NTNU – Trondheim
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Head Race Tunnel Melado Hydropower Plant, Chile

Optimization in Excavation and Rock Support

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Hydropower Development
Submission date: June 2012
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Program for MSc-thesis

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The Melado Hydropower project located in the Melado basin within the VII region of Chile, 450 km south east of Santiago is currently at the planning stage. The distance from the intake to the pressure shaft is 18.8 km, of which 16.6 km is underground in two head race tunnels. Both tunnels should be able to convey 100 m^3/s.

This thesis is to focus on review, analysis and evaluation of potential improvement of the design, excavation and support of the two head race tunnels. The analysis is to based on applying state of the art Norwegian knowledge for evaluating the tunnel layouts as described in the feasibility report.

Particularly, the thesis is to cover the following points:

- The geological conditions in the area.
- Analysis of alternatives for tunnel alignment and rock support.
- Implementation of the best alternatives for rock support based on empirical methods and numerical modeling.
- Estimation of total cost and expected advance rate.
- Optimisation of tunnel cross section.
- Analysis of the strengths and weaknesses of TBM excavation for this project.

The thesis work starts January 11, 2012 and is to be completed by June 11, 2012.

The Norwegian University of Science and Technology (NTNU)
Department of Geology and Mineral Resources Engineering

January 12, 2012/revised March 14, 2012

Bjørn Nilsen
Professor of geological engineering, main supervisor

ACKNOWLEDGMENT

I would like to express my sincere gratitude to my advisor Professor Bjørn Nielsen, for his wise guidance, patience and good will to answer all kind of geological inquiries at short notice.

I would like to express as well my sincere thanks to Professor Amund Bruland and his invaluable information and practical counseling in so many aspects related to tunneling.

It is highly appreciated for me the unselfish help provided by the PhD. Student Pawan Shrestha in aspects related to rock numerical modeling.

Sincere thanks go to the Chilean company Colbún which sponsored me during the whole master program and all the executives who trusted in me. Also, my gratitude to Sr. Engineer Matias Cespedes for all the information provided during the development of this thesis.

Finally, I want to thank the patience, the goodwill and effort provided by my wife who joined me in this adventure throughout these two years.

Cristobal Manquehual

June, 2012.

EXECUTIVE SUMMARY

The present thesis is to focus on the two head race tunnels within the Melado HPP, Chile.

The upstream tunnel is called *Castillo* 7800 m long and the downstream tunnel is called *Vallical* 8800 m long.

The feasibility study of this project was carried in 2010. The resulting report from this study was used as a basis for this thesis. Along with all the background information obtained from the feasibility studies, there are some other sources gotten from field investigation tests carried out after the end of the feasibility report.

Based on all the information collected, a new rock mass classification was carried out, identifying all the geological stretches for both tunnels. This classification was undertaken according to the RMR system (Bieniawski, 1973) and the Q-system (NGI, 1974).

The support required for both tunnels was estimated mainly based on empirical rock support methods. Numerical model by means of software Phase 2 as well as ground behavior analysis were carried out to provide a better understanding of actual problems expected during excavation that are not always cover by the empirical methods.

Once there was a clear understating of the rock masses involved in both tunnels along with their corresponding rock support, some analysis related to the tunnels were performed to improve the economical benefit of the whole project.

The first economic analysis undertaken was the optimum cross section for Drill & Blast excavation method. A detail analysis that systematizes the cost and advance rate involved in the round cycle of the Drill & Blast excavation method and rock support was used. A thorough analysis of energy losses was also carried out as well, making a distinction between unlined, shotcrete lined and reinforced shotcrete ribs tunnel stretches. The existing cross section of 6.9 m horse shoe shape tunnel turned out to be very small. The new optimum cross section for Castillo tunnel was 8.7 m diameter and 9 m diameter for Vallical tunnel.

The second economic analysis was a tunnel alignment change in order to improve some blast design parameters in stretches where their tunnel axis orientation was close to predominant joint set. In any of the two tunnel stretches analyzed turned out to be economically convenient to extend the tunnel for this purpose, even though the safety factor for the crew was not included.

Also, the convenience of extra adits was carried out. In fact, the layout of the feasibility report included one extra adit for each tunnel in order to have an earlier tunnel completion. The findings suggest that these adits are not economically convenient as long as the project’s electromechanical equipments take 34 months from their manufacturing to their assembly on project cavern.

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1. INTRODUCTION

The head race tunnel is by far the most important item in the HPP Melado, accounting for 50% of the total project cost based on its feasibility report.

This thesis is to focus on the two head race tunnels within the Melado HPP, which together account 16.6 km out of 18.8 km head race long. A review of the design, excavation and support of the existing layout given by the feasibility report will be undertaken based on state of the art knowledge.

2. GENERAL PROJECT DESCRIPTION

2.1 Location

The *Guaiquivilo-Melado* project is located within the VII *Maule* Region, Chile. From *Santiago*, the *Guaiquivilo-Melado* project is around 300 km south in a straight line, but around 450 km through the existing roads.

The following picture shows the location of the VII Maule Region, its capital Talca and the

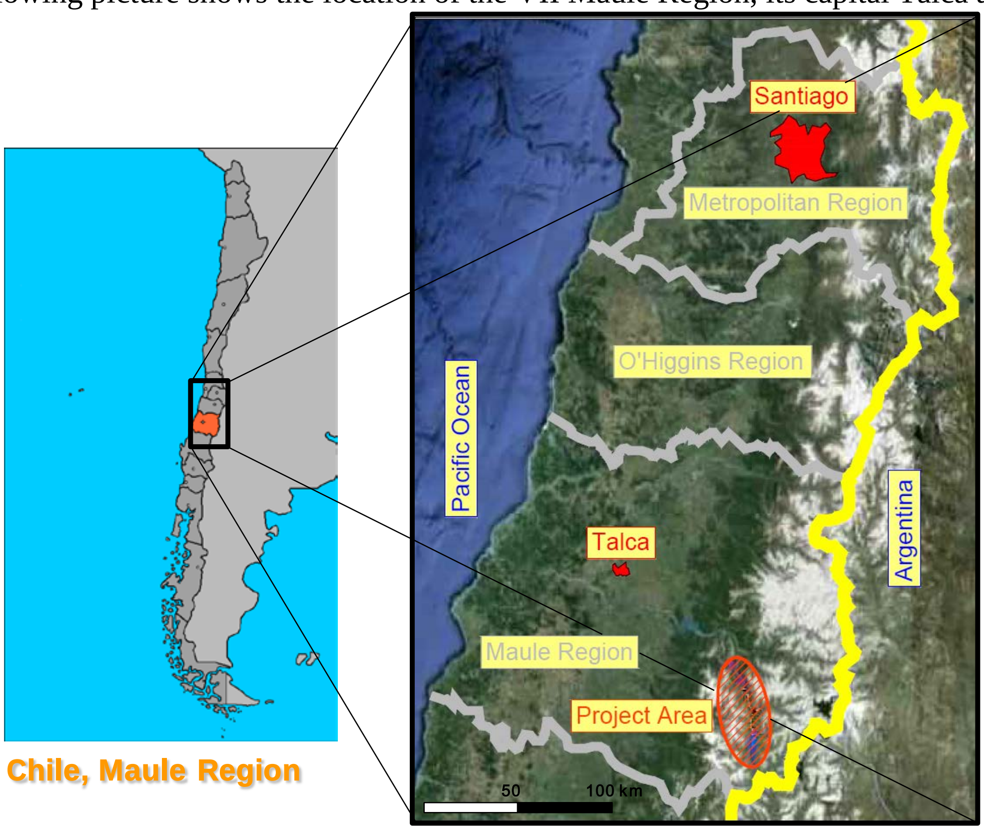


Figure 2-1: General location of project area

2.2 Access

The access to the project site from Talca is trough the international route CH-115 eastbound until its km 90 where the route L-399-K must be taken turning right (south). Following the latter route for 30 km the road reaches the *Melado* river through the hillsides of the Vallical brook. The following picture illustrates the above track description from Talca.

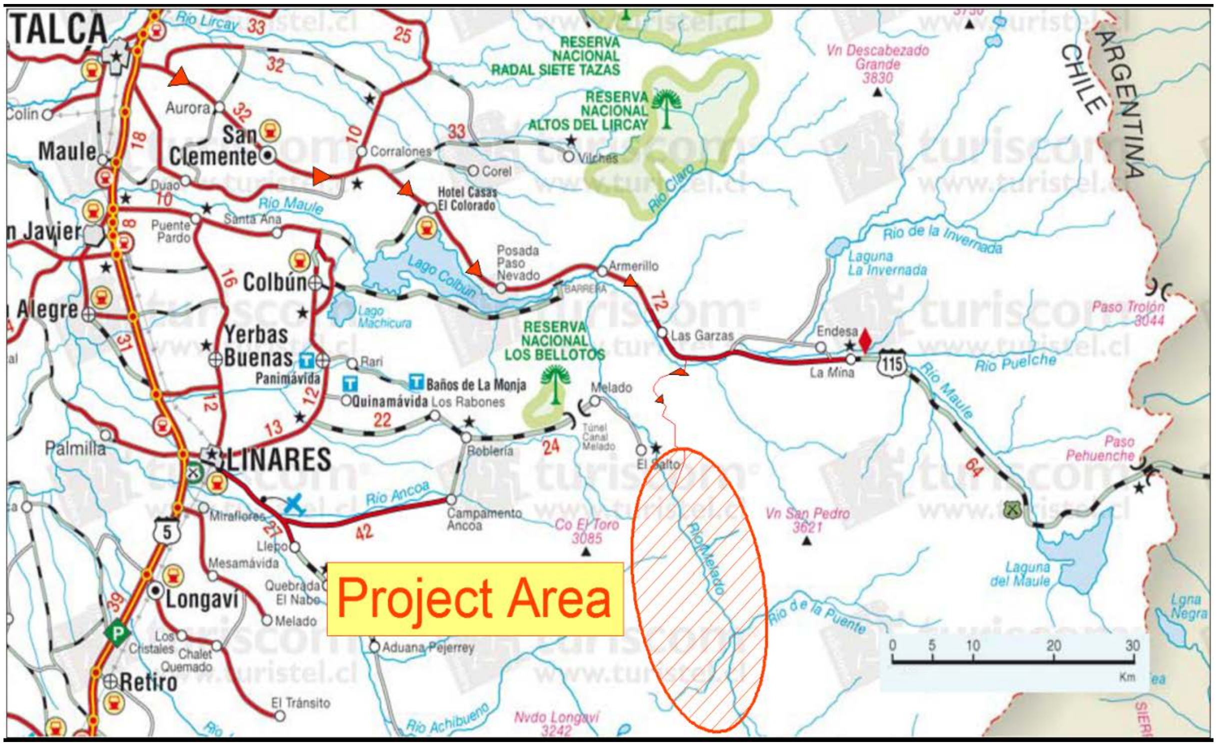


Figure 2-2: Access road to the project

2.3 Project Details

The "*Guaiquivilo-Melado*" project is a complex of six hydropower projects, of which four are regulated by a dam projected upstream of them. Those four projects run parallel to either the *Guaiquivilo* River (*Calabozos* HPP & *Lleuques* HPP) or the *Melado* River (*Catalinas* HPP & *Melado* HPP). The sequence given is to downstream.

Along with those four, two other run of river hydropower projects are included in the complex located in *Guaiquivilo*'s tributaries. These are *Cristales* HPP & *La Puente* HPP.

The following picture shows a top view of the *Guaiquivilo-Melado* project.

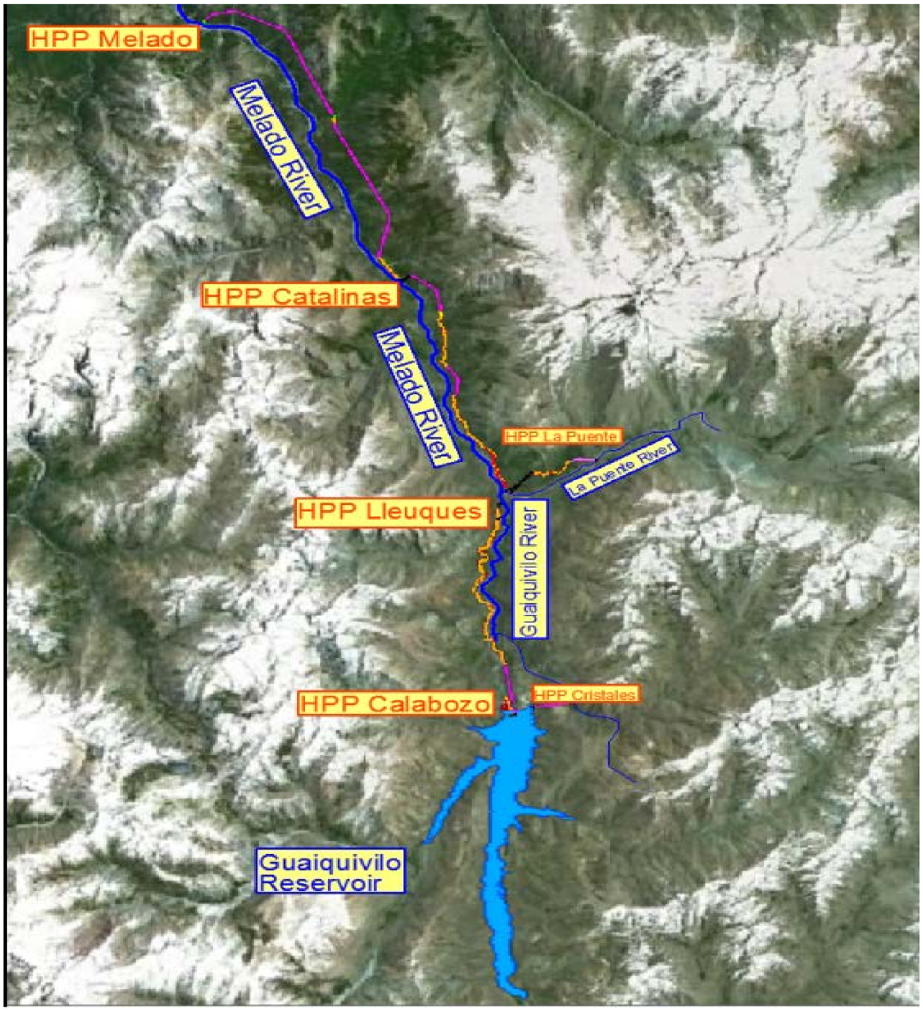


Figure 2-3: Top view of the *Guaiquivilo Melado* Project.

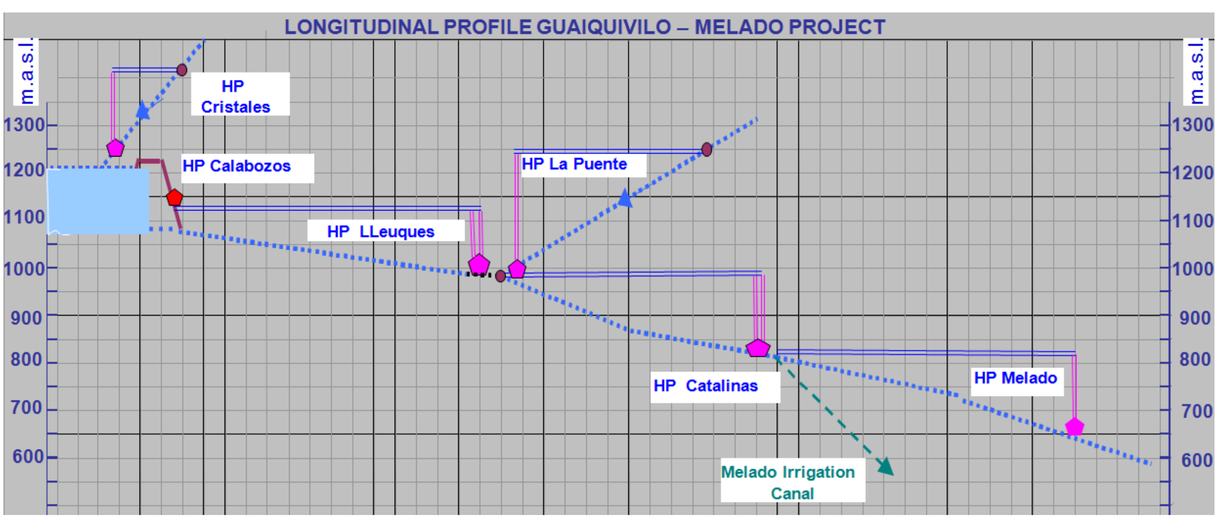


Figure 2-4: A longitudinal profile of the *Guaquivilo-Melado* provided by *Colbún S.A.*

Melado hydropower project is therefore at the downstream end of the complex regulated partly by the Guaiquivilo dam.

The following table shows the salient features of each project at the end of their feasibility reports:

		CALABOZO	LLEUQUES	LOS CRISTALES	LA PUENTE	CATALINAS	MELADO	Total
Design Discharge q_d	m^3/s	80	80	10	15	100	**100**	
Net Head Hn	m	99	108	41	234	145	**125**	
Installed Power P_{ins}	MW	71	78	4	31	131	**108**	421
Annual Average Energy Generation E	GWh	285	406	11	165	782	**625**	2 273

Table 2-1: Salient features *Guaiquivilo – Melado* complex

With regard to *Melado* HPP, it is important to point out that the design discharge is 100 m^3/s and therefore, its two head race tunnels on analysis must be able to convey up to that discharge.

The following image shows the two tunnels involved in the *Melado* project.

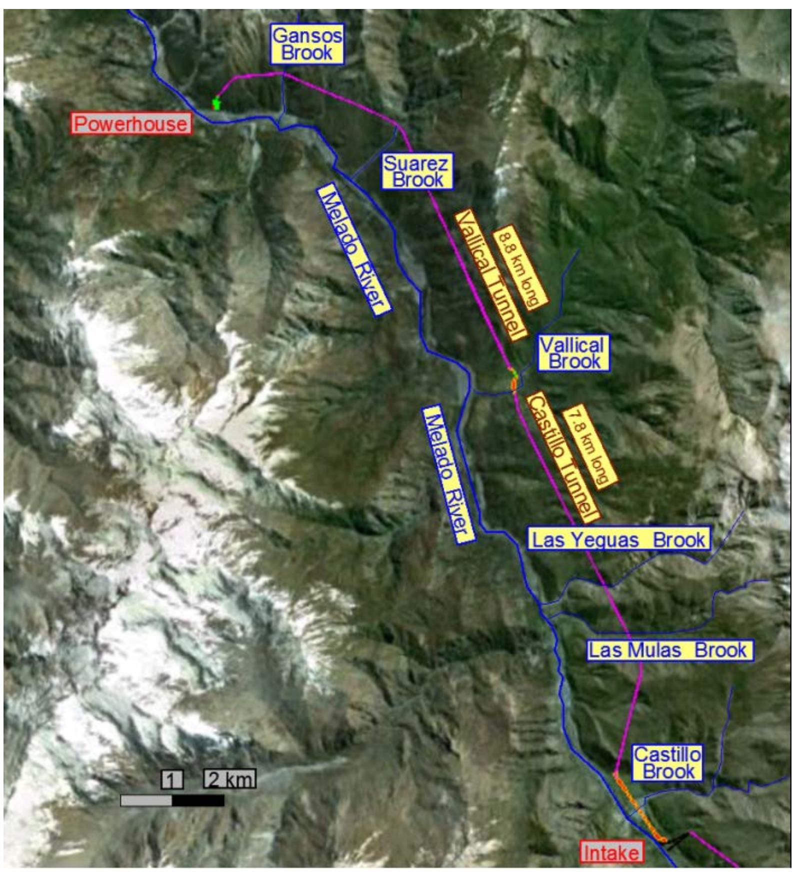


Figure 2-5: Layout of the *Melado* HPP and its two head race tunnels (Castillo & Vallical).

The existing layout of Melado HPP consists of:

- Off-stream intake (Taking water directly from upstream tail race canal Las Catalinas HPP)

- 1700 head race canal

- **7800 m long Head race *Castillo* Tunnel.**

- 560 m canal (Crossing the Vallical brook).

- **8800 m long Head race *Vallical* Tunnel.**

- Pressure shaft 120 m deep.

- Cavern (83.7 m long x 20.5 m wide x 33.27 m high)

- Tail race tunnel 125 m long.

Specifically about the two tunnels, both have a horseshoe shaped excavation and the cross section diameter is 6.9 m obtained from an economic analysis. The hydraulic conveyance has been determined as free surface, using at design discharge 73% of the total height.

2.4 About Basin & Water Resources

The *Melado* River is a tributary to the *Maule* river, coming from its southern bank. Melado Watershed at the joint with Maule River is around 2250 km^2. The river runs mainly northbound. From its start at *Del Dial* Lake until its joint with *La Puente* River, it is called *Guaiquivilo*. Downstream that joint, it is called Melado.

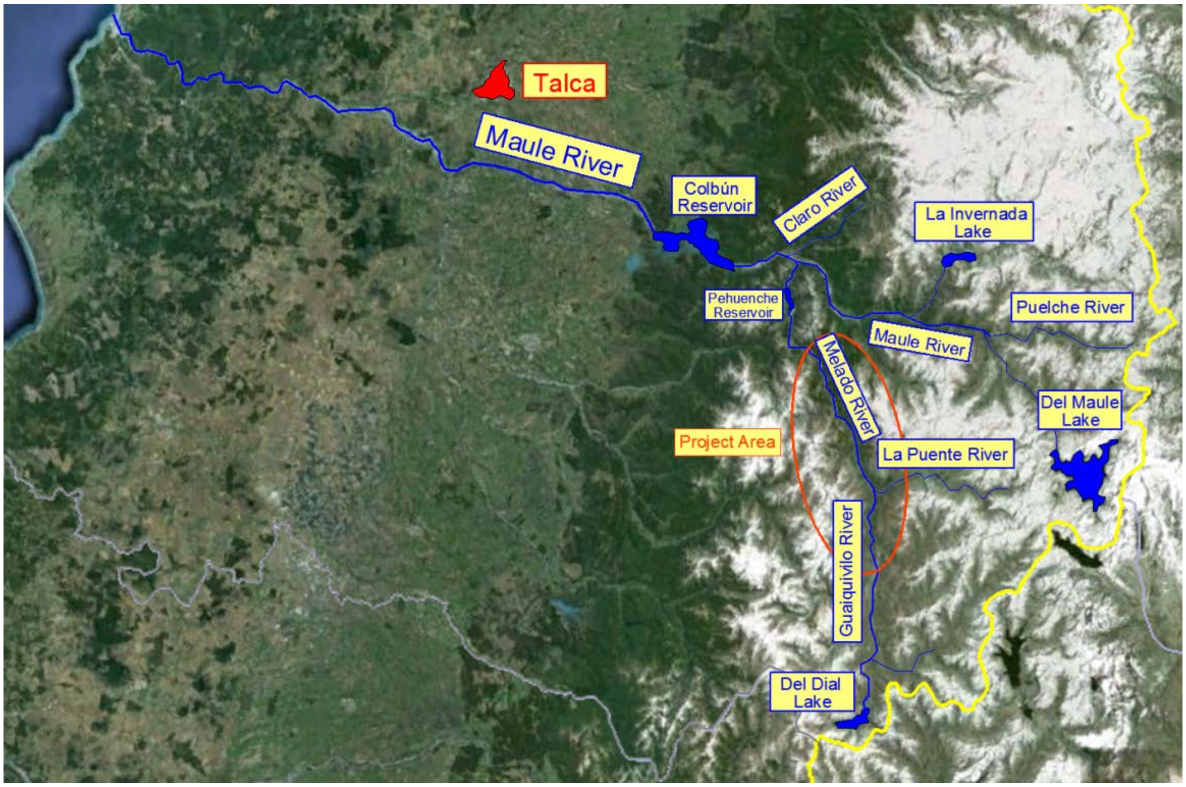


Figure 2-6: Melado basin

The *Melado* HPP takes the water from the tailrace tunnel of the upstream hydropower project *Catalinas*. The hydrology behavior of the basin, and particularly the potential generation from the *Melado* HPP was determined in previous studies [7], where the resulting input discharges for generation in Melado HPP are shown below.

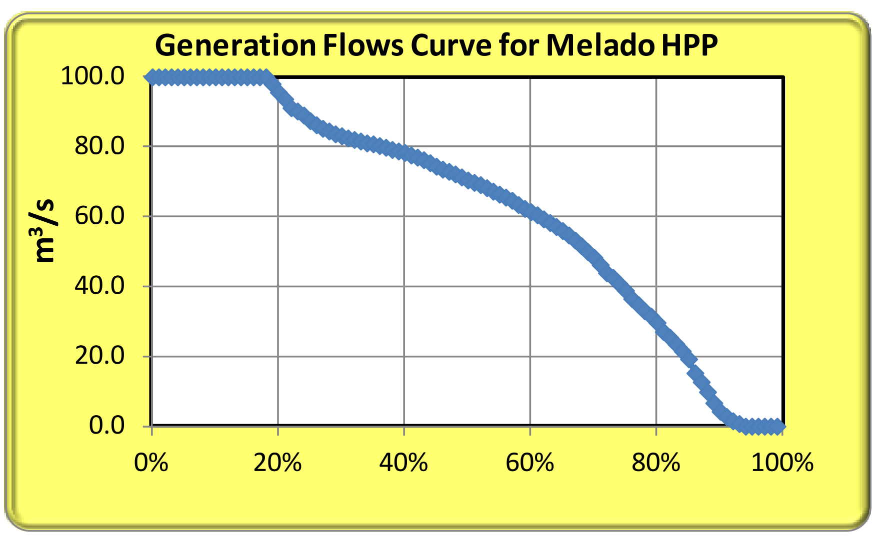


Figure 2-7: Usable water for turbine. Weekly data from 60/61 to 2005/06; Q_d = 100 m^3/s.

This information will be relevant for the energy losses calculation within the tunnel economic cross section.

3. GEOLOGY

3.1 General Geology & Topography

The general geology and topography for the Castillo tunnel is shown in the next image:

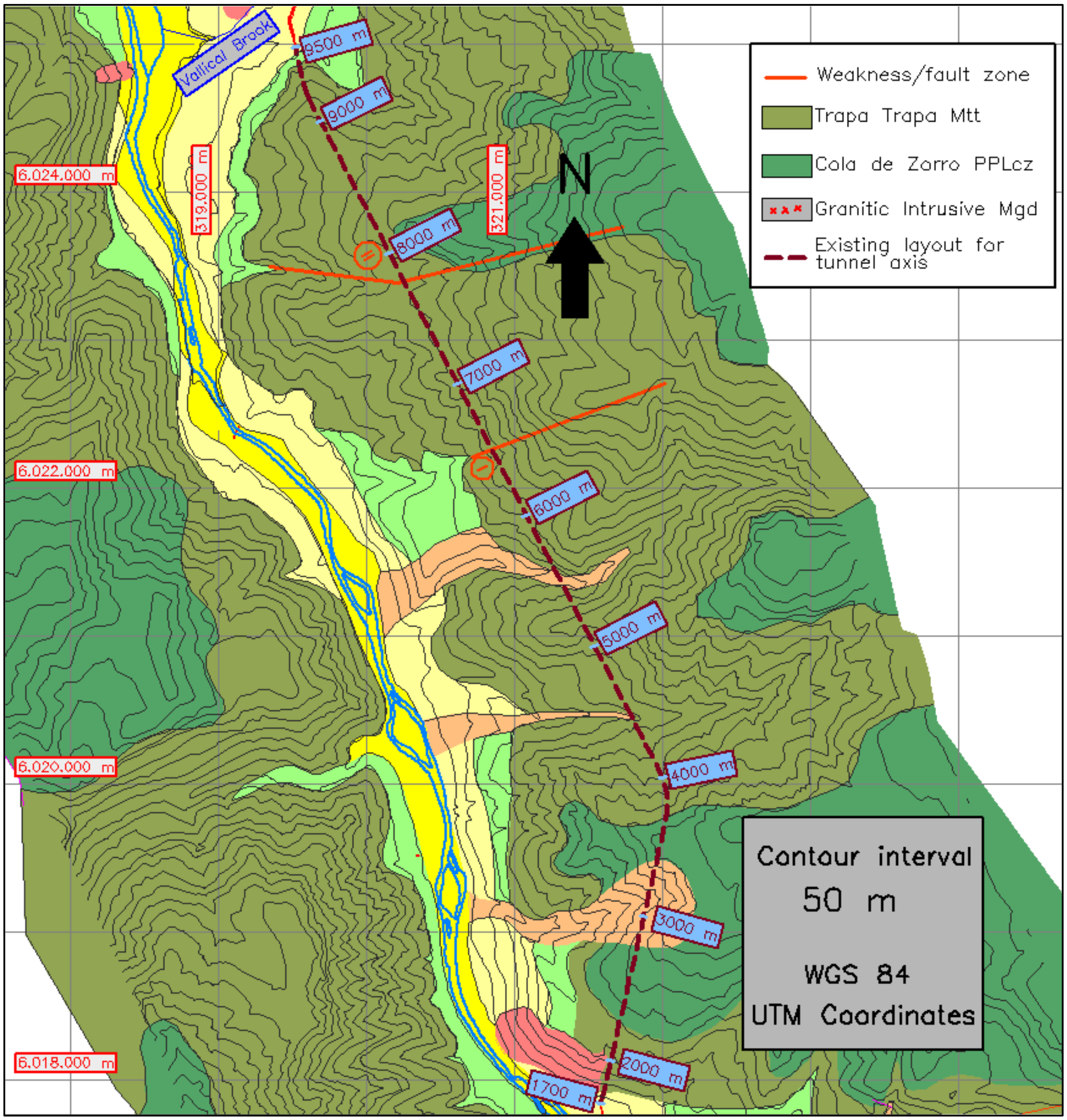


Figure 3-1: Top view of the general geology for Castillo tunnel.

Note that the *Cola de Zorro* rock formation does not cut the contours, suggesting that is almost horizontal.

The following figures show the longitudinal profile of Castillo tunnel:

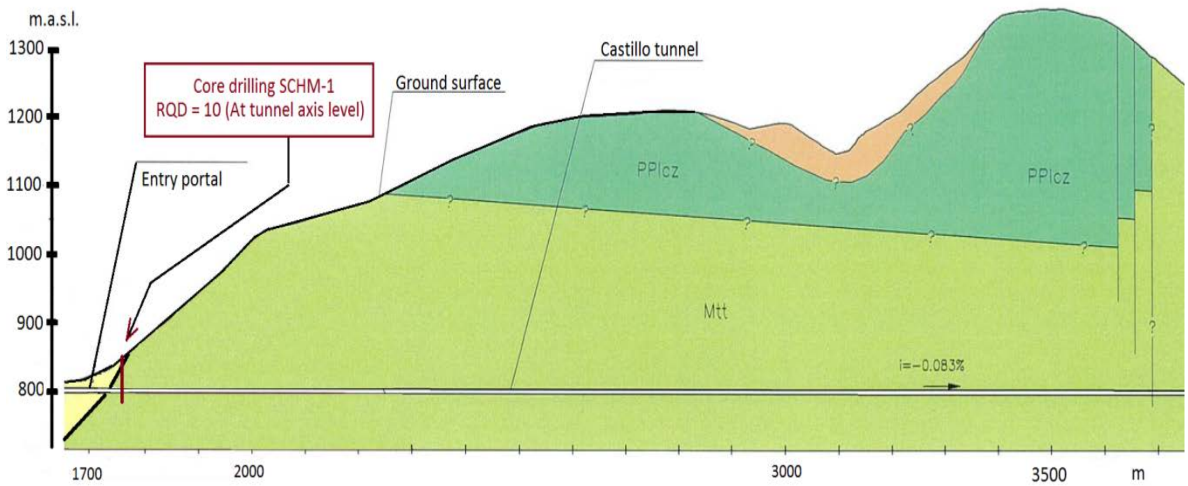


Figure 3-2: Castillo tunnel longitudinal profile. Chainage 1700 – 3700 m.

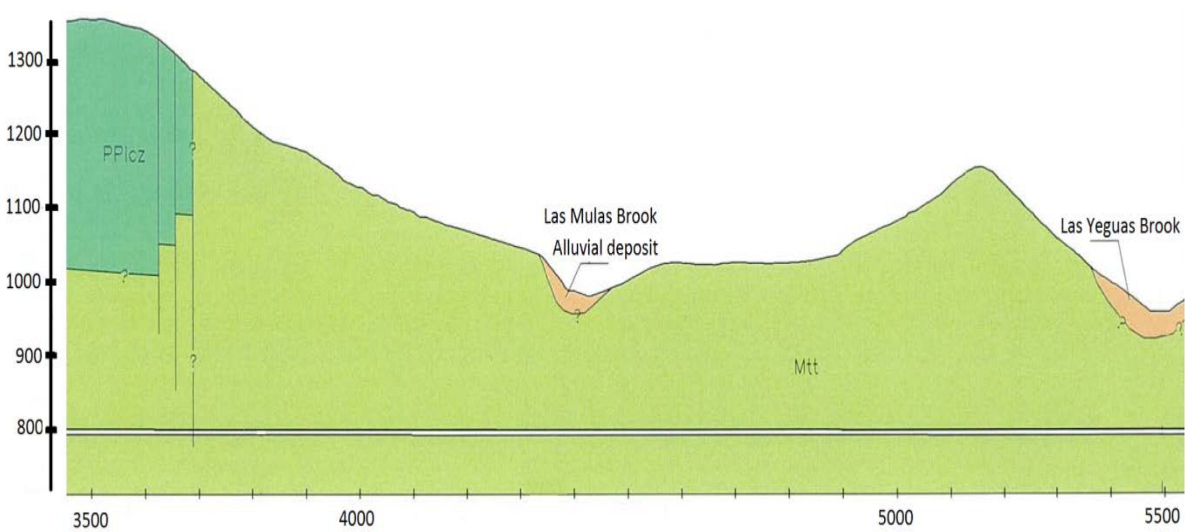


Figure 3-3: Castillo tunnel longitudinal profile. Chainage 3700 – 5500.

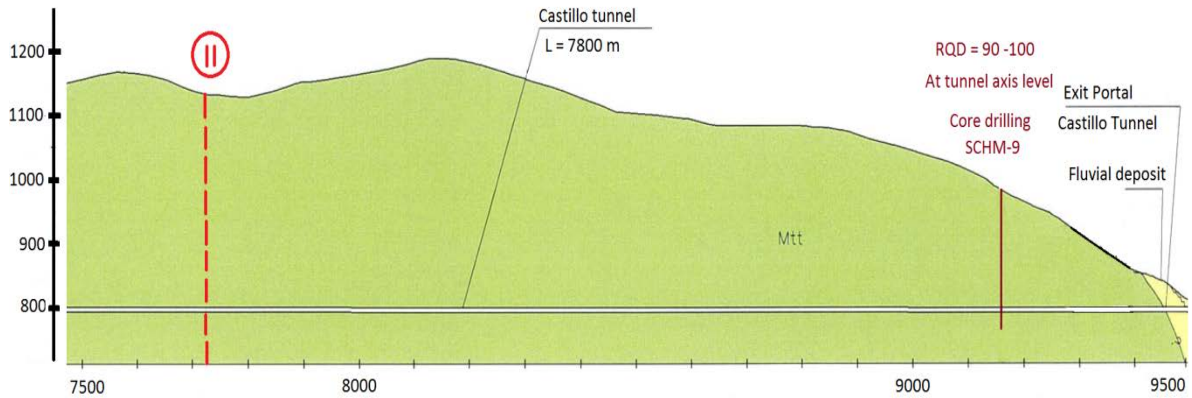


Figure 3-4: Castillo tunnel longitudinal profile

It follows from the longitudinal profiles that the maximum overburden for Castillo tunnel is around 560 m at chainage 3500 m. Also, one can visualize that there are not sudden changes in tunnel overburden. The two depressions in ground surface above tunnel axis are two brooks, but both are higher than 150 m overburden.

The general geology and topography for the Vallical tunnel is shown in the next image:

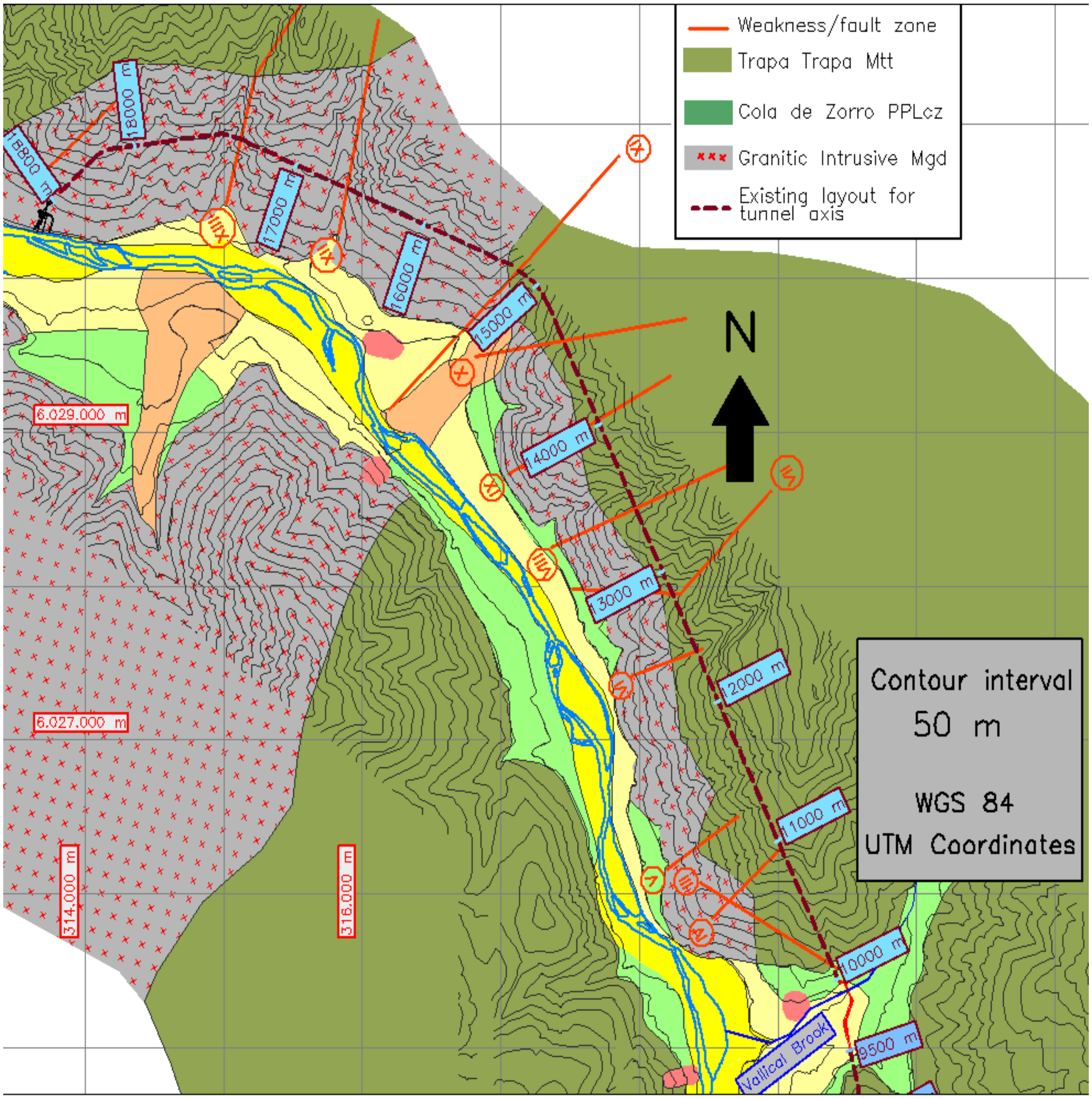


Figure 3-5: Top view of the general geology for Castillo tunnel.

The following for images show the type of rock involved for the Vallical tunnel.

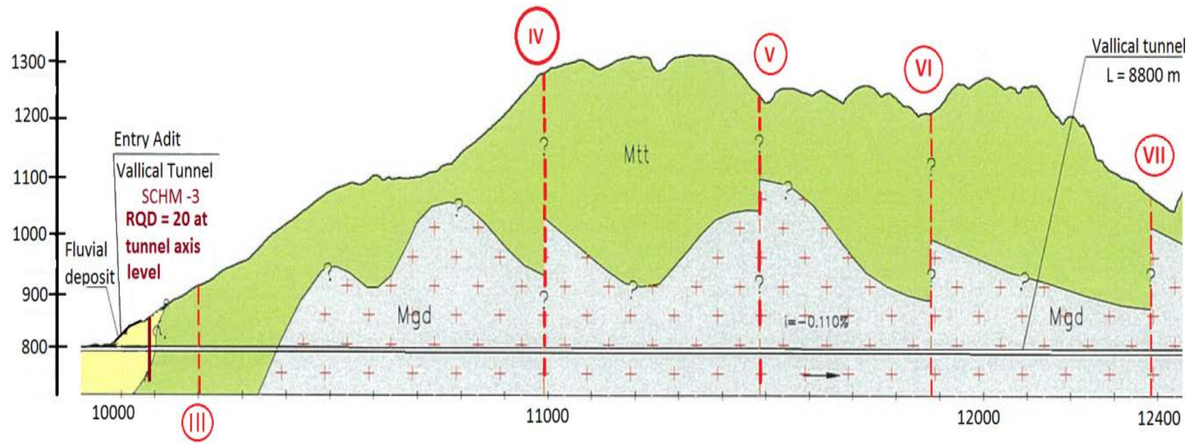


Figure 3-6: Vallical tunnel longitudinal profile Chainage 10,000 – 12,400 m.

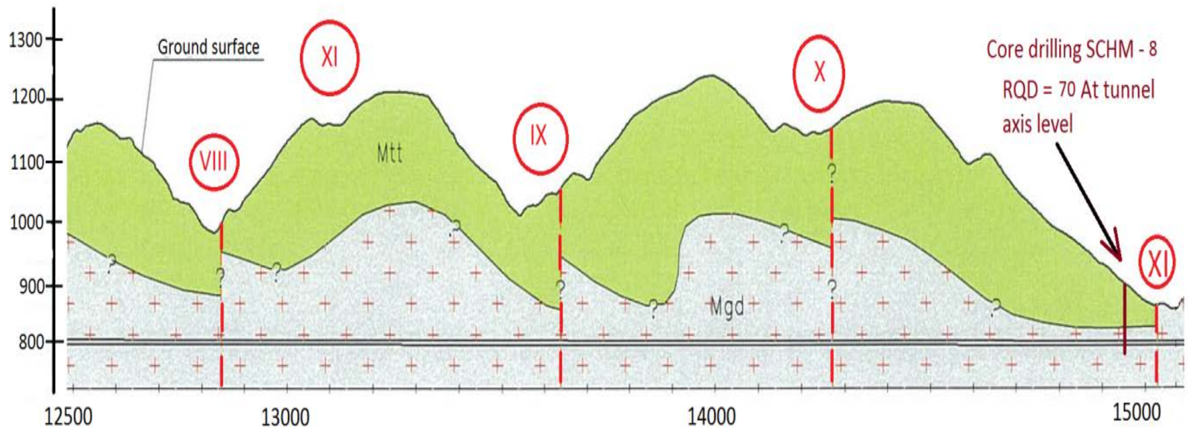


Figure 3-7: Vallical tunnel longitudinal profile Chainage 12,500 – 15,100 m.

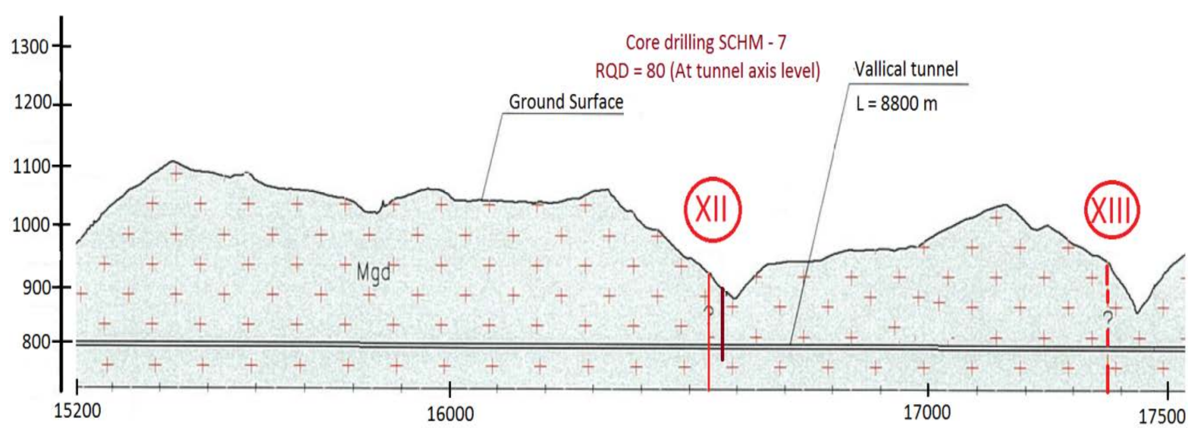


Figure 3-8: Vallical tunnel longitudinal profile Chainage 15,200 – 17,500 m.

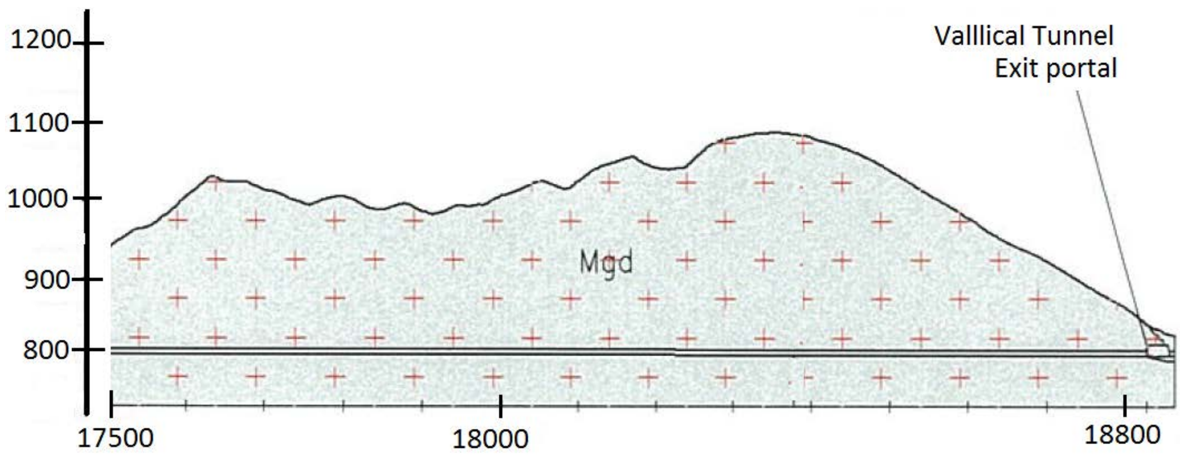


Figure 3-9: Vallical longitudinal profile: 17,500 – 18,800 m

3.2 Lithology

3.2.1 Trapa Trapa (Mtt) rock formation

As it is described in the geology report [4], this rock formation is a sequence of Andesitic lava, volcano-clastic rocks and continental sediments. Includes Andesite, Basalt Andesite, Dacite calc-alkaline. Additionally Volcanic Breccias and Tuffs can be found.

The rock formation would have formed during the boundary between Miocene – Pliocene around 16.5 and 5.3 million years ago.

3.2.2 Granitic Intrusive (Mgd)

This unit intrudes the area where emplaces the downstream end of Vallical Tunnel, with discrete outcrops and abundant colluvial, also rich in granitic blocks recognized throughout the right side of the "Melado" valley.

Batholiths Melado: On the project site, there are intrusive bodies composed from granodioritic to **tonalite** composition with acid or basic facies.

All the granitic material located on the project area intrudes volcanic and volcano-clastic material, which belongs to Trapa -Trapa rock formation.

3.2.3 Cola de Zorro (PPlcz) Rock Formation

It is defined by a volcanic sequence of Andesite and Basalts, widespread and subhorizontal tectonic.

The rock formation is constituted essentially by Andesitic-basaltic lava and pyroclastic material of similar composition, along with frequent bench of agglomerates and Breccias.

Glacial and fluvial erosions have created sudden cliffs. Also this unit is affected by tectonic blocks due to a normal fault.

Stratigraphic correlations and radiometric data say that this rock formation is from Pliocene Superior – Pleistocene between 3.5 and 11 million years ago.

3.3 Distribution

The existing tunnel layout crosses the following rock formations:

***Castillo* Tunnel: 7800 m**

1700-9500 (7800 m): *Trapa Trapa* Rock Formation (Bed rock from Andesitic-Basaltic lava).

***Vallical* Tunnel: 8800 m**

10050 - 10350 (300 m) : *Trapa-Trapa* rock formation

10350 - 18850 (8500 m) : Granitic Intrusive

3.4 Intact Rock Properties

For practical reasons, it is necessary to quantify some mechanical properties of the rock that can not be carried out by knowing only the type of rock formation, but rather by the rock type. Threfore it is necessary to select one rock type that best represent each rock formation.Andesite is an igneous volcanic rock, of intermediate composition that best represents the Trapa-Trapa rock formation. Granite is an intrusive acid rock that is quite often found in Batholith Melado.

3.4.1 Uniaxial Compressive Strength

Is the most commonly used method for testing the mechanical character of the rocks**.** A piece of core is loaded axially from both ends until failure occurs without confinement.

Granite is a well known rock of being strong. The values provided in reference [8] with an average value of Uniaxial Compresive strength equal to 154 MPA (71 test of rock worlwide) is quite conservative compared to the literature in general like rock lab (Statistical commercial software) that suggests values higher than 250 MPA. But the first value is consistent with the geology report carried out during feasibility studies [4] where a value larger than 150 MPA was suggested. Therefore an UCS conservative value of 150 MPA will be considered.

With regard to Andesite, reference [5] includes one failure example (Slabbing) in underground opening excavated in Andesite located in El Teniente Mine, Chile. This mine lies about 200 km northern of *Melado* project area. A good predicted failure compared to field measurements was achieved using an UCS of 150 MPA as an input parameter. This value is quite consistent with UCS value given by reference [8].

Reference [6] summarizes case histories of failure in tunnels, including the main parameters involved (Type of rock, UCS, stress situation, etc). Among those cases, there is one excavated in a “blocky Andesite” also from El Teniente Mine, where an UCS of 100 MPA was considered for failure analysis.

For similar volcanic rocks like basalt, rock lab software assigns a UCS value between 100 and 250 MPA.

For Andesite, a conservative value of 100 MPA will be adopted.

With regard to weakness zone, reference [8] provides some values depending on the degree of weathering. A UCS value of 2.5 MPA will be considered for all the weakness zones.

3.4.2 Elasticity Modulus

It is the gradient of the stress-strain curve during uniaxial testing.

Reference [8] collects some test results for some typical rock types from around the world. Specifically for the relevant rock types involved in the project, the results are given below:

Andesite: Intact rock sample		
E_{ci}	GPA	31
Number of tests: 6		

Granite: Intact rock sample		
E_{ci}	GPA	48
Number of tests: 71		

Those values will be adopted for intact rock E-modulus.

3.5 Joint Characteristics

This chapter describes the system of mechanical discontinuities in the rock mass.

The most relevant joint characteristics are available in the geology report [4] along with core drilling loggings which are summarized in the following description:

3.5.1 Roughness

The roughness of joint walls is characterized by a large scale waviness (Planar / Undulating, stepped) and a small scale smoothness (Rough / smooth / slickensided).

The following description was obtained from field mapping [4], for the two main rock formation involved in the project:

Trapa Trapa: Joint walls are mainly with no waviness (planar) and a smooth surface.

Granitic Intrusive: Joint walls are planar with rough surface.

Even though in core drilling SCHM-9 done in the Trapa-Trapa rock formation is visualized a rough wall surface in most of the cracks, the feasibility report [4] described *Trapa Trapa* rock formation with smooth surface. The last opinion was kept here since it is possible that core drilling SCHM-9 does not represent fairly this rock formation.

3.5.2 Alteration and filling

From field mapping documented in the geology report [4] plus core drillings undertaken in the project area, it was possible to obtain relevant data about the joint walls and how they are filled.

Trapa – Trapa: Slightly weathered joint walls filled with clay and chlorite coating.

Granitic Intrusive: From fresh to slightly weathered joint walls. There is hard and fine coating materials in close walls.

3.5.3 Joint spacing & Openness

From field mapping carried out during the feasibility report a range of joint spacing values are given for each rock type.

Trapa – Trapa: 100 - 300 mm (Equally spaced for the three joint sets).

Granitic intrusive: 60 - 600 mm (Equally spaced for the three joint sets).

With regards to openness, close joints are described for both Granitic intrusive and Trapa Trapa rock formation based on field mapping during the feasibility studies. This qualitative description could be interpreted as 0.1 – 0.5 mm from Bieniawski's classification.

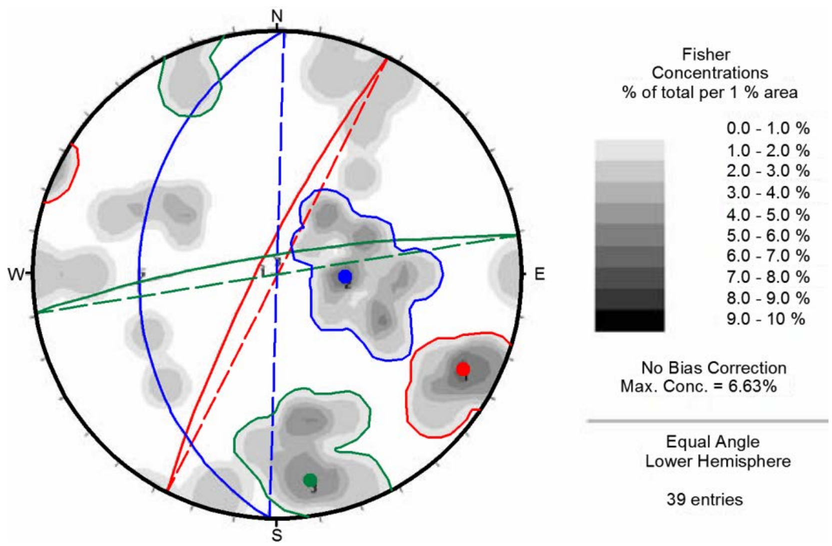
3.6 Jointing System & Orientation

As reference [8] defines is the occurrence of joint sets forming the system or pattern of joints as well as amount or intensity of joints.

The main characteristic of the joints are described for each rock formation directly involved in the project area.

The geology report [4] provides information about the orientation of the main joint sets by means of two stereonets (spherical projection), one for each tunnel. From this information, it was highlighted for each joint set the corresponding Fisher concentration zone, the Strike (Dashed line for each joint set), the Great Circle (continuous line) and the representative Pole.

Castillo stereonet



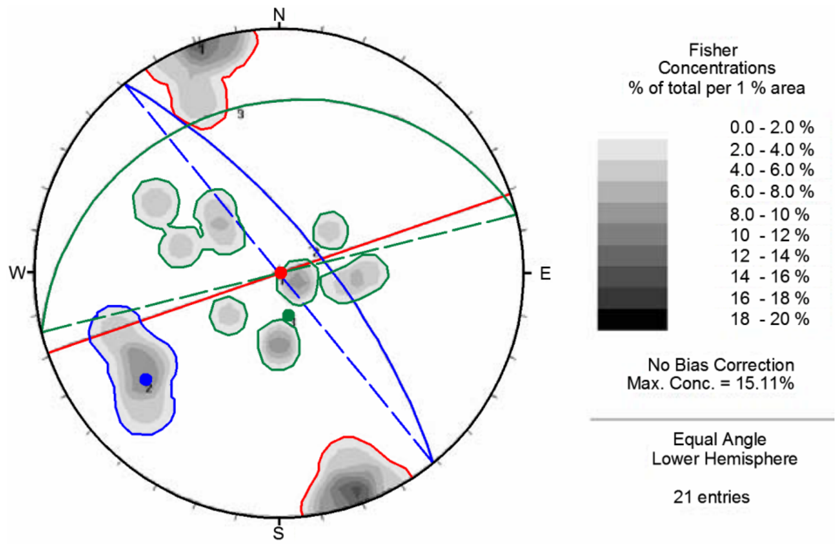
The average values for each joint set are given in the next table:

Main Joint sets Castillo Tunnel								
Joint Set	Strike			Dip Angle	Dip Direction			
1	N	27	E	81	N	297	E
2	N	2	E	31	N	272	E
3	N	81	E	81	N	351	E

The report also complement this information with the following statement: “The Jointing system for the Castillo Tunnel is defined by three main joint sets perpendicular to each other”. But from the image it can be identified along with the three joint sets some random joints. For practical purposes, the rock involved in the Castillo tunnel (Trapa Trapa rock formation) will be considered on this report as three joint sets with random joints.

Vallical Stereonet (Only Granitic intrusive)

For the second tunnel (Mostly granitic intrusive rock), there is also a stereonet showing the orientation of joint sets for the batholith *Melado*:



Main Joint sets Vallical Tunnel			
Joint Set	Strike	Dip Angle	Dip Direction
1	N 71 E	90	N 161 E
2	N 39 W	70	N 51 E
3	N 76 E	20	N 346 E

In the feasibility report [4], as well as it states in the Castillo tunnel, it is said that there are three joint sets perpendicular to each other visualized from field mapping. However, in this case the third joint set could be interpreted as random joints that only have in common a gentle dip angle. But all these random joints were considered as one joint to be consistent with the description. According to the rock mass classification Q system, there is not much difference between “two joint sets plus random joints” and “three joint sets” as it will be shown later in chapter 4.

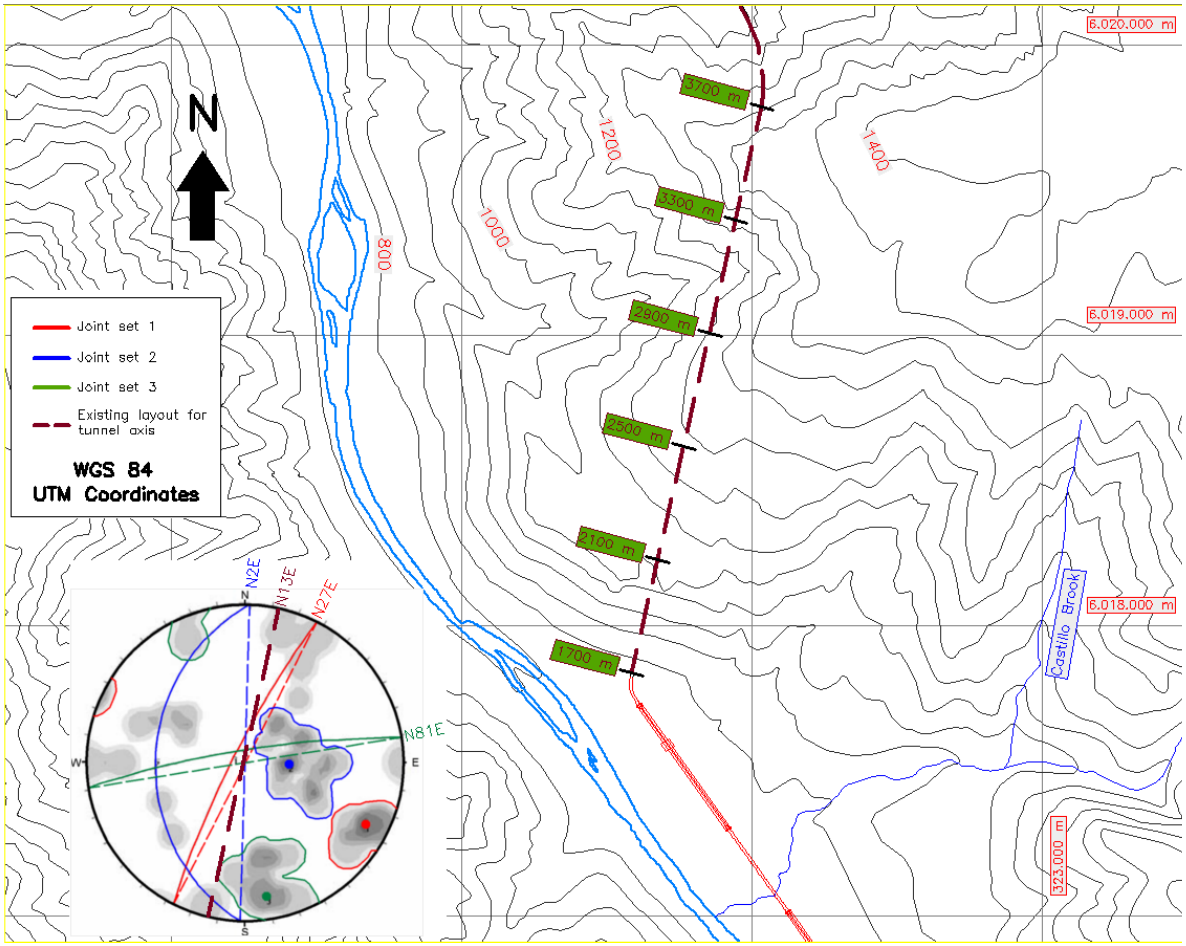


Figure 3-10: Tunnel axis along with stereonet: Chainage 1700 – 3700 m.

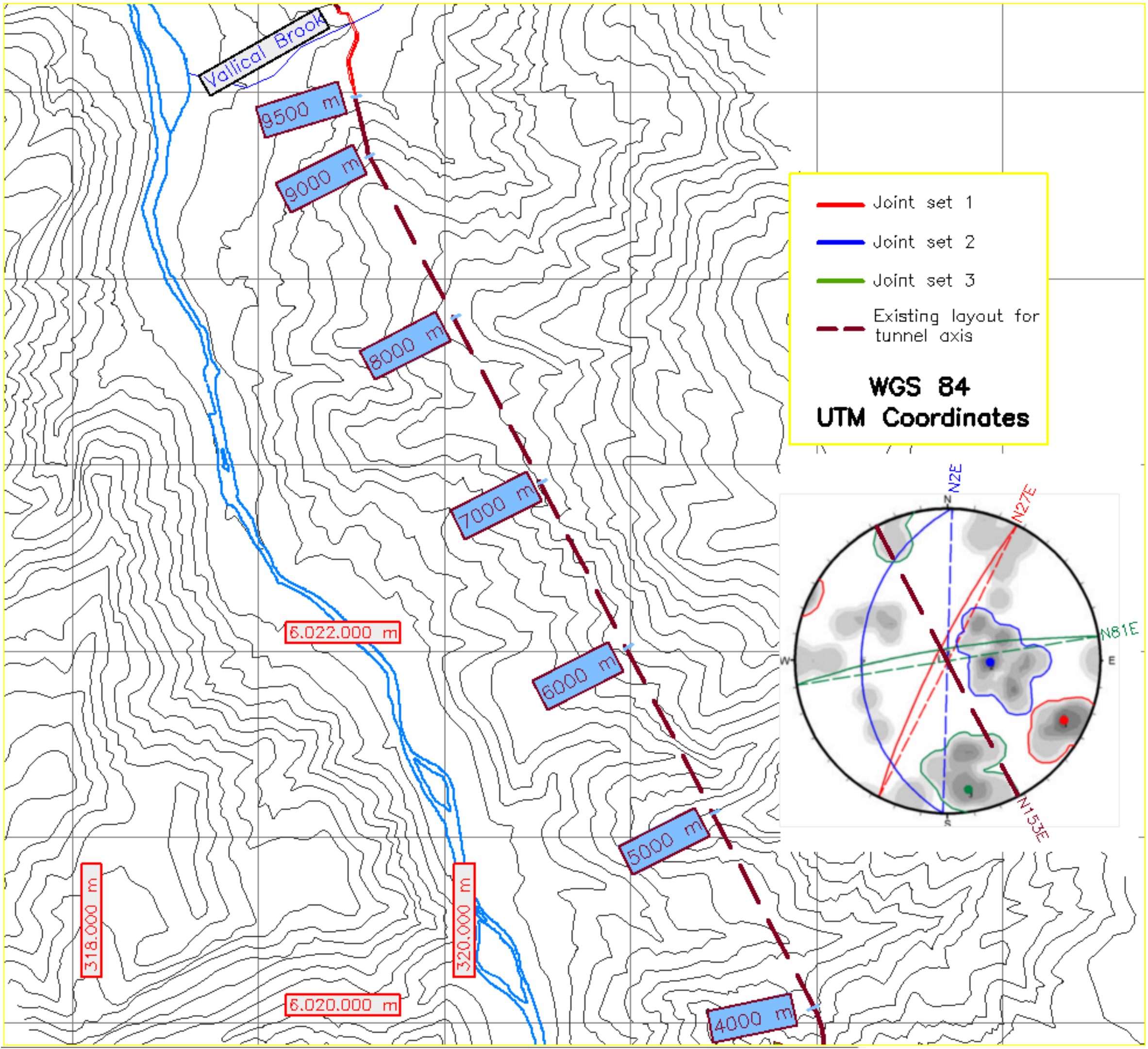


Figure 3-11: Tunnel axis along with stereonet: Chainage 4000 – 9500 m.

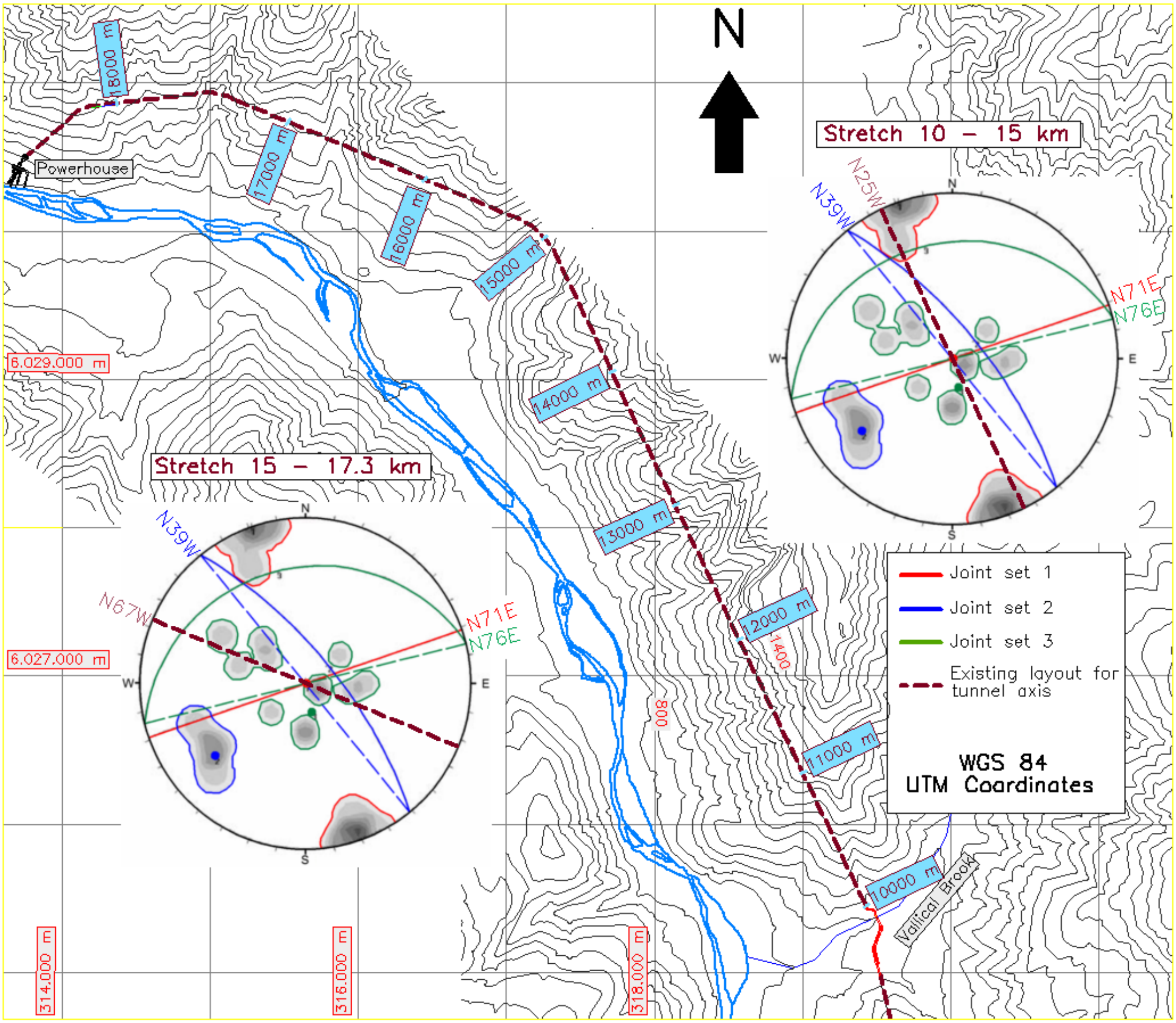


Figure 3-12: Tunnel axis along with stereonet: Chainage 10000 – 17400 m

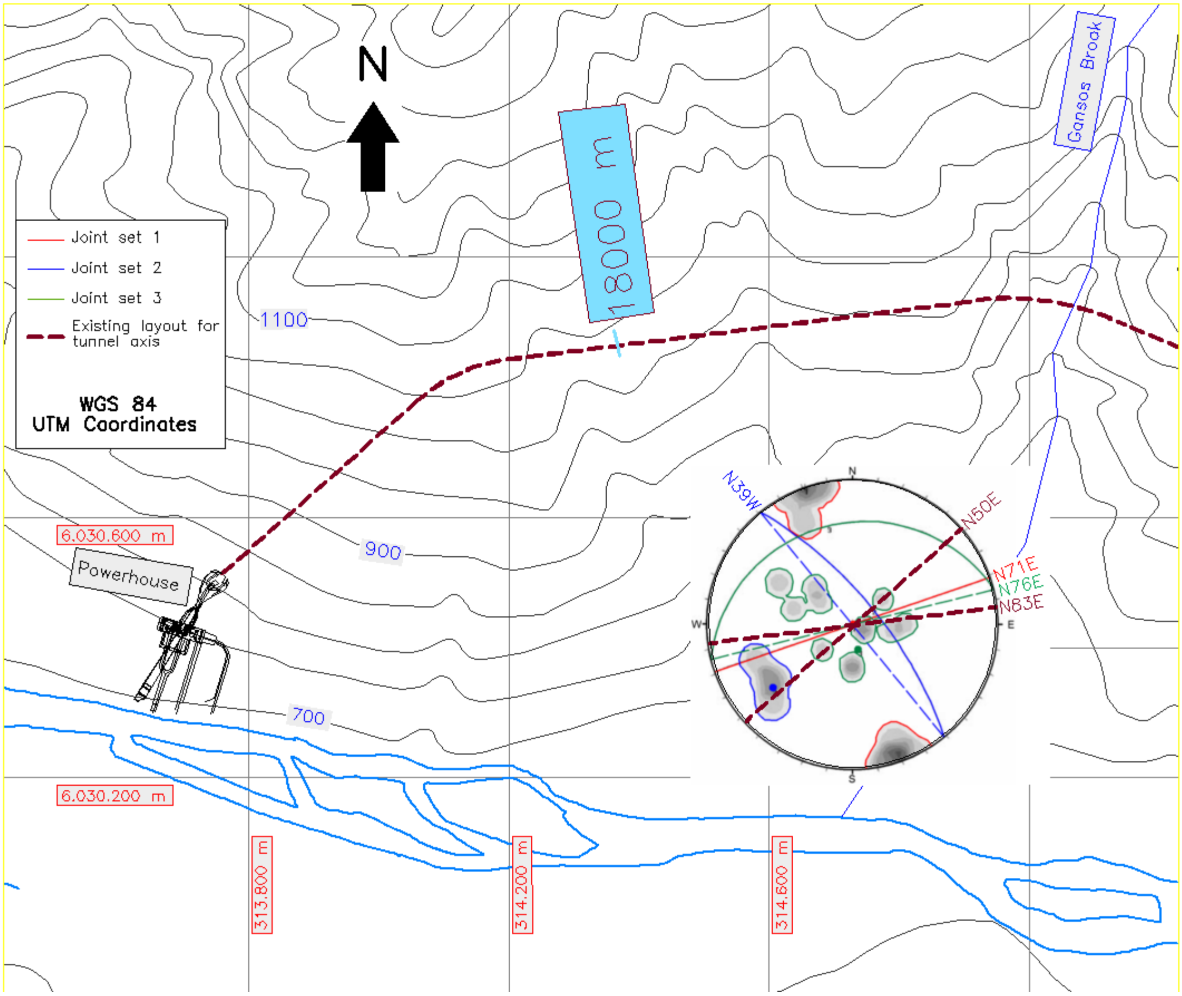


Figure 3-13: Tunnel axis along with stereonet: Chainage 17400 – 18800 m

3.7 Degree of jointing

Several methods can be used to measure the density of joints in the rock mass. Since there is log data for 8 core drillings carried out for this project, RQD will be the main parameter used to determine the degree of jointing in the rock masses involved in the project.

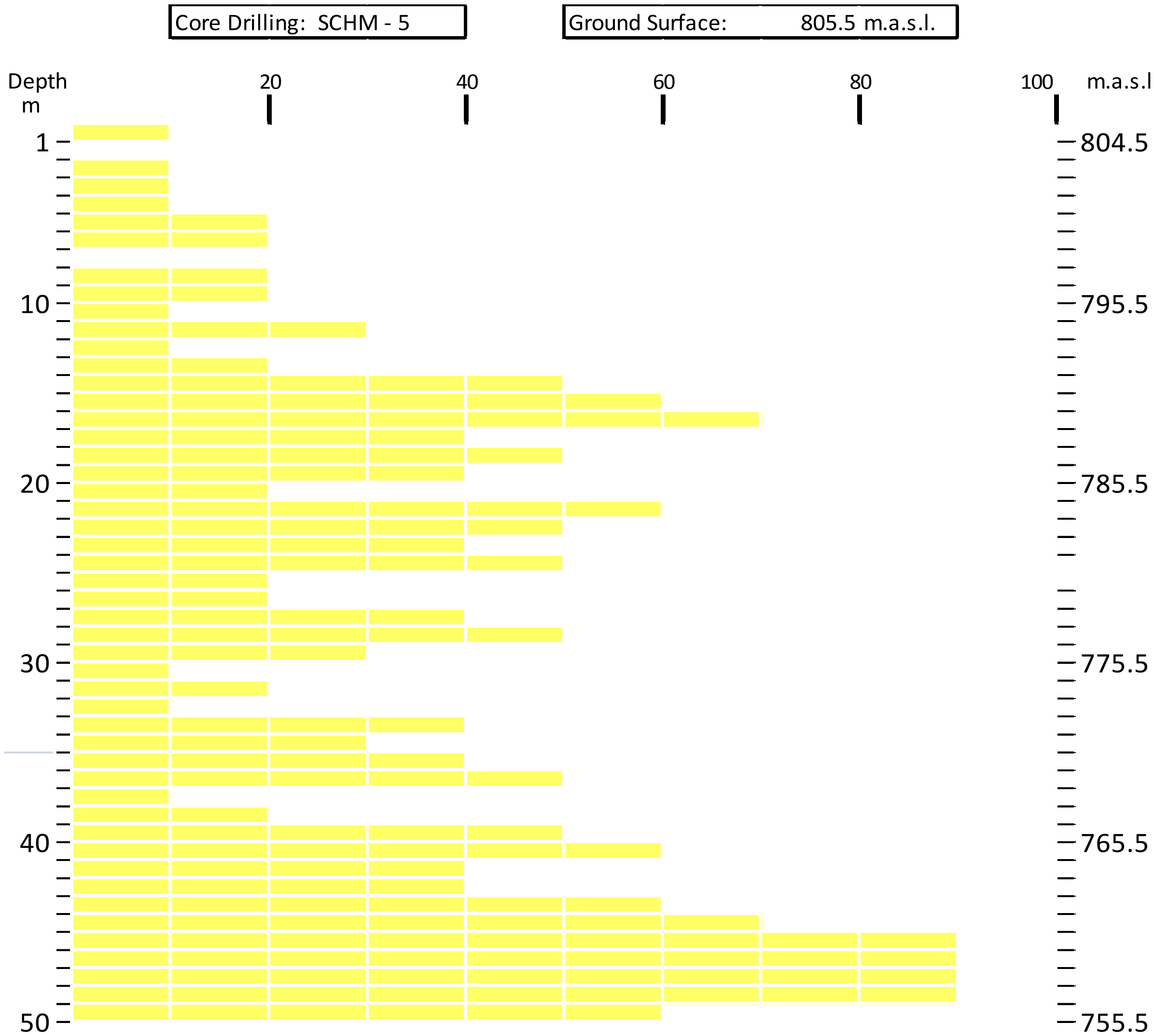
RQD is one dimensional classification system for characterizing the degree of jointing in a core drilling. It is defined as core bits larger than 10 cm divided by the measured length.

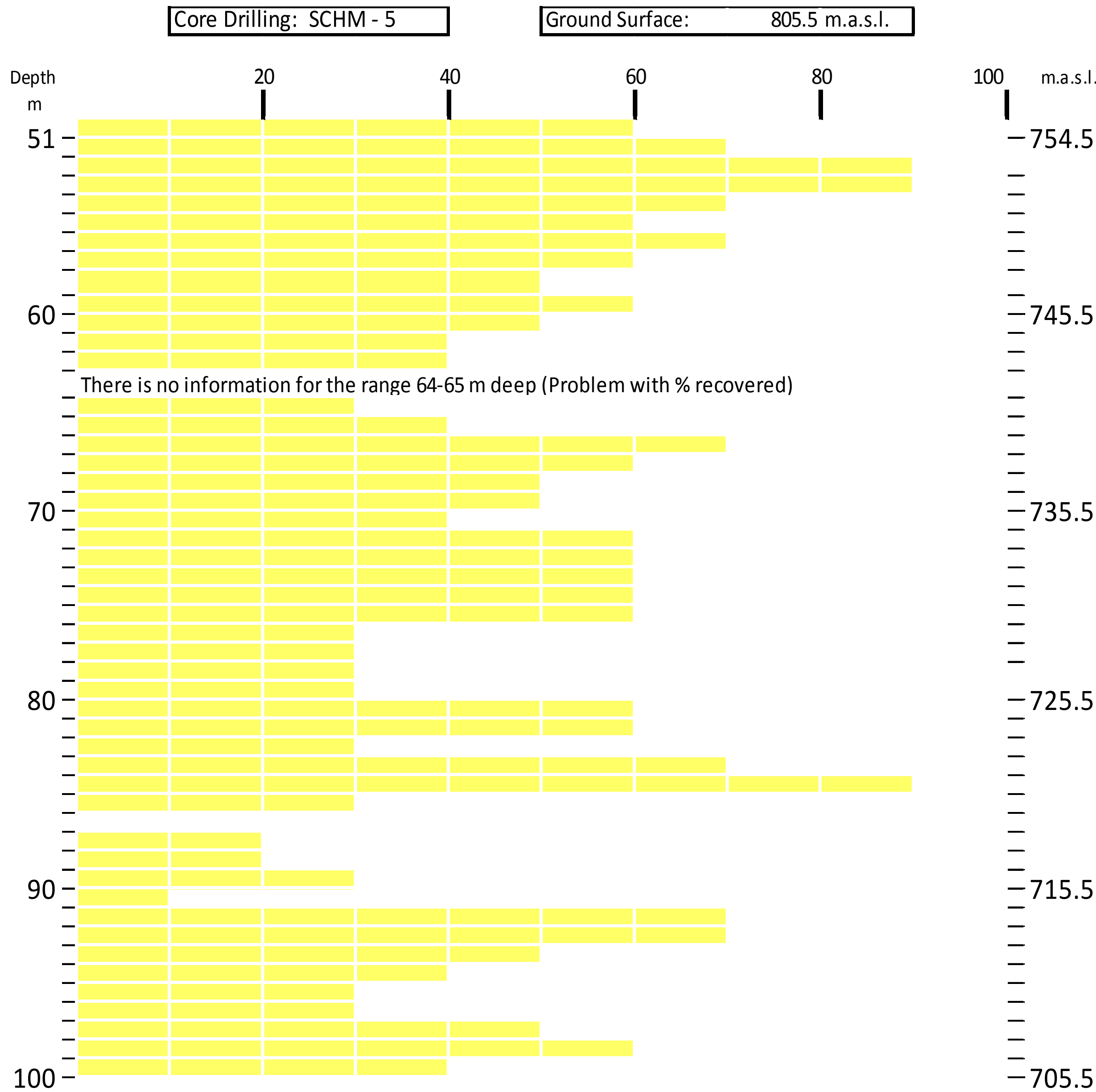
3.7.1.1.1 Core Drillings

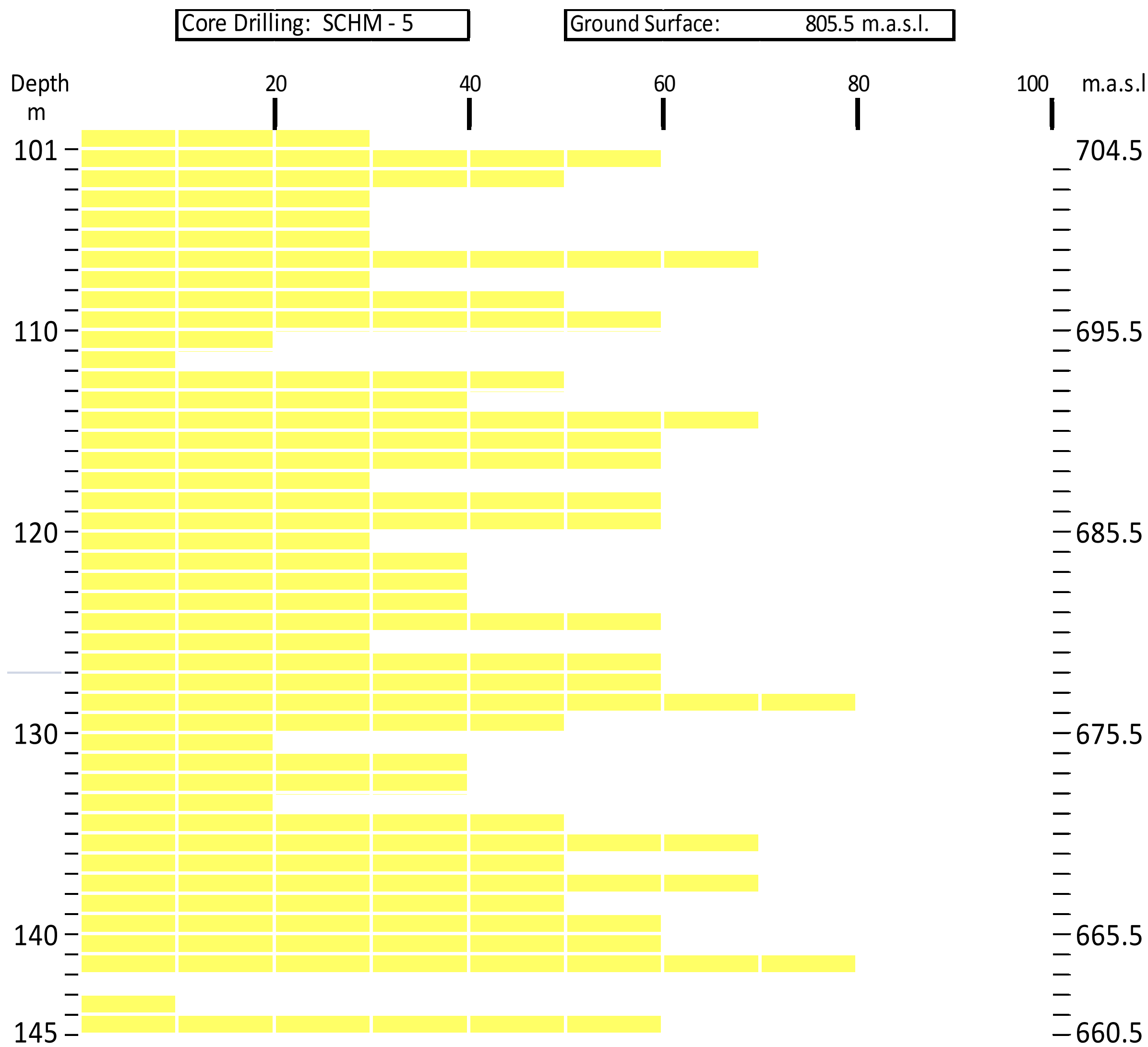
Seven core drillings are available in the project area. Three are related to Castillo Tunnel and four to Vallical tunnel. It is important to stand out that only four were carried out before the feasibility report. These are core drillings: SCHM 1, SCHM 2, SCHM 3 & SCHM 9.

Figure 3-14: Overview of Core Drilling Locations

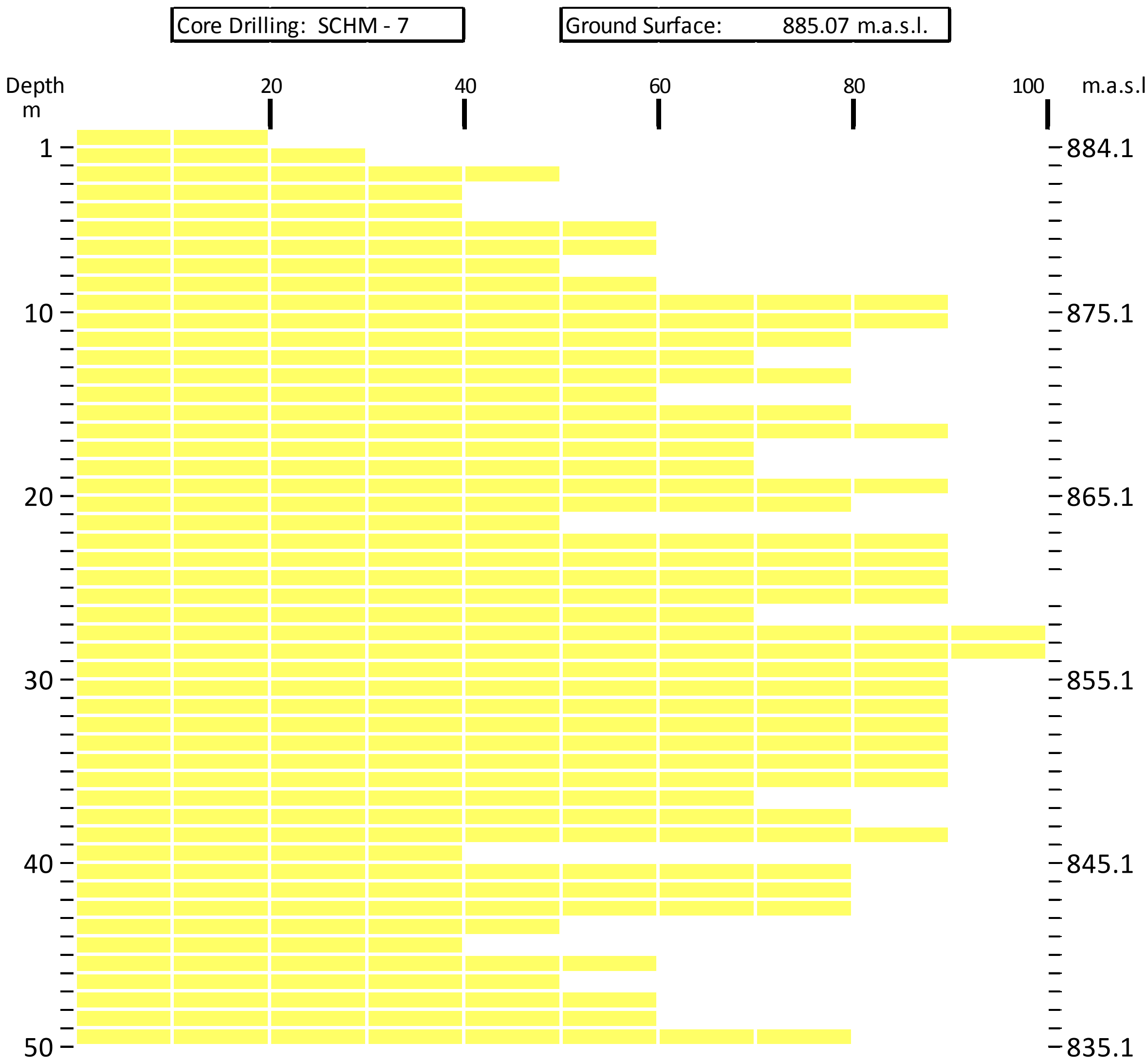
Core drilling 5 (145.3 m deep): Medium grain Granite with chlorite, calcite and hematite veins. Low mafic inclusion. The deeper, the higher content of pirite.

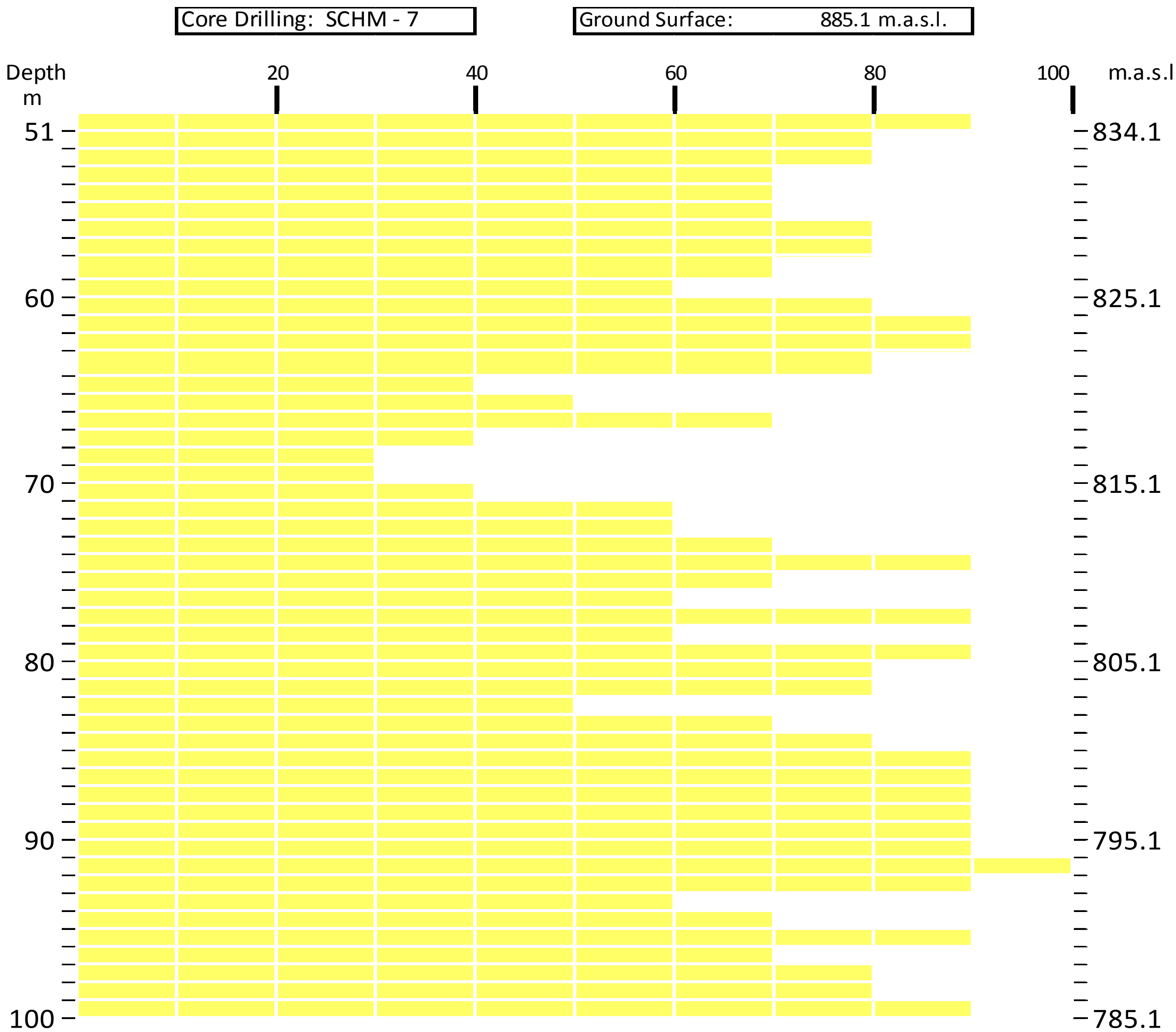



Core Drilling: SCHM - 5
Ground Surface: 805.5 m.a.s.l.
Depth
m
20
40
60
80
100
m.a.s.l.
51
60
70
80
90
100
754.5
745.5
735.5
725.5
715.5
705.5
There is no information for the range 64-65 m deep (Problem with % recovered)


Core Drilling: SCHM - 5
Ground Surface: 805.5 m.a.s.l.
Depth
m
20
40
60
80
100
m.a.s.l.
101
110
120
130
140
145
704.5
695.5
685.5
675.5
665.5
660.5

Core drilling 7 (105 m deep): Medium grain Granite with calcite veins.




Core Drilling: SCHM - 7
Ground Surface: 885.1 m.a.s.l.
Depth m
20
40
60
80
100
m.a.s.l.
51
60
70
80
90
100
834.1
825.1
815.1
805.1
795.1
785.1

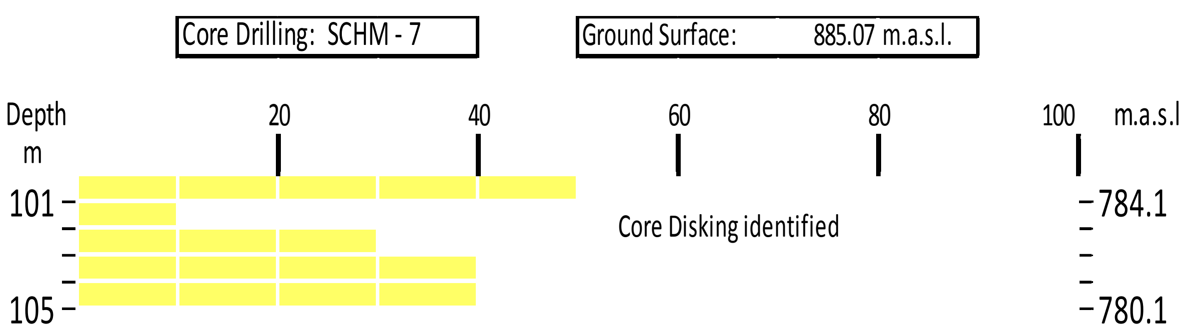
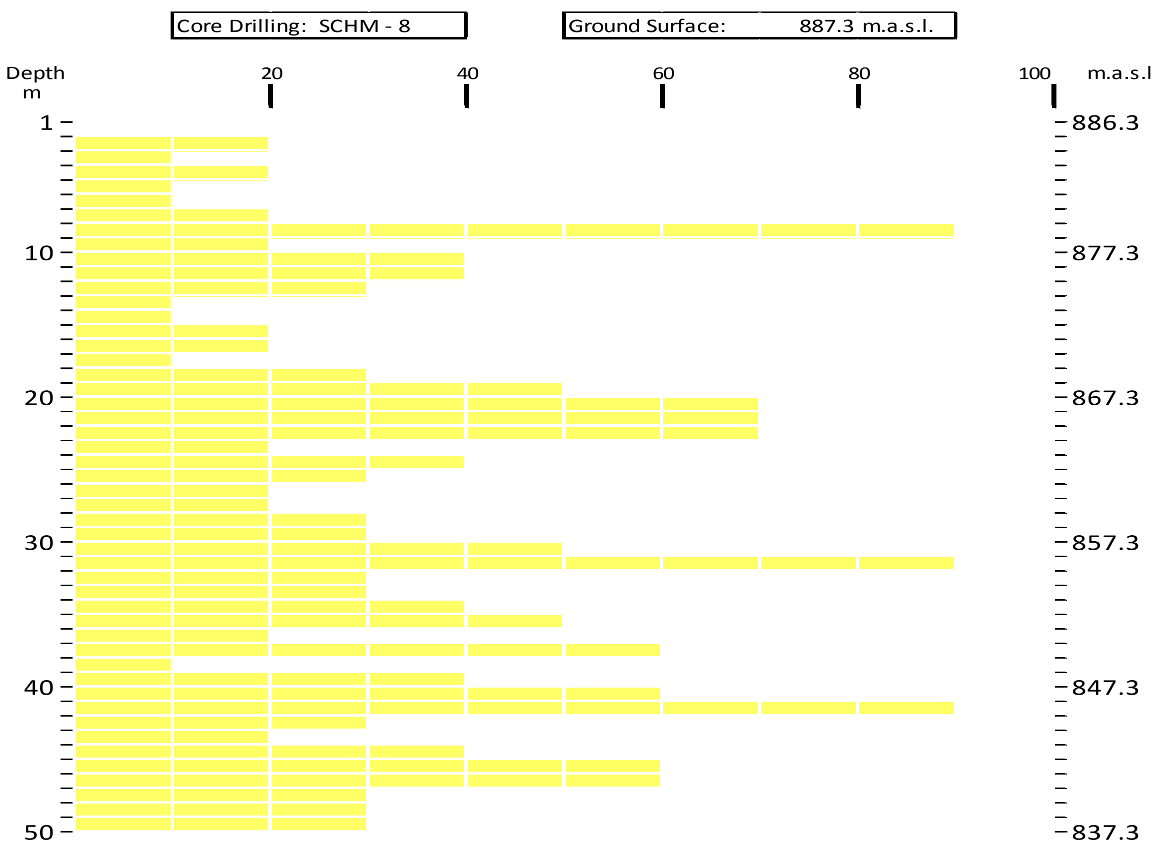
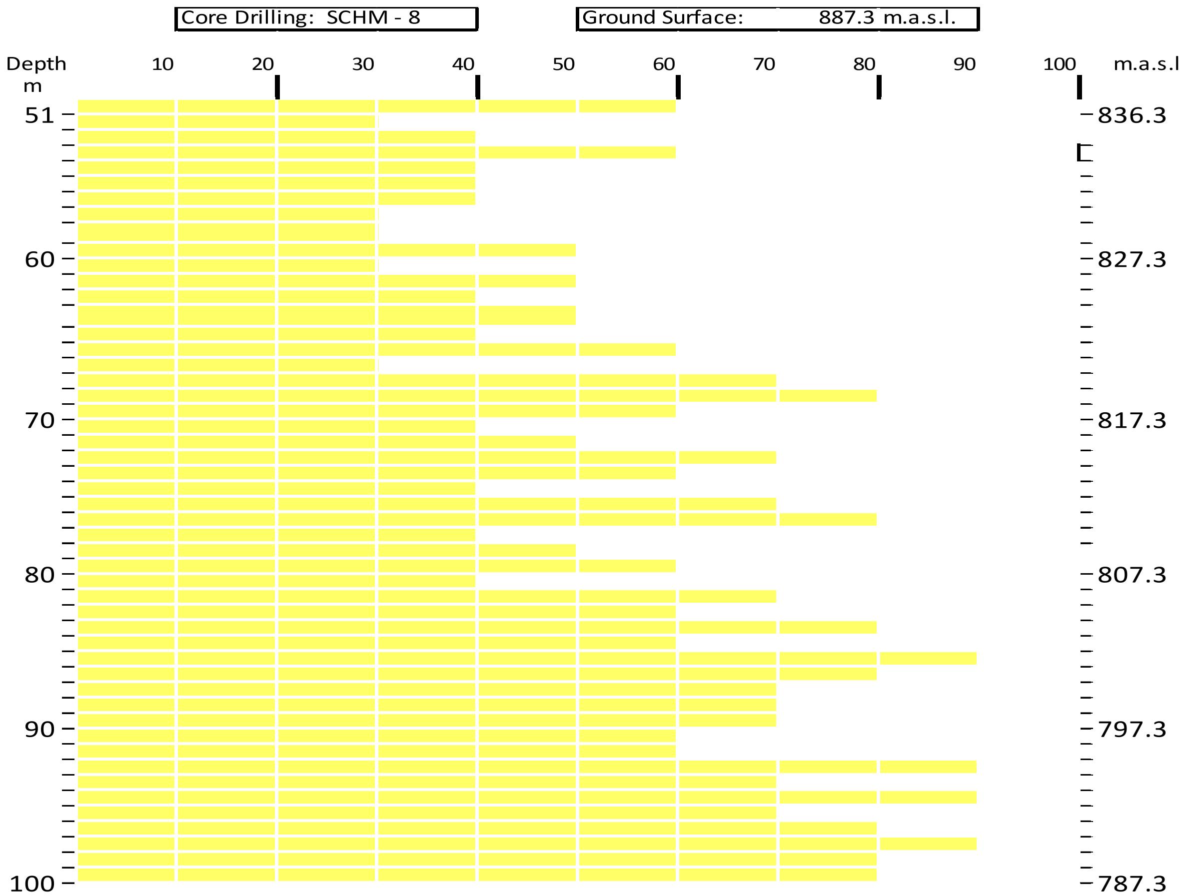

Core Drilling: SCHM - 7
Ground Surface: 885.07 m.a.s.l.
Depth m
20
40
60
80
100
m.a.s.l.
101
105
Core Disking identified
784.1
780.1

Figure 3-15: Core drilling SCHM-7 (83.7 – 94 m deep)

Core drilling 8 (105 m deep): 0 - 40.5 m deep is constituted by fragments of granite, granodiorite, volcanic breccia and andesite. 40.5 – 105 m deep is andesitic basalt.




Core Drilling: SCHM - 8
Ground Surface: 887.3 m.a.s.l.
Depth m
10 20 30 40 50 60 70 80 90 100
m.a.s.l.
51
60
70
80
90
100
836.3
827.3
817.3
807.3
797.3
787.3

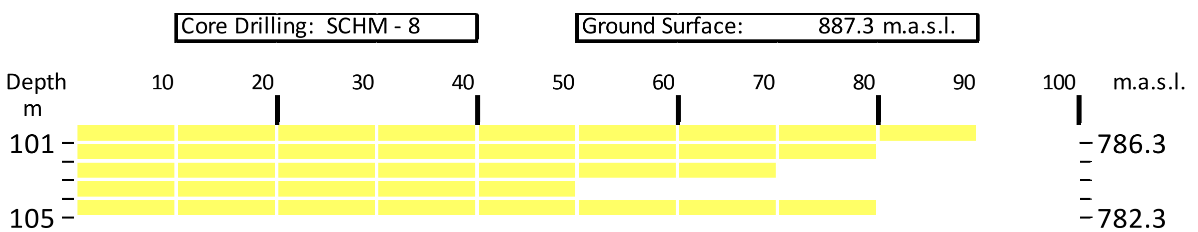
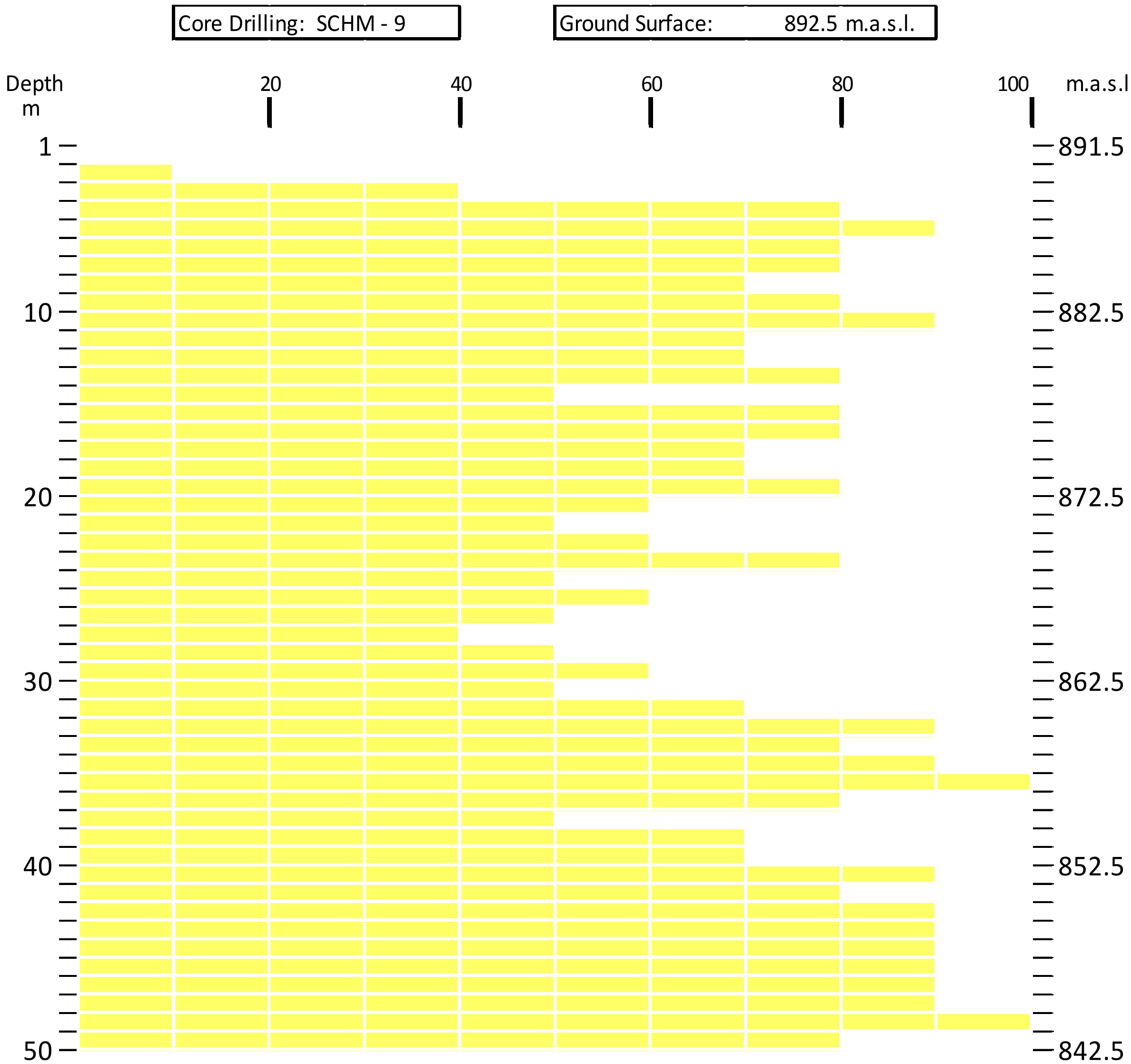
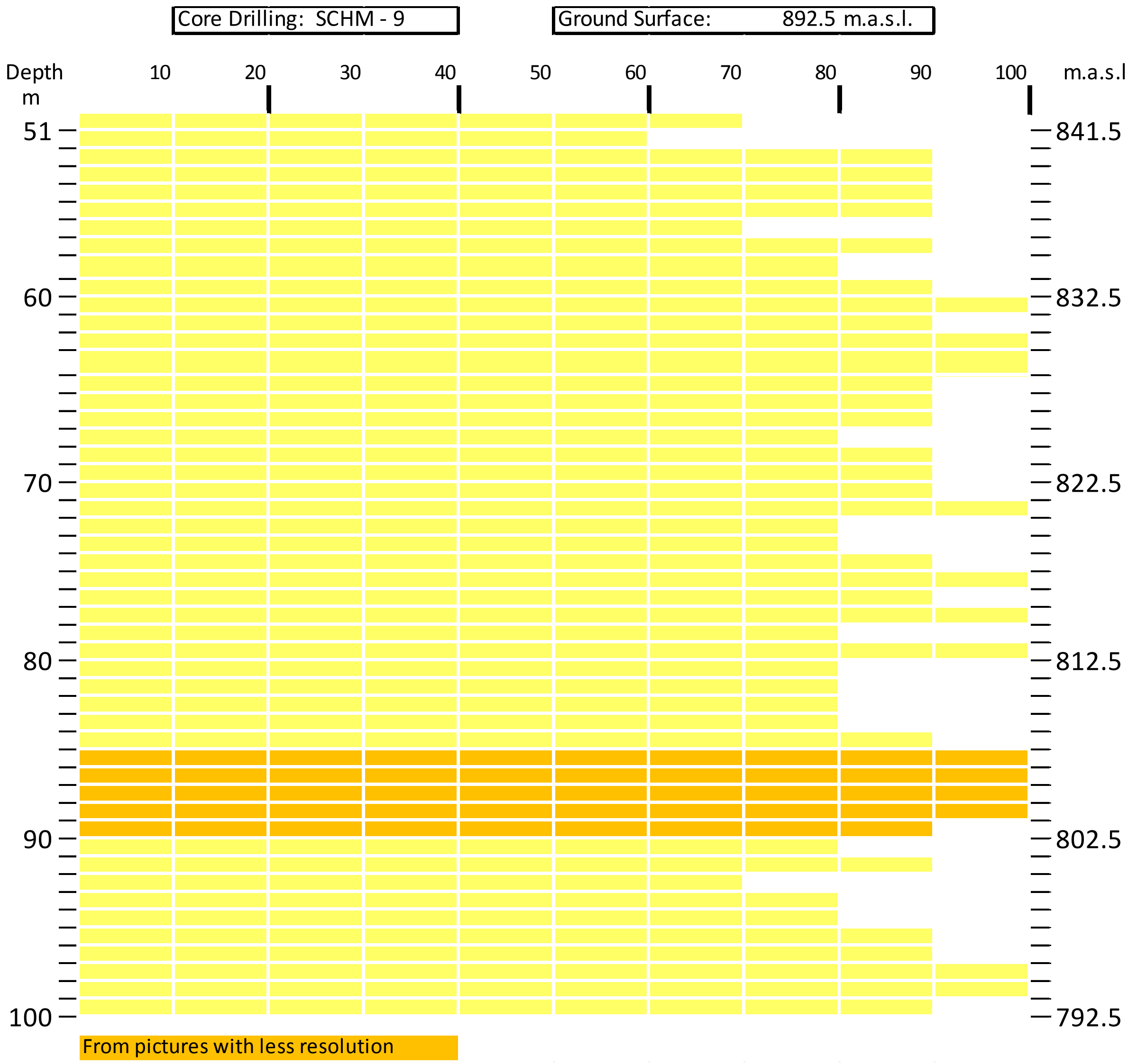

Core Drilling: SCHM - 8
Ground Surface: 887.3 m.a.s.l.
Depth m
10 20 30 40 50 60 70 80 90 100
m.a.s.l.
101
105
786.3
782.3

Figure 3-16: Core drilling SCHM-8. 76.1 – 83.05 m.

Core drilling 9 (105 m deep): Andesitic lava.




Core Drilling: SCHM - 9
Ground Surface: 892.5 m.a.s.l.
Depth
m
10
20
30
40
50
60
70
80
90
100
m.a.s.l.
51
60
70
80
90
100
841.5
832.5
822.5
812.5
802.5
792.5
From pictures with less resolution

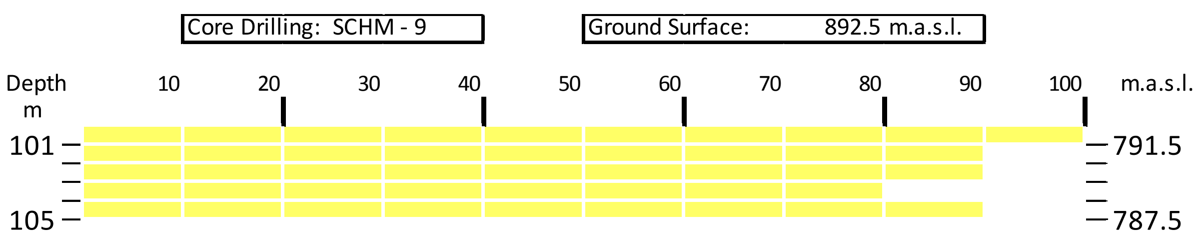
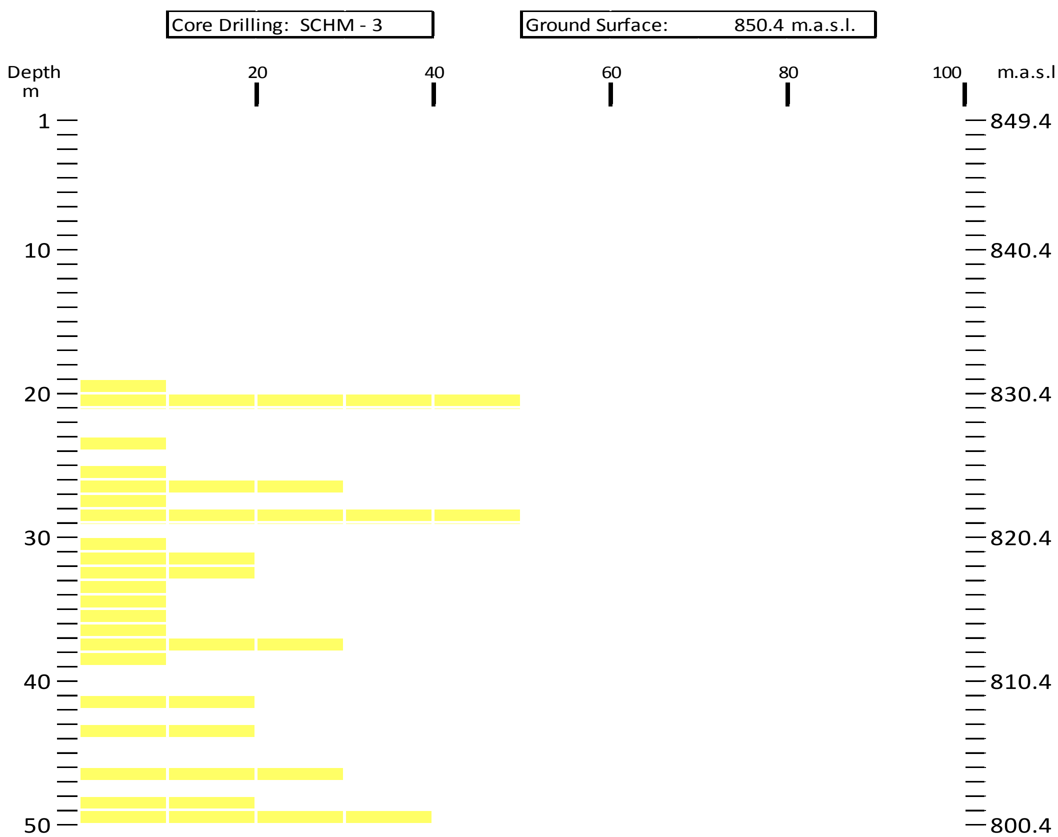

Core Drilling: SCHM - 9
Ground Surface: 892.5 m.a.s.l.
Depth
m
10
20
30
40
50
60
70
80
90
100
m.a.s.l.
101
105
791.5
787.5

Figure 3-17: Core drilling SCHM-9 (89.2 – 92.55 m deep)

Core drilling 3: Ancient Fluvial Deposit & Marginal Aluvial



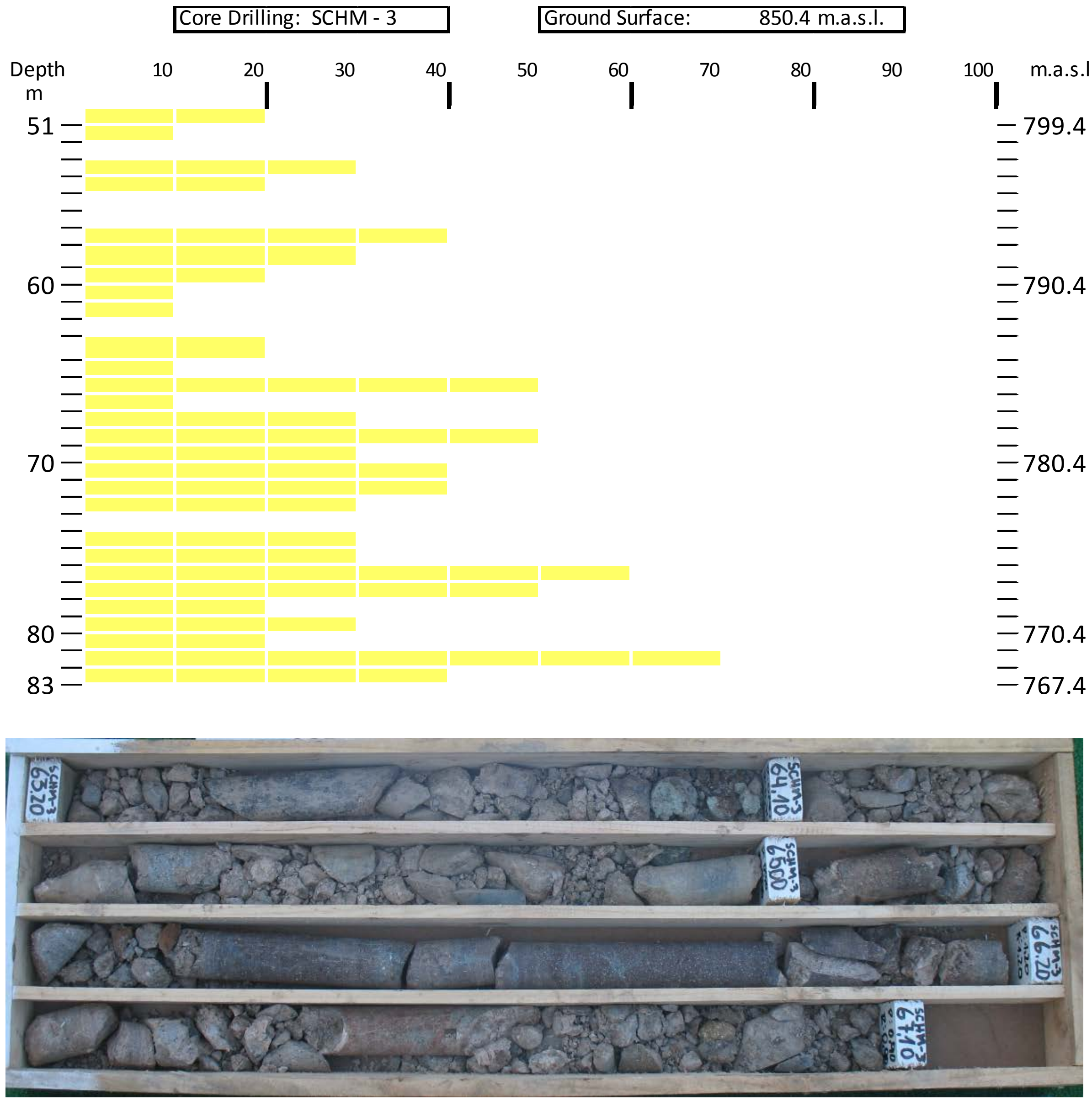
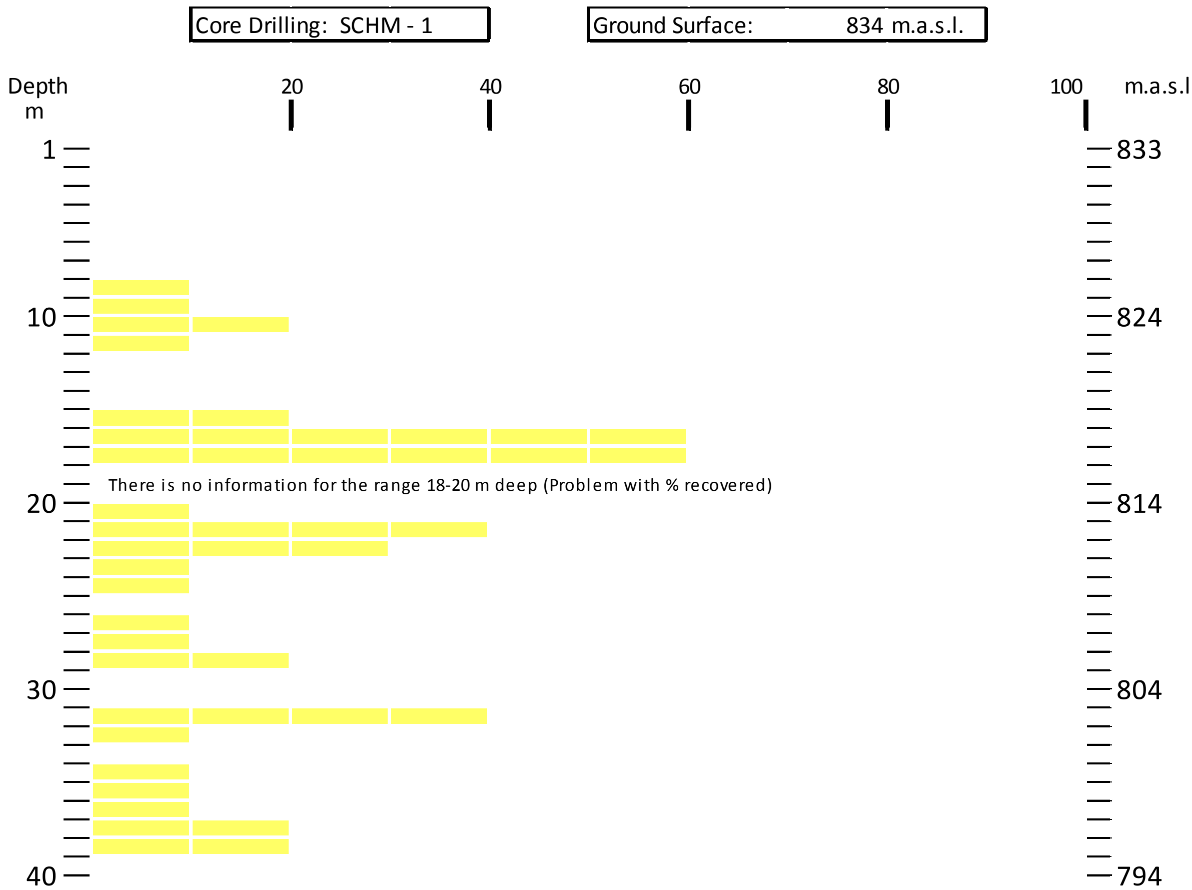
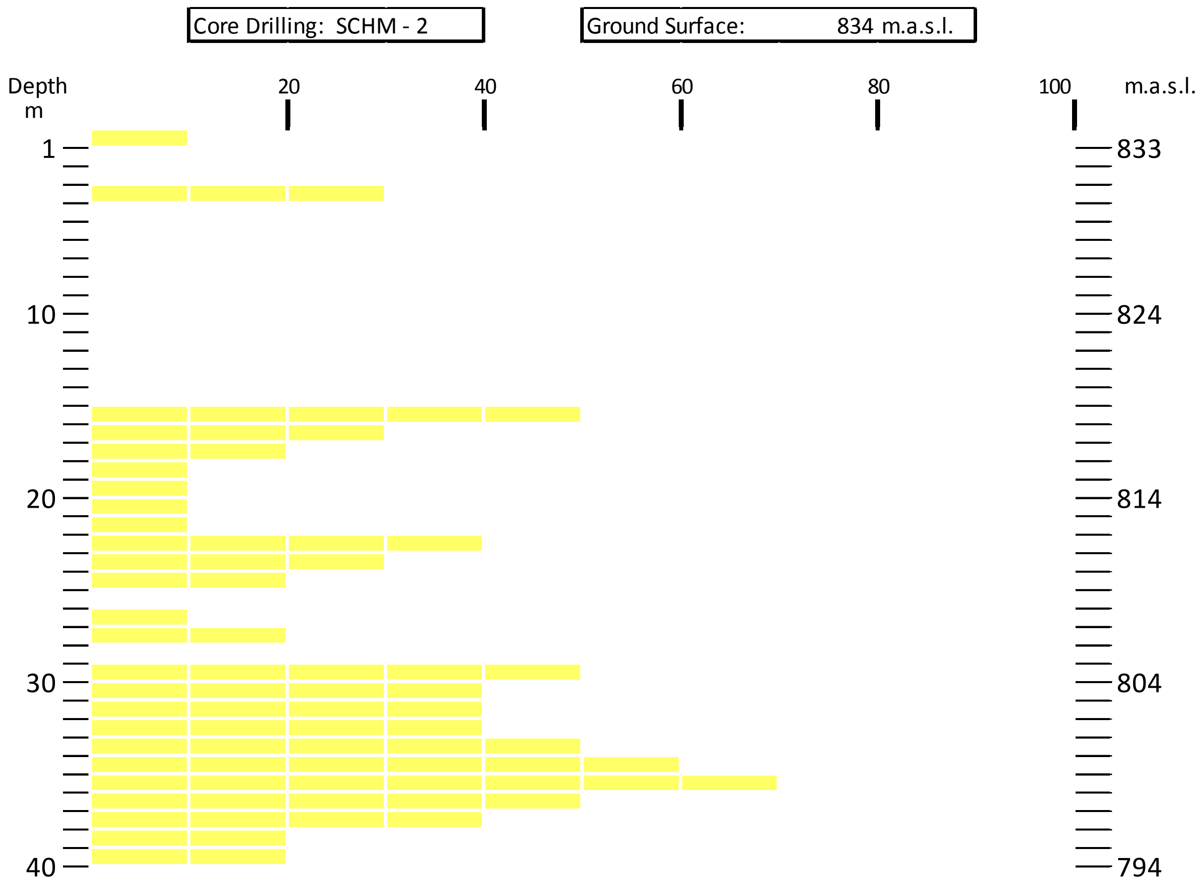


Figure 3-18: Core drilling SCHM-3. 63.2 – 67.2 m deep.

Core drilling 1 (40 m deep): Basic Intrusive. Andesitic composition.



Core drilling 2 (40 m deep): Andesitic Lava (Trapa Trapa).



3.8 Weakness Zones

One can visualize from figures 3.1 and 3.5 that most of the inferred weakness/fault zones are almost perpendicular relatively to the tunnel axis. These zones are mostly related to steep and narrow brooks. Based on the arguments given above, one could tend to think that these zones will be short stretches for the tunnels. To be on the conservative side, each fault/weakness zone will be considered as 100 m long.

Unfortunately there is not a clear character of the weakness zones, but one could extrapolate from the weathering on the Trapa-Trapa rock formation which is a volcanic rock that contains clay coating, that these weakness zones could reach the tunnel 200 m below ground surface. There are cases documented where tuff layer in basalt and similar volcanic rocks has reached several 100 m below surface.

3.9 Water Leakage

Water leakage is the measure of quantity of water that is coming inside the tunnel in liter/minute. It has the great influence for stability of tunnel during construction and operation. In case the clay formed in the Trapa-Trapa rock formation has swelling properties, water ingress could cause severe problems.

In the case of the Batholith Melado rock formation, there could also be water ingress problems due to low overburden near fault zones. This is not a coincidence, because the design criterion during the feasibility studies was to avoid tunnel overburden below 50 m because of surface weathering. And this restriction along with the effort to reduce the tunnel length triggers that the tunnel bends are exactly below brooks where inferred weakness are placed.

The decision of 50 m as a minimum overburden is quite reasonable after visualizing the result of some core drillings where the first 40 m is only crushed rock.

3.10 In situ Stress

The stress situation is a relevant input in rock stability and is governed by a combination of gravitational, topographic, tectonic and residual stresses. Magnitude and direction of in situ stresses prior to excavation are very important, but they are only possible to know well by performing rock stress measurements like triaxial stress measurements by drill hole overcoring test or hydraulic fracturing test. Any of these tests have been carried out so far in the project site, but some approximations can be applied in order to have a first approach.

3.10.1 Stress situation prior to excavation:

The first step is to assume that vertical stressσ_Z is a result of gravity alone.

$$\sigma_Z = \gamma * z$$

γ = Specific gravity of the rock.

z = depth below surface (overburden height).

The specific gravity assigned for granite is 0.027 MPA/m and for Andesite is 0.029 MPA/m.

The second step is to find a value for horizontal stress. As a first approach, one can assume that the ratio between horizontal and vertical stress k follows a certain correlation with depth z. The following chart try to link these variables based on average horizontal and vertical stresses measured from around the world.

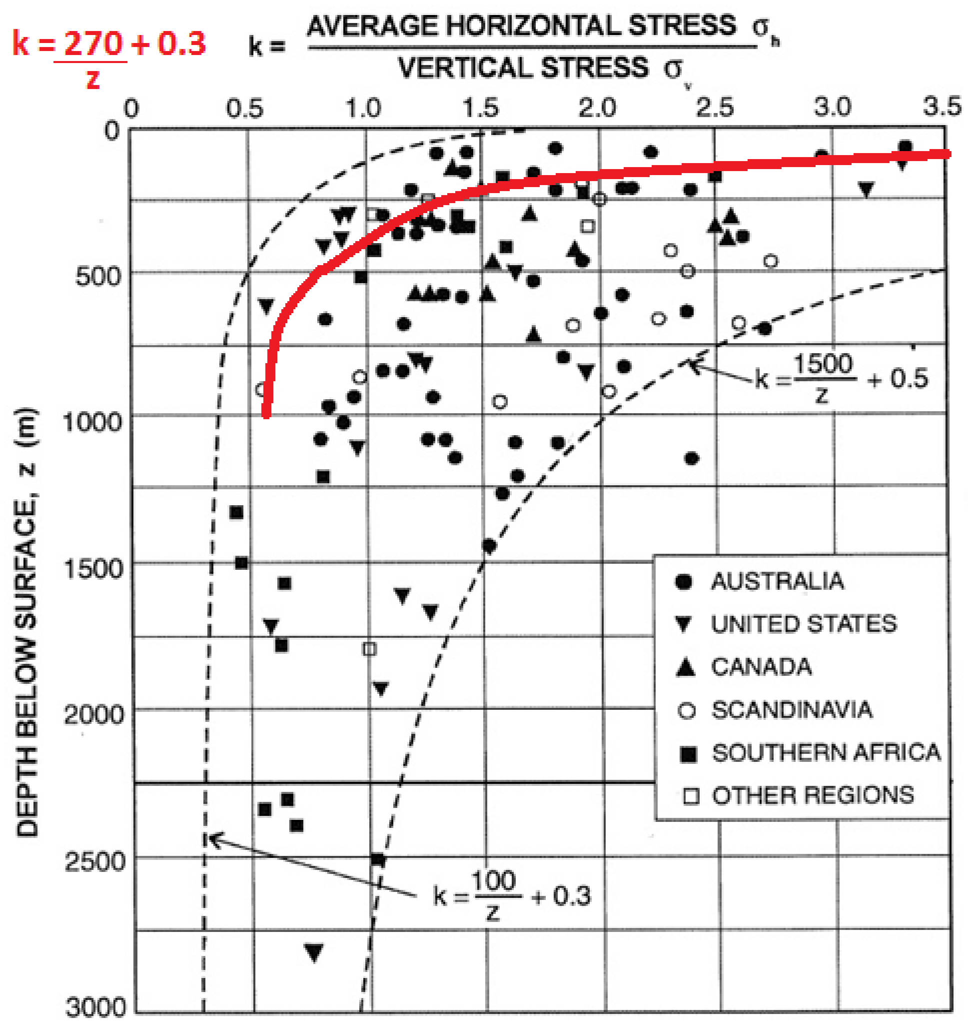


Figure 3-19: Variations of ratio of average horizontal to vertical stress with depth (From Hoek & Brown, 1980).

From the last chart, it is possible to conclude that the correlation between k and z has this general relation:

$$k = \frac{a}{z} + b \quad \text{where } 100 < a < 1500 \text{ ; } 0.3 < b < 0.5$$

Tendency to core disking was found in several core drillings, suggesting a high k value. One evidence of core disking was found at 100 m deep in core drilling SCHM-7 (Batholith Melado: Granite) and another in core drilling SCHM-1 (Andesite) at 32 m deep. A k value equal to 3 at 100 m was adopted. A combination of a and b values that meet this restriction is when a = 270 and b = 0.3.

The equation for k is shown in the previous char in red. And the horizontal stress is:

$$\sigma_H = k * \sigma_z$$

Figure 3-20: Core drilling SCHM-7: 97-65 – 104.5 m deep.

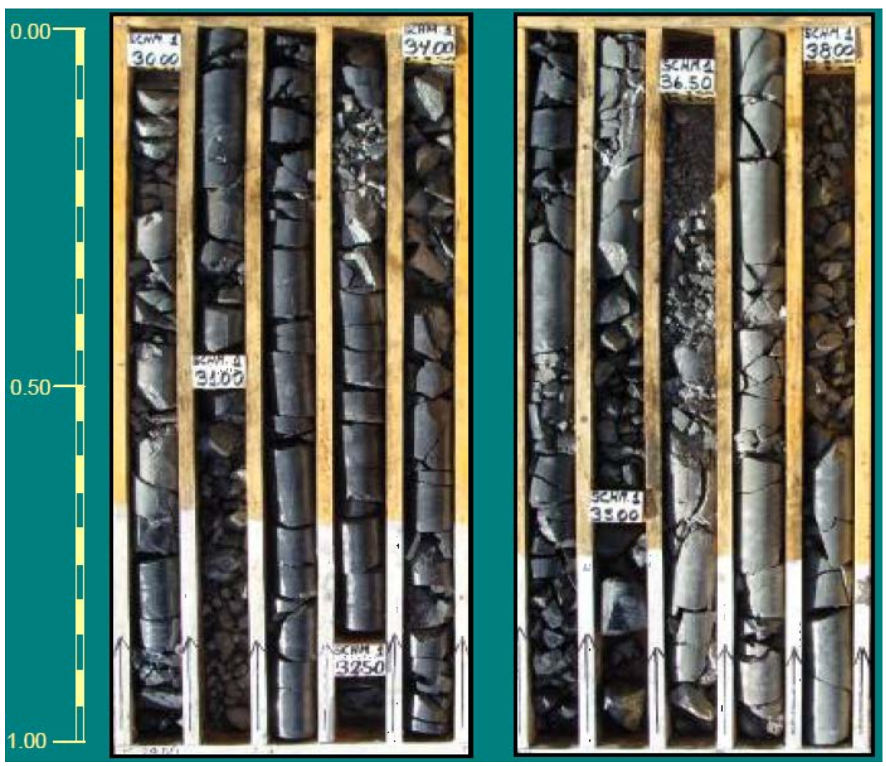


Figure 3-21: Core drilling SCHM-1: 30 – 38 m.

The last step is to assume that there is no topography effect on stress direction and therefore the major principal stress σ_1 is either horizontal or vertical depending on the resulting magnitudes of them and the minor principal stress σ_3 is perpendicular to it.

With these three steps one can have an idea of the principal stress magnitudes.

The overburden along Castillo and Vallical tunnels varies mainly between 100 m and 600 m. In order to provide an idea of the stress situation along the tunnels, it will be estimated the maximum and minimal principal stresses for these extreme values.

For 100 m overburden, the stress situation for each rock type is given below:

Stress Situation: Granite		
z	100	m
ϒ	0.027	MPA/m
σ_V	2.7	MPA
k= σ_h /σ_V	3	
σ_h	8.1	MPA
σ_1	8.1	MPA
σ_3	2.7	MPA

Stress Situation: Andesite		
z	100	m
ϒ	0.029	MPA/m
σ_V	2.9	MPA
k= σ_h /σ_V	3	
σ_h	8.7	MPA
σ_1	8.7	MPA
σ_3	2.9	MPA

It is important to highlight that based on the last results; at 100 m overburden the major principal stresses would be horizontal in both cases. It may not be the wholly true because topography effect should still be relevant and major principal stress should be parallel to ground surface. As long as the ground surface is flat, the last results are valid.

For 600 m overburden, the stress situation for each rock type is given below:

Stress Situation: Granite		
z	600	m
ϒ	0.027	MPA/m
σ_V	16.2	MPA
k= σ_h /σ_V	0.75	
σ_h	12.2	MPA
σ_1	16.2	MPA
σ_3	12.2	MPA

Stress Situation: Andesite		
z	600	m
ϒ	0.029	MPA/m
σ_V	17.4	MPA
k= σ_h /σ_V	0.75	
σ_h	13.1	MPA
σ_1	17.4	MPA
σ_3	13.1	MPA

It is important to mention that in this case the major principal stresses would be vertical instead of horizontal in both rock types. It is expected that at a higher depth underground vertical stress becomes larger than horizontal stress.

3.10.2 Stresses surrounding rock excavation

An underground excavation changes the initial stress condition. For circular opening and anisotropic stress condition, Kirsch's equations can be used to obtain the maximum and minimum tangential stresses:

$$\sigma_{\theta(max)} = 3 * \sigma_1 - \sigma_3$$

$$\sigma_{\theta(min)} = 3 * \sigma_3 - \sigma_1$$

The maximum tangential stress will occur on tunnel contour parallel to major principal stress and minimal tangential stress will occur on tunnel contour parallel to minor principal stress.

Even though the planned tunnel is not circular, but rather it has a horse shoe tunnel shape, the latter equations provide good results as long as there are not sharp corners on tunnel contour.

Finally, it will be estimated the ratio between the maximum tangential stress $\sigma_{\theta(max)}$ and the rock mass strength σ_c which is an indicator of eventual stress-induced problem. The last parameter is given in chapter 3.4 for each rock type.

100 m overburden: Granite		
$\sigma_{\theta\ MAX}$	21.6	MPA
$\sigma_{\theta\ MIN}$	0.0	MPA
σ_c	150	MPA
$\sigma_{\theta\ MAX} / \sigma_c$	0.14	

100 m overburden: Andesite		
$\sigma_{\theta\ MAX}$	23.2	MPA
$\sigma_{\theta\ MIN}$	0.0	MPA
σ_c	100	MPA
$\sigma_{\theta\ MAX} / \sigma_c$	0.23	

600 m overburden: Granite		
$\sigma_{\theta\ MAX}$	36.5	MPA
$\sigma_{\theta\ MIN}$	20.3	MPA
σ_c	150	MPA
$\sigma_{\theta\ MAX} / \sigma_c$	0.24	

600 m overburden: Andesite		
$\sigma_{\theta\ MAX}$	39.2	MPA
$\sigma_{\theta\ MIN}$	21.8	MPA
σ_c	100	MPA
$\sigma_{\theta\ MAX} / \sigma_c$	0.39	

At this stage, without analyzing the ground behavior of each rock formation, it is not possible to judge fairly the last results, but one could say that in case there is blocky Andesite and blocky Granite and topographic effect is insignificant, there should not be large stress-induced problems taking into account the most unfavorable anisotropic stress conditions.

4. ROCK MASS CLASSIFICATION

4.1 Q-System

4.1.1 Introduction

The Q-system was developed by NGI in 1974. NGI defines the Q-method as a numerical description of the rock mass quality with respect to rock stability, and is also used for estimating required permanent rock support in tunnels and caverns. In this chapter, only the rock mass classification is carried out. The corresponding rock support is developed in chapter 6.

From 1974, new updates have been carried out in order to include modern tunnel support. This empirical method is now based on data from more than 1250 examples from existing tunnels around the world.

4.1.2 Methodology

The Q-value is obtained by the following equation:

$$Q = \frac{RQD}{J_n} * \frac{J_r}{J_a} * \frac{J_w}{SRF}$$

RQD = Rock Quality Designation.

J_n = Joint set number

J_r = Joint set roughness (Roughness of the most unfavorable joint).

J_a = Joint alteration number (Degree of alteration or filling along the weakest joint).

J_w = Joint water reduction

SRF = Stress Reduction Factor

The quotient $\frac{RQD}{J_n}$ represents the block size of the rock mass.

The quotient $\frac{J_r}{J_a}$ represents the inter-block shear strength.

The quotient $\frac{J_w}{SRF}$ represents the active stresses.

The rating for each input parameter is shown in the following six tables:

***RQD** (Rock Quality Designation)*

Very poor	RQD = 0 - 25%
Poor	25 - 50
Fair	50 - 75
Good	75 - 90
Excellent	90 - 100
Notes: *(i) Where RQD is reported or measured as < 10 (including 0), a nominal value of 10 is used to evaluate Q* *(ii) RQD intervals of 5, i.e. 100, 95, 90, etc. are sufficiently accurate*	

Table 4-1: Rating of RQD value in the Q-system

***Jn** (joint set number)*

Massive, no or few joints	Jn = 0.5 - 1
One joint set	2
One joint set plus random joints	3
Two joint sets	4
Two joint sets plus random joints	6
Three joint sets	9
Three joint sets plus random joints	12
Four or more joint sets, heavily jointed, "sugar-cube", etc.	15
Crushed rock, earthlike	20
Notes: *(i) For tunnel intersections, use (3.0 x Jn); (ii) For portals, use (2.0 x Jn)*	

Table 4-2: Joint set number

Jr (Joint roughnes number)			
a) Rock-wall contact b) Rock-wall contact before 10 cm shear		c) No rock-wall contact when sheared	
Discontinous joints	Jr =4	Zone containing clay minerals thick enough to prevent rock-wall contact	Jr = 1
Rough or Irregular, undulating	3		
Smooth, undulating	2	Sandy, gravelly or crushed zone thick enough to prevent rock-wall contact	1.0
Slickensided, undulating	1.5		
Rough or Irregular, planar	1.5	Notes: I) Add 1.0 if the mean spacing of the relevant joint set is greater than3 m. II) Jr = 0.5 can be used for planar, slickensided joints having linieations, provided the lineations are oriented for minimum strength.	
Smooth, planar	1		
Slickensided, planat	0.5		
Note: I) Descriptions refer to small scale features,and intermediate scale features, in that order.			

Table 4-3: Joint roughness number

Ja (joint alteration number)

<table>
<tr><td rowspan="6">Contact between joint walls</td><td colspan="2">JOINT WALL CHARACTER</td><td colspan="2">Condition</td><td>Wall contact</td></tr>
<tr><td rowspan="3">CLEAN JOINTS</td><td>Healed or welded joints:</td><td colspan="2">filling of quartz, epidote, etc.</td><td>Ja = 0.75</td></tr>
<tr><td>Fresh joint walls:</td><td colspan="2">no coating or filling, except from staining (rust)</td><td>1</td></tr>
<tr><td>Slightly altered joint walls:</td><td colspan="2">non-softening mineral coatings, clay-free particles, etc.</td><td>2</td></tr>
<tr><td rowspan="2">COATING OR THIN FILLING</td><td>Friction materials:</td><td colspan="2">sand, silt, calcite, etc. (non-softening)</td><td>3</td></tr>
<tr><td>Cohesive materials:</td><td colspan="2">clay, chlorite, talc, etc. (softening)</td><td>4</td></tr>
<tr><td rowspan="6">Some or no wall contact</td><td rowspan="2" colspan="2">FILLING OF:</td><td rowspan="2">Type</td><td>Some wall contact</td><td>No wall contact</td></tr>
<tr><td>Thin filling (< 5 mm)</td><td>Thick filling</td></tr>
<tr><td colspan="2">Friction materials</td><td>sand, silt calcite, etc. (non-softening)</td><td>Ja = 4</td><td>Ja = 8</td></tr>
<tr><td colspan="2">Hard cohesive materials</td><td>compacted filling of clay, chlorite, talc, etc.</td><td>6</td><td>5 - 10</td></tr>
<tr><td colspan="2">Soft cohesive materials</td><td>medium to low overconsolidated clay, chlorite, talc</td><td>8</td><td>12</td></tr>
<tr><td colspan="2">Swelling clay materials</td><td>filling material exhibits swelling properties</td><td>8 - 12</td><td>13 - 20</td></tr>
</table>

Table 4-4: Joint alteration number.

Jw (Joint water reduction factor)		
Dry excavations or minor inflow, i.e. < 5 l/min locally	$\rho_w < 1 kg/cm^2$	Jw = 1
Medium inflow or pressure, occasional outwash oj joint fillings	1-25	0.66
Large inflow or high pressure in competent rock with unfilled joints	2.5 - 10	0.5
Large inflow or high pressure, considerable outwash of joint fillings	2.5 - 10	0.3
Exceptionally high inflow or water pressure at blasting, decaying with time	> 10	0.2 - 0.1
Exceptionally high inflow or water pressure continuing without noticeable decay	> 10	0.1 - 0.05
Note I) The last four factors are crude estimates, increase Jw if drainage measures are installed II) Special problems cause by ice formation are not considered.		

Table 4-5: Joint wáter reduction factor.

SRF *(Stress Reduction Factor)*

		SRF
Weakness zones intersecting excavation	Multiple weakness zones with clay or chemically disintegrated rock, very loose surrounding rock (any depth)	SRF = 10
	Single weakness zones containing clay or chemically disintegrated rock (depth of excavation < 50 m)	5
	Single weakness zones containing clay or chemically disintegrated rock (depth of excavation > 50 m)	2.5
	Multiple shear zones in competent rock (clay-free), loose surrounding rock (any depth)	7.5
	Single shear zones in competent rock (clay-free), loose surrounding rock (depth of excavation < 50 m)	5
	Single shear zones in competent rock (clay-free), loose surrounding rock (depth of excavation > 50 m)	2.5
	Loose, open joints, heavily jointed or "sugar-cube", etc. (any depth)	5

Note: *(i) Reduce these SRF values by 25 - 50% if the relevant shear zones only influence, but do not intersect the excavation.*

		σ_c / σ_1	σ_θ / σ_c	SRF
Competent rock, rock stress problems	Low stress, near surface, open joints	> 200	< 0.01	2.5
	Medium stress, favourable stress condition	200 - 10	0.01 - 0.3	1
	High stress, very tight structure. Usually favourable to stability, may be except for walls	10 - 5	0.3 - 0.4	0.5 - 2
	Moderate slabbing after > 1 hour in massive rock	5 - 3	0.5 - 0.65	5 - 50
	Slabbing and rock burst after a few minutes in massive rock	3 - 2	0.65 - 1	50 - 200
	Heavy rock burst (strain burst) and immediate dynamic deformation in massive rock	< 2	> 1	200 - 400

Notes: *(ii) For strongly anisotropic stress field (if measured): when $5 < \sigma_1/\sigma_3 < 10$, reduce σ_c to $0.75\ \sigma_c$. When $\sigma 1/\sigma 3 > 10$, reduce σ_c to $0.5\sigma_c$*

(iii) Few case records available where depth of crown below surface is less than span width. Suggest SRF increase from 2.5 to 5 for low stress cases

			σ_θ / σ_c	SRF
Squeezing rock	*Plastic flow of incompetent rock under the influence of high pressure*	Mild squeezing rock pressure	1 - 5	5 - 10
		Heavy squeezing rock pressure	> 5	10 - 20
Swelling rock	*Chemical swelling activity depending on presence of water*	Mild swelling rock pressure		5 - 10
		Heavy swelling rock pressure		10 - 15

Table 4-6: Stress reduction factor.

Based on the description undertaken in chapter 3, all the Q-system input parameters can be determined.

The following chart provides the rock class according to the Q-system when it is known the Q-value.

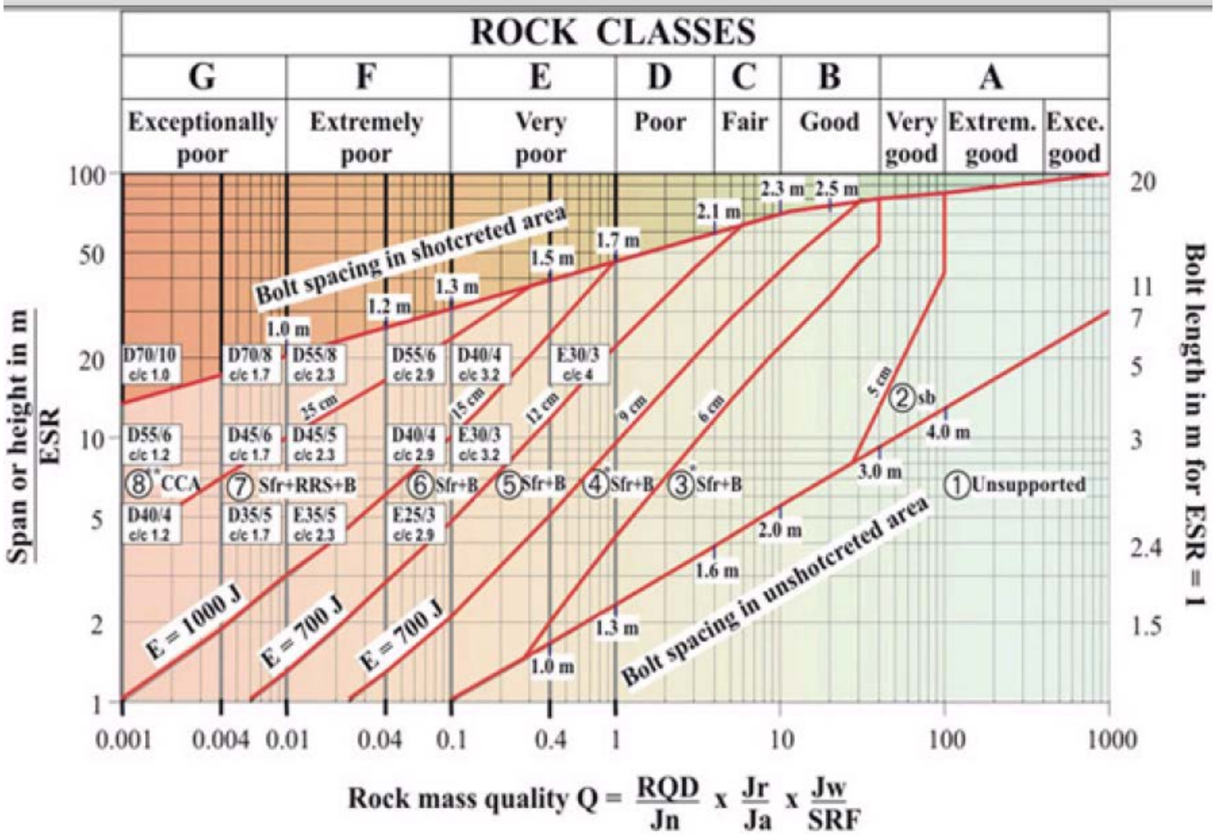


Figure 4-1: Updated Q-system Chart, NGI.

In order to obtain the Q-value in walls, the last chart can also be used, and the corresponding Q value is related to the roof Q-value in the following way:

$Q_{WALL} = 5*Q_{ROOF}$ if $Q_{ROOF} > 10$

$Q_{WALL} = 2.5*Q_{ROOF}$ if $0.1 < Q_{ROOF} < 10$

$Q_{WALL} = Q_{ROOF}$ if $Q_{ROOF} < 0.1$

4.2 RMR System

4.2.1 Introduction

Rock mass rating was also included to have a benchmark for comparison to the Q-system.

RMR classification system developed by Bieniawski in 1973 includes the following six parameters.

- Uniaxial compressive strength of intact rock material.

- Rock Quality Designation RQD

- Spacing of discontinuities

- Condition of discontinuities

- Groundwater conditions

- Orientation of discontinuities

4.2.2 Methodologhy

The rating of each variable is given below:

A. Classification parameters and their ratings

PARAMETER			Range of values // RATINGS						
1	Strength of intact rock material	Point-load strength index	> 10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range uniaxial compr. strength is preferred		
		Uniaxial compressive strength	> 250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
		RATING	15	12	7	4	2	1	0
2	Drill core quality RQD		90 - 100%	75 – 90%	50 - 75%	25 - 50%	< 25%		
		RATING	20	17	13	8	5		
3	Spacing of discontinuities		> 2 m	0.6 - 2 m	200 - 600 mm	60 - 200 mm	< 60 mm		
		RATING	20	15	10	8	5		
4	Condition of discontinuities	Length, persistence	< 1 m	1 - 3 m	3 - 10 m	10 - 20 m	> 20 m		
		Rating	6	4	2	1	0		
		Separation	none	< 0.1 mm	0.1 - 1 mm	1 - 5 mm	> 5 mm		
		Rating	6	5	4	1	0		
		Roughness	very rough	rough	slightly rough	smooth	slickensided		
		Rating	6	5	3	1	0		
		Infilling (gouge)	none	Hard filling		Soft filling			
			-	< 5 mm	> 5 mm	< 5 mm	> 5 mm		
		Rating	6	4	2	2	0		
		Weathering	unweathered	slightly w.	moderately w.	highly w.	decomposed		
		Rating	6	5	3	1	0		
5	Ground water	Inflow per 10 m tunnel length	none	< 10 litres/min	10 - 25 litres/min	25 - 125 litres/min	> 125 litres /min		
		$p_w / \sigma 1$	0	0 - 0.1	0.1 - 0.2	0.2 - 0.5	> 0.5		
		General conditions	completely dry	damp	wet	dripping	flowing		
		RATING	15	10	7	4	0		

p_w = joint water pressure; $\sigma 1$ = major principal stress

Table 4-7: RMR input parameters and their rating

B. Rating adjustment for discontinuity orientations

		Very favourable	Favourable	Fair	Unfavourable	Very unfavourable
RATINGS	Tunnels	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	-60

Table 4-8: Rating adjustment for discontinuity orientation.

C. Rock mass classes determined from total ratings

Rating	100 – 81	80 – 61	60 - 41	40 - 21	< 20
Class No.	I	II	III	IV	V
Description	VERY GOOD	GOOD	FAIR	POOR	VERY POOR

Table 4-9: Rock mass classed base on RMR.

In order to compare the results between the Q-system and the RMR system, the following equation will be used:

$$Q = e^{(RMR-44)/9} \text{ (Bieniawski, 1989)}$$

4.3 Results:

In Trapa-Trapa rock formation (represented by Andesite) a distinction is made between Massive, Blocky and Weakness zones.

In Granitic intrusive rock formation (represented by granite) a distinction is made between blocky and weakness zone. (No massive granite is expected as it will be shown in chapter 6 related to ground behavior).

Since in the RMR system an explicit rating is considered for discontinuity orientation, two rock mass classifications are given for “Blocky Trapa-Trapa” (or “Blocky Andesite”) and Blocky Granitic Intrusive (Or “Blocky Granite”). These two discontinuity orientations are “fair” when the strike of the predominant joint set compared to the tunnel axis is less than 20 degrees apart) and “favorable”, when the strike of the predominant joint set is higher than 20 degrees apart with regard to the tunnel axis.

4.3.1 Trapa-Trapa Rock Formation: “Blocky Andesite”

The following rock mass classification belongs to the Castillo Tunnel crossing the Trapa-Trapa rock formation in between weakness zones.

Q-System = Trapa - Trapa Rock Formation (Andesite) in Castillo Tunnel "Blocky Andesite"		
Input parameter	Rating	Comment
RQD	90	RQD (20 m average) at the level of the tunnel axis from core drilling SCHM-9
Jn	12	3 joint sets plus random joints
Jr	1.0	Smooth joint surface & No waviness (Planar)
Ja	4	Clay and Chlorite coating (Contact between joint walls)
Jw	1.0	Minor inflow q < 5 l/min excluding weakness zones
SRF	1.0	In general Medium stress: $0.22 < \sigma_{\theta\,MAX}/\sigma_c < 0.37$; $6.2 < \sigma_c/\sigma_1 < 12.2$
Q_{ROOF}	**1.9**	**Class D (Poor)**
Q_{WALL}	**4.7**	**Class C (Fair)**

The RMR system makes an explicit distinction for discontinuity orientation. Hence, two RMR classifications are given for “blocky Andesite” along the Castillo Tunnel, fair (Chainage 1850 - 3900) and favorable orientation for the rest of the Castillo tunnel in between weakness zones.

RMR System = Trapa - Trapa Rock Formation (Andesite) in Castillo Tunnel (Fair orientation) "Blocky Andesite"			
A	Classification parameters & their ratings		
1	Strength of intact rock material	10	σ_c = 100 MPA
2	RQD	20	(RQD = 90) from core drilling SCHM - 9
3	Spacing of Discontinuities	8	100-300 mm
4.1	Length, Persistance	2	3-10 m (No information available; Average value adopted)
4.2	Separation	4	0.1 - 1 mm (Assumed from "close joint" description)
4.3	Roughness	1	Smooth
4.4	Infilling (Gouge)	2	Soft filling < 5 mm
4.5	Weathering	3	Moderately weathered
4	Condition of Discontinuities	12	
5	Ground Water	10	Inflow 10 l/min
B	Rating Adjustment for discontinuity orientations		
Tunnel		-5	Fair
	RMR	**55.0**	Fair (RMR rating)
	Q-value (From RMR)	3.4	Poor (Q-System)

RMR System = Trapa - Trapa Rock Formation (Andesite) in Castillo Tunnel (Favourable orientation) "Blocky Andesite"			
A	Classification parameters & their ratings		
1	Strength of intact rock material	10	σ_c = 100 MPA
2	RQD	20	(RQD = 90) from core drilling SCHM - 9
3	Spacing of Discontinuities	8	100-300 mm
4.1	Length, Persistance	2	3-10 m (No information; Average value adopted)
4.2	Separation	4	0.1 - 1 mm (Assumed from "close joint" description)
4.3	Roughness	1	Smooth
4.4	Infilling (Gouge)	2	Soft filling < 5 mm
4.5	Weathering	3	Moderately weathered
4	Condition of Discontinuities	12	Sum up from 4.1 to 4.5
5	Ground Water	10	Inflow 10 l/min
B	Rating Adjustment for discontinuity orientations		
	Tunnel	-2	Favourable
	RMR	**58.0**	Fair (RMR rating)
	Q (From RMR)	4.7	Fair (Q-System)

Trapa-Trapa is also found in Vallical tunnel, and the main difference compared to the “blocky Andesite” in Castillo tunnel is that core drilling SCHM-8 is considered instead of core drilling SCHM-9 and higher water inflow problems are expected due to low overburden.

Trapa - Trapa Rock Formation (Andesite) in Vallical Tunnel		
Input parameter	Rating	Comment
RQD	70	RQD (20 m average) at the level of the tunnel axis from core drilling SCHM-8
Jn	12	3 joint sets plus random joints
Jr	1.0	Smooth joint surface & No waviness (Planar)
Ja	4	Clay and Chlorite coating (Contact between joint walls)
Jw	0.66	Medium inflow predicted (Low overburden)
SRF	1.0	In general Medium stress: $\sigma_{\theta\ MAX}/\sigma_c = 0.22$; $\sigma_c/\sigma_1 = 12.2$
Q_{ROOF}	**1.0**	**Class D/E Between Poor and Extremely Poor**
Q_{WALLS}	**2.4**	**Class D (Poor)**

RMR System = Trapa - Trapa Rock Formation in Vallical Tunnel (Chainage 10,100 - 10,300 m) " Blocky Andesite"			
A	Classification parameters & their ratings		
1	Strength of intact rock material	10	σ_c = 100 MPA
2	RQD	13	(RQD = 70) From Core drilling SCHM-8
3	Spacing of Discontinuities	8	100 - 300 mm
4.1	Length, Persistance	2	3-10 m (No information available; Average value adopted)
4.2	Separation	4	0.1 - 1 mm (Assumed from "close joint" description)
4.3	Roughness	1	Smooth
4.4	Infilling (Gouge)	2	Soft filling < 5 mm
4.5	Weathering	3	Moderately weathered
4	Condition of Discontinuities	12	
5	Ground Water	7	Inflow 10-25 l/min
B	Rating Adjustment for discontinuity orientations		
	Tunnel	-2	Favourable
	RMR	**48.0**	Fair (RMR rating)
	Q (From RMR)	1.6	Poor (Q-System)

4.3.2 Trapa-Trapa Rock Formation: Massive Andesite

An exception in *Trapa-Trapa* rock mass classification in between weakness zones was made for the Chainage 7770 - 9400 m (Near the downstream end of Castillo tunnel), where the core drilling SCHM-9 carried out in the area showed very good results with an RQD between 90 and 100 for the deepest 45 m.

The following rock mass classification belongs to the Castillo Tunnel stretch just described:

Q-System = Trapa - Trapa Rock Formation "Massive Andesite" : Chainage 7770 - 9400 m		
Input parameter	Rating	Comment
RQD	90	RQD average at the level of the tunnel axis from core drilling SCHM 9
Jn	1	No joint set
Jr	1.0	smooth joint surface & No waviness (Planar)
Ja	4	Clay and Chlorite coating (Contact between joint walls)
Jw	1.0	Minor inflow q < 5 l/min excluding weakness zones
SRF	1.0	Competent rock; Medium stress ; favourable stress conditions
Q_{ROOF}	**23**	**Classe B Good**
Q_{WALL}	**56**	**Class A Very Good**

RMR System = Trapa - Trapa Rock Formation (Andesite) in Castillo Tunnel "Massive Andesite" Chainage 7770 - 9400 m			
A	Classification parameters & their ratings		
1	Strength of intact rock material	10	σc = 100 MPA
2	RQD	20	(RQD = 90)
3	Spacing of Discontinuities	20	> 2 m
4.1	Length, Persistance	2	3-10 m (No information available; Average value adopted)
4.2	Separation	4	0.1 - 1 mm (Assumed from "close joint" description)
4.3	Roughness	1	Smooth
4.4	Infilling (Gouge)	2	Soft filling < 5 mm
4.5	Weathering	3	Moderately weathered
4	Condition of Discontinuities	12	
5	Ground Water	10	Inflow 10 l/min
B	Rating Adjustment for discontinuity orientations		
	Tunnel	-2	Favourable
	RMR	**70.0**	
	Q (From RMR)	18.0	

4.3.3 Trapa-Trapa Rock Formation: Weakness/fault zones

All the fault/weakness zones found in Trapa-Trapa rock formation are represented by the following Q- system rock mass classification:

Q-System = Weakness Zone: Trapa-Trapa (Andesite)		
Input parameter	Rating	Comment
RQD	10	From Core drilling SCHM-1
Jn	20	Crushed rock
Jr	1.0	Smooth joint surface & No waviness (Planar)
Ja	6	Clay coating & argilic alteration
Jw	0.3	Large inflow eventually (Volcanic rock)
SRF	10.0	weakness zones with clay
Q_{ROOF}	**0.003**	**Class G Exceptionally poor**
Q_{WALL}	**0.003**	**Class G Exceptionally poor**

RMR System = Trapa - Trapa Rock Formation (Andesite) Weakness/fault zones			
A	Classification parameters & their ratings		
1	Strength of intact rock material	1	σ_c = 3 MPA
2	RQD	5	(RQD = 10) From core drilling SCHM-1
3	Spacing of Discontinuities	5	< 60 mm
4.1	Length, Persistance	0	Crushed rock (Not related to length)
4.2	Separation	1	1 - 5 mm (Assumption of longer separationin crushed rock compared to blocky Andesite)
4.3	Roughness	1	Smooth
4.4	Infilling (Gouge)	2	Soft filling < 5 mm
4.5	Weathering	1	Highly weathered (Heavily fractured; clay coating)
4	Condition of Discontinuities	4	
5	Ground Water	4	Dripping (Volcanic rock)
B	Rating Adjustment for discontinuity orientations		
	Tunnel	-12	Very unfavourable
	RMR	**7.0**	Very Poor (RMR Rating)
	Q (From RMR)	0.02	Extremely Poor (Q-System)

4.3.4 Granitic Intrusive: “Blocky Granite”

Granitic intrusive is only found in Vallical tunnel. The rock mass classification is given in the following three tables for this rock formation where no weakness zone is expected:

Q-System = Granitic Intrusive Rock Formation (Granite) in Vallical Tunnel "Blocky Granite"		
Input parameter	Rating	Comment
RQD	80	RQD (20 m average) at the level of the tunnel axis from Core drilling number SCHM-7
Jn	9	3 joint sets
Jr	1.5	Planar surface, rough joint surface
Ja	2	Contact between joint walls; Clean joints; Slightly altered joint walls
Jw	1.0	Minor inflow q < 5 l/min
SRF	1.0	In general Medium stress : $0.14 < \sigma_{\theta\,MAX}/\sigma_c < 0.24$; $9.3 < \sigma_c/\sigma_1 < 18.5$
Q_{ROOF}	**6.7**	**Class C (Fair)**
Q_{WALL}	**16.7**	**Class B (Good)**

The RMR discontinuity orientation triggers two different classifications for “blocky-granite”. Fair Andesite between Chainage 10,000 -15,000 m and 17400-18500 m and favorable orientation for the rest of Vallical tunnel where no weakness zone is expected.

RMR System = Granitic Intrusive (Granite) in Vallical Tunnel (Fair Orientation) "Blocky granite"			
A	Classification parameters & their ratings		
1	Strength of intact rock material	12	σ_c = 150 MPA
2	RQD	17	(RQD = 80) From Core drilling SCHM-7
3	Spacing of Discontinuities	10	60 - 600 mm
4.1	Length, Persistance	2	3-10 m (No information available; Average value adopted)
4.2	Separation	4	0.1 - 1 mm (Assumed from "close joint"description)
4.3	Roughness	3	slightly rough
4.4	Infilling (Gouge)	4	Soft filling < 5 mm
4.5	Weathering	5	Slightly weathered
4	Condition of Discontinuities	18	
5	Ground Water	10	< Inflow 10 l/min
B	Rating Adjustment for discontinuity orientations		
	Tunnel	-5	Fair
	RMR	**62.0**	Good (RMR Rating)
	Q (From RMR)	7.4	Fair (Q-system)

RMR System = Granitic Intrusive (Granite) in Vallical Tunnel (Favorable Orientation) "Blocky Granite"			
A	Classification parameters & their ratings		
1	Strength of intact rock material	12	σ_c = 150 MPA
2	RQD	17	(RQD = 80) From Core drilling SCHM-7
3	Spacing of Discontinuities	10	60 - 600 mm
4.1	Length, Persistance	2	3-10 m (No information available; Average value adopted)
4.2	Separation	4	0.1 - 1 mm (Assumed from "close joint"description)
4.3	Roughness	3	slightly rough
4.4	Infilling (Gouge)	4	Soft filling < 5 mm
4.5	Weathering	5	Slightly weathered
4	Condition of Discontinuities	18	
5	Ground Water	10	< Inflow 10 l/min
B	Rating Adjustment for discontinuity orientations		
	Tunnel	-2	Favorable
	RMR	**65.0**	Good (RMR Rating)
	Q (From RMR)	10.3	Good (Q-system)

4.3.5 Granitic Intrusive: Weakness/fault zones

All the fault/weakness zones found in Granitic Intrusive rock formation are represented by the following Q- system rock mass classification:

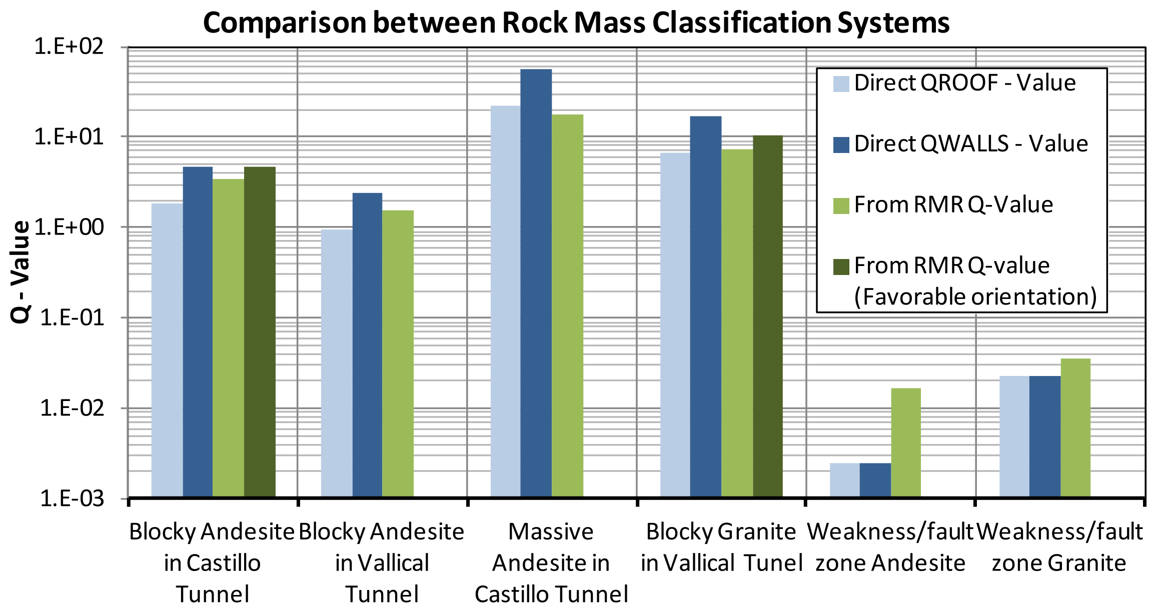
Q-System = Weakness Zone: Granitic Intrusive		
Input parameter	Rating	Comment
RQD	10	First 10 m from core drillings SCHM 5 & 7
Jn	20	Crushed rock
Jr	1.5	**rough joint** surface & No waviness (**Planar**)
Ja	2	Contact between joint walls; Clean joints, non softening minerals coating; clay free content
Jw	0.3	Large infloe eventually (Low overburden)
SRF	5.0	Weakness zone; Shear zone in competent rock (clay-free)
Q_{ROOF}	**0.02**	**Class F Extremely Poor**
Q_{WALL}	**0.02**	**Class F Extremely Poor**

Finally, it is also given the RMR rating for Granitic Intrusive weakness/fault zones:

RMR System = Granitic Intrusive (Granite) Weakness/fault zones			
A	Classification parameters & their ratings		
1	Strength of intact rock materia	1	σc = 3 MPA
2	RQD	5	(RQD = 10) First 10 m from core drilling SCHM 5 & 7
3	Spacing of Discontinuities	5	< 60 mm
4.1	Length, Persistance	0	Crushed rock (Not related to length)
4.2	Separation	1	1 - 5 mm (Assumption of longer separation in crushed rock compared to blocky granite)
4.3	Roughness	3	Slightly rough
4.4	Infilling (Gouge)	4	Hard filling < 5 mm
4.5	Weathering	3	Moderately weathering
4	Condition of Discontinuities	11	
5	Ground Water	4	Dripping (Low overburden ≈ 50 m)
B	Rating Adjustment for discontinuity orientations		
	Tunnel	-12	Very unfavourable
	RMR	**14.0**	
	Q (From RMR)	0.04	

4.4 Comparison & Discussion

The following chart summarizes the main results in this chapter:



From the chart, it can be inferred the following:

1- RMR and Q rock classification systems are quite consistent with each other, having almost no difference in blocky and massive rock formations.

2- Blocky granite has a higher Q-Value than “Blocky Andesite”.

3- The larger differences between RMR and Q rock classification systems lie on weakness/fault zones, where Q-values from RMR system are higher.

With regard to point 1, one can add that in general, massive and blocky rocks are well described and classified by RMR, even it could be considered better than Q-system in one aspect which is that RMR is explicitly sensitive to discontinuity orientation. However, RMR has some parameters that are not intuitive for poor rock conditions (faults or weakness zones). Concepts like “persistence” depending on discontinuity length, “adjustment for discontinuity orientation” and “UCS of intact rock” included in the RMR system are not intuitive parameters for crushed rocks. Q-system provides a more general overview when one is rating a crushed rock, without requesting input parameters that are more appropriate in blocky and massive rocks.

With regard to point 2, it is possible to say that 2 “Blocky Granite” obtained a better Q-value compared to “Blocky Andesite” because of:

- The expected Uniaxial Compressive Strength UCS is higher.

- The joint walls are less weathered and with a higher surface roughness.

- The infilling material is harder (sandy) than in the case of “Blocky Andesite” where clay was found.

- "Blocky Granite" does not have random joints along with the three joint sets.

Finally, point 3 leads to conclude that if Q-system is adopted, a more conservative decision is made. Therefore, Q-system rock mass classification will be used to classify the tunnels, but Q-values will be corrected by RMR system in cases where favorable orientation is expected and Q-system is not quantifying it. The correction factor for the preliminary Q-system value will be the quotient between the new and preliminary Q-value from the RMR system.

Based on this, the Q value adopted for Castillo tunnel is shown below:

Castillo Tunnel: Rock Mass Classification							
Chainage	**Stretch Length m**	**Main Rock type**	**RMR**	**Q (RMR)**	**Comment**	**Q_{ROOF}**	**Q_{ROOF} ADOPTED**
1700 - 1850	150	Andesite	7	0.02	Surface weathering	0.003	**0.003**
1850 - 3900	2050	Andesite	55	3.4	Fair orientation - Poor/Fair Andesite	1.9	**1.9**
3900 - 4330	430	Andesite	58	4.7	Good Orientation - Poor/Fair Andesite	1.9	**2.6**
4330 - 4430	100	Andesite	7	0.02	Inferred fault zone around Las Mulas brook	0.003	**0.003**
4430 - 5350	920	Andesite	58	4.7	Good Orientation - Poor/Fair Andesite	1.9	**2.6**
5350 - 5450	100	Andesite	7	0.02	Inferred fault zone around Las Yeguas brook	0.003	**0.003**
5450 - 6380	930	Andesite	58	4.7	Good Orientation - Poor/Fair Andesite	1.9	**2.6**
6380 - 6480	100	Andesite	7	0.02	Inferred fault zone I from aerial photos	0.003	**0.003**
6480 - 7670	1190	Andesite	58	4.7	Good Orientation - Poor/Fair Andesite	1.9	**2.6**
7670 - 7770	100	Andesite	7	0.02	Inferred fault zone II from aerial photos	0.003	**0.003**
7770 - 9400	1630	Andesite	70	18.0	Good Orientation - Good Andesite	23	**23**
9400 - 9500	100	Andesite	7	0.02	Surface Weathering	0.003	**0.003**
	7800						

Table 4-10: Final rock mass classification for Castillo tunnel.

And for Vallical tunnel, the rock mass classification per tunnel stretch is given below:

Vallical Tunnel: Rock Mass Classification							
Chainage	**Stretch Length m**	**Main Rock type**	**RMR**	**Q (RMR)**	**Comment**	**Q_{ROOF}**	**Q_{ROOF} ADOPTED**
10000 - 10100	100	Andesite	7	0.02	Surface weathering & inferred fault zone III	0.003	0.003
10100 - 10300	200	Andesite	48	1.6	Good orientation - Blocky Andesite	1.0	1.0
10300 - 10350	50	Transition	7	0.02	Weakness zone - rock formation change	0.003	0.003
10350 - 10850	500	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
10850 - 10950	100	Granite	14	0.04	Inferred fault zone IV	0.02	0.02
10950 - 11300	350	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
11300 - 11400	100	Granite	14	0.04	Inferred fault zone V	0.02	0.02
11400 - 12550	1150	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
12550 - 12650	100	Granite	14	0.04	Inferred fault zone VI	0.02	0.02
12650 - 13000	350	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
13000 - 13100	100	Granite	14	0.04	Inferred fault zone VII	0.02	0.02
13100 - 13400	300	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
13400 - 13500	100	Granite	14	0.04	Inferred fault zone VIII	0.02	0.02
13500 - 14150	650	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
14150 - 14250	100	Granite	14	0.04	Inferred fault zone IX	0.02	0.02
14250 - 14750	500	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
14750 - 14850	100	Granite	14	0.04	Inferred fault zone X	0.02	0.02
14850 - 15350	500	Granite	62	10.3	Good orientation - Blocky Granite	6.7	9.3
15350 - 15450	100	Granite	14	0.04	Inferred fault zone XI	0.02	0.02
15450 - 16550	1100	Granite	65	10.3	Good orientation - Blocky Granite	6.7	9.3
16550 - 16650	100	Granite	14	0.04	Inferred fault zone XII	0.02	0.02
16650 - 17400	750	Granite	65	10.3	Fair Orientation - Blocky Granite	6.7	6.7
17400 - 17500	100	Granite	14	0.04	Inferred Fault zone XIII	0.02	0.02
17500 - 18300	800	Granite	62	7.4	Fair Orientation - Blocky Granite	6.7	6.7
18300 - 18700	400	Granite	65	10.3	Good orientation - Blocky Granite	6.7	9.3
18700 - 18800	100	Granite	14	0.04	Surface weathering	0.02	0.02
	8800						

Table 4-11: Final rock mass classification for Vallical tunnel.

Putting together all the different geological stretches found in both tunnels Castillo & Vallical, the rock mass classification based on the Q-system is:

Rock mas Classification Q-system			
From roof description	**Stretch m**	**Q_{ROOF}**	**Q_{WALLS}**
Exceptionally Poor	800	0.003	0.003
Extremely Poor	1100	0.02	0.02
Poor	200	1.0	2.4
Poor	2970	1.9	4.7
Poor	2550	2.6	6.5
Fair	5100	6.7	16.7
Fair	2250	9.3	23.3
Good	1630	23	113
	16600		

Table 4-12: Rock mass classification for both tunnels together.

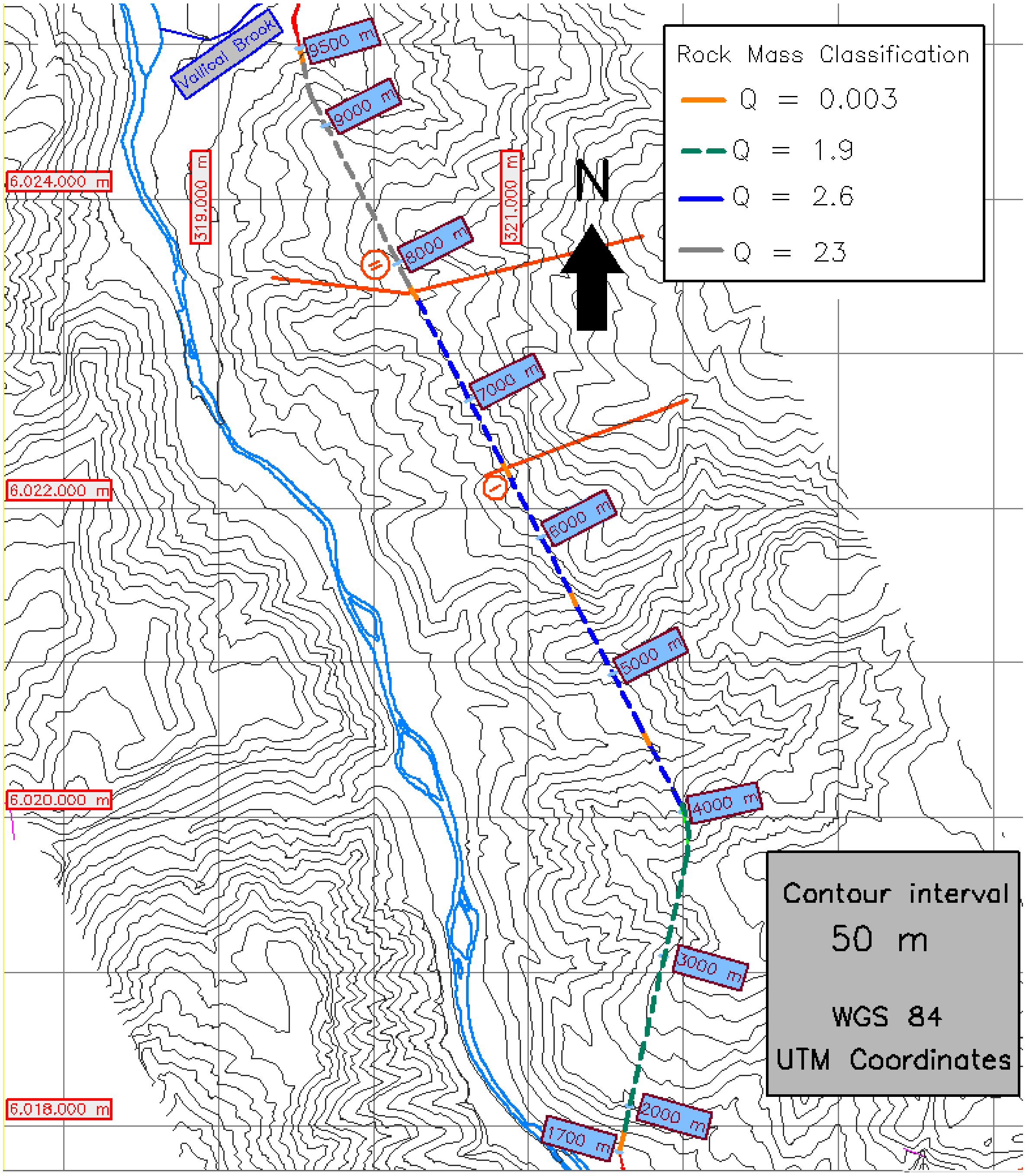


Figure 4-2: Rock mass classification for Castillo tunnel.

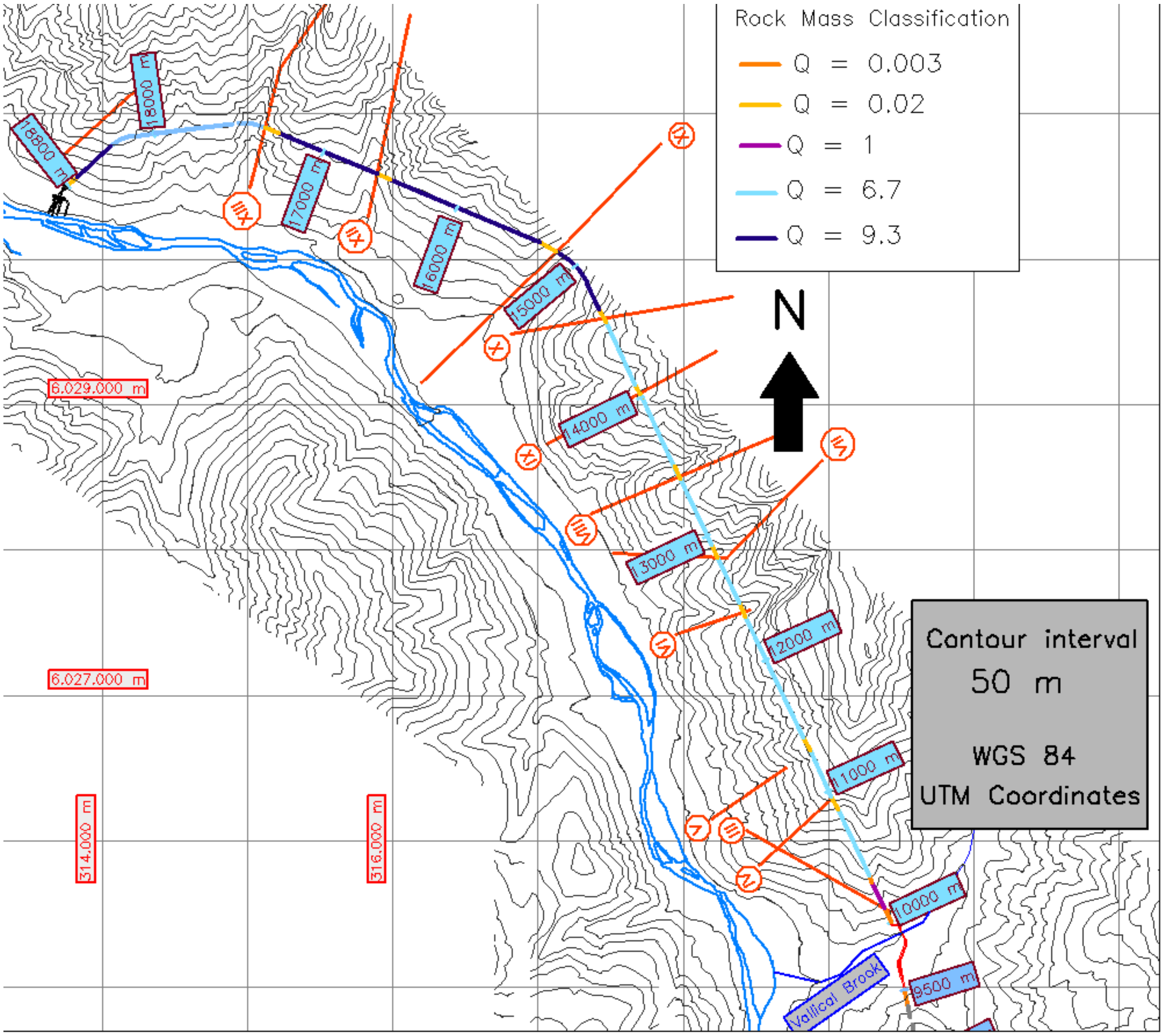


Figure 4-3: Rock mass classification for Vallical tunnel.

5. ROCK MASS

5.1 Ground Behavior

5.1.1 Introduction

As continuous or discontinuous materials behave differently, it is important to determine which type the rock mass in question is. The same type of ground will behave differently for different sizes of opening compared to the size of the blocks. The "continuity factor" CF defined as the ratio between the tunnel diameter D_t and block diameter D_b [8] provides an idea about the rock mass behavior.

CF < 5 means a massive rock where the rock properties dominate.

5 < CF < 100 means that the ground is discontinuous or blocky.

CF > 100 means that the rock is heavily jointed and the material behaves like a soil.

5.1.2 Methodology

Continuity Factor:

As it was said before, the continuity factor is the ratio between the tunnel diameter D_t and block diameter $D_{b.}$ The tunnel diameter obtained from an economic analysis during the feasibility studies is 6.9 m for both tunnels. This value will be kept for rock mass behavior analysis. The procedure to determine the block diameter is shown below:

Block diameter D_b

The equivalent block diameter can be obtained from the block volume V_b:

$$D_b = (V_b)^{1/3}$$

V_b= Blok volume.

In turn, the block volume V_b can be estimated by the Block Shape factor β which quantifies the magnitude difference between the distinct joint set spacing values and the Volumetric Joint Count J_V which is the number of joints in a rock mass volume of 1 m^3.

$$V_b = \frac{\beta}{(J_V)^3}$$

β = It is the block shape factor.

J_V = The volumetric joint count.

The block shape factor varies between 27 and above 1000, being the lowest value for equi-dimensional blocky rocks and the highest values for long and flat rocks.

The volumetric joint set J_V can be estimated in several ways. Below, it will be shown how to estimate it from the Rock Quality Designation RQD and from joint spacing values:

J_V from RQD:

$$J_V = 35 - \frac{RQD}{3.3}$$

J_V from Joint Spacing:

There is a distinction between joint sets alone and joint sets plus random joints.

Joint sets without random joints:

$$J_V = \sum_i^n \frac{1}{S_i}$$

S_i = Spacing related to joint set i.

Joint sets with random joints:

$$J_V = \sum_i^n \frac{1}{S_i} + \frac{N}{5}$$

N = Number of random joints.

5.1.3 Results

The following tables show the adopted values of input parameters and the corresponding results for rock masses in between weakness zones.

Continuity factor CF for Andesite from Joint spacing calculations				
Variable		Magnitude	Unit	Comment
Joint Spacing	Si	0.2	m	100 - 300 mm (Equally spaced)
Volumetric Joint count	Jv	15.2		From 3 joint sets with random joints
Block shape factor	β	30		Equi-dimensional
Block Volume	V_b	0.01	m^3	
Block Diameter	D_b	0.20	m	
Tunnel diameter	Dt	6.9	m	
Continuity factor	CF	**34**		Blocky zone

Continuity factor CF for Andesite from RQD				
Variable		Magnitude	Unit	Comment
Rock Quality Designation	RQD	70		From Core drilling SCHM - 8 at tunnel axis
Volumetric Joint count	Jv	11.7		
Block shape factor	β	30		Equi-dimensional
Block Volume	V_b	0.02	m^3	
Block Diameter	D_b	0.27	m	
Tunnel diameter	Dt	6.9	m	
Continuity factor	CF	**26**		Blocky Zone

Continuity factor CF for Granite from Joint spacing calculations				
Variable		Magnitude	Unit	Comment
Joint Spacing	Si	0.33	m	60 - 600 mm (Equally spaced)
Volumetric Joint count	Jv	9.1		From 3 joint sets
Block shape factor	β	30		Equi-dimensional
Block Volume	V_b	0.04	m^3	
Block Diameter	D_b	0.34	m	
Tunnel diameter	Dt	6.9	m	
Continuity factor	CF	**20**		Blocky zone

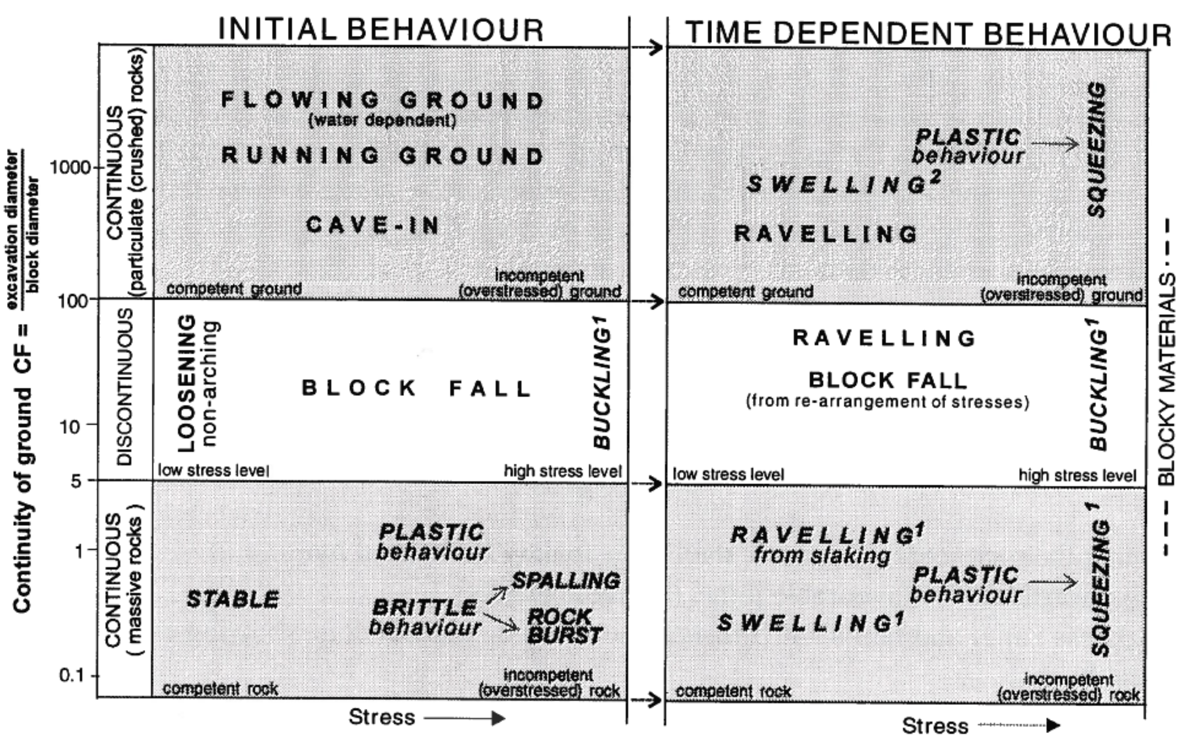
Continuity factor CF for Granite from RQD				
Variable		Magnitude	Unit	Comment
Rock Quality Designation	RQD	80		From Core drilling SCHM - 7 at tunnel axis
Volumetric Joint count	Jv	8.3		
Block shape factor	β	30		Equi-dimensional
Block Volume	V_b	0.05	m^3	
Block Diameter	D_b	0.37	m	
Tunnel diameter	Dt	6.9	m	
Continuity factor	CF	**19**		Blocky Zone

5.1.4 Comments & Discussion

From the last four tables, it is possible to deduct that both rock masses would behave as blocky rocks since their Continuity factor are always between 5 and 100.

It is important to point out that the Continuity Factor of "Granitic Intrusive" rock formation (represented by granite), obtained from two different methods, is lower than the Trapa-Trapa rock formation (represented by Andesite), meaning that the first one is less jointed than the second one.

The next chart shows the main types of instability problem once it is known the Continuity Factor CF and the stress situation:



Based on the calculations carried out in this chapter and taking into account the results in chapter 3.10 (related to in situ stress) where no high stress levels are expected in any of the tunnels, one can conclude that block fall is the most relevant instability problem in between weakness zones.

It is clear that the corresponding CF value for very poor rock mass conditions will be above 100 due to crushed rock. In the case of fault zones located in the Trapa-Trapa rock formation, problems with flowing ground could be possible.

5.2 Rock Mass Properties

The reduction in rock mass strength due to discontinuities must be quantified in order to have a good understanding of the rock mass behavior. The Geological Strength Index GSI is a useful system to know the reduction in rock mass strength and several important parameters related to the rock mass depend on it.

5.2.1 GSI

The degree of jointing and the surface condition of the discontinuities are the two input parameters for this system. The following chart shows how they are rating:

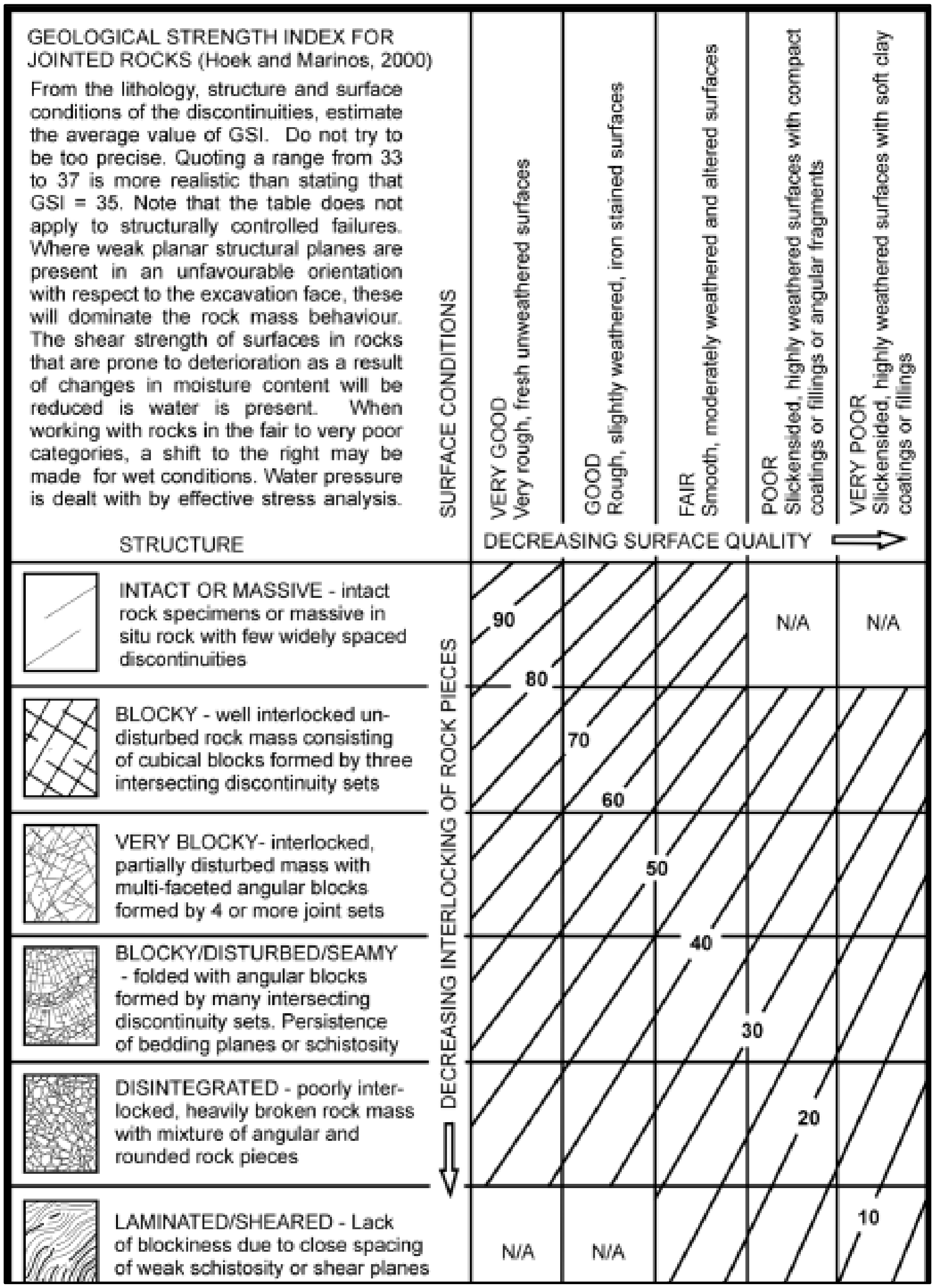


Figure 5-1: GSI Characterization of rock masses: Hoek & Marinos, 2000

Based on the description given in chapter 3, it is possible to classify the different rock masses involved in the project. With regard to the interlocking classification in GSI, it is possible to say that both Andesite (*Trapa-Trapa* rock formation) and Granite (Batholith *Melado*) in between weakness/fault zones are "blocky". Related to surface condition one could make a difference, rating Granite as "Good" (rough joint surface, slightly altered) and Andesite as "Fair" (Smooth joint walls and altered surface with clay).

In weakness/fault zones, it is possible to say that both rock types could be classified as “disintegrated”. Finally, with regard to surface conditions in weakness zones, one could make a difference rating the Batholith Melado (granite) weakness zone as “Poor”(hard infilling) and Trapa-Trapa rock formation as “very poor” (clay coating).

Along with this qualitative classification of interlocking and surface conditions for each rock mass that gives a fixed rectangle in the GSI system, one can combine it with some empirical relations between the GSI system and information from the rock mass classification:

GSI = RMR – 5   RMR > 23   (Biewniawski, 1989)

GSI = 9*Ln(Q’) + 44   (Lien and Lunde)

Q’ is the Q value with SRF and J_W equal to 1.

"Blocky" and "Fair" Andesite (Trapa Trapa rock formation)			
Variable		Magnitude	Comment
From Qualitative description in GSI system	GSI	45 - 65	Range for "Blocky" interlocking and "Fair" surface conditions
From value	Q'	1.9	Q-system with SRF = 1 & Jw = 1
GSI value from Q-System (SRF = 1 & Jw = 1)	GSI	49.7	
Rock mass rating	RMR	55.0	(Fair orientation)
GSI value from RMR system	GSI	50.0	
GSI adopted for "Blocky" and "Fair" Andesite	GSI	**50**	

"Blocky" and "Good" Granite (Batholith Melado rock formation)			
Variable		Magnitude	Comment
From Qualitative description in GSI system	GSI	55 - 75	Range for "Blocky" interlocking and "Good" surface conditions
From value	Q'	6.7	Q-system with SRF = 1 & Jw = 1
GSI value from Q-System (SRF = 1 & Jw = 1)	GSI	61.1	
Rock mass rating	RMR	62.0	(Fair orientation)
GSI value from RMR system	GSI	57.0	
GSI adopted for "Blocky" and "Good" Granite	GSI	**60**	

"Desintegrated" and "Very Poor" Andesite (Trapa Trapa rock formation)			
Variable		Magnitude	Comment
From Qualitative description in GSI system	GSI	5 - 20	Range for "Desintegrated" interlocking and "Very Poor" surface conditions
From value	Q'	0.08	Q-system with SRF = 1 & Jw = 1
GSI value from Q-System (SRF = 1 & Jw = 1)	GSI	21.6	
GSI adopted for "Desintegrated" and "Very poor" Andesite	GSI	**20**	

"Desintegrated" and "Poor" Granite (Batholith Melado)			
Variable		Magnitude	Comment
From Qualitative description in GSI system	GSI	15 - 28	Range for "Desintegrated" interlocking and "Poor" surface conditions
From value	Q'	0.38	Q-system with SRF = 1 & Jw = 1
GSI value from Q-System (SRF = 1 & Jw = 1)	GSI	35.2	
GSI adopted for "Desintegrated" and "Poor" Granite	GSI	**30**	

Summary of GSI Values adopted		
GSI adopted for "Blocky" and "Fair" Andesite	GSI	**50**
GSI adopted for "Blocky" and "Good" Granite	GSI	**60**
GSI adopted for "Desintegrated" and "Very poor" Andesite	GSI	**20**
GSI adopted for "Desintegrated" and "Poor" Granite	GSI	**30**

5.2.2 Modulus of Deformation

This parameter is very relevant for the mechanical behavior of rock mass. There is no in situ measured in the field with regard to deformation modulus. Nevertheless, some indirect methods based on statistical analysis and rock mass classification systems will be considered to quantify this parameter.

$\mathrm{E_m} = 2 * \mathrm{RMR} - 100$ (RMR > 50) Bieniawski, 1978

$\mathrm{E_m} = 10^{(\mathrm{RMR}-10)/40}$ (RMR < 50) Serafim & Pereira, 1983

$E_m = 25 * LOG_{10}(Q)$ (Q > 1) Grimsta & Barton, 1993

$E_m = \sqrt{\frac{\sigma_c}{100}} * 10^{(GSI-10)/40}$ $(\sigma_c \leq 100\ MPA)$ Hoek & Brown, 1998

$E_m = \frac{E_{ci}}{60} * \sqrt{\sigma_{ci}}$ Phanti, 2006

Note that not always an equation will be used in order to respect the limited range of applicability.

The results for each rock type and rock mass conditions are given in the following 4 tables:

"Blocky" Trapa-Trapa rock formation (Andesite)				
Variable		Magnitude	Unit	Comment
Elasticity modulus	Ei	31	GPa	
Uniaxial compressive stregth	σ_{ci}	100	MPA	
Deformation Modulus (From σ_{ci})	Em	5.2	GPa	Panthi (Statistical)
Rock mass classification	Q	1.9		Only Q_{ROOF} considered
Deformation Modulus (From Q)	Em	6.8	GPa	Q > 1
Rock mass rating	RMR	55		Fair orientation
Deformation Modulus (From RMR)	Em	10	GPa	RMR > 50 equation
Geological Strength Index	GSI	50		"Blocky" & "Fair"
Deformation modulus (from σc ; GSI)	Em	10	GPa	σc < = 100 MPA
Value adopted for Blocky Andesite	Em	**8.00**	GPa	Simple average (4)

Note that when the deformation modulus was obtained from the Q-system, the input value considered belongs to the roof Q-value only. When the RMR system was used, only fair orientation was considered, representing both fair and favorable discontinuity orientation. These assumptions are also valid for the following three tables.

"Blocky" Batholith Melado rock formation (Granite)				
Variable		Magnitude	Unit	Comment
Elasticity modulus	Ei	48	GPa	
Uniaxial compressive stregth	σ_{ci}	150	MPA	
Deformation Modulus (From σ_{ci})	Em	9.8	GPa	Panthi (Statistical)
Rock mass classification	Q	6.7		Only Q_{ROOF} considered
Deformation Modulus (From Q)	Em	20.6	GPa	Q > 1
Rock mass rating	RMR	62		Fair orientation
Deformation Modulus (From RMR)	Em	24	GPa	RMR > 50 equation
Value adopted for Blocky Granite	Em	**18.1**	GPa	Simple average (3)

Weakness/fault zone Trapa-Trapa rock formation (Andesite)				
Variable		Magnitude	Unit	Comment
Rock mass rating RMR	RMR	7		
Deformation Modulus (From RMR)	Em	0.8	GPa	RMR < 50 equation
Geological Strength Index	GSI	20		"Desintegrated" & "Very Poor"
Deformation modulus (from σc ; GSI)	Em	2	GPa	σc < = 100 MPA
Value adopted for Weakness zone Andesite	Em	**1.3**	GPa	Simple average (2)

Weakness/fault zone Batholith Melado rock formation (Granite)				
Variable		Magnitude	Unit	Comment
Rock mass rating RMR	RMR	14		
Deformation Modulus (From RMR)	Em	1.3	GPa	RMR < 50 equation
Value adopted for Weakness zone Granite	Em	**1.3**	GPa	

Summary: Deformation Modulus for each rock type			
Value adopted for Blocky Andesite	Em	**8.00**	GPa
Value adopted for Blocky Granite	Em	**18.1**	GPa
Value adopted for Weakness zone Andesite	Em	**1.3**	GPa
Value adopted for Weakness zone Granite	Em	**1.3**	GPa

5.2.3 Shear Strength of jointed rock mass

The strength of a jointed rock mass is a key parameter to predict the rock mass strength, especially in blocky or jointed rocks. Below, it will be shown the Hoek-Brown failure criterion along with all the rock mass parameters that it requires.

5.2.3.1 Generalized Hoek-Brown failure criterion

This failure criterion is:

$$\sigma_1 = \sigma_3 + \sigma_c * \left(m_b * \frac{\sigma_3}{\sigma_c} + s\right)^a$$

σ_1 = Axial effective principal stress.

σ_3 = Confining effective principle stress.

σ_c = Uniaxial compressive strength of the rock mass.

S = Empirical constant analogous to the “cohesive strength” of the rock mass.

m_b = Empirical constant analogous to the “friction angle” of the rock mass.

a = Empirical constant related to surface of the jointed rock and interlocking.

The three empirical parameters can be obtained from the GSI system, except the parameter m_b that also requires a constant m_i which is related to the intact rock. The equations are given below:

$m_b = m_i * \exp\left(\frac{GSI-100}{28}\right)$

$s = \exp\left(\frac{GSI - 100}{9}\right)$

a = 0.65 – $\frac{GSI}{200}$

These three parameters are for the estimation of the peak strength parameters of jointed rock masses.

Finally, it is also necessary to know the residual Hoek-Brown parameters for the plastic behaviour of the rock mass.

$m_r = 0.65 * m_b$ Ribacchi, 2000)

$s_r = 0.04 * s$ (Ribacchi, 2000)

An analysis about the adopted values for m_i will be undertaken first and then the results of the three empirical parameters for each rock type will be given:

The following table obtained from reference [5] shows the m_i values for different igneous rocks, among them Granite and Andesite.

Rock type	Class	Group	Texture			
			Course	Medium	Fine	Very fine
IGNEOUS	Light		Granite 33		Rhyolite (16)	Obsidian (19)
			Granodiorite (30)		Dacite (17)	
	Dark		Diorite (28)		Andesite 19	
			Gabbro 27	Dolerite (19)	Basalt (17)	
			Norite 22			
	Extrusive pyroclastic type		Agglomerate (20)	Breccia (18)	Tuff (15)	

Hoek, 1983

And m_i value of 33 is assigned for granite and a m_i value of 19 is assigned for Andesite. In the case of granite, the value suggested is consistent with the value given by “Rock lab” data, (which comes along “Phase 2” commercial software). This software suggests for granite a m_i equal to 32 ±3. In the case of Andesite, the same software suggests a value of

25 ± 5 which is a little higher than the value suggested from the last table. The values from the last table will be adopted for a conservative decision.

Andesite m_i = 19

Granite m_i = 33

"Blocky" Andesite Hoek-Brown Parameters	
GSI	50
m_i	19
m_b	3.2
s	0.0039
a	0.4
m_r	2.07
s_r	0.00015

"Blocky" Granite Hoek-Brown Parameters	
GSI	60
m_i	33
m_b	7.9
s	0.0117
a	0.35
m_r	5.14
s_r	0.00047

Weakness/fault zone Andesite	
GSI	20
m_i	19
m_b	1.1
s	0.0001
a	0.55
m_r	0.71
s_r	0.00001

Weakness/fault zone Granite Hoek-Brown Parameters	
GSI	30
m_i	33
m_b	2.7
s	0.0004
a	0.5
m_r	1.76
s_r	0.00002

6. ROCK SUPPORT

6.1 Introduction

Rock support is provided to improve the stability in an underground opening. Rock support is determined by rock mass quality and external geological conditions but also by technology available and local past experience. The following description of different rock supports will try to include the most common supports used nowadays.

Scaling:

This is the process of taking out loose material in the tunnel contour immediately after blasting (and ventilation). This method is placed in this section because blocks can be scaled down instead of being held by rock support. This procedure can be done manually over the muck pile for small tunnels, but for the range of tunnel cross sections under analysis it is recommended mechanized scaling from Jumbo. The latter alternative is also safer for the crew during the drill and blast process.

Rock Bolting:

It transfers load from the unstable zone near the tunnel contour to the confined interior of the rock mass.

The most common bolts used for rock support are described below:

At the working face:

Mechanical anchor

The bolt is anchored by an expansion of it at the far end of the tunnel contour. On this end, the bolt has two wedge shaped steel blades and a conical nut so that tightening the bolt triggers an expansion against the drillhole wall.

Resin Anchor (Polyester cartridge)

After the bolthole is done, two polyester cartridges are inserted to it up to the bottom. Thereafter, a rebar is put by rotating it which creates the mix and chemical reaction between the resin and the catalyst. Once it sets, the bolts are tensioned. It is possible to grout the rest of the hole against corrosion.

Behind the working face:

Untensioned grouted bolts (Dowels)

After the bolthole is done, the grouting is placed, filling up the hole before the bolt is inserted. This causes that bolts are surrounded by grout and therefore well protected against corrosion. Unlike the other bolt described, these are passive reinforcement elements that require some ground displacement to be activated.

<u>Shotcrete:</u>

This type of rock support is obtained by spraying concrete on the rock surface. It is usually combined with wire mesh (Mesh reinforced shotcrete) or steel fibers (Fibercrete which in both cases is to obtain a higher compressive strength and a higher ductility behavior (More energy absorption).

In areas of weakness zones heavy rock support will be considered as:

<u>Lattice Girder:</u>

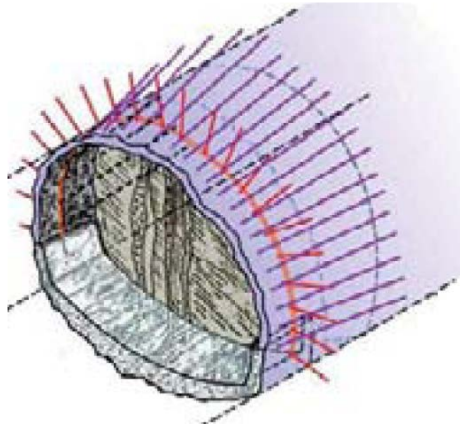
As defined by the US Federal Highway Administration, Lattice Girder is a support made up of steel reinforcement bars laced together usually in a triangular pattern and rolled to match the shape of the opening. Lattice girder embedded in shotcrete can be an alternative to the concrete lining for extremely poor rock conditions.

<u>Steel Arch:</u>

It is formed by constructing rings made of rolled steel beams (There are different shapes) perpendicular to the tunnel axis. Steel ribs are often installed with shotcrete for the blocking (lagging) material.

<u>Shotcrete Ribs</u>:

Shotcrete reinforced ribs consist of steel rebar arches (Commonly 6 rebars of 16-20 mm diameter each one) anchored to the rock mass with radial bolts and 3 layers of shotcrete. As it is said in reference [8], the first layer of shotcrete, commonly between 100 to 150 mm, is put with steel fibers and before the rebars. The following two layers of unreinforced shotcrete which in total have a thickness of 150-300 mm are put after the rebars in order to cover them.



Pre-injection:

This method aims to improve stability by means of grouting ahead of the tunnel face before blasting. It suits for poor rock conditions and high water pressure.

Concrete Lining:

It is a full lining of the tunnel with concrete, establishing an arch capable to take considerable load. Reinforcement, anchorage and possibly concrete invert should be eventually included depending on the case. Cast in place concrete lining is implemented while excavating a tunnel through long weakness zones.

From the literature, there are documented experiences about some of the rock support described above and the convenience of one over another.

Related to Shotcrete:

- As it is said in reference [8] steel fiber shotcrete compared to wire mesh is being more and more common worldwide since it minimizes the labor intensive process of mesh installation.

- Wet mix shotcrete (Water is added before delivery into a positive displacement) has shown some advantages compared to dry mix method (Water is added to the mix at the nozzle) like higher production capacity, reducing rebound, improving work conditions, etc.

Related to heavy rock support:

There is consistency in the literature that lattice girders are easier and faster to install than steel arches. Lattice girder is considered [16] as an improvement of the steel arch being easier to manufacture, transport, storage mainly because it is lighter and more flexible.

On the other Hand, reference based on experience in Chile, [15] describes several advantages of shotcrete reinforced ribs over lattice girder like quicker to install, easier to repair (Add more shotcrete or add more steel rebars is not difficult in case it is required), etc. The only hindrance visualized in the installation of the Shotcrete reinforced ribs in this project is the likely lack of crew's experience that will affect the efficiency since it is not a well known solution in Chile for weakness zones.

Cast in place concrete lining will not be considered for the *Melado* Project because it is not expected long stretches of very poor rock conditions since all the identified weakness zones are almost perpendicular to the tunnel axis.

Pre-injection should be considered for the two weakness zones identified in the Trapa-Trapa rock formation (Andesite/Basalt/Breccia) since it is not uncommon to have significant water leakage through weakness zones located in these rock types.

All of these experience will be considered for the final assignment of rock support in each tunnel stretch.

6.2 Empirical Method

This is a statistical approach for assessing the stability of tunnels. The two best known rock support systems worldwide are covered here.

6.2.1 RMR

The empirical support estimation based on RMR system is given in the following table:

Shape: horseshoe; Width: 10 m; Vertical stress: below 25 MPa; Excavation by drill & blast

Rock	Excavation	Support		
Mass class		*Rock bolts* (20 mm diam., fully bonded)	*Shotcrete*	*Steel sets*
1. Very good rock RMR: 81-100	*Full face:* 3 m advance	Generally no support required except for occasional spot bolting		
2. Good rock RMR: 61-80	*Full face:* 1.0-1.5 m advance; Complete support 20 m from face	Locally bolts in crown, 3 m long, spaced 2.5 m with occasional wire mesh	50 mm in crown where required	None
3. Fair rock RMR: 41-60	*Top heading and bench:* 1.5-3 m advance in top heading; Commence support after each blast; Commence support 10 m from face	Systematic bolts 4 m long, spaced 1.5-2 m in crown and walls with wire mesh in crown	50-100 mm in crown, and 30 mm in sides	None
4. Poor rock RMR: 21-40	*Top heading and bench:* 1.0-1.5 m advance in top heading; Install support concurrently with excavation - 10 m from face	Systematic bolts 4-5 m long, spaced 1-1.5 m in crown and walls with wire mesh	100-150 mm in crown and 100 mm in sides	Light ribs spaced 1.5 m where required
5. Very poor rock RMR < 21	*Multiple drifts:* 0.5-1.5 m advance in top heading; Install support concurrently with excavation; shotcrete as soon as possible after blasting	Systematic bolts 5-6 m long, spaced 1-1.5 m in crown and walls with wire mesh. Bolt invert	150-200 mm in crown, 150 mm in sides, and 50 mm on face	Medium to heavy ribs spaced 0.75 m with steel lagging and forepoling if required. Close invert

6.2.2 Q-System

The Q-system determines the temporary and permanent rock support required by two inputs value, the Q-value obtained from the rock classification system and the "Equivalent dimension" D_e which weighs the purpose of the tunnel and the tunnel span.

$$D_e = \frac{D_t}{ESR}$$

D_t = Diameter of the underground excavation.

ESR = Excavation support ratio = 1.6 for water tunnels.

The Q-System chart (was provided in Figure 4.1)

6.2.3 Results:

The estimated rock support RMR does not provide in detail the suitable rock support as Q does like rock bolt spacing, shotcrete rib spacing, shotcrete thickness, etc. Furthermore, the Q system is sensitive to the span (tunnel diameter) and the type of tunnel purpose (included in the Excavation Support Ratio ESR factor) which makes it a more valuable tool when one is analyzing different alternatives.

The Q System is chosen to select the support required to avoid instability problems, but RMR is not totally neglected, because it was indirectly included in the rock mass classification system by means of the correction factor of discontinuity orientation.

Using the Q-system chart, the rock support required for both tunnels Castillo and Vallical is given below:

Final rock support assigened by Q-system 6.9 m tunnel diameter (Both tunnels)						
Rock Class (From Q_{ROOF})	**Stretch m**	**Q_{ROOF}**	**Reinforcement Category in roof**	**Q_{WALLS}**	**Reinforcement Category in walls**	**Rock Support Summary: Roof & Walls**
Exceptionally Poor	800	0.003	8	0.003	8	Reinforced Shotcrete ribs D40/4 c/c 1.2 m ; systematic bolting (Roof & Walls)
Extremely Poor	1100	0.02	8	0.02	8	Reinforced Shotcrete ribs E35/5 c/c 2.3 m ; systematic bolting (Roof & Walls)
Poor	200	1.0	5	2.4	4	7 cm Fibercrete in roof; 7 cm shotcrete in walls; systematic bolting (Roof & Walls)
Poor	2970	1.9	4	4.7	3	7 cm shotcrete in roof; Systematic bolting (Roof & Walls)
Poor	2550	2.6	4	6.5	1	7 cm shotcrete and bolts (both in roof only)
Fair	5100	6.7	1	16.7	1	Unsuppurted
Fair	2250	9.3	1	23.3	1	Unsuppurted
Good	1630	23	1	113	1	Unsuppurted
	16,600					

With regard to the weakness zones (the lowest two Q-values in the last chart) it is important to point out that:

D40/4 c/c 1.2 = 4 rebars in double layer in 40 cm thick lining with c/c spacing 1.2 m.

D35/5 c/c 2.3 = 5 rebars in double layer in 35 cm thick lining with c/c spacing 2.3 m.

Also, the Q-system chart specifies the radial bolt spacing input parameter depending on the Q-system value. All the relevant Q-system values for both Castillo and Vallical tunnels with the corresponding bolt spacing is summarized in the following chart:

Q_{ROOF}	Radial Bolt spacing in shotcreted area m
0.003	1.0
0.02	1.2
1.0	1.7
1.9	2.0
2.6	2.0
6.7	2.2
9.3	2.3
23	2.4

6.2.4 Complementary Information

Even though, the Q-system provides important information about the rock support to be assigned for instability problems, there are some details that must be complemented.

Radial bolt length:

In Norwegian tunnels the following expression is often used to find the bolt length

$$Lb = 1.4 + 0.184*D_t \qquad (m)$$

Dt = Tunnel diameter (m).

Lb (6.9 m tunnel diameter) = 2.67 m ≈ 3 m.

Rebar diameter:

When in the Q-system a certain stretch of a tunnel is in reinforcement category 8, it provides the number of rebars, but not the diameter of each one. Reference [8] suggests that rebars has a typical dimension of 20 mm diameter. This value is adopted for subsequent analysis.

Radial bolt diameter:

It will depend on the specific requirement in the tunnel and the type of bolt to be considered, but if a tunnel requires bolts with 20 tones ultimate tensile strength, the bolt should be around 25 mm diameter.

Concrete invert:

In general, invert in Norwegian water tunnels are not frequently concreted and only rough cleaning is carried out once the tunnel excavation has finished. Meanwhile, in Chile there is a certain tradition to have a concrete invert. A key difference between these countries is the cross section area in each place, where for several reasons (interest rate, tunnel total cost, energy price, etc), at a same discharge, in general tunnels in Chile are smaller and hence flow velocities higher (Flow velocity in tunnels above 2 m/s at design discharge is very often). The latter aspect can cause potential invert erosion problems in case it is left unlined and therefore invert is commonly concreted. This Chilean tradition has been kept in this thesis.

6.3 Analytical Method: Numerical Modeling

6.3.1 Introducion

The rock mass behavior, the stress situation and eventual tunnel inward deformation are handled more precisely in a numerical model analysis.

Phase 2 Software:

It is a 2D elasto-plastic finite element program for calculating stresses and displacements around underground openings.

Phase 2 uses a plane strain analysis where two principle in-situ stresses are in the plane of the excavation and no shear stresses and strains are out of plane.

Even though, it is ideally for continuous rock deformation analysis, the input parameters belong to the rock mass in order to quantify it fairly.

Two cases will be analyzed in order to identify eventually some problems that are not being visualized by the empirical methods.

6.3.2 Theory:

Some parameters must be explained before the cases are developed.

Dilation: It is a measure of the increase in volume of the material when sheared. When Hoek-Brown parameters are chosen, this input parameter varies between 0.33 m_b for soft rock to 0.66 m_b for hard rock.

Disturbance factor: It is related to the blast damage. It varies between 0 and 1, being the lowest value for an excavation carried out by TBM, and 1 for a very poor blast design tunneling.

6.3.3 Case 1: Blocky Andesite – Chainage 3500 m

6.3.3.1 Elastic analysis:

Elastic analysis is carried out first to check if rock support is required.

6.3.3.1.1 Input parameters:

The field stress considered here is the same as included in chapter 3.10. The way to put these values in Phase 2 software is shown in the next figure:

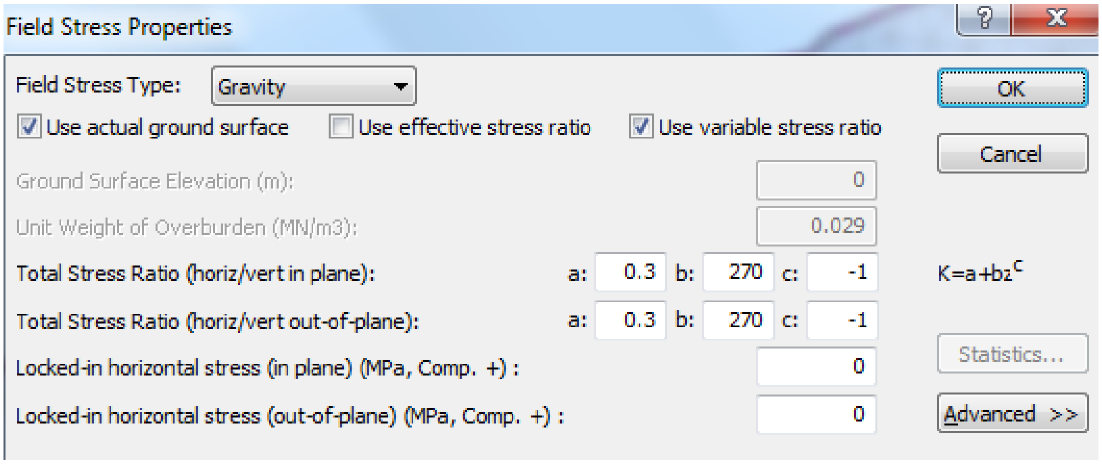


Figure 6-1: Field stress input parameters inserted in Phase 2.

The properties of the blocky Andesite were obtained in chapter 5.2. The way to include these parameters in Phase 2 is shown in the next figure:

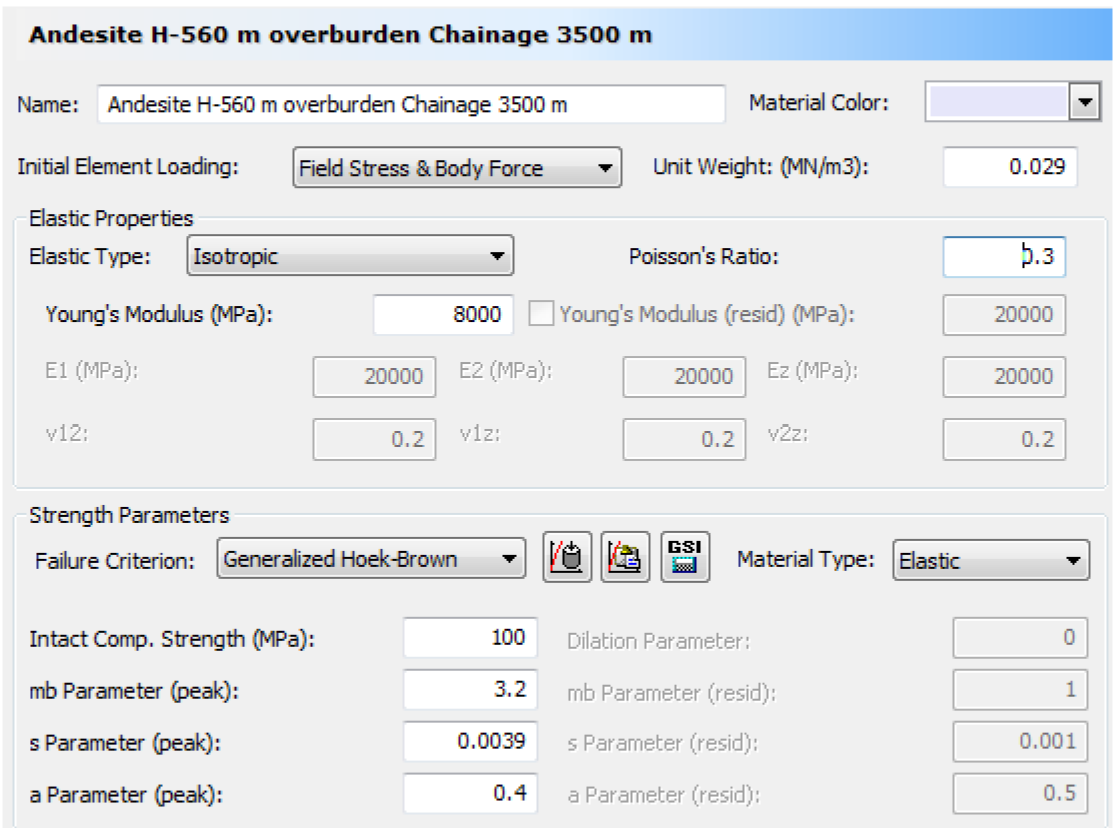


Figure 6-2: Rock mass properties for blocky Andesite inserted in Phase 2.

6.3.3.1.2 Results:

The following picture shows the virgin stress situation before underground excavation is carried out:

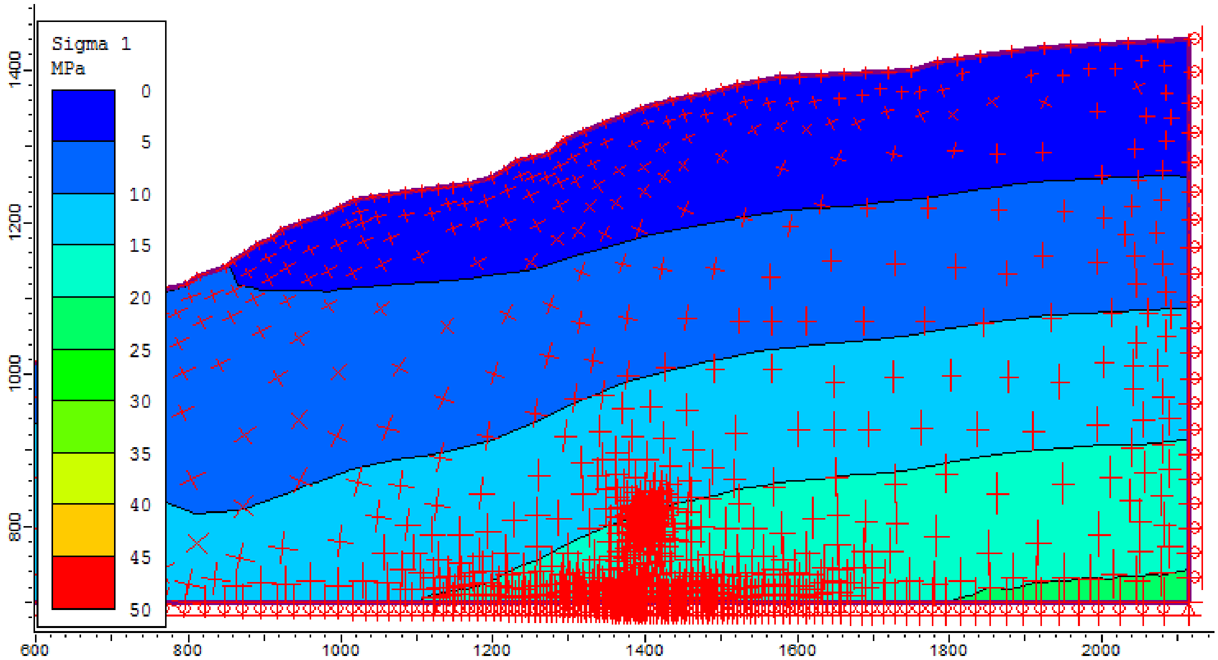


Figure 6-3: In situ stress situation before excavation

It is visualized from the previous illustration that topographic effect is not that significant at 560 m of overburden measured from the tunnel roof, because the major principal stress is almost vertical and the minor principal stress is almost horizontal. The next picture shows in detail the stress situation before and after excavation.

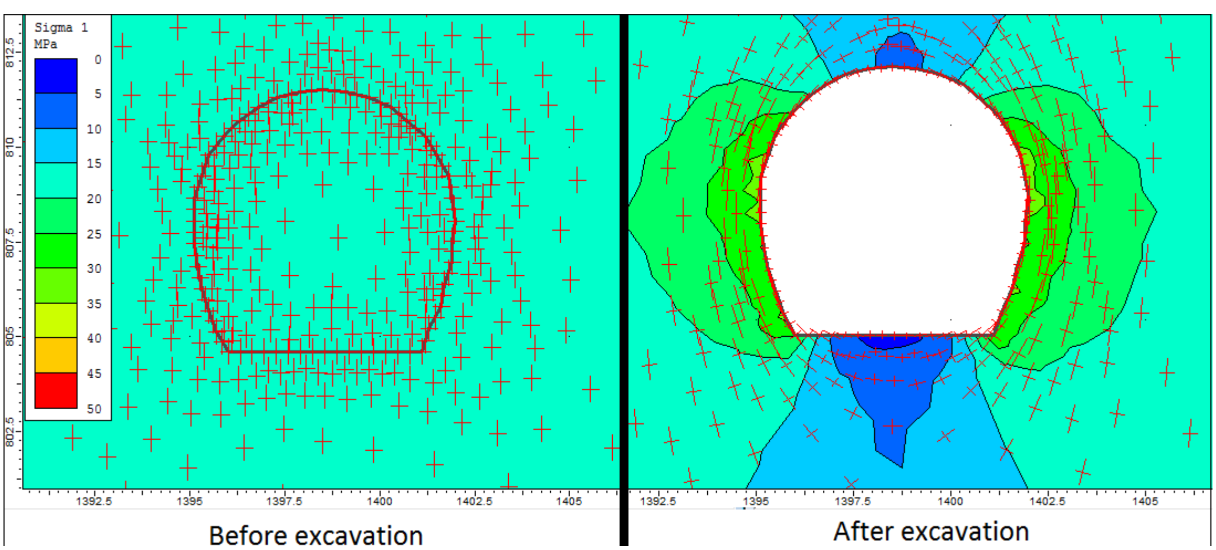


Figure 6-4: Change in stress situation due to excavation

Before excavation	After excavation
σ_1 = 15.3 MPA	$\sigma_{\theta MAX}$ = 33 MPA
σ_3 = 7.4 MPA	$\sigma_{\theta MIN}$ = 4 MPA

The maximum tangential stress takes place on the excavation boundary parallel to the major principal stress σ_1. That is why the higher stresses after excavation occur on both side walls.

Finally it is shown the strength factor in order to visualize if rock support is required.

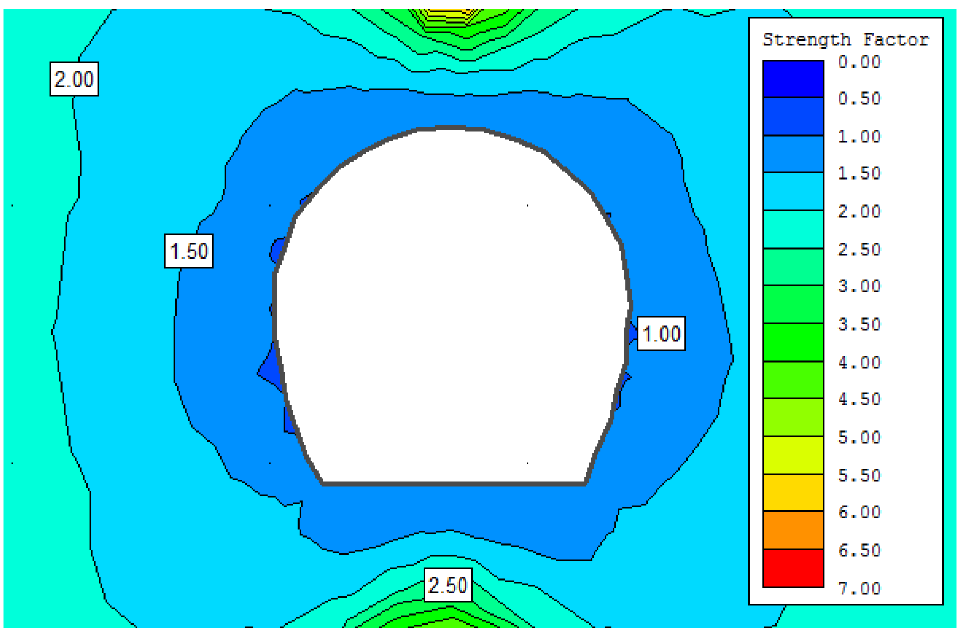


Figure 6-5: Strength factor for blocky Andesite in elastic anaysis.

As it can be seen from the last figure, there are some areas next to the walls where the strength factor is below one.

According to the Q-system, this tunnel should have systematic bolting in walls and roof and unreinforced shotcrete in roof. Below it will be checked if this rock support is enough to improve the tunnel stability. Also, a concrete slab in invert will be included.

6.3.3.2 Plastic Analysis:

Compared to the elastic analysis, some new input parameters are required like residual values for Hoek-Brown failure criterion and the input parameters of rock support:

6.3.3.2.1 Input parameters:

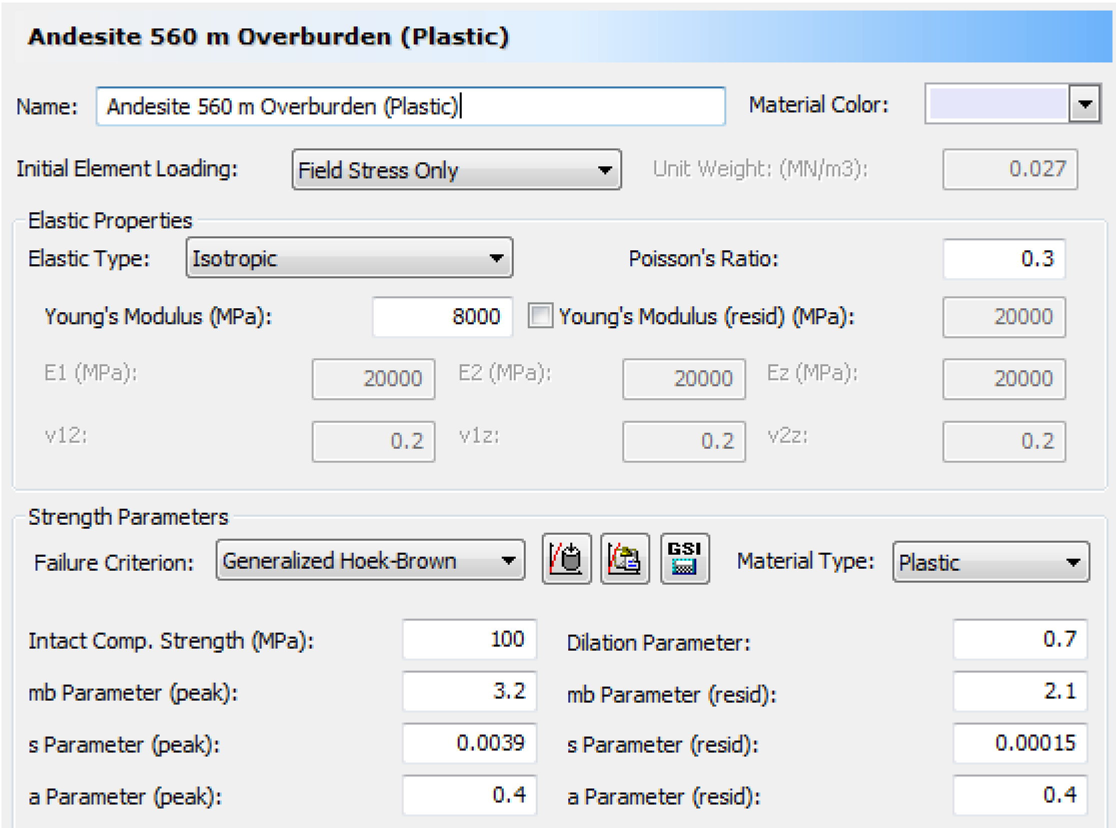


Figure 6-6: Rock mass parameters for blocky Andesite in plastic analysis.

Unreinforced shotcrete in roof:

The key input parameters for unreinforced shotcrete are given below:

Deformation Modulus: 30,000 MPA

Thickness: 7 cm.

Compressive strength: 25 MPA.

The way to include these parameters in software Phase 2 along with some others that are necessary is given below:

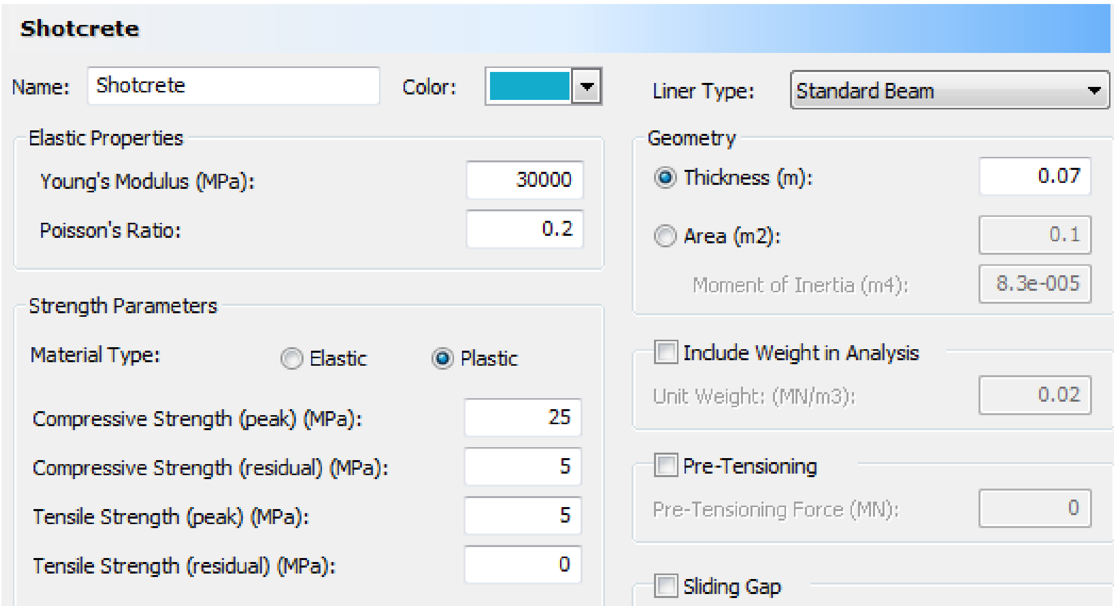


Figure 6-7: Unreinforced shotcrete input parameters:

<u>Radial Bolts:</u>

The key inputs for radial bolts from the Q-system and other complementary sources are summarized below:

Bolt length = 3 m.

Bolt diameter = 25 mm. (End anchored rock bolt was chosen with 20 tones of maximum strength and 25 % of pre-tensioning).

Square pattern installation: 2 m x 2 m.

The way to include these parameters in software Phase 2 is shown in the following two figures:

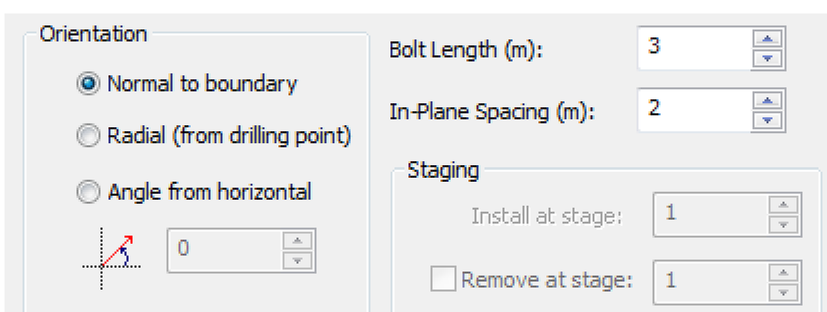


Figure 6-8: Radial bolt input parameters:

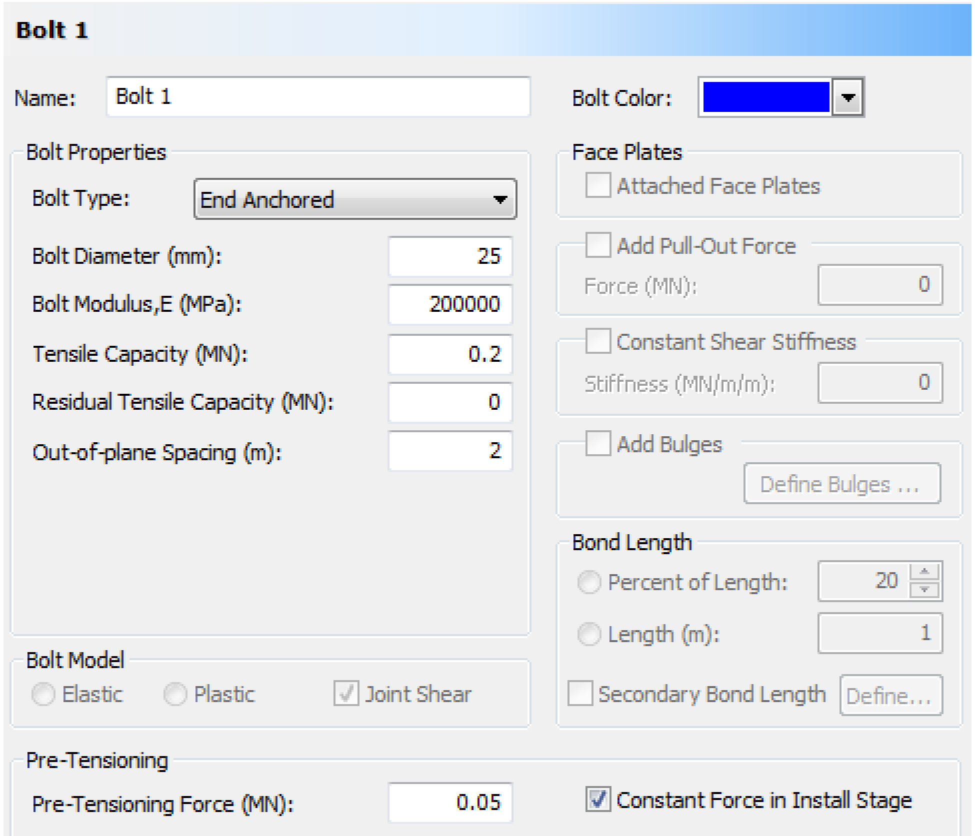
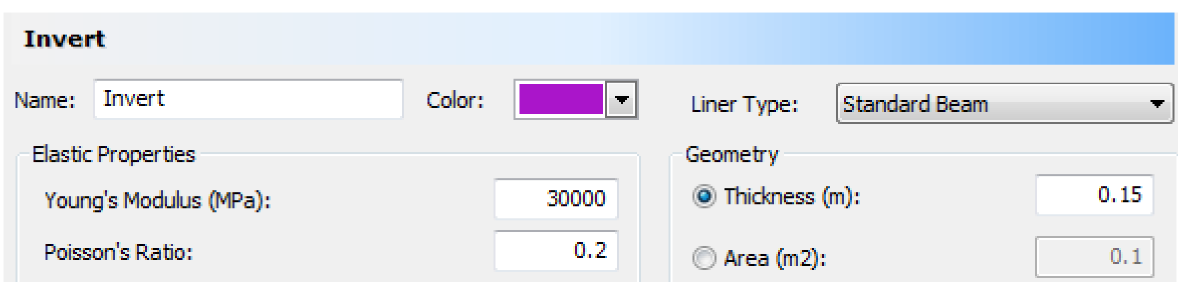


Figure 6-9: Radial bolt input parameters

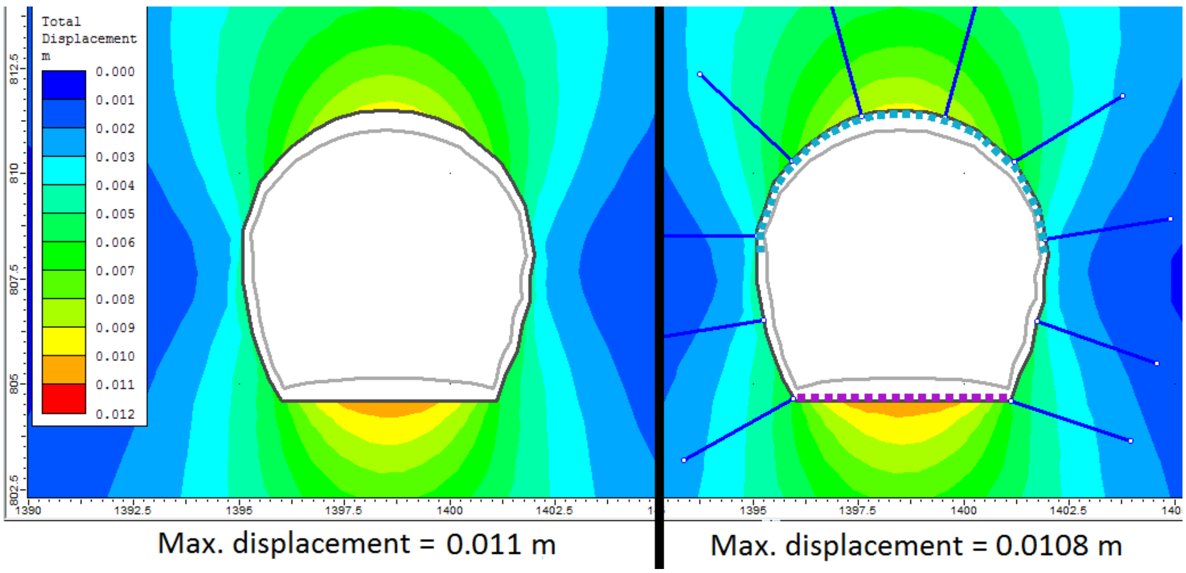
6.3.3.2.2 Concrete Invert:

As it was said before, concrete invert is included only because of Chilean tradition. A thickness of 15 cm is considered.



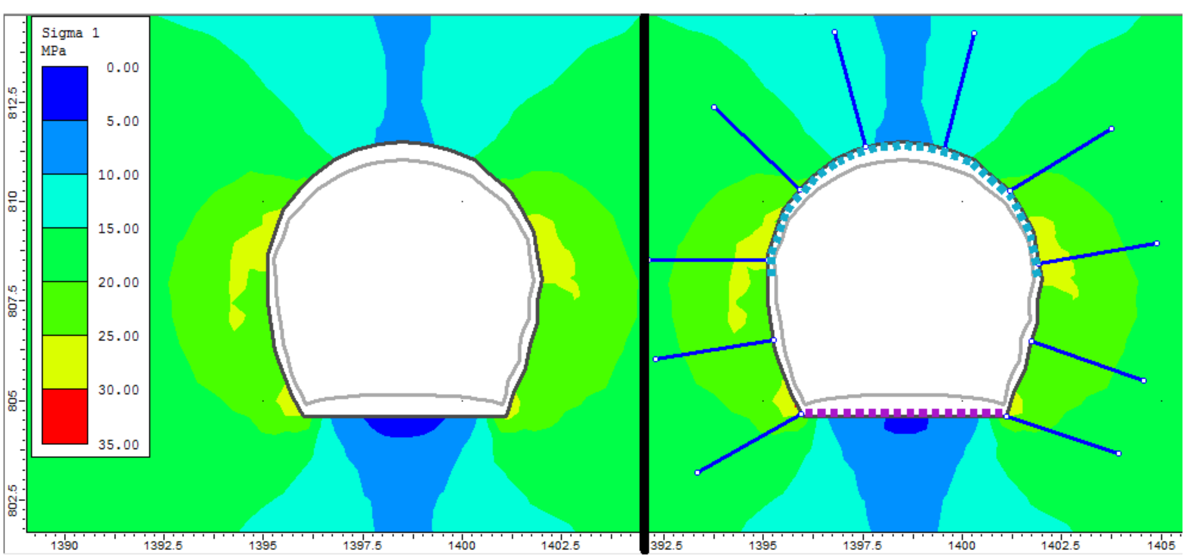
6.3.3.2.3 Results:

Total displacement:



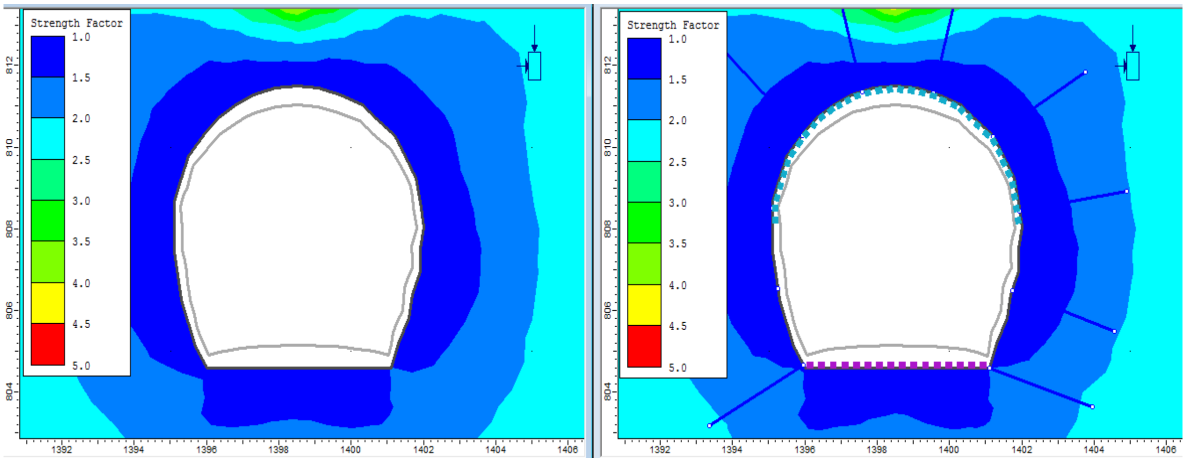
There is almost no difference in displacement between with and without rock support. In both cases the maximum displacement occurs at the middle of invert with a similar magnitude.

Sigma 1:



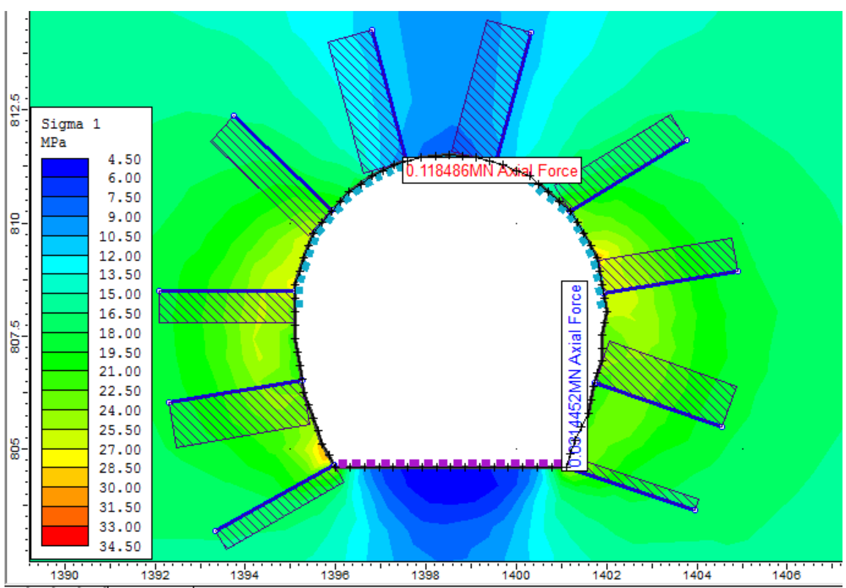
There is a slight difference near the invert caused by the concrete slab between without (left) and with support which increased the minimum tangential stress at the bottom. For the rest of tunnel contour there is no difference.

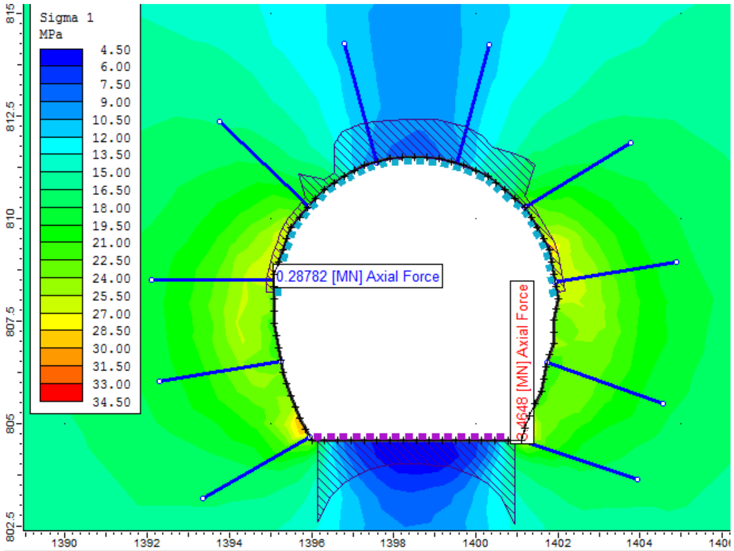
Strength factor:



There is no visible difference between with and without rock support.

Finally it is shown that not yielded support was found neither in bolts nor in shotcrete.





6.3.3.2.4 Comment about case 1:

At almost 600 m below surface, the topographic effect is not relevant.

It is difficult to visualize in a continuous rock model the support effect for tunnel excavation in rock of strong mass quality. The results above do not mean that rock support is not necessary; it just means that the instability problems in blocky rocks are not well predicted in a continuous rock model.

6.3.4 Case 2: Weakness Zone: Andesite Chainage 6400:

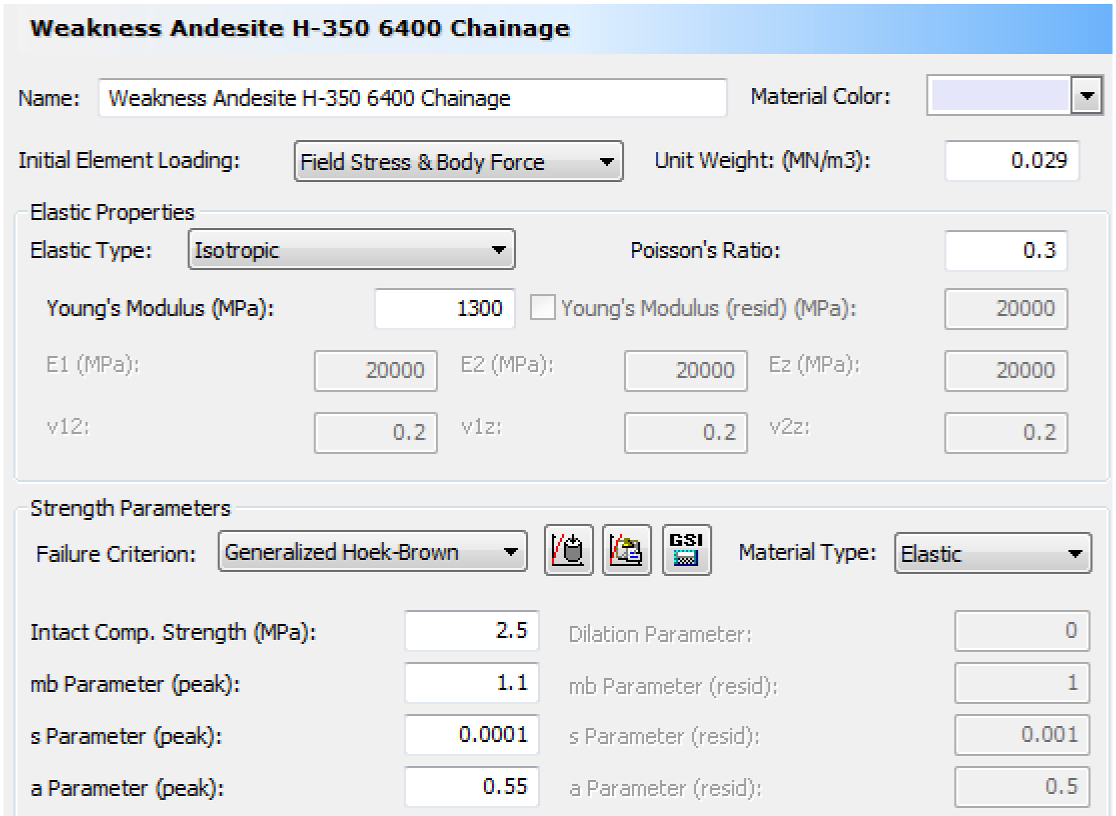
6.3.4.1 Elastic Analysis

Elastic analysis is carried out first to check if rock support is required.

6.3.4.1.1 Input parameters

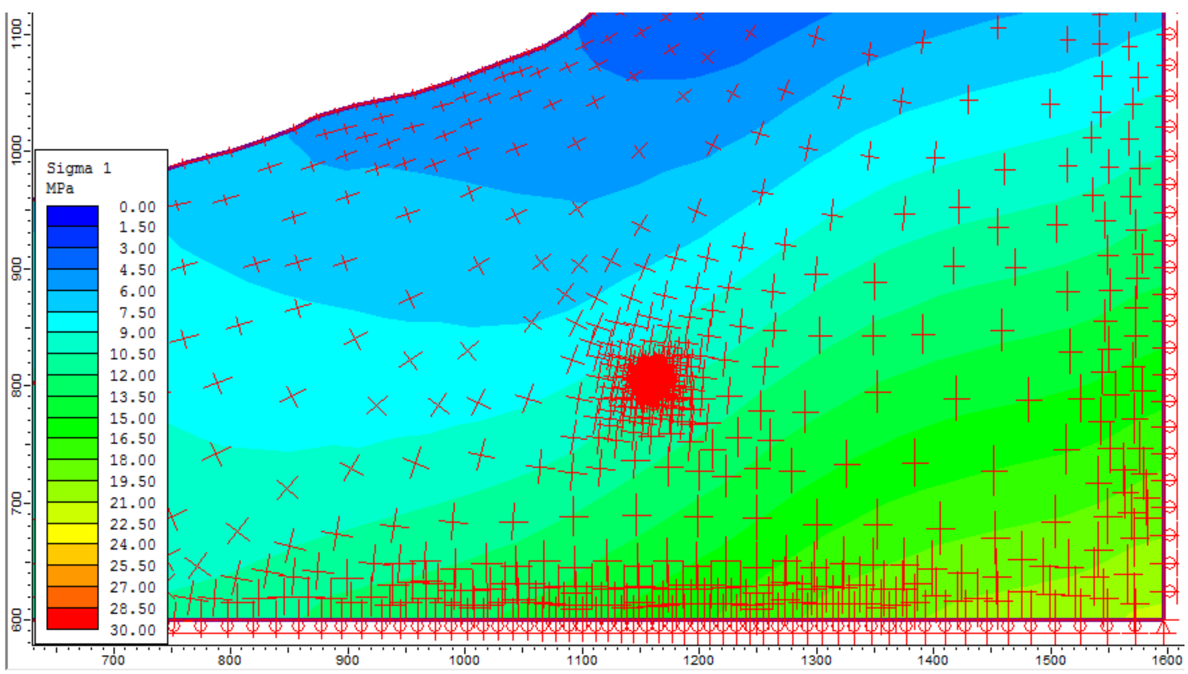
The virgin stress distribution is considered exactly the same as this case.

The properties of the weakness zone in Trapa-Trapa rock formation (Andesite) were obtained in chapter 5.2. The way to include these parameters in Phase 2 is shown in the next figure:

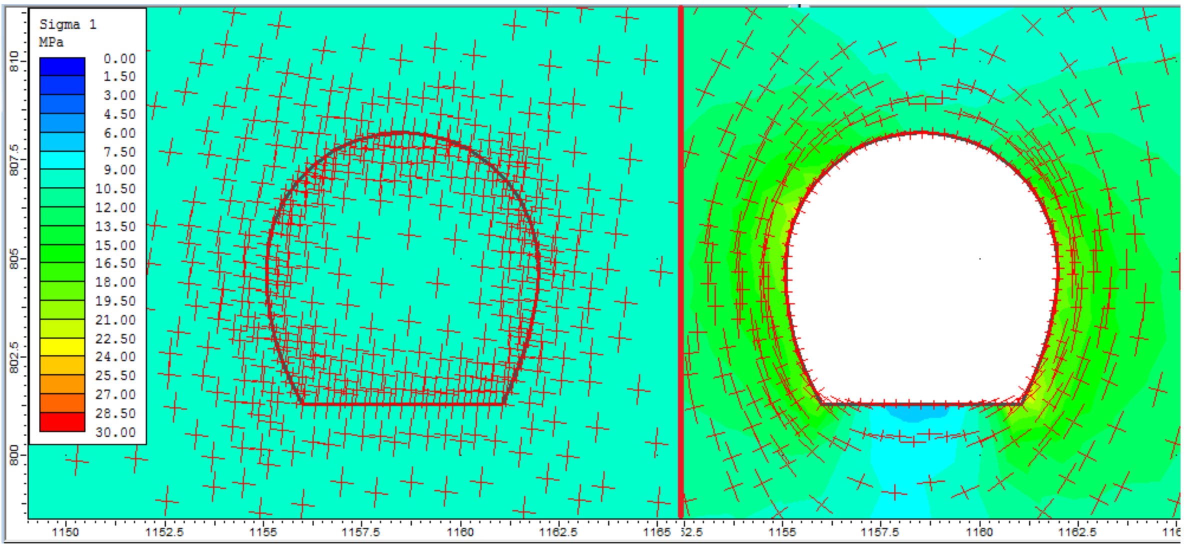


6.3.4.1.2 Results:

The following picture shows the virgin stress situation before underground excavation is carried out:



From the last picture, one can visualize that there is still some topographic effect 350 m below surface. The next picture shows in detail the stress situation before and after excavation:

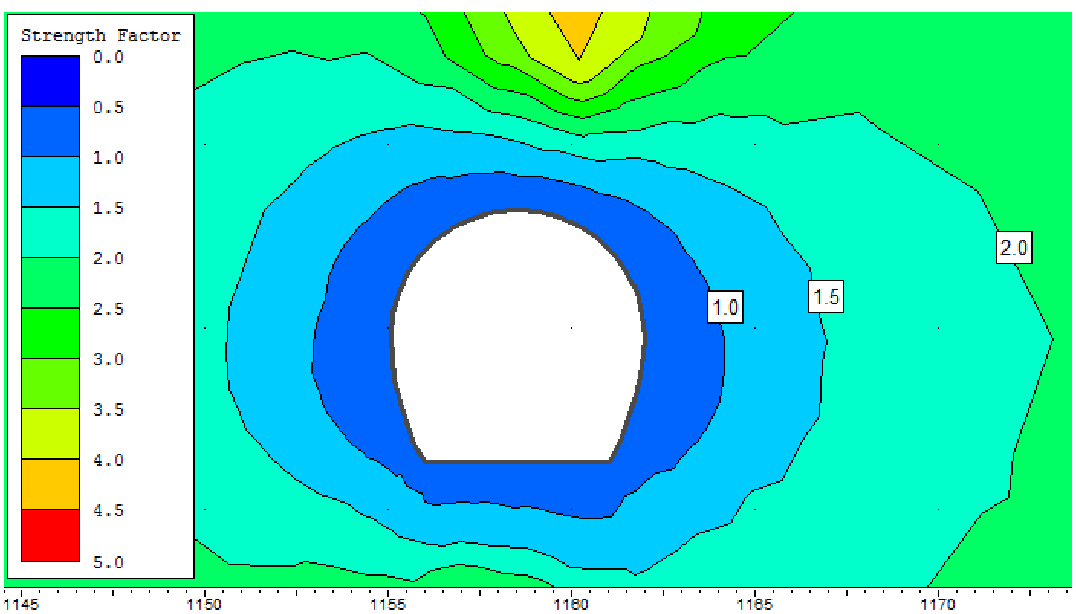


$\sigma_1 = 10.2$ MPA
$\sigma_3 = 6.8$ MPA

$\sigma_{\Theta max} = 22.8$ MPA
$\sigma_{\Theta min} = 11.1$ MPA

Before excavation (left image), the major principal stress σ_1 is slightly tilted. The maximum tangential stresses after excavation are located in bottom right corner and upper left wall consistent with σ_1.

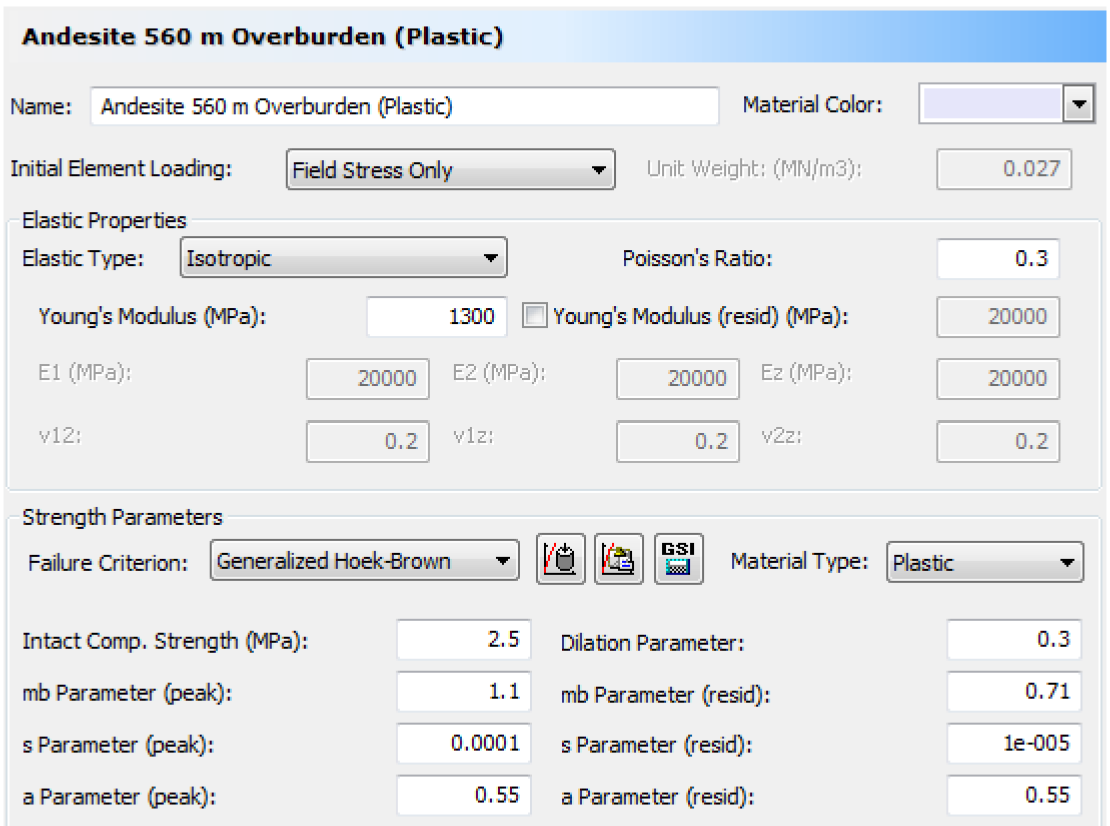
Finally it is shown the strength factor in order to visualize if rock support is required.



The last figure clearly shows that rock support is required since along the whole tunnel contour the strength factor is below 1. The rock support suggested by the Q-system for this case is D40/4 c/c 1.2 and systematic bolting. Along with that rock support, a slab of 15 cm in invert is also included.

6.3.4.2 Plastic Analysis

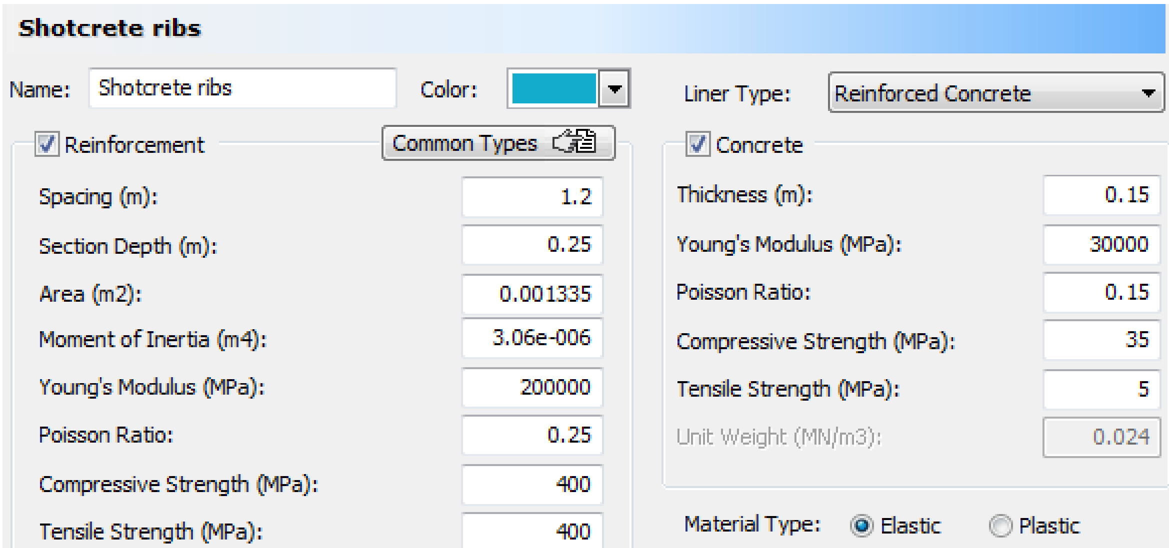
6.3.4.2.1 Input parameters:

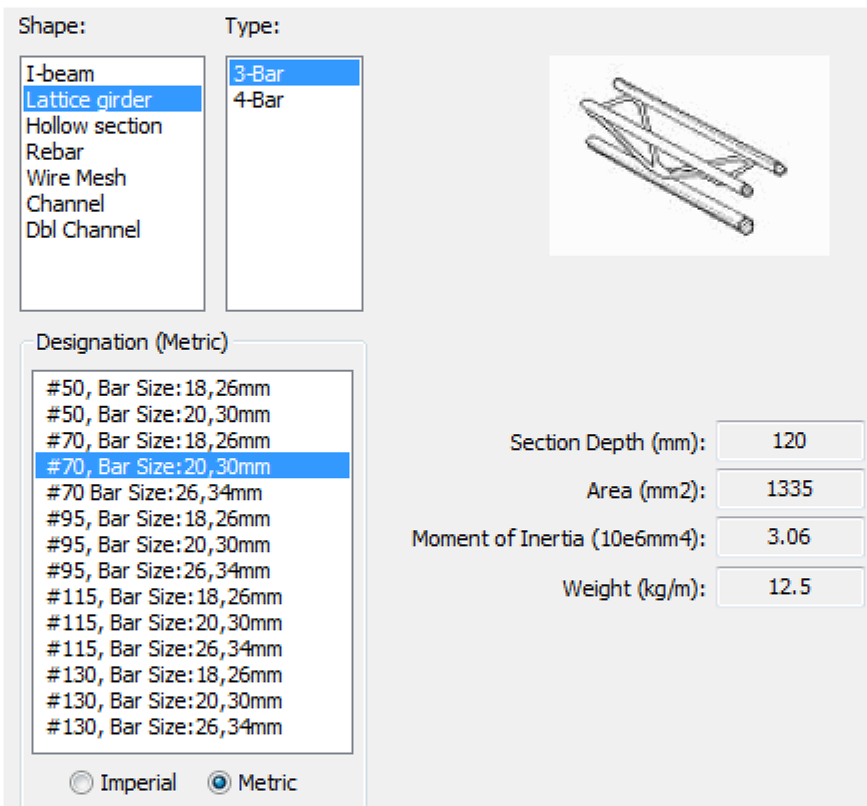


Shotcrete ribs:

The shotcrete ribs needed for this case is 4 rebars in double layer in 40 cm thick lining with c/c spacing 1.2 m. The systematic bolting is in a pattern of 1 m x 1 m. (Q-value = 0.003).

Lattice girder rock support was used instead of rebars reinforcement, since its application is easiest in software Phase 2, and the support behavior is similar. The input parameters inserted in software Phase 2 are shown in the following two figures:





<u>Concrete Invert</u>

The input values for concrete invert (15 cm slab thickness) are exactly the same as the first case “blocky” Andesite.

Radial Bolts:

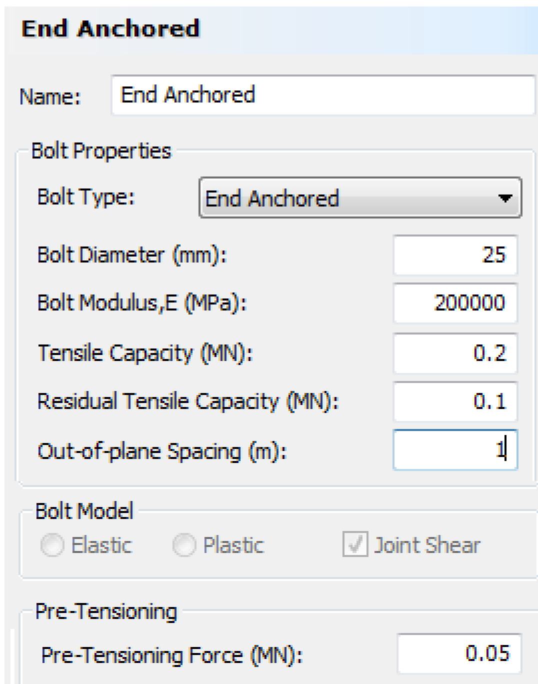
The key inputs for radial bolts from the Q-system and other complementary sources are summarized below:

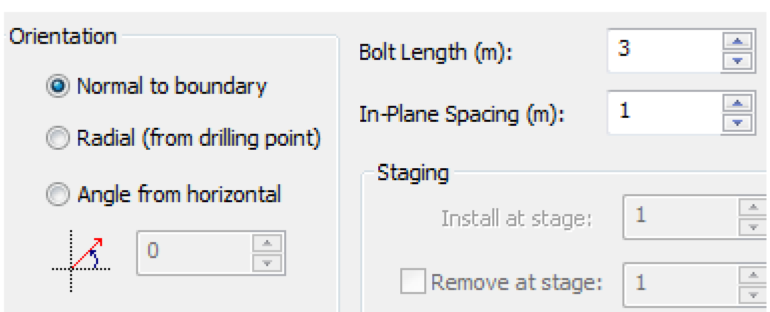
Bolt length = 3 m.

Bolt diameter = 25 mm. (End anchored rock bolt was chosen with 20 tones of maximum strength and 25 % of pre-tensioning).

Square pattern installation: 1 m x 1 m.

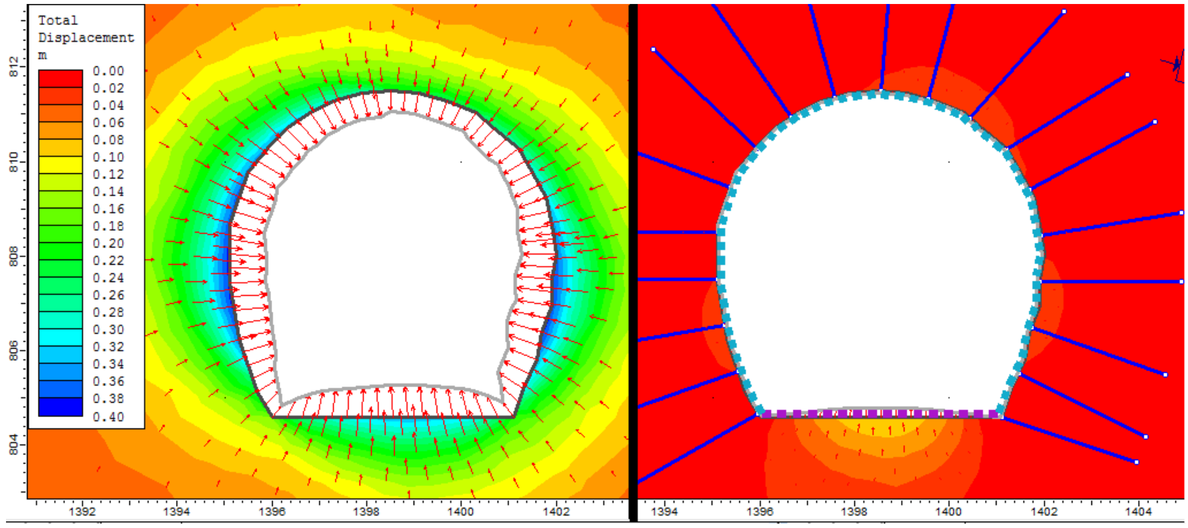
The way to include these parameters in software Phase 2 is shown in the following two figures:





6.3.4.2.2 Results:

Total displacement:

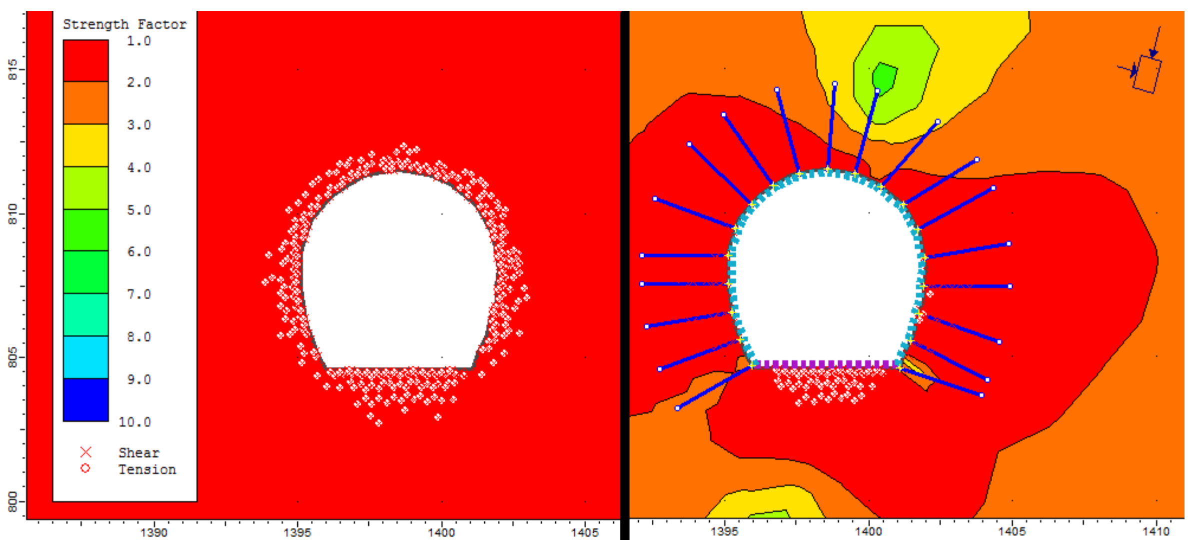


Unsupported **Shotcrete ribs D40/4 c/c 1.2 m & Systematic bolting**

Exaggeration factor: 2

Displacement is strongly reduced from 40 cm in the case of unsupported tunnel invert to 8 cm.

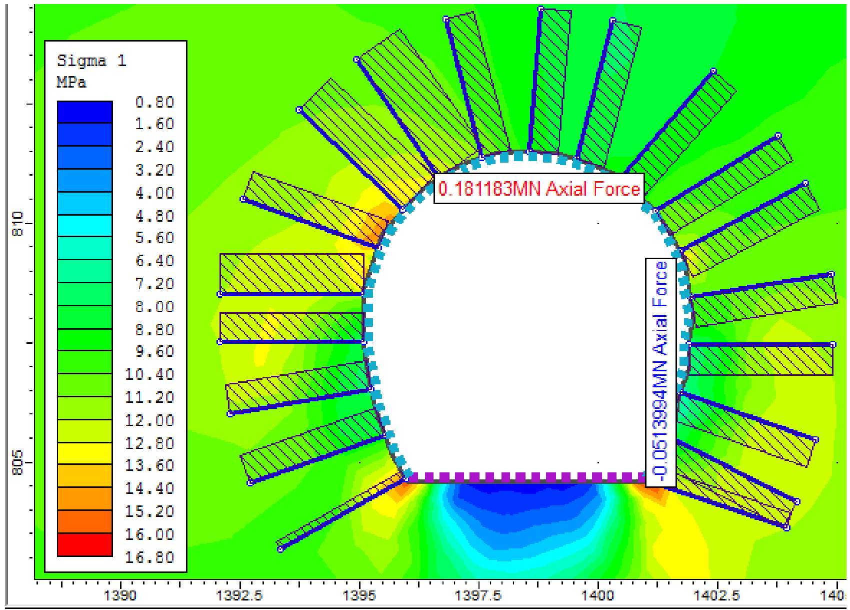
Strength factor:

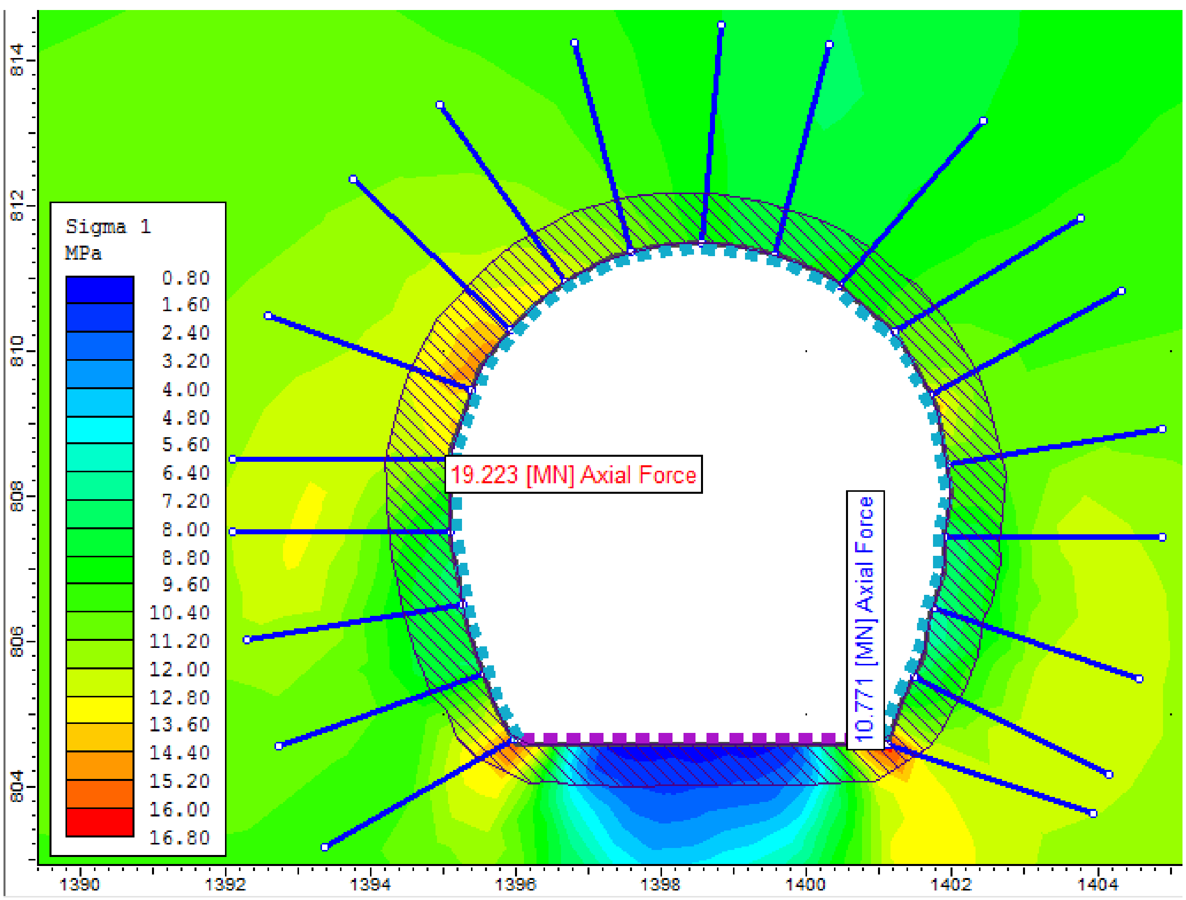


Unsupported **Shotcrete ribs D40/4 c/c 1.2 m & systematic rock bolting**

The zone of plastic yielding is significantly reduced by the rock support. The supported tunnel has only yield elements at the bottom where a slab of 15 cm was included.

Finally, it is shown that neither rock bolts nor shotcrete ribs (represented by lattice girder) are yielded.





6.3.4.2.3 Comments about case 2:

The cross section analyzed in Andesite fault zone clearly shows the need of heavy rock support. The shotcrete rib solution provided by the Q-system improves significantly the stability problem without having yielded elements. A warning is given for the invert, where even with a concrete slab there is a displacement of 8 cm that could crack it.

6.3.5 Comments about Numerical Modelling

Numerical model was used to visualize some aspects of the rock mass that are not predicted by empirical approaches.

At intermediate-seated underground opening and non steep hillsides as the two cases analyzed here (Underground excavation 350 & 560 m below surface), topographic effect is not that significant, finding that the major principal stress is almost vertical in both cases.

Also, case 2 showed that a thicker concrete invert could be required for weakness zones in Andesite where reinforced shotcrete ribs are suggested. An alternative would be a full concrete lining.

7. OPTIMUM CROSS SECTION ANALYSIS

7.1 Introduction:

A larger cross section will have a higher investment cost and a longer construction period but the hydraulic energy losses will be lower which increases the future incomes of the project. In contrast, a smaller cross section will trigger a lower investment cost and a shorter construction period but the hydraulic energy losses will be higher, reducing the future incomes of the project.

The existing diameter for both tunnels Castillo and Vallical is 6.9 m obtained from an economical analysis undertaken as part of the feasibility studies. This chapter aims to redo this study based on new geological information, new support estimation, new advance rate estimation and a thorough analysis of energy losses.

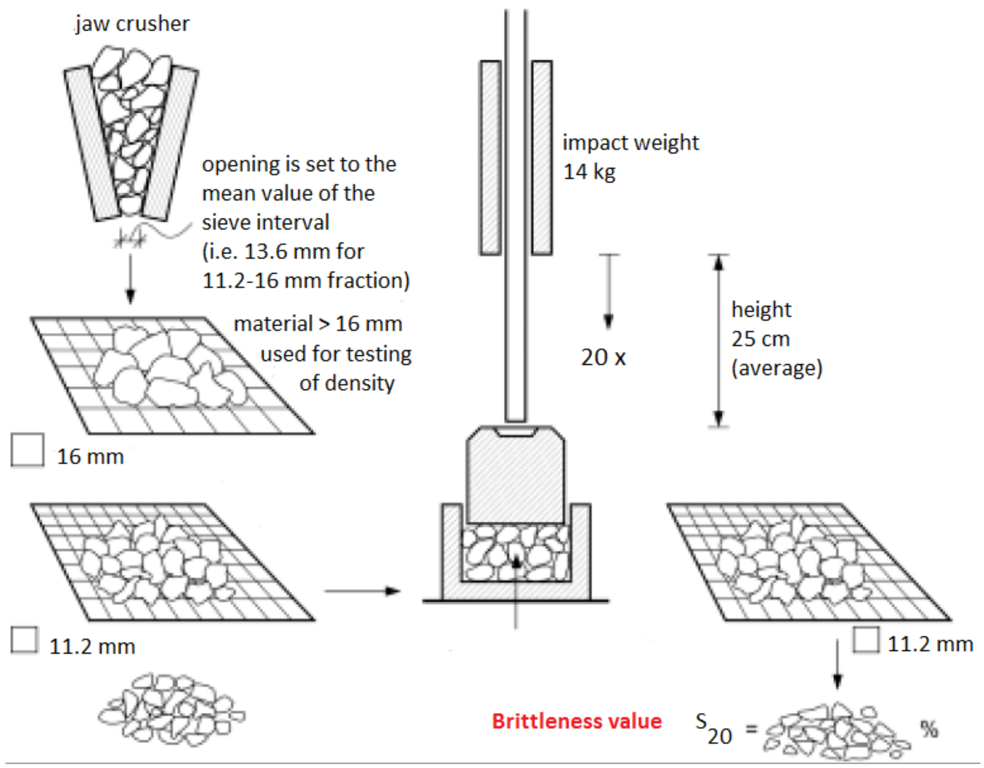
7.2 Theory:

Some input parameters used to quantify and standardize the excavation cost and advance rate are not always known outside Norway and they will be explained below.

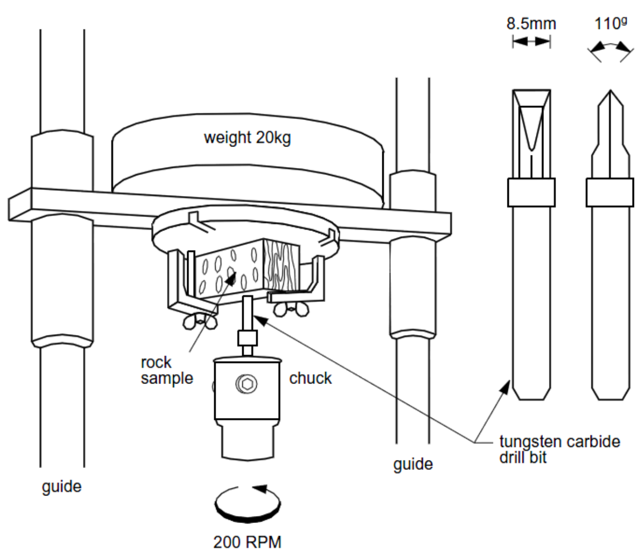
7.2.1 Drillability

It is a measurement on how easy is to penetrate the rock for a certain drilling hammer. Drillability is governed by the capacity of the equipment and the character of the rock mass. Drilling rate index DRI is estimated by the Brittleness Value and the Sievers J-value.

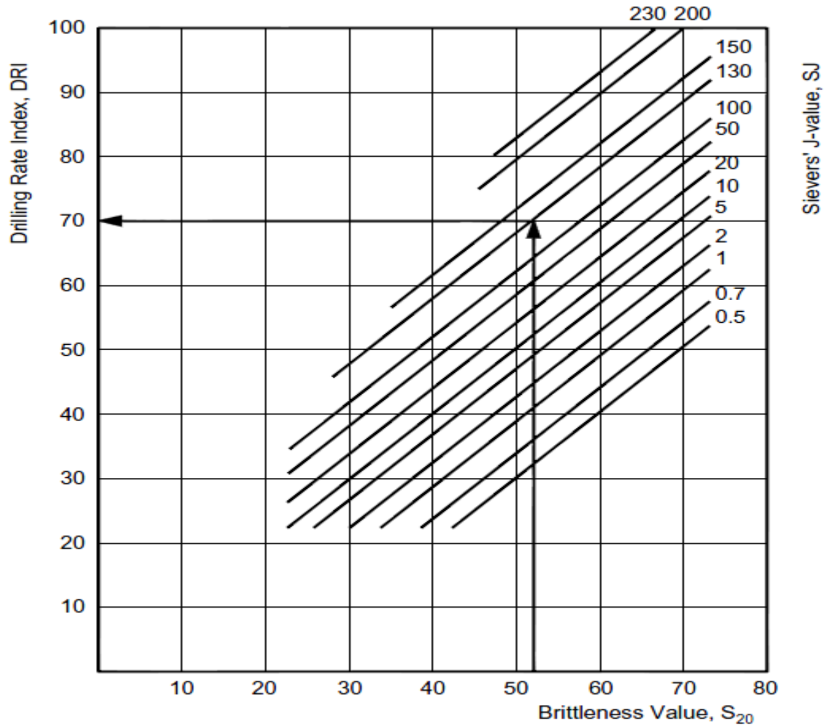
The Brittleness Value is the percentage of the material that passes the 11.2 mm sieve after 20 drops of the 14 kg piston. The higher the brittleness value, the higher the DRI.



The Siever J-value SJ is the penetration measured in 1/10 mm after 200 rotations. The higher the SJ value, the higher the DRI.



The way DRI value is linked to the Siever J value and the brittleness value is shown below:



Drillability value calculated from the Brittleness Value S_{20} and the Sievers J- value SJ.

Good drillability	DRI = 65	For example mica schist
Medium drillability	DRI = 49	For example granite
Poor drillability	DRI = 37	For example gneiss

A DRI value of 49 will be considered for the cross section economic analysis in Andesite (Trapa- Trapa rock formation) and Granite (Melado Batholith). Drilling hammer AC COP 1838 was chosen.

7.2.2 Blastability

It is the amount of explosives needed to break the rock to a certain degree of fragmentation, where 50% of the blasted rock size is under 250 mm.

The rock blastability Index SPR is determined as follows:

$$SPR = \frac{0.736 * I_a^{0.6} * LT^{0.7}}{\frac{c}{1000}^{0.4} * \frac{w^{0.25}}{c} * \rho^2}$$

C = Average dry sonic velocity between perpendicular and parallel to foliation m/s.

I_a = Dry sonic velocity ratio between parallel and perpendicular to foliation m/s.

ρ = Rock density g/cm^3.

w = Detonation velocity of explosives m/s.

LT = Charging density of explosives. (Amount of explosives per volume unit of Drillhole, g/cm^3).

Blastability is influenced by:

Anisotropy (represented by I_a)

Density (represented by ρ)

Sonic velocity (represented by C).

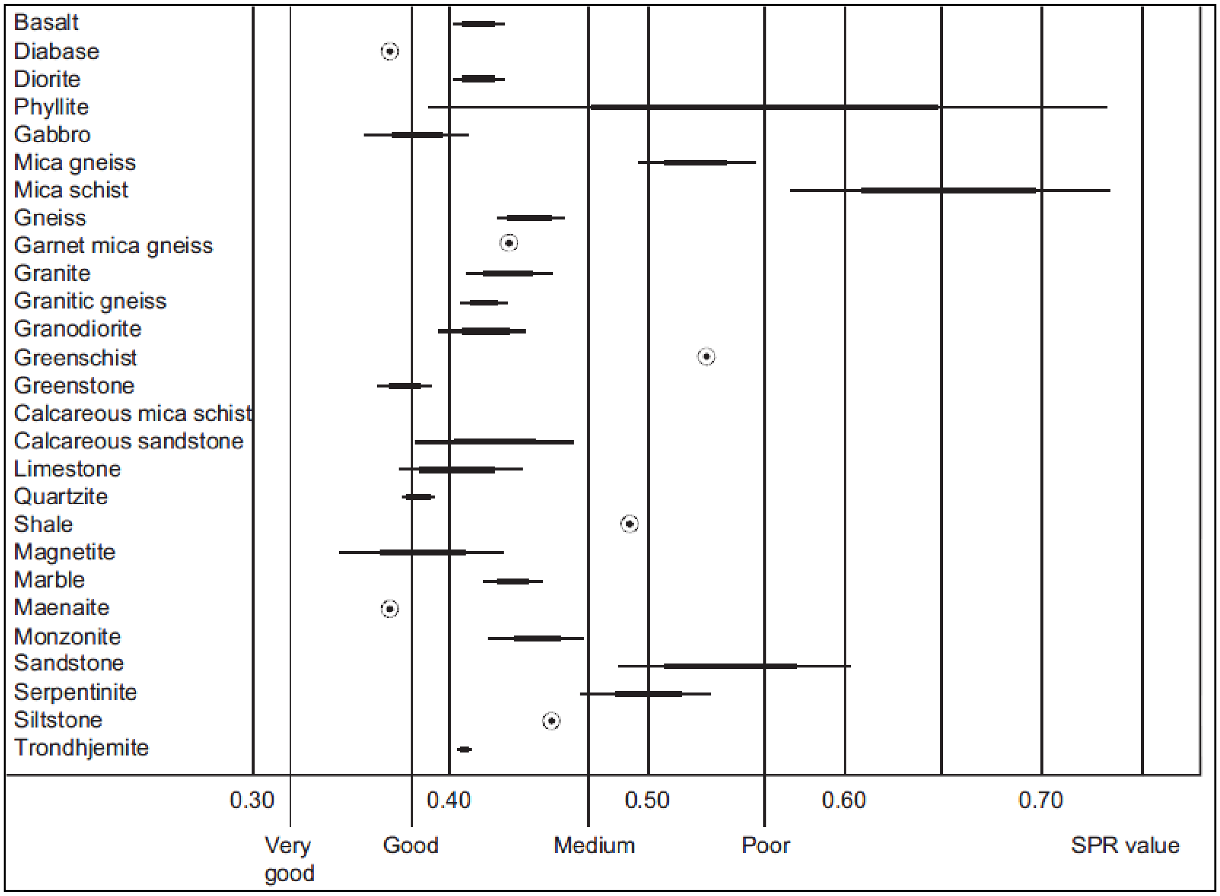
Mineralogy and grain binding (Indirectly represented by C).

Charging density and explosives (Represented by LT)

Detonation velocity of the explosives (Represented by w).

The inputs w and LT depend on the type of explosives. If Emulsion is used its values are w = 4250 m/s; LT= 0.9 g/cm^3.

Since there is not laboratory data, SPR value cannot be determined accurately, but Engineering Geological Laboratory at NTNU has collected data for different rock types as it is shown below:



SPR value tested in the Engineering Geological Laboratory at NTNU.

From the last chart, it is possible to visualize that both, the average SPR value for granite (representing the Batholith Melado) and for Basalt (Representing the Trapa – Trapa rock formation) are between medium and good blastability, but the index SPR does not take into consideration the variation of the rock mass fracturing and orientation of fractures which causes a reduction not quantified in blastability. Therefore, the same document suggests the reader to describe the blastability in a qualitative manner: Good, Medium and Poor blastability.

Good blastability *SPR = 0.38*	Coarse grained homogeneous granites, syenites and quartz diorites. For example "Swedish granite".
Medium blastability *SPR = 0.47*	For example gneiss.
Poor blastability *SPR = 0.56*	Metamorphic rocks with schistose structure, often with high content of mica and a low compressive strength. For example mica schist in the Rana region in Norway.

For the cross section economic analysis, poor blastability will be considered in weakness zones and medium blastability will be considered for both Granite and Andesite where fair and good rock mass conditions are expected.

When drillability and/or blastability are involved in the prognosis model of a variable, it is given its good and poor drillability and/or blastability. Medium values are computed by a simple average between the corresponding good and poor values.

7.2.3 Drill Bit Lifetime:

This variable tries to quantify the lifetime of the drill bit. The main input parameter is the hardness of the rock minerals which is related to the rock abrasiveness. Drill Bit Lifetime varies from 100 to 1800, representing very high steel wear to very low steel wear respectively. Quartz and plagioclase are among the most important minerals for a high wear bit, where the first one is expected to be found in significant percentage in the Batholith Melado and the second one in the Trapa-Trapa rock formation (represented by Andesite) . A drill bit lifetime of 200 has been assigned for both rock formations.

7.3 Methodology

A detail analysis that systematizes the cost and advance rate involved in the round cycle of the Drill & Blast excavation method was carried by Zare, Shokrollah (2007) PHd. thesis [11] & [12]. These reports are based on Norwegian tunneling experience and include recent advances in equipment and methods. These studies were taken in the absence of local prognosis models.

Related to rock support cost, the required support is based on the previous chapter and the unit price considered for each support method is taken from the *Melado* feasibility report (2010) which reflects the real rock support cost for this specific project.

With regard to rock support advance rate, the Department of Civil and Transport Engineering at NTNU provides a prognosis model [13] that was used in the absence of a local rock support prognosis model.

Finally, related to energy losses, the procedure applied is mainly the suggested by reference [14] approach on how to estimate the Manning roughness coefficient and the real cross section area in tunnels (Overbreak included). Some other sources are used to quantify the hydraulic effect of shotcrete lining [1] and shotcrete ribs [2].

The economic range of analysis is from 6.9 m ($\approx$ A = 40 m^2) to 9.3 m ($\approx$ A = 69 m^2) diameter for both tunnels to cover the optimum cross section for each tunnel.

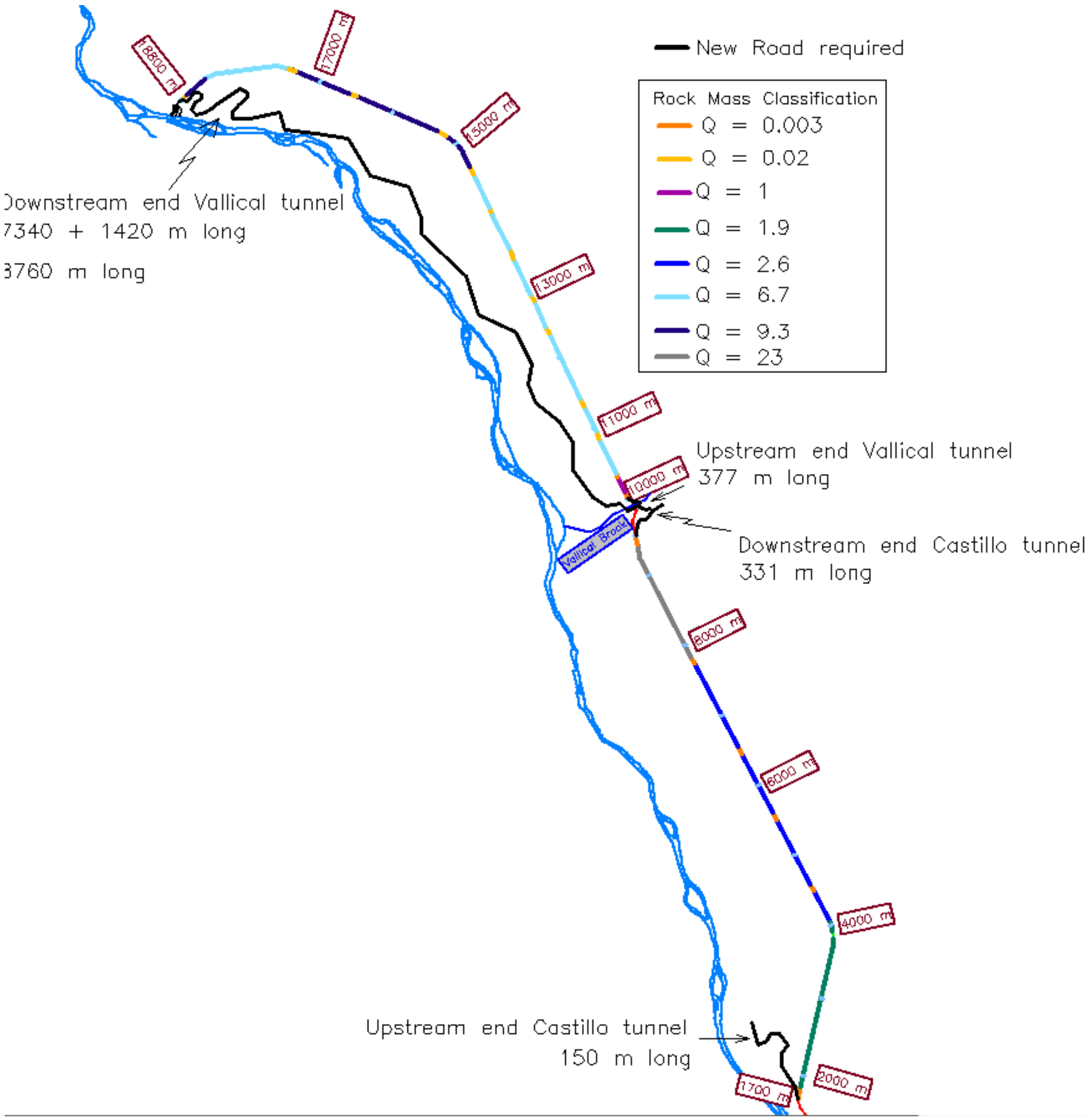


Figure 7-1:Layout for optimum cross section

Note that there is no extra adit considered for this analysis.

7.3.1 Assumptions for Drill & Blast

Even though the Drill & Blast cost and advance rate documents cover a wider range of tunnel cross sections, some simplifications can be adopted related to the economic range of analysis between 40 and 69 m^2.

Trackless transport: Only cross sections smaller than 16 m^2 are recommended to be constructed with rail bound transport. Therefore for the whole range of cross sections analyzed, trackless transport will be chosen.

3 boom wheel mounted jumbo: A drilling jumbo with three drilling hammers is recommended for cross sections between 20 and 80 m^2. This range covers the whole range of the current analysis.

Three boom tunneling rig: Picture obtained from Atlas Copco "Underground rock excavation".

Scaling from drilling jumbo: Only tunnel cross sections smaller than 20 m^2 should be scaled from muck pile with a crowbar. Larger cross sections, as the range under analysis here should be scaled from Jumbo.

No load and haul niches: These niches are recommended for cross sections smaller than 30 m^2, which is lower than the minimum cross section under analysis here. Only turning niches will be considered between 40 and 65 m^2 (each 300 meter), and no niches above that.

Some other criteria not recommended necessarily by the cost and/or advance rate prognosis model are adopted by different reasons which are given below:

Large-hole parallel cut: This type of cut is dominating in Norway [10]. The round length is not dependent on the tunnel cross section. It has a Good pull. It triggers less throw and spreading of the much pile, reducing the loading time. Finally it has a good fragmentation

Emulsion slurry as explosive: There is no restriction suggested by the cost or advance rate prognosis models about which kind of explosive should be used for tunnel cross sections larger than 16 m^2 and both documents provide details for ANFO and Emulsion. However, both in the advance rate prognosis model in Drill and Blast and the PHd thesis “Conventional tunnel construction” **1997 [17]** agree that emulsion is faster to install, it can deal with water leakage problems, requires less ventilation and causes less toxic gases than ANFO.

Charged Drillhole diameter = 48 mm: The cost and advance rate prognosis models provide information for both 48 mm and 64 mm. In general, a larger charged drill-hole triggers a better pull "p_r" (efficiency ratio between effective round length and drilled length) [10], but also causes a higher overbreak [17]. The resulting economic convenience of either alternative is not clear and 48 mm was chosen arbitrarily.

Low skill level:

Number of empty holes N_g = 4 with a diameter = 102 mm each. The need of enough space for rock expansion that will be blasted is given at the beginning of the explosion by the cut. This empty area required to secure a full throw out depends on blastability and drilled length. For poor blastability and 5 m drilled length, [10] the blast design suggests 4 large holes with a diameter of 102 mm each. No difference was made for 3 m drilled length, where free space requirement should be less.

Blastability and drilled length will depend on rock mass conditions. In poor rock mass conditions where the degree of fracturing is very high, a poorer blastability is expected [10] . With regard to drilled length in the same rock conditions, the general rule is to reduce the drilled length (and consequently the round length) because of immediate rock support required as it is shown in the next illustration.

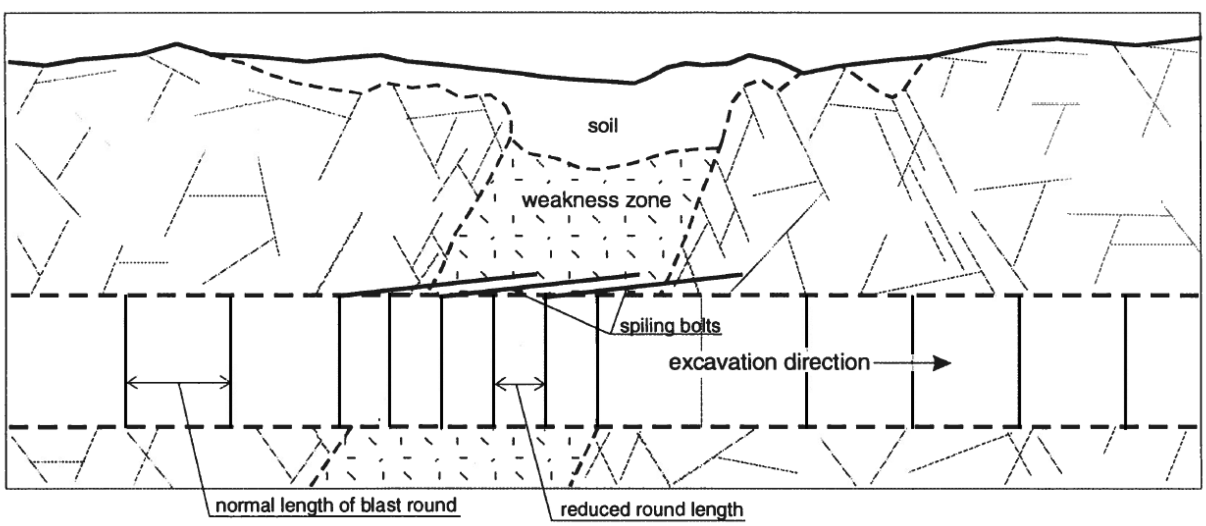


Figure 7-2: Reduced round length in poor rock conditions [8].

For this economic analysis, poor blastability and 3 m drilled length is adopted for extremely poor rock mass conditions which correspond to weakness/fault zones in either Trapa-Trapa rock formation or Batholith Melado. Meanwhile in fair and good rock mass conditions, in both Trapa-Trapa rock formation and Batholith Melado, medium blastability and 5 m drilled length is assigned.

Finally, some economic decisions are made to determine the cost and advance rate of each tunnel:

No extra adit is considered in tunnel layouts: This means that each tunnel (Castillo & Vallical) is excavated only from its two ends. The economic convenience of this decision will be demonstrated in the next chapter.

7.3.2 Excavation Cost

As it was said before, a statistical approach carried out by Zare, S. PHd thesis [12] will be used to estimate the excavation cost when Drill and Blast technique is used.

The cost in kroner is updated from 2005 to 2009, by an inflation of 10% among those years. Afterward a conversion rate of 6.19 between NOK and USD is used. The final conversion rate is 5.7.

Excavation cost in Drill and Blast technique comprises:

Drilling Costs

Charging (Explosive and detonator) Costs

Scaling Costs

Loading Costs

Hauling Costs

Tip Costs

Ventilation Costs

Electrical, Water Supply & Miscellaneous Costs

Labor Costs

Niche Costs

Unforeseen Costs

For this economic analysis, the excavation cost will vary due to:

Tunnel cross section area, which affects:

All the costs mentioned above, except Electrical & Water Supply.

Blastability, which affects:

Charging costs

Scaling costs

Drilled length l_h which affects:

Charging Costs

Scaling costs

Labor costs

Tunnel length from one adit (Heading tunnel length) which affects:

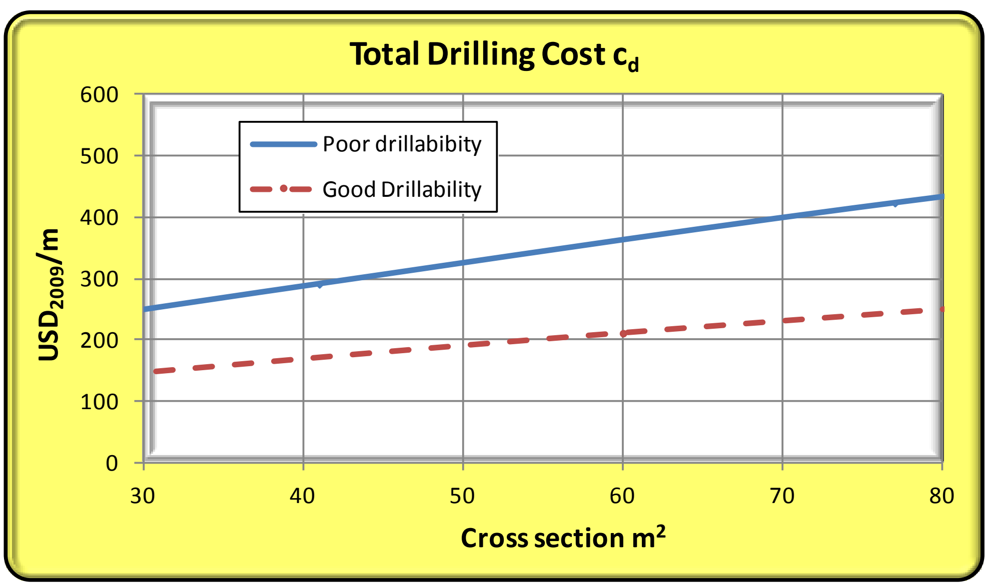
Hauling Cost

Ventilation Cost

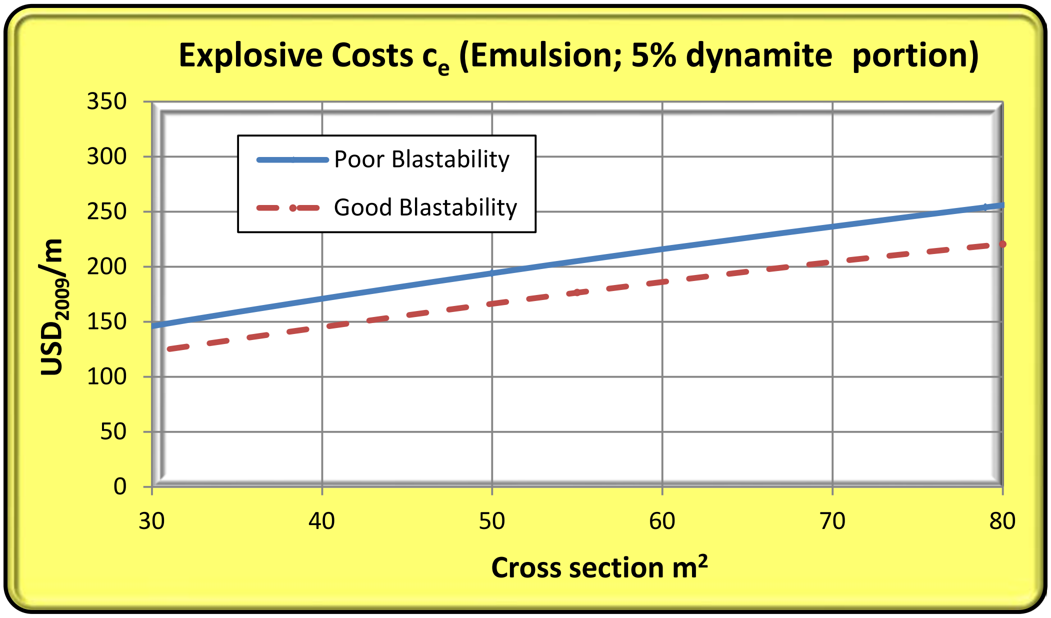
Electrical & Water supply Cost

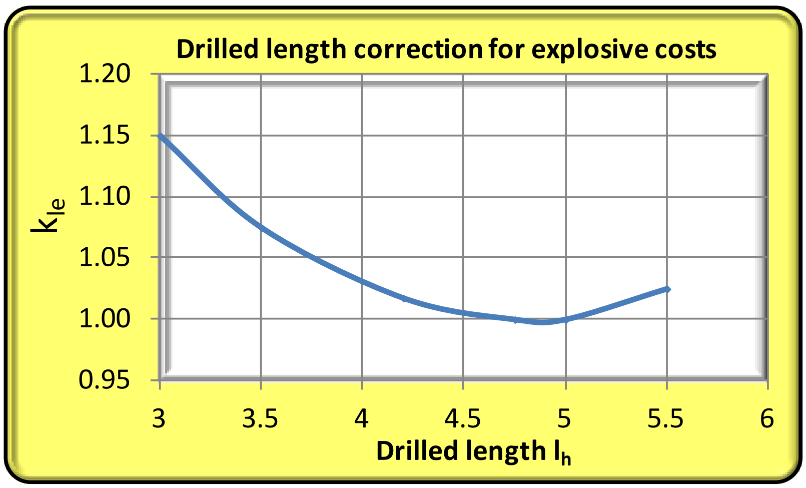
Labor Cost

7.3.2.1 Drilling Cost:

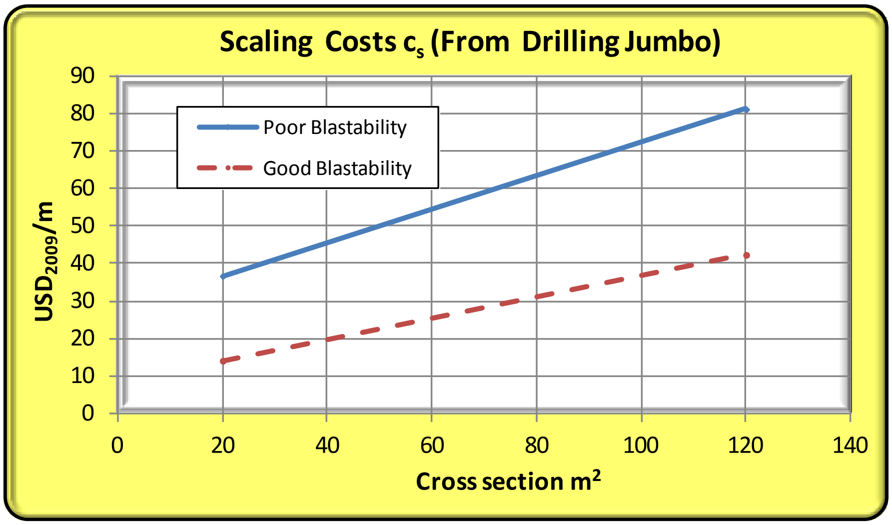


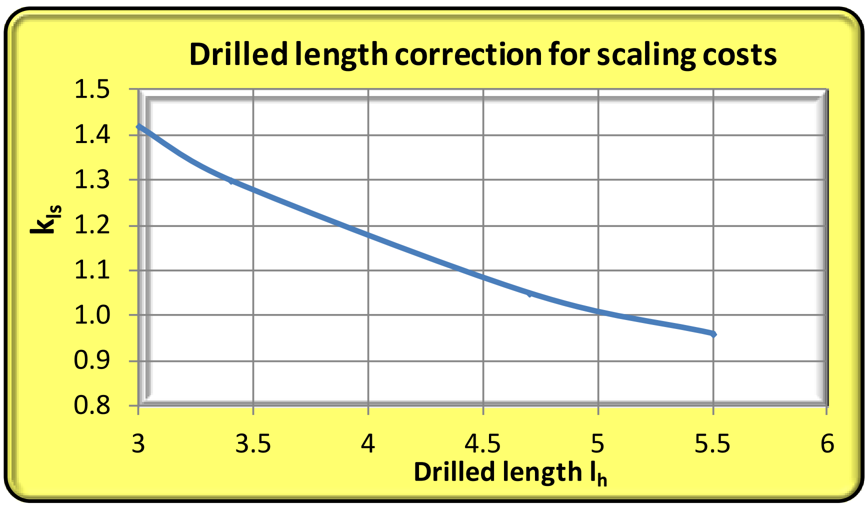
7.3.2.2 Explosive costs:



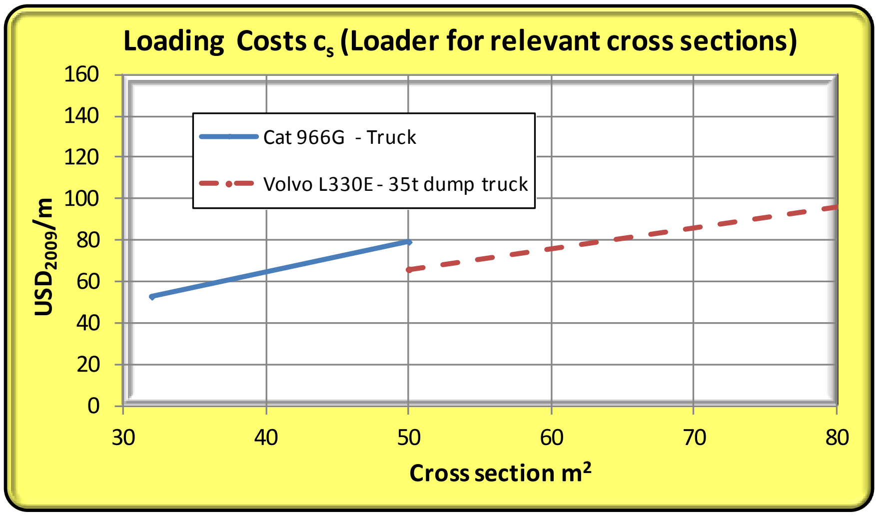


7.3.2.3 Scaling cost



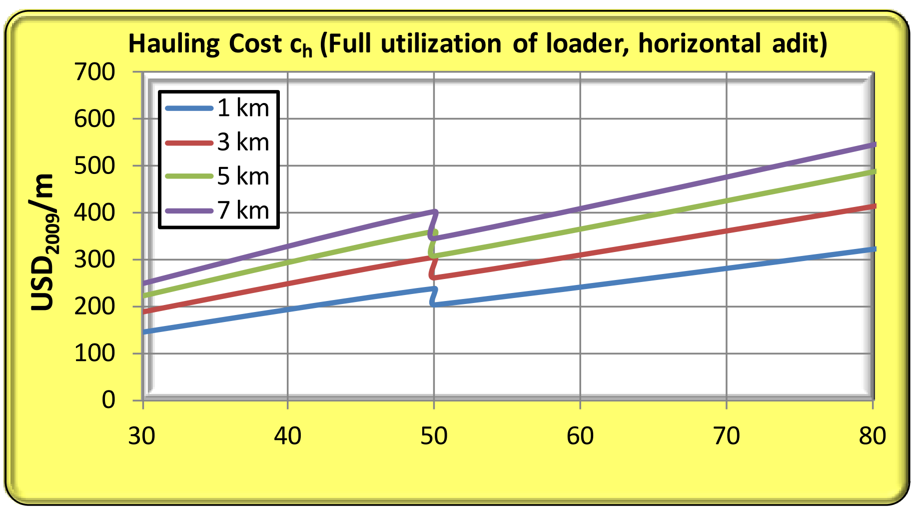


7.3.2.4 Loading cost



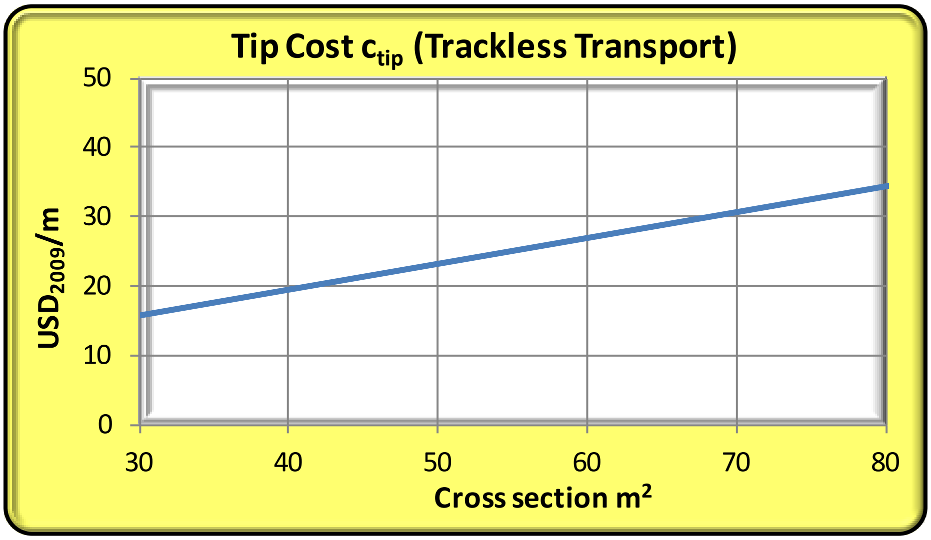
Volvo L – 330 (Recommended for cross section larger than 50 m^2).

7.3.2.5 Hauling Cost:

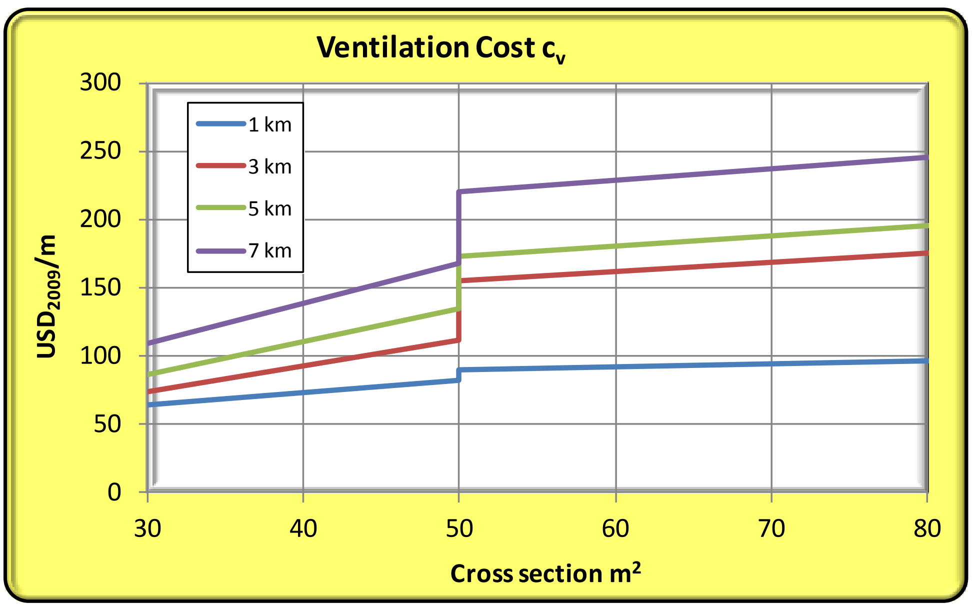


The drop at 50 m^2 is explained because of change in technology.

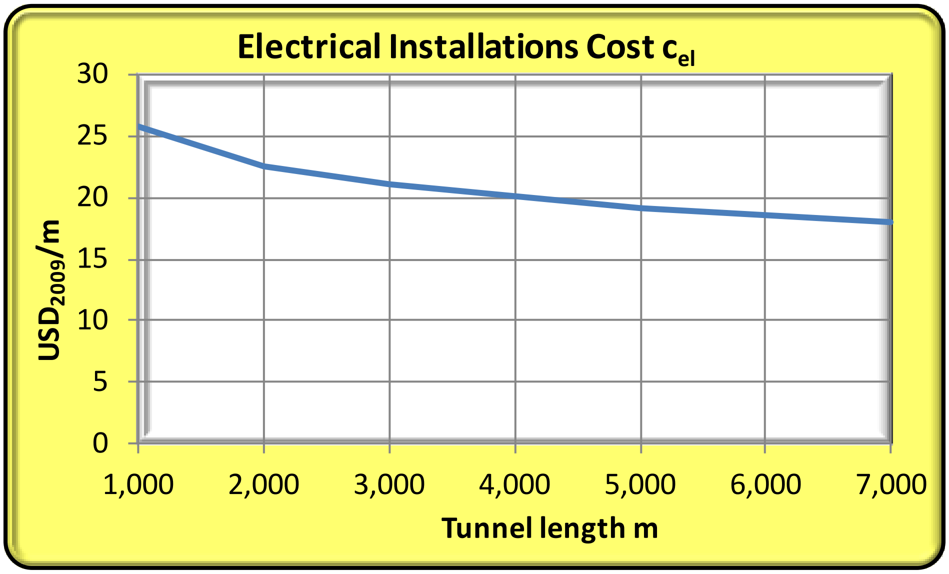
7.3.2.6 Tip Cost:



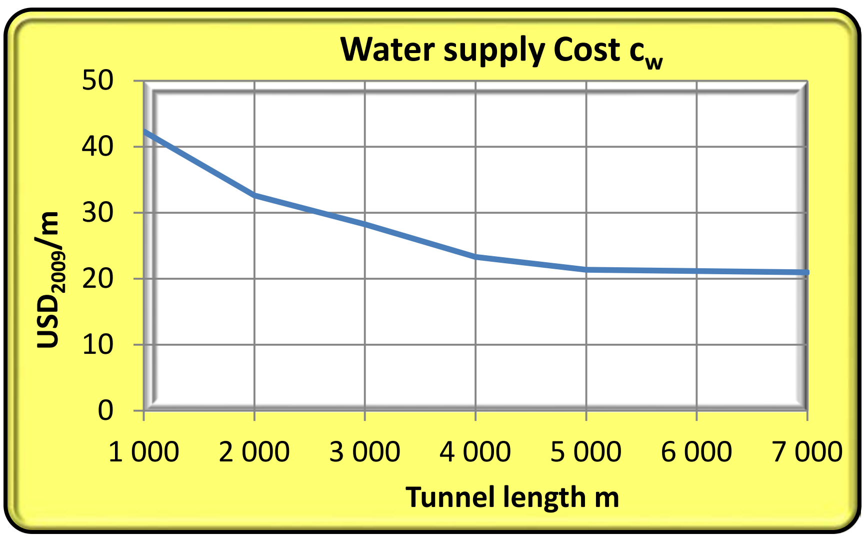
7.3.2.7 Ventilation Cost:



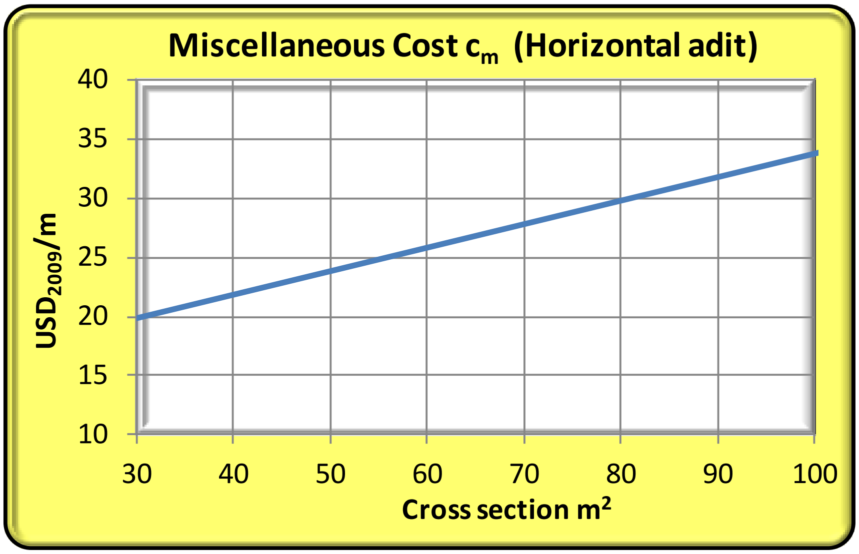
7.3.2.8 Electrical installation:



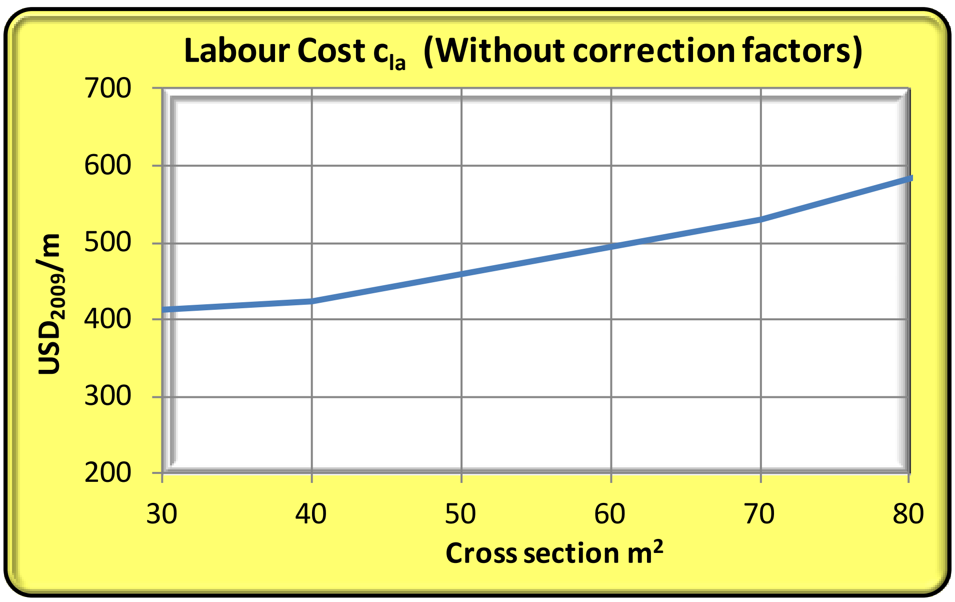
7.3.2.9 Water supply cost:

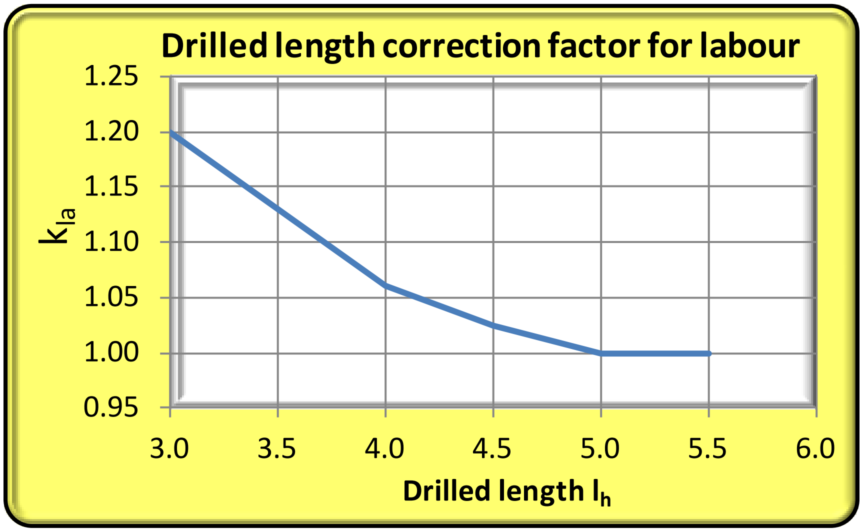


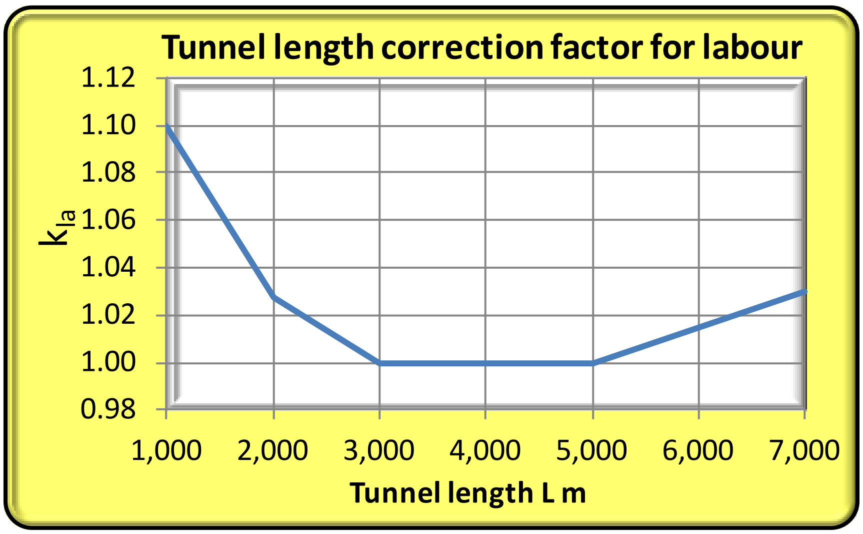
7.3.2.10 Miscellaneous:

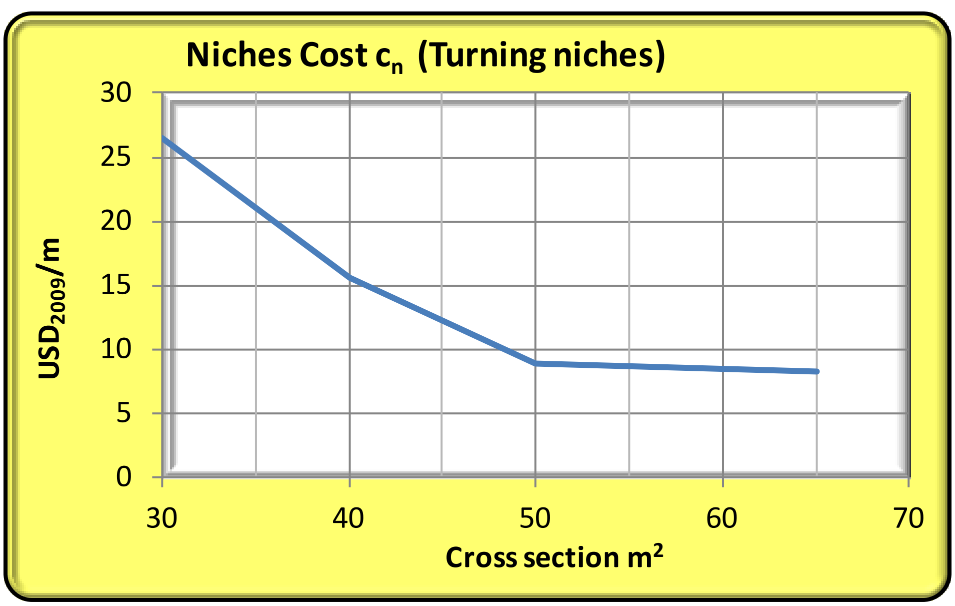


7.3.2.11 Labour




Drilled length correction factor for labour
1.25
1.20
1.15
1.10
1.05
1.00
0.95
k_{la}
3.0
3.5
4.0
4.5
5.0
5.5
6.0
Drilled length l_h


Tunnel length correction factor for labour
1.12
1.10
1.08
1.06
1.04
1.02
1.00
0.98
k_{la}
1,000
2,000
3,000
4,000
5,000
6,000
7,000
Tunnel length L m


Niches Cost c_n (Turning niches)
30
25
20
15
10
5
0
USD_{2009}/m
30
40
50
60
70
Cross section m^2

Finally, the cost is updated from 2005 to 2009, by an inflation of 10% among those years. Afterward a conversion rate of 6.19 between NOK and USD is used. The final conversion rate is 5.7.

7.3.3 Rock Support cost

The rock support considered here will be related to what was carried out in the previous chapter. Some other complementary sources were considered when details are not provided. But only, the following rock support will be considered for the economic analysis:

Unreinforced Shotcrete

End anchored rock bolts

Dowells (grouted bolts)

Fibercrete

Reinforced shotcrete ribs (With rebars)

Concrete invert

Wire-Mesh for invert

The unit price of each item was obtained from the feasibility report in US dollar May, 2009.

Fibrecrete	USD/m^3	985
Unreinforced Shotcrete	USD/m^3	895
Rebars (20 mm diameter)	USD/kg	4.8
Bolts (25 mm, 3 m) galvanized end anchored	USD/unit	70
Bolts (32 mm, 6 m) fully grouted	USD/unit	230
Concrete invert	USD/m^3	325
Wire mesh type ACM C-139 for invert	USD/m^2	8.8

These costs are the installed cost for the owner. This means direct cost and indirect costs like transport, installation and constructor company profit.

All the rock support included in the economic evaluation is mentioned below:

Q_{ROOF} (0.003): Reinforced Shotcrete ribs D40/4 c/c 1.2 m; Systematic bolting (Roof & Walls)

Q_{ROOF} (0.02): Reinforced Shotcrete ribs E35/5 c/c 2.3 m; Systematic bolting (Roof & Walls)

Note that for the range of analysis between 6.9 m diameter and 9.3 m diameter which is related to the "Equivalent Dimension" D_e from 4.3 to 5.6 (Vertical axis in the Q-system) no change is visualized in reinforcement category. This is very important, because it means that the following table suits for the whole range of cross section areas:

Fixed values for Shotcrete ribs		
Shotcrete ribs Rebar (20 mm diameter)	2.5	kg/m/rebar
Number of rebars in Shotcrete Ribs (Q = 0.003)	4	Unit
Number of rebars in Shotcrete Ribs (Q = 0.02)	5	Unit
Spacing Shotcrete Ribs RRS (Q = 0.003)	1.2	m
Spacing Shotcrete Ribs RRS (Q = 0.02)	2.3	m
Fibrecrete thickness before shotcrete ribs (Q= 0.003 & Q = 0.02)	0.15	m
Unreinforced shotcrete thickness to cover ribs (Q = 0.003)	0.25	m
Unreinforced shotcrete thickness to cover ribs (Q = 0.02)	0.2	m

Along with reinforced shotcrete ribs, Spiling bolts are always considered in weakness zones and are installed along with RRS Reinforced Shotcrete Ribs. Spiling bolt spacing is given by NFF Publication 19 with a range between 0.2 and 0.6 m. As it was said before, drilled length in weakness zones was set to 3 m, and with a pull of 91% (48 mm drill-hole diameter), the resulting round length is 2.73 m. The latter value is relevant when counting the number of spiling bolts per meter tunnel length. The adopted values for Spiling bolts are given below:

Fixed values for Spiling Bolts		
Spacing Spiling Bolts (6 m long, 32 mm diameter)	0.4	m
Round length in poor rock mass conditions (3 m * 91%)	2.73	m

Either dowels or end-anchored bolts 25 mm diameter is adopted. The Dowel bolt cost is considered equal to end anchored bolt.

Rock bolts length of 3 m is considered for all the radial bolts as it was determined in the rock support chapter.

Fixed value for Radial bolts		
Radial bolt length (25 mm diameter)	3	m

7.3.4 Advance rate

The advance rate defines two important inputs which are the total length from each portal (heading length) and the tunnel construction period. The first input is a variable for the excavation cost influencing the Hauling Cost, Ventilation cost, Electrical cost, etc. The tunnel construction period is an input for the interest cost under construction that is also estimated for the economic analysis.

From the document Drill and Blast tunneling advance rate Shokrollah, Zare (2007) was predicted the advance rate for each geological stretch of the tunnel.

From reference [11]

This prognosis model considers four major operations in the round cycle:

I Drilling, Charging & Blasting

II Ventilation

III Loading and Hauling

IV Scaling and Rock Support

The operations I and III are divided in three categories:

1- Rig time = All unproductive operations regularly repeated from round to round like driving the drilling jumbo to and from the face.

2- Proportional Operation time = It is the productive time such as drilling (proportional to specific drilled meters), loading time (proportional to the amount of broken rock), etc.

3- Incidental lost time = It is the lost time that occurs randomly during tunnel operation like machine breakdown, delays because of shift change, etc.

All the rock support required at the tunnel face will be included within the round cycle. Support that can be carried out behind the face is not included in the round cycle.

7.3.4.1 Assumptions:

The effective working time per week = 120 hr.

One month for each portal construction period.

Advance rate of access roads to adits is 50 m/day

2/3 of the total estimated bolts were considered at the tunnel face and 1/3 behind the face for fair and good rock mass conditions, while all the bolts were considered at the face for weakness zones. This fact affects the advance rate, where the bolts installed at the tunnel face delays the excavation while the bolts installed behind the face don’t.

Fibercrete is installed always at the tunnel face. Unreinforced shotcrete is always installed behind the face.

All the heavy rock support estimated for poor rock conditions was considered at the tunnel face.

All the spiling bolts considered will be installed at the tunnel face.

As suggested by the document [11], fracture can lead to additional time consumption during drilling and up to 10% could be added because of this problem.

Some definitions used here in order to distinguish the different tunnel advance stretches (heading length) are explained below:

"Castillo 1" = Castillo tunnel stretch excavated from the Castillo brook portal.

"Castillo 2" = Castillo tunnel stretch excavated from Vallical brook portal.

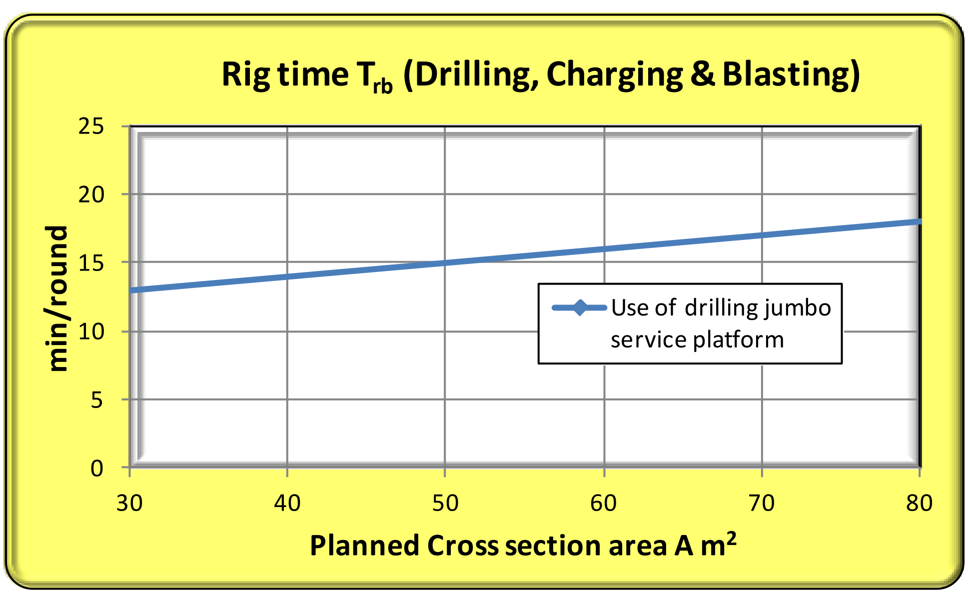
"Vallical 1" = Vallical tunnel stretch excavated from Vallical brook.

"Vallical 2" = Vallical tunnel stretch excavated from its downstream end.

7.3.4.2 Drill and Blast:

I Drilling, Charging and Blasting

Rig Time:



Proportional Operation Time:

Drilling time

$$\text{Drilling time: } T_h + T_g + T_f + T_K + T_{sa}$$

T_h= Drilling time for charged holes.

T_g= Drilling time for empty holes.

T_f= Time for moving drilling hammers.

T_K = Time for changing bit.

T_{sa} = Lack of simultaneousness.

$$T_h = \frac{l_h * N_h}{v_h * N_m}$$

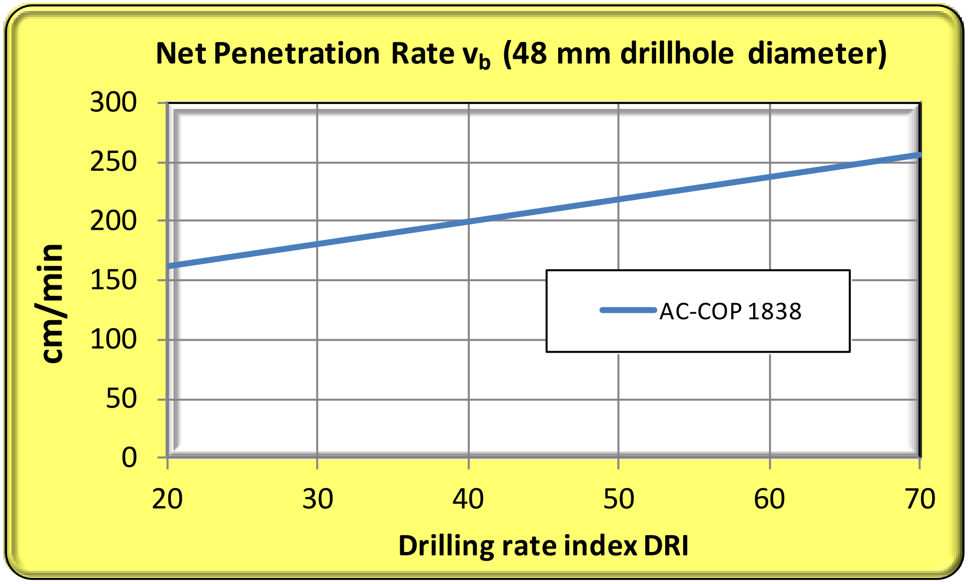
l_h = Drilled length (3 m in weakness zone and 5 m in fair & good rock mass conditions).

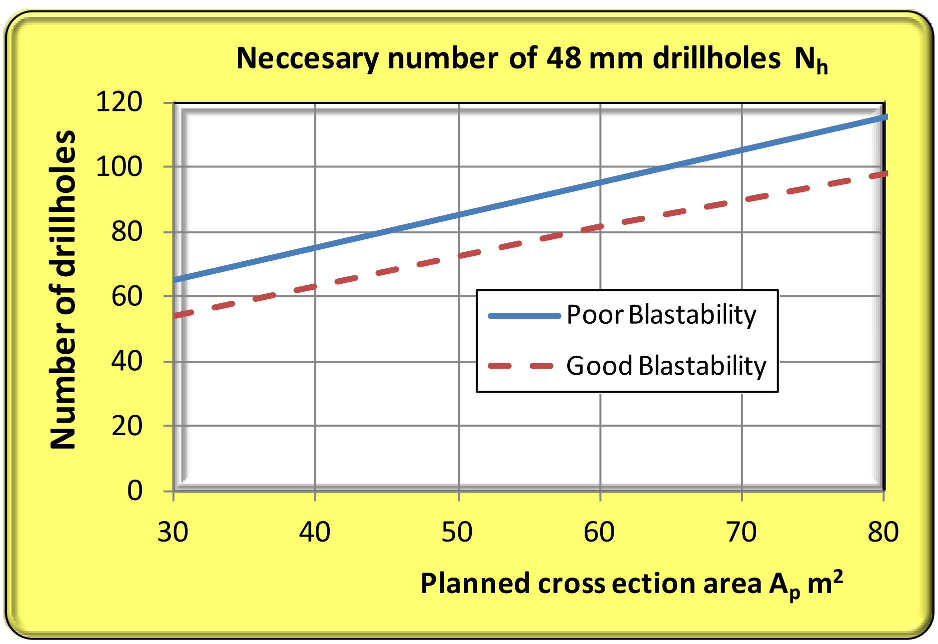
N_h = Number of charged holes.

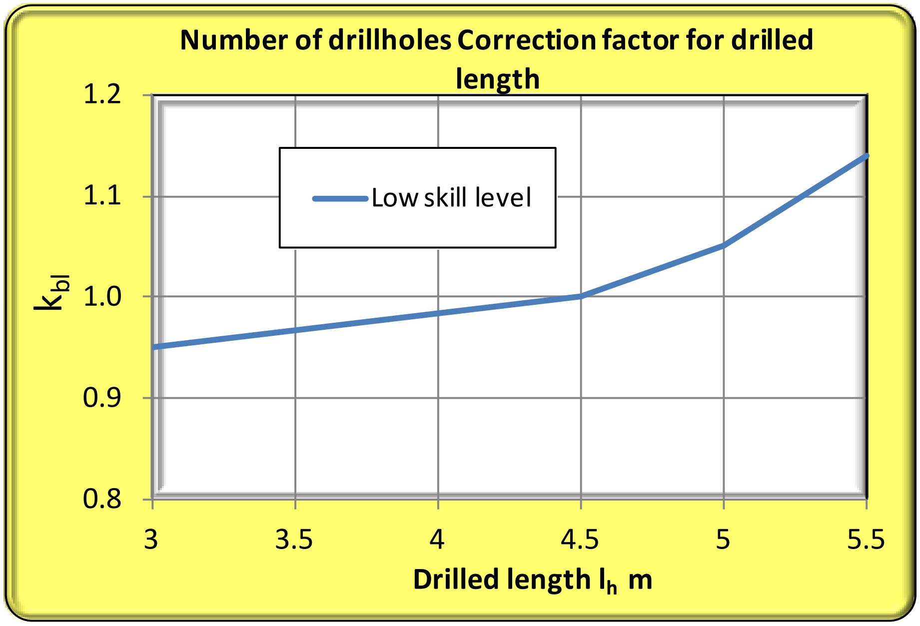
v_h= Net penetration rate for charged holes.

N_m = Number of drilling hammers = 3.

In the following chart the net penetration rate for the drilling hammer AC-COP 1838 is shown. Note that the net penetration rate is represented by v_b instead of v_h as in the equation mentioned above. The difference between them is a correction factor in the penetration rate because of charged drillholes diameter k_{hv}, but this factor is 1 for 48 mm diameter and therefore v_b and v_h become the same value.


Net Penetration Rate v_b (48 mm drillhole diameter)
300
250
200
150
100
50
0
cm/min
AC-COP 1838
20
30
40
50
60
70
Drilling rate index DRI


Neccesary number of 48 mm drillholes N_h
120
100
80
60
40
20
0
Number of drillholes
Poor Blastability
Good Blastability
30
40
50
60
70
80
Planned cross ection area A_p m^2



$$T_g = \frac{l_g * N_g}{v_g * N_m} * 1.25$$

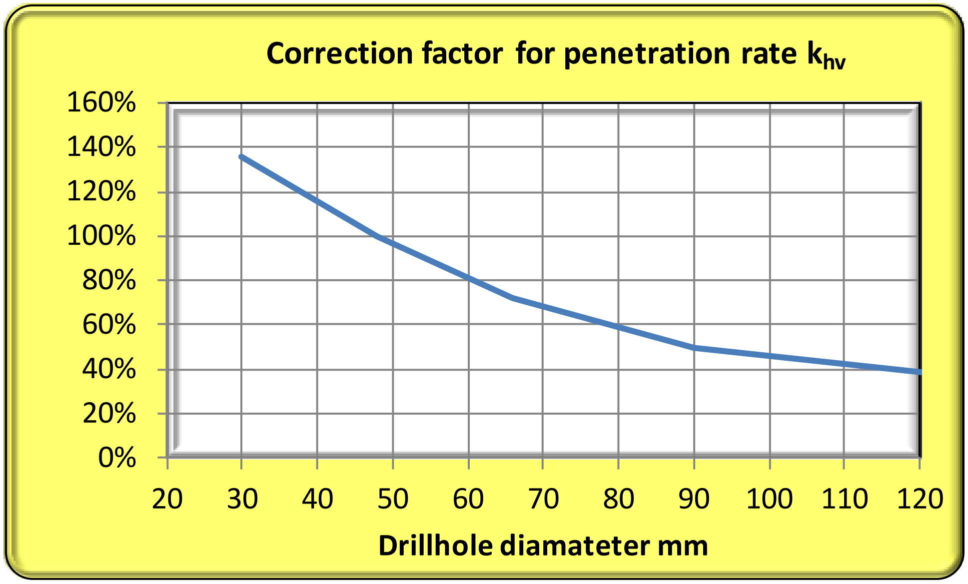
l_g = Drilled length of large holes = l_h.

N_g = Number of large holes = 4.

v_g = Net penetration rate for empty holes.

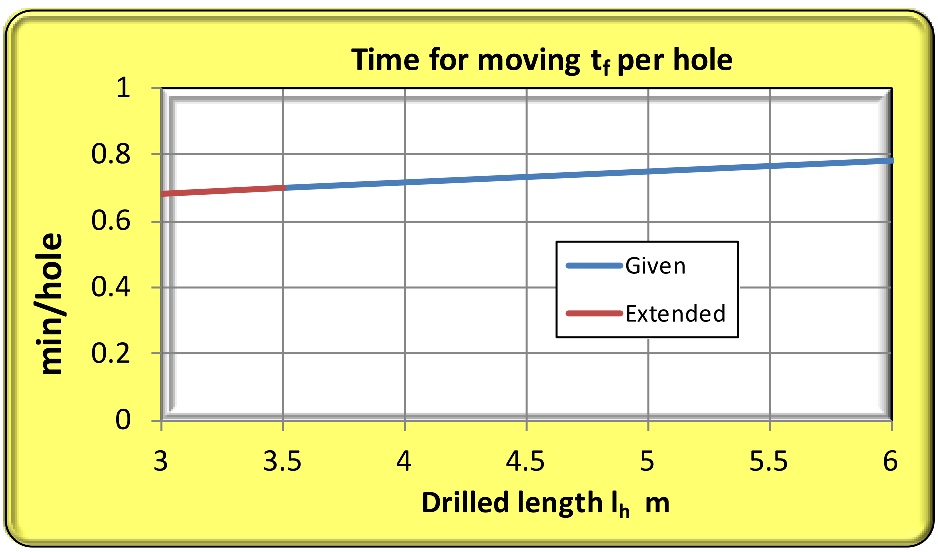
$v_g = v_b * k_{hv}$

Values of the correction factor k_{hv} for drill-hole diameters different from 48 mm are shown in the next chart:



$$T_f = \frac{t_f * (N_h + 2 * N_g)}{N_m}$$

t_f = Time for moving per hole.



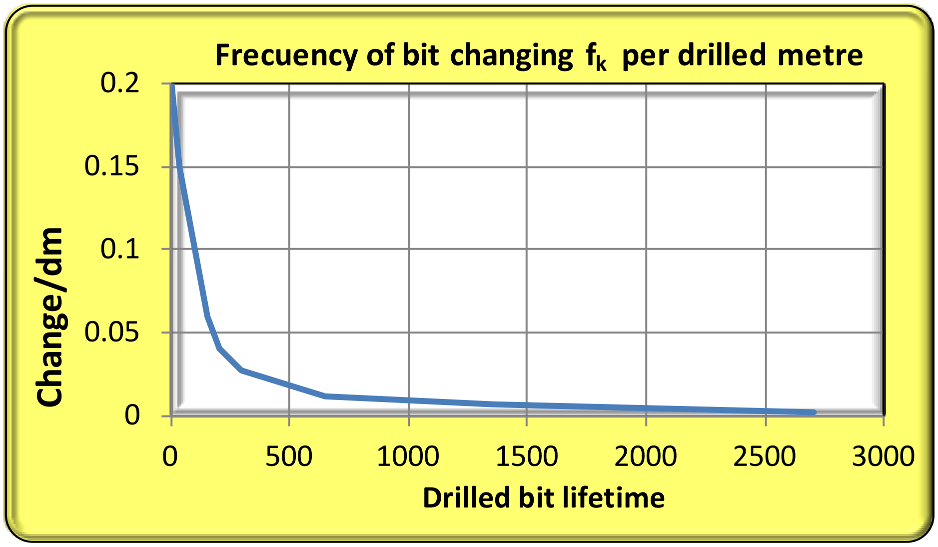
Changing of Bits:

$$T_K = \frac{l_h * (N_h + 2 * N_g) * f_k * t_k}{N_m}$$

t_k= Unit time for bit changing = 3 minutes.

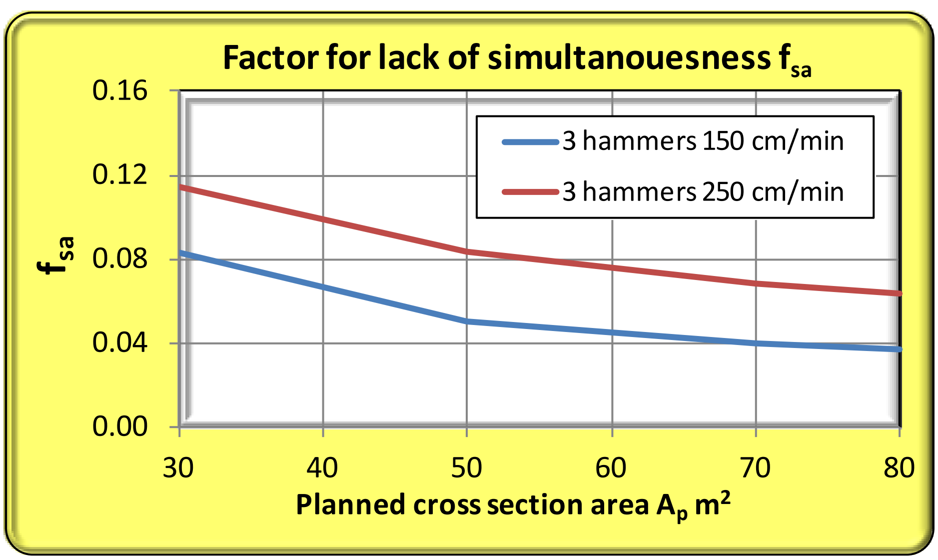
f_k = Frequency of bit changing factor per drilled meter.

Next chart shows f_k values depending on Drilled Bit Lifetime. Drilled bit lifetime equals to 200 has been adopted for both rock mass formations:



Lack of simultaneousness:

$$T_{sa} = f_{sa} * (T_h + T_g + T_f)$$

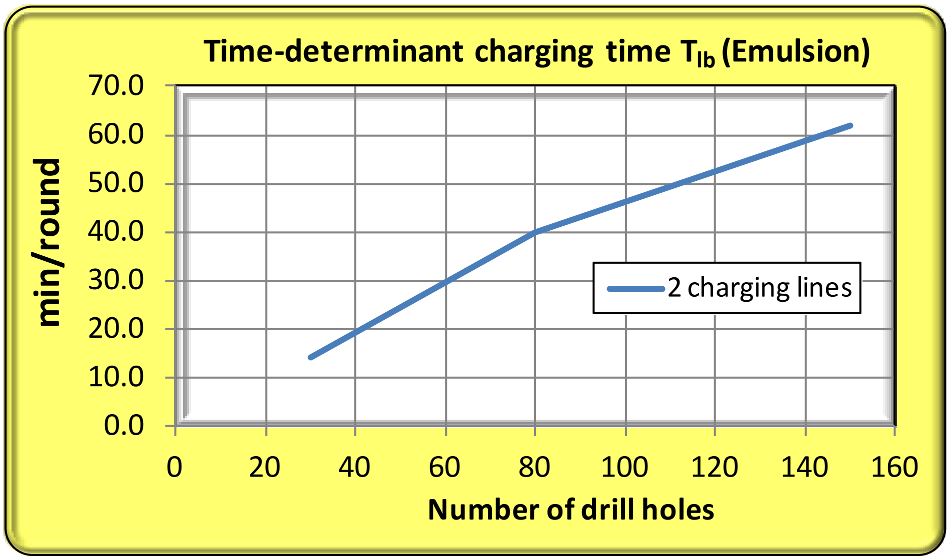


Charging time: $k_{II} * T_{lb}$

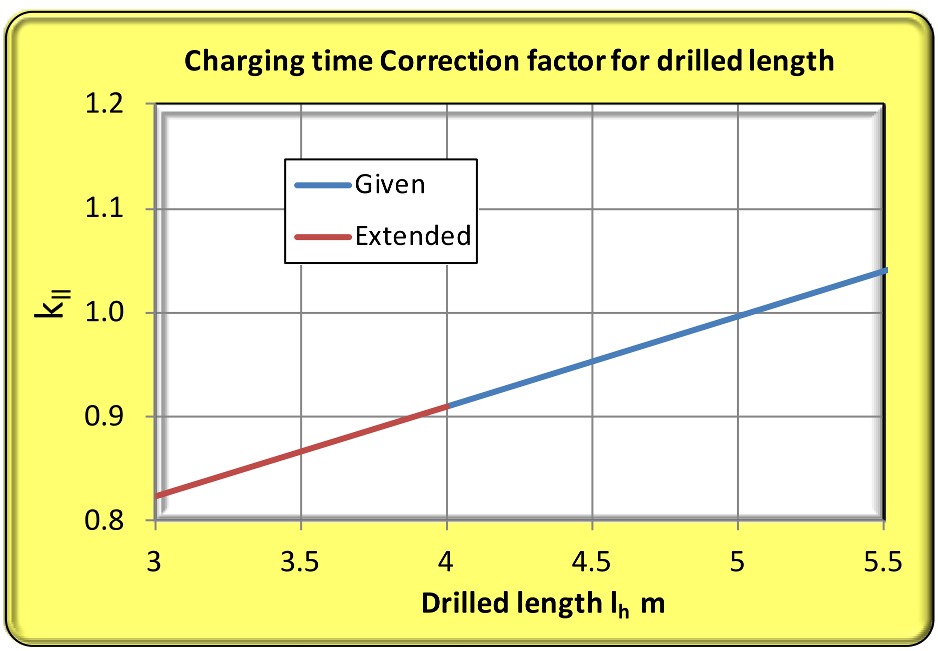
k_{II} = It is a correction factor for drilled length in charging time.

T_{lb} = Time determinant charging time without a correction factor.

The following chart shows T_{lb} values for emulsion and two charging lines as a function of different number of charged drillholes.



And the correction factor k_{II} as a function of drilled length is shown below.

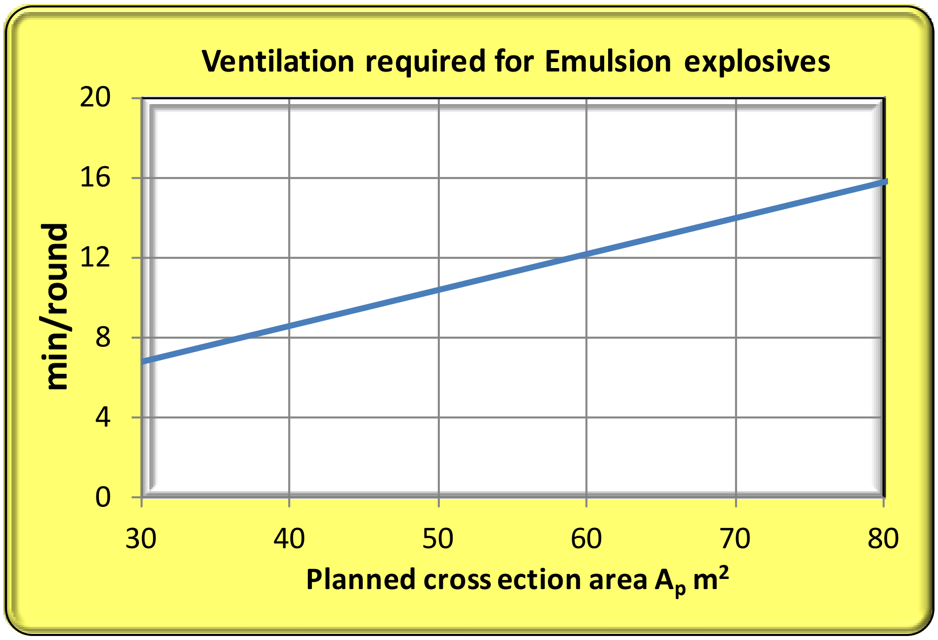


Note that the available information until 4 m drilled length was extended until 3 m drilled length in order to have a prognosis model for poor rock mass conditions, where the drilled length (and therefore the round length) is reduced.

Incidental Lost Time

Incidental lost time T_{tb} = 0.111*(operational Drilling + operational Charging + Rig time)

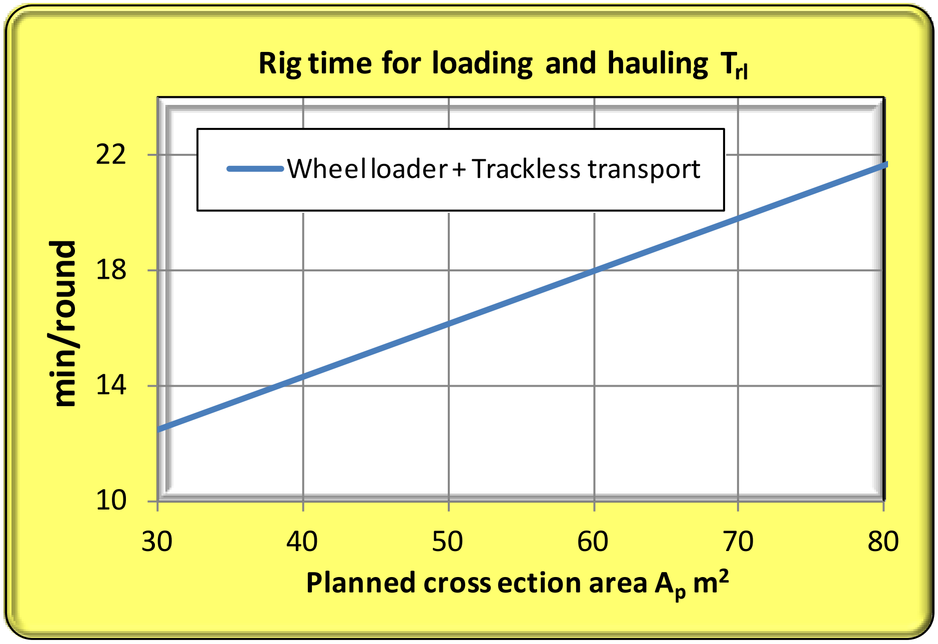
II Ventilation Break



III Loading and hauling:

Rig Time:

The rig time for Loading and hauling T_{rl} is shown in the next table for the relevant cross sections in this economic analysis:



Proportional Operational Time:

$$T_{lt} = \frac{V_r}{Q_l} * 60$$

V_r = Rock volume which includes overbreak.

Q_l = Normalized gross loading capacity.

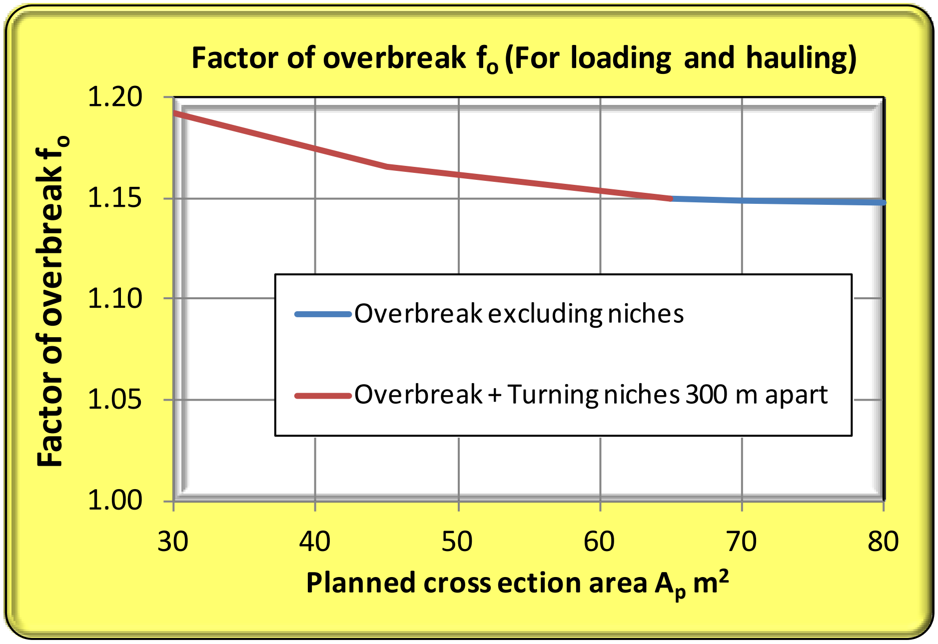
$$V_r = A_p * l_h * p_r * f_o$$

A_p= Planned tunnel cross section area.

p_r= Pull = 91% for 48 mm charged drill-hole diameter.

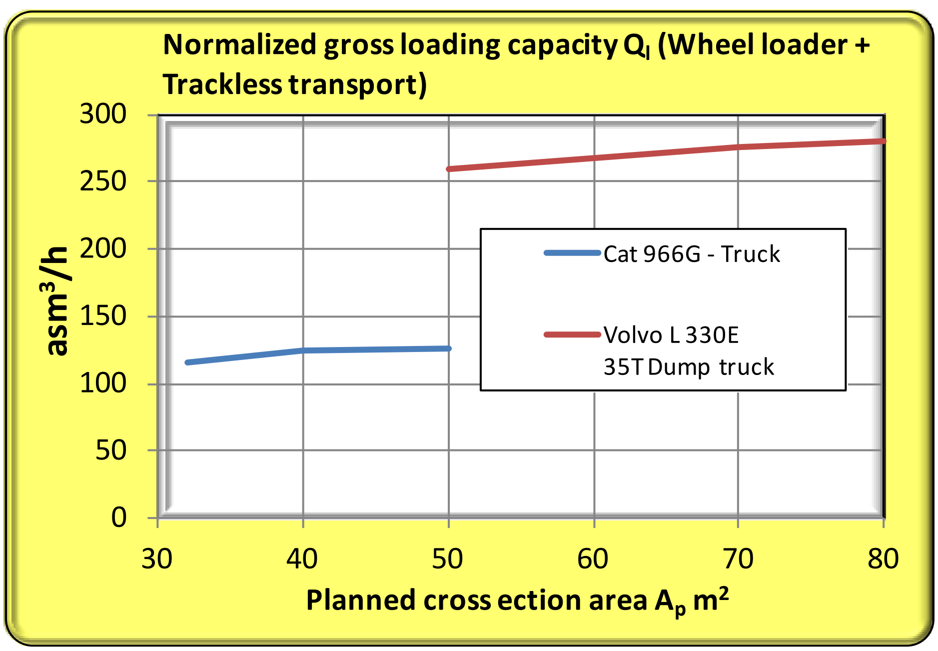
f_o= Overbreak factor.

The following chart shows the overbreak factor as a function of cross section area.



Note that the overbreak factor includes niches only for cross sections smaller than 65 m^2. It is important to point out that the overbreak factor here is for lauding and hauling purposes, taking into account the blasted cross section area with an invert being cleaned down and should not be confused with the overbreak considered for the hydraulic tunnel design which a lower overbreak is estimated as shown in the energy losses chapter (7.3.5).

The normalized gross loading capacity Q_l is shown for the relevant cross sections:



The advance rate prognosis model [11], provides more alternatives in between the two chosen loading and hauling combinations, but for practical reasons only these two were selected. It is important to highlight that the chosen alternatives quantify the fact that the larger the cross section, the faster is the capacity of moving blasted rock due to a different bucket size, truck trailer size, etc.

Incidental Time:

The incidental time T_{tl} is estimated as follows:

$$T_{tl} = 0.111 * (T_{lt} + T_{rl})$$

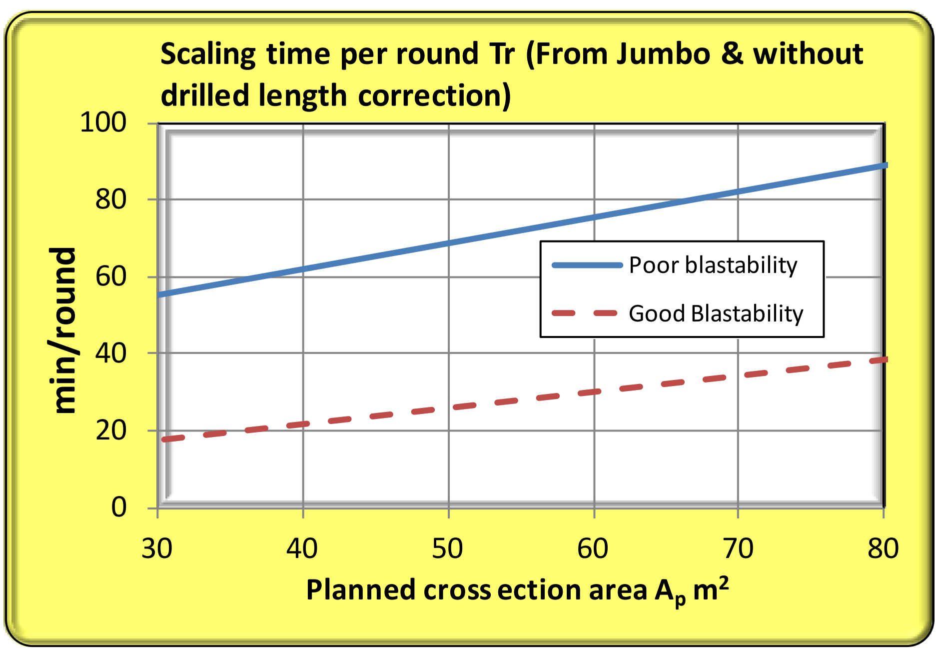
Scaling and Rock Support IV:

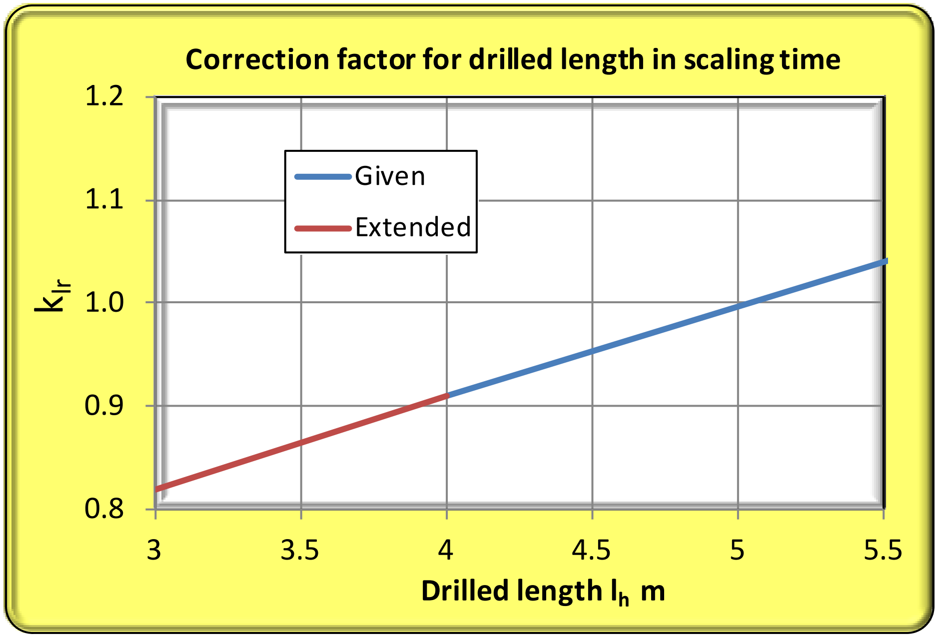
Scaling time: $T_r * K_{lr}$

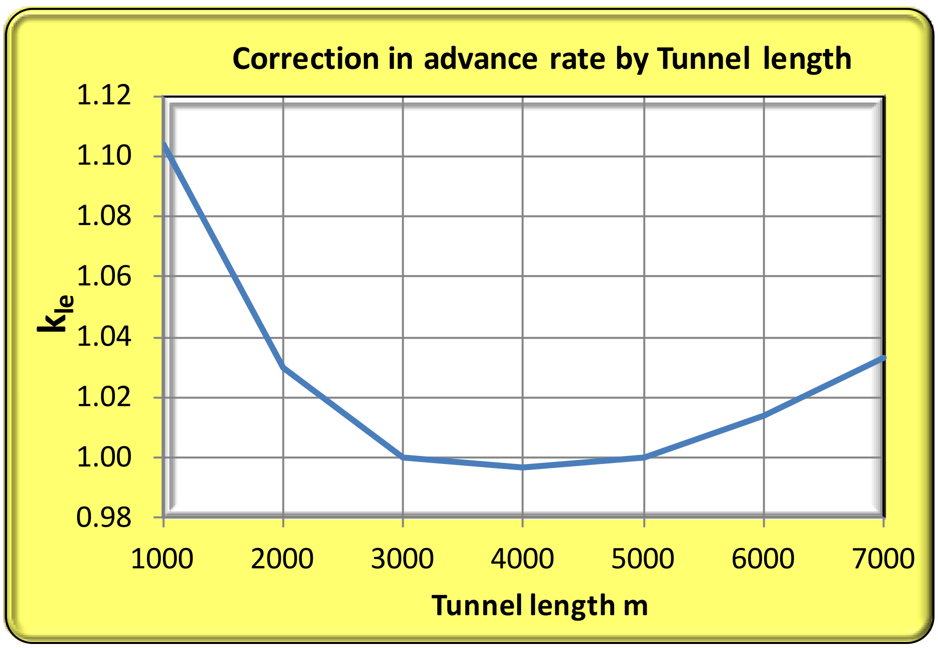
T_r = Scaling time per round without drilled length correction factor.

K_{lr}= Correction factor for drilled length in scaling time.

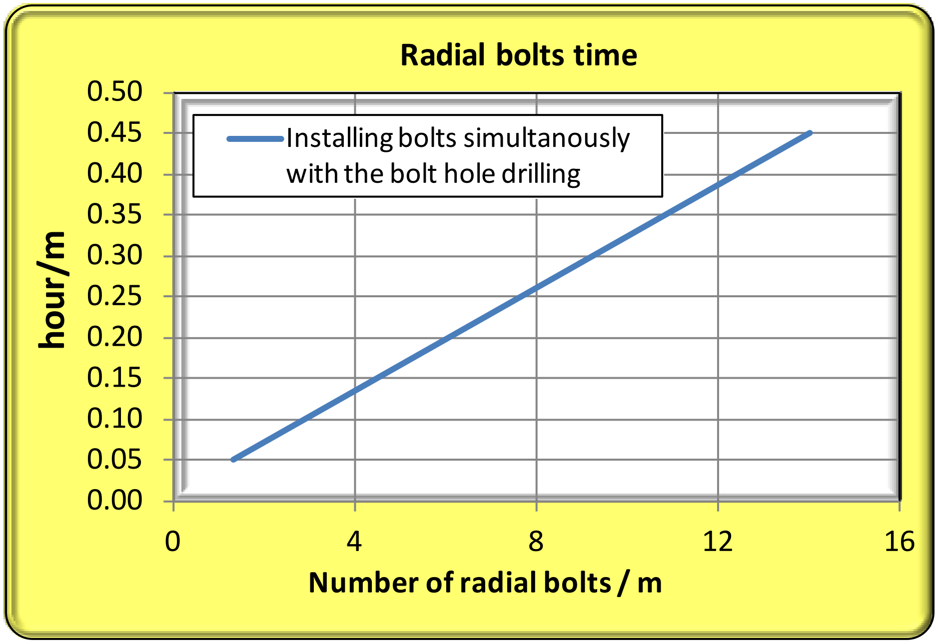
The following chart shows T_r values for the relevant cross section area and blastability. Only scaling from jumbo was considered, because scaling from pile is not recommended for the tunnel sizes under the economic analysis:

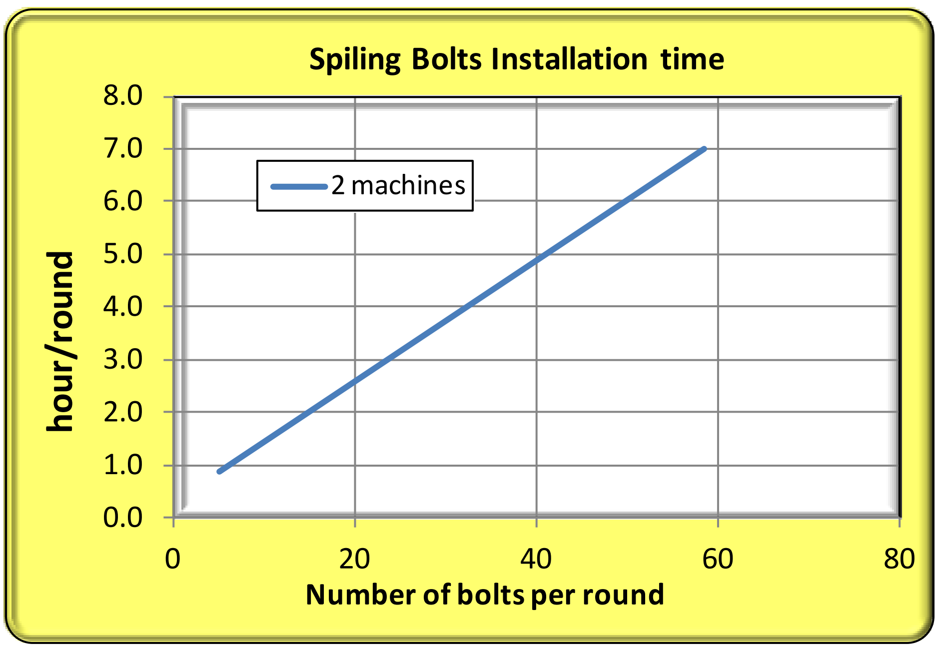


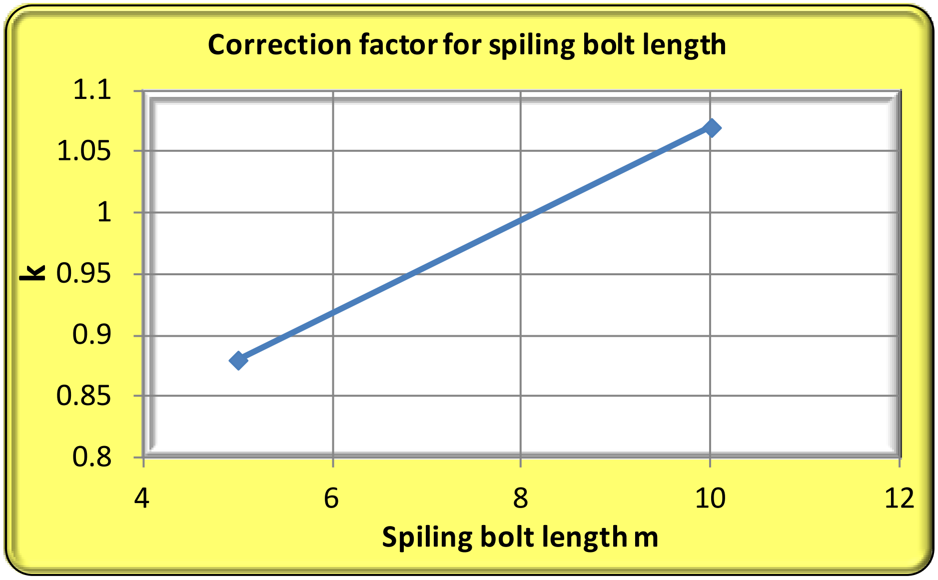


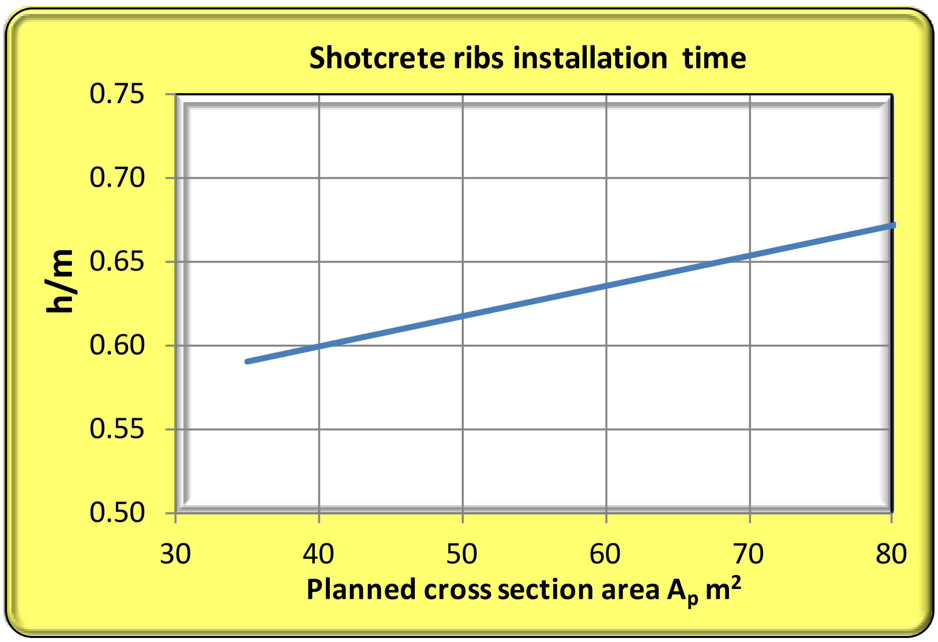


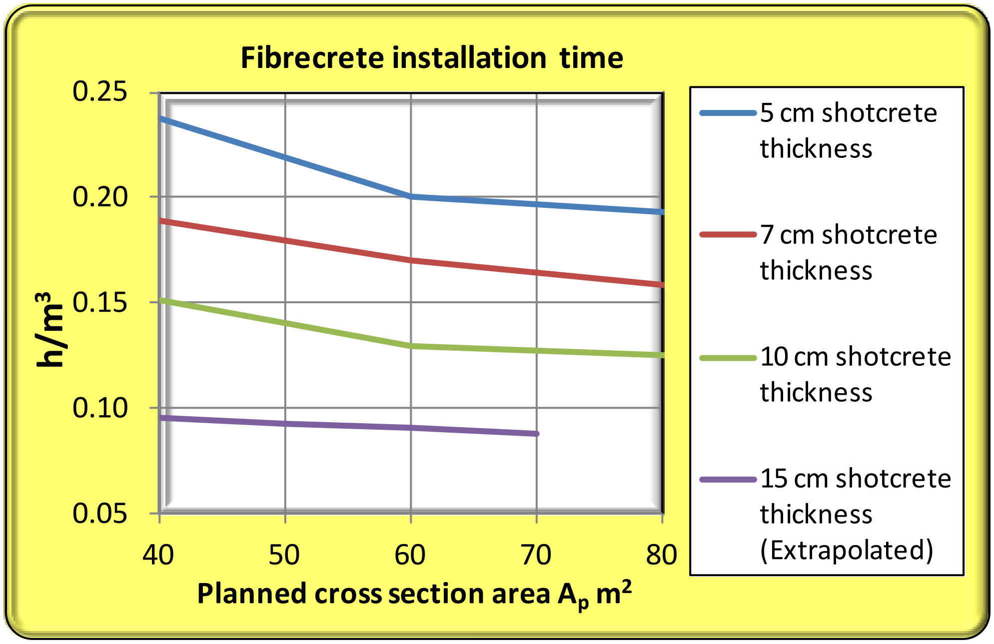
Rock support:

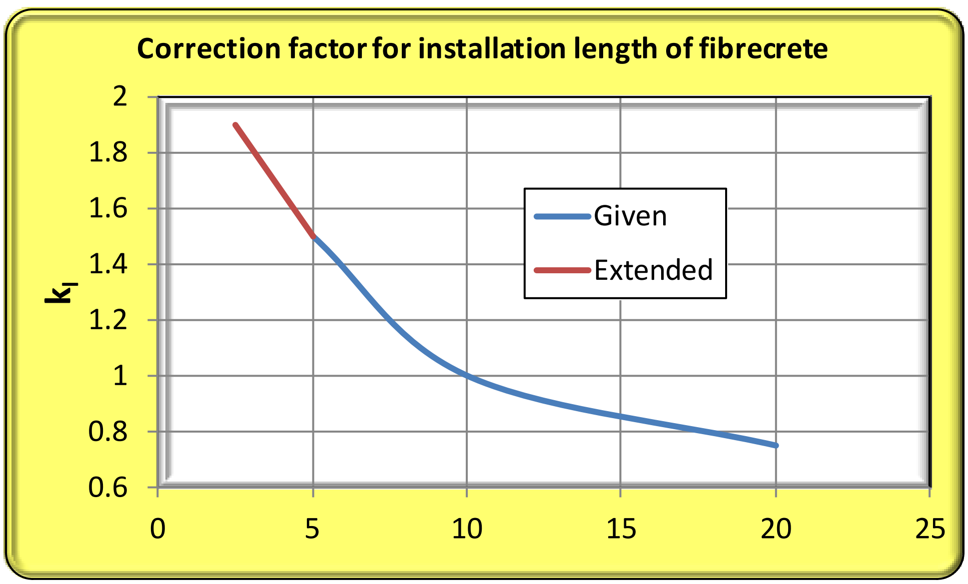

Radial bolts time
Installing bolts simultanously with the bolt hole drilling
hour/m
0.50
0.45
0.40
0.35
0.30
0.25
0.20
0.15
0.10
0.05
0.00
0
4
8
12
16
Number of radial bolts / m


Spiling Bolts Installation time
2 machines
hour/round
8.0
7.0
6.0
5.0
4.0
3.0
2.0
1.0
0.0
0
20
40
60
80
Number of bolts per round


Correction factor for spiling bolt length
k
1.1
1.05
1
0.95
0.9
0.85
0.8
4
6
8
10
12
Spiling bolt length m


Shotcrete ribs installation time
h/m
0.75
0.70
0.65
0.60
0.55
0.50
30
40
50
60
70
80
Planned cross section area Ap m2





7.3.5 Cost of Energy Losses

The cost of energy losses along the project lifetime and updated to the plant commissioning is estimated as follows:

$$\boldsymbol{E_L = \frac{g * \eta * q_d^3}{A_f^2 * M^2 * R_h^{4/3}} * T' * P_e * D \qquad USD/m}$$

g = Gravity force = 9.8 m/s^2.

η = Global efficiency, including turbine, generator and transformer efficiencies = 90%.

q_d = Power plant design discharge = 100 m^3/s.

M = Manning roughness coefficient.

A_f = Hydraulic cross section area in m^2.

R_h = Hydraulic radius in m.

T' = Utilization time for head losses in hours.

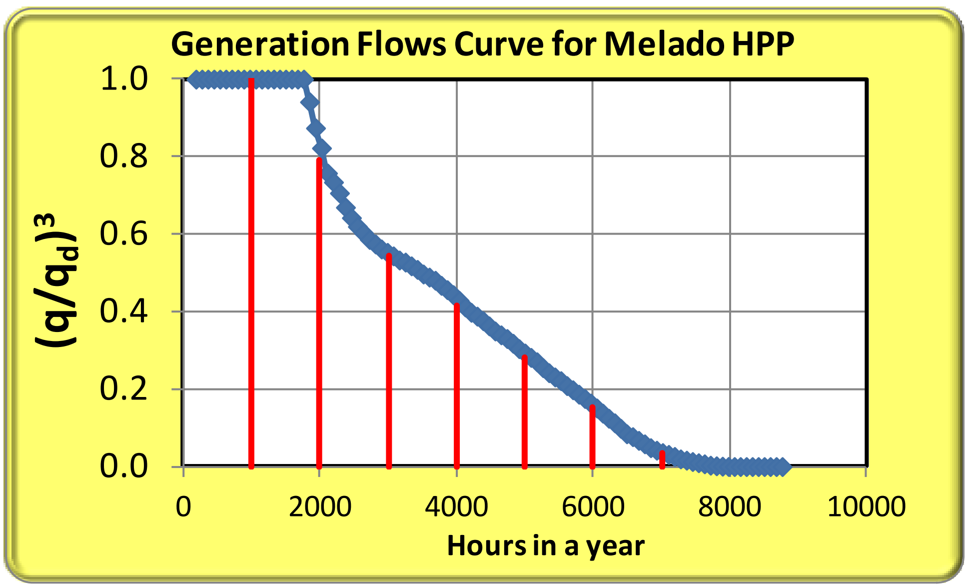
P_e= Long term Energy price = 0.065 USD/KWh.

D = Capitalization factor.

T' is estimated by the following equation:

$$T' = \int_0^T \left(\frac{q}{q_d}\right)^3 dt$$

Graphically, it is the hatched area in the following chart:



T'= 3681 hours.

The capitalization factor D can be estimated as:

$$D = \frac{(1+i)^n - 1}{i * (1+i)^n}$$

Where i is the interest rate and n is the project lifetime period. If one considers an annual interest rate of 10% with a project lifetime of 40 years, the capitalization factor is:

Annual Interest Rate	i	%	10%
Economic life time	n	year	40
Capitalization factor	D		9.78

7.3.5.1 Manning Roughness Value & Hydraulic Cross Section Area

A thorough analysis is carried to estimate the energy losses in the two tunnels and particularly the Manning roughness coefficient value and the final hydraulic area for different tunnel stretches. These stretches are defined by the rock mass quality and their corresponding rock support required to improve stability which leads to different Manning roughness coefficient and hydraulic areas throughout the tunnel.

Particularly, hydraulic flow area and Manning roughness value will be obtained for unlined, shotcrete lining and shotcrete ribs tunnels, but this Manning roughness value is not the final one because it must be weighed with the concrete invert in all cases.

Einstein equation is used to estimate the equivalent Manning coefficient for the total cross section:

$$M_{GLOBAL} = \left(\frac{\sum P_i}{\sum \frac{P_i}{M_i^{3/2}}} \right)^{2/3}$$

P_i = Wetted perimeter with related to a Manning roughness M_i.

M_i = Manning roughness coefficient for the wetted perimeter P_i.

7.3.5.1.1 Unlined Tunnel

Manning Roughness Coefficient

The blast design plays a huge role in the Manning roughness coefficient when no rock support is required and the tunnel contour is left unlined. Among the most influential parameters that govern the resulting tunnel roughness after blasting are:

Rod length (Eccentricity).

Specific drilling (Drillhole spacing in contour)

Charging (Tunnel contour)

The rod length has an important role in the Manning roughness coefficient. As it is said in reference [10], certain deviation at the bottom of contour drllholes (eccentricity) measured from the planned cross section is unavoidable since some space is required at tunnel boundary to drill these holes. The problem is illustrated in the following picture:

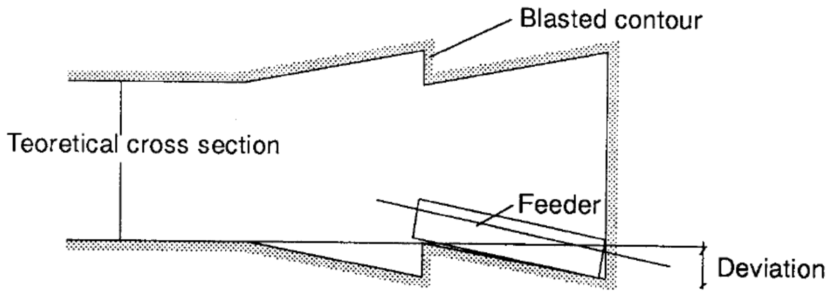
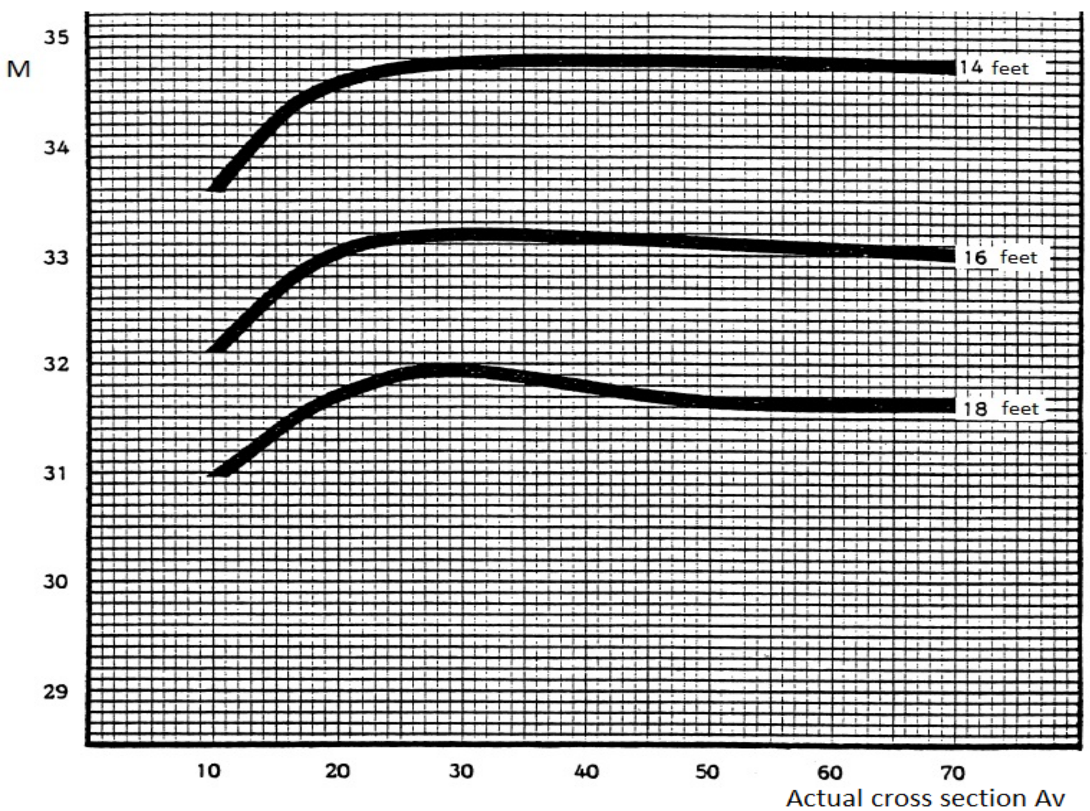
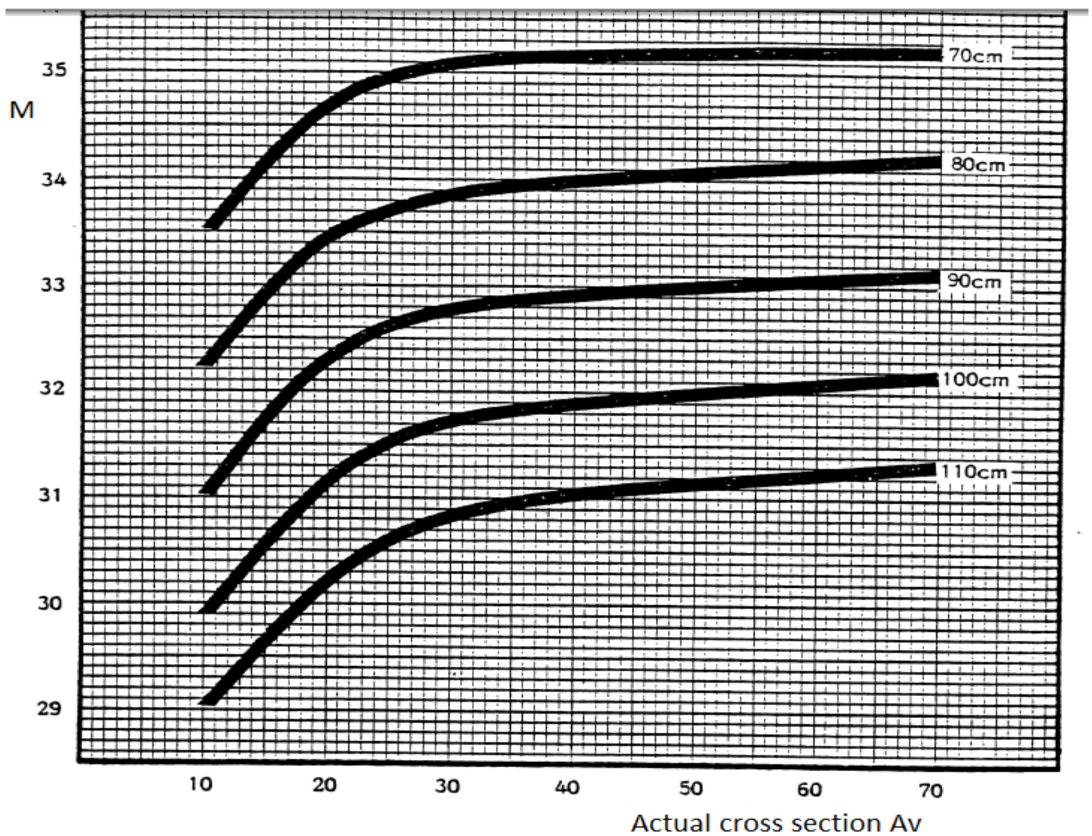


Image from PHd Thesis (Zare, S.) [12]: Drill & Blast Tunnelling

A longer rod length will lead to a higher eccentricity and the saw-toothed shape at blasted contour will be more marked. Therefore, the longer the rod length the rougher the contour. This conclusion was demonstrated in the Norwegian report "Head Losses in power plant tunnels" (*Falltap I Kraftverks Tunneler Rapport 1985)***:**

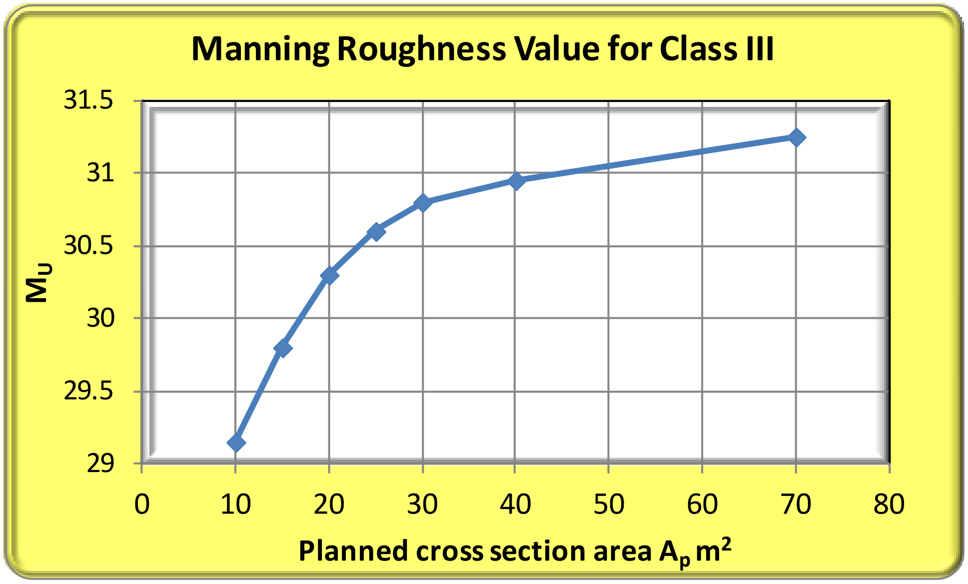


Specific drilling has also an impact on contour roughness, particularly in Contour holes and Row Nearest Contour holes. The following chart shows a clear correlation between the contour hole spacing and Manning roughness coefficient for different cross sections:



Finally, charging also has an effect on Manning roughness, because if tunnel contour is heavily loaded, a considerable overbreak and a rough contour is achieved.

Even though, one could have a good correlation between each blasting design variable and the Manning roughness coefficient individually, it is not enough to estimate the resulting Manning roughness for a set of different chosen values in the blast design. But, in the same Norwegian report, it was given a classification of different classes, where a set of relevant variables are included with different specifications depending on the class type I, II or III. Class III was chosen for all the range of different cross sections included in the economic analysis which is the least demanding class alternative in terms of measures for improving contour roughness. The Manning roughness coefficient in unlined tunnels M_u for different tunnel cross section areas is given in the next chart for class III:



Class III Specifications:

Rod length: 18 feet (5.49 m).

Charging Pattern:

The next table summarizes the charging pattern for the drillholes depending on where they are located:

Invert	Cut	Easers	Row nearest contour	Contour	Charging Pattern Class 3
90%	90%	70%	70%	70%	Charged length (% of total drilled length)
100%	100%	100%	70%	40%	Charging Density (% of drilled area within charged length)

Drilling pattern:

Drillhole diameter = 48 mm

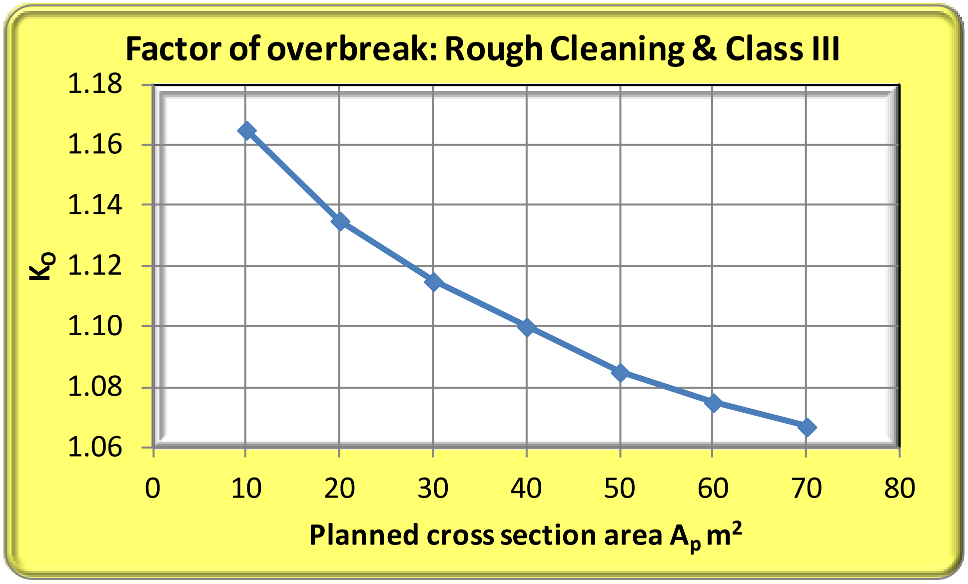
Type of hole		Burden, *V*	Spacing, *E*
Contour	*Good blastability*	0.8 - 1.0 m	0.7 - 1.0 m
	Poor blastability	0.7 - 0.9 m	0.6 - 0.9 m
Row nearest contour	*Good blastability*	1.0 m	1.1 m
	Poor blastability	0.9 m	1.0 m
Invert hole	*Good blastability*	1.0 m	1.0 m
	Poor blastability	0.8 m	0.8 m
Easer	*Good blastability*	F_s = 1.8 m^2	
	Poor blastability	F_s = 1.3 m^2	

The burden and spacing values for the contour are given as intervals where the lowest value applies for a 20 m^2 cross section and the highest one for 120 m^2. It means that for a cross section of 70 m^2 in good blastability condition, burden should be around 0.9 m and spacing should be around 0.85 m.

It is important to point out that the Manning roughness coefficient estimation undertaken below does not apply well in rocks with thin foliation or bedding and in cases of heavily fractured rock, but they are not the case for most of these two tunnels under analysis.

Hydraulic Area

The hydraulic design cross section area should include the overbreak for a fair economic evaluation, but not all the overbreak is used by the water for conveyance in unlined tunnels. Water is not able to follow sudden changes throughout the tunnel contour, and therefore the effective flow area is less than the total overbreak. The design hydraulic overbreak is estimated as 75 % of total overbreak [14]. The following chart shows for rough cleaning tunnel invert and class III blast design, the hydraulic overbreak:



The chart above is for rough cleaning invert and unlined walls and roof. As it was mentioned before, the invert will be concrete lined and therefore the resulting hydraulic area A_{V_1} is reduced to:

$$A_{V\_1} = K_o * A_p - t_{INVERT} * L_{INVERT}$$

A_p = Planned cross section area.

t_{INVERT} = Invert thickness equals to 30 cm = 15 cm concrete invert + 15 cm granular soil.

L_{INVERT}= Invert with.

$$L_{INVERT} = D_1 * \left(\sqrt{3} - 1\right)$$

D_1 = Unlined tunnel diameter related to the cross section areaA_{V_1}.

7.3.5.1.2 <u>Shotcrete</u>

Manning Roughness Coefficient

Shotcrete lining over an unlined tunnel causes a smoother tunnel contour and a reduction in cross section area. Elfman,S. (1997) provides a procedure on how shotcrete lining influences the head losses in water tunnels. This is achieved by obtaining a new v friction factor f value which includes the shotcrete effect. The new Manning roughness value is obtained by:

$$M_s = \sqrt{\frac{124.45}{f*(4*R_h)^{1/3}}}$$

f = Friction factor with shotcrete.

R_h = Hydraulic radius m.

And the friction factor for shotcrete is:

$$f = 0.03 + (f_o - 0.03) * \alpha^Y$$

f_o = It is the friction factor of the tunnel before shotcreting.

α = It is the ratio between area after and before shotcreting.

Y = It is an empirical coefficient that corrects the roughness differences measured in two different cases.

Once it is known f_o, it is possible to know the Y value by means of a linear interpolation between the following values

f_o	Y
0.11	10
0.0755	52

A distinction is made between the resulting Manning value from shotcrete in roof only M_{S_2} and the Manning value from shotcrete in roof and walls M_{S_1}. The difference relies on the final cross section area and therefore α value is different.

7.3.5.1.3 Hydraulic Area

The cross section area with shotcrete has a smaller diameter D_2 that can be estimated as:

$$D_2 = D_1 - 2 * t_{SHOTCRETE}$$

D_1 = Unlined tunnel diameter with concrete invert.

$t_{SHOTCRETE}$ = Shotcrete thickness.

And the resulting cross section area with shotcrete is obtained by the new diameter D_2 (Horse shoe tunnel shape):

$$A_{V\_2.1} = \frac{(D_2)^2}{2} * \left[\frac{7 * \pi}{12} - 1 + \frac{\sqrt{3}}{2}\right]$$

If shotcrete is only placed on roof the new hydraulic area is:

$$A_{V\_2.2} = A_{V\_1} - \frac{\pi * D_1 * t_{SHOTCRETE}}{2}$$

7.3.5.1.4 Shotcrete ribs

Manning Roughness Coefficient

One inconvenience of shotcrete ribs in water tunnels (And particularly for hydropower purposes) is its higher roughness on surface contour caused by the ribs.

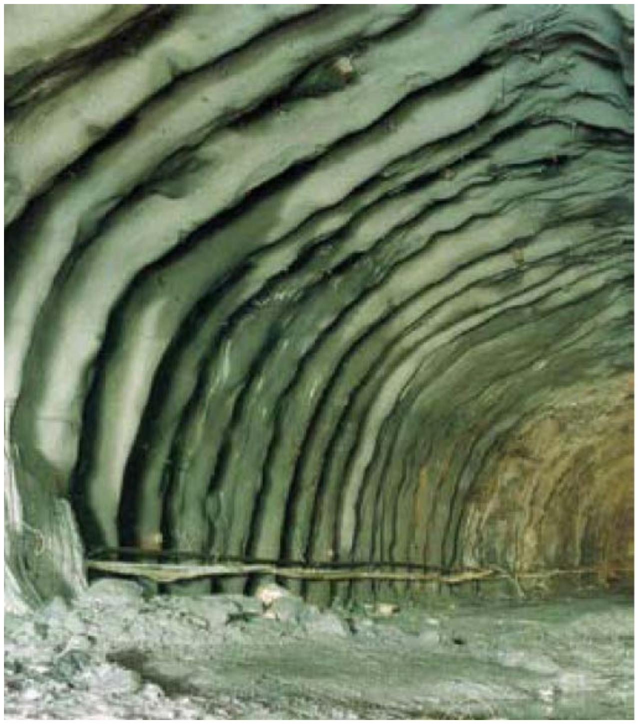
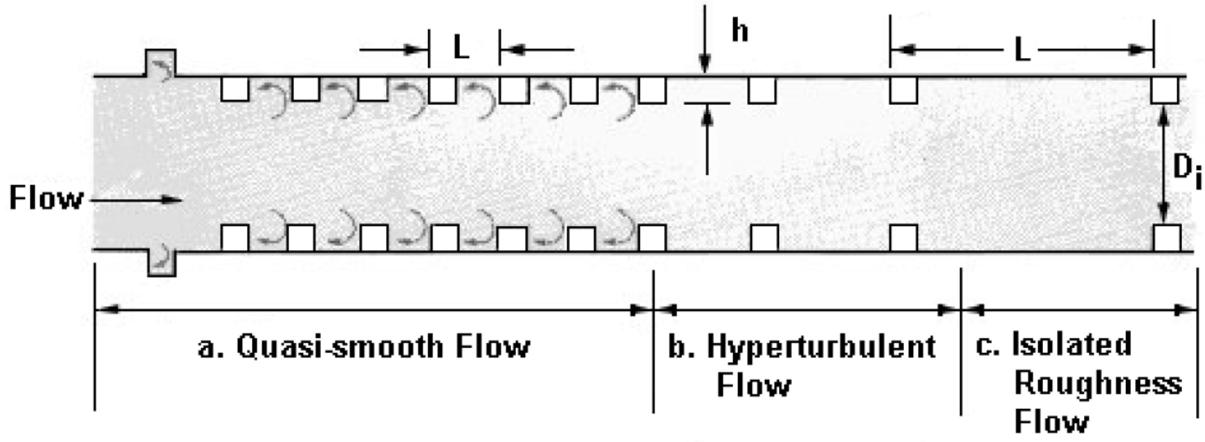


Figure 7-3: Resulting tunnel contour when shotcrete ribs are applied.

Research analysis on this matter is still underway, but there are some approaches that have been applied for other purposes and seem to match quite well in shotcrete ribs energy losses. One example is the analysis carried out by the US Federal Highway Administration in an effort to have a better understanding of internal roughness elements placed within a culvert as an energy dissipator. This analysis is at least sensitive to roughness element spacing (shotcrete rib spacing) and roughness element height (rib thickness).

The following illustration given by Morris (1963) shows the different hydraulic regimes depending on the two variables mentioned above:



The shotcrete ribs resemblance to the previous figure is the roughness element which has a height h corresponding to the coating of rebars in shotcrete ribs and the spacing of roughness elements representing the ribs spacing. Finally, the culvert surface without roughness elements is equivalent to the fibercrete lining in shotcrete ribs before the installation of rebars.

In practice, only two different types of reinforced shotcrete ribs are considered for rock support in weakness zones:

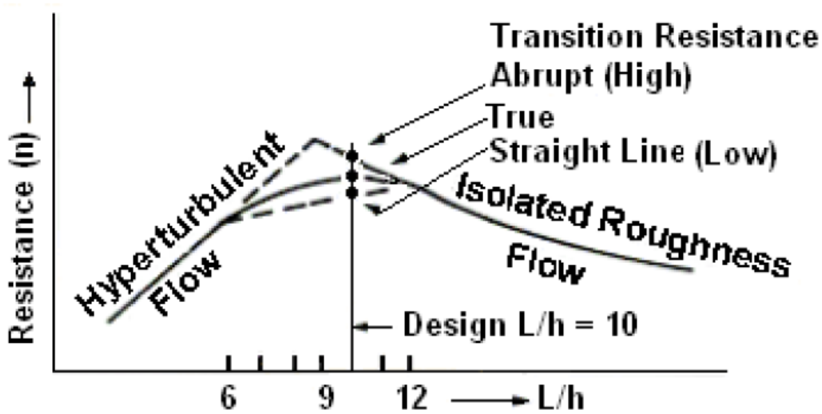
1- D40/4 c/c 1.2 = 4 rebars in double layer in 40 cm thick lining with c/c spacing 1.2 m.

2- D35/5 c/c 2.3 = 5 rebars in double layer in 35 cm thick lining with c/c spacing 2.3 m.

In both cases, a first fibercrete lining thickness of 15 cm is applied. Therefore the first case will have a shotcrete coating over rebars equals to 25 cm and for the second case equals to 20 cm. These latter values are considered as the height h of the ribs for each case.

Quasi Smooth Flow suits for roughness element with a height in an order of magnitude similar to the roughness element spacing (L/h < = 2) which is not the case of reinforced shotcrete ribs.

The following picture provides a distinction between “Hyper-turbulent flow” and “Isolated roughness flow” in box culverts which is different from circular culverts, but the behavior difference with regard to resistance as a function of roughness element spacing is the same.



As it is shown in the previous chart and taking into account a fixed height h, a longer spacing between rings (or ribs) in a “Hyper-Turbulent Flow” triggers a higher resistance in contrast to what happens in the “Isolated Roughness Flow” where longer ring (rib) spacing causes a lower resistance. Therefore, it is very important to identify what flow situation is governing. The same document suggests for design purposes to take the lowest Manning roughness n (1/M) value resulting between “Hyper-Turbulent Flow” and “Isolated Roughness Flow”. The way to estimate those n values are given below:

Hyper-turbulent flow n_{HT} value:

$$n_{HT} = \frac{\alpha D_i^{1/6}}{2\log\left(\frac{r_i}{L}\right) + 1.75}$$

Each variable is described below and between parentheses is given the equivalence in reinforced shotcrete ribs:

α = Constant factor = 0.0898 (SI)

D_i = Inside diameter of roughness rings (Inside diameter of roughness ribs).

L = Roughness element spacing (Roughness rib spacing).

r_i = Culvert radius based on the inside diameter of roughness rings (ribs) measured from crest to crest.

Note that in Hyper-turbulent flow regime, the resulting n_{HT} value is independent of culvert surface roughness (Or in the case of shotcrete ribs, the n_{HT} value is independent of fibercrete lining).

Isolated-Roughness flow n_{IR} value:

$$n_{IR} = n\left(\frac{D_i}{D}\right)^{1/6}\left(1+67.2C_D\left(\frac{L_r}{P}\right)\left(\frac{h}{L}\right)\right)^{1/2}$$

The variables are explained below with the equivalence in shotcrete ribs between parentheses

C_D = Drag coefficient for the roughness shape = 1.9 for sharp edge rectangular shape.

L_r = Total peripheral length of roughness elements. (Since shotcrete rib is not placed in invert L_r represents the walls and roof perimeter only).

P = Total wetted perimeter, which means L_r plus invert width.

D = Nominal diameter of the culvert. (Tunnel diameter with only fibercrete lining)

n = Manning roughness coefficient value for the culvert surface without roughness rings. (In Shotcrete ribs, n is the Manning roughness coefficient value with fibercrete lining and without the rebars along with their coating of shotcrete).

The culvert surface Manning roughness (Equivalent to the Tunnel with only fibercrete lining in shotcrete ribs) value n specifically for the purpose of estimating the "Isolated roughness flow" was set to 0.015 (Or 66.7). In box culverts analysis of roughness element effect, reference [8] provides a warning about the maximum value that can be assigned for culvert surface in Isolated roughness flow where the total friction comes from both the roughness elements (In this case ribs) and friction on the culvert surface (In this case tunnel with fibercrete lining only). This Manning boundary value is 0.015 for culvert surface, because the equation was obtained for smooth culvert surface. Fibercrete lining in tunnel will never reach this smooth limit, but in order to use this equation, the boundary value was used. Behind this assumption, there is a convincement that friction force from ribs is much more important than roughness in between them.

Finally, the Manning roughness coefficient in shotcrete ribs M_R is the minimum value between $M_{HT} = \frac{1}{n_{HT}}$ and $M_{IR} = \frac{1}{n_{IR}}$.

7.3.5.1.5 Hydraulic Cross Section Area:

The cross section area with fibercrete has a smaller diameter D_3 that can be estimated as:

$$D_3 = D_1 - 2 * t_{FIBERCRETE}$$

And the resulting cross section area with fibercrete is A_{v_3}

$$\boldsymbol{A_{v\_3} = \frac{(D_3)^2}{2} * \left[\frac{7 * \pi}{12} - 1 + \frac{\sqrt{3}}{2}\right]}$$

7.4 Economic Evaluation

Investment Cost is the sum of the excavation cost and the rock support cost.

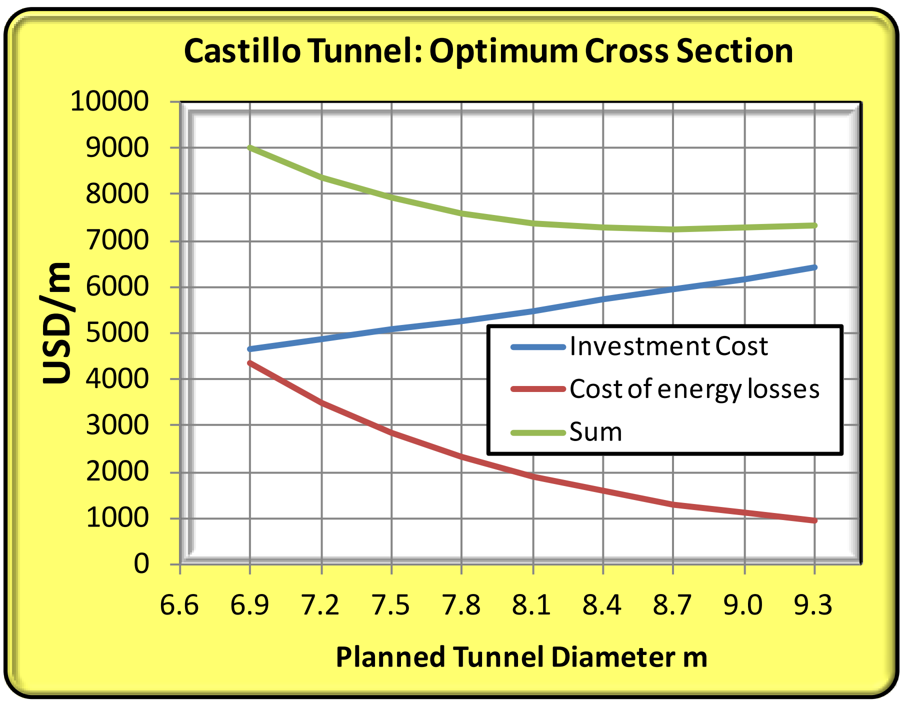
The tunnel completion is 1 year in advance of plant commissioning.

The construction period is out of the critical path.

7.5 Results

7.5.1 Castillo Tunnel

The optimum cross section for Castillo tunnel based on the methodology exposed here is 64.3 m^2 (8.7 m diameter for a horse shoe tunnel shape). The following chart shows the investment cost updated to the plant commissioning and the cost of energy losses for the same date. Also it is shown the sum of the latter two costs in order to indicate where the cross section, which minimizes that sum, is located.



The details of the last chart are shown below:

Castillo Tunnel											
Planned Cross Section area	A_p	m^2	40.4	44.0	47.8	51.7	55.7	59.9	64.3	68.8	73.5
Planned Diameter	D_p	m	6.9	7.2	7.5	7.8	8.1	8.4	8.7	9.0	9.3
Gross Investment Cost	I_g	USD/m	3,549	3,713	3,881	4,026	4,188	4,353	4,522	4,684	4,865
Planning & Administration 20%	I_{pa}	USD/m	710	743	776	805	838	871	904	937	973
Gross Investment Cost + Planning & Administration	$I_g + I_{pa}$	USD/m	4,259	4,456	4,657	4,831	5,025	5,224	5,427	5,620	5,838
Tunnel Construction period	n_{TRI}	Trimester	4.3	4.5	4.7	4.2	4.3	4.5	4.6	4.8	4.9
Investment Cost updated at commencement	I	USD/m	4,650	4,875	5,107	5,264	5,486	5,712	5,945	6,168	6,419
Cost of energy losses	E_L	USD/m	4,358	3,496	2,833	2,314	1,901	1,574	1,311	1,099	1,119
Cost sum	$I + E_L$	USD/m	**9,008**	**8,371**	**7,940**	**7,579**	**7,387**	**7,287**	**7,256**	**7,267**	**7,538**

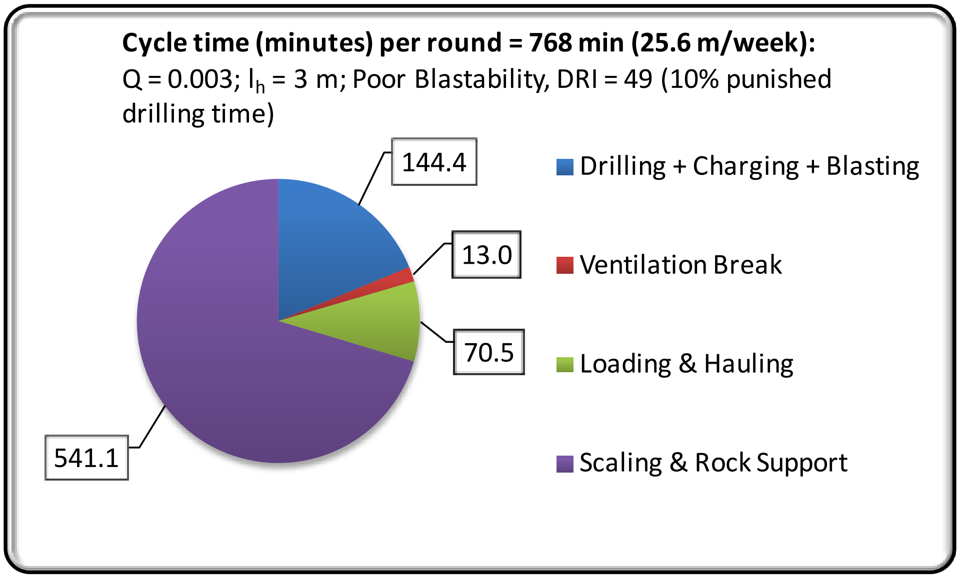
Note that there is a drop in Tunnel construction period between 47.8 m^2 and 51.7 m^2 and this is due to different technology equipments considered for cross sections larger than 50 m^2

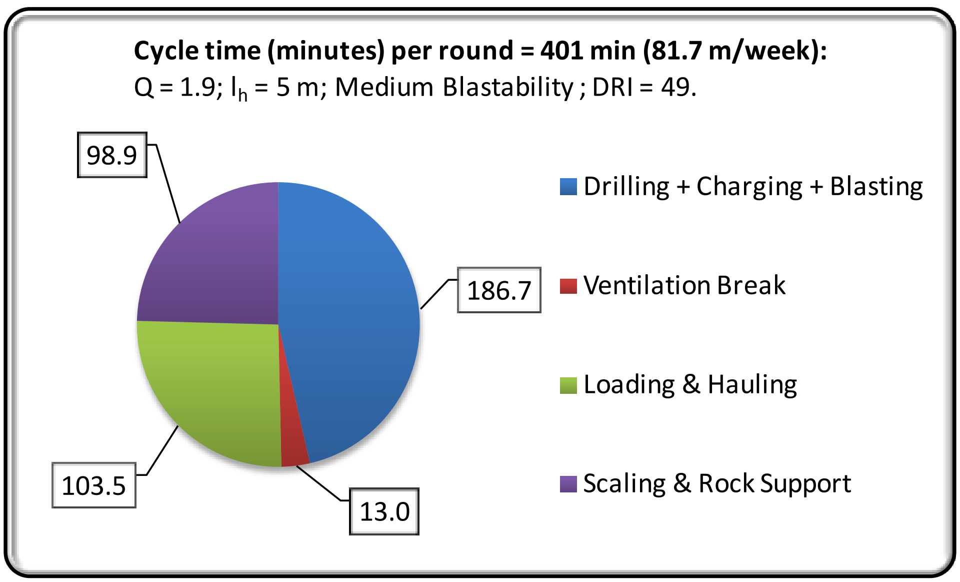
7.5.1.1.1 Advance rate:

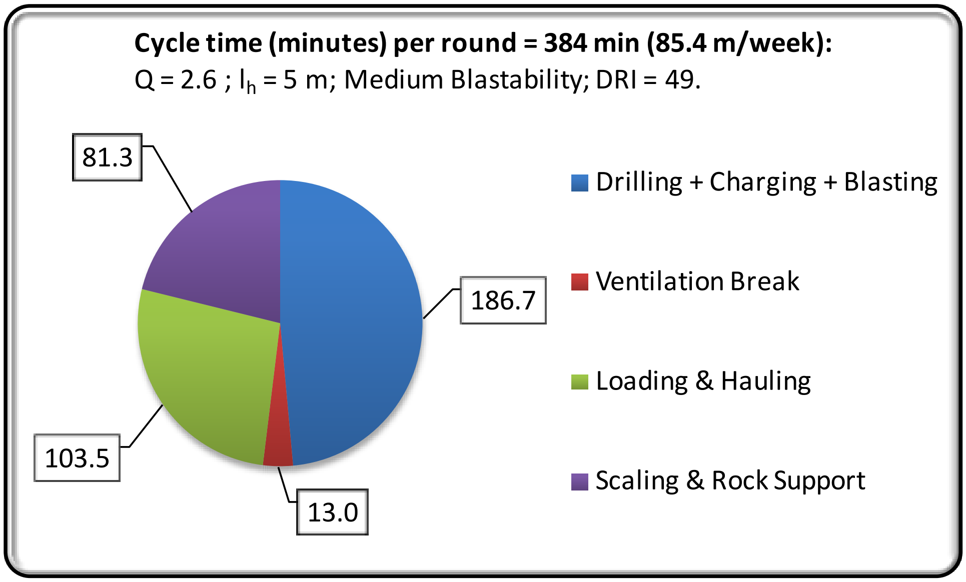
The following table shows the different situations where the advance rate was estimated for the Castillo Tunnel:

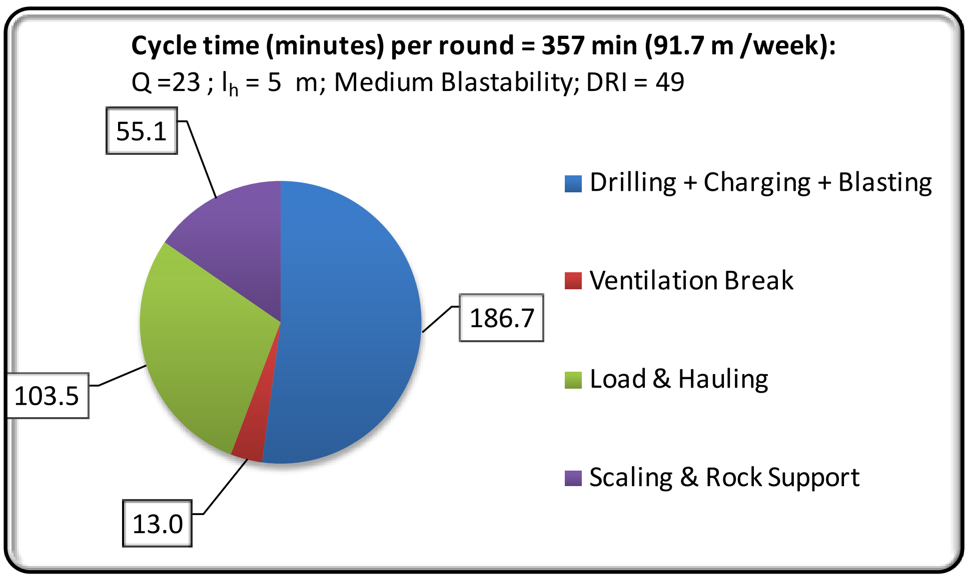
Castillo tunnel: 8.7 m diameter (64.3 m^2)			
Rock Mass Classification Q_{ROOF}	Blastability	Drilling Time punishment fractured rock	Drilled length m
0.003	Poor	1.1	3
1.9	Medium	1.0	5
2.6	Medium	1.0	5
23	Medium	1.0	5

And the details are given in the following four charts:


Cycle time (minutes) per round = 768 min (25.6 m/week):
Q = 0.003; l_h = 3 m; Poor Blastability, DRI = 49 (10% punished drilling time)
144.4
13.0
70.5
541.1
Drilling + Charging + Blasting
Ventilation Break
Loading & Hauling
Scaling & Rock Support


Cycle time (minutes) per round = 401 min (81.7 m/week):
Q = 1.9; l_h = 5 m; Medium Blastability ; DRI = 49.
98.9
186.7
103.5
13.0
Drilling + Charging + Blasting
Ventilation Break
Loading & Hauling
Scaling & Rock Support





Before tunnel construction starts, access road and portal must to be finished before. As it was mentioned in the advance rate assumptions, each portal construction takes 1 month and the corresponding access road construction time to each portal is given in the following two tables where “Castillo 1” represents the tunnel stretch that is constructed from the upstream end (or from Castillo brook) of Castillo tunnel and “Castillo 2” represents the tunnel stretch excavated from the downstream end (or Vallical brook):

Access to Castillo 1 portal		
Ustream end	150	m
Road advance rate	50	m/day
time elapsed	3.0	day
Time elapsed	0.1	month

Access to Castillo 2 portal		
Ustream end	331	m
Road advance rate	50	m/day
time elapsed	6.6	day
Time elapsed	0.2	month

Castillo tunnel: 8.7 m diameter (64.3 m^2)								
Rock Mass Classification Q_{ROOF}	Identification of stretch	Chainage m	Stretch Length m	Advance rate m/week	Advance rate month/stretch	Cumulative time elapsed Castillo 1 month	Cumulative time elapsed Castillo 2 month
0.003	Surface weathering	1700 - 1850	150	25.6	1.4	2.5	
1.9	Fair orientation - Poor/Fair Andesite	1850 - 3900	2050	81.7	5.9	8.3	
2.6	Good Orientation - Poor/Fair Andesite	3900 - 4330	430	85.4	1.2	9.5	
0.003	Inferred fault zone around Las Mulas brook	4330 - 4430	100	25.6	0.9	10.4	
2.6	Good Orientation - Poor/Fair Andesite	4430 - 5350	920	85.4	2.5	12.9	
0.003	Inferred fault zone around Las Yeguas brook	5350 - 5450	100	25.6	0.9	13.8	
2.6	Good Orientation - Poor/Fair Andesite	5450 - 6380	930	85.4	2.5	13.9	13.9
0.003	Inferred fault zone I from aerial photos	6380 - 6480	100	25.6	0.9		11.4
2.6	Good Orientation - Poor/Fair Andesite	6480 - 7670	1190	85.4	3.3		10.5
0.003	Inferred fault zone II from aerial photos	7670 - 7770	100	25.6	0.9		7.2
23	Good Orientation - Good Andesite	7770 - 9400	1630	91.7	4.1		6.3
0.003	Surface Weathering	9400 - 9500	100	25.6	0.9		2.1
			7800				

The first stretch of “Castillo 1” (advance from Castillo brook or upstream end), Chainage 1700 – 1850 m, is completed after 2.5 months (green color), which is the sum of the first stretch equal to 150 m long in 1.4 months and 1.1 months related to road construction and portal. The time consumption for the following stretches until the end (brown cell) is only related to the advance rate in the tunnel.

For “Castillo 2” (access from Vallical brook or downstream end) the computation is exactly the same. The green cell is the sum of the previous construction times (road and portal from Vallical brook access) plus the time spent to excavate the corresponding stretch. The following stretches until the end (brown cell) is only related to the advance rate in the tunnel.

Castillo 1 & 2 meet each other after 13.9 months. This defines the tunnel construction period and the tunnel length from each portal. 13.9 months ≈ 4.6 trimesters.

Castillo tunnel: 8.7 m diameter (64.3 m_2)					
Rock Mass Classification Q_{ROOF}	**Comment**	**Chainage m**	**Stretch Length m**	**Cumulative distance Castillo 1 m**	**Cumulative distance Castillo 2 m**
0.003	Surface weathering	1700 - 1850	150	150	
1.9	Fair orientation - Poor/Fair Andesite	1850 - 3900	2050	2200	
2.6	Good Orientation - Poor/Fair Andesite	3900 - 4330	430	2630	
0.003	Inferred fault zone around Las Mulas brook	4330 - 4430	100	2730	
2.6	Good Orientation - Poor/Fair Andesite	4430 - 5350	920	3650	
0.003	Inferred fault zone around Las Yeguas brook	5350 - 5450	100	3750	
2.6	Good Orientation - Poor/Fair Andesite	5450 - 6380	930	**3764**	**4036**
0.003	Inferred fault zone I from aerial photos	6380 - 6480	100		3120
2.6	Good Orientation - Poor/Fair Andesite	6480 - 7670	1190		3020
0.003	Inferred fault zone II from aerial photos	7670 - 7770	100		1830
23	Good Orientation - Good Andesite	7770 - 9400	1630		1730
0.003	Surface Weathering	9400 - 9500	100		100
			7800		

The tunnel length of "Castillo 1" is equal to 3764 m and the tunnel length of "Castillo 2" is 4036 m.

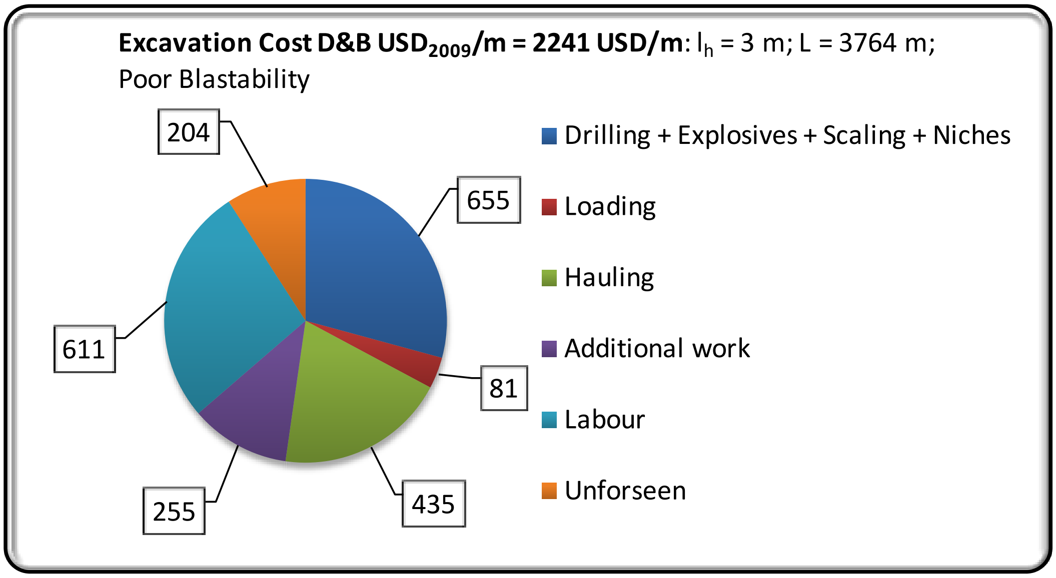
7.5.1.1.2 Excavation Cost:

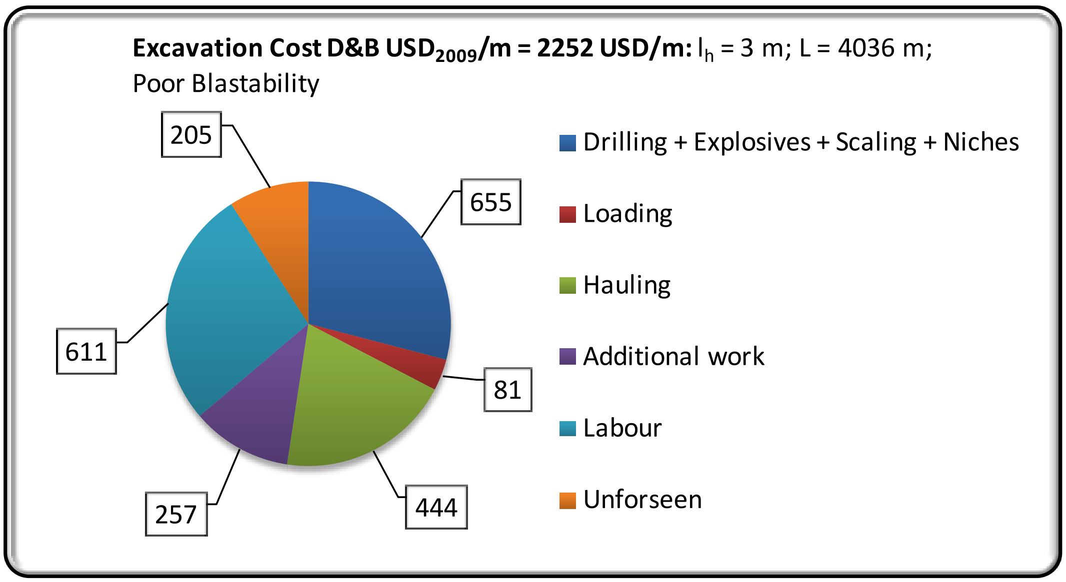
The following table shows the different situations where the excavation cost was estimated for the Castillo Tunnel:

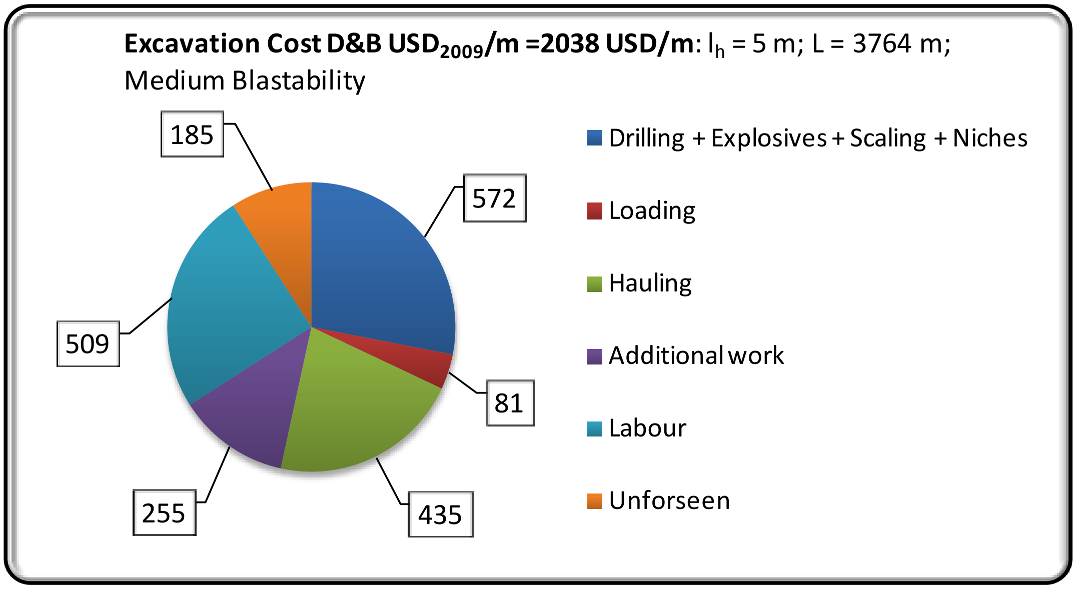
D & B Excavation Cost : 64.3 m^2				
Stretch	I_h m	Blastability	Q range	Length reference m
Castillo 1	3	Poor	0.003	3,764
Castillo 2	3			4,036
Castillo 1	5	Medium	1.9 -23	3,764
Castillo 2	5			4,036

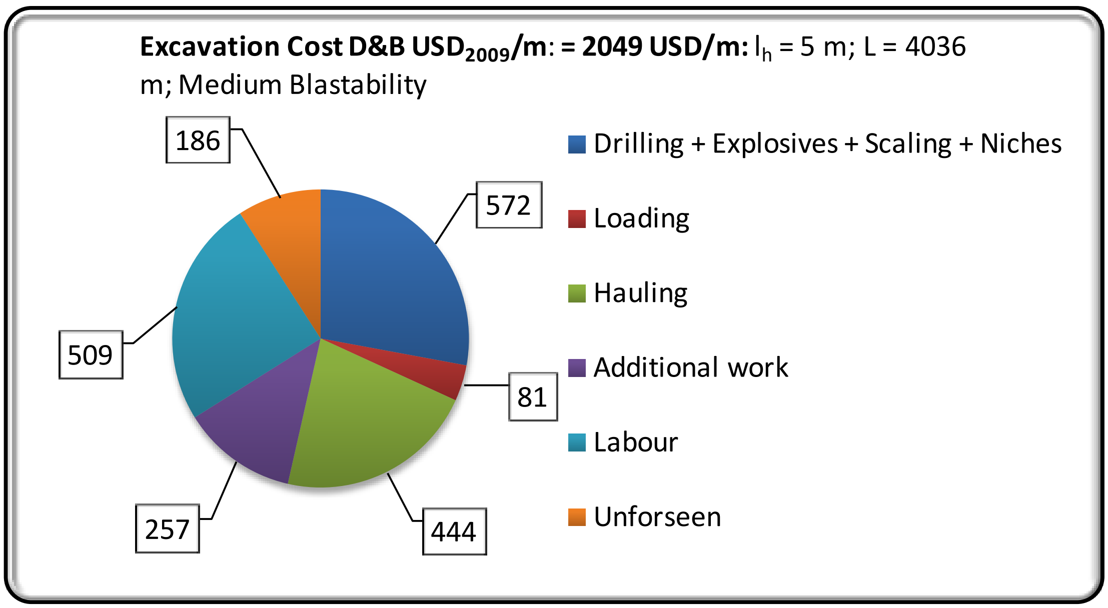
The length reference is the tunnel length used as input in the excavation cost. It is measured from the access portal to the place where one advance stretch meets the other construction stretch from other adit.

Note that the construction stretch from each portal or "Length reference" comes from the advance rate chapter results in Castillo tunnel.









7.5.1.1.3 Rock support Cost:

Cost of Rock Support for Castillo Tunnel: 64.3 m2 (8.7 m diameter horse shoe shape)						
Q_{ROOF}	Radial Bolts USD/m	Fibercrete USD/m	Shotcrete USD/m	Rebars USD/m	Spiling bolts USD/m	Total Rock Support USD/m
0.003	1,594	3,364	4,247	902	5,228	**15,334**
1.9	399	0	856	0	0	**1,255**
2.6	239	0	856	0	0	**1,095**
23	0	0	0	0	0	**0**

A summary of the total investment cost involved in the Castillo tunnel is shown below:

Castillo tunnel: 8.7 m diameter (64.3 m^2)							
Rock Mass Classification Q_{ROOF}	Chainage m	Stretch Length m	D & B Cost USD/m	Invert Cost Concrete + wire mesh USD/m	Cost Rock Support USD/m	Total Gross Cost USD/m	Total Cost/stretch USD
0.003	1700 - 1850	150	2241	367	15334	17942	2,691,262
1.9	1850 - 3900	2050	2038	367	1255	3659	7,501,486
2.6	3900 - 4330	430	2038	367	1095	3500	1,504,925
0.003	4330 - 4430	100	2241	367	15334	17942	1,794,174
2.6	4430 - 5350	920	2038	367	1095	3500	3,219,840
0.003	5350 - 5450	100	2252	367	15334	17953	1,795,274
2.6	5450 - 6380	930	2049	367	1095	3511	3,265,068
0.003	6380 - 6480	100	2252	367	15334	17953	1,795,274
2.6	6480 - 7670	1190	2049	367	1095	3511	4,177,882
0.003	7670 - 7770	100	2252	367	15334	17953	1,795,274
23	7770 - 9400	1630	2049	367	0	2416	3,937,267
0.003	9400 - 9500	100	2252	367	15334	17953	1,795,274
		7800					**35,273,002**

Gross investment cost: 35,273,002 / 7800 = 4522 USD/m

7.5.1.1.4 Energy losses

Manning roughness coefficient and hydraulic flow area depend on the rock support adopted. Therefore, first of all the details to obtain these two variables are presented for the Castillo tunnel optimum cross section (64.3 m^2) in different rock support. After that, the energy losses for the whole tunnel are given.

A summary of all the rock support adopted along the Castillo tunnel is given below:

Case	Q_{ROOF}	Reinforcement Category in roof	Q_{WALLS}	Reinforcement Category in walls	Final rock support adopted for Roof and Walls
1	0.003	8	0.003	8	Reinforced Shotcrete ribs + systematic bolting (Roof & Walls) D40/4 c/c 1.2 m
2	1.9	4	4.7	3	7 cm shotcrete in roof ; Systematic bolting (Roof & Walls)
3	2.6	4	6.5	1	7 cm shotcrete in roof and bolts in roof only
4	23	1	113	1	Unsupported

Hydraulic flow area and Manning roughness coefficient computation details are given for each case:

Case 1: (Q_{ROOF} = 0.003):

Reinforced Shotcrete ribs D40/4 c/c 1.2 m + systematic bolting (Roof & Walls)

The hydraulic flow area for this case is:

Castillo Tunnel: 64.3 m^2 (8.7 m diameter)	Q_{ROOF} =	0.003	
Theoretical cross section	A_p	64.3	m^2
Factor of hydraulic overbreak (rough cleaning)	k_o	1.07	
Blasted cross section + concrete invert	A_{v_1}	67.0	m^2
Tunnel diameter related to A_{V_1}	D_1	8.88	m
Tunnel diameter with 15 cm fibrecrete thickness	D_3	8.58	m
Final shotcrete ribs cross section area	A_{v_3}	62.5	m^2

The hydraulic flow area in reinforced shotcrete rib rock support D40/4 C/C 1.2 is 62.5 m^2 based on a theoretical cross section equals to 64.3 m^2.

The lowest Manning roughness value for case 1 in Castillo Tunnel is obtained from the Hyper-turbulent flow regime. The detail values are given in the next table:

Manning roughness value: Case 1: Hyper-turbulent Flow			
Shotcrete rib spacing	L	m	1.2
rib thickness (Or height)	h	m	0.25
Constant factor	α		0.0898
Inside diameter of shotcrete ribs	D_i	m	8.08
Radius based on inside diameter D_i	r_i	m	4.04
Shotcrete rib Manning value	n_{HT}	-	0.045
Shotcrete rib Manning value	M_{HT} ($1/n_{HT}$)	-	**22.0**

In order to have the equivalent Manning roughness coefficient of the stretch, the Manning value gotten from the last table for walls and roof shotcrete ribs must be weighed with the Manning roughness of concrete invert.

Castillo Tunnel: 64.3 m^2 (8.7 m diameter) ;	Q_{ROOF} =		0.003 m
Concrete invert Manning value	M_1	-	60
Invert width	P_1	m	6.3
Shotcrete rib Manning value	$M_{HT} = M_2$	-	22.0
Walls & roof Perimeter	P_2	m	22.5
Einstein equation	M_{GLOBAL}	-	24.9

The resulting Manning value M_{GLOBAL} for a shotcrete rib rock support D40/4 C/C 1.2 and concrete invert in Castillo tunnel is 24.9.

Case 2: (Q_{ROOF} = 1.9)

Shotcrete 7 cm thickness in roof; Systematic bolting (Roof & Walls)

Only shotcrete is relevant for hydraulic purposes, which is shotcrete 7 cm thickness in roof.

The final hydraulic area for this case is:

Castillo Tunnel: 64.3 m^2 (8.7 m diameter); Case 2			
Theoretical cross section	A_p	64.3	m^2
Factor of hydraulic overbreak (rough cleaning)	k_o	1.07	
Blasted cross section + concrete invert	A_{v_1}	67.0	m^2
A_{V_1} + Shotcrete in roof	$A_{V\_2.2}$	66.0	m^2

The Manning roughness value for shotcrete in roof is:

Castillo Tunnel: 64.3 m^2 ; Manning roughness value; Shotcrete in roof only		
Cross section before shotcrete in roof	A_{v_1}	67.0
Cross section area with shotcrete in roof	$A_{V\_2.2}$	66.0
ratio between area after and before shotcreting	$\alpha = A_{V\_2.2} / A_{V\_1}$	0.985
Hydraulic radius (unlined contour except invert)	R_h	2.3
Friction factor without shotcrete	f_o	0.061
Empirical correction coefficient	y	69.1
(Friction With shotcrete in roof)	f	0.041
Manning value with shotcrete in roof	M_{S_2}	38.0

The Manning roughness coefficient in unlined tunnel is given by the theoretical cross section, considering class 3 in blast design:

M_U (A_p = 64.3 m^2) = 31.2

Finally, the equivalent Manning roughness coefficient is obtained by Einstein equation, including the shotcrete in roof, unlined tunnel in walls and concrete in invert.

Castillo Tunnel: 64.3 m^2 (8.7 m diameter) ; Case 2			
Concrete invert Manning value	M_1	-	60
Invert width	P_1	m	6.5
Manning value with shotcrete in roof only	$M_{S\_2} = M_2$	-	38.0
Roof Perimeter	P_2	m	13.9
Manning value for unlined walls	$M_3 = M_U$		31.2
Walls perimeter	P_3	m	9.3
Einstein equation	M_{GLOBAL}	-	38.0

Case 3: (Q_{ROOF} = 2.6)

Shotcrete 7 cm thickness in roof and bolts in roof only

For energy losses calculation, case 3 is exactly the same as Case 2.

Case 4: (Q_{ROOF} = 23)

Unsupported

The hydraulic flow area for this case has already been given in the previous cases, because it is the first step in the computation. The hydraulic flow area in this case is 67 m^2 and the details are given in the next table:

Castillo Tunnel: 64.3 m^2 (8.7 m diameter): Case 4 (Unsupported)			
Theoretical cross section	A_p	64.3	m^2
Factor of hydraulic overbreak (rough cleaning)	k_o	1.07	
Blasted cross section + concrete invert	A_{v_1}	67.0	m^2

The Manning roughness coefficient in unlined tunnel is given by the theoretical cross section:

M_U (A_p = 64.3 m^2) = 31.2

Finally, the equivalent Manning roughness coefficient in unlined walls and roof and concrete in invert is:

Castillo Tunnel: 64.3 m^2 (8.7 m diameter) ; Case 4			
Concrete invert Manning value	M_1	-	60
Invert width	P_1	m	6.5
Manning value for unlined tunnel	$M_U = M_2$	-	31.2
Walls & roof Perimeter	P_2	m	23.2
Einstein equation	M_{GLOBAL}	-	34.4

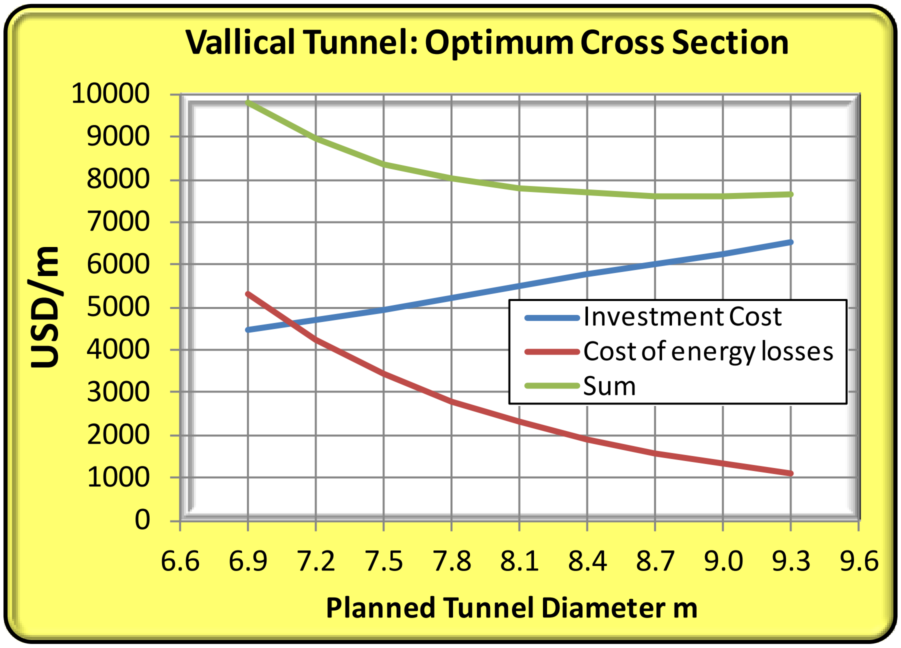
Summary: Castillo Tunnel: 64.3 m^2 (8.7 m diameter)				
Case	Rock Mass Classification Q_{ROOF}	Equivalent Manning roughness M_{GLOBAL}	Hydraulic Area A_h m^2	Final rock support adopted for Roof and Walls
1	0.003	24.9	62.5	Reinforced Shotcrete ribs D40/4 c/c 1.2 m + systematic bolting (Roof & Walls)
2	1.9	38.0	66.0	7 cm shotcrete in roof ; Systematic bolting (Roof & Walls)
3	2.6	38.0	66.0	7 cm shotcrete in roof; bolts in roof only
4	23	34.4	67.0	Unsupported

Rock Mass Classification Q_{ROOF}	**Chainage m**	**Stretch Length m**	**Equivalent Manning Coefficient M_{GLOBAL}**	**Final Hydraulic flow Area A_h m^2**	**Final Hydraulic Radius R_h m**	**Energy Production losses E_L USD/m**	**Long term energy losses per stretch USD**
0.003	1700 - 1850	150	24.9	62.5	2.16	3040	456,052
1.9	1850 - 3900	2050	38.0	66.0	2.22	1135	2,327,437
2.6	3900 - 4330	430	38.0	66.0	2.22	1135	488,194
0.003	4330 - 4430	100	24.9	62.5	2.16	3040	304,034
2.6	4430 - 5350	920	38.0	66.0	2.22	1135	1,044,508
0.003	5350 - 5450	100	38.0	66.0	2.22	1135	113,534
2.6	5450 - 6380	930	38.0	66.0	2.22	1135	1,055,862
0.003	6380 - 6480	100	24.9	62.5	2.16	3040	304,034
2.6	6480 - 7670	1190	38.0	66.0	2.22	1135	1,351,049
0.003	7670 - 7770	100	24.9	62.5	2.16	3040	304,034
23	7770 - 9400	1630	34.4	67.0	2.23	1333	2,172,866
0.003	9400 - 9500	100	24.9	62.5	2.16	3040	304,034
		7800					**10,225,640**

10,225,640 USD / 7800 m = 1311 USD/m

7.5.2 Vallical Tunnel

The optimum cross section for Vallical tunnel based on the methodology exposed here is 68.8 m^2 (9 m diameter for a horse shoe tunnel shape). The following chart shows the investment cost updated to the plant commissioning and the cost of energy losses for the same date. Also it is shown the sum of the latter two costs in order to indicate where the cross section, which minimizes that sum, is located.



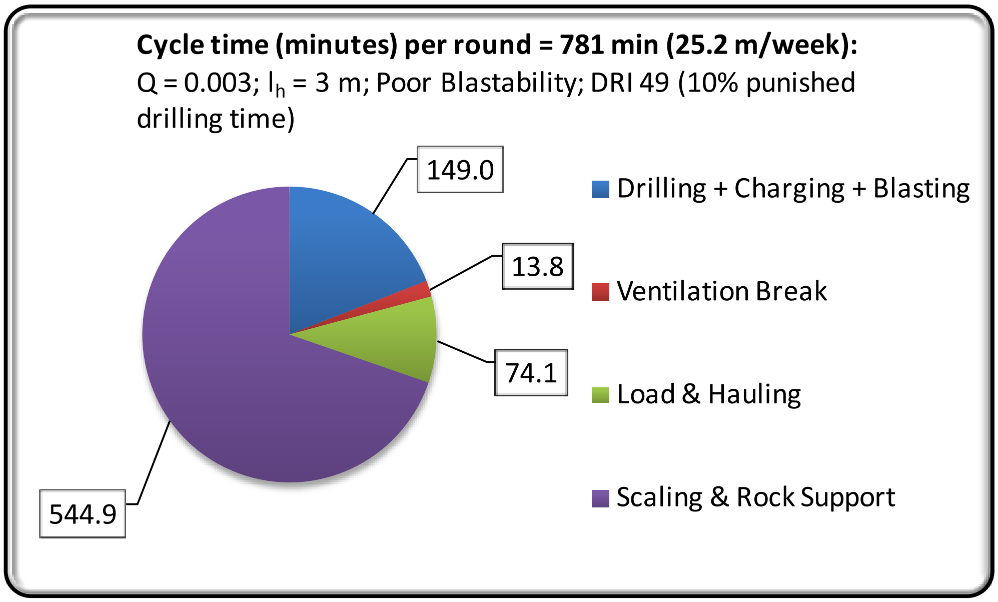
Vallical Tunnel											
Planned Cross Section area	A_p	m^2	40.4	44.0	47.8	51.7	55.7	59.9	64.3	68.8	73.5
Planned Diameter	D_p	m	6.9	7.2	7.5	7.8	8.1	8.4	8.7	9.0	9.3
Gross Investment Cost	I_g	USD/m	3,358	3,514	3,674	3,919	4,090	4,312	4,479	4,640	4,820
Planning & Administration 20%	I_{pa}	USD/m	672	703	735	784	818	862	896	928	964
Gross Investment Cost + Planning & Administration	$I_g + I_{pa}$	USD/m	4,029	4,217	4,409	4,703	4,908	5,174	5,375	5,568	5,784
Tunnel Construction period	n_{TRI}	Trimester	6.0	6.2	6.4	6.0	6.2	6.4	6.6	6.8	6.9
Investment Cost updated at plant comissioning	I	USD/m	4,490	4,710	4,938	5,240	5,479	5,793	6,032	6,260	6,519
Cost of energy losses	E_L	USD/m	5,294	4,241	3,433	2,800	2,297	1,900	1,579	1,322	1,115
Sum	$I + E_L$	USD/m	**9,784**	**8,951**	**8,371**	**8,040**	**7,776**	**7,693**	**7,611**	**7,583**	**7,634**

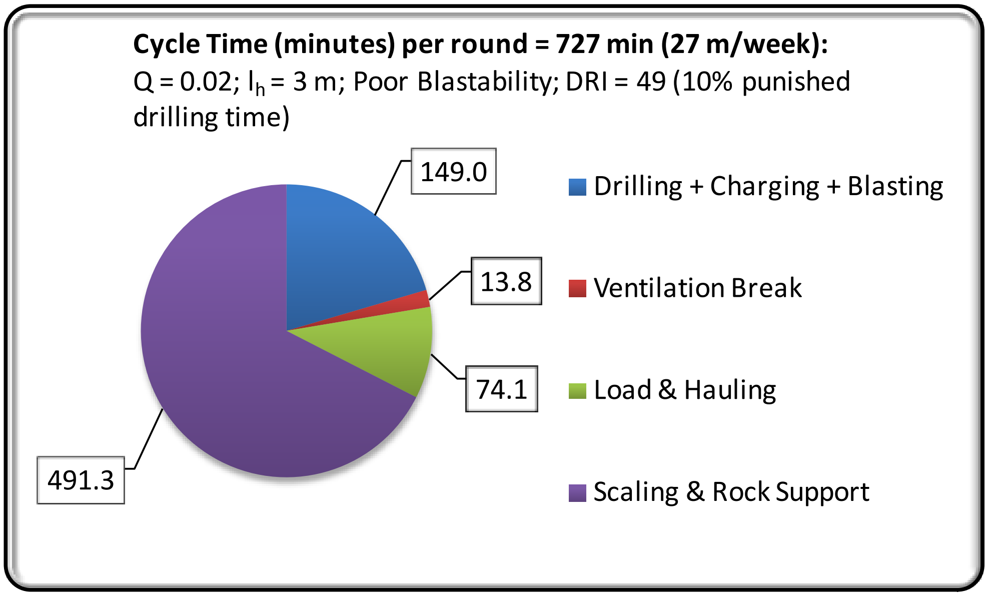
7.5.2.1.1 Advance rate:

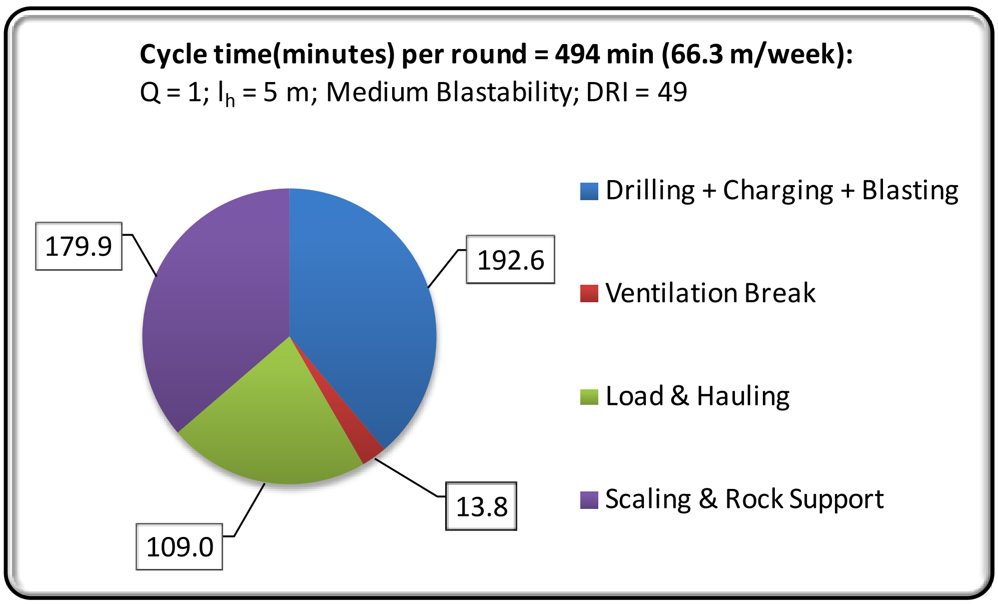
The following table shows the different situations where the advance rate was estimated for the Vallical Tunnel:

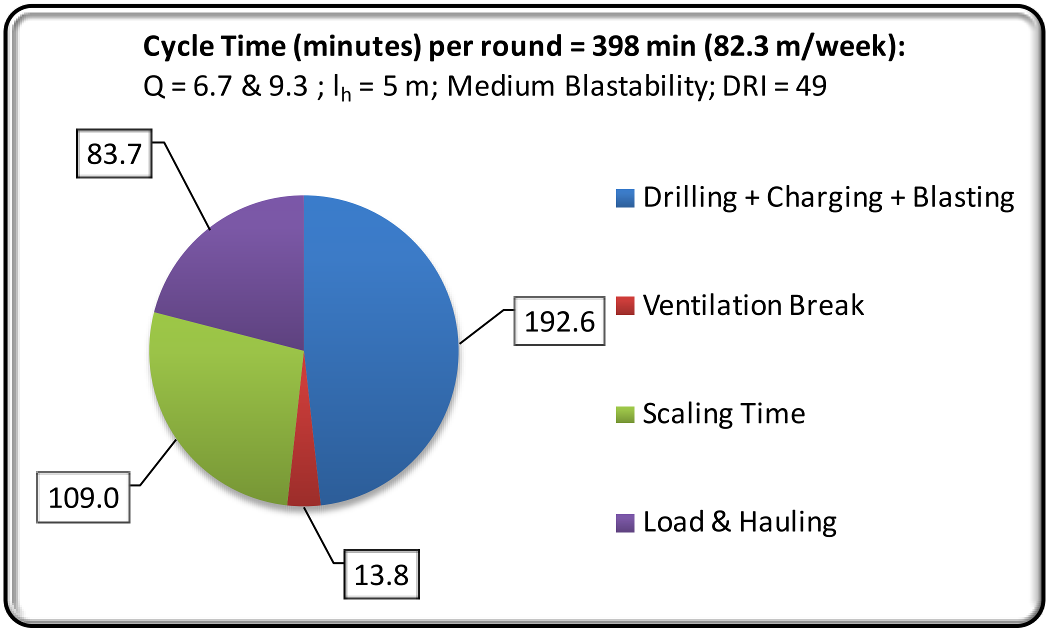
Vallical Tunnel: 68.8 m^2 (9 m diameter horse shoe shape)			
Rock Mass Classification Q_{ROOF}	Blastability	Drilled length m	Drilling time punishment 10% (fractured rock)
0.003	Poor	3	1.1
0.02	Poor	3	1.1
1.0	Medium	5	1.0
6.7	Medium	5	1.0
9.3	Medium	5	1.0

And the details are given in the following four charts:




Cycle Time (minutes) per round = 727 min (27 m/week):
Q = 0.02; l_h = 3 m; Poor Blastability; DRI = 49 (10% punished drilling time)
149.0
13.8
74.1
491.3
Drilling + Charging + Blasting
Ventilation Break
Load & Hauling
Scaling & Rock Support


Cycle time(minutes) per round = 494 min (66.3 m/week):
Q = 1; l_h = 5 m; Medium Blastability; DRI = 49
179.9
192.6
13.8
109.0
Drilling + Charging + Blasting
Ventilation Break
Load & Hauling
Scaling & Rock Support



Before Vallical tunnel construction starts, access road and portal must to be finished before. As it was mentioned in the advance rate assumptions, each portal construction takes 1 month and the corresponding access road construction time to each portal is given in the following two tables where "Vallical 1" represents the tunnel stretch that is constructed from the upstream end (or from Vallical brook) of Vallical tunnel and "Vallical 2" represents the tunnel stretch excavated from the downstream end:

Access to Vallical 1 portal		
Ustream end	372	m
Road advance rate	50	m/day
time elapsed	7.4	day
Time elapsed	0.2	month

Access to Vallical 2 portal		
Ustream end	8760	m
Road advance rate	50	m/day
time elapsed	175	day
Time elapsed	5.8	month

Tunnel Vallical : 68.8 m^2 (9 m diameter)							
Rock Mass Classification Q_{ROOF}	Identification of stretch	Chainage m	Stretch Length m	Advance rate m/week	Advance rate month/stretch	Cumulative time elapsed Vallical 1 month	Cumulative time elapsed Vallical 2 month
0.003	Surface weathering & inferred fault zone 3	10000 - 10100	100	25.2	0.9	2.2	
1.0	Good orientation - Fair Andesite	10100 - 10300	200	66.3	0.7	2.9	
0.003	Weakness zone - Change in rock formation	10300 - 10350	50	25.2	0.5	3.3	
6.7	Fair Orientation - Fair Granite	10350 - 10850	500	82.3	1.4	4.8	
0.02	Inferred fault zone 4	10850 - 10950	100	27.0	0.9	5.6	
6.7	Fair Orientation - Fair Granite	10950 - 11300	350	82.3	1.0	6.6	
0.02	Inferred fault zone 5	11300 - 11400	100	27.0	0.9	7.5	
6.7	Fair Orientation - Fair Granite	11400 - 12550	1150	82.3	3.3	10.7	
0.02	Inferred fault zone 6	12550 - 12650	100	27.0	0.9	11.6	
6.7	Fair Orientation - Fair Granite	12650 - 13000	350	82.3	1.0	12.6	
0.02	Inferred fault zone 7	13000 - 13100	100	27.0	0.9	13.5	
6.7	Fair Orientation - Fair Granite	13100 - 13400	300	82.3	0.9	14.3	
0.02	Inferred fault zone 8	13400 - 13500	100	27.0	0.9	15.2	
6.7	Fair Orientation - Fair Granite	13500 - 14150	650	82.3	1.8	17.0	
0.02	Inferred fault zone 9	14150 - 14250	100	27.0	0.9	17.9	
6.7	Fair Orientation - Fair Granite	14250 - 14750	500	82.3	1.4	19.3	
0.02	Inferred fault zone 10	14750 - 14850	100	27.0	0.9	20.2	
9.3	Fair Orientation - Fair Granite	14850 - 15350	500	82.3	1.4	**20.3**	**20.3**
0.02	Inferred fault zone 11	15350 - 15450	100	27.0	0.9		18.9
9.3	Good orientation - Fair Granite	15450 - 16550	1100	82.3	3.1		18.1
0.02	Inferred fault zone 12	16550 - 16650	100	27.0	0.9		15.0
9.30	Good orientation - Fair Granite	16650 - 17400	750	82.3	2.1		14.1
0.02	Inferred Fault zone 13	17400 - 17500	100	27.0	0.9		12.0
6.7	Fair Orientation - Fair Granite	17500 - 18300	800	82.3	2.3		11.1
9.3	Good orientation - Fair Granite	18300 - 18700	400	82.3	1.1		8.8
0.02	Surface weathering	18700 - 18800	100	27.0	0.9		7.7
			8800				

The first stretch of "Vallical 1" (advance from Vallical brook or upstream end), chainage 10,000 – 10,100 m, is completed after 2.2 months (green color), which is the sum of the first stretch equals to 100 m long excavated in 0.9 months plus 1.3 months related to access road to the portal (377 m long) and the portal construction itself. The time consumption for the following stretches until the end (brown cell) is only related to the advance rate in the tunnel.

The computation for "Vallical 2" (advance from downstream end) is exactly the same. The green cell is the sum of the previous construction times (road and portal from Vallical brook access) plus the time spent to excavate the corresponding stretch. Note that Vallical 2 starts several months after Vallical 1 because of the access road (8,760 m long) needed for the downstream end of Vallical tunnel. The following tunnel stretches until the end (brown cell) is only related to the advance rate in the tunnel.

Vallical 1 & 2 meet each other after 20.3 months ≈ 6.8 trimesters (See summary table above).. This defines the tunnel construction period and the length from each tunnel access. 20.3 months

Tunnel Vallical : 68.8 m^2 (9 m diameter)					
Rock Mass Classification Q_{ROOF}	**Identification of stretch**	**Chainage** m	**Stretch Length** m	**Cumulative Length Vallical 1** m	**Cumulative Length Vallical 2** m
0.003	Surface weathering & inferred fault zone 3	10000 - 10100	100	100	
1.0	Good orientation - Fair Andesite	10100 - 10300	200	300	
0.003	Weakness zone - Change in rock formation	10300 - 10350	50	350	
6.7	Fair Orientation - Fair Granite	10350 - 10850	500	850	
0.02	Inferred fault zone 4	10850 - 10950	100	950	
6.7	Fair Orientation - Fair Granite	10950 - 11300	350	1300	
0.02	Inferred fault zone 5	11300 - 11400	100	1400	
6.7	Fair Orientation - Fair Granite	11400 - 12550	1150	2550	
0.02	Inferred fault zone 6	12550 - 12650	100	2650	
6.7	Fair Orientation - Fair Granite	12650 - 13000	350	3000	
0.02	Inferred fault zone 7	13000 - 13100	100	3100	
6.7	Fair Orientation - Fair Granite	13100 - 13400	300	3400	
0.02	Inferred fault zone 8	13400 - 13500	100	3500	
6.7	Fair Orientation - Fair Granite	13500 - 14150	650	4150	
0.02	Inferred fault zone 9	14150 - 14250	100	4250	
6.7	Fair Orientation - Fair Granite	14250 - 14750	500	4750	
0.02	Inferred fault zone 10	14750 - 14850	100	4850	
9.3	Fair Orientation - Fair Granite	14850 - 15350	500	**4883**	**3917**
0.02	Inferred fault zone 11	15350 - 15450	100		3450
9.3	Good orientation - Fair Granite	15450 - 16550	1100		3350
0.02	Inferred fault zone 12	16550 - 16650	100		2250
9.30	Good orientation - Fair Granite	16650 - 17400	750		2150
0.02	Inferred Fault zone 13	17400 - 17500	100		1400
6.7	Fair Orientation - Fair Granite	17500 - 18300	800		1300
9.3	Good orientation - Fair Granite	18300 - 18700	400		500
0.02	Surface weathering	18700 - 18800	100		100
			8800		

The tunnel length of “Vallical 1” is equal to 4883 m and the tunnel length of “Vallical 2” is 3917 m.

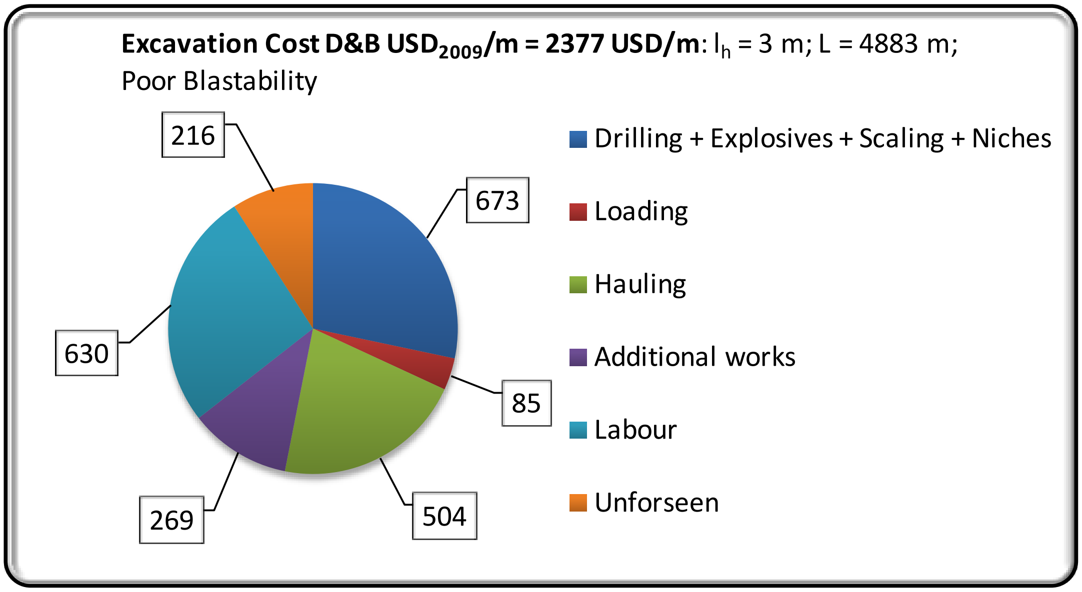
7.5.2.1.2 Excavation cost

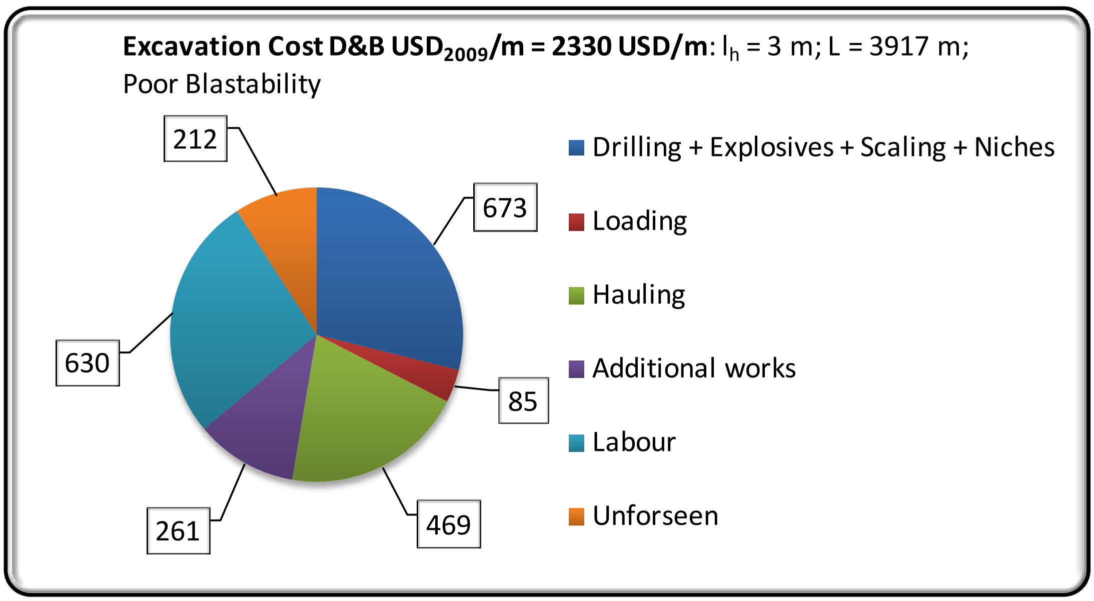
The following table shows the different situations where the excavation cost was estimated for the Vallical Tunnel:

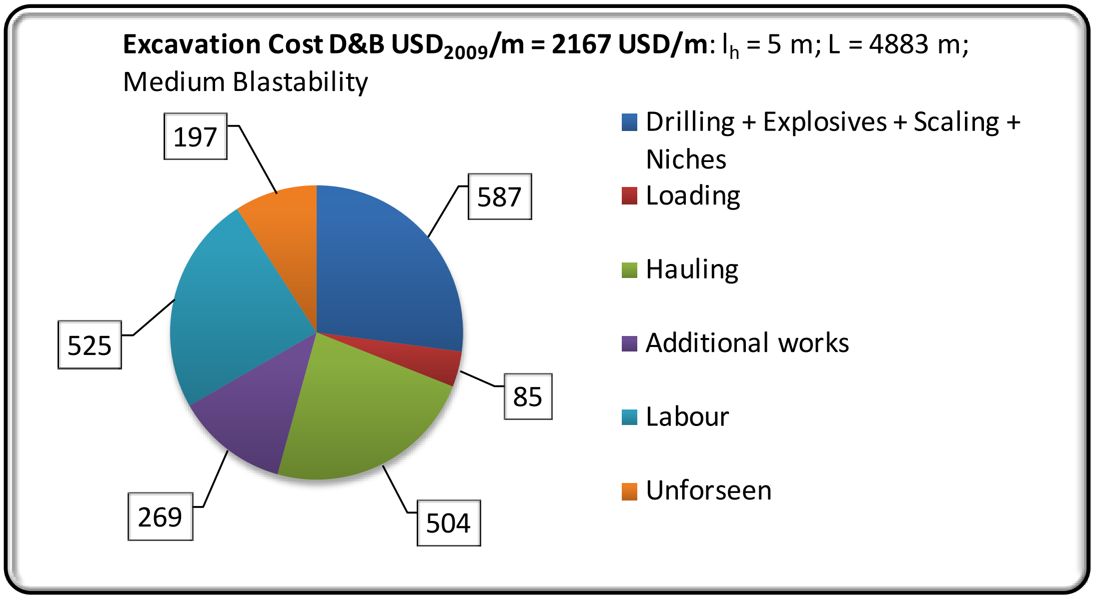
	D & B Excavation Cost :		68.8	m^2
Stretch	Drilled length l_h m	Blastability	Q range	Length reference m
Vallical 1	3	Poor	0.003 - 002	4,883
Vallical 2	3	Poor	0.003 - 002	3,917
Vallical 1	5	Medium	1 -9.3	4,883
Vallical 2	5	Medium	1 -9.3	3,917

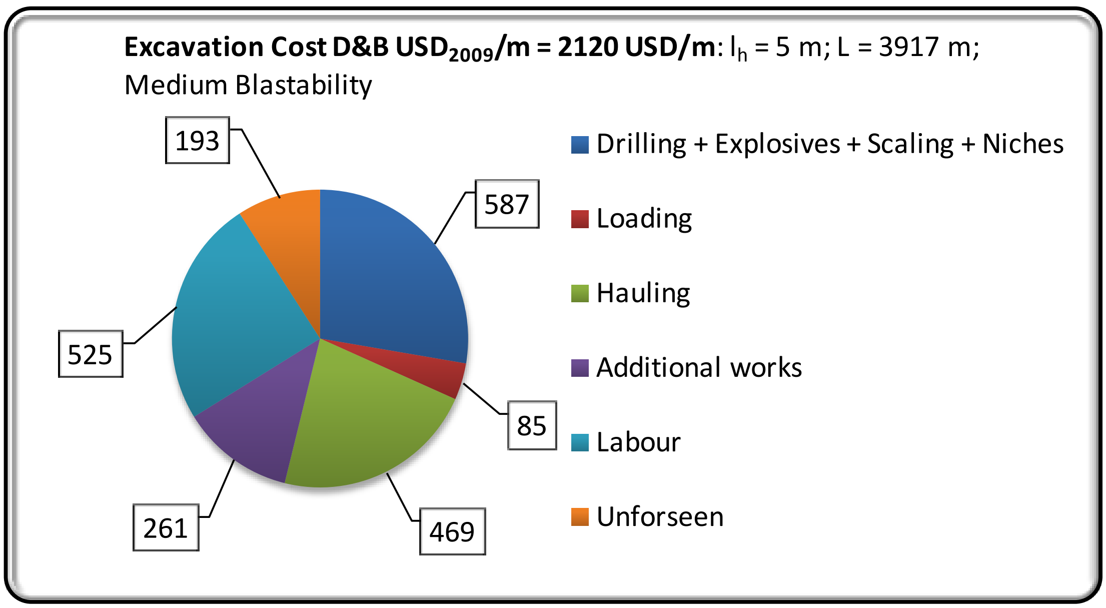
The length reference is the tunnel length used as input in the excavation cost. It is measured from the access portal to the place where one advance stretch meets the other construction stretch from other adit.

Note that the construction stretch from each portal or “Length reference” comes from the advance rate chapter results in Vallical tunnel.


Excavation Cost D&B USD$_{2009}$/m = 2377 USD/m: l_h = 3 m; L = 4883 m;
Poor Blastability
216
673
630
85
269
504
Drilling + Explosives + Scaling + Niches
Loading
Hauling
Additional works
Labour
Unforseen


Excavation Cost D&B USD$_{2009}$/m = 2330 USD/m: l_h = 3 m; L = 3917 m;
Poor Blastability
212
673
630
85
261
469
Drilling + Explosives + Scaling + Niches
Loading
Hauling
Additional works
Labour
Unforseen





7.5.2.1.3 Rock Support Cost

Cost of Rock Support for Vallical Tunnel: 68.8 m^2 (9 m diameter horse shoe shape)						
Rock mass Classidication Q_{ROOF}	Radial Bolts USD/m	Fibercrete USD/m	Shotcrete USD/m	Rebars USD/m	Spiling bolts USD/m	Total Rock Support USD/m
0.003	1,649	3,480	4,393	933	5,408	15,863
0.02	1,145	3,480	1,834	608	5,408	12,475
1.0	571	974	590	0	0	2,135
6.7	247	0	0	0	0	247
9.3	247	0	0	0	0	247
23	0	0	0	0	0	0

A summary of the total investment cost involved in the Vallical tunnel is shown below:

Tunnel Vallical : 68.8 m^2 (9 m diameter)							
Rock Mass Classification	**Chainage**	**Stretch Length**	**D& B Cost**	**Invert Cost concrete + wire mesh**	**Cost Rock Support**	**Total Unit Cost**	**Total Cost/stretch**
Q_{ROOF}	**m**	**m**	**USD/m**	**USD/m**	**USD/m**	**USD/m**	**USD**
0.003	10000 - 10100	100	2377	379	15863	18619	1,861,915
1.0	10100 - 10300	200	2167	379	2135	4682	936,314
0.003	10300 - 10350	50	2377	379	15863	18619	930,957
6.7	10350 - 10850	500	2167	379	247	2794	1,396,770
0.02	10850 - 10950	100	2377	379	12475	15231	1,523,114
6.7	10950 - 11300	350	2167	379	247	2794	977,739
0.02	11300 - 11400	100	2377	379	12475	15231	1,523,114
6.7	11400 - 12550	1150	2167	379	247	2794	3,212,571
0.02	12550 - 12650	100	2377	379	12475	15231	1,523,114
6.7	12650 - 13000	350	2167	379	247	2794	977,739
0.02	13000 - 13100	100	2377	379	12475	15231	1,523,114
6.7	13100 - 13400	300	2167	379	247	2794	838,062
0.02	13400 - 13500	100	2377	379	12475	15231	1,523,114
6.7	13500 - 14150	650	2167	379	247	2794	1,815,801
0.02	14150 - 14250	100	2377	379	12475	15231	1,523,114
6.7	14250 - 14750	500	2167	379	247	2794	1,396,770
0.02	14750 - 14850	100	2377	379	12475	15231	1,523,114
9.3	14850 - 15350	500	2120	379	247	2747	1,373,270
0.02	15350 - 15450	100	2330	379	12475	15184	1,518,414
9.3	15450 - 16550	1100	2120	379	247	2747	3,021,194
0.02	16550 - 16650	100	2330	379	12475	15184	1,518,414
9.30	16650 - 17400	750	2120	379	247	2747	2,059,905
0.02	17400 - 17500	100	2330	379	12475	15184	1,518,414
6.7	17500 - 18300	800	2120	379	247	2747	2,197,232
9.3	18300 - 18700	400	2120	379	247	2747	1,098,616
0.02	18700 - 18800	100	2330	379	12475	15184	1,518,414
		8800					**40,830,310**

Average gross investment Cost = 40,830,210 USD / 8800 m = 4,640 USD/m

7.5.2.1.4 Energy Losses

Compared to Castillo tunnel in terms of energy losses, the Vallical tunnel has a different tunnel length, rock support and the optimum cross section (68.8 m^2). As it was done for Castillo tunnel, details about the figures to obtain the Manning roughness coefficient and design hydraulic area are presented for the Vallical tunnel optimum cross section (68.8 m^2) for each tunnel stretch with different rock support. After that, the energy losses for the whole tunnel are given.

Vallical Tunnel: 68.8 m^2 (9 m diameter horse shoe shape tunnel)					
Case	**Q_{ROOF}**	**Reinforcement Category in roof**	**Q_{WALLS}**	**Reinforcement Category in walls**	**Final rock support adopted for Roof and Walls**
1	0.003	8	0.003	8	Reinforced Shotcrete ribs + systematic bolting (Roof & Walls) D40/4 c/c 1.2 m
2	0.02	8	0.02	8	Reinforced Shotcrete ribs ; systematic bolting (Roof & Walls) E35/5 c/c 2.3 m
3	1.0	5	2.4	4	7 cm Fibercrete in roof; 7 cm shotcrete in walls; systematic bolting (Roof & Walls)
4	6.7	1	16.7	1	Bolts in roof only
5	9.3	1	23	1	Bolts in roof only

Hydraulic flow area and Manning roughness coefficient computation details are given for each case:

Case 1: (Q_{ROOF} = 0.003)

Reinforced Shotcrete ribs D40/4 c/c 1.2 m + systematic bolting (Roof & Walls)

Vallical Tunnel: 68.8 m^2 (9 m diameter) Case 1			
Theoretical cross section	A_p	68.8	m^2
Factor of hydraulic overbreak (rough cleaning)	k_o	1.07	
Blasted cross section + concrete invert	A_{V_1}	71.5	m^2
Tunnel diameter related to A_{V_1}	D_1	9.17	m
Tunnel diameter with 15 cm fibrecrete thickness	D_3	8.87	m
Final shotcrete ribs cross section area	A_{V_3}	66.9	m^2

The hydraulic flow area in reinforced shotcrete rib rock support D40/4 C/C 1.2 is 66.9 m^2 based on a theoretical cross section equals to 68.8 m^2.

The lowest Manning roughness value for case 1 in Vallical Tunnel is obtained from the Hyper-turbulent flow regime. The detail values are given in the next table:

Manning roughness value: Case 1: Hyper-turbulent Flow			
Shotcrete rib spacing	L	m	1.2
rib thickness (Or height)	h	m	0.25
Constant factor	α		0.0898
Inside diameter of shotcrete ribs	D_i	m	8.37
Radius based on inside diameter D_i	r_i	m	4.19
Shotcrete rib Manning value	n_{HT}	-	0.045
Shotcrete rib Manning value	M_{HT} $(1/n_{HT})$	-	**22.2**

In order to have the equivalent Manning roughness coefficient for case 1, the Manning value obtained from the last table for walls and roof shotcrete ribs must be weighed with the Manning roughness of concrete invert.

Vallical Tunnel: 68.8 m^2 (9 m diameter) ; Case 1 (Q_{ROOF} = 0.003)			
Concrete invert Manning value	M_1	-	60
Invert width	P_1	m	6.5
Shotcrete rib Manning value	$M_{HT} = M_2$	-	22.2
Walls & roof Perimeter	P_2	m	23.2
Einstein equation	M_{GLOBAL}	-	25.1

The resulting Manning value M_{GLOBAL} for a shotcrete rib rock support D40/4 C/C 1.2 and concrete invert in Vallical tunnel is 25.1.

Case 2: (Q_{ROOF} = 0.02)

Reinforced Shotcrete ribs E35/5 c/c 2.3 m; systematic bolting (Roof & Walls)

The resulting hydraulic flow area for Case 2 “A_{V_3}” which is shotcrete ribs in walls and roof along with concrete in invert is exactly the same as case 1. This is due to the same fibercrete thickness (15 cm) considered before the installation of rebars and the following layers of shotcrete.

A_{V_3} = 66.9 m^2.

The lowest Manning roughness value for case 2 in Vallical Tunnel is obtained from the “Isolated Roughness flow” regime. The detail values are given in the next table:

Vallical Tunnel = Manning roughness value: Case 2: Isolated roughness Flow			
Tunnel lined with fibrecrete 15 cm thickness Manning Value	n		0.015
Inside diameter of shotcrete ribs	D_i	m	8.47
Tunnel diameter with only fibercrete lining	D	m	8.87
Drag coefficient for roughness shape	C_D		1.9
Rib perimeter (Walls & roof)	L_r	m	22.2
Total wetted perimeter = L_r + invert width	P	m	28.4
Rib thickness (Or height)	h	m	0.2
Shotcrete rib spacing	L	m	2.3
Shotcrete rib Manning value	$\mathbf{n_{IR}}$		0.046
Shotcrete rib Manning value	$\mathbf{M_{IR}}$		21.6

In order to have the equivalent Manning roughness coefficient of this stretch, the Manning value obtained from the last table for walls and roof shotcrete ribs must be weighed with the Manning roughness of concrete invert.

Vallical Tunnel: 68.8 m^2 (9 m diameter) ; Case 2 (Q_{ROOF} = 0.02)			
Concrete invert Manning value	M_1	-	60
Invert width	P_1	m	6.5
Shotcrete rib Manning value	$M_{HT} = M_2$	-	21.6
Walls & roof Perimeter	P_2	m	23.2
Einstein equation	M_{GLOBAL}	-	24.5

The resulting Manning value M_{GLOBAL} for a shotcrete rib rock support D35/5 C/C 2.3 and concrete invert in Vallical tunnel is 24.5.

Case 3 (Q_{ROOF} = 1)

Fibercrete 7 cm thickness in roof; 7 cm shotcrete in walls; systematic bolting (Roof & Walls)

Only shotcrete (or fibercrete) is relevant for hydraulic energy losses purposes. There is no distinction between shotcrete and fibercrete with regard to Manning roughness value.

The hydraulic flow area for this case is:

Vallical Tunnel: 68.8 m^2 (9 m diameter) Case 3			
Theoretical cross section	A_p	68.8	m^2
Factor of hydraulic overbreak (rough cleaning)	k_o	1.07	
Blasted cross section + concrete invert	A_{V_1}	71.5	m^2
A_{V_1} plus 7 cm shotcrete tickness walls & roof	$A_{V\_2.1}$	69.3	m^2

Manning roughness value = Vallical Tunnel: 68.8 m^2 ; Shotcrete in roof & walls		
Cross section before shotcrete in roof & walls	A_{v_1}	71.5
Cross section area with shotcrete in roof & walls	$A_{V\_2.1}$	69.3
Ratio between area after and before shotcreting	$\alpha = A_{V\_2.2} / A_{V\_1}$	0.985
Hydraulic radius (unlined contour except invert)	R_h	2.3
Friction factor without shotcrete	f_o	0.061
Empirical correction coefficient	y	70.1
(Friction With shotcrete in roof & walls	f	0.034
Manning value with shotcrete in roof & Walls	M_{S_1}	42.0

Finally, the equivalent Manning roughness coefficient of this case is obtained from the Manning value gotten from the last table for roof & walls shotcrete lining and weighed with the concrete invert Manning roughness.

Vallical Tunnel: 68.8 m^2 (9 m diameter) ; Case 3 (Q_{ROOF} = 1)			
Concrete invert Manning value	M_1	-	60
Invert width (shotcrete included)	P_1	m	6.6
Shotcrete in roof & walls Manning value	$M_{HT} = M_2$	-	42.0
Walls & roof Perimeter (shotcrete included)	P_2	m	23.7
Einstein equation	M_{GLOBAL}	-	44.7

Case 4: (Q_{ROOF} = 6.7)

Bolts in roof only.

For hydraulic purposes, this case is the same as unlined tunnel in roof and walls.

Vallical Tunnel: 68.8 m^2 (9 m diameter) Case 4			
Theoretical cross section	A_p	68.8	m^2
Factor of hydraulic overbreak (rough cleaning)	k_o	1.07	
Blasted cross section + concrete invert	A_{V_1}	71.5	m^2

The Manning roughness coefficient in unlined tunnel is given by the theoretical cross section:

M_U (A_p = 64.3 m^2) = 31.2

Vallical Tunnel: 68.8 m^2 (9 m diameter) ; Case 4 (Q_{ROOF} = 6.7)			
Concrete invert Manning value	M_1	-	60
Invert width (Unlined tunnel)	P_1	m	6.7
Manning tunnel in unlined walls & roof	$M_{HT} = M_2$	-	31.2
Walls & roof Perimeter (unlined)	P_2	m	24.0
Einstein equation	M_{GLOBAL}	-	34.4

Case 5: (Q_{ROOF} = 9.3)

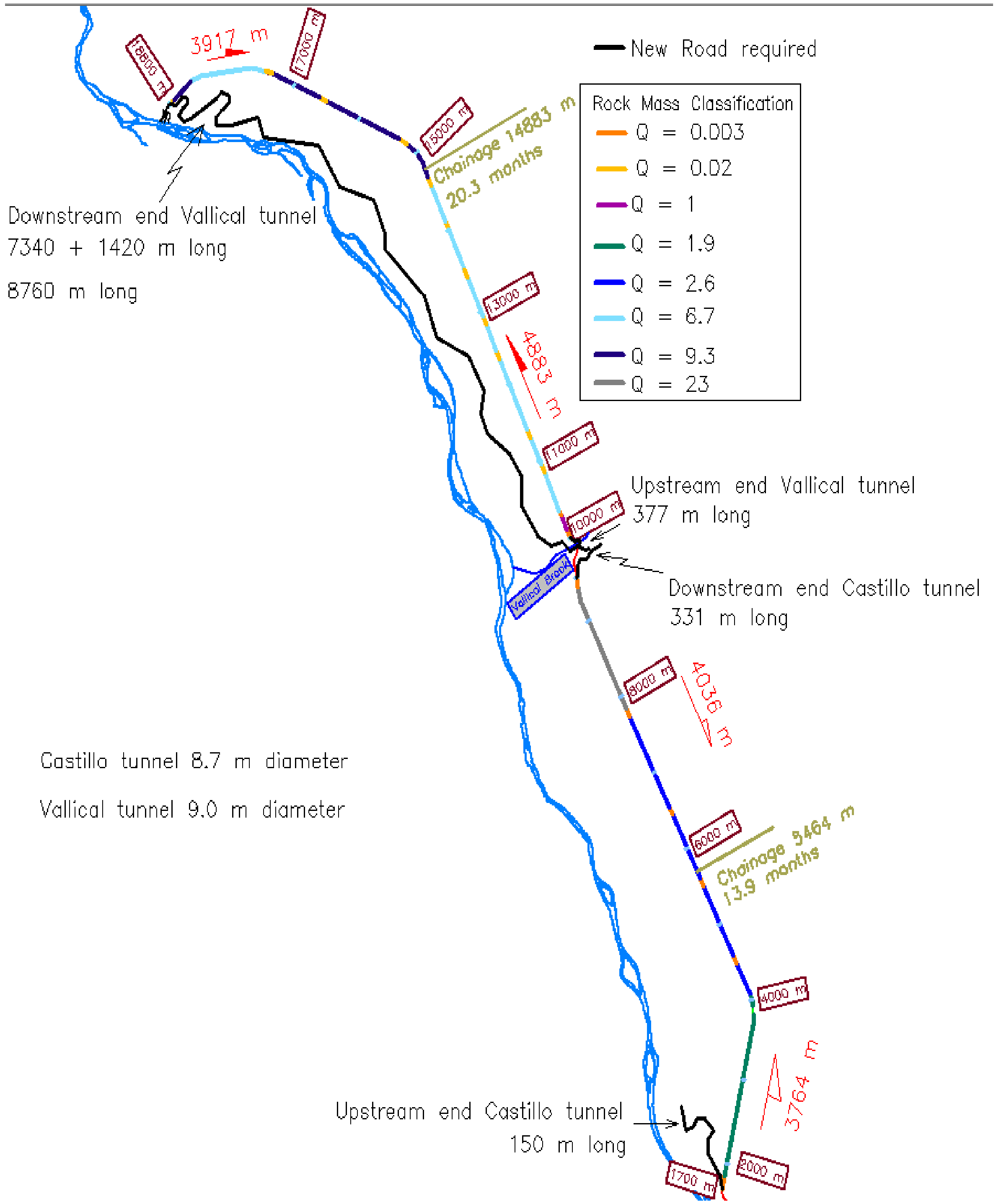
Bolts in roof only.

This case can be considered equal to case 4. (Unlined walls and roof tunnel and concrete invert).

Summary: Vallical Tunnel: 68.8 m^2 (9 m diameter)				
Case	**Rock Mass Classification Q_{ROOF}**	**Equivalent Manning roughness M_{GLOBAL}**	**Hydraulic Area A_h m^2**	**Final rock support adopted for Roof and Walls**
1	0.003	25.1	66.9	Reinforced Shotcrete ribs D40/4 c/c 1.2 m + systematic bolting (Roof & Walls)
2	0.02	24.5	66.9	Reinforced Shotcrete ribs E35/5 c/c 2.3 m ; systematic bolting (Roof & Walls)
3	1.0	44.7	69.3	7 cm Fibercrete in roof; 7 cm shotcrete in walls; systematic bolting (Roof & Walls)
4	6.7	34.4	71.5	Bolts in roof only
5	9.3	34.4	71.5	Bolts in roof only

Rock Mass Classification Q_{ROOF}	Chainage m	Stretch Length m	Equivalent Manning Coefficient M_{GLOBAL}	Final Hydraulic flow Area A_h m^2	Final Hydraulic Radius R_h m	Energy Production losses E_L USD/m	Long term energy losses per stretch USD
0.003	10000 - 10100	100	25.1	66.9	2.23	2515	251,521
1.0	10100 - 10300	200	44.7	69.3	2.27	719	143,865
0.003	10300 - 10350	50	25.1	66.9	2.23	2515	125,761
6.7	10350 - 10850	500	34.4	71.5	2.31	1117	558,560
0.02	10850 - 10950	100	24.5	66.9	2.23	2640	263,970
6.7	10950 - 11300	350	34.4	71.5	2.31	1117	390,992
0.02	11300 - 11400	100	24.5	66.9	2.23	2640	263,970
6.7	11400 - 12550	1150	34.4	71.5	2.31	1117	1,284,688
0.02	12550 - 12650	100	24.5	66.9	2.23	2640	263,970
6.7	12650 - 13000	350	34.4	71.5	2.31	1117	390,992
0.02	13000 - 13100	100	24.5	66.9	2.23	2640	263,970
6.7	13100 - 13400	300	34.4	71.5	2.31	1117	335,136
0.02	13400 - 13500	100	24.5	66.9	2.23	2640	263,970
6.7	13500 - 14150	650	34.4	71.5	2.31	1117	726,128
0.02	14150 - 14250	100	24.5	66.9	2.23	2640	263,970
6.7	14250 - 14750	500	34.4	71.5	2.31	1117	558,560
0.02	14750 - 14850	100	24.5	66.9	2.23	2640	263,970
9.3	14850 - 15350	500	34.4	71.5	2.31	1117	558,560
0.02	15350 - 15450	100	24.5	66.9	2.23	2640	263,970
9.3	15450 - 16550	1100	34.4	71.5	2.31	1117	1,228,832
0.02	16550 - 16650	100	24.5	66.9	2.23	2640	263,970
9.30	16650 - 17400	750	34.4	71.5	2.31	1117	837,840
0.02	17400 - 17500	100	24.5	66.9	2.23	2640	263,970
6.7	17500 - 18300	800	34.4	71.5	2.31	1117	893,696
9.3	18300 - 18700	400	34.4	71.5	2.31	1117	446,848
0.02	18700 - 18800	100	24.5	66.9	2.23	2640	263,970
		8800					**11,635,647**

11,635,647 / 8,800 = 1322 USD/m



New Road required
Rock Mass Classification
Q = 0.003
Q = 0.02
Q = 1
Q = 1.9
Q = 2.6
Q = 6.7
Q = 9.3
Q = 23
3917 m
17000 m
15000 m
Chainage 14883 m
20.3 months
13000 m
4883 m
11000 m
10000 m
Downstream end Vallical tunnel
7340 + 1420 m long
8760 m long
Upstream end Vallical tunnel
377 m long
Downstream end Castillo tunnel
331 m long
Vallical Brook
8000 m
4036 m
6000 m
Chainage 5464 m
13.9 months
4000 m
3764 m
Castillo tunnel 8.7 m diameter
Vallical tunnel 9.0 m diameter
Upstream end Castillo tunnel
150 m long
1700 m
2000 m

8. ALTERNATIVES

8.1 Convenience of Tunnel Alignment Change

8.1.1 Introduction:

In general, an underground tunnel parallel to a predominant joint set is undesired, because it will cause drilling problems, poorer blastability and stability problems.

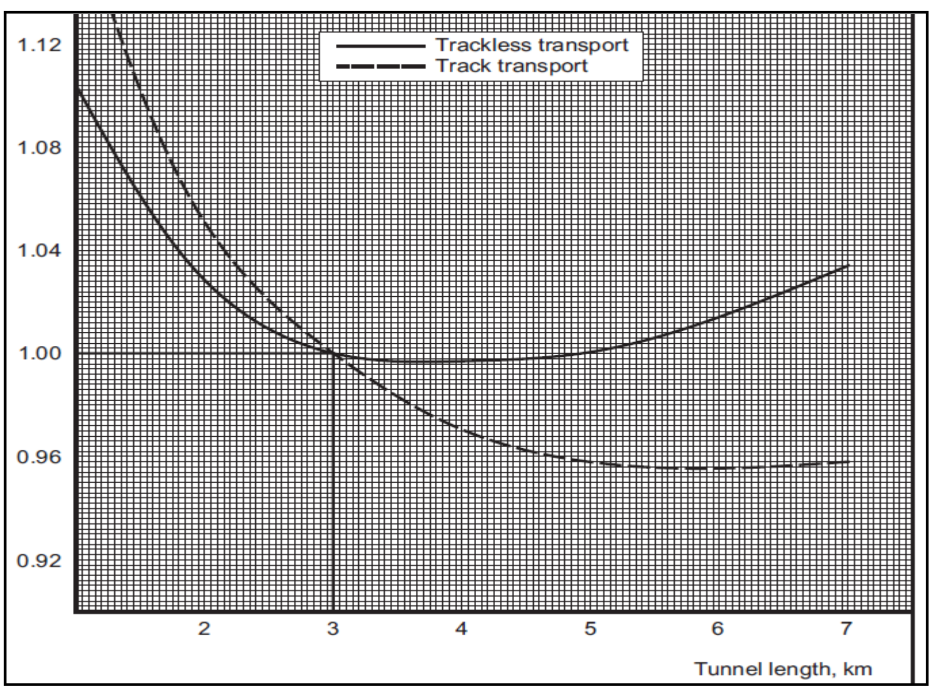
One potential solution to a parallelism between a predominant joint set and the tunnel alignment is to re-orient the last one in a way that increases the angle between the tunnel alignment and a certain joint direction. As a criterion 20 degrees apart between the tunnel alignment and the strike of a certain joint set is considered acceptable for avoiding stability problems caused by discontinuity orientation.

This chapter will analyze if some re-orientations applied to the existing tunnel layout are economically convenient.

8.1.2 Methodology

In practice blastability, drillability, total tunnel length and rock support are theoretically influenced by a re-orientation in tunnel alignment. These four variables influence the advance rate and the total cost, but some practical simplifications can be undertaken.

The Influence in advance rate from a slight change (less than 300 m) in tunnel length from one adit is insignificant as can be visualized in the next chart:



Theory

Blastability:

Scaling Cost

Explosives & Detonators Cost

Total tunnel length (heading length) from one adit:

Hauling Cost

Ventilation Cost

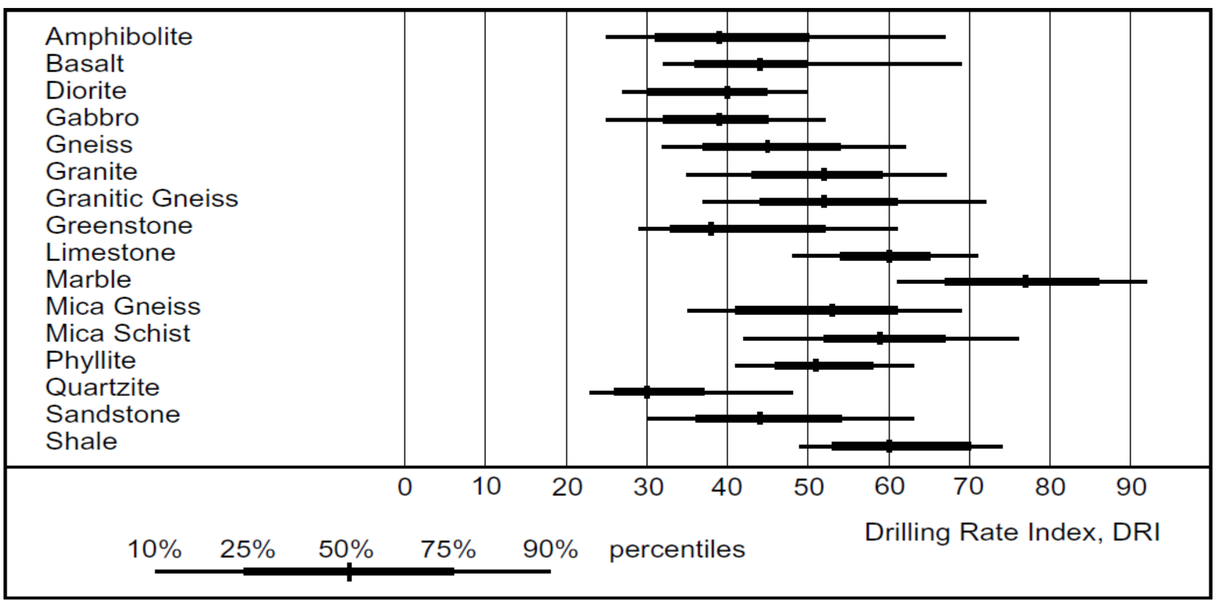
Electrical & Water supply Cost

Labor Cost

Drillability

Drilling Cost

Safety

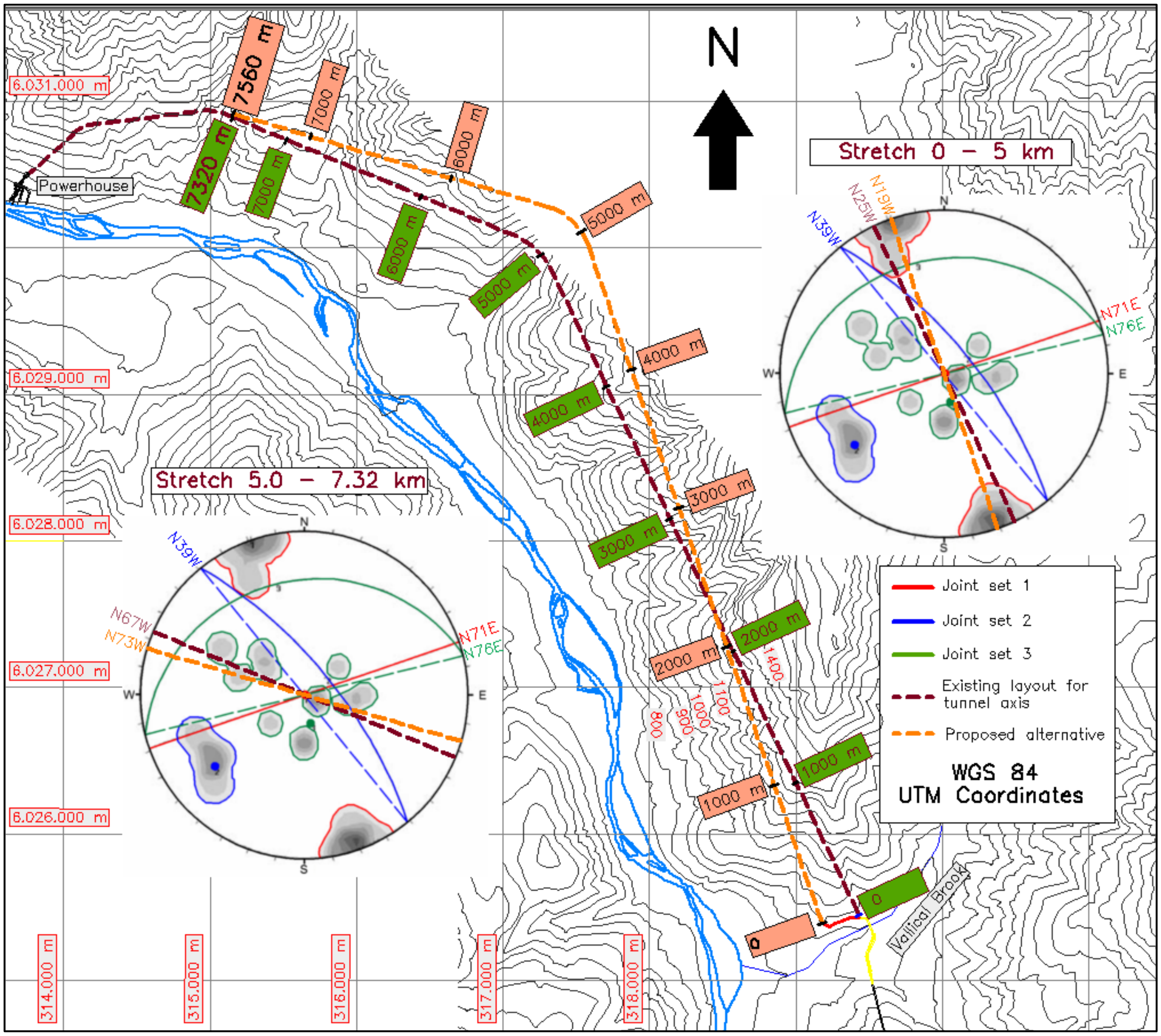


Drillability recorded for different rock types taken from Shokrollah, Z Drill & Blast Tunneling. Advance rate.

From the last chart, it is possible to obtain an average DRI value for granite (representing the Batholith Melado) around 52 and Basalt around 44. (Basalt is the best representation of the Trapa Trapa rock formation in the absence of Andesite information). For alternative analysis.

8.1.3 Case 1: Vallical Tunnel Chainage 10,000 – 17,320 m

This alternative consists of a re-orientation to the Vallical tunnel alignment in 6 degrees (clockwise direction) in order to move it further away from the “Joint set 2” strike.



This new alignment will bring some other changes like:

Extend the tunnel length in 240 m (7560 m – 7320 m).

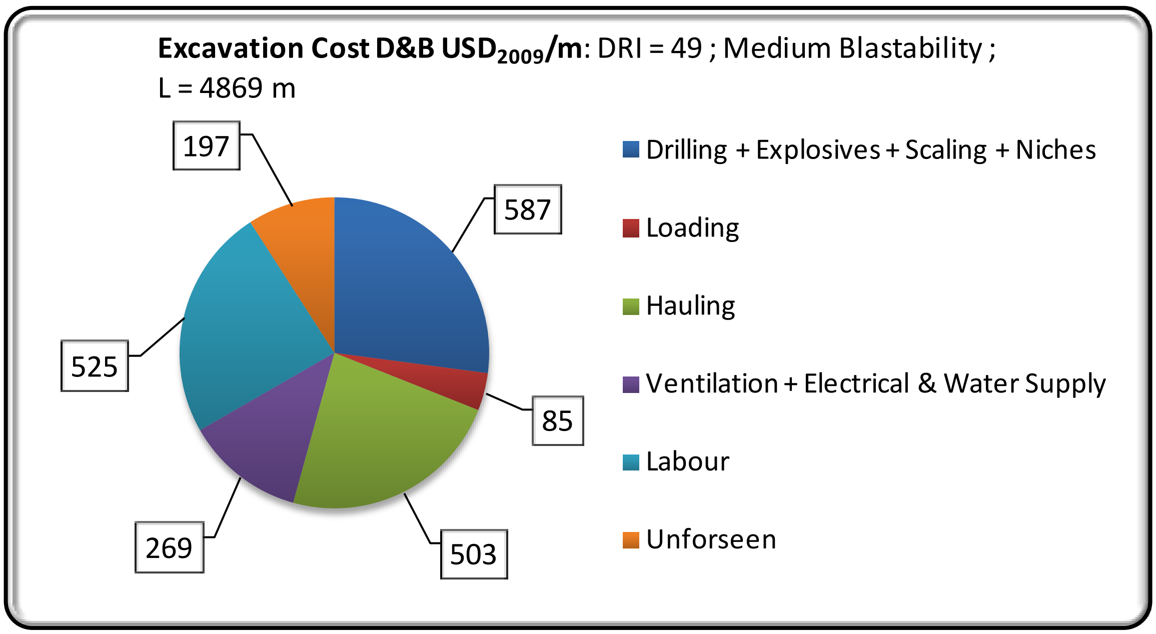
Extend the planned canal that crosses the Vallical brook located just upstream the Vallical tunnel in 270 m (red line).

The tunnel extension will compromise both fair rock mass and weakness zones, but the improvement because of the change in alignment only involves the tunnel in fair rock mass. (It is not expected any distinctive orientation in weakness zones, only crushed rock which makes no difference between the tunnel alignment alternatives analyzed).

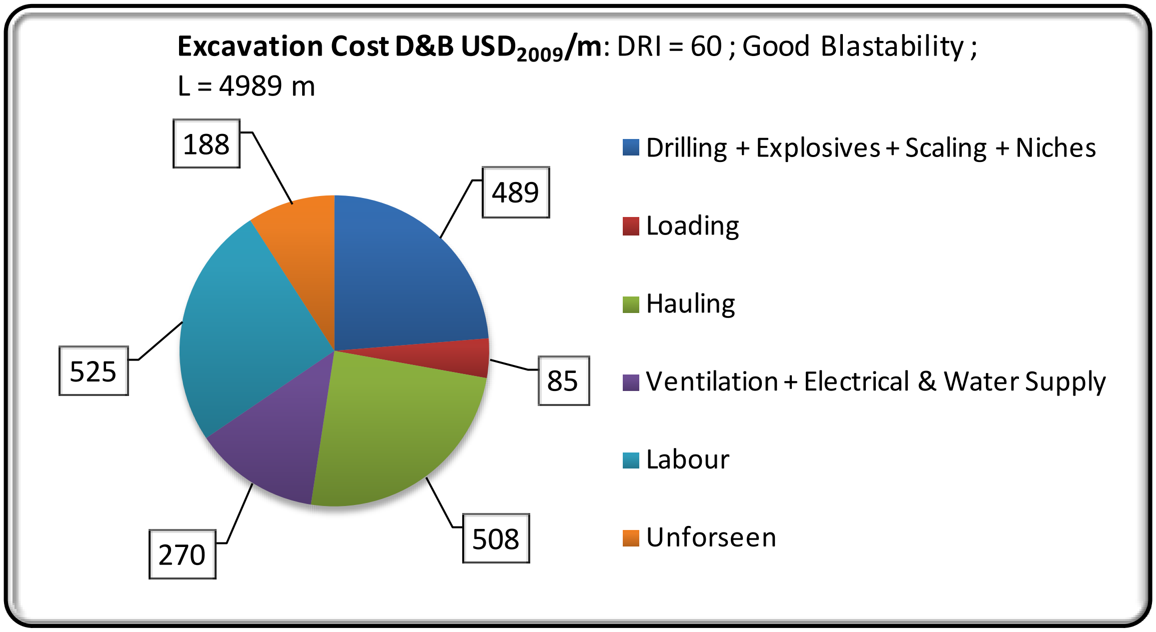
The improvements of drillability and blastability are along the first 5000 m, excluding the last 2320 m because the last stretch has more than 20 degrees apart from the “Joint set 2” strike in the current situation.

For the current tunnel alignment, medium blastability and drillability (DRI = 49) has been considered, and the analysis will focus on changing from medium to good blastability in rock mass conditions just mentioned.

A longer tunnel not only means a cost rise because of the extension itself, but also for the whole tunnel: hauling, ventilation, Electrical & water supply and labor costs will increase.



Cost per round = **2166 USD/m**



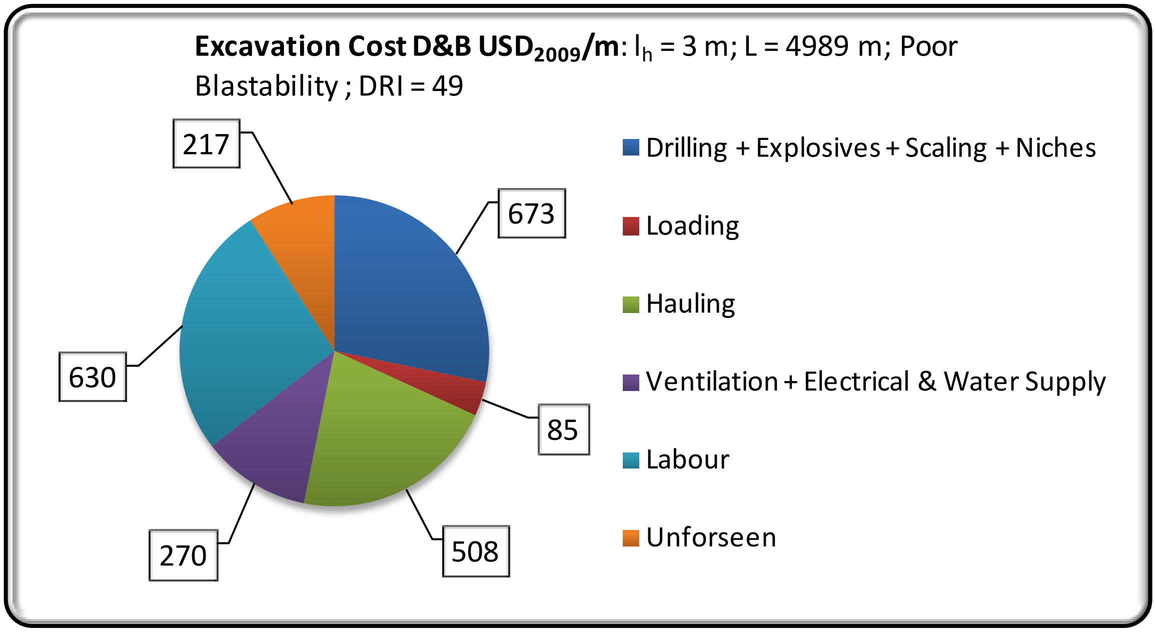
Cost per round: **2065 USD/m**

Category	Unit	DRI = 60 Blastability = Good Tunnel length = 4989 m	DRI = 49 Blastability = Medium Tunnel length = 4869 m	Cost reduction Δ Cost
Drillability Cost	USD/m	258	323	65
Explosives Cost	USD/m	201	217	16
Scaling Cost	USD/m	31	48	17
Total Cost	USD/m	**489**	**587**	**98**

Cost reduction: (2166 USD/m – 2065 USD/m) * 4150 m = **419,150 USD**

On the other hand, the tunnel extension has an investment cost that must be considered.

As it was said before there will be an extension not only in fair rock mass, but also in weakness zone. The tunnel length



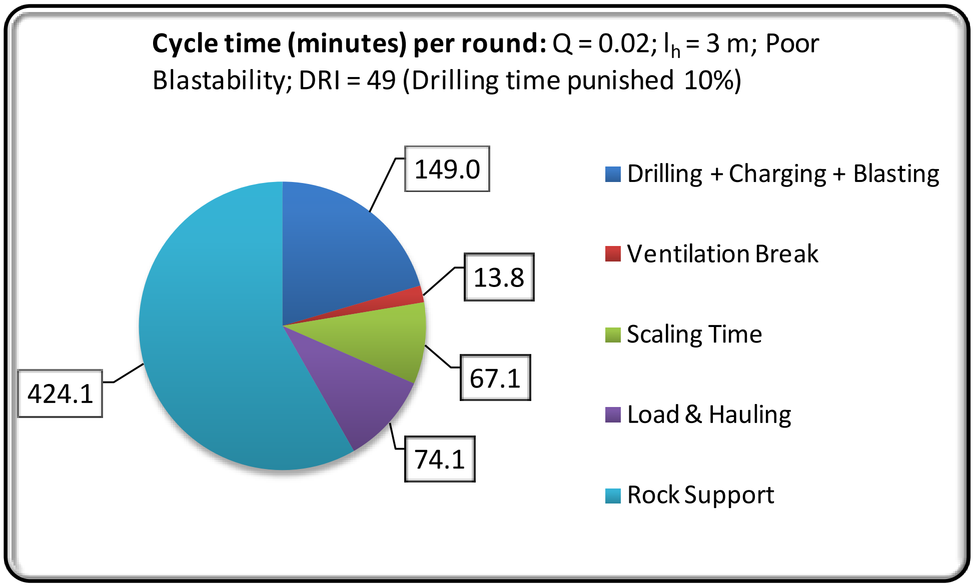
Cost = 2382 USD/m

	Stretch	Rock mass classification Q - Method	Excavation Cost	Rock Support Cost	Total Unit Investment Cost	Gross Investment Cost
	m		USD/m	USD/m	USD/m	USD
Extension Poor Rock	34	0.02	2382	12,475	14,857	511,471
Extension Good Rock	206	9.3	2065	247	2,312	475,369
Sum	**240**					**986,839**

And also it is required an extension of the canal just upstream the Vallical tunnel:

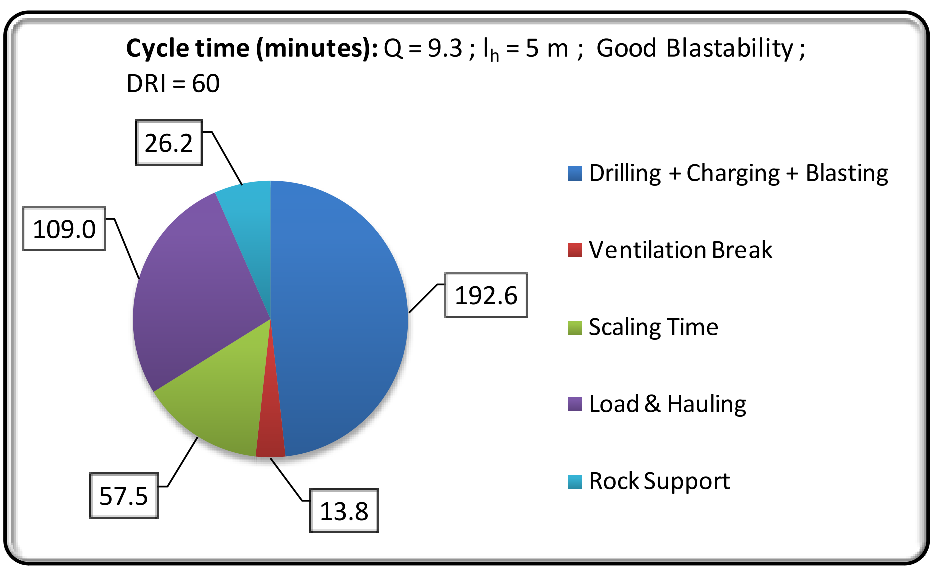
	Stretch	Total Unit Cost	Total Cost
	m	USD/m	USD
Canal cost	270	3500	**945,000**

Extension Tunnel Construction **Period**



Round cycle: 727 min

Working hour per week		120	h/week
Drilled length	l_h	3	m
Pull (Drillhole diameter = 48 mm)	p_r	0.91	
Round length	p_r*l_h	2.73	m
Cycle Time		727	min
Advance rate		27.0	m/week

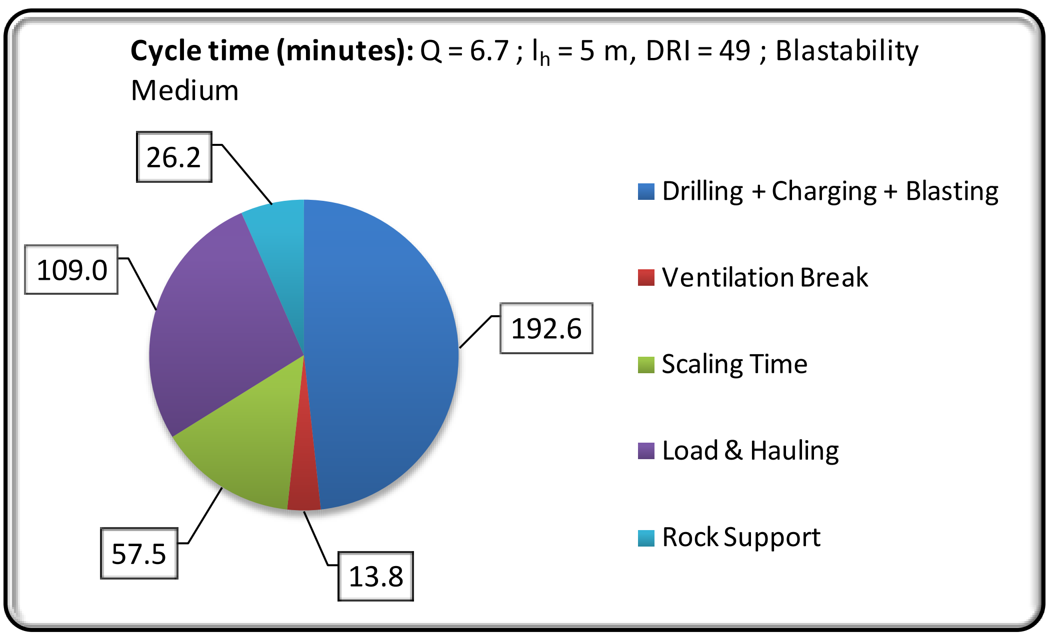


Cycle time = 368 min.

Working hour per week		120	h/week
Drilled length	l_h	5	m
Pull (Drillhole diameter = 48 mm)	p_r	0.91	
Round length	p_r*l_h	4.55	m
Cycle Time		368	min
Advance rate		89.1	m/week

	Stretch	Rock Mass Classification Q - Method	Advance rate	Advance rate
	m		m/week	week
Extension Poor Rock	34	0.02	27.0	1.28
Extension Good Rock	206	9.30	89.1	2.31
Sum	**240**			**3.6**

The advance rate improvement is shown below:



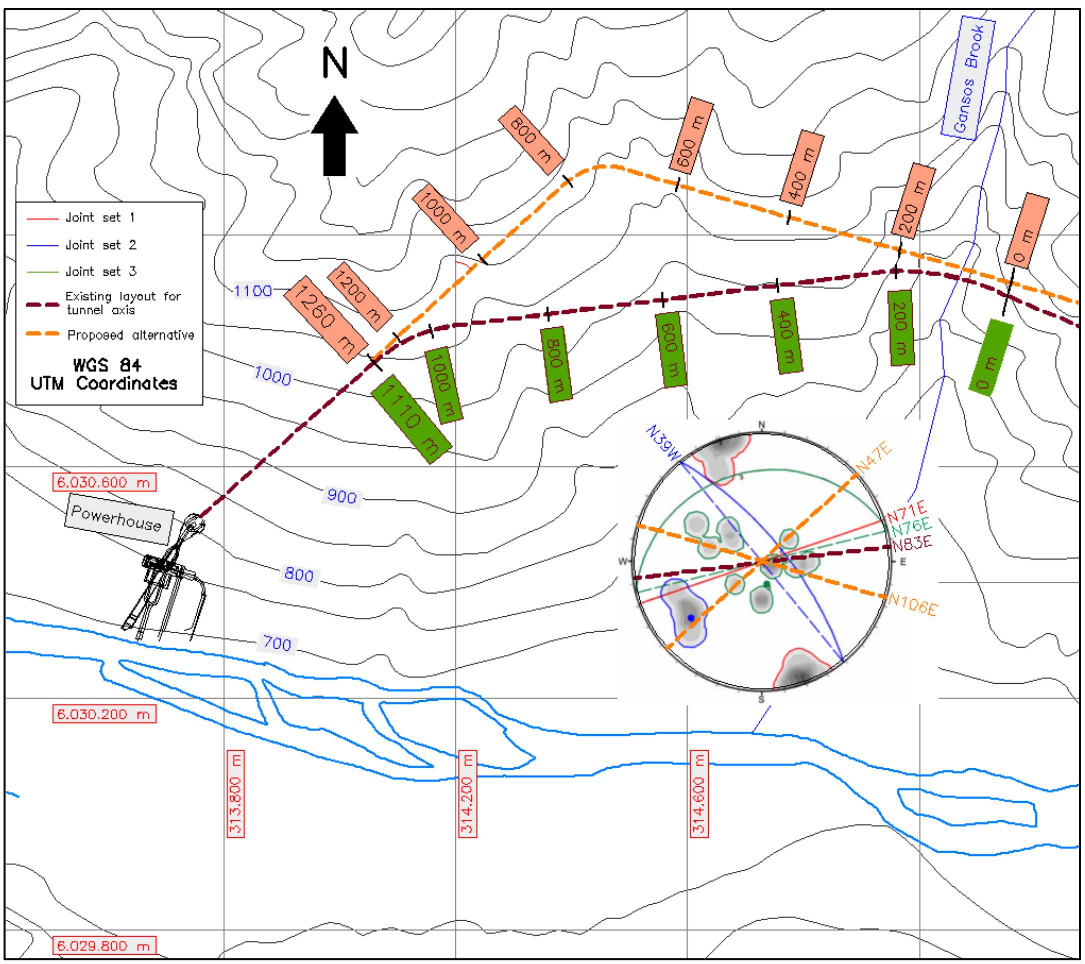
Working hour per week		120	h/week
Drilled length	l_h	5	m
Pull (Drillhole diameter = 48 mm)	p_r	0.91	
Round length	p_r*l_h	4.55	m
Cycle Time		398	min
Advance rate		82.3	m/week

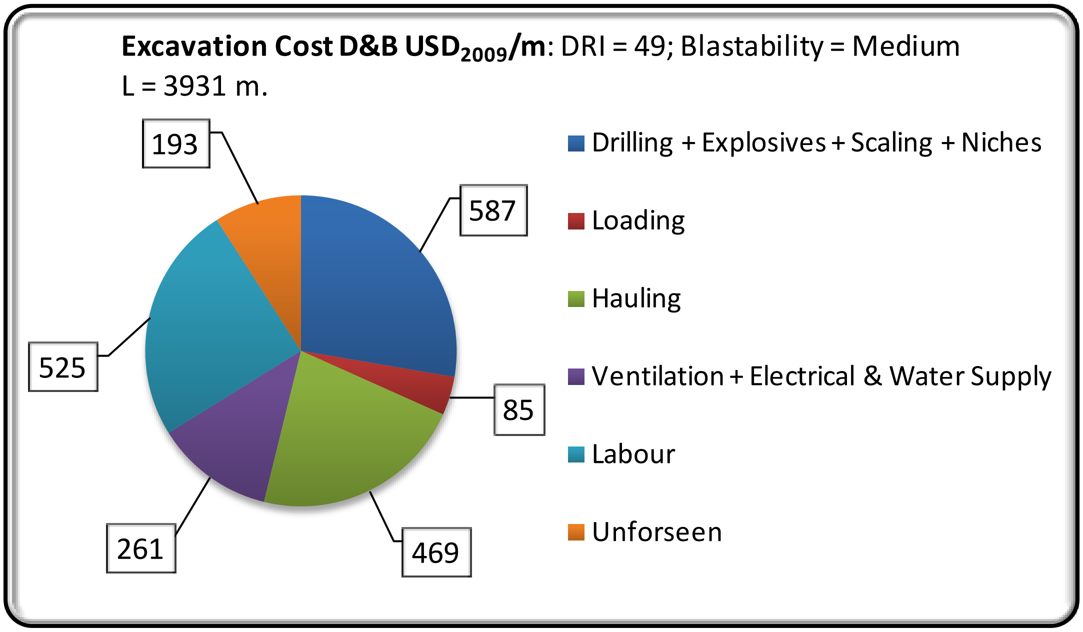
$$\frac{4150\ m}{82.3\ m/week} - \frac{4150\ m}{89.1\ m/week} = 3.9 \text{ week}$$

There is almost no difference in time spent.

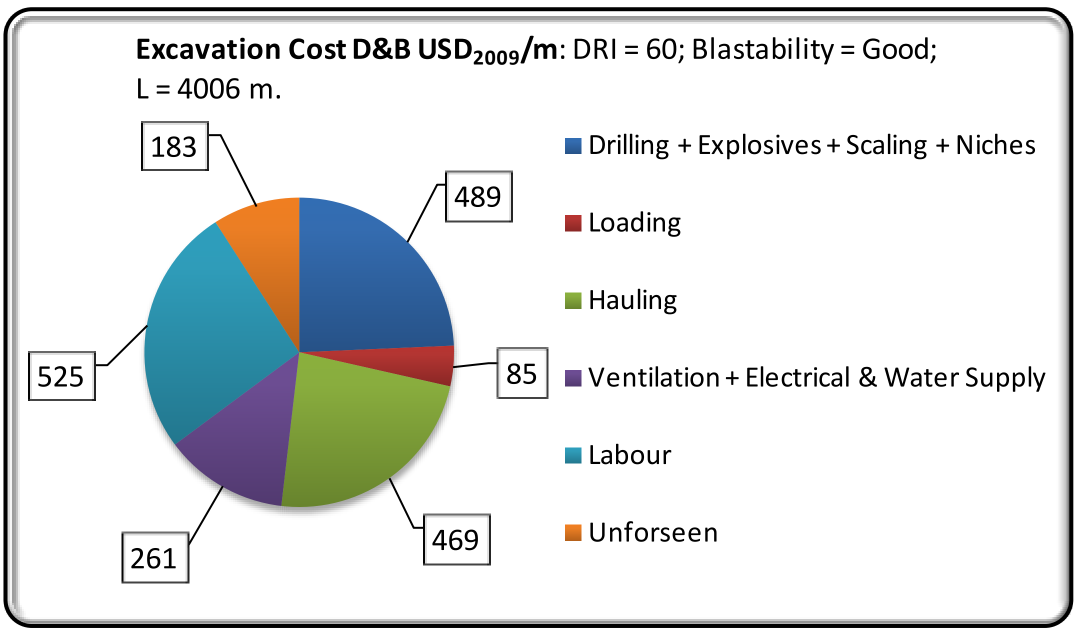
8.1.4 Case 2: Vallical Tunnel Chainage 17,400 – 18,500 m

This alternative consists of an extension of the tunnel length at the end of the Vallical tunnel. The idea is to avoid the almost parallelism between the tunnel alignment and the joint sets 1 and 3.





Total cost 2121 USD/m.



Total cost = 2017 USD/m

From the last two charts it is possible to deduct that the tunnel length factor is not a relevant variable in this analysis because the tunnel extension in 75 m (150/2 m) did not change the unit cost (USD/m) of Hauling, Ventilation, Electrical & Water Supply where tunnel length was the only variable changed which has an influence on them.

The main difference among these two charts are found in the first item which is “Drilling, explosives, scaling and niches” where both drillability in Drilling cost and blastability in Explosives and scaling costs have affected them.The Niches cost is 0 in both cases because the optimum cross section area is above 65 m^2 and therefore is big enough to have dual transit of dump trucks.

The next table shows in detail the difference in unit cost between

Category	Unit	DRI = 60 Blastability = Good Tunnel length = 4006 m	DRI = 49 Blastability = Medium Tunnel length = 3931 m	Cost reduction Δ Cost
Drillability Cost	USD/m	258	323	65
Explosives Cost	USD/m	201	217	16
Scaling Cost	USD/m	31	48	17
Total Cost	USD/m	**489**	**587**	**98**

Cost reduction: (2121 USD/m – 2017 USD/m) * 1100 m = **114,400 USD.**

Rock support

Extension Tunnel (150 m)					
Stretch	Rock Mass Classification Q-Method	Excavated Cost	Rock Support Cost	Total Cost	Gross Investment Cost
m		USD/m	USD/m	USD/m	USD
150	9.3	2017	247	2264	**339,660**

From the previous alternative which also involves the Vallical tunnel is possible to obtain the advance rate for the existing situation which is 82.3 m/week when drillability is medium DRI = 49 and blastability is also medium. Change in advance rate because of tunnel length from one adit was not considered.

$$\frac{1100\ m}{82.3\ m/week} - \frac{1100\ m}{89.1\ m/week} = 1.02 \text{ week}$$

Stretch	Advance rate	Advance rate
m	m/week	week
150	89.1	**1.7**

8.2 Convenience of extra adits

The feasibility study conceived one extra adit for both Castillo and Vallical tunnels. Both adits are located approximately at half of total tunnel length in each case.

The main benefits of considering an extra adit in a tunnel are an earlier tunnel completion and an investment cost reduction because of shortening tunnel stretches to be excavated.

In case a tunnel construction is within the project critical path, the benefit of an earlier tunnel completion becomes more relevant since it influences the plant commissioning and therefore the beginning of the revenues. The latter issue does not apply for this specific project, because tunnels are not within the critical path when extra adits are considered. This is due to the fact that there is a lag between tunnel completions and commissioning of project, mainly because of the supply and assembly of electromechanical equipments period which would take around 34 months starting from the same date as tunnels start construction.

In the case of Castillo tunnel the adit has a length of 963 m, requires an access road of 272 m and meets the Castillo tunnel at Chainage 6,100 m (measured from the intake) which is around 4,400 away from Castillo brook and 3400 m away from Vallical brook.

In the case of Vallical tunnel, the adit is 650 long, requires an access road of 874 m and meets the Vallical tunnel at Chainage 14,450 m which is around 4,400 m away from the Vallical brook and 4,400 m away from the downstream end of the Vallical tunnel.

If adits were not included in the construction layout, each tunnel (Castillo & Vallical) would be excavated from its two ends only. When an extra adit has met the main tunnel, it creates four advance stretches.

Once the extra adit reaches the main tunnel, the advance rate considered is normal for each side of the main tunnel. In practice this is not real, because some downtime occurs when two tunnel faces are being excavated from only one adit.

The portal construction period was considered as 1 month for all the cases.

For this analysis, medium blastability and drillability is considered for the water tunnels (Castillo & Vallical) as it was carried out in the optimum cross section since alignment of the main tunnels are kept.

Cross section area assumed for the adit is 40 m^2. This was considered as the minimum area for the largest equipment needed to construct a tunnel of 64.3 m^2 in the case of Castillo tunnel and 68.8 m^2 in the case of Vallical tunnel.

Only direct cost was considered for both alternatives (with and without the extra adit) since there should not be significant difference in indirect cost, administration and planning cost, constructor profit, etc. between them.

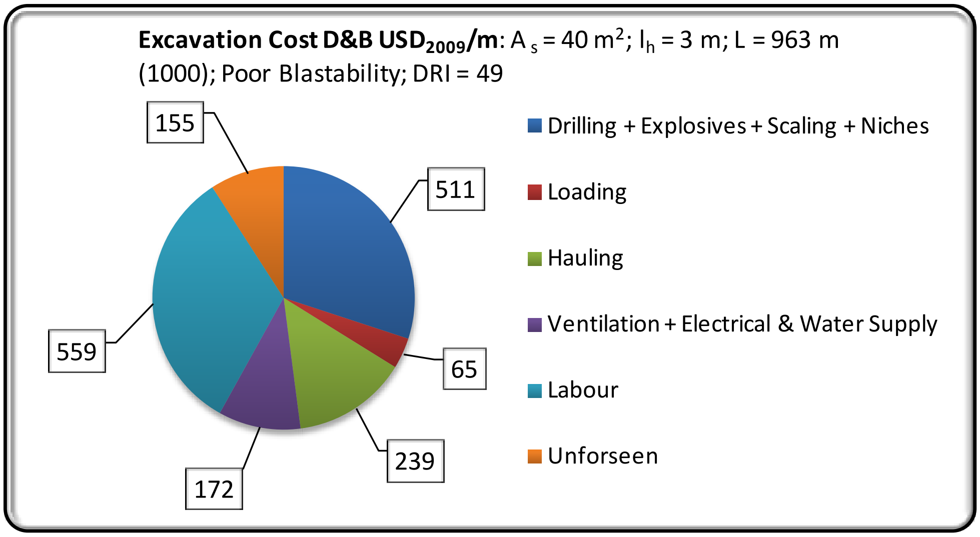
8.2.1 Castillo Adit

8.2.1.1 Disadvantages

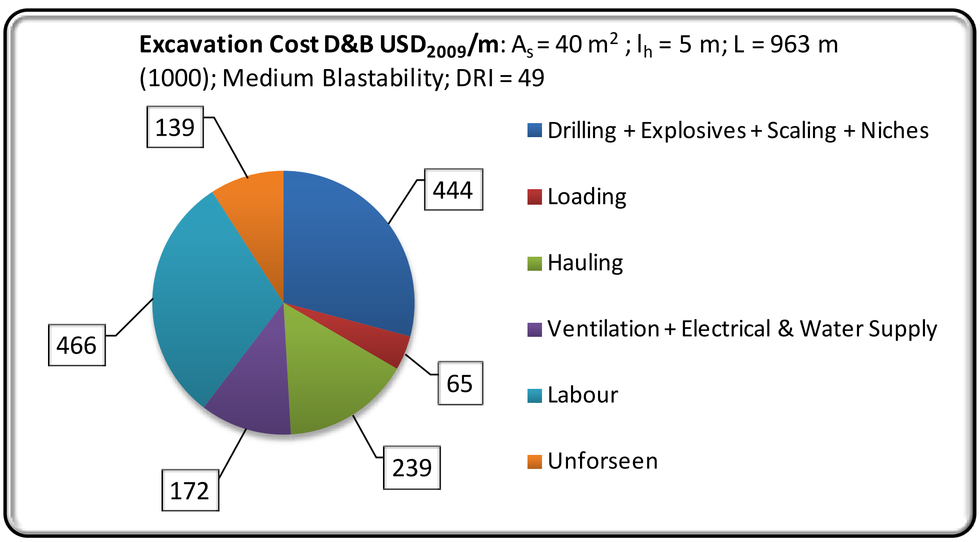
8.2.1.1.1 Castillo Extra Adit Cost

The extra adit was considered as 100 m of poor rock mass condition (surface weathering at the starting stretch) and 863 of good rock mass quality for the rest before meeting the Castillo tunnel. No weakness zone is visualized for the adit.

The resulting cost for the poor rock mass quality stretch is 1700 USD/m and it breaks down as follows:



The resulting cost for the fair rock mass quality stretch is 1525 USD/m and it breaks down as follows:



*

From the previous two charts, it is possible to identify which costs are different: They are “Drilling + Explosives + Scaling + Niches”, “Labor” and “Unforeseen” costs. The first one is due to the difference in blastability and particularly it affects the explosives and scaling costs. The labor cost is affected by drilled length which is different because of change in rock mass quality. The unforeseen cost is explained by the change in the last two, because it was estimated as 10% of the total excavation cost.

Rock Support Cost

The rock support cost is determined in the same way as was done for the optimum cross section analysis. Particularly for 40 m^2 and the rock conditions found for the extra adit, the relevant items are included in the next table:

Q_{ROOF}	Radial Bolts USD/m	Fibercrete USD/m	Shotcrete USD/m	Rebars USD/m	Spiling bolts USD/m	Total Rock Support USD/m
0.003	1,257	2,652	3,349	711	4,122	12,091
23	0	0	0	0	0	0

The summary of the costs involved due to the extra adit for the Castillo tunnel is presented below:

Castillo Extra Adit 963 m; A = 40 m^2						
Chainage	Stretch m	Rock mass classification Q-System	Excavation Cost	Rock Support USD/m	Total Unit Cost USD/m	**Cost USD**
0-100	100	0.003	1700	12,091	13,791	**1,379,114**
100-963	863	23	1525	0	1,525	**1,316,075**
Sum	**963**					**2,695,189**

In addition to the extra adit construction cost, it is needed an access road to the adit which unit cost was estimated from an average of different roads obtained from the feasibility report, resulting a value equal to 1325 USD/m.

Cost access road to Adit		
Unit Cost USD/m	length m	Cost USD
1325	272	**360,400**

8.2.1.2 Advantages

This chapter aims to quantify the benefits of an extra adit which are related to an earlier tunnel completion and an investment cost reduction because of shortening tunnel stretches to be excavated for the Castillo tunnel.

8.2.1.2.1 Cost reduction in Castillo tunnel

With the extra adit, Castillo tunnel is excavated by four tunnel faces. This tunnel length reduction per adit reduces the unit cost of excavation. The first step is to determine the length of these stretches which is the same as to find the place where they meet each other. It is an iterative process to determine this because cost advance rate depends on tunnel stretch length and vice-versa.

The result of advance rate per geology stretch in the Castillo tunnel when an extra adit is taken into account is shown in the next table:

Rock mass classification Q-System	Tunnel length m	Tunnel length reference m	Drilled lenth l_h m	Blastability	Drillability DRI	Tunnel length reference m	Advance rate m/week
0.003	2678	2678				2678	25.5
	1722	2685	3.0	Poor	49		
	1232	2195			(10% punished	2195	25.3
	2168	2168			drilled time)		
1.9	2678	2678				2678	80.8
	1722	2685					
2.6	2678	2678	5.0	Medium	49	2678	84.4
	1232	2195				2195	83.3
	2168	2168					
23	2168	2168				2168	89.3

Tunnel length reference is the length that was used as input in the advance rate computation. Note that not always the length to be excavated in the Castillo tunnel coincides with the reference length used for advance rate estimate, since the length of the extra adit must be considered in the Castillo tunnel stretches where from this extra adit it is excavated (Castillo 3 & Castillo 4 shown in the table below).

Advance rate is not sensitive to small differences in tunnel length from one adit as 2684 m and 2679 m long or 2195 and 2168 m long. Between two similar tunnel lengths, one was taken as reference.

Once the advance rate for each stretch is known, it is possible to know accurately where the different advance stretches are met. This is shown in the following table:

Rock mass classification Q_{ROOF}	Identification of stretch	Chainage	Stretch Length m	Cumulative distance Castillo 1 m	Cumulative distance Castillo 2 m	Cumulative distance Castillo 3 m	Cumulative distance Castillo 4 m
0.003	Surface weathering	1700 - 1850	150	150			
1.9	Fair Andesite - Fair tunnel orientation	1850 - 3900	2050	2200			
2.6	Fair Andesite - Good Orientation	3900 - 4330	430	2630			
0.003	Inferred fault zone around Las Mulas brook	4330 - 4430	100	**2678**		**1722**	
1.9	Fair Andesite - Good tunnel Orientation	4430 - 5350	920			1670	
0.003	Inferred fault zone around Las Yeguas brook	5350 - 5450	100			750	
2.6	Fair Andesite - Good tunnel Orientation	5450 - 6380	930			650	280
0.003	Inferred fault zone I from aerial photos	6380 - 6480	100				380
2.6	Fair Andesite - Good tunnel Orientation	6480 - 7670	1190		**2168**		**1232**
0.003	Inferred fault zone II from aerial photos	7670 - 7770	100		1830		
23	Good Andesite - Good Orientation	7770 - 9400	1630		1730		
0.003	Surface Weathering	9400 - 9500	100		100		
	Sum	**1700 - 9500**	**7800**				
						Start	**End**

From the previous table, it is possible to say that:

Castillo 1 & 3 meet each other at chainage 4378 m which is 2678 m away from the Castillo brook (Or upstream end of Castillo tunnel).

Castillo 2 & 4 meet each other at chainage 7332 m which is 2168 m away from the Vallical brook (Or downstream end of Castillo tunnel).

Extra adit meets Castillo tunnel at Chainage 6100, and the first stretch toward Castillo brook is 650 m long and toward Vallical brook is 280 m long.

Once the length of each stretch is known, it is possible to obtain the excavation cost which is:

D & B Excavation Cost :			64.3	m^2	
Stretch	Drilled length l_h m	Blastability	Rock mass classification Q-system	Length reference m	Excavation Cost USD/m
Castillo 1 (L = 2678 m)	3	Poor	0.003	2678	**2187**
Castillo 2 (L = 2168 m)				2168	**2156**
Castillo 3 (L = 1722 m)				2685	**2187**
Castillo 4 (L = 1232 m)				2195	**2157**
Castillo 1 (L = 2678 m)	5	Medium	1 -23	2678	**1983**
Castillo 2 (L = 2168 m)				2168	**1950**
Castillo 3 (L = 1722 m)				2685	**1983**
Castillo 4 (L = 1232 m)				2195	**1952**

The rock support cost is the same as considered in the economic cross section analysis. Below, just the final rock support cost is presented for 64.3 m^2 or 8.7 m diameter for a “horse shoe” shape tunnel:

Rock mass classification Q_{ROOF}	Total Rock Support USD/m
0.003	15,334
1.9	1,255
2.6	1,095
23	0

Finally, the Castillo tunnel has a new cost of **34,689,965 USD.** Details for each stretch are shown below:

Rock mass classification Q $_{ROOF}$	Stretch Length m	Chainage	D & B Cost USD/m	Cost concrete invert + wire mesh USD/m	Cost Rock Support USD/m	Total Gross Cost/m	Total Cost/stretch USD
0.003	150	1700 - 1850	2,187	367	15,334	17,888	2,683,162
1.9	2050	1850 - 3900	1,983	367	1,255	3,604	7,388,736
2.6	430	3900 - 4330	1,983	367	1,095	3,445	1,481,275
0.003	100	4330 - 4430	2,187	367	15,334	17,888	1,788,774
1.9	920	4430 - 5350	1,983	367	1,095	3,445	3,169,240
0.003	100	5350 - 5450	2,187	367	15,334	17,888	1,788,774
2.6	930	5450 - 6380	1,974	367	1,095	3,435	3,195,008
0.003	100	6380 - 6480	2,157	367	15,334	17,858	1,785,774
2.6	1190	6480 - 7670	1,951	367	1,095	3,413	4,061,776
0.003	100	7670 - 7770	2,157	367	15,334	17,858	1,785,774
23	1630	7770 - 9400	1,950	367	0	2,317	3,775,897
0.003	100	9400 - 9500	2,157	367	15,334	17,858	1,785,774
	7800					**SUM**	**34,689,965**

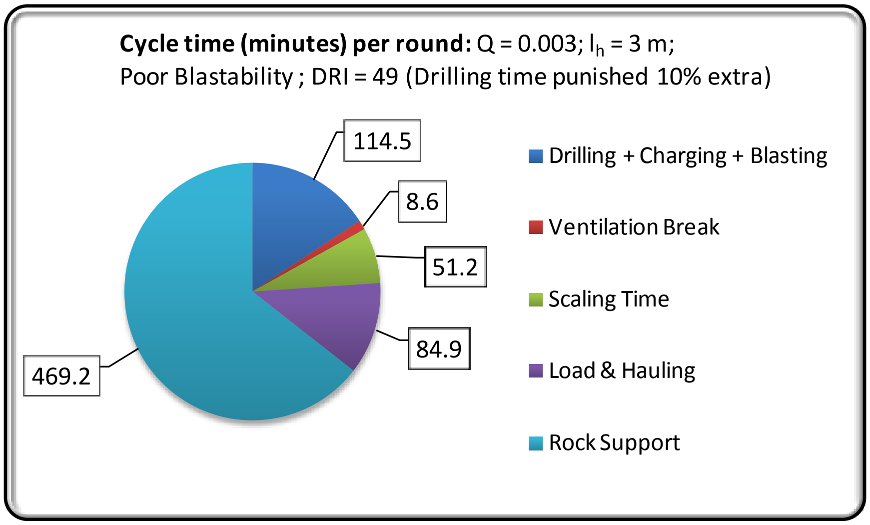
This final cost for Castillo tunnel is almost 600,000 USD cheaper than the cost obtained when no extra adit was computed. The exact cost in that case is 35,273,002 USD and the details are shown in the optimum cross section analysis.

8.2.1.2.2 <u>Earlier tunnel completion</u>

As long as the extra adit does not reach the Castillo tunnel, only two tunnel faces will be working on the Castillo tunnel. Once the extra adit reaches the Castillo tunnel, four tunnel faces will be working on the Castillo Tunnel. Therefore it is of prime importance to determine when the extra adit reaches the Castillo tunnel.

Advance rate Castillo Adit

In poor rock mass condition the adit advance rate is shown in the next chart:



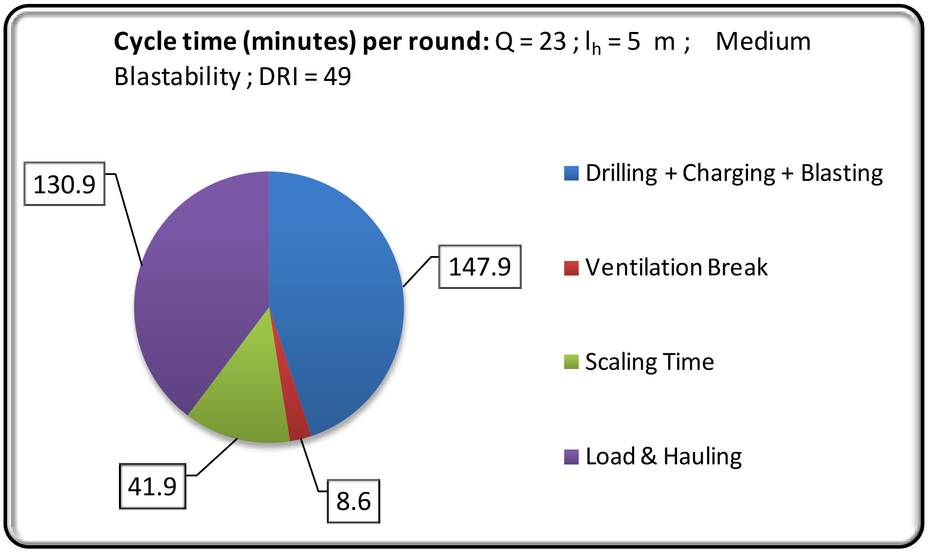
Cycle time 728 min

Correction factor for tunnel length (963 m) k_{le} = 1.11

Total cycle time: (728-469.2) *(1.11-1) + 728 = **755 min.** (Tunnel length only affects the excavation advance rate).

Working hour per week		120	h/week
Drilled length	l_h	3	m
Pull	p_r	91%	
Round length	p_r*l_h	2.73	m
Cycle Time min		**755**	min
Advance rate		**26.0**	m/week

And for the tunnel stretch crossing good rock mass, the cycle time is 330 min (without tunnel length correction factor) and it breaks down as follows:



Cycle time = 147.9 + 8.6 + 41.9 + 130.9 = 330 min

Correction factor for tunnel length (963 m) k_{le} = 1.1

Total cycle time: 330 * 1.11 = **364 min**

Working hour per week		120	h/week
Drilled length	l_h	3	m
Pull	p_r	91%	
Round length	p_r*l_h	4.55	m
Cycle Time min		**364**	min
Advance rate		**90.1**	m/week

Stretch	m	Rock mass classification Q-System	Advance rate m/week	Construction time month/stretch
0-100	100	0.003	26.0	0.9
100-963	863	23	90.1	2.2
Sum	**963**			**3.1**

Also the access road from the existing road to the extra adit requires some days to be constructed and the details are shown below:

Access road	272	m
Road advance rate	50	m/day
Construction period	5.44	days
Construction period	0.2	month

Finally one month was considered as portal construction period.

Therefore, construction Castillo tunnel from the extra adit starts after 4.3 months (3.1 month + 0.2 month + 1 month). The extra adit comes out in the Castillo tunnel at Chainage 6100 m, in a stretch where the rock is classified as 2.6 in the Q system between chainage 5450 m and 6380 m (6380 m – 5450 m = 930 m long). Hence, in the Castillo tunnel, 650 m of this 930 m stretch will be constructed towards upstream and 280 m towards downstream at an advance rate equal to 84.4 m/week for either direction.

The time elapsed to reach the Chainage 5450 m and 6380 m respectively in the Castillo tunnel from Chainage 6100 m is provided in the next table:

Advance rate Chainage 5450 -6380	To Castillo Brook (Or upstream)		To Vallical brook (Or downstream)	
	Distance	time elapsed	Distance	time elapsed
m/week	m	month	m	month
-	0	4.3	0	4.3
84.4	**650**	6.1	**280**	5.1

With regard to the two Castillo tunnel ends, they also require an access road. Access road required to the upstream end adit is 150 m and the downstream end adit is 331 m.

Upstream end access road		
Upstream end	150	m
Road advance rate	50	m/day
time elapsed	3	day
time elapsed	0.1	month

Downstream end access road		
Upstream end	331	m
Road advance rate	50	m/day
time elapsed	7	day
time elapsed	0.2	month

In addition to access road construction time, one month is added in each Castillo tunnel end for portal construction time before tunnel excavation starts. Therefore the upstream end or “Castillo 1” starts 1.1 months (0.1 month + 1 month) after road construction initiation. Likewise, the downstream end or “Castillo 2” starts 1.2 months after road construction initiation.

Finally, the time elapsed at the end of each stretch from each adit is shown in the following table:

Chainage	Stretch Length m	Advance rate m/week	Advance rate month/stretch	Cumulative time elapsed Castillo 1 month	Cumulative time elapsed Castillo 2 month	Cumulative time elapsed Castillo 3 month	Cumulative time elapsed Castillo 4 month
1700 - 1850	150	25.5	1.4	2.5			
1850 - 3900	2050	80.8	5.9	8.4			
3900 - 4330	430	84.4	1.2	9.6			
4330 - 4430	100	25.5	0.9	**10.0**		**10.0**	
4430 - 5350	920	84.4	2.5			**8.6**	
5350 - 5450	100	25.5	0.9			6.0	
5450 - 6380	930	84.4	2.6			6.1	5.1
6380 - 6480	100	25.3	0.9				**6.0**
6480 - 7670	1190	83.3	3.3		**8.3**		**8.3**
7670 - 7770	100	25.3	0.9		7.4		
7770 - 9400	1630	89.3	4.3		6.5		
9400 - 9500	100	25.3	0.9		2.2		
1700 - 9500	**7800**						
						Start	**End**

From the previous table, it is possible to say that:

The first stretch of "Castillo 1" tunnel (Chainage 1700 - 1850) is completed after 2.5 months (green color), which is the sum of the first stretch equal to 150 m long in 1.4 months and 1.1 months related to road construction and portal. The time consumption for the following stretches until the end (brown cell) is only related to the advance rate in the tunnel.

For "Castillo 2", "Castillo 3" and "Castillo 4" the computation is exactly the same. The green cell is the sum of the previous construction time consumptions plus the time spent to excavate the corresponding stretch. The following stretches until the end (brown cell) is only related to the advance rate in the tunnel.

Castillo 1 & 3 meet each other after 10 months and Castillo 2 & 4 meet each other after 8.3 months.

Therefore, Castillo tunnel is completed after **10.0 months** (or 3.3 trimesters). This is 3.9 months (or 1.3 trimesters) faster than eliminating the extra adit.

8.2.1.3 Evaluation

This chapter aims to trades off the advantages and disadvantages of an extra adit in economic terms for the Castillo tunnel.

About the cost, not only the gross investment cost already estimated must be considered when evaluating both alternatives, but also the interest on the capital invested (or interest under construction) must be included in order to quantify the benefit of an earlier tunnel completion which is the case of the extra adit. This extra cost is called compounding the gross investment.

The total cost is assumed to be paid evenly distributed and monthly. This monthly payment is the total gross investment I_g divided by the number of months n.

The compounding factor R moves forward all the regular monthly payment to the end of the tunnel construction:

$$R = \frac{(1+i)^n - 1}{i}$$

Where i represents the monthly interest during construction and n represents in this case the number of months.

$i_{ANNUAL} = 10\% \Rightarrow i_{MONTHLY} = (1+ i_{ANNUAL})^{1/12} - 1 = 0.8\%$.

Finally the value obtained is again compounded to the commissioning of the plant which would occur 1 year after the tunnel completion.

Castillo Tunnel is not as part of critical path			Without Adit	With Adit	Difference
Access Road + Portal + Castillo Tunnel	n	Month	13.9	10.0	3.9
Gross Investment Cost Castillo Tunnel		USD	35,273,002	34,689,926	583,076
Adit Cost (963 m)		USD	0	2,695,189	-2,695,189
Access road to adit (272 m)		USD	0	360,400	-360,400
Total Gross Investment Cost	I_g	USD	35,273,002	37,745,515	-2,472,513
Equivalent monthly payment	I_g/n	USD	2,537,626	3,774,551	-1,236,925
Compounding factor	R		14.6	10.4	4.3
Investment cost compounding	$R*I_g/n$	USD	37,145,857	39,129,168	-1,983,311
Total Cost at Commissioning	$R*I_g/n*(1+i)^{12}$	USD	40,860,443	43,215,741	**-2,355,298**

Therefore, if the Castillo tunnel is not part of the critical path, an extra adit is not convenient because an earlier completion benefit and a cost reduction benefit because of a shorter advance stretches does not pay off the extra cost of the adit itself and its required access road.

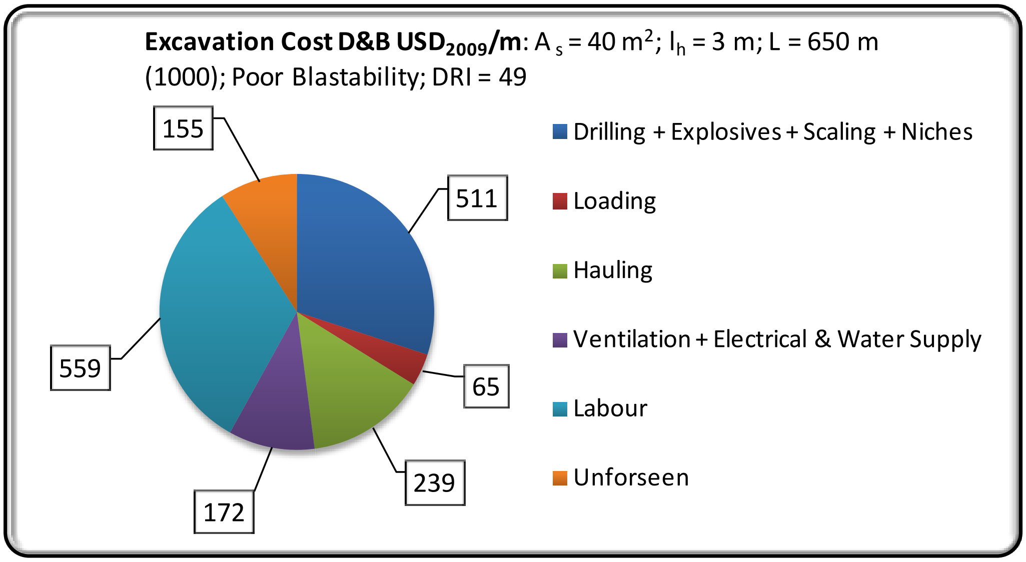
8.2.2 Vallical Adit

8.2.2.1 Disadvantage

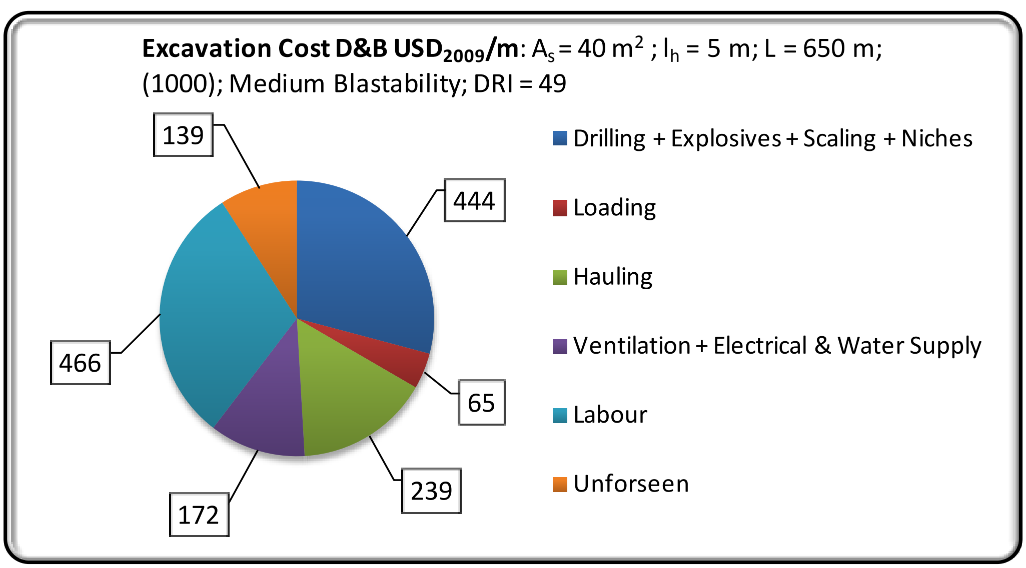
8.2.2.1.1 <u>Vallical Extra Adit Cost</u>

The extra adit was considered as 100 m of poor rock mass condition (surface weathering at the starting stretch) and 550 of fair rock mass quality for the rest before meeting the Vallical tunnel. No weakness zone is visualized for the adit.

The resulting cost for the poor rock mass quality stretch is 1700 USD/m and it breaks down as follows:



511 + 65+ 239 +172 + 559 + 155 = 1700 USD/m



444 + 65 + 239 + 172 + 466 +139 = 1525 USD/m

Rock support

The rock support cost is determined in the same way as was done for the optimum cross section analysis. Particularly for 40 m^2 and the rock conditions found for the extra adit, the relevant items are included in the next table:

Vallical extra adit A = 40 m^2						
Q_{ROOF}	Radial Bolts USD/m	Fibercrete USD/m	Shotcrete USD/m	Rebars USD/m	Spiling bolts USD/m	Total Rock Support USD/m
0.02	873	2,652	1,398	464	4,122	9,509
6.67	0	0	0	0	0	0

The summary of the costs involved due to the extra adit for the Vallical tunnel is presented below:

Vallical Adit 650 m ; A = 40 m^2						
Chainage	Rock mass classification Q-System	length m	Excavation Cost USD/m	Rock Support USD/m	Total Unit Cost USD/m	**Cost USD**
0 - 100	0.02	100	1700	9,509	11,209	1,120,872
100 - 650	6.7	550	1525	0	1,525	838,750
						1,959,622

As it was said before, the estimated average unit cost for road is 1325 USD/m. This is used to quantify the cost of the access road needed for the extra adit.

Cost access road to extra Adit		
Unit Cost	Road length	Cost
USD/m	m	USD
1325	834	**1,105,050**

8.2.2.2 Advantage

This chapter aims to quantify the benefits of an extra adit which are related to an earlier tunnel completion and an investment cost reduction because of shortening tunnel stretches to be excavated for the Vallical tunnel.

8.2.2.2.1 Cost reduction in Vallical tunnel

In Vallical tunnel one extra adit also will trigger a tunnel length reduction per adit compared to the alternative without extra adits, since four tunnel faces will be working in parallel instead of two from the ends. It is vital to know the advance rate from each tunnel face in order to find the place where they meet each other which determines the length of each stretch.

The result of advance rate per geology stretch in the Vallical tunnel when an extra adit is taken into account is shown in the next table:

Rock mass classification Q-System	Tunnel length m	Tunnel length reference m	Drilled lenth l_h m	Blastability	Drillability DRI	Advance rate m/week
0.003	3019	3019	3.0	Poor	49 (10% punished drilling time)	25.1
0.02	3019	3019				27.0
	2194	2194				26.7
	1431	2081				26.7
	2156	2806				26.9
1.0	3019	3019	5.0	Medium	49	66.1
6.7	3019	3019				82.1
	2194	2194				80.3
	1431	2081				80.0
	2156	2806				81.6
9.3	2194	2194				80.3
	2156	2806				81.6

Tunnel length reference is the length that was used as input in the advance rate computation. Some tunnel length references are longer than the corresponding tunnel length. Those cases are including the adit length in addition to the tunnel length for the excavation advance rate.

Once the advance rate for each stretch is known, it is possible to know accurately where the different advance stretches are met. This is shown in the following table:

Q_{ROOF}	Identification of stretch	Chainage	Length m	Cumulative distance Vallical 1 m	Cumulative distance Vallical 2 m	Cumulative distance Vallical 3 m	Cumulative distance Vallical 4 m
0.003	Surface weathering & inferred fault zone 3	10000 - 10100	100	100			
1.0	Good orientation - Fair Andesite	10100 - 10300	200	300			
0.003	Weakness zone - Change in rock formation	10300 - 10350	50	350			
6.7	Fair Orientation - Fair Granite	10350 - 10850	500	850			
0.02	Inferred fault zone 4	10850 - 10950	100	950			
6.7	Fair Orientation - Fair Granite	10950 - 11300	350	1300			
0.02	Inferred fault zone 5	11300 - 11400	100	1400			
6.7	Fair Orientation - Fair Granite	11400 - 12550	1150	2550			
0.02	Inferred fault zone 6	12550 - 12650	100	2650			
6.7	Fair Orientation - Fair Granite	12650 - 13000	350	3000			
0.02	Inferred fault zone 7	13000 - 13100	100	**3019**		**1431**	
6.7	Fair Orientation - Fair Granite	13100 - 13400	300			1350	
0.02	Inferred fault zone 8	13400 - 13500	100			1050	
6.7	Fair Orientation - Fair Granite	13500 - 14150	650			950	
0.02	Inferred fault zone 9	14150 - 14250	100			300	
6.7	Fair Orientation - Fair Granite	14250 - 14750	500			200	300
0.02	Inferred fault zone 10	14750 - 14850	100				400
6.7	Fair Orientation - Fair Granite	14850 - 15350	500				900
0.02	Inferred fault zone 11	15350 - 15450	100				1000
9.3	Good orientation - Fair Granite	15450 - 16550	1100				2100
0.02	Inferred fault zone 12	16550 - 16650	100		**2194**		**2156**
9.30	Good orientation - Fair Granite	16650 - 17400	750		2150		
0.02	Inferred Fault zone 13	17400 - 17500	100		1400		
6.7	Fair Orientation - Fair Granite	17500 - 18300	800		1300		
9.3	Good orientation - Fair Granite	18300 - 18700	400		500		
0.02	Surface weathering	18700 - 18800	100		100		
			8800				
						Start	**End**

From the previous table, it is possible to say that:

Vallical 1 & 3 meet each other at chainage 13,019 m which is 3,019 m away from the Vallical brook (Or upstream end of Vallical tunnel).

Castillo 2 & 4 meet each other at chainage 16,606 m which is 2194 m away from the downstream end of Vallical brook.

Extra adit meets Castillo tunnel at Chainage 14,250 m, and the first stretch toward Vallical brook is 200 m long and toward Vallical brook is 300 m long.

Once the length of each stretch is known, it is possible to obtain the excavation cost which is:

D & B Excavation Cost : 68.8 m^2						
Stretch	Tunnel length m	l_h m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 1	3019	3	Poor	0.003 - 002	3019	2297
Vallical 2	2194	3	Poor	0.003 - 002	2194	2244
Vallical 3	1431	3	Poor	0.003 - 002	2081	2237
Vallical 4	2156	3	Poor	0.003 - 002	2806	2284
Vallical 1	3019	5	Medium	1 -23	3019	2087
Vallical 2	2194	5	Medium	1 -23	2194	2032
Vallical 3	1431	5	Medium	1 -23	2081	2024
Vallical 4	2156	5	Medium	1 -23	2806	2073

The rock support cost is the same as considered in the economic cross section analysis. Below, just the final rock support cost is presented for 68.8 m^2 or 9.0 m diameter for a “horse shoe” shape tunnel:

Rock mass classification Q_{ROOF}	Total Rock Support USD/m
0.003	15,863
0.02	12,475
1.0	2,135
6.7	247
9.3	247

Finally, the Vallical tunnel has a new cost of **40,045,110 USD.** Details for each stretch are shown below:

Rock mass classification Q_{ROOF}	Stretch Length m	Chainage	D & B Cost USD/m	Cost concrete invert + wire mesh USD/m	Rock Support Cost USD/m	Total Unit Cost USD/m	Total Cost/stretch USD
0.003	100	10000 - 10100	2297	379	15,863	18,539	1,853,915
1.0	200	10100 - 10300	2087	379	2,135	4,602	920,314
0.003	50	10300 - 10350	2297	379	15,863	18,539	926,957
6.7	500	10350 - 10850	2087	379	247	2,714	1,356,770
0.02	100	10850 - 10950	2297	379	12,475	15,151	1,515,114
6.7	350	10950 - 11300	2087	379	247	2,714	949,739
0.02	100	11300 - 11400	2297	379	12,475	15,151	1,515,114
6.7	1150	11400 - 12550	2087	379	247	2,714	3,120,571
0.02	100	12550 - 12650	2237	379	12,475	15,091	1,509,114
6.7	350	12650 - 13000	2024	379	247	2,651	927,689
0.02	100	13000 - 13100	2237	379	12,475	15,091	1,509,114
6.7	300	13100 - 13400	2024	379	247	2,651	795,162
0.02	100	13400 - 13500	2237	379	12,475	15,091	1,509,114
6.7	650	13500 - 14150	2024	379	247	2,651	1,722,851
0.02	100	14150 - 14250	2237	379	12,475	15,091	1,509,114
6.7	500	14250 - 14750	2053	379	247	2,680	1,339,970
0.02	100	14750 - 14850	2284	379	12,475	15,138	1,513,814
6.7	500	14850 - 15350	2073	379	247	2,700	1,349,770
0.02	100	15350 - 15450	2284	379	12,475	15,138	1,513,814
9.3	1100	15450 - 16550	2073	379	247	2,700	2,969,494
0.02	100	16550 - 16650	2284	379	12,475	15,138	1,513,814
9.30	750	16650 - 17400	2032	379	247	2,659	1,993,905
0.02	100	17400 - 17500	2244	379	12,475	15,098	1,509,814
6.7	800	17500 - 18300	2032	379	247	2,659	2,126,832
9.3	400	18300 - 18700	2032	379	247	2,659	1,063,416
0.02	100	18700 - 18800	2244	379	12,475	15,098	1,509,814
	8800						**40,045,110**

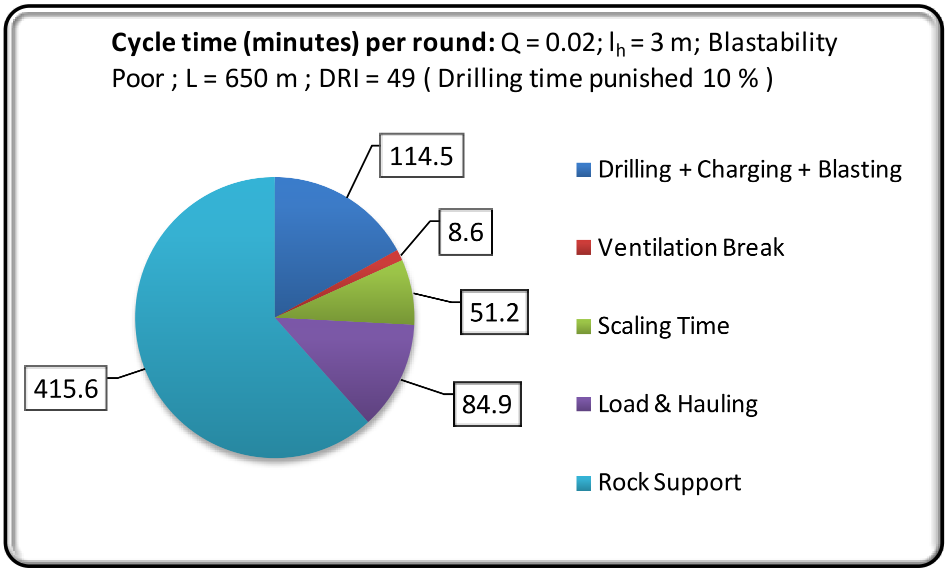
This final cost for Vallical tunnel is almost 800,000 USD cheaper than the cost obtained when no extra adit was computed. The exact cost in that case is 40,830,310 USD and the details are shown in the optimum cross section analysis.

8.2.2.2.2 Earlier tunnel completion

As long as the extra adit does not reach the Castillo tunnel, only two tunnel faces will be working on the Castillo tunnel. Once the extra adit reaches the Castillo tunnel, four tunnel faces will be working on the Castillo Tunnel. The extra adit will let to finish earlier the tunnel and its improvement relies on how fast the extra adit reaches the Vallical tunnel.

Therefore it is needed to know access road advance rate to the adit and the adit advance rate.

In poor rock mass condition the adit advance rate is shown in the next chart:

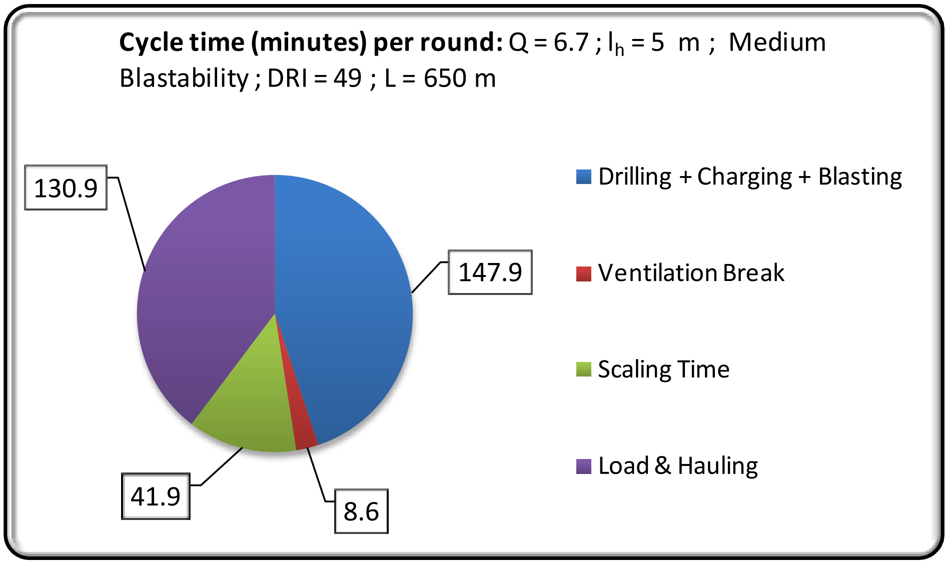


$$114.5 + 8.6 + 51.2 + 84.6 + 415.6 = 675 \text{ min}$$

Correction factor for tunnel length (650 m) k_{le} = 1.143 (Reference [8] only has information from 1000 m and longer tunnel, but curve was extended).

Total cycle time: (675– 415.6) *(1.143 - 1) + 675 = **712 min.** (Tunnel length only affects the excavation advance rate).

Working hour per week		120	h/week
Drilled length	l_h	3	m
Pull	p_r	91%	
Round length	p_r*l_h	2.73	m
Cycle Time		**712**	min
Advance rate		**27.6**	m/week



147.9 + 8.6 + 41.9 + 130.9 = 329 min

Correction factor for tunnel length (650 m) k_{le} = 1.143

329* 1.143 = **376 min**

Working hour per week	120	h/week
Drilled length	5	m
Pull	91%	
Round length	4.55	m
Cycle Time min	**376**	min
Advance rate	**87.0**	m/week

Chainage	length m	Rock mass classification Q-System	Advance rate m/week	Construction time month/stretch
0 - 100	100	0.02	27.6	0.8
100 - 650	550	6.7	87.0	1.5
Sum	**650**			**2.3**

Also, the access road from the existing road to the extra adit requires a significant period of time to be constructed and the details are shown below:

Road construction time before extra adit		
Road length from road "Vallical brook - Cavern" to adit	834	m
Road length stretch of Vallical brook - Cavern	4,660	m
Total road length	5,494	m
Advance rate road	50	m/day
Construction period	110	day
Construction period	3.7	month

Finally one month was considered for portal construction period.

Hence, construction Vallical tunnel from the extra adit starts after 7 months (2.3 month + 3.7 month + 1 month).

The extra adit comes out in the Vallical tunnel at Chainage 14,450 m, in a stretch where the rock is classified as 6.7 in the Q system between chainage 14,250 m and 14,750 m (14,750 m – 14,250 m = 500 m long). Therefore, in the Vallical tunnel, 200 m of this 500 m stretch will be constructed toward upstream and 300 m toward downstream at an advance rate equal to 81.6 m/week for either direction.

The time elapsed to reach the Chainage 14,250 m and 14,750 m respectively in the Vallical tunnel from Chainage 14,450 m is provided in the next table:

Advance rate Chainage 14250 -14750	To Vallical Brook (Or upstream)		To downstream	
	Distance	time elapsed	Distance	time elapsed
m/week	m	month	m	month
-	0	7.0	0	7.0
81.6	**200**	7.6	**300**	7.8

With regard to the two Vallical tunnel ends, they also require an access road. Access road required to the upstream end adit is 377 m and the downstream end adit is 8760 m.

Vallical 1: Acces road		
Road length	377	m
Advance rate	50	m/day
time elapsed	8	day
time elapsed	0.3	month

Vallical 4: Acces road		
Road length	8760	m
Advance rate	50	m/day
time elapsed	175	day
time elapsed	5.8	month

In addition to access road construction time, one month is added in each Vallical tunnel end for portal construction time before tunnel excavation starts. Therefore the upstream end or “Vallical 1” starts 1.3 months (0.3 month + 1 month) after road construction initiation. Likewise, the downstream end or “Castillo 2” starts 6.8 months (5.8 months + 1 month) after road construction initiation.

Finally, the time elapsed at the end of each stretch from each adit is shown in the following table:

Chainage	Stretch Length m	Rock Mass Q_{ROOF}	Advance rate m/week	Advance rate month/stretch	Cumulative time elapsed Vallical 1 month	Cumulative time elapsed Vallical 2 month	Cumulative time elapsed Vallical 3 month	Cumulative time elapsed Vallical 4 month
10000 - 10100	100	0.003	25.1	0.9	2.2			
10100 - 10300	200	1.0	66.1	0.7	2.9			
10300 - 10350	50	0.003	25.1	0.5	3.4			
10350 - 10850	500	6.7	82.1	1.4	4.8			
10850 - 10950	100	0.02	27.0	0.9	5.6			
10950 - 11300	350	6.7	82.1	1.0	6.6			
11300 - 11400	100	0.02	27.0	0.9	7.5			
11400 - 12550	1150	6.7	82.1	3.3	10.8			
12550 - 12650	100	0.02	27.0	0.9	11.6			
12650 - 13000	350	6.7	82.1	1.0	12.6			
13000 - 13100	100	0.02	26.7	0.9	**12.8**		**12.8**	
13100 - 13400	300	6.7	80.0	0.9			12.1	
13400 - 13500	100	0.02	26.7	0.9			11.2	
13500 - 14150	650	6.7	80.0	1.9			10.3	
14150 - 14250	100	0.02	26.7	0.9			8.4	
14250 - 14750	500	6.7	81.6	1.4			7.6	7.8
14750 - 14850	100	0.02	26.9	0.9				8.7
14850 - 15350	500	6.7	81.6	1.4				10.1
15350 - 15450	100	0.02	26.9	0.9				11.0
15450 - 16550	1100	9.3	81.6	3.1				14.2
16550 - 16650	100	0.02	26.9	0.9		**14.6**		**14.6**
16650 - 17400	750	9.30	80.3	2.2		14.2		
17400 - 17500	100	0.02	26.7	0.9		12.1		
17500 - 18300	800	6.7	80.3	2.3		11.2		
18300 - 18700	400	9.3	80.3	1.2		8.9		
18700 - 18800	100	0.02	26.7	0.9		7.7		
	8800							
							Start	**End**

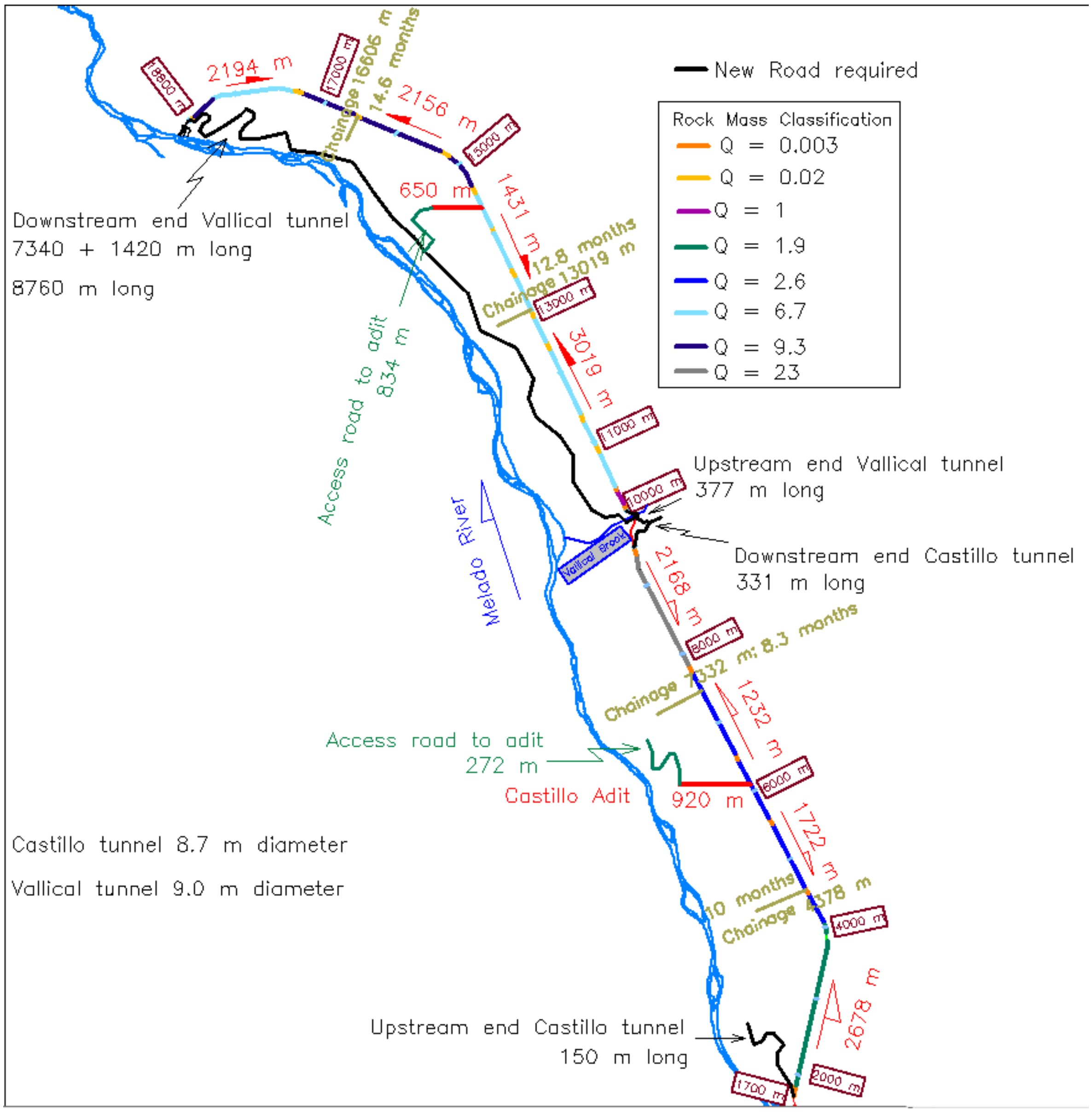
From the previous table, it is possible to say that:

The first stretch of "Vallical 1" tunnel (Chainage 10000 - 10100) is completed after 2.2 months (green color), which is the sum of the first stretch equal to 100 m long in 0.9 months and 1.3 months related to road construction and portal. The time consumption for the following stretches until the end (brown cell) is only related to the advance rate in the tunnel.

For "Vallical 2", "Vallical 3" and "Vallical 4" the computation is exactly the same. The green cell is the sum of the previous construction time consumptions plus the time spent to excavate the corresponding stretch. The following stretches until the end (brown cell) is only related to the advance rate in the tunnel.

Vallical 1 & 3 meet each other after 12.8 months and Vallical 2 & 4 meet each other after 14.6 months.

Therefore, Castillo tunnel is completed after **14.6 months** (or 3.3 trimesters). This is 5.6 months (or 1.9 trimesters) faster than eliminating the extra adit.



8.2.2.3 Evaluation

This chapter aims to trades off the advantages and disadvantages of an extra adit in economic terms for the Vallical tunnel.

The evaluation procedure is exactly the same as that applied in the extra adit for Castillo tunnel. The results are shown below:

Vallical Tunnel is not part of critical path			Without Adit	With Adit	Difference
Tunnel + Portal Constraction Period	n	Month	20.3	14.6	5.6
Gross Investment Cost Vallical Tunnel		USD	40,830,310	40,045,110	785,200
Extra adit cost (650 m)		USD	0	1,959,622	-1,959,622
Access Road to Vallical Adit (834 m)		USD	0	1,105,050	-1,105,050
Total Gross Investment Cost		USD	40,830,310	43,109,782	-2,279,472
Compounding factor	R		21.9	15.9	6.0
Investment cost compounding	$R*I_g/n$	USD	44,123,443	45,533,569	-1,410,126
Total Cost at Commissioning	$R*I_g/n*(1+i)^{12}$	USD	**48,535,787**	**50,086,926**	**-1,551,139**

Based on the calculations undertaken here, an extra adit for the Vallical tunnel is not economically convenient as it occurred for the Castillo tunnel. In both situations a delay in plant commissioning were not quantified since neither Castillo nor Vallical tunnel are in the critical path.

8.3 Convenience of TBM

This section will analyze qualitatively the convenience of TBM utilization in the Melado Hydropower Project. In general, there is a consensus about the following advantages using TBM excavation which are:

- It requires less rock support.

- It gives smoother tunnel walls and reduced head loss in water tunnels.

- Longer tunnel sections can be excavated between adits.

- It has higher tunnelling capacity.

- It gives better working conditions for the crew.

Also, there is a consensus about the following disadvantages of TBM:

- More (better) geological information from the pre-investigation stage is required.

- It is more sensitive to tunneling problems in poor rock mass conditions.

- It is a less flexible method than drill & blast method.

Also, in general the cost of TBM is higher than drill and blast but it could be traded off by a smoother contour and a faster advance rate.

Based on the above, and the facts about this project, one can mention the following with regard to TBM in this project that:

The benefit of higher advance rate is not so much appreciated in this project since tunnel construction is not in the critical path. If there was an earlier plant commissioning because of a higher advance rate, TBM excavation method would be very appreciated.

The benefit of eliminating adits is not appreciated in this project because with Drill & Blast is also achieved.

The uncertainties related to the character of weakness zone are not favorable for TBM, where a wrong tunnel mapping can even cause that the machine get stuck and support at the face is in general more difficult to install.

Of course, there are some benefits that Drill & Blast will never achieved like a smoother tunnel contour and better working conditions for the crew, but based on the facts given above, the recommendation is not to considerer TBM.

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9. CONCLUSION & RECOMENDATION

The first task of this thesis was to improve the evaluation of the engineering geological conditions which are the key input parameters for a tunnel planning. The way to achieve this was by means of a combination between a correlation of different statistical methods, analysis of field data and the incorporation of input data of similar rock conditions.

Among the findings that are completely new for this project one can point out:

Blocky Granite/ Blocky Andesite:

Based on the rock mass description, tunnel diameter and core drillings log data, there is consistency that both rock types that represents the two relevant rock formations would behave as blocky **in between weakness zones**, meaning that the main challenge for these stretches would be block fall and could become as buckling in case of high stress level.

Eventual problems in invert for inferred weakness zones located in *Trapa-Trapa* rock formation:

The numerical modeling showed that a high level of in situ stress in poor rock mass conditions could lead to lift the invert where a thicker concrete slab could be required.

Eventual water ingress in inferred *Trapa-Trapa* weakness zones:

From different documented cases, the weathering in volcanic rocks can reach the 200 m deep. The literature suggests that when tuff layers or breccias are placed within volcanic rocks can be a sign of severe water problems. Pre-injection and probe holes could be considered.

Swelling problems in Trapa-Trapa weakness zone:

As it was described in the geology chapter, the rock mass in *Trapa-Trapa* rock formation contain clay coating and its content in weakness zone should be much higher that could trigger swelling problems and therefore heavier rock support.

Eventual water leakage in Batholith Melado fault zones:

The main problem in some inferred fault zones located in Batholith Melado are the low overburden, having two weakness zones a rock cover of around 60 m. Probe holes and pre-injection could be considered.

High stress situation found in core drillings:

In general, core disking suggests high in situ stress values. Two core drilling showed tendency to core disking.

The second main task was to implement the state of the art rock support for this project in order to reduce costs and improve efficiency.

Reinforced shotcrete ribs (rebars) is a heavy rock support alternative that has been implemented in Chile with succeed and is clearly more flexible to installed than steel ribs.

Fiber reinforced shotcrete has completely replaces the mesh reinforced shotcrete, minimizing the intensive process of mesh installation.

In case, it is found that in fact weakness zone is an important source of water leakage, probe holes and pre-injection could be carried out.

The third and final task was to implement all the input data collected in the first two tasks in thorough economic analysis that concluded that:

No extra Adit for Castillo and Vallical tunnels are needed as long as they are not in the project critical path of construction. So far, electromechanical equipments would take longer to

Both, Castillo and Vallical tunnels should have a larger diameter compared to the current tunnel diameter equal to 6.9 m. Castillo tunnel should have a diameter of 8.7 m and Vallical tunnel should have a diameter of 9 m.

Further detailed geological investigation required:

To know well the character of the weakness zones, especially in Trapa-Trapa rock formation.

In situ field stress could be carried whenever it possible. Eventually galleries to the cavern could be a good chance to confirm this and review the project design.

Analyze the limitation of the Norwegian systematization of Drill and blast costs and advance rate with regard to organization and skill level.

For a better model prediction in advance rate, some tests could be carried out like:

- Brittleness test

- Siever’s miniature drill test

- Abrasion test

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APPENDIX A: Details about Excavation Cost and Advance rate

6.9 m tunnel diameter (Q = 0.003)

Cross Section	A_s	40.4	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	76	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	63	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	
Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**45.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**16.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**13.4**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.088	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**6.2**	min
Drilling Time		**90.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**90.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	29.1	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**23.9**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.0**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**10.8**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**114.5**	min
Time for Ventilation break	**II**	**8.7**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	124	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	129	asm^3
Loading Time	$\mathbf{T_{lt}}$	**62.6**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**14.4**	min
Incidental Time for Loading and Hauling	T_{tl}	**8.5**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**85.6**	min
Scaling Time without correction factor	T_r	62.4	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**51.4**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**260**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**259.5**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	40.4	m^2
Performance of Rebars + 2 layers of shotcrete	**0.60**	h/m
Total Performance Shotcrete ribs	**1.09**	h/m
Shotcrete ribs time consumption per round	**179.1**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**729**	min
Advance rate	**27.0**	m/week

6.9 m tunnel diameter (Q=0.02)

Cross Section	A_s	40.4	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	76	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	63	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**27.2**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	**15.3**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**8.0**	min
Lack of simultaneousness	f_{sa}	0.088	
Extra Time of Simultaneousness	T_{sa}	**4.2**	min
Drilling Time	T_{bb}	**59.8**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**65.8**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	29.1	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**23.9**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.0**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**10.8**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**114.5**	min
Time for Ventilation break	**II**	**8.7**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	124	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.17	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	129	asm^3
Loading Time	**T_{lt}**	**62.6**	min
Rig Time For Loading and Hauling	**T_{rl}**	**14.4**	min
Incidental Time for Loading and Hauling	T_{tl}	**8.5**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**85.6**	min
Scaling Time without correction factor	T_r	62.4	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**51.4**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**260**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**259.5**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	40.4	m^2
Performance of Rebars + 2 layers of shotcrete	**0.60**	h/m
Total Performance Shotcrete ribs	**1.09**	h/m
Shotcrete ribs time consumption per round	**179.1**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**675**	min
Advance rate	**29.1**	m/week

6.9 m tunnel diameter (Q = 1)

Cross Section	A_s	40.4	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	76	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	63	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**45.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**16.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**13.4**	min
Lack of simultaneousness	f_{sa}	0.088	
Extra Time of Simultaneousness	T_{sa}	**6.2**	min
Drilling Time		**90.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**90.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	29.1	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**29.0**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.0**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.8**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**147.9**	min

Time for Ventilation break	**II**	**8.7**	min

Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	124	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	216	asm^3
Loading Time	$\mathbf{T_{lt}}$	**104.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**14.4**	min
Incidental Time for Loading and Hauling	T_{tl}	**13.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**131.9**	min

Scaling Time without correction factor	T_r	42.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**42.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**331**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**329.8**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		452	min
Advance rate		**72.4**	m/week

6.9 m tunnel diameter (Q = 1.9)

Cross Section	**A_s**	40.4	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	76	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	63	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	**v_g**	98	cm/min
Drilling Time for charged Holes	T_h	**45.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	**t_f**	0.75	min/hole
Time for Moving (Charged and Empty holes)	**T_f**	**16.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	**f_k**	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	**T_k**	**13.4**	min
Lack of simultaneousness	**f_{sa}**	0.088	
Extra Time of Simultaneousness	**T_{sa}**	**6.2**	min
Drilling Time		**90.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**90.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	29.1	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**29.0**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.0**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.8**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**147.9**	min
Time for Ventilation break	**II**	**8.7**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	124	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	216	asm^3
Loading Time	$\mathbf{T_{lt}}$	**104.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**14.4**	min
Incidental Time for Loading and Hauling	T_{tl}	**13.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**131.9**	min
Scaling Time without correction factor	T_r	42.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**42.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**331**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**329.8**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		452	min
Advance rate		**72.4**	m/week

6.9 m tunnel diameter (Q = 2.6)

Cross Section	A_s	40.4	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	76	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	63	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**45.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**16.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**13.4**	min
Lack of simultaneousness	f_{sa}	0.088	
Extra Time of Simultaneousness	T_{sa}	**6.2**	min
Drilling Time		**90.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**90.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	29.1	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**29.0**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.0**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.8**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**147.9**	min

Time for Ventilation break	**II**	**8.7**	min

Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	124	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	216	asm^3
Loading Time	$\mathbf{T_{lt}}$	**104.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**14.4**	min
Incidental Time for Loading and Hauling	T_{tl}	**13.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**131.9**	min

Scaling Time without correction factor	T_r	42.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**42.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**331**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**329.8**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	2.7	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.10	h/m
Radial Bolting time consumption per round	**26.2**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		356	min
Advance rate		**92.0**	m/week

6.9 m tunnel diameter (Q = 6.7)

(Same advance rate for Q = 9.3 & Q = 23)

Cross Section	A_s	40.4	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	76	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	63	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	
Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	45.4	min
Drilling Time for empty holes	T_g	8.5	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	16.7	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	13.4	min
Lack of simultaneousness	f_{sa}	0.088	
Extra Time of Simultaneousness	T_{sa}	6.2	min
Drilling Time	T_{bb}	**90.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	90.2	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	29.1	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**29.0**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.0**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.8**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**147.9**	min
Time for Ventilation break	**II**	**8.7**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	124	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	216	asm^3
Loading Time	$\mathbf{T_{lt}}$	**104.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**14.4**	min
Incidental Time for Loading and Hauling	T_{tl}	**13.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**131.9**	min
Scaling Time without correction factor	T_r	42.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**42.1**	min
Net Round cycle time **I + II + III + IV**	T_{nr}	**331**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**329.8**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		330	min
Advance rate		**99.3**	m/week

7.2 m tunnel diameter (Q = 0.003)

Cross Section	A_s	44	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	79	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	65	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**28.6**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	**15.9**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**8.4**	min
Lack of simultaneousness	f_{sa}	0.082	
Extra Time of Simultaneousness	T_{sa}	**4.1**	min
Drilling Time	T_{bb}	**62.1**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**68.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	30.6	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**25.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**11.3**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**119.2**	min
Time for Ventilation break	**II**	**9.3**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	125	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	140	asm^3
Loading Time	$\mathbf{T_{lt}}$	**67.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.1**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**91.7**	min
Scaling Time without correction factor	T_r	64.8	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**53.4**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**274**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**272.9**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	44.0	m^2
Performance of Rebars + 2 layers of shotcrete	**0.61**	h/m
Total Performance Shotcrete ribs	**1.10**	h/m
Shotcrete ribs time consumption per round	**180.2**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**743**	min
Advance rate	**26.4**	m/week

7.2 m tunnel diameter (Q = 0.02)

Cross Section	A_s	44	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	79	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	65	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**28.6**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	**15.9**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**8.4**	min
Lack of simultaneousness	f_{sa}	0.082	
Extra Time of Simultaneousness	T_{sa}	**4.1**	min
Drilling Time	T_{bb}	**62.1**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**68.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	30.6	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**25.2**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.3**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**11.3**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**119.2**	min

Time for Ventilation break	**II**	**9.3**	min

Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	125	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	140	asm^3
Loading Time	$\mathbf{T_{lt}}$	**67.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.1**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**91.7**	min

Scaling Time without correction factor	T_r	64.8	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**53.4**	min

Net Round cycle time	I + II + III + IV	T_{nr}	274	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**272.9**	min
Rock Support: Radial Bolts				
Round length			2.73	m
Number of Radial Bolts			12	Unit/m
Percentage of bolts at the face			100%	Unit/m
Radial Bolt length			3	m
Radial Bolt Installation after drilling Bolt holes?			No	
Radial Bolting time consumption			0.64	h/m
Radial Bolting time consumption per round			**105.4**	min
Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)				
Fibrecrete Thichness			15	cm
Length of sprayed area			2.5	m
Quantity of Fibrecrete required			2.7	m^3/m
Performance Fibrecrete			0.18	h/m^3
Performance Fibrecrete			**0.49**	h/m
Cross section Area			44.0	m^2
Performance of Rebars + 2 layers of shotcrete			**0.61**	h/m
Total Performance Shotcrete ribs			**1.10**	h/m
Shotcrete ribs time consumption per round			**180.2**	min
Rock Support: Spiling Bolts				
Number of Spiling Bolt/round			18	Unit/m
Spiling Bolt length m			6	m
Spiling bolting time consumption			**2.19**	h/round
Spiling bolting time consumption per round			**131.2**	min
Working hour per week			120	h/week
Round length			2.73	m
Cycle Time min			**690**	min
Advance rate			**28.5**	m/week

7.2 m diameter (Q = 1)

Cross Section	A_s	44	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	79	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	65	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	47.7	min
Drilling Time for empty holes	T_g	8.5	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	17.5	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	14.0	min
Lack of simultaneousness	f_{sa}	0.082	
Extra Time of Simultaneousness	T_{sa}	6.0	min
Drilling Time		**93.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	93.7	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	30.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**30.5**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**15.4**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**154.0**	min
Time for Ventilation break	**II**	**9.3**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	125	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.17	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	234	asm^3
Loading Time	**T_{lt}**	**112.4**	min
Rig Time For Loading and Hauling	**T_{rl}**	**15.1**	min
Incidental Time for Loading and Hauling	T_{tl}	**14.1**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**141.6**	min
Scaling Time without correction factor	T_r	44.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**44.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**349**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**348.1**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		471	min
Advance rate		**69.6**	m/week

7.2 m diameter (Q = 1.9)

Cross Section	A_s	44	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	79	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	65	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**47.7**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**17.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**14.0**	min
Lack of simultaneousness	f_{sa}	0.082	
Extra Time of Simultaneousness	T_{sa}	**6.0**	min
Drilling Time		**93.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**93.7**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	30.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**30.5**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**15.4**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**154.0**	min
Time for Ventilation break	**II**	**9.3**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	125	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.17	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	234	asm^3
Loading Time	**T_{lt}**	**112.4**	min
Rig Time For Loading and Hauling	**T_{rl}**	**15.1**	min
Incidental Time for Loading and Hauling	T_{tl}	**14.1**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**141.6**	min

Scaling Time without correction factor	T_r	44.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**44.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**349**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**348.1**	min
	Rock Support: Radial Bolts			
Round length			4.55	m
Number of Radial Bolts			4.5	Unit/m
Percentage of bolts at the face			67%	Unit/m
Radial Bolt length			3	m
Radial Bolt Installation after drilling Bolt holes?			No	
Radial Bolting time consumption			0.16	h/m
Radial Bolting time consumption per round			**43.8**	min
Working hour per week			120	h/week
Round length		p_r*l_h	4.55	m
Cycle Time min			392	min
Advance rate			**83.6**	m/week

7.2 m diameter (Q = 2.6)

Cross Section	A_s	44	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	79	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	65	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	47.7	min
Drilling Time for empty holes	T_g	8.5	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	17.5	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	14.0	min
Lack of simultaneousness	f_{sa}	0.082	
Extra Time of Simultaneousness	T_{sa}	6.0	min
Drilling Time		**93.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	93.7	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	30.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**30.5**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**15.4**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**154.0**	min
Time for Ventilation break	**II**	**9.3**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	125	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	234	asm^3
Loading Time	$\mathbf{T_{lt}}$	**112.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.1**	min
Incidental Time for Loading and Hauling	T_{tl}	**14.1**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**141.6**	min
Scaling Time without correction factor	T_r	44.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**44.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**349**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**348.1**	min
	Rock Support: Radial Bolts			
Round length			4.55	m
Number of Radial Bolts			2.7	Unit/m
Percentage of bolts at the face			67%	Unit/m
Radial Bolt length			3	m
Radial Bolt Installation after drilling Bolt holes?			No	
Radial Bolting time consumption			0.10	h/m
Radial Bolting time consumption per round			**26.2**	min
Working hour per week			120	h/week
Round length		p_r*l_h	4.55	m
Cycle Time min			374	min
Advance rate			**87.5**	m/week

7.2 m diameter (Q = 6.7)

(Same advance rate for Q = 9.3 & Q = 23)

Cross Section	A_s	44	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	79	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	65	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	47.7	min
Drilling Time for empty holes	T_g	8.5	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	17.5	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	14.0	min
Lack of simultaneousness	f_{sa}	0.082	
Extra Time of Simultaneousness	T_{sa}	6.0	min
Drilling Time		**93.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	93.7	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	30.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**30.5**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**15.4**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**154.0**	min
Time for Ventilation break	**II**	**9.3**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	125	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.17	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	234	asm^3
Loading Time	$\mathbf{T_{lt}}$	112.4	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	15.1	min
Incidental Time for Loading and Hauling	T_{tl}	14.1	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**141.6**	min
Scaling Time without correction factor	T_r	44.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**44.1**	min
Net Round cycle time **I + II + III + IV**	T_{nr}	**349**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**348.1**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		348	min
Advance rate		**94.1**	m/week

7.5 m diameter (Q = 0.003)

Cross Section	$\mathbf{A_s}$	47.8	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	83	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	68	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_l	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**30.0**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.68	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**16.6**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**8.8**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.076	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**3.9**	min
Drilling Time	$\mathbf{T_{bb}}$	**64.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**70.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	32.2	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**26.5**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.7**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**11.7**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**123.8**	min
Time for Ventilation break	**II**	**10.0**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	126	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	152	asm^3
Loading Time	$\mathbf{T_{lt}}$	**72.6**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**98.1**	min
Scaling Time without correction factor	T_r	67.4	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**55.5**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**287**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**286.7**	min
	Rock Support: Radial Bolts			
Round length			2.73	m
Number of Radial Bolts			18.1	Unit/m
Percentage of bolts at the face			100%	Unit/m
Radial Bolt length			3	m
Radial Bolt Installation after drilling Bolt holes?			No	
Radial Bolting time consumption			0.97	h/m
Radial Bolting time consumption per round			**159.0**	min
Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)				
Fibrecrete Thichness			15	cm
Length of sprayed area			2.5	m
Quantity of Fibrecrete required			2.7	m^3/m
Performance Fibrecrete			0.18	h/m^3
Performance Fibrecrete			**0.49**	h/m
Cross section Area			47.8	m^2
Performance of Rebars + 2 layers of shotcrete			**0.61**	h/m
Total Performance Shotcrete ribs			**1.11**	h/m
Shotcrete ribs time consumption per round			**181.3**	min
Rock Support: Spiling Bolts				
Number of Spiling Bolt/round			18	Unit/m
Spiling Bolt length m			6	m
Spiling bolting time consumption			**2.19**	h/round
Spiling bolting time consumption per round			**131.2**	min
Working hour per week			120	h/week
Round length			2.73	m
Cycle Time min			**758**	min
Advance rate			**25.9**	m/week

7.5 m diameter (Q = 0.02)

Cross Section	A_s	47.8	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	83	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	68	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**30.0**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	**16.6**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**8.8**	min
Lack of simultaneousness	f_{sa}	0.076	
Extra Time of Simultaneousness	T_{sa}	**3.9**	min
Drilling Time	T_{bb}	**64.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**70.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	32.2	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**26.5**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.7**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**11.7**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**123.8**	min

Time for Ventilation break	**II**	**10.0**	min

Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	126	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	152	asm^3
Loading Time	$\mathbf{T_{lt}}$	**72.6**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**98.1**	min

Scaling Time without correction factor	T_r	67.4	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**55.5**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**287**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**286.7**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	47.8	m^2
Performance of Rebars + 2 layers of shotcrete	**0.61**	h/m
Total Performance Shotcrete ribs	**1.11**	h/m
Shotcrete ribs time consumption per round	**181.3**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**705**	min
Advance rate	**27.9**	m/week

7.5 m diameter (Q = 1)

Cross Section	A_s	47.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	83	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	68	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**50.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**18.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**14.6**	min
Lack of simultaneousness	f_{sa}	0.076	
Extra Time of Simultaneousness	T_{sa}	**5.8**	min
Drilling Time		**97.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**97.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	32.2	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**32.1**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.7**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.0**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**159.9**	min
Time for Ventilation break	**II**	**10.0**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	126	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	253	asm^3
Loading Time	$\mathbf{T_{lt}}$	**120.9**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**15.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**151.9**	min
Scaling Time without correction factor	T_r	46.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**46.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**368**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**367.0**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		489	min
Advance rate		**66.9**	m/week

7.5 m diameter (Q = 1.9)

Cross Section	A_s	47.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	83	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	68	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	
Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**50.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**18.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**14.6**	min
Lack of simultaneousness	f_{sa}	0.076	
Extra Time of Simultaneousness	T_{sa}	**5.8**	min
Drilling Time		**97.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**97.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	32.2	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**32.1**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.7**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.0**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**159.9**	min
Time for Ventilation break	**II**	**10.0**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	126	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.16	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	253	asm^3
Loading Time	**T_{lt}**	120.9	min
Rig Time For Loading and Hauling	**T_{rl}**	15.8	min
Incidental Time for Loading and Hauling	T_{tl}	15.2	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**151.9**	min
Scaling Time without correction factor	T_r	46.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**46.1**	min

Net Round cycle time **I + II + III + IV**	T_{nr}	**368**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**367.0**	min
Rock Support: Radial Bolts			
Round length		4.55	m
Number of Radial Bolts		4.5	Unit/m
Percentage of bolts at the face		67%	Unit/m
Radial Bolt length		3	m
Radial Bolt Installation after drilling Bolt holes?		No	
Radial Bolting time consumption		0.16	h/m
Radial Bolting time consumption per round		**43.8**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		411	min
Advance rate		**79.7**	m/week

7.5 m diameter (Q = 2.6)

Cross Section	A_s	47.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	83	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	68	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**50.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**18.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**14.6**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.076	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**5.8**	min
Drilling Time		**97.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**97.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	32.2	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**32.1**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.7**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.0**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**159.9**	min
Time for Ventilation break	**II**	**10.0**	min
Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	126	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	253	asm^3
Loading Time	$\mathbf{T_{lt}}$	**120.9**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**15.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**151.9**	min
Scaling Time without correction factor	T_r	46.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**46.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**368**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**367.0**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	2.7	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.10	h/m
Radial Bolting time consumption per round	**26.2**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		393	min
Advance rate		**83.3**	m/week

7.5 m diameter (Q = 6.7)

(Same advance rate for Q = 9.3 & Q = 23)

Cross Section	A_s	47.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	83	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	68	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**50.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**18.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**14.6**	min
Lack of simultaneousness	f_{sa}	0.076	
Extra Time of Simultaneousness	T_{sa}	**5.8**	min
Drilling Time		**97.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**97.2**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	32.2	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**32.1**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**14.7**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.0**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**159.9**	min

Time for Ventilation break	**II**	**10.0**	min

Type of Loader and Transport Equipment		Cat 966G - Truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	126	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	253	asm^3
Loading Time	$\mathbf{T_{lt}}$	**120.9**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**15.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**15.2**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**151.9**	min

Scaling Time without correction factor	T_r	46.3	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**46.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**368**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**367.0**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		367	min
Advance rate		**89.3**	m/week

7.8 m diameter (Q = 0.003)

Cross Section	**A_s**	51.7	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	87	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	72	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_l	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	**v_g**	98	cm/min
Drilling Time for charged Holes	T_h	**31.4**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	**t_f**	0.68	min/hole
Time for Moving (Charged and Empty holes)	**T_f**	**17.3**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	**f_k**	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	**T_k**	**9.1**	min
Lack of simultaneousness	**f_{sa}**	0.071	
Extra Time of Simultaneousness	**T_{sa}**	**3.8**	min
Drilling Time	**T_{bb}**	**66.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**73.4**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	33.8	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time $\mathbf{T_{lb}}$ **min** (Per Round)	T_l	**27.8**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.1**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**12.2**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**128.5**	min
Time for Ventilation break	**II**	**10.7**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	261	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	164	asm^3
Loading Time	$\mathbf{T_{lt}}$	**37.6**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**16.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**6.0**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**60.1**	min
Scaling Time without correction factor	T_r	70.0	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**57.7**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**257**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**256.3**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	51.7	m^2
Performance of Rebars + 2 layers of shotcrete	**0.62**	h/m
Total Performance Shotcrete ribs	**1.11**	h/m
Shotcrete ribs time consumption per round	**182.5**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**729**	min
Advance rate	**27.0**	m/week

7.8 m diameter (Q = 0.02)

Cross Section	$\mathbf{A_s}$	51.7	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	87	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	72	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**31.4**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.68	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**17.3**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**9.1**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.071	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**3.8**	min
Drilling Time	$\mathbf{T_{bb}}$	**66.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**73.4**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	33.8	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**27.8**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.1**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**12.2**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**128.5**	min
Time for Ventilation break	**II**	**10.7**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	261	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	164	asm^3
Loading Time	$\mathbf{T_{lt}}$	**37.6**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**16.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**6.0**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**60.1**	min
Scaling Time without correction factor	T_r	70.0	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**57.7**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**257**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**256.3**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	51.7	m^2
Performance of Rebars + 2 layers of shotcrete	**0.62**	h/m
Total Performance Shotcrete ribs	**1.11**	h/m
Shotcrete ribs time consumption per round	**182.5**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**675**	min
Advance rate	**29.1**	m/week

7.8 m diameter (Q = 1)

Cross Section	A_s	51.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	87	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	72	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**52.3**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**19.0**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**15.2**	min
Lack of simultaneousness	f_{sa}	0.071	
Extra Time of Simultaneousness	T_{sa}	**5.7**	min
Drilling Time		**100.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**100.7**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	33.8	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**33.6**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.1**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.6**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**166.0**	min

Time for Ventilation break	**II**	**10.7**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	261	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	273	asm^3
Loading Time	$\mathbf{T_{lt}}$	**62.7**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**16.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**8.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**87.9**	min

Scaling Time without correction factor	T_r	48.4	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**48.2**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**313**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**312.1**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		435	min
Advance rate		**75.4**	m/week

7.8 m diameter (Q = 1.9)

Cross Section	A_s	51.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	87	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	72	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**52.3**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**19.0**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**15.2**	min
Lack of simultaneousness	f_{sa}	0.071	
Extra Time of Simultaneousness	T_{sa}	**5.7**	min
Drilling Time		**100.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**100.7**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	33.8	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**33.6**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.1**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.6**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**166.0**	min
Time for Ventilation break	**II**	**10.7**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	261	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.16	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	273	asm^3
Loading Time	**T_{lt}**	**62.7**	min
Rig Time For Loading and Hauling	**T_{rl}**	**16.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**8.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**87.9**	min
Scaling Time without correction factor	T_r	48.4	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**48.2**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**313**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**312.1**	min
	Rock Support: Radial Bolts			
Round length			4.55	m
Number of Radial Bolts			4.5	Unit/m
Percentage of bolts at the face			67%	Unit/m
Radial Bolt length			3	m
Radial Bolt Installation after drilling Bolt holes?			No	
Radial Bolting time consumption			0.16	h/m
Radial Bolting time consumption per round			**43.8**	min
Working hour per week			120	h/week
Round length		p_r*l_h	4.55	m
Cycle Time min			356	min
Advance rate			**92.0**	m/week

7.8 m diameter (Q = 2.6 & Q = 6.7)

Cross Section	A_s	51.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	87	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	72	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**52.3**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**19.0**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**15.2**	min
Lack of simultaneousness	f_{sa}	0.071	
Extra Time of Simultaneousness	T_{sa}	**5.7**	min
Drilling Time		**100.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**100.7**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	33.8	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**33.6**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.1**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.6**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**166.0**	min
Time for Ventilation break	**II**	**10.7**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	261	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.16	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	273	asm^3
Loading Time	**T_{lt}**	62.7	min
Rig Time For Loading and Hauling	**T_{rl}**	16.5	min
Incidental Time for Loading and Hauling	T_{tl}	8.8	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**87.9**	min
Scaling Time without correction factor	T_r	48.4	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**48.2**	min
Net Round cycle time **I + II + III + IV**	T_{nr}	**313**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**312.1**	min
Rock Support: Radial Bolts			
Round length		4.55	m
Number of Radial Bolts		2.7	Unit/m
Percentage of bolts at the face		67%	Unit/m
Radial Bolt length		3	m
Radial Bolt Installation after drilling Bolt holes?		No	
Radial Bolting time consumption		0.10	h/m
Radial Bolting time consumption per round		**26.2**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		338	min
Advance rate		**96.8**	m/week

7.8 m diameter (Q = 9.3 & Q = 23)

Cross Section	A_s	51.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	87	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	72	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**52.3**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**19.0**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**15.2**	min
Lack of simultaneousness	f_{sa}	0.071	
Extra Time of Simultaneousness	T_{sa}	**5.7**	min
Drilling Time		**100.7**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**100.7**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	33.8	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**33.6**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.1**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**16.6**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**166.0**	min
Time for Ventilation break	**II**	**10.7**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	261	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.16	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	273	asm^3
Loading Time	**T_{lt}**	**62.7**	min
Rig Time For Loading and Hauling	**T_{rl}**	**16.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**8.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**87.9**	min
Scaling Time without correction factor	T_r	48.4	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**48.2**	min
Net Round cycle time **I + II + III + IV**	T_{nr}	**313**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**312.1**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		312	min
Advance rate		**105.0**	m/week

8.1 m diameter (Q = 0.003)

Cross Section	A_s	55.7	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	91	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	75	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_l	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**32.8**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	**18.0**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**9.5**	min
Lack of simultaneousness	f_{sa}	0.068	
Extra Time of Simultaneousness	T_{sa}	**3.8**	min
Drilling Time	T_{bb}	**69.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**76.1**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	35.3	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**29.1**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.5**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**12.6**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**133.3**	min
Time for Ventilation break	**II**	**11.4**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	265	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	176	asm^3
Loading Time	$\mathbf{T_{lt}}$	**39.9**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**17.2**	min
Incidental Time for Loading and Hauling	T_{tl}	**6.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**63.5**	min
Scaling Time without correction factor	T_r	72.7	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**59.9**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**268**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**267.4**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	55.7	m^2
Performance of Rebars + 2 layers of shotcrete	**0.63**	h/m
Total Performance Shotcrete ribs	**1.12**	h/m
Shotcrete ribs time consumption per round	**183.7**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**741**	min
Advance rate	**26.5**	m/week

8.1 m diameter (Q = 0.02)

Cross Section	A_s	55.7	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	91	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	75	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	32.8	min
Drilling Time for empty holes	T_g	5.1	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	18.0	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	9.5	min
Lack of simultaneousness	f_{sa}	0.068	
Extra Time of Simultaneousness	T_{sa}	3.8	min
Drilling Time	T_{bb}	**69.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	76.1	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	35.3	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**29.1**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.5**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**12.6**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**133.3**	min

Time for Ventilation break	**II**	**11.4**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	265	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	176	asm^3
Loading Time	$\mathbf{T_{lt}}$	**39.9**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**17.2**	min
Incidental Time for Loading and Hauling	T_{tl}	**6.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**63.5**	min

Scaling Time without correction factor	T_r	72.7	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**59.9**	min

Net Round cycle time	I + II + III + IV	T_{nr}	**268**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**267.4**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	55.7	m^2
Performance of Rebars + 2 layers of shotcrete	**0.63**	h/m
Total Performance Shotcrete ribs	**1.12**	h/m
Shotcrete ribs time consumption per round	**183.7**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**688**	min
Advance rate	**28.6**	m/week

8.1 m diameter (Q = 1)

Cross Section	A_s	55.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	91	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	75	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**54.7**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**19.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**15.8**	min
Lack of simultaneousness	f_{sa}	0.068	
Extra Time of Simultaneousness	T_{sa}	**5.7**	min
Drilling Time		**104.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**104.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	35.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**35.2**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.5**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.2**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**172.2**	min

Time for Ventilation break	**II**	**11.4**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	265	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	293	asm^3
Loading Time	$\mathbf{T_{lt}}$	**66.5**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**17.2**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**93.0**	min

Scaling Time without correction factor	T_r	50.6	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**50.4**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**327**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**326.3**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		449	min
Advance rate		**73.0**	m/week

8.1 m diameter (Q = 1.9)

Cross Section	A_s	55.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	91	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	75	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**54.7**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**19.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**15.8**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.068	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**5.7**	min
Drilling Time		**104.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**104.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	35.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**35.2**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.5**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.2**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**172.2**	min

Time for Ventilation break	**II**	**11.4**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	265	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.16	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	293	asm^3
Loading Time	**T_{lt}**	**66.5**	min
Rig Time For Loading and Hauling	**T_{rl}**	**17.2**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**93.0**	min

Scaling Time without correction factor	T_r	50.6	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**50.4**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**327**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**326.3**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	4.5	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.16	h/m
Radial Bolting time consumption per round	**43.8**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		370	min
Advance rate		**88.5**	m/week

8.1 m diameter (Q = 2.6 & Q = 6.7)

Cross Section	A_s	55.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	91	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	75	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**54.7**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**19.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**15.8**	min
Lack of simultaneousness	f_{sa}	0.068	
Extra Time of Simultaneousness	T_{sa}	**5.7**	min
Drilling Time		**104.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**104.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	35.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**35.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.5**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.2**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**172.2**	min
Time for Ventilation break	**II**	**11.4**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	265	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	293	asm^3
Loading Time	$\mathbf{T_{lt}}$	**66.5**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**17.2**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**93.0**	min
Scaling Time without correction factor	T_r	50.6	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**50.4**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**327**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**326.3**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	2.7	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.10	h/m
Radial Bolting time consumption per round	**26.2**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		352	min
Advance rate		**92.9**	m/week

8.1 m diameter (Q = 9.3 & Q = 23)

Cross Section	A_s	55.7	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	91	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	75	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**54.7**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**19.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**15.8**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.068	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**5.7**	min
Drilling Time		**104.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**104.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	35.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**35.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.5**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.2**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**172.2**	min
Time for Ventilation break	**II**	**11.4**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	265	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.16	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	293	asm^3
Loading Time	$\mathbf{T_{lt}}$	66.5	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	17.2	min
Incidental Time for Loading and Hauling	T_{tl}	9.3	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**93.0**	min
Scaling Time without correction factor	T_r	50.6	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**50.4**	min
Net Round cycle time **I + II + III + IV**	T_{nr}	**327**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**326.3**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		326	min
Advance rate		**100.4**	m/week

8.4 m tunnel diameter (Q = 0.003)

Cross Section	A_s	59.9	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	95	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	78	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_l	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**34.2**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	**18.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**9.8**	min
Lack of simultaneousness	f_{sa}	0.066	
Extra Time of Simultaneousness	T_{sa}	**3.8**	min
Drilling Time	T_{bb}	**71.6**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**78.7**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	36.9	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**30.4**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.9**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**13.1**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**138.1**	min
Time for Ventilation break	**II**	**12.2**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	268	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	189	asm^3
Loading Time	$\mathbf{T_{lt}}$	**42.3**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.0**	min
Incidental Time for Loading and Hauling	T_{tl}	**6.7**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**66.9**	min
Scaling Time without correction factor	T_r	75.5	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**62.2**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**279**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**278.7**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	59.9	m^2
Performance of Rebars + 2 layers of shotcrete	**0.63**	h/m
Total Performance Shotcrete ribs	**1.13**	h/m
Shotcrete ribs time consumption per round	**184.9**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**754**	min
Advance rate	**26.1**	m/week

8.4 m tunnel diameter (Q = 0.02)

Cross Section	**A_s**	59.9	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	95	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	78	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	**v_g**	98	cm/min
Drilling Time for charged Holes	T_h	**34.2**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	**t_f**	0.68	min/hole
Time for Moving (Charged and Empty holes)	**T_f**	**18.7**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	**f_k**	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	**T_k**	**9.8**	min
Lack of simultaneousness	**f_{sa}**	0.066	
Extra Time of Simultaneousness	**T_{sa}**	**3.8**	min
Drilling Time	**T_{bb}**	**71.6**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**78.7**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	36.9	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time $\mathbf{T_{lb}}$ **min** (Per Round)	T_l	**30.4**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.9**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**13.1**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**138.1**	min
Time for Ventilation break	**II**	**12.2**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	268	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	189	asm^3
Loading Time	$\mathbf{T_{lt}}$	**42.3**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.0**	min
Incidental Time for Loading and Hauling	T_{tl}	**6.7**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**66.9**	min
Scaling Time without correction factor	T_r	75.5	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**62.2**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**279**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**278.7**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	59.9	m^2
Performance of Rebars + 2 layers of shotcrete	**0.63**	h/m
Total Performance Shotcrete ribs	**1.13**	h/m
Shotcrete ribs time consumption per round	**184.9**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**700**	min
Advance rate	**28.1**	m/week

8.4 m tunnel diameter (Q = 1)

Cross Section	A_s	59.9	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	95	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	78	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**57.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**20.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**16.4**	min
Lack of simultaneousness	f_{sa}	0.066	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**107.9**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**107.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	36.9	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**36.8**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.9**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.8**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**178.5**	min
Time for Ventilation break	**II**	**12.2**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	268	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	315	asm^3
Loading Time	$\mathbf{T_{lt}}$	**70.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.0**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**98.2**	min
Scaling Time without correction factor	T_r	52.8	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**52.7**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**342**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**340.7**	min
Rock Support: Shotcrete (Or Fibrecrete)				
Shotcrete Thickness cm			7	cm
Length of sprayed area m			5	m
Shotcrete (or fibrecrete) required			0.8	m3/m
Performance shotcrete			**0.28**	h/m3
h/m			0.22	h/m
Performance shotcrete (Or fibrecrete) per round			**61.1**	min
Rock Support: Radial Bolts				
Round length			4.55	m
Number of Radial Bolts			6.3	Unit/m
Percentage of bolts at the face			67%	Unit/m
Radial Bolt length			3	m
Radial Bolt Installation after drilling Bolt holes?			No	
Radial Bolting time consumption			0.22	h/m
Radial Bolting time consumption per round			**61.4**	min
Working hour per week			120	h/week
Round length		p_r*l_h	4.55	m
Cycle Time min			463	min
Advance rate			**70.7**	m/week

8.4 m tunnel diameter (Q = 1.9)

Cross Section	A_s	59.9	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	95	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	78	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**57.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**20.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**16.4**	min
Lack of simultaneousness	f_{sa}	0.066	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**107.9**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**107.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	36.9	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**36.8**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.9**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.8**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**178.5**	min

Time for Ventilation break	**II**	**12.2**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	268	asm^3/h
Factor of Overbreak,excluding niches	f_o	1.15	
Pull (drill hole diameter = 48 mm)	p_r	0.91	
Round Volume (Overbreak Included)	V_r	315	asm^3
Loading Time	T_{lt}	**70.4**	min
Rig Time For Loading and Hauling	T_{rl}	**18.0**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**98.2**	min

Scaling Time without correction factor	T_r	52.8	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**52.7**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**342**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**340.7**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	4.5	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.16	h/m
Radial Bolting time consumption per round	**43.8**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		384	min
Advance rate		**85.2**	m/week

8.4 m tunnel diameter (Q = 2.6 ; Q = 6.9; Q = 9.3)

Cross Section	A_s	59.9	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	95	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	78	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**57.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**20.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**16.4**	min
Lack of simultaneousness	f_{sa}	0.066	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**107.9**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**107.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	36.9	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**36.8**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.9**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.8**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**178.5**	min
Time for Ventilation break	**II**	**12.2**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	268	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	315	asm^3
Loading Time	$\mathbf{T_{lt}}$	**70.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.0**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**98.2**	min
Scaling Time without correction factor	T_r	52.8	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**52.7**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**342**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**340.7**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	2.7	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.10	h/m
Radial Bolting time consumption per round	**26.2**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		367	min
Advance rate		**89.3**	m/week

8.4 m (Q = 23)

Cross Section	A_s	59.9	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	95	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	78	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**57.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**20.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**16.4**	min
Lack of simultaneousness	f_{sa}	0.066	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**107.9**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**107.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	36.9	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**36.8**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**15.9**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**17.8**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**178.5**	min
Time for Ventilation break	**II**	**12.2**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	268	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	315	asm^3
Loading Time	$\mathbf{T_{lt}}$	**70.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.0**	min
Incidental Time for Loading and Hauling	T_{tl}	**9.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**98.2**	min
Scaling Time without correction factor	T_r	52.8	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**52.7**	min
Net Round cycle time **I + II + III + IV**	T_{nr}	**342**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**340.7**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		341	min
Advance rate		**96.2**	m/week

8.7 m diameter (Q = 0.003)

Cross Section	**A_s**	64.3	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	100	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	82	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	**v_g**	98	cm/min
Drilling Time for charged Holes	T_h	**36.0**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	**t_f**	0.68	min/hole
Time for Moving (Charged and Empty holes)	**T_f**	**19.6**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	**f_k**	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	**T_k**	**10.3**	min
Lack of simultaneousness	**f_{sa}**	0.063	
Extra Time of Simultaneousness	**T_{sa}**	**3.8**	min
Drilling Time	**T_{bb}**	**74.8**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**82.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		3	
(Without Correction Factor)	T_{lb}	39.0	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**32.1**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.4**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**13.7**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**144.4**	min
Time for Ventilation break	**II**	**13.0**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	271	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	202	asm^3
Loading Time	$\mathbf{T_{lt}}$	**44.6**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**7.0**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**70.5**	min
Scaling Time without correction factor	T_r	78.5	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**64.6**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**293**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**291.8**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	64.3	m^2
Performance of Rebars + 2 layers of shotcrete	**0.64**	h/m
Total Performance Shotcrete ribs	**1.14**	h/m
Shotcrete ribs time consumption per round	**186.2**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**768**	min
Advance rate	**25.6**	m/week

8.7 m diameter (Q = 0.02)

Cross Section	A_s	64.3	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	100	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	82	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	36.0	min
Drilling Time for empty holes	T_g	5.1	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	19.6	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	10.3	min
Lack of simultaneousness	f_{sa}	0.063	
Extra Time of Simultaneousness	T_{sa}	3.8	min
Drilling Time	T_{bb}	**74.8**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**82.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	39.0	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**32.1**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.4**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**13.7**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**144.4**	min

Time for Ventilation break	**II**	**13.0**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	271	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	202	asm^3
Loading Time	$\mathbf{T_{lt}}$	**44.6**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**7.0**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**70.5**	min

Scaling Time without correction factor	T_r	78.5	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**64.6**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**293**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**291.8**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	64.3	m^2
Performance of Rebars + 2 layers of shotcrete	**0.64**	h/m
Total Performance Shotcrete ribs	**1.14**	h/m
Shotcrete ribs time consumption per round	**186.2**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**715**	min
Advance rate	**27.5**	m/week

8.7 m diameter (Q = 1)

Cross Section	A_s	64.3	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	100	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	82	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**60.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**21.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**17.2**	min
Lack of simultaneousness	f_{sa}	0.063	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**112.8**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**112.8**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	39.0	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**38.8**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.4**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**18.7**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**186.7**	min

Time for Ventilation break	**II**	**13.0**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	271	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	337	asm^3
Loading Time	$\mathbf{T_{lt}}$	**74.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**10.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**103.5**	min

Scaling Time without correction factor	T_r	55.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**55.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**358**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**357.3**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		480	min
Advance rate		**68.3**	m/week

8.7 m tunnel diameter (Q =1.9)

Cross Section	A_s	64.3	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	100	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	82	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**60.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**21.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**17.2**	min
Lack of simultaneousness	f_{sa}	0.063	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**112.8**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**112.8**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	39.0	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**38.8**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.4**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**18.7**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**186.7**	min

Time for Ventilation break	**II**	**13.0**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	271	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	337	asm^3
Loading Time	$\mathbf{T_{lt}}$	**74.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**10.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**103.5**	min

Scaling Time without correction factor	T_r	55.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**55.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**358**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**357.3**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	4.5	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.16	h/m
Radial Bolting time consumption per round	**43.8**	min

Working hour per week		120	h/week
Round length	$p_r{*}l_h$	4.55	m
Cycle Time min		401	min
Advance rate		**81.7**	m/week

8.7 m diameter (Q =2.6; Q = 6.9 ; Q = 9.3)

Cross Section	A_s	64.3	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	100	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	82	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**60.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**21.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**17.2**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.063	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**5.6**	min
Drilling Time		**112.8**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**112.8**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	39.0	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**38.8**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.4**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**18.7**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**186.7**	min
Time for Ventilation break	**II**	**13.0**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	271	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	337	asm^3
Loading Time	$\mathbf{T_{lt}}$	**74.4**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**18.8**	min
Incidental Time for Loading and Hauling	T_{tl}	**10.3**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**103.5**	min
Scaling Time without correction factor	T_r	55.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**55.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**358**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**357.3**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	2.7	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.10	h/m
Radial Bolting time consumption per round	**26.2**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		384	min
Advance rate		**85.4**	m/week

8.7 m diameter (Q = 23)

Cross Section	A_s	64.3	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	100	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	82	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**60.0**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**21.5**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**17.2**	min
Lack of simultaneousness	f_{sa}	0.063	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**112.8**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**112.8**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	39.0	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**38.8**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.4**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**18.7**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**186.7**	min
Time for Ventilation break	**II**	**13.0**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	271	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	337	asm^3
Loading Time	$\mathbf{T_{lt}}$	74.4	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	18.8	min
Incidental Time for Loading and Hauling	T_{tl}	10.3	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**103.5**	min
Scaling Time without correction factor	T_r	55.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**55.1**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**358**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**357.3**	min
Working hour per week			120	h/week
Round length		p_r*l_h	4.55	m
Cycle Time min			357	min
Advance rate			**91.7**	m/week

9 m diameter (Q = 0.003)

Cross Section	A_s	68.8	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	104	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	86	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_l	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**37.4**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	**20.3**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**10.7**	min
Lack of simultaneousness	f_{sa}	0.059	
Extra Time of Simultaneousness	T_{sa}	**3.7**	min
Drilling Time	T_{bb}	**77.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**84.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		3	
(Without Correction Factor)	T_{lb}	40.3	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**33.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.8**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.1**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**149.0**	min
Time for Ventilation break	**II**	**13.8**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	275	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	216	asm^3
Loading Time	$\mathbf{T_{lt}}$	**47.1**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**19.6**	min
Incidental Time for Loading and Hauling	T_{tl}	**7.4**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**74.1**	min
Scaling Time without correction factor	T_r	81.5	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**67.1**	min

Net Round cycle time	I + II + III + IV	T_{nr}	**304**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**303.3**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	68.8	m^2
Performance of Rebars + 2 layers of shotcrete	**0.65**	h/m
Total Performance Shotcrete ribs	**1.14**	h/m
Shotcrete ribs time consumption per round	**187.5**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**781**	min
Advance rate	**25.2**	m/week

9 m diameter (Q = 0.02)

Cross Section	**A_s**	68.8	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	104	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	86	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	**v_g**	98	cm/min
Drilling Time for charged Holes	T_h	**37.4**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	**t_f**	0.68	min/hole
Time for Moving (Charged and Empty holes)	**T_f**	**20.3**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	**f_k**	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	**T_k**	**10.7**	min
Lack of simultaneousness	**f_{sa}**	0.059	
Extra Time of Simultaneousness	**T_{sa}**	**3.7**	min
Drilling Time	**T_{bb}**	**77.2**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**84.9**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	40.3	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**33.2**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.8**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.1**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**149.0**	min

Time for Ventilation break	**II**	**13.8**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	275	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	216	asm^3
Loading Time	$\mathbf{T_{lt}}$	**47.1**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**19.6**	min
Incidental Time for Loading and Hauling	T_{tl}	**7.4**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**74.1**	min

Scaling Time without correction factor	T_r	81.5	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**67.1**	min

Net Round cycle time	I + II + III + IV	T_{nr}	**304**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**303.3**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	68.8	m^2
Performance of Rebars + 2 layers of shotcrete	**0.65**	h/m
Total Performance Shotcrete ribs	**1.14**	h/m
Shotcrete ribs time consumption per round	**187.5**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**727**	min
Advance rate	**27.0**	m/week

9 m diameter (Q = 1)

Cross Section	A_s	68.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	104	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	86	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**62.3**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**22.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**17.8**	min
Lack of simultaneousness	f_{sa}	0.059	
Extra Time of Simultaneousness	T_{sa}	**5.5**	min
Drilling Time		**116.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**116.4**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	40.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**40.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.8**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**19.2**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**192.6**	min
Time for Ventilation break	**II**	**13.8**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	275	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	360	asm^3
Loading Time	$\mathbf{T_{lt}}$	**78.5**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**19.6**	min
Incidental Time for Loading and Hauling	T_{tl}	**10.9**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**109.0**	min
Scaling Time without correction factor	T_r	57.7	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**57.5**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**373**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**371.9**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		494	min
Advance rate		**66.3**	m/week

9 m diameter (Q = 1.9)

Cross Section	A_s	68.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	104	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	86	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	62.3	min
Drilling Time for empty holes	T_g	8.5	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	22.2	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	17.8	min
Lack of simultaneousness	f_{sa}	0.059	
Extra Time of Simultaneousness	T_{sa}	5.5	min
Drilling Time		**116.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	116.4	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	40.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**40.2**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.8**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**19.2**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**192.6**	min

Time for Ventilation break	**II**	**13.8**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	275	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	360	asm^3
Loading Time	$\mathbf{T_{lt}}$	**78.5**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**19.6**	min
Incidental Time for Loading and Hauling	T_{tl}	**10.9**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**109.0**	min

Scaling Time without correction factor	T_r	57.7	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**57.5**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**373**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**371.9**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	4.5	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.16	h/m
Radial Bolting time consumption per round	**43.8**	min

Working hour per week		120	h/week
Round length	$p_r{*}l_h$	4.55	m
Cycle Time min		416	min
Advance rate		**78.8**	m/week

9 m diameter (Q = 2.6; Q = 6.9 ; Q = 9.3)

Cross Section	**A_s**	68.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	104	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	86	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	**v_g**	98	cm/min
Drilling Time for charged Holes	T_h	**62.3**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	**t_f**	0.75	min/hole
Time for Moving (Charged and Empty holes)	**T_f**	**22.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	**f_k**	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	**T_k**	**17.8**	min
Lack of simultaneousness	**f_{sa}**	0.059	
Extra Time of Simultaneousness	**T_{sa}**	**5.5**	min
Drilling Time		**116.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**116.4**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	40.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time $\mathbf{T_{lb}}$ **min** (Per Round)	T_l	**40.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.8**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**19.2**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**192.6**	min
Time for Ventilation break	**II**	**13.8**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	275	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	360	asm^3
Loading Time	$\mathbf{T_{lt}}$	**78.5**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**19.6**	min
Incidental Time for Loading and Hauling	T_{tl}	**10.9**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**109.0**	min
Scaling Time without correction factor	T_r	57.7	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**57.5**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**373**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**371.9**	min
	Rock Support: Radial Bolts			
Round length			4.55	m
Number of Radial Bolts			2.7	Unit/m
Percentage of bolts at the face			67%	Unit/m
Radial Bolt length			3	m
Radial Bolt Installation after drilling Bolt holes?			No	
Radial Bolting time consumption			0.10	h/m
Radial Bolting time consumption per round			**26.2**	min
Working hour per week			120	h/week
Round length		p_r*l_h	4.55	m
Cycle Time min			398	min
Advance rate			**82.3**	m/week

9 m diameter (Q = 23)

Cross Section	A_s	68.8	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	104	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	86	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**62.3**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**22.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**17.8**	min
Lack of simultaneousness	f_{sa}	0.059	
Extra Time of Simultaneousness	T_{sa}	**5.5**	min
Drilling Time		**116.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**116.4**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	40.3	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**40.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**16.8**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**19.2**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**192.6**	min
Time for Ventilation break	**II**	**13.8**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	275	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.15	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	360	asm^3
Loading Time	**T_{lt}**	**78.5**	min
Rig Time For Loading and Hauling	**T_{rl}**	**19.6**	min
Incidental Time for Loading and Hauling	T_{tl}	**10.9**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**109.0**	min
Scaling Time without correction factor	T_r	57.7	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**57.5**	min
Net Round cycle time **I + II + III + IV**	T_{nr}	**373**	min
Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**371.9**	min
Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		372	min
Advance rate		**88.1**	m/week

9.3 m diameter (Q = 0.003)

Cross Section	A_s	73.5	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	109	
Drilled length	l_h	3	m
Correction factor for drilled length	k_b	0.8	
Number of drillholes excluding large holes	N	90	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_l	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DR	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	39.3	min
Drilling Time for empty holes	T_g	5.1	min
Time for Moving Between Holes	t_f	0.68	min/hole
Time for Moving (Charged and Empty holes)	T_f	21.2	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	11.2	min
Lack of simultaneousness	f_{sa}	0.057	
Extra Time of Simultaneousness	T_{sa}	3.8	min
Drilling Time	T_{bb}	**80.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**88.5**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	41.6	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**34.2**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**17.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.6**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**154.7**	min
Time for Ventilation break	**II**	**14.6**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	277	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.15	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	230	asm^3
Loading Time	**T_{lt}**	**49.9**	min
Rig Time For Loading and Hauling	**T_{rl}**	**20.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**7.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**78.1**	min
Scaling Time without correction factor	T_r	84.7	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**69.7**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**317**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**316.4**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	18.1	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.97	h/m
Radial Bolting time consumption per round	**159.0**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	73.5	m^2
Performance of Rebars + 2 layers of shotcrete	**0.66**	h/m
Total Performance Shotcrete ribs	**1.15**	h/m
Shotcrete ribs time consumption per round	**188.9**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**796**	min
Advance rate	**24.7**	m/week

9.3 m diameter (Q=0.02)

Cross Section	A_s	73.5	m^2
Skill Level		Low	
Blastability		Poor	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	109	
Drilled length	l_h	3	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	90	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**39.3**	min
Drilling Time for empty holes	T_g	**5.1**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.68	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**21.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**11.2**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.057	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**3.8**	min
Drilling Time	$\mathbf{T_{bb}}$	**80.4**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1.1	
Final Drilling Time per round	T_b	**88.5**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	41.6	min
Correction Factor (Drilled length)	k_{ll}	0.82	
Charging Time **T_{lb} min** (Per Round)	T_l	**34.2**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**17.3**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**14.6**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**154.7**	min

Time for Ventilation break	**II**	**14.6**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	277	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	230	asm^3
Loading Time	$\mathbf{T_{lt}}$	**49.9**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**20.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**7.8**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**78.1**	min

Scaling Time without correction factor	T_r	84.7	min
Correction factor for drilled length	k_{lr}	0.82	
Scaling time	**IV**	**69.7**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**317**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**316.4**	min

Rock Support: Radial Bolts		
Round length	2.73	m
Number of Radial Bolts	12	Unit/m
Percentage of bolts at the face	100%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.64	h/m
Radial Bolting time consumption per round	**105.4**	min

Rock Support: Shotcrete Ribs (Fibrecrete + Rebars + shotcrete)		
Fibrecrete Thichness	15	cm
Length of sprayed area	2.5	m
Quantity of Fibrecrete required	2.7	m^3/m
Performance Fibrecrete	0.18	h/m^3
Performance Fibrecrete	**0.49**	h/m
Cross section Area	73.5	m^2
Performance of Rebars + 2 layers of shotcrete	**0.66**	h/m
Total Performance Shotcrete ribs	**1.15**	h/m
Shotcrete ribs time consumption per round	**188.9**	min

Rock Support: Spiling Bolts		
Number of Spiling Bolt/round	18	Unit/m
Spiling Bolt length m	6	m
Spiling bolting time consumption	**2.19**	h/round
Spiling bolting time consumption per round	**131.2**	min

Working hour per week	120	h/week
Round length	2.73	m
Cycle Time min	**742**	min
Advance rate	**26.5**	m/week

9.3 m tunnel diameter (Q = 1)

Cross Section	A_s	73.5	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	109	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	90	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**65.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**23.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**18.6**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.057	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**5.6**	min
Drilling Time		**121.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**121.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	41.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**41.4**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**17.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**20.0**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**200.0**	min
Time for Ventilation break	**II**	**14.6**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	277	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	384	asm^3
Loading Time	$\mathbf{T_{lt}}$	**83.1**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**20.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**11.5**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**115.1**	min
Scaling Time without correction factor	T_r	60.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**60.0**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**390**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**388.8**	min

Rock Support: Shotcrete (Or Fibrecrete)		
Shotcrete Thickness cm	7	cm
Length of sprayed area m	5	m
Shotcrete (or fibrecrete) required	0.8	m3/m
Performance shotcrete	**0.28**	h/m3
h/m	0.22	h/m
Performance shotcrete (Or fibrecrete) per round	**61.1**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	6.3	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.22	h/m
Radial Bolting time consumption per round	**61.4**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		511	min
Advance rate		**64.1**	m/week

9.3 m tunnel diameter (Q = 1.9)

Cross Section	A_s	73.5	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	109	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	90	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	v_g	98	cm/min
Drilling Time for charged Holes	T_h	**65.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	t_f	0.75	min/hole
Time for Moving (Charged and Empty holes)	T_f	**23.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	f_k	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	T_k	**18.6**	min
Lack of simultaneousness	f_{sa}	0.057	
Extra Time of Simultaneousness	T_{sa}	**5.6**	min
Drilling Time		**121.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**121.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	41.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**41.4**	min

Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**17.3**	min

Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**20.0**	min

Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**200.0**	min

Time for Ventilation break	**II**	**14.6**	min

Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	277	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	384	asm^3
Loading Time	$\mathbf{T_{lt}}$	**83.1**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**20.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**11.5**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**115.1**	min

Scaling Time without correction factor	T_r	60.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**60.0**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**390**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**388.8**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	4.5	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.16	h/m
Radial Bolting time consumption per round	**43.8**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		433	min
Advance rate		**75.7**	m/week

9.3 m tunnel diameter (Q = 2.6; Q = 6.9; Q = 9.3)

Cross Section	A_s	73.5	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	109	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	90	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**65.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**23.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**18.6**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.057	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**5.6**	min
Drilling Time		**121.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**121.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	41.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**41.4**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**17.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**20.0**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**200.0**	min
Time for Ventilation break	**II**	**14.6**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	277	asm^3/h
Factor of Overbreak,excluding niches	$\mathbf{f_o}$	1.15	
Pull (drill hole diameter = 48 mm)	$\mathbf{p_r}$	0.91	
Round Volume (Overbreak Included)	V_r	384	asm^3
Loading Time	$\mathbf{T_{lt}}$	**83.1**	min
Rig Time For Loading and Hauling	$\mathbf{T_{rl}}$	**20.5**	min
Incidental Time for Loading and Hauling	T_{tl}	**11.5**	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**115.1**	min
Scaling Time without correction factor	T_r	60.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**60.0**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**390**	min

Tunnel length	L	4150	m
Correction factor for tunnel length	k_{le}	1.00	
Standard advance rate per round		**388.8**	min

Rock Support: Radial Bolts		
Round length	4.55	m
Number of Radial Bolts	2.7	Unit/m
Percentage of bolts at the face	67%	Unit/m
Radial Bolt length	3	m
Radial Bolt Installation after drilling Bolt holes?	No	
Radial Bolting time consumption	0.10	h/m
Radial Bolting time consumption per round	**26.2**	min

Working hour per week		120	h/week
Round length	p_r*l_h	4.55	m
Cycle Time min		415	min
Advance rate		**78.9**	m/week

Tunnel diameter = 9.3 (Q = 23)

Cross Section	A_s	73.5	m^2
Skill Level		Low	
Blastability		Medium	
Drillhole diameter	d_h	48	mm
Number of charged drillholes	N_b	109	
Drilled length	l_h	5	m
Correction factor for drilled length	k_{bl}	0.8	
Number of drillholes excluding large holes	N_h	90	
Diameter for large drillholes	d_g	102	mm
Number of large drillholes	N_g	4	
Type of Drilling hammer		AC-COP 1838	
Number of drilling hammer		3-boom wheel mounted jumbo	
Rock Drillability	DRI	49	

Penetratrion rate 48 mm drillholes	v_b	217	cm/min
Correction of penetration rate for d_h	k_{hv}	1.00	
Penetration rate charged holes	v_h	217	cm/min
Correction of penetration rate for d_g	k_{gv}	0.45	
Penetration Rate large holes	$\mathbf{v_g}$	98	cm/min
Drilling Time for charged Holes	T_h	**65.4**	min
Drilling Time for empty holes	T_g	**8.5**	min
Time for Moving Between Holes	$\mathbf{t_f}$	0.75	min/hole
Time for Moving (Charged and Empty holes)	$\mathbf{T_f}$	**23.2**	min
Drill bit lifetime (1800 = Very Low rock wear quality; 100 = Very high rock wear quality)		200	
Bit Changing Factor	$\mathbf{f_k}$	0.040	frec/m
Unit time for changing bit	t_k	3.0	min
Time for changing Bit	$\mathbf{T_k}$	**18.6**	min
Lack of simultaneousness	$\mathbf{f_{sa}}$	0.057	
Extra Time of Simultaneousness	$\mathbf{T_{sa}}$	**5.6**	min
Drilling Time		**121.3**	min
Fracture rock factor (To punish up to 10% for weakness zones)		1	
Final Drilling Time per round	T_b	**121.3**	min

Type of Explosive		Emulsion	
Number of Charging lines		2	
(Without Correction Factor)	T_{lb}	41.6	min
Correction Factor (Drilled length)	k_{ll}	1.00	
Charging Time **T_{lb} min** (Per Round)	T_l	**41.4**	min
Rig Time Trb (Service Platform on Jumbo for Charging) Drilling + Charging + Blasting	T_{rb}	**17.3**	min
Incidental lost Time (For Drilling, Charging; Blasting and Rig time)	T_{tb}	**20.0**	min
Total Time until blasting (Drilling + Charging + Blasting + rig + Incidental Time)	**I**	**200.0**	min
Time for Ventilation break	**II**	**14.6**	min
Type of Loader and Transport Equipment		Volvo L 330E 35T Dump truck	
Gross Loading Capacity (f_{ul} = 1)	Q_l	277	asm^3/h
Factor of Overbreak,excluding niches	**f_o**	1.15	
Pull (drill hole diameter = 48 mm)	**p_r**	0.91	
Round Volume (Overbreak Included)	V_r	384	asm^3
Loading Time	**T_{lt}**	83.1	min
Rig Time For Loading and Hauling	**T_{rl}**	20.5	min
Incidental Time for Loading and Hauling	T_{tl}	11.5	min
Total Loading & Hauling (Loading + Rig Time + Incidental time)	**III**	**115.1**	min
Scaling Time without correction factor	T_r	60.2	min
Correction factor for drilled length	k_{lr}	1.00	
Scaling time	**IV**	**60.0**	min

Net Round cycle time	**I + II + III + IV**	T_{nr}	**390**	min
Tunnel length		L	4150	m
Correction factor for tunnel length		k_{le}	1.00	
Standard advance rate per round			**388.8**	min
Working hour per week			120	h/week
Round length		p_r*l_h	4.55	m
Cycle Time min			389	min
Advance rate			**84.3**	m/week

APPENDIX B: Excavation cost detail

Tunnel diameter = 8.7 m

Stretch	l_h m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 1	3	Poor	0.003 - 002	3,764	2241

Cross section	A_s	64.3	m^2
Tunnel length	L	3764	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1754	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1268	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1458	NOK/m
Scaling cost (without correction)	c_{sb}	320	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	454	NOK/m
Drilling + Explosives + Scaling Costs	$C_{dt} = c_d + c_e + c_s$	**3666**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	**457**	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2305	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	c_{ht}	**2467**	NOK/m
Ventilation costs	c_v	975	NOK/m
Electrical installation costs	c_{el}	115	NOK/m
Water supply costs	c_w	124	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1446**	NOK/m
Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**3463**	NOK/m
Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m
Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**11546**	NOK/m
Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**12700**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,713	655
Loading	457	81
Hauling	2,467	435
Additional work	1,446	255
Labour	3,463	611
Unforseen	1,155	204
	12,700	**2240**

Tunnel 8.7 m diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 2	3	Poor	0.003 - 002	4,036	2252

Cross section	A_s	64.3	m^2
Tunnel length	L	4036	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	**1754**	NOK/m

Explosive type		Emulsion	
Exposive costs	c_{eb}	1268	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	**1458**	NOK/m

Scaling cost (without correction)	c_{sb}	320	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	**454**	NOK/m

Drilling + Explosives + Scaling Costs $c_{dt} = c_d + c_e + c_s$	**3666**	NOK/m

Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	**457**	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2357	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	c_{ht}	**2519**	NOK/m

Ventilation costs	c_v	990	NOK/m
Electrical installation costs	c_{el}	113	NOK/m
Water supply costs	c_w	121	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1456**	NOK/m

Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**3463**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**11608**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**12769**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,713	655
Loading	457	81
Hauling	2,519	444
Additional work	1,456	257
Labour	3,463	611
Unforseen	1,161	205
	12,769	**2253**

8.7 m tunnel diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 1	3	Poor	0.003 - 002	4,905	2293

Cross section	A_s	64.3	m^2
Tunnel length	L	4905	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1754	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1268	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1458	NOK/m
Scaling cost (without correction)	c_{sb}	320	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	454	NOK/m
Drilling + Explosives + Scaling Costs	$\mathbf{c_{dt} = c_d + c_e + c_s}$	**3666**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	$\mathbf{c_l}$	457	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2524	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	$\mathbf{c_{ht}}$	2687	NOK/m

Ventilation costs	c_v	1035	NOK/m
Electrical installation costs	c_{el}	109	NOK/m
Water supply costs	c_w	119	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1495**	NOK/m

Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**3463**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**11815**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**12996**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,713	655
Loading	457	81
Hauling	2,687	474
Ventilation + Electrical & Water Supply	1,495	264
Labour	3,463	611
Unforseen	1,181	208
	12,996	**2293**

8.7 m tunnel diameter

Stretch	Ih m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 2	3	Poor	0.003 - 002	3,895	2246

Cross section	A_s	64.3	m^2
Tunnel length	L	3895	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1754	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1268	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1458	NOK/m
Scaling cost (without correction)	c_{sb}	320	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	454	NOK/m
Drilling + Explosives + Scaling Costs	$\mathbf{c_{dt} = c_d + c_e + c_s}$	**3666**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	$\mathbf{c_l}$	457	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2330	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	$\mathbf{c_{ht}}$	2492	NOK/m

Ventilation costs	c_v	982	NOK/m
Electrical installation costs	c_{el}	114	NOK/m
Water supply costs	c_w	122	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1451**	NOK/m
Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab}*k_{la}*k_{La}$	**3463**	NOK/m
Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m
Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**11575**	NOK/m
Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**12733**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,713	655
Loading	457	81
Hauling	2,492	440
Ventilation + Electrical & Water Supply	1,451	256
Labour	3,463	611
Unforseen	1,158	204
	12,733	**2246**

8.7 m tunnel diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 1	5	Medium	1 -23	3,764	2038

Cross section	A_s	64.3	m^2
Tunnel length	L	3764	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1754	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1180	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1180	NOK/m
Scaling cost (without correction)	c_{sb}	235	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	261	NOK/m
Drilling + Explosives + Scaling Costs	$\mathbf{c_{dt} = c_d + c_e + c_s}$	**3196**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	$\mathbf{c_l}$	457	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2305	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	$\mathbf{c_{ht}}$	2467	NOK/m

Ventilation costs	c_v	975	NOK/m
Electrical installation costs	c_{el}	115	NOK/m
Water supply costs	c_w	124	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1446**	NOK/m

Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**2886**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**10498**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**11548**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,243	572
Loading	457	81
Hauling	2,467	435
Additional work	1,446	255
Labour	2,886	509
Unforseen	1,050	185
	11,548	**2037**

8.7 m tunnel diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 2	5	Medium	1 - 23	4,036	2049

Cross section	A_s	64.3	m^2
Tunnel length	L	4036	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	**1754**	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1180	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	**1180**	NOK/m
Scaling cost (without correction)	c_{sb}	235	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	**261**	NOK/m
Drilling + Explosives + Scaling Costs	$c_{dt} = c_d + c_e + c_s$	**3196**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	**457**	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2357	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	c_{ht}	**2519**	NOK/m

Ventilation costs	c_v	990	NOK/m
Electrical installation costs	c_{el}	113	NOK/m
Water supply costs	c_w	121	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1456**	NOK/m

Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab}*k_{la}*k_{La}$	**2886**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**10561**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**11617**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,243	572
Loading	457	81
Hauling	2,519	444
Additional work	1,456	257
Labour	2,886	509
Unforseen	1,056	186
	11,617	**2049**

8.7 m tunnel diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 1	5	Medium	1 -23	4,905	2089

Cross section	A_s	64.3	m^2
Tunnel length	L	4905	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	**1754**	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1180	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	**1180**	NOK/m
Scaling cost (without correction)	c_{sb}	235	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	**261**	NOK/m
Drilling + Explosives + Scaling Costs	$c_{dt} = c_d + c_e + c_s$	**3196**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	**457**	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2524	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	c_{ht}	**2687**	NOK/m

Ventilation costs	c_v	1035	NOK/m
Electrical installation costs	c_{el}	109	NOK/m
Water supply costs	c_w	119	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1495**	NOK/m
Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**2886**	NOK/m
Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m
Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**10767**	NOK/m
Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**11844**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,243	572
Loading	457	81
Hauling	2,687	474
Ventilation + Electrical & Water Supply	1,495	264
Labour	2,886	509
Unforseen	1,077	190
	11,844	**2089**

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 2	5	Medium	1 -23	3,895	2043

Cross section	A_s	64.3	m^2
Tunnel length	L	3895	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1754	NOK/m

Explosive type		Emulsion	
Exposive costs	c_{eb}	1180	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1180	NOK/m

Scaling cost (without correction)	c_{sb}	235	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	261	NOK/m

Drilling + Explosives + Scaling Costs	$c_{dt} = c_d + c_e + c_s$	**3196**	NOK/m

Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	457	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2330	NOK/m
Tip Costs	c_{tip}	162	NOK/m
Sum : Total Hauling Cost	c_{ht}	2492	NOK/m

Ventilation costs	c_v	982	NOK/m
Electrical installation costs	c_{el}	114	NOK/m
Water supply costs	c_w	122	NOK/m
Miscellaneous costs	c_m	232	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1451**	NOK/m

Labour costs	c_{lab}	2886	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab}*k_{la}*k_{La}$	**2886**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**47**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**10528**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**11581**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,243	572
Loading	457	81
Hauling	2,492	440
Ventilation + Electrical & Water Supply	1,451	256
Labour	2,886	509
Unforseen	1,053	186
	11,581	**2043**

Tunnel 9 m diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 1	3	Poor	0.003 - 002	3,768	2324

Cross section	A_s	68.8	m^2
Tunnel length	L	3785	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1828	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1320	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1518	NOK/m
Scaling cost (without correction)	c_{sb}	331	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	470	NOK/m
Drilling + Explosives + Scaling Costs	$c_{dt} = c_d + c_e + c_s$	**3817**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	**482**	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2458	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	c_{ht}	**2630**	NOK/m

Ventilation costs	c_v	994	NOK/m
Electrical installation costs	c_{el}	115	NOK/m
Water supply costs	c_w	123	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1476**	NOK/m
Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**3571**	NOK/m
Type of Niches		Turning niches	
Niches	c_n	0	NOK/m
Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**11976**	NOK/m
Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**13174**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,817	673
Loading	482	85
Hauling	2,630	464
Ventilation + Electrical & Water Supply	1,476	260
Labour	3,571	630
Unforseen	1,198	211
	13,174	**2324**

Tunnel 9 m diameter

Stretch	Ih m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 2	3	Poor	0.003 - 002	4,032	2335

Cross section	A_s	68.8	m^2
Tunnel length	L	4015	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1828	NOK/m

Explosive type		Emulsion	
Exposive costs	c_{eb}	1320	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1518	NOK/m

Scaling cost (without correction)	c_{sb}	331	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	470	NOK/m

Drilling + Explosives + Scaling Costs $c_{dt} = c_d + c_e + c_s$	**3817**	NOK/m

Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	482	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2505	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	c_{ht}	2677	NOK/m

Ventilation costs	c_v	1007	NOK/m
Electrical installation costs	c_{el}	113	NOK/m
Water supply costs	c_w	121	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1485**	NOK/m

Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**3571**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**0**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**12032**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**13236**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,817	673
Loading	482	85
Hauling	2,677	472
Ventilation + Electrical & Water Supply	1,485	262
Labour	3,571	630
Unforseen	1,203	212
	13,236	**2335**

9 m tunnel diameter

Stretch	Ih m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 1	3	Poor	0.003 - 002	4,883	2377

Cross section	A_s	68.8	m^2
Tunnel length	L	4883	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1828	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1320	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1518	NOK/m
Scaling cost (without correction)	c_{sb}	331	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	470	NOK/m
Drilling + Explosives + Scaling Costs	$C_{dt} = c_d + c_e + c_s$	**3817**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	482	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2684	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	C_{ht}	2855	NOK/m

Ventilation costs	c_v	1053	NOK/m
Electrical installation costs	c_{el}	109	NOK/m
Water supply costs	c_w	119	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1525**	NOK/m

Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**3571**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**0**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**12251**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**13476**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,817	673
Loading	482	85
Hauling	2,855	504
Additional works	1,525	269
Labour	3,571	630
Unforseen	1,225	216
	13,476	**2377**

9 m tunnel diameter

Stretch	Ih m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 2	3	Poor	0.003 - 002	3,917	2330

Cross section	A_s	68.8	m^2
Tunnel length	L	3917	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Poor	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1828	NOK/m

Explosive type		Emulsion	
Exposive costs	c_{eb}	1320	NOK/m
Correction factor for drilled length	k_{le}	1.15	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1518	NOK/m

Scaling cost (without correction)	c_{sb}	331	NOK/m
Correction for drilled length	k_{ls}	1.42	
Scaling cost	c_s	470	NOK/m

Drilling + Explosives + Scaling Costs $\mathbf{c_{dt} = c_d + c_e + c_s}$	**3817**	NOK/m

Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	$\mathbf{c_l}$	482	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2485	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	$\mathbf{c_{ht}}$	2657	NOK/m

Ventilation costs	c_v	1001	NOK/m
Electrical installation costs	c_{el}	114	NOK/m
Water supply costs	c_w	122	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1481**	NOK/m

Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.2	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**3571**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**0**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**12008**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**13209**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,817	673
Loading	482	85
Hauling	2,657	469
Additional works	1,481	261
Labour	3,571	630
Unforseen	1,201	212
	13,209	**2330**

9 m tunnel diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 1	5	Medium	1 -23	3,768	2114

Cross section	A_s	68.8	m^2
Tunnel length	L	3785	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1828	NOK/m
Explosive type		Emulsion	
Exposive costs	c_{eb}	1229	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1229	NOK/m
Scaling cost (without correction)	c_{sb}	245	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	271	NOK/m
Drilling + Explosives + Scaling Costs	$c_{dt} = c_d + c_e + c_s$	**3329**	NOK/m
Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	482	NOK/m
Hauling costs (roadway not I ncluded here)	c_h	2458	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	c_{ht}	2630	NOK/m

Ventilation costs	c_v	994	NOK/m
Electrical installation costs	c_{el}	115	NOK/m
Water supply costs	c_w	123	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1476**	NOK/m
Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**2976**	NOK/m
Type of Niches		Turning niches	
Niches	c_n	**0**	NOK/m
Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**10893**	NOK/m
Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**11983**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,329	587
Loading	482	85
Hauling	2,630	464
Ventilation + Electrical & Water Supply	1,476	260
Labour	2,976	525
Unforseen	1,089	192
	11,983	**2114**

9 m tunnel diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Castillo 2	5	Medium	1 -23	4,032	2125

Cross section	A_s	68.8	m^2
Tunnel length	L	4015	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	**1828**	NOK/m

Explosive type		Emulsion	
Exposive costs	c_{eb}	1229	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	**1229**	NOK/m

Scaling cost (without correction)	c_{sb}	245	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	**271**	NOK/m

Drilling + Explosives + Scaling Costs $c_{dt} = c_d + c_e + c_s$	**3329**	NOK/m

Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	**482**	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2505	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	c_{ht}	**2677**	NOK/m

Ventilation costs	c_v	1007	NOK/m
Electrical installation costs	c_{el}	113	NOK/m
Water supply costs	c_w	121	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1485**	NOK/m
Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**2976**	NOK/m
Type of Niches		Turning niches	
Niches	c_n	**0**	NOK/m
Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**10949**	NOK/m
Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**12044**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,329	587
Loading	482	85
Hauling	2,677	472
Ventilation + Electrical & Water Supply	1,485	262
Labour	2,976	525
Unforseen	1,095	193
	12,044	**2125**

9 m tunnel diameter

Stretch	lh m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 1	5	Medium	1 -23	4,883	2167

Cross section	A_s	68.8	m^2
Tunnel length	L	4883	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1828	NOK/m

Explosive type		Emulsion	
Exposive costs	c_{eb}	1229	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1229	NOK/m

Scaling cost (without correction)	c_{sb}	245	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	271	NOK/m

Drilling + Explosives + Scaling Costs	$\mathbf{c_{dt} = c_d + c_e + c_s}$	**3329**	NOK/m

Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	$\mathbf{c_l}$	**482**	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2684	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	$\mathbf{c_{ht}}$	**2855**	NOK/m

Ventilation costs	c_v	1053	NOK/m
Electrical installation costs	c_{el}	109	NOK/m
Water supply costs	c_w	119	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1525**	NOK/m

Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**2976**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**0**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**11168**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**12284**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,329	587
Loading	482	85
Hauling	2,855	504
Additional works	1,525	269
Labour	2,976	525
Unforseen	1,117	197
	12,284	**2167**

9 m tunnel diameter

Stretch	Ih m	Blastability	Q range	Length reference m	Excavation Cost USD/m
Vallical 2	5	Medium	1 -23	3,917	2120

Cross section	A_s	68.8	m^2
Tunnel length	L	3917	m
Excavation method		Trackless	
Drillability (Good = 65 ; Poor = 37)		49	
Blastability (Good, Medium or Poor)		Medium	
Drill Hole diameter	d_h	48	mm
Type of Jumbo		3-boom wheel mounted jumbo	
Total Drilling Cost	c_d	1828	NOK/m

Explosive type		Emulsion	
Exposive costs	c_{eb}	1229	NOK/m
Correction factor for drilled length	k_{le}	1.00	
Correction factor for dynamite proportion	k_{de}	1.0	
Corrected explosive costs	$c_e = c_{eb} * k_{le} * k_{de}$	1229	NOK/m

Scaling cost (without correction)	c_{sb}	245	NOK/m
Correction for drilled length	k_{ls}	1.11	
Scaling cost	c_s	271	NOK/m

Drilling + Explosives + Scaling Costs $c_{dt} = c_d + c_e + c_s$	**3329**	NOK/m

Loading & Hauling equipment		Volvo L330E - 35t dump truck	
Loading Cost	c_l	482	NOK/m

Hauling costs (roadway not I ncluded here)	c_h	2485	NOK/m
Tip Costs	c_{tip}	172	NOK/m
Sum : Total Hauling Cost	c_{ht}	2657	NOK/m

Ventilation costs	c_v	1001	NOK/m
Electrical installation costs	c_{el}	114	NOK/m
Water supply costs	c_w	122	NOK/m
Miscellaneous costs	c_m	244	NOK/m
Additional Costs	$c_a = c_v + c_{el} + c_w + c_m$	**1481**	NOK/m

Labour costs	c_{lab}	2976	NOK/m
Correction factor for drilled length	k_{la}	1.0	
Correction factor tunnel length	k_{La}	1.0	
Labour	$c_{la} = c_{lab} * k_{la} * k_{La}$	**2976**	NOK/m

Type of Niches		Turning niches	
Niches	c_n	**0**	NOK/m

Sum Elemental Cost	$c_t = c_{dt} + c_l + c_{ht} + c_a + c_{la} + c_n$	**10925**	NOK/m

Unforseen costs	k_u	1.1	
Standard Cost	$k_u * c_t$	**12018**	NOK/m

	NOK/m	USD/m
Drilling + Explosives + Scaling + Niches	3,329	587
Loading	482	85
Hauling	2,657	469
Additional works	1,481	261
Labour	2,976	525
Unforseen	1,093	193
	12,018	**2120**