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TBM Tunneling in Hard Rock Conditions

Verification of the NTNU and the Q_{tbm}
estimation models and assessment of TBM
performance at the Upper Kontum
Hydroelectric Powerplant

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Abstract

The use of Tunnel Boring Machines when excavating tunnels allows for a highly industrialized form of production, while at the same time introducing new kinds of risks and challenges in regards of performance estimation and advance rates. The work in this thesis is focused on verifying the NTNU-and Q_{TBM}-models for performance estimation. In addition, there has been conducted an assessment of the time required for achieving high utilization after a longer period of stand-still due to maloperation of a TBM. A general assessment of the project execution is included to set the performance in context with its surroundings. A field study at the Upper Kontum Hydroelectric Powerplant in Vietnam was conducted to gather data and to do a general assessment of the work at the site. Geological back-mapping of parts of the tunnel and penetration tests with the TBM was conducted, in addition to assessing the general quality of the production system. Geology and performance data for the entire tunnel length was reviewed in cooperation with the project geologists to ensure a correct assessment and analysis.

The data gathered suggested that both estimation models gave quite accurate results for net penetration rate in the geology found at site, while also suggesting that the Q_{TBM}-model in certain cases is too influenced by the applied cutter thrust. There should be expected approximately six months of reduced utilization before an increase is observed when restarting a project, due to the complexity of ensuring all systems work properly after maloperation of the TBM. The utilization given by the NTNU-model appears to be somewhat optimistic, while utilization given by the Q_{TBM}-model does not provide a realistic estimation. The use of wider cutter tips appears to ensure a significantly higher expected lifetime than what is suggested by the Cutter Lifetime Index.

In total, TBM performance can be accurately estimated when using the estimation models. There should however be extensive knowledge regarding the functional structure of the models, to ensure that their limitations are well understood. Replacing contractors and taking over machines should be done with extensive care, as the potential limitations on utilization may cause severe delays and cost overruns.

Sammendrag

Bruk av tunnelboremaskiner for driving av tunneler legger til rette for en effektiv form for produksjon, samtidig som annen risiko og utfordringer knyttet til fremdriftestimering og produktivitet oppstår. Arbeidet I denne avhandlingen fokuserer på verifikasjon av NTNU- og Q_{TBM}-modellene for fremdriftsestimering. I tillegg har det blitt gjennomført en analyse av tidsaspektet for å oppnå høy maskinutnyttelse ved gjenoppstart av prosjekter, hvor feilstyring av maskinen har ført til store skader på utstyret. En generell vurdering av prosjektgjennomførelsen er også inkludert for å sette den generelle prestasjonen i sammenheng med prosjektets omgivelser. Et feltstudieopphold ved "Upper Kontum Hydroelectric Powerplant" ble gjennomført for innsamling av data og for å gjøre en generell vurdering av arbeidet på byggeplassen. Det ble gjennomført en geologisk kartlegging i deler av tunnelen og kjøring av matekrafttester med TBMen, samt en gjennomgang av kvaliteten på produksjonssystemet på plassen. Geologi- og produksjonsdata for hele tunnelen ble gjennomgått i samarbeid med prosjektgeologene for å sikre korrekt vurdering og analyse av informasjonen.

Den innsamlede dataen indikerer at begge fremdriftsmodellene ga nøyaktige resultater i de geologiske forholdene på plassen, samtidig som de indikerte at Q_{TBM}-modellen er noe for lettpåvirkelig av matekraft. Det burde forventes en periode på seks måneder med nedgang i maskinutnyttelse før den øker igjen ved gjenoppstart, grunnet kompleksiteten med å få alle systemer til å fungere problemfritt etter feilstyring av TBMen. Maskinutnyttelsen som gis av NTNU-modellen ser ut til å være noe optimistisk, mens utnyttelsen gitt av Q_{TBM} ikke representerer noe realistisk estimat. Bruken av større kutterbredder ser ut til å gi en vesentlig høyere kutterlevetid enn hva som angis av kutterlevetidsindeksen.

Prestasjonen til TBM kan estimeres nøyaktig ved bruk av fremdriftsmodellene. En burde likevel ha inngående kunnskap om den funksjonelle oppbyggingen av modellene, for å sikre at begrensinger i modellene er vel forstått. Overtakelse av kontrakter med utstyr andre entreprenører har operert burde gjøres med omfattende forsiktighet, da potensielle begrensinger på maskinutnyttelser kan føre til alvorlige forsinkelser og kostnadsoverskridelser.

Preface and Acknowledgements

The work in this thesis is the culmination of achieving a Master of Science in Civil and Environmental Engineering and concludes my five years as a student at the Norwegian University of Science and Technology. The subject of the thesis has been the driving of tunnels with the use of Tunnel Boring Machines. A field study in Central-Vietnam was conducted to gather data for the research, while the analysis was conducted at the Department of Civil and Environmental Engineering at NTNU.

I would like to express my gratitude to the following people for supporting the work in my thesis:
My main supervisor Associate Professor Pål Drevland Jakobsen, co-supervisor Dr. Francisco Javier Macias and Professor Amund Bruland for advising and assessing the work conducted in the thesis. Fellow student Ola Hobbelstad, who accompanied me to the site in Vietnam and who I gathered data in cooperation with. I would also like to thank the site personnel from The Robbins Company for their support and help, especially site manager Greg Adams, who ensured that we got to achieve our intended goals for the stay. Sindre Log of The Robbins Company Norway, who helped facilitate the stay at the project site.

Trondheim, June 10^{th} 2018
Egil Zahl Johansen

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Chapter 1

Introduction

As the rapid economic development continues across the globe, enormous investments in civil infrastructure is needed due to the increased demand of services as water-supply, electricity, transportation and waste-water collection. The introduction of Tunnel Boring Machines (TBM) for Hard Rock Conditions has given engineers a gentler way of tunneling in urban areas sensitive to vibrations and settlement issues. It has also proven to be a potential faster way of tunneling due to the industrialized character of the boring process, given preferable geology and that the machine is designed correctly to handle this. The continued development and increased knowledge of the use of TBMs has made possible projects previously deemed out-of-reach from an engineering point of view, as well as It has also opened the mind to new civil projects that had not been thought of previously.

At the same time as technology improves and knowledge regarding the use of TBMs increases, the limits of what kind of projects that can be taken on are continuously expanded. With this increasing complexity of the projects that are initiated, more extensive knowledge regarding prediction models for advance rates are needed. The execution time for tunnels is a key component in the cost estimation of the project itself, for the planning of the execution of the construction, and also for the profitability of the project as an investment from an owner. If the current trend of more complex projects is set to continue, or even accelerate, verification and improvement of the prediction models is needed to handle these challenges.

1.1 General Remarks

Estimation models for Hard Rock Tunnel Boring are a necessary tool for planning and executing projects using a TBM. Due to technological development and an increased scope of geological conditions they are being applied, they continuously need to be verified and updated to ensure their accuracy. Both the NTNU model and the Q_{TBM}-model give estimations regarding the net penetration rate and the overall performance, when taking into account all other activities that is part of the boring operation. They also rely on laboratory testing and field observations, which requires and throughout understanding of how to interpret the results and the limitations of the information provided.

1.2 Objective of the Thesis

The objective of the work in this Master Thesis is to gather quantitative data for analysis and doing qualitative assessments of conditions that may have influenced the data. The purpose is to attain information that may be used in the verification of both estimations models and provide insight in areas where the estimations deviates from the actual performance. Data regarding what influences the net penetration rate, as rock-mass fracturing, rock parameters and machine operation will be gathered. The gross penetration rate, or weekly advance rate, will be subject to a more comprehensive review, by collecting data regarding the production system as a whole. This includes all maintenance activities, shut-downs of operation and cutter consumption, and assessing what may have influenced the data regarding these issues. The project assessed was the construction of the headway tunnel for the Upper Kontum Hydroelectric Powerplant in Central Vietnam.

1.3 Scope and Limitations

The scope of the work in this thesis is a 5-week field study at the project site in Central Vietnam where data would be collected and the analysis of production data for the entire execution of the project. The analysis and the assessment of the data, and laboratory testing of geological samples, was conducted at the Department of Civil and Environmental Engineering at the Norwegian University of Science and Technology (NTNU) and the SINTEF laboratories.

As quite comprehensive amount data regarding the boring process and the surroundings were available, a wide scope of issues has been covered in the thesis. Though not all elements were equally thoroughly examined, it was still done at a level where one could successfully identify the root-cause of several interconnected issues regarding the tunneling. There are many other widely used estimation models, such as the "Modified Colorado School of Mines"(Yagiz, 2002), that has not been covered by this thesis, including theoretical models using other parameters than what is utilized in the NTNU and Q_{TBM}-model(Yagiz, 2008a).

1.4 Structure of the Thesis

The thesis consists of a total of 5 chapters.

Chapter two covers the theory and background for the estimation models and the project reviewed. A detailed description of how the models function is laid out to emphasize how the input parameters are processed to give and output in form of an estimation. A brief introduction to the project is given. A detailed description of the history regarding the TBM is included, as this is one of the central issues covered in the thesis. Some information regarding the contractual arrangements is also discussed, as incentive mechanisms may in some cases reward unfortunate behavior, which in our case would inflict upon the data collected at the project.

In chapter three, a description of all the research methodology is given. The standard field testing procedure is laid out, with focus on adaptions to project specific considerations that had to be made. Laboratory testing, how data received from the project was sorted and analyzed and literature review is also covered.

All the results and discussions are given in chapter four. As the thesis covers a wide range of issues, the result and discussion are completed in each section before moving on to the next subject. The chapter starts with a general review of the geological mapping by The Robbins Company and its results when used as an input in the estimation models. It then goes on to the mapping conducted during the field study and the resulting estimations given by this input. The chapter then continues with a comparison of the mapping during the field study and that of The

Robbins Company in the same area, with the purpose of looking for discrepancies. Continuing, the results from the tests conducted and chip sampling are covered. The section covering the mapping methodology is more of theoretical focus, where it gives a mathematical rationale and look at the consequences of different analytical lengths. The utilization of the TBM is then covered, looking at all the aspects, both internal and external, that affected this parameter. The last subject covered is the data available on cutter consumption and how it related to the laboratory results and estimations.

Chapter five provides some conclusive remarks and suggestions for further work, based on issues unveiled from the field study in Vietnam.

Chapter 2

Background and Theory

2.1 The NTNU-Model

The NTNU prediction model for hard rock TBMs is an empirical model based on including relevant machine and rock parameters to estimate the penetration rate. It uses performance data from previous tunneling projects, laboratory tests of core samples and geological mapping to give an estimate for penetration rate in future projects. The purpose of combining this input is to give an as objective assessment as possible of the penetration rate as possible by taking into account time consuming practical issues one faces during tunneling, that a pure theoretical model will not be able to anticipate. The model is intended to be used as a tool for:

- Estimating net penetration and cutter wear
- Estimating time consumption and excavation costs
- Assessing risk linked to variation in rock mass boreability and machine parameters, including its impact on time consumption and costs
- Establishing and managing contract price regulation
- Verifying machine performance
- Verifying variation in geological conditions

The model can be used at every stage of a project, from feasibility studies to construction, and in eventual claims and disputes(Macias, 2016).

The NTNU model was published for the first time in 1976 and has gone through several revisions and expansions since then. All versions can be seen in table 2.1.

Table 2.1: NTNU-model versions (Macias, 2016)

Edition	**Year**
1^{st} edition	1976
2^{nd} edition	1979
3^{rd} edition	1983
4^{th} edition	1986
5^{th} edition	1994
6^{th} edition	2000

The 7th edition was published by Francisco Javier Macias in 2016 and is currently the newest version of the model. A renewal of the data the model is based on is necessary at regular intervals, as increased material quality, higher degree of automation and other technological improvements influence the general performance of the TBM. These improvements also make more complex projects feasible to execute, thus widening the scope of conditions one operates in.

Several machine and rock parameters are included in the model to ensure that as many aspects as possible are considered when estimating the penetration rate. When estimating the net penetration rate, parameters in table 2.2 are used.

Table 2.2: Parameters for estimating NPR (Macias, 2016)

ROCK PROPERTIES		**MACHINE**
Intact Rock	**Rock Mass**	**PARAMETERS**
Drilling Rate Index, DRI	Rock Mass Fracturing Factor (k_s)	TBM diameter
Porosity		Cutter diameter
		Number of cutters
		Gross average cutter thrust
		Average cutter spacing
		Cutterhead rpm

Rock properties includes both the intact rock alone, to assess its influence on penetration rate, and the rock mass as a whole, to include the effects of discontinuities created after its formation(Moon & Oh, 2012)(Bruland, 1998). The relevant machine parameters included all take into account some effect on the net penetration. An increased diameter will increase the area of the tunnel face, thus increasing that volume that needs to be excavated. Increased cutter diameter will increase the contact area between the rock and the cutter, thus reducing the tension in the rock. An increased number of cutters will increase the amount of destructive work per revolution of the cutter head, and thus increase the net penetration rate. The gross average cutter thrust is the most influential parameter to the net penetration rate, which increases approximately exponentially with this parameter. If the cutter spacing is increased, the induced fractures created by the passing cutter will have a smaller possibility of hitting each other, thus reducing the amount of material excavated per revolution of the cutter head. The net penetration increases with the cutterhead RPM, but reaches an optimum at a certain point.

The cutter wear is also dependent on the properties of the rock and machine parameters which can be seen in table 2.3

Table 2.3: Parameters for estimating Cutter Wear (Macias, 2016)

ROCK PROPERTIES	MACHINE PARAMETERS
Cutter Life Index, CLI	TBM diameter
	Cutter diameter
Content of abrasive minerals	Number of cutters
	Cutterhead rpm
	Gross average cutter thrust

The cutter wear, or indirectly the cutter consumption, heavily affects the machine utilization and thus the gross penetration rate of the machine. Changing a cutter is very time consuming and costly, and is in many cases the most influential time-consuming activity with the exception of boring. Cutter wear is only dependent on the rock properties in the model. When increasing the TBM diameter, the average lifetime of the cutters will increase. This is due to the fact that center and gauge cutters have shorter lifetimes than the face cutter. When increasing the diameter, the percentage of these cutters as a portion of all cutters will decrease, which leads to an increase in the average lifetime. An increase in the cutter diameter gives a more solid cutter and thereby increases the lifetime of the cutter. When there are installed more cutters than what the model suggests, the lifetime of the cutters will be prolonged. Increasing the cutterhead RPM will increase the amount of work done by the cutter per minute, thus decreasing the lifetime. The gross average cutter thrust will heavily affect the lifetime at low values of the CLI, by decreasing the lifetime of the cutters when increasing the thrust. An aspect not covered in the model is the influence of gross average cutter thrust for higher values of CLI and the rock mass fracturing on the cutter lifetime. The thrust level is restricted by the rock mass fracturing, since extensive vibrations that may damage the machine will occur for high cutter thrust levels in high values of k_s. Lower thrust values will lead to less wear on the cutter rings, thus increasing the lifetime of these. There has been some critique that the model is not well suited for jointed faulted hard rock comditions, which can be related to this (Yagiz, 2008b).

As we can see, there are certain parameters that will increase the net penetration rate when increasing its value, while at the same time decreasing the cutter lifetime. There thus exists an optimum where the gain in the increased net penetration rate lost due to lower utilization of the TBM by increased cutter consumption. This optimum may be influenced by other machine and rock properties, even though only some parameters are assumed interdependent in the model.

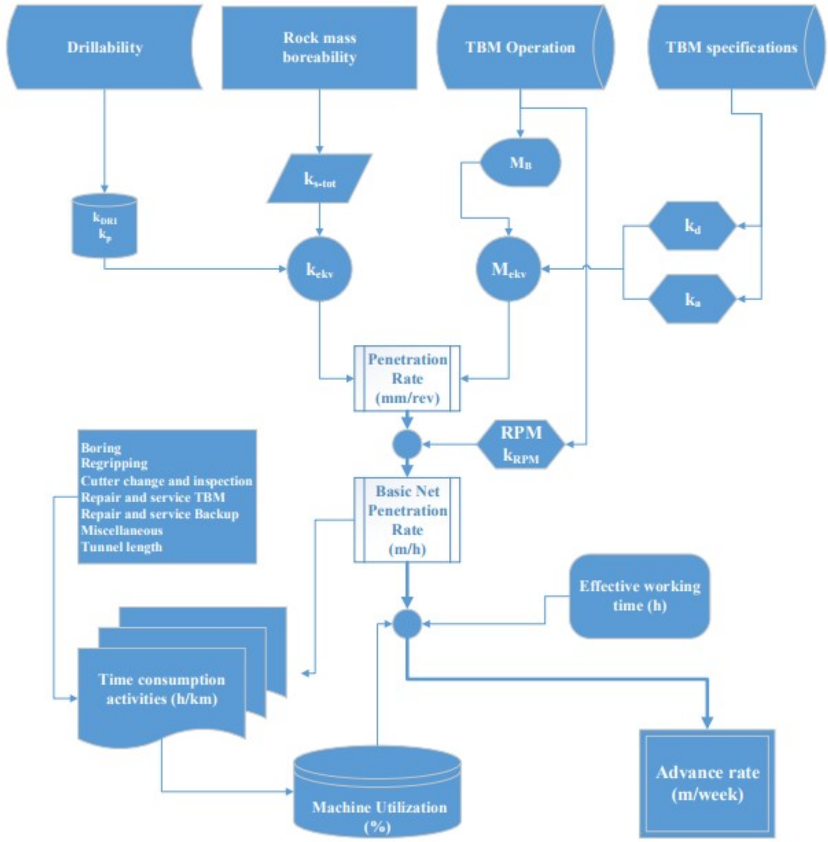

Figure 2.1: Flowchart of functional structure of NTNU-model (Macias, 2016)

The flowchart in figure 2.1 shows the functional structure of the model. The intact rock and the rock mass properties are combined to give an equivalent rock mass fracturing factor. This is done by identifying up to three sets of fractures in the rock mass and combining this into one factor that describes the fracturing of the rock mass. This multiplied by correction factors for the measured DRI and the porosity of the rock, giving us the equivalent fracturing factor. As for the TBM operation, its gross average thrust per cutter are corrected by factors for the average spacing and the diameter of the cutters. This gives us an equivalent gross average thrust per cutter, which is combined with the penetration rate to give us the advancement per revolution of the cutter head.

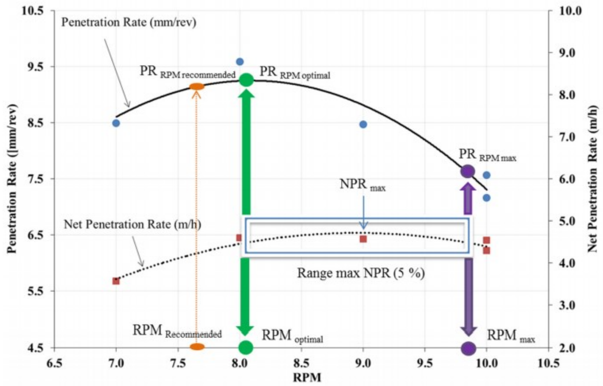

Figure 2.2: Connection between NPR and BPR (Macias, 2016)

The penetration rate per revolution as can be seen from figure 2.2 is dependent on the cutterhead velocity, and reaches an optimum at a certain point. The continued increase in RPM can compensate for the decrease in penetration per revolution, and we will reach an optimum in net penetration rate at a higher RPM than for the optimal penetration rate. At this point, we have gotten the basic net penetration rate, which is the progress the TBM is making while in operation.

Activities other than boring will also consume portions of the number of working hours, thus bringing down the machine utilization. Except for cutterhead velocity and gross average thrust per cutter at low values of CLI, the lifetime of the cutters is modeled to solely depend on the rock properties and the TBM specifications, while being independent of the TBM operation. This will not be the case in reality, as the accumulated wear on the cutters if operating at maximum gross thrust and cutterhead velocity will be higher per meter of tunnel than if operated at recommended values. Thus, operating the TBM in certain ways may increase the net penetration rate while decreasing the gross penetration rate or advancement per week. Other time-consuming activities are merely time that has to be used for maintenance and occasional breakdowns of some support system. The tunnel length factor takes into account the learning curve for the crew operating the machine and gives an interval for the extra time consumption for inexperienced vs experienced crews.

The NTNU model is designed to estimate progress rates for "open gripper" TBMs. These are not subject to the frictional forces on shielded TBMs that influences the average net cutter thrust, and eventual halts in boring due to installation of concrete shell lining in the tunnel. The frictional forces may be very site specific and has to be estimated at each project. For concrete shell lining of the tunnel, it has to be assumed that the logistical system is dimensioned to deliver enough concrete elements to the TBM for installation, so that the boring operation is not disturbed(The Robbins Company, n.d.)(Herrenknecht AG, n.d.). Single shield TBMs and double shield in single shield mode will have to stop boring when installing a new segment of elements. This can be considered in the model by adding time to the re-gripping operation for the segment to be installed(Maidl, Schmid, Ritz, & Herrenknecht, 2008).

The functional description of the NTNU model gives an overview over what the philosophy behind the model is, but not how all the input parameters are interconnected, and especially not how cutter lifetime is connected to all the input. Figures 2.3, 2.4 and 2.5 shows the interconnection between all parameters, and how some input both influences the net penetration rate and the cutter wear, and thus indirectly gross penetration rate.

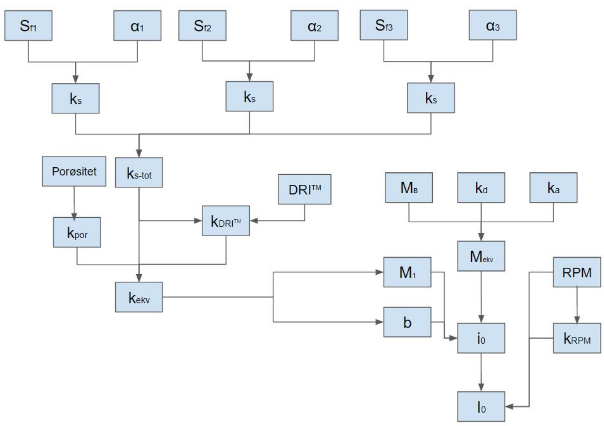

Figure 2.3: Detalied connection between parameters for geology and machine parameters in the NTNU Model

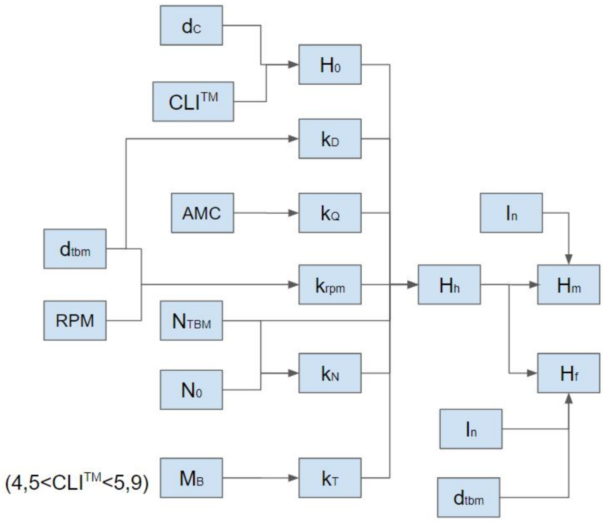

Figure 2.4: Detalied connection between parameters for cutter lifetime in the NTNU Model

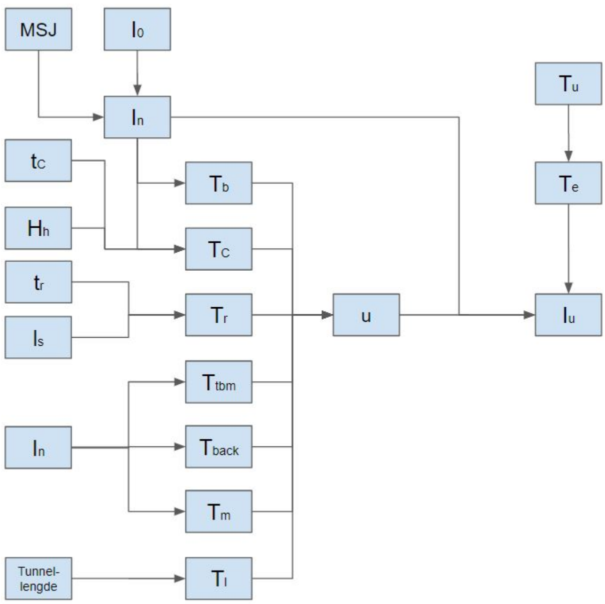

Figure 2.5: Detalied connection between parameters for gross advance rate in the NTNU Model

2.2 The Q_{TBM}-model

The Q_{TBM} model for TBM progress estimation was developed in 1999 and is based on utilizing the Q-system for rock mass classification to estimate the time needed for boring a tunnel, in addition to other parameters(Barton, 1999). The system was developed at NGI between 1971 to 1974 by mapping the use of support measures in Norwegian tunnels and caverns, with the purpose of creating a general experience-based system for support measures in underground structures.

2.2.1 Q-System

The system has been revised several times due to development in support measures and installation techniques and increased knowledge regarding the use of these, to keep up with the state of the art on these issues(NGI, 2015). Currently, the Q-system utilizes 6 parameters to calculate the Q-value, which in turn has a recommended scope of support measures based on this value, the roof span and the intended use of the tunnel or cavern. The value is given by the equation
$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{j_w}{SRF}$
with each parameter representing

- RQD = Degree of Jointing (Rock Quality Designation)
- J_n = Joint set number
- J_r = Joint roughness number
- J_a = Joint alteration number
- J_w = Joint water reduction factor
- SRF = Stress Reduction Factor

The RQD is decided either my counting the number of joints per m^3 of rock mass or by summarizing the sections in 1 m of core samples longer than 10 cm(Deere, 1964). Deciding the RQD in foliated or schistose rock may be challenging as the planes in these rocks may in some cases represent planes of weakness, while in other not. Under some circumstances, core samples may be intact right after it is recovered from the rock, but crack into pieces after some time when the sample has dried up. Deciding what value of RQD to use in this case is challenging, and must be considered in the support design. For progress estimation in boring, only short-term considerations have to be taken into account, as the TBM operation passes by each area relatively fast. In soft rocks, the RQD may appear to be high as these rocks tends to deform rather than crack in many cases. In rocks that are weakly consolidated or highly weathered and non-cohesive that can be defined as soil from an engineering geology point of view, the value should be set to 10. This is also the case when hitting materials such as clay, as the material will act as a weakness zone compared to its surroundings. The Joint Set Number J_n gives a value for the number of joint sets in an area. When mapping this in a tunnel, only the sets discovered in a restricted area should be counted as the value will become too high if a longer stretch is surveyed. Together, the $\frac{RQD}{J_n}$ factor gives us the relative block size in the rock masses.

The Joint Roughness Number J_r describes the surfaces of the weakness planes, or indirectly the friction between two surfaces. The parameter describes in two scales. The small scale, which takes into account unevenness in cm or mm, or what can be felt by sliding a finger over the surface. The large scale considers unevenness or amplitudes when looking at sections up to a meter. In cases with infill in the joints, the Jr is set to the lowest value if the amount of infill is so large that there will be no contact even at 10 cm shear deformation. These two scales combined will give a recommended value for J_n, with exceptions in certain cases. The Joint Alteration Number J_a considers the joint infill, both its thickness and the friction angle (or strength) of the mass. This takes into consideration both situations where the thickness of the infill may be a thin layer and the surfaces are smooth, thus ensuring no contact between the surfaces, and also situations with thick infill and coarse surfaces where there also may or may not be contact between the surfaces. Infills susceptibility to water is a case that also have to be considered, as exposure to water may lead to large swelling pressures in some cases, while significantly lower the strength of the material in other cases. The $\frac{J_r}{J_a}$ factor thus describes the actual joint friction that will be observed in these cases. The tangent inverse of this number will in addition give a fair approximation of the actual friction angle for each case.

The Joint Water Reduction Factor J_w takes into account the effect of water pressure reducing the normal forces and thereby the friction, and the softening or wash out of the infill. This may cause the blocks to shear more easily. When deciding the value of the factor, it has to be considered there are outside factors influencing the inflow of water while mapping the tunnel. Seasonal differences in precipitation, temporary lowering of the ground water table and freezing due to cold weather are all effects that may influence the inflow. The Strength Reduction Factor (SRF) takes into account the relationship between the stress situation around an underground opening and the strength of the rock. Especially the effect of the stresses in the ground, if these are in a favorable direction or not. The value is decided at first by classifying which category the area to be mapped belongs to. These are “Weakness zones that intersect the underground opening which may or may not be able to transfer stresses in the surrounding rock mass”, “Competent rock with stability problems due to high stresses or lack of stresses”, “Squeezing rock with plastic deformation of incompetent rock under the influence of moderate or high rock stresses” and “Swelling rock; chemical swelling activity depending on the presence of water”. Subsequently, different stress situations related to each classification is described in detail with an associated value. The $\frac{J_w}{SRF}$ factor considers effects on the stress situation in total and how this affects the given Q-value in total.

The resulting Q-value may range from 0,001 to 1000, which in combination with the span of the tunnel or cavern and intended use will give a recommended extent of support measures. The measures range from spatial bolting, to the use of fiber-reinforced concrete, reinforced concrete arches to full line casting. The recommended support measure for each value may change in the future, as new techniques, lower cost for gentler blasting and new kinds of support measures all may influence the necessity of this(NGI, 2015).

2.2.2 Q_{TBM}-model

The Q_{TBM} model utilizes the parameters from the Q-system and expands on these by including factors that are descriptive for the TBM-rock interaction(Barton, 1999).
$Q_{TBM} = \frac{RQD_0}{J_n} \times \frac{J_r}{J_a} \times \frac{j_w}{SRF} \times \frac{SIGMA}{F^{10}/20^9} \times \frac{20}{CLI} \times \frac{q}{20} \times \frac{\sigma_\theta}{5}$
The three first factors are the same as for the Q-system, with the exception of the RQD-value. Here, RQD_o is used, which is the value in the tunneling direction and the worst-case direction one can choose to take the sample from. This is to include the effect of the angle of the plane of weaknesses on the penetration rate.

The SIGMA parameter is set to describe the rock mass strength by incorporating the Q-value (using RQD_o), density of the rock and the uniaxial compressive strength or the point load strength. Which of the two latter parameters are used depends on the foliation of the rocks. The F parameter is the net average cutter thrust in metric tons used through the zone and describes the load transferred from the cutter to the rock.

The Cutter Lifetime Index (CLI)(NTH, 1983) is included to take into account the effect of wear and cutter change on the progress of the boring. Cutter change and inspection is in most cases the main reason for time consumption other than boring in a TBM project, thus making it a key parameter in most cases for estimating progress. The quartz content q is also included in this respect, as this is a significant and common contributor to the cutter wear.

The σ_θ includes the induces biaxial stress situation at the tunnel face. The chipping process depends on a shear failure that occurs due to the difference in the highest and lowest stress in the rock induces respectively by the cutter and the weight of the rock. If the biaxial stress at the tunnel face is high, the chipping process will not happen in an ideal way.

The penetration rate is calculated with the use of the Q_{TBM}-value based the following equation:
$PR = 5 \times (Q_{TBM})^{-0.2} (m/h)$
The equation indicates that an increase in the value reduces the penetration rate, which is given in m/h. The advance rate, or the long-term performance, is given by multiplying the PR with the number of work hours to the power of a parameter m.

$AR \approx 5 \times (Q_{TBM})^{-0.2} \times T^m$

The m-parameter is calculated with the following equation:

$m = m_1 (\frac{D}{20})^{0.20} (\frac{20}{CLI})^{0.15} (\frac{q}{20})^{0.10} (\frac{n}{2})^{0.05}$

The m_1 will be a negative number decided by the Q-value, thus having the same mathematical function as for calculating PR, with a decreasing impact on AR at a higher number of work-hours. The TBM diameter D is included as a factor in the long term-performance, indicating that an increase in diameter will have a negative impact on the advance rate. A lower CLI and a higher quartz content both also contribute to larger decrease in AR. Porosity of the rock is also included as a factor, where an increase in porosity decreases the AR. The relationship between the AR and the PR over time will be as illustrated in figure 2.6.

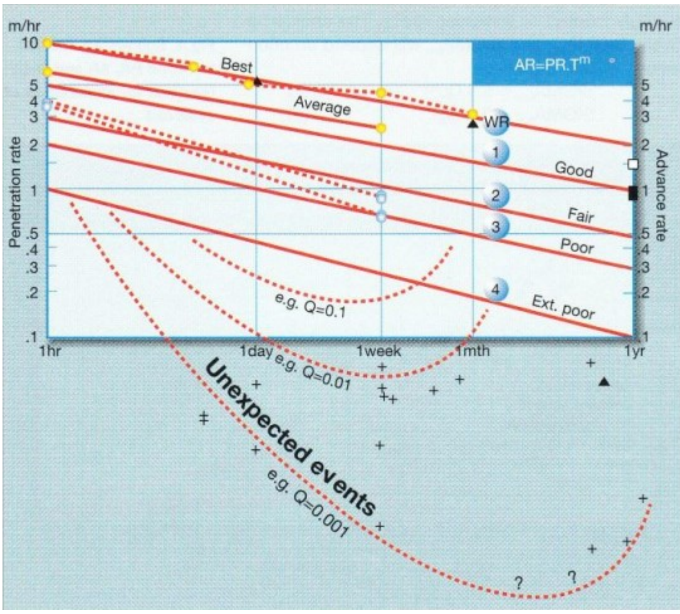

Figure 2.6: Connection between PR and AR over time (Barton, 1999)

As we can see from the plot, the AR after one full year of work-hours is expected to range between one tenth to one fifth of the PR in terms of m/h bored, with the exception of boring through extremely challenging conditions with low Q-value. Looking at the PR and AR as a function of the Q-value, we get the following relationship as shown in figure 2.7.

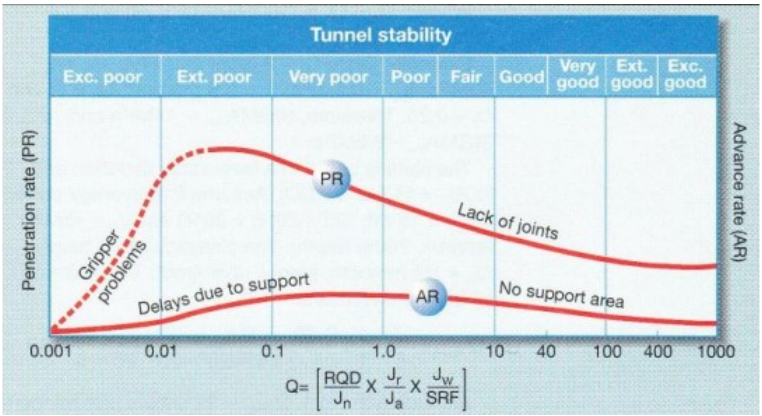

Figure 2.7: Connection between Q-Value and AR/PR (Barton, 1999)

Favorable rock mass conditions are expected at Q-values between 0,1 and 10, as the rock will neither require too much work with support measures or having too low penetration rate due to intact rock mass(Barton, 1999).

2.3 Geology at the Site

The geology along the tunnel route consists of two main stratums. The stratum observed from the intake and 9 kilometers into the tunnel consists of Granite Biotite. The rock-parameter test results can be seen in table 2.4

Table 2.4: Rock paremeters in Granite Biotite based on sample from chainage 7790

DRI^{TM}	CLI^{TM}	Density (g/cm^3)	UCS
N/A	6.0	N/A	192.6

The stratum from the 9-kilometer chainage to the surge chamber consists of a mix of Biotite Gneiss, Granite Gneiss and Gneiss. The rock-parameter test results can be seen in table 2.5

Table 2.5: Average rock paremeters in Granitic Gneiss based on five test sites in chainage 11600-12050 m

DRI^{TM}	CLI^{TM}	Density (g/cm^3)	UCS
39	5.5	2.66	191.9

The respective mineralogies for each site are given in table 2.6.

Table 2.6: Mineralogy for both sections tested in the tunnel

	Quartz	**Mica**	**Plagioclase**	**K-Feldspar**	**Other**
Chainage 7790 m	33%	2%	36%	26%	3%
Chainage 11855 m	33%	3%	34%	20%	10%

The top 2-10 meters of the rock is heavily weathered and has to be surpassed to access solid rock for tunneling. The tunnel depth ranges from 250 m to 800 m below ground, with exception from the intake area. Several weakness zones were predicted before boring started and has been surpassed, among them one fault zone. None of these were however of an extent that gave a Q-value of less than 1 according to what the project geologist registered, but were secured with the use of steel ring beams, rebar, rebar-grids and shotcrete. The crown was stable enough to advance one meter and then stop to install these. There was identified two fracture sets that appeared to be present in the entire tunnel. One major $N190^oE/85^o$ and one minor $N65^oE/85^o$.

2.4 Project Description

The tunnel is part of the Upper Kon Tum hydroelectric powerplant in Vietnam. The reservoir and intake are located in Dak Tang Village in Kon Plong County on the Dak Nghe River and the powerplant is located Dak Koi Village in Kon Ray County, with outlet into the Song Xa Lo River. The headrace tunnel is approximately 17.5 kilometers long with a 180-meter surge shaft to the powerplant at the end. The first 7.3 kilometers of the tunnel is excavated by conventional drill and blast while the last 10.5 kilometers are driven by use of a TBM. The total vertical drop throughout the tunnel system is around 680 meters, with an installed capacity of 220MW giving a planned annual production of approximately 1 TWh.

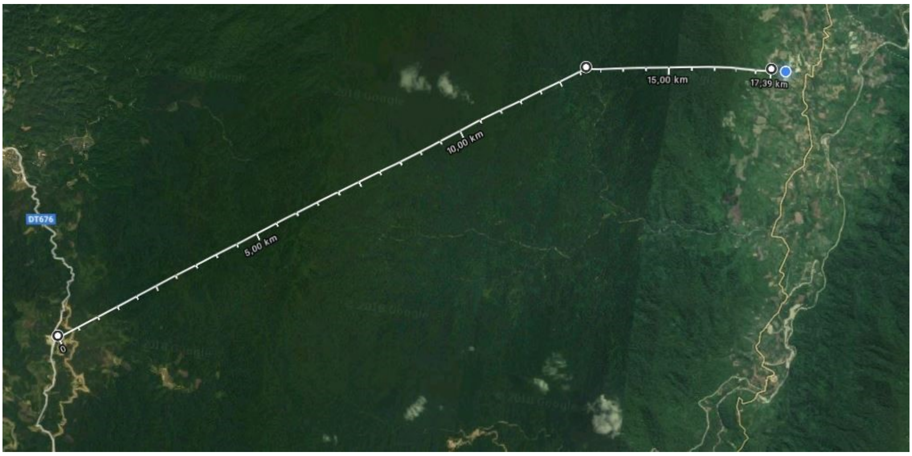

Figure 2.8: Map tunnel path

The tunnel is set to have a direction of N49^oE from the intake for about 13 kilometers, before it turns in a N91^oE direction towards the surge tunnel and the power station. The machine itself is a main beam (open gripper) of the Robbins 160 series. The TBM has a diameter of 4.5 meters with 30 432-millimeter (17 inch) cutters installed on the cutterhead. The stroke length is 1855 millimeters and the installed power installed power in the cutterhead is 1980 kW. The maximum thrust is 14800 kN, with a targeted operating gross thrust per cutter of 267 kN. There is equipment for the installation of steel ring beams, bolts and shotcrete behind cutterhead shield, for advancements through geologically unstable areas. The machine has been used at two tunnels prior to the ongoing one and has been refurbished by Robbins between each use. The original contractor for the TBM tunnel was “China Railway 18^{th} Bureau Group”, subcontracted from the Chinese hydropower contractor “Power Construction Corporation of China Huadong Engineering Corporation Limited (Powerchina Huadong)”. The sub-contractor was given the project based on supposedly extensive experience in TBM tunneling. However, the sub-contractor chose to replace its workers with operators from the 1^{st} and 6^{th} “… Bureau group” which lacked the experience to operate a TBM. The TBM was wrongfully operated which lead to a need for extensive repairs after barely 1 kilometer of tunneling. The TBM contractor was eventually fired from the project, which lead to a 2.5-year standstill before Robbins took over operations of the TBM in September 2016 in a joint venture with Vietnamese "Civil Contractor 47". Approximately 7.2 kilometers has been driven between September 2016 and February 2018.

2.5 State of the TBM

The TBM has been inside the tunnel since mid-2011 under hot and extremely humid conditions, which are unfavorable in regards of corrosion, especially of electronical components as sensors, connections and power supply to many systems. This affects the utilization of the TBM, as many of these systems fail when boring, with its vibrations and shaking, commenced again after the stand-still. Low-quality parts delivered by "Robbins China" has also lead to an increase in down-time on the project. Even though all this should not affect the penetration rate in theory, the operators try to minimize the stress on all the systems by operating more gently than what would have been the case with a new machine. The chief electrician also informed that the fluctuations of the frequency in the Vietnamese power grid are causing malfunctions on many electronical components, also contributing to more down-time. In total, the state of the technical systems required an increased focus on preventive maintenance and control, which affected the overall TBM operation.

2.6 Contractual Arrangements

There were two distinct contractual periods related to Robbins presence at the job site. The first period was from June 2016 to November 2017. It is unknown what incentive mechanisms were in place during this period, but the period was affected by a low TBM utilization and a cutter consumption way higher than what was anticipated by Robbins. From November 2017 and onwards, the project was under a new contractual scheme. The deadline for finishing the boring was set to January 2019, with a bonus-package for delivering on time. All the workers on the project would recieve full pay out 2018, thus having an incentive to finish as early as possible, as they could receive double pay by transferring to another project.

Chapter 3

Research Methodology

The collection of qualitative and quantitative research data for the thesis was done at the project site. The collection consists of mode data from completed sections of the tunnel and mapping by using the Q-system and the fracturing factor as described in the NTNU model, core samples to estimate the DRI and CLI, and net penetration and RPM tests with the subsequent mapping of these sections of these sections, including analyzing the chips from these tests. The testing is done in accordance with the descriptions of other completed tests. The purpose of the tests is to gain knowledge of the influence of cutter thrust and RPM to the net penetration and basic penetration of the TBM in the given geological conditions.

3.1 Field Work

3.1.1 Net Penetration Test

A penetration test is a procedure to evaluate the TBM performance in a given geological zone, to gain knowledge of how the rock responds to variations in gross average cutter thrust in terms of basic and net penetration rate. The testing should be done in combination with an engineering geological mapping of the same chainage and testing of the drillability by taking core samples. All other parameters should be kept as stable as possible, to avoid disturbing the output from the test. The penetration test should also ideally be executed in a geology with as small variations as possible for the same reason.

- Measurement of the cutterhead penetration over a given time at various thrust levels and constant RPM.
- Registration of the average cutterhead torque of each cutter load level
- Registration of other relevant data such as cutter wear state, whether the test is at the start, middle or end of the stroke, cutterhead vibration level, etc.
- Measurement or registration of the net penetration rate, cutter thrust level and cutterhead torque of the previous and following strokes.
- Collecting a complete chip sample for the penetration test and chip samples for the previous and following strokes.

The thrust increments should include at least four different values, typically 10 percentage points increments from 70% to 100%(Bruland, 1998). Cutter thrust up to 105% or 110% can also be included if applicable. Round numbers may be used to simplify the registration process. If the cutter thrust is given in cylinder pressure (bar), rounded numbers for pressures can be used instead and converted into kN/cutter later. The penetration should be measured over approximately 30 revolutions and at least 3 minutes for small diameter machines at each thrust level. The actual measurement should be taken at one of the thrust cylinders in addition to the computer measurement, as these are in direct contact with the cutterhead. This will ensure the correctness of the data. The applied torque for each thrust level is recorded by logging the average amperage to the cutterhead. The applied voltage must also be checked, in case it deviates from the rated voltage.

The applied gross average thrust was given directly by the computer in the operator cabin. The basic penetration rate was calculated by dividing the progress rate (mm/min) by the average RPM.

When the gross thrust with its associated basic penetration has been recorded, their log_{10} values are plotted and a regression line on the form $log_{10}(i_0) = A_R \times log_{10}(M_1) + B_R$, where the regression constants A_R and B_R are given. For the equation $i_0 = (\frac{M_t}{M_1})^b$, $b = A_R$ and $M_1 = 10^{\frac{-B_R}{A_R}}$ are calculated by setting $i_0 = 1$ and combining both equations.

The cutter wear state and if the test is started at the beginning or the end of a grip should also be recorded, as these are factors that may influence the test results.

The operator cabins computer displayed directly the gross thrust, torque, penetration rates in mm/rev and m/h, and the RPM. All parameters except the penetration rate were updated every second, leaving it up to the person recording the parameter to estimate an average over a given period of time. The penetration rate gave an average for a 30-second interval, thus making recording of performance easier. The distance was recorded by the use of a laser surveying system, thus giving accurate measurements(Bruland, 1998).

3.1.2 RPM Test

The collection of data and samples are performed the same way for RPM tests as for penetration test. However, for each thrust level, ideally five different cutterhead velocities are used and the associated performance is logged for each of these speeds. The speed should be varied in 10 percentage points increments, typically from 70% to 110%, to get a satisfying logging of data over a significant span of cutterhead velocities. A complete test procedure thus consists of five RPM tests for each of the five thrust levels, in total 25 tests to be performed in total(Eide, 2015)(Macias, 2016). Depending on the penetration rate, one might have to regrip during the test procedure, as each test should be approximately 30 revolutions and longer that 3 minutes. This has to be noted, as a fully extended thrust cylinder may behave differently in regard to vibrations and other factors(Eide, 2015).

3.1.3 Chip Analysis

The analysis of the chips broken off during boring may give valuable information of rock breaking mechanism and information regarding the influence of TBM operation parameters, if conducted during penetration tests. They may also provide information regarding the rock drillability(Bruland, 1998). When conducting the sampling, all data regarding the operation parameters should be registered. The section should also be subject to a detailed engineering geological back-mapping to ensure, to relate the rock mass properties to the chips and the general performance. The chips should be sampled as close to the cutterhead. The sample size should consist of 25-30 pieces, choosing the 20 largest of these to be analyzed. The analyzing includes measurement of the height, width and length of the sampled chips recorded like shown in table 3.1.

Table 3.1: Sample Table for Chip Analysis (Bruland, 1998)

Chip no.	Height h_{ch} (mm)	Width w_{ch} (mm)	Length l_{ch} (mm)
1	24	65	267
2	43	65	209
.....			
19	16	67	193
20	26	66	204
Mean size (mm)	28.5	66.6	217.2
Standard Deviation, mm	7.9	5.8	37.2
Standard Deviation, %	27.7	8.7	15.4

If sampling is done during a penetration test, the measurements can be included with the gross cutter thrust ad the basic penetration rate as shown in table 3.2.

Table 3.2: Sample Table for Chip Analysis (Bruland, 1998)

Thrust M_t (kN/cutter)	Penetration i_0 (mm/rev)	Average Chip Size (mm)			Chipping Frequency f_{ch} (rev^{-1})
		h_{ch}	w_{ch}	l_{ch}	
173.4	2.00	26.8	55.8	215.5	0.075
183.1	2.20	24.9	64.4	212.0	0.088
192.7	2.51	22.3	62.9	205.6	0.112
202.3	2.58	25.4	66.1	233.7	0.101
221.6	3.41	28.5	66.6	217.2	0.119

The chipping frequency is calculated by the equation $f_{ch} = \frac{1}{\frac{h_{ch}}{i_0}}$. The size of the chips is plotted towards the gross cutter thrust per cutter for analyzing as shown in figure 3.1. Trends can thus be seen if there is any influence of the cutter thrust.

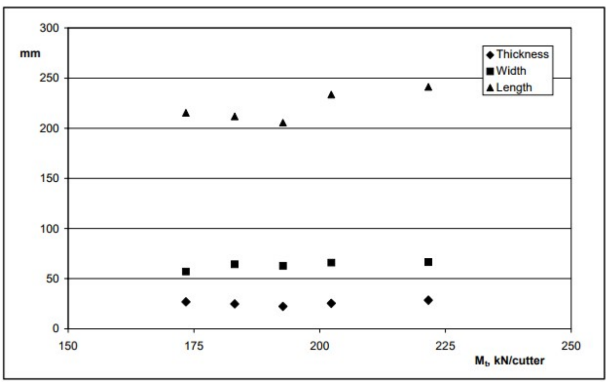

Figure 3.1: Sample Table for Chip Analysis (Bruland, 1998)

The shape factor of the chips can also provide information regarding the breaking process. The chip is described by the relationship $f_{hw} = \frac{h_{ch}}{w_{ch}}$ and is shown in figure 3.2.

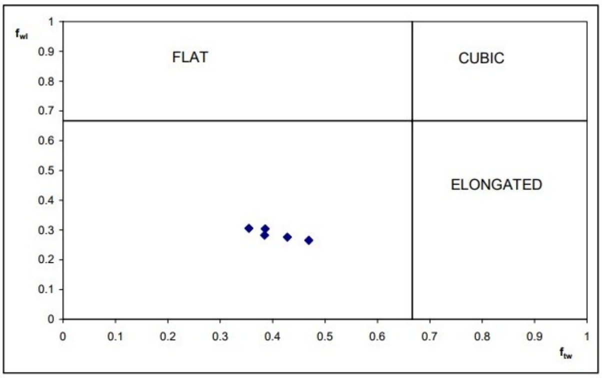

Figure 3.2: Sample Table for Chip Analysis (Bruland, 1998)

In addition to the volume of the chips broken off, the ration between depth of the kerf from passing measured from the thickest part of the chip and the thickness of the chip give valuable information regarding the chip breaking process(Bruland, 1998).

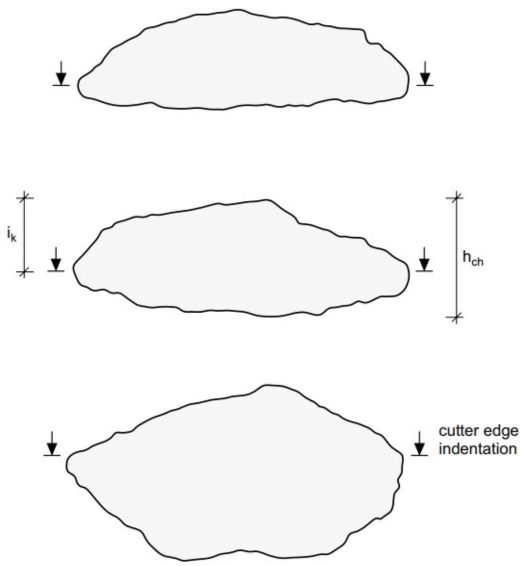

Figure 3.3: Example of chipping under different cutter loads (Bruland, 1998)

The upper example shows the least efficient breakage process, as the kerfs, or crushing zones from the disc cutters, are deeper indicating that a lot of energy goes is used for this purpose before a crack propagates to a neighboring kerf, relatively shallow compared to the kerf depth. The kerf depth factor is given by the equation $f_{kd} = \frac{i_k}{h_{ch}}$and will in this case give a high value.The bottom example shows a relatively deep propagation of the cracks to a neighboring kerf in comparison to the kerf depth. This gives a low kerf depth factor, indicating an efficient rock breaking process. Thus, analyzing both the size of the chips and the shape of the chips is necessary obtain a complete understanding of the efficiency of the rock breaking process.

The chipping frequency is defined as the inverse number of the number of revolutions necessary to penetrate the depth of the thickest chip, and is given by $f_{ch} = \frac{1}{\frac{h_{ch}}{i_0}}$. The chipping frequency will increase with an increase in gross cutter thrust, as the basic net penetration will increase, decreasing the denominator of the equation. In practical terms, the required number of revolutions will decrease to chip away one layer across the face of the tunnel(Bruland, 1998).

3.1.4 Engineering Geological Back-Mapping

The mapping was conducted in cooperation with another student, where NTNU fracture class, Q-value and RMR was registered during surveying. The mapping procedure was sought to be as systematical as possible to avoid to avoid systematic errors while mapping, In addition, each section was assessed in cooperation to minimize the influence of subjectivity that may have significant influence on the results(Seo et al., 2015). The chainage of the tunnel was marked every 5 meters by the TBM surveying team. There was no way to cross-reference these chainages to external reference points, so it had to be assumed that they were correct and that they coincided with the TBM data recordings when comparing mapping to performance.

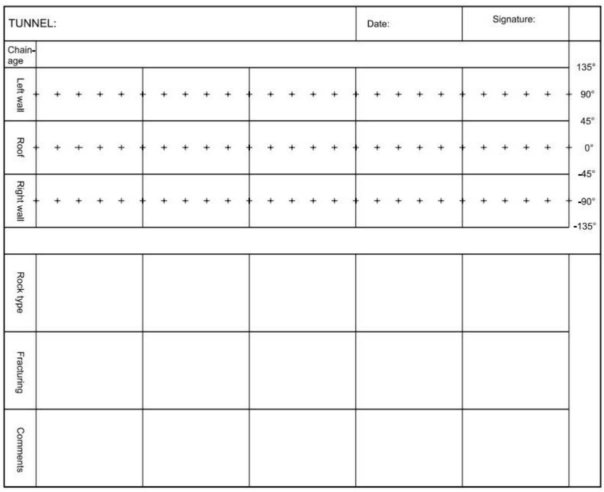

Figure 3.4: Sheet for use when mapping (Bruland, 1998)

When performing the mapping, 5 meters sections were assessed at a time using the sheet seen in figure 3.4. Fractures for the NTNU model were drawn on a standardized mapping sheet, and strike and dip were taken on a regular basis. For the Q-value, the joint volume for RQD were assessed per meter for left wall, crown and right wall, to get an as accurate approximation as possible. Only joint volume and not directional RQD was registered. There were for the most part some breakage around the fractures, thus exposing surfaces for assessment of the number of sets, filling and roughness. Each section was mapped in cooperation and deviance in assessments was discussed to ensure an as equal assessment between each section.

The mapping was conducted from the right side of the tunnel relative to drilling direction. The roof was to a large extent covered by the ventilation channel, thus making it harder to follow fractures around the circumference. The conveyer belt was located on the left side of the tunnel and was not approached during operation due to safety reasons. The humidity in the tunnel caused a lot of dust to stick to the walls, making especially some sections challenging to give an equal assessment to other not covered by a thick layer. All area mapped, with exception of parts of chainage 8700-8750, were in the Biotite Granite formation. The fracturing characteristics of the rock-mass did not appear to change between the geological formations.

Performance data from the TBM computer provided information regarding the operating parameters needed for input to the estimation models and the actual performance in these areas. The data was given as an average per excavation meter. There was an issue regarding the wrong average being calculated when the cutter head was pulled back for inspection or cutter change. Most of this data is assumed sorted out by not including data for excavation meters where the torque is less than 100 kNm. It was assessed that this would not sort out sections with intact rock, which would also give low numbers for the cutter head torque. This removal of data came in addition to lacking data on a general basis, causing some sections to have up to 50% data loss. Most sections had more than 80% intact data.

3.2 Laboratory Testing

Due to the small amount of test material available for laboratory testing, only Uniaxial Compressive Strength, Mineralogy and Cutter Lifetime Index test were performed. UCS was chosen as a confirmation parameter whether or not the rock was equal throughout the tunnel or not. The UCS is also included in the Q_{TBM} model. The remaining material from preparing the UCS-test sample could be used for CLI and Mineralogy(Bieniawski & Bernede, 1979)(Dahl, Bruland, Jakobsen, Nilsen, & Grøv, 2012).

3.2.1 Uniaxial Compressive Strength

The UCS test is a common test procedure for deciding the strength of a certain material. A cylindrical sample with the length of 2-3 times the diameter is compressed along the length axis, and the strength is given by the peak tension before the sample fails. The procedure can be seen in figure 3.5.

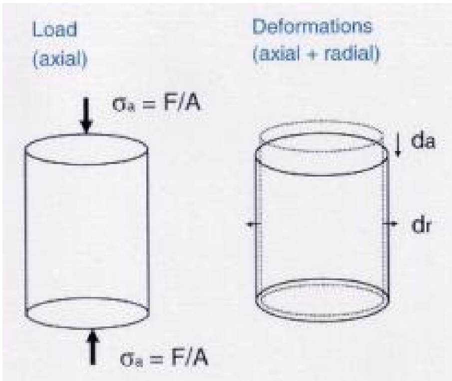

Figure 3.5: Test procedure for determining compressive strength (Jacobssen, 2004)

The test is conducted at SINTEF in accordance with the ISRM standard(Bieniawski & Bernede, 1979). Five samples were tested and the failure documented. Different modes of failure are typically connected to different peak tensions. Low peak tensions are connected to simple shear failures while higher peak tensions are connected to multiple shear and multiple fracturing. Shear and fracturing failures are typically connected to that failures happen along discontinuities in the sample, while failure along the axis are connected to the intact rock(Szwedzicki, 2007). Illustration can be seen in figure 3.6.

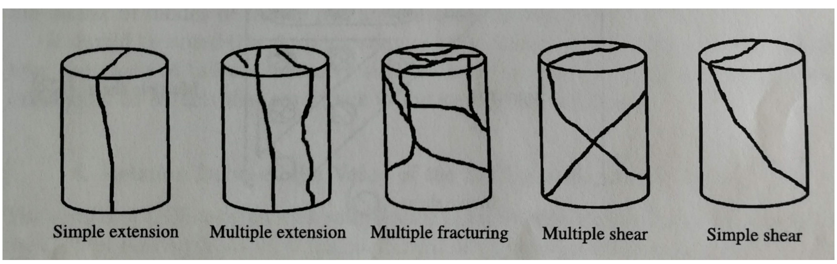

Figure 3.6: Modes of Failure (Szwedzicki, 2007)

3.2.2 Cutter Lifetime Index

The CLI is given by combining the results from Sievers J-testing and the Abrasion Value for cutter steel (AVS). Sievers J is given by the indentation in 1/10 mm after 200 revolutions by the testing procedure seen in figure 3.7.

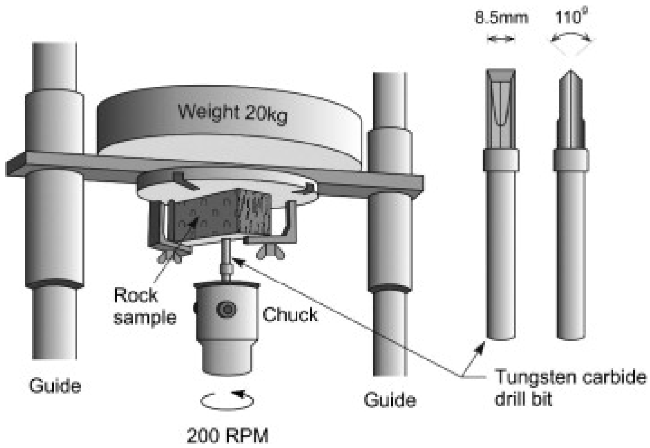

Figure 3.7: Test procedure for determining the Sievers' J-value (Zare & Bruland, 2013)

The AVS value is given by the weight loss in mg after 100 revolutions of a standardized cutter steel sample. The test apparatus can be seen in figure 3.8.

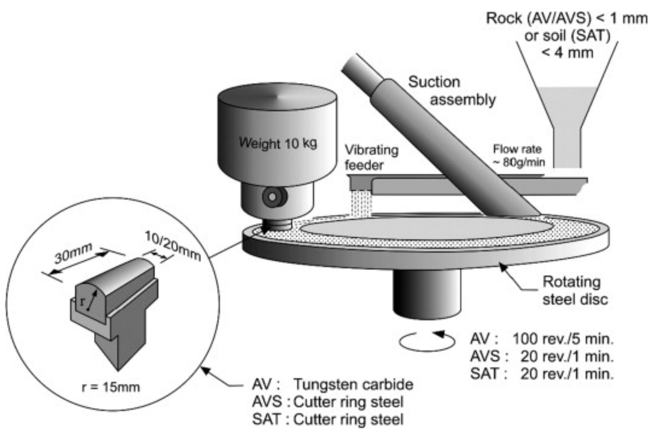

Figure 3.8: Test procedure for determining the AVS-value (Zare & Bruland, 2013)

The CLI value is then calculated by the use of the equation $CLI = 13.84 \times \frac{SJ}{AVS}^{0.3847}$. A lower value suggests a lower cutter lifetime in boring hours, which is solely a function (in the NTNU model) of the cutter diameter and the CLI value(NTH, 1983).

3.3 Data analysis

3.3.1 TBM Utilization and Cutter Consumption

The TBM performance was recorded on a daily basis by Robbins to assess overall performance. The parameters recorded was chainages bored per day, number of boring hours and other time consumption, cutters and bucket lips changed, and average machine performance for RPM, thrust pressure, motor ampere and propel flow.

- Chainages bored
- Boring hours
- All time-consuming activities
- Cutters and bucket-lips changed
- RPM
- Cutterhead motor amperage
- Cylinger pressure (thrust)
- Propel flow

The time consumption was categorized as seen in table 3.3

Table 3.3: All time-consuming activities and the category of which they were allocated.

Symbol:	Activity:	
T_b	BORING	PROBE DRILLING
T_r	GRIPPING	SHOTCRETE
T_c	CUTTER INSP/CHANGE	TUNNEL SERVICE
$T_{Maintenance}$	TOWING	ELEC.BREAKDOWN
Not included	MAINTENANCE	MECH.BREAKDOWN
	SHIFT CHANGE/MEALS	HYD.BREAKDOWN
	TUNNEL CONVEYOR	INDUSTRY WATER SUPPLY
	TBM CONVEYORS	H.V POWER SUPPLY
	OVERLAND CONVEYOR	INVERT RAILS INSTALL
	BELT EXTENSION	SURVEY
	GROUND SUPPORT	LOCO/TRANSPORT
		OTHER

The categorization used on the project made comparison to all time-consuming activities other than boring, re-gripping and cutter change possible. The categories were however not defined detailed enough to distribute all time consumption between T_{TBM}, T_{Back}, $T_{Miscellaneous}$ and T_{length}, so they were all summarized into the category $T_{Maintenance}$. All categories not related to the boring process, with the exception of "Ground Support", "Probe Drilling" and "Others" were included in $T_{Maintenance}$. Utilization was calculated on the basis of the total time consumption excluding "Ground support", "Probe Drilling" and "Others". Time consumption for belt extension was summarized for all months and divided per meter tunnel to more correctly include this parameter from a utilization point of view. All performance parameters of interest were calculated on a monthly basis to assess the overall performance.

3.4 Literature Review

A literature review was conducted in the prelude of writing the project thesis, which had the main focus of TBM as a production system and the NTNU model, that this thesis is partly based on. This has been supplemented by additional literature regarding Q_{TBM} and the rock-breaking process. Reports from independent engineering entities regarding the project has been of key interest, as these puts the quantitative results into context.

Chapter 4

Results and Discussion

4.1 Review of Robbins Geological Back-Mapping

Production data and results from the engineering geological mapping from the first part of the tunnel through gneiss was made available for review by Robbins. The assessment of the mapping was solely based on comparing TBM performance with what the NTNU-model and QTBM-model predictions, as it was not possible to map these sections again due to safety constraints.

4.1.1 NTNU Fracture Class Results

The project registered the NTNU fracture class continuously as the TBM was advancing. The classification definitions used was in accordance with version 6, as this was the current version when the project commenced. The plot of all actual net penetration rates to their respective fracture class can be seen in figure 4.1.

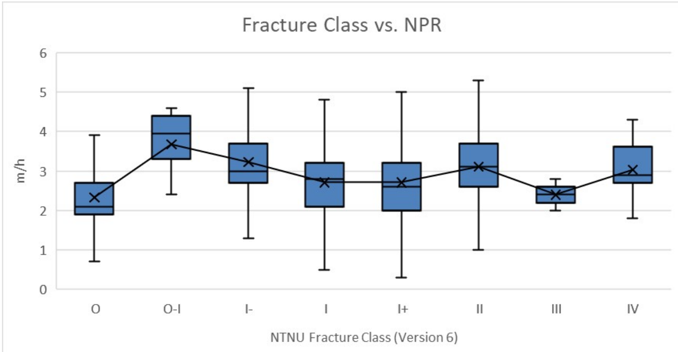

Figure 4.1: Relationship between fracture class given by Robbins and actual NPR for the given areas. Box contains 50% of observations, line inside box shows median value, “x” shows average value, whiskers shows total spread in data.

Looking at all the data combined, there appear to be no correlation between registered fracture class and estimated NPR. There was however a large spread in the length observed per class, which may cause some uncertainty in the averages given.

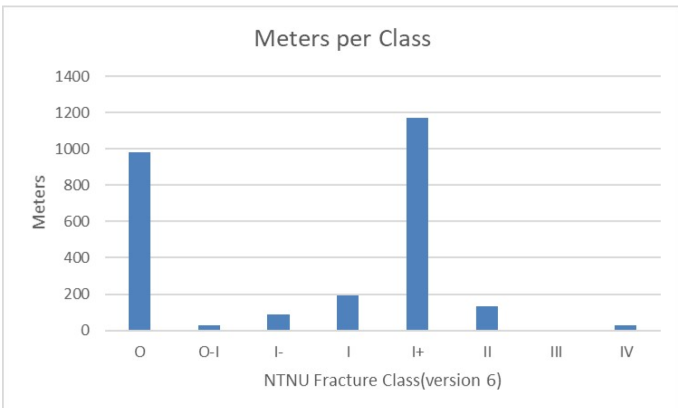

Figure 4.2: Frequency of observations per class.

As seen in figure 4.2, the majority of all observations either class O or I+, while there are still enough observations for statistical significance for class I and II as well. The small observed difference in average NPR for class O and I+, which corresponds to average spacing indefinite-240 cm and 60-30 cm, suggest that there is little to no connection between the way the NTNU fracture class has been estimated and actual tunneling performance. Looking at sections internally per fracture class, one gets a better understanding of the spread of the data.

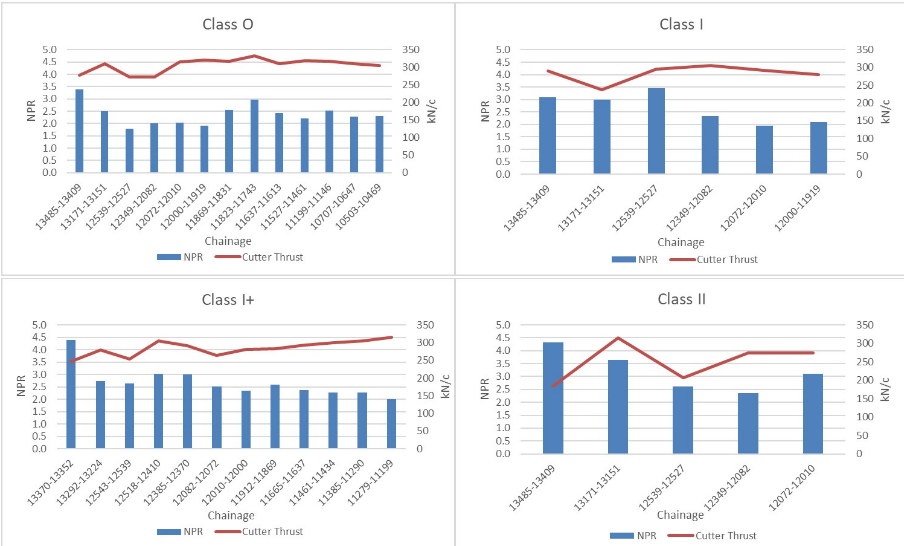

Figure 4.3: Resulting NPR and cutter thrust for different sections per fracture class.

As can be seen in figure 4.3, there is no apparent connection between fracture class and TBM performance when looking at several sections per class. The NPR varies significantly despite that the cutter thrust is approximately constant and that the given classification is the same. The extremes in the NPR may be parts of several weakness zones or marked single joints that exists in the tunnel, but this cannot explain all deviations from the expected performances per classification. There is in addition no correlation for cutter thrust between different fracture classes, of which most are in the interval 275-300 kN/c. There is an exception for class II, which has a slightly lower average cutter thrust. One would have observed way higher NPRs at these cutter thrust levels if the classification were in accordance with version 6(Bruland, 1998).

4.1.2 NTNU Fracture Class Discussion

The mapping itself, and indirectly the classification given, was not done in accordance with the procedure described in version 6. The fracture class had been assessed by all fractures regardless of direction, as one would when estimating the joint volume for RQD, and given an average distance based on this. An example of fractures included can be seen in figure 4.4.

Figure 4.4: Picture showing all fractures included when evaluating the fracture class by Robbins.

This procedure gave a higher fracture class than what a correct assessment after the NTNU-model would have given. In addition, the influence of different sets and their angle towards the tunnel axis is not included. Looking at the results in figure 4.2, the distribution suggests the fracture class to be approximately a discrete event being either class O or I+. Given that all observations are in the same geological stratum and that sections with the identified classes are intertwined, and not separated in each end of the formation, the actual distribution between classes should have been something closer to a normal or triangular distribution. This supports the assertion that the way of registering the fracture class is not done correctly.

Solely based on comparing the NPR and the observed cutter thrust levels with the estimations given by the NTNU-model, all observations of NPR lower than 2.0 m/h and the majority of all observations lower than 2.5 m/h appears to actually be class O. The remaining observations, with a few exceptions, does not appear to belong to the class listed to. According to site personnel, the intention behind registering the NTNU-fracture class was not to obtain data for penetration rate estimations for later projects, but rather as a tool to describe how the rock-mass appeared after it has been passed. The surface left behind after the TBM has passed by does show a significant difference despite belonging to the same fracture class (when fracture class is estimated correctly). As can be seen in figure 4.5, there is a clear difference in how fractured the rock-mass appear to be despite both being class O, thus the need for a system to describe this. No samples from this area was taken to investigate if the intact rock parameters were any different.

Figure 4.5: Difference in tunnel surface under equal rock-mass conditions

The value however of registering this way can be questioned, as there is no apparent application of the data. The extent of safety measures installed is detirmined by the Q-system and it has no value as an description of the rock-mass for use in the NTNU-model. It only provides confusion if someone in the aftermath of the project completion tries to utilize the data to estimate NPRs for a coming project. The use of terminology should thereby be reconsidered, as it is not the actual fracture class that is being registered.

4.1.3 Q_{TBM} Results

The project registered Q-value continuously as the TBM advanced. The value was registered in various stretches, ranging from 2 m to 30 m, depending what was deemed necessary. The minor stretches assessed was where there were significant changes of the rock stability and rock support had to be installed, while longer stretches with even rock conditions were given an average value.

The extra parameters that required to calculated Q_{TBM} can be seen in table 4.1. The values given are based on information provided by site personnel and available laboratory testing results from SINTEF. The UCS was set to a somewhat lower value as the samples were collected in locations with intact rock that probably represented the higher end of the scale of the rock-mass quality. The net cutter thrust was based on an assesment given by one of the site geologists.

Table 4.1: Input parameters for Q_{TBM}

Parameter	**Value**
$\sigma_c(MPa)$	170
$F(metrictons)$	27/varying
CLI^{TM}	5.55
Q (%)	27
$\sigma_\rho(MPa)$	15

The cutter thrust is plotted for both actual and target cutter thrust values, to highlight the difference one would estimate in advance of a project versus the prediction using the actual cutter thrust. As the TBM performance data was registered per excavation meter, it was decided to estimate the average performance for both the Q_{TBM} penetration rate and the actual penetration rate by using a moving average of 20 meters to normalize the data.

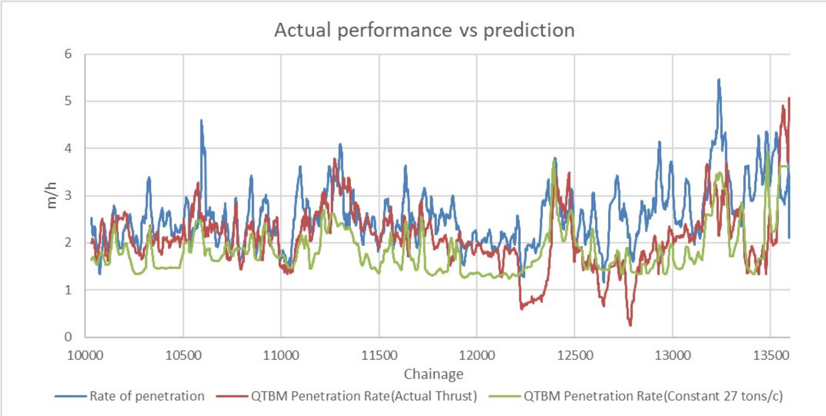

Figure 4.6: Actual performance compared to Q_{TBM} prediction for both constant and actual thrust.

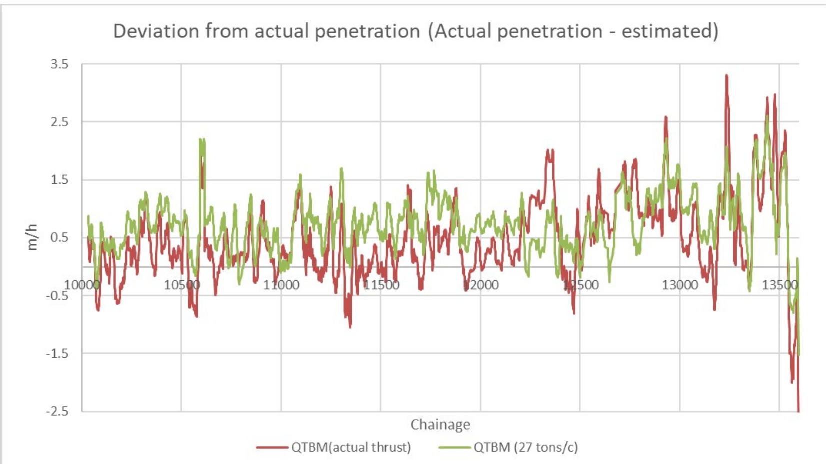

Figure 4.7: Actual penetration rate subtracted the estimated deviation rate.

As seen in figure 4.6, the overall estimation of the penetration rate performance appears to correlate with both ways of calculating Q_{TBM}, giving a conservative estimate. The deviation is smaller when using the actual cutter thrust, which can be seen in figure 4.7. The deviation using actual thrust is on average between chainage 10000-12500 0.31 m/h, while the deviation is higher between chainage 12500-13600 is 0.90 m/h. The estimate using constant cutter thrust is on average more conservative, thus giving higher deviations of 0.67 and 0.95 respectively for the chainages. The reason for the difference in between the two chainage intervals is unknown, and there was no additional information available that could help explain the difference.

4.1.4 Q_{TBM} Discussion

The estimations seem to have given a good prediction, especially when considering that the sigma-theta value is assumed constant even though this changes with the weight of the overlying bedrock. As the cutter thrust is highly influential on the estimated Q_{TBM}-value, comparing this value to the actual and estimated penetration rates for both constant and actual thrust will highlight the influence of the other parameters assumed constant. It also shows the difference between the results one would get estimating the performance before tunneling starts and after tunneling is completed, by using respectively the assumed necessary thrust level and the actual thrust.

Q_{TBM} using constant thrust

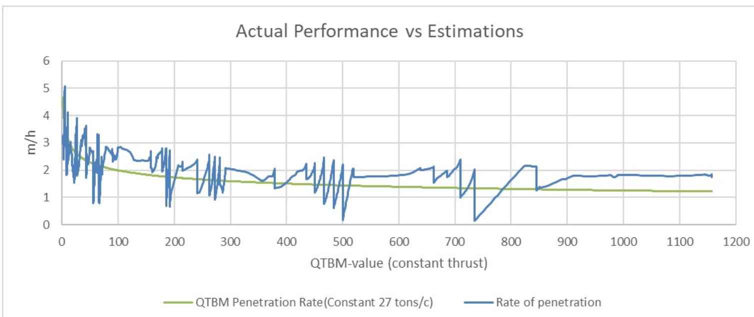

Figure 4.8: Actual NPR and estimated NPR as a function of the Q_{TBM}-value.

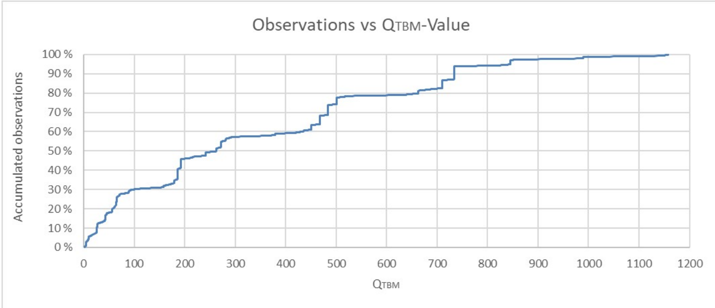

Figure 4.9: Accumulation of observations per Q_{TBM}-value.

Comparing the actual NPR to the corresponding Q_{TBM}-value gives us an understanding for which conditions the estimations are most accurate. Figure 4.8 shows a clear correlation between actual performance and the estimation based on a constant cutter thrust, and clearly shows that the estimation on average is conservative. A high variation in the NPR appears to coincide with the intervals of the Q_{TBM}-value with the most observations, as seen in figure 4.9. One would have expected this to not be the case, especially for higher values of Q_{TBM}, as the rock-mass conditions should be stable in these areas. The influence of the actual cutter thrust, of which the actual cutter thrust highly depends on, is likely the cause of this variation.

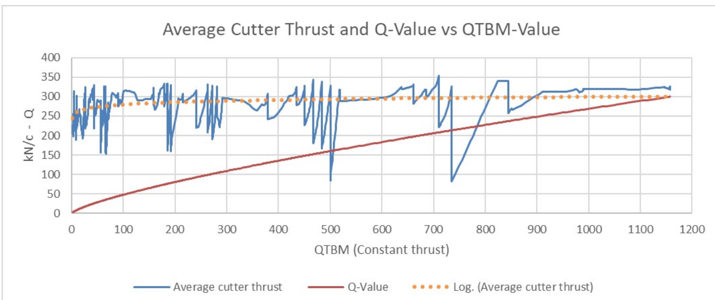

Figure 4.10: Cutter thrust and Q-value as a function of the Q_{TBM}-value.

The Q-value, which describes the rock mass in the Q_{TBM}-model and can be seen in figure 4.10, is clearly the most influential parameter when using constant thrust as input. There is a correlation between the actual cutter thrust and the Q-value, which appears to be strongest at values of Q less than 50.

Q_{TBM} using actual thrust

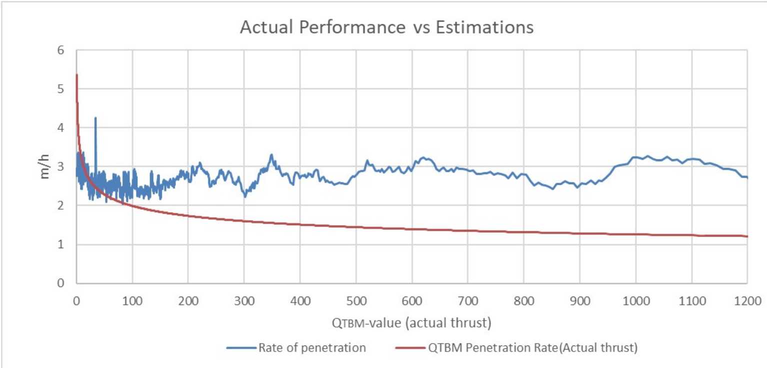

Figure 4.11: Actual NPR and estimated NPR as a function of the Q_{TBM}-value.

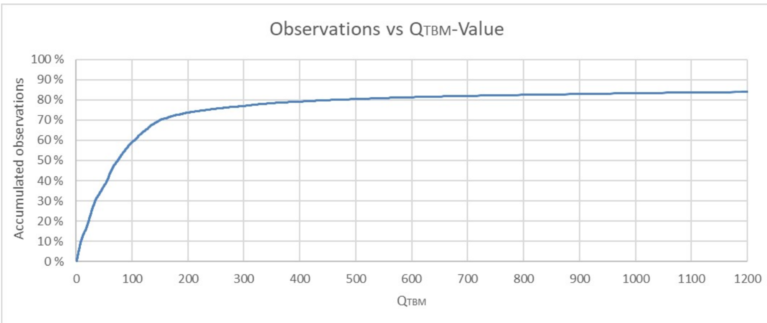

Figure 4.12: Accumulation of observations per Q_{TBM}-value.

Figure 4.11 shows the actual NPR and the estimated penetration rate per Q_{TBM}-value. The deviation is clearly smaller for a value less than 100, whereof also 60% of the observation are done, as can be seen in figure 4.12. The absence of the volatility in the plotted actual NPR which is seen in figure 4.8 indicates that the cutter thrust was the main reason for the volatility. For values of Q_{TBM} more than 100, there is a trend of increasing deviation with increase of the Q_{TBM}-value. An interesting observation is that Q_{TBM} is capable of correctly predicting the NPR for 60% of the observations, while it underestimates this for Q_{TBM}-values between 100-1200, despite the fact that the actual NPR lies within the same interval as for values between 0-100.

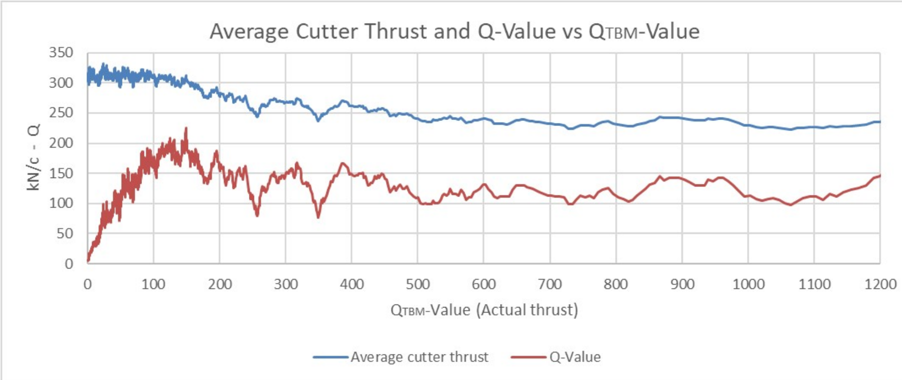

Figure 4.13: Cutter thrust and Q-value as a function of the Q_{TBM}-value.

As the cutter thrust is highly influential on the Q_{TBM}-value, the connection between this and the deviation to the actual NPR for the shows an interesting relationship. As seen in figures 4.11 and 4.13, the deviation is to a large degree driven by the cutter thrust as soon as this drops below 300 kN per cutter. This indicates that the estimated NPR given by Q_{TBM} is too dependent on the applied cutter thrust, and that the parameters set to describe the rock-mass, the Q-value, is not influential enough to compensate the estimated value when the cutter thrust decreases. This is supported by the fact that the actual NPR stays approximately the same while the thrust is reduced by 50-60 kN per cutter and the Q-Value drops from a peak at 200 to a place between 100-150. Based on the comparison of back-mapping in section 4.3, the estimation of the Q-value appears to be equally assessed as the independent study, thus minimizing the probability that any variation is caused by wrongful mapping.

When comparing figure 4.11 and 4.13, is that for Q_{TBM}-values from 0 to 100-120, the Q-value is the major influential parameter on the estimated NPR. For Q_{TBM}-values over 100-120, the cutter thrust appears to be the most influential. Another observation is that the relationship between the Q-value, which describes the rock-mass, and the cutter thrust is not behaving as one would expect. An increase in the fracturing of the rock-mass is in most cases coupled with a decrease in the cutter thrust to reduce the vibrations in the TBM. Despite the decrease, one would still have a higher NPR than for the opposite situation. This is the situation for the

Q_{TBM}-Values higher than 100-120, but not below these values. For Q_{TBM} less than 100-120, we see that the cutter thrust behaves independently of the rock-mass conditions with an approximate constant value, even though the estimated NPR in this area behaves as expected.

Summarized, using the actual cutter thrust as an input to the Q_{TBM}-model appears to give a correct rendering of the actual NPR when one operates close to the targeted cutter thrust. However, when the cutter thrust is lowered to reduce the vibrations caused by better boring conditions, the estimated NPR is too highly influenced by this to give a correct estimation. The Q-value appears in some cases to be a good parameter for boreability, while in others not, which of course is a fundamental weakness when using it to estimate the TBM performance. Given the low number of meters mapped and analyzed, and given the assumptions made when analyzing, any significant conclusion should not be drawn from the results.

4.2 Engineering Geological Back-Mapping

As the fracture class registration by The Robbins Company could not be used, only mapping from the field study at the site could be used for the NTNU-model. Based on strike and dip measurements taken over the entire length of the mapped area, there was identified two distinctive sets that were used as input to the model. The predominant set J1 was identified with a strike of $N190^oE$ and a dip of 85^o, and minor set J2 was identified to $N65^oE$ and a dip of 85^o (nomenclature for strike and dip in accordance with project standard). The sets respectively gave an angle towards the tunnel axis of 49^o and 6^o, seen in figure 4.14.

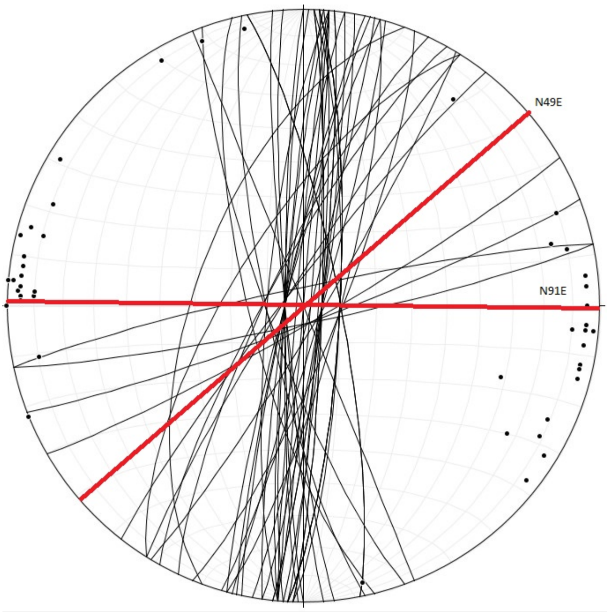

Figure 4.14: Stereonet Plot from Measurements of Strike and Dip, with Tunneling Directions shown in red

4.2.1 NTNU-Model Results

The rock mass fracturing factor was estimated for each 5-meter interval before it was accumulated to 50-meter section of which the TBM performance would be predicted. This was in accordance with previous studies of correctly assessing the factor. As the spacing distance was sporadic, while having a low degree of fracturing, the fracture spacing was estimated using the "n+1" approach. This is an alternative approach to achieve a more accurate description of the spacing in these conditions(Seo et al., 2015). The resulting k_{s-avg} can be seen in figure 4.15.

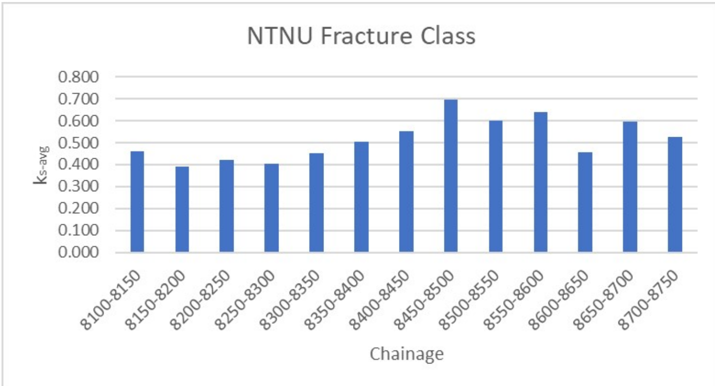

Figure 4.15: Results from Mapping In Accordance With NTNU-Model Fracture Class

Verifying the mapping can only be done by assessing the TBM performance in terms of cutter thrust and NPR. This can be seen in figure 4.16.

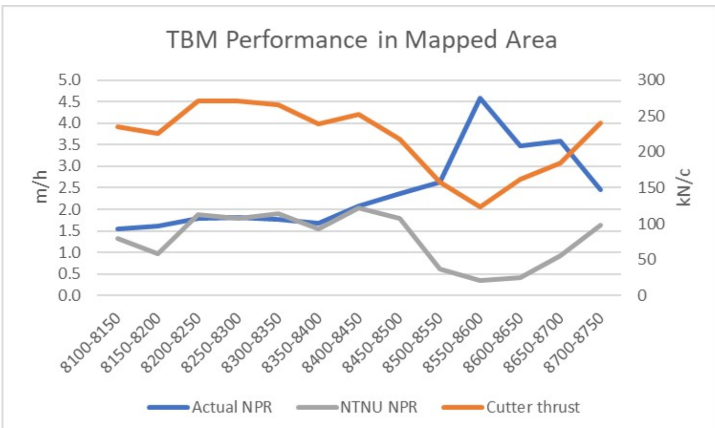

Figure 4.16: NPR performance and estimation compared to the actual cutter thrust utilized.

The estimated NPR is quite close to the actual NPR, with the exception of chainages 8500-8700. As the NTNU model is highly dependent on applied cutter thrust, the estimated NPR is significantly lower than actual NPR despite the fact that the fracturing factor is higher in these chainages.

4.2.2 NTNU-Model Discussion

There are several factors which may have affected the assessment during the mapping. Firstly, both students had no experience in mapping in TBM-driven tunnels, thus having no prior knowledge to help in their judgment. Taking into account the learning curve for this kind of work, one should assume a higher variation in the assessment first sections mapped than in the latter. It is nearly impossible to avoid the influence of subjectivity, especially considering lack of previous experience in mapping. There was also an issue with TBM performance data, which led to a considerable loss in percentage of the number of meters with recorded. Firstly, the data logging did not record all performance data for all excavation meters. The reason for this is unclear. Secondly, every time the cutter head was pulled back for inspection or cutter change, this was logged as high NPR at low cutter thrusts. This data could be identified by looking at the cutter head torque, which would have a low value when this happened. It was decided that registered excavation meter with a torque of less than 100 kNm was sorted out. Some of the 50-meter sections lacked performance data for up to 50% of the excavation meters, so the average for the entire section is based on the remaining data for this section. Due to the similarity of the rock mass conditions in chainage 8100-8500, it was deemed to be a reasonable approach for assessing performance where data was lacking.

As seen in figure 4.16, there is a clear deviation between actual and estimated performances in the 50-meter sections in chainages 8500-8700. The combination of a low cutter thrust and a very high NPR would normally indicate that this is a heavily fractured area, with a k_{s-avg} of more than five. This was obviously not the case, as the majority of the area was self-supporting without any rock support installed. The chainages in question were close to an expected weakness zone and lies in the transition zone between the Biotite Granite formation and the Granitic Gneiss, which makes it possible that this has affected the rock parameters. This assumption is supported by what can be interpreted from the interaction between the rock-mass and the TBM. The surface left behind after the TBM had passed were in and close to chainages 8500-8700 was uneven and the fracturing induced by the TBM appeared to have spread outside the cross-section of the tunnel. This can be seen in figure 4.17. Pieces of rock could easily be loosened by hand in these areas. The opposite was the case outside this area, where the TBM left a smooth and completely intact surface except for around the fracture sets, as seen in figure 4.17.

Figure 4.17: NPR performance and estimation compared to the actual cutter thrust utilized.

Results from chainages 8100-8500 and 8700-8750 does give an accurate estimation of the actual NPR. Given that the uncertainty of the parameters in the NTNU model, as the DRI and the subjectivity of the interpretation of the rock mass conditions, there is normally a low probability of getting an estimating this close to the actual conditions. The interpretation of the rock mass conditions, where it has to be evaluated if fractures have contributed to the breaking process of the TBM, was in this case more obvious than in other tunnels. The TBM induces a 30-45^o fracture from the tunneling axis towards the fracture plane, leaving in most cases a highly distinct fracture around the circumference of the tunnel. Other fractures could be identified in the tunnel, but it was in most cases evident that these did not contribute to the breaking process due to the lack of rock breaking onto the fracture plane.

4.2.3 Q_{TBM} Results

The Q-value was estimated on a 5-meter basis and averaged over 50-meter sections. The result can be seen in figure 4.18. Only the areas not covered in rock support were mapped, thus giving a higher average Q-value than what was the case for some sections.

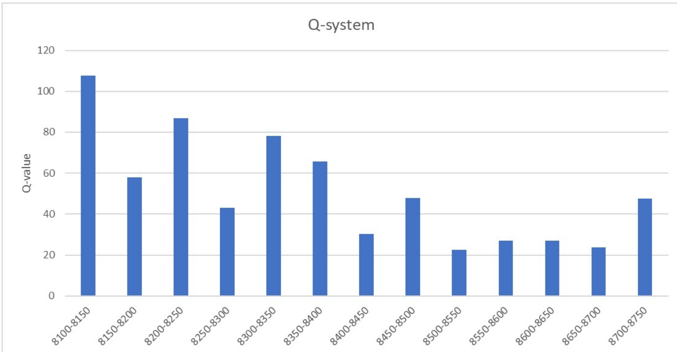

Figure 4.18: Results from Q-system mapping per 50-meter section

As all parameters except cutter thrust and Q-value are kept constant, the results of the mapping can be verified by comparing the estimated NPR with the actual NPR.

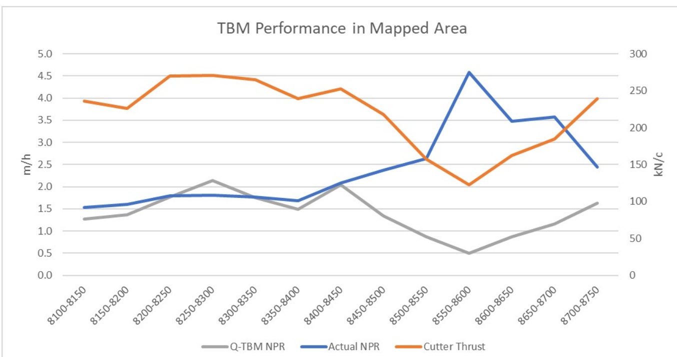

Figure 4.19: Q_{TBM} NPR estimation compared to actual performance and cutter thrust

As seen in figure 4.19, the deviation from the actual performance is negligible in chainage 8100-8450. As the estimated NPR is clearly correlating with the cutter thrust, the drop in this value leads to a reduction in the estimated NPR.

4.2.4 Q_{TBM} Discussion

The geological back-mapping indicates relatively good rock-mass conditions, whereof the average values of are well within the area where no rock support is needed. The weakness zone in chainage 8550-8700 is to some degree identified by the system, but equal Q-values for this area is also seen in chainage 8400-8450 and 8500-8550. The original purpose of the Q-value is to assess the necessity for rock support measures, while it acts as the rock-mass parameter for the Q_{TBM} model. The Q_{TBM} model gives accurate estimations for chainage 8100-8400, while it deviates significantly for the remaining sections. The cutter thrust does seem to be too influential on the model, making significant reductions in the Q-value almost without influence. This is especially clear in the section from 8450-8750, where it does not seem like that the large reduction in Q-value has any effect on the estimated NPR. As for the NTNU-model, it is possible that that the rock parameters changes when passing through the weakness zone and that the uniaxial compressive strength is in fact lower in this area.

As for uncertainties in the mapping, there was an issue with dust sticking to the wall. This dust layer was in some case so thick that the evaluation of the RQD for calculating the Q-value was challenging to do equally between sections. The influence of subjectivity may also have contributed to the uncertainty. The mapping was also done for two different purposes, both for assessing the necessity of rock support and for the use in the Q_{TBM} model. For the first case, the conservative approach would be to interpret the rock-mass so that the Q-value was lower. For the latter case, a lower Q-value would give an increased estimated NPR, thus making a high estimate of the Q-value the more conservative case. Still, it is doubtful that this has had any actual influence on the estimated NPR in this case given the discussed influence of cutter thrust. It may be more influential in cases where the cutter thrust is more equal and when the mapper has an economic incentive one way or the other. The estimation of the RQD-value was also based on using joint volume and not oriented RQD, which is also a source of error.

4.2.5 Comparison of Estimation Models Discussion

Both estimation models give relatively good estimations in chainage 8100-8400. In chainage 8400-8500, the Q_{TBM} model deviates significantly while the NTNU-model gives a lower estimate than what the actual NPR is. The estimates are off in the weakness zone, while getting closer for chainage 8700-8750. The results do however indicate that the Q_{TBM} model is too influenced by the applied cutter thrust. This can be seen in chainage 8450-8550, where the Q_{TBM} estimation falls significantly despite the fact that boring conditions are getting better. The NTNU model does also underestimate the NPR in this area, but the estimation is clearly not only influenced by the reduction in cutter thrust. The reason for this can be seen in how the influence of cutter thrust is modeled as a function of the rock-mass conditions. The NTNU model with the equation $i_o = \frac{M_t}{M_1}^b$, where M_1 and b are functions of k_{ekv}. The Q_{TBM}-model however assumes that all rock-mass conditions respond equally to the cutter thrust by the $\frac{1}{\frac{F^{10}}{20^9}}$ fraction. This clearly leads to an underestimation in cases where the TBM operator reduces cutter thrust when hitting rock with good boreability, as the parameter describing the rock mass does not sufficiently compensate for this. This highlights the problem of using a “single equation model”, as these are built under the assumption that real-world problems behave mathematically ideally. It may be the case in narrowly defined intervals of given parameters but is clearly not the case when facing actual problems. Still, both models appear to give good estimates under conditions where the rock-mass has a low fracturing, as was the case in this tunnel. Empirical models can more easily be adapted to take into account different parameters in a better way, thus giving a more flexible tool to estimate the performance.

4.3 Comparison of Back-Mapping

4.3.1 Results

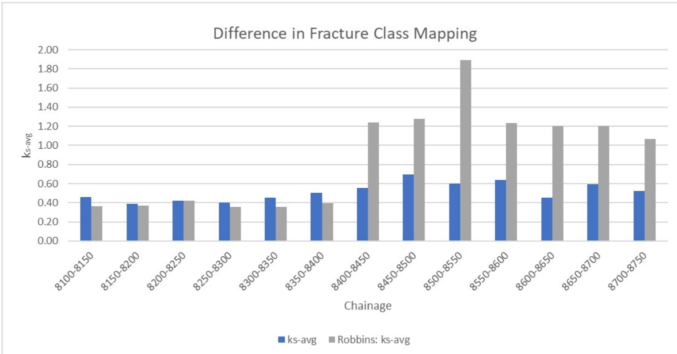

Figure 4.20: Resulting fracture class from Robbins mapping versus independent mapping

Figure 4.20 shows the results from the mapping done during the visit at site versus the mapping done by Robbins. The assessment is approximately equal in chainage 8100-8400, while it deviates significantly in chainage 8400-8750.

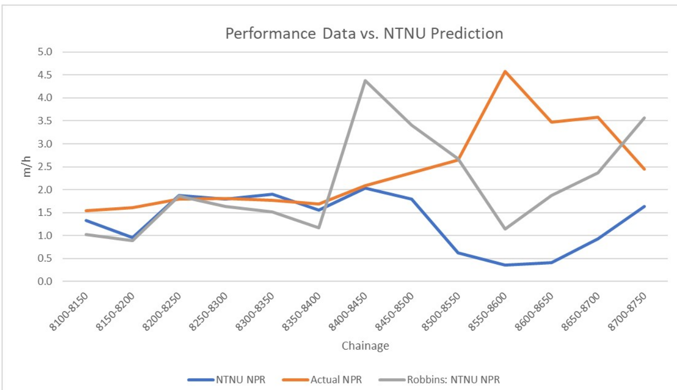

Figure 4.21: NTNU-model NPR estimations for Robbins and independent fracture class mapping

Using the results from figure 4.20 as input in the NTNU-model, we get the estimated NPR seen in figure 4.21. The results show an approximately equal estimation in chainage 8100-8400. The Robbins mapping indicate a highly fractured rock-mass for chainage 8400-8500, which does not lie in the weakness zone. The corresponding estimated NPR for this area is thereby significantly higher than the actual NPR.

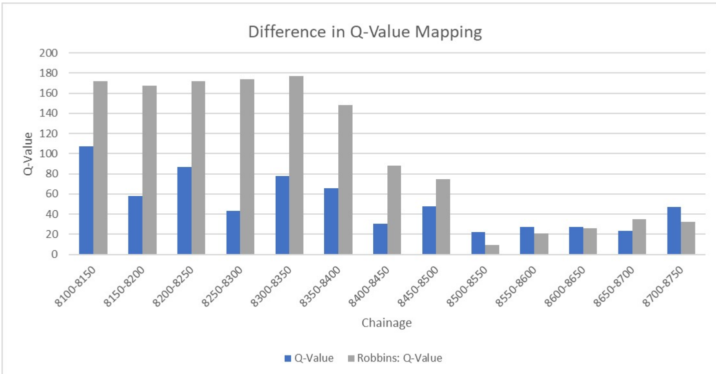

Figure 4.22: Resulting Q-system value from Robbins mapping versus independent mapping

The results from mapping with the Q-system can be seen in figure X. The mapping conducted by Robbins shows a higher value given in areas with good rock-mass conditions compared to the student mapping, while the value is more equal for the other areas.

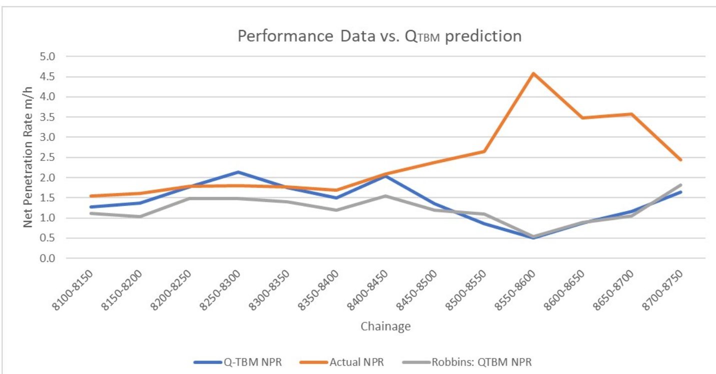

Figure 4.23: Q_{TBM} NPR estimations for Robbins and independent fracture class mapping

The resulting estimated NPR can be seen in figure 4.23. In chainage 8100-8400, the estimations are approximately equal, while the Robbins estimating result is more conservative than the other estimation. Both estimations given an inaccurate estimate in chainage 8400-8750, where the estimates are too low.

4.3.2 Discussion

As previously discussed in section 4.1, the mapping procedure for fracture class by Robbins was not conducted in accordance with the NTNU-model. In chainage 8100-8400 where the rock was intact and the surface was smooth, the resulting k_{s-avg} was equal to that of the actual k_{s-avg}. In chainage 8400-8500, where the rock is still intact while the surface is not smooth, the given k_{s-avg} is way higher than what the actual conditions suggest. This is reflected by deviation between the actual performance and the estimation by Robbins. The fact that the approximation for 8500-8550 is accurate should thus not be treated as anything other than a coincidence, where portions of the section lies within the weakness zone. This contributes to increase the average NPR, thus coinciding with estimation based on an incorrect mapping. This is supported by the fact that chainage 8700-8750, whereof most of the section is outside the weakness zone, has a way too high estimation when normal thrust levels are applied.

The mapping using the Q-system shows in general a significantly higher Q-value in chainage 8100-8500, while the evaluation is more equal in chainage 8500-8750. The mapping conducted by Robbins was expected to give the lowest value, as the purpose was to assess the necessity for rock support during boring. One would thereby expect a conservative estimation, especially when considering that the persons mapping are in the tunnel at all times. This was though only the case for chainage 8100-8500 which consisted mostly of intact rock, thus making the exact value trivial. For chainage 8500-8750, the Robbins value is for most sections lower or equal to the other mapping.

It has to be taken into account that the mapping could not be conducted for areas covered with shotcrete, thus giving a higher Q-value for the mapping conducted during the field surveys. The big difference in Q-value does however not significantly affect the estimated NPR. The estimation by Robbins is slightly more conservative on average, while it is not as affected by the difference in Q-value as one would expect. Continuing, it is clearly shown for chainage 8450-8550 that the necessity for rock support and the boreability are not coupled, as the model is not able to correctly estimate the NPR in this case.

Summarized, the results from the comparison shows that the way the fracture class has been mapped by Robbins is not in accordance with the NTNU model. This is in accordance with the analysis of mapping done for the rest of the tunnel. Mapping using the Q-system appears to have been done correctly, as the methodology by the site geologists is in accordance with the instructions provided by the Q-system Handbook(NGI, 2015).

4.4 Laboratory Testing of Core Samples

The testing of the samples was conducted at the SINTEF laboratories 15.05.2018. The results can be seen in table 4.2.

4.4.1 Results

Table 4.2: Results from UCS testing

Test nr.	**Compressive Strength** (MPa)
1-1	200.1
1-2	175.5
1-3	214.5
1-4	179.4
1-5	193.3
Average	192.6
St. Deviation	15.8

All failures were of the “multiple shear” category. Pictures from pre- and post-testing can be seen in figure 4.24.

Figure 4.24: Pictures before and after testing showing failure mode

The AVS and Sievers' J tests gave the results seen in table 4.3.

Table 4.3: Results from AVS and Sievers' J testing

Test nr.	**Sievers' J** (1/10 mm)	**AVS** (mg)
1	1.8	21
2	4.4	22
3	1.6	
4	1.9	
Average	2.4	21.5
St. Deviation	1.30	0.71

The resulting values correspond to a CLI-value of 6.0. The mineralogy of the rock at the from the same sample can be seen in table 4.4.

Table 4.4: Resulting mineral composition from sample at chainage 7790 m

Quartz	**Mica**	**Plagioclase**	**K-Feldspar**	**Pyroxene**	**Chlorite**
33%	2%	36%	26%	2%	1%

The results indicate a similar composition in both the Granite Biotite and the Granitic Gneiss, where Quartz, Plagioclase and K-Feldspar are the predominant minerals.

4.4.2 Discussion

The results from the laboratory testing suggests that the boring conditions are on average equal throughout the tunnel. Despite the fact that the other test was taken 7 km in distance away and in another geological formation, the difference in test results were insignificant. The mineral composition in both areas does not change significantly. This supports the assumptions done in the analysis that the tunnel, from an engineering point of view, can be treated as if the boring conditions are equal throughout the tunnel.

4.5 Test Procedures

As chip sampling was to be conducted during the test, the test procedure had to be adapted. The sampling had to be performed at the tunnel conveyor behind the back-up as the muck would be within reach at this point. Communication was thus an issue, since the sampler could not know at which test increment was currently being utilized. The solution was that the sampler would call the operator cabin from a cooling-cabin located at the middle of the back-up and instruct when each test thrust level increment could start. The sampling would then occur approximately 5 minutes after the call, to ensure that the operator had time to stabilize the thrust and that muck from previous strokes would not be on the conveyor. When the sampling was finished, the next call would be made to set a new thrust force level. At the same time, one person sat with the TBM operator in the cabin monitoring the performance data. The data would be assessed after three minutes of operating at each thrust level, to ensure that the breaking process was not influenced by previous thrust force levels. The computer onboard displayed gross thrust force, penetration per minute, cutterhead torque and cutterhead RPM directly, and the value noted would be what was considered an average reading for the period after three minutes of operation. As the collection of the samples involved bending over the tunnel conveyor with the possibility of getting stuck, it was decided to not perform any additional chip samplings in the tunnel due to safety concerns after test 1. Tests 2-4 were done in accordance with the NTNU-model descriptions.

4.5.1 Penetration Test 1

The first penetration test was conducted 08.03.2018 from chainage 8091.5 to 8089.6. The preceding stroke to the test showed stable conditions with an NPR around 3.2 m/h with thrust per cutter at 280 kN and RPM at 10.3. The test started with a stroke extension of 150 mm.
The test itself was not conducted under ideal conditions, as what was interpreted by the operator as potential weak rock-mass conditions were encountered. This lead to the RPM being reduced significantly due to concerns from the operator of damaging the cutters for test 3 and 4, which is not in accordance with the test procedure. The TBM did also shut down between test 1 and 2, and 2 and 3, as there was a malfunction with a conveyor sensor. Test 5 was supposed to be conducted for a thrust force at 5000 kN, but was changed due to low amounts of usable muck for already for test 4. Results can be seen in table 4.5.

Table 4.5: Test Results from Penetration Test 1

Tunnel: Upper Kon Tum		Chainage: 8091.460-8089.638		Date: 08.03.2018	
Test	Average Applied Thrust M_t (kN)	Average Applied Torque T_t (kNm)	Average penetration after 3 minutes (mm/min)	RPM	Penetration rate i_0 (mm/rev)
1	8400	700	50	10	5.00
2	7500	530	38	9.6	3.96
3	7000	400	22.3	7.5	2.97
4	6000	200	10.7	7.4	1.45
5	8400	800	62	10.2	6.08
RPM: Varying		Comments: Previous stroke and sucseeding stroke equal, Mt=8200, RPM=10.2, i=52mm/min, NPR 3.2 m/h. Newly inspected and changed cutters. Stroke extension at 150 mm at start.			

As the test procedure requires a constant RPM during the testing, only results for 1,2 and 4 were used for calculating M_1 and b. This gave the log-log plot for gross thrust per cutter and penetration rate as seen in figure 4.25.

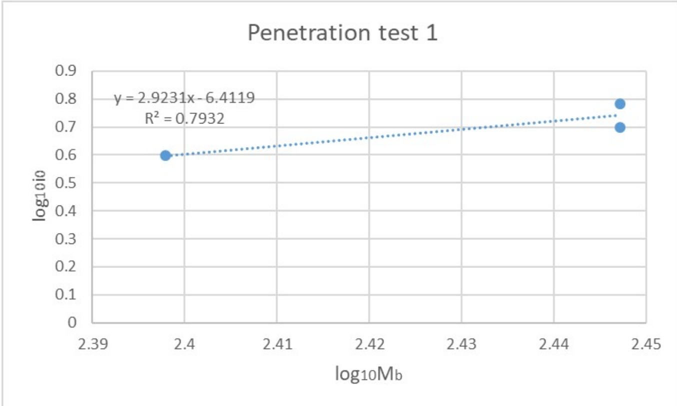

Figure 4.25: Loglog-plot of Test 1 Results

The curve gave the following values seen in table 4.6 for the penetration coefficients.

Table 4.6: Penetration Coefficients from Test 1

	$\mathbf{M_1}$ **(kN)**	**b**
Result	156	2.92

4.5.2 Penetration Test 2

The second penetration test was conducted 17.03.2018 from chainage 7948 to 7949. The preceding and succeeding strokes to the test showed stable conditions with an NPR around 1.8 m/h with thrust per cutter at 287 kN and RPM at 10.3. The test started with a stroke extension of 400 mm. The test was conducted under ideal conditions as no fractures were hit during the procedure. The results did however show somewhat varying results for penetration rate at different thrust intervals. The results from the second penetration test can be seen in table 4.7.

Table 4.7: Test Results from Penetration Test 2

Tunnel: Upper Kon Tum		Chainage: 7948-7949		Date: 17.03.2018	
Test	Average Applied Thrust M_t (kN)	Average Applied Torque T_t (kNm)	Average penetration after 3 minutes (mm/min)	RPM	Penetration rate i_0 (mm/rev)
1	8800	500	27.83	10.3	2.70
2	8000	420	22.63	10.3	2.20
3	7400	370	20.8	10.3	2.02
4	6800	320	14.37	10.3	1.40
5	7400	520	32.3	10.3	3.14
6	8200	800	53.52	10.3	5.20
RPM: 10.3		Comments: Previous stroke and sucseeding stroke equal, Mt=8600, RPM=10.3, i=34mm/minh. Medium to high cutter wear. Stroke extension at 400 mm at start.			

As test was conducted under even rock-mass conditions and there were no problems regarding keeping the RPM constant. The log-log plot of the test results can be seen in figure 4.26.

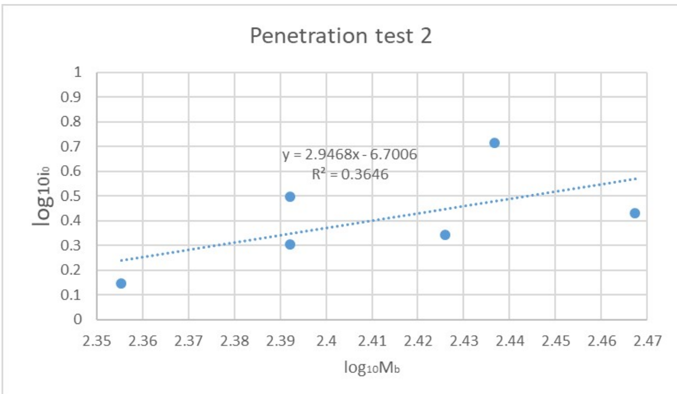

Figure 4.26: Loglog-plot of Test 2 Results

The regression line gave the penetration parameters seen in table 4.8.

Table 4.8: Penetration Coefficients from Test 2

	$\mathbf{M_1}$ **(kN)**	**b**
Result	188	2.95

4.5.3 Penetration Test 3

The third penetration test was conducted 22.03.2018 from chainage 7883 to 7884. The preceding stroke showed a NPR of 2.6 m/h and the succeeding stroke to the test showed stable conditions with an NPR around 1.8 m/h. The test started with a stroke extension of 500 mm.

The test was assumed to be under ideal conditions as the NPR was stable for the first 500 mm of the stroke. After the third increment, the cutter thrust became hard to stabilize and the results from the two next increments were excluded from the analysis. The final increment was however stable and was included in the analysis. The RPM was held somewhat stable, even though the RPM was reduced for increment number four. The results from the third penetration test can be seen in table 4.9.

Table 4.9: Test Results from Penetration Test 3

Tunnel: Upper Kon Tum		Chainage: 7948-7949		Date: 17.03.2018	
Test	Average Applied Thrust M_t (kN)	Average Applied Torque T_t (kNm)	Average penetration after 3 minutes (mm/min)	RPM	Penetration rate i_0 (mm/rev)
1	8800	500	27.83	10.3	2.70
2	8000	420	22.63	10.3	2.20
3	7400	370	20.8	10.3	2.02
4	6800	320	14.37	10.3	1.40
5	7400	520	32.3	10.3	3.14
6	8200	800	53.52	10.3	5.20
RPM: 10.3		Comments: Previous stroke and sucseeding stroke equal, Mt=8600, RPM=10.3, i=34mm/minh. Medium to high cutter wear. Stroke extension at 400 mm at start.			

As test was conducted under even rock-mass conditions and there were no problems regarding keeping the RPM constant. The log-log plot of the test results can be seen in figure 4.27.

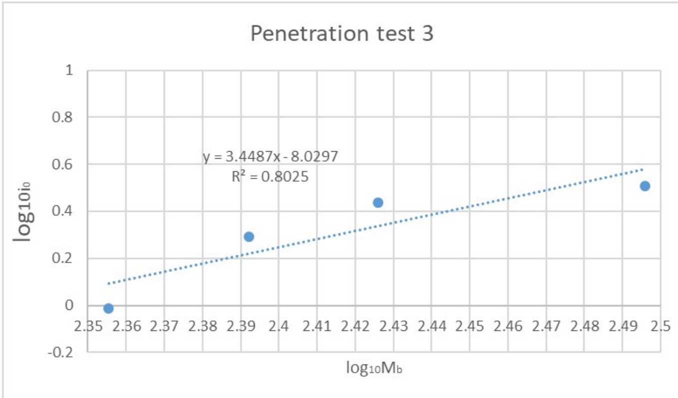

Figure 4.27: Loglog-plot of Test 3 Results

The regression line gave the penetration parameters seen in table 4.10.

Table 4.10: Penetration Coefficients from Test 3

	M_1(kN)	**b**
Result	213	3.45

4.5.4 Penetration Test 4

The fourth penetration test was conducted 22.03.2018 from chainage 7948 to 7949. The preceding and succeeding strokes to the test showed stable conditions with an NPR around 1.8 m/h with thrust per cutter at 287 kN and RPM at 10.3. The test started with a stroke extension of 400 mm. The test was conducted under ideal conditions as no fractures were hit during the procedure. There was no unexpected variation during the test. The results from the fourth penetration test can be seen in table 4.11.

Table 4.11: Test Results from Penetration Test 4

Tunnel: Upper Kon Tum		Chainage: 7881-7882		Date: 22.03.2018	
Test	**Average Applied Thrust M_t (kN)**	**Average Applied Torque T_t (kNm)**	**Average penetration after 3 minutes (mm/min)**	**RPM**	**Penetration rate i_0 (mm/rev)**
1	9600	530	33.6	10.3	3.26
2	8800	430	26.6	10.3	2.58
3	8000	350	18	10.3	1.75
4	7200	260	11.3	10.3	1.10
RPM: 10.3		Comments: Previous stroke and sucseeding stroke equal, M_t=8600, RPM=10.3, NPR=1.8 m/h. Low Cutter wear. Test start at 400mm stroke extension.			

The log-log plot of the test results can be seen in figure 4.28.

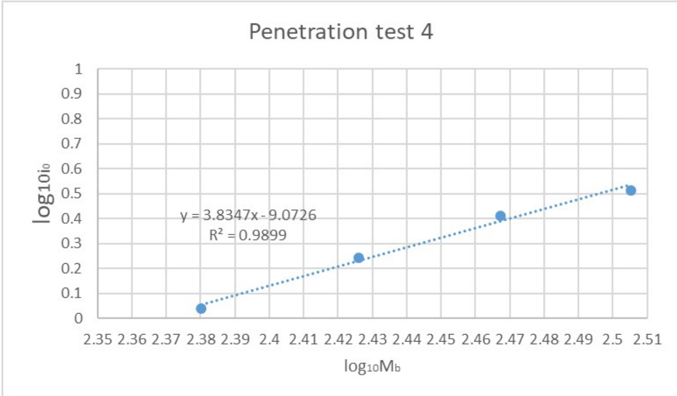

Figure 4.28: Loglog-plot of Test 4 Results

The regression line gave the penetration parameters seen in table 4.12.

Table 4.12: Penetration Coefficients from Test 4

	$\mathbf{M_1}$ **(kN)**	**b**
Result	232	3.83

4.5.5 Summary

The chainages where the tests were conducted were mapped and the rock-mass classified in accordance with the NTNU-model. The results can be seen in table 4.13.

Table 4.13: Summary of results from penetration tests

Test Number	M_1 (kN/c)	b	k_s	k_{ekv}
1	156	2.92	0.44	0.37
2	188	2.95	0.36	0.30
3	213	3.45	0.36	0.30
4	232	3.83	0.36	0.30

Tests 2, 3 and 4 were conducted in solid rock-mass with no sign of fracturing. Test 1 was conducted through a fracture. A fracturing factor was given on the best judgement based on general knowledge of the TBM responded to different rock-mass conditions. The results were plotted with the NTNU 2016 test results in figures 4.29 and 4.30.

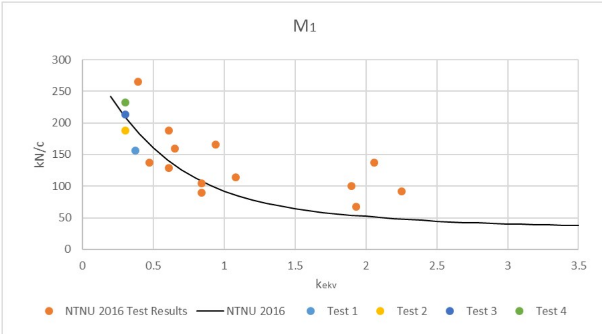

Figure 4.29: Results from Kontum tests plotted with M_1 results used in version 7 of the NTNU model

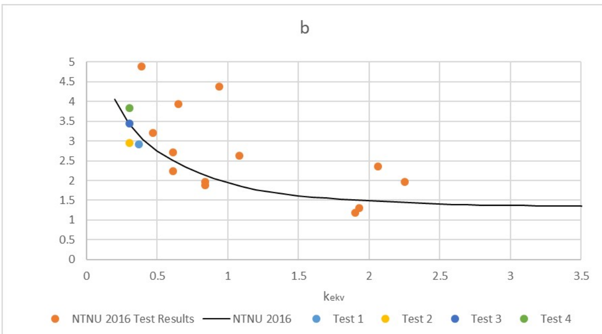

Figure 4.30: Results from Kontum tests plotted with b results used in version 7 of the NTNU model

All tests results have a lower k_{ekv}-value than that of any of the results included in version 7 of the NTNU model. They appear to be in accordance with the graph giving the actual value used for modelling NPR.

4.5.6 Discussion

Despite two out of four were not conducted under ideal conditions, all test still gave results supporting the conclusions from the tests conducted for version 7 of the NTNU model. A source of uncertainty was that the DRI used to calculate k_{ekv} was based on samples taken 7 kilometers away from the testing area. The tests were also conducted in another geological formation, consisting of Granite Biotite instead of Granitic Gneiss. The different formations are assumed to behave equally, as previously discussed. This assertion is supported by the fact that the UCS test results were equal in both formations. But as the results form the basis of the penetration curve, they should still be sought to be as accurate as possible.

4.6 Chip Analysis

4.6.1 PT1 Results

The average ship size for different cutter thrust increments can be seen in figure 4.31. The results indicate a slight increase in height and length as a function of the increase in cutter thrust.

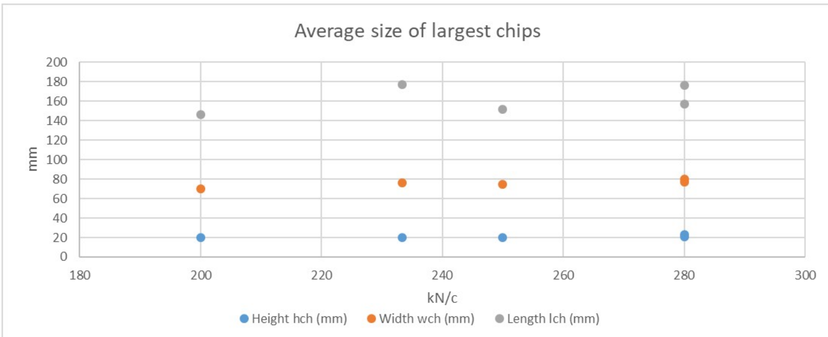

Figure 4.31: Average Size of the Largest Chips

The ship shape factor can be seen in figure 4.32. The resulting location in the diagram suggests a normal shape with a tilting towards the “flat” area.

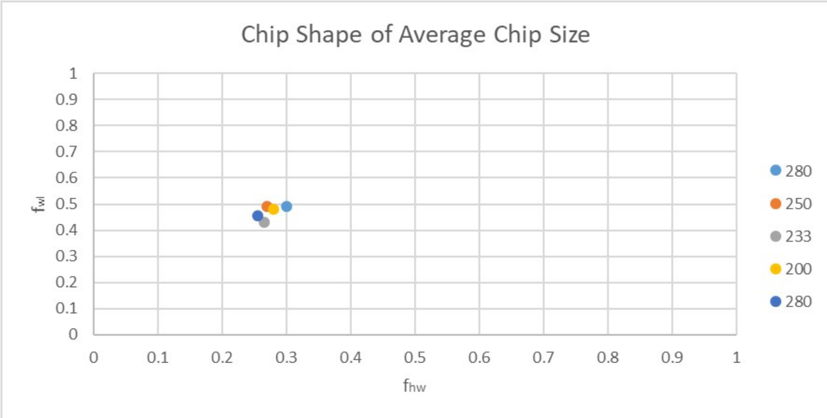

Figure 4.32: Chip Shape of Average Chip Size

The resulting chip cubic volume can be seen in figure 4.33. There is an indication of an increased cubic volume with in increase in cutter thrust.

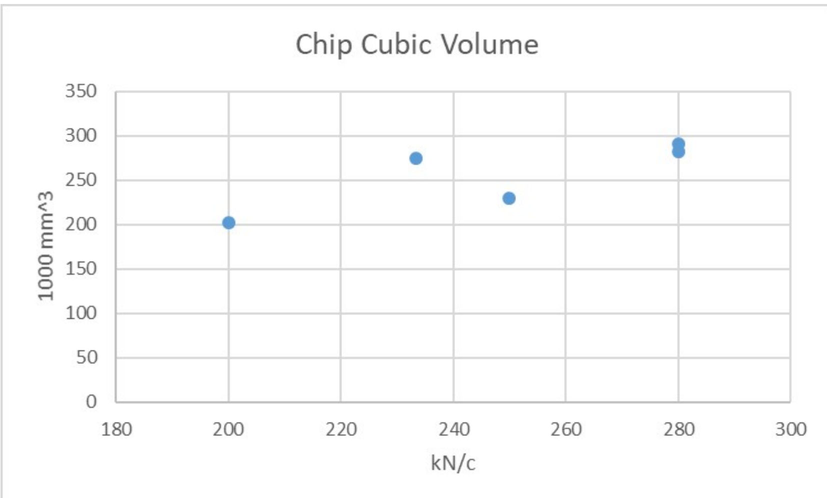

Figure 4.33: Cubic Chip Volume per Cutter Thrust Level

The indentation can be seen in figure 4.34. The results in total are very stochastic. Given that the two middle values for cutter thrust were under a higher fractured condition, the remaining results indicate a lower indentation with an increase in cutter thrust. This suggests a more efficient rock-breaking process.

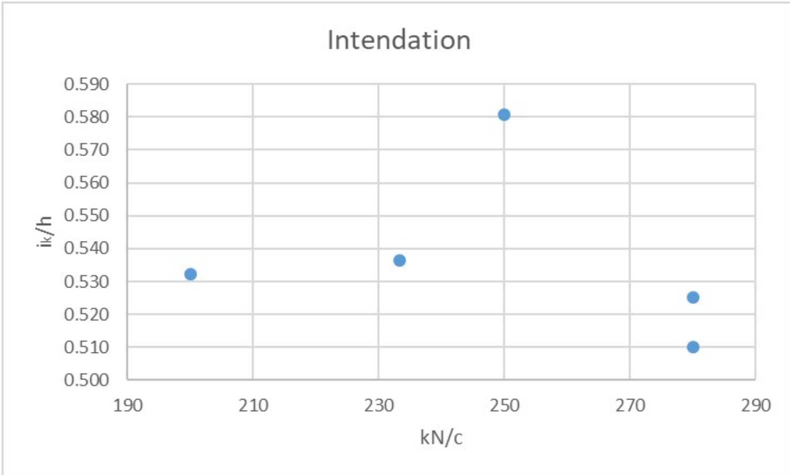

Figure 4.34: Indentation per Cutter Thrust Level

The resulting chipping frequency can be seen in figure 4.35. An increase in the frequency with an increase in cutter thrust suggests a more efficient rock-breaking process, as fewer but thicker chips are released per revolution.

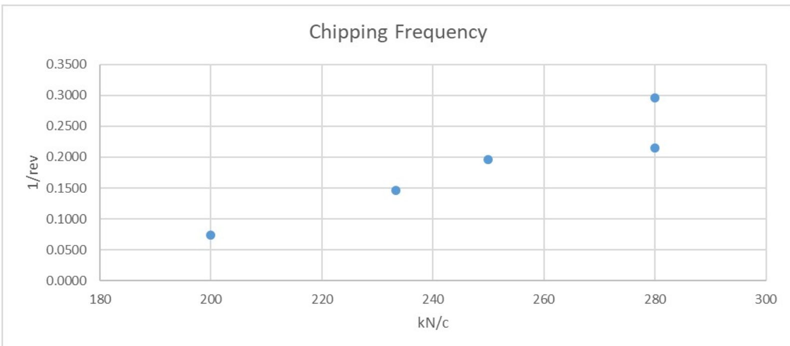

Figure 4.35: Chipping Frequency per Cutter Thrust Level

The resulting specific energy use can be seen in figure 4.36. The results does indicate a reduced energy use with an increase in cutter thrust. There were however problems getting an exact read of the ampere levels to the cutter head, in addition to changes in the rock-mass conditions and change in the RPM. It is thereby not a too reliable result as many parameters were changing during the test.

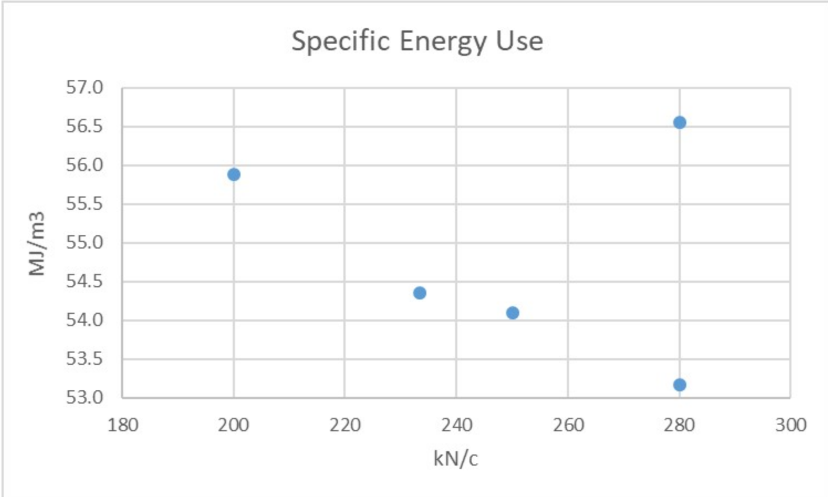

Figure 4.36: Specific Energy Use per Cutter Thrust Level

4.6.2 Normal Operations Results

The resulting average ship size can be seen in figure 4.37. The chip dimensions are significantly larger compared to those of PT1.

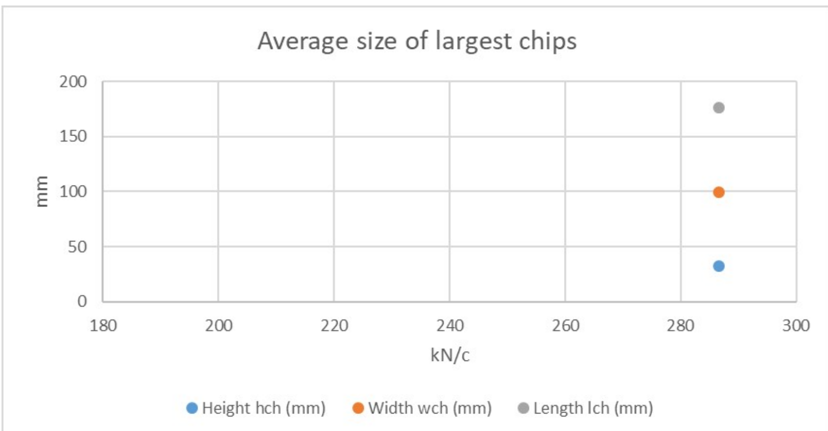

Figure 4.37: Average Size of the Largest Chips

The ship shape factor can be seen in figure 4.38. The resulting location in the diagram suggests a normal shape with a tilting towards the “flat” area.

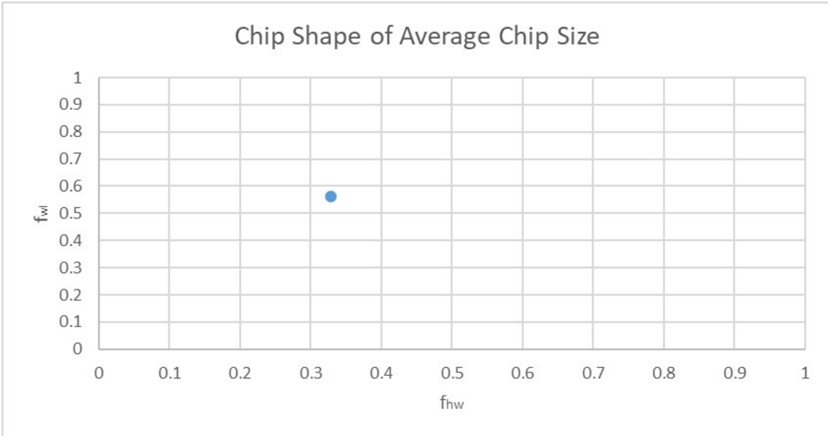

Figure 4.38: Chip Shape of Average Chip Size

The resulting chip cubic volume can be seen in figure 4.39. The volume is considerably larger compared to he results in PT1.

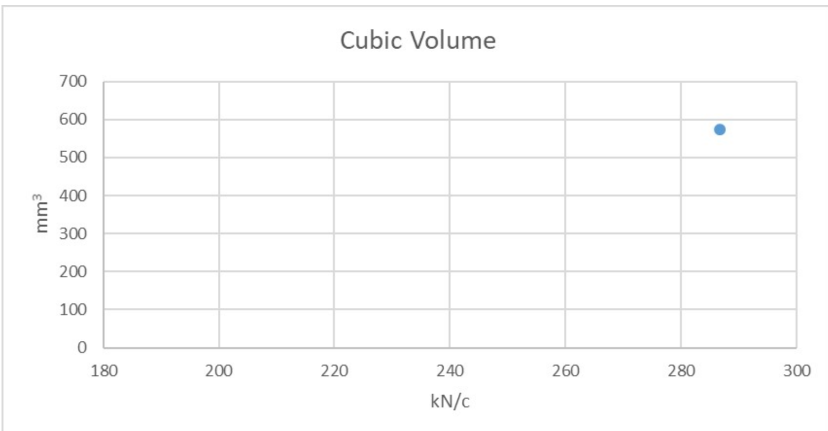

Figure 4.39: Cubic Chip Volume per Cutter Thrust Level

The indentation can be seen in figure 4.40. The significantly lower value compared to the PT1 suggests a more efficient rock-breaking process.

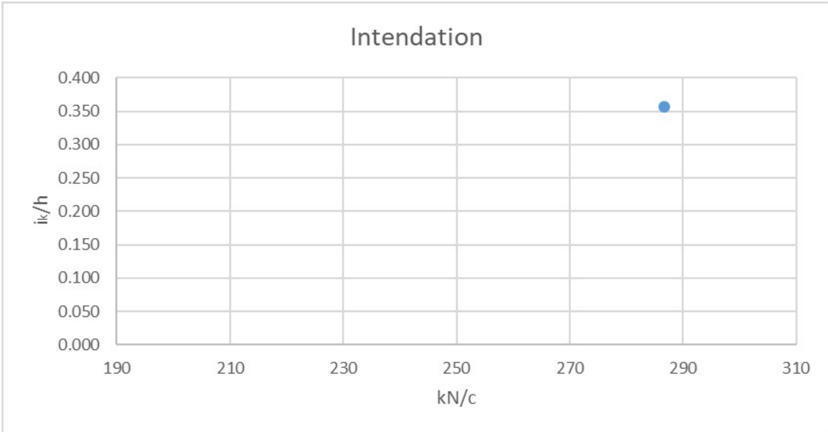

Figure 4.40: Indentation per Cutter Thrust Level

The resulting chipping frequency can be seen in figure 4.41. The chipping frequency in considerably lower than in PT1.

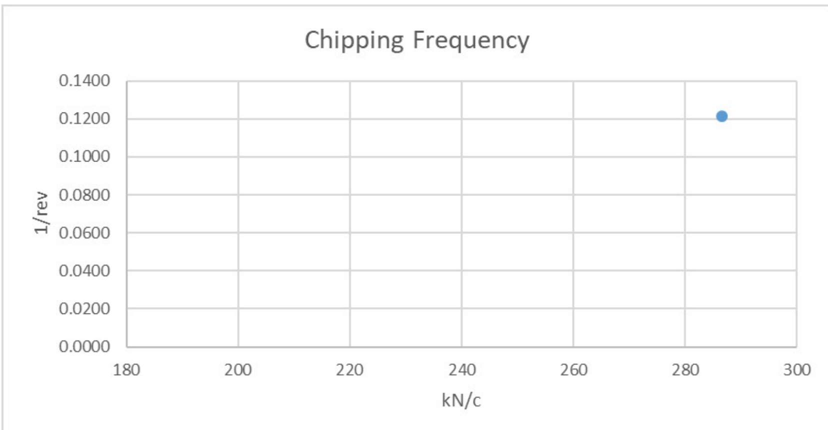

Figure 4.41: Chipping Frequency per Cutter Thrust Level

The resulting specific energy use can be seen in figure 4.42.

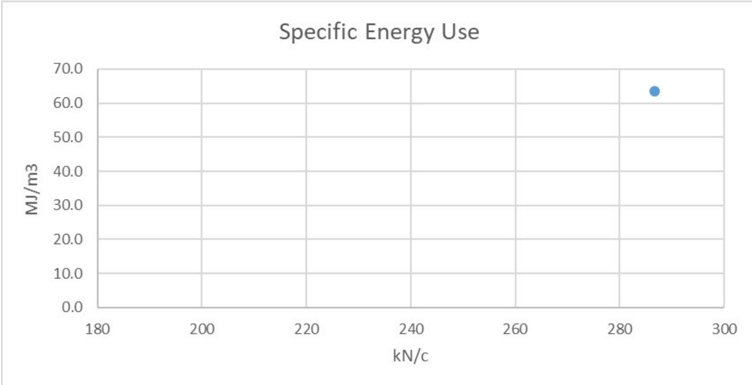

Figure 4.42: Specific Energy Use per Cutter Thrust Level

4.6.3 Discussion

The results from the penetration test shows a decreasing indentation with increased cutter thrust levels, suggesting a more effective rock breaking process. The test was however not conducted under ideal conditions, thus having a fracturing heavily influence the results. The samples taken during normal operation in intact rock gives a kerf depth factor of 0.35. The significantly lower value is likely caused by a more equal distribution of the thrust between the cutters than for fractured rock. The degree of cutter wear may also be very influential, as it both widens the contact area and less thrust gets distributed to this cutter. Energy use per cubic meter of rock excavated appears to be decreasing as cutter thrust increases. The results should not be utilized to draw any general conclusions, due to the low amount of data. There were two attempts of running av RPM test. Both failed due to that only four out of six cutterhead engines were operational. When the RPM was lowered, the torque increased to a level where the amperage could damage the engine.

4.7 Mapping Methodology

4.7.1 Results

The use of different analytical lengths when mapping has proven to have significant impact on the estimated average fracturing factor and thus the resulting estimated NPR(Seo et al., 2015).

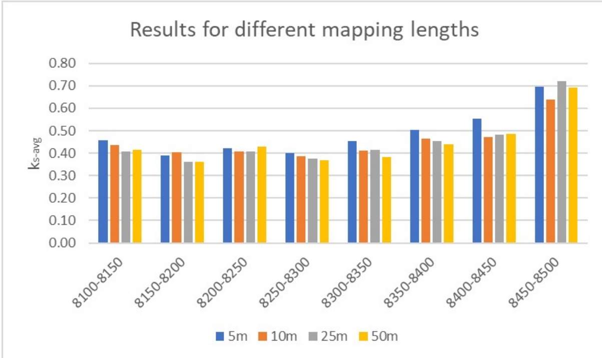

Figure 4.43: Results of different mapping lengths per analyzed section in Kontum.

Figure 4.43 shows the resulting average fracturing factor based on different analytical lengths accumulated to 50-meter sections. There is a clear trend of a reduced estimated fracturing factor for increased mapping lengths. When averaged and used as input for the NTNU model, we get the following results.

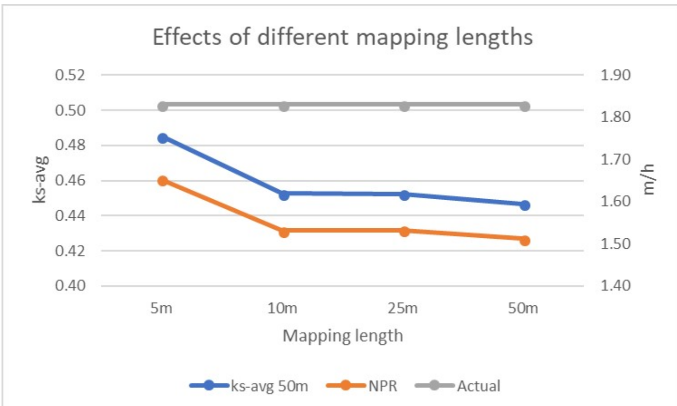

Figure 4.44: Accumulated results for chainage 8100-8500.

Figure 4.44 shows that the use of 5 m sections give the best resulting estimated NPR, with a 10% underestimation of the actual NPR. The remaining mapping lengths respectively give 16%, 16% and 17% underestimation. This is in accordance with previous studies, suggesting that 5-meter mapping length gives the most accurate result. The NTNU-model is also supposed to give a conservative estimate, which is in line with the deviation seen in this case.

4.7.2 Discussion

The results indicated that the use of 5-meter sections were the most appropriate mapping lengths for the NTNU model. A general issue causing variation in the results from geological back-mapping is the influence of subjectivity when it comes to interpreting the rock-mass. This was not considered to be a great concern in the Kontum geology, as the characteristics of the fractures clearly indicated which had contributed to the rock-breaking process and which had not. This thus reduced the probability that too few fractures had been included when estimating the fracturing factor, thus making it improbable that the use of longer mapping lengths gave a more accurate estimation.
There was however some contradiction in the results compared to previous studies when increasing the mapping lengths.

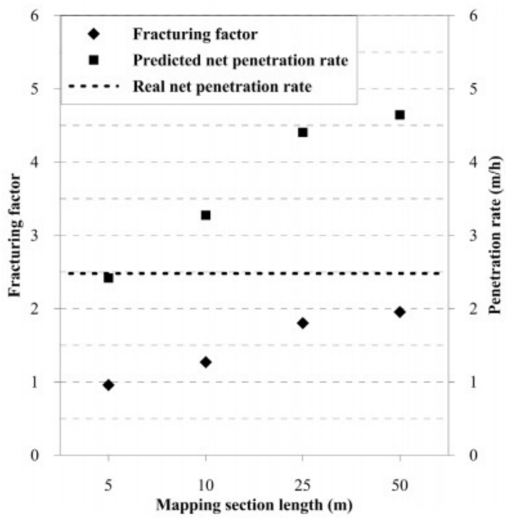

Figure 4.45: Results from Faraoe Island Study (Seo et al., 2015)

Figure 4.45 shows that the results from the previous study indicate an increase in the mapping lengths also increases the calculated average fracturing factor. The results from Kontum indicate that the fracturing factor decreases with increasing mapping lengths. At first, it was assumed that the deviation could be the result of statistical variation due to the short distance mapped in Kontum. To verify if this was the case, an analysis of a simulated rock-mass was conducted.

The rock-mass was simulated in excel by assuming that the occurrence of fractures followed a Poisson-distribution. 10 km kilometers were simulated with different average fracture distances to see the effect of a low fractured versus a highly fractured rock-mass. It was assumed only one fracture set with an angle to the tunnel axis of 49 degrees, alike the J1 set identified in Kontum.

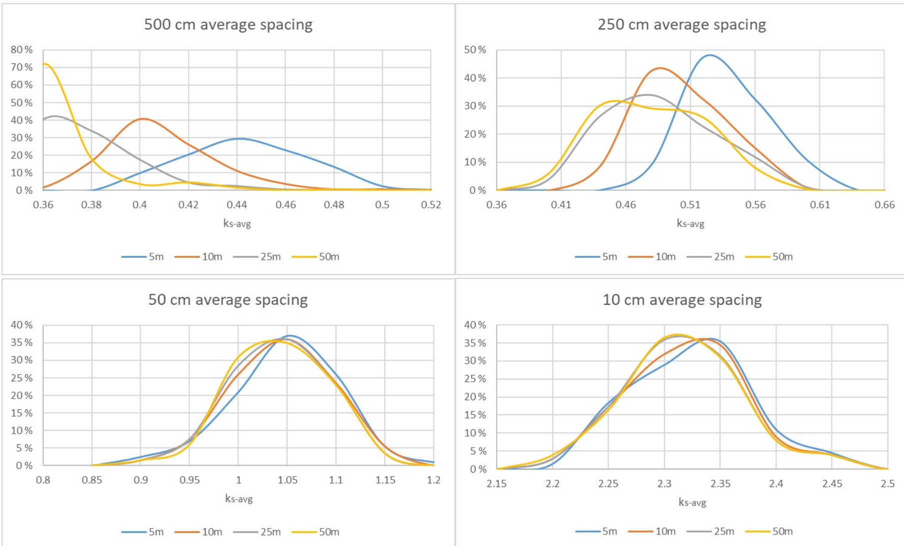

Figure 4.46: Effect of analytical lengths under different rock-mass fracturing conditions, given systematic fracturing.

The simulation results in figure 4.46 clearly show that the average value of the fracturing factor is highest for 5-meter mapping lengths and decreases with increased mapping lengths. The deviation in the average fracturing factor between different mapping lengths decreases with the increase in rock-mass fracturing. This implies that the use of different mapping lengths becomes less significant as the general rock-mass fracturing increases. The variation does decrease with longer mapping lengths, while the deviation becomes between lengths becomes significantly smaller with an increase in mapping length. The simulation supports the results from Kontum, thus indicating that the calculated average fracturing factor should increase with shorter mapping lengths. This is though under the assumption that the fracturing is

following a Poisson distribution throughout the entire tunnel.
According to one of the engineering geologists present at the Faraoe Island tunnel, the characteristic of the rock-mass fracturing was different than for Kontum. The rock-mass had a more sporadic occurrence of fractures, where there as an example could be 20-30 m without fractures followed by 5 m with 20 fractures. This was simulated in the model by assuming that the occurrence of heavily fractured 5-meter sections would occur on average every 25 m.

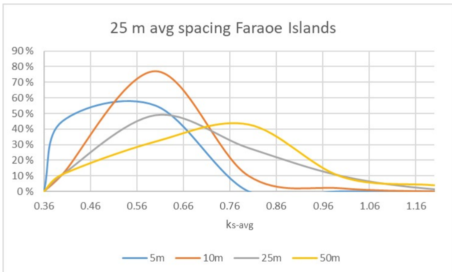

Figure 4.47: Simulation results for rock-mass conditions met in the Faraoe Islands.

As can be seen in figure 4.47, the modelling of this fracture characteristics gives the lowest fracturing factor estimate for 5-meter mapping lengths and increasing estimates for increasing mapping lengths. This is in accordance with the results from the study at the Faraoe Islands. In addition, the variation in the resulting data appears to be increasing with an increase in mapping lengths.
In total, the results from the simulations shows that the consequence of using longer mapping lengths than 5 m are different depending on the rock-mass fracturing characteristics. Even though both projects gave the best results when using 5-meter sections, it is not given that the use of this length provides the ideal description of the all rock-mass fracturing characteristics.

4.8 TBM Utilization

4.8.1 Results

Data for a total of 20 months, from June 2016 to February 2018, was analyzed for assessing the TBM performance. July 2017 was excluded from the assessment, as TBM operation was shut down almost the entire month due to other work that had to be executed in the tunnel. Figure 4.48 shows that the overall performance for the TBM has improved from the start of the project to February 2018. The accumulated length of tunnel bored is seen in figure 4.49.

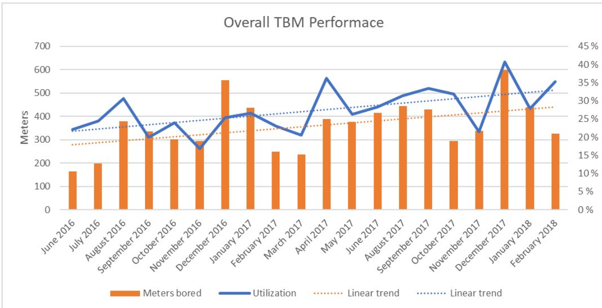

Figure 4.48: Overall TBM Performance showing the monthly boring length and monthly utilization, including the linear trend of the general performance.

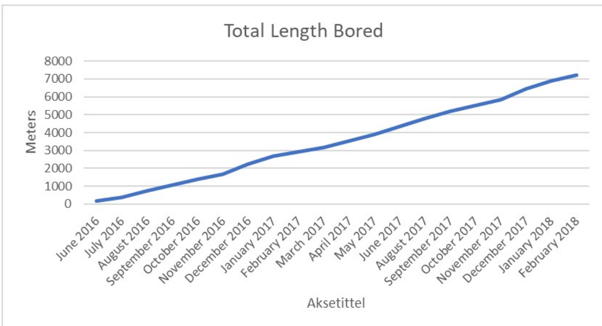

Figure 4.49: Total length bored from current contractors boring commencement until February 2018.

The utilization can be divided into two significant intervals. June 2016-November 2016 shows declining trend in utilization with a loss of 1.1 percentage point per month. November 2016-February 2018 shows an increasing utilization of 0.9 percentage point per month.

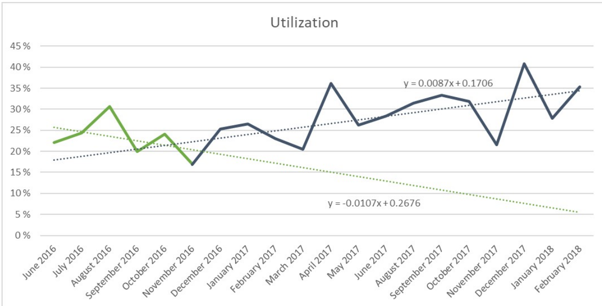

Figure 4.50: Development in the TBM utilization showing two distinct trends, respectively for the first 6 months and the last 14 months.

The time consumption for maintenance is shown in figure 4.51, normalized to the number of hours per kilometer on a monthly basis. The NTNU model for a low-quality system is shown for comparison. This includes the ''Tunnel length factor" given in the model, in addition to time consumption for TBM, back-up systems and miscellaneous activities.

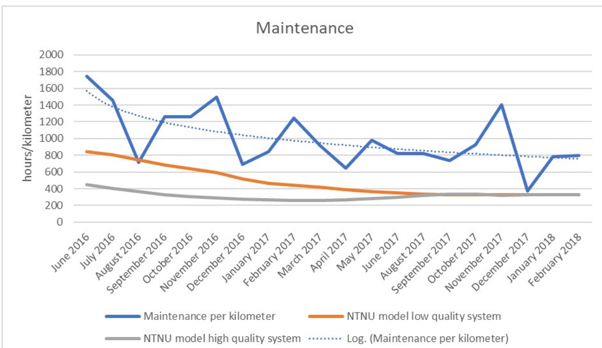

Figure 4.51: The total number of hours other than boring, re-gripping and cutter change normalized to h/km on a monthly basis. The total time consumption for a low-quality system as defined in the NTNU-model is plotted as a comparison.

As shown in table 4.14, the estimates given for time consumption for boring and cutter inspection and change are accurate. This is though despite the fact that the average cutter lifetime is 50% higher than what given by the model, which can be seen in section 4.9. Time consumed for regripping is about 50% higher, while time spent on maintenance is respectively three times and twice the estimation for the first and second 10-month period of the project.

Table 4.14: Comparison of the number of work-hours to each activity for the first and last 10 months of operation with the output from the NTNU-model.

Total time consumption		T_b (h/km)	T_r (h/km)	T_c (h/km)	Maintenance (h/km)	Sum (h/km)
June 2016	**March 2017**	422	59	256	1070	1807
April 2017	**February 2018**	486	58	222	793	1559
NTNU Model		454	42	209	339	1043
Percentage of time consumption						
June 2016	**March 2017**	23 %	3 %	14 %	59 %	
April 2017	**February 2018**	31 %	4 %	14 %	51 %	
NTNU Model		44 %	4 %	20 %	33 %	

The results from estimating the utilization with the Q_{TBM} model is seen in figure 4.52. The estimation was done both for with the m_1 parameter both for the conditions at site and for the conditions during the world record in boring given in the model description. The model assumes a gradual deterioration, with a slighter decrease in utilization as the number of work-hours increases. The actual utilization is on average higher than the estimation in all months except June 2016. The utilization decreases at a similar rate as the estimation the first six months but increases after this and surpasses the world record case. It should be noted that the Q_{TBM} model also includes time consumed for rock-support, which is not included for estimating the utilization in Kontum. Certain months would thereby have a utilization closer to what is estimated for Kontum.

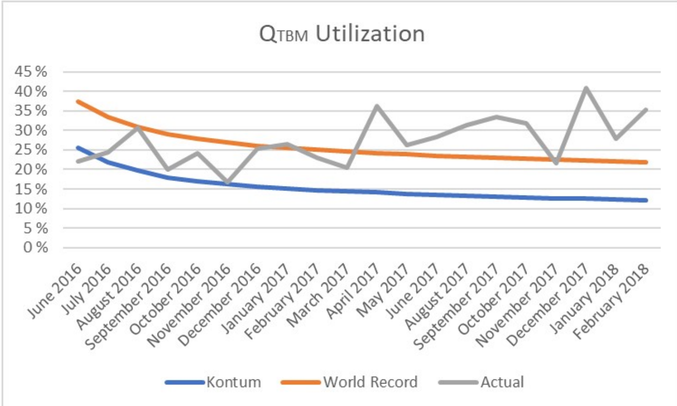

Figure 4.52: Comparison of the Q_{TBM} estimated utilization for Kontum with the actual utilization and the world-record case (Barton, 1999)

4.8.2 Discussion

Given the state of the TBM when the boring recommenced, there was an expectation that the utilization and gross advance rate would be significantly lower than for a new TBM. The NTNU model suggested a utilization of 44% for a low-quality system, which is higher than the best monthly utilization for all recorded months. The average utilization performance to this date is 27%, which accounts to 61% of the expected performance. The corresponding advance rate is 594 meters per month, of which only two of the recorded months has managed to achieve. The current monthly average is 360 meters, which also accounts for 61% of expected performance. These numbers are based on the average rock mass fracturing mapped in chainage 8100-8750 as input for the NTNU model. This suggest that the NTNU model predicts the net penetration rate for this project accurately, while the time consumption for other activities are significantly lower than expected. There has not been conducted mapping of the rock-mass fracturing in accordance with the NTNU model throughout the entire tunnel, but it is assumed somewhat equal conditions based on the assessment of general TBM performance data and lab testing. This is supported by the clear correlation between the utilization and the advance rates seen in figure 4.48.

Looking closer at the utilization, there is a clear downward trend the first 6 months of operations before this turns around and leads to an increased trend in utilization. This is the opposite case of what is usually seen in TBM projects, where there is a high increase in utilization the first months due to a “learning curve” for the workers of the projects and a slight decrease in utilization over time due to wear of the machine. There may be several reasons to the development of the utilization. Firstly, the TBM has been inside the tunnel for more than six years under hot and humid conditions. This is unfavorable in regards of corrosion of mechanical parts, electrical engines and equipment, sensors, and so on. Secondly, the TBM was operated by non-competent personnel while being built by the Chinese contractor. There were large damages to the cutter head after only one kilometer of boring and a second cutter head was kept in reserve at site in case the current one would fail. The damages suggest that the entire TBM had been under significantly larger stress than what was intentioned for the machine, thus leaving it in a poorer state from a mechanical point of view. In combination, these conditions are likely to have been the root cause of many of the problems when the boring commenced. From what can be interpreted from figure 4.50, it took around six months of boring before a enough problems were rooted out so that the utilization started increasing again. The steady increase after six months suggests that the continued replacement of deteriorated components had a significant impact on the utilization. It should also be taken into account that extensive repairs were conducted prior boring commencement in June 2016.

The total time consumption for maintenance per kilometer is significantly higher than what the NTNU model suggests, even with a high estimate for miscellaneous activities and the low quality/low skill estimate for the tunnel length factor. The deviation is nearly at 800 hours/kilometer at project start while decreasing to about 400 hours/kilometer in February 2018. Despite the improvement, the average number of hours used is about twice as high as the most conservative estimate. As the trend in the number of hours used for maintenance appears to be flattening out, which is also the case for the NTNU model estimate, there may fundamental reasons for why the number of hours used for maintenance are this high. One reason may be the high temperature and humidity in the tunnel, especially during boring, affects the general productivity of the workers at site. Damages to the casings holding the cutter axle in the cutter head required more frequent repairs to avoid excessive cutter use, which also contributed to the decrease in general utilization. According to the chief electrician, there was also an issue regarding fluctuations in the frequency in the Vietnamese power grid. As electrical components as sensors were vulnerable to this, they often failed and had to be changed at a higher rate than what was common. This also contributes to the general productivity of the TBM, as this would shut down if any TBM or conveyor sensors failed. There were also occasional power failures in the local electricity grid caused by lightning. A previous SINTEF report has discredited this element as the main or a major factor for reduced utilization(Trinh, 2013), while it still has an impact on the performance. There was also an issue with low quality parts delivered by the Chinese division of Robbins which led to an increase in down-time. This was the case with the TBM conveyor belt, which was changed twice during the five-week stay at the project site, in addition to several extensive repairs conducted to keep it operational. It took about eight hours to change one belt in addition to approximately 24 hours for the vulcanization process when merging the belt. One change alone contributes to 4-5 percentage points reduction in utilization per month, while some of this is gained by the fact that other maintenance activities can be conducted at the same time.

Looking at the time consumption in hours per kilometer in table 4.14, it is evident that the model makes realistic estimates for time used for boring and cutter change. The actual time consumed for regripping is about 50% higher than what is estimated. An explanation for this is that the forms the TBM operator used to keep record of activities did so on a ten-minute basis. Even though the regripping only took six minutes, it would be registered as ten minutes in the daily logs. It is again shown here that the maintenance is heavily underestimated. One would expect the time used for cutter inspection and change would be significantly lower, as the average lifetime in fact is 50% higher than what is assumed. An explanation for this may be that the time used for inspecting the casings holding the cutters were higher than normal, as there was severe damages to the cutterhead.

There are some factors that affected the performance parameters that should be taken into account. Time registered for “Ground Support”, which in some portions of the tunnel were quite extensive, have reduced the calculated working time significantly in some cases. Boring does however happen during this activity, as the crew would advance the TBM approximately one meter before installing support measures. The meters advanced would thus be registered for the actual boring time, while the actual boring time was reduced, thus inflating this number. During breakdowns, heavy repairs or other work that required shutting down operations, general maintenance on the TBM would occur concurrently as this work was being done. Given the large number of hours spent on these activities per month on this project, the time registered for “Cutter inspection and change” would in certain months be drastically reduced.

The Q_{TBM} estimation for the utilization does not realistically represent the actual development in the utilization on this project. Despite the fact that this TBM is performing significantly worse than for a new machine, it still surpasses the best-case scenario provided by the model. Especially the "learning-curve" when commencing boring has a significant impact on actual utilization, and is not included in the model. As commented in the results, the time consumed for rock-support has not been included when calculation the actual utilization in Kontum. This is however included in the Q_{TBM}-model, which is thereby a source of error. Time used for this purpose was only extensive for a limited number of months, and does not affect the general results or or the difference in trend when it comes to both the estimation and actual utilization. The lack of possibility to consider project specific considerations leads to a wrongful estimation of the actual performance, in addition to the fact that it clearly giving a way too conservative estimate. The development in utilization may be realistic for a new TBM operating under ideal conditions. Still, the model is built on the assumption that projects operate under conditions so that the development behaves ideally from a mathematical point of view.

4.9 Cutter Consumption

4.9.1 Results

TBM performance and time-consumption data for a total number of 96 weeks was recorded. Of these, only 60 weeks were used in the analysis. The data sorted out was due to lacking registration of TBM performance parameters and no registration of cutter changes for some weeks. In addition, weeks where there was suspected that not all cutter changes were registered were sorted out. These were identified by having 10-20 times the lifetime per cutter compared to the average life time. Weeks with fewer than 25 boring hours were also sorted out, to reduce the variability in the resulting numbers. Total cutter data can be seen in table 4.15.

Table 4.15: Cutter consumption data

Parameter	Value
Length (m)	5816
Cutters	1160
h/c	2.66
m/c	6.26
m^3/c	97

As the rock-mass fracturing was not mapped in correctly, the NPR has been used to represent this parameter. The NPR is based dividing the number of meters excavated on the number of boring hours per week.

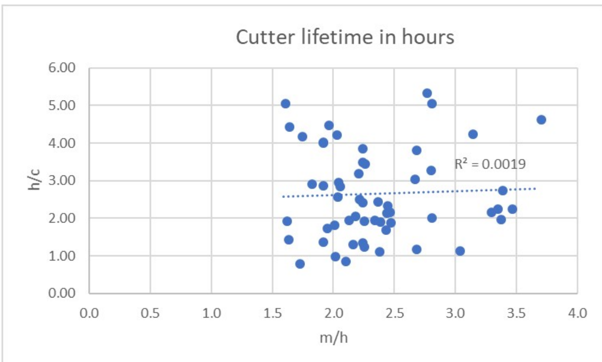

Figure 4.53: Scatter plot of cutter lifetime in hours and NPR

Lifetime based on rock-mass fracturing can be seen in figure 4.53. The result shows that the average lifetime per cutter is independent of the boring conditions. This is also the assumption in the NTNU model. The average cutter lifetime is 2.66 h, while the CLI of 5.5-6.0 suggests a lifetime of 1.75-1.81 h.

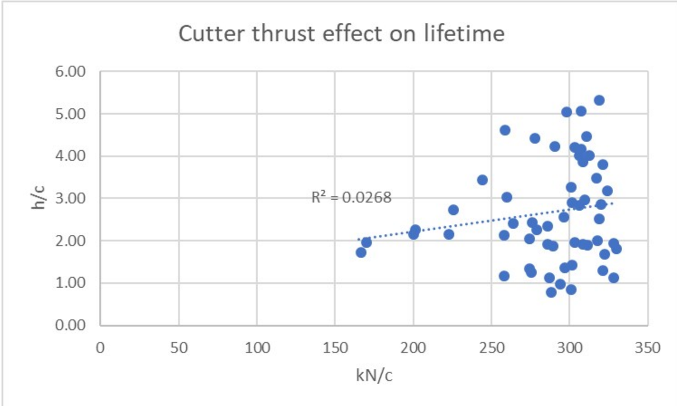

Figure 4.54: Scatter plot of cutter lifetime in hours and cutter thrust

The effect of cutter thrust on cutter lifetime can be seen in figure 4.54. The cutter lifetime in boring hours does not appear to be influenced by changes in applied cutter thrust for the given values, based on the value of R-squared. The NTNU model does assume an influence of the cutter thrust for CLI less than 6.0, which is the case on this project. It is though stated that the assumed reduction in lifetime in the model should be used with care.

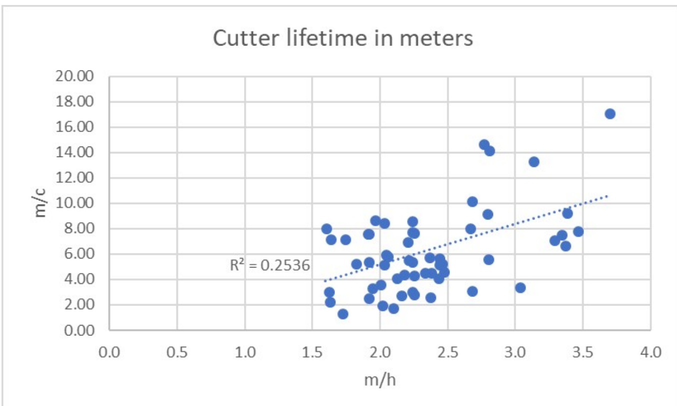

Figure 4.55: Scatter plot of cutter lifetime in meters and NPR

The cutter lifetime in excavation meters can be seen in figure 4.55. The result shows a correlation between the rock-mass fracturing and the cutter lifetime in meters. The NTNU-model assumes a linear relationship between NPR and the number of excavation meters, where the slope is equal to the average lifetime per cutter. The slope in figure 4.55 indicates that this value is 3.21, 20% higher than the actual average cutter lifetime of 2.66.

4.9.2 Discussion

The NTNU model basic cutter lifetime for 432 mm diameter cutters is based on cutter tip widths of 12-15 mm, while the cutters on this project are 19 mm. The lifetime should, in general, increase with an increase in cutter tip width as there is more material that has to be worn off. This may be the reason for the general increased lifetime seen on the project. The actual lifetime indicates a CLI of 14. It is not likely that the laboratory test results would have given such an error for the CLI value, thereby leaving an increased cutter tip width as the only plausible explanation. The NTNU model assumes that the cutter lifetime is independent of the rock-mass fracturing during boring. The results in figure 4.53 supports this assumption, as there is no correlation between the NPR, as an indirect parameter for rock-mass, and the average cutter lifetime in hours. The results in figure 4.54 do indicate that the applied cutter thrust does not influence the cutter life time. The NTNU model however does assume that the lifetime is affected by the cutter thrust for CLI-values less than 6.0. As the CLI is 5.5 in this case, the results conflict with this assumption. A source of uncertainty is that the sample used for CLI may have been taken at a location with a high portion of hard minerals. This can be identified by the color of the rock, as lighter materials tend to have a high percentage of quartz and Plagioclase Feldspar. Whether it can account for the big difference between predicted and actual lifetime is however uncertain, but it may contribute to some degree.

A source of uncertainty is that the data is averaged over the period of one week instead of being divided into different rock-mass conditions. Changes rock-mass conditions is directly linked to changes in applied cutter thrust, though not in a 1:1 relationship. Small increases in rock-mass fracturing leads to a significant increase in NPR and relatively small decrease in cutter thrust. When averaged, the changes in cutter thrust will be insignificant compared to the general variation in the data, while changes in NPR will have a significant influence on the calculated average.

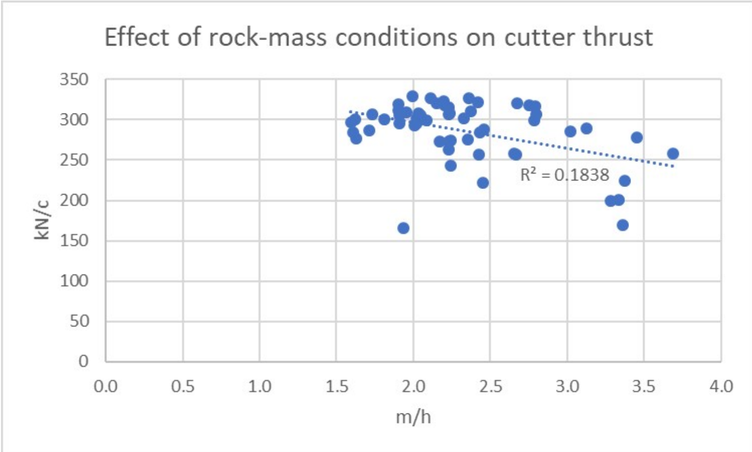

Figure 4.56: Scatter plot of the NPR and the cutter thrust

As seen in figure 4.56, the spread in cutter thrust versus the NPR does indicate that the latter is not a perfect indirect parameter for the rock-mass conditions. It does however clearly show the connection between decrease in cutter thrust for improved boring conditions. Continuing, the TBM performance data for cutter thrust was registered daily based on the operators estimated average. This may have caused some difference in the registered performance versus the actual performance. These factors may have influenced the results when looking at the effect of cutter thrust and should thus be used with care for further work.

The cutter lifetime measured in meters does clearly correlate with the rock-mass conditions. Using this parameter as a description of the lifetime in meters is thereby unfortunate, as one may get situations where the cutter lifetime appears to be higher while the rock may in fact more abrasive. The results do also indicate that the lifetime in meters increases with a higher rock-mass fracturing. These results thereby supports the NTNU-model approach, where the cutter lifetime is estimated in h/c. Another issue is that the CLI is assumed to be the same throughout the entire tunnel. The laboratory testing does support this assumption, while it has to be considered that small samples necessarily does not represent the entire tunnel. It is thereby likely that the use of wider cutter tips has led to a significant increase of the cutter lifetime.

Chapter 5

Conclusions and Further Work

Review of back-mapping conducted by Robbins

The way the NTNU fracture class has been registered at the project is obviously not correct. There is no connection between what was registered and the TBM performance. The mapping should for future purposes be registered correctly so that it will have an actual value for the company and potentially for academic use. The Q-system mapping appears to be in accordance with the instructions provided by NGI.

Model performance

The resulting estimation from the NTNU model based on the mapping conducted at the project site suggests that the model appears to be accurate under the geological conditions at site. The Q_{TBM}-model did also provide accurate results, despite that it revealed some weaknesses. This was especially related to how the influence of cutter thrust was modeled and the poor representation of the rock-mass at certain instances.

Mapping Methodology

The use of different analyzing lengths when back-mapping a tunnel evidently has different consequences depending on the characteristics of the rock-mass fracturing in the tunnel. Despite that both cases indicated a more correct estimation using 5-meter sections, it is not given that this will be the case in all sorts of geology.

TBM Utilization

Given the same or equal circumstances as has been the case in this project, actual utilization of half of what a new machine should be capable to accomplish should be expected. This was the case despite having an experienced maintenance and operation crew. A decline in utilization of more than one percentage point should be expected the first six months, before it starts to increase at a rate less than one percentage point per month. If this is due to maloperation of the TBM or that the TBM has been inside the tunnel in unfavorable conditions for several years, or a combination, is unknown. The number of hours spent on maintenance per kilometer should be expected to be double of what expected from a low-quality system over the course of the project. This may be less if issues like unstable frequency in power grids and the use of low quality parts are not a problem. The performance for the boring operation, hereunder boring, regripping and cutter inspection and change, should be expected to be that of a new machine, given that there are no limiting factors to cutter thrust. Estimating the utilization with Q_{TBM} should be done with care and with extensive knowledge of how the model responds to different input. It should also only be used for projects operating under ideal conditions, as there is no flexibility to adjust the model to specific conditions. One should also expect very conservative estimates for the utilization. Use of the model should be avoided in general.

Cutter consumption

The average cutter lifetime on the project was significantly higher than what the laboratory testing indicated. It appears that the reason for this was the use of a wider cutter tip than what was used in the projects forming the basis for the NTNU model. There was no connection between the applied cutter thrust and the rock-mass fracturing with the cutter lifetime in the data for the project, despite the CLI being in the range where cutter thrust should have some influence.

Further Work

There should be conducted studies on projects where maloperation of the TBM has caused reductions in utilization due to severe damages on the machine. The purpose would be able to distinguish what portion on this project is related to the standstill in humid conditions and what portion is caused by the maloperation, as situations like this are likely to occur in the future. There should also be conducted studies in other fracturing conditions to see the consequence of different analytical mapping lengths, as the knowledge about this currently is limited. Continuing, the effect of cutter tip width in regards of cutter lifetime, NPR and weekly advance rate should be studied more extensively, as it appears that the effects of this is not sufficiently considerer in the NTNU model. There should also be conducted studies much one can influence the test result of DRI and CLI by being selectively choosing the sample area, as there appears to be little knowledge of this subject.

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Appendices

A - Master Thesis Agreement

B - Technical Design Report

C - Report 1 SINTEF Site Visit and Project Assessment

D - Report 2 SINTEF Site Visit and Project Assessment

E - SINTEF Drillability Test Report

F - SINTEF Test Results of Core Samples from Site

Appendix A

NTNU

Agreement concerning MSc theses and supervision

This Agreement confirms that the topic for the MSc thesisis approved, the supervisory issues are agreed and the parties to this Agreement (student, supervisor and department) understand and accept the guidelines for MSc theses. This Agreement is also subject to Norwegian law, the examination regulations at NTNU, the supplementary provisions and the regulations for the MSc Engineering Education programme.

1. Personal information

Family name, first name: **Johansen, Egil Zahl**	Date of birth **May 5, 1993**
Email address **Egil.joh93@gmail.com**	Phone **92248973**

2. Department and programme of study

Faculty **Faculty of Engineering**
Department **Department of Civil and Environmental Engineering**
Programme of study **Civil and Environmental Engineering**

3. Duration of agreement

Starting date **January 15, 2018**	Submission deadline* **July 23, 2018**
If part-time study is approved, state percentage:	

** Including 6 weeks ekstra for staying abroad. + 1 week extra for Easter*
All supervision must be completed within the duration of the agreement.

4. Thesis working title

Analysis of TBM Performance at Upper Kon Tum Hydro Powerplant Headrace Tunnel *Comparing performance with the NTNU and the Q-TBM models*

5. Supervision

Supervisor **Pål Drevland Jakobsen**
Co-supervisor: **Francisco Javier Macias** (JMC Rock Engineering)

Standardized supervision time is **25 hours** for 30 credits (siv.ing) and **50 hours** for 60 credits (MST) theses.

6. Thematic description

Background:
Egil Zahl Johansen is studying Civil Engineering at the Norwegian University of Science and Technology (NTNU), with specialization in Underground Construction. The intended purpose of the master thesis perform a study of the TBM performance at a suitable TBM project in order to validate the newly revised NTNU hard rock TBM prognosis model (Macias, 2016), and to compare this to the results from the Q-TBM model.

Objective:
The main objective in this thesis would be to compare TBM responses due to variations in TBM operations and rock properties. As part of this Master study project rock samples may be gathered during production for laboratory testing in Norway. The aim of the lab tests would be to correlate rock properties, e.g. strength and abrasivity, to the actual TBM performance.

Short description:
The thesis work would consist of 2 - 3 months of fieldwork at the project site. During this time, an extensive

field study would be performed. The field study would include recording essential TBM operational and performance data, collecting rock samples, perform mapping of sections of the tunnel and obtaining valuable data and knowledge from the experienced people at the site. Parts of this work will be done in cooperation with student Ola Hobbelstad, who will be focusing on the use of support measures compared to what the Q-system
indicates.
After finishing up the field study the collected samples from site will be taken back to Norway for lab testing.

Goal:
By evaluating the TBM performances and cutter/structural wear in relation to the ground properties at site the aim of the thesis would be to:
- Investigate opportunities to increase the efficiency of the TBM operations by reducing the machine/cutter wear and/or increase production.
- Contribute to the NTNU models of TBM performance in hard rock tunnelling.
- Contribute to the understanding of the difference in reults between the NTNU and Q-TBM models.

7. Other Agreements

Supplementary agreement	**Not applicable**
Approval required (REK, NSD)	**Not applicable**
Risk assessment (HES) done	**Yes**

Appendix (list)

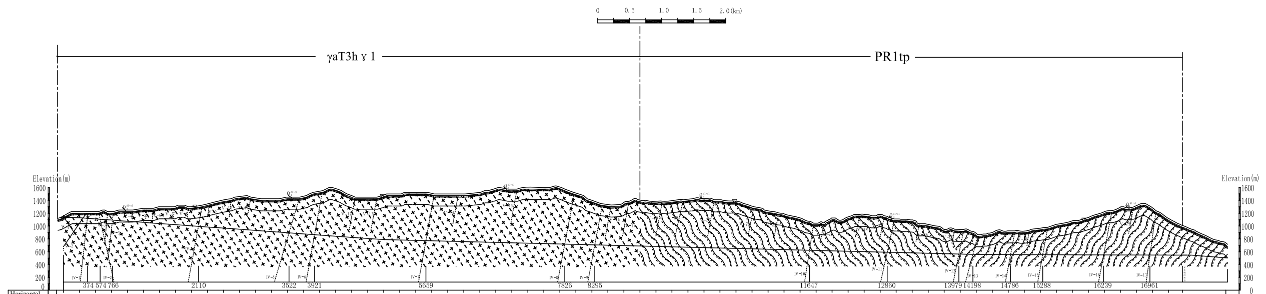
Risk assessment

8. Signatures

Conditions	Date	Signatures
I have read and accept the guidelines for MSc theses		E.Z. Fale... Student
I take the responsibility for the supervision of the student in accordance with the guidelines or MSc theses		Pål D. Jakobsen Supervisor
I take the responsibility for the co-supervision of the student in accordance with the guidelines for MSc theses		Co-supervisor
Department/Faculty approves the plan for the MSc thesis		Department/Faculty

Appendix B

Engineering Geological Profile along the Headrace Tunnel of Thuong Kon Tum Hydropower Project



Horizontal distance (m)	0 1000 2000 3000 4000 5000 6000 7000 8000 9000 10000 11000 12000 13000 14000 15000 16000 17000 17467.98				
Profile direction	N 49° E		N 49° E～N 89° W		N 89° W
Classification of surrounding rocks	Class III, accounting for 65%; Class II, 20%; Class IV，10%; Class V in small amount, 5%	Class II, accounting for 55%; Class III, 25%; Class I, 10%; Classes IV and V locally, accounting for 5% respectively	Class II, accounting for 65%; Class III, 15%; Class I, 10%; Classes IV and V locally, accounting for 5% respectively	Class III, accounting for 65%; Class II, 20%; Class IV, 10%; Class V in small amount, accounting for 5%	Class II, accounting for 60%; Class III, 20%; Class I, 10%; Classes IV and V locally, accounting for 5% respectively

Description of geological conditions

一、Strata and lithology

The tunnel runs through: *Stra.γaT3h¦Γ1*, which is mainly composed of biotite granite and distributed along the upstream half section (the first 9km length) of the tunnel in the water conveyance route, the fresh rocks (IIB) are in grayish white and blocklike structure, and compact and hard. The main mineral compositions are: quartz (30 %), plagioclase (30%), orthoclase (30~40%), mica (4~10%) and others in small quantity; *Stra.PR1tp,* which is distributed in the downstream half section (after the 9km length), the main lithology is biotite gneiss, granite gneiss and gneiss, etc.; *Stra. N2-Q1,* which is distributed in the middle section of the water conveyance route, outcrops in small area at EL. 1400m above the cavern, and is of basalt; and the *overlying residual slope wash* (Q_4^{dl+el}), which is generally 5~30m thick.

二、Geological structure

The geological discontinuities develop after upstream 3km length of the water conveyance route, totally seventeen Class IV faults and twenty-nine Class V faults. Most faults are approx. orthogonal to the tunnel axis and with high dip angle (75~85¡ ã). The faults may impact the cavern stability and construction, especially Fault Ⅳ-13 (attitude N20~30¡ ã E NW ∠80¡ ã , affected zone 8~10m wide, a small included angle (23¡ ã) to the tunnel axis) that has great impacts on the cavern stability. A large fault is concluded to pass through the gully at the 10.9 ～12.1km section (calculated from 3km) of the water conveyance route. The fault is plotted based on surface survey and physical exploration, without verification by borehole and exploratory adit, and is basically drawn with a high dip angle and in one direction, resulting in a relatively low reliability and uncertainty. The location may be changed during construction.

3．Classification of the surrounding rocks

In the data provided, the surrounding rocks in the headrace tunnel haven't been classified. According to the geological data available and in combination with engineering experience, the initial classification of the surrounding rocks in the headrace tunnel is put forward. The tunnel has mainly the surrounding rocks of Classes II~III, accounting for 70~80 % of the total tunnel, then, surrounding rocks of Classes IV ~V and I, accounting for 10~15%.

4. Hydrogeology

The water conveyance route is relatively long, where the landform is undulant is developed with gullies. There is a large gully at the 10.9～12.1km (about 3km long) section of the tunnel, the geological discontinuities develop along the tunnel and is water enriched. During construction, great attention should be paid to the groundwater regime, and water gushing is likely to take place along the gully or faults. In view of the geological structure, characteristics, latent activity as well as the relation of the surface water to groundwater in the region, the water conveyance system area is mainly of the water-bearing complex composed of invaded rocks and metamorphic rocks. The water table is generally shallow underground (6~28.1m) along the downstream half of the water conveyance route. As the headrace tunnel runs all below the water table, big water gushing is likely to occur in construction.

图 例

- Residual slope wash
- Biotite granite
- Gneiss
- Inferred faults and their serial numbers (IV-1)
- PR1tp Stratigraphical age
- Boundary between bedrock and overburden
- Lower limit of inferred weak weathering
- Lower limit of inferred weak weathering
- Inferred groundwater line

说明：

1．The topographical contour of the profile is made according to the topographic map of 1：20000.

2． Only two faults on the intake profile are measured and the rest are all inferred according to the bidding documents.

3．The weathering borderline and water table are inferred according to the bidding documents.

NO: H131B-3D3-01

PROJECT: Thuong Kontum Hydro Power Project								
Contract No: 885/2010/HD-VSH-TH				Package No: TKT-4.2.1				
Employer: VINH SON SONG HINH HYDROPOWER JOINT STOCK COMPANY				Leader of Consortium: HYDROCHINA HUADONG ENGINEERING CORPORATION; Partner of Consortium: CHINA RAILWAY 18TH BUREAU GROUP				
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Appendix C

SINTEF

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Report

TBM & Rock Engineering Expert for Upper Kontum hydropower project – Trip 1

Trip date: from 22nd to 27th July 2013 2013

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Report

TBM & Rock Engineering Expert for Upper Kontum hydropower project – Trip 1

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ABSTRACT

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SINTEF

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Summary

After a field trip to the project site in July 2013, TBM and Rock engineering Experts have commenting on the following issues: the construction issues at the TBM tunnel, including both technical and management issues, construction issues at the 5 km drill and blast headrace tunnel, location of the powerhouse complex, and initial comments on the measurement of the in-situ stress.

The TBM tunnel: The construction issues at the TBM tunnel can be divided into two groups, which are technical and management issues. After the major repair program from April to June 2013, it can be stated that the TBM generally meets the technical requirements for operation. The remaining technical issue with the TBM is the high consumption of the cutter. This issue will be investigated and handled by the TBM and the main Contractor. The Project Management Unit needs to follow up the case as it affects the performance of the TBM. The largest problem, which has the strongest influence to the overall constructability of the TBM is the lacking of competence of the Contractor to the operation, maintenance, and construction organisation and management. In addition to that, the financial situation of the Contractor is problematic. In order to improve the management and financial issues of the Contractor, the Project Management Unit needs to improve regarding the personnel, competence, and method of working. It might be necessary to strengthen the Unit with a TBM Expert. The internal policy within the VSH needs to be revised to increase the real power of the Project Management Unit, closely coordinating and dividing responsibilities and action between the levels of the company. From this improvement, some recommended action plans can be carried out effectively. The purpose of the action plans is that the Contractor must feel real pressure to change, such as adding experienced personnel, re-organising the construction and management. In brief, the real problem causing the poor performance of the TBM is not the TBM itself, but the lacking experience of the Contractor. In order to improve the situation, first of all, the Owner especially the Project Management Unit needs to be improved for effectively managing the Contractor.

The 5 km drill and blast headrace tunnel: The largest problem in this tunnel now is the ingress of the groundwater causing a slow progress of tunnelling. Based on the observation of the small streams on the surface of the excavated and the remaining part of the tunnel, it is anticipated that the groundwater problem can be encountered in at least two sections ahead. Thus, the Contractor needs to sufficiently prepare pump capacity, pipe, and generator. The discussed method on site to seal joints with major flow can be implemented for testing.

Location of the powerhouse complex: The powerhouse is moved 80-85 m further into the mountain comparing to the planned location due to a large weakness zone. The new location has rock mass quality from "Good" to "Very Good", suitable for large span openings. The Contractor considers 3 alternatives for the orientation of the powerhouse complex, which are almost North-South, 15 degrees (N15E), and 80 degrees (N80W). The Contractor's considerations about joint set orientation affects to the stability of the opening are theoretically appropriated. However, observations at the site indicated that joint frequency is low and rock mass quality in all direction is relatively isotropic. Thus, it is not necessary to rotate the axis of the powerhouse complex. Keeping the original axis or rotating by 15 degrees (N15E) as proposed by the Contractor are both acceptable.

The stress measurement and results: The Contractor measured stress at 4 locations, which are transportation tunnel, starting point of the steel lined section, adit tunnel, and intake tunnel. From the measurement result, the Contractor concludes that the steel lined section needs to be extended further upstream, and this will significantly increase the construction time and cost of the project. After discussion at the site, the rock engineering expert has some initial comments that the stress measurement was not sufficient due to the following reasons:

- The measurement does not meet its purpose: the main purpose of the stress measurement is to check the stress situation at around chainage Km14+500 for considering tunnel route. Thus, the stress measurement shall carry out from around chainage Km17+000 toward upstream to evaluate the stress development and obtain reliable stress information for analyses;
- The measured result at the starting point of the steel lined section was significantly different to the 3 remaining locations. The measurement at the point gave a low result and negatively affecting to the steel lined consideration;
- Comparing the rock overburden with the Norwegian experience, the measured result at the point was much lower than normal. Thus, it is possible that the measurement was locally affected by a discontinuity or by some unknown factor, and the result is not representative.

1 General

Upon request from Vinh Son – Song Hinh Hydropower Joint Stock Company (VSH), a TBM expert (Mr. Beat Lutiger, from Amberg Engineering) and a rock engineering expert (Ph.D. Trinh, Quoc Nghia from SINTEF) carried out a field trip to the construction site for investigation from 22nd to 26th July 2013. This is part of a long-term cooperation between VSH and SINTF for the Upper Kontum hydropower project (220 MW, H = 900 m). The purpose of the site visit was to collect information and assist the VSH in evaluating the following issues:

- Investigating problems causing the poor performance of the TBM: up to date, the TBM is progressing very slowly and much behind the schedule. Thus, the VSH is very much concern about the total schedule of the project, as the TBM tunnel is in critical path now. The TBM expert will collect the information and evaluate the technical issue of different systems of the TBM such as mechanical, electrical, air compressor, hydraulics,.... Issues of construction organisation and management of the Contractor and Project Management Unit for the TBM are also evaluated;
- Evaluating the construction issues during the construction of the 5 km headrace tunnel (drill and blast), from the intake: currently, the most challenging issue in this tunnel section is the excessive groundwater flow (80 to 90 l/s), much larger than anticipated in the contract (20 l/s). The groundwater problem slows down the construction progress from more than 120 m/month to only 60-70 m/month. The rock engineering expert will collect the information at the site as well as geological report for a total review. Based on the review, the expert will discuss with the Contractor and the Project Management Unit to find out the optimum solution for the case, which fit the current condition and the equipment capacity of the Contractor;
- Evaluating the new location of the powerhouse, which is 80-85 m into the mountain comparing to the previous location;
- In addition to the above work, the rock engineering expert is requested to provide initial comments to the stress measurement carried out by the Contractor, recommending further actions.

2 Issues of concern at the TBM tunnel section

As mentioned earlier, the TBM progress causes a significant delay in the construction schedule for the project. According to the contract, the Contractor shall excavate 650 m/month in average. However, after 10 months of tunnelling (from 10/2012 to 7/2013), the total length of the TBM excavation is only 1075 m, meaning about 100 m/month.

After a thorough investigation of the main and auxiliary systems of the TBM as well as site organisation and management, the experts will report on the following issues:

- Technical condition of the TBM;
- Working model and construction organisation of the Contractor;
- Management aspect of the VSH.

2.1 Technical issues of the TBM

The tunnelling equipment, which consists of a Robbins TBM MB 1612-285-2 equipped with a back-up system and auxiliary equipment for rock support and logistics, is in line with the current state of technology for the excavation of long tunnels in hard rock. The logistic system consisting of a track-bound supply and evacuation purpose supplemented with a continuous conveyor for muck transport is designed well in balance with conditions of the project and provide sufficient transport capacity for high advance rates even for long transport distances.

In order to reach high advance rates, the following points are crucial:

- Geology: affects the heading rates insofar as the rock strength limits the penetration and the abrasiveness accelerates the wear. The used TBM corresponds in its lay-out to the state of the art for driving the in-situ rock formations. Considering the abrasiveness of the present rock, the cutter head and the cutting tools or the cutting rings are subject to high wear and it is assumed that the change of the cutters and cutter head maintenance will continue to consume a lot of time. Therefore, it is all the more important to reduce the number of cutter changes by using exclusively high-quality cutting tools. It is recommended to use original disc cutters and discs only.

- Organisation/Guidance: a highly mechanised excavation as TBM requires a tight organisation and guidance of the excavation and workshop personnel. The maintenance and logistic works to be done, the subsequent installation of the belt systems, the prolongation of the high-voltage cables and ducts, track maintenance etc. have to be well planned to accommodate anticipated and experienced ground conditions. All corresponding tasks have to be coordinated well.

- Personnel: The TBM operators have to be trained specifically for this TBM and its systems in order to reduce standstills and repair costs caused by operating errors.

- Availability of the tunnelling equipment: On long excavation stretches, availability of the tunnelling equipment can only be achieved by consistent maintenance works on the job site. The corresponding tasks have to be accomplished according to the instructions in the maintenance manuals. In particular, daily checks of the cutter head must not be omitted. These activities must not be neglected when the excavation works fall behind construction schedule.

- Excavation Logistics: With an increasing excavation length, the capacity of track-bound transport systems will be affected (decreased). Thus, timely planning and setup of alternative stations is necessary. The correct installation of the line inventory has to be verified.

- Job-Site Logistics: Since the TBM is the only available cutting tool on site, a breakdown of the TBM causes a standstill of the overall project. Therefore, a reliable spare parts supply is a basic prerequisite. Besides the spare parts management, i.e. the timely ordering of spare parts, a correct stock of spare parts according to instructions has to be guaranteed. Usually, spare parts should be available as close to the job site as possible so that they can be transported to the site within the time needed for the removal of the required part.

- Special Technical Aspects: The climate conditions under which excavation is under taken have to be carefully considered. With increasing excavation length and rock cover, a pre-cooling of the ventilation air at the portal or a cooling of the fresh air on the back-up shall be elaborated. In order to ensure a sufficient cooling of the drive systems, an additional cooling of the water supply is recommended.

Details of the comments on different parts of the TBM can be found in Attachment 1.

2.2 Working model and construction management of the Contractor

From the official meeting (under the Contractor's arrangement), we were informed that the working model in the project is that the Contractor buys the TBM from Robbins and gives responsibility to operate the machine to a sub-contractor.

In addition, the contract between the Contractor and Robbins implies that Robbins will be responsibility for installation, operation guidance, and maintenance for the first 2 km. It means that the Contractor's operation team will be guided to operate the machine according to the recommended procedure from the manufacturer. The guidance from Robbins is based on the assumption that the operation teams have sufficient experience in operating TBM. The contract implies that during the excavation of the first 2 km, Robbins will be responsible for investigation and fixing any technical failure of the TBM (not from abuse of the user). After completing thí 2 km of tunnelling, Robbins will not take responsibility for any technical failure. This is a significant different with the understanding of the VSH that Robbins is responsible for installation, operation, training, and excavation for the first 2 km.

The organisation can be presented as in Figure 1.

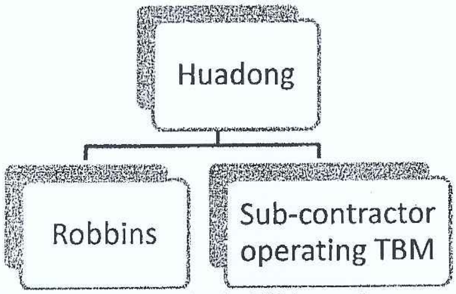

Figure 1: Contractor's organisation chart for the TBM operation.

With the organisation as presented in Figure 1, it can be seen that the Robbins personnel cannot directly instruct the operation team. Except for the normal guidance, Robbins personnel have no authority to intervene, suggest, or change the way the sub-contractor is operating the TBM. When observing erroneous operation, Robbins needs to inform Huadong and Huadong notifies the sub-contractor. In many occasions, the notification becomes too late. During the construction, it can be observed that the cooperation between Robbins personnel and the sub-contractor is very loose. Thus, the Contractor's organisation now has some serious issues need to be improved, being such as:

- The Sub-contractor does not have sufficient experience operating TBM;
- The main Contractor – Huadong is mis-understanding the context of "normal operation guidance" and "thoroughly training their team from inexperience to be skilful in operating TBM";
- The working organisation and cooperation between Robbins and the sub-contractor is ineffective.

Based on the contract (between the Contractor and VSH), the VSH understanding is that the TBM will be operated by Railway Bureau 18 with approved experience record. At the moment, the Huadong uses Bureau 1 and 6, and the competence and experience of the team is insufficient. This is a serious breach of contract that the VSH must take action to eliminate the dis-qualified personnel, teams, or sub-contractor.

Furthermore, the Huadong is of the understanding that Robbins is responsible for "thoroughly training their team from inexperience to be skilful in operating TBM", within the first 2 km. This is a mis-interpretation. As in the meeting with VSH in Quy Nhon, Vietnam dated 28/7/2013, Robbins clearly stated that they have no responsibility for training during the first 2 km. Their responsibility is limited to "normal operation guidance from the manufacturer" and maintenance for the first 2 km. According to Robbins, it is IMPOSSIBLE to train a person from unskilled to be skilful TBM operator in only 2 km tunnel. To be skilful, it would be needed to operate the TBM for at least 12 to 15 km. Therefore, in order to properly follow the "normal operation guidance" from the manufacturer, the operation team must have certain experience. This is an important point that the VSH, through its Project Manager Unit must follow-up.

2.3 Management issues of the Owner

In the meeting with VSH in Quy Nhon, Vietnam, on 28th July 2013, the Contractor presented several reasons for the poor performance of the TBM, the two following reasons were particularly stressed:

- Poor quality of the supplied electricity;
- The critical financial situation of the Contractor;

According to our observations at the site, we believe that only the second reason is relevant. The statistic from the Project Management Unit indicated that for the first half of 2013, electricity break down was recorded at 68 incidents with total time of 111 hours. This is a significant number, but it is not the main reason causing poor performance of the TBM. We believe that the financial difficulty of the Contractor is one of the main reasons. In our opinion, the most serious reason for the delay, which was not mentioned by the Contractor, is the present of an incompetent sub-contractor to operate the TBM. The situation leads to the following consequences:

- Sub-contrator non-qualified operational team: observations at the site showed many mistakes during operation of the TBM. Some of the typical mistakes that we observed at the site were pulling out the TBM without lifting the TBM legs; jamming the steel bar to the conveyor system, etc. These indicate the non-qualified personnel and may cause serious consequences for the TBM;
- Poor construction organisation and management:
  - Too many people inside the tunnel, and many of them have no contribution to the TBM operation;
  - The Contractor has no shift supervisor (shift boss), who holds responsibility of the shift. During the tunnelling in general, especially with TBM, this person is important and has almost all authority in the tunnel, includes the responsibility for operation, maintenaning the TBM, and securing health, safety, and environment. Before the shift starts, the supervisor knows exactly the number of people and responsibility of each individual in the shift. If in doubt, the supervisor can check any of the personnel for their understanding of their work. This is to ensure that the work in the shift is carried out according the plan and meeting the technical requirement. With this criterion and observations at the site, we believe that there is no such position in the tunnel. This is the first thing that the Contractor should improve to increase the performance of the TBM;
  - Lack of spare parts management: in fact, the TBM had to stop drilling on several occations to wait for the spare parts replacement and repair. The site is remotely located, and it is a logistical challenge to get timely spare part delivery at the site due to transportation, custom clearance, etc.. Not least, the financial situation of the Contractor delays spare part deliveries due to their late payment. A typical example that we observed at the site during our visit is that the Contractor needs to stop the TBM for 4 days to wait for the glue to seal the conveyor. The conveyor extending and sealing is an active plan, but this plan does not smoothly combine with other activities such as cutter changes. This leads to the situation that the TBM needs to stop 2 times, first to change the cutter and then to extend the conveyor. For an experienced Contractor, such activities would be carefully planned to minimise the TBM down time;
  - In the meeting with VSH in Quy Nhon (28/7/2013), the Contractor confirmed that the current model of the spare part preparation will be the same for the first 3 km. After this 3 km, a new model will be considered by the Contractor. As discussed earlier, the current model for the spare parts is having negative impact to the TBM performance. If all of the

down time has been recorded, all parties would have realised the seriousness of this problem. Thus, the VSH and Project Management Unit should carry out necessary actions so that the Contractor changes this model as soon as possible, without waiting for the completion of the first 3 km;

- As mentioned earlier, after the first 2 km, Robbins will not be responsible for any technical failure of the TBM. The next questions are what will happen with this inexperienced Contractor if similar technical failure occurs, as it happened so far? Does the Contractor have sufficient qualified personnel to replace the Robbins personnel? Is the infrastructure of the Contractor sufficient to replace the Robbin's (storages, workshop with standard equipment,...) after Robbins is pulling out of the site?

