DEPARTMENT OF MINES AND ENERGY
SOUTH AUSTRALIA

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MOUNT OWEN COAL PROJECT PRELIMINARY GEOTECHNICAL REPORT

GEOLOGICAL SURVEY

by

S WALKER and L J MORRIS

GROUNDWATER & ENGINEERING GEOLOGY BRANCH

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CONTENTS

TABLES	PAGE NO

FIGURES

APPENDICES

APPENDIX FIGURES

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ABSTRACT

Field investigations were carried out to provide preliminary geotechnical and hydrogeological parameters for the design of a new open pit coal mine at Mount Owen in the Hunter Valley of New South Wales. This work was done in association with the Joint Coal Board of NSW, between February and May 1991.

A wide range of hydrogeological and geotechnical factors were considered. Analysis indicates none of these factors to be excessively detrimental to the establishment of the proposed mine.

Geotechnical factors of principal importance to safe, economic mining at Mount Owen include : material strengths, characteristics of rock-mass defects, groundwater pressures, and bedding dips. These dictate the geometry of the pit walls to be constructed.

The coal seams were confirmed to be semi-confined aquifers. Groundwater pressures in a coal seam beneath the pit floor will need to be reduced before mining proceeds beyond 25 metres depth to reduce the risk of lowwall heave.

Limits are not placed by rock strengths on planned pit slopes for the proposed 30 metre deep pit. However, defect orientations limit the slope angles of the highwall and endwalls and safe design angles have been specified. Blasting of all rock overburden will be required but the soil and some of the weathered rock can be stripped without blasting.

A modest amount of further work is required to provide confirmation of parameters important to mine design considerations including: geotechnical characterization of rocks in the weathered zone, additional work on the unweathered rock zone (including undisturbed coal core), and more extensive piezometric instrumentation and hydraulic testing. Clarification of an apparent disagreement between bedding dips measured in the drill cores and those from large scale cross sections is also required.

Additional work is also needed to address issues not covered by this report, such as: excavation characteristics of the soil and weathered rock zones; impact of groundwater; blast design; equipment selection; and extension of mine plans to any areas, seams and/or depths other than those of the proposed initial pit.

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SUMMARY

General

This report contains the results of a preliminary geotechnical and hydrogeological assessment of conditions to be encountered in a proposed open pit coal mine at Mount Owen in the Hunter Valley of NSW, Australia.

Work Programme

A limited field study involved the drilling of 14 boreholes, of which 6 were partially cored in HQ size by the triple tube wireline method using a tungsten tipped bit and air flush. Piezometers were constructed in 9 boreholes using 50 mm diameter uPVC casing, slotted and gravel packed over the test interval, and sealed with cement grout back to the surface. Coring was almost entirely limited to fresh rock and coal intervals and very little information was obtained on the soil or weathered rock zones.

A small number of rock strength tests using a Point Load Test frame were carried out on the fresh rock.

The defects (joints) in the drill core were oriented, wherever possible, relative to the bedding dip direction, and the nature of the defects was logged.

Geophysical logging was undertaken but satisfactory correlation of the sonic velocity log to rock strengths could not be established.

Although the results of all of the fieldwork are recorded in this report, budgetary constraints made it necessary that the majority of the drilling and testing was undertaken or supervised by Joint Coal Board (JCB) personnel. The South Australian Department of Mines and Energy (SADME) Engineering Geologist supervised most of Borehole 128 and Borehole 206, and conducted the Point Load Testing.

Results

SADME personnel have analyzed all data from this programme.

The extremes of strength of fresh rock materials ranged from very weak to strong. However the large majority of the sandstone, mudstone and siltstone which comprise the interburden fall within the moderately weak to moderately strong range. The very weak materials were limited to thin zones. The strong rocks were confined to thin zones with siderite cement. Blasting will be required below the zone of extensive rock mass weathering to enable overburden removal. Bedding is reported to have an average dip of approximately 17°.

Rock mass defects, which are generally very widely spaced, were oriented relative to bedding. There appear to be 6 main defect sets. No single defect set is oriented such that highwall or endwall stability should be adversely affected, and the recommended maximum batter slopes of:

- 60° highwall overall
- 60° south-west endwall
- 70° north-east endwall

would not be significantly affected by slab failure or from wedge failures resulting from combinations of defects.

Some weak zones parallel to bedding dip were noted in the core. If these were to occur in the lowwall their orientation and low strength could contribute to lowwall slab sliding. They appear to be discontinuous, however, with only a single intersection occurring in the lowwall of the proposed pit.

Variable quality groundwater is present at an average depth of 7 metres. The coal seams are identified as aquifers with a permeability range estimated to be from 0.3 to 1 metre per day (moderate permeability). The interburden rocks also have appreciable (but lower) permeability.

Groundwater modelling is beyond the brief given to SADME, but it is anticipated that the worst case situation could be stated as:

- trafficability and materials handling could be impaired during excavation of the saturated weathered rock in the initial box cut
- little groundwater will flow onto the pit floor, particularly if the water released from the Middle Liddell coal seam is collected in pit-floor sumps
- pressure relief from the Lower Liddell seam will be required prior to mining beyond 25 metres depth in order to eliminate the risk of floor heave. This should be readily achieved.

There are no indications that either hydrogeological or geotechnical factors would be excessively detrimental to the establishment or operation of the proposed open pit coal mine at this site.

CONCLUSIONS

1. Within the limitations of this preliminary investigation by SADME, there are no indications that either hydrogeological or geotechnical factors would be excessively detrimental to the establishment or operation of the proposed open pit coal mine at this site to 30 metres total depth.

2. There is a discrepancy between the observed bedding dips measured in the drill core and the mean dips taken from the cross sections which were provided by JCB. There are a number of possible causes including faulting. This matter requires early resolution.

3. A moderate amount of further work is required to confirm mine design parameters which are recommended in this report. It is, however, concluded that for design purposes:

   (a) the weathered and fresh rock, beneath the soil cover, are fully saturated;
   (b) the soil and at least some of the weathered rock should be amenable to excavation by scraper, dozer-assisted scraper or ripper-scraper methods;
   (c) the effect of groundwater on excavation of the weathered rock zone during box cutting is not known, but this material should be substantially dewatered by drainage towards the box cut when subsequent strips are removed;
   (d) the strengths of fresh rock lie mainly in the moderately weak to moderately strong range and defects are widely spaced. Blasting will be required for economic removal of the overburden;
   (e) rock strength would be sufficient to permit design of very steep highwalls but defect orientations put limits on highwall and endwall slope angles;
   (f) the pit lowwall will require depressurization prior to mining beyond 25 metres depth. This appears to be achievable by passive drainage. Failure to do so may lead to lowwall heave;
   (g) weak zones parallel to bedding may exist in the lowwall, but they appear to be discontinuous and to be unlikely to affect lowwall stability;
   (h) the coal seams are aquifers of moderate permeability. The interburden is of low permeability. The coal seams will behave as semiconfined aquifers.

4. Additional work may be required, following discussions with the mine design engineers, to address other matters which may include:

   (a) thickness of the soil and weathered rock zones, their properties and excavation characteristics
   (b) quantification of the impact (if any) of groundwater on mine design and operation, along with related environmental considerations
   (c) collection of the data required to permit blast design to be matched to the selected excavation and haulage equipment.

RECOMMENDATIONS

1) Recommended maximum overall pit slope angles are summarized below.

RECOMMENDED OVERALL PIT-SLOPE ANGLES FOR PRELIMINARY MINE DESIGN PURPOSES

PIT SLOPE ELEMENT	RECOMMENDED SLOPE ANGLE
SOIL STRENGTH MATERIALS (From 3 to 8 metres thick)	30°
WEATHERED ROCK MATERIALS (Indicated thickness range of 4 to 12 metres)	60° Note: These materials should be separated from those of the high-wall by a bench.
HIGHWALL (IN FRESH ROCK)	60°
NE ENDWALL (IN FRESH ROCK)	70°
SW ENDWALL (IN FRESH ROCK)	60°

Further work is required to confirm these parameters.

2) It is considered advisable to undertake early resolution of the possible discrepancy between dips observed in the drill core and those shown on cross sections through the deposit.

3) It is recommended that groundwater pressures in the lower Liddell seam be reduced prior to mining below 25 m depth.

4) Additional work is recommended to address issues related to:

(a) excavation characteristics and volumes of the soil and weathered rock zones
(b) establishing the impact of groundwater on the mine and the environment
(c) blast design
(d) equipment selection.

This work should be directed towards solving specific problems identified during the mine design stage.

5) Some broad scale screening of the conceptual plans for mining beyond 30 metres depth and/or to mine other seams or in other areas of the deposit should be undertaken during the next investigation.

FURTHER WORK

Although a number of broad issues are identified for further study, the amount of work required to resolve these matters is considered modest.

Subject to an assessment of the sensitivity of mining operations, and hence costs, to these issues the further work for the proposed open pit would be likely to include:

(a) resolution of the apparent discrepancy between core dips and those from the cross sections;

(b) core drilling, logging and testing of materials in the weathered rock zone, including laboratory testing. Diamond-drilled HQ sized core is recommended;

(c) additional core drilling in the unweathered rock zone, with logging, testing (including laboratory testing), and sonic and dipmeter wireline logging. some undisturbed coal core is required for geotechnical examination;

(d) installation of piezometers designed to permit more extensive hydraulic testing of aquifer and aquitard lithologies. Installation of wells for long term pump tests may be required. Limited groundwater modelling should be undertaken to quantify groundwater inflows and the influence of groundwater on slope stability.

1.0 INTRODUCTION

1.1 General

The Mount Owen Coal Deposit lies approximately 20 km to the NNW of Singleton in the Hunter Valley of NSW (see Figure 1). The deposit is owned by the Hunter Valley Coal Corporation (HVCC). A large resource of high grade coal has been partially delineated, of which approximately 31 million tonnes may be accessible by open cut mining methods to 100 m depth. The coal occurs as multiple seams in a structurally complex environment.

The topography of the mine lease area is subdued, consisting of flood plains and low rolling hills. The land is predominantly used for cattle grazing. Thick, mature forests cover part of the southeast corner of the lease.

1.2 Previous Work

Prior to the Joint Coal Board of NSW (JCB) becoming involved as Project Managers, only a moderate amount of work had been done. 127 boreholes had been drilled (the first 25 of which were fully cored) over a period of approximately 12 months to February 1991. The locations of many of these holes were far from optimized. Almost no attempt was made to gather information of an engineering geological nature during this drilling episode.

In February 1991 the owners of the deposit (HVCC) appointed the JCB to manage the project and carry out investigations to establish the feasibility of mining in the area and where best to site the initial mine excavation.

As a result of extensive discussions with consultants in the fields of coal quality, mining and geotechnics, the HVCC decided on March 1 that mining would commence in the southern central part of the deposit area. Conceptually, this first cut is to be about 30 m deep, about 100 m wide and approximately 800 m along strike (see Figure 2).

1.3 Regional Geology

The Mount Owen Coal Deposit is near the north-eastern boundary of the Hunter Coalfield; an area approximately 200 km by 70 km in the north part of the Permian Sydney Basin. The tectonic setting is thought to be associated with the development of a retroarc basin of a volcanic island arc complex (Beckett, 1988).

The deposit lies within the Foybrook Formation of the Vane Subgroup - part of the Late Permian Wittingham Coal Measures. The depositional environment of the Foybrook Formation was a "river dominated delta system, which prograded to the south and south-west from the New England Fold Belt". (Beckett, 1988; p.32).

Major thrust faults exist immediately to the northeast and southwest (Hunter Thrust and Hebden Thrust, respectively) of the deposit (see Figure 1). Both of these faults trend NW-SE, dip to the NE and have a sense of movement of NE block up (see Figure 1).

All the major coal seams of the Foybrook Formation are present in the lease area. These range from the Lower Hebden seam to the Lemmington Seams. They are shown in the stratigraphic column (Fig. 3) along with seam name abbreviations used throughout this report. The sediments separating the coal seams are predominantly fine to medium grained, lithic, clayey sandstones interbedded with siltstones and mudstones.

The sediments have been folded into a broad anticline and syncline with axes trending approximately E and plunging about 10 degrees to the E.

Some faults of both normal and reverse sense of movement have been indicated from recent drilling and interpretation by the JCB, but these are reportedly not of major significance.

The inferred traces of the faults of regional significance are shown on Figure 2.

TABLE 1: DETAILS OF BOREHOLES

Borehole Number	Easting	Northing	Collar EL	Depth Drilled (m)	Cored Depth (m)	Hole Diam. (mm)	50mm PVC Standpipe	Slotted Section	Gravel Packed Interval	Geophysically Logged	Field Permeability Test
128	309238.0	1414580.7	125.6	39.61	4.12 to 39.61	100	NO	N/A	N/A	YES	N/A
129	309277.6	1414614.8	124.7	57.51	33.17 to 57.51	100	YES	45.5 to 57.5	38 to 57.5	YES	YES
160	309178.1	1414467.5	125.8	51.14	5.79 to 51.14	100	NO	N/A	N/A	YES	N/A
161	309147.7	1414538.6	121.5	56.58	15.2 to 56.58	100	NO	N/A	N/A	YES	N/A
170	309129.2	1414609.5	121.55	40	NIL	115	YES	32 to 40	30 to 40	NO	YES
171	309133.8	1414617.8	121.6	33	NIL	115	YES	21 to 33	18 to 33	NO	YES
172	309270.1	1414670.3	125.25	41	NIL	115	YES	35 to 41	31 to 40	NO	YES
173	309290.1	1414624.3	124.5	33	NIL	115	YES	21 to 33	18 to 33	NO	YES
174	309465.4	1414749.4	125.8	62	NIL	115	YES	50 to 62	46 to 62	NO	YES
175	309475.8	1414752.3	125.9	44	NIL	115	YES	36 to 44	20 to 44	NO	YES
176	309725.1	1414928.2	130.9	32.61	13.14 to 32.61	100	NO	N/A	N/A	YES	N/A
178	309032.3	1414349.6	123.0	42	NIL	115	YES	30 to 42	24 to 42	NO	YES
179	309459.5	1414837.2	126.4	42	NIL	115	YES	36 to 42	34 to 42	NO	YES
206	NOT AVAILABLE			54.61	14.83 to 54.61	100	NO	N/A	N/A	YES	N/A

2.0 SCOPE OF WORK

The JCB has requested the SA Department of Mines and Energy (SADME) to provide consulting services in the geotechnical and hydrogeological fields, for this initial mine development.

The work consisted of:

1. Advice on the type and amount of work required to gauge the engineering properties of the rock materials and to estimate relevant hydrogeological parameters.

2. Supervision of some of this work on site, which included logging of drill cores, installing and testing piezometers.

3. Interpretation of the resulting data.

4. Recommendations for preliminary mine design parameters and of further work required to aid in ensuring coal is recovered both economically and safely.

3.0 FIELD PROGRAMME

The engineering geology field programme involved drilling a total of 14 boreholes (including a total of 205m of HQ core), limited field testing of cored rock materials, piezometric instrumentation of nine (9) boreholes, down-hole geophysical logging and hydrogeological tests on piezometers.

A Senior Engineering Geologist from SADME was on site during the periods February 25 to March 7 and March 21 to March 25, at the request of JCB.

During the first of these periods the selection of the mine site was finalized by JCB and the HVCC, leading to a major revision of the drilling and testing program. Drilling commenced on March 4 and progress by the contractor (Thomson-Holland Pty Ltd) was slow. Only a single cored borehole (No. 128) was supervised prior to SADME's departure from the site. This hole was abandoned at 39.61 metres depth due to hole collapse following equipment failure.

In the second period, SADME supervised the completion of the drilling program (Borehole 206) carried out the Point Load Strength Tests, and relogged the drill core from the six cored boreholes.

The installation of the piezometers and the insitu permeability test work and supervision was hence carried out by JCB personnel.

3.1 Drilling and Geotechnical Core Logging

Six of the boreholes were cored and the remaining eight were non-cored (See Table 1). The cores were described in terms of lithology, structural defects, material strengths and rock mass characteristics. Core logs are given in Appendix A. Borehole locations are shown on Figure 2.

A number of factors combined to cause the core logging to be somewhat less than ideal in a geotechnical sense;

- drilling practice was such that extremely weak strata were often ground completely away or severely disturbed.

- intervals with coal were unavailable for geotechnical logging, having been removed for other purposes.

- due to budgetary constraints and a slow rate of core-drilling, it was deemed impossible for the SADME engineering geologist to be on site while all the fully cored holes were being drilled. Consequently, apart from boreholes 128 and most of 206, the cores were logged some days to weeks after they had been drilled. They had been placed in boxes and transported some distance to the core storage area. Very little of the core had been wrapped and few representative samples sealed. As a result of these factors, significant drying and disturbance had occurred making it difficult to estimate in-situ rock strength and the Rock Quality Designation (RQD).

3.2 Materials Testing

It was initially planned that samples would be sent to a laboratory for unconfined compressive strength tests (UCS) and defect shear strength determination (Direct Shear). However, on site Point Load Tests were performed instead of the UCS tests, and no suitable defects on which low shear strength would be anticipated were recovered, making Direct Shear testing unnecessary.

Representative samples of the rock materials from Boreholes 128, 161, 176 and 206 were tested using a Point Load Test frame belonging to the Joint Coal Board. Results of these tests are given in Appendix C and Section 4.1.

3.3 Piezometric Instrumentation and Tests

This section of the field work was carried out by JCB personnel.

Nine of the boreholes were instrumented with standpipe piezometers. The purpose of these instruments was to measure piezometric pressures in various water-bearing strata and for testing to estimate the rock-mass permeabilities. Field permeability tests were performed on these instruments. Details of these are given in Appendix B. The results are discussed in Section 4.4.

3.4 Geophysical Logging

The cored holes were logged with a comprehensive suite of geophysical tools. These included natural gamma, dual-spaced density, neutron-neutron, caliper and sonic logs. Only the sonic logs were used specifically for geotechnical purposes. Interpretation is discussed in Section 4.1.1, and in Appendix C.

4.0 RESULTS

4.1 Rock Strengths

The materials in the cores principally correspond to those of the fresh rock sections of both the planned high wall and low wall. As can be seen from the core logs (Appendix A) the strengths of these materials lie predominantly in the moderately weak (MW) range with lesser proportions of those in the weak (W) range and the moderately strong (MS) range (see Appendix E - General Explanatory Notes).

Thirty five (35) point load tests were performed on selected core samples. Point load strength measurements ranged from $Is_{(50)} = 0.19$ MPa to $Is_{(50)} > 1.9$ MPa (off scale on the gauge). Using the relationship $Is_{(50)} \times 24 = q_u$[1] these equate to an unconfined compressive strength range of 4.5 to 45 MPa (ie. Weak to Moderately Strong). Results and methodology are given in Appendix C, and summarized in Figure 4.

Although only a small number of core samples were preserved in a suitable state for strength testing, it is considered that a reasonable coverage of sample strengths has been obtained. Rocks in the Very Weak (VW) and Weak (W) strength class ranges (as indicated on the bore logs) would have been very fragile and difficult to test with the equipment available. Rocks in the Strong (S) class range or stronger are rare on this site, and are confined to the occasional thin bands of siltstone or mudstone which have sideritic cement. Representative samples of rocks between these two extremes were tested.

4.1.1 Correlation of Sonic Velocity with Rock Strength

Downhole sonic velocity logging was performed in each of the cored holes. An attempt has been made to relate measured rock strengths with logged sonic transit times for the Mt Owen site. The details of this are given in

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1 Although experience indicates the relationship to be widely applicable, no confirmation was undertaken and the unconfined compressive strength range should be taken as indicative only.

Appendix C. At this stage, however, no useable relationship between sonic values and rock strengths can be stated with any confidence. The reasons for this include:

- there are only a small number of strength values measured.
- the strength values show a wide scatter when plotted against transit time.
- the relationship between Point Load Strength ($Is_{(50)}$) and Unconfined Compressive Strength (q_u) has not been established for this site.
- the "noisy" nature of the sonic logs makes it difficult to relate reliable values of transit time to corresponding rock strength measurements.

4.2 Rock Mass Defects

Rock mass defects in the cores were logged for relevant physical attributes including type, orientation (where possible), roughness, planarity, amount of infill material and infill characteristics, evidence of shearing and any other noticeable characteristic. No idea of continuity or extent of any of the defects can, however, be gained from their exposure in a borehole. One hundred and twenty (120) defects were described in the cores.

4.2.1 Defect Types

The predominant type of defects seen in the cores were clean joints. The joints generally had little roughness, but some ranged up to roughness 5 (see Appendix A). Most defects were relatively planar (on the scale of the core sample) but many were slightly curved. Although it is difficult to gauge from disturbed core, it is considered that the joints are generally tight. Few contain infillings or coatings on surfaces but many are polished or slickensided. Other types were crushed zones, sheared zones, zones in which there were frequent similar joints and weak zones parallel to bedding. (Refer Appendix A - Borehole logs).

4.2.2 Defect Orientations

The method used to orient the defects requires the assumption that the highwall will be excavated parallel to the coal seam strike at any given point. It therefore becomes possible to consider defect orientation in relation to the orientation of bedding in the rocks as measured in the core, and to relate the defects so oriented to the faces created by the excavation of the mine.

From previous drilling it was possible to establish the local bedding plane orientation from the level of intersection of the same stratigraphic unit in three boreholes located at apices of a triangle. This method assumes that there is no faulting of the strata between boreholes. This procedure was also used to establish the average dip of 17° used throughout this report.

Measurement of the orientation of each defect relative to bedding surface(s) observed in adjacent core allows the orientation in three dimensions to be approximately calculated. The method is more fully explained in Appendix D.

It proved possible to estimate orientations for only 74 defects which reflects both the low quality of some of the coring and the relatively massive nature of some of the sandstone strata. All oriented defects were plotted onto an equal area lower hemisphere stereographic projection as poles to planes. Six (6) main sets of joints are indicated.

The six joint sets and a hypothetical 60° highwall are plotted on a lower hemisphere stereographic projection in Figure 5.

The small total number of oriented defects, the uncertainties associated with the relative orientations of large scale cross bedding (cross-bedding is usually indistinguishable from normal bedding in drill core) and a defect frequency bias caused by all the boreholes being vertical must all be taken into account in assessing the statistical relevance of the defect orientation data.

The results of the defect orientation indicate that the following conclusions can be drawn:

- If highwall slopes are excavated at 60° they will not undercut either individual joint sets or wedge combinations of joint sets.

- If intermediate highwall benches are cut with batters at 65° or steeper, the individual benches may be affected significantly by either slab or wedge failures.

4.2.3 Weak Zones Parallel to Bedding

Only three significant weak zones parallel to bedding were found in the drill cores from two of the boreholes (129 and 206). Due to severe disturbance during drilling none of these zones were suitable for sampling and testing by Direct Shear.

In addition there were six measurable core losses in two of the holes (206 and 128) ranging in thickness from 0.08 m to 0.63 m. All but one of these core losses were interpreted to be due to grinding, crushing, shearing and/or washing away of the core due to poor drilling practice. However, the 0.63 m core loss in borehole 206, from 50.98 to 51.61 m depth, is interpreted as corresponding to mudstone having an extremely low strength.

Of these four weak zones, three occur in borehole 206 (350 m to the south of the proposed initial pit) and one in borehole 129. All except one of them occur either immediately above or immediately below the Lower Liddell seam. This fourth one (at 36.51 to 37.11 m depth in Borehole 206) occurs approximately 3 metres below the Middle Liddell seam.

This would appear to indicate that weak seams parallel to bedding are likely to be discontinuous on the scale of drillhole spacing, and not of sufficient frequency to prove detrimental to lowwall performance in the proposed pit in which the MLD seam is to be mined. Future core drilling should however, be of better quality in order to verify this conclusion.

4.2.4 Defect Spacings

Joints logged in the cored boreholes tended to occur in clusters of between 3 to 8 relatively closely spaced (0.02 to 0.2m) joints separated by moderate to large vertical distances (2 to 10m) of little or no jointing. Most of the clusters of joints occur in the finer grained rocks (siltstones and mudstones). Joints are much less prevalent in the sandstone units, and usually exist singly or in smaller groups of 2 or 3 closely spaced joints (0.05 to 0.2m) separated by 2 to 10m of intact core.

The actual spacing of defects of a similar orientation (ie those within the sets shown on Fig 5) should be corrected from the above apparent spacings to an actual spacing by a geometric correction factor (Terzaghi, 1965) but there are insufficient oriented defects to make the results meaningful.

4.3 Bedding Dip Observations

It was felt necessary to compare bedding dip angles observed and measured in the drill core with the mean bedding dip angles obtained from the major stratigraphic units in adjacent boreholes which were used by JCB to produce cross sections (eg Fig 6). The results are summarized in Table 2. Any major disagreement between these dips would require clarification.

Measurements of bedding dips in the core (124 measurements) range from 10 to 42 degrees and have a mean of 23.5 degrees (see Table 2).

It was thought that a significant bias could have arisen from the inclusion of sandstone cross bedding with normal bedding plane dips.

Consequently bedding dips from the fine grained rocks, where cross bedding would be unlikely to occur or be confused with normal bedding, were analyzed separately. The result leads to a mean dip of 22.9 degrees. Some degree of local steepening of bedding dips is indicated. The extent is not known.

Such local steepening could be caused by a number of geological processes including depositional steepening, faulting or differential compaction of the coal measures. This matter should be resolved at an early opportunity.

TABLE 2: ANOMALOUS BEDDING DIP OBSERVATIONS

SOURCE OF INFORMATION	RANGE OF DIP	NO OF OBSERVATIONS	MEAN DIP
JCB CROSS SECTIONS	16-22	3[1]	17[2]
CORE LOGS (all rocks)	10-42	124	23.5
CORE LOGS (siltstones and mudstones)	10-38	65	22.9

Note 1: From three cross sections judged to be representative of the subject area.
2: Bulk of subject area.

4.4 Groundwater

The coal measures are saturated with groundwater to between 3m and 10m of the natural surface, as measured in the standpipe piezometers. The groundwater table appears to be approximately parallel to the topography. Hydraulic gradients are typically 1 in 15 to 1 in 30. Static Water Levels (SWL) are given for the 9 piezometers in Appendix B.

4.4.1 Permeabilities

Interpretation of the results of the Rising Head Tests of the piezometers (see Appendix B) indicate the permeabilities shown in Table 3. These hydraulic parameters have been plotted onto a typical cross-section in Figure 6. In general, the coal seams have moderate permeabilities and the sandstone interburdens have low permeabilities. The coal seams are expected to behave as leaky aquifers, and the interburden as aquitards, although some of the weak seams (Section 4.2.3) may comprise permeability barriers.

TABLE 3: ESTIMATED PERMEABILITY

Seam/Stratigraphic Interval	No. of Tests	Estimated Permeability (m/day)	Average Thickness (m)	Estimated Transmissivity (m^2/day)
Overburden of MLD Seam	2	0.02	20 (say)	0.4
Middle Liddell Coal Seam	2	0.3	5	1.5
Interburden from LLD to MLD	1	0.05	14	0.7
Lower Liddell Coal Seam	3	0.3	1.5	0.45
Interburden from Barrett Coal Seam to LLD	1	0.2	15	3.0
Barrett Coal Seam	2	1.0	3	3.0

Note: Abbreviations are defined on Figure 3.

It is emphasized that these permeabilities result from a small number of quick, inexpensive tests and can be considered to be approximate only. In order to design a detailed footwall depressurization system more extensive testing will be necessary to provide design values with a greater level of confidence.

4.4.2 Groundwater Quality

No samples of groundwater were taken for chemical analysis during the current programme. The information in Table 4 comes from section 9.5.3 of a report on the lease area by Resource Planning Pty Ltd (Resource Planning, 1990).

TABLE 4: GROUNDWATER QUALITY DATA

Acidity (pH): Neutral to mildly acidic.
Total Dissolved Solids: 2000 mg/l to 16,100 mg/l
Dissolved Iron: 0.3 mg/l to 18.8 mg/l

It is our present expectation that some dewatering or depressurization of the coal measures will become necessary (see section 5.5.1) and a significant volume of poor quality water could be produced. Calculations of the projected quantities and the likely quality of this water are likely to be required before looking at options for discharge to the environment.

5.0 DISCUSSION

It must be emphasized that the present investigation is of a limited nature and is specifically relevant to the planned initial pit (see Figure 2). The amount and type of work undertaken in this study is consistent with establishing the geotechnical feasibility of the project, as proposed. It provides sufficient guidance for preliminary pit design and planning considerations (and to a lesser degree equipment selection) in only the most fundamental parameters. Further work will be required to confirm or refine critical preliminary design assumptions and decisions. Further work will also be required when it is desired to extend the mining operation to other seams, adjacent areas, a deeper pit or to make any other significant change from the initial pit outlined in Section 1.2.

During this study by SADME there has been no opportunity to discuss pit design options with the engineers. It is therefore necessary to make some assumptions about box cut, highwall and low wall design, and about practices which may be employed in excavation.

5.1 Considerations for Excavations Above the Level of Oxidation (LOX)

This programme of work provided no opportunities to examine or test the soil or weathered rock zones of the overburden, and no accurate positioning of the base of each of these materials appears to exist. The thicknesses shown on Figure 6 are estimated from summary logs of the 9 open holes drilled for the piezometers, which were logged by JCB personnel.

The water table lies at 3 to 10 metres depth and is generally below the soil materials which will be exposed in the near surface excavations. However, the weathered rock zone and the underlying fresh rock will be fully saturated during the box cutting. Successive mining strips should be dewatered to some degree by drainage into previous strips. Estimates of groundwater seepage should be undertaken and provision should be made for collection and disposal of seepage.

The behaviour of the soil materials and the saturated weathered rock during excavation can not be determined from data available to SADME. While it is considered likely that scraper, dozer assisted scraper, or ripper - scraper methods could be employed some additional investigation is required to confirm both the method and estimate the productivity, as well as to establish the depth limits to which such methods could be used.

The stability of the slopes cut in the weathered rock can be expected to remain satisfactory if the rate of drainage of groundwater is sufficiently rapid, corresponding to the rate of stripping. This issue requires further consideration.

The design slopes indicated within these materials (which includes the oxidized coal in the box cut low wall) will require confirmation before being used beyond the preliminary design stage. The suggested slopes are shown on Figure 6 and listed in Table 5, but the upper bench height and the degree of flattening of the slope in the soil zone will depend on local conditions.

Materials from the soil and weathered rock zones should not be dumped at the toe or in the base of high or steep spoil dumps, as this practice is often found to be the principal cause of spoil dump failures.

TABLE 5: RECOMMENDED OVERALL PIT-SLOPE ANGLES FOR PRELIMINARY MINE DESIGN PURPOSES

PIT SLOPE ELEMENT	RECOMMENDED SLOPE ANGLE
SOIL STRENGTH MATERIALS (From 3 to 8 metres thick)	30°
WEATHERED ROCK MATERIALS (Indicated thickness range of 4 to 12 metres)	60° Note: These materials should be separated from those of the high-wall by a bench.
HIGHWALL (IN FRESH ROCK)	60°
NE ENDWALL (IN FRESH ROCK)	70°
SW ENDWALL (IN FRESH ROCK)	60°

5.2 Bedding Dips

There remains uncertainty as to the true bedding dips likely to be encountered during mining. Computer modelling based on numerous borehole intersections has indicated that dips of major correlated units (ie coal seams) average about 17°. However, numerous measurements of bedding dip in the drill core indicate steeper average dips of around 23°.

This discrepancy may be due to a number of factors, including depositional steepening or the possible existence of faults.

Early resolution is required as the result has potential to significantly impact on major aspects of the planned open-pit operation, such as:

- lowwall stability
- recoverable coal quantities
- dilution factors/washing operations
- operational considerations of mining steeper and/or faulted seams.

5.3 Highwall

Factors affecting highwall stability and performance as the mining extends down dip include:

rock substance strength
rock mass defect orientations and strength
groundwater

The strength of the rocks (predominantly in the moderately weak to moderately strong range - see Figure 4) is considered to be sufficient to ensure stability and safety of the planned highwall slope for the 30 metre deep mine, provided the weathered rock and soil materials comprise a separate bench.

It appears that the only significant combinations of rock defects likely to affect the stability, and thus design, of the highwall are:

- a wedge combination formed by intersection of joint sets 3 and 4. The line of intersection of these sets plunges at about 65° in the opposite direction to bedding dip (ie. close to the direction of dip of the highwall).
- Set 1, dipping at approximately 70°, parallel to the strike of the highwall. These sets are shown on Figure 5.

The rock mass permeability and effective porosity of the fresh rock materials in the highwall are both estimated to be low. Groundwater movement will be primarily restricted to flow along defects in the rock mass, but given that these will open slightly in response to stress reduction during overburden removal, the highwall should be relatively free draining. It is considered unlikely, however, that much groundwater will drain into the pit from this source.

In consideration of these factors the overall highwall design angle should not exceed 60°. If benches are required in the highwall, some wedge failures of the steeper batter slopes should be expected.

5.4 Endwalls

The factors affecting endwall stability and performance during mining are the same as those outlined for highwalls:

rock substance strength
rock mass defect orientations and strength
groundwater.

Two of these factors can be considered in a manner identical to that adopted in Section 5.3. These are the rock substance strength and groundwater considerations. Neither of these two factors is considered to be likely to have any adverse impact on the stability, and hence design, of the endwalls.

However, the affect that orientations of the defects in the rock mass will have on an excavated slope changes with different orientations of the excavated slope. Two different orientations of endwall need to be considered; the endwall at the south-west end (dipping NNE) and the endwall at the north-west end (dipping WSW).

5.4.1 South-West Endwall

Defects of set 4 (see Figure 5) dip approximately parallel to the direction of dip of the proposed SW endwall, at about 70°. In addition, the line of intersection of joint sets 1 and 3 (plunging at about 60° in a similar direction to the SW endwall dip) defines a wedge combination which may cause significant instability if this endwall is constructed at an overall angle steeper than 60°. Other wedge combinations exist with much shallower dipping intersections which plunge in the direction of dip of this endwall but these are expected to be of sufficient strength to ensure their stability.

5.4.2 North-East Endwall

No individual defect set or combinations of defect sets shown on Figure 5 have the potential to adversely affect the stability of this endwall. The overall design slope of this pit wall will be dictated by local mining practices. An overall slope not exceeding 70° is recommended for design purposes.

5.5 Lowwall

Factors affecting the stability of the lowwall as the mining proceeds down dip include:

the dip of the mined coal seam and floor rocks
rock strength and height of the slope
orientation and shear strength of rock mass defects
groundwater pressures.

Bedding dips in the area are moderate, and reportedly average about 17 degrees (see Section 5.2).

Rocks strengths are adequate, generally in the moderately weak to moderately strong range (see Fig. 4).

Only four rock mass defects, with the potential to affect lowwall stability, have been delineated during the current programme (see Section 4.2.3). Three of these defects may be inferred to be correlatable. They occur in very close proximity to the lower Liddell coal seam, both above the below it. Correlation between the two boreholes (129 and 206) would imply a large aerial extent of a significant weak zone parallel to bedding. This correlation, over a distance of hundreds of metres, requires checking.

The fourth weak zone occurs in hole 206 (well outside the initial pit) at about 3 m below the base of the Middle Liddell coal seam. It does not correlate with any other significant defect encountered during the current drilling programme.

There is no evidence that such weak zones would adversely affect the stability of the lowwall of the proposed initial pit. However, the probable existence of other such zones, and their likely effects on future lowwall stability assessments, should be subject to some examination and scrutiny during subsequent investigations.

There is, however, potential for pit floor heave due to high groundwater pressures in the lowwall materials, as discussed below.

5.5.1 Groundwater and Lowwall Heave

A critical pit floor depth, with respect to pit-floor heave, is reached when the uplift forces provided by confined groundwater are balanced by the load provided by the confining rocks. The Lower Liddell seam, although it is thin, is an aquifer with an average of 14m of rock confining it in the proposed pit floor (see Fig. 7).

At the site of the proposed pit the critical pit depth with respect to floor heave is estimated to be 25 metres.

To remove the potential for pit floor heave at a planned pit depth of 30m, pressure levels in the Lower Liddell seam would have to be reduced to about 20m above the planned pit floor, prior to proceeding beyond a pit depth of 25 metres. This represents a modest degree of depressurization for a 30 metre deep pit, and should be readily achievable, although more work is required to design a scheme and to confirm the aquifer parameters.

For planning and costing purposes allowance can be made for a depressurization scheme comprising, nominally, 100 mm diameter open boreholes to 20 metres depth drilled into the low wall on 300 metre centres along strike, and repeated at an offset of 150 metres per strip. Water quantities and qualities should be established for water management purposes.

The problems associated with groundwater should be investigated in detail, however, prior to deepening the pit beyond 30 metres.

5.6 Excavation Properties

As stated above, SADME is unable to comment on the excavatability of the soil and weathered rock comprising the top 10-15 metres (approximately) of the overburden. The materials strengths and handling properties need to be established in a future investigation.

The fresh rock, with strength predominantly moderately weak to moderately strong, will require blasting for economic loading and haulage with any equipment.

6.0 REFERENCES

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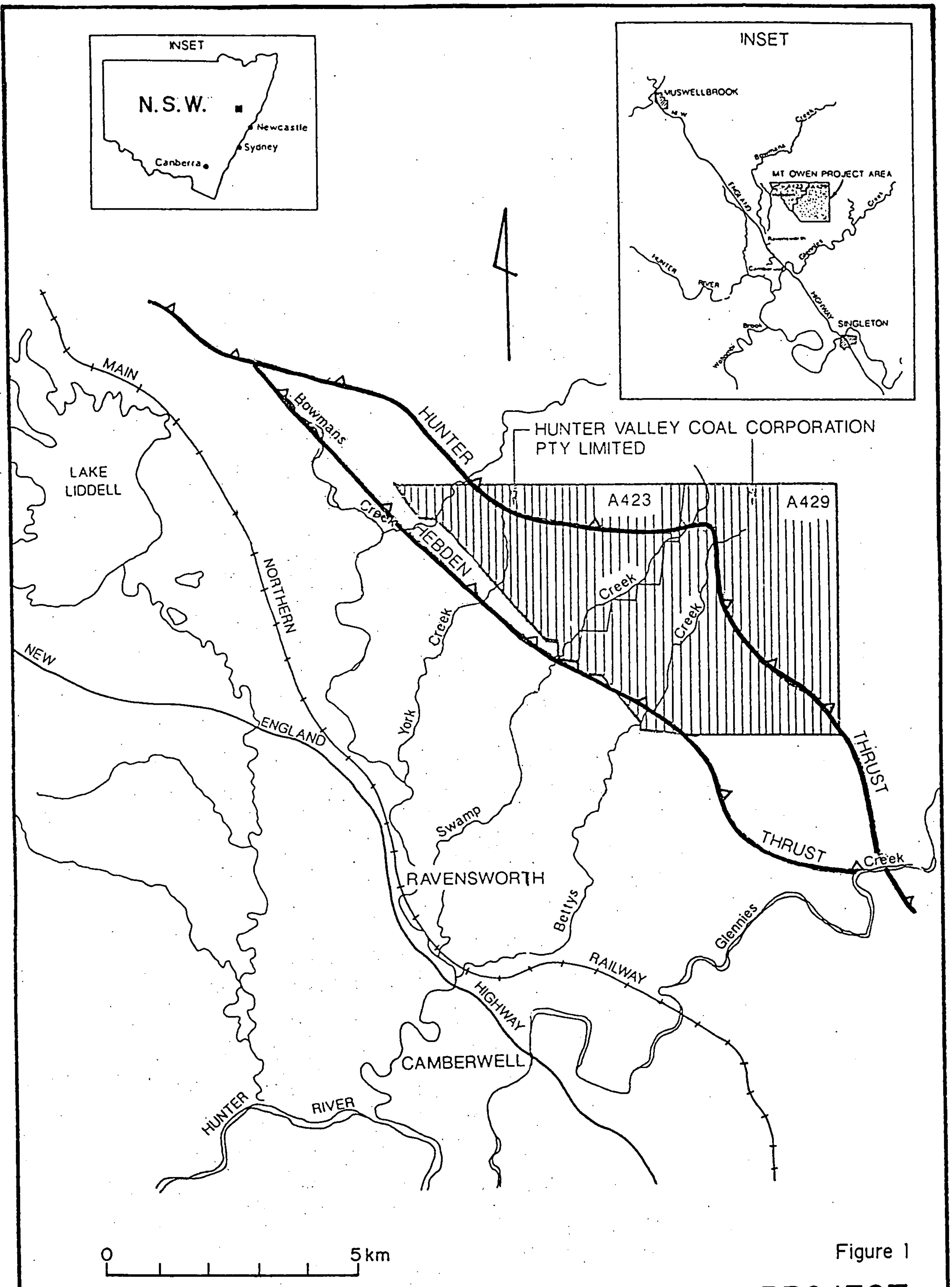

Figure 1

MT OWEN PROJECT
LOCALITY PLAN

SADME S22183

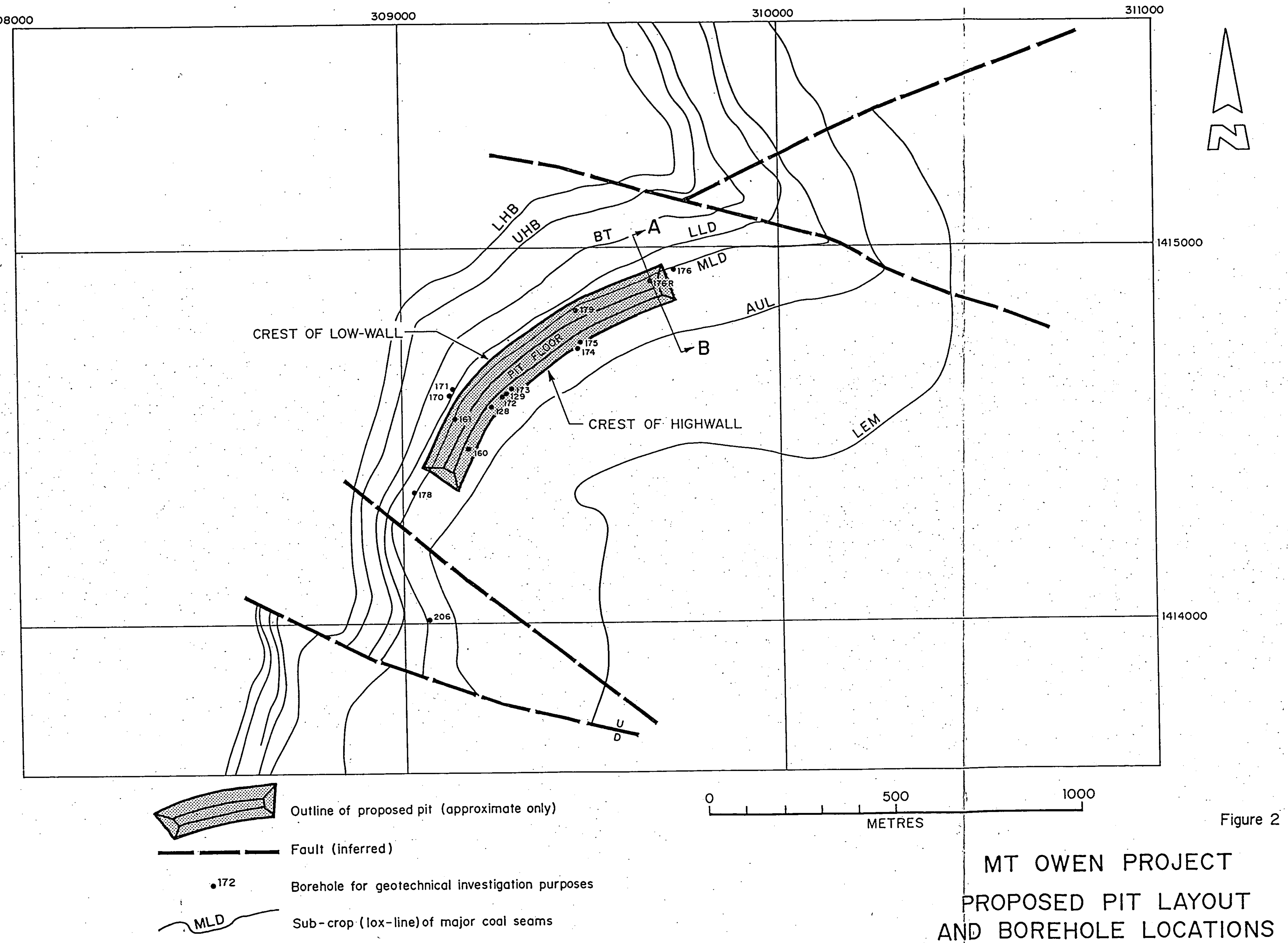

Figure 2

MT OWEN PROJECT

PROPOSED PIT LAYOUT AND BOREHOLE LOCATIONS

S.A. Dept of Mines and Energy 24-4-91

91-339

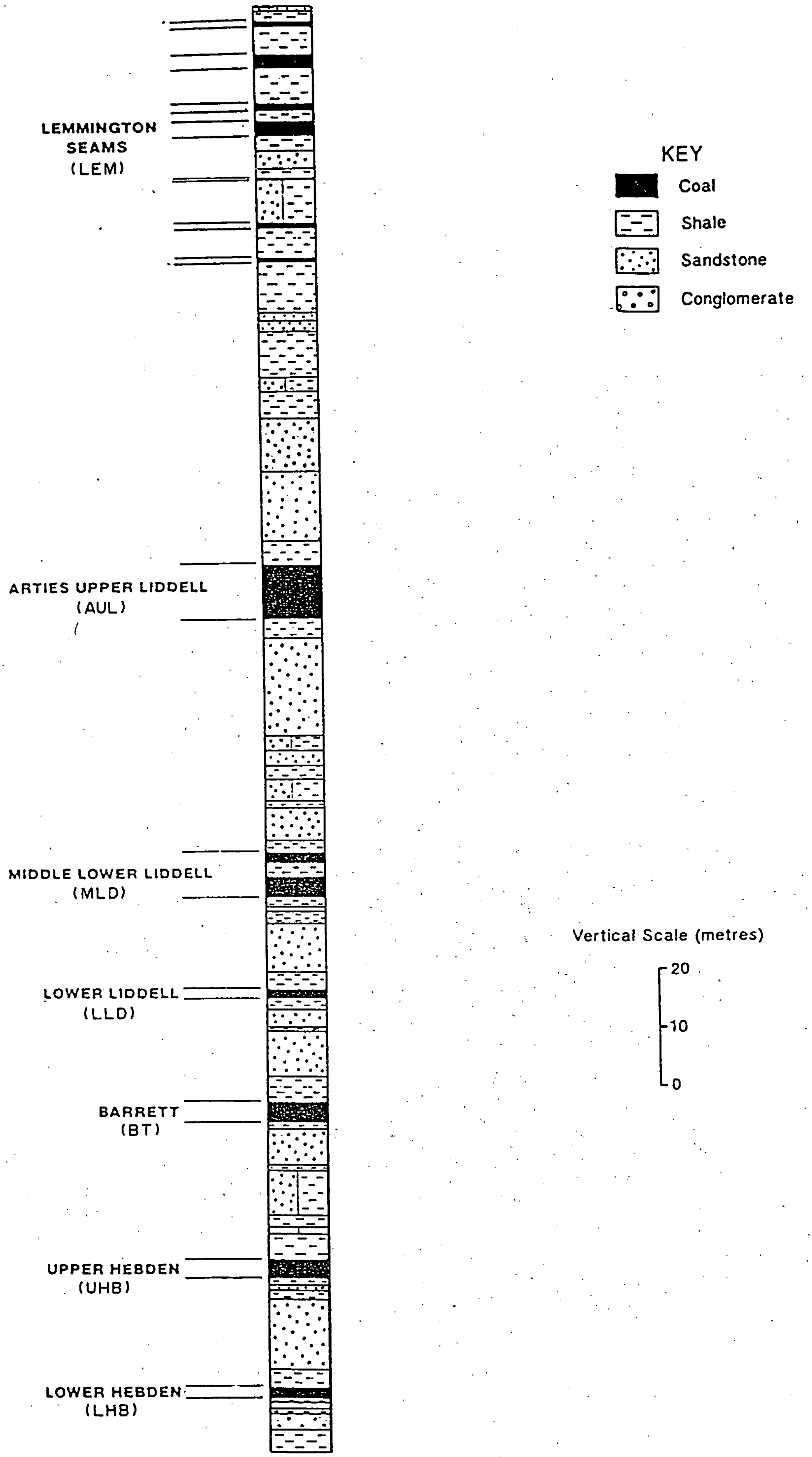

Figure 3

MT OWEN PROJECT
TYPICAL STRATIGRAPHIC COLUMN

SADME S22184

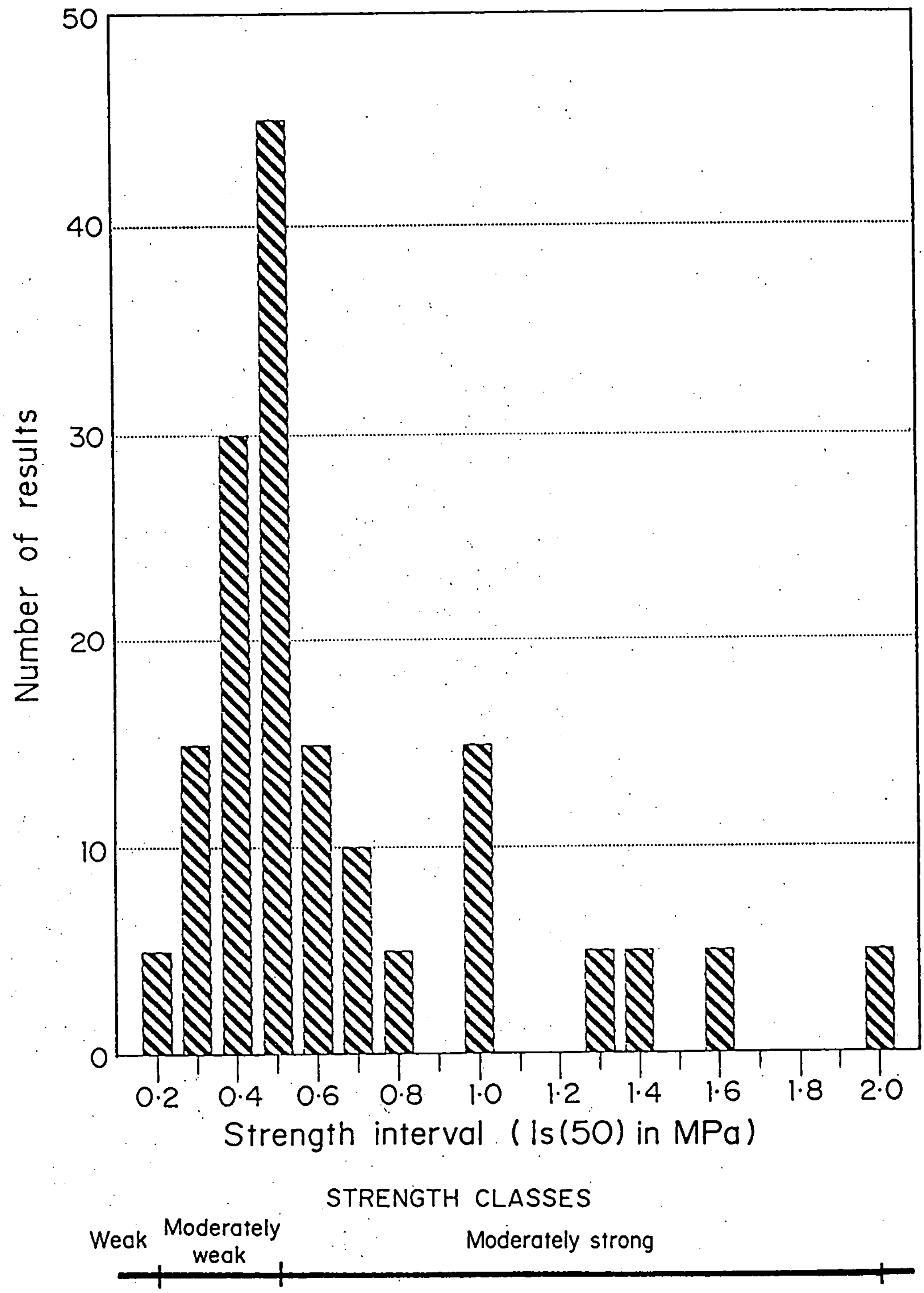

Figure 4

MT OWEN PROJECT
DISTRIBUTION OF ROCK STRENGTHS FROM POINT LOAD TESTS

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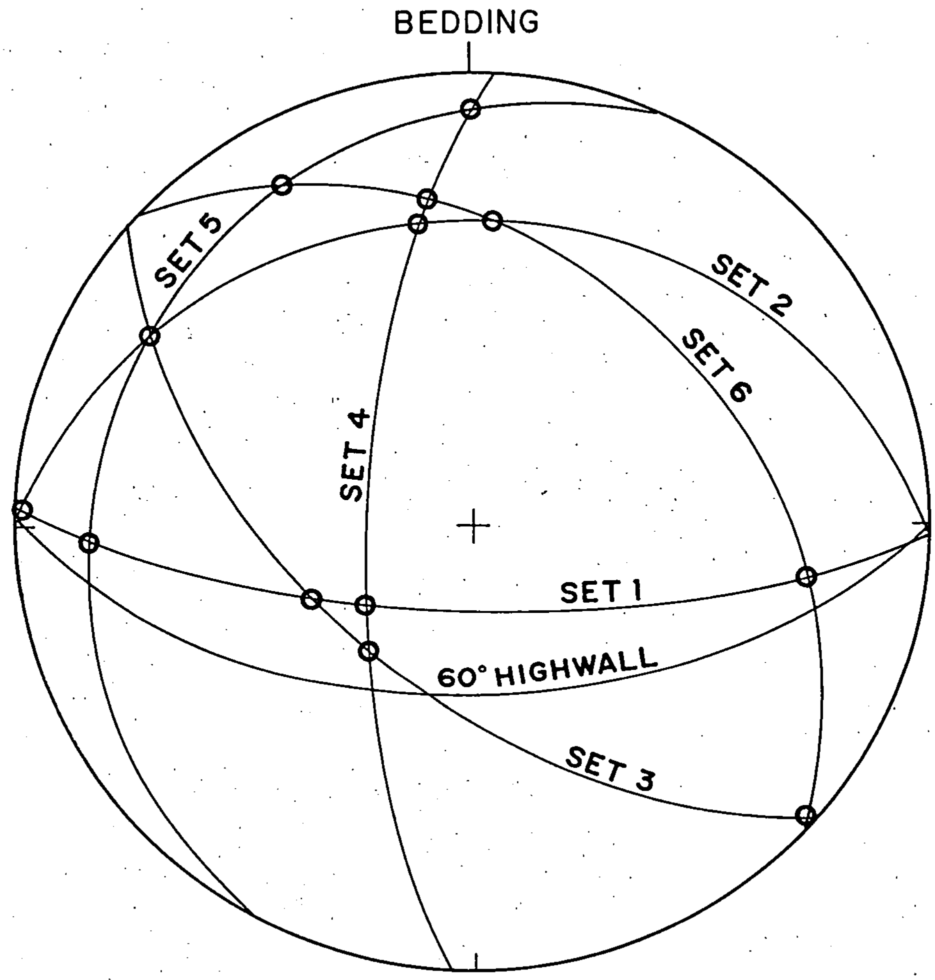
S22190

MT OWEN PROJECT
MAIN JOINT SETS



Intersection of joint sets

Schmidt net, lower hemisphere projection

Figure 5

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S22185

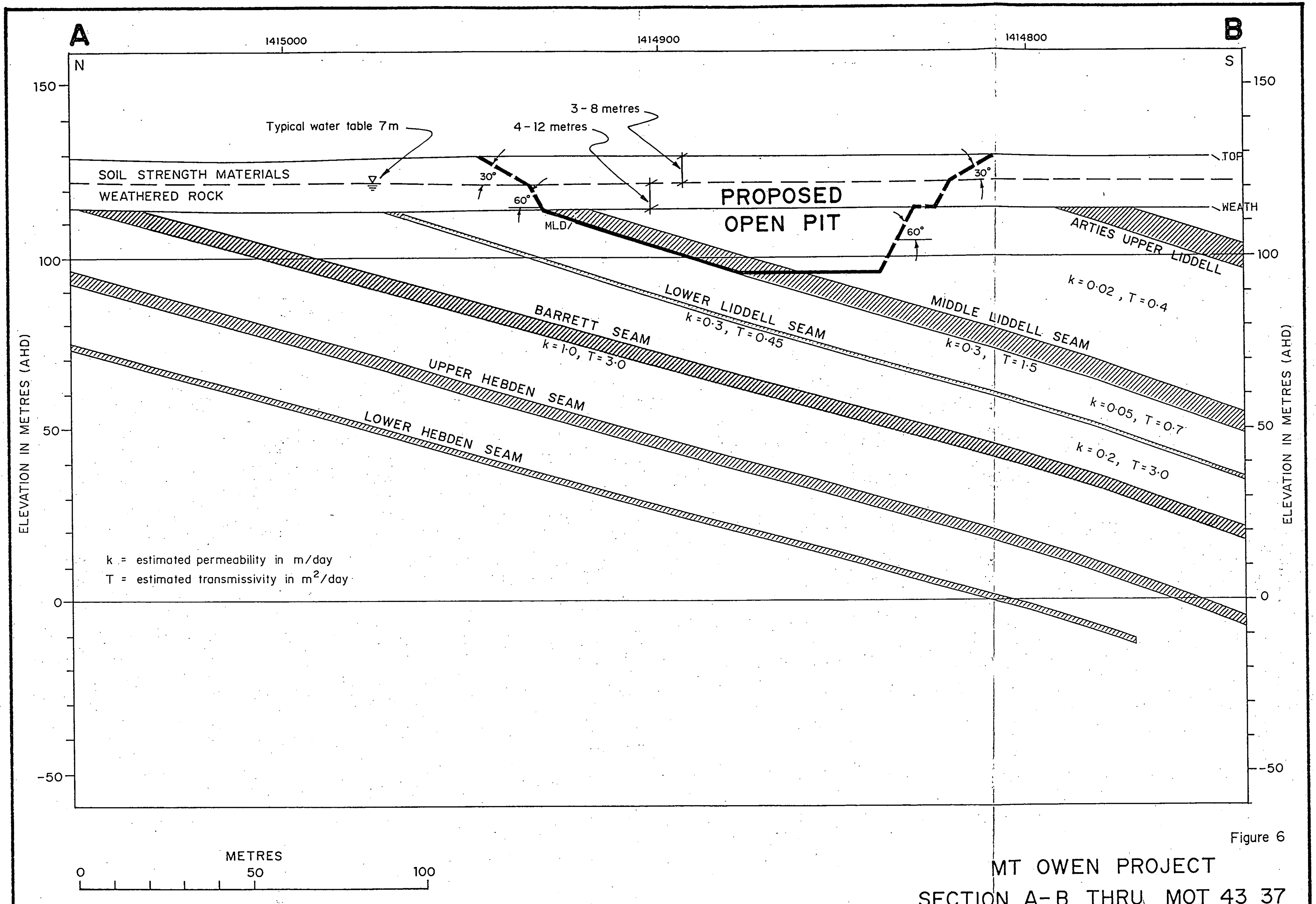

Figure 6

MT OWEN PROJECT
SECTION A-B THRU MOT 43 37

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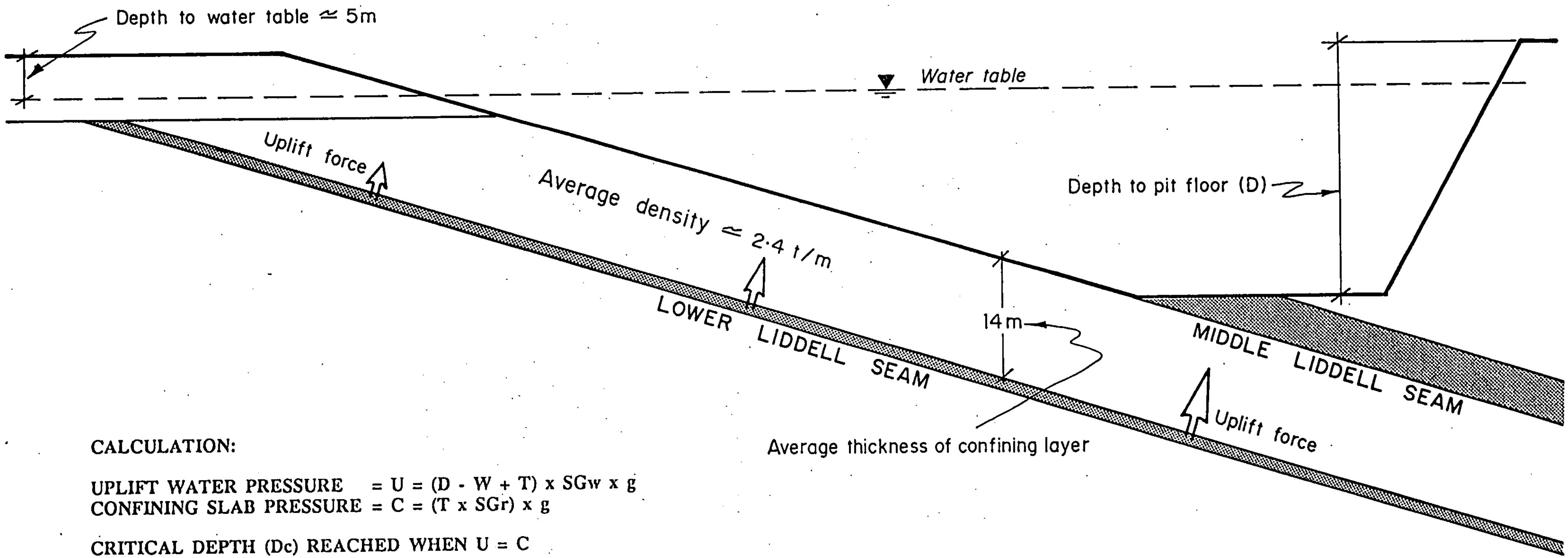


CALCULATION:

UPLIFT WATER PRESSURE = U = (D - W + T) x SGw x g
CONFINING SLAB PRESSURE = C = (T x SGr) x g

CRITICAL DEPTH (Dc) REACHED WHEN U = C

ie (Dc + 9) (9.81) = 14 x 2.4 x 9.81

Dc = 24.6m

where: D = depth to pit floor
W = depth to water table = 5m
T = thickness of confining layer = 14m
SGw = specific gravity of water = 1.0 t/m^3
SGr = estimated specific gravity of confining rocks = 2.4 t/m^3
g = acceleration due to gravity

Figure 7

MT OWEN PROJECT

CALCULATION OF FLOOR HEAVE POTENTIAL

S.A. Dept of Mines and Energy 24-4-91 S22188

APPENDIX A

GEOTECHNICAL CORE LOGGING AND GEOPHYSICAL LOGGING

1.0 INTRODUCTION

The drill cores were geotechnically logged and the following characteristics recorded:

1. Gross lithology and description
2. Core recovery/core loss
3. Rock Quality Designation (RQD)
4. Defect descriptions
5. Fracture to core axis angles
6. Bedding to core axis angles
7. Strength
8. Weathering
9. Orientation Angle (B).

Ideally, geotechnical logging of drill cores should be performed as soon as practicable after the core splits are extracted from the inner barrel and before the cores have been removed from them. The cores from boreholes 128 and 206 were logged in this way. This was not practicable during most of the current programme, however, due to time and budget constraints which limited the on-site time available to the SADME Senior Engineering Geologist. All other cores were geotechnically logged some days to weeks after drilling, boxing and transporting to a storage area. After severe disturbance of this kind it is often difficult to accurately gauge core recovery, RQD and rock strength.

The geotechnical logging by SADME has been plotted onto log sheets by JCB, and these are included in this Appendix.

All cored holes were logged with a suite of geophysical probes including:

- dual spaced density
- multi-channel sonic velocity
- neutron-neutron
- natural gamma
- caliper.

A borehole dipmeter was not used in this study, but it is a recommendation that its inclusion in future programmes be considered favourably.

No geophysical logs have been included with the geotechnical report.

2.0 GEOTECHNICAL LOGGING CONVENTIONS

2.1 Lithology and Rock Description

The main rock types encountered were coal, sandstone, siltstone, mudstone and sideritic siltstone. Descriptions of these rocks included colour, grain sizes of sand constituents, approximate percentages of different rock/soil types where they are interlaminated, approximate percentages of different mineral constituents where appropriate and, where possible, interpretation of rock type where core was lost.

2.2 Core Recovery

Core recovery was measured as the length of core recovered in each run divided by the length of the run and is expressed as a percentage. Significant core losses are shown in the graphic log column of the borehole log sheets. This core loss is shown to scale and in a position which is interpreted if not known exactly.

2.3 Rock Quality Designation

The Rock Quality Designation (RQD) is calculated according to the definition of Deere (1964). It is defined as the percentage of core recovered in intact pieces 100mm or more in length divided by the total length of the run.

i.e. $$\text{RQD (\%)} = \frac{100 \times (\text{length of core in pieces} > 100\text{mm})}{\text{length of run}}$$

The RQD number is a method of quantifying rock quality for engineering purposes and according to Deere, can be interpreted as follows:

RQD	**Rock Quality**
less than 25%	Very Poor
25% to 50%	Poor
50% to 75%	Fair
75% to 90%	Good
90% to 100%	Very Good

2.4 Defect Descriptions

The defect descriptions consist of:

1) defect type, eg joint;
2) roughness and planarity;
3) filling (if any), clean if none;
4) approximate orientation, if FCA not measurable.

Defects are classified according to the classification outlined in Australian Standard AS-1726 (1981). All defects in the drill cores are described either as bedding, joints, crushed zones or sheared zones. Bedding, where no actual break occurs, is an incipient weakness in most rocks.

Roughness and planarity are estimated according to a ten-fold classification based on the suggested methods of the ISRM (Figure A1) for roughness (R on logs) and a five-fold classification of planarity (P on logs) as follows:

1) Planar
2) Slightly curved
3) Curved
4) Folded
5) Crenulated.

2.5 Bedding to Core Axis and Fracture to Core Axis Angles (BCA and FCA)

The numbers in these columns are measurements in degrees of the smallest angle between a bedding surface (BCA) or a fracture surface (FCA), and a line parallel to the core axis.

2.6 Rock Strength

Strengths were estimated where appropriate and inserted in the borehole log sheets under the heading of "Description Lithological". The strength description and degree of cementation applies only to rock substances. These qualitative descriptions are related to quantitive measurements of rock strength according to Table A1 (from BS5930, 1981).

TABLE A1 - Rock Strength Classes (To BS-5930)

STRENGTH CLASS	DESCRIPTION	POINT LOAD STRENGTH (MPa)	APPROXIMATE EQUAL UNCONFINED COMPRESSIVE STRENGTH (q_u, MPa)
VW	Very Weak	<0.05	<1.25
W	Weak	0.05 to 0.2	1.25 to 5.0
MW	Moderately Weak	0.2 to 0.5	5.0 to 12.5
MS	Moderately Strong	0.5 to 2.0	12.5 to 50
S	Strong	2.0 to 4.0	50 to 100
VS	Very Strong	4.0 to 8.0	100 to 200

2.7 Weathering

The state of weathering of the cores was gauged according to Australian Standard AS1726 (1981), summarized below.

Term	Abbreviation	Definition
Fresh	Fr	No weathering effects visible to the naked eye.
Slightly Weathered	SW	Visible change in appearance but no significant loss in strength.
Moderately Weathered	MW	Visible change in appearance and with significant loss in strength.
Highly Weathered	HW	Considerable change in appearance and loss in strength. Material still a rock but normally very weak.
Completely Weathered	CW	Soil properties and often a complete change of appearance.
Altered	A	Chemical and physical alteration to rock fabric caused by temperature, pressure or injection of other material.

2.8 Orientation Angle (B)

The orientation angle (B) is used to relate the dip direction of a defect to the dip direction of bedding (or some other consistent reference plane in the hole). It is measured clockwise (looking down hole) from the bottom of the reference plane to the bottom of the defect to be oriented (see Appendix D).

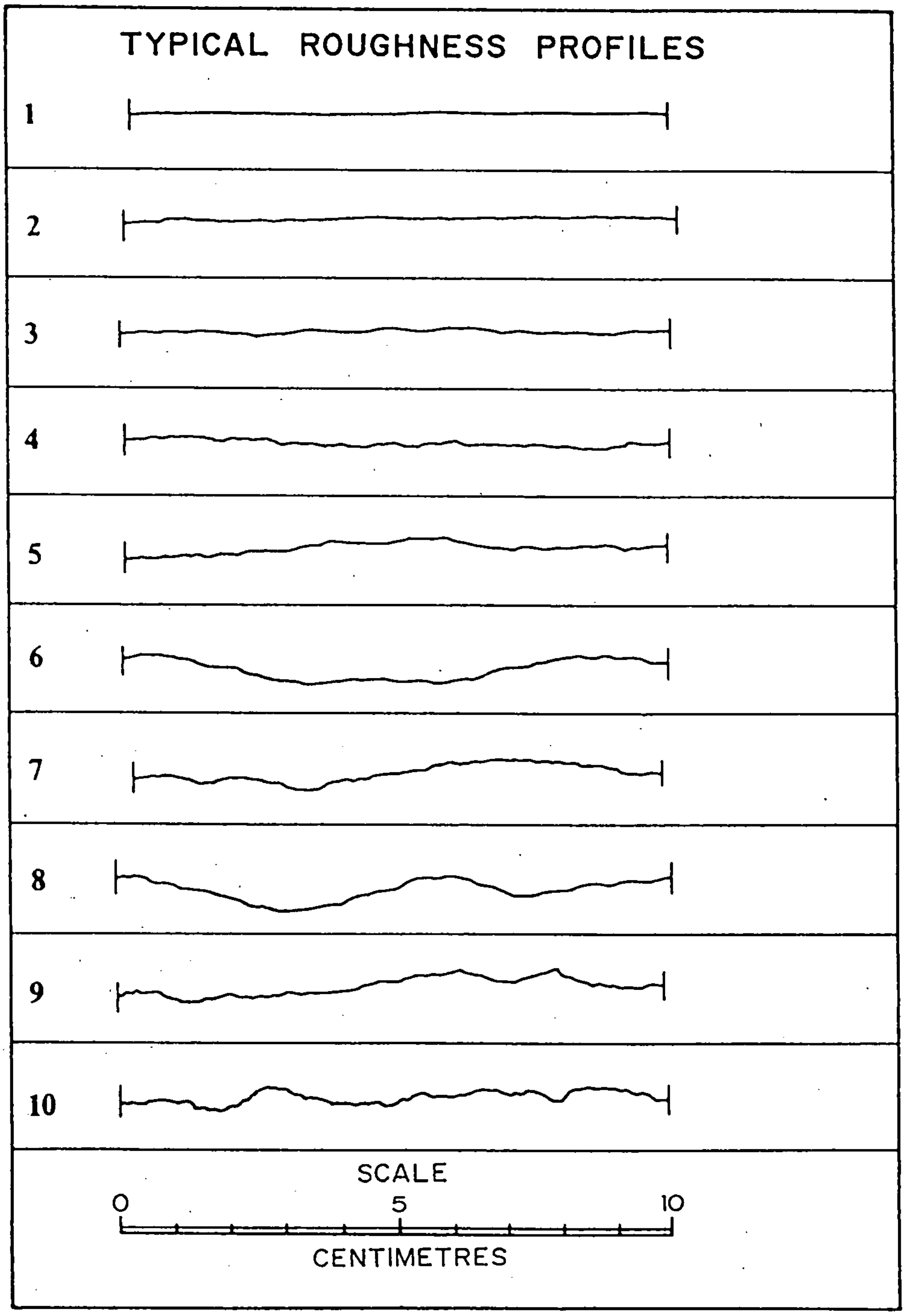


Figure A1

MT OWEN PROJECT

ROUGHNESS PROFILES FOR
DEFECT DECRIPTION IN
GEOTECHNICAL CORE LOGGING

S.A. Dept of Mines and Energy 24-4-91

S22187

BOREHOLE CORE LOG	LOGGED BY: SAM WALKER DATE 21/3/91 CHECKED BY DATE	Job No

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: 20/3/91 | Drill rig | Drilling Fluid: Air
Hole completed 22/3/91 | Mounted on | Barrel Type 3.0m, HQ, TT

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
				11	NON-CORED TO 14.83m DEPTH					
				12						
				13						
				14						
CORE LOSS				15						
SANDSTONE, light grey, fine grained, clayey, MWstrength, moderately weathered.							62			15.48m to 15.50m DOLERITE, high strength
SILTSTONE, light grey, slightly sandy, MWstrength, moderately to highly weathered.				16				67	93	
SANDSTONE, Light grey, fine to very fine grained, clayey, MW strength, moderately weathered.				17	17.65m to 17.75m JOINTED ZONE. About 10 joints spaced 5 to 10mm, R2, P2, clean, probably due to core drying	85	69			
				18			67			
SANDSTONE, light brown, medium to coarse grained, abundant lithic fragments MW strength, HW-MW				19				95	99	
					19.35 JOINT, R2, P1, clean probably parallel to bedding.	63	63			
				20						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

NOTE: Cores were logged one full day after drilling. Cores had been placed in boxes. Strengths are only estimated.

JOINT COAL BOARD

Borehole No: 206 Sheet 1 of 5 Figure

BOREHOLE CORE LOG

LOGGED BY: SAM WALKER	DATE 21/3/91	
CHECKED BY	DATE	Job No.

Borehole dip (from horizontal) 90° — Borehole dip direction — Location AMG E N EL — Location Mine E N

Hole commenced: 20/3/91 — Drill rig — Drilling Fluid:
Hole completed 22/3/91 — Mounted on — Barrel Type

DESCRIPTION LITHOLOGICAL	DEPTH	DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
SANDSTONE, as above							
SILTSTONE, light grey, slightly sandy, MW strength, moderately weathered	21						
SANDSTONE, light grey, fine grained, clay matrix, MW strength, MW							
SILTSTONE, light grey, slightly sandy, MW strength, SW-MW	22				95	100	
MUDSTONE, light grey, silty, MW strength, SW	23						
	24	CORES FROM 23.91m TO 33.45m DEPTH WERE UNAVAILABLE (COAL SAMPLE)					
COAL: black, v. brittle, abundant fractures, occ. clay and silt bands	25–29						
MUDSTONE:	30						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

NOTE: Cores were logged one full day after drilling. Cores had been placed in boxes. Strengths are only estimated, significant drying has occurred.

JOINT COAL BOARD

Borehole No: 206 Sheet 2 of 5 Figure

BOREHOLE CORE LOG

LOGGED BY: SAM WALKER DATE 21/3/91

CHECKED BY DATE

Job No

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: | Drill rig | Drilling Fluid:

Hole completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL

- MUDSTONE: lt grey, silty, MW strength
- SILTSTONE, light grey, very clayey, MS strength
- MUDSTONE, light grey, silty to very silty, MW strength
- SIDERITIC SILTSTONE, jointed, MS strength
- MUDSTONE AND COAL, thinly interlaminated, MW strength
- SILTSTONE, interbedded with MUDSTONE light grey, MS strength, SW-Fr
- SANDSTONE, light grey, fine grained, clay matrix, MS strength

DESCRIPTION STRUCTURAL

CORES FROM 23.91m TO 33.45m DEPTH WERE NOT AVAILABLE (COAL SAMPLE)

DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
34.58JOINT,R2,P4,very thin clay coatings, slickensided	67				
34.82JOINT,R1,P3,clean	47				B=75
34.79JOINT,R2,P2,clean	62				B=261
35.24JOINT,R1,P1,0.5mm clay coatings,slickensided	58	70	85	100	B=0
35.29JOINT,(as at 35.24m)	58	72			B=0
35.53to35.64,CRUSHED ZONE,	48	69			
35.77to35.81,CRUSHED ZONE,	72				
35.95JOINT,R1,P2, clean slickensided	60				
35.98JOINT,R1,P2,0.5mm calcite infill, slickensided	60	73			
36.04JOINT,R2,P3, clean	46				
36.43JOINT,R1,P2, clean	42				
36.51to37.11,extremely weak zone, cleavage parallel to beds, puggy in parts					
37.17to37.18,JOINTED ZONE,R5, P3, clean			68	97	
37.28JOINT,R2,P1,clean	49				B=315
37.47) JOINTS,R1,P1,clean,	50				
37.48) slickensided	45				
37.72to37.73 CRUSHED ZONE, fragments 1mmto20mm	45				
37.93-38.06 CRUSHED ZONE, a/a	48	55			B=18
38.49JOINT,R2,P3,clean	47	62			B=37
38.65JOINT,R1,P1,slickensided 0.5mm calcite infill	58				B=341

HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 206 Sheet 3 of 5 Figure

REHOLE CORE LOG	LOGGED BY : SAM WALKER	DATE 21/3/91	
	CHECKED BY	DATE	Job No .

rehole dip 90°
om horizontal) | Borehole dip direction | Location AMG E N EL
Location Mine E N

e commenced: | Drill rig | Drilling Fluid:
e completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
SANDSTONE, as above, medium grain size					40.15JOINT,R3,P2,clean	67	67			B=0
					40.35JOINT,R2,P1,2mm calcite infill	33				B=180
SANDSTONE, light grey, clay matrix, fine grained, MW strength					40.58JOINT,R1,P1,clean slickensided	41				B=55
				41	41.69JOINT,R2,P2,0.5mm calcite infill	21		97	103	B=187
					42.51JOINT,R1,P2,clean	62				B=28
					43.14JOINT,R1,P1,clean slickensided	78				B=308
				42	43.19JOINT,R1,P2,clean	62				B=308
					43.20JOINT,R3,P3,0.5mm calcite infill	10	65 48			B=134
					43.36JOINT,R1,P3,clean slickensided	68	68	100	90	B=278
				43	43.69JOINT,R1,P1,clean	59	59			B=0
				44						
SANDSTONE, brownish grey, sideritic,MS strength					44.51JOINT,R3,P2,clean		67			
					44.70JOINT,R3,P2,up to 0.5mm calcite coatings	34	64 73	90	95	B=65
SANDSTONE, light grey, fine to medium grained,MS strength					44.80)JOINT,R2,P3,clean	48				B=9
				45	44.81)slickensides	48				B=9
							66			
SILTSTONE, sideritic,MS strength										
SILSTONE, light grey,clayey, MS strength				46						
MUDSTONE, light grey, silty, MWstrength				47	47.02JOINT,R1,P1,clean slickensided	47	74 62			B=0
					47.25CRUSHED ZONE, angular pieces to 10mm	90	65 58	82	93	
SILTSTONE, sideritic,MS strength					47.39)JOINT,R1,P1,clean	75				B=297
				48	47.42)slickensided	75				B=297
					47.46JOINT,R2,P3,clean	32	62			B=37
					47.77to47.83 FRACTURED ZONE, angular pieces up to 40mm		53			
					48.57JOINT,R3,P2,clean	53	60	50	86	B=0
SILTSTONE, highly carbonaceous, (STONY COAL),MW strength				49	48.87to48.89 SHEARED ZONE, parallel to bedding, clayey and puggy, EL strength, disturbed	68				
COAL, black, highly cleated and fractured. MW strength							64			
				50						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 206 Sheet 4 of 5 Figure

BOREHOLE CORE LOG

LOGGED BY : SAM WALKER DATE 21/3/91

CHECKED BY DATE

Job No

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: | Drill rig | Drilling Fluid:
Hole completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
COAL, black, vitreous, highly cleated and fractured.		LLD						0	61	
CORE LOSS, VW strength, MUDSTONE				51						
MUDSTONE, light grey, very silty, VW strength					51.91JOINT,R2,P1,clean					B=330
SANDSTONE, light grey, silty, W strength				52				92	99	
MUDSTONE, light grey, very silty, W strength					52.99JOINT,R5,P3,clean					B=0
SILTSTONE, dark brownish grey, sideritic, MS strength, Fr				53	53.18JOINT,R5,P2,clean					B=180
SILTSTONE, light grey, sandy, MS strength								100	101	
SANDSTONE, Light grey, fine grained, MS strength				54						
					END OF HOLE AT 54.61m DEPTH					
				55						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 206 Sheet 5 of 5 Figure

BOREHOLE CORE LOG

LOGGED BY : SAM WALKER DATE 22/3/91

CHECKED BY DATE

Job No

Borehole dip (from horizontal) 90°

Borehole dip direction

Location AMG E N EL

Location Mine E N

Hole commenced:

Hole completed

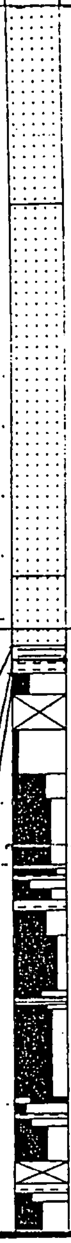
Drill rig

Mounted on

Drilling Fluid:

Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
				11						
				12	NON-CORED TO 15.2m DEPTH					
				13						
				14						
SANDSTONE, light grey, medium to coarse grain size, W strength, HW-EW				15						
MUDSTONE, light grey, very silty, W strength, HW-EW										
SILTSTONE, black, carbonaceous, W strength, HW				16	CORES FROM 16.13m TO 21.20m DEPTH HAVE BEEN REMOVED FOR COAL QUALITY ANALYSIS PRIOR TO GEOTECHNICAL LOGGING.					
				17						
				18						
		MLD		19						
				20						



HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

JOINT COAL BOARD

NOTE: CORES WERE LOGGED SOME DAYS AFTER DRILLING. CORES HAD BEEN BOXED AND TRANSPORTED TO STORE CAUSING SIGNIFICANT DRYING AND DISTURBANCE. CONSEQUENTLY, ROCK STRENGTHS ARE DIFFICULT TO ESTIMATE.

Borehole No: 161 Sheet 1 of 5 Figure

BOREHOLE CORE LOG	LOGGED BY : SAM WALKER	DATE 22/3/91	
	CHECKED BY	DATE	Job No

Borehole dip (from horizontal) 90°	Borehole dip direction	Location AMG E N EL	Location Mine E N
Hole commenced:	Drilling rig	Drilling Fluid	
Hole completed	Mounted on	Barrel Type	

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
					COAL REMOVED PRIOR TO GEOTECHNICAL LOGGING.					
MUDSTONE, light grey, silty, MW strength, SW-MW				21						
				22						
				23						
				24						
COAL, black, silty, MW strength				25			61			
SILTSTONE, light grey, thinly bedded, MW strength, MW							63			
SANDSTONE, light grey, fine grained, massive				26						
				27						
SANDSTONE, light grey, coarse grained, lithic, clayey, matrix, MW strength, HW				28						
SIDERITIC SILTSTONE, dark grey, MS strength				29			68			
SILTSTONE, light grey, clayey, MW strength, SW				30			72 65			

HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

JOINT COAL BOARD

NOTE: The geologist on site during drilling verbally reported that core recovery was 100% for entire hole and there were very few natural defects. (RQD's generally very high)

Borehole No: 161 Sheet 2 of 5 Figure

BOREHOLE CORE LOG	LOGGED BY: SAM WALKER DATE 22/3/91	
	CHECKED BY DATE	Job No.

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: | Drill rig | Drilling Fluid
Hole completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
SILTSTONE, light grey, clayey, MW strength, SW				31						
				32			80, 68, 78, 72			
SIDERITIC SILTSTONE, dark grey, fractured, MS strength				33						
MUDSTONE, light grey, slightly carbonaceous, MW strength										
COAL, thinly interlaminated with MUDSTONE (40%)				34	34.01 JOINT, R1,P3, clean slickensided	30				NO B
CLAYSTONE, light grey, W strength SW-Fr					34.07 JOINT, R1,P3, clean	60				NO B
MUDSTONE, grey, silty, slightly carbonaceous, MW strength				35						
COAL, black, vitreous, MW strength		LLO			COAL REMOVED, 35.20m to 36.10m					
MUDSTONE, light grey, minor carbonaceous filaments, silty MWstrength				36						
				37	37.76 JOINT, R2,P1, clean	18	75			B=269
					37.90 JOINT, R1,P1, clean	15				B=269
				38	38.00 JOINT, R1,P1, clean	12				B=269
					38.48 JOINT, R1,P3, clean	55				NO B
SILTSTONE, light grey, MW strength, Fr				39						
MUDSTONE, light grey, MWstrength Fr					39.60 JOINT, R2,P2, clean	14	74			B=278
					39.80 JOINT, R2,P2, clean	48				B=93
				40						

SEE NOTE: Sheets 1 and 2

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 161 Sheet 3 of 5 Figure

BOREHOLE CORE LOG

LOGGED BY: SAM WALKER DATE: 22/3/91

CHECKED BY DATE

Job No.

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: | Drill rig | Drilling Fluid
Hole completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
MUDSTONE, light grey, MW strength, Fr				41			68 71			
SANDSTONE, Light grey, MW strength, Fr				42			65 70			
				43			77			
				44	44.50 JOINT, R5, P2, clean	5				NO B
				45			52			
				46			76			
SILTSTONE, light grey, MW strength Fr				47						
MUDSTONE, light grey, MW strength, Fr				48						
SILTSTONE, light grey, MW strength, Fr				49						
				50						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

SEE NOTE: Sheets 1 and 2

JOINT COAL BOARD

Borehole No: 161 Sheet 4 of 5 Figure

BOREHOLE CORE LOG	LOGGED BY : SAM WALKER DATE 22/3/91 CHECKED BY DATE	Job No

Borehole dip from horizontal : 90° — Borehole dip direction — Location AMG E N EL — Location Mine E N

Hole commenced: — Drill rig — Drilling Fluid
Hole completed — Mounted on — Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
SILTSTONE, light grey, MW strength Fr				51						
				52	CORES FROM 52.0 TO 53.8 HAVE BEEN REMOVED FOR COAL QUALITY TESTING PRIOR TO LOGGING.					
COAL: black, bright (80%), v. brittle, tr claystone - lt brn, med soft		BT		53						
SILTSTONE, light grey, MW Strength Fr				54						
				55						
				56						
	56.58 END OF BORE AT 56.58			57	END OF HOLE AT 56.50m DEPTH.					

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

SEE NOTE: sheets 1 and 2

JOINT COAL BOARD

Borehole No: 161 Sheet 5 of 5 Figure

BOREHOLE CORE LOG

LOGGED BY : SAM WALKER DATE 4/3/91

CHECKED BY DATE

Job No

Borehole dip (from horizontal) 90°
Borehole dip direction
Location AMG E N EL
Location Mine E N

Hole commenced:
Hole completed
Drill rig
Mounted on
Drilling Fluid
Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
SILT, light brown to off-white, clayey, stiff, ML				1	NO CORES UNTIL 4.12m BLADE BIT USED					
Sandier towards base				2						
SANDSTONE, off-white, fine grained, very low strength, EW-CW.				3						
				4						4.12
SANDSTONE, off-white to light tan, fine to medium grained, MW strength, HW				5	5.23 JOINT, R3,P1,clean 5.71 JOINT, R2,P2, clean, some Fe staining	22 23	67	75	100	
				6						6.61
strength from 7.32m to 7.61m depth				7			76			
				8				87	100	
SANDSTONE, light brown, coarse grained, MW strength.				9	9.76 to 9.78 CW zone, loose sand 9.89 to 9.91 CW zone, loose sand					9.61
				10						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 128 Sheet 1 of 4 Figure

BOREHOLE CORE LOG	LOGGED BY: SAM WALKER DