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Rock Engineering and Mine Backfill Specialists

February 2, 2006

Yukon Zinc Corporation
701 - 475 Howe Street
Vancouver, British Columbia
V6C 2B3

Attn.: Mr. Richard Goodwin, P. Eng.
Vice President, Mining

Dear Sir:

RE: GEOMECHANICAL MINE DESIGN ASSESSMENT OF THE WOLVERINE DEPOSIT WOLVERINE PROJECT – YUKON ZINC CORPORATION.

Please find attached a copy of the report entitled "Geomechanical Mine Design Assessment of the Wolverine Deposit - Wolverine Project – Yukon Zinc Corporation". This report presents the analyses and results of geotechnical investigations for the feasibility assessment of the Wolverine Deposit.

I trust that this report meets with your present requirements. Should you have any questions with regard to the content of this report, please do not hesitate to contact me.

Yours truly

ROCKLAND LTD.

Khosrow Aref, Ph. D., P. Eng.

Principal

Attachment

GEOMECHANICAL MINE DESIGN ASSESSMENT OF THE WOLVERINE DEPOSIT

WOLVERINE PROJECT – YUKON ZINC CORPORATION.

February 2, 2006

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1.0 INTRODUCTION

The bankable feasibility study of the Wolverine deposit of Yukon Zinc Corporation (YZC) was initiated in 2005. Dr. Khosrow Aref, P. Eng. of Rockland Ltd was retained to provide geotechnical input to the study and, in particular, to analyze the collected geotechnical data, assess stope and pillar sizes and recommend ground support systems. The following report presents the results of various geotechnical investigations and provides the geomechanical mine design guidelines for the stope excavations and mine development headings. The guidelines include recommendations for maximum span designs, stand up times, ground support requirements, pillar design and geomechanical aspects of mining method.

2.0 OBJECTIVE

2.1 General

Mining has always been, and will continue to be, an activity associated with uncertainty. In the case of new projects, the purpose of a feasibility study is to develop a long-term plan that can be implemented at a level of risk acceptable to the parties concerned. Ground control is one of the basic parameter that has an impact on a mining program. Its importance, related to design and operation, will vary from project to project because of the site specific nature of the orebody.

An optimum stoping operation will normally be function of the following parameters:

- The size and shape of stopes associated with a certain level of dilution.
- The extraction sequence and inter-dependence of individual stopes.
- Ground support requirements, including regional support.
- Required development for access and haulages.

These parameters are inter-related and will have a significant affect on the economics of the proposed mining operations.

2.2 Statement of Problem

According to the recent mineral resource update, the measured and indicated ore reserve category of the Wolverine deposit estimated at 4.52 million tonnes. The reserves are contained in three separate orebodies with different thicknesses. Two thicker massive sulphide orebodies of Wolverine and Lynx are separated with a thinner orebody of Saddle in the middle. Though the overall stratigraphy is relatively consistent across the Wolverine deposit, the orebodies are highly deformed, showing a number of generations of faults with several orientations. YZC plan to mine this massive sulphide deposit at the rate of 1,250 tonnes per day.

2.3 Units

The S.I. unit system was adopted for all data presented in this report. Since some data are commonly expressed in other unit systems, both S.I. and the equivalent unit are included.

3.0 SITE INVESTIGATION

3.1 Visit

Three site visits were undertaken during the course of this investigation. The first visit was preliminary in nature and intended to review the typical drill hole cores and collected geotechnical data. The second site visit was carried out at the early stage of the test mining program and was designed to assess the geotechnical aspects of decline and suitability of the ground support system. Three instruments and equipments: a tape extensometer, the point load test apparatus and pull test equipment were purchased for the rock mechanics program. During the site visit, training of the staff with these instruments / equipments for their applications was carried out. The third site visit was planned towards the end of the test mining program and was directed to evaluate the geotechnical aspects of the decline and test stope area. During each site visit a number of meetings and discussions were also held with the project geologists / geotechnical technicians, engineers / mine planners and mining contractors. A summary of collected information is included in the relevant sections of this report.

3.2 Information Available

The following information was made available for use in this investigation:

- Geological drill hole logs: from WV 00-113 to WV 00-120, from WV 04-121 to WV 04-131, from WV 05-132 to WV 05-188, from 95-01 to WV 97-99, from WV 00-01 to 00-03.
- Geotechnical drill holes logs: 9501GEO to 95GEO24, WV 96-25, WV 97-103, GeoTech WV 05-133 to GeoTech WV 05-189.
- Drill holes information (coordinates, dip and azimuth).

The following Auto Cad drawing and files were provided for the investigation:

- Wolverine deposit drilling at a scale of 1:1500
- Preliminary geology of decline and test stope area at a scale of 1:500
- Decline excavation and test stope area
- Typical mining methods for the Wolverine project

YZC geological staff has collected the geotechnical data based on a document prepared by Golder Associates. Geotechnical parameters that were measured in the core include recovery, RQD, fracture per run, point load index or field strength index, weathering index, presence of faults or broken core, joint set number, joint alteration number, minimum average discontinuities spacing, condition of discontinuities and ground water. A review of geotechnical holes indicated that these parameters were not collected for all holes. The recent geotechnical holes, as included in Section 4.6, were used for this investigation.

3.3 Test Mining Program

Test mining provides the best method of confirming and / or refining mining, ground support requirements and cost. The ground support requirements, and the mine design itself could be estimated on the basis of geotechnical information gathered from the exploration drill core. However, the reliability of geotechnical data collected in drill hole can be compared to the actual conditions underground during the test mining. In addition, the opportunity to observe the performance of the underground openings and ground support provide the best estimates for the feasibility costing.

A test mining program at Wolverine was carried out to provide detailed information on the geological, geotechnical, mining, recovery and cost aspects of the project. The program consisted of a decline which was collared in the southeast of the deposit. The decline is approximately 450m at a -15% grade and has dimensions of approximately 5m wide by 5m high in cross section. The decline was sized for production and will be utilized as the main haulage ramp during the operation. The decline passed through a number of rock types and extended into the ore to expose the HW and to evaluate the proposed mining method. The passage is considered to be representative of much of the other areas of the deposit.

The test mining program had a number of geomechanical objectives:

- Evaluate rock mass quality of various rock types;
- Evaluate the maximum unsupported span in various rock types;
- Evaluate the ground support performance in various rock types; and
- Evaluate ground water inflow.

4.0 SITE CHARACTERIZATION

The site characteristics including, geology, rock properties, rock mass quality and in situ stress are required for the purpose of geomechanical assessment. The site characterization is discussed in the following sections for the entire project site and also the area encountered during the decline development.

4.1 Geology

The geology of the Wolverine deposit, described in the following sections, has been prepared based on discussions with the project geologists and a report written by Dessureau (2005).

The Wolverine deposit is a polymetallic (Zn-Ag-Pb-Cu-Au) volcanogenic massive sulphide deposit hosted within Devonian-Carboniferous argillitic sediments, and rhyolitic volcaniclastic rocks. The deposit can be divided into three major orebodies based on the mineralization style and thickness; the Wolverine orebody, the Saddle orebody, and the Lynx orebody. The Wolverine and Lynx orebodies are thick massive sulphide generally ranging from 3 to over 10m true thickness. They are separated by the Saddle orebody with relatively thin massive sulphide, generally ranging from less than 1 to 4m true thickness. The deposit dips 30^0 to 50^0 to northeast with 35^0 being the average.

4.1.1 Stratigraphy and Rock Types

The overall stratigraphy is relatively consistent across the Wolverine deposit and can be divided into 4 units. The lowermost unit (Unit 1) consists of the FW volcaniclastic, carbonaceous sedimentary and porphyritic intrusive rocks. Stratigraphically above unit 1 is Unit 2, which consists of interbedded argillite, rhyolite, magnetite-silica, magnetite-carbonate, and carbonate-pyrite exhalite followed by Unit 3, which consists of fragmental rhyolite, and Unit 4, which consists of interbedded carbonaceous argillite and greywacke, with lesser basalt and rhyolite.

The Wolverine deposit occurs at the base of Unit 2 at / or near the contact with Unit 1. Unit 2 stratigraphy, although locally faulted and disrupted, is relatively consistent across the deposit. The lower most marker horizon is the massive sulphide mineralization or the orebody. Immediately above the massive sulphide, the HW zone, in both the Wolverine and Lynx orebodies are several meters of graphitic to carbonaceous, massive to well-bedded to laminated argillite. In the HW zone, the next two significant marker horizons are the calcite-pyrite exhalites (EXCP) and the magnetite exhalites (EXMT), separated by tens of meters of interbedded argillite and siliceous sandstone. The immediate FW is generally rhyolitic rock type which is intensely altered with chlorite and/or sericite. The typical rock types of the ore HW and FW zones are presented in Table 1.

Table 1: Summary of Rock Types.

Location	Description	Typical Rock Type
Immediate HW	Interbedded carbonaceous to siliceous argillite and rhyolite.	Typical examples are Argillite (ARMS), Carbonaceous Argillite (ARCB), Carbonaceous Argillite with Rhyolite Lapilli (ARRT), Graphite Argillite (ARGR), Siliceous Argillite (ARSI)
Ore zone	Massive to semi massive sulphide	Typical examples are SSMS, PYMS and SPMS.
Immediate FW	Rhyolite tuff and argillite, chloritic and sericitic alteration strongly foliated	Typical examples are Chlorite Rhyolite Tuff (RHCT), Sericitic Lapilli Tuff (RHSR), Chloritic Lapilli Tuff (RHCL)

4.1.2 Structure and Fabric

There are several dominant foliations present in the Wolverine stratigraphy. These foliations are visible in all units within the Wolverine stratigraphy, but are more obvious in some units, and less pronounced in others. Though these foliations are generally consistent in their orientation, they are locally disrupted around the major fault zones. The Saddle orebody is highly deformed with several generations of faults. These faults have a number of orientations and show significant displacements.

4.1.3 Geology of the Test Mining Area

Several underground visits were made to observe the ground condition along the decline and test stope areas. The following is a brief description of major lithologies encountered during the test mining. Figure 1 presents the locations of these lithologies on the HW, back and FW along the decline and test stope areas.

ARMS (Argillite) - The argillite is a dark grey to black, carbonaceous to graphitic fine grain sedimentary rock type. It occurs as thick beds to fine laminations. It often contains 1-5% disseminated pyrite, and up to 10% siliceous bands.

ARGR (Graphitic Argillite) - The graphitic argillite consists of black, massive and unconsolidated, fine argillitic mud. This often contains 1-5% quartz and carbonate veins.

RHFS (Siliceous Siltstone) - The siliceous siltstone is a light grey to greenish grey unit, consisting of 0.5-2cm siliceous beds interbedded with 1-2mm sericite laminations. This rock type is generally only a few meters thick and occurs below the EXCP rock type.

EXCP (Calcite Pyrite Exhalite) – The calcite pyrite exhalite consists of wispy to irregular bands of calcite, siderite, silica, and pyrite. This rock type occurs tens of meters above the massive sulphide orebody.

SSMS (Massive Sulphide) – These include massive sulphide, pyrite and sphalerite-rich, laminated massive sulphide, sphalerite-rich massive sulphide, pyrite and sphalerite-rich, replacement texture massive sulphide, replacement textured massive chalcopyrite, stringer style chalcopyrite.

ARCL (Chlorite Altered Argillite) – The chlorite altered argillite rock type is well foliated and locally intensely deformed.

RHSR (Sericite Altered Rhyolite or Argillite) – The sericite altered rhyolite or argillite occurred in the HW between the massive sulphide and the argillite. This unit was contained between two or more faults and is only found within a 20m space in the HW. This rock type is intensely sericite altered, and shows very little primary textures.

4.2 Rock Mass Quality

Two rock mass classification systems should ideally be implemented in any rock mass characterization exercise. Two widely used rock mass classifications are Bieniawski's RMR (1976, 1989) and the rock tunnelling index, Q, of Barton et al, (1974). A detailed description of these classification systems are given in the standard rock mechanics books. For the purpose of this investigation, both RMR and Q classification systems were used.

4.2.1 Rock Mass Rating 'RMR'

The Rock Mass Rating, RMR, also known as the Geomechanics Classification, was developed by Bieniawski in 1972-1973 and subsequently modified as more case histories became available and to conform with international standard and procedures. The system has gained wide acceptance in the design of tunnels, chambers, mines, slopes and foundations. The following six parameters are used in the RMR system:

Table 2 - Rock Mass Parameters.

Factors	Factors	Range
uniaxial compressive strength of intact rock	A1	0-15
Rock quality designation, RQD	A2	3-20
spacing of discontinuities	A3	5-20
Condition of discontinuities	A4	0-30
Ground water conditions	A5	0-15
orientation of discontinuities	B	(-12) – 0

Further descriptions of each parameter are available in the standard rock engineering books. A numerical value is selected for each parameter, and the sum of the ratings, yields the Rock Mass Rating, RMR:

$$RMR = A1 + A2 + A3 + A4 + A5 + B$$

Based on this relationship, Bieniawski proposed the following rock mass classifications:

Table 3 - Classification of Rock Mass Rating.

Rock Mass Class	Description	RMR
I	Very Good Rock	81 – 100
II	Good Rock	61 – 80
III	Fair rock	41 – 60
IV	Poor Rock	21 – 40
V	Very Poor Rock	0 –21

Bieniawski, (1989), suggests that poor blasting can reduce RMR by up 20%

4.2.2 Rock Tunnelling Quality Index 'Q'

On the basis of an evaluation of a large number of case histories of underground excavations, Barton et al. (1974) proposed a rock mass Quality Index (Q) for the determination of rock mass characteristics and tunnel support requirement. The Q index is established through the following relations:

$$Q = RQD/J_n * J_r/J_a * J_w/SRF$$

Where:

RQD = 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

Detailed description of Q parameters is given in the standard engineering books. The parameters RQD, J_n, J_r and J_a are normally measured during geotechnical core logging. J_w is estimated from previous experience and drilling reports of water levels. SRF is determined by empirical methods, which relate the estimated in situ stress and rock strength. Barton (1974) proposed the following classifications of rock mass quality based upon the evaluation of Q:

Table 4 - Rock Mass Description Based upon Tunnelling Quality Index.

Tunnelling Quality Index Q	Rock Mass Description
0.001 – 0.01	Exceptionally Poor
0.01 – 0.1	Extremely Poor
0.1 – 1	Very Poor
1 – 4	Poor
4 – 10	Fair
10 – 40	Good
40 – 100	Very Good
100 – 400	Extremely Good
400 – 1000	Exceptionally Good

The physical significance of the system is that the term RQD/Jn, represents, in principal, the average block size of the mass, J_r/J_a, represents the roughness and frictional characteristics of the joint walls or filling materials, J_w/SRF is a complicated empirical factor describing the "active stress". Barton et al. (1974) developed a now widely utilised chart based on Q. A version of this chart based on Grimstad and Barton (1993) is presented in Figure 2. The excavation support ratio (ESR) was introduced to extend the applicability of this empirical chart to temporary excavations and mining conditions. The ESR is a factor used by Barton to account for different degrees of allowable instability (risk) based on the excavation service life and usage. ESR ranges from about 1-3 for permanent or temporary mine opening and up to 300 for severe rock bursting. These support recommendations are largely applicable for low to moderately high stress conditions, from which the chart itself was developed.

4.3 Rock Strength

The Unconfined Compressive Strength (UCS) of rock is one of the major parameters in the application of the rock mass classification systems. In order to establish the UCS, representative core samples are normally collected and tested in a laboratory. Alternatively, point load strength test is a convenient way of determining the UCS. The tests are conducted on representative core and / or lump samples of the various rock types. The point load strength result correlates well with the UCS of rock.

The point load strength test was carried out on the core samples of several drill holes. The results of these tests are included in Appendix I and a summary for the HW, ore and FW zones is presented in Table 5. The point load strength index ($I_{s(50)}$) were used for the purpose of rock mass classification of various zones at Wolverine.

Table 5 – Summary of the Point Load Strength Test Results for HW, Ore and FW Zones.

Location	Rock Type	Drill Hole	Number of Tests	$I_{s(50)}$ Mean Value (MPa)
Above HW	EXCP	WV 05-175, WV 05-186, WV 05-187, WV 05-178	55	3.25
	EXSP	WV 05-176, WV 05-177, WV 05-178,	60	2.2
HW	RHFS	WV 05-178	9	3.93
	RHMS	WV 05-175, WV 05-178	27	2.83
	ARSI	WV 05-177, WV 05-178, WV 05-180, WV 05-188	59	1.69
Immediate HW	ARMS	WV 05-174, WV 05-175, WV - 05-177, WV 05-180	65	1.80
	ARCB	WV 05-174	6	1.43
	ARCL	WV 05-176	13	2.06
Ore Zone	PYMS	WV 05-177	19	3.43
	PMMS	WV 05-176	10	5.62
	PMSM	WV 05-176	6	4.2
	SSMS	WV 05-174, WV 05-175, WV 05-176, WV 05-177, WV 05-180	160	4.86
	CPMS	WV 05-176	8	4.66
Immediate FW Zone	RHCL	WV 05-186	8	1.33
	RHCT	WV 05-176	19	1.45
	QCVN	WV 05-174, WV 05-175, WV 05-176, WV 05-177, WV 05-180, WV 05-186, WV 05-187	41	3.14

The grade and estimated range of UCS according to the International Society of Rock Mechanics (1981) for various rock types are presented in Table 6.

Table 6 - Field Estimates of Uniaxial Compressive Strength.

Grade*	Term	Uniaxial Comp. Strength (MPa)	Point Load Index (MPa)	Field Estimate of Strength	Examples
R6	Extremely Strong	>250	> 10	Specimen can only be chipped with a geological hammer	Fresh basalt, chert, diabase, gneiss, granite, quartzite
R5	Very Strong	100 -250	4 – 10	Specimen requires many blows of a geological hammer to fracture it	Amphibolite, sandstone, basalt, gabbro, gneiss, granodiorite, peridotite, rhyolite, tuff
R4	Strong	50 – 100	2 – 4	Specimen requires more than one blow of geological hammer to fracture it	Limestone, marble, sandstone, schist
R3	Medium Strong	25 – 50	1 – 2	Cannot be scraped or peeled with a pocket knife, specimen can be fractured with a single blow from a geological hammer	Concrete, phyllite, schist, siltstone
R2	Weak	5 – 25	**	Can be peeled with a pocket knife with difficulty, shallow indentation made by firm blow with point of geological hammer	Chalk, claystone, potash, marl, siltstone, shale, rock salt
R1	Very Weak	1 – 5	**	Crumbles under firm blows with point of geological hammer, can be peeled with pocket knife	Highly weathered or altered rock, shale
R0	Extremely Weak	0.25 -1	**	Indented by thumbnail	Stiff fault gouge

*Grade according to Brown (1981).
**Point load tests on rocks with a uniaxial compressive strength below 25MPa are likely to yield highly ambiguous results.

Using the classification presented in Table 6, the range of rock point load strength, grade and term are summarized in Table 7.

Table 7 – Range of Point Load Strength Index, Grade and Term for HW, Ore and FW Zone.

Location	Rock Type	$I_{s(50)}$ Mean Value (MPa)	Grade	Term
Above HW	EXCP, EXSP	2.2-3.25	R4	Strong
HW	RHFS, RHMS, ARSI	1.69-3.93	R3-R4	Medium strong -strong
Immediate HW	ARMS, ARCB, ARCL	1.43 to 2.06	R3	Medium strong
Ore zone	SSMS, PYMS, PMMS, PMSM, CPMS	3.43 - 5.62	R4-R5	Strong-very strong
Immediate FW zone	RHCL, RHCT,	1.33 to 1.45	R3	Medium strong
	QCVN	3.14	R4	Strong

With the exception of QCVN (Quartz Carbonate Vein), the point load strength index ranges from 1.43 to 2.06MPa and 1.33 to 1.45MPa for the immediate HW and FW respectively. When the UCS falls below 25MPa ($I_{s(50)}$ < 1MPa), the point load test results will not be accurate. The lower ranges have been recorded in the drill hole logs. Using the results of point load index strength test and drill hole logs, the representative ranges of UCS for the HW and FW zones are from “extremely weak to medium strong” and “weak to medium strong” respectively. The ore zone with average point load strength values, ranging from 3.43 to 4.86MPa, is rated as “strong to very strong” rock types. Therefore, according to the point load strength tests, examination of drill hole cores and review of core logs, the ore zone is more competent than the immediate HW and FW zones.

The point load strength test is mainly used to predict or/and verify the UCS of rock. On average, UCS is 20-25 times the point load strength. A program of UCS tests in a commercial laboratory should be carried out to establish the correlation factor between the point load strength and UCS of various rock types. Further, as mining advances at Wolverine, representative rock samples should be collected and the point load strength test should be carried out to provide a range of rock strength.

4.4 In-Situ Stress

The magnitude and the orientation of in-situ stress will influence ground behaviour. There have been no in-situ stress measurements conducted at Wolverine project site and the author is not aware of any direct measurements made in the local vicinity. Therefore, an estimate of in-situ stress based upon previous experience and published data was made.

Arjang and Herget (1997) have reviewed the results of stress measurements conducted in a number of mines in the Canadian Shield. To a depth of 1000 m, the ratios of σ_{Hmax} / σ_v and σ_{Hmin}/σ_V indicate large dispersions. Arjang and Herget (1997) have suggested the following relationships for the average far field stresses in the Canadian Shield:

$$\sigma_V : 0.026 \text{ MPa} / (\text{m})$$

$$\sigma_{Hmax}/\sigma_V : 7.44 \times \text{Depth (m)}^{-0.198}$$

$$\sigma_{Hmin} / \sigma_V : 2.81 \times \text{Depth (m)}^{-0.120}$$

Assuming a range of mining depth of 200 to 400m, the major (σ_{Hmax}) and intermediate (σ_{Hmin}) principal stresses are 2.6 to 2.3 times and 1.5 to 1.4 times the vertical stress respectively.

In order to more accurately estimate the actual stress levels in the rock as mining activity progresses, it is recommended to make measurements of the pre-mining stress during the mine development phase.

4.5 Ground Water

The following ground water description was prepared based upon a memorandum by Gartner Lee (2006) and underground observations during the site visits.

- The water table in the hill and rock ridges surrounding the mine site appears to be between the surface and 50 to 100m depth.

- Minimum mine inflow rate of 7L/s was calculated based on infiltration of 40% of annual precipitation during a normal year over a catchment of about 1.1 km^2.
- Water dripping in some parts of the decline, probably due to discharging from the faults, was present and required nearly constant pumping to ensure good working conditions.
- According to the water pumping data during the test mining, a rate of less than 5L/min, indicating, in general, dry condition for the decline, was recorded.

Therefore, depending upon a number of parameters (e.g. underground fault locations, fault characteristics, ground water table location, etc.) during the underground development, a range of “dry to medium inflow” conditions could occur.

4.6 Rock Mass Quality based on the Drill Holes

A total number of 57 drill holes were used to assess the rock mass quality. The location and distribution of these drill holes in the Wolverine, Saddle and Lynx orebodies are shown in Figure 3 and Table 8. In order to characterize and compare the rock mass quality across the Wolverine deposit, each orebody was divided into the upper, center and lower zones. Figure 4 presents these divisions for the Wolverine, Saddle and Lynx orebodies.

Table 8 – Drill Holes Used for the Geomechanical Assessment.

	Lynx	Saddle	Wolverine
Upper Deposit	WV05-149	WV05-135	WV05-132
	WV05-153	WV05-136	WV05-150
	WV05-184	WV05-137	WV05-155
	WV05-185	WV05-138	WV05-163
		WV05-141	WV05-165
		WV05-143	
		WV05-145	
		WV05-146	
		WV05-147	
		WV05-148	
Center Deposit	WV05-151	WV05-139	WV05-133
	WV05-154	WV05-140	WV05-157
	WV05-159	WV05-142	WV05-160
	WV05-176	WV05-144	WV05-162
	WV05-177	WV05-156	WV05-167
	WV05-178		WV05-168
	WV05-180		WV05-170
			WV05-187
			WV05-188
			WV05-189
Lower Deposit	WV05-158		WV05-134
	WV05-161		WV05-152
	WV05-166		WV05-164
	WV05-169		WV05-171
	WV05-172		WV05-186
	WV05-173		
	WV05-174		
	WV05-175		
	WV05-179		
	WV05-181		
	WV05-182		
	WV05-183		

YZC staff collected the rock mass classification parameters (RQD, J_n, J_r and J_a) during the geotechnical core logging. The majority of these ratings were initially collected based upon the core run. Subsequently, YZC staff adjusted the information based upon the rock units to reflect the geomechanical domains.

The geotechnical parameters were reviewed to characterize the rock mass quality of the Lynx, Saddle and Wolverine orebodies. The objective was to identify any distinct geomechanical differences between these orebodies and their HW, ore and FW zones. The geomechanical domains of approximately 5m above and below the ore zone were considered for the HW and FW. The range and the most typical rating of each parameter were established. Table 9 presents the range and typical ratings of these parameters for various zones of the Wolverine deposit.

Table 9 – Summary of Range and Typical Rock Mass Ratings for the Wolverine Deposit.

Orebody	Zone	HW/Ore /FW	RQD			Jn			Jr			Ja		
			low	average	high	low	typical	high	low	typical	high	low	typical	high
Lynx	Upper	HW	0	5	29	3	3	20	1	1.5	3	0.75	2	6
		Ore	0	40	100	0.5	0.5	20	0.5	3	3	1	2	4
		FW	0	7	65	3	3	20	1	1	3	1	4	4
	Center	HW	0	9	78	3	3	20	1	1	3	1	4	4
		Ore	0	45	100	0.5	0.5	20	1	3	3	1	2	4
		FW	0	8	89	2	3	20	1	1	3	2	4	4
	Lower	HW	0	9	38	2	6	20	1	3	3	0.5	2	4
		Ore	0	56	100	0.5	0.5	20	1	3	3	1	2	4
		FW	0	30	100	2	3	20	0.5	1	3	1	4	4
Saddle	Upper	HW	0	5	43	3	3	20	1	1	3	0.75	4	12
		Ore	17	64	94	0.5	0.5	3	0.5	1.5	3	2	3	8
		FW	0	19	48	2	3	15	1	1	3	0.5	4	8
	Center	HW	0	20	49	3	6	20	1	1.5	3	2	4	4
		Ore	0	50	100	0.5	0.5	3	1.5	1.5	3	2	2	3
		FW	0	32	82	3	3	15	1	1	3	2	4	4
Wolverine	Upper	HW	0	13	67	3	3	20	0.5	1	3	1	4	4
		Ore	14	44	74	0.5	3	3	1	3	3	1	2	4
		FW	0	6	29	2	3	6	1	1	3	1	4	4
	Center	HW	0	1	10	2	20	20	0.5	2	3	1	4	4
		Ore	0	25	83	0.5	0.5	20	0.5	3	3	1	2	4
		FW	0	24	97	2	6	20	0.5	1	3	1	4	4
	Lower	HW	0	17	78	3	3	20	1	1	3	2	4	4
		Ore	4	60	98	0.5	0.5	6	1	1.5	3	1	2	4
		FW	0	24	85	2	3	20	1	1	3	1	4	4

Assuming dry to minor inflow condition (J_w =1) and the medium stress environment (SRF=1), Q values were calculated for the typical ratings. Table 10 presents Q values and corresponding rock mass description for various zones.

Table 10 – Summary of Typical Rock Mass Quality for the Wolverine Deposit.

Orebody	Zone	HW/Ore /FW	Q	Description
Lynx	Upper	HW	1.3	Poor
		Ore	120	Extremely good
		FW	0.6	Very poor
	Center	HW	0.8	Very poor
		Ore	135	Extremely good
		FW	0.7	Very poor
	Lower	HW	2.3	Poor
		Ore	168	Extremely good
		FW	2.5	Poor
Saddle	Upper	HW	0.4	Very poor
		Ore	64	Very good
		FW	1.6	Poor
	Center	HW	1.3	Poor
		Ore	75	Very good
		FW	2.6	Poor
Wolverine	Upper	HW	1.1	Poor
		Ore	22	Good
		FW	0.5	Very Poor
	Center	HW	0.03	Extremely poor
		Ore	75	Very good
		FW	1	Poor
	Lower	HW	1.4	Poor
		Ore	90	Very good
		FW	2	Poor

Therefore, the rock mass quality ratings highlight the distinct separation between ore and immediate HW/FW rock zones. The ore rock quality can be described as “good to extremely good”. With the exception of the “extremely poor” HW rock quality in the center zone of the Wolverine orebody, the rock mass quality in the immediate HW and FW is rated as “very poor to poor”. In the upper zones, Saddle’s HW is poorer than Wolverine and Lynx, however; it has a higher rock mass quality for the FW zone. In the center zones, Saddle’ HW has a higher rock mass quality than Lynx and Wolverine. In lower zones, Lynx and Wolverine have similar HW and FW zones. A comparison of the ore rock mass quality indicates that Lynx has a higher rating than Saddle and Wolverine orebodies for all zones.

4.7 Rock Mass Quality of the Test Mining Area

Several underground visits were made to identify various geomechanical zones along the decline and test stope areas. The major lithologies encountered during the test mining were: ARMS (Argillite), ARGR (Graphitic Argillite), RHFS (Siliceous Siltstone), EXCP (Calcite Pyrite Exhalite), SSMS (Massive Sulphide), ARCL (Chlorite Altered Argillite) and RHSR (Sericite Altered Rhyolite or Argillite). The rock mass quality was estimated based on the Q (Barton, 1974) and the RMR (Bieniawski, 1989) rock mass classifications. A total number of eight locations, representing typical rock mass quality along the decline and test stope areas, were selected. Table 11 presents a summary of rock mass classification results at the selected locations. The photographic records of underground observations at each location are presented in Photographs 1 to 29 and their locations are identified in Figure 1.

Table 11 – Rock Mass Quality along the Decline and Test Stope Area.

Location*	Rock Type**	Q	Description***	RMR	Description	Photograph
1	ARMS	0.01-1	Extremely Poor – Very Poor	18-35	Very Poor - Poor	1 to 4
2	EXCP	1.9-9	Poor-Fair	52-70	Fair -Good	5 to 7
3	EXCP	1.9-9	Poor-Fair	52-70	Fair -Good	8 to 10
4	ARMS	0.08-0.6	Extremely Poor – Very Poor	20-35	Very Poor-Poor	11 to 13
5	SSMS	4.1-6.6	Fair	49-67	Fair-Good	14 to 16
6	ARMS – SSMS	4.1-6.6	Fair	46-64	Fair-Good	17 to 21
7	SSMS	4.1-6.6	Fair	49-67	Fair-Good	22 to 25
8	SSMS	4.1-6.6	Fair	49-67	Fair – Good	26 to 29

*See locations in Figure 1
**ARMS = Argillite, EXCP= Calcite-Pyrite Exhalite, SSMS = Massive Sulphide ore zone
***Assuming SRF= 1 (low stress – medium stress) and J_w = 1 (dry to minor inflow)

Similar to the geotechnical core logging results, there are significant differences between the rock mass quality of the HW and ore. The ore (SSMS) rock quality rated as “fair” and “fair to good” based upon the Q and RMR rock mass classifications respectively. However, the immediate HW (ARMS) has a lower rock mass quality, ranging from “extremely poor to very poor” and “very poor to poor” based on the Q and RMR rock mass classifications respectively. It should be noted that the joint orientation adjustment factor was not included in the RMR calculation.

5.0 ASSESSMENT OF EXCAVATION GEOMETRY AND STABILITY

5.1 General

The stability of an underground excavation is dependent upon a large number of parameters. These parameters include structural features, fault and shear zones, rock quality, state of stress, excavation geometry, support characteristics, etc. Each parameter has a different influence upon the stability. Due to this complexity, no single method is available for the stability evaluation. Empirical, analytical and numerical techniques are normally thus used to assess the stability.

Rock mass classification systems are normally used to assess stability. The two most widely rock mass classifications in the mining industry are Bieniawski's RMR (1976, 1989) and Barton et al. Q (1974). Both methods incorporate geological, geometric and design / engineering parameters to derive a qualitative value of the rock mass quality. The similarities between RMR and Q stem from the use of identical or very similar parameters for calculating the final rock mass quality rating. The differences between the systems lie in the weight given to similar parameters in each scheme. Ideally, both rock mass classification systems should be used and compared in any project.

The RMR and Q rock mass classifications were employed to assess stability at Wolverine. Both classifications, through extensive case histories, specify the maximum stable unsupported span against various rock mass qualities.

5.2 Empirical Evaluation

Barton et al. (1974) and Barton (1988, 1994) described the application of the Q system for rock mass classification as the determination of no-support limits for various types of excavations. Figure 5 presents the case histories of database for Q-system for supported and unsupported cases. Where Q values are known, the maximum unsupported span of excavations could be established.

Based upon the geotechnical core logging, the typical Q ranges of the immediate HW, ore and FW zone have been established (Section 4.6). With the exception of the "extremely poor" HW rock quality in the center of the Wolverine orebody, the rock mass quality in the immediate HW and FW has a range of 0.4 to 2.3 (Table 10). According to Figure 5 and using an ESR of 1.6, the maximum unsupported span ranges from approximately 2.5 to 5m respectively. The ore has Q values ranging from approximately 22 to 168 (Table 10). Again, using Figure 5 and an ESR of 1.6, the maximum unsupported span ranges from 11 to 24m.

These spans are possible in the absence of continuous, adversely-oriented geological structures which normally control stability. In other words, the suggested unsupported span is applicable where the Q values are consistently equal or greater than specified values. Areas with lower Q values are obviously unable to sustain the suggested spans. In such conditions, provision must be made for additional ground support.

6.0 GROUND SUPPORT

6.1 Bolts

For any support system specification; three major design parameters should be considered:

- bolt capacity;
- bolt length; and,
- bolt spacing.

Empirical methods are widely used for the specification of rock support systems in mining projects. The empirical methods can be divided into two major categories:

- Rules of Thumb
- Rock Mass Classification System

Rules of thumb were developed based upon various project experiences gained in the construction of tunnels, caverns and mine openings. The support specifications based on rock mass classification systems distinguish between different rock masses and specify the rock support system accordingly. It is proven that the rock mass classification system, in conjunction with monitoring and sound engineering judgement, provides an excellent guideline for the design of ground support systems.

6.1.2 Rules of Thumb

Rules of thumb, developed from industry experience, have been used for many years to specify the bolting specifications for mining projects. Although other methods, such as rock mass classifications, are normally used to establish support guidelines for underground excavations, Rules of thumb could provide the preliminary ground support design for projects, or where detailed rock mass characterization are not available. These rules should be optimized and refined during the actual mining operation.

The rock arch concept is one of the approaches for assessing the structural stability of an excavation. The formation of stress arches above underground excavations occurs in most mines. The formation of the arches is the result of the stress redistribution in the rock as the opening is formed. The rock in the arch is subjected to compressive stresses. The location of the stable arch beyond the excavation is dependent upon the rock mass properties and excavation span. In order to maintain the stability of the natural arch, the de-stressed rock between the excavation boundary and natural arch boundary must be stabilized. This can be achieved by reinforcing the rock by bolts that anchor above the stressed arch boundary. Therefore, the bolt specifications should be related to dimension of de-stressed zone above the excavation which is related to the span of excavation. A number of equations have been developed for bolt specifications. Typical equations are shown in Table 12:

Table 12 - Equations for Estimating of Rockbolt Length Based upon Rules of Thumb.

Rockbolt Equation**	Location of Application/ Reference
$L = 1.4 + 0.184 * W$	Norwegian institute for rock blasting techniques IFF (1979)
$L = 1.8 + 0.013 * W^2$	Based on experience on Snowy Mountain Project. Pender et al (1963)
$L = 0.3 * W$	based on experience with the Australian Tunnelling Method Rabcewicz (1955)
$L = 0.5 * W$	based on experiences at Mount Isa Mine

** L: length of bolt (in meters), W: width of opening (in meters)

The length of rock bolts is designed according to the span of the excavation. Normally, the length of the bolt ranges from one third to one half of the span of the excavation. Based on experience gained during the test mining program at Wolverine, the length of bolt equivalent to the one half of excavation span is recommended.

In order that bolts act together rather than individually, they must be located within a certain distance of each other. For 1.8 and 2.4m (6' and 8') long bolts, this distance should not be greater than 1.5m (5') otherwise spacing is insufficient for bolt interaction. Again, based on experience gained during the test mining program, the spacing of 1.2 by 1.2m (4' by 4') and 1.5 by 1.5m (5' by 5') for the back and walls respectively were selected for application at Wolverine.

It is also essential that patterns are regular and examined to meet the most critical conditions expected. This rule can be relaxed somewhat for temporary openings that will be open for less than 6 months. Bolt spacing should be uniform to preserve the interlocking nature of the rock in the back. This is particularly important where the rock quality deteriorates. It should be noted that where adverse ground conditions are encountered during development, additional support should be installed. Major intersections require additional support as specified above.

6.1.3 Rock Mass Classification System

As indicated previously, Barton (1974) has developed a relationship between rock mass quality, opening size and support requirements. This relationship is used as the basis for assessing the support needs in different mine openings. Inputs include a range of opening dimensions; and the predicted effect of mining induced-stress (through adjustments to the SRF factor). Support recommendations based on the Q-system have evolved over the years as more and more case histories have been added to the database. Barton (1988) presented a tabulated series of detailed support recommendations based on different combinations of rock quality, Q, and the Equivalent Span (Span/ESR) ratio. Grimstad et al (1993) proposed a summary graph, presented in Figure 6 based on these recommendations. This graph was developed for permanent support in civil engineering for tunnels, shafts and caverns and therefore, is likely to be conservative for mining applications.

The empirical method by Grimstad et al (1993) was employed to estimate the ground support requirements. According to the proposed mining method at Wolverine (see Section 8), the permanent mine development headings will be 4m wide. In Figure 6, a drift with Span/ESR ratio of 2.5 and Q value of 0.4-2.3 require ground support consisting of fibre reinforced shotcrete (FRS) or mesh reinforced shotcrete and bolts. In the poorer ground condition, where Q values drop to approximately 0.4, FRS and bolts are required. The shotcrete thickness should be 50mm (2”) in the back. Further information on the application of shotcrete is given in the next section. In a better ground quality (Q>1), pattern bolting is expected to be sufficient. In the ore zone, where Q values range from 22 to 168 and using the Span/ESR ratio of 2.5, application of pattern bolting will be sufficient.

The suggested ground support recommendations are applicable where the Q values are consistently equal or greater than specified values. Areas with lower Q values require additional ground support. It should be noted that the excavation support ratio is related to the use for which the excavation is intended, and the extent to which some degree of instability is acceptable. The span calculated and ground support recommended by this method does not apply to multiple opening situations. This is primarily due to the effect of stress redistribution. Barton’s method is not sensitive to such changing conditions and therefore only provides an appraisal of initial conditions prior to full production mining.

6.2 Shotcrete

Shotcrete was employed at a number of locations during the test mining program. The regular shotcrete has been initially employed with unsatisfactory results. Subsequently, in order to improve the performance of shotcrete, Fondu Cement, as shotcrete accelerator, was added. Though the performance was improved, the application of shotcrete with Fondu, in general, appeared to be patchy with limited thickness. The Superstick product was also experimented. The result appeared to be better than the previous applications of the shotcrete. Therefore, a high quality FRS should be used at Wolverine. The importance of shotcrete quality is discussed below.

Since the publication of Q, much progress has been made in developing high performance shotcrete. Melbey and Garshol (1999) summarized their recent experience with high performance, wet-sprayed concrete with steel fibre and presented a chart relating rock mass quality (Q) and shotcrete thickness. Their experience is compared in Figure 7 with data from Grimstad et al. (1993). The center line of each range from small (lower limit) to large (upper limit) spans represent tunnels with a span of about 10m. This figure indicates that today's shotcrete technology allows stable excavation with significantly less shotcrete than proposed by Grimstad et al. (1993). Therefore, it is critical to use the high quality FRS at Wolverine.

Hoek et al. (1997) provide a table of recommended shotcrete applications in underground mining for different rock mass conditions (Table 13). This table provides a simple link between rock-mass description, behaviour and recommendations of shotcrete. It can also serve to check the designs obtained by other means. The support recommendations cover the whole spectrum of anticipated excavation behaviour including wedge-type instability in low-stressed rock to moderately violent rock fracturing during rock burst. According to this table, for highly jointed igneous or metamorphic rocks in low stress conditions, similar to the ground condition as anticipated at Wolverine, 50mm (2") SFR shotcrete is required. This is similar to the ground support recommendation based on the Q rock mass classification (see section 6.1.2).

Table 13 - Shotcrete Recommendations in Hard-Rock Underground Mining (after Hoek et al., 2005).

Rock mass designation	Rock Mass Behaviour	Support Recommendations	Shotcrete Applications
Massive metamorphic or igneous rock. Low stress conditions.	No spalling, slabbing or failure.	None.	None.
Massive sedimentary rock. Low stress conditions.	Surfaces of some shales, siltstones, or claystones may slake as a result of moisture content change.	Sealing surface to prevent slaking.	Apply 25 mm thickness of plain shotcrete to permanent surfaces as soon as possible after excavation. Repair shotcrete damage due to blasting.
Massive rock with single wide fault or shear zone	Fault gouge may be weak and erodible and may cause stability problems in adjacent jointed rock.	Provision of support and surface sealing in vicinity of weak fault of shear zone	Remove weak material to a depth equal to width of fault or shear zone and grout rebar into adjacent sound rock. Weld mesh can be used if required to provide temporary rockfall support. Fill void with plain shotcrete. Extend steel fibre reinforced shotcrete laterally for at least width of gouge zone.
Massive metamorphic or igneous rock. High stress conditions	Surface slabbing, spalling and possible rockburst damage.	Retention of broken rock and control of rock mass dilation	Apply 50 mm shotcrete over weld mesh anchored behind bolt faceplates, or apply 50 mm of steel fibre reinforced shotcrete on rock and install rockbolts with faceplates; then apply second 25 mm shotcrete layer. Extend shotcrete application down sidewalls where required.
Massive sedimentary rock. High stress conditions.	Surface slabbing, spalling and possible squeezing in shales and soft rocks.	Retention of broken rock and control of squeezing.	Apply 75 mm layer of fibre reinforced shotcrete directly on clean rock. Rockbolts or dowels are also needed for additional support
Metamorphic or igneous rock with a few widely spaced joints. Low stress conditions.	Potential for wedges or blocks to fall or slide due to gravity loading.	Provision of support in addition to that available from rockbolts or cables.	Apply 50 mm of steel fibre reinforced shotcrete to rock surfaces on which joint traces are exposed.
Sedimentary rock with a few widely spaced bedding planes and joints. Low stress conditions.	Potential for wedges or blocks to fall or slide due to gravity loading. Bedding plane exposures may deteriorate in time	Provision of support in addition to that available from rockbolts or cables. Sealing of weak bedding plane exposures.	Apply 50 mm of steel fibre reinforced shotcrete on rock surface on which discontinuity traces are exposed, with particular attention to bedding plane traces.
Jointed metamorphic or igneous rock. High stress conditions.	Combined structural and stress controlled failures around opening boundary.	Retention of broken rock and control of rock mass dilation	Apply 75 mm plain shotcrete over weld mesh anchored behind bolt faceplates or apply 75 mm of steel fibre reinforced shotcrete on rock, install rockbolts with faceplates and then apply second 25 mm shotcrete layer. Thicker shotcrete layers may be required at high stress concentrations.
Bedded and jointed weak sedimentary rock. High stress conditions.	Slabbing, spalling and possibly squeezing.	Control of rock mass failure and squeezing.	Apply 75 mm of steel fibre reinforced shotcrete to clean rock surfaces as soon as possible, install rockbolts, with faceplates, through shotcrete, apply second 75 mm shotcrete layer.
Highly jointed metamorphic or igneous rock. Low stress conditions.	Ravelling of small wedges and blocks defined by intersecting joints.	Prevention of progressive ravelling.	Apply 50 mm of steel fibre reinforced shotcrete on clean rock surface in roof of excavation. Rockbolts or dowels may be needed for additional support for large blocks.

Highly jointed and bedded sedimentary rock. Low stress conditions.	Bed separation in wide span excavations and ravelling of bedding traces in inclined faces.	Control of bed separation and ravelling.	Rockbolts or dowels required to control bed separation. Apply 75 mm of fibre reinforced shotcrete to bedding plane traces before bolting.
Heavily jointed igneous or metamorphic rock, conglomerates or cemented rockfill. High stress conditions.	Squeezing and 'plastic' flow of rock mass around opening.	Control of rock mass failure and dilation.	Apply 100 mm of steel fibre reinforced shotcrete as soon as possible and install rockbolts, with face-plates, through shotcrete. Apply additional 50 mm of shotcrete if required. Extend support down sidewalls if necessary.
Heavily jointed sedimentary rock with clay coated surfaces. High stress conditions.	Squeezing and 'plastic' flow of rock mass around opening. Clay rich rocks may swell.	Control of rock mass failure and dilation.	Apply 50 mm of steel fibre reinforced shotcrete as soon as possible, install lattice girders or light steel sets, with invert struts where required, then more steel fibre reinforced shotcrete to cover sets or girders. Forepoling or spiling may be required to stabilise face ahead of excavation. Gaps may be left in final shotcrete to allow for movement resulting from squeezing or swelling. Gap should be closed once opening is stable.
Mild rockburst conditions in massive rock subjected to high stress conditions.	Spalling, slabbing and mild rockbursts.	Retention of broken rock and control of failure propagation.	Apply 50 to 100 mm of shotcrete over mesh or cable lacing which is firmly attached to the rock surface by means of yielding rockbolts or cablebolts.

6.3 Ground Support of the Test Mining Area

Table 14 presents the ground support system employed at different locations in the decline and test stope area. Though the steel arch and timber support system has been used at the portal and a location in Argillite, the main support elements were resin rebar and split set in conjunction with mesh and occasional FRS. In very poor to poor rock quality (ARMS), resin rebar did not provide sufficient anchorage; however, split set, mesh with occasional application of FRS was successful. Photographs 2, 3, 4 and 12 show the steel arches and timber ground support in extremely poor to very poor rock mass quality. In the poor to fair ground rock quality (EXCP and SSMS), resin rebar with mesh was implemented. These types of ground support are shown in photographs 6, 7, 8, 28 and 29. The main type of ground support on the walls was split set.

Table 14 – Rock Mass Quality along the Decline and Test Stope Area.

Location*	Rock Type**	Q	Description	Ground Support (in the back)	Bolt Length m (ft)	Pattern m (ft)	Photograph
1	ARMS	0.01-1	Extremely Poor – Very Poor	Steel arch, split set, timber, shotcrete	2.4 (8')	1.2 to 1.8 (4' to 6') apart	2,3,4
2	EXCP	1.9-9	Poor-Fair	Resin rebar, mesh	2.4 (8')	1.2 by 1.2 (4' by 4')	5,8,7
3	EXCP	1.9-9	Poor-Fair	Resin rebar, mesh	2.4 (8')	1.2 by 1.2 (4' by 4')	8,9
4	ARMS	0.08-0.6	Extremely Poor – Very Poor	Timber, split set, mesh, shotcrete	1.8 and 2.4 (6' and 8')	1.2 by 1.2 (4' by 4')	11,12
5	SSMS	4.1-6.6	Fair	Resin rebar, split set, mesh, shotcrete	1.8 and 2.4 (6' and 8')	1.2 by 1.2 (4' by 4')	15,16
6	ARMS – SSMS	4.1-6.6	Fair	Resin rebar, split set, mesh	1.8 and 2.4 (6' and 8')	1.2 by 1.2 (4' by 4')	18,19,20
7	SSMS	4.1-6.6	Fair	Resin rebar, splits, mesh	1.8 and 2.4 (6' and 8')	1.2 by 1.2 (4' by 4')	22,24
8	SSMS	4.1-6.6	Fair	Resin rebar, split set, mesh	1.8 and 2.4 (6' and 8')	1.2 by 1.2 (4' by 4')	28,29

*See locations in Figure 1
**ARMS = Argillite, EXCP= Calcite-Pyrite Exhalite, SSMS = Massive Sulphide
Average span in the decline= 5m (16')

The support guideline for the Wolverine deposit was prepared based upon the rock mass classification and experience gained during the test mining program. Tables 15 and 16 present the support guideline based upon the geomechanical zones / rock types for the back and walls respectively.

The experience during the decline excavation in the "very poor to poor" ground condition suggest that the back should be supported immediately after the blast. The application of a thin layer of FRS to the rock surfaces will improve the stability. During the decline excavation, sloughing of the back occurred at two locations, at the end of main decline and the south access drive. It appears that the direction of the foliation has profound affects on the back stability. Where decline traveled parallel to foliation, no major failure occurred. Once decline perpendicular to the foliation failure has occurred. The influence of structural directions should be examined during the mine development at Wolverine.

Table 15 – General Guidelines on Support Type and Density for the Back.

Geomechanics zone / Rock Type	Function	Opening	Operating Life	Span m (ft)	Support Type	Bolt Length m (ft)	Spacing* m (ft)	Shotcrete*** cm (in)	Comments
Argillite or similar rock type	Access	Decline	Long	5 (16’)	Split set+ mesh some resin rebar	2.4 (8’)	1.2 by 1.2 (4’ by 4’)	50 (2”)	Also requires timber support, steel set arches 5m (16’) span and straps for occasional use
	Intersection	Decline intersection	Long	>5(16’)	Split set+ mesh +some resin rebar	2.4 to 4.5 (8’ to 15’)	1.5 by 1.5 (5’ by 5’) to 1.8m by 1.8m (6’ by 6’)	50 (2”)	use of Final support depends on geometry of intersection
EXCP or similar rock type		Decline	Long	5(16’)	Resin rebar+ Mesh +some split set	2.4(8’)	1.2 by 1.2 (4’ by 4’)		
Ore	Stope	Herringbone Primary	Short	4 (13’)	Resin rebar + mesh and some split set	1.8 to 2.4 (6’ and 8’)	1.2 by 1.2 (4’ by 4’)		Close spacing because men constantly working under exposed back
		Herringbone Secondary	Short	4 (13’)	Resin rebar+ mesh and some split set	1.8 to 2.4 (6’ and 8’)	1.2 by 1.2 (4’ by 4’)		Close spacing because men constantly working under exposed back

*Spacing values quoted represent minimum. Condition will occur where specific block of ground support density require additional support, resulting in higher densities.
**Split set is SS33 – Resin rebar is #7 – Mesh is #8 gauge
***Shotcrete is the fibre reinforced shotcrete

Table 16 – General Guidelines on Support Type and Density for the Walls.

Geomechanics zone / Rock Type	Function	Opening	Operating Life	Span m (ft)	Support Type	Bolt Length m (ft)	Spacing* m (ft)	Shotcrete*** Cm (in)
Argillite or similar rock type	Access	Decline	Long	5 (16')	Split set and some mesh	2.4 (8')	1.5 by 1.5 (5' by 5')	25 (1")
	Intersection	Decline intersection	Long	>5(16')	Split set and mesh	2.4 (8')	1.5 by 1.5 (5' by 5')	25 (1")
EXCP or similar rock type		Decline	Long	5(16')	split set and some mesh	2.4 (8')	1.8 by 1.8 (6' by 6')	
Ore	Stope	Herringbone Primary	Short	4(13')	Split set and some mesh	1.8 to 2.4 (6' and 8')	1.8 by 1.8 (6' by 6')	
		Herringbone Secondary	Short	4 (13')	Split set and some mesh	1.8 to 2.4 (6' and 8')	1.8 by 1.8 (6' by 6')	

*Spacing values quoted represent minimum. Condition will occur where specific block of ground support density require additional support, resulting in higher densities.
**Split set is SS33 – Resin rebar is #7 – Mesh is #8 gauge
***Shotcrete is the fibre reinforced shotcrete

7.0 PILLAR Design

7.1 General

Pillar strength against pillar stress has been employed as the criterion for pillar stability assessment. Coates (1981) defines pillars as the in-situ rock between two or more underground openings. The term height or thickness is restricted to the dimension normal to the plane of working; the length of pillar is the greatest dimension in the plane of the working and the width the lesser dimension. The function of the pillars is to ensure stability of the back during the mining. The pillars will consist of mineable ore that may or may not be recovered depending upon the mining method adopted.

When estimating an appropriate size for pillars, there are three factors to consider. First, the load that is applied to the pillar must be determined. Second, the strength of the pillar must be assessed and a suitable Safety Factor (SF) applied. The third aspect is to examine the reaction of the HW and FW (to pillar stresses).

7.2 Pillar Stress

Pillar load can be estimated using the Tributary Area theory. This theory assumes that each pillar will be loaded by the normal stress acting over the area of the back or roof, tributary to that pillar.

For the general case, pillar stress can be estimated from:

$$\sigma_p = \sigma_n / (1-R)$$

Where:

σ_p = the average pillar stress in the direction normal to the orebody.

σ_n = the normal component of the pre-mining stress field.

The extraction ratio (R) is determined by areas measured in the plane of the orebody and given by:

$$R = 1 - W_p^2 / (W_r + W_p)^2$$

For square room and pillar mining, and

$$R = 1 - W_p / (W_r + W_p)$$

For panel and pillar mining

Where:

W_p = the width of the pillar.
W_r = the width or span of the adjacent room or stope.

The Tributary Area theory will generally produce higher stresses than more accurate numerical methods and can for certain geometries produce non-conservative results. The method, however, does provide a reasonable starting point and was used to carry out a preliminary evaluation of the stress levels that may be encountered during mining.

7.3 Pillar Strength

Pillar strength is a function of the size of the pillar, the quality of the rock mass within the pillar and the strength of the intact rock. Although various methods exist for estimating pillar strength, they generally have the following format:

$$\sigma_p = k\,\sigma_c\,(W_p\,\alpha\,/H_p\,\beta)$$

Where:

σ_p = the estimated pillar strength
k = a scaling factor
σ_c = uniaxial compressive strength of intact rock
W_p = pillar width
H_p = pillar height
α and β = site specific constants

The conventional approach for the design of pillars is material and scale dependent. Where the rock mass quality is known, a relationship developed by Hoek and Brown (1980) can be used. This relationship in terms of the pillar strength to specimen strength versus pillar aspect ratio is plotted in Figure 8. It is evident from this relationship that the strength of a pillar is reduced as its slenderness increases. The Hoek and Brown relationship does not consider geological weakness such as faults. In these cases, induced stresses may cause slip along the fault surfaces and hence leads to pillar instability.

Where the width to height ratio of a pillar and UCS are known, the average pillar strength could be calculated. Then, factor of safety (SF), defined as the ratio between average pillar strength to pillar stress, is established. A SF of 1.0 or less implies that the pillar is theoretically unstable and failure could propagate across the entire pillar, resulting in its collapse. Since entry type mining method will be used at Wolverine, SF in excess of 1.5 should be considered for the pillar design.

According to the proposed mining method (Section 8), the pillar dimensions of 4m (width) by 4m (height) will be used at Wolverine. SF was calculated for the range of ore rock mass quality of “good to extremely good” (Table 11). Employing the relationship in Figure 8, using pillar width / height ratio of 1 and the range of ore rock mass quality, the average pillar strength was calculated. Then, the pillar stress was estimated based on the panel and pillar mining formula (See section 7.2). Table 17 presents SF determination for the range of rock mass quality.

Table 17 – Factor of Safety Determination.

Scenario	Width/Height Ratio	Average Pillar Strength / UCS	Average Pillar Strength*	Pillar Stress ($2\gamma z$)	SF (Pillar Strength / Stress)
1	1	$0.3\sigma_c$	30	10-20	3 - 1.5
2	1	$1.45\sigma_c$	145	10-20	14.5 – 7.3

*Estimated $\sigma_{c=}$ 100MPa for the ore zone
**mining depth range from 200 to 400m

According to this analysis SF is greater than 1.5 for the range of ore rock mass quality and pillars are classified as “stable”.

A number of assumptions were made in the SF calculations. As mining progresses and additional information becomes available, the validity of these assumptions should be examined. For example, the good quality rock condition was assumed for the stoping areas, where the rock mass quality changes significantly, the pillar stability could also change. In addition, the average stresses were calculated based on the empirical analysis. The numerical modelling should be employed during the mining operation. The modelling will allow pillar stresses, and in particular where multiple openings are present, to be more accurately evaluated. In addition, as panel mining progresses and during the pillar recovery, the induced stresses will change. The numerical modelling can assist in the pillar and stope stability assessment during the mining operation.

8.0 GEOMECHANICAL ASPECTS OF MINING

Three different variants of the drift and fill mining methods have been selected at Wolverine (Goodwin, 2006). Drift and Fill Mining with Side Slash (DFSS), Drift and Fill with Retreat Panels (DFRP), Drift and Fill with Primary and Secondary Panels (DFPS). Mining method selection will be determined by horizontal ore thickness, as shown in Table 18.

Table 18 – Selection of Mining Method by Ore Thickness.

Horizontal Ore Thickness (m)	Mining Method
>7	Drift and Fill with Side Slash (DFSS)
4 to 7	Drift and Fill with Retreat Panels (DFSS)
<4	Drift and Fill with Primary and Secondary Panels (DFPS)

Figure 9 shows the three mining methods on cross-sectional views.

The primary reasons for selecting these mining methods are as follows:

- A high percentage extraction of the deposit can be achieved as no permanent pillars are required and thinner zones are mineable.
- Most of the mining backs will be in ore, providing a competent back for most stope headings.
- The poor ground of the HW will have minimal exposure, controlling external dilution and enhancing safety for the workers.
- High productivity can be maintained due to multiple working faces.

8.1 Drift and Fill with Primary and Secondary Panels (DFPS)

The DFPS mining method will be employed where the horizontal thickness of the orebody is greater than 7m. The method requires a FW stope drift of 4 m wide in the ore along the FW contact of the ore. The stoping panels will then be excavated at 4 m wide in a "herringbone" fashion at an angle of approximately 45º (Figure 9). These panels will be driven from the FW drift and extend into the argillitic HW contact. The primary and secondary sequence will be used for the panel excavation. The primary panels will initially be mined and backfilled with waste rock and paste fill. The secondary panels will then be mined between the backfilled primary stopes, with ore in the back and the exposed backfill of the two adjacent primary panels as walls. The secondary panels will then also be backfilled as tightly as possible. The mining proceeds in a retreat fashion and filling the FW drift simultaneously.

Key geomechanical aspects of the DFPS mining method can be summarized as follows:

- Herringbone panels will be excavated in ore which, according to the geotechnical core logging, will have the range of "good to extremely good" rock mass quality. The ground support will mainly include resin rebar and mesh as specified in Tables 15 and 16.
- As herringbone panels approach the end of panels, the HW will be exposed. Typically, according to the geotechnical core logging, HW has the range of "very poor to poor" rock mass quality and short unsupported span. Therefore, where HW is unsupported, excessive dilution is anticipated at the ends of the herringbone panels once it is exposed.
- Careful blasting is required adjacent to the pillar line and as lift approaches the extremity of the orebody and panel reach HW. The use of low density explosive or similar explosive type should be investigated.
- Tight backfilling is required in primary panels if additional support in the secondary stope is to be minimized.
- Cemented paste fill placed in the panels should be placed as tight as possible.
- The backfill bulkheads should be placed as close to the FW stope drift as possible to minimize the unsupported span.

- During secondary mining, stopes should be retreated in a series of blocks in order to minimize excessive fill exposure.

An area that is expected to require specific care and attention during the implementation of herringbone mining is the intersection between the FW drift and the panels. At this point, the intersection span could open up from about 5 to 10m wide. All equipment, materials and muck pass through this point. The continued stability of this area is therefore very important. Thus, it is recommended that the area be knitted together with a combination of 2.4 to 4.5m (8' to15') resin rebars, depending upon the span and rock quality, and tied together with straps.

It is possible that the last round of the herringbone panels will be drilled and blasted to double length with extension steel such that the ore is completely blasted without having to control the back. The drift end will be mucked as completely as possible, using a remote controlled scooptram if required. Continuous unraveling of the HW will occur. Mucking will continue until dilution is excessive, rendering the muckpile uneconomic, at which point the panel will be closed for filling.

8.2 Drift and Fill with Retreat Panels (DFRP)

The DFRS mining method will be employed where the ore thickness range from 4 to 7m. The method is similar to the DFPS mining method; however, panels are mined adjacent to each other and mining retreat from extremity of the orebody towards the stope access (Figure 9). Therefore, each panel should be backfilled, cured then mining can proceed in the adjacent panel. The bulkhead for each stope pour will be placed in the FW drift and both the panel and a portion of the FW drift will be filled.

The backfill is the major component of the proposed drift and fill mining methods and following aspects should be considered:

- Since mining progresses in the adjacent panels, the fill strength becomes critical. Therefore, tests should be carried out to establish the required fill strength.

- The backfill strength required is controlled by a number of factors including grading of fill particles; cement content; moisture content; mineralogy and chemistry of fill particles; curing time and placement techniques
- The backfill strength required is a function of stresses generated within the fill mass. In turn these are due to self weight; degree of arching between solid rock walls; blast damage and ground movement.

8.3 Drift and Fill with Side Slash (DFSS)

DFSS will be employed in the area which has less than 4 m horizontal thickness. A FW drift will initially be driven to the ultimate extent of ore. Then, the ore is slashed incrementally and mining retreats toward the stope access (Figure 9). The slashing process will be achieved with horizontal drill jumbo holes and broken ore will be mucked out remotely. The maximum stoping width will depend upon the reach of the jumbo drill. A 4m drill steel is assumed, limiting this mining method to a maximum horizontal ore thickness of 4 m at the average dip of 34º. The incremental blast of 6 to 8m has been planned during the retreat mining. Since the exposed HW in the stope is unsupported, it will become unstable. The geotechnical characteristics of the HW will determine the stability of the unsupported span. Where unsupported HW collapses, a bulkhead will be placed in the FW drift and the stope void will be filled as tightly as possible with paste fill.

8.4 Discussion

The proposed guidelines for the DFPS, DFRP and DFSS mining methods should be verified with known conditions and failures observed at the mine. To further refine the proposed mining method, it is important to increase the geotechnical database and quantitative data during the underground development. A program of systematic monitoring of rock quality and ground conditions should be instigated. In addition, a record of stoping performance should be maintained. A number of stress meters are recommended to monitor the stress changes as mining advances. The stress meter should be installed in the HW of the stope to monitor the relaxation of the stope back as the herringbones are being developed. The stress gauges should also be installed in the pillars to monitor stress increase as mining advances. In addition, numerical modelling should be carried out to evaluate the stope and pillar sizes. This process will allow optimization of the proposed mining methods.

As with any mining operation, the original design tool "evolves" over time to reflect refinement in the geological models and mining methods. The evolution of the mine's ground control program must take place to reflect an increase in the understanding of structural controls, stress conditions, and rock mass strength as mining progresses. Therefore, refined and optimized geomechanical mine design guidelines would be based upon actual site and operating conditions.

9.0 MINING EFFICIENCY

Dilution and recovery, though reasonably simple to define, are complex and dynamic processes. A number of different causes for dilution can be identified, some of which are:

- Diamond drilling – adequacy, accuracy
- Interpretation accuracy
- Production holes – setup, deviation, pick up
- Blasting - design, procedure, efficiency
- Ground support
- Block design-dimension, sequence, development
- Rock quality
- Mining method
- Mine supervision
- Incentive system

However, dilution can be divided into two broad categories:

1. Planned dilution (or designed dilution or primary dilution);
2. Additional dilution (or unplanned dilution or secondary dilution).

Planned dilution is defined as the material below cut-off that lies within the mining lines as determined by the mining method and stope layout. Additional dilution is that additional rock below cut off that is derived from the outside the mining line and is a result of poor mining practices, wall slough, etc.

A complicating factor that arises in any dilution or recovery estimate is time. As mine progresses through its various stages from pre-production development to primary stoping to pillar recovery and mine closure, the amount of dilution and recovery will change. This concept is illustrated in Figure 10.

During the early stage of mining, dilution can arise from various process associated with development, lack of knowledge of the orebody, and general learning curve associated with opening a new mine. As primary panels get underway, dilution should become constant (i.e. within some band of variation). However, as secondary panel gets underway and / or the mine approaches closure, dilution can expect to increase. Conversely, recovery will decrease. Individual stopes will follow a similar pattern, with dilution increasing as a stope is progressively mined out. In addition, further dilution (and the potential for reduced recovery) may occur if the period over which a stope is mined is extended.

9.1 Potential Source of Dilution at Wolverine

Dilution is a complex variable which is difficult to quantify for the proposed mining methods at Wolverine. The amount of dilution will vary during the various stages of mining. The following have been identified as the potential sources of dilution:

- Undercutting of the weak geologic structures of the HW at the ends as panels are mined out. This is expected to be the most common source for dilution.
- Excessive panel size inducing HW instability.
- Variation in the rock quality in the HW and FW.
- Blast damage in the panel due to high powder factors.

Knowledge of orebody geometry ahead of the advancing stope / panels will be a very important aspect of dilution control. Where a roll occurs in the back of a panel, bringing the poorer quality rock into the stope, will be extremely critical to the planning of the stope and designing ground support.

During primary stoping, recoveries are expected to be very high, reduced only by unexpected wall sloughs or the ends' failures. As stoping progresses, however, and more pillar mining is undertaken, recovery can be expected to decrease.

10.0 INSTRUMENTATION AND MONITORING

The empirical and numerical geomechanical mine design give relative rather than absolute answers and require calibration against existing excavation behaviour. In order to validate selected input parameters and failure criteria, an instrumentation and monitoring program should be implemented during the mining operation. The main goal of the program is to verify the panel / pillar design and ensure the adequacy of the recommended ground support. The instrumentation and monitoring results can be compared with empirical and / or numerical model predicted displacement and stresses. Then, the calibrated model may be used to optimize the mine layouts and for the design purpose of similar geomechanical domains at the mine.

In addition, Instrumentation, installed to monitor a local safety concern, is valuable. Such instruments however do not necessarily assist in gaining any overall understanding of ground response to increase extraction in given stoping block or on a mine wide basis.

The following instrumentation and monitoring program is recommended for the Wolverine project:

- Visual observation and monitoring,
- Tape extensometer,
- Point load test
- Pull test

While it sounds simple and intuitive, visual observation is often the best and most comprehensive monitor of changes in ground conditions in a mine. The key to establishing a good system is the reliable recording of the observations made. A binder should be kept in the engineering office with daily inspection sheets filed chronologically. The engineers, supervisors, and surveyors should document any observations made by recording and where possible by photographing the specific location and visual observation made. This may include loading of rockbolts plates, loading or bagging of mesh, fresh loose on the ground, fresh water inflows, or the formation of cracks in ground or shotcrete.

A suitable instrument for monitoring ground movement or convergence of a drift is a tape extensometer. Tape extensometer is a simple, portable instrument and used to accurately measure changes in distance between two points. The changes in distance between these two points over period of time can be monitored with accuracy, reliability and repeatability. Normally, five permanent anchor points will be installed around the perimeter of a drift profile and well marked. The initial readings of point to point will be taken and recorded. These should then be measured on a regular basis to mark any change in the initial readings, monitoring walls and back convergences. Because tape extensometers are very inexpensive, they can be generously placed throughout the mine and monitored as frequently as required to ensure that the ground response is understood. They will also last the life of the operation.

The pull test is the method which is commonly used to determine the effectiveness of ground support element. Bolts could be tested on a regular basis at any time after installation by applying a load to the collar and increasing it until the bolt slips. YZC owns a pull test equipment which was purchased during the test mining program. The pull test should be carried out randomly on various bolts during the mine development at Wolverine.

11.0 CONCLUSIONS AND RECOMMENDATIONS

A geomechanical mine design assessment for the bankable feasibility study of the Wolverine deposit has been carried out. Based on the results of the evaluation, a number of conclusions have been reached. These are as follows:

- A total number of 57 drill holes were used to assess the rock mass quality of the Wolverine deposit. In order to characterize and compare the rock mass quality across the deposit, each orebody was divided into the upper, center and lower zones;
- The rock mass quality ratings highlight the distinct separation between ore and immediate hangingwall and footwall zones. The ore rock quality can be described as "good to extremely good". With the exception of the "extremely poor" hangingwall rock quality in the center zone of the Wolverine orebody, the rock mass quality in the immediate hangingwall and footwall is rated as "very poor to poor";

- According to the point load strength tests, examination of drill hole cores and review of core logs, the ore zone has a higher uniaxial compressive strength and is much more competent than the immediate hangingwall and footwall zones;
- The ground support guidelines for the back and walls are specified based upon the geomechanical domains / rock types. The main ground support systems in the ore and hangingwall are resin rebar / mesh and split set / fibre reinforced shotcrete respectively. In the very poor ground conditions, provision should be made for steel set arches and timber support. The main type of ground support on the walls is split set;
- Fibre reinforced shotcrete will be a major component of the ground support system in the poor rock mass quality. Since the performance of shotcrete depends on its quality, the application of high quality fibre reinforced shotcrete should only be considered;
- The stability of pillars was evaluated based upon a number of assumptions and empirical methods. The numerical modelling should be employed for pillar and stope stability assessment during the mining operation. The modelling will allow pillar stresses, and in particular where multiple openings are present, to be more accurately evaluated;
- Three different variants of the drift and fill mining methods: Drift and Fill Mining with Side Slash, Drift and Fill with Retreat Panels, Drift and Fill with Primary and Secondary Panels, have been proposed for the Wolverine Deposit. All three are geomechanically viable;
- All mine workings should be backfilled as tightly as possible to improve stability within the stope and reduce stress build-up in mine pillars. The backfill will be combination of paste fill and waste fill;
- Unplanned dilution will occur. As mining progresses, from the primary to secondary panels, dilution can be expected to increase. The major source of dilution includes undercutting of the weak geologic structures in the hangingwall and ends as panels are mined out;
- Knowledge of orebody geometry and major structures ahead of the advancing stope / panels will be an important aspect of the ground support specification, mine planning and dilution control at Wolverine;
- As the mine is developed, a program of instrumentation and systematic monitoring should be instigated. In addition, a record of stoping performance should be

maintained. This information will be used to adjust the various geomechanical mine design parameters in order to improve prediction; and,

- The various recommendations contained in this report are based upon an interpretation of geotechnical data from drill core and from geomechanical assessment of the decline. As more information becomes available, the interpretation must be updated. If significant changes in interpretation occur, the results of the analysis should be reviewed.

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VISUAL RECORDS OF INSPECTED LOCATIONS IN THE DECLINE

Photograph 1: Location 1
View of highly foliated graphitic argillite at the portal

Photograph 2: Location 1
View of portal and square steel set at the portal

Photograph 3: Location 1
View of graphitic Argillite and massive argillite at the face.

Photograph 4: Location 1
View of Arch steel set support at the portal - Note very poor rock quality of the FW

Photograph 5: Location 2
View of the Argillite and EXCP contact on the wall

Photograph 6: Location 2
View of the back in EXCP. Note screen and split sets ground support

Photograph 7: Location 2
View of the back in EXCP. Note screen, split sets and a resin rebar

Photograph 8: Location 3
View of the Argillite wall cut by deformed quartz veins

Photograph 9: Location 3
View of the wall in EXCP

Photograph 10: Location 3
View of the wall in EXCP

Photograph 11: Location 4
View of the Shotcreted back in Argillite

Photograph 12: Location 4
View of the timber set support in the argillitic back

Photograph 13: Location 4
View of the wall in Argillite

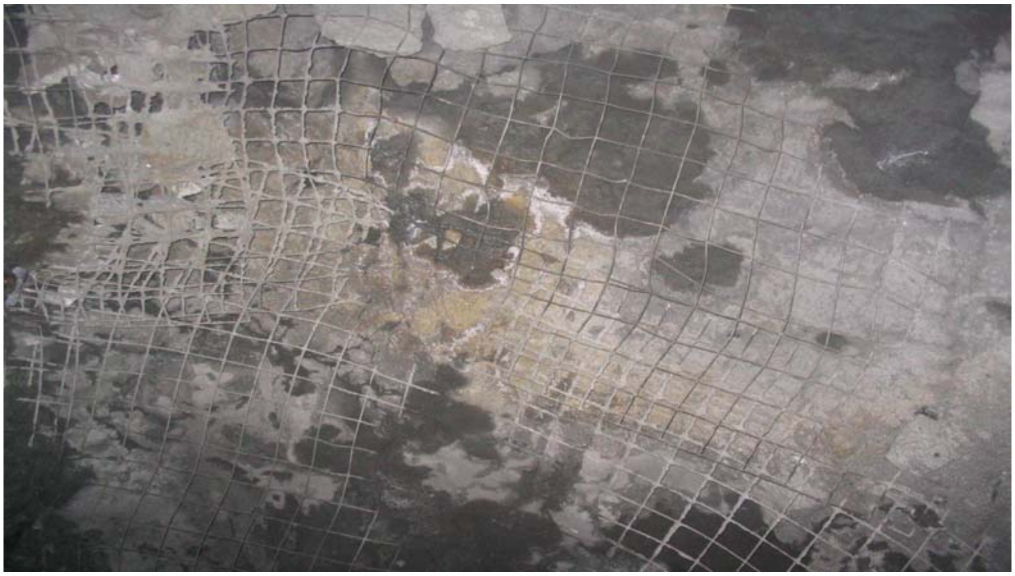
Photograph 14: Location 5
View of chlorite altered Argillitic FW

Photograph 15: Location 5
View of Shotcreted wall and back

Photograph 16: Location 5
View of the back

Photograph 17: Location 6
View of the wall in the massive sulphide and internal argillitic bands.

Photograph 18: Location 6
View of the upper part of wall in massive sulphide and argillite contact and argillite in the back



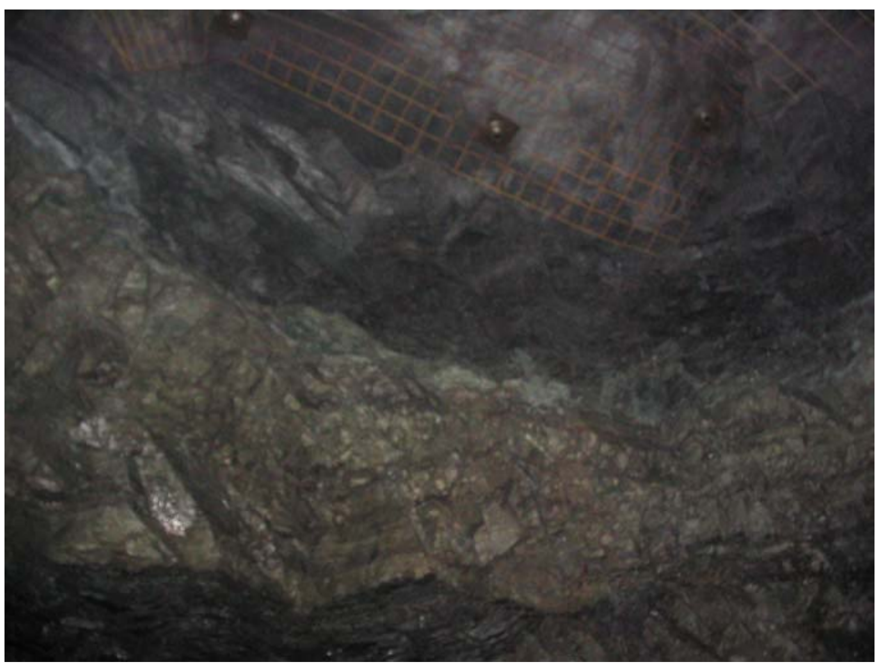
Photograph 19: Location 6
View of the Shotcreted back with split set and screen ground support

Photograph 20: Location 6
View of the Shotcreted back with screen and split set ground support

Photograph 21: Location 6
View of the right wall in the massive sulphide with internal argillite bands in the bottom part and argillite in the upper part.

Photograph 22: Location 7
View of the wall - Note the contact between massive ore zone and an argillite band in between.

Photograph 23: Location 7
View of the exposed HW in Argillite.

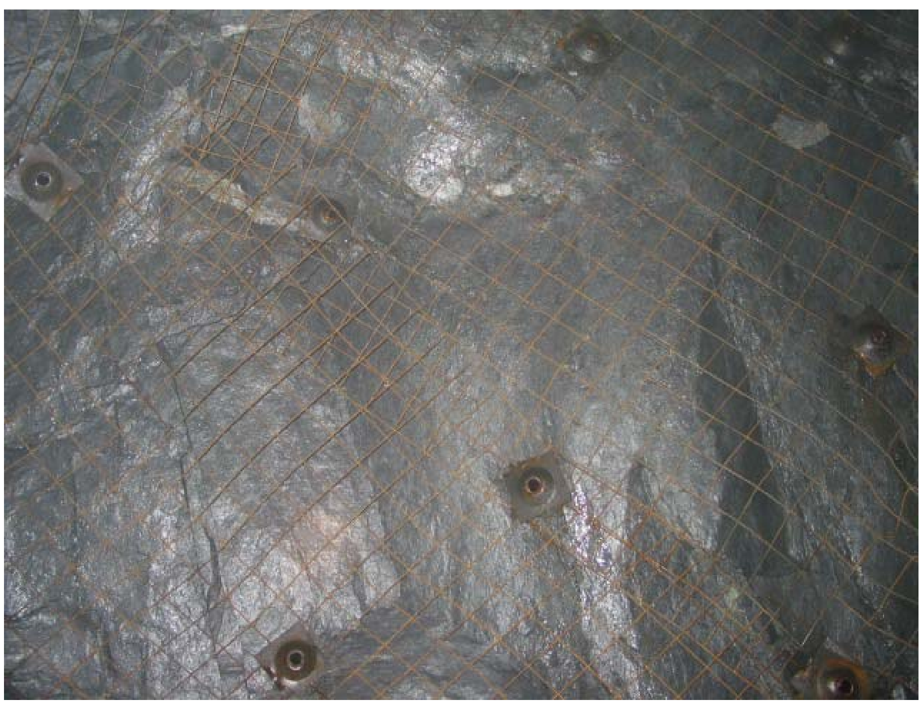


Photograph 24: Location 7
View of the face and exposed argillitic HW in the back. Note the split set and screen ground support in the back. (Note good Stability of HW between two contacts)

Photograph 25: Location 7
View of the wall in the ore zone and argillitic HW.

Photograph 26: Location 8
View of the wall in the ore zone

Photograph 27: Location 8
View of the wall in the ore zone



Photograph 28: Location 8
View of the back in the ore zone – Note the split sets and screen ground support

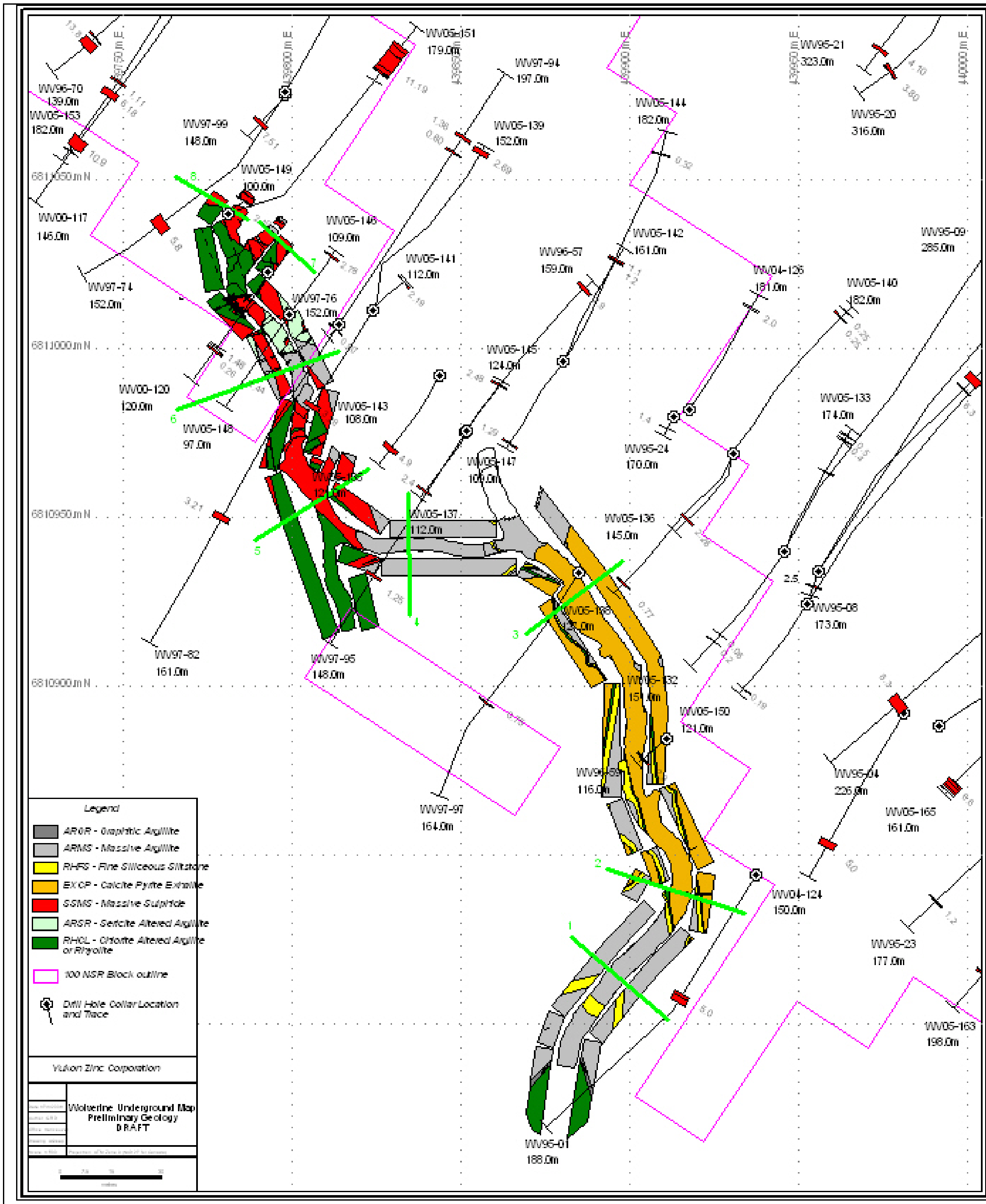
Photograph 29: Location 8
View of the back in the ore zone – Note the split sets and screen ground support

Photograph 30
View of the pillar between the herringbone panels

Photograph 31
View of the pillar between the herringbone panels

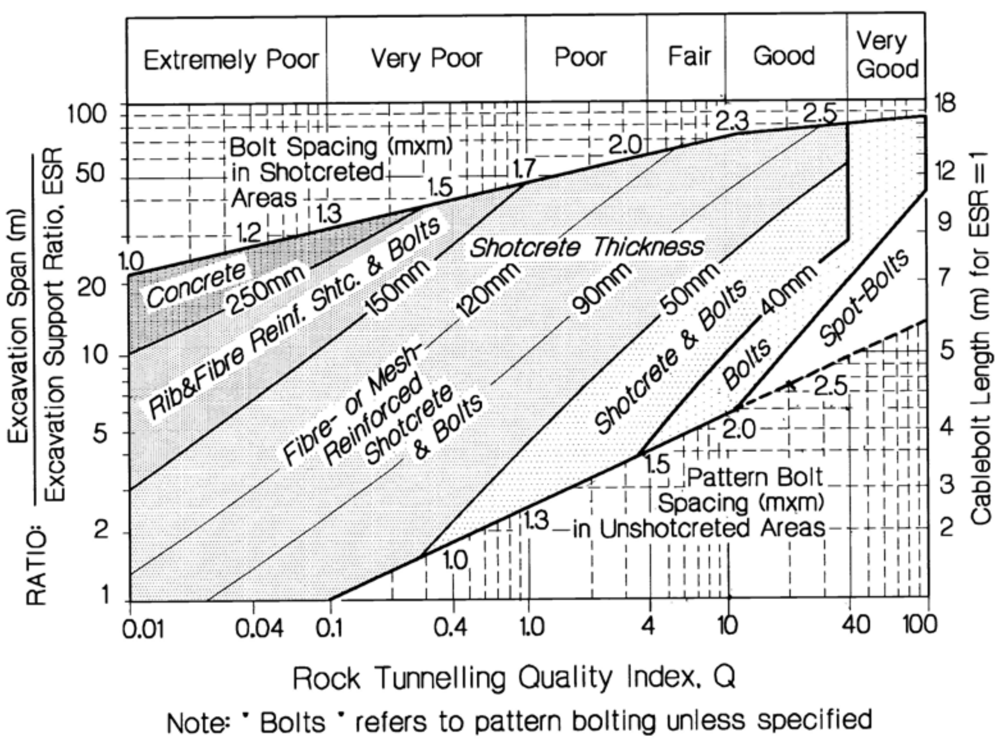
Photograph 32
View of the back at the intersection of herringbone panels and decline - Note ground support consisting of split sets, screen, and shotcrete

Photograph 33
View of the back at the intersection of the decline and herringbone panels
Note ground support consisting of split sets, screen, and shotcrete.



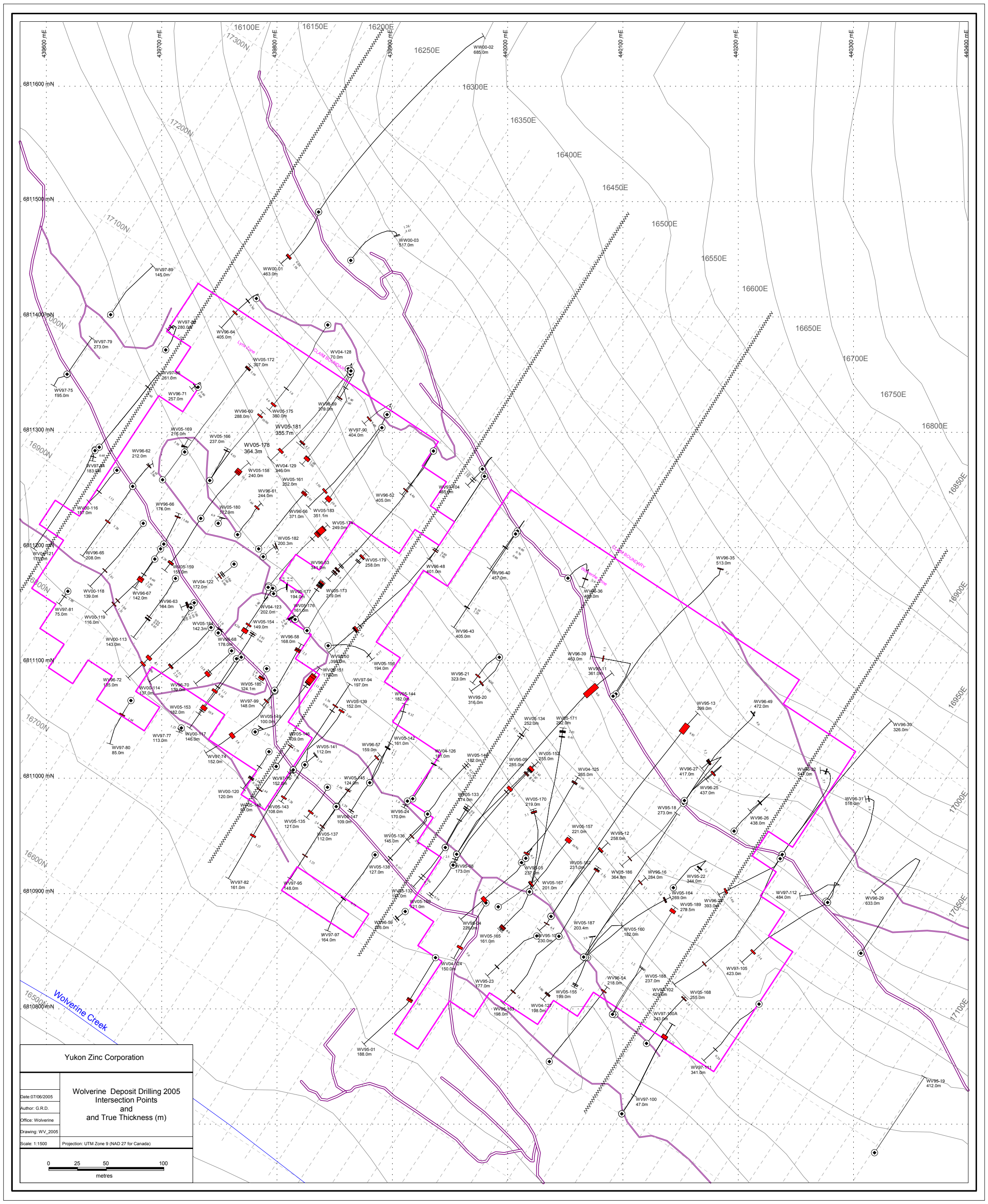
Lithology of the Back and Walls
Observations Locations along the Decline and Test Mining Areas

Figure 1

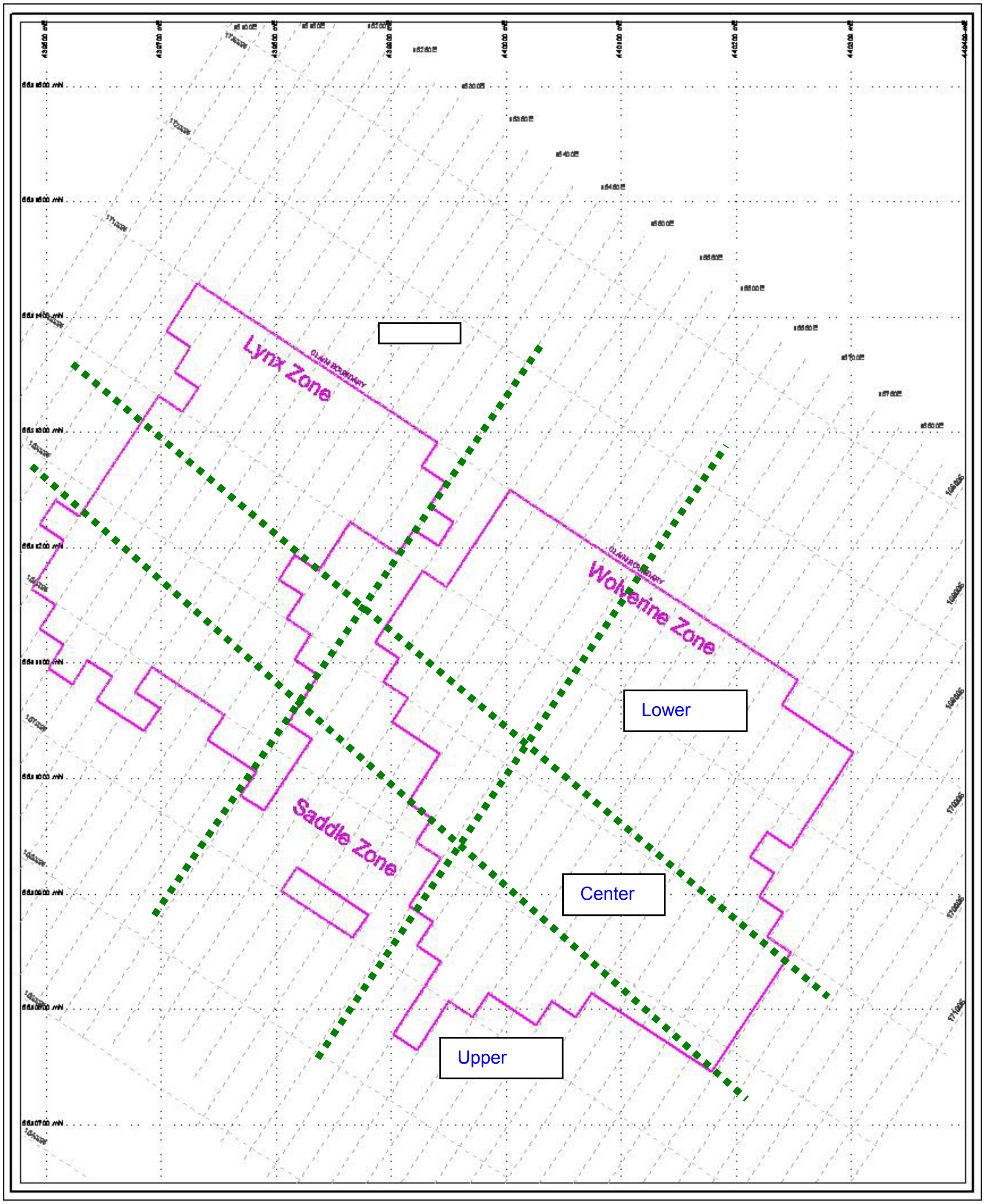


Grimstad et al., 1993

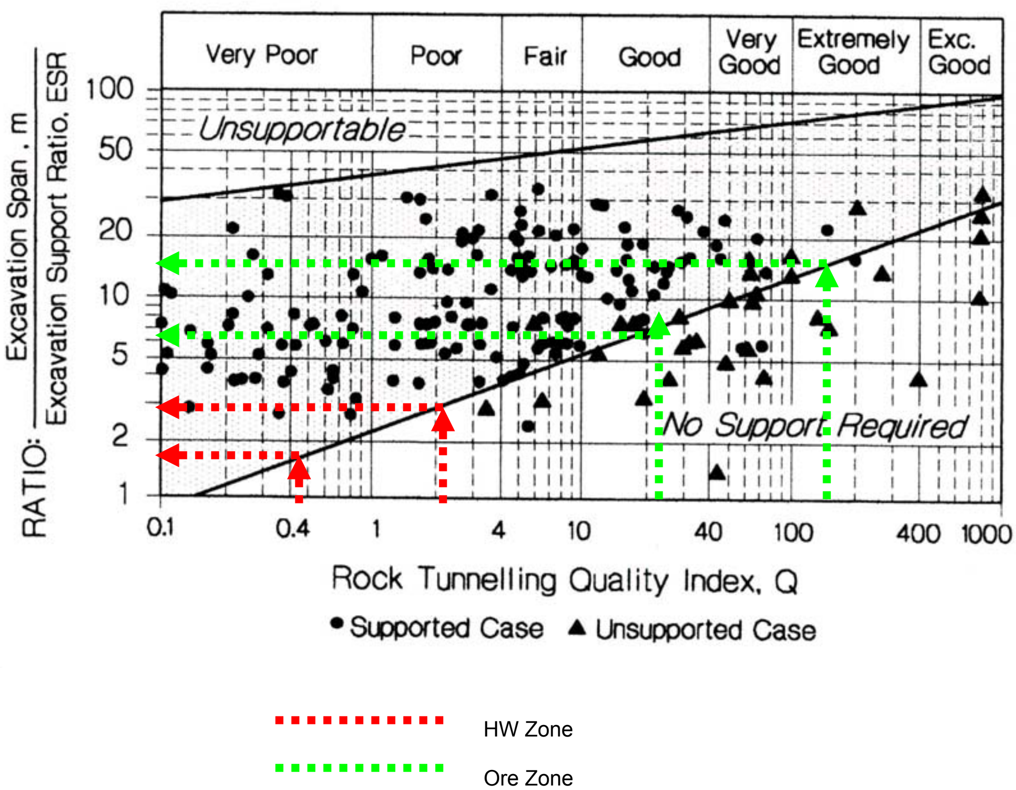
Tunnelling Support Guidelines | **Figure 2**



Drill Holes Distribution in the Wolverine Deposit | **Figure 3**



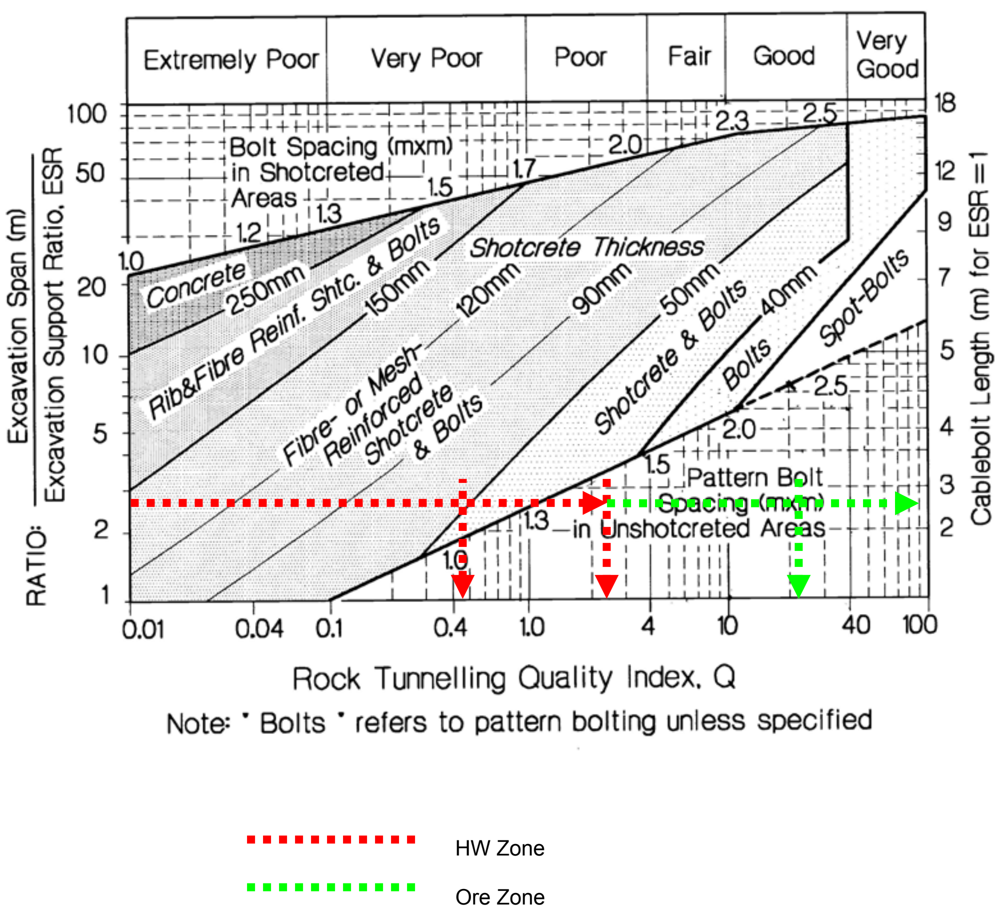
Divisions Used for the Rock Mass Characterisation of the Wolverine, Saddle and Lynx Orebodies	Figure 4



(After Barton, 1988)

Unsupported Span Based on the Case History Database for Q-System

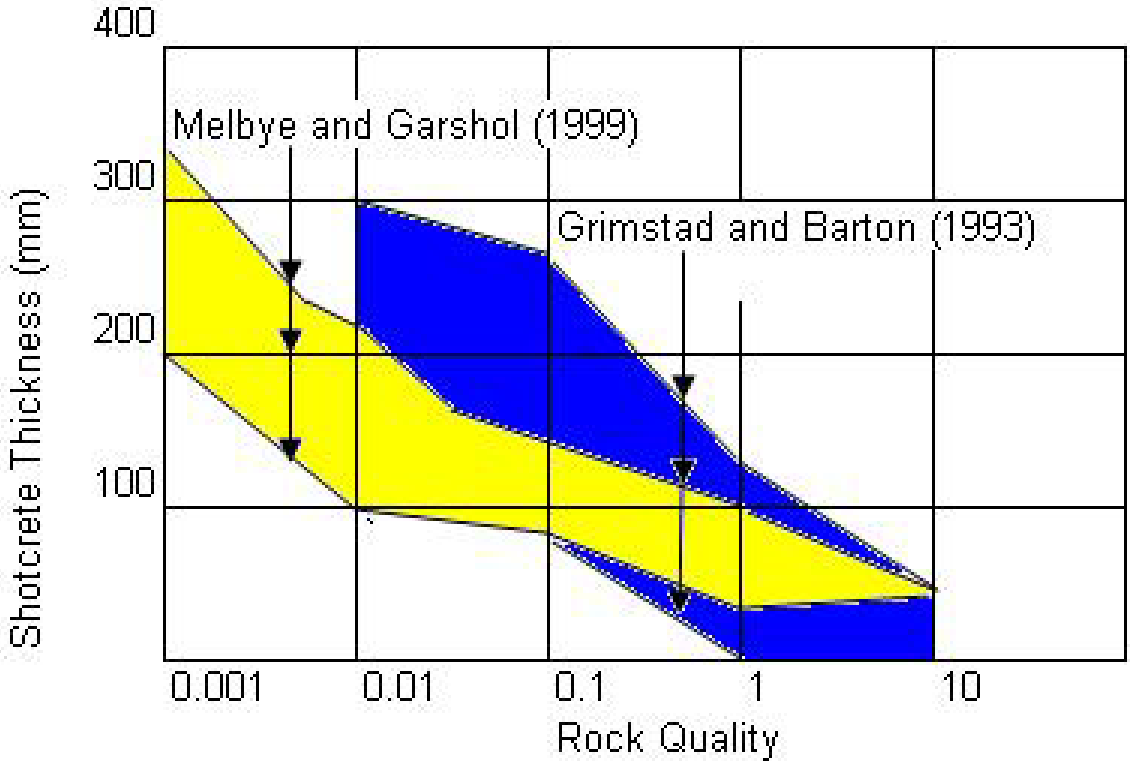
Figure 5



Grimstad et al., 1993

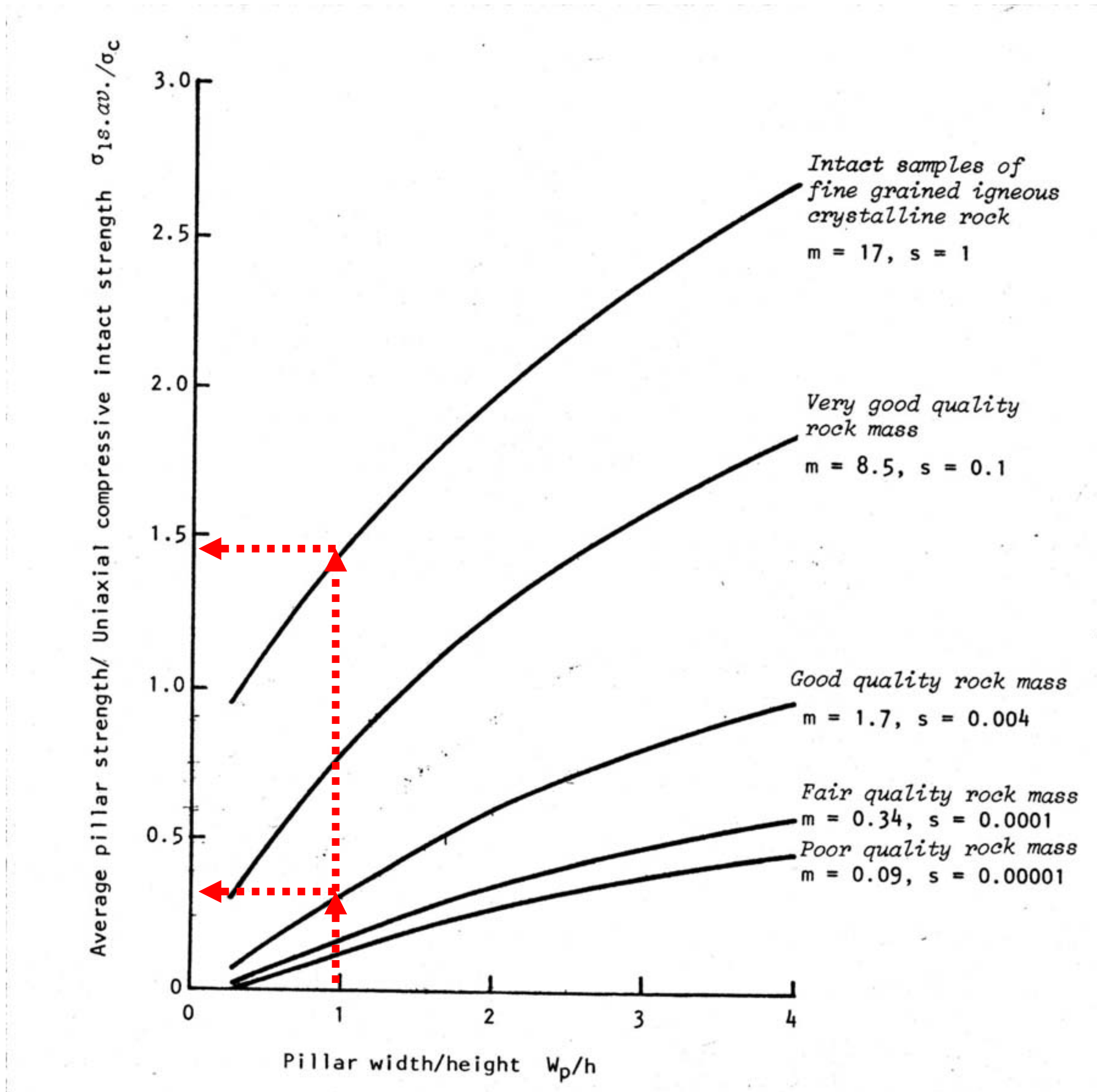
Ground Support System for the HW and Ore Zones based on the Q Rock Mass Classification System

Figure 6



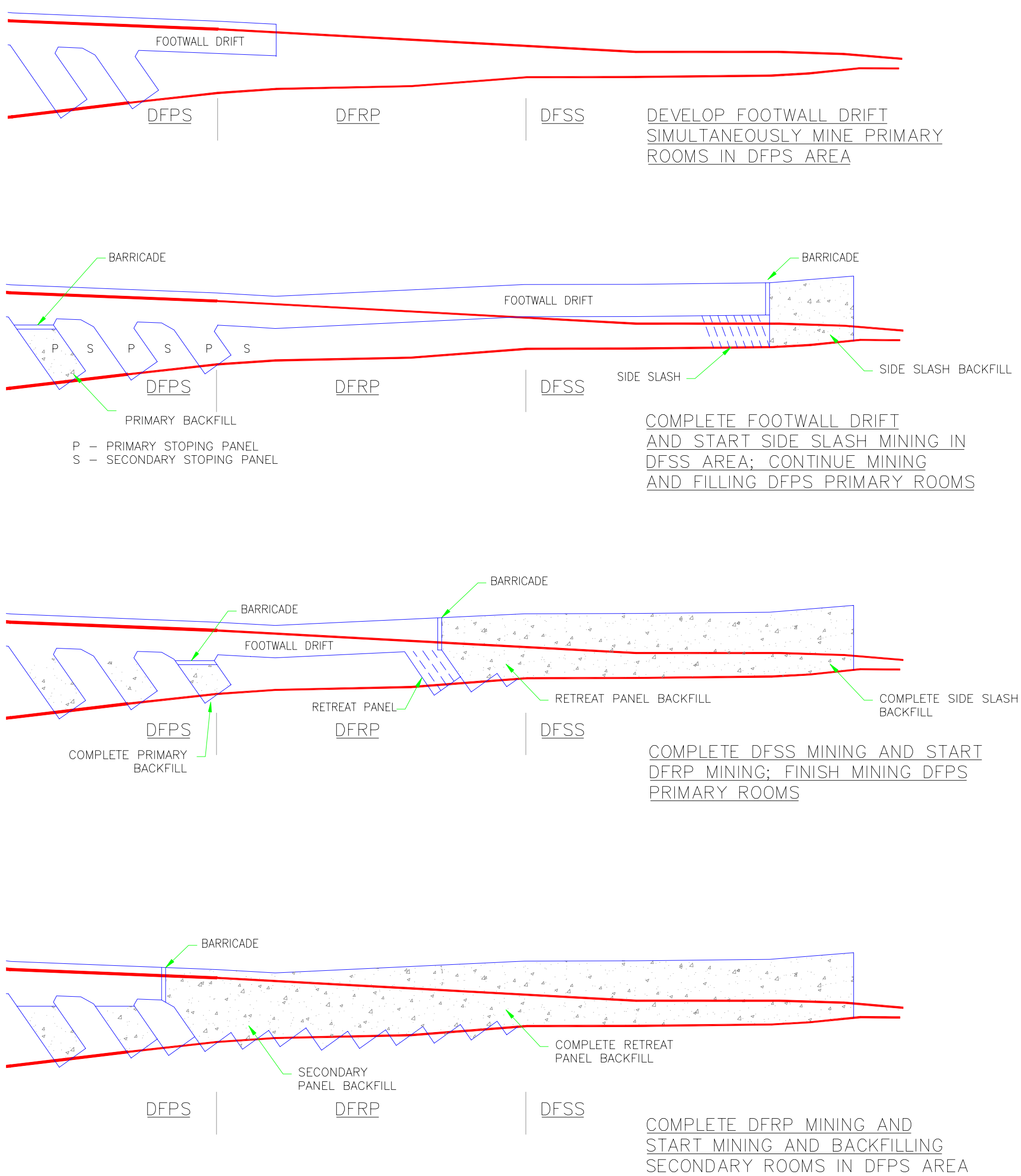
Recent Trends in High Performance Wet-Mix Sprayed Concrete Thickness (Melbye and Garshol 199) compared with Recommendation based on Grimstad and Barton (1993)

Figure 7



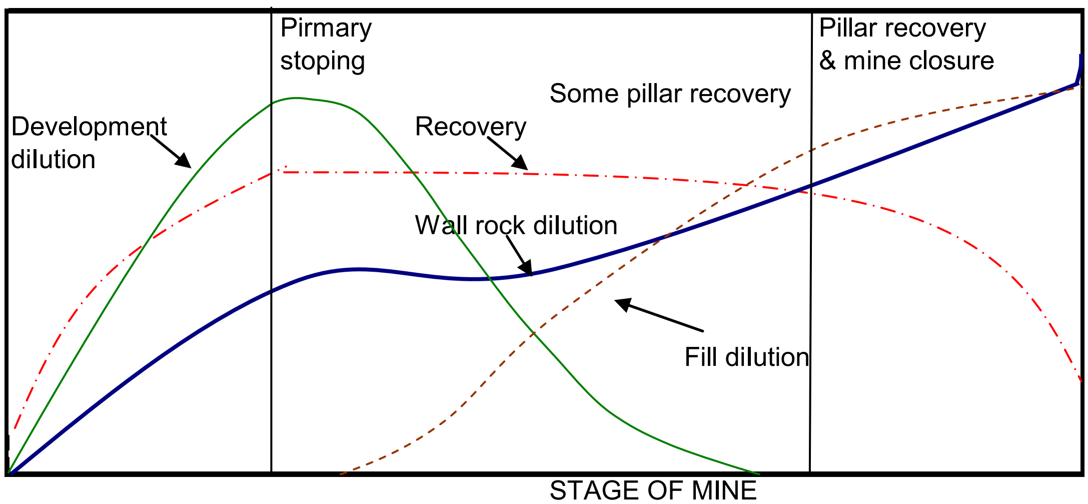
Influence of Pillar Width to Height Ratio on Pillar Strength

Figure 8



Typical Level Stope Sequence

Figure 9



Dilution Type vs. Stage of Mining

Figure 10

APPENDIX I

POINT LOAD STRENGTH TESTS

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 25/2005	WV05-176	147.2	SSMS	dT	OFL	#####	50				#####	1.00	#####	
2	Sept 25/2005	WV05-176	147.2	SSMS	d//	13980	13.52	50				5.41	1.00	5.41	
3	Sept 25/2005	WV05-176	147.4	SSMS	d	10140	9.81	50				3.92	1.00	3.92	
4	Sept 25/2005	WV05-176	147.4	SSMS	d	10520	10.17	50				4.07	1.00	4.07	
5	Sept 25/2005	WV05-176	147.5	SSMS	d	37780	36.53	50				14.61	1.00	14.61	
6	Sept 25/2005	WV05-176	147.6	SSMS	d	18640	18.02	50				7.21	1.00	7.21	
7	Sept 25/2005	WV05-176	147.6	SSMS	d	20160	19.49	50				7.80	1.00	7.80	
8															
9															
10															
11															
12															
13															
14															
15															
16															
17															
18															
19															
20															

(1) 10 tests per rock unit. If rock anisotropic, test in direction which gives the greatest and least strength values

(2) d = diameteral a = axial b = bloc l = Irregular Lump Test // = parallel to plane of weakness T = Perpendicular to plane of weakness

(3) De-dia= core diameter in the diametral test

(4) D-axial= Distance between platen contact point in the axial test

(5)W-axial=core diameter in axial test

Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 25/2005	WV05-176	149.6	PMMS	d	16740	16.19	50				6.48	1.00	6.48	
2	Sept 25/2005	WV05-176	149.6	PMMS	d	19980	19.32	50				7.73	1.00	7.73	
3	Sept 25/2005	WV05-176	149.7	PMMS	d	13660	13.21	50				5.28	1.00	5.28	
4	Sept 25/2005	WV05-176	149.8	PMMS	d	14600	14.12	50				5.65	1.00	5.65	
5	Sept 25/2005	WV05-176	149.8	PMMS	d	13880	13.42	50				5.37	1.00	5.37	
6	Sept 25/2005	WV05-176	150.2	PMMS	d	15800	15.28	50				6.11	1.00	6.11	
7	Sept 25/2005	WV05-176	150.2	PMMS	d	12520	12.11	50				4.84	1.00	4.84	
8	Sept 25/2005	WV05-176	150.7	PMMS	d	12020	11.62	50				4.65	1.00	4.65	
9	Sept 25/2005	WV05-176	150.6	PMMS	d	17000	16.44	50				6.58	1.00	6.58	
10	Sept 25/2005	WV05-176	151.4	PMMS	d	11140	10.77	50				4.31	1.00	4.31	
11															
12															
13															
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19															
20															

(1) 10 tests per rock unit. If rock anisotropic, test in direction w hich gives the greatest and least strength values

(2) d = diameteral a = axial b = bloc l = Irregular Lump Test // = parallel to plane of w eakness T = Perpendicular to plane of w eakness

(3) De-dia= core diameter in the diametral test

(4) D-axial= Distance betw een platen contact point in the axial test

(5)W-axial=core diameter in axial test

Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation									
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1	Oct 6/2005	WV05-177	151.9	EXSP	dT	1360	1.32	50				0.53	1.00	0.53	
2	Oct 6/2005	WV05-177	152.0	EXSP	d//	4080	3.95	50				1.58	1.00	1.58	
3	Oct 6/2005	WV05-177	152.4	EXSP	d//	1220	1.18	50				0.47	1.00	0.47	
4	Oct 6/2005	WV05-177	153.7	EXSP	dT	8400	8.12	50				3.25	1.00	3.25	
5	Oct 6/2005	WV05-177	153.7	EXSP	d//	5100	4.93	50				1.97	1.00	1.97	
6	Oct 6/2005	WV05-177	154.8	EXSP	dT	4940	4.78	50				1.91	1.00	1.91	
7	Oct 6/2005	WV05-177	154.8	EXSP	d//	4820	4.66	50				1.86	1.00	1.86	
8															
9	Oct 6/2005	WV05-177	161.7	EXSP	d//	4520	4.37	50				1.75	1.00	1.75	
10	Oct 6/2005	WV05-177	161.8	EXSP	dT	4220	4.08	50				1.63	1.00	1.63	
11	Oct 6/2005	WV05-177	161.8	EXSP	d//	4160	4.02	50				1.61	1.00	1.61	
12	Oct 6/2005	WV05-177	161.9	EXSP	dT	4700	4.54	50				1.82	1.00	1.82	
13															
14	Oct 6/2005	WV05-177	173.5	EXSP	dT	11520	11.14	50				4.46	1.00	4.46	
15	Oct 6/2005	WV05-177	173.5	EXSP	d//	15880	15.36	50				6.14	1.00	6.14	
16	Oct 6/2005	WV05-177	173.5	EXSP	dT	13280	12.84	50				5.14	1.00	5.14	
17	Oct 6/2005	WV05-177	173.6	EXSP	d//	16880	16.32	50				6.53	1.00	6.53	
18															
19															
20															

(1) 10 tests per rock unit. If rock anisotropic, test in direction which gives the greatest and least strength values

(2) d = diameteral a = axial b = bloc l = Irregular Lump Test // = parallel to plane of weakness T = Perpendicular to plane of weakness

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(5)W-axial=core diameter in axial test

Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1	Oct 12/2005	WV05-180	153.1	SSMS	d	16600	16.05	50				6.42	1.00	6.42	
2	Oct 12/2005	WV05-180	153.1	SSMS	d	24820	24.00	50				9.60	1.00	9.60	
3	Oct 12/2005	WV05-180	153.8	SSMS	d	20000	19.34	50				7.74	1.00	7.74	
4	Oct 12/2005	WV05-180	153.8	SSMS	d	24900	24.08	50				9.63	1.00	9.63	
5	Oct 12/2005	WV05-180	154.7	SSMS	d	13020	12.59	50				5.04	1.00	5.04	
6	Oct 12/2005	WV05-180	154.7	SSMS	d	8400	8.12	50				3.25	1.00	3.25	
7	Oct 12/2005	WV05-180	155.4	SSMS	d	24180	23.38	50				9.35	1.00	9.35	
8	Oct 12/2005	WV05-180	155.5	SSMS	d	19820	19.17	50				7.67	1.00	7.67	
9	Oct 12/2005	WV05-180	156.0	SSMS	d	8320	8.05	50				3.22	1.00	3.22	
10	Oct 12/2005	WV05-180	156.2	SSMS	d	13420	12.98	50				5.19	1.00	5.19	
11	Oct 12/2005	WV05-180	156.4	SSMS	d	15300	14.80	50				5.92	1.00	5.92	
12	Oct 12/2005	WV05-180	157.5	SSMS	d	27680	26.77	50				10.71	1.00	10.71	
13	Oct 12/2005	WV05-180	157.8	SSMS	d	13900	13.44	50				5.38	1.00	5.38	
14	Oct 12/2005	WV05-180	158.2	SSMS	d	11700	11.31	50				4.53	1.00	4.53	
15	Oct 12/2005	WV05-180	158.4	SSMS	d	20000	19.34	50				7.74	1.00	7.74	
16	Oct 12/2005	WV05-180	159.4	SSMS	d	25680	24.83	50				9.93	1.00	9.93	
17	Oct 12/2005	WV05-180	159.4	SSMS	d	19800	19.15	50				7.66	1.00	7.66	
18	Oct 12/2005	WV05-180	160.0	SSMS	d	25480	24.64	50				9.86	1.00	9.86	
19															
20															

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Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1	Oct 12/2005	WV05-180	161.2	SSMS	d	12920	12.49	50				5.00	1.00	5.00	
2	Oct 12/2005	WV05-180	161.3	SSMS	d	18800	18.18	50				7.27	1.00	7.27	
3	Oct 12/2005	WV05-180	161.7	SSMS	d	23960	23.17	50				9.27	1.00	9.27	
4	Oct 12/2005	WV05-180	161.8	SSMS	d	21280	20.58	50				8.23	1.00	8.23	
5	Oct 12/2005	WV05-180	162.0	SSMS	d	15700	15.18	50				6.07	1.00	6.07	
6	Oct 12/2005	WV05-180	162.2	SSMS	d	18660	18.04	50				7.22	1.00	7.22	
7	Oct 12/2005	WV05-180	163.0	SSMS	d	7160	6.92	50				2.77	1.00	2.77	

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	155.8	ARSI	dT	4380	4.24	50				1.69	1.00	1.69	
2	Oct 6/2005	WV05-177	155.8	ARSI	d//	3760	3.64	50				1.45	1.00	1.45	
3	Oct 6/2005	WV05-177	156.4	ARSI	dT	9500	9.19	50				3.67	1.00	3.67	
4	Oct 6/2005	WV05-177	156.4	ARSI	d//	5860	5.67	50				2.27	1.00	2.27	
5	Oct 6/2005	WV05-177	156.7	ARSI	dT	1840	1.78	50				0.71	1.00	0.71	
6	Oct 6/2005	WV05-177	156.7	ARSI	d//	8060	7.79	50				3.12	1.00	3.12	
7	Oct 6/2005	WV05-177	156.8	ARSI	dT	11920	11.53	50				4.61	1.00	4.61	
8	Oct 6/2005	WV05-177	156.8	ARSI	d//	5700	5.51	50				2.20	1.00	2.20	
9	Oct 6/2005	WV05-177	157.1	ARSI	dT	7640	7.39	50				2.96	1.00	2.96	
10	Oct 6/2005	WV05-177	157.1	ARSI	d//	5220	5.05	50				2.02	1.00	2.02	
11	Oct 6/2005	WV05-177	159.0	ARSI	dT	7940	7.68	50				3.07	1.00	3.07	
12	Oct 6/2005	WV05-177	160.1	ARSI	d//	3740	3.62	50				1.45	1.00	1.45	
13	Oct 6/2005	WV05-177	160.8	ARSI	dT	1940	1.88	50				0.75	1.00	0.75	
14	Oct 6/2005	WV05-177	160.8	ARSI	d//	2820	2.73	50				1.09	1.00	1.09	
15	Oct 6/2005	WV05-177	160.9	ARSI	dT	3700	3.58	50				1.43	1.00	1.43	
16	Oct 6/2005	WV05-177	160.9	ARSI	d//	5280	5.11	50				2.04	1.00	2.04	
17	Oct 6/2005	WV05-177	161.0	ARSI	dT	6260	6.05	50				2.42	1.00	2.42	
18	Oct 6/2005	WV05-177	161.6	ARSI	d//	1680	1.62	50				0.65	1.00	0.65	
19															
20															

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(4) D-axial= Distance between platen contact point in the axial test

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 25/2005	WV05-176	158.4	RHCT	dT	460	0.44	50				0.18	1.00	0.18	Invalid Break
2	Sept 25/2005	WV05-176	159.1	RHCT	dT	1420	1.37	50				0.55	1.00	0.55	
3	Sept 25/2005	WV05-176	159.1	RHCT	dT	2420	2.34	50				0.94	1.00	0.94	
4	Sept 25/2005	WV05-176	159.1	RHCT	d//	1780	1.72	50				0.69	1.00	0.69	
5	Sept 25/2005	WV05-176	159.2	RHCT	d//	1260	1.22	50				0.49	1.00	0.49	
6	Sept 25/2005	WV05-176	159.2	RHCT	d//	340	0.33	50				0.13	1.00	0.13	
7	Sept 25/2005	WV05-176	159.5	RHCT	dT	2660	2.57	50				1.03	1.00	1.03	
8	Sept 25/2005	WV05-176	159.5	RHCT	d//	1620	1.57	50				0.63	1.00	0.63	
9	Sept 25/2005	WV05-176	159.9	RHCT	dT	5960	5.76	50				2.31	1.00	2.31	
10	Sept 25/2005	WV05-176	159.9	RHCT	d//	4620	4.47	50				1.79	1.00	1.79	
11	Sept 25/2005	WV05-176	160.1	RHCT	dT	7040	6.81	50				2.72	1.00	2.72	
12	Sept 25/2005	WV05-176	160.1	RHCT	d//	8040	7.77	50				3.11	1.00	3.11	
13	Sept 25/2005	WV05-176	160.3	RHCT	dT	3860	3.73	50				1.49	1.00	1.49	
14	Sept 25/2005	WV05-176	160.3	RHCT	d//	3800	3.67	50				1.47	1.00	1.47	
15	Sept 25/2005	WV05-176	160.5	RHCT	dT	6400	6.19	50				2.48	1.00	2.48	
16	Sept 25/2005	WV05-176	160.5	RHCT	d//	9280	8.97	50				3.59	1.00	3.59	
17	Sept 25/2005	WV05-176	160.7	RHCT	dT	8180	7.91	50				3.16	1.00	3.16	
18	Sept 25/2005	WV05-176	161.5	RHCT	d//	1320	1.28	50				0.51	1.00	0.51	
19	Sept 25/2005	WV05-176	161.7	RHCT	dT	3940	3.81	50				1.52	1.00	1.52	
20	Sept 25/2005	WV05-176	161.9	RHCT	d//	800	0.77	50				0.31	1.00	0.31	

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Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	162.0	ARSI	dT	1360	1.32	50				0.53	1.00	0.53	
2	Oct 6/2005	WV05-177	162.0	ARSI	d//	2500	2.42	50				0.97	1.00	0.97	
3	Oct 6/2005	WV05-177	162.1	ARSI	d//	2320	2.24	50				0.90	1.00	0.90	
4															
5	Oct 6/2005	WV05-177	164.6	ARSI	dT	1580	1.53	50				0.61	1.00	0.61	
6	Oct 6/2005	WV05-177	164.6	ARSI	d//	2120	2.05	50				0.82	1.00	0.82	
7	Oct 6/2005	WV05-177	164.9	ARSI	dT	4500	4.35	50				1.74	1.00	1.74	
8	Oct 6/2005	WV05-177	164.9	ARSI	d//	5820	5.63	50				2.25	1.00	2.25	
9	Oct 6/2005	WV05-177	166.0	ARSI	dT	4480	4.33	50				1.73	1.00	1.73	
10	Oct 6/2005	WV05-177	166.0	ARSI	d//	7040	6.81	50				2.72	1.00	2.72	
11	Oct 6/2005	WV05-177	166.7	ARSI	dT	12060	11.66	50				4.66	1.00	4.66	
12	Oct 6/2005	WV05-177	166.7	ARSI	d//	6720	6.50	50				2.60	1.00	2.60	
13	Oct 6/2005	WV05-177	167.0	ARSI	dT	5160	4.99	50				2.00	1.00	2.00	
14	Oct 6/2005	WV05-177	168.0	ARSI	d//	3000	2.90	50				1.16	1.00	1.16	
15															
16	Oct 6/2005	WV05-177	171.4	ARSI	dT	4400	4.25	50				1.70	1.00	1.70	
17	Oct 6/2005	WV05-177	172.0	ARSI	d//	3180	3.08	50				1.23	1.00	1.23	
18															
19															
20															

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Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	162.7	QTVN	d	10080	9.75	50				3.90	1.00	3.90	
2	Oct 6/2005	WV05-177	162.9	QTVN	d	6840	6.61	50				2.65	1.00	2.65	
3	Oct 6/2005	WV05-177	163.1	QTVN	d	8380	8.10	50				3.24	1.00	3.24	
4	Oct 6/2005	WV05-177	163.2	QTVN	d	5180	5.01	50				2.00	1.00	2.00	
5	Oct 6/2005	WV05-177	163.3	QTVN	d	13960	13.50	50				5.40	1.00	5.40	
6	Oct 6/2005	WV05-177	163.3	QTVN	d	6660	6.44	50				2.58	1.00	2.58	
7	Oct 6/2005	WV05-177	163.3	QTVN	d	16920	16.36	50				6.54	1.00	6.54	
8	Oct 6/2005	WV05-177	163.6	QTVN	d	10940	10.58	50				4.23	1.00	4.23	
9	Oct 6/2005	WV05-177	164.2	QTVN	d	4280	4.14	50				1.66	1.00	1.66	
10															
11	Oct 6/2005	WV05-177	177.2	QTVN	d	4380	4.24	50				1.69	1.00	1.69	
12	Oct 6/2005	WV05-177	177.3	QTVN	d	6080	5.88	50				2.35	1.00	2.35	
13	Oct 6/2005	WV05-177	177.3	QTVN	d	5080	4.91	50				1.96	1.00	1.96	
14															
15															
16															
17															
18															
19															
20															

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Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	168.5	PYMS	d	8080	7.81	50				3.13	1.00	3.13	
2	Oct 6/2005	WV05-177	168.5	PYMS	d	12400	11.99	50				4.80	1.00	4.80	
3	Oct 6/2005	WV05-177	168.7	PYMS	d	14400	13.92	50				5.57	1.00	5.57	
4	Oct 6/2005	WV05-177	168.7	PYMS	d	10480	10.13	50				4.05	1.00	4.05	
5	Oct 6/2005	WV05-177	168.9	PYMS	dT	10720	10.37	50				4.15	1.00	4.15	
6	Oct 6/2005	WV05-177	168.9	PYMS	d//	12640	12.22	50				4.89	1.00	4.89	
7	Oct 6/2005	WV05-177	169.3	PYMS	dT	8240	7.97	50				3.19	1.00	3.19	
8	Oct 6/2005	WV05-177	169.4	PYMS	d//	3260	3.15	50				1.26	1.00	1.26	
9	Oct 6/2005	WV05-177	169.4	PYMS	dT	8320	8.05	50				3.22	1.00	3.22	
10	Oct 6/2005	WV05-177	170.0	PYMS	dT	14060	13.60	50				5.44	1.00	5.44	
11	Oct 6/2005	WV05-177	170.1	PYMS	d//	4980	4.82	50				1.93	1.00	1.93	
12															
13	Oct 6/2005	WV05-177	172.2	PYMS	d	6860	6.63	50				2.65	1.00	2.65	
14	Oct 6/2005	WV05-177	172.8	PYMS	d//	2300	2.22	50				0.89	1.00	0.89	
15	Oct 6/2005	WV05-177	172.8	PYMS	dT	15940	15.41	50				6.17	1.00	6.17	
16	Oct 6/2005	WV05-177	172.8	PYMS	d//	12940	12.51	50				5.01	1.00	5.01	
17	Oct 6/2005	WV05-177	173.0	PYMS	dT	8820	8.53	50				3.41	1.00	3.41	
18	Oct 6/2005	WV05-177	173.0	PYMS	d//	7820	7.56	50				3.02	1.00	3.02	
19															
20															