Based on reasons above (lacking of experience and financially difficulty), the Project Management Unit shall implement measures to closely monitor and follow-up the Contractor's performance and also assist him in detecting the real problems to be solved accordingly. Measures from the Project Manage Unit can be:

- Record all failures during the operation to demonstrate the lack of experience of the operation team. The personnel from the Project Management Unit must be sufficient in number to follow all working shift of the TBM, and they need to carefully study the operation manual from the manufacturer on TBM and auxiliary systems;
- When observing few "serious mistake" or only one "very serious mistake", the Project Management Unit, based on the appropriate clauses in the contract, can stop the Contractor and ask him for improvement. With this measure, the Contractor will have pressure to use qualified teams or hire expert to work with the team and improving the situation;
- Record waiting time for spare parts and distinguish this from the time of the electricity break down in order to highlight the real problem that the Contractor may be intentional or unintentional stated;
- Request the Contractor to organise monitoring program for the spare parts. The monitoring program shall record in/out time of any spare part show exactly the number of spare parts available in the store at any time;
- The main Contractor using Bureau 1 and 6 instead of Bureau 18 leads to an inexperienced contractor working in the tunnel. The VSH needs to use contractual tools to request the main Contractor to replace Bureau 1 and 6 with a qualified, approved sub-contractor as Bureau 18 or similar. The VSH should immediately remove from site sub-contractors or personnel, who are not fulfilling the technical qualification. In order to implement this measure, VSH must have an approach with contractual evidences so that the Contractor cannot object. Whenever the Contractor promise their solution, there must be a deadline to meet, and actions and the correspondent deadlines to be monitored closely;
- With the financial situation of the Contractor, the Project Manager Unit may provide assistances. One solution is the "consignment option" from Robbins. According to Robbins proposal, the Robbins offers to provide containers of spare parts at the site and the Contractor can take the necessary parts needed. When the project is completed, Robbins will take back the unused spare parts. With this option, the Project Management Unit can assist the Contractor by paying directly to Robbins all or part of the used spare parts. Thus, the Contractor has less financial pressure for the construction;

In order to implement the listed measures, the Project Management Unit must be strengthened significantly, both in number of personnel as well as competence. The personnel should be trained with sufficient knowledge and experience for TBM, studying carefully the manual from the manufacturer for understanding the technical requirements,... It might be necessary to hire a TBM expert with hand-on experience working with the supervision team for training and assisting. The VSH can also review the policy to their Project Management Unit, in the way that their power and responsibility at the site is increased for more effective management.

3 Construction issues of the 5 km drill and blast headrace tunnel

Currently, the largest problem slowing down the tunnel excavation is the excessive groundwater inflow. Up to date (26/7/2013, from chainage K0+000 to K2+600), 9 large inflow points have been encountered. The measured discharge of each point varies from 15 to 85 l/s. Whilst there is a trend that the discharge is reduced with time. The time elapsed from encountering water inflow until it is reduced can be a few days to more than 4 weeks. The tunnel section with most inflow is from chainage K1+650 to K2+600, as shown in Figure 2.

Based on the location of the groundwater inflow points and the location of creeks on the surface, there is a coherence between these inflow points and surface creeks consisting weakness zones. It might be that the streams being oriented parallel and close to the tunnel have stronger impact than the ones cutting across the tunnel. Based on the surface observations as shown in Figure 2, the risk of groundwater inflow still exist in at least two areas, around K3+400 and K4+800. To prepare for this situation, the Contractor must prepare for sufficient capacity for pump, pipe, and electrical generator.

With the current equipment capacity of the Contractor (hand held drilling), the pre-grouting is not possible due to the fact that the method would require more than 12 m drilling hole. If there is no intention of improving equipment, then the Contractor must accept the fact that they have to deal with excessive groundwater inflow during the excavation. The only solution now is preparing sufficient capacity of pump, pipe, and electrical generator (for the electrical break down situation).

Observations at the site on 26/7/2013 indicate that the Contractor is not prepared enough for the electrical break down situation. The Contractor has 4 generators at site, however only one is functioning. The Project management Unit and specifically the supervisor personnel at this location should execute tight supervision and monitoring. A risk analyses ought to be undertaken to clearify consequences and mitigative actions to be taken for scenarios like break down of electrical power supply. One consequence of the electrical grid break down could be groundwater may raise quickly causing flood in the tunnel. With all the related activities such as mobilising the equipment in the flooded part of the tunnel, pumping,... the construction can be delayed for many weeks.

According to general experience in tunnelling, controlling the groundwater inflow after excavation is very costly and ineffective. Post-grouting is time consuming and in addition to that the equipment capacity of the Contractor is not sufficient. A proposal presented by Mr. Bui Khoi Hung – an independent geological expert – includes inserting a pipe to drain the water flow, sealing the joint and then closing the pipe. We recommended to discus this with the Contractor. Reference can be searched among other Contractors in Vietnam that have been carrying out such a solution in similar situation to identify technical requirements, method, cost, and effectiveness. In the contract between VSH and the main Contractor, the Contractor will be eligible to claim for additional cost if the groundwater flow is more than 20 l/s. Thus, the Contractor may not have economical motivations to reduce the groundwater flow.

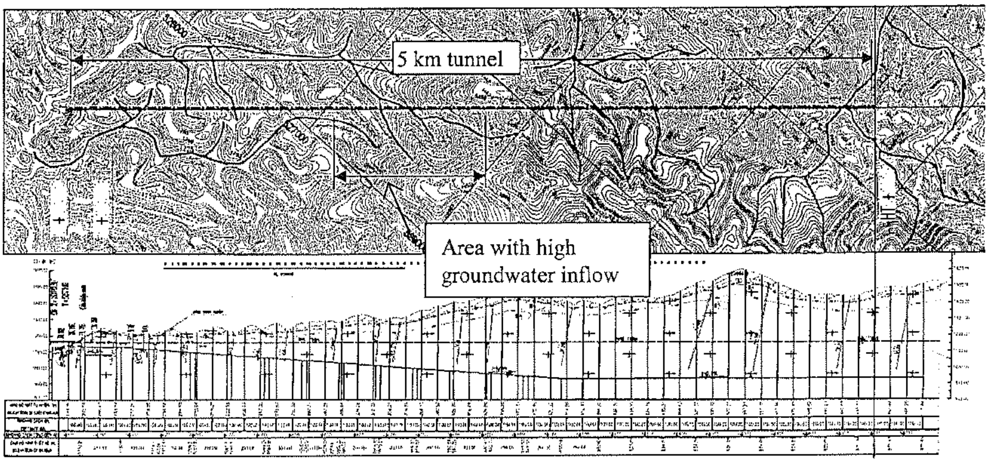

Figure 2: Plan and longitudinal section – The 5 km drill and blast tunnel from the intake.

4 Location of the powerhouse complex

Concerning the location of the powerhouse, the Contractor has prepared a comprehensive report to justify its chosen siting of the powerhouse complex. In the report, the following criteria are brought forward for their evaluation:

- Location and arrangement should provide a smooth transition of the flow through the powerhouse complex;
- The location should be in competent rock mass quality, away from the large fault zone that was encountered during the excavation;
- The powerhouse axis should create a large angle with the strike of the main joint set;
- The longitudinal axis of the powerhouse should be almost parallel with the maximum horizontal stress direction. However, in view of the small stress in the project area, the stress does not constitute the controlling factor of the project.

From the above criteria, the Contractor proposed a new location which is 85 m further into the mountain compared with the previous planned location. The Contractor considered 3 options for the axis of the powerhouse complex:

- Option 1: axis oriented almost North-South, N1.1E;
- Option 2: the axis is 15 degrees to the North-South, N15E; and
- Option 3: the axis is almost perpendicular to the North-South, N80W.

After site inspection, the rock engineering expert has following comments:

- The criteria presented by the Contractor are appropriate;
- It is agreed with the Contractor about the stress condition can be neglected. In fact, there is no clear indication of the negative effect from the stress to the stability of the underground structures (tunnel, adits, and shaft). Thus, the effect from the in-situ stress to the choice of powerhouse axis can be neglected;

- It is appropriate to locate the powerhouse complex away from the fault zone. A distance of 80 to 85 m may be slightly conservative. The rock mass in the new location is observed to be "Good" to "Extremely Good" quality (evaluated based on the Q-system) and suitable for large span openings;
- We agree that the powerhouse axis should "make a large angle" (preferably perpendicular) to the strike of the main joint set to increase the stability of the caverns. This criterion is theoretically appropriate. However, this criterion should be considered only if the joint set creates a certain rock mass quality in different directions. At the new chosen location, the rock mass appears as massive with low frequency joints, and the rock mass is relatively isotropic. The rock mass can be seen in the Figure 3;

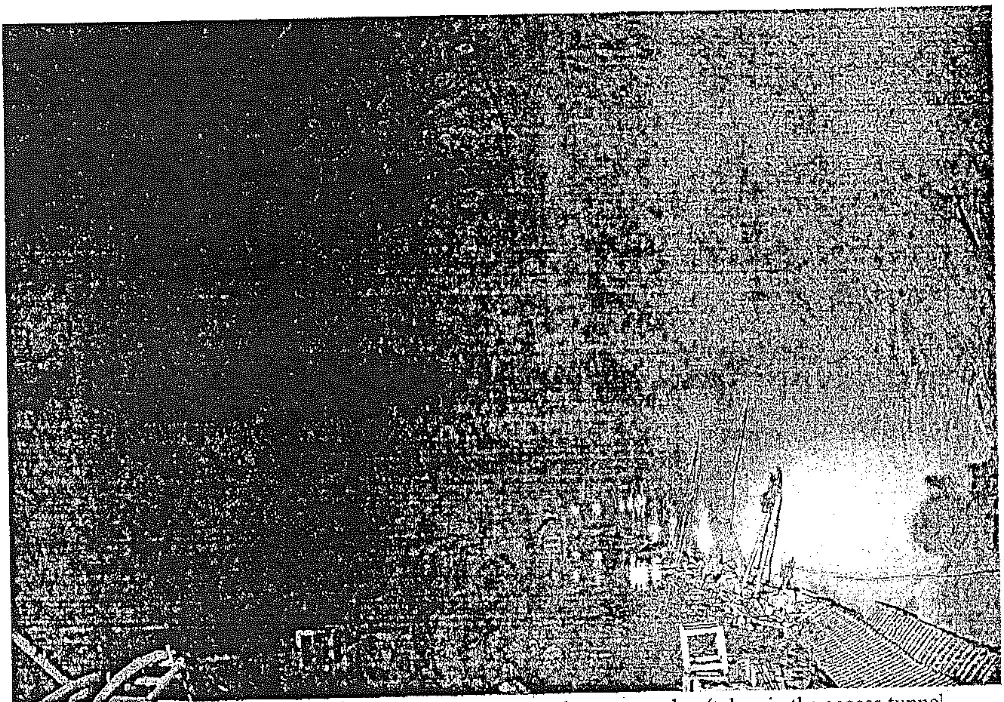

Figure 3: Rock mass quality at the new location of the powerhouse complex (taken in the access tunnel, looking out).

- The rock mass quality is found to be "Very Good" to "Extremely Good" based on the Q-system (Barton et al., 1974 and Barton 2002), and the Q-value is approximately 100. Thus, with 40 m span, the powerhouse can be supported with spot bolting or systematic bolting plus shotcrete if needed, as shown in Figure 4;

With the above considerations and especially the isotropy of the rock mass quality, we believe that the change of the powerhouse axis is not necessary. Thus, the powerhouse axis can be kept as original – option 1 (almost North-South) or option 2 (N15E). Option 3 (N80W) will create an unnecessary complicated situation for rearranging the underground structures.

General information

Ch.:	in the traffic tunnel, K1+745
Proj.:	Upper Kontum HPP
Loc.:	Kontum, Vietnam
Own.:	VSH
Ctr.:	Huadong
Csl.:	Huadong
Date:	26.07.2013
Geo.:	Nghia Trinh

Geometrical inputs

Span	=	40	m
ESR	=	1	
Equivalent dim.	=	40	m

Plotting

Q range	0.001	124.444
Span	40	40
Q range	124.4	124.4
Span	40	1

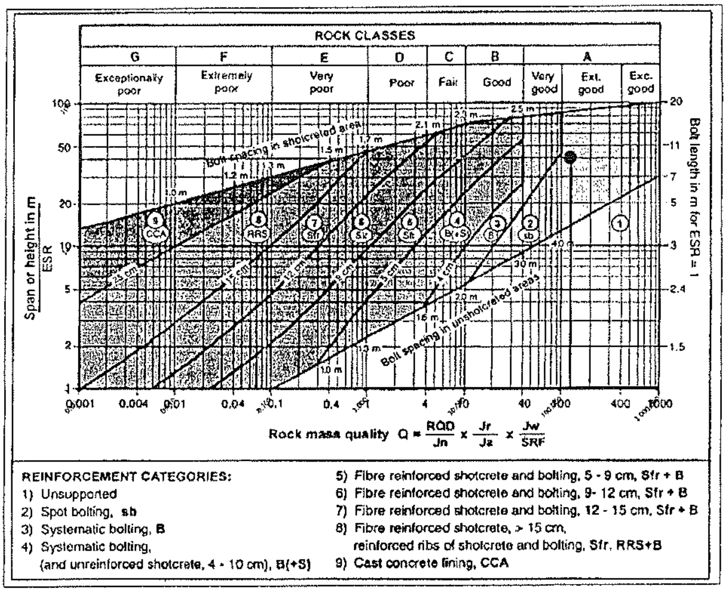

Figure 4: Recommended rock support for the powerhouse, according to the Q-system.

5 Preliminary comments on the in-situ stress measurements

The contractor has carried out stress measurements at 4 locations, including access tunnel, starting point of the planned steel lining section, adit tunnel, and headrace tunnel at the intake. Results of the stress measurements are presented in Table 1. Based on the measurements, the Contractor concludes that the stress situation is low. Thus, it is necessary to extent the steel lining section toward upstream, leading to a significant extension of the construction time and cost. After reviewing the stress measurement work, the rock engineering expert believes that the measurement does not meet the requirement due to the following reasons:

- The stress measurements do not meet one of its main purposes, which is to quantify the stress conditions in the "valley" area (around chainage K14+500). The stress conditions in this area are important for an evaluation of the tunnel route and adjustment if necessary. To serve such purpose, it was agreed during the technical design that stress measurements will be performed along with the TBM tunnelling. The stress measurements can be carried out from chainage K17+000 with certain interval toward upstream for obtaining reliable stress information for early considerations;
- The stress result at the starting point of steel lining was significant different from the other 3 results, giving an unfavourable condition for the steel lining design. Looking at Table 1, the third row from the bottom, the ratio between sigma z and "dead-weight stress", this ratio is around 1 in all the three other locations. The ratio is significant lower in the penstock location – only 0.53. After discussion with the Contractor through a meeting at their office, the Contractor mentioned that there could be a particular fault zone that affects the result. According to our investigation of the drill core, there is no observation of such discontinuity as described by the Contractor;
- In addition to that, the rock cover at that location is about 75% of the internal water pressure. Experience from many hydropower projects being built in Norway indicates that a rock cover of about 50% of the static water head would be sufficient to prevent the hydraulic fracturing. This is one more point to doubt the measurement result at this location;

- From the above discussion, the rock engineering expert is of the opinion that the measurement results may not be accurately representative for the in-situ rock stress condition. It is recommended to carry out additional stress measurements at locations decided by VSH, as well as considering using another company to do the test.

Table 1: Results of stress measurement carried out by the Contractor – July 2013.

Group No.	Access tunnel σJTD group	2# adit σ2HD group	Penstock 0-025.00m group	Penstock 0+290.00m group
Tunnel depth (m)	1+120.00m	0+539.60m	0-025.00m	0+290.00m
Burial depth (m)	348.9	228.1	450	330
Dead-weight stress γh (MPa)	9.07	5.93	10.92	8.32
Horizontal stress σ_x (MPa)	14.07	5.93	10.22	10.08
Horizontal stress σ_y (MPa)	9.85	6.45	7.56	6.51
Vertical stress σ_z (MPa)	7.58	5.10	5.75	9.02
σ_z/dead-weight stress γh	0.91	0.99	0.53	1.16
$(\sigma_x+\sigma_y)/2\sigma_z$	1.58	1.21	1.55	0.92
σ_3/dead-weight stress γh	0.63	0.80	0.51	0.55

6 Conclusion and recommendations

Through a thorough site investigation, the TBM and rock engineering experts have the following conclusions and recommendations:

- Technical condition of the TBM: the Robbins TBM MB 1612-285-2 equipped with a back-up system and the necessary auxiliary equipment for rock support and logistics is in line with the current state of technology for the excavation of long tunnels in hard rock. In order to improve the excavation rate, there are certain aspects that need to be taken care of, such as geological investigation to find out the cutter consumption and quality, contractor organisation and guidance, more training or more competence personnel for operation and maintenance team, availability of the spare parts, excavation and job-site logistic, extra cooling system for air and water in future;

- The largest problem causing the construction delay of the TBM tunnel now is the lacking of experience with the Contractor, and especially the Sub-contractor operating the TBM is critically unexperienced in the work. To solve this problem effectively, the VSH and especially the Project Management Unit must be strengthened to better deal with the situation. Such strengthening may be in policy, quantity, and quality of the personnel with good knowledge of the TBM operation. A hands-on TBM expert working closely with the team should be considered. From the change of the Owner side, an effective pressure can be created for the Contractor to be improved;
- Groundwater inflow in the 5 km intake tunnel: with limited capacity of equipment, pre-grouting is not possible. The proposed drain pipe and then sealing the leakage can be considered for testing. However, reference should be made to contractors in Vietnam, who have carried out the same work before for further investigation of the method, requirements, cost and effectiveness. Due to the contract, the Contractor at Upper Kontum HPP may not have financial motivation to seal the leakage;
- Location and orientation of the powerhouse complex: the new location of the powerhouse is in "Good" to "Extremely Good" rock mass quality. In addition, the rock mass quality is relatively isotropic. Thus, the orientation of the powerhouse can be kept as original or at 15 degrees as proposed by the Contractor;
- Stress measurements: the stress measurements carried out by the contractor does not provide enough information for evaluating the stress condition at the valley area (around chainage K14+500). The results of the measurement gave a significant lower value in one important location, comparing to other location. Stress result of this particular point heavily affects the construction cost and time of the project. With some evidences to question the accuracy of the result, it is recommended to measure stress again by an independent and competent company. The necessary stress measurements to evaluate the stress condition in the valley should also be carried out.

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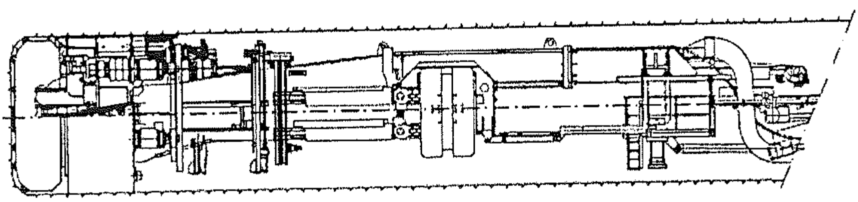
Vietnam – Thuong Kon Tum Hydropower Project

Summary of Site Visit Investigating the TBM and Back-up System

Site visit of 23. July to 26. July 2013

Report no.: AN2000/1142/R02



Regensdorf, 22. August 2013

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Content page

1. Project

The tunnel which is part of the Thuong Kon Tum Hydropower Project is about 17,548 m long. The stretch excavated by a TBM with a diameter of 4.53 m is estimated to be 13,085 m long. The first 420 m were excavated by drill-and-blast method. The counter-attack with a slope of 8% is realised by drill-and-blast, as well. Until now, about 2000 m of the counter-attack have already been excavated. The TBM advance started in November 2012, and on 23. July 2013 it reached Tm 1480. In parts, the back-up system has been completed and the continuous conveyor as well as the outside installation rails and primary ventilation have been installed. The geology consists of compact granite.

2. TBM and Back-up System

A rebuilt hard-rock high-performance open gripper TBM (Robbins model MB1612-285-2) with a diameter of 4.53 m as well as a back-up system and auxiliary equipment is used. The TBM was built in 1998 with a bore diameter of 4.94 m for the 10 km long tunnel of Metro West in the USA. Later, the diameter of the machine and its back-up system was modified to 4.5 m and the machine bored another 1 km long tunnel in the USA. Afterwards, the TBM and its back-up system were rebuilt and refurbished in the USA to be used for the Thuong Kon Tum Hydropower Project.

2.1. TBM

The Robbins TBM 160 series is a high-performance main-beam machine and has advanced by about 1060 m until 23. July 2013.

The machine is built with the best components such as:

- Main bearing by SKF (or Rothe Erde)
- 6 x 330 kW main drive motors by Elin (Germany)
- Gear reducers by Lohmann & Stolterfoht
- Hydraulic components by Rexroth and Robbins
- Cutter head for 17" diameter disc cutters designed and manufactured by Robbins
- 2 roof drills located behind the cutter head support (by Boart Longyear)

In Europe, rebuilt and refurbished Robbins TBMs have been used for building long tunnels without major problems.

2.1.1. Cutter Head

The cutter head has a typical Robbins design for hard-rock types and was in a very good condition on 23. July 2013. Mr. Nghia Quoc Trinh and Mr. Beat Lutiger inspected it on the job site. Some bended wearing plates have to be observed and critical ones have to be cut (risk of cutter damage).

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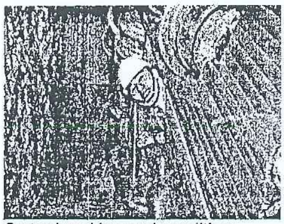
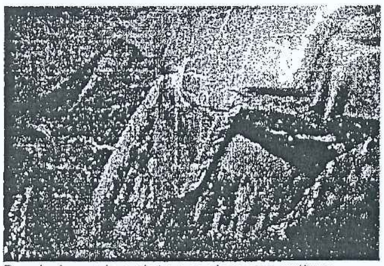
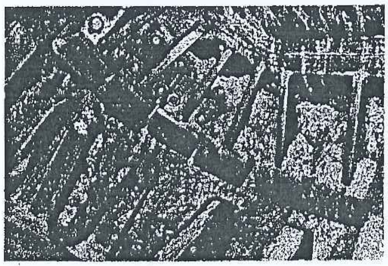


Cutter head in good condition





Bended wearing plate can damage cutters

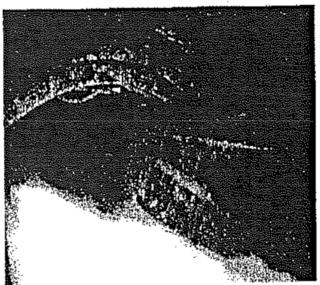
2.1.2. Cutter Head Drive Unit (VFD)

The main drive motors and gear reducers are water-cooled. The water flow rates and motor over-temperature is monitored and the information provided to the operator. Insufficient water flow rates or motor over-temperature will result in an automatic shutdown of the equipment. In order to guarantee a sufficient temperature gradient on the TBM for the cooling, cool enough or cooled water has to be pumped into the excavation process from the portal.

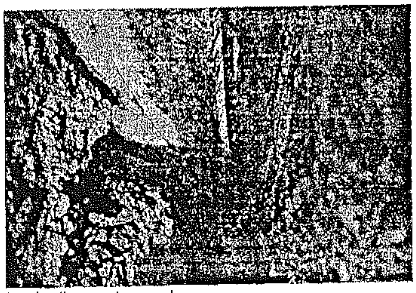
2.1.3. Lubrication System

The lubrication system provides automatic gear oil lubrication for the inner and outer seals, the main bearing, the bull gear and the drive pinions of the machine. The system provides automatic cycles to distribute the oil flow from the main bearing to the inner and outer seals in measured quantities.

The function of the lubrication system is ensured when there is some leaking oil on the inner and outer seal retainer rings.



Leaking oil on inner seal

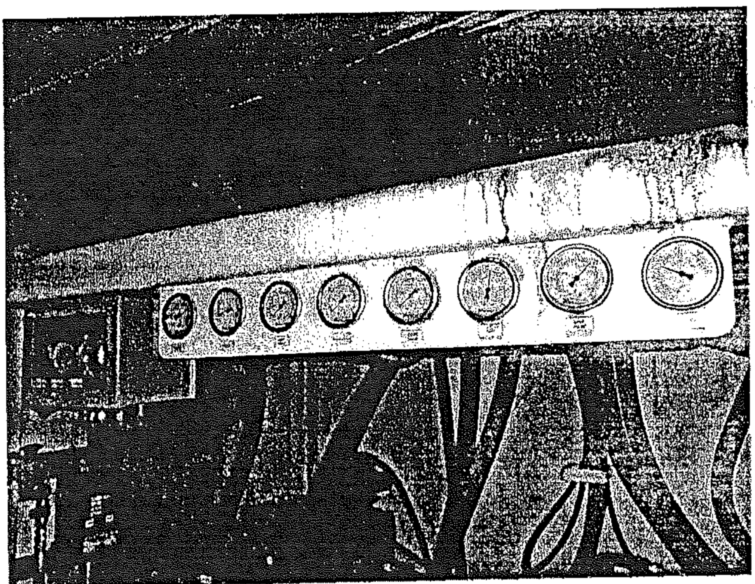


Leak oil on outer seal

2.1.4. Hydraulic Low-/High-Pressure System

High-quality industrial grade hydraulic components are used permanently. Most hydraulic valves are supplied by Rexroth or equivalent international suppliers of special hydraulic devices.

In the picture below, the gauges for the operating pressure of the hydraulic system can be seen.



Pressure gauge on TBM power pack

The major hydraulic circuit includes the gripper, thrust, steering/torque cylinders, and the conveyor drive system. Double-ended electric motor drive pumps are at each end of the motors.

Low-pressure / low-volume and high-pressure / high-volume pumps are incorporated. Variable and fixed volume pumps are used depending on the circuit requirements. Hydraulic logic components may be cartridge and/or sub-plate mounted.

The condition and the lay-out of the hydraulic aggregates consisting of pumps and drive motors and the interaction with the valves are all correct. The hydraulic system is water-cooled. The water flow rates and oil over-temperature are monitored and information is provided to the operator. Insufficient water flow rates or oil over-temperature will result in an automatic shutdown of the equipment. For this reason, it is important that the water coming from outside is sufficient and not too warm. Robbins has installed additional water coolers on the back-up.

The propel cylinders, thrust cylinders, the cylinders at the roof, side and rear support as well as the torque cylinders are in a very good condition and built according to Robbins standard.

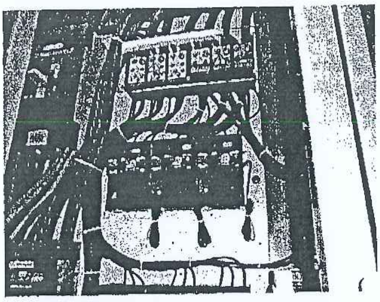
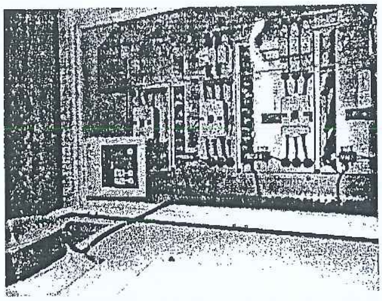
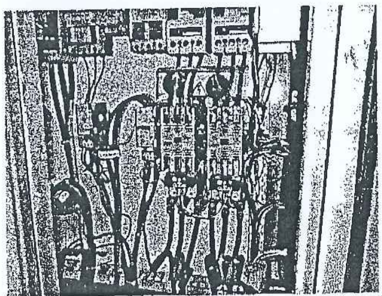
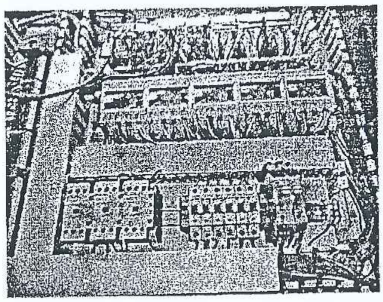

Extended propel cylinder


Roof support cylinder

2.1.5. Electrical Systems

Most of the installed electrical components are made by ABB, Eaton and Scae. The complete electrical system is neatly mounted in electrical cabinets on the back-up, built to IP55 standard, as it is Robbins standard. All added roof protection for the transformers, electrical cabinets and HT power switch are installed.

All motors of the main drive are started with VFD. An equipment monitoring device including a display at the operator's console is installed to enable the fast detection of problems and corrective action.

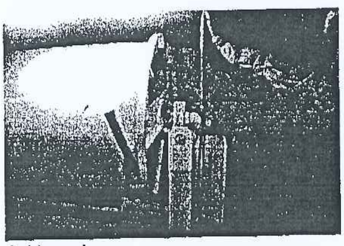
Examples: electrical components properly installed

2.1.6. TBM Conveyor and Muck Shut

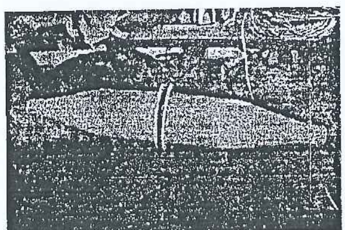
The belt from the TBM conveyor and the muck shut are in a good condition.

2.1.7. High-Voltage Cable Reel and HAT Cable Connection

A high-voltage cable reel with a capacity of 250 m of flexible high-voltage cable (3 x 50 + 3 x 25 -12/20) is mounted on the back-up. The high-voltage cable reel was broken at the time of the inspection. The contractor worked without cable reel.



Cable reel



Protection for HT-cable connecters

2.1.8. Cutter Change and Maintenance

The investigation of the cutter maintenance reports from 5. July 2013 to 13. July 2013 (Appendix 1) showed that many cutter changes happened on the approximately 1000 m advance. The wear of the cutter disc in the present granite is enormous. It cannot be verified if the contractor only uses original Robbins cutter discs. In long tunnels through granite (75–195 MPa), a capacity of 200–300 m^3/cutter disc is reached. In the current excavation for the Kon Tum Project, a capacity of 65 m^3 per cutter has been reached, which is an unusually low value. The high consumption cannot only be attributed to the hard rock.

The contractor proceeds with the change of the gage cutters as it is explained in the Robbins Appendix 2. The cutters are marked with different colours and thus are signed where they had been installed on the cutter head. During the repair in the cutter shop, the mechanic makes this mark. Too many of the cutters are blocked for the stretch of 1000 m. Probably, many of the blocked cutters (especially gage cutters on the positions no. 30 and no. 29) are damaged due to inadequate TBM steering. On the first 1000 m, there are many ridges or gouges in the bored tunnel. Most of them are the result of abrupt TBM corrections or bad operating. After a cutter change, the TBM has to be advanced very slowly to the drill face until all cutters are operative.

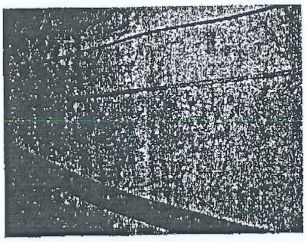
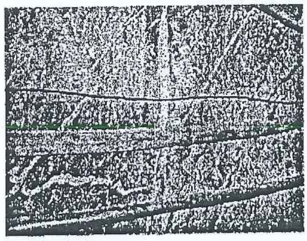
For a more precise analysis of the cutter consumption, a systematic exploration of the rock conducting laboratory tests with drill cores is recommended to identify the actual rock parameters.

2.1.9. Steering the TBM

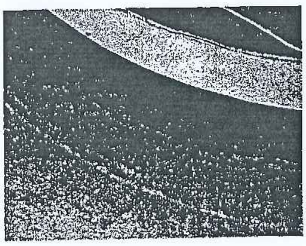
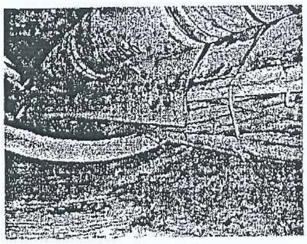
Steering the TBM in such hard granite demands great care from the TBM operator. The Robbins manual description has to be exactly followed. Corrections have to be made slowly and carefully. Otherwise, the outside gage cutters will be damaged and can only be used for less metres. In the appendix, the blocked outside cutters can be seen. If the outside cutters no. 30 or 29 have been blocked or worn to the limit, the TBM has to be pulled back and then, it is necessary to make an overcut in order to make space for the new cutters.

ROBBINS: *Proper steering can be defined as maintaining line a grade without noticeable variation in the tunnel walls due to the steering corrections. No ridges or gouges will be apparent in the walls if proper steering has been accomplished. If such ridges or gouges are visible, it is a very apparent indication that over-steering has occurred and action should be taken to prevent recurrence.*

ROBBINS: ***4-8.5 Limitation on Movements***
No steering correction, either vertically or horizontally, should be made that will result in displacing the gage cutter more than 1/8 inch (3 mm) at any one movement; and so on.

Ridges or gouges from over-steering and inappropriate operating.

Advance view: the machine has to be pulled back for the gage cutter change.
ATTENTION: in this manner, the cutter head supports risk to be damaged, especially the side-supports, with counter-pushing while the TBM advances.

2.2. Back-up Components

The back-up system uses a single-track rolling structure designed for use with tunnel conveyor muck removal.

The back-up consists of:

- Bridge structure assembled between TBM and back-up
- Shotcrete robot by ALIVA assembled at the bridge structure
- Bridge conveyor
- Rolling decks
- Track-laying area
- Towing system
- Monorail and material handling hoist
- Ventilation system
- Dust scrubber system

- Cable reel
- Utility piping

Walkways are maintained throughout the back-up for safe access and emergency exit. A muck car passage way through the back-up with adequate clearance exists, as well.

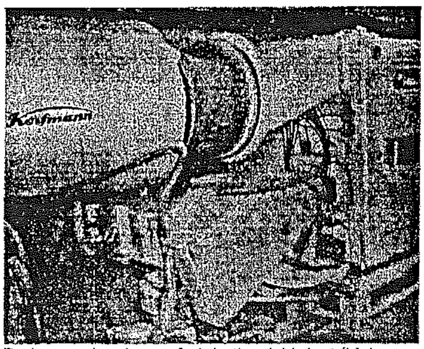
Throughout the back-up system, a space is maintained for the guidance system and laser.

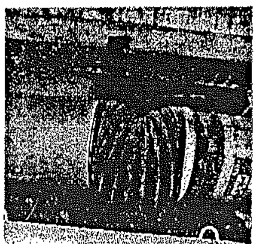
2.2.1. Secondary Ventilation System

There is enough overlap of the dust scrubber air vent line and the fresh air vent line. A booster fan (3 x 11 kW) for fresh air is mounted to push the fresh air from the main ventilation line. Exhausts are aligned to the TBM cutter head area. (The system can be seen in "Ventilation System Project Kon Tum".)

2.2.2. De-dusting System

The dry-type de-duster with fan (capacity of 250 m^3/min) and two silencers are installed along with the necessary rigid ducting from the TBM to the scrubber location on the back-up. The de-duster sucks air from the rear of the cutter head support zone and blows out clean air at the rear of the back-up. The de-duster capacity of 250 m^3/min meets the requirements of a bore diameter of 4.5 m. A correct and accurate installation as well as a good quality of the rigid ducts are important.





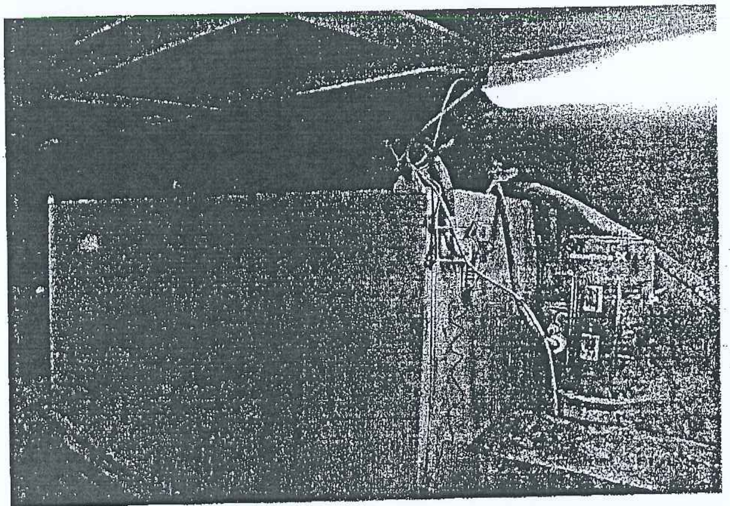
Bad examples: Loss of air in the rigid duct (higher quality to be installed)

2.2.3. Fresh-Air Ventilation (Primary)

The ventilation system is set up for use with "bag line" type vent ducting with a diameter of 1.5 m. The storage cassette is at the end of the back-up. A calculation and statement regarding the primary ventilation can be seen in chapter 7.

2.2.4. Water System

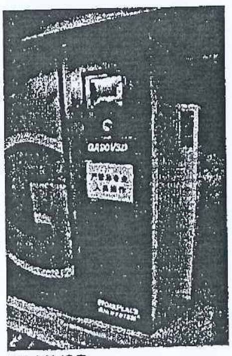
A fresh water tank (closed) with inlet float valve and a supply water pump with 8 kW is installed. This water system is approved on many back-ups.



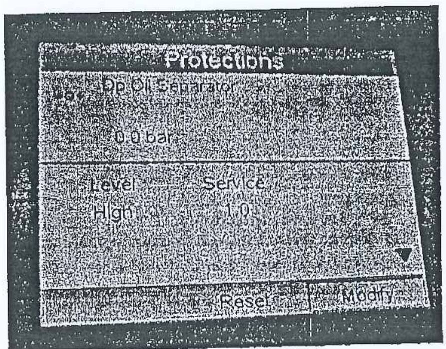
Closed water tank

2.2.5. Industrial Air

For the industrial air, two air-cooled GA90VSD atlas compressors by Atlas Copco, each with 16 m³/min (newest fabrication) are installed on the back-up. For the wet shotcrete application with one robot, the consumption is about 10 m³/min. One compressor can be on stand-by. The additional air pressure vessel is about 5 m³.



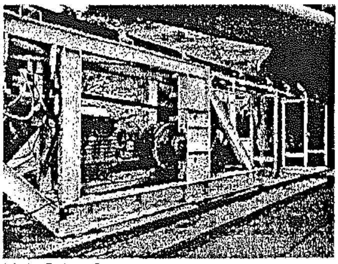
GA90VSD

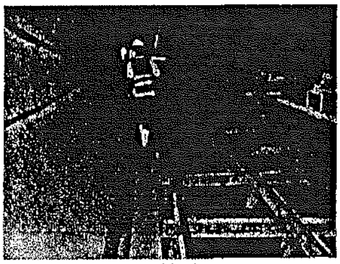


Error on display of one of the GA90

3. Continuous Belt Conveyor

The continuous belt conveyor with a belt width of 610 mm has a mucking capacity of 300 t/h. The belt storage cassette is manufactured for 500 m long steel cable belt ST 1600. The installed power of the main drive reaches 600 kW (2 x 300 kW) with VFD. It is planned to install an additional booster station for the whole tunnel length. Inside the tunnel, the conveyor steel structure is mounted by knee brace to the tunnel wall. The system of the continuous belt conveyor is very simple and has been approved on many projects all over the world.


Main Drive Conveyor


Structure is mounted by knee

4. Spare Parts

Taking into account the large transport distances and the long delivery times of individual components, it is very important to organise an adequate stock of spare parts. A good and efficient stock system is indispensable.

4.1. Major Spare Parts

The first category is the one of major spare parts which guard against long delays caused by failure of major items on the tunnelling system. At least the following major spare parts have to be supplied by Robbins and should be stored in the Robbins warehouse:

- Main bearing
- Ring gear

4.2. Minor Spare Parts

The second category is the one of minor spare parts required for normal operation. Robbins recommends that all normal-wear items, including one of all hydraulic and electrical components, bearings, wear rings, filter elements, seals, fittings, hoses, spray nozzles etc. be stored on site. At least the following minor spare parts have to be supplied by Robbins and be stored at the job site:

- Motor of cutter head main drive
- Gearbox of cutter head main drive

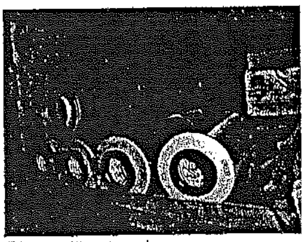
- Main thrust cylinder
- Misc. hydraulic components
- Misc. electrical components
- Wear items

The contractor has bought spare parts and additional spare parts are ordered (list in Appendix 3). Robbins has proposed consignment spares (list in Appendix 4).

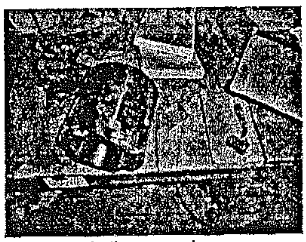
4.3. Stock of Spare Parts

The climate at the job site in Kon Tum is rather damp and warm. The spare parts remain stored for a long time. Therefore, it is recommended to wrap the spare parts in oil paper or to apply good anti-corrosive protection. Filter packages must not be opened. It should be taken care of proper disposal storage.

A hydraulic hose repair container has to be organised if this has not already been done.



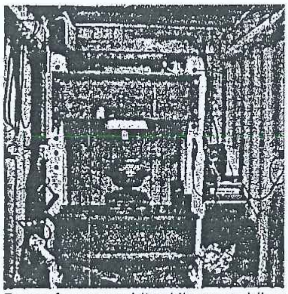
Filters without package

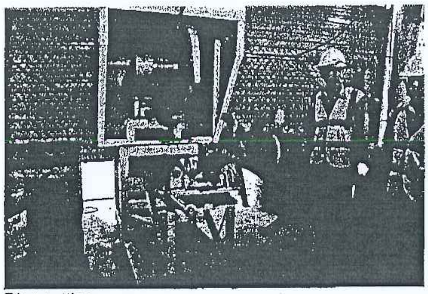


Rusty part in the magazine

5. Cutter Repair, Cutter Stock

For the cutter repair, the contractor installed an excellent cutter repair shop. The cutter specialist has the best tools, instruments and installations of Robbins available. As could be seen on the visit, the cutter repair and refurbishment is done in a professional way. The contractor uses original Robbins spare parts. As far as the cutter discs are concerned, it cannot be said for sure whether the contractor only uses the product from the USA, i.e. whether the discs have the best Robbins standard.


Press for assembling/disassembling

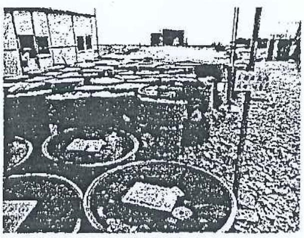

Disc cutting

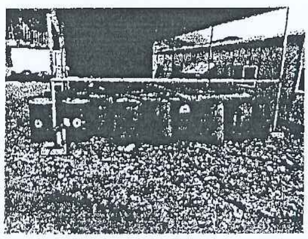
Stock of cutters ready for use: In order to ensure a continuous advance with the 4.5 m diameter TBM in granite, the stock of repaired cutters should amount to:

- 6 p of twin cutters
- 20 p of face cutters
- 30 p of gage cutters
- A good stock of parts, such as about 180 p of disc rings, 40 p of hubs, 60 p of covers, 40 p of axes, 80 p of bearings, 200 p of split rings etc. is indispensable.

6. Oil and Grease Stock

An adequate stock of grease and different oils for various purposes for the TBM and the back-up components is essential for such a project.




Stock of grease and oil

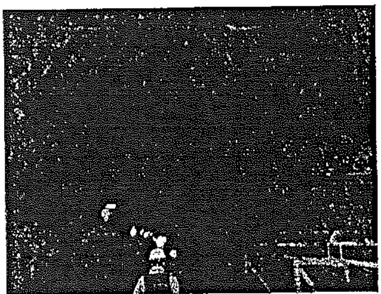
7. Fresh-Air Ventilation (Primary)

The primary ventilation system is set up for use with a "bag line" type vent ducting with a diameter of 1.5 m. Outside, 4 GIA fans of the type AVH 125.75.4.8, each with 75 kW, are installed in series.

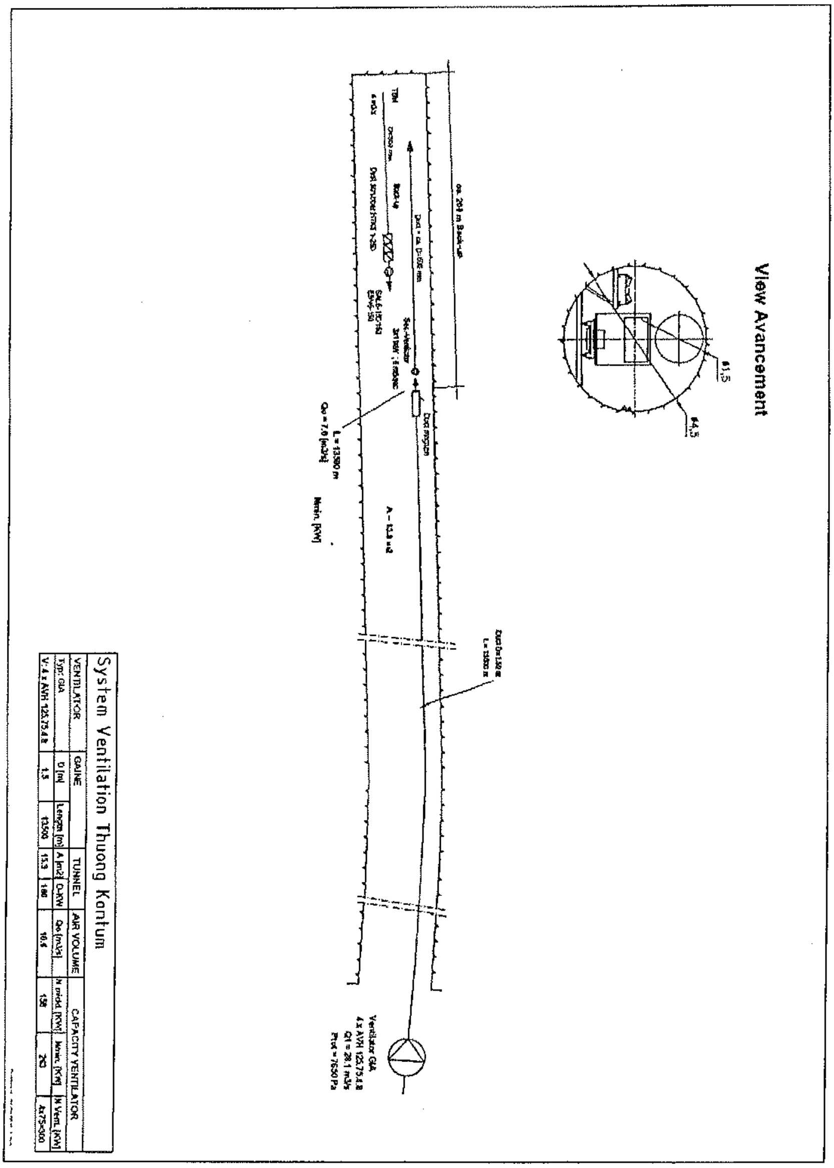
A scheme and the calculation of the primary ventilation system can be found below.

The installed ventilation system is sufficient for about 13,500 m. It is very important that the duct be mounted in a taut manner. It is recommended to reinstall the first 400 m of the duct to guarantee a straight alignment.

The specifications of the ducts with a diameter of 1.5 m and the fans can be seen in Appendix 5 and 6. The maximum allowed working pressure of the ducts is about 17,100 to 26,100 Pa. The ducts have to resist a working pressure of up to 9000 Pa.


Duct better aligned on the first 400 m

Kon Tum Project Ventilation System



AMBERG ENGINEERING

Calculation of Kon Tum Project Ventilation

	Input	Output
dust classification (A/B/X)	X	
friction coeff. A=0.018/B=0.024	.018	
leackage ar. f* A=10/B=20 [mm2/m2]	6	
site height [m]	535	
temperature [°C]	20	
air density [kgm-3]		1.14
air volume at front [cbm/s]	7.00	
duct length [m]	13500	
duct diameter ϕ [m]	1.50	
pressure at duct end [Pa]	100	
pressure factor $\pi 0$		11.20
L/D factor		9000.00
pressure factor $\pi 1$		829.32
loss factor $\Omega 1$		4.01
ventilator efficiency δ	0.80	
Zeta loss factor	0.70	
ventilator volume supply [cbm/s]		28.09
stat. pressure loss [Pa]		7407
dyn. pressure loss [Pa]		144
addit. losses [Pa]		101
total pressure loss [Pa]		7651
minimum inst. capacity [kW]		263

8. TBM Advance

Some operating reports of TBM 1612-285-2 dated between 10. July and 21. July were investigated. From this results that the TBM is operated as it is explained in the attachment 2 (Appendix 7) of the Robbins "comparison sheet of boring diameter and rock class in TBM excavation section of Thuong Kon Tum Hydropower Station". In this manner, the max. cutter load, the drive amperes and the cutter head rpm are optimised. For a stroke of 1.8 m, the TBM takes about 35 min. That makes a cutter penetration of about 5 mm per rpm. This rate is the optimum that can be reached in the granite of Thuong Kon Tum.

9. Summary Statement of the Expert

The excavation installation, which consists of a Robbins TBM MB 1612-285-2 equipped with a back-up system and the necessary auxiliary equipment for rock support and logistics is in line with the current state of technology for the excavation of long tunnels in hard rock. The logistic system made up of track-bound supply and evacuation with a continuous conveyor is also designed in an optimal way considering the boundary conditions of the project and guarantees sufficient transport capacity for high advance rates even for long transport distances.

Due to the high degree of mechanisation, the long transport distances to and on the job site, the chosen excavation installation implies great demands on the job-site organisation and the excavation team.

In order to reach high advance rates, the following points are crucial:

Geology

The in-situ rock affects the heading rates insofar as the rock strength limits the penetration and the abrasiveness accelerates the wear. The used TBM corresponds in its lay-out to the state of the art for driving the in-situ rock formations. Considering the abrasiveness of the present rock, the cutter head and the cutting tools or the cutting rings are subject to high wear and it is assumed that the change of the cutters and cutter head maintenance will continue to consume a lot of time. Therefore, it is all the more important to reduce the number of cutter changes by using exclusively high-quality cutting tools. It is recommended to use original disc cutters and discs only.

Organisation/Guidance

As opposed to a drill-and-blast advance, a highly mechanised excavation requires a tight organisation and guidance of the excavation and workshop personnel. The maintenance and logistic works to be done, the subsequent installation of the belt systems, the prolongation of the high-voltage cables and ducts, track maintenance etc. have to be planned with anticipation and the corresponding tasks have to be coordinated well.

Personnel

The machine operators have to be trained specifically for the machine in order to reduce standstills and repair costs caused by operating errors.

Availability of the Excavation Installation

On long excavation stretches, the high availability of the excavation installation can only be guaranteed by the consistent execution of maintenance works on the job site. The corresponding tasks have to be accomplished according to the instructions in the maintenance manuals. In particular, daily checks of the cutter head must not be omitted. These activities must not even be neglected when the excavation works fall behind construction schedule.

Excavation Logistics

With an increasing excavation length, the transport capacity of track-bound transport systems decreases. The timely planning and setup of alternative stations is necessary. The correct installation of the line inventory has to be verified.

Job-Site Logistics

Since the TBM is the only available cutting tool on site, a breakdown of the TBM causes a standstill of the overall project. Therefore, a reliable spare parts supply is a basic prerequisite. Besides the spare parts management, i.e. the timely ordering of spare parts, a correct storage according to instructions has to be guaranteed.

Usually, spare parts should be available as close to the job site as possible so that they can be transported to the site within the time needed for the removal of the required part.

Special Technical Aspects

The climatic excavation conditions have to be considered. With increasing excavation length and rock cover, a pre-cooling of the ventilation air at the portal or a cooling of the fresh air on the back-up shall be examined.

In order to ensure a sufficient cooling of the drive systems, an additional cooling of the water supply is recommended.

Authors: B. Lutiger / T. Murer

Enclosures:
Appendices 1 to 7

Cutter Inspection / Maintenance

Appendix: No. 1

HYDROCHINA HUADONG CR18G CONSORTIUM
(Composition B. Luliger)
13.01.2013 _ 13.07.2013

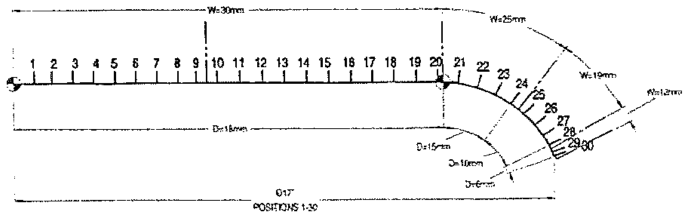
No.	Date	Chainage																															Meter	Cutter Changed
12	13.01.05	707.905	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position		29W, 28W, 27W, 26W, 25W, 24W, 23W
			0	0	10	0	0	0	0	0	0	5	11	24	24	25	25	20	26	24	20	19	15	11	9	0	10	0	10	0	7	0	Wear [mm]		22W, 21W, 20W,15W, 1/3W, 2/4W, 6/8W
13	13.01.07	727.673	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	19.768	2/4W
			1	0	14	0	2	1	2	2	5	9	14	25	25	27	29	20	27	27	21	20	16	13	11	0	10	0	0	0	0	5	Wear [mm]		
14	13.01.14	736.774	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	9.101	1 piece
			5	0	15	1	5	5	5	5	7	15	16	26	27	29	30	22	29	27	24	20	19	15	13	1	11	0	0	0	0	5	Wear [mm]		
15	13.01.15	740.455	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	3.681	16W
			5	1	15	1	5	5	5	6	7	15	19	27	28	29	20	24	29	28	25	20	19	15	13	1	12	0	0	0	0	9	Wear [mm]		
16	13.01.16	745.482	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	5.027	
			7	1	16	2	6	5	6	6	10	15	19	28	29	29	20	25	29	29	25	22	20	15	13	1	12	1	0	0	0	9	Wear [mm]		
16a	13.01.17	759.870	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	14.388	30W, 17W, 14W, 13W
			0	2	19	4	10	10	10	10	14	19	20	29	29	24	23	28	20	20	25	24	20	16	14	2	13	1	0	0	0	10	Wear [mm]		
17	13.01.19	767.707	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	7.837	30B, 29W, 28W, 19W, 18W
	13.01.20		5	0	11	5	11	10	13	12	15	20	21	23	18	25	20	26	20	24	26	24	21	18	14	3	13	1	0	0	0	10	Wear [mm]		
17a	13.01.21	786.165	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	18.458	
			7	0	14	6	15	15	15	17	20	24	25	25	21	25	23	30	24	25	28	25	22	18	15	5	14	1	1	1	1	10	Wear [mm]		
18	13.01.22	789.886	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	3.721	30W, 12W, 15W
			0	0	15	6	18	15	15	17	20	25	25	25	21	25	23	15	25	25	20	25	22	19	15	5	14	3	1	1	1	13	Wear [mm]		
18a	13.01.23	796.521	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	6.635	
			0	0	15	7	18	18	18	19	21	25	25	25	24	25	24	20	25	25	20	25	25	19	15	6	15	4	1	1	1	14	Wear [mm]		
19	13.01.24	805.919	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	9.398	27W, 26W, 25W, 24W
			0	1	15	0	0	0	0	0	0	0	15	25	25	25	24	21	25	25	20	25	25	19	15	6	15	5	1	1	1	14	Wear [mm]		23W, 22W, 21W, 20W
19a	13.01.25	811.964	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	6.045	
			2	1	20	0	0	0	0	0	0	0	15	27	25	25	25	21	25	25	20	25	25	20	19	7	15	5	2	2	1	14	Wear [mm]		
20	13.01.28	830.052	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	18.088	30B, 28W, 19W, 18W, 10W, 6/8W
			0	3	8	1	2	2	3	3	3	2	15	13	8	20	20	21	18	20	20	20	20	20	0	8	0	6	3	2	3	14	Wear [mm]		17W, 16W,14W, 13W, 11W,
20a	13.01.29	834.313	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	4.261	
			0	3	9	1	2	2	4	3	4	2	15	13	9	20	20	21	19	20	20	20	20	20	0	8	0	7	3	2	4	14	Wear [mm]		
20b	13.01.30	858.743	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	24.43	
			5	3	15	2	5	5	6	9	10	7	16	15	10	21	24	21	20	21	20	20	20	20	0	8	0	8	3	2	4	14	Wear [mm]		
21	13.02.01	869.641	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	10.898	28W, 22W
			6	4	10	3	5	5	6	9	0	7	17	16	11	21	25	21	21	21	20	20	20	20	0	9	0	8	4	2	4	14	Wear [mm]		
21b	13.02.02	891.989	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	22.348	30B, 16W
			0	4	15	5	10	10	10	13	4	10	20	20	14	25	11	21	24	24	24	20	20	20	0	10	0	8	5	2	5	14	Wear [mm]		
21c	13.02.05	899.778	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	7.789	
			1	5	15	5	14	11	14	15	6	10	20	20	15	25	13	23	25	25	24	20	20	20	0	10	0	8	5	2	5	14	Wear [mm]		
22	13.02.06	922.316	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	22.538	30B, 28W, 22W, 20W
			0	5	9	7	15	15	15	17	20	14	24	20	16	25	15	23	25	25	25	20	23	20	3	10	2	10	6	4	5	15	Wear [mm]		
23	13.02.08	948.693	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	26.377	30 pieces
24	13.02.14		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	Wear [mm]		
24a	13.02.15	965.719	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	17.026	
			1	0	4	0	4	2	4	4	3	4	1	1	1	0	0	0	1	0	0	0	1	0	0	0	0	0	0	0	0	0	Wear [mm]		
24b	13.02.16	978.468	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	12.749	
			4	0	11	2	6	5	6	6	6	5	6	4	3	3	4	5	5	2	2	1	1	1	0	0	0	0	0	0	0	3	Wear [mm]		
25	13.02.18	985.575	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	7.107	
			5	0	13	3	8	7	8	10	9	8	6	5	6	4	5	6	5	2	2	2	2	1	1	0	0	0	0	0	0	4	Wear [mm]		
25a	13.02.18	989.403	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	3.828	30B, 24W, 23W, 22W, 21W, 20W
	13.02.20		0	0	13	3	8	7	5	0	0	0	0	5	6	4	5	6	5	2	2	2	2	1	1	0	0	0	0	0	0	4	Wear [mm]		

25b	13.02.21	998.494	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	9.091	
			2	0	18	5	13	12	9	3	3	2	1	7	8	7	7	9	9	6	6	3	5	6	3	1	2	0	1	1	2	7	Wear [mm]		
25c	13.02.22	1016.241	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	17.747	
			5	2	19	5	15	14	11	5	6	4	3	8	10	9	7	11	9	10	6	5	6	6	4	1	2	0	2	1	2	7	Wear [mm]		
26	13.02.22	1024.908	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	8.667	30B, 5/7W
			0	3	20	6	15	15	11	5	6	5	3	9	10	9	8	11	10	10	6	5	6	6	4	0	3	0	2	1	2	7	Wear [mm]		
26a	13.02.23	1030.381	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	5.473	
			0	3	20	6	15	15	12	5	7	5	4	9	10	10	8	11	10	10	6	6	6	6	4	0	3	0	2	1	3	7	Wear [mm]		
27	13.02.23	1040.151	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	9.77	30B
			0	3	20	6	16	15	13	7	9	7	5	10	10	11	10	14	10	10	7	6	6	6	4	0	3	0	2	1	3	7	Wear [mm]		
28	13.02.26	1043.842	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	3.691	20W
			0	4	21	7	17	16	14	9	10	8	10	10	12	11	10	14	12	11	7	7	7	7	4	0	3	0	3	2	4	8	Wear [mm]		
28a	13.02.27	1052.133	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	8.291	27W, 28W, 30W
			0	4	8	8	19	18	14	10	13	9	11	12	13	13	12	15	12	13	9	8	8	7	4	0	3	0	3	2	4	8	Wear [mm]		
28b	13.02.28	1073.911	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	21.778	
			2	4	10	1	22	19	18	13	15	12	12	12	13	13	12	16	14	13	9	9	8	8	5	0	4	0	4	2	5	10	Wear [mm]		
28c	13.02.28	1081.728	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	7.817	
			4	4	13	3	22	20	18	13	15	12	16	13	14	15	13	17	15	14	10	9	9	8	5	1	4	0	6	2	5	11	Wear [mm]		
29	13.03.01	1091.569	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	9.841	
			4	5	19	4	23	21	20	15	18	14	14	14	15	16	14	18	16	17	11	10	10	8	6	1	5	1	6	2	6	11	Wear [mm]		
29a	13.03.02	1091.569	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	0	22A, 23W, 24W, 25W, 26W,
			4	0	10	4	0	0	0	5	8	14	14	14	15	16	14	18	16	17	11	10	10	8	6	1	5	1	6	2	6	11	Wear [mm]		28A, 29W
29b	13.03.04	1095.687	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	4.118	
29c			4	0	10	4	0	0	0	5	10	14	14	14	15	16	14	18	16	17	11	10	10	9	6	1	5	1	6	2	6	11	Wear [mm]		
29c	13.03.05	1112.084	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	16.397	
			6	0	10	6	2	2	2	8	14	17	16	15	17	16	16	19	17	17	12	12	12	10	7	2	6	1	6	3	7	13	Wear [mm]		
29d	13.03.05	1116.536	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	4.452	
			6	0	10	6	3	3	3	8	15	17	17	15	17	17	16	19	17	18	14	12	12	10	7	2	6	1	6	3	7	13	Wear [mm]		
30	13.03.06	1129.876	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	13.34	28A, 30B
			0	2	9	7	8	6	6	10	19	20	19	18	19	20	17	20	18	18	14	12	13	11	7	2	6	2	7	3	7	14	Wear [mm]		
30a	13.03.07	1138.697	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	8.821	17W
			0	2	11	7	9	8	8	13	20	20	20	18	20	20	18	22	18	19	15	15	13	11	8	4	6	2	8	5	8	15	Wear [mm]		
31	13.03.07	1150.391	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	11.694	28W, 1/3A, 6/7W
			1	2	20	8	11	9	9	15	20	22	23	19	21	20	19	23	20	19	15	15	13	11	9	0	6	0	8	5	9	15	Wear [mm]		
32	13.03.08	1160.160	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	9.769	24N, 30A
			16	2	20	8	12	10	10	15	22	22	23	20	21	20	19	23	20	19	15	15	14	13	9	0	6	0	8	7	9	6	Wear [mm]		
33	13.03.09	1172.323	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	12.163	30A
			0	2	20	9	14	13	17	20	25	24	23	20	20	19	20	24	21	20	16	15	15	13	7	0	6	0	8	7	9	8	Wear [mm]		
33a	13.03.10	1178.688	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	6.365	
			3	3	20	10	16	16	20	20	26	24	26	21	24	23	23	26	22	20	18	23	16	18	9	2	7	2	8	8	9	9	Wear [mm]		
34	13.03.11	1181.232	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	2.544	
			3	3	20	10	16	16	20	20	26	24	26	21	24	23	23	26	22	20	18	23	16	18	9	2	7	2	8	8	9	9	Wear [mm]		
34a	13.03.11	1187.502	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	6.27	
			17	4	24	11	20	18	22	26	31	29	29	24	24	24	23	27	24	23	19	18	18	18	11	2	8	2	10	9	9	13	Wear [mm]		
35	13.03.14	1187.502	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	0	30C, 29W, 28W, 27W, 26W, 25W
			0	0	11	0	0	0	0	0	0	0	21	24	24	24	23	21	24	23	19	18	18	18	11	2	8	2	10	9	9	13	Wear [mm]		24W, 23W, 22W,21W, 20W, 15W
35a	13.03.15	1192.864	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	5.362	
			2	1	12	1	0	1	2	1	2	2	21	24	26	24	24	20	26	26	21	18	17	16	11	3	8	4	11	8	11	20	Wear [mm]		
36	13.03.15	1199.745	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	6.881	1A,3A, 5A,7A
			3	1	12	2	4	3	5	6	5	5	22	24	28	25	25	23	26	27	21	20	21	16	12	5	8	4	10	9	12	27	Wear [mm]		
37	13.03.16	1205.340	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	5.595	
			5	4	14	7	7	5	7	7	8	7	23	27	29	26	25	24	26	27	23	23	26	17	12	5	8	5	10	9	12	17	Wear [mm]		
37a	13.03.17	1211.007	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	5.667	30W, 29W, 17W, 18W
	13.03.18		1	0	16	7	11	8	12	12	14	12	24	27	30	31	27	25	27	27	23	23	26	18	13	5	10	6	13	1	13	18	Wear [mm]		1A,3A, 2A,4A

38	13.03.23	1227.470	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	16.463	
	13.03.24		9	0	20	7	16	26	17	21	22	20	28	32	23	20	31	30	32	31	27	25	28	23	14	8	10	10	1	6	2	10	Wear [mm]		
38a	13.03.25	1223.272	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	12.265	13W, 14W, 15W, 16W,17A, 19W
		Wear	0	0	0	0	0	0	0	0	0	0	28	0	23	0	0	0	0	0	27	25	28	23	14	8	10	10	1	6	2	10	Wear [mm]		21W-28W, 30W, (15p)
38b	13.03.26	1226.620	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	3.348	
			5	1	11	1	2	1	2	2	2	2	11	17	24	23	21	18	16	22	27	26	27	23	15	8	10	10	2	6	2	10	Wear [mm]		
39	13.03.27	1234.357	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	7.737	5B,7B, 18N
			6	1	14	1	5	5	4	5	5	6	13	20	17	24	21	15	18	23	29	26	26	23	16	10	11	10	2	7	2	10	Wear [mm]		
40	13.03.28	1244.788	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	10.431	11N, 12W, 29O, 30W
			2	0	16	3	9	8	8	9	10	10	15	22	20	26	23	21	21	26	20	15	26	23	17	1	13	1	3	8	3	11	Wear [mm]		
41	13.03.30	1263.642	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	18.854	2A,4A, 19N
			8	2	16	3	12	16	12	15	16	15	17	10	20	29	26	23	23	27	21	16	29	25	17	2	13	2	5	8	4	11	Wear [mm]		
42	13.03.31	1289.918	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	26.276	30W, 29O, 11A
			2	0	16	5	14	16	13	16	17	15	20	10	22	29	27	23	23	27	21	10	29	25	17	2	13	2	15	8	10	11	Wear [mm]		
43	13.04.01	1298.601	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	8.683	28W, 27W, 26W, 25W, 24W, 23W, 22W, 21W,
			2	0	16	5	14	16	13	16	17	15	20	10	22	29	27	23	23	27	21	10	29	25	17	2	13	2	15	8	10	11	Wear [mm]		20W 17W, 18W, 13W, 11C,10W, 9W, 6/8O
44	13.04.04	1303.389	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	4.788	6/8O
			4	0	11	0	1	1	1	1	2	1	12	11	22	20	16	26	24	20	22	23	17	16	18	3	23	2	15	10	10	12	Wear [mm]		
45	13.04.04	1329.685	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	26.296	3 Pc
			6	2	13	2	3	3	3	3	4	3	14	13	24	20	18	26	24	23	22	23	17	17	18	3	23	2	15	10	10	12	Wear [mm]		
46	13.04.13	1333.296	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	29.907	19N, 30W
			0	1	18	2	10	12	13	12	14	11	18	20	26	20	21	15	20	24	25	26	20	18	1	0	1	0	13	10	12	14	Wear [mm]		
47	13.04.20	1341.968	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	8.672	1AC,3AC
			3	1	19	2	11	14	14	14	16	13	20	20	27	21	22	18	20	25	27	27	20	22	2	1	1	1	15	5	14	6	Wear [mm]		
48	13.04.23	1351.219	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	9.251	1AC,3AC, 2O,4O, 12W, 18W, 28W
49	13.04.24		0	2	10	4	13	15	16	16	18	14	21	20	28	23	22	19	21	25	28	25	20	22	2	1	1	1	13	15	5	14	Wear [mm]		30W
50	13.04.25	1371.336	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	20.117	
			1	2	13	5	16	18	20	19	21	17	22	23	24	25	22	20	22	26	28	27	22	22	2	1	2	1	13	15	5	14	Wear [mm]		
51	13.07.08	1384.636	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	13.3	W28, W27, W26, W25, W24, W23,
			2	1	6	1	1	7	1	3	8	12	24	22	23	20	18	22	24	20	21	11	23	20	4	2	2	2	10	16	5	15	Wear [mm]		W22, W21
52	13.07.10	1388.638	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	4.002	
			4	1	8	1	2	2	2	4	10	12	25	24	23	20	19	22	24	20	21	13	23	23	4	2	3	2	10	16	5	15	Wear [mm]		
53	13.07.11	1393.154	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	4.516	
			5	1	9	3	4	4	5	5	10	12	25	24	23	20	20	22	25	21	21	14	23	19	5	3	4	2	11	16	5	15	Wear [mm]		
54	13.07.12	1405.813	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	12.659	
			6	1	11	4	5	5	5	7	11	16	26	25	24	20	20	23	26	22	20	15	25	19	7	3	4	2	11	16	8	15	Wear [mm]		
55	13.07.13	1410.152	30	29	28	27	26	25	24	23	22	21	20	19	18	17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	Position	4.339	
			6	1	12	4	5	6	6	8	13	16	27	25	25	22	20	25	26	25	20	15	25	20	7	4	4	3	13	16	8	15	Wear [mm]		

APPENDIX: No. 2

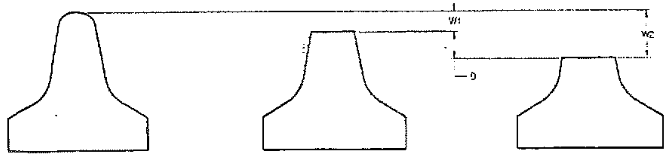
CUTTER CHANGING GUIDELINE
(TO BE USED IN CONJUNCTION WITH WRITTEN INSTRUCTIONS 5107 6028 58)
SEE SHEET 2

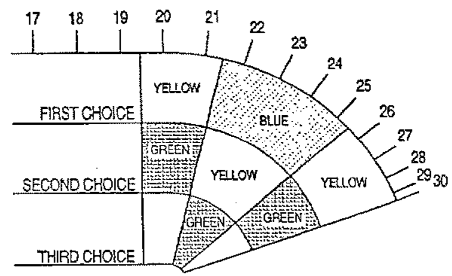


IT IS RECOMMENDED THAT CUTTERS BE CHANGED IN GROUPS TO AVOID EXCESSIVE LOADS ON INDIVIDUAL ASSEBMBLIES AND TO MAKE THE MOST EFFICIENT USE OF CUTTER CHANGING TIME. IF A CUTTER BLOCKS OR CHIPS PREMATURELY, IT IS RECOMMENDED TO REPLACE IT WITH ONE THAT IS PARTIALLY WORN TO AVOID EXCESSIVE WEAR BETWEEN RINGS. IF THE LIMITS OF DIMENSIONS "D" ARE EXCEEDED IT COULD CAUSE A FAILURE AND POSSIBLE WIPEOUT OF MULTIPLE ASSEMBLIES.

BEARING COLOR CODE LEGEND:
BLUE= NEW BEARING
YELLOW = SECOND RING PER BEARING
GREEN = THIRD OR MORE RING PER BEARING

EACH TIME A CUTTER IS CHANGED OUT OF POSITION 25-30 THE WORN CUTTER SHOULD BE MOVED TO ANOTHER INNER POSITION WHERE THE WEAR LIMITS ARE GREATER.





TITLE	CUTTER CHANGE GUIDE, VIETNAM
SIZE	D
DWG NO.	1058513
REV	2
SHEET	1 of 2
SCALE	1:4

Robbins

Cutter Changing Guideline – 5107 6028 58 Rev. 2

Robbins

Date of Issue: 29 March 2001

Date of Revision:

R1: 23 May 2008

R2: 22 July 2011

Approved: 23 May 2008

Robbins has established cutter change guidelines to maintain the optimum dress of cutters on the cutterhead, maximize cutter life, and to increase machine availability for boring. The following guidelines are to be used to determine which cutter positions to change:

1. When inspecting cutters, first measure and record the cutter wear on all cutters using cutter wear gage tools. Assess the wear across the entire cutterhead. Cutter wear gage tools for standard rings are as follows:
   - a. 17" Ring AM1724-1 — Wear Gage B96575-1
   - b. 17" Ring B70458-5 — Wear Gage B66117-2
   - c. 19" Ring 5107602255 — Wear Gage B48242-1 Rev. B
   - d. 20" Ring B97504-1 — Wear Gage B104424-1

   For rings not listed here, please contact The Robbins Company for the correct wear gage.
2. Always remove a cutter from its position when any of the following conditions exist:
   - a. Excessive ring wear. It is important to not allow cutter disc wear to exceed the limits as shown on Cutter Changing Guideline drawing. This causes the cutter to experience excessive loads and can cause premature bearing damage. Excessive wear in the gage positions can cause the machine to become stuck, or damage bucket lips.
   - b. Excessive wear between adjacent cutters: large variations in wear between adjacent cutters cause the rock face to have peaks and valleys. Cutters in "valley" positions have experienced excessive loads and premature bearing failure. Adhere to the limits shown on the Cutter Changing Guideline drawing.
   - c. Flat spots or broken ring.
   - d. Rough rotation is an indication of bearing failure.
   - e. Oil or grease leakage: remove cutter and send to cutter repair shop to determine cause of oil or grease leak. Do not attempt to refill cutter while on the cutterhead.

The Robbins Company Locations: Solon, OH • Kent, WA
5866 S. 194th St. • Kent, WA 98032 U.S.A
Phone: (253) 872-0500 • Fax: (253) 872-0199

Cutter Changing Guideline – 5107 6028 58 Rev. 2

Robbins

3. Cutters in the transition positions experience higher loads than any other location on the cutterhead. For this reason, it is best to put cutters with the least worn bearings in the transition positions. To identify the cutters according to bearing wear, a color code has been established as follows:
   - a. Blue: new bearing.
   - b. Yellow: bearing has seen one ring and now has second ring installed.
   - c. Green: bearing has seen two rings and now has third ring installed.
   - d. No color: bearing has seen at least four rings.

   These colors must be painted on the cutters in the cutter shop. Use this color code to install cutters on the cutterhead as shown on the Cutter Changing Guideline drawing.
4. It is recommended to change cutters in groups so as to avoid excessive loads on individual cutters and to make the most efficient use of cutter changing time.
5. Should one cutter block or stop turning prematurely, it must be replaced immediately to avoid catastrophic failure (wipeout) of adjacent cutters. It is recommended to replace the cutter with a partially worn cutter that is reserved for that purpose. The worn cutter should have similar wear to the adjacent cutters so it complies with limits on wear difference between adjacent cutters. These limits vary by position.
6. Whenever a cutter requires changing before its wear limit has been exceeded, this cutter can be re-installed in an area of the cutterhead where the wear limits are greater, always complying with the limit on wear difference between adjacent cutters.
7. Installing cutters
   - a. Clean the mating surfaces of the cutter and cutter housing. Surfaces to be free of foreign material. Liberal use of water spray and wire brushing is required.
   - b. Clean all threads so they are free of any foreign material.
   - c. Lightly oil threaded fasteners with SAE 30 wt. oil and install cutter and mounting hardware.
   - d. Tighten M30 Class 10.9 fasteners to 1734 N·m (1279 ft·lb) & M24 Class 10.9 fasteners to 872 N·m (643 ft·lb) using either a hydraulic or manual torque wrench. Air wrenches are only to be used for initial tightening.

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CUTTER CHANGE GUIDE, VIETNAM

D 1058513 2

1:1

Robbins

Appendix: No. 3

No1 Spare Parts List (arrived at jobsit)				
S/N 序号	DESCRIPTION 品名	PART NUMBER 零件号	QTY 数量	REMARK 备注
1	ROLLER BEARING	0506010006	1	in using
2	O-RING	0663210150	2	√
3	ADAPTER SLEEVE	0517221100	3	in using
4	DISC FOR BRAKE ASSY	3035032875	1	√
5	BEARING HOUSING	3035330175	1	√
6	SEAL RING	303530176	1	√
7	COVER	3035330177	1	√
8	TORSION SHAFT	5750000241	1	√
9	SPEED SWITCH	5750000455	1	√
10	HUB	5750000923	1	√
11	FILTER ELEMENT	5750001361	6	0
12	FILTER ELEMENT	5750001362	6	5
13	VANE PUMP	5750010366	1	√
14	DOUBLE VANE PUMP	5750010367	1	√
15	DOUBLE VANE PUMP	5750010368	1	in using
16	FILTER ELEMENT	5750010379	2	√
17	FLOW/METER SWITCH	5750011633	1	√
18	DIRECTIONAL VALVE	5750012187	1	√
19	SEAL KIT FOR 5750110234	5750110234	1	√
20	RETURN IDLER 30"	A39908-4	3	√
21	FILTER ELEMENT	A65368-1	24	0
22	FILTER ELEMENT	A65368-2	12	0
23	POWER SUPPLY 24VDC	A78734-1	2	0
24	SEAL KIT FOR 5750110234	A121599-1	1	√
25	BREAK PADS	3035032874-2	4	√
26	PUMP VARIABLE VOLUME	5750010349	1	0
27	PUMP VARIABLE VOLUME	5750010349	1	0
28	FLOW METER/ SWITCH	A105916-1	1	√
29	FLOW METER/ SWITCH	A105917-1	1	0
30	STUD,MAIN BEARING M36X440	5750007767	33	√
31	STUD, RING GEAR M36X480	5750007768	21	√
32	STUD,CUTTERHEAD,M36X750	5750007769	26	√
33	BEARING, ROLLER (A14701-3)	0506010006	1	√
34				
35	VALVE, DIRECTIONAL CONTROL	5750012188	1	√
36	YOKE SET - FOR BRAKE ASSEMBLY	3035032874	1	0
37	PUMP LUBE RETURN	5750013503	1	0
38	SAFETY RELAY	1028204	1	√
39	STARTER COMBO	1028903	1	√
40	STARTER COMBO	1028904	1	√
41	STARTER COMBO	1029001	1	√
42	STARTER,70-100A	A108214-11	1	√
43	STARTER, 4-6A	A108214-4	1	√
44	STARTER,24-40A	A108214-8	1	√
45	STARTER,35-50A	A108214-9	1	0
46	DIGITAL INPUT MODULE	A116347-1	2	√
47	DIGITAL OUTPUT MODULE	A116347-2	2	√

Appendix: No. 3

No1 Spare Parts List (arrived at jobsit)				
S/N 序号	DESCRIPTION 品名	PART NUMBER 零件号	QTY 数量	REMARK 备注
48	BUSNODE MODULE	A116347-3	1	√
49	ANALOG OUTPUT MODULE	A116347-5	1	0
50	SOCKET W/HOLD DOWN CLIP	A34227-11	2	√
51	RELAY 24VDC	A34227-14	2	√
52	TRANSDUCER 5A	A62125-4	1	0
53	SAFETY RELAY EXPANDER	1028332	1	√
54	POWER SUPPLY 24VDC	A79007-8	1	√
55	CONTRACTOR 4 POLE 24VDC	A95052-1	2	√
56	FUSE	A99193-8	3	√
57	TIME DELAY FUSE	A99508-12	5	√
58	TIME DELAY FUSE	A99508-28	13	√
59	TIME DELAY FUSE	A99508-32	5	√
60	INDICATING LIGHT RED	A99761-8	2	√
61	ANALOG INPUT MODULE CUBE 20	A116347-4	1	0
62				
63	COMPUTER, INDUSTRIAL PROFACE 8GB	B118660-1	1	√
64	CCTV PROC/DVR,CHINA	A119721-1	2	√
65	POTENTIMETER CONVERTER	A101396-1	1	√
66	LCD TOUCHSCREEN MONITOR	1001286	1	√
67	DIRECTIONAL VALVE	A113320-1	1	√
68	DIRECTIONA VALVE	A60000-4	1	√
69	DIRECTIONAL VALVE	5750012193	1	√
70	PROP. PRESS RE/REL VALVE	2000456	1	0
71	PRESSURE TRANSDUCER	2003433	1	√
72	C-TIP	D120135-15	15	0
73	POWERFLEX CUSHION	D120135-16	1	0
74	TENSIONING SPRING	D120135-17	2	√
75	30" POLE	D120135-7	1	√
76	SEAL SET	DM1173-21	2	√
77	FLAGE HOUSING	5750000393	2	0
78	FELT STRIP	5750000670	2	√
79	SHCS M24X110	9126719500	2	0
80	PIPE PLUG	653-933	4	√
81	HYDRAULIC MOTOR	A92266-1	1	√
82	HHCS M30X320 GR 10.9	A22151-83208	20	√
83	HHCS M24X60 GR10.9	A22152-70608	8	√
84	PLAIN HEX NUT , M30 GR10	A22154-88	40	√
85	HARDWASHER M30	A22158-8	20	√
86	RUBBER SLEEVE	B76469-1	20	0
87	SPACER, WELDMENT	1033736	4	0
88	WEDGE 17" B.L.W.L.HOUSING	5107602225	20	7
89	CLAMP BLOCK 17" DIA. B.L.W.L.	B76468-1	20	7
90	WEDGE,17" M24 (5107602057)	C65665-1	12	0
91	HHCA,M24*200 GR10.9	A22151-72008	12	√
92	FILTER ELEMENT	5750001826	6	√
93	FLOW DIVIDER	5750010908	1	√
94	FLOW METER/ SWITCH	A105915-1	1	√

Appendix: No. 3

No1 Spare Parts List (arrived at jobsit)				
S/N 序号	DESCRIPTION 品名	PART NUMBER 零件号	QTY 数量	REMARK 备注
95	INTERNAL GEAR PUMP	2003258	1	√
96	ELEMENT, FILTER	5750005022	3	1
97				
98	AIR BREATHER	A94481-1	6	√
99	FILTER ELEMENT	A106117-2	6	√
100	ELECTRIC MOTOR 330KW	2004238	1	√
101	BEARING	0506211800	2	√
102	SAFESET PUMP	1026128	1	0

CAT.	PART NUMBER	DESCRIPTION	QTY.	REMARK
B	CUTTERHEAD ASSY			pro. Lub
	H01085-200025	PIN, SPRING 20 x 25mm	12	
	1033736	SPACER, WELDMENT	4	√
	A18182-1	SPRAY NOZZLE	10	15
	A110088-1	MONIFLOW UNION（ROTATION UNION)	1	2
C	MAIN BEARING AND SEAL ASSY			
	5750007765	OUTER SEAL	3	√
	5750007766	INNER SEAL	2	√
E	MAIN DRIVE ASSY			
	5750010086	GEAR REDUCER	1	√
	5750000241	TORSION SHAFT	1	√
	D110056-1	COUPLING ASSEMBLY	1	√
	D109727-1	SHEAR TUBE		
	2004546	SHEAR TUBE		
	3035300352	SHEAR TUBE	20	√
	3035032874	YOKE SET	1	√
	1026128	SAFESET PUMP	1	√
	A120103-1	EMI FILTER	2	√
F	GRIPPER, TORQUE, PROPEL ASSY			
	5750110232	SEAL KIT, PROPEL CYLINDER	1	√
	A121498-1	SEAL KIT, GRIPPER CYLINDER	1	√
	A62346-1	WEAR STRIPS	8	√
H	REAR LEG ASSY			
I	CONVEYOR ASSY			
	A39907-24	TROUGHING IDLER, 30"	8	√
	5750000455	SPEED SWITCH	1	2
	5750000456	WHEEL ASSEMBLY, SPEED SWITCH	1	2
	506010006	ROLLER BEARING	3	√
	A92266-1	HYDRAULIC MOTOR	1	√
	5750000417	scraper plow	4	√
	D120124-30S	SECONDARY BELT SCRAPER	8	√
	D120135-15	C-TIP	15	√
	D120135-16	POWERFLEX CUSHION	1	√
	A47667-5	BELT, CONVEYOR 30" x 163'	1	√
	A79833-7	TAKE-UP BEARING	1	√
J	LUBE SCHEMATIC			
	2003258	PUMP INTERNAL GEAR	1	√
	5750013503	PUMP, LUBE RETURN	1	2
	A105915-1	FLOWMETER/SWITCH	1	√

CAT.	PART NUMBER	DESCRIPTION	QTY.	REMARK
	A105916-1	FLOWMETER/SWITCH	1	√
	5750001826	FILTER ELEMENT	6	√
	5750001405	FILTER ELEMENT	16	√
	5750005022	FILTER ELEMENT	9	√
	5750001825	PRESSER FILTER	2	√
	5750010908	FLOW DIVIDER	1	√
	5750010348	PRESSURE RELIEF VALVE	4	√
	A69002-3	PRESSURE RELIEF VALVE	2	√
	2003344	PRESSURE TRANSDUCER	2	√
	5750010916	PROXIMITY SWITCH GREASE	1	√
	5750001609	PROXIMITY SWITCH SEAL LUBE	1	√
K	HYDRAULIC SHCEMATIC			
	5750010349	PUMP	1	√
	5750010366	PUMP, VANE	1	√
	5750010368	PUMP DOUBLE VANE	1	√
	5750010367	PUMP DOUBLE VANE	1	√
	A104905-1	PROPORTIONAL PRESS. RELIEF	1	√
	2000456	PROPORTIONAL PRESS. RELIEF	1	√
	5750012193	VALVE DIRECTIONAL	1	√
	A28434-4	VALVE DIRECTIONAL	1	√
	A113320-1	VALVE DIRECTIONAL	1	√
	5750012187	VALVE DIRECTIONAL	1	√
	5750012188	VALVE DIRECTIONAL	1	√
	A60000-4	VALVE DIRECTIONAL	1	√
	2003433	PRESSURE TRANSDUCER	1	√
	2003433	PRESSURE TRANSDUCER	5	√
	2003434	PRESSURE SWITCH	1	√
	2001929	HYD. LIQUID LEVEL SWITCH	1	√
	5750008893	TEMPERATURE SWITCH	1	√
	A98937-1	TEMPERATURE TRANSDUCER	3	√
L	WATER SCHEMATIC			
	A106117-2	FILTER ELEMENT	6	√
	2004065	WATER PUMP	1	√
	WATER PUMP SEAL SETS		2	√
	A90243-1	VFD TANK LOW LEVEL SWITCH	1	√
	2002423	WATER LIQUID LEVEL SWITCH	1	√
N	AIR SCHEMATIC			
	A98780-4	SERVICE KIT	1	√
	N/A	SCRUBBER FILTERS	16	√

CAT.	PART NUMBER	DESCRIPTION	QTY.	REMARK
	D120244-2	DRY DEDUSTING UNIT 250m^3/min	1	√
			100m	
O	POSITION MONITORING			
	C96780-1	CYLINDER, POSITION INDICATING	1	√
	C96780-4	POSITION SENSOR ASSEMBLY	1	√
	C96780-5	CONNECTOR W/10M CABLE	1	√
	C72104-1	TRANSDUCER 0-100"	2	√
P	BRIDGE/TRANSFER CONVEYOR			
	D120135-15	C-TIP	15	√
	A120271-1	IDLER, TROUGHING, CEMA B, 30" B.W.	4	√
	A120271-2	IDLER, TROUGHING, CEMA B, 30" B.W.	4	√
	A120271-3	IDLER, TROUGHING, CEMA B, 30" B.W.	20	√
	A120272-1	RETURN ROLLER	20	√
	80996	2" GUIDE ROLLER	10	√
	1000631	BELT	1件大约400m	√
	5750000106	TAIL PULLEY ASSEBLY	1	√
	D118691-1	DRIVE ROLLER 400V	1	√
	D118691-2	LAGGING	1	√
	D120668-1	DRIVE ROLLER 30 HP	1	√
	D120668-2	LAGGING	1	√
	5750000414	TAIL PULLEY	1	√
	D120358-1	scraper plow	4	√
	D120124-30S	SECONDARY BELT SCRAPER	8	√
	D120270-1	scraper plow	8	√
	D113226-1	primary belt scraper	8	√
	D113218-1	SECONDARY BELT SCRAPER	8	√
	D86236-1?	primary belt scraper	8	√
	4002096	SECONDARY BELT SCRAPER	8	
Q	MONORAIL ANS HOIST ASSEMBLY			
	TBD	HOIST, SPARE PARTS PACKAGE	1	√
R	TOWING SYSTEM			
	A76991-22	SEAL KIT, TOWING CYLINLER	1	√
S	ELECTRICAL SCHEMATIC			
	A108214-9	Starter, 35-50A 120V	1	√
	A62125-4	Transducer, 5A:4-20ma	1	√
	A78734-1	Power Supply, 24V 20A	10	√
	A79007-8	Power Supply, 24V 4.2A	2	√
	A116347-4	Analog Input Module	1	√
	A116347-5	Analog Output Module	1	√

Appendix: No. 4

CAT.	PART NUMBER	DESCRIPTION	QTY.	REMARK
	A97742-10	VFD Unit, Profibus	1	√
	5750 0011 95	Telephone	1	√
	5750000987	CONVEYOR PROXIMITY SPEED/SWITCH	5	√
	A104506-6	CONTACTOR 32A	2	√
	1030023	RELAY	5	√
	A98083-1	DIODE SUPPRESSOR	5	√
	1028888	ESTOP TBM	1	√
	1030498	PROFIBUS CONNECTOR	1	√
		SIEMENS 6*V1 830-OEH10	100m	√
T	AIR MONITORING			
	B76733-7	SENSOR, METHANE	2	√
	A106347-1	SENSOR, HYDROGEN SULFIDE	1	√
	A98899-6	SENSOR, CARBON MONOXIDE	1	√
	A98898-4	SENSOR, OXYGEN	1	√
U	ROCK/PROBE DRILL			
Y	GUIDANCE SYSTEM			
	A115800-13	RMT CONTROL UNIT IN TBM	1	√
	A115800-14	RMT CONTROL UNIT FOR BACK SIGH	1	√
	A115800-15	RMT TARGET FOR TBM	1	2
	A115800-16	BACK SIGHT & TARGET	1	2
	A115800-17	SCU COMM CABLE 200M	1	√
	A115800-18	RCU-3 CABLE 35M	1	√
	A115800-20	RMT CABLE 15M FOR TBM TARGET	1	√
	A115800-21	RMT CABLE 2M FOR BACK SIGHT	1	√
	A115800-25	CABLE & ACCESSORIES SET	1	√
			1	
			2	
Z	SHOTCRETE			
	TBD	FLAT SEAL FOR DL10/18	2	√
	TBD	FLANGE RING FOR DL12/18	2	√
	TBD	PUMP HOSE FOR DL18	1	√
	TBD	DELASCOIL 2L	1	√
	TBD	FUSE FOR VEHICLE 30A	1	√
	TBD	FUSE FOR VEHICLE 20A	1	√
	TBD	FUSE FOR VEHICLE 10A	1	√
	TBD	FUSE FOR VEHICLE 5A	2	√
	TBD	FUSE FOR VEHICLE 3A	1	√
	TBD	CENTRE ZERO RELAY 12V	2	√
	TBD	TIMING RELAY	1	√
	TBD	RELAY 12V	1	√

CAT.	PART NUMBER	DESCRIPTION	QTY.	REMARK
	TBD	FILTER INSERT AGITATOR SPM500	1	√
	TBD	FILTER ELEMENT 175	1	√
	TBD	BEARING FLANGE	1	√
	TBD	BEARING FLANGE COUPLING	1	√
	TBD	O-RING 130x150MM	1	√
	TBD	RUBBER SEAL 70x24x25	1	√
	TBD	BOW ELEMENT W/RUBBER	1	√
	TBD	SEAL A	1	√
	TBD	SPONGE RUBBER BALL	2	√
	TBD	SPONGE CUBE	1	√
	TBD	WEAT PARTS SET	1	√
	TBD	CONVEYING PISTON SLEEVE	2	√
	TBD	GUIDE RING 130x150,20.5	2	√
	TBD	PRESSURE RING 170x105x20	1	√
	TBD	BUSHING 140.4x170x50	1	√
	TBD	RING 140x155x9x12	1	√
	TBD	NOZZLE PIPE 4"	10	√

APPENDIX: No. 5

SwedVent
UNDERGROUND VENTILATION SYSTEM

Flexible Ducting
Technical specification

General
All seams and other joining works are fully welded (vulcanised) 40 mm wide. No sewing. Suspension hooks are welded to the fabric at a distance of c/c 0,75 m. On duct diameters >Ø1800 mm, two parallel lines of suspension hooks are mounted. Every duct length is marked with information regarding quality, length, diameter, max allowed working pressure and manufacturing date.

Coupling A
GIA SwedVent ZIP-joints
Both duct ends made with a split zipper (plastic zipper) of heavy duty PVC-type. Protection-sleeve which on the outside hides the zipper and on the inside seals the joint according to the principle "the higher pressure the tighter joint". With ZIP-joints, the ducting is delivered on standard pallets regardless diameter.

Coupling B
GIA SwedVent steel clamps
One end of each duct length is fitted with a steel ring, vulcanised to the fabric. One steel clamp made of galvanized 1,5 mm steel with locking device (threaded crank) closes the joint.

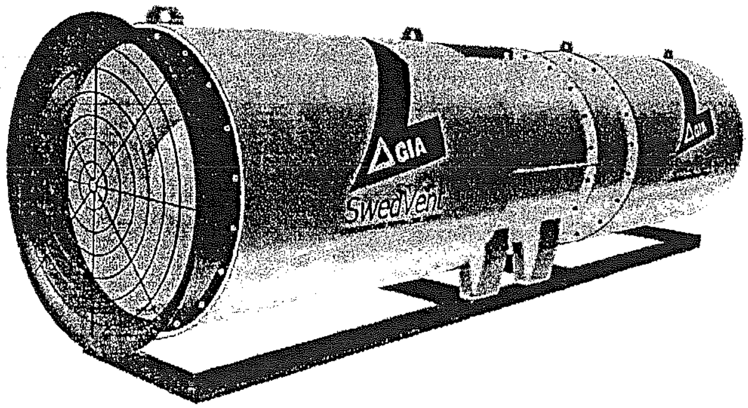
** "RS" stands for Rip-Stop. The base fabric is made with an enhanced yarn which dramatically increases the tear strength. This feature eliminates the ducting to further-tear longitudinally under normal conditions.*		Titan FR-RSX* (yellow)	Airolite FR-RSX* (yellow)	Titan FRA-RSX* (black)	Airolite FRA-RSX* (black)
Base fabric		Polyester			
Yarn thickness (dtex)		2200/1430	3300/1100	2200/1100	3300/1100
Yarns/cm	warp	3,5 (x1430)	3,5	3,5 (x1430)	3,5
	weft	3,5 (x2200)	3,5+1	3,5 (x2200)	3,5+1
Coating		Plastizised PVC			
Tensile strength (N/5 cm) (DIN 53354)	warp	2400	1400	2400	1400
	weft	3000	1700	3000	1700
Tear strength (N) (DIN 53363)	warp	550	270	550	270
	weft	750	340	750	340
Total weight (gram/m2) (DIN 53352)		600	500	600	500
Flame resistance (DIN4102-B1)		yes			
Cold crack (DIN 53361)		-30 C			
Environmental resistance		Resistant against rotting, humus acid, diesel- and nitrous gases, UV-light			
Conductivity (ISO 284)		—	—	1x10^{6}Ω	1x10^{6}Ω

DIAMETER (mm)	Max allowed working pressures (kPa) valid for new ducting. Safety factor 3.			
400	98,0	64,4	98,0	64,4
500	78,4	51,5	78,4	51,5
600	65,3	42,9	65,3	42,9
700	56,0	36,8	56,0	36,8
800	49,0	32,2	49,0	32,2
900	43,6	28,6	43,6	28,6
1000	39,2	25,8	39,2	25,8
1100	35,6	23,5	35,6	23,5
1200	32,7	21,5	32,7	21,5
1300	30,2	19,8	30,2	19,8
1400	28,0	18,4	28,0	18,4
1500	26,1	17,1	26,1	17,1
1600	24,5	16,1	24,5	16,1
1700	23,1	15,2	23,1	15,2
1800	21,8	14,3	21,8	14,3
2000	19,6	12,9	19,6	12,9
2200	17,8	11,7	17,8	11,7
2400	16,3	10,7	16,3	10,7
2500	15,7	10,4	15,7	10,4
2600	15,1	9,9	15,1	9,9
2800	14,0	9,2	14,0	9,2
3000	13,1	8,6	13,1	8,6

Note: All specifications are subject to changes without notification.

APPENDIX: No 6

Ventilation
GIA High Pressure Fans



High pressure fans designed especially for delivering air through ducts with extensive length in mining and tunnelling. Together with GIA SwedVent ducting and GIA SwedVent calculations, a complete ventilation system is offered.

Technical features

- Available in nine sizes with inner diameter from ø630 to ø2240 mm.
- Capacity of 1,5 to 200 m^3/s.
- System pressure of up to 4 200 Pa/stage. At higher system pressures two or more fans are mounted in series.
- 1-5 fan units per fanstation.
- Heavy duty design with fully welded flanges.
- Low sound levels.
- Low energy consumption.
- Available with ISO classified high quality motors from 10 to 500 kW.
- Delivered with different types of starters as well as automatic air flow control system.
- Lubrication of the engine bearings is done from outside the casing.
- Thermistors are mounted directly onto the motor windings for sensing the heat of the winding.
- Impellers with aerodynamically designed and individually adjustable blades.
- Statically and dynamically balanced impeller.
- Precise hub-casing design (small gap between the casing and the blade tip).
- Aerodynamically designed guide vanes in the hub-casing wich eliminates swirl and turbulence.
- Anti rust protection designed to withstand acid and salt environments
- Silencers with mineral wool as sound absorbing material.
- For high efficient noise reduction, the silencers are provided with a centre core which efficiently also reduces the high frequency sound.
- Silencers are also a safe protection against splinter if a fan should burst.
- The inlet bell is bell-shaped and pressure turned and has a protection grille of spidernet model.
- The duct adapter is designed to connect the flexible ducting to the fan station.
- The support frame supports the station and is adapted for the number of fan units in each fan station.

Standard

CE mark according to European standards.

Options

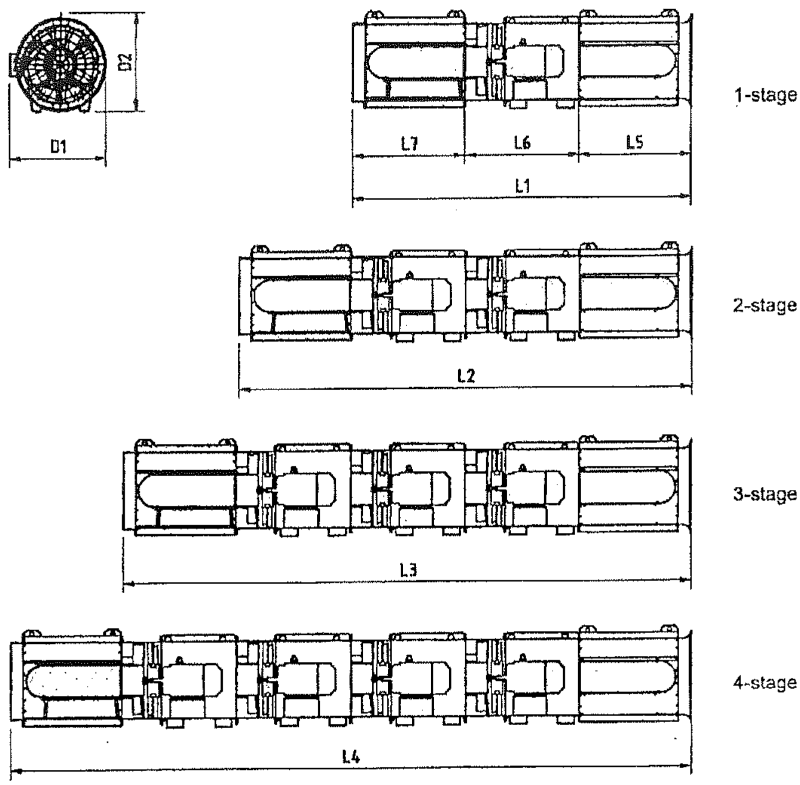
- *Starters*
  -Direct starters
  -Dahlander starters (for two-speed motors)
  -Frequency converters.
  GIA evaluates and recommends which of the three types will be most suitable.
- *Vibration monitor.*
- *Air flow measuring devices.*
- *Support frame for roof mounting on 20" container.*

Other options and dimensions on request.

Ventilation High Pressure Fans

Ventilation High Pressure Fans

Technical data



Dimensions (mm)

	L1	L2	L3	L4	L5	L6	L7	D1	D2
AVH63	3 398	4 448	5 498	6 548	1 198	1 050	1 150	871	904
AVH71	3 504	4 704	5 904	7 104	1 154	1 200	1 150	955	980
AVH90	4 084	5 464	6 844	8 224	1 354	1 380	1 350	1 172	1 155
AVH100	4 189	5 574	6 959	8 344	1 404	1 385	1 400	1 275	1 255
AVH125	6 000	7 600	9 200	10 800	2 250	1 600	2 150	1 543	1 578
AVH140	6 000	7 600	9 200	10 800	2 250	1 600	2 150	1 728	1 815
AVH160	7 200	9 000	10 800	12 600	2 750	1 800	2 650	1 941	1 985
AVH180	7 303	9 203	11 103	13 003	2 753	1 900	2 650	2 144	2 229
AVH224	8 470	10 570	12 670	14 770	3 250	2 100	3 120	2 595	2 590

Weight (kg)

	Fan unit (depending on motor)	Silencer GS	Silencer GSC	Inlet bell	Duct adapter
AVH63	approx. 360 kg	93	114	14	15
AVH71		112	133	14	21
AVH90	approx. 1 000 kg	148	177	17	26
AVH100		187	215	20	28
AVH125	approx. 1 500 kg	315	370	43	46
AVH140	approx.1 460 kg	330	385	52	43
AVH160	approx. 2 100 kg	476	540	61	58
AVH180	approx. 3 550 kg	585	652	70	55
AVH224	approx. 5 800 kg	1 474	----	67	69

Silencer GS = without centre core (inner baffle)
Silencer GSC = with centre core (inner baffle)

Ventilation High Pressure Fans

Functional data

Type	Inner diameter	Nominal power
AVH63	ø 630 mm	9-25 kW
AVH71	ø 710 mm	11-37 kW
AVH90	ø 900 mm	37-90 kW
AVH100	ø 1 000 mm	45-110 kW
AVH125	ø 1 250 mm	45-160 kW
AVH140	ø 1 400 mm	45-160 kW
AVH160	ø 1 600 mm	75-315 kW
AVH180	ø 1 800 mm	132-500 kW
AVH224	ø 2 240 mm	132-500 kW

The casings are treated with anti-rust protection, a 60 µm acrylic primer and a 100 µm acrylic top layer.

Bearings, SKF with "auto-lubricators system 24" on the outside of the casing.

Capacity

	Pressure from fan station (kPa)				approx. Flow rate
	1-stage	2-stage	3-stage	4-stage	
AVH63	1,3 - 2,2	2,5 - 4,5	4,0 - 6,5	5,0 - 7,5	3 - 9 m³/s
AVH71	1,3 - 2,3	2,6 - 4,6	4,1 - 6,6	5,2 - 8,5	4 - 13 m³/s
AVH90	3,5 - 4,3	7,0 - 8,6	10,1 - 12,5	12,8 - 16,0	8 - 22 m³/s
AVH100	3,8 - 4,4	7,3 - 8,7	10,3 - 12,6	12,9 - 16,2	10 - 24 m³/s
AVH125	1,5 - 2,6	2,9 - 5,1	4,5 - 7,7	6,0 - 9,5	14 - 42 m³/s
AVH140	1,2 - 2,2	2,3 - 4,3	3,5 - 6,4	4,6 - 9,5	20 - 48 m³/s
AVH160	1,3 - 2,2	2,5 - 4,3	3,8 - 6,4	5,1 - 8,5	22 - 70 m³/s
AVH180	1,5 - 3,5	3,0 - 7,0	5,0 - 10,3	———	40 - 120 m³/s
AVH224	1,0 - 2,0	1,5 - 4,0	2,0 - 5,7	———	- 200 m³/s

Impellers

Type	Speed (rpm)	Hub	Blades
AVH63	3 000 / 3 600	ø 380 mm	8 pcs
AVH71			
AVH90		ø 560 mm	
AVH100			
AVH125	1 500 / 1 780	ø 800 mm	
AVH140			
AVH160			10 pcs
AVH180		ø 1 000 mm	
AVH224	1 000 / 1 200		10/12 pcs

Quality control
GIA works according to the quality system ISO 9001 and the fans are delivered with CE marking affixed.
Before delivery every fan is tested (capacity, vibration etc.) and accompanied with a testreport. Allowed vibrations <2 mm/s.

In combination with

Flexible ducting
Manufactured from PVC-coated polyester fabrics in four qualities.

Titan		Aerolite	
FR-RSX	FRA-RSX	FR-RSX	FRA-RSX

Available in diameters from ø300 - ø3000 mm in unit length from 10 m up to 150 m.
There are three types of coupling, steel clamps, zip-joints and velcro.
Repair sleeves are available in length of 1, 3 and 5 metres.

Contact us for further information.

TCV Underground Control System
A complete solution for
- Tracking
- Communication
- Ventilation

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Standard technical specifications GIA SwedVent high pressure tunnelling fans with accessories

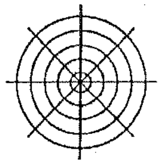
FANS

FAN MODEL	AVH63	AVH71	AVH90	AVH100	AVH112	AVH125	AVH140	AVH160	AVH180	AVH224
Motor	F-class, IP54, 400/440 V, 50/60 Hz									
Nominal power	9-25 kW	11-37 kW	37-90 kW	45-110 kW	9-37 kW	45-160 kW	45-160 kW	75-315 kW	132-500 kW	132-500 kW
Impeller speed	3000/3600 rpm				1500/1780 rpm					1000/1200 rpm
Bearings	SKF									
Bearing lubrication	SKF "auto-lubricators system 24" on casing outside									
Impeller hub	Ø380 mm, cast aluminium	Ø380 mm, welded steel	Ø560 mm, welded steel	Ø560 mm, welded stee	Ø560 mm, cast aluminium	Ø800 mm, cast aluminium	Ø800 mm welded steel	Ø800 mm, welded steel	Ø1000 mm, welded steel	Ø1000 mm, welded steel
Impeller blades	8 pcs, cast aluminium, adjustable at stillstand							10 pcs, cast aluminium, adjustable at stillstand		
Gap between blade top and fan casing	1,5 mm	1,5 mm	1,5 mm	1,5 mm	2,0 mm	2,5 mm	2,5 mm	3,0 mm	3,0 mm	3,0 mm
Casing inner Ø	630 mm	710 mm	900 mm	1000 mm	1120 mm	1250 mm	1400 mm	1600 mm	1800 mm	2240 mm
Fan casing thickness	5 mm	5 mm	6 mm	6 mm	6 mm	6 mm	8 mm	8 mm	8 mm	10 mm
Casing antirust protection	60 µ acryl primer + 100 µ acryl top layer									
Casing flanges	laser-cut, 8x50 mm, welded	laser-cut, 10x60 mm, welded				laser-cut, 10x70 mm, welded				
Turbulence elimination	Guide vanes welded to inside casing + 5 straight longitudinal "fins"									
CE-marking	yes									
Performance test	full test before delivery, capacity measure-ments									
Allowed vibrations	< 2mm/s									

SILENCERS

MODEL	GS63	GS71	GS90	GS100	GS112	GS125	GS140	GS160	GS180	GS224
Inner Ø	630 mm	710 mm	900 mm	1000 mm	1120 mm	1250 mm	1400 mm	1600 mm	1800 mm	2240 mm
Outer Ø	790 mm	870 mm	1060 mm	1160 mm	1280 mm	1410 mm	1560 mm	1760 mm	1960 mm	2400 mm
Length	1000 mm	1000 mm	1200 mm	1200 mm	1200 mm	2000 mm	2000 mm	2400 mm	2400 mm	3000 mm
Casing flanges	laser-cut, 8x70 mm, welded					laser-cut, 8x80 mm, welded				
Casing thickness	3 mm	3 mm	3 mm	3 mm	3 mm	3 mm	4 mm	4 mm	4 mm	5 mm
Sound absorbing material		Heavy mineral wool								
Self supporting to fan casing		yes								
Casing antirust protection		60 µ acryl primer + 100 µ acryl top layer								
"Super silencer"	centre baffle, l=800 mm, Ø380 mm		centre baffle, l=1000 mm, Ø550 mm			centre baffle, l=1800 mm, Ø640 mm		centre baffle, l=2200 mm, Ø640 mm		centre baffle, l=2500 mm, Ø640 mm

INLET BELLS

MODEL	GS63	GS71	GS90	GS100	GS112	GS125	GS140	GS160	GS180	GS224
Intake	"Bell shaped", pressure turned									
Casing thickness	2 mm						3 mm			
Protection grille "spider net model										

DUCT ADAPTERS

MODEL	GS63	GS71	GS90	GS100	GS112	GS125	GS140	GS160	GS180	GS224
Length	150 mm	150 mm	150 mm	150 mm	150 mm	150 mm	200 mm	200 mm	200 mm	200 mm
Casing thickness	3 mm						4 mm			
	Ø9 mm ring welded to adapter end for securing ducting									

FREQUENCY INVERTERS

Nominal power	11-500 kW
Electrical supply	400/440 V, 50/60 Hz
Adjustable fan speed	250 rpm up to full speed
Protections	motor current, short circuit, phase failure, over- and under voltage
Operating controls	LED-display with control buttons mounted on outside casing door
Cooling	By cooling fan
Capsuling	IP 21
Standard extra protection	All circuit boards laquered for protection against moisture and dust
CE-marking	yes
Others	The inverters fulfil the EU-regulations for start of electrical equipment
Optional	Inverters built in in an IP54 cabinet. Circuit breaker, automatic fuses, cabinet heater are also optional.
Optional	EMC-filter

DAHLANDER STARTERS

Nominal power	11-132 kW
Electrical supply	400/440 V, 50/60 Hz
Possible fan speed	Two-speed (half/full)
Protections	Motor current, short circuit
Operating controls	Control buttons mounted on outside casing door
Capsuling	IP 54 cabinet
CE-marking	yes
Others	The starters fulfil the EU-regulations for start of electrical equipment
Optional	Circuit breaker, automatic fuses, cabinet heater
Optional	Cabinet heater

DIRECT STARTERS

Nominal power	11-132 kW
Electrical supply	400/440 V, 50/60 Hz
Possible fan speed	One-speed
Protec-tions	Motor current, short circuit
Operating controls	Control buttons mounted on outside casing door
Capsuling	IP 54 cabinet
CE-marking	yes
Others	The starters fulfil the EU-regulations for start of electrical equipment
Optional	Cabinet heater

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APPENDIX: No. 7

上崑嵩水电站TBM掘进段围岩类别与主要掘进参数对应表

Comparison sheet of boring parameter and rock class in TBM excavation section of Thuong Kon Tum Hydropower Station

围岩类别 Rock classification	刀盘转速 Cutter head rpm (rpm)	刀盘扭矩 Cutter head torgue (KN.m)	刀盘推力 Cutter head thrust (KN)	推进油缸压力 Propel cylinder pressure(bar)	支撑油缸压力 Gripper cylinder pressure(bar)	顶部支撑压力 roof support pressure(bar)	侧支撑压力～左 Side support pressure-left (bar)	侧支撑压力～右 Side support pressure-right (bar)
I	9～11	1600～1800	6800～7200	220～230	250	20～40	30～40	30～40
II	8～11	1800～2000	6500～7000	210～225	250	20～40	30～40	30～40
III	7～10	2000～2200	6000～6500	195～210	250	20～40	30～40	30～40
IV	6～8	2200～2500	5000～6000	160～195	200	30～50	40～50	40～50
V	/	/	/	/	/	/	/	/

Appendix D

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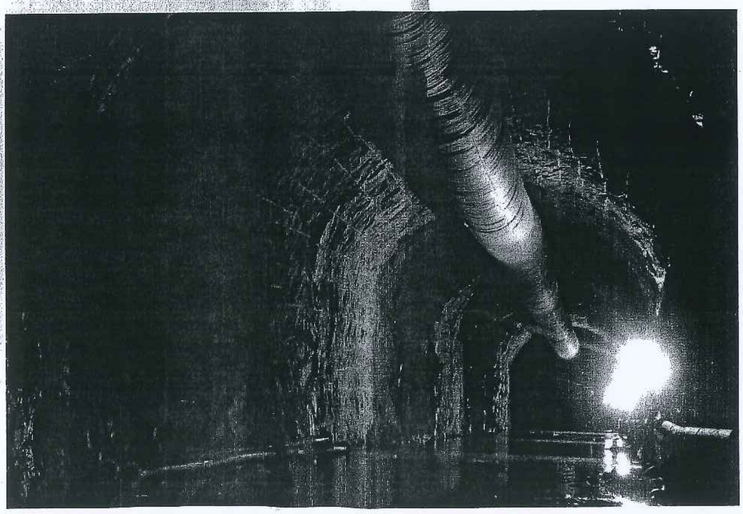
Report

Upper Kontum HPP – Technical advisor trip 1, 2014

Subtitle

Author(s)
Nghia Trinh



SINTEF Building and Infrastructure
Rock Engineering
2014-03-24

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ABSTRACT

A rock engineering expert from SINTEF was invited for a technical assistance to VSH. The task was carried out from 17/2 to 23/2/2014. The details of the task are:

- Inspecting at headrace tunnel from the intake (drill and blast tunnel);
- Inspecting the construction site at the TBM tunnel;
- Inspecting the construction site at the powerhouse complex;
- Attend and provide technical assistance to the Client (VSH) in a meeting with the Contractor (Huadong) concerning the steel lining of the headrace tunnel for the Upper Kontum hydropower project.

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1	2014-03-24	Inspection and meeting report

SINTEF

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APPENDICES

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1 General information

In November and December 2013, SINTEF received a request from Vinh Son - Song Hinh Hydropower Joint Stock Company (VSH) for carrying out rock stress measurements for Upper Kontum HPP. The stress measurements were carried out using 3D-over coring method. Results of the stress measurement were emailed to VSH at the end of 1/2014, and an official report was submitted to VSH on 2/2014.

After the stress measurements, a rock engineering expert from SINTEF (Ph.D. Nghia Trinh) was invited to assist VSH in inspecting the construction and attending the meeting between VSH and Huadong to discuss the steel lining option for the headrace tunnel. The task was carried out from 17/2 to 23/2/2014. The details of the task are:

- Carry out a construction inspection to the headrace tunnel from the intake, the TBM tunnel, and the powerhouse complex:
  - Inspecting and giving opinion about the post-grouting work and other risks of construction delay and tunnel stability if necessary at headrace tunnel from the intake;
  - Inspecting the construction site at the TBM tunnel and giving comments about the TBM performance;
  - Inspecting the construction site at the powerhouse complex and giving comments about the risks of construction delay and tunnel and cavern stability if necessary;
- Attend and provide technical assistance to the Client (VSH) in a meeting with the Contractor (Huadong) concerning the steel lining of the headrace tunnel for the Upper Kontum hydropower project.

This report is established to present the results of the trip in February 2014.

2 Inspection at the drill and blast tunnel

The drill and blast tunnel was inspected on 18/2/2014. Based on his own site observations and information obtained from the meeting with the Contractor as well as Project Management Unit, the rock engineering expert has the following comments:

- Up to date (18/2/2014), the drill and blast tunnel is excavated to chainage K2+940, and there is about 2 km more to be excavated. The construction progress for this tunnel is as shown in Figure 1;
- The most challenging problem in construction of this tunnel is the excessive groundwater flow into the tunnel. This tunnel was designed with 1 descending excavation face, thus all inflow of water along the tunnel will accumulate at the excavation face. The amount of groundwater reached to 180 l/s in 12/2013, creating a very difficult situation for the construction;
- The project owner VSH decided to carry out post-grouting to seal large inflow locations. Some locations with large inflow were sealed and the discharge is reduced by 90% from 90-100 l/s to about 10 l/s at location K2+652. On 18/2/2014 (inspection date), the total groundwater flow to the excavation face is estimated to be about 140 l/s. With a positive result from the post-grouting work, the remain large inflow locations will be attempted to be sealed by a continuing post-grouting effort;
- For the purpose of improving the construction condition, VSH agreed to the Contractor to change the tunnel inclination from descending 7-8% to 0%;
- With the two mentioned measures (post-grouting and 0% inclination), the construction progress is increased significantly in 1/2014, with 107 m/month as shown in Figure 1;

Even though the construction progress has improved significantly, there are some issues to be noted as follow:

- Blasting the tunnel into a water bearing joint/zone will increase area for the flow, and in most cases will increase the groundwater flow. Thus, large water bearing zones in front of the excavation face must be sealed before excavation. General experience in the tunnel construction shows that pre-grouting is much faster, cheaper, and effective than post grouting;
- The most effective way to detect potential water bearing zones ahead of the excavation face is by probe drilling. Thus, VSH and Contractor should consider including a drilling machine as soon as possible. The current hand-held drilling machine for tunnelling is not capable to drill more than 3.5 m length. This supplemental drilling equipment is also necessary to deal with possible weakness zones during construction. In the meantime, when the drilling machine is not available, it is necessary to closely observe the geological condition in the excavation face and in the blast holes to early detect the geological condition in front of the excavation face;
- To set up a program for geological observation, it is necessary to carry out geological mapping in the tunnel systematically. After each blast round, as soon as the blasted rock is removed, scaling done, and the temporary rock support is installed, the Contractor should provide half an hour for the engineering geologist to carry out the geological mapping for the last blast round. Geological information such as joint system, rock type, rock mass quality, etc. at the tunnel face, roof, and wall of the excavated tunnel section will be recorded, sketched, and described in a prescribed format. A template will include required information to be filled out during mapping to secure that this work is uniform throughout the tunnel construction. Groundwater flow in the excavation face, roof, and walls must also be marked in the sketch and described in the mapping sheet;
- Engineering geologist must observe the excavation face when all holes for the next blast are completed to update information of water observed from the drill holes;
- These tunnel mapping sheets must be archived systematically to assist responsible personnel in evaluating the groundwater condition. If there is a doubt of the groundwater bearing zone in front of the excavation face, the charging work cannot commence until pre-grouting has been carried out. The blasting work may start when the pre-grouting mets pre-set technical requirements;
- The application of the above procedure is technically simple but contracturely complicated, and it may cause contracturely conflicts. At the moment, the Contractor is not willing to cooperate for the pre-grouting work, as he claims that this work brings more benefit to VSH than to the Contractor, and pre-grouting work causes delay to the Contractor's progress. VSH should evaluate the situation and work with the Contractor to develop a solution. One possibility is to compensate the Contractor for dealing with large water inflow such as pumping cost, and cost for post-grouting. Those costs can be compared to the cost for pre-grouting in order to make decision;
- The post-grouting has some positive results now, so this work should be continued together with an overall evaluation. Groundwater monitoring should be carried out closely to see if the post-grouting can reduce the inflow or the inflow is forced to move from one location to another, and the total discharge is not reduced. An example is at chainage K2+652; the inflow before grouting is from 90 to 100 l/s, whilst after the grouting, the discharge at this location is largely reduced, and remaining inflow is about 10 l/s. However the total flow towards the excavation face is not significantly reduced (from 180 l/s before grouting to only 140 l/s);
- The pumping system and drainage pipes should be properly maintained. Observation in the tunnel is that there is a large leakage in the pipe at 500 m behind the tunnel face. From this point, the water escapes from the pipe and flows back to the excavation face. The water circulates rather than being pumped out of the tunnel. The used energy for pumping this water is thus wasted. It is noted that the cost for this pumping will, in one way or another, be paid by the Client;
- During discussion with the Contractor, it is found that the power supply to this tunnel is problematic. The contractor claims that if the power supply is without interruption, then they can achieve an excavation progress of 110 to 120 m/month with 0% inclination tunnel;
- The upgraded system for power supply was approved by VSH. Plan for this work is to place all the existing generators to the tunnel entrance, and a new generator of 2.5 MW will be added and supplemented with transformer. To improve the construction progress, this upgraded power supply

system should be installed as soon as possible. The Contractor may blame the performance of the new power system, as similar blames were experienced. Thus, it is necessary to reach an agreement that the Contractor is responsible for the design, building, as well as operate and maintain the power supply to ensure that the system meets the contractor's requirements;

- The change of the tunnel inclination from 7-8% downward to 0% brings an advantage condition for collecting and pumping the water, increasing the tunnel excavation progress. However, it will also affect the designed arrangement of the tunnel system. Thus, VSH should request the Contractor to complete their new tunnel design with necessary modifications, so that VSH can evaluate and approve;

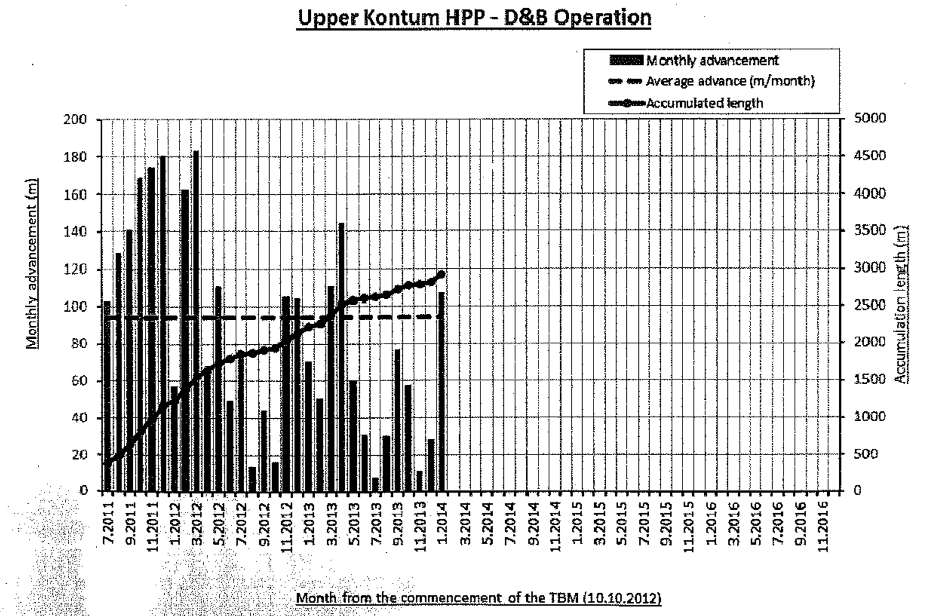

Figure 1: Construction progress of the drill and blast tunnel.

3 Inspection at the TBM tunnel

The TBM tunnel was inspected on 19/2/2014. From the visual observation in the tunnel and meeting with the contractor, the rock engineering expert has following comments:

- To the date of the visit – 19/2/2014, the TBM tunnelling reached chainange K15+890. Monthly tunnelling progress as well as the accumulated tunnelling length of this TBM tunnel is as shown in Figure 2. It can be seen that the TBM tunnelling performs far below expected and planned progress. The average monthly advance is only 117 m/month while it was expected more than 500 m/month. Thus, it is necessary to find a "break through" solution to improve the progress;
- The TBM operation team is recently replaced due to pressure from the VSH. The new operation team is from Railway Bureau 18, which has more experience on TBM tunnelling than the previous team. The new TBM operation team was established in November 2013. For three months from 12/2013 to 2/2014, the new team has carried out a total technical check of the TBM and its auxiliary

equipment. They also carried out operation tests and the advance rate was not more than 10 m a day. The tunnel advance was therefore low during January and February 2014. The TBM tunnelling operation still needs to be closely supervised with regular meetings for status reporting and updating;

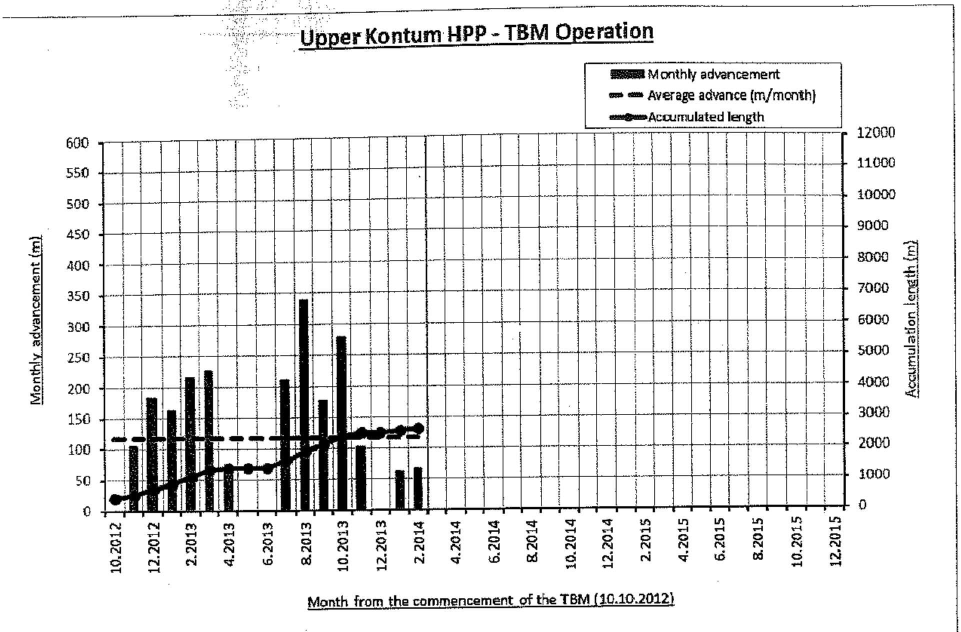

Figure 2: Monthly advance rate and the accumulated length of the TBM tunnel.

- On the 12/2/2014 – seven days before the expert visit, the TBM encountered a weakness zone of about 4 m width. The material in the zone was blocky breccia in completely weathered material constituting dark-greenish clay. The weakness zone had a limited width and the quality of the zone was not a threat to the stability and could not flow into the TBM tunnel;
- A lesson learned for the Contractor as well as the Project Management Unit from this incident is that when the TBM approaches a weakness zone, the operation team must operate the TBM in accordance with the "Guidance for TBM excavation in poor zones". This document has been submitted to the Client earlier, but it was not used properly at this event on 12/2/2014. The operation team did not reduce the thrust and rotation speed of the TBM when it approached the weakness zone, and there was no discussion with the geological engineer at the site. Given that this zone was larger with extremely poor geological condition with high groundwater pressure, the material would collapse and flow into the tunnel, which could have collapsed onto the TBM causing severe damage and may jam the TBM;
- Such situation has happened in some TBM projects such as Pinglin and Gotthard, as shown in Figure 3 and 4. A TBM that is trapped in a weakness zone is very complicated to move, and it is timely and costly to release it. To release the TBM, it can be done by one of the following measures:
    - Excavating a pilot tunnel through the cutter head (if possible). The weakness zone can be strengthened (grouting) from the pilot tunnel and excavating around the TBM to free it, as shown in Figure 3;

- Alternatively, a bypass tunnel can be excavated to pass through the zone. Access to the TBM face is provided from the other side of the weakness zone. The zone is strengthened from this access and the TBM can continue its advance, as shown in Figure 4;

- From the descriptions, it can be seen that it is expensive and time consuming to overcome the situation if a TBM is trapped in a weakness zone. The cost and time will be out of control for both involved parties (Owner and Contractor) in case of a trapped TBM. Thus, maximum effort must be put into preventing such situation from happening. Carrying out a careful excavation procedure as described in the "Guidance for TBM excavation in poor zones" is therefore a pre-requisite. In addition to this, it is necessary to install probe drilling and pre-grouting equipment at the TBM as soon as possible. This issue has been mentioned few times in earlier reports. It is required that VSH will cooperate with the Contractor, so that the equipment can be installed and such "expensive lessons" as trapped TBM can be avoided;
- To increase the Contractor's capacity in dealing with adverse geological conditions during tunnelling, it is recommended that the Contractor should repair the auxiliary equipment for steel rib installation. This equipment was provided with the TBM, but it was not able to operate properly due to some obstacle within its operating space. In addition, the TBM operation team is not familiar with the equipment. Efforts to be undertaken on repair works and training of TBM operators will improve the TBM progress (with smoother full cycling of advancing for about a meter – depending on the encountered geological condition – and installing steel ribs, shotcreteing and so on);
- The Contractor uses labour for rock support work, such as steel rib installation, manual shotcreting, etc. This is more time consuming than mechanised work. Involving labour for such work in poor geological condition causes concerns on safety issues. Observations at the site show that labour works the full length of a shift underneath unsupported roof even in weakness zone. Rock blocks from the weakness zone may fall down without advanced warning. It is an urgent demand for the fixing of the equipment for the steel rib installation;

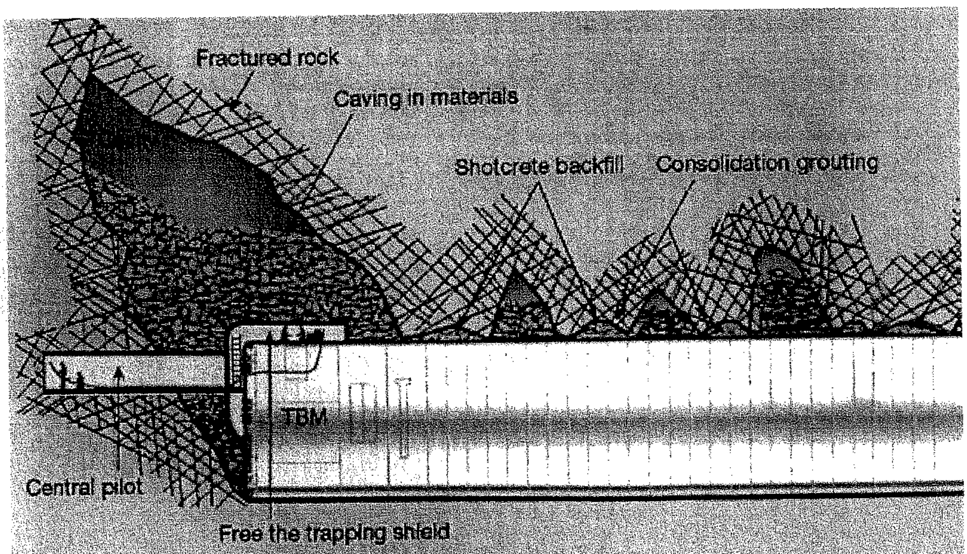

Figure 3: Trapped TBM and excavation method at Pinglin project (from Shen et al., 1999).

- Beside the listed technical details, the largest problem is the poor performance of the TBM. This issue has been discussed thoroughly between VSH, the Contractor, and Robbins in a meeting in China, in January 2014. The meeting concluded that this relates partly to the incompetence of the operation team and partly to some technical faults of the TBM. The operation team has been changed to a more competence team from Railway Bureau 18 few months before the meeting, now the focus is on the technical faults. Based on the minutes of the meeting, it is not clear if the technical faults originate to the manufacturer or imprudent operation. The meeting concluded that the TBM will not be able to operate with expected advance rate (more than 500 m/month), rather estimated to approximately 300 m/month. However, it gave no clear indication whether Robbins has any responsibility nor any time schedule was presented;
- The advance rate of the TBM will not meet the demand from the project owner, thus VSH needs to study construction options as proposed from VSH and Contractor, which are (a) improving the current TBM and operation team; or (b) supplement with a second TBM;
- A meeting directly with Robbins company is recommended to have their opinion of the current situation as well as evaluating the two listed options;

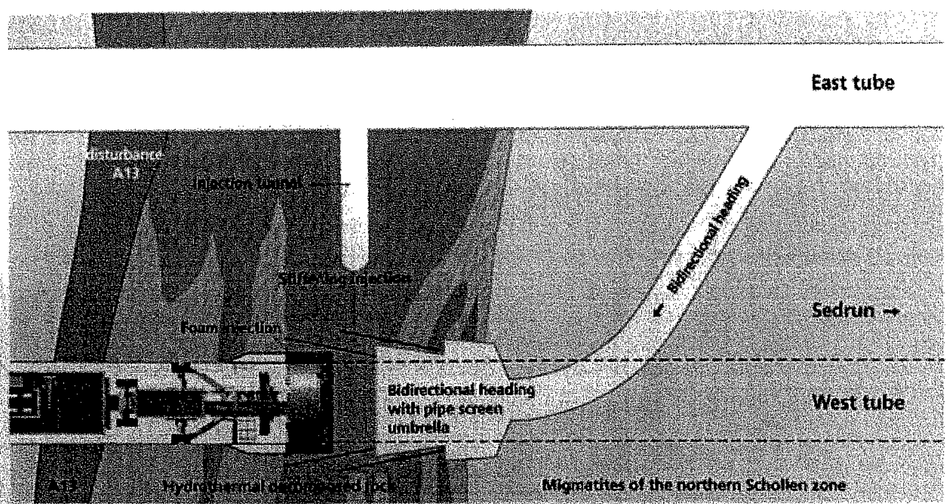

Figure 4: Plan view of a TBM tunnelling in a weakness zone and excavation method at Amsteg, Gotthard Base Tunnel project – Switzerland (from Heinz Ehrbar, 2008).

4 Powerhouse complex

The powerhouse complex was inspected on 19/2/2014. Top heading and first benches were carried out for the transformer cavern. Rock support consisted of systematic bolting and shotcrete with steel mesh are carrying out. The powerhouse cavern was excavated at top heading level, and rock support (systematic bolt and shotcrete) is installing for the roof. No bench has been excavated yet. From the site observation and meetings, the rock engineering expert has following comments:

- After every excavation stage (top heading and benching), detailed geological mapping must be carried out before the rock surface is covered by shotcrete and impossible to observe the rock mass condition. Geological mapping must be carried out regularly and the mapping sheets should be kept systematically;

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- Information from the mapping sheets must be used to update the cavern design. All of the design calculation such as empirical method (Q, RMR), numerical modelling, or wedge failure analyses, etc. need to be reviewed with the updated geological information;
- The updated calculation, analyses must be submitted to the VSH for consideration and approval;

5 Attending the meeting between VSH and the Contractor

On 22/2/2014, the rock engineering expert attended a meeting between VSH and the Contractor. The main content of the meeting are:

- The Contractor presented their stress measurements by hydraulic fracturing method in drill holes. The tests were carried out in five boreholes in different locations, whereas 4 locations in the powerhouse complex and pressure tunnel, 1 location in the drill and blast tunnel at the intake;
- Based on the results of the measurements, the Contractor concluded that the in-situ rock stress condition along the tunnel system is low. The results showed that the minor principal stress is smaller than the theoretical vertical stress estimated from the rock overburden and rock unit weight. From the measured results, the Contractor concluded that the current tunnel layout will need 5.5 km of steel lining to compensate for the low stress. The construction time and cost will therefore be largely increased. It is estimated to take 10 years for the steel lining work only;
- The Contractor also proposed an alternative layout with lifting up of the tunnel system to allow that more adits can be added. The tunnelling method will include both drill and blast and TBM. The Contractor did not indicate the length of steel lining for the new alternative. It is likely that the length of steel lining will be the same or longer than 5.5 km because the tunnel is located shallower. One advantage of the new alternative is that more adits and construction faces are created to shorten the construction time;

- Following the presentation from the Contractor, the rock engineering expert from SINTEF presented an independent rock stress measurement carried out by SINTEF, using 3D over-coring in drill holes. Locations for the SINTEF measurement were at P0+025 and P0+290 – the same locations as the Contractor;
- The designed water head in the tunnel at these two locations is 610 and 635 m, i.e. 6.1 and 6.35 MPa respectively;
- Results of the measurement are:
    - Location P0-025:
      $\sigma 1$ = 15.7MPa ± 4.0 MPa, trending N138°and plunging 16°, i.e. approximately horizontal.
      $\sigma 2$ = 12.1MPa ± 1.3MPa, trending N23°, and plunging 57°.
      $\sigma 3$ = 7.7 MPa ± 3.6 MPa, trending N237°, and plunging 28° i.e. approximately horizontal.

    - Location P0+290:
      $\sigma 1$ = 29.6MPa ± 2.2 MPa , trending N175°and plunging 14°, i.e. approximately horizontal.
      $\sigma 2$ = 12.1MPa ± 1.1MPa, trending N62°, and plunging 58°.
      $\sigma 3$ = 10.1 MPa ± 1.9 MPa, trending N273°, and plunging 29° i.e. approximately horizontal.
- With these results, at location P0+025, the ratio between in-situ minor principal stress to the water head is $\sigma 3/(\gamma H)$ = 7.7 (MPa)/ 6.1(MPa) = 1.26. At location P0+290, the ratio is 10.1/6.35 = 1.59;
- Commenting on the Contractor's conclusion that the in-situ stress along the tunnel is low, the rock engineering expert does not agree. Some evidences of high stress conditions were shown in the meeting. The first evidence related to some core disking due to high stress as shown in Figure 5. The core disking was observed from the core of the earlier drill core. Unfortunately it has been impossible to identify the location of the core disking. The second evidence relates to rock slabbing at a section of the TBM tunnel – around K16+545, as shown in Figure 6;

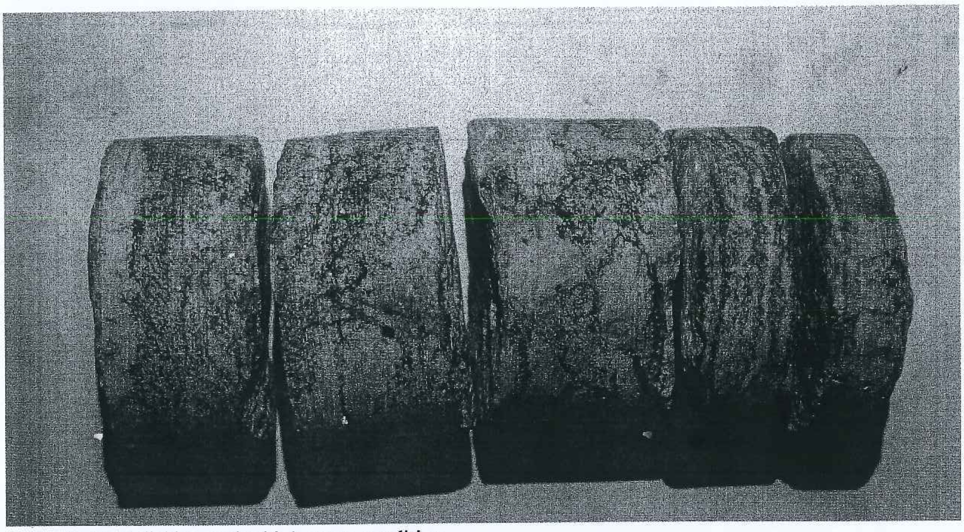

Figure 5: Core disking under high stress condition.

Figure 6: Rock slabbing/bursting around K16+545 under high stress condition.

- The rock slabbing or rock burst in the TBM tunnel can be explained as follows:
    - For a circular tunnel with sigma 1 and sigma 3 as in Figure 7, according to the analytical solution from Kirsh (1898), maximum tangential stress will be at point A (at a point 90 degrees from sigma 1) and minimum tangential stress will be at point B;
    - Magnitude of the tangential stress is $\sigma_t(\max) = 3\times\sigma_1 - \sigma_3$; $\sigma_t(\min) = 3\times\sigma_3 - \sigma_1$;

- o At point P0+025, the measured in-situ stresses are σ1 = 15,7MPa and σ3 = 7,7 MPa, then the calculated tangential stresses are σ_t(max) = 39,4 MPa and σ_t(min) = 7,4 MPa;
- o At point P0+290, the measured in-situ stresses are σ1 = 29,6MPa and σ3 = 10,1 MPa, then the calculated tangential stresses are σ_t(max) = 78,7 MPa and σ_t(min) = 0,7 MPa;
- o The uniaxial compressive strength of the intact rock ranged from 70 to 230 MPa, with average of 170 MPa (SINTEF, 2014);
- o The ratio between the calculated maximum tangential stress to the uniaxial compressive strength is about 0.45. Thus, the rock slabbing would occur after more than a hour, according to description in row H and J in Table 1. The rock slabbing / rock burst in the TBM tunnel is well suite with the general experience within the tunnelling industry in hard rock around the world;
- o The rock slabbing shown in Figure 6 is almost at the same location as the calculated maximum tangential stress shown in Figure 8. The stress measurement results and the actual observations fit well;

- Considering the situation based on different sources of information, VSH decided to keep the original design of the steel lining;
- The rock engineering expert also recommended that the Contractor should update their cavern design to secure that the cavern is also safe under the new information of the in-situ stress obtained from the SINTEF measurement;

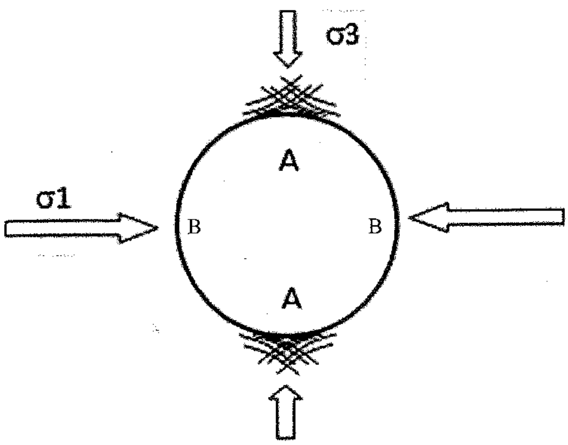

Figure 7: A circular tunnel and the stress concentration.

Kirsch's formulae to calculate the stress around a circular tunnel under in-situ stresses sigma 1 and sigma 3:

$$\sigma_r = \sigma_1\left[\left(\frac{1+2k}{2}\right)\left(1-\frac{r^2}{R^2}\right)+\left(\frac{1-k}{2}\right)\left(1-4\frac{r^2}{R^2}+3\frac{r^4}{R^4}\right)\cos 2\theta\right]$$

$$\sigma_\theta = \sigma_1\left[\left(\frac{1+2k}{2}\right)\left(1+\frac{r^2}{R^2}\right)-\left(\frac{1-k}{2}\right)\left(1+3\frac{r^4}{R^4}\right)\cos 2\theta\right]$$

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$$\tau_\theta = \sigma_1\left[\left(\frac{1-k}{2}\right)\left(1+2\frac{r^2}{R^2}-3\frac{r^4}{R^4}\right)\sin 2\theta\right]$$

Where:

- σ_r – Radial stress,
- σ_θ – Tangential stress,
- τ – Shear stress,
- σ_1 – Major principal stress,
- σ_3 – Minor principal stress,
- k $= \sigma_3/\sigma_1$,
- R – Distance from centre to the studied point,
- r – The radius of the opening,
- θ – The angle between σ_1 and R,

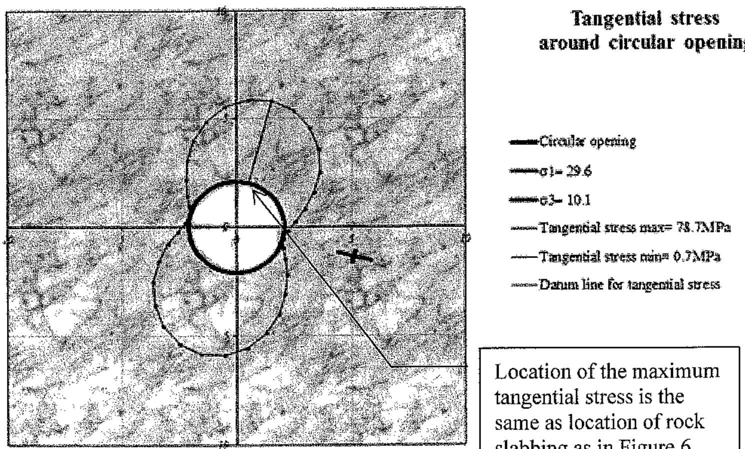

Figure 8: Calculation of tangential stresses in the TBM tunnel at K16+545, using in-situ tress values measured at P0+290.

Table 1: Relationship between stress and possible rock problems in a competent, massive rock. This table is extracted from Table 6 for stress reduction factor in the Q-system (NGI, 2013).

	b) Competent, mainly massive rock, stress problems	σ_c/σ_1	σ_θ/σ_c	SRF
F	Low stress, near surface, open joints	>200	<0.01	2.5
G	Medium stress, favourable stress condition	200-10	0.01-0.3	1
H	High stress, very tight structure. Usually favourable to stability. May also be unfavourable to stability dependent on the orientation of stresses compared to jointing/weakness planes*	10-5	0.3-0.4	0.5-2 2-5*
J	Moderate spalling and/or slabbing after > 1 hour in massive rock	5-3	0.5-0.65	5-50
K	Spalling or rock burst after a few minutes in massive rock	3-2	0.65-1	50-200
L	Heavy rock burst and immediate dynamic deformation in massive rock	<2	>1	200-400

6 Conclusions and recommendations

From the site observations and the meetings with the Contractor and the Project Owner, the rock engineering expert has main conclusions and recommendations as follow:

- Drill and blast tunnel:
  - Tunnel mapping must be carried out systematically after each blast round. Information from the tunnel mapping can be used to evaluate the rock mass condition as well as pre-grouting requirement;
  - Supplement with a probe drilling equipment if possible;
  - Continue with the post-grouting work and monitor the total groundwater flow;
  - The approved electrical supply should be installed as soon as possible;
  - The Contractor should update the new design for the tunnel system considering the changes of the inclination in the drill and blast tunnel. The updated design must be submitted to VSH for consideration and approval;

- TBM tunnel:
  - The probe drilling equipment should be installed at the TBM as soon as possible;
  - The pre-grouting equipment should be installed at the TBM as soon as possible;
  - When approaching a weakness zone, the excavation procedure for weakness zone must be strictly followed. The procedure has already been submitted and approved by VSH;
  - The equipment for steel rib installation, and shotcrete equipment must be repaired as soon as possible;
  - Studying further two alternatives of improving the current TBM or add a second TBM for improving the excavation progress;
  - When the TBM tunnel is progressed to the low rock cover area (around chainage K14+000), in-situ stress measurement should be carried out. This is to check if the rock stress is sufficient for preventing water leakage from the tunnel;

- Powerhouse complex:
  - Tunnel mapping should be carried out systematically;
  - Update the cavern design with the updated geological information;
  - Update the cavern design with the updated in-situ stress measured by SINTEF;

- The updated calculations, analyses must be submitted to VSH for review, considerations, and approval.

References

Heinz Ehrbar, 2008: "Gotthard base tunnel, Switzerland experiences with different tunnelling methods", Seminário Internacional "South American Tunnelling" – 2008, Brasil.

NGI, 201: Handbook "Using the Q-system" web-version, available at http://www.ngi.no/upload/6700/Q-method%20Handbook%202013%20web-version.pdf (access on March 2014).

Shen, C.P., Tsai, H. C., Hsieh, Y.S. & Chu, B. 1999. The methodology through adverse geology ahead of Pinglin large TBM. Proc. RETC. Orlando, FL. Hilton & Samuelson (eds). Ch.8: 117-137. Littleton, CO: Soc. for Mining, Metallurgy, and Exploration, Inc.

SINTEF, 2014: "Project report: Stress measurement at Thuong Kontum HPP, Vietnam", report number SBF2014F00223, submitted to VSH on February 2014.

Appendix E

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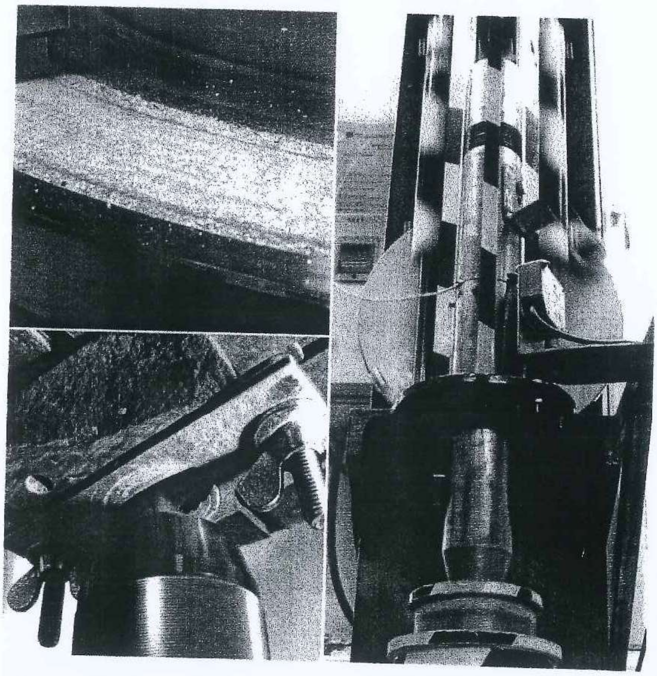
14002IG - Restricted

Test report

Drillability

Investigated TBM drillability properties of 2 rock samples from Thuong Kontum HEP, Vietnam

Author
Joakim Eggen



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KEYWORDS
TBM Drillability

Test report

Drillability

Investigated TBM drillability properties of 2 rock samples from Thuong Kontum HEP, Vietnam

VERSION
1

DATE
2014-01-29

AUTHOR
Joakim Eggen

CLIENT
Vinh Son-Song Hinh Hydropower Joint Stock Company

CLIENT'S REF.
Nguyen Van Thanh

PROJECT NO.
102006082

NUMBER OF PAGES/APPENDICES
18

TEST OBJECT
2 rock core samples

TEST OBJECT RECEIVED
2014-01-09

TEST PROGRAM
DRI™, CLI™, DTA

TEST LOCATION
Geological Engineering laboratory

DATE OF TEST
From 2014-01-10
To 2014-01-23

ABSTRACT

The samples are analysed in order to determine Drilling Rate Index™ (DRI™), Cutter Life Index™ (CLI™) and Quartz content by Differential Thermal Analysis (DTA).

