *11*, esp. values concerning discharging. Dumping took 0.6 min, reversing 0.4 min and one haul and return cycle 14.4 min. For details see calc_trench_dump.xls and **12.2.4.**

[90] calc_bloc3_haul.xls

9.2.2 Number of trucks

The very conservative value of $prod_L$ = 600 m³ / h used for actual calculations is around 100 to 150 m³ / h lower than the average actual production rate [91] and is about ⅔ of the theoretical possible production rate per hour. This led to a load time per bucket (cyc_L) which was around 50 % longer than the measured value. Dumping (D) seems to be included in the reverse time (rev) because it is equal to the measured reverse and dumping times.

Regarding the actual calculation the necessary number of trucks seems to be very high and is always rounded down. Despite that sometimes too many trucks are used for actual loading conditions and this result in queuing (see **Fig. 9.10**). Furthermore the average haul distance in *Plan d'Exploitation / Short Term Mine Planning 2009* is 500 m longer than in reality which increases the number of necessary trucks additionally.

Following table should visualize the different results between the actual and the modified calculation. To provide an hourly production of 600 m³ / h, via actual approach for a distance of 1,500 m between load and dump site 3.9 trucks are needed and for 2,500 m 5.4 trucks. Using the modified approach, only 3.3 and 4.9 trucks are required for the same distances.

[91] calc_prod_bloc _09_01.xls, calc_prod_bloc_09_02.xls

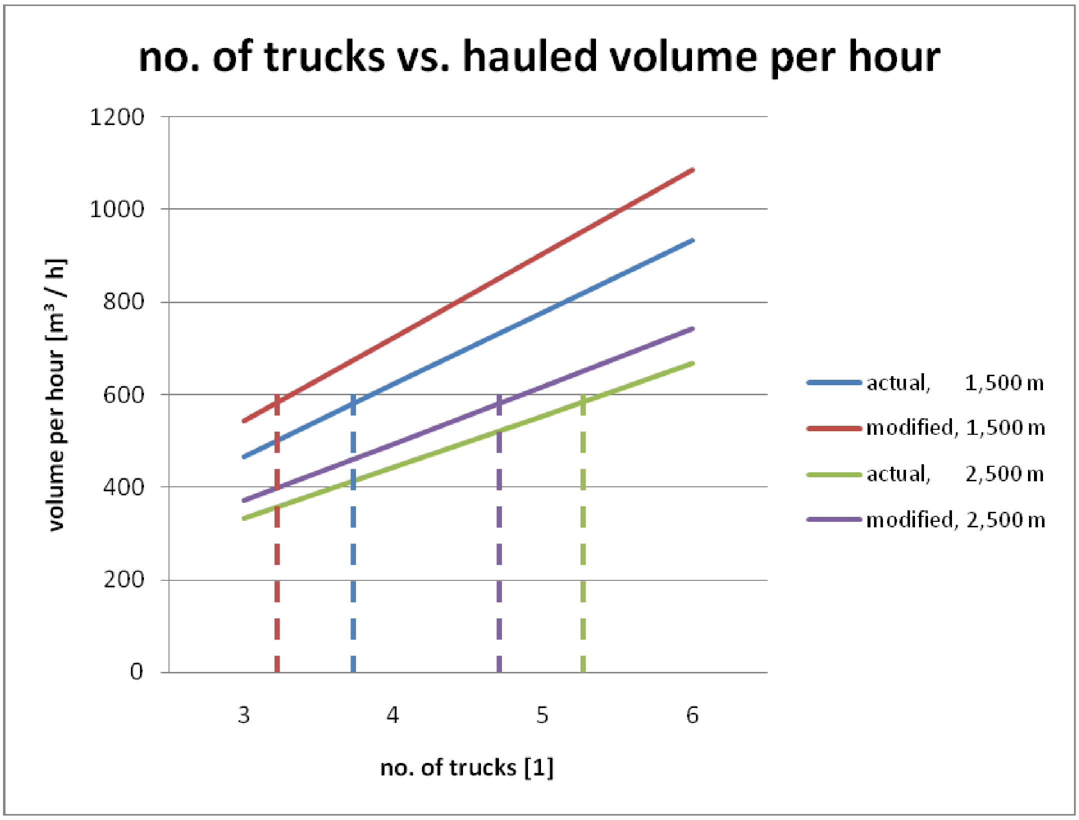


Figure 9.15: Number of trucks per hauled cubes, calculated via actual and modified approach

Besides using modified calculation, significant changes of the haul distance (e.g. different dump location, stretched bloc), real load and haul distances, changes of rock type and therefore different loading conditions and times should find recognition and could help to precise the number of necessary trucks. Furthermore narrow roads which do not allow two trucks to pass at the same time (see **Fig. 6.11**) and some extra waiting time has to be added to the haul and return cycle of a truck (cyc_T). Despite that feedback from operators on site is essential. It should be discussed if capacities given in the technical description are used instead of 12.4 m³ per bucket and 37.0 m³ per truck load.

The re-evaluation of *bloc 3* for different distances (planned 2,600 m and real 2,200 m) with both methods can been found in **12.2** or calc_no_trucks.xls.

9.3 Auxiliary equipment

9.3.1 Push time

Time measurement of different piles showed that there are no significant differences in the average duration of 2.8 min for pushing 1, 5 or 10 loads, only a big deviation appeared between minimum and maximum times for piles with 5 and 10 loads because limits between loads have not been clear during observation.

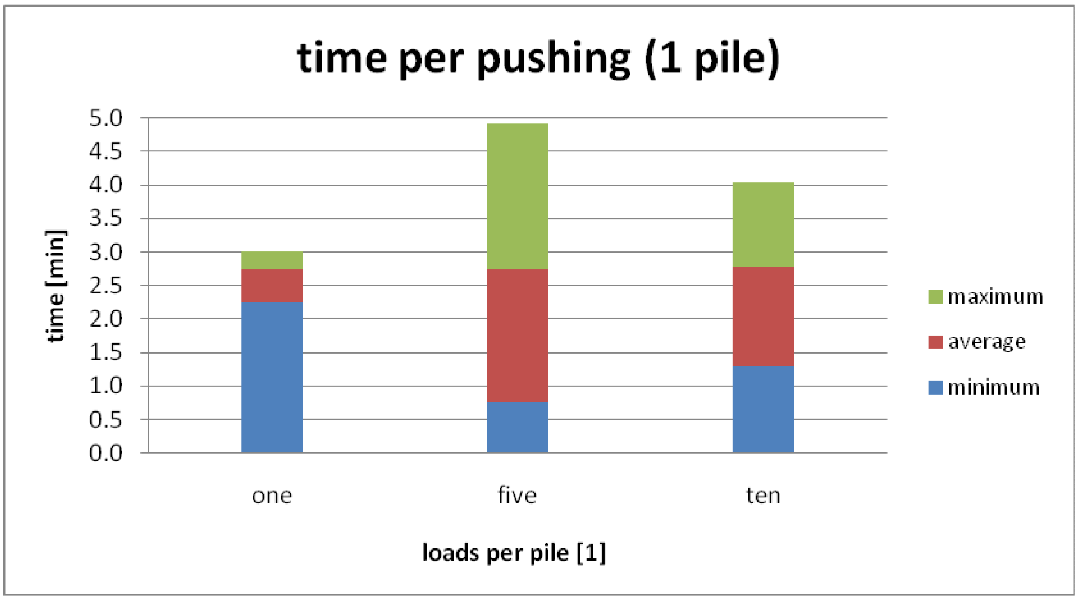


Figure 9.16: Measured push times at dump site for different pile loads

The same time for pushing can lead to the consideration to change the actual practised pushing process of waiting besides the discharge position and pushing of single loads but therefore dumping close to the edge. Summarised pushing activity would provide more time for other work like improvement of road conditions at the dump site. But this would make necessary that truck operators keep the safety distance while discharging because they are not waved into position.

10 Summary

This chapter summarises all findings and conclusions concerning the main topics defined in the introduction.

10.1 Change from inclined to vertical blast holes

The main argument to use inclined boreholes in the first two rows is the argument that it would not be possible to break the big toe burden. Measurements and observations have shown that vertical boreholes do not lead to excessive front row burden or boulders because there is already fragmented material from former blasts at the face. The use of vertical boreholes does not require a drill rig orientation perpendicular to the face and is therefore independent from position for set-up. Therefore small bench widths and uneven crest are not problematic any more. Less deviated boreholes will provide an optimum energy distribution of the explosives and tendencies for collapse in soft material can be worked against. Charging of vertical boreholes does not require a direct contact between emulsion cartridges and the hole's wall. The use of inclined boreholes should be limited to possible massive burden and the use of the front end loader for material removal.

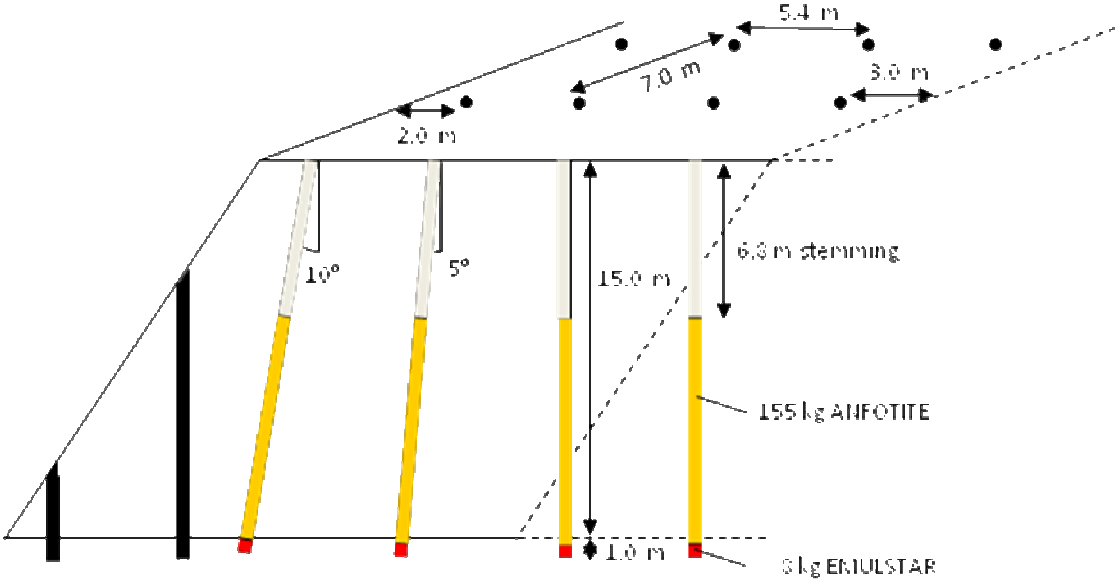


Figure 10.1: Sketch of the actual drill and blast pattern at *Trimouns*

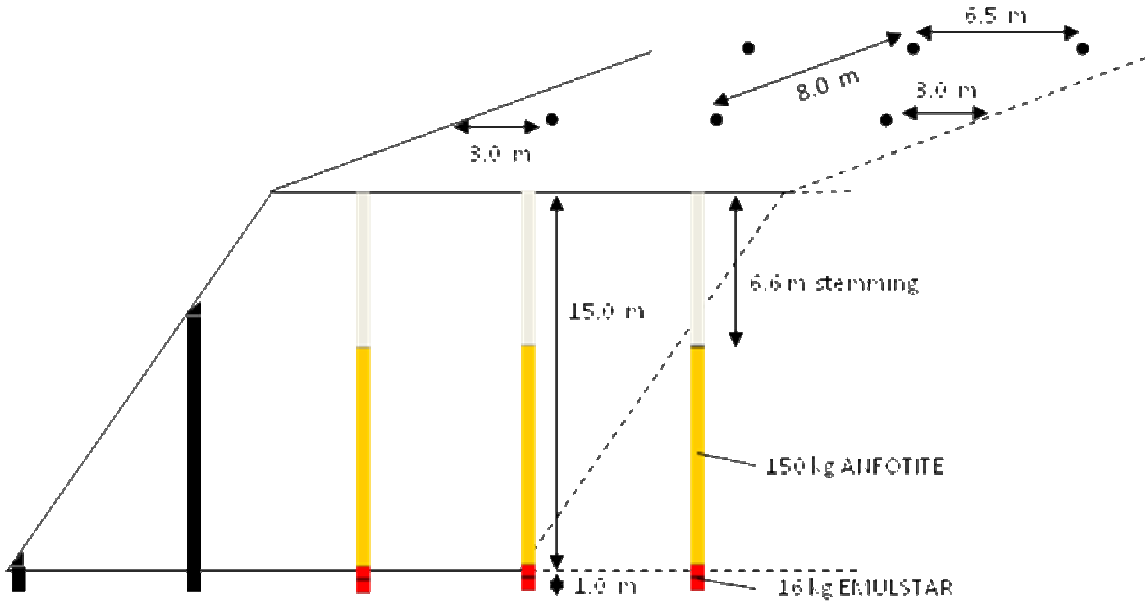


Figure 10.2: Proposed use of vertical boreholes including the recommended increased burden and spacing for dolomite

10.2 Increase of the drill and blast pattern (burden and spacing)

Tests for drilling and blasting and later the removal of this material have shown that the drill and blast pattern can be increased from 5.4 x 7.0 m in certain areas without decreasing the loading performance. It is possible to blast homogeneous material like pure dolomite with a spacing of 6.5 m and a burden of 8.0 m if 2 cartridges of emulsion are used without significantly decreasing the fragmentation (see **Fig. 10.2**). For mixed material like schistose dolomite or pure schist a grid of 6.0 x 7.5 m and the use of only one cartridge would provide good results. For problematic zones like schist with big dolomite or marble boulders an increased drill and blast pattern is not advised, eventually smaller distances and the use of intermediate stemming could avoid re-blasting.

geology		dolomite	schist (with dolomite)	schist with marble or dolomite blocs
burden	[m]	6.5	6.0	≤ 5.4
spacing	[m]	8.0	7.5	≤ 7.0
EMULSTAR	[kg]	16	8	≥ 8

Table 10.1: Summary of the proposed drill and blast pattern

The tested and proposed increase of burden and spacing and the change of the amount of explosives, result in general in a lower powder factor and therefore decreased explosives costs. Furthermore, fewer boreholes minimise time and costs for drilling and charging. E. g. to blast 1,000 m³ actually 27 holes and 287 kg of explosives are necessary. Using an increased pattern only 19 or 22 holes have to be drilled and fewer explosives (between 230 and 264 kg) have to be charged.

These recommendations are resulting from only 5 loaded test blasts on *bloc 3* and *11*. 6 further test blasts on *bloc 4* were not loaded due to changes of the work schedule, which did not allow the removal. Therefore it is strongly advised to verify possible changes in further measurements.

10.3 Increase of the blast size (holes and cubes per blast)

Blasts should be as big as possible to increase productivity due to minimisation of unproductive time. This would make drilling in advance necessary and requires more charging capacities. The opening time, occurrence of water in the boreholes and the time charging a borehole are limiting the number of holes per blast. Decrease of borehole depth or its continuity is not really an observed problem, but a rise of the water level could increase the time of blowing water or the use of more expensive emulsion cartridges. Therefore bigger blasts and drilling in advance should be mainly practiced in geological structure without water like dolomite and smaller blast units used for schist or marble where the occurrence of water is more likely. With the arrival of the explosives truck more holes per hour should be able to be filled with explosives with lower manual effort and without increasing the number of people working on site.

Mining activities would individually profit from this change. Close equipment would not need to be moved to a safe location and production can continue until and right after the break. E.g. the time for daily removal of the drill rig could be used to make at least one to two holes. Furthermore the effective drill time could be increased if there are no limitations to the number of holes and no need for the driller to assist charging. It would not be necessary to have someone guarding the load site after charging and blocking the pit while blasting every day. Fewer evacuations of the pit’s working personal would save additional driving and offer the possibility of spending the break on-site. The possible occurrence of delays due to complications while blasting would be minimised by fewer blasts.

10.4 Introduction of a systematic drill and blast planning approach

Due to the actual drill and blast pattern which is the same for all geological conditions a systematic drill and blast planning is not necessary, especially if acceptable results occur from these blasts. The use of a grid and amount of explosives adapted to geology would make a rough geological mapping of the actual blast side necessary which would be precise due to the experience and knowledge people have on site.

Nevertheless a general blast planning with e. g. *BlastMetrix3D* is not advised due to the difficulties experienced while using the system and especially with the current practise of daily blasting. Besides suboptimal working conditions, like small working benches or limited access to blast sides with bloc advance, too much time is necessary for the site's preparation and the processing problems resulting from those. Information receiving from *BlastMetrix3D,* like borehole or inclination, is not relevant because the bench height is established while determining the bloc limit and the use of the front rows' inclination is following a fixed scheme not depending on actual burden or inclination.

Some improvements concerning the blast documentation would be the increase of the data consistency between different sources. Additional information concerning specific drill and blast conditions, like changes of borehole depth or geology over one blast site, should be noted as well. Those could be used with properties of later process' steps like loading conditions which could actualise the blast result and help to determine a pattern for the occurrence of boulders to avoid them and necessary re-blasting. Furthermore this could help to establish the limit of a possible blast size increase.

10.5 Reduction the number of haulage trucks in use

The calculation of the necessary number of trucks via modified approach has shown that the required load performance should be generally possible to achieve with a lower number of trucks in use. The main target should be the elimination of all queue time (see **Fig. 10.3**) which allows the excavator to prepare material and / or even the load site without a truck waitin g. To know the needed amount of trucks further improvement like recognition of additional factors, e. g. changes in the haul distance, rock type and load conditions, narrow roads which lead to extra waiting time and increase of the haul and return cycle time and the operators' feedback should be used. The reduction of only one hauling unit would decrease the operating costs around 60 to 110 € / h.

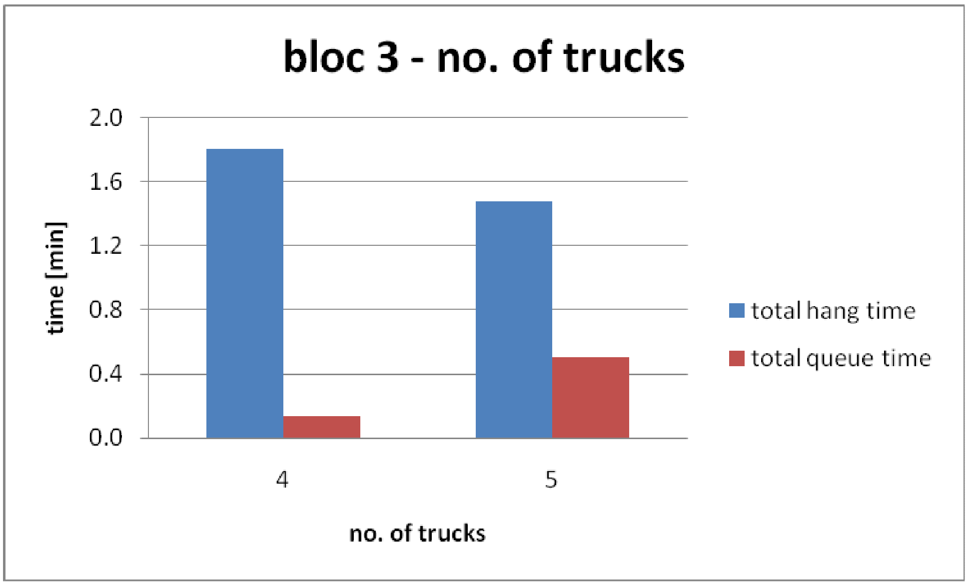


Figure 10.3: Number of trucks and their effect on haul and queue time, *bloc 3*

10.6 Reduction of dozer use

Before thinking of reducing the number of dozers – *D 275 A2* at load and *PR 764* at dump site – in use, improvements concerning actual work pattern should have priority and procedures should be evaluated. Despite the availability of dozing equipment, driving conditions on the load and dump site are bad, resulting in percussion while driving and decrease of speed. This increases equipment wear, esp. tires and suspension, and makes driving uncomfortable.

Actually the use of the bulldozer *D 275 A2* cannot be reduced due to the loader's inability and / or time to prepare the load site. A possible solution would be the decrease of the number of trucks in use to provide the possibility for the *R 994 B* to clean and even the load site. Nevertheless the use of an assisting dozer at load site cannot be avoided completely. Therefore dozing should be done if there is enough space possible to or in case of a narrow work bench as quick as possible or during break time that the actual load and haul is not disturbed.

Most of the time the *PR 764* is positioned perpendicular to the dump's crest, waits with running engine until a truck dumps its load and pushes each single load over the edge. This allows discharging closer to the crest but minimises the time for dozing activities at dump site. If more than one pile, as long as the loads are dumped in one line, is pushed, the time in between can be used effectively, e. g. for road maintenance.

Because neither the *D 275 A2* nor the *PR 764* was observed regularly operating with the ripping unit and all work is done by using the front blade, it should be evaluated if it is really necessary to have two track dozers on site. Because the ripping unit is not part of the general dozing process, at least one of the two track dozers could be replaced by a more flexible wheel dozer.

11 Appendix – Observations

11.1 Drilling and blasting

11.1.1 Data inconsistency

title: data inconsistency - drilling and blasting (bloc 3, 4 & 11)

	date	parameter	Sarda	Extraction	Logimine
bloc 3	27/07/2009	terrain	schist / marble	dolomite	dolomite
	27/07/2009	no. of holes	16	16	24
	28/07/2009	no. of holes	24	24	21
	30/07/2009	terrain	dolomite / schist	dolomite	dolomite
	30/07/2009	stemm height	4.5	4.4	4.4
	31/07/2009	no. of holes	27	30	27
	03/08/2009	terrain	shist	dolomite	dolomite
	03/08/2009	no. of holes	21	21	20
	04/08/2009	no. of holes	33	33	35
	04/08/2009	stemm height	5.5	5.4	5.4
	05/08/2009	no. of holes	47	47	48
	05/08/2009	stemm height	5.5	5.4	5.4
	07/08/2009	depth	12.9	13.9	13.9
	07/08/2009	stemm height	5.5	5.4	5.4
bloc 11	10/08/2009	no. of holes	n.s.	24	19
	11/08/2009	cubes	8,670	8,675	8,675
	11/08/2009	no. of holes	17	17	18
	12/08/2009	no. of holes	missing	18	16
	12/08/2009	anfo	missing	0	2,525
	13/08/2009	anfo	missing	0	2,525
	14/08/2009	diameter	165	0	0
	24/08/2009	no. of holes	22	22	25
	24/08/2009	stemm height	5.5	5	5
	24/08/2009	anfo	2,700	50	2,700
	25/08/2009	no. of holes	18	18	20
bloc 4	01/09/2009	zone	BLOC4	BLOC12	BLOC4
	01/09/2009	no. of holes	24	21	25
	02/09/2009	no. of holes	21	21	20
	03/09/2009	no. of holes	24	24	25
	04/09/2009	no. of holes	24	24	25
	04/09/2009	stemm height	6	5.6	5.6
	04/09/2009	anfo	3,600	0	3,600
	07/09/2009	no. of holes	24	24	25
	07/09/2009	stemm height	6	5.6	5.6
	08/09/2009	no. of holes	24	24	25
	08/09/2009	stemm height	6	5.6	5.6

Table 11.1: Data inconsistency for drilling and blasting (1 / 2)

	Sarda	Extraction	Logimine
ref.:	rtm_b3_57_2707.xls	mail_blasts_2009.csv	log_b3_57_2707.xls
	rtm_b3_59_2807.xls	mail_explosives_2009.csv	log_b3_59_2807.xls
	rtm_b3_61_2907.xls		log_b3_61_2907.xls
	rtm_b3_63a_3007.xls		log_b3_63_3007.xls
	rtm_b3_63b_3007.xls		log_b3_65_3107.xls
	rtm_b3_65_3107.xls		log_b3_67_0308.xls
	rtm_b3_67_0308.xls		log_b3_69_0408.xls
	rtm_b3_69_0408.xls		log_b3_71_0508.xls
	rtm_b3_71_0508.xls		log_b3_74_0708.xls
	rtm_b3_74_0708.xls		log_b11_87_2508.xls
	rtm_b4_95_0109.xls		log_b11_77_1108.xls
	rtm_b4_97_0209.xls		log_b11_81_1308.xls
	rtm_b4_99_0309.xls		log_b11_83_1408.xls
	rtm_b4_101_0409.xls		log_b11_85_2408.xls
	rtm_b4_103_0709.xls		log_b4_95_0109.xls
	rtm_b4_105_0809.xls		log_b4_97_0209.xls
	rtm_b4_107_0909.xls		log_b4_99_0309.xls
	rtm_b4_109_1009.xls		log_b4_101_0409.xls
	rtm_b11_77_1108.xls		log_b4_103_0709.xls
	rtm_b11_81_1308.xls		log_b4_105_0809.xls
	rtm_b11_83_1408.xls		log_b4_107_0909.xls
	rtm_b11_85_2408.xls		log_b4_109_1009.xls
	rtm_b11_87_2508.xls		

Table 11.2: Data inconsistency for drilling and blasting (2 / 2)

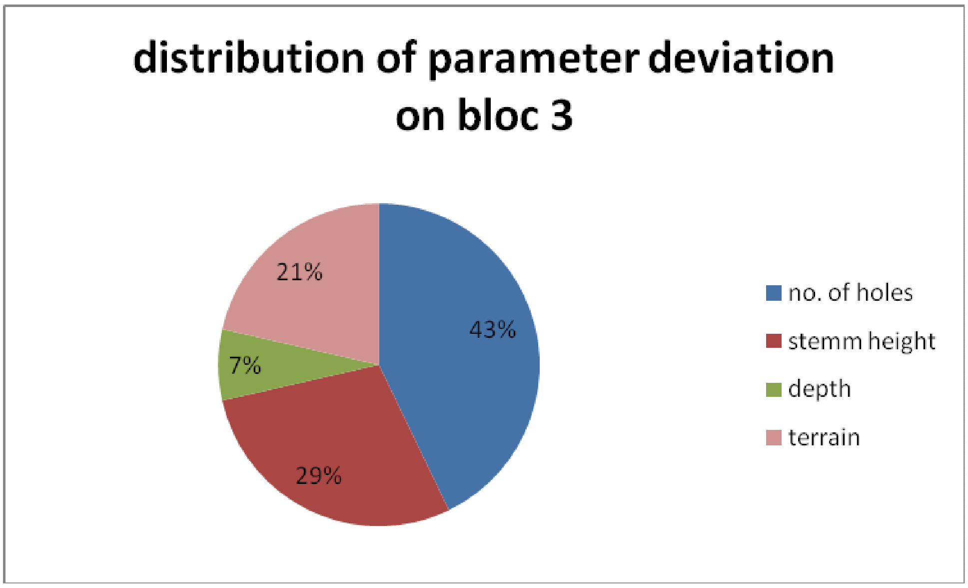


Figure 11.1: Distribution of drill and blast parameter deviation on *bloc 3*

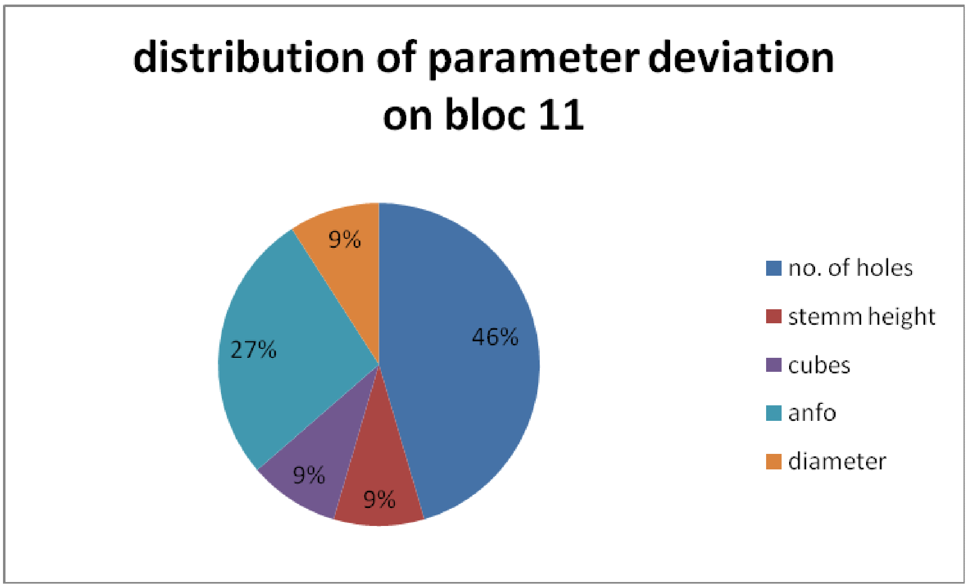


Figure 11.2: Distribution of drill and blast parameter deviation on *bloc 11*

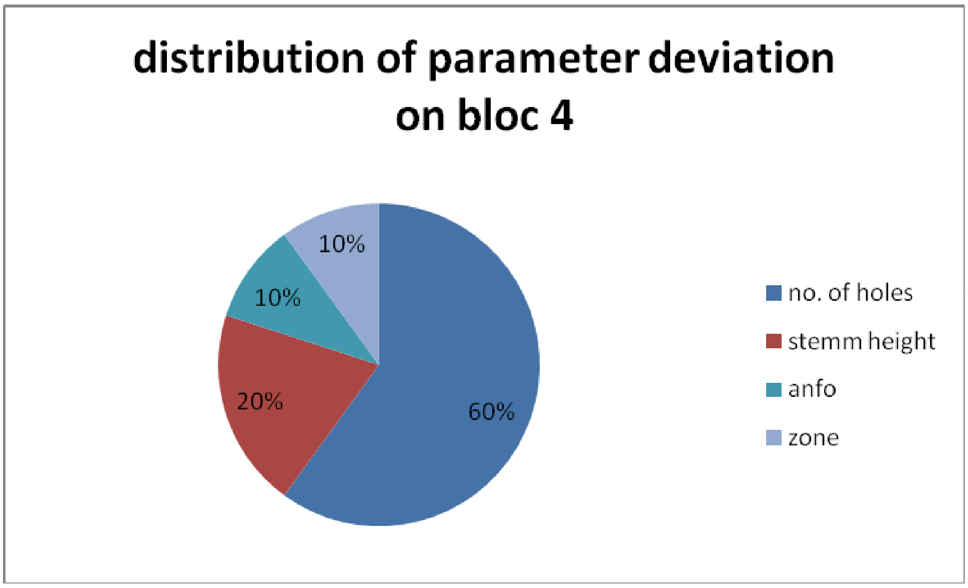


Figure 11.3: Distribution of drill and blast parameter deviation on *bloc 11*

11.1.2 Back break

Figure 11.4: Overview back break on *bloc 3* (pic 1 or **Fig. 11.6** and pic 2 or **Fig. 11.7**) and *bloc 4* (pic 3 or **Fig. 11.8**)

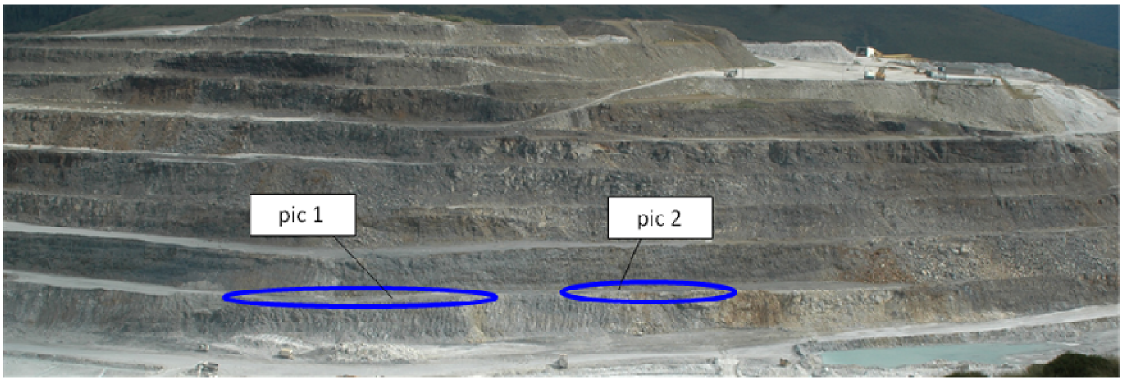


Figure 11.5: Overview back break on *bloc 11* (pic 1 or **Fig. 11.9** and pic 2 or **Fig. 11.10**)

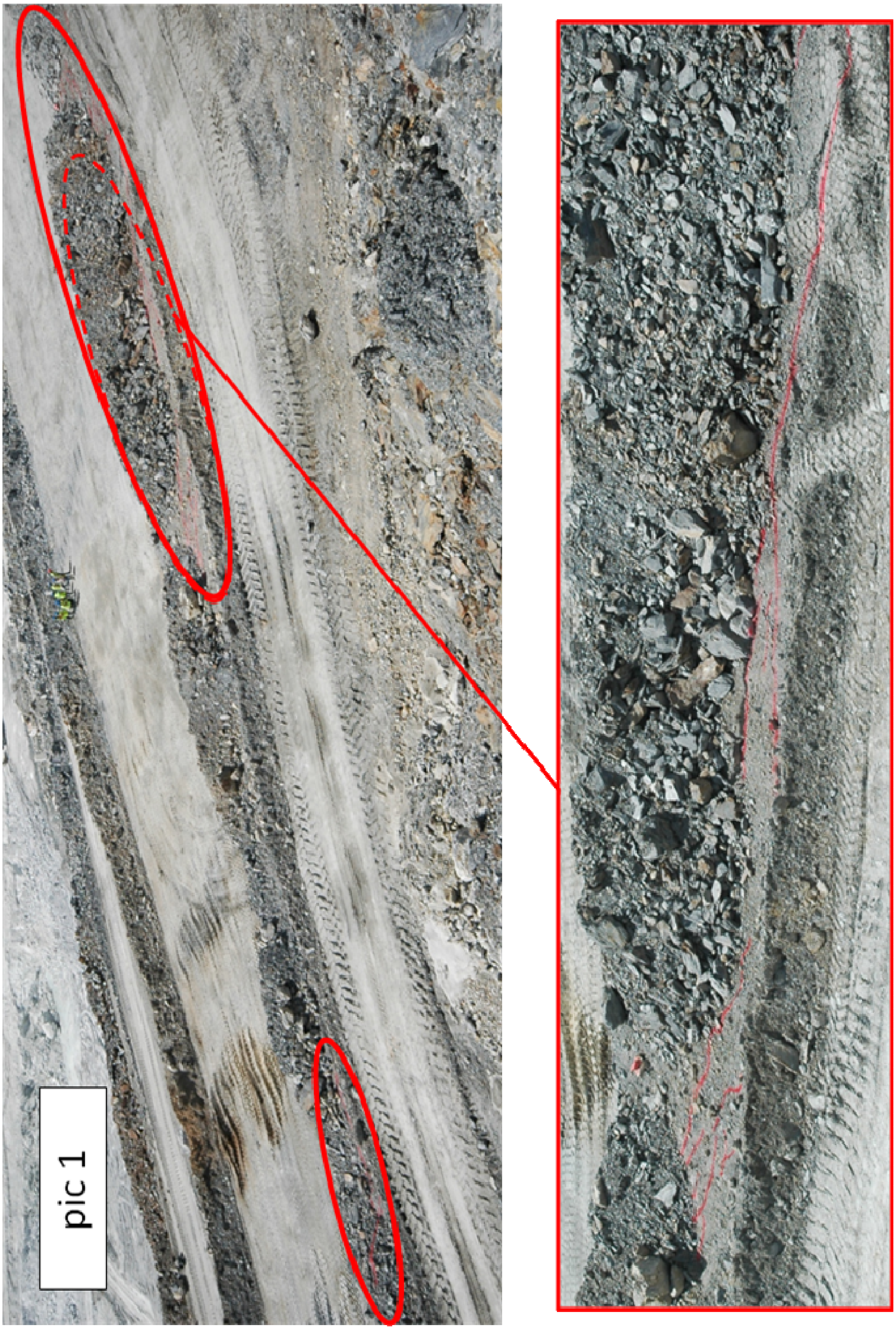


Figure 11.6: Back break after loading on *bloc 3*, North

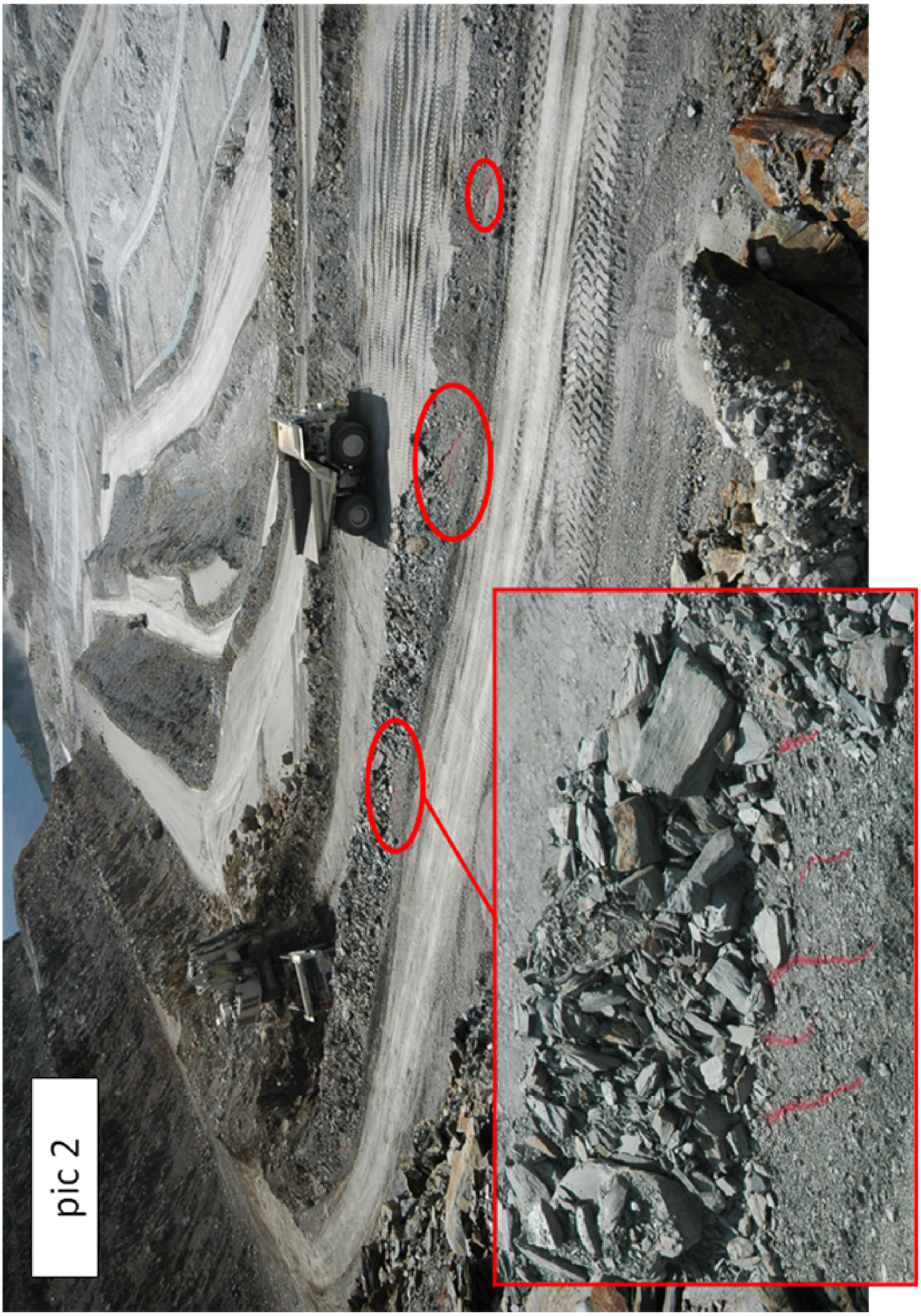


Figure 11.7: Back break after loading on *bloc 3*, South

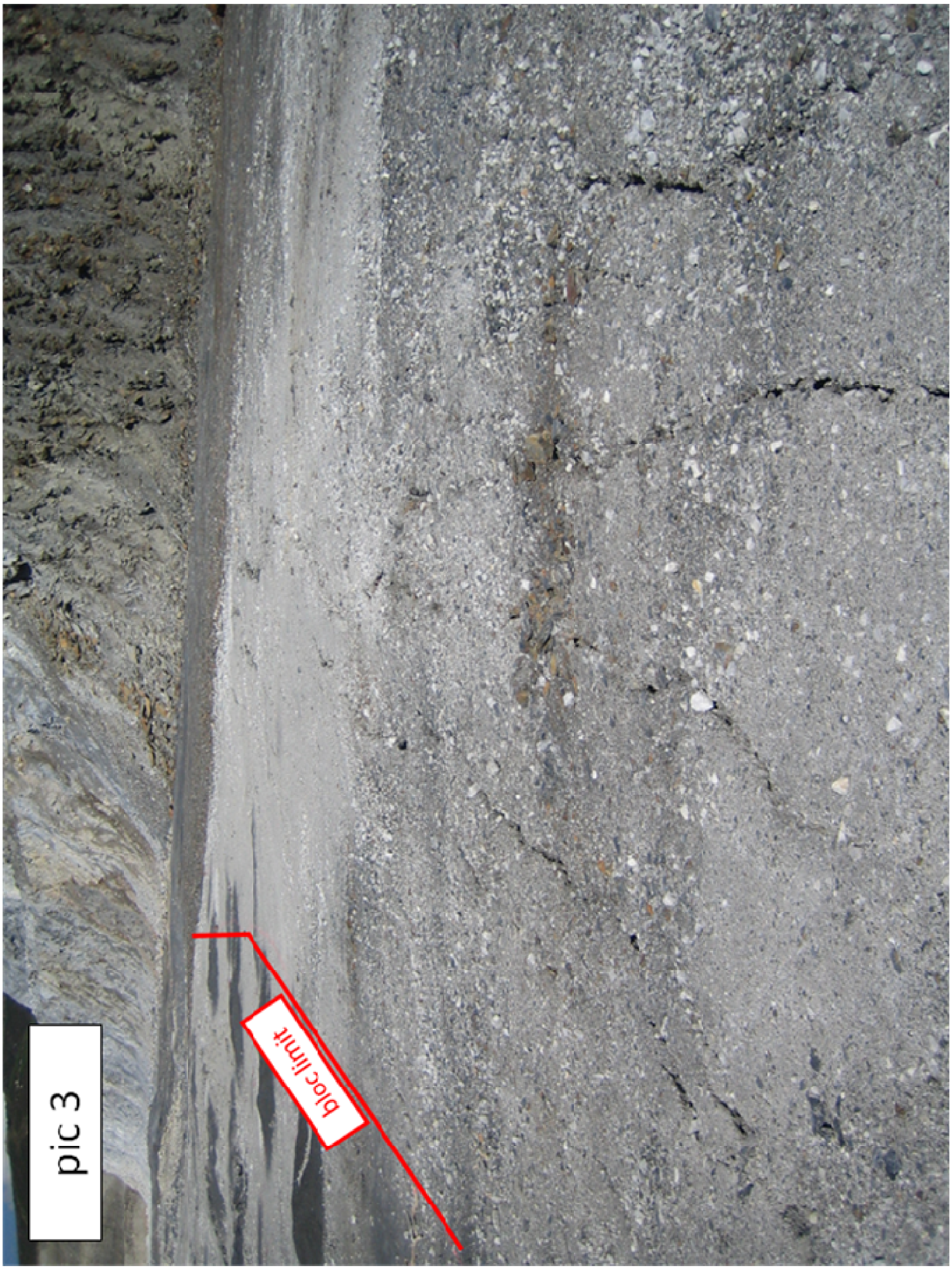


Figure 11.8: Back break before loading on *bloc 4*, South

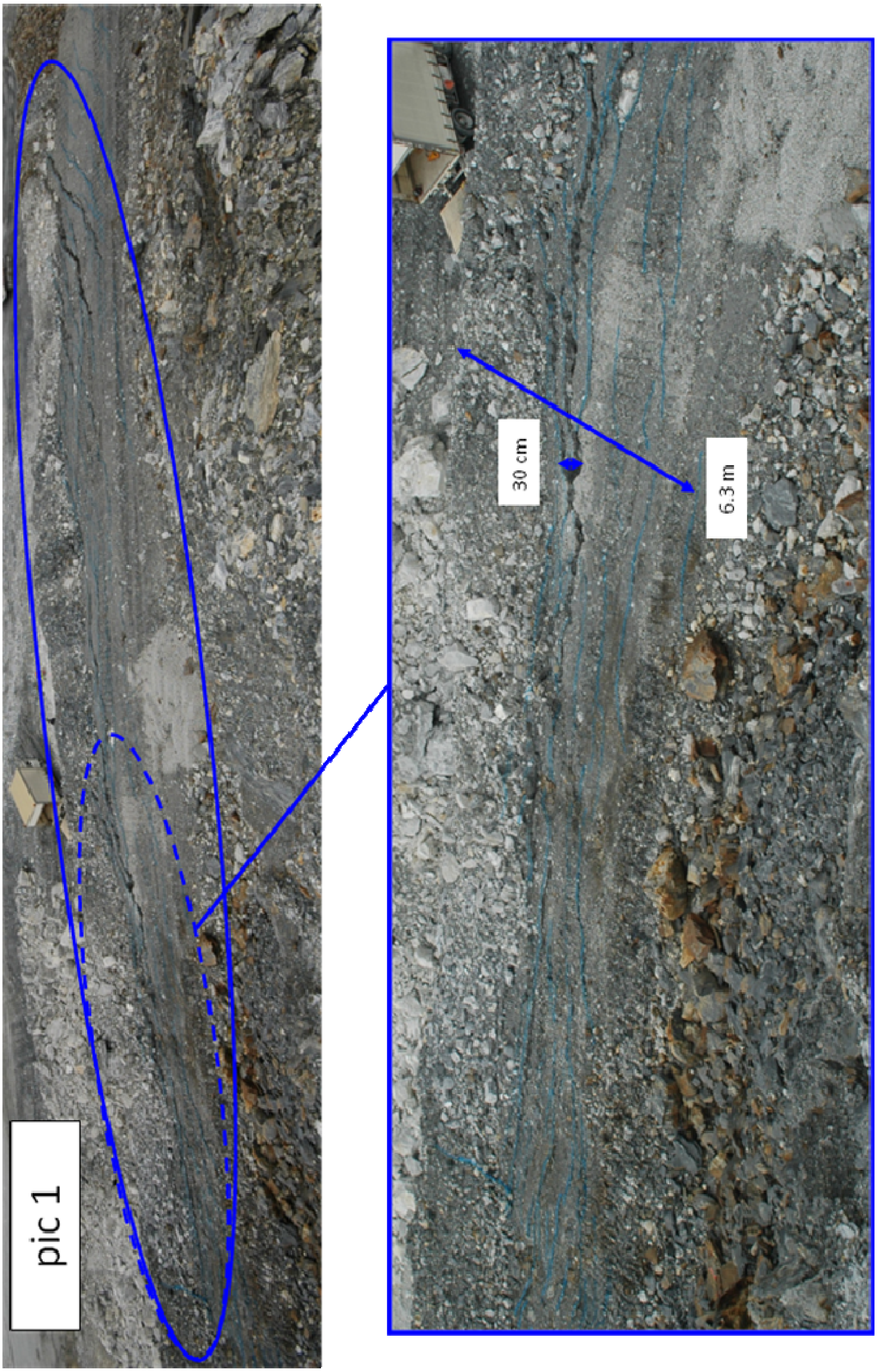


Figure 11.9: Back break after loading on *bloc 11*, North

Figure 11.10: Back break after loading on *bloc 11*, South

11.2 Loading and hauling

11.2.1 Data inconsistency

title: data inconsistency - loading and hauling (bloc 3)

	cubic meter [m³]			dev.	operating hour [h]			dev.
	A	B	diff.	[%]	A	B	diff.	[%]
R 994 B	134,187	135,290	-1,103	-0.8	171	194	-23	-13.5
C 777C-1	7,420	7,481	-61	-0.8	55	63	-8	-14.5
C 777C-2	20,461	20,630	-169	-0.8	149	149	0	0.0
HD 985-1	26,142	25,954	188	0.7	177	176	1	0.6
HD 985-2	21,989	21,686	303	1.4	148	146	2	1.4
HD 985-3	28,902	28,696	206	0.7	189	188	1	0.5
HD 985-4	29,273	29,070	203	0.7	194	194	0	0.0
total	134,187	133,517	670	0.5	913	915	-2	-0.2

A = calc_prod_bloc_09_01.xls

B = calc_prod_bloc_09_02.xls

ref.:
log_b11_sum_exc.pdf
log_b11_sum_truck.pdf
log_b3_sum_exc.pdf
log_b3_sum_truck.pdf

rtm_haul_bloc_0809.xls
rtm_load_bloc_0809.xls

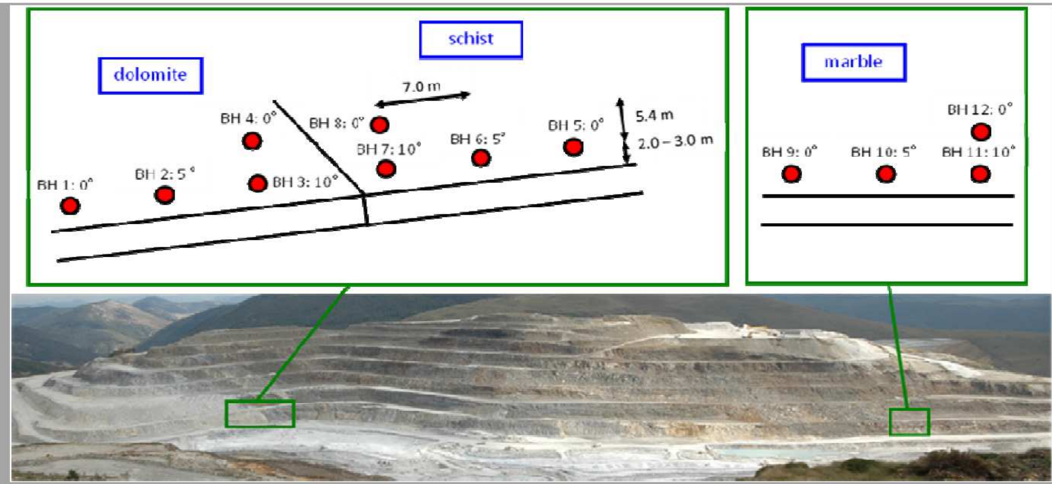
Figure 11.11: Data inconsistency for loading and hauling (*bloc 3*)

12 Appendix – Measurements and calculations

12.1 Drilling and blasting

12.1.1 Drill time

title:	drill time
date(s):	06/10/2009, 07/10/2009
start time(s):	both 08:25
location:	1st bench after trench
	(1-8 North, 9-12 South)
material:	dolomite, schist, marble
drill rig:	D 25 KS



no.	material	row	depth	inclin.
1	dolomite	1	16 m	0°
2		1	16 m	5°
3		1	16 m	10°
4		2	16 m	0°
5	schist	1	16 m	0°
6		1	16 m	5°
7		1	16 m	10°
8		2	16 m	0°
9	marble	1	16 m	0°
10		1	16 m	5°
11		1	16 m	10°
12		2	16 m	0°

time [min] for drilling & additional activities		
before drilling	drive	0.9
	park	0.5
	rig down	0.5
drilling	16 m	17.6
	continuity	0.9
	2nd rod	1.4
after drilling	continuity [2]	1.3
	2nd rod	1.0
	1st rod	0.7
	rig up	0.4
	unpark	0.4
total [1]		25.5

[1] (drilling & add. activities)
[2] not really tested while drilling in marble
→ not included in average values

drill time [min] per geology					
rock type		dol.	schist	mar.	av.
drilling	per 1 m	1.0	0.7	1.6	1.1
	per 16 m	15.9	11.9	25.2	17.6
total (incl. add. activities)		27.8	18.9	28.5	25.1

pressure [bar] per geology				
rock type	dol.	schist	mar.	av.
receptor pressure	21	21	22	21
working pressure	23	23	23	23
rotary pressure	42	43	37	41
thrust pressure	20	20	21	21

head speed [RPM] per geology				
rock type	dol.	schist	mar.	av.
head speed	11	20	6	12

Table 12.1: Summary of drill time measurements [92]

[92] calc_time_drill.xls

12.1.2 Charge time

title: average charging time
comment: not added is the time to receipt explosives and detonators from storage (~ 2 h for 3.25 t of explosives)

no. of holes	[1]	1	16	20	24	40	1	16	20	24	40
burden	[m]	5.4					5.4				
spacing	[m]	7.0					7.0				
bench height	[m]	15.0					15.0				
sub-drill	[m]	1.0					1.0				
cubes	[m^3]	567	9,072	11,340	13,608	22,680	567	9,072	11,340	13,608	22,680
anfo	[kg]	155	2,480	3,100	3,720	6,200	150	2,480	3,100	3,720	6,200
emulsion	[kg]	8	128	160	192	320	16	128	160	192	320
stemm height	[m]	6	101	126	151	252	7	106	132	158	264
persons working	[1]	2					2				
positioning of truck	[min]	2	8	10	13	21	2	8	10	13	21
unload	[min]	3	45	56	67	112	3	45	56	67	112
measure of depth	[min]	3	41	51	61	102	3	41	51	61	102
1st cartridge + detonator	[min]	1	19	23	28	47	1	19	23	28	47
2nd cartridge	[min]						1	19	23	28	47
anfo	[min]	2	26	33	39	65	2	25	33	39	65
stemming	[min]	4	63	78	94	156	4	54	67	80	40
total	[min]	14	202	252	302	504	11	157	197	237	395
	[h]	0.2	3.4	4.2	5.0	8.4	0.2	2.6	3.3	3.9	6.6

Table 12.2: Theoretical charging time for different drill and blast pattern and number of holes [93]

[93] calc_time_charge.xls

12.1.3 Test blasts [94]

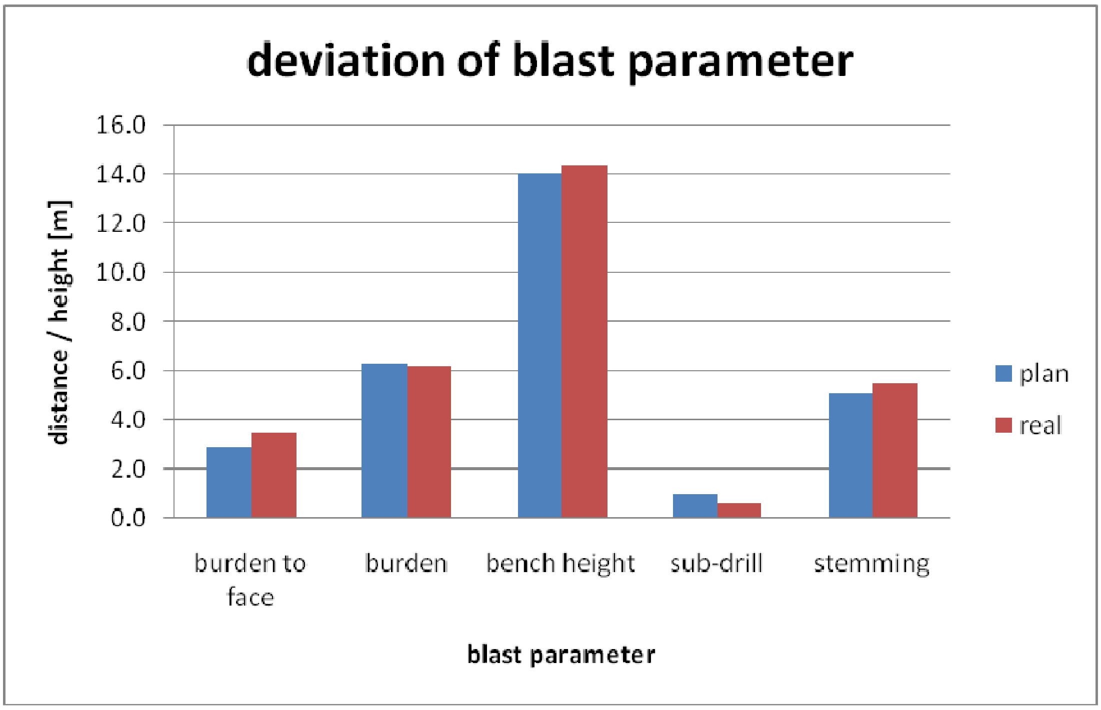


Figure 12.1: Deviation of blast parameter between planned and real values

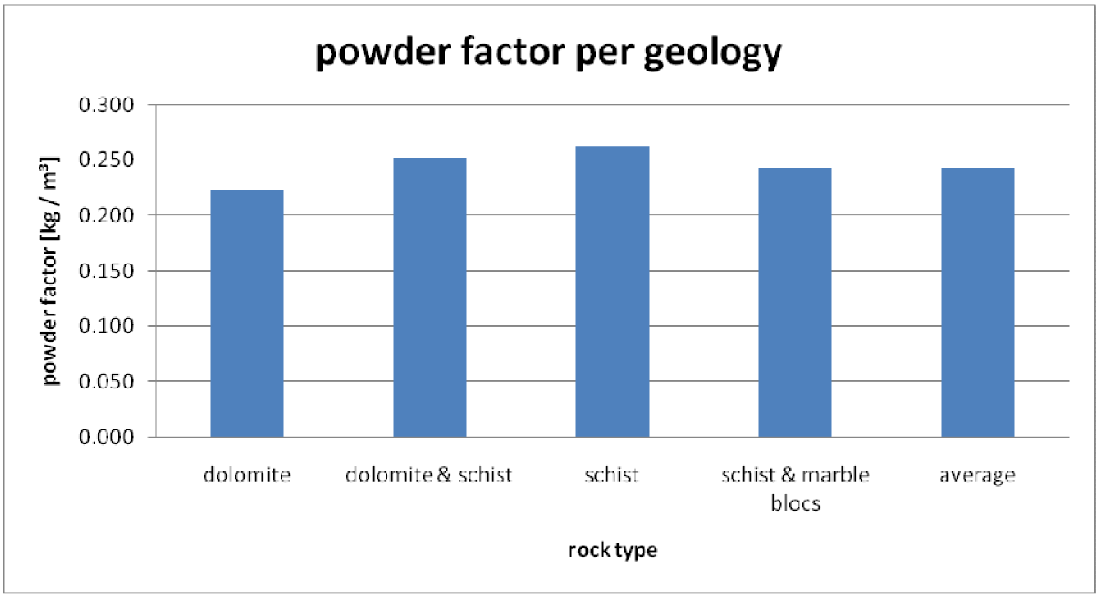


Figure 12.2: Average powder factor per geology

[94] calc_bm_sum.xls

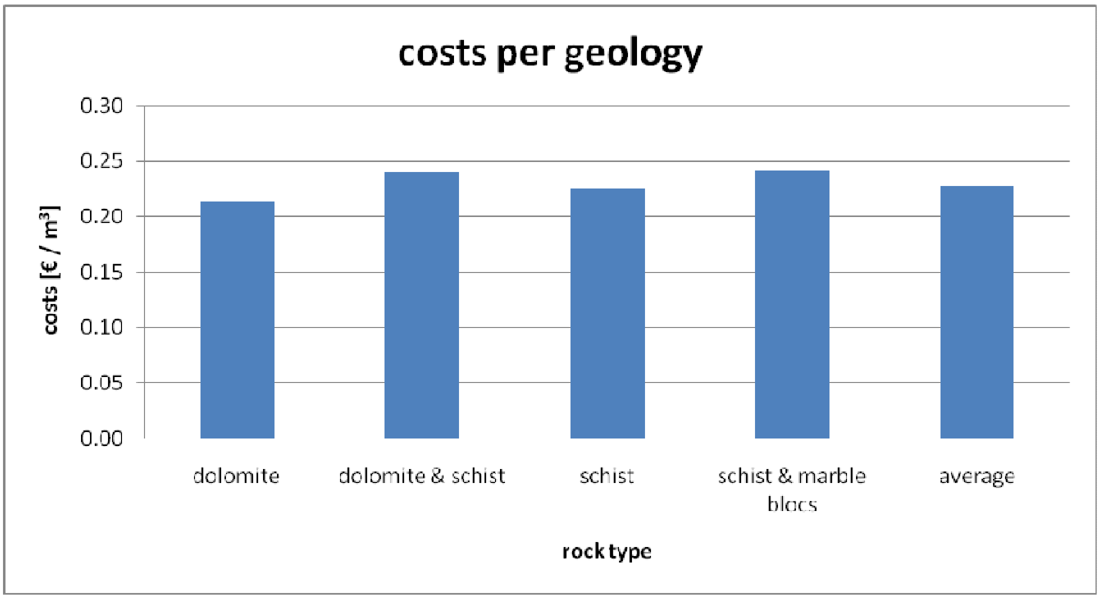


Figure 12.3: Average explosive cost per geology

12.2 Loading and hauling

12.2.1 Load and haul measurements [95]

title: summary load & haul measurements

bloc		3	11	trench	average
loader		R 994 B			
no. of trucks	[1]	5.1	4.0	3.0	4.0
av. distance	[m]	2,131	1,355	1,145	2,131
time at load site (excl. time leaving)	[min]	2.4	2.2		2.3
total time at load site (LS)	[min]	2.8	2.6		2.7
total time at dump site (DS)	[min]	1.1		1.0	1.1
hang time (ex. 1st bucket)	[min]	1.0	1.0		1.0
total hang time	[min]	1.5	1.4		1.5
queue time upon arrival	[min]	0.2	0.2		0.2
queue time upon loading	[min]	0.2	0.1		0.2
total queue time (LS)	[min]	0.4	0.3		0.4
waiting time while haul (H)	[min]	0.0		0.0	0.0
waiting time while return (R)	[min]	0.2		0.8	0.5
total waiting time	[min]				0.9
reverse time upon arrival	[min]	0.1	0.0		0.1
reverse time upon loading	[min]	0.4	0.3		0.4
reverse time at load site (LS)	[min]	0.5	0.4		0.4
reverse time at dump site (DS)	[min]	0.5		0.4	0.4
total reverse time	[min]	0.9			0.9
total time while haul (H)		5.3			5.3
total time while return (R)		4.6			4.6
time per load cycle	[min]	3.1	2.8		2.9
load cycle per hour	[1]	19.6	21.6		20.6
time per haul & return cycle	[min]	13.9	11.0	14.4	13.1
haul & return cycle per hour	[1]	4.3	5.5	4.2	4.6
time per dumping (U)	[min]	0.7		0.6	0.7
buckets per truck	[1]	4.1	3.9		4.0
time per bucket	[sec]	30.0	28.4		29.2
buckets per hour	[1]	120	128		124
fill factor per bucket (without last)	[1]	2.5	2.7		2.6
weight per bucket	[t]	26.1			26.1
th. loading performance per R 994 B [1]	[m³ / h]	806	809		807
th. hauling performance per truck [2]	[m³ / h]	159	202	154	172

[1] (haul and return cycle per hour) * (no.of trucks) * (cap_T) with cap_T = 37 m³

[2] (haul and return cycle per hour) * (cap_T)

Table 12.3: Summary of all load and haul measurements

[95] calc_sum_load_haul.xls

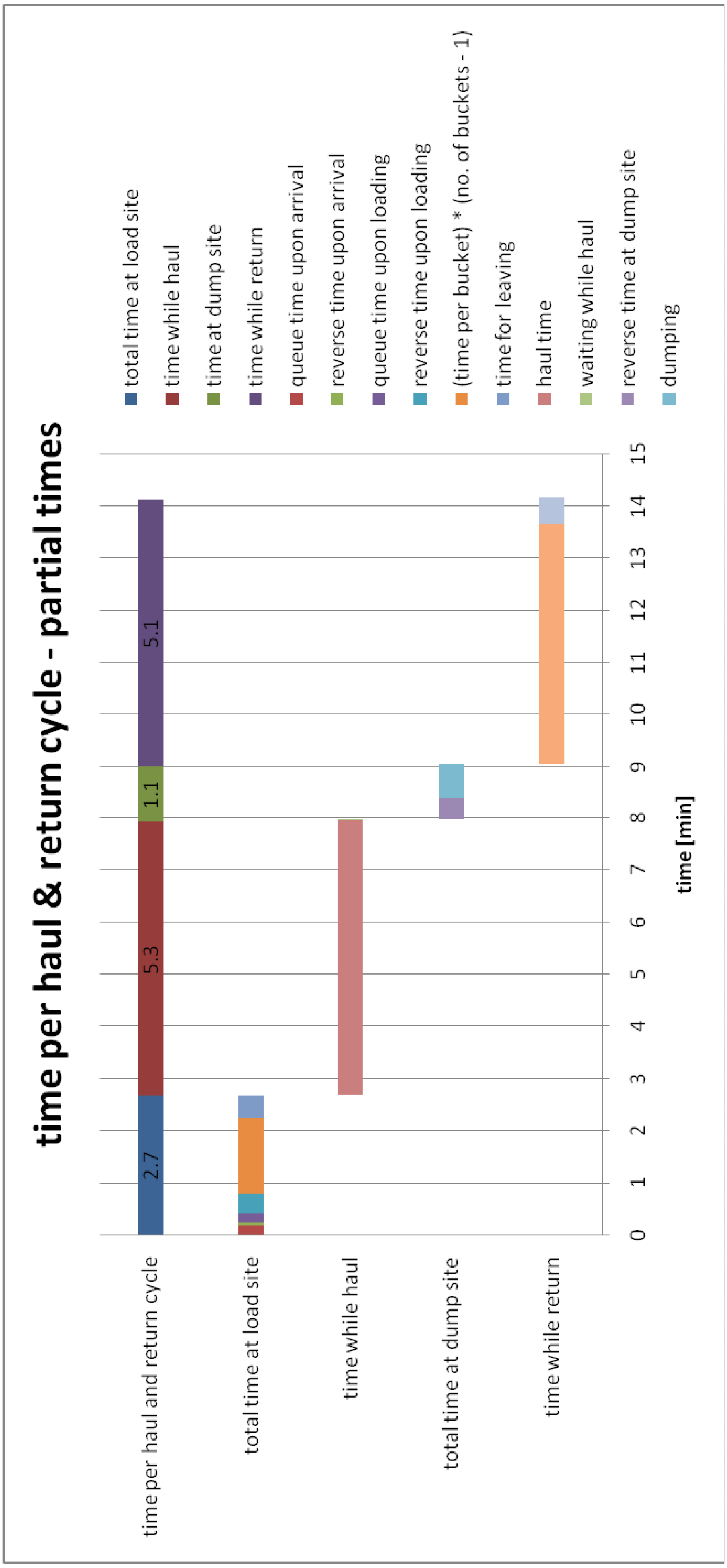


Figure 12.4: Total per load and haul cycle, incl. partial times

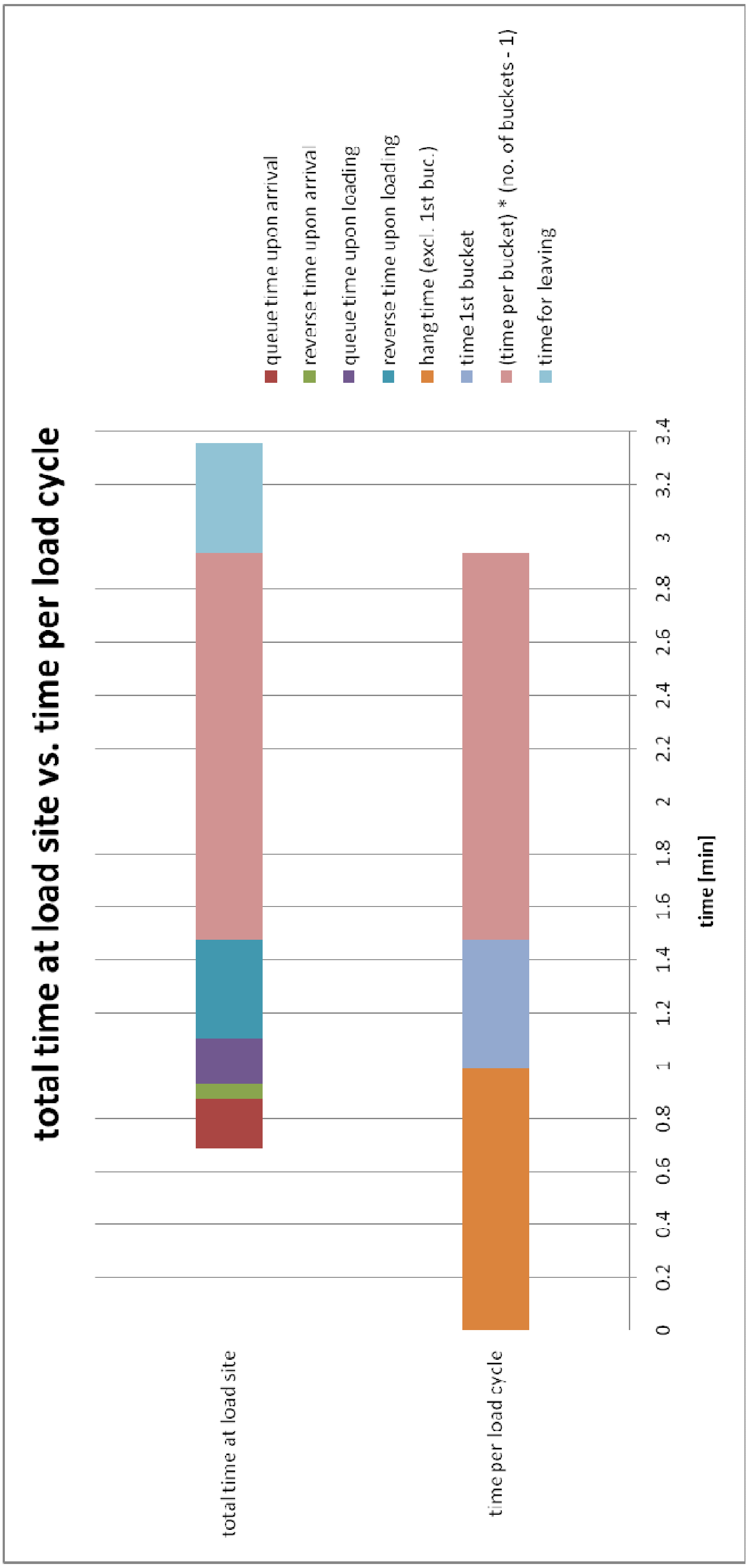


Figure 12.5: Total time at load site vs. time per load cycle

12.2.2 Load and haul measurements at load site – *bloc 3* [96]

overview:		load & haul measurements (load site)	
bloc			3
loader			R 994 B
no. of trucks		[1]	4.8
av. distance		[m]	2,184
time at load site (excl. leaving)		[min]	2.5
time at load site		[min]	2.9
hang time (ex. 1st bucket)		[min]	1.0
total hang time		[min]	1.5
queue time upon arrival		[min]	0.2
queue time upon loading		[min]	0.2
total queue time		[min]	0.4
reverse time upon arrival		[min]	0.1
reverse time upon loading		[min]	0.4
total reverse time at load site		[min]	0.5
time per load cycle		[min]	3.1
load cycle per hour		[1]	19.6
time per haul & return cycle		[min]	14.3
haul & return cycle per hour		[1]	4.2
buckets per truck		[1]	4.0
time per bucket		[s]	30
buckets per hour		[1]	118
fill factor per bucket (without last)		[1]	2.5
time per	normal loading	[s]	29
bucket activity	ripping	[s]	32
	cleaning	[s]	32
	stones	[s]	33
percent per	normal loading	[%]	71
bucket activity	ripping	[%]	21
	cleaning	[%]	7
	stones	[%]	1
fill factor per	normal loading	[fill f.]	2.6
bucket activity	ripping	[fill f.]	2.4
	cleaning	[fill f.]	2.4
	stones	[fill f.]	2.1

Table 12.4: Summary of load and haul measurements, *bloc 3* at load site

[96] calc_bloc3_load.xls

material (blast design)	no. of trucks [1]			
	slice 1	slice 2	slice 3	slice 4
dolomite (RS)			4	
dolomite (JT)		4	5	5
dolomite + graphite (RS)	5			5
schist (RS)	5	5	5	5
schist + dolomite (RS)	5		5	
schist + marble (JT + RS)	6	4		

material (blast design)	distance [m]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)	2,005	2,025	2,045	2,065	2,035
dolomite (JT)	2,005	2,025	2,045	2,065	
dolomite + graphite (RS)	2,095	2,115	2,135	2,155	2,125
schist (RS)	2,170	2,190	2,210	2,230	2,200
schist + dolomite (RS)	2,230	2,250	2,270	2,290	2,260
schist + marble (JT + RS)	2,270	2,290	2,310	2,330	2,300
average	2,125	2,145	2,165	2,185	2,131

material (blast design)	hang time (ex. 1st bucket) [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			1.2		1.2
dolomite (JT)		1.5	0.8	0.6	1.0
dolomite + graphite (RS)	0.8			2.1	1.5
schist (RS)	1.0	0.8	0.9	0.8	0.8
schist + dolomite (RS)	0.9		1.0		0.9
schist + marble (JT + RS)	1.6	1.2			1.4
average	0.9	1.1	1.0	1.2	1.0

material (blast design)	total hang time [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			1.6		1.6
dolomite (JT)		2.0	1.4	1.1	1.5
dolomite + graphite (RS)	1.3			2.6	2.0
schist (RS)	1.5	1.3	1.3	1.3	1.3
schist + dolomite (RS)	1.5		1.5		1.5
schist + marble (JT + RS)	2.1	2.0			2.0
average	1.4	1.6	1.5	1.7	1.5

material (blast design)	queue time upon arrival [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			0.0		0.0
dolomite (JT)		0.1	0.1	0.5	0.2
dolomite + graphite (RS)	0.4			0.5	0.4
schist (RS)	0.4	0.2	0.1	0.1	0.2
schist + dolomite (RS)	0.3		0.0		0.2
schist + marble (JT + RS)	0.4	1.6			1.0
average	0.4	0.1	0.1	0.3	0.2

material (blast design)	reverse time upon arrival [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			0.0		0.0
dolomite (JT)		0.1	0.1	0.1	0.1
dolomite + graphite (RS)	0.2			0.0	0.1
schist (RS)	0.1	0.0	0.0	0.0	0.0
schist + dolomite (RS)	0.1		0.0		0.1
schist + marble (JT + RS)	0.1	1.8			1.0
average	0.1	0.1	0.0	0.0	0.1

Table 12.5: Load and haul measurements per material, blast design and slice, *bloc 3* at load site (1 / 4)

material (blast design)	queue time upon loading [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			0.1		0.1
dolomite (JT)		0.1	0.2	0.2	0.1
dolomite + graphite (RS)	0.3			0.8	0.5
schist (RS)	0.2	0.1	0.1	0.1	0.1
schist + dolomite (RS)	0.3		0.1		0.2
schist + marble (JT + RS)	0.6	1.2			0.9
average	0.3	0.1	0.1	0.3	0.2

material (blast design)	reverse time upon loading [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			0.5		0.5
dolomite (JT)		0.6	0.4	0.3	0.4
dolomite + graphite (RS)	0.5			0.4	0.4
schist (RS)	0.4	0.3	0.3	0.3	0.3
schist + dolomite (RS)	0.6		0.4		0.5
schist + marble (JT + RS)	0.7	0.4			0.5
average	0.5	0.4	0.4	0.3	0.4

material (blast design)	total queue time [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			0.2		0.2
dolomite (JT)		0.1	0.3	0.7	0.4
dolomite + graphite (RS)	0.8			1.3	1.0
schist (RS)	0.6	0.3	0.2	0.1	0.3
schist + dolomite (RS)	0.6		0.1		0.3
schist + marble (JT + RS)	0.9	2.3			1.6
average	0.7	0.2	0.2	0.7	0.4

material (blast design)	total reverse time at load site [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			0.6		0.6
dolomite (JT)		0.7	0.6	0.3	0.5
dolomite + graphite (RS)	0.8			0.4	0.6
schist (RS)	0.5	0.3	0.3	0.3	0.3
schist + dolomite (RS)	0.7		0.5		0.6
schist + marble (JT + RS)	0.8	1.8			1.3
average	0.7	0.5	0.5	0.3	0.5

material (blast design)	time per load cycle [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			3.0		3.0
dolomite (JT)		3.6	2.9	2.6	3.0
dolomite + graphite (RS)	2.8			4.2	3.5
schist (RS)	3.0	2.6	2.7	2.6	2.7
schist + dolomite (RS)	3.6		3.1		3.4
schist + marble (JT + RS)	3.9	4.7			4.3
average	3.1	3.1	2.9	3.1	3.1

material (blast design)	load cycle per hour [1]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			19.9		19.9
dolomite (JT)		16.5	20.6	23.1	20.1
dolomite + graphite (RS)	21.1			14.3	17.7
schist (RS)	20.3	23.3	22.6	23.0	22.3
schist + dolomite (RS)	16.8		19.2		18.0
schist + marble (JT + RS)	15.5	12.8			14.1
average	19.4	19.9	20.6	20.1	20.1

Table 12.6: Load and haul measurements per material, blast design and slice, *bloc 3* at load site (2 / 4)

material (blast design)	time per haul & return cycle [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			12.1		12.1
dolomite (JT)		13.5	14.5	12.9	13.6
dolomite + graphite (RS)	14.1			16.3	15.2
schist (RS)	14.7	12.8	13.6	13.2	13.6
schist + dolomite (RS)	18.2		15.8		17.0
schist + marble (JT + RS)	18.0	20.3			19.1
average	15.7	13.1	14.0	14.2	14.3

material (blast design)	haul & return cycle per hour [1]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			5.0		5.0
dolomite (JT)		4.5	4.1	4.6	4.4
dolomite + graphite (RS)	4.3			3.7	4.0
schist (RS)	4.1	4.7	4.4	4.5	4.4
schist + dolomite (RS)	3.3		3.8		3.5
schist + marble (JT + RS)	3.3	3.0			3.1
average	3.9	4.6	4.3	4.3	4.2

material (blast design)	buckets per truck [1]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			4.0		4.0
dolomite (JT)		4.3	4.0	4.1	4.1
dolomite + graphite (RS)	4.2			4.0	4.1
schist (RS)	3.9	3.7	3.8	3.9	3.8
schist + dolomite (RS)	4.1		4.0		4.0
schist + marble (JT + RS)	4.3	4.6			4.4
average	4.1	4.0	3.9	4.0	4.0

material (blast design)	fill factor per bucket [1]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			2.6		2.6
dolomite (JT)		2.3	2.7	2.6	2.5
dolomite + graphite (RS)	1.8			2.6	2.2
schist (RS)	2.6	2.7	2.8	2.9	2.7
schist + dolomite (RS)	2.3		2.5		2.4
schist + marble (JT + RS)	2.6	1.8			2.2
average	2.2	2.5	2.6	2.7	2.5

without last bucket if fill factor is 1

material (blast design)	time per bucket [s]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			28		28
dolomite (JT)		30	31	30	30
dolomite + graphite (RS)	29			32	30
schist (RS)	31	29	28	28	29
schist + dolomite (RS)	39		32		36
schist + marble (JT + RS)	32	44			38
average	33	30	30	30	30

material (blast design)	buckets per hour [1]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			130		130
dolomite (JT)		119	116	122	119
dolomite + graphite (RS)	126			113	120
schist (RS)	118	124	127	130	125
schist + dolomite (RS)	92		113		102
schist + marble (JT + RS)	112	82			97
average	112	122	121	122	119

Table 12.7: Load and haul measurements per material, blast design and slice, *bloc 3* at load site (3 / 4)

material (blast design)	time at load site (excl. leaving) [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			2.1		2.1
dolomite (JT)		2.3	2.5	2.6	2.5
dolomite + graphite (RS)	3.0			3.4	3.2
schist (RS)	2.7	1.9	1.8	1.8	2.1
schist + dolomite (RS)	3.4		2.1		2.8
schist + marble (JT + RS)	3.5	6.3			4.9
average	3.0	2.1	2.1	2.6	2.5

material (blast design)	time at load site [min]				
	slice 1	slice 2	slice 3	slice 4	average
dolomite (RS)			2.5		2.5
dolomite (JT)		2.8	3.0	3.0	2.9
dolomite + graphite (RS)	3.4			3.8	3.6
schist (RS)	3.1	2.3	2.3	2.2	2.5
schist + dolomite (RS)	3.9		2.5		3.2
schist + marble (JT + RS)	3.9	6.7			5.3
average	3.4	2.6	2.6	3.0	2.9

Table 12.8: Load and haul measurements per material, blast design and slice, *bloc 3* at load site (4 / 4)

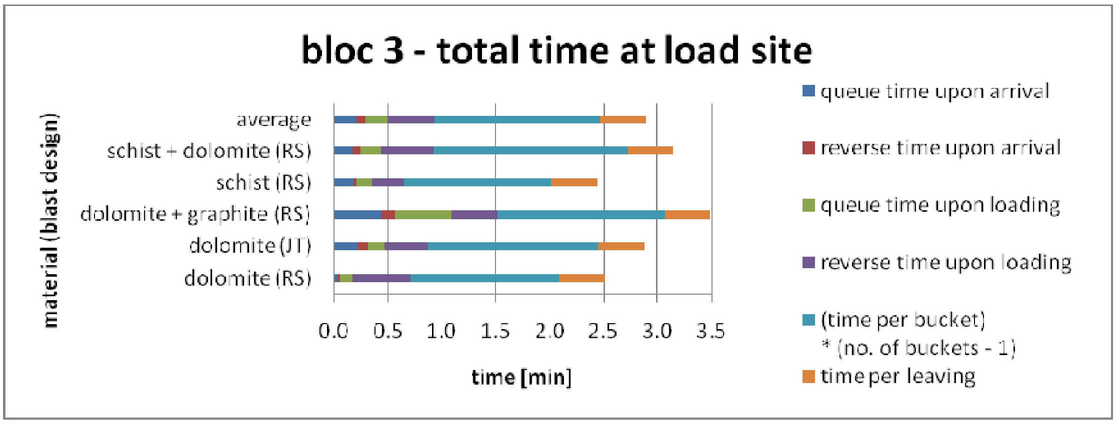


Figure 12.6: Total time at load site per material and blast design, *bloc 3*

	activity	normal loading	ripping	cleaning	stones	average
		time per bucket activity [s]				
slice 1	dolomite (RS)					
	dolomite (JT)					
	dol. + graph. (RS)	28	29	32	33	30
	schist (RS)	29	35	30	53	37
	schist + dol. (RS)	34	43	41		39
slice 2	dolomite (RS)					
	dolomite (JT)	30	29	35		31
	dol. + graph. (RS)					
	schist (RS)	28	31	28	33	30
	schist + dol. (RS)					
slice 3	dolomite (RS)	27	30	28		28
	dolomite (JT)	30	33	34	25	30
	dol. + graph. (RS)					
	schist (RS)	28	30	28	20	27
	schist + dol. (RS)	32	33	33		32
slice 4	dolomite (RS)					
	dolomite (JT)	28	32	36	30	31
	dol. + graph. (RS)	30	38	33	35	34
	schist (RS)	28	27	25		27
	schist + dol. (RS)					

Table 12.9: Time per bucket activity, material, blast design and slice, *bloc 3*

	activity	activity per bucket [%] normal loading	ripping	cleaning	stones
slice 1	dolomite (RS)				
	dolomite (JT)				
	dol. + graph. (RS)	75	13	8	4
	schist (RS)	80	14	5	1
	schist + dol. (RS)	37	51	13	
slice 2	dolomite (RS)				
	dolomite (JT)	70	23	7	
	dol. + graph. (RS)				
	schist (RS)	69	20	8	3
	schist + dol. (RS)				
slice 3	dolomite (RS)	82	10	8	
	dolomite (JT)	62	23	13	2
	dol. + graph. (RS)				
	schist (RS)	76	19	3	2
	schist + dol. (RS)	66	30	4	
slice 4	dolomite (RS)				
	dolomite (JT)	74	19	6	1
	dol. + graph. (RS)	72	18	8	3
	schist (RS)	89	7	4	
	schist + dol. (RS)				

Table 12.10: Bucket activity per material, blast design and slice, *bloc 3*

		fill factor per bucket activity (incl. last buc.) [1]				
		normal loading	ripping	cleaning	stones	average
slice 1	dolomite (RS)					
	dolomite (JT)					
	dol. + graph. (RS)	1.8	1.7	1.8	2.0	1.8
	schist (RS)	2.7	2.5	2.6	2.0	2.4
	schist + dol. (RS)	2.4	2.1	2.3		2.3
slice 2	dolomite (RS)					
	dolomite (JT)	2.4	2.0	2.1		2.2
	dol. + graph. (RS)					
	schist (RS)	2.8	2.4	2.4	2.0	2.4
	schist + dol. (RS)					
slice 3	dolomite (RS)	2.7	2.4	2.5		2.5
	dolomite (JT)	2.7	2.6	2.7	2.0	2.5
	dol. + graph. (RS)					
	schist (RS)	2.9	2.5	3.0	2.3	2.7
	schist + dol. (RS)	2.6	0.0	2.5		1.7
slice 4	dolomite (RS)					
	dolomite (JT)	2.6	2.7	2.7	2.0	2.5
	dol. + graph. (RS)	2.6	2.6	2.8	2.5	2.6
	schist (RS)	2.9	3.0	2.0		2.6
	schist + dol. (RS)					

Table 12.11: Bucket fill factor (incl. last bucket) per activity, material, blast design and slice, *bloc 3*

overview:		load & haul measurements (load site)	
bloc		3	
truck		HD 985-5	C 777 D
queue time upon arrival	[min]	0.3	0.6
queue time upon loading	[min]	0.3	0.5
total queue time	[min]	0.5	1.0
reverse time upon arrival	[min]	0.2	0.1
reverse time upon loading	[min]	0.4	0.4
total reverse time at load site	[min]	0.6	0.6
time per load cycle	[min]	3.2	3.1
load cycle per hour	[1]	18.5	19.3
time per haul & return cycle	[min]	15.0	14.7
haul & return cycle per hour	[1]	4.0	4.1
buckets per truck	[1]	4.1	4.0

Table 12.12: Comparison between *HD 985-5* and *C 777 D*, *bloc 3*

12.2.3 Load and haul measurements at load site – *bloc 11* [97]

title:			load & haul measurements (load site)
bloc			11
loader			R 994 B
no. of trucks		[1]	4.0
av. distance		[m]	1,300
time at load site (excl. leaving)		[min]	2.2
total time at load site		[min]	2.6
hang time (ex. 1st bucket)		[min]	1.0
total hang time		[min]	1.4
queue time upon arrival		[min]	0.2
queue time upon loading		[min]	0.1
total queue time		[min]	0.3
reverse time upon arrival		[min]	0.0
reverse time upon loading		[min]	0.3
total reverse time at load site		[min]	0.4
time per load cycle		[min]	2.8
load cycle per hour		[1]	21.6
time per haul & return cycle		[min]	11.0
haul & return cycle per hour		[1]	5.5
buckets per truck		[1]	3.9
time per bucket		[s]	28
buckets per hour		[1]	128
fill factor per bucket (without last)		[1]	2.7
time per	normal loading	[s]	28
bucket activity	ripping	[s]	33
	cleaning	[s]	32
	stones	[s]	31
percent per	normal loading	[%]	87
bucket activity	ripping	[%]	6
	cleaning	[%]	7
	stones	[%]	1
fill factor per	normal loading	[1]	2.4
bucket activity	ripping	[1]	2.3
	cleaning	[1]	2.2
	stones	[1]	2.1

Table 12.13: Summary of load and haul measurements, *bloc 11* at load site

[97] calc_bloc11_load.xls

material (blast design)	no. of trucks [1]			
	slice 1	slice 3	slice 4	slice 5
schist + dolomite (RS)		4	4	
schist + dolomite (JT)	4	4		
dolomite (RS)				4
dolomite (JT)			4	4

material (blast design)	distance [m]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)	1,260	1,300	1,320	1,340	1,305
schist + dolomite (JT)	1,370	1,410	1,430	1,450	1,415
dolomite (RS)	1,260	1,300	1,320	1,340	1,305
dolomite (JT)	1,370	1,410	1,430	1,450	1,415
average	1,315	1,355	1,375	1,395	1,360

material (blast design)	hang time (ex. 1st bucket) [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		0.9	1.3		1.1
schist + dolomite (JT)	1.1	0.9			1.0
dolomite (RS)				0.9	0.9
dolomite (JT)			0.7	0.8	0.7
average	1.1	0.9	1.0	0.9	1.0

material (blast design)	total hang time [s]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		1.3	1.7		1.5
schist + dolomite (JT)	1.6	1.4			1.5
dolomite (RS)				1.5	1.5
dolomite (JT)			1.2	1.3	1.2
average	1.6	1.4	1.7	1.4	1.4

material (blast design)	queue time upon arrival [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		0.2	0.1		0.2
schist + dolomite (JT)	0.1	0.1			0.1
dolomite (RS)				0.2	0.2
dolomite (JT)			0.0	0.4	0.2
average	0.1	0.2	0.1	0.3	0.2

material (blast design)	reverse time upon arrival [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		0.0	0.0		0.0
schist + dolomite (JT)	0.0	0.1			0.1
dolomite (RS)				0.0	0.0
dolomite (JT)			0.0	0.1	0.0
average	0.0	0.1	0.0	0.0	0.0

material (blast design)	queue time upon loading [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		0.1	0.1		0.1
schist + dolomite (JT)	0.1	0.1			0.1
dolomite (RS)				0.3	0.3
dolomite (JT)			0.0	0.2	0.1
average	0.1	0.1	0.1	0.2	0.1

material (blast design)	reverse time upon loading [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		0.3	0.3		0.3
schist + dolomite (JT)	0.5	0.4			0.5
dolomite (RS)				0.3	0.3
dolomite (JT)			0.2	0.4	0.3
average	0.5	0.4	0.2	0.3	0.3

Table 12.14: Load and haul measurements per material, blast design and slice, *bloc 11* at load site (1 / 3)

material (blast design)	total queue time [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		0.3	0.2		0.3
schist + dolomite (JT)	0.2	0.2			0.2
dolomite (RS)				0.5	0.5
dolomite (JT)			0.1	0.5	0.3
average	0.2	0.3	0.2	0.5	0.3

material (blast design)	total reverse time at load site [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		0.5	0.3		0.4
schist + dolomite (JT)	0.6	0.5			0.6
dolomite (RS)				0.3	0.3
dolomite (JT)			0.2	0.5	0.4
average	0.6	0.5	0.3	0.4	0.4

material (blast design)	time per load cycle [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		2.6	2.9		2.7
schist + dolomite (JT)	3.0	2.7			2.8
dolomite (RS)				3.1	3.1
dolomite (JT)			2.4	2.8	2.6
average	3.0	2.7	2.7	2.9	2.8

material (blast design)	load cycle per hour [1]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		23.4	20.5		21.9
schist + dolomite (JT)	20.3	21.8			21.1
dolomite (RS)				19.6	19.6
dolomite (JT)			24.9	21.5	23.2
average	20.3	22.6	22.7	20.5	21.7

material (blast design)	time per haul & return cycle [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		11.1	10.1		10.6
schist + dolomite (JT)	11.6	10.9			11.3
dolomite (RS)				12.6	12.6
dolomite (JT)			9.3	11.3	10.3
average	11.6	11.0	9.7	11.9	11.0

material (blast design)	haul & return cycle per hour [1]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		5.4	5.9		5.7
schist + dolomite (JT)	5.2	5.5			5.3
dolomite (RS)				4.8	4.8
dolomite (JT)			6.4	5.3	5.9
average	5.2	5.4	6.2	5.0	5.5

material (blast design)	buckets per truck [1]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		3.9	3.6		3.7
schist + dolomite (JT)	4.0	4.0			4.0
dolomite (RS)				4.0	4.0
dolomite (JT)			4.0	4.0	4.0
average	4.0	3.9	3.8	4.0	3.9

material (blast design)	fill factor per bucket [1]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		2.8	2.9		2.8
schist + dolomite (JT)	2.5	2.8			2.6
dolomite (RS)				2.4	2.4
dolomite (JT)			2.8	2.7	2.7
average	2.5	2.8	2.8	2.6	2.7

without last bucket if fill factor is 1

Table 12.15: Load and haul measurements per material, blast design and slice, *bloc 11* at load site (2 / 3)

material (blast design)	time per bucket [s]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		25	28		26
schist + dolomite (JT)	27	29			28
dolomite (RS)				33	33
dolomite (JT)			27	30	28
average	27	27	27	32	28

material (blast design)	buckets per hour [1]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		146	128		137
schist + dolomite (JT)	133	123			128
dolomite (RS)				109	109
dolomite (JT)			135	119	127
average	133	135	132	114	128

material (blast design)	time at loading site (excl. leaving) [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		2.0	1.7		1.9
schist + dolomite (JT)	2.3	2.1			2.2
dolomite (RS)				2.4	2.4
dolomite (JT)			1.7	2.9	2.3
average	2.3	2.1	1.7	2.7	2.2

material (blast design)	total time at load site [min]				
	slice 1	slice 3	slice 4	slice 5	aver.
schist + dolomite (RS)		2.5	2.1		2.3
schist + dolomite (JT)	2.7	2.6			2.6
dolomite (RS)				2.8	2.8
dolomite (JT)			2.1	3.3	2.7
average	2.7	2.5	2.1	3.1	2.6

Table 12.16: Load and haul measurements per material, blast design and slice, *bloc 11* at load site (3 / 3)

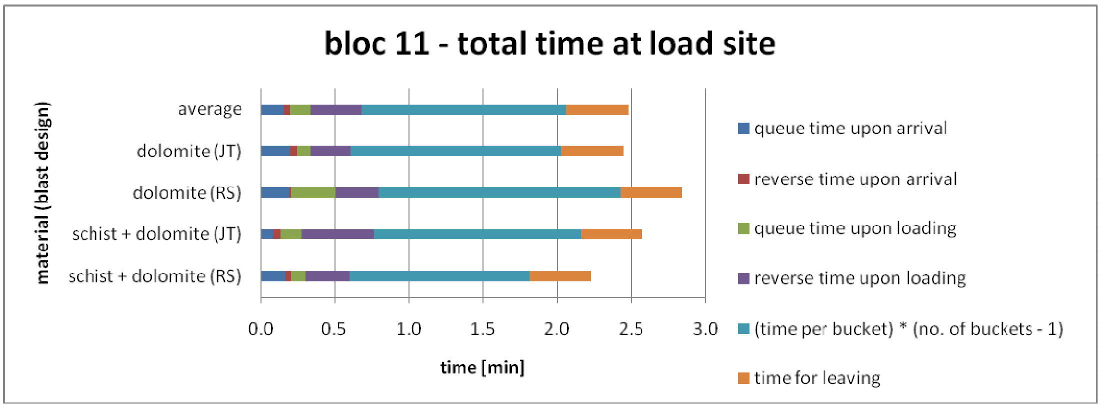


Figure 12.7: Total time at load site per material and blast design, *bloc 11*

		time per bucket activity [s]				
		normal loading	ripping	cleaning	stones	average
slice 1	schist + dolo. (RS)					
	schist + dolo. (JT)	26	40	30		32
	dolomite (RS)					
	dolomite (JT)					
slice 3	schist + dolo. (RS)	25		25		25
	schist + dolo. (JT)	29	34	30		31
	dolomite (RS)					
	dolomite (JT)					
slice 4	schist + dolo. (RS)	28	30	30	30	29
	schist + dolo. (JT)					
	dolomite (RS)					
	dolomite (JT)	26	30	28		28
slice 5	schist + dolo. (RS)					
	schist + dolo. (JT)					
	dolomite (RS)	31	33	41	33	34
	dolomite (JT)	29	34	35		33

Table 12.17: Time per bucket activity, material, blast design and slice, *bloc 11*

		activity per bucket [%]			
		normal loading	ripping	cleaning	stones
slice 1	schist + dolo. (RS)				
	schist + dolo. (JT)	93.8	3.1	3.1	0.0
	dolomite (RS)				
	dolomite (JT)				
slice 3	schist + dolo. (RS)	97.1	1.4	1.4	0.0
	schist + dolo. (JT)	92.0	6.9	1.1	0.0
	dolomite (RS)				
	dolomite (JT)				
slice 4	schist + dolo. (RS)	89.2	4.3	4.3	2.2
	schist + dolo. (JT)				
	dolomite (RS)				
	dolomite (JT)	91.2	1.5	7.4	0.0
slice 5	schist + dolo. (RS)				
	schist + dolo. (JT)				
	dolomite (RS)	72.8	11.2	13.6	2.4
	dolomite (JT)	76.8	10.7	12.5	0.0

Table 12.18: Bucket activity per material, blast design and slice, *bloc 11*

		fill factor per bucket activity (incl. last buc.) [1]				
	material (blast design)	normal loading	ripping	cleaning	stones	average
slice 1	schist + dolo. (RS)					
	schist + dolo. (JT)	2.5	3.0	2.0		2.5
	dolomite (RS)					
	dolomite (JT)					
slice 3	schist + dolo. (RS)	2.8	3.0	2.0		2.6
	schist + dolo. (JT)	2.7	2.8	3.0		2.9
	dolomite (RS)					
	dolomite (JT)					
slice 4	schist + dolo. (RS)	2.9	2.6	2.8	2.3	2.7
	schist + dolo. (JT)					
	dolomite (RS)					
	dolomite (JT)	2.9	2.0	2.4		2.4
slice 5	schist + dolo. (RS)					
	schist + dolo. (JT)					
	dolomite (RS)	2.5	2.3	2.3	3.0	2.5
	dolomite (JT)	2.7	2.4	2.7		2.6

Table 12.19: Bucket fill factor (incl. last bucket) per activity, material, blast design and slice, *bloc 11*

title:		load & haul measurements (load site)	
bloc		11	
truck		HD 985-5	C 777 D
queue time upon arrival	[min]	0.1	0.0
queue time upon loading	[min]	0.2	0.1
total queue time	[min]	0.3	0.1
reverse time upon arrival	[min]	0.0	0.0
reverse time upon loading	[min]	0.3	0.3
total reverse time at load site	[min]	0.4	0.3
time per load cycle	[min]	2.8	2.3
load cycle per hour	[1]	21.2	26.6
time per haul & return cycle	[min]	10.9	10.5
haul & return cycle per hour	[1]	5.5	5.7
buckets per truck	[1]	3.9	3.5

Table 12.20: Comparison between *HD 985-5* and *C 777 D*, *bloc 11*

title: load & haul measurements (load site)

bloc			11	
loader			R 994 B	C 992 G
type			backhoe excavator	wheel loader
no. of trucks		[1]	4.0	3.0
time at load site (excl. leaving)		[min]	2.2	4.4
time at load site		[min]	2.6	4.8
hang time (ex. 1st bucket)		[min]	1.0	2.7
total hang time		[min]	1.4	3.8
queue time upon arrival		[min]	0.2	0.0
queue time upon loading		[min]	0.1	0.0
total queue time		[min]	0.3	0.0
reverse time upon arrival		[min]	0.0	0.0
reverse time upon loading		[min]	0.3	0.2
total reverse time at load site		[min]	0.4	0.2
time per load cycle		[min]	2.8	7.9
load cycle per hour		[1]	21.6	7.6
time per haul & return cycle		[min]	11.0	22.7
haul & return cycle per hour		[1]	5.5	2.6
buckets per truck		[1]	3.9	4.9
time per bucket		[s]	28	63
buckets per hour		[1]	128	57
fill factor per bucket (without last)		[1]	2.7	2.4
time per bucket activity	normal loading	[s]	28	64
	ripping	[s]	33	64
	cleaning	[s]	32	
	stones	[s]	31	45
percent per bucket activity	normal loading	[%]	87	26
	ripping	[%]	6	72
	cleaning	[%]	7	0
	stones	[%]	1	3
fill factor per bucket activity	normal loading	[1]	2.4	2.7
	ripping	[1]	2.3	2.2
	cleaning	[1]	2.2	0.0
	stones	[1]	2.1	3.0

Table 12.21: Comparison between *R 994 B* and *C 992 G*, *bloc 11*

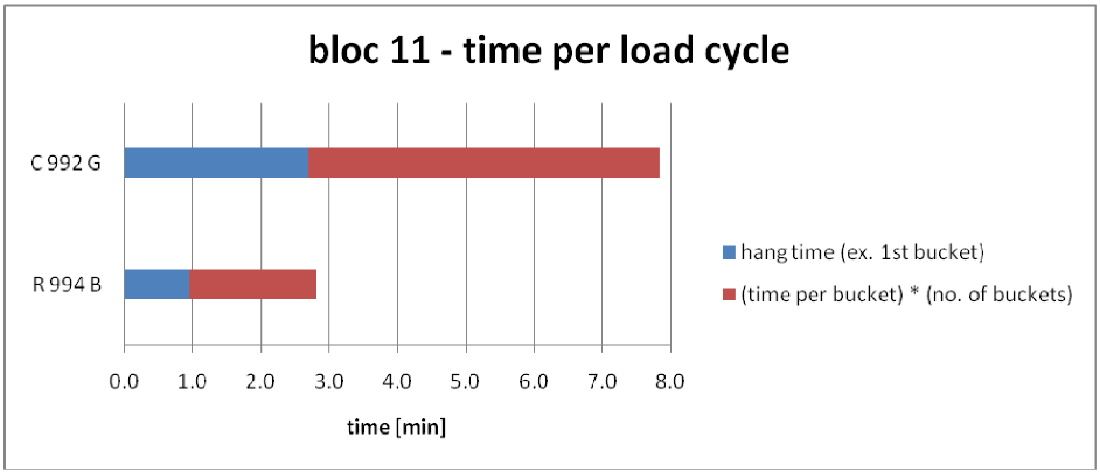


Figure 12.8: Time per load cycle for *R 994 B* and *C 992 G*, *bloc 11*

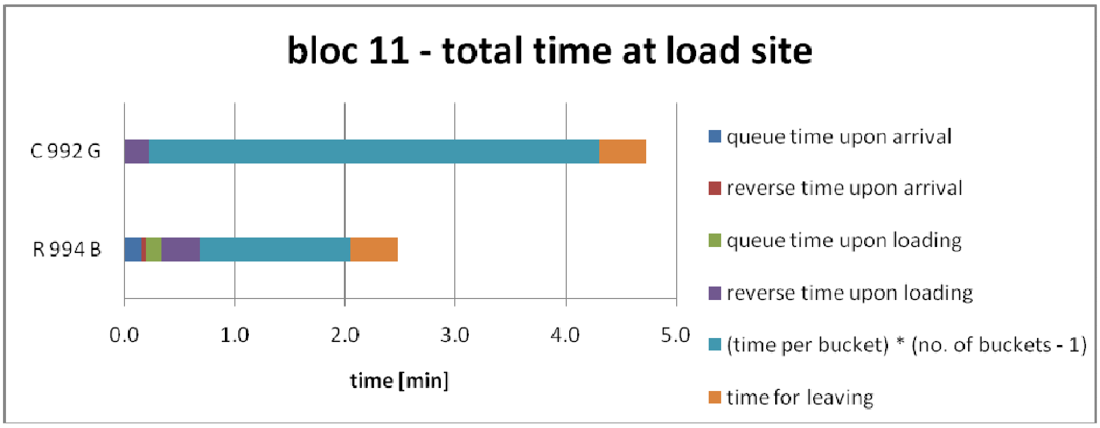


Figure 12.9: Total time at load site for *R 994 B* and *C 992 G*, *bloc 11*

12.2.4 Load and haul measurements on truck – *bloc 3* [98]

title:		load & haul measurements (on truck)					
bloc			3				
rock			dolomite	dolomite + graphite	schist	schist + marble	average
no. of trucks			5.5	0.5	0.5	0.8	2.2
time per dumping (U)		[min]	0.7	0.7	0.7	0.8	0.7
reverse time at load site (LS)		[min]	0.6	0.4	0.3	1.1	0.4
reverse time at dump site (DS)		[min]	0.7	0.4	0.3	0.3	0.5
total reverse time		[min]	1.3	0.8	0.6	1.5	0.9
total queue time (LS)		[min]	0.3	0.2	0.2	3.4	0.2
waiting time while haul (H)		[min]	0.1	0.0	0.0	0.0	0.0
waiting time while return (R)		[min]	0.2	0.2	0.0	0.0	0.2
total waiting time		[min]	0.5	0.4	0.2	3.4	0.4
time at load site (excl. leaving)		[min]	2.5	2.3	2.0	7.8	2.3
total time at load site (LS)		[min]	2.9	2.8	2.4	8.3	2.7
total time at dump site (DS)		[min]	1.4	1.0	1.0	1.1	1.1
total time while haul (H)		[min]	5.1	4.7	6.0	6.4	5.3
total time while return (R)		[min]	4.0	4.7	5.1	5.8	4.6
time per haul & return cycle		[min]	12.9	13.8	14.1	21.5	13.6
haul & return cycle per hour		[1]	4.7	4.3	4.3	2.8	4.4
buckets per truck		[1]	4.3	4.1	4.1	5.3	4.2
time per bucket		[sec]	27	33	28	48	30
buckets per hour		[1]	132	109	127	75	123
weight per bucket		[t]	24	27	27	20	26
distance	A	[m]	310	400	475	575	395
	B	[m]	825	825	825	825	825
	C	[m]	845	845	845	845	845
	D	[m]	55	25	65	45	48
	total	[m]	2,035	2,095	2,210	2,290	2,113
haul time	A	[min]	1.4	1.2	1.4	1.8	1.3
	B	[min]	1.3	1.6	1.6	1.6	1.5
	C	[min]	1.9	2.3	2.3	2.3	2.2
	D	[min]	0.8	0.4	0.7	0.6	0.6
	total	[min]	5.4	5.5	6.0	6.4	5.6
return time	D	[min]	0.2	0.2	0.4	0.5	0.3
	C	[min]	2.0	2.4	2.1	3.1	2.2
	B	[min]	1.0	1.2	1.1	1.6	1.1
	A	[min]	1.1	0.9	1.4	0.7	1.1
	total	[min]	4.3	4.6	5.1	5.8	4.7
haul speed	A	[km/h]	13.8	19.9	17.8	18.8	17.2
	B	[km/h]	38.2	31.8	31.2	30.9	33.7
	C	[km/h]	27.1	21.6	22.0	21.9	23.6
	D	[km/h]	5.0	3.6	6.9	4.5	5.2
	total	[km/h]	22.8	22.7	21.7	21.6	22.4
return speed	D	[km/h]	15.1	7.7	11.0	5.4	11.3
	C	[km/h]	25.6	21.4	24.0	16.4	23.7
	B	[km/h]	50.8	41.2	46.8	31.8	46.2
	A	[km/h]	17.1	28.2	17.5	49.7	20.9
	total	[km/h]	28.6	27.2	25.9	23.6	27.2

Table 12.22: Summary of load and haul measurements, *bloc 3* on truck

[98] calc_bloc3_truck.xls

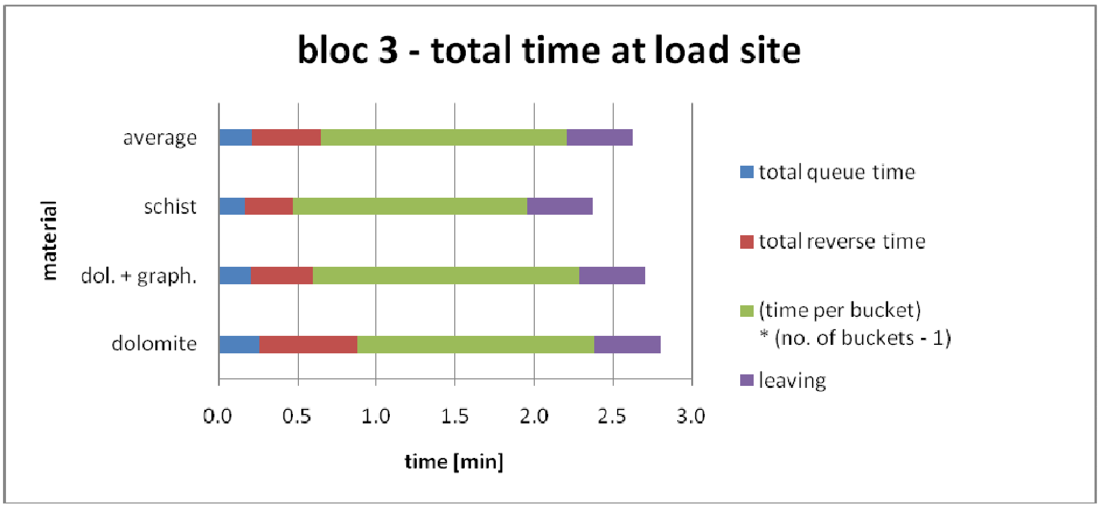


Figure 12.10: Total time at load site per material, *bloc 3*

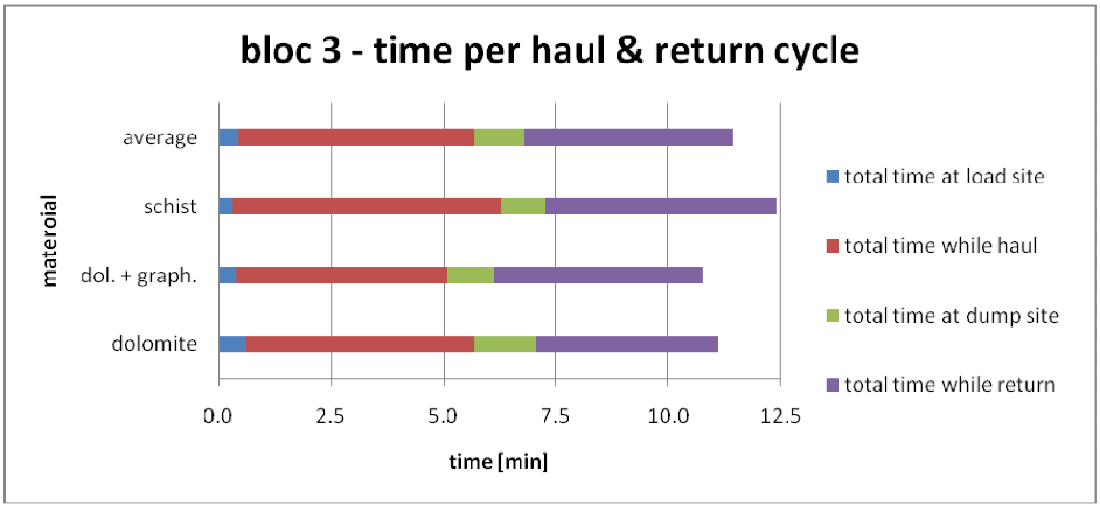


Figure 12.11: Haul and return cycle time per material, *bloc 3*

12.2.5 Load and haul measurements at dump site – *trench* [99]

title:	load & haul measurements (waste dump)		
bloc			trench
no. of trucks		[1]	3
av. distance		[m]	1,145
time per dumping (U)		[min]	0.6
reverse time at dump site		[min]	0.4
waiting time at dump site (DS)		[min]	1.0
waiting time while haul (H)		[min]	0.0
waiting time while return (R)		[min]	0.8
total waiting time		[min]	0.8
time per haul & return cycle		[min]	14.4
haul & return cycle per hour		[1]	4.2
distance	C	[m]	455
	D	[m]	125
	total	[m]	1,500
haul time	C	[min]	1.1
	D	[min]	1.0
return time	D	[min]	1.0
	C	[min]	1.1
haul speed	C	[km/h]	25.7
	D	[km/h]	7.3
return speed	D	[km/h]	8.2
	C	[km/h]	25.2

Table 12.23: Summary of load and haul measurements, *trench* at dump site

[99] calc_trench_dump.xls

12.3 Number of trucks [100]

title: number of trucks calculated via actual & modified approach

			actual	modified	actual	modified
cap_L		[m³]	12.4	12.4	12.4	12.4
cap_T		[m³]	37.0	37.0	37.0	37.0
d		[m]	2,600	2,600	2,200	2,200
v		[min]	21.0	21.0	21.0	21.0
rev		[min]	2.0	1.3	2.0	1.3
D		[min]	0.0	0.7	0.0	0.7
$prod_L$		[m³ / h]	600	893	600	893
cyc_L		[min]	1.2	0.8	1.2	0.8
$prod_T$		[m³ / h]	108	119	122	136
cyc_T		[min]	20.6	18.6	18.3	16.3
n_T		[1]	5.6	7.5	4.9	6.6
$prod_L$	$n_T = 3$	[m³ / h]	324	358	365	408
	$n_T = 4$	[m³ / h]	432	478	486	545
	$n_T = 5$	[m³ / h]	540	597	608	681
	$n_T = 6$	[m³ / h]	648	716	729	817

ref.: rtm_plan_exploitation_2009.doc
calc_sum_load_haul.xls

Figure 12.12: Number of trucks calculated via actual and modified approach

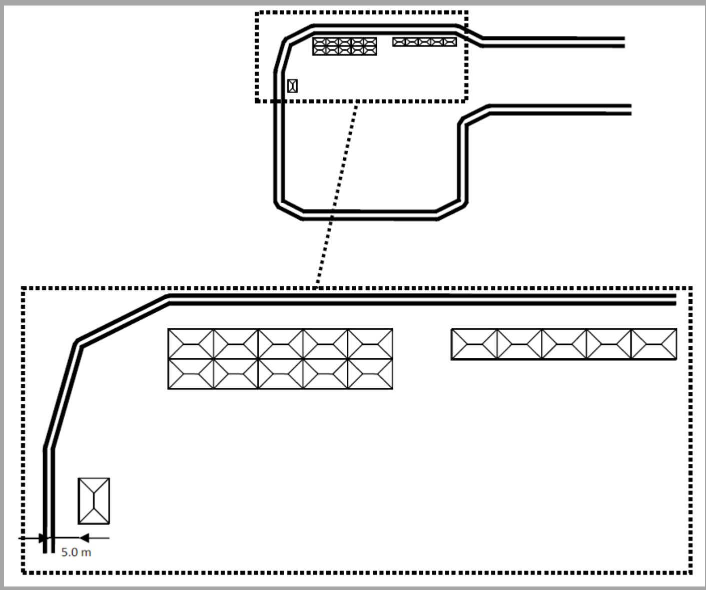
[100] calc_no_trucks.xls

12.4 Auxiliary Equipment

12.5 Push time

title:	push measurements (dump site)				
date:	05/10/2009	start time(s):	08:55, 10:00, 10:47	material:	unclean talc
bloc:	trench	slice:	1	blast design:	no blast
dozer:	PR764-T2	no. of trucks:	3	trucks:	H1, H2, H4

sketch:



	loads per pile	average [min]	std. dev. [min]	minimum [min]	maximum [min]
time per pushing (1 pile)	one	2.8	0.4	2.3	3.0
	five	2.7	1.0	0.8	4.9
	ten	2.8	0.9	1.3	4.0

Table 12.24: Summary of push measurements at waste dump [101]

[101] calc_time_push.xls

References

Published and electronic sources

BlastMetriX3D 2007, *3G Software & Measurement GmbH*, Bench face measurement and planning of blasts using metric 3D images, viewed 06. August 2009, http://www.3gsm.at/downloads/BMX_Info_en_2.0.pdf, pp. 3-5, 10 of 15

Calmein, M., Jones, L., Pons, R., Robert, J.-F., Vinandy, G. 2005, *The epic of Luzenac talc*, Talc de Luzenac, France.

Howsen, M. P. 2000, 'Resource modelling at Talc de Luzenac, France', in P. W. Scott & C. M. Bristow (eds), *Industrial Minerals and Extractive Industry Geology*.

Rio Tinto Minerals n.d.a, *Facts and Figures*, viewed 06. August 2009, http://rtm-intranet.corp.riotinto.org/operations/Europe/Facts_and_Figures_Luzenac_Operations.aspx

Rio Tinto Minerals n.d.b, *Luzenac Operations*, viewed 06. August 2009, http://rtm-intranet.corp.riotinto.org/operations/Europe/luzenac_operations_landing_page.aspx

Technical description – D 275 A2 n.d., *Komatsu*, viewed 06. August 2009, http://www.schwickert-baumaschinen.de/_mediafiles/191-komatsu_dozer_d_275_a-2_engl..pdf

Technical description – C 777 D n.d., *Caterpillar*, viewed 06. August 2009, http://korea.cat.com/cmms/images/C229910.pdf

Technical description – C 997 G n.d., *Caterpillar*, viewed 06. August 2009, http://deutschland.cat.com/cmms/images/C198766.pdf

Technical description – HD 985-5 n.d., *Komatsu*, viewed 06. August 2009, http://www.komatsu.eu/new_equipment/displayFile.ashx?fileId=15636

Technical description – PR 764 n.d., *Liebherr*, viewed 06. August 2009, http://liebherr.com/catXmedia/em/Documents/7a5a60b9-550e-4619-bc8b-7c41b643e254.pdf

Technical description – R 994 B n.d., *Liebherr*, viewed 06. August 2009, http://www.monnis.com/File/liebherr_R994B.pdf

Wyllie, C. W. & Mah, C. W. 2007, *Rock slope engineering: civil and mining*, 4^{th} edn, E. Hoek, E. & J. Bray (eds), Institute of Mining and Metallurgy, USA.

Digital sources (on attached CD)

BlastMetriX3D models and related files

Name	Reference / description
bm_bloc11_blast85.pdf	*Bloc 11*, blast 85
bm_bloc11_blast85.smb	*Bloc 11*, blast 85
bm_bloc11_blast87.pdf	*Bloc 11*, blast 87
bm_bloc11_blast87.smb	*Bloc 11*, blast 87
Bm_bloc11_blasts.pdf	*Bloc 11*, blasts 85 and 87
bm_bloc11_holes.smb	*Bloc 11*, blasts 85 and 87
bm_bloc11_holes_boulder.smb	*Bloc 11*, blasts 85 and 87, boulder
bm_bloc11_ref.jm3	*Bloc 11*
bm_bloc11_ref.smb	*Bloc 11*
bm_bloc3_01_blast59.pdf	*Bloc 3*, blast 59
bm_bloc3_01_blast59.smb	*Bloc 3*, blast 59
bm_bloc3_01_blast63.pdf	*Bloc 3*, blast 63
bm_bloc3_01_blast63.smb	*Bloc 3*, blast 63
bm_bloc3_01_blasts.pdf	*Bloc 3*, blast 59 and 63
bm_bloc3_01_holes.smb	*Bloc 3*, blast 59 and 63
bm_bloc3_01_ref.jm3	*Bloc 3*, North
bm_bloc3_01_ref.smb	*Bloc 3*, North
bm_bloc3_02_blast61.pdf	*Bloc 3*, blast 61
bm_bloc3_02_blast61.smb	*Bloc 3*, blast 61
bm_bloc3_02_ref.jm3	*Bloc 3*, South
bm_bloc3_02_ref.smb	*Bloc 3*, South
bm_bloc4_blast101_limit.pdf	*Bloc 4*, blast 101, incl. limit
bm_bloc4_blast101_limit.smb	*Bloc 4*, blast 101, incl. limit
bm_bloc4_blast103_limit.pdf	*Bloc 4*, blast 103, incl. limit
bm_bloc4_blast103_limit.smb	*Bloc 4*, blast 103, incl. limit
bm_bloc4_blast105_limit.pdf	*Bloc 4*, blast 105, incl. limit
bm_bloc4_blast105_limit.smb	*Bloc 4*, blast 105, incl. limit
bm_bloc4_blast107_limit.pdf	*Bloc 4*, blast 107, incl. limit
bm_bloc4_blast107_limit.smb	*Bloc 4*, blast 107, incl. limit
bm_bloc4_blast109_limit.pdf	*Bloc 4*, blast 109, incl. limit
bm_bloc4_blast109_limit.smb	*Bloc 4*, blast 109, incl. limit

bm_bloc4_blast99_limit.pdf	*Bloc 4*, blast 99, incl. limit
bm_bloc4_blast99_limit.smb	
bm_bloc4_blasts_limit.pdf	*Bloc 4*, blast 99, 101, 103, 105, 107 and 109, incl. limit
bm_bloc4_holes_limit.smb	
bm_bloc4_ref.jm3	*Bloc 4*
bm_bloc4_ref.smb	
bm_boulder_bloc11_01.idx	Boulder surveyed via total station, *bloc 11*
bm_holes_bloc11.idx	Boreholes surveyed via total station, *bloc 11*
bm_holes_bloc3_01.idx	Boreholes surveyed via total station, *bloc 3*
bm_holes_bloc3_02.idx	Boreholes surveyed via total station, *bloc 3*
bm_holes_bloc4_01.idx	Boreholes surveyed via total station, *bloc 4*
bm_holes_bloc4_02.idx	Boreholes surveyed via total station, *bloc 4*
bm_holes_boulder_limit.xls	Boreholes and boulder per bloc and blast

Calculations done by the author

Name	**Reference / description**
calc_activity_FOD_09.xls	Activities & FOD 2009 (feb - aug)
calc_blast_exp_09.xls	Drill and blast parameter 2009
calc_bloc11_load.xls	Load and haul measurements at load site (*bloc 11*)
calc_bloc3_load.xls	Load and haul measurements at load site (*bloc 3*)
calc_bloc3_truck.xls	Load and haul measurements on truck (*bloc 3*)
calc_bm_sum.xls	Summary of drill and blast documentation
calc_costs_09.xls	Costs per hour (apr - aug)
calc_expl_09.xls	Use of explosives 2009
calc_factors_09.xls	Utilisation, standby & availability 2009 (apr - aug)
calc_macro.xls	Macro of load and haul measurements
calc_no_trucks.xls	Number of trucks calculated via actual and modified approach
calc_prod_bloc_09_01.xls	Production per bloc (RTM)
calc_prod_bloc_09_02.xls	Production per bloc (Logimine)
calc_sum_load_haul.xls	Summary of load and haul measurements
calc_survey_holes.xls	Opening time and water filling of boreholes
calc_time_charge.xls	Charge time measurements
calc_time_dril.xls	Drill time measurements
calc_time_push.xls	Push measurements (dump site)
calc_trench_dump.xls	Load and haul measurements at dump site (*trench*)

Videos taken by the author

Name	Reference / description
video_b11_83_1408.wmv	Blast 83, *bloc 11*, no ejection
video_b11_85_2508_01.wmv	Blast 85, *bloc 11*, ejection of 1 hole
video_b11_85_2508_02.wmv	Charging of borehole with explosives
video_b11_85_2508_03.wmv	Blowing out of water filled borehole
video_b11_85_2508_05.wmv	
video_b3_59_2807.wmv	Blast 59, *bloc 3*, ejection of 1 hole
video_b3_61_2907.wmv	Blast 61, *bloc3*, ejection of 1 hole
video_b3_63_3007.wmv	Blast 63, *bloc 3*, no ejection
video_b4_101_0409.wmv	Blast 101, *bloc 4*, no ejection
video_b4_105_0809.wmv	Blast 105, *bloc 4*, no ejection
video_b4_107_0909.wmv	Blast 107, *bloc 4*, ejection of 2 holes
video_b4_109_1009.wmv	Blast 109, *bloc 4*, no ejection
video_cycle_b3s3_1208_03.wmv	Loading and hauling, *bloc 3*, slice 3, uneven dump site
video_dump_b3_2608_01.wmv	Dumping, *bloc 3*, dozer waving in
video_haul_b11_0209_01.wmv	Hauling, *bloc 11*, dozer waving in
video_load_b11s3_0409_01.wmv	Loading, *bloc 11*, slice 3, queuing
video_load_b11s4_0709_01.wmv	Loading, *bloc 11*, slice 4, normal loading, uneven load site
video_load_b3s3_1208_01.wmv	Loading, *bloc 3*, slice 3, normal loading
video_load_b3s3_1208_02.wmv	Loading, *bloc 3*, slice 3, queuing
video_load_b3s3_1408_04.wmv	Loading, *bloc 3*, slice 3, uneven load site
video_load_b3s4_2608_01.wmv	Loading, *bloc 3*, slice 4, easy front and difficult last rows
video_load_b3s4_2608_02.wmv	Loading, *bloc 3*, slice 4, ripping
video_load_b3s4_2608_03.wmv	Loading, *bloc 3*, slice 4, easy front and difficult last rows
video_load_b3s4_2608_04.wmv	Loading, *bloc 3*, slice 4, easy front and difficult last rows
video_load_b5_2508_02.wmv	Loading via C 777 G, *bloc 3*, slice 2

Information extracted from Logimine

Name	Reference / description
log_b11_77_1108.xls	Tir n°77 du 11/08/2009 poste 1, viewed 01. August 2009
log_b11_81_1308.xls	Tir n°81 du 13/08/2009 poste 1, viewed 01. August 2009
log_b11_83_1408.xls	Tir n°83 du 14/08/2009 poste 1, viewed 01. August 2009
log_b11_87_2508.xls	Tir n°75 du 10/08/2009 poste 1, viewed 01. August 2009
log_b11_HD4.pdf	CYCLE LIST HD.4 - BLOC11, 23/7/2009 to 23/9/2009
log_b11_sum_exc.pdf	Production Engins de Chargement du 22/07/2009 au 22/09/2009 dans la zone : BLOC11, viewed 23. August 2009
log_b11_sum_truck.pdf	Statistiques de Production du 22/07/2009 au 22/09/2009 dans la zone : BLOC11, viewed 23. August 2009
log_b3_57_2707.xls	Tir n°57 du 27/07/2009 poste 1, viewed 01. August 2009
log_b3_59_2807.xls	Tir n°59 du 28/07/2009 poste 1, viewed 01. August 2009
log_b3_61_2907.xls	Tir n°61 du 29/07/2009 poste 1, viewed 01. August 2009
log_b3_63_3007.xls	Tir n°63 du 30/07/2009 poste 1, viewed 01. August 2009
log_b3_65_3107.xls	Tir n°65 du 31/07/2009 poste 1, viewed 01. August 2009
log_b3_67_0308.xls	Tir n°67 du 03/08/2009 poste 1, viewed 01. August 2009
log_b3_69_0408.xls	Tir n°69 du 04/08/2009 poste 1, viewed 01. August 2009
log_b3_71_0508.xls	Tir n°71 du 05/08/2009 poste 1, viewed 01. August 2009
log_b3_74_0708.xls	Tir n°74 du 07/07/2009 poste 1, viewed 01. August 2009
log_b3_HD4.pdf	CYCLE LIST HD.4 - BLOC3, 23/7/2009 to 23/9/2009
log_b3_sum_exc.pdf	Production Engins de Chargement du 22/07/2009 au 22/09/2009 dans la zone : BLOC3, viewed 23. August 2009
log_b3_sum_truck.pdf	Statistiques de Production du 22/07/2009 au 22/09/2009 dans la zone : BLOC3, viewed 23. August 2009
log_b4_101_0409.xls	Tir n°101 du 04/09/2009 poste 1 , viewed 23. August 2009
log_b4_103_0709.xls	Tir n°103 du 07/09/2009 poste 1 , viewed 23. August 2009
log_b4_105_0809.xls	Tir n°105 du 08/09/2009 poste 1 , viewed 23. August 2009
log_b4_107_0909.xls	Tir n°107 du 09/09/2009 poste 1 , viewed 23. August 2009
log_b4_109_1009.xls	Tir n°109 du 10/09/2009 poste 1 , viewed 23. August 2009
log_b4_95_0109.xls	Tir n°95 du 01/09/2009 poste 1, viewed 23. August 2009
log_b4_97_0209.xls	Tir n°97 du 02/09/2009 poste 1, viewed 23. August 2009
log_b4_99_0309.xls	Tir n°99 du 03/09/2009 poste 1, viewed 23. August 2009

Information provided by Rio Tinto Minerals

Name	Reference / description
rtm_anfo.pdf	Anfotite N° 1, Technical data sheet, copy received summer 2009
rtm_b11_77_1108.xls	Blast documentation received from Sarda, R.
rtm_b11_81_1308.xls	Blast documentation received from Sarda, R.
rtm_b11_83_1408.xls	Blast documentation received from Sarda, R.
rtm_b11_85_2408.xls	Blast documentation received from Sarda, R.
rtm_b11_87_2508.xls	Blast documentation received from Sarda, R.
rtm_b3_57_2707.xls	Blast documentation received from Sarda, R.
rtm_b3_59_2807.xls	Blast documentation received from Sarda, R.
rtm_b3_61_2907.xls	Blast documentation received from Sarda, R.
rtm_b3_63a_3007.xls	Blast documentation received from Sarda, R.
rtm_b3_63b_3007.xls	Blast documentation received from Sarda, R.
rtm_b3_65_3107.xls	Blast documentation received from Sarda, R.
rtm_b3_67_0308.xls	Blast documentation received from Sarda, R.
rtm_b3_69_0408.xls	Blast documentation received from Sarda, R.
rtm_b3_71_0508.xls	Blast documentation received from Sarda, R.
rtm_b3_74_0708.xls	Blast documentation received from Sarda, R.
rtm_b4_101_0409.xls	Blast documentation received from Sarda, R.
rtm_b4_103_0709.xls	Blast documentation received from Sarda, R.
rtm_b4_105_0809.xls	Blast documentation received from Sarda, R.
rtm_b4_107_0909.xls	Blast documentation received from Sarda, R.
rtm_b4_109_1009.xls	Blast documentation received from Sarda, R.
rtm_b4_95_0109.xls	Blast documentation received from Sarda, R.
rtm_b4_97_0209.xls	Blast documentation received from Sarda, R.
rtm_b4_99_0309.xls	Blast documentation received from Sarda, R.
rtm_budget2008.pdf	Operating costs (mine), December 2008, partial copy received summer 2009
rtm_coeff_auxiliary_0809.xls	P:\Tlf Rapports Techniques\Carriere\2009\2009\08-Août\Découverture\Autres engins découverture\Coefficients autres engins découverture 06.09.xls
rtm_coeff_drill_0809.xls	P:\Tlf Rapports Techniques\Carriere\2009\2009\08-Août\Perforation et minage\Ratios consommation perfo-minage.xls
rtm_coeff_haul_0809.xls	P:\Tlf Rapports Techniques\Carriere\2009\2009\08-Août\Découverture\Engins de roulage\Coefficients engins de roulage.xls
rtm_coeff_load_0809.xls	P:\Tlf Rapports Techniques\Carriere\2009\2009\08-Août\Découverture\Engins de chargement\Coefficients engins de chargement.xls

rtm_costs_C777C-1.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient egins\ Evolution Coûts de revient engins en €\Roulage Découverture\ CAT.777CN°1.xls
rtm_costs_C777C-1.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient egins\ Evolution Coûts de revient engins en €\Roulage Découverture\ CAT.777CN°2.xls
rtm_costs_C992G.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient egins\ Evolution Coûts de revient engins en €\Chargement Découverture\ CAT.992G.xls
rtm_costs_D25KS.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\ Evolution Coûts de revient engins en €\Foration\Tamrock D25 KS.xls
rtm_costs_D275A2.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\Evolution Coûts de revient engins en €\Bulldozers\ KOMT.D275 A2.xls
rtm_costs_HD985-1.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\ Evolution Coûts de revient engins en €\Roulage Découverture\Komatsu HD 985 n°1.xls
rtm_costs_HD985-2.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\ Evolution Coûts de revient engins en €\Roulage Découverture\Komatsu HD 985 n°2.xls
rtm_costs_HD985-3.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\ Evolution Coûts de revient engins en €\Roulage Découverture\Komatsu HD 985 n°3.xls
rtm_costs_HD985-4.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\ Evolution Coûts de revient engins en €\Roulage Découverture\Komatsu HD 985 n°4.xls
rtm_costs_R994B.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\ Evolution Coûts de revient engins en €\Chargement Découverture\L994 N°2.xls
rtm_costs_summary.xls	O:\Methodes\BUREAU METHODE CARRIERE\Coûts de revient engins\ Coûts de revient engins 2008\SOMMAIRE.xls
rtm_D25KS.pdf	Drill rig D 25 KS, Technical description and contract of sale (2000), partial copy received summer 2009
rtm_drill_blast_0509.xls	P:\TIf Rapports Techniques\Carriere\2009\2009\05-Mai\Perforation et minage\Ratios consommation perfo-minage.xls
rtm_drill_blast_0609.xls	P:\TIf Rapports Techniques\Carriere\2009\2009\06-Juin\Perforation et minage\Ratios consommation perfo-minage.xls
rtm_drill_blast_0709.xls	P:\TIf Rapports Techniques\Carriere\2009\2009\07-Juillet\Perforation et minage\Ratios consommation perfo-minage.xls
rtm_drill_blast_0809.xls	P:\TIf Rapports Techniques\Carriere\2009\2009\08-Août\Perforation et minage\Ratios consommation perfo-minage.xls
rtm_emul.pdf	Emulstar 5000, Technical data sheet, copy received summer 2009
rtm_FOD_hours_0809.xls	P:\TIf Rapports Techniques\Carriere\2009\2009\08-Août\Découverture\ Autres engins découverture\Consommations et hres marche engins 07.09.xls

rtm_haul_bloc_0809.xls	P:\Tlf Rapports Techniques\Carriere\2009\2009\08-Août\Découverture\Engins de roulage\Roulage découverture par bloc août 2009.xls
rtm_load_bloc_0809.xls	P:\Tlf Rapports Techniques\Carriere\2009\2009\08-Août\Découverture\Engins de chargement\Roulage découverture par bloc août 2009.xls
rtm_map_01.dwg	O:\Geologie_Trimouns\Topographie\Topo_annuelle\Fin_2008 renommé\MODTOUT habillé_renommé.dwg
rtm_mine_process_2008.xls	Lanely, C. 2008, Mine Flow Chart and Capacity 2008, Trimouns Mine, Luzenac Operations, RTM LO.
rtm_plan_exploitation_2009.doc	Plan d'Exploitation / Short Term Mine Planning 2009, Trimouns Mine, Luzenac Operations, RTM LO.
rtm_plan_exploitation_2009.ppt	Plan d'Exploitation / Short Term Mine Planning 2009, Trimouns Mine, Luzenac Operations, RTM LO.
rtm_res_audit.pdf	Eggleston, T., da Silva, H., Kirkland, K. 2008, 'Resource and Reserve Audit, Trimouns Mine', amec, Project No.: 159365**.**
rtm_trimouns.ppt	Paris, P. 2006, Trimouns Talc – Chlorite Ore Body, Trimouns Mine, Luzenac Operations, RTM LO.

Information received via mail

mail_bh_1310.pdf	mail from paul-alain.pitach@riotinto.com, 19. October 2009, measurements follow-up
mail_bh_1410.pdf	mail from paul-alain.pitach@riotinto.com, 19. October 2009, measurements follow-up
mail_bh_1510.pdf	mail from paul-alain.pitach@riotinto.com, 19. October 2009, measurements follow-up
mail_bh_1610.pdf	mail from paul-alain.pitach@riotinto.com, 19. October 2009, measurements follow-up
mail_blasts_2009.csv	mail received from otto.vanderende@riotinto.com, 28. October 2009, Blast 2009.csv
mail_contrat_exp_2009.pdf	mail from didier.anglade@riotinto.com, 23. March 2009, RE: Contrat
mail_effet_arriere.pdf	mail from paul-alain.pitach@riotinto.com, 06. October 2009
mail_explosives_2009.csv	mail received from otto.vanderende@riotinto.com, 28. October 2009, Explosiv 2009.csv
mail_hopper.doc	mail from paul-alain.pitach@riotinto.com, October 2009, Trémie guillotine de bourrage
mail_marble_block.pdf	mail from paul-alain.pitach@riotinto.com, 24. September 2009