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(5)W-axial=core diameter in axial test

Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	176.2	SSMS	d	3580	3.46	50				1.38	1.00	1.38	
2	Oct 6/2005	WV05-177	176.4	SSMS	d	4200	4.06	50				1.62	1.00	1.62	
3	Oct 6/2005	WV05-177	177.0	SSMS	d	5040	4.87	50				1.95	1.00	1.95	
4															
5	Oct 6/2005	WV05-177	180.8	SSMS	d	25320	24.48	50				9.79	1.00	9.79	
6	Oct 6/2005	WV05-177	181.0	SSMS	d	9880	9.55	50				3.82	1.00	3.82	
7	Oct 6/2005	WV05-177	181.0	SSMS	d	9440	9.13	50				3.65	1.00	3.65	
8	Oct 6/2005	WV05-177	181.1	SSMS	d	20000	19.34	50				7.74	1.00	7.74	
9	Oct 6/2005	WV05-177	181.1	SSMS	d	21480	20.77	50				8.31	1.00	8.31	
10	Oct 6/2005	WV05-177	181.6	SSMS	d	2280	2.20	50				0.88	1.00	0.88	porous
11	Oct 6/2005	WV05-177	181.7	SSMS	d	5560	5.38	50				2.15	1.00	2.15	porous
12	Oct 6/2005	WV05-177	182.1	SSMS	d	2240	2.17	50				0.87	1.00	0.87	
13	Oct 6/2005	WV05-177	182.5	SSMS	d	4220	4.08	50				1.63	1.00	1.63	
14	Oct 6/2005	WV05-177	182.8	SSMS	d	3840	3.71	50				1.49	1.00	1.49	
15	Oct 6/2005	WV05-177	183.6	SSMS	d	2580	2.49	50				1.00	1.00	1.00	
16	Oct 6/2005	WV05-177	184.2	SSMS	d	2120	2.05	50				0.82	1.00	0.82	
17	Oct 6/2005	WV05-177	184.8	SSMS	d	1320	1.28	50				0.51	1.00	0.51	
18	Oct 6/2005	WV05-177	185.3	SSMS	d	6100	5.90	50				2.36	1.00	2.36	
19	Oct 6/2005	WV05-177	185.8	SSMS	d	1940	1.88	50				0.75	1.00	0.75	
20	Oct 6/2005	WV05-177	186.0	SSMS	d	6460	6.25	50				2.50	1.00	2.50	

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Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	186.2	SSMS	d	3360	3.25	50				1.30	1.00	1.30	
2	Oct 6/2005	WV05-177	186.5	SSMS	d	4980	4.82	50				1.93	1.00	1.93	
3	Oct 6/2005	WV05-177	186.9	SSMS	d	5680	5.49	50				2.20	1.00	2.20	
4	Oct 6/2005	WV05-177	187.8	SSMS	d	4560	4.41	50				1.76	1.00	1.76	
5															
6															
7															

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	177.4	PYMS	d	5380	5.20	50				2.08	1.00	2.08	
2	Oct 6/2005	WV05-177	177.5	PYMS	d	5100	4.93	50				1.97	1.00	1.97	
3															
4															
5															
6															
7															
8															
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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Aug 11/2005	WV05-174	209.4	PMSM	d//	880	0.85	36				0.66	0.8	0.56	Brecciated
2	Aug 11/2005	WV05-174	209.4	PMSM	d//	760	0.73	36				0.57	0.8	0.48	Brecciated
3	Aug 11/2005	WV05-174	209.5	PMSM	d//	560	0.54	36				0.42	0.8	0.35	Brecciated
4	Aug 11/2005	WV05-174	209.9	PMSM	d//	2460	2.38	36				1.84	0.8	1.56	
5	Aug 11/2005	WV05-174	210	PMSM	d//	1280	1.24	36				0.96	0.8	0.81	
6	Aug 11/2005	WV05-174	211.5	PMSM	d//	5560	5.38	36				4.15	0.8	3.52	
7	Aug 11/2005	WV05-174	211.5	PMSM	d//	5040	4.87	36				3.76	0.85	3.19	Invalid Break
8	Aug 11/2005	WV05-174	211.6	PMSM	d//	4220	4.08	36				3.15	0.85	2.67	
9	Aug 11/2005	WV05-174	211.6	PMSM	d//	2740	2.65	36				2.04	0.85	1.73	
10	Aug 11/2005	WV05-174	211.7	PMSM	d//	1580	1.53	36				1.18	0.85	1.00	
11	Aug 11/2005	WV05-174	211.7	PMSM	d//	1740	1.68	36				1.30	0.85	1.10	
12	Aug 11/2005	WV05-174	211.8	PMSM	d//	1040	1.01	36				0.78	0.85	0.66	
13															
14															
15															
16															
17															
18															
19															
20															

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Aug 19/2005	WV05-174	214.2	SSMS	d	4040	3.91	36				3.01	0.8	2.56	
2	Aug 19/2005	WV05-174	214.5	SSMS	d	2460	2.38	36				1.84	0.8	1.56	
3	Aug 19/2005	WV05-174	214.5	SSMS	d	2980	2.88	36				2.22	0.8	1.89	
4	Aug 19/2005	WV05-174	215.5	SSMS	d	8080	7.81	36				6.03	0.8	5.12	
5	Aug 19/2005	WV05-174	215.5	SSMS	d	6500	6.29	36				4.85	0.8	4.12	
6	Aug 19/2005	WV05-174	215.6	SSMS	d	6900	6.67	36				5.15	0.8	4.37	
7	Aug 19/2005	WV05-174	215.6	SSMS	dT	8880	8.59	36				6.63	0.85	5.62	
8	Aug 19/2005	WV05-174	215.7	SSMS	d	7480	7.23	36				5.58	0.85	4.74	
9	Aug 19/2005	WV05-174	215.7	SSMS	d	8680	8.39	36				6.48	0.85	5.50	
10	Aug 19/2005	WV05-174	215.7	SSMS	d	6040	5.84	36				4.51	0.85	3.82	
11	Aug 19/2005	WV05-174	215.9	SSMS	d	6140	5.94	36				4.58	0.85	3.89	
12	Aug 19/2005	WV05-174	216.0	SSMS	d	9540	9.23	36				7.12	0.85	6.04	
13	Aug 19/2005	WV05-174	216.2	SSMS	d	2040	1.97	36				1.52	0.85	1.29	
14	Aug 19/2005	WV05-174	216.2	SSMS	d//	5020	4.85	36				3.75	0.85	3.18	
15	Aug 19/2005	WV05-174	216.4	SSMS	d	6100	5.90	36				4.55	0.85	3.86	
16	Aug 19/2005	WV05-174	216.4	SSMS	d	8600	8.32	36				6.42	0.85	5.44	
17	Aug 19/2005	WV05-174	216.5	SSMS	d	17880	17.29	36				13.34	0.85	11.32	
18	Aug 19/2005	WV05-174	216.6	SSMS	d	11080	10.71	36				8.27	0.85	7.01	
19	Aug 19/2005	WV05-174	216.6	SSMS	d	14520	14.04	36				10.83	0.85	9.19	
20	Aug 19/2005	WV05-174	216.6	SSMS	d	13220	12.78	36				9.86	0.85	8.37	

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Aug 19/2005	WV05-174	216.8	SSMS	d	7420	7.18	36				5.54	0.8	4.70	
2	Aug 19/2005	WV05-174	216.8	SSMS	d	10420	10.08	36				7.77	0.8	6.60	
3	Aug 19/2005	WV05-174	216.9	SSMS	d	9520	9.21	36				7.10	0.8	6.03	

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Aug 20/2005	WV05-174	233.4	ARMS	d	10540	10.19	36				7.86	0.8	6.67	
2	Aug 20/2005	WV05-174	233.4	ARMS	d	9860	9.53	36				7.36	0.8	6.24	
3	Aug 20/2005	WV05-174	233.4	ARMS	d	10940	10.58	36				8.16	0.8	6.93	
4	Aug 20/2005	WV05-174	233.6	ARMS	d	11220	10.85	36				8.37	0.8	7.10	
5	Aug 20/2005	WV05-174	233.6	ARMS	d	11100	10.73	36				8.28	0.8	7.03	
6	Aug 20/2005	WV05-174	233.6	ARMS	d	9160	8.86	36				6.83	0.8	5.80	
7	Aug 20/2005	WV05-174	233.8	ARMS	d	2780	2.69	36				2.07	0.85	1.76	
8															
9															
10															
11															
12															
13															
14															
15															
16															
17															
18															
19															
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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1															
2	Aug 20/2005	WV05-174	237.5	ARMS	d//	1300	1.26	36				0.97	0.8	0.82	
3	Aug 20/2005	WV05-174	237.6	ARMS	d//	1060	1.03	36				0.79	0.8	0.67	
4	Aug 20/2005	WV05-174	238.7	ARMS	d//	1080	1.04	36				0.81	0.8	0.68	
5	Aug 20/2005	WV05-174	238.7	ARMS	dT	8460	8.18	36				6.31	0.8	5.36	
6	Aug 20/2005	WV05-174	238.9	ARMS	d//	740	0.72	36				0.55	0.8	0.47	
7	Aug 20/2005	WV05-174	239.0	ARMS	d//	1520	1.47	36				1.13	0.85	0.96	
8	Aug 20/2005	WV05-174	239.7	ARMS	d//	2340	2.26	36				1.75	0.85	1.48	
9	Aug 20/2005	WV05-174	239.9	ARMS	dT	6800	6.58	36				5.07	0.85	4.31	
10	Aug 20/2005	WV05-174	240.0	ARMS	dT	5600	5.42	36				4.18	0.85	3.55	
11	Aug 20/2005	WV05-174	240.0	ARMS	dT	3720	3.60	36				2.78	0.85	2.36	
12	Aug 20/2005	WV05-174	240.0	ARMS	d//	360	0.35	36				0.27	0.85	0.23	
13	Aug 20/2005	WV05-174	240.5	ARMS	d//	940	0.91	36				0.70	0.85	0.60	
14	Aug 20/2005	WV05-174	240.6	ARMS	dT	3960	3.83	36				2.95	0.85	2.51	
15	Aug 20/2005	WV05-174	240.6	ARMS	dT	7560	7.31	36				5.64	0.85	4.79	
16	Aug 20/2005	WV05-174	240.9	ARMS	d//	6560	6.34	36				4.89	0.85	4.15	
17	Aug 20/2005	WV05-174	246.1	ARMS	d//	1740	1.68	36				1.30	0.85	1.10	
18	Aug 20/2005	WV05-174	246.1	ARMS	dT	1660	1.61	36				1.24	0.85	1.05	
19	Aug 20/2005	WV05-174	246.8	ARMS	dT	1660	1.61	36				1.24	0.85	1.05	
20	Aug 20/2005	WV05-174	246.8	ARMS	dT	7380	7.14	36				5.51	0.85	4.67	

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Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1															
2	Aug 20/2005	WV05-174	246.9	ARMS	dT	9720	9.40	36				7.25	0.8	6.15	
3	Aug 20/2005	WV05-174	247.0	ARMS	d//	1860	1.80	36				1.39	0.8	1.18	
4	Aug 20/2005	WV05-174	247.0	ARMS	d//	3080	2.98	36				2.30	0.8	1.95	

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Aug 20/2005	WV05-174	239.1	QCVN	d	5720	5.53	36				4.27	0.8	3.62	Brecciated
2	Aug 20/2005	WV05-174	239.1	QCVN	d//	2400	2.32	36				1.79	0.8	1.52	Brecciated
3	Aug 20/2005	WV05-174	239.3	QCVN	d	3360	3.25	36				2.51	0.8	2.13	Brecciated
4	Aug 20/2005	WV05-174	239.3	QCVN	d	5740	5.55	36				4.28	0.8	3.63	
5	Aug 20/2005	WV05-174	239.5	QCVN	d	3040	2.94	36				2.27	0.8	1.92	
6	Aug 20/2005	WV05-174	239.5	QCVN	d	6140	5.94	36				4.58	0.8	3.89	
7	Aug 20/2005	WV05-174	239.5	QCVN	d	1860	1.80	36				1.39	0.85	1.18	Invalid Break
8	Aug 20/2005	WV05-174	239.5	QCVN	d	3680	3.56	36				2.75	0.85	2.33	
9	Aug 20/2005	WV05-174	239.5	QCVN	d	8280	8.01	36				6.18	0.85	5.24	
10															
11															
12															
13															
14															
15															
16															
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18															
19															
20															

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Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1	Sept 28/2005	WV05-175	357.7	QCVN	d	1780	1.72	47				0.78	0.97	0.76	
2	Sept 28/2005	WV05-175	357.7	QCVN	d	5500	5.32	47				2.41	0.97	2.33	
3	Sept 28/2005	WV05-175	357.7	QCVN	d	7620	7.37	47				3.34	0.97	3.23	
4	Sept 28/2005	WV05-175	358.1	QCVN	d	4980	4.82	47				2.18	0.97	2.11	
5	Sept 28/2005	WV05-175	358.5	QCVN	d	7820	7.56	47				3.42	0.97	3.32	
6	Sept 28/2005	WV05-175	359.1	QCVN	d	7800	7.54	47				3.41	0.97	3.31	
7	Sept 28/2005	WV05-175	360.2	QCVN	dT	9360	9.05	47				4.10	0.97	3.97	
8	Sept 28/2005	WV05-175	360.2	QCVN	d//	9520	9.21	47				4.17	0.97	4.04	
9	Sept 28/2005	WV05-175	360.4	QCVN	dT	12340	11.93	47				5.40	0.97	5.24	
10	Sept 28/2005	WV05-175	360.4	QCVN	d//	9100	8.80	47				3.98	0.97	3.86	
11															
12															
13															
14															
15															
16															
17															
18															
19															
20															

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Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 28/2005	WV05-175	343.0	RHMS	d//	2880	2.78	47				1.26	0.97	1.22	
2	Sept 28/2005	WV05-175	343.1	RHMS	dT	5220	5.05	47				2.29	0.97	2.22	
3	Sept 28/2005	WV05-175	343.1	RHMS	d//	4660	4.51	47				2.04	0.97	1.98	
4	Sept 28/2005	WV05-175	343.1	RHMS	dT	5280	5.11	47				2.31	0.97	2.24	
5	Sept 28/2005	WV05-175	343.2	RHMS	d//	17040	16.48	47				7.46	0.97	7.23	
6	Sept 28/2005	WV05-175	343.4	RHMS	dT	7720	7.47	47				3.38	0.97	3.28	
7	Sept 28/2005	WV05-175	343.8	RHMS	d//	3860	3.73	47				1.69	0.97	1.64	
8	Sept 28/2005	WV05-175	344.2	RHMS	dT	8080	7.81	47				3.54	0.97	3.43	
9	Sept 28/2005	WV05-175	344.2	RHMS	d//	2740	2.65	47				1.20	0.97	1.16	
10	Sept 28/2005	WV05-175	344.8	RHMS	dT	9760	9.44	47				4.27	0.97	4.14	
11	Sept 28/2005	WV05-175	344.8	RHMS	d//	12980	12.55	47				5.68	0.97	5.51	
12	Sept 28/2005	WV05-175	345.8	RHMS	dT	8960	8.66	47				3.92	0.97	3.80	
13	Sept 28/2005	WV05-175	347.6	RHMS	d//	5240	5.07	47				2.29	0.97	2.22	
14	Sept 28/2005	WV05-175	349.2	RHMS	dT	4400	4.25	47				1.93	0.97	1.87	
15	Sept 28/2005	WV05-175	349.5	RHMS	d//	1840	1.78	47				0.81	0.97	0.78	
16	Sept 28/2005	WV05-175	350.4	RHMS	dT	6200	6.00	47				2.71	0.97	2.63	
17	Sept 28/2005	WV05-175	351.4	RHMS	d//	5460	5.28	47				2.39	0.97	2.32	
18	Sept 28/2005	WV05-175	351.6	RHMS	dT	5180	5.01	47				2.27	0.97	2.20	
19	Sept 28/2005	WV05-175	352.0	RHMS	d//	5740	5.55	47				2.51	0.97	2.44	
20	Sept 28/2005	WV05-175	354.5	RHMS	dT	9800	9.48	47				4.29	0.97	4.16	

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(3) De-dia= core diameter in the diametral test

(4) D-axial= Distance betw een platen contact point in the axial test

(5)W-axial=core diameter in axial test

Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 28/2005	WV05-175	355.2	EXCP	dT	4060	3.93	47				1.78	0.97	1.72	
2	Sept 28/2005	WV05-175	355.5	EXCP	d//	10140	9.81	47				4.44	0.97	4.30	
3	Sept 28/2005	WV05-175	355.5	EXCP	dT	7540	7.29	47				3.30	0.97	3.20	
4	Sept 28/2005	WV05-175	355.5	EXCP	d//	11240	10.87	47				4.92	0.97	4.77	
5	Sept 28/2005	WV05-175	355.7	EXCP	dT	12560	12.15	47				5.50	0.97	5.33	
6	Sept 28/2005	WV05-175	355.7	EXCP	d//	12600	12.18	47				5.52	0.97	5.35	
7	Sept 28/2005	WV05-175	355.7	EXCP	dT	3520	3.40	47				1.54	0.97	1.49	
8	Sept 28/2005	WV05-175	356.0	EXCP	d//	3580	3.46	47				1.57	0.97	1.52	
9	Sept 28/2005	WV05-175	356.8	EXCP	dT	3560	3.44	47				1.56	0.97	1.51	
10	Sept 28/2005	WV05-175	356.8	EXCP	d//	4200	4.06	47				1.84	0.97	1.78	
11															
12															
13															
14															
15															
16															
17															
18															
19															
20															

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 28/2005	WV05-175	357.2	ARMS	dT	7540	7.29	47				3.30	0.97	3.20	
2	Sept 28/2005	WV05-175	357.2	ARMS	d//	7620	7.37	47				3.34	0.97	3.23	
3															
4	Sept 29/2005	WV05-175	365.8	ARMS	dT	2120	2.05	47				0.93	0.97	0.90	
5	Sept 29/2005	WV05-175	366.0	ARMS	d//	2160	2.09	47				0.95	0.97	0.92	
6	Sept 29/2005	WV05-175	366.3	ARMS	dT	3520	3.40	47				1.54	0.97	1.49	
7	Sept 29/2005	WV05-175	366.3	ARMS	d//	5060	4.89	47				2.22	0.97	2.15	
8	Sept 29/2005	WV05-175	367.1	ARMS	dT	4260	4.12	47				1.86	0.97	1.81	
9	Sept 29/2005	WV05-175	367.1	ARMS	d//	4380	4.24	47				1.92	0.97	1.86	
10	Sept 29/2005	WV05-175	367.7	ARMS	dT	1040	1.01	47				0.46	0.97	0.44	
11	Sept 29/2005	WV05-175	367.7	ARMS	d//	2380	2.30	47				1.04	0.97	1.01	
12	Sept 29/2005	WV05-175	368.0	ARMS	dT	2000	1.93	47				0.88	0.97	0.85	
13	Sept 29/2005	WV05-175	368.0	ARMS	d//	1960	1.90	47				0.86	0.97	0.83	
14	Sept 29/2005	WV05-175	368.9	ARMS	dT	2000	1.93	47				0.88	0.97	0.85	
15	Sept 29/2005	WV05-175	368.9	ARMS	d//	1940	1.88	47				0.85	0.97	0.82	
16	Sept 29/2005	WV05-175	369.4	ARMS	dT	980	0.95	47				0.43	0.97	0.42	
17	Sept 29/2005	WV05-175	369.7	ARMS	d//	1360	1.32	47				0.60	0.97	0.58	
18															
19															
20															

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 29/2005	WV05-175	360.6	SSMS	d	8840	8.55	47				3.87	0.97	3.75	
2	Sept 29/2005	WV05-175	360.6	SSMS	d	3940	3.81	47				1.72	0.97	1.67	
3	Sept 29/2005	WV05-175	361.3	SSMS	d	11380	11.00	47				4.98	0.97	4.83	
4	Sept 29/2005	WV05-175	361.3	SSMS	d	8520	8.24	47				3.73	0.97	3.62	
5	Sept 29/2005	WV05-175	361.4	SSMS	d	9980	9.65	47				4.37	0.97	4.24	
6	Sept 29/2005	WV05-175	361.5	SSMS	d	6540	6.32	47				2.86	0.97	2.78	
7	Sept 29/2005	WV05-175	362.0	SSMS	d	12600	12.18	47				5.52	0.97	5.35	
8	Sept 29/2005	WV05-175	362.0	SSMS	d	7900	7.64	47				3.46	0.97	3.35	
9	Sept 29/2005	WV05-175	363.1	SSMS	d	19660	19.01	47				8.61	0.97	8.34	
10	Sept 29/2005	WV05-175	363.2	SSMS	d	6880	6.65	47				3.01	0.97	2.92	
11	Sept 29/2005	WV05-175	363.3	SSMS	d	14600	14.12	47				6.39	0.97	6.20	
12	Sept 29/2005	WV05-175	363.7	SSMS	d	6920	6.69	47				3.03	0.97	2.94	
13	Sept 29/2005	WV05-175	363.7	SSMS	d	9600	9.28	47				4.20	0.97	4.07	
14	Sept 29/2005	WV05-175	364.1	SSMS	dT	12940	12.51	47				5.66	0.97	5.49	
15	Sept 29/2005	WV05-175	364.2	SSMS	d//	14460	13.98	47				6.33	0.97	6.14	
16	Sept 29/2005	WV05-175	365.0	SSMS	dT	11580	11.20	47				5.07	0.97	4.91	
17	Sept 29/2005	WV05-175	365.0	SSMS	d//	11460	11.08	47				5.02	0.97	4.86	
18	Sept 29/2005	WV05-175	365.2	SSMS	dT	14420	13.94	47				6.31	0.97	6.12	
19	Sept 29/2005	WV05-175	365.2	SSMS	d//	12080	11.68	47				5.29	0.97	5.13	
20	Sept 29/2005	WV05-175	365.5	SSMS	d//	5140	4.97	47				2.25	0.97	2.18	

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 13/2005	WV05-178	232.3	EXCP	d//	10260	9.92	47				4.49	0.97	4.35	
2	Oct 13/2005	WV05-178	232.3	EXCP	d//	6060	5.86	47				2.65	0.97	2.57	
3	Oct 13/2005	WV05-178	232.3	EXCP	d//	16660	16.11	47				7.29	0.97	7.07	
4	Oct 13/2005	WV05-178	232.3	EXCP	d//	6640	6.42	47				2.91	0.97	2.82	
5	Oct 13/2005	WV05-178	246.1	RHFS	d//	21060	20.37	47				9.22	0.97	8.94	
6	Oct 13/2005	WV05-178	246.3	RHFS	d//	9020	8.72	47				3.95	0.97	3.83	
7	Oct 13/2005	WV05-178	246.3	RHFS	dT	5560	5.38	47				2.43	0.97	2.36	
8	Oct 13/2005	WV05-178	249.1	RHFS	dT	14680	14.20	47				6.43	0.97	6.23	
9	Oct 13/2005	WV05-178	249.1	RHFS	d//	7740	7.48	47				3.39	0.97	3.29	
10	Oct 13/2005	WV05-178	245.9	RHFS	d	6520	6.30	47				2.85	0.97	2.77	
11	Oct 13/2005	WV05-178	245.9	RHFS	d	13920	13.46	47				6.09	0.97	5.91	
12	Oct 13/2005	WV05-178	255.0	RHFS	dT	9920	9.59	47				4.34	0.97	4.21	
13	Oct 13/2005	WV05-178	255.6	RHFS	d	10500	10.15	47				4.60	0.97	4.46	
14	Oct 13/2005	WV05-178	266.4	ARSI	dT	5860	5.67	47				2.57	0.97	2.49	
15	Oct 13/2005	WV05-178	266.9	ARSI	dT	2120	2.05	47				0.93	0.97	0.90	
16	Oct 13/2005	WV05-178	266.9	ARSI	d//	3980	3.85	47				1.74	0.97	1.69	
17	Oct 13/2005	WV05-178	267.0	ARSI	dT	12260	11.86	47				5.37	0.97	5.20	
18	Oct 13/2005	WV05-178	267.0	ARSI	d//	8640	8.35	47				3.78	0.97	3.67	
19	Oct 13/2005	WV05-178	283.8	ARSI	dT	1180	1.14	47				0.52	0.97	0.50	
20	Oct 13/2005	WV05-178	284.5	ARSI	d//	5980	5.78	47				2.62	0.97	2.54	

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 13/2005	WV05-178	285.0	ARSI	d//	860	0.83	47				0.38	0.97	0.37	
2	Oct 13/2005	WV05-178	286.0	ARSI	dT	1420	1.37	47				0.62	0.97	0.60	
3	Oct 13/2005	WV05-178	286.2	ARSI	d//	920	0.89	47				0.40	0.97	0.39	
4	Oct 13/2005	WV05-178	286.8	ARSI	dT	2000	1.93	47				0.88	0.97	0.85	
5	Oct 14/2005	WV05-178	290.0	ARCB	dT	4200	4.06	47				1.84	0.97	1.78	
6	Oct 14/2005	WV05-178	290.0	ARCB	d//	980	0.95	47				0.43	0.97	0.42	
7	Oct 14/2005	WV05-178	293.2	EXSP	dT	6240	6.03	47				2.73	0.97	2.65	

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Nov 8/2005	WV05-186	293.3	ARSI	dT	8700	8.41	35				6.87	0.84	5.75	
2	Nov 8/2005	WV05-186	294.9	EXCP	dT	2020	1.95	35				1.59	0.84	1.33	
3	Nov 8/2005	WV05-186	295.3	EXCP	d//	2660	2.57	35				2.10	0.84	1.76	
4	Nov 8/2005	WV05-186	296.3	QTVN	dT	3440	3.33	35				2.72	0.84	2.27	
5	Nov 8/2005	WV05-186	296.7	QTVN	d//	5660	5.47	35				4.47	0.84	3.74	
6	Nov 8/2005	WV05-186	297.8	ARSI	dT	4160	4.02	35				3.28	0.84	2.75	
7	Nov 8/2005	WV05-186	298.6	ARSI	d//	1220	1.18	35				0.96	0.84	0.81	
8	Nov 8/2005	WV05-186	298.7	ARSI	dT	2140	2.07	35				1.69	0.84	1.41	
9	Nov 8/2005	WV05-186	300.1	ARSI	d//	2460	2.38	35				1.94	0.84	1.62	
10	Nov 8/2005	WV05-186	300.8	EXCP	dT	1400	1.35	35				1.11	0.84	0.92	
11	Nov 8/2005	WV05-186	301.5	EXCP	dT	5060	4.89	35				3.99	0.84	3.34	
12	Nov 8/2005	WV05-186	302.5	EXCP	dT	3000	2.90	35				2.37	0.84	1.98	
13	Nov 8/2005	WV05-186	304.3	EXCP	dT	2160	2.09	35				1.71	0.84	1.43	
14	Nov 8/2005	WV05-186	305.7	EXCP	d//	1020	0.99	35				0.81	0.84	0.67	
15	Nov 8/2005	WV05-186	306.1	EXCP	dT	1660	1.61	35				1.31	0.84	1.10	
16	Nov 8/2005	WV05-186	306.3	EXCP	d//	860	0.83	35				0.68	0.84	0.57	
17	Nov 8/2005	WV05-186	307.4	EXCP	dT	920	0.89	35				0.73	0.84	0.61	
18	Nov 8/2005	WV05-186	307.6	EXCP	d//	700	0.68	35				0.55	0.84	0.46	
19	Nov 8/2005	WV05-186	332.3	PMSM	dT	22600	21.85	35				17.84	0.84	14.93	
20	Nov 8/2005	WV05-186	335.4	RHCL	dT	1640	1.59	35				1.29	0.84	1.08	

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Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Nov 8/2005	WV05-186	334.9	RHCL	d//	8160	7.89	35				6.44	0.84	5.39	
2	Nov 8/2005	WV05-186	335.1	RHCL	dT	2800	2.71	35				2.21	0.84	1.85	
3	Nov 8/2005	WV05-186	335.3	RHCL	d//	1180	1.14	35				0.93	0.84	0.78	
4	Nov 8/2005	WV05-186	335.4	RHCL	d//	1180	1.14	35				0.93	0.84	0.78	
5	Nov 8/2005	WV05-186	335.5	RHCL	dT	2500	2.42	35				1.97	0.84	1.65	
6	Nov 8/2005	WV05-186	335.6	RHCL	d//	1180	1.14	35				0.93	0.84	0.78	
7	Nov 8/2005	WV05-186	335.8	RHCL	dT	2860	2.77	35				2.26	0.84	1.89	

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 31/2005	WV05-187	133.4	EXCP	dT	10560	10.21	47				4.62	0.97	4.48	
2	Oct 31/2005	WV05-187	133.8	EXCP	d//	5920	5.72	47				2.59	0.97	2.51	
3	Oct 31/2005	WV05-187	133.9	EXCP	dT	11240	10.87	47				4.92	0.97	4.77	
4	Oct 31/2005	WV05-187	134.1	EXCP	d//	6040	5.84	47				2.64	0.97	2.56	
5	Oct 31/2005	WV05-187	135.1	EXCP	d//	10720	10.37	47				4.69	0.97	4.55	
6	Oct 31/2005	WV05-187	135.4	EXCP	dT	3060	2.96	47				1.34	0.97	1.30	
7	Oct 31/2005	WV05-187	136.5	EXCP	dT	14060	13.60	47				6.15	0.97	5.97	
8	Oct 31/2005	WV05-187	136.5	EXCP	d//	6020	5.82	47				2.64	0.97	2.56	
9	Oct 31/2005	WV05-187	136.7	EXCP	dT	11840	11.45	47				5.18	0.97	5.03	
10	Oct 31/2005	WV05-187	136.7	EXCP	d//	6020	5.82	47				2.64	0.97	2.56	
11	Oct 31/2005	WV05-187	138.0	EXCP	dT	8720	8.43	47				3.82	0.97	3.70	
12	Oct 31/2005	WV05-187	138.3	EXCP	d//	4800	4.64	47				2.10	0.97	2.04	
13	Oct 31/2005	WV05-187	138.3	EXCP	dT	4460	4.31	47				1.95	0.97	1.89	
14	Oct 31/2005	WV05-187	139.4	EXCP	d//	13220	12.78	47				5.79	0.97	5.61	
15	Oct 31/2005	WV05-187	147.6	EXCP	d//	2620	2.53	47				1.15	0.97	1.11	
16	Oct 31/2005	WV05-187	147.6	EXCP	dT	4440	4.29	47				1.94	0.97	1.88	
17	Oct 31/2005	WV05-187	148.3	EXCP	d//	2080	2.01	47				0.91	0.97	0.88	
18	Oct 31/2005	WV05-187	148.8	EXCP	dT	8380	8.10	47				3.67	0.97	3.56	
19	Oct 31/2005	WV05-187	148.9	EXCP	dT	12500	12.09	47				5.47	0.97	5.31	
20	Oct 31/2005	WV05-187	148.9	EXCP	d//	3340	3.23	47				1.46	0.97	1.42	

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Mean Is(50) T =   Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 31/2005	WV05-187	149.6	EXCP	dT	10840	10.48	47				4.75	0.97	4.60	
2	Oct 31/2005	WV05-187	150.0	EXCP	d//	9700	9.38	47				4.25	0.97	4.12	
3	Oct 31/2005	WV05-187	150.4	EXCP	dT	11500	11.12	47				5.03	0.97	4.88	
4	Oct 31/2005	WV05-187	151.0	EXCP	d//	8460	8.18	47				3.70	0.97	3.59	
5	Oct 31/2005	WV05-187	151.4	EXCP	dT	10120	9.79	47				4.43	0.97	4.30	
6	Oct 31/2005	WV05-187	151.7	EXCP	d//	10660	10.31	47				4.67	0.97	4.52	
7	Oct 31/2005	WV05-187	151.9	EXCP	dT	10520	10.17	47				4.61	0.97	4.46	

Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1	09-Aug-05	WV05-174	191	ARCB	d//	1960	1.90	50	n/a	n/a	n/a	0.76	1	0.76	
2	09-Aug-05	WV05-174	191	ARCB	d//	1820	1.76	50	n/a	n/a	n/a	0.70	1	0.70	
3	09-Aug-05	WV05-174	191.2	ARCB	a	4220	4.08	50	25	50	39.90	2.56	0.9	2.29	
4	09-Aug-05	WV05-174	192.5	ARCB	d//	3400	3.29	50	n/a	n/a	n/a	1.32	1	1.32	
5	09-Aug-05	WV05-174	192.5	ARCB	d//	3540	3.42	50	n/a	n/a	n/a	1.37	1	1.37	
6	09-Aug-05	WV05-174	192.5	ARCB	d//	7680	7.43	50	n/a	n/a	n/a	2.97	1	2.97	
7	09-Aug-05	WV05-174	205	PYSM	d//	5680	5.49	36	n/a	n/a	n/a	4.24	0.85	3.60	
8	09-Aug-05	WV05-174	205	PYSM	d//	15940	15.41	36	n/a	n/a	n/a	11.89	0.85	10.09	
9	09-Aug-05	WV05-174	205	PYSM	d//	5740	5.55	36	n/a	n/a	n/a	4.28	0.85	3.63	
10	09-Aug-05	WV05-174													
11	09-Aug-05	WV05-174													

(1) 10 tests per rock unit. If rock anisotropic, test in direction which gives the greatest and least strength values

(2) d = diameteral   a = axial   b = bloc   I = Irregular Lump Test   // = parallel to plane of weakness   T = Perpendicular to plane of weakness

(3) De-dia= core diameter in the diametral test

(4) D-axial= Distance between platen contact point in the axial test

(5)W-axial=core diameter in axial test

Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1	Oct 12/2005	WV05-180	106.6	ARSI	dT	5260	5.09	50				2.03	1.00	2.03	
2	Oct 12/2005	WV05-180	106.6	ARSI	d//	5040	4.87	50				1.95	1.00	1.95	
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(5)W-axial=core diameter in axial test

Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 12/2005	WV05-180	107.5	QCVN	d	6120	5.92	50				2.37	1.00	2.37	
2	Oct 12/2005	WV05-180	108.0	QCVN	d	3660	3.54	50				1.42	1.00	1.42	
3															
4	Oct 12/2005	WV05-180	115.3	QCVN	d	4800	4.64	50				1.86	1.00	1.86	
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Mean Is(50) T = Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 24/2005	WV05-176	115.2	EXSP	d//	3800	3.67	50				1.47	1.00	1.47	
2	Sept 24/2005	WV05-176	115.8	EXSP	d//	6860	6.63	50				2.65	1.00	2.65	
3	Sept 24/2005	WV05-176	115.9	EXSP	dT	10400	10.06	50				4.02	1.00	4.02	
4	Sept 24/2005	WV05-176	115.9	EXSP	d//	10940	10.58	50				4.23	1.00	4.23	
5	Sept 24/2005	WV05-176	116.0	EXSP	dT	5380	5.20	50				2.08	1.00	2.08	
6	Sept 24/2005	WV05-176	116.0	EXSP	dT	6320	6.11	50				2.44	1.00	2.44	
7	Sept 24/2005	WV05-176	116.1	EXSP	dT	8500	8.22	50				3.29	1.00	3.29	
8	Sept 24/2005	WV05-176	116.1	EXSP	d//	4500	4.35	50				1.74	1.00	1.74	
9	Sept 24/2005	WV05-176	116.1	EXSP	d//	3840	3.71	50				1.49	1.00	1.49	
10	Sept 24/2005	WV05-176	116.3	EXSP	dT	6280	6.07	50				2.43	1.00	2.43	
11	Sept 24/2005	WV05-176	116.3	EXSP	dT	5740	5.55	50				2.22	1.00	2.22	
12	Sept 24/2005	WV05-176	116.3	EXSP	d//	5000	4.84	50				1.93	1.00	1.93	
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(5)W-axial=core diameter in axial test

Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation									
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 12/2005	WV05-180	135.6	ARMS	d	3660	3.54	50				1.42	1.00	1.42	brecciated
2	Oct 12/2005	WV05-180	141.4	ARMS	d//	5100	4.93	50				1.97	1.00	1.97	
3	Oct 12/2005	WV05-180	142.5	ARMS	dT	7460	7.21	50				2.89	1.00	2.89	
4	Oct 12/2005	WV05-180	143.0	ARMS	d//	3700	3.58	50				1.43	1.00	1.43	
5	Oct 12/2005	WV05-180	143.1	ARMS	dT	5960	5.76	50				2.31	1.00	2.31	
6	Oct 12/2005	WV05-180	143.2	ARMS	d//	4960	4.80	50				1.92	1.00	1.92	
7	Oct 12/2005	WV05-180	143.6	ARMS	dT	9340	9.03	50				3.61	1.00	3.61	
8															
9	Oct 12/2005	WV05-180	168.9	ARMS	dT	1780	1.72	50				0.69	1.00	0.69	
10	Oct 12/2005	WV05-180	169.0	ARMS	d//	900	0.87	50				0.35	1.00	0.35	
11	Oct 12/2005	WV05-180	169.4	ARMS	dT	1780	1.72	50				0.69	1.00	0.69	
12	Oct 12/2005	WV05-180	169.6	ARMS	d//	2660	2.57	50				1.03	1.00	1.03	
13	Oct 12/2005	WV05-180	169.8	ARMS	dT	720	0.70	50				0.28	1.00	0.28	
14	Oct 12/2005	WV05-180	170.0	ARMS	d//	2380	2.30	50				0.92	1.00	0.92	
15	Oct 12/2005	WV05-180	170.6	ARMS	dT	7260	7.02	50				2.81	1.00	2.81	
16	Oct 12/2005	WV05-180	170.9	ARMS	d//	2420	2.34	50				0.94	1.00	0.94	
17	Oct 12/2005	WV05-180	171.4	ARMS	dT	3180	3.08	50				1.23	1.00	1.23	
18	Oct 12/2005	WV05-180	171.5	ARMS	d//	1560	1.51	50				0.60	1.00	0.60	
19	Oct 12/2005	WV05-180	171.6	ARMS	dT	3240	3.13	50				1.25	1.00	1.25	
20	Oct 12/2005	WV05-180	171.8	ARMS	d//	1440	1.39	50				0.56	1.00	0.56	

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Mean Is(50) T =

Mean Is(50) // =

Point Load Sample Data							Point Load Strength Calculation								
Test(1)	Date	Drill Hole	Depth	Rock Type	Type of	P'	P	De-dia (3)	D- axial(4)	W-axial (5)	De-axial	Is	F	Is(50)	Comment
(#)		(#)	(m)		Test (2)	(KPa)	KN	(mm)	(mm)	(mm)	(mm)	(MPa)		(MPa)	
1	Sept 24/2005	WV05-176	140.8	QCVN	d//	6640	6.42	50				2.57	1.00	2.57	
2	Sept 24/2005	WV05-176	141.0	QCVN	dT	14520	14.04	50				5.62	1.00	5.62	
3	Sept 24/2005	WV05-176	141.0	QCVN	d//	4660	4.51	50				1.80	1.00	1.80	
4	Sept 24/2005	WV05-176	141.0	QCVN	d//	12600	12.18	50				4.87	1.00	4.87	
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Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 24/2005	WV05-176	143.7	ARCL	d//	5580	5.40	50				2.16	1.00	2.16	
2	Sept 24/2005	WV05-176	143.7	ARCL	dT	2540	2.46	50				0.98	1.00	0.98	
3	Sept 24/2005	WV05-176	143.8	ARCL	dT	7340	7.10	50				2.84	1.00	2.84	
4	Sept 24/2005	WV05-176	143.9	ARCL	d//	1400	1.35	50				0.54	1.00	0.54	
5	Sept 24/2005	WV05-176	144.0	ARCL	d//	2680	2.59	50				1.04	1.00	1.04	
6	Sept 24/2005	WV05-176	144.1	ARCL	dT	5220	5.05	50				2.02	1.00	2.02	
7	Sept 24/2005	WV05-176	144.1	ARCL	d//	2660	2.57	50				1.03	1.00	1.03	
8	Sept 24/2005	WV05-176	144.1	ARCL	dT	4580	4.43	50				1.77	1.00	1.77	
9	Sept 24/2005	WV05-176	144.2	ARCL	d//	5700	5.51	50				2.20	1.00	2.20	
10	Sept 24/2005	WV05-176	144.2	ARCL	dT	7220	6.98	50				2.79	1.00	2.79	
11	Sept 24/2005	WV05-176	144.2	ARCL	d//	9460	9.15	50				3.66	1.00	3.66	
12	Sept 24/2005	WV05-176	144.4	ARCL	dT	8620	8.34	50				3.33	1.00	3.33	
13	Sept 24/2005	WV05-176	144.5	ARCL	d//	6140	5.94	50				2.37	1.00	2.37	
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Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Oct 6/2005	WV05-177	144.2	ARMS	dT	5520	5.34	50				2.14	1.00	2.14	
2	Oct 6/2005	WV05-177	144.2	ARMS	d//	3440	3.33	50				1.33	1.00	1.33	
3	Oct 6/2005	WV05-177	145.4	ARMS	d//	1000	0.97	50				0.39	1.00	0.39	
4	Oct 6/2005	WV05-177	145.4	ARMS	dT	1020	0.99	50				0.39	1.00	0.39	
5															
6	Oct 6/2005	WV05-177	174.3	ARMS	d//	10800	10.44	50				4.18	1.00	4.18	
7	Oct 6/2005	WV05-177	174.0	ARMS	a	16780	16.23	50	35	50	47.22	7.28	0.97	7.07	
8	Oct 6/2005	WV05-177	174.7	ARMS	dT	5940	5.74	50				2.30	1.00	2.30	
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Point Load Sample Data							Point Load Strength Calculation								
Test(1) (#)	Date	Drill Hole (#)	Depth (m)	Rock Type	Type of Test (2)	P' (KPa)	P KN	De-dia (3) (mm)	D- axial(4) (mm)	W-axial (5) (mm)	De-axial (mm)	Is (MPa)	F	Is(50) (MPa)	Comment
1	Sept 24/2005	WV05-176	146.0	CPMS	d	19600	18.95	50				7.58	1.00	7.58	
2	Sept 24/2005	WV05-176	146.1	CPMS	d	14040	13.58	50				5.43	1.00	5.43	
3	Sept 24/2005	WV05-176	146.3	CPMS	d	13280	12.84	50				5.14	1.00	5.14	
4															
5	Sept 25/2005	WV05-176	148.5	CPMS	d	7620	7.37	50				2.95	1.00	2.95	
6	Sept 25/2005	WV05-176	148.5	CPMS	d	8840	8.55	50				3.42	1.00	3.42	
7	Sept 25/2005	WV05-176	148.6	CPMS	d	5220	5.05	50				2.02	1.00	2.02	
8	Sept 25/2005	WV05-176	149.2	CPMS	d	16460	15.92	50				6.37	1.00	6.37	
9	Sept 25/2005	WV05-176	149.2	CPMS	d	12080	11.68	50				4.67	1.00	4.67	Invalid Break
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