Drilling Rate Index™ (DRI™) and Cutter Life Index™ (CLI™) are determined in accordance with: *http://www.drillability.com, SINTEF/NTNU (2003), Suggested Methods for determining DRI™, BWI™ and CLI™.*

The trademarked acronyms and terms Drilling Rate Index™, Cutter Life Index™, DRI™ and CLI™ are unique for test results and calculated indices originating from the NTNU/SINTEF laboratory and can only be obtained by testing samples at our reference laboratory.

The test results relate only to the items tested

SINTEF

PREPARED BY
Joakim Eggen

SIGNATURE

APPROVED BY
Simon Alexander Hagen

SIGNATURE

REPORT NO.
14002IG

CLASSIFICATION
Restricted

SINTEF

Table of contents

1 Executive summary

Test results, calculated indices and classifications are given in the following table.

Sample No. (given by SINTEF)	1	2
Sample ID (given by the Client)	PO+290	PO-025
Brittleness Value (S_{20})	50.1 Medium	44.3 Medium
Sievers' J-Value (SJ)	2.6 Very high surface hardness	1.7 Extremely high surface hardness
Abrasion Value Cutter Steel (AVS)	21.0 Medium	26.0 High
Drilling Rate Index™ (DRI™)	43 Medium	35 Low
Cutter Life Index™ (CLI™)	6.2 Low	4.9 Extremely low
Quartz content (DTA)	27 %	27%

Classification of S_{20}, SJ, AV and AVS according to Dahl, F., et al. 2012. Classifications of properties influencing the drillability of rocks, based on the NTNU/SINTEF test method. Tunnelling and Underground Space Technology 28 (2012). 150-158.

Classification of DRI™, BWI™ and CLI™ according to Project Report "13A-98 Drillability Test Methods", published by the Department of Civil and Transport Engineering at the Norwegian University of Science and Technology.

2 Results DRI™ and CLI™

TEST RESULTS

Sample No. (given by SINTEF) Sample ID (given by Client)		1 PO+290	2 PO-025
Brittleness Value (S_{20}, 11.2 - 16.0 mm)	[%]	50.1	44.3
Flakiness (f)		1.33	1.31
Compaction index		0	1
Density (ρ)	[g/cm^3]	2.64	2.68
Sievers' J-Value (SJ)	[mm/10]	2.6	1.7
Abrasion Value Cutter Steel (AVS)	[mg]	21.0	26.0
Quartz content (DTA)	[weight %]	27	26

CALCULATED INDICES

Drilling Rate Index™ (DRI™)	43	35
Cutter Life Index™ (CLI™)	6.2	4.9

CLASSIFICIATION

Category	DRI™	BWI™	CLI™
Extremely Low	≤ 25	≤ 10	< 5
Very Low	26 - 32	11 - 20	5.0 - 5.9
Low	**33 - 42**	**21 - 30**	**6.0 - 7.9**
Medium	**43 - 57**	**31 - 44**	**8.0 - 14.9**
High	**58 - 69**	**45 - 55**	**15 - 34**
Very High	70 - 82	56 - 69	35 - 74
Extremely High	≥ 83	≥ 70	≥ 75

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3 Comments and remarks on NTNU/SINTEF Drillability tests and test methods

3.1 Brittleness Value (S_{20})

Rock brittleness or the ability to be crushed by repeated impacts is determined by the Brittleness Value. The Brittleness Value test is normally performed on three extractions from one representative and homogenized sample of crushed and sieved rock material and should hence be regarded as representative for the tested rock sample.

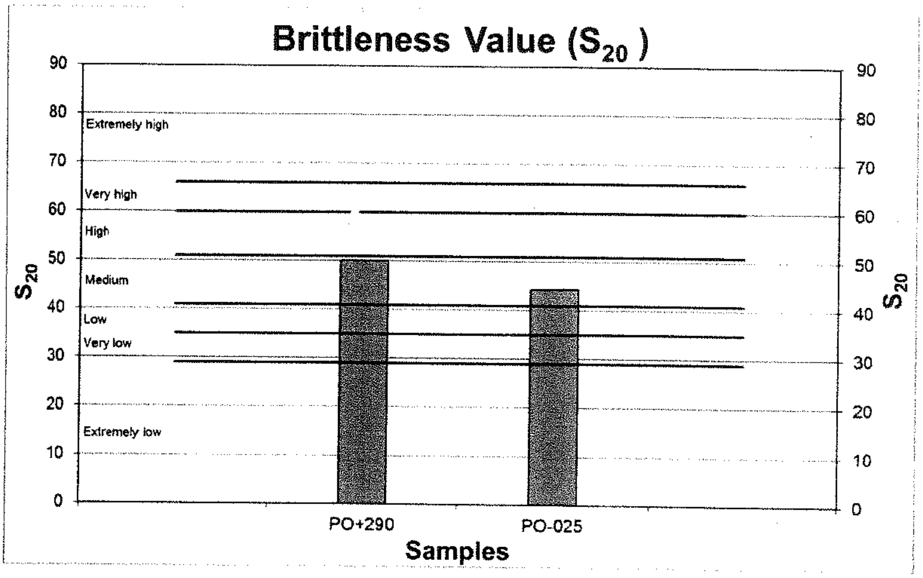

Table 1 Classification of rock brittleness or the ability to be crushed by repeated impacts according to Dahl. F., et al. TUST 28 (2012) 150 -158.

Category – brittleness	Brittleness Value [%]	Cumulative percentage
Extremely low	≤ 29.0	0 – 5 %
Very low	29.1 – 34.9	5 – 15 %
Low	35.0 – 40.9	15 – 35 %
Medium	41.0 – 50.9	35 – 65 %
High	51.0 – 59.9	65 – 85 %
Very high	60.0 – 65.9	85 – 95 %
Extremely high	≥ 66.0	95 – 100 %

3.2 Sievers' J-Value (SJ)

Rock surface hardness or the resistance to indentation is determined by the Sievers' J-Value. The standard number of Sievers' J drillings performed on each sample is 4 to 8, depending on the variation in the texture of the sample. We try to place the holes in soft and hard layers according to a visual interpretation of the composition of the rock. E.g. 60% light and 40% dark layers in a sample would result in 3 holes in the light layer(s) and 2 holes in the dark layer(s). We also try to avoid the soft/hard combination, but we do not always succeed in that matter due to e.g. thin layers of alternating mineral composition.

The Sievers' J charts on pages 11 and 12 show elapsed time in seconds. The Sievers' J-Value is defined as the penetration depth after 200 revolutions and the rotation of the drill bit is hence stopped when this is achieved. As may be seen in some of the graphs, 200 revolutions occur after approximately 67 seconds of drilling.

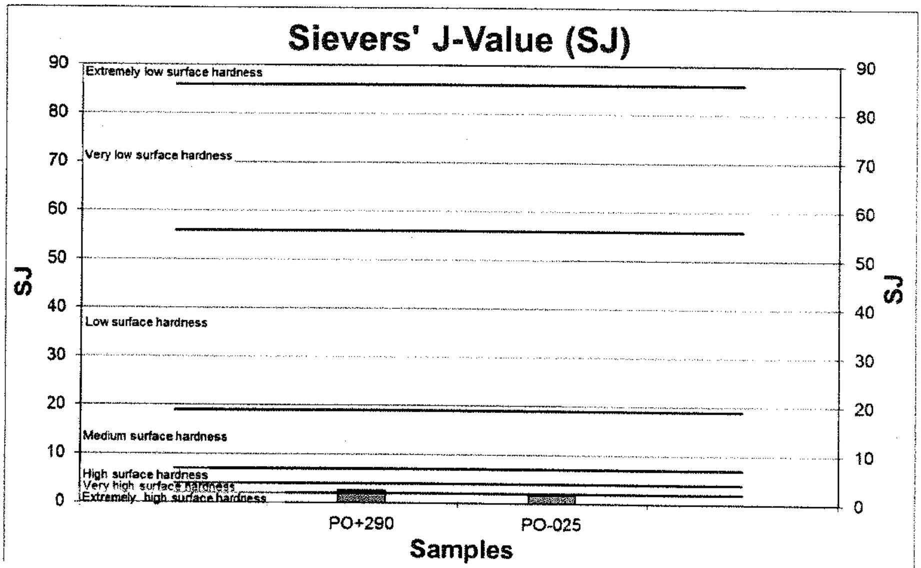

Table 2 Classification of rock surface hardness or the resistance to indentation according to Dahl. F., et al. TUST 28 (2012) 150 -158.

Category – surface hardness	SJ-Value [mm/10]	Cumulative percentage
Extremely high	≤ 2	0 – 5 %
Very high	2.0 – 3.9	5 – 15 %
High	4.0 – 6.9	15 – 35 %
Medium	7.0 – 18.9	35 – 65 %
Low	19.0 – 55.9	65 – 85 %
Very low	56.0 – 85.9	85 – 95 %
Extremely low	≥ 86.0	95 – 100 %

3.3 Abrasion Value Cutter Steel (AVS)

Rock abrasivity or the ability to induce wear on cutter ring steel is determined by the Abrasion Value Cutter Steel. Abrasion test material is taken from the extractions used for the Brittleness test. The AVS tests use test pieces of cutter ring steel. Quartz and other hard minerals will cause abrasion on the test pieces. Grain size, shape and binding are other factors that are believed to have substantial influence on the abrasiveness of the rock.

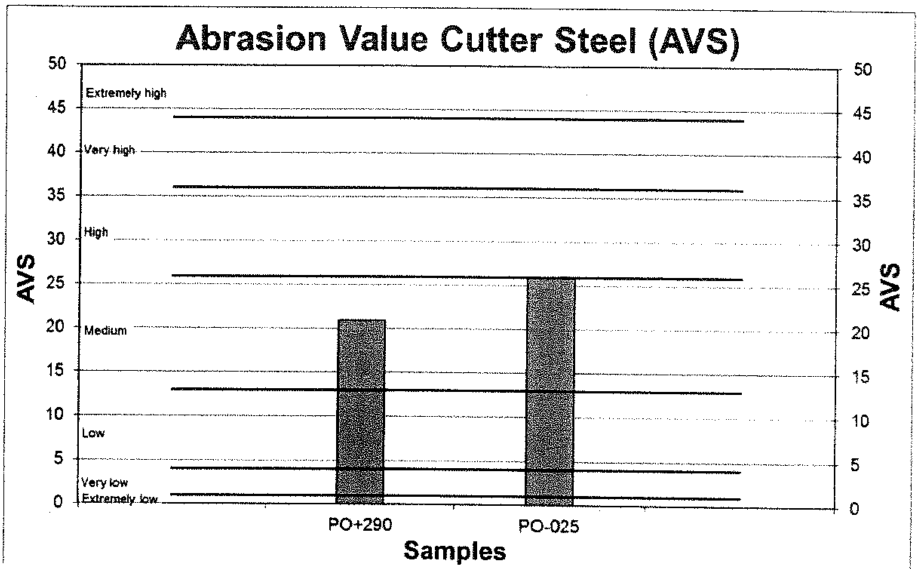

Table 3 Classification of rock abrasivity or the ability to induce wear on cutter ring steel according to Dahl.F., et al. TUST 28 (2012) 150 -158.

Category – cutter steel abrasion	AVS [weight loss mg]	Cumulative percentage
Extremely low	≤ 1.0	0 – 5 %
Very low	1.1 – 3.9	5 – 15 %
Low	4.0 – 12.9	15 – 35 %
Medium	13.0 – 25.9	35 – 65 %
High	26.0 – 35.9	65 – 85 %
Very high	36.0 – 43.9	85 – 95 %
Extremely high	≥ 44.0	95 – 100 %

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3.4 Drilling Rate Index™

The Drilling Rate Index™ is assessed on the basis of the Brittleness Value (S_{20}) and the Sievers' J-Value (SJ). The DRI™ may be described as the Brittleness Value corrected for the rock surface hardness (SJ).

None of the tested samples showed DRI™ classified as extreme.

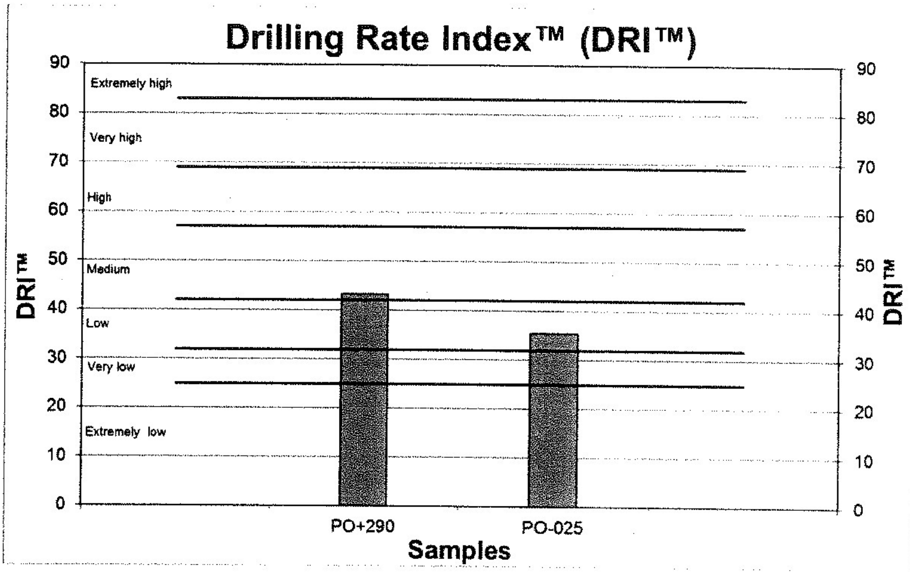

Table 4 Classification of Drilling Rate Index™ according to Project Report "13A-98 Drillability Test Methods", published by the Department of Civil and Transport Engineering, NTNU.

Category – Drilling rate	Drilling Rate Index™	Cumulative percentage
Extremely low	≤ 25	0 – 5 %
Very low	26 – 32	5 – 15 %
Low	33 – 42	15 – 35 %
Medium	43 – 57	35 – 65 %
High	58 – 69	65 – 85 %
Very high	70 – 82	85 – 95 %
Extremely high	≥ 83	95 – 100 %

3.5 Cutter Life Index™

The Cutter Life Index™ is assessed on the basis of Sievers' J-Value (SJ) and the Abrasion Value Cutter Steel (AVS). The CLI™ expresses lifetime of TBM disc cutter steel.

Sample No. 2, PO-025 showed an extremely low CLI™. The cause of the extreme value is the combination of extremely high surface hardness and high abrasion on cutter steel, as shown in Table 5.

Table 5. Samples showing extremely low CLI™ and the associated SJ and AVS values

Sample ID	Cutter Life Index™ (CLI™)	Sievers' J-Value (SJ)	Abrasion Value Cutter Steel (AVS)
PO-025	4.9 Extremely low	1.7 Extremely high surface hardness	26.0 High

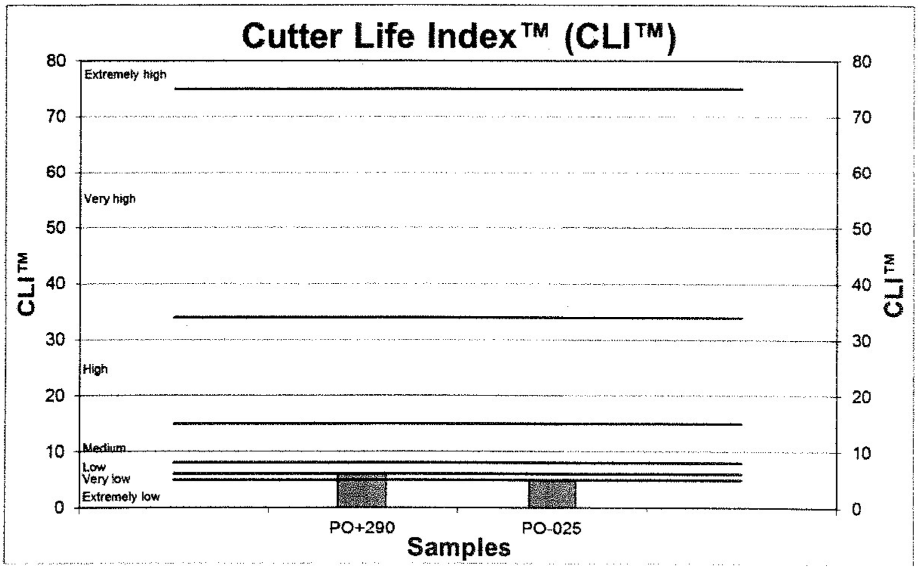

Table 6 Classification of Cutter Life Index™ according to Project Report "13A-98 Drillability Test Methods", published by the Department of Civil and Transport Engineering, NTNU

Category – Cutter life	Cutter Life Index™	Cumulative percentage
Extremely low	< 5	0 – 5 %
Very low	5.0 – 5.9	5 – 15 %
Low	6.0 – 7.9	15 – 35 %
Medium	8.0 – 14.9	35 – 65 %
High	15 – 34	65 – 85 %
Very high	35 – 74	85 – 95 %
Extremely high	≥ 75	95 – 100 %

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4 Individual values from tests used to determine DRI™ and CLI™

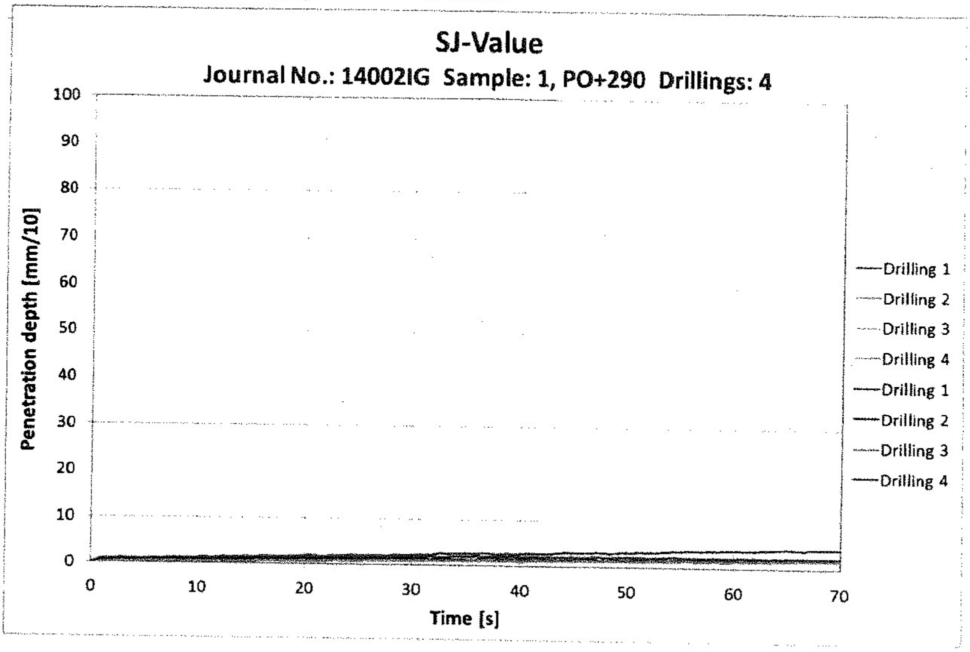
Sample: *PO+290*

Test No.	Brittleness Value S_{20} [%]	Sievers' J-Value SJ [1/10 mm]	Abrasion Value Cutter Steel AVS [mg]
1	53.0	4.2	18
2	45.2	2.0	24
3	52.2	1.8	
4		2.4	
Mean	50.1	2.6	21.0
Stdev	4.31	1.10	4.24

Sample: *PO-025*

Test No.	Brittleness Value S_{20} [%]	Sievers' J-Value SJ [1/10 mm]	Abrasion Value Cutter Steel AVS [mg]
1	45.3	2.2	25
2	43.2	0.9	27
3	44.5	1.7	
4		2.1	
Mean	44.3	1.7	26.0
Stdev	1.10	0.57	1.41

5 Sievers' J-Value drillings presented as charts



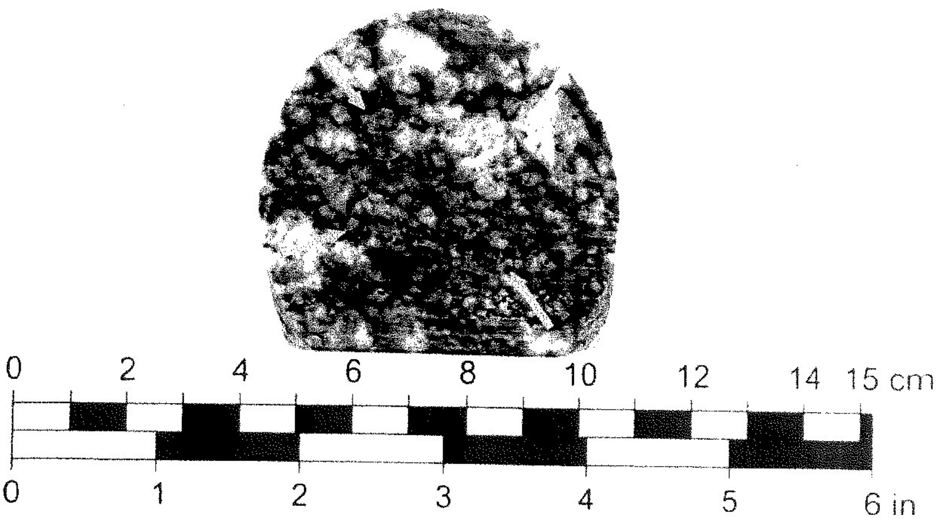


Photo of the Sievers' J (SJ) specimen subsequent to completed testing. Yellow arrows indicate the position of the drillings.

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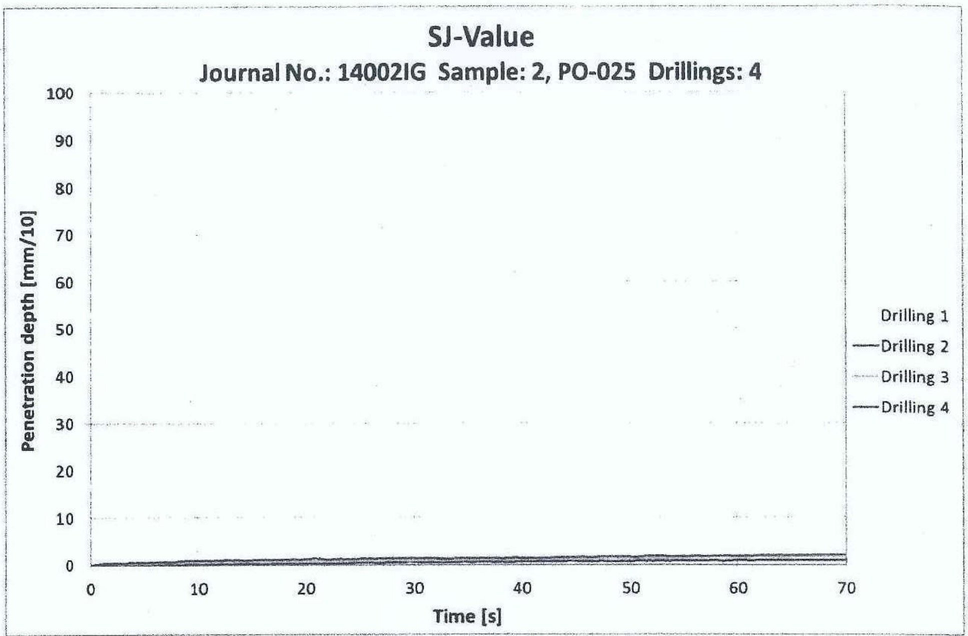


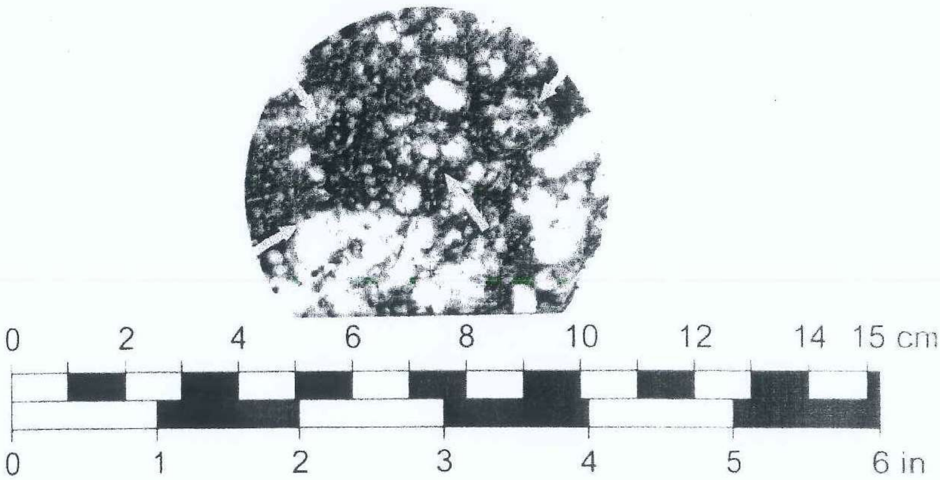


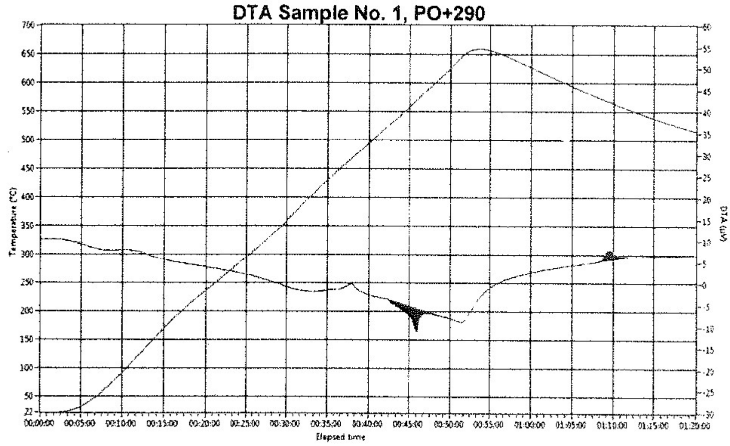
Photo of the Sievers' J (SJ) specimen subsequent to completed testing. Yellow arrows indicate the position of the drillings.

6 Results Differential Thermal Analysis (DTA)

Sample No. (given by SINTEF)	1	2
Sample ID (given by the Client)	PO+290	PO-025
Quartz content [weight %] (DTA)	27	26

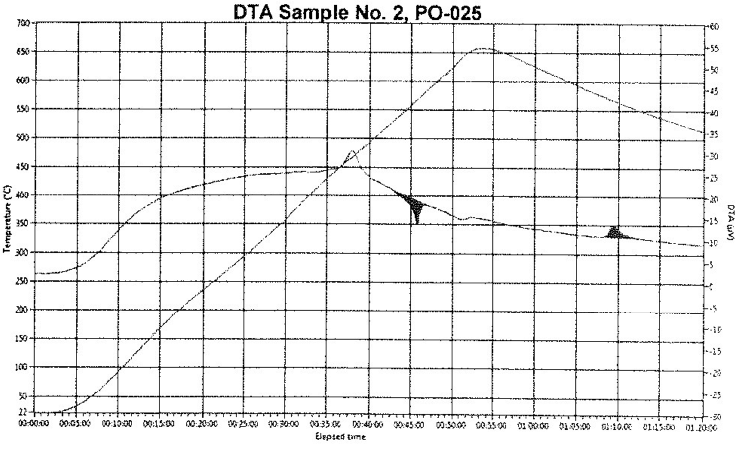
6.1 DTA presented as charts

DTA Sample No. 1, PO+290



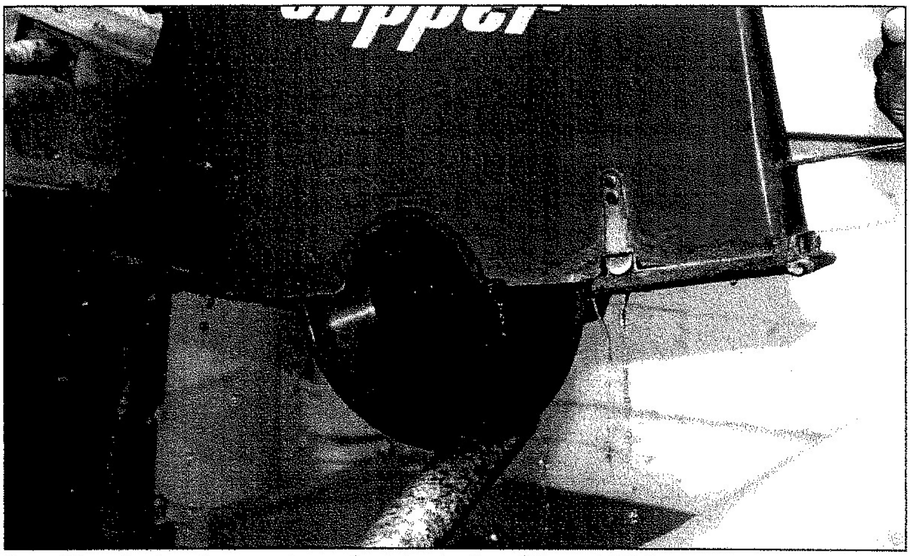
The endothermic reaction (by heating) and exothermic (by cooling) for quartz at 574° C are marked with blue colour.

DTA Sample No. 2, PO-025

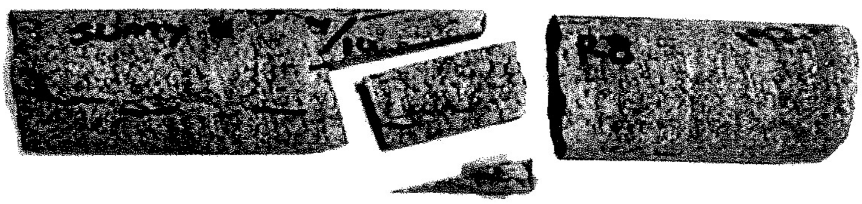


The endothermic reaction (by heating) and exothermic (by cooling) for quartz at 574° C are marked with blue colour.

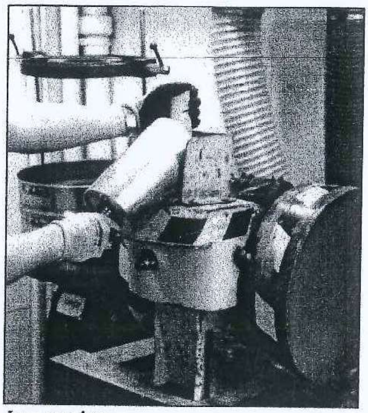
7 Photographs of the test methods, equipment and methodology



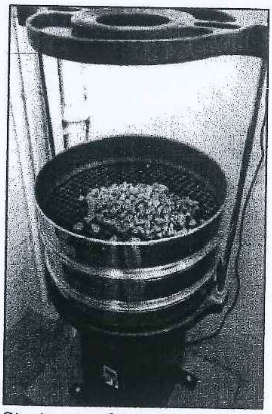
Pre-cutting of a piece from a rock core for determination of Sievers' J-Value.



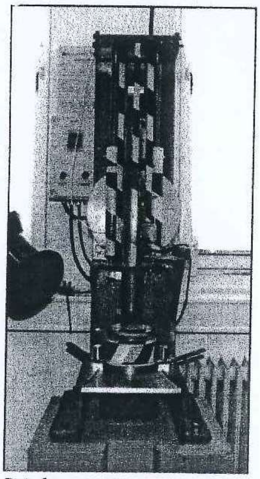
A section of a rock core sample showing the orientation of the pre-cut Sievers' J piece inrelation to the core axis and foliation.



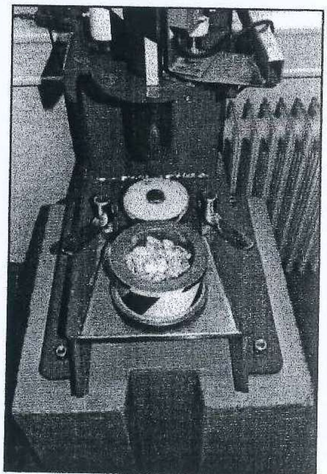
Jaw crusher.



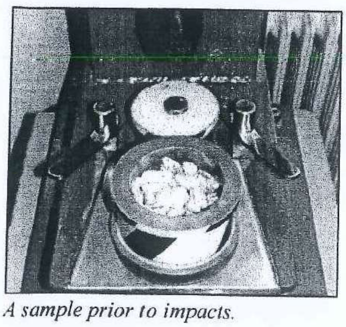
Sieving machine.



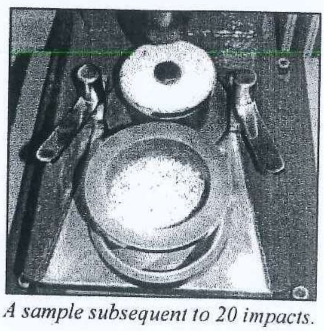
Brittleness test equipment.



Brittleness test. Mortar with sample.



A sample prior to impacts.



A sample subsequent to 20 impacts.

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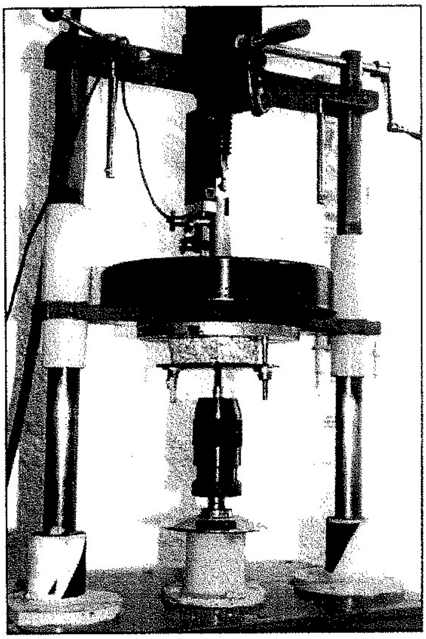
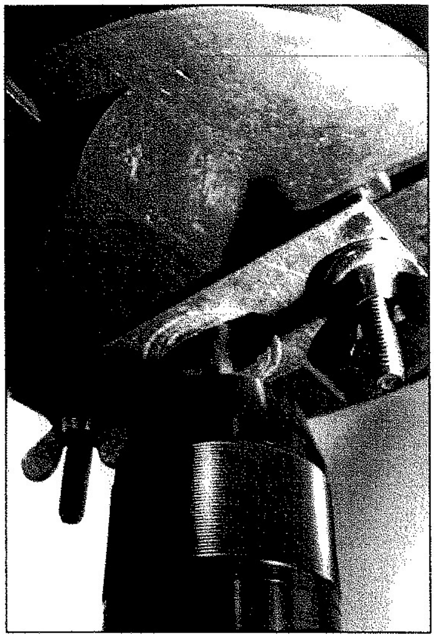
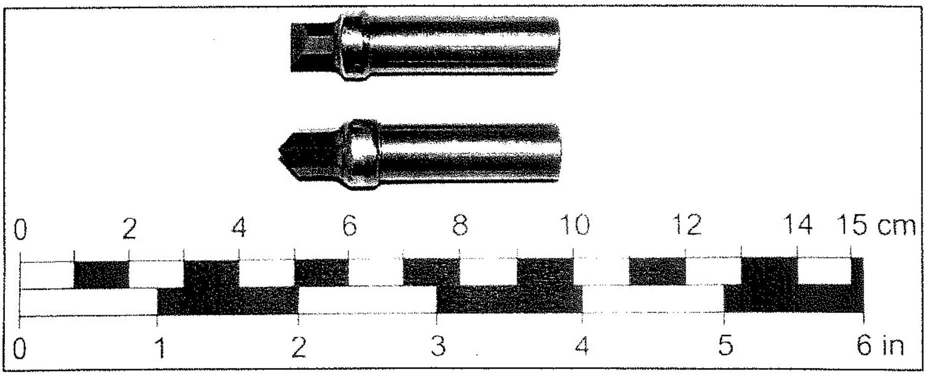


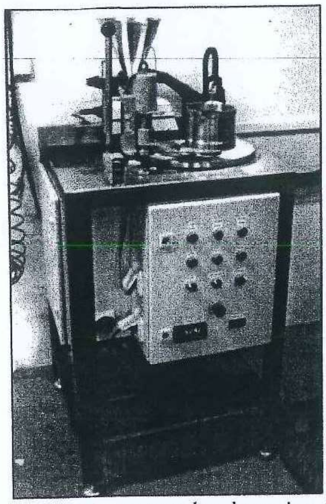
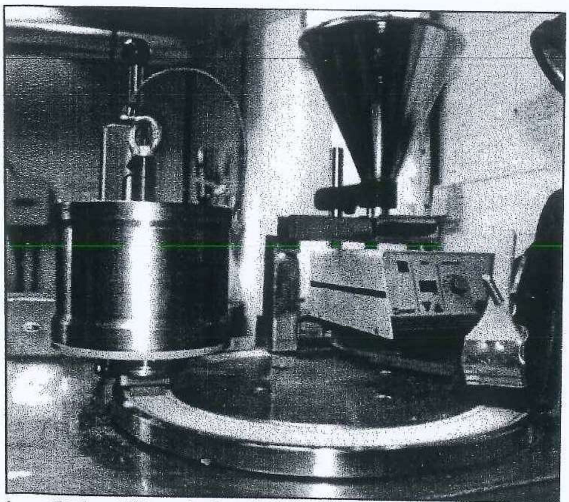
Sievers' J-Value test equipment.



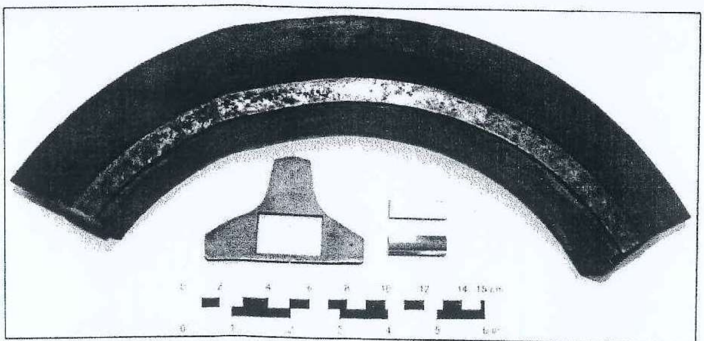
Close up of drill and Sievers' J piece.



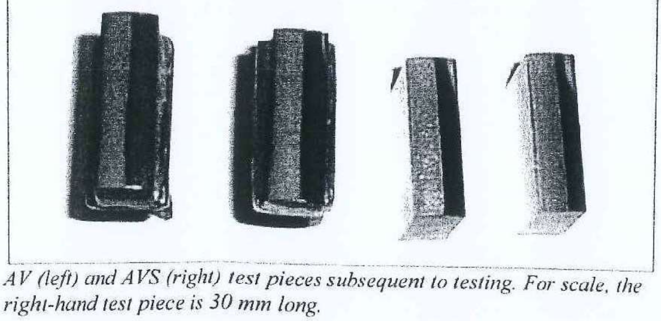
Miniature drills used to determine Sievers' J-Value.

Test equipment used to determine Abrasion Value (AV) and Abrasion Value Cutter Steel (AVS).

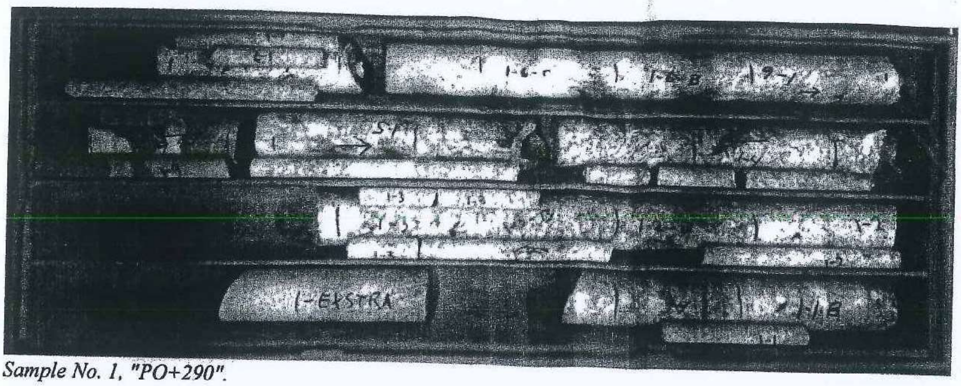


Part of a cutter ring, a 10 mm slice taken from the same ring, and two prepared AVS test pieces which are cut out of the centre of the slice.

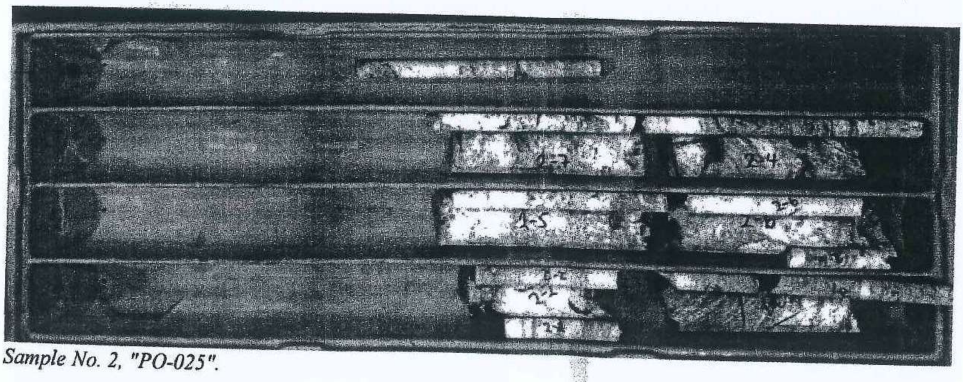


AV (left) and AVS (right) test pieces subsequent to testing. For scale, the right-hand test piece is 30 mm long.

8 Photographs of the received rock samples



Sample No. 1, "PO+290".



Sample No. 2, "PO-025".

Appendix F

TESTRESULTAT

Prøve nr. (gitt av SINTEF)		1
Prøvemerking (gitt av oppdragsgiver)		A
Sievers' J (SJ)	[mm/10]	2,4
Slitasjeverdi kutterstål (AVS)	[mg]	21,5

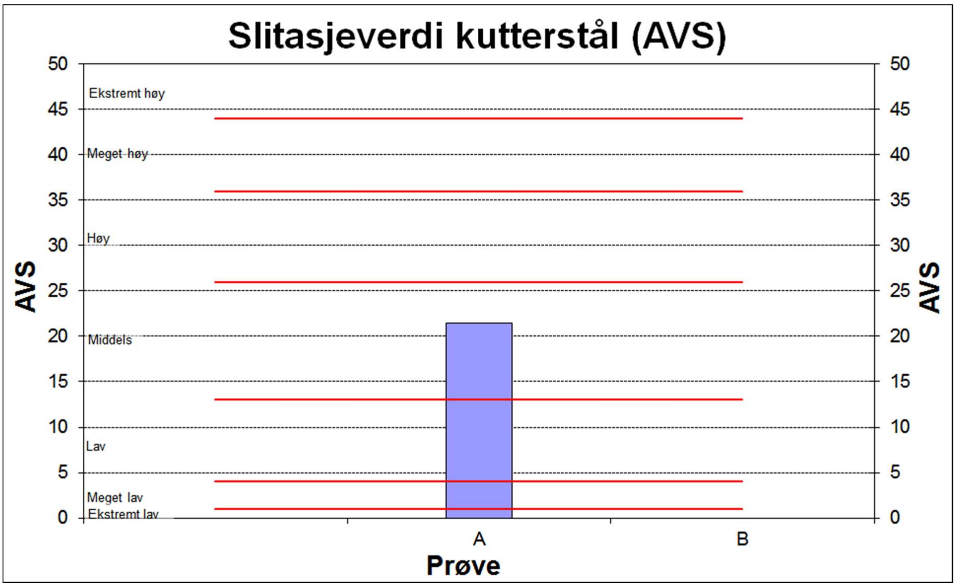
BEREGNEDE INDEKSER

Kutterringlevetid (CLI™)	6,0

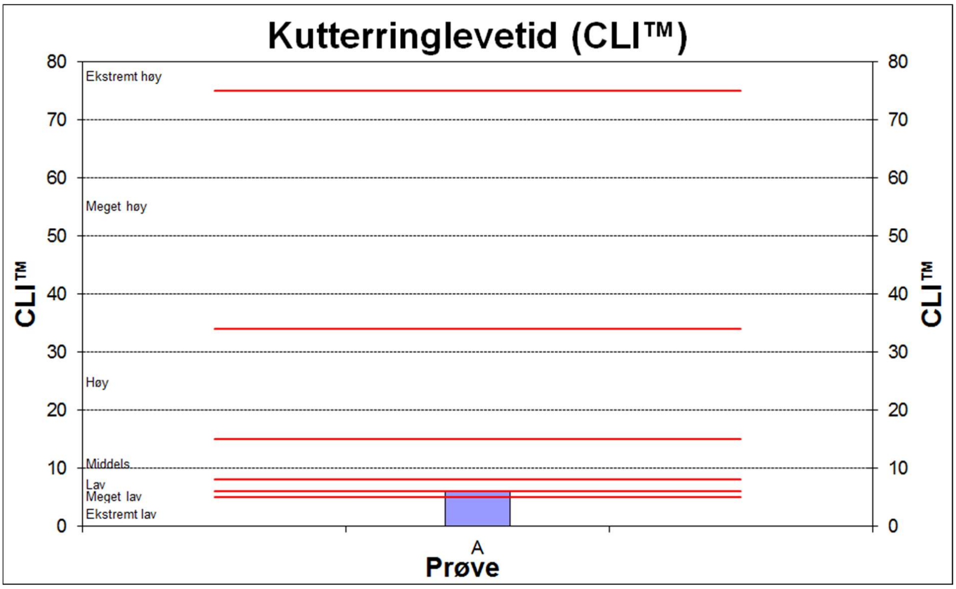
KLASSIFISERING

Kategori	CLI™
Ekstremt lav	< 5
Meget lav	5.0 - 5,9
Lav	**6.0 - 7,9**
Middels	**8.0 - 14,9**
Høy	**15 - 34**
Meget høy	35 - 74
Ekstremt høy	≥ 75

Test nr.	Sievers' J SJ [1/10 mm]	Slitasjeverdi kutterstål AVS [mg]
1	1,8	21
2	4,4	22
3	1,6	
4	1,9	
Middel	2,4	21,5
Standardavvik	1,30	0,71


Slitasjeverdi kutterstål (AVS)
AVS
Ekstremt høy
Meget høy
Høy
Middels
Lav
Meget lav
Ekstremt lav
50
45
40
35
30
25
20
15
10
5
0
A
B
Prøve


Sievers' J (SJ)
SJ
Ekstrem lav overflatehardhet
Meget lav overflatehardhet
Lav overflatehardhet
Middels overflatehardhet
Høy overflatehardhet
Meget høy overflatehardhet
Ekstremt høy overflatehardhet
90
80
70
60
50
40
30
20
10
0
A
Prøve


Kutterringlevetid (CLI™)
80
70
60
50
40
30
20
10
0
Ekstremt høy
Meget høy
Høy
Middels
Lav
Meget lav
Ekstremt lav
CLI™
A
Prøve

Prøve nr.	Diameter [mm]	Lengde [mm]	Vekt [g]	Lengde/diameter forhold	Densitet [kg/m^3]	Trykkfasthet [MPa]	Bruddvinkel [°]	Bruddtype (visuell evaluering)
1-1	43,1	110,9	431,0	2,57	2663	200,1	19	Mangfoldig skjærbrudd
1-2	43,2	101,1	397,2	2,34	2688	175,5	21	Mangfoldig skjærbrudd
1-3	43,2	111,9	438,9	2,59	2683	214,5	23	Mangfoldig skjærbrudd
1-4	43,2	111,8	436,5	2,59	2669	179,4	24	Mangfoldig skjærbrudd
1-5	43,2	89,9	352,1	2,08	2679	193,3	21	Mangfoldig skjærbrudd
Gjennomsnitt					**2676**	**192,6**	**22**	
St. avvik					10	15,8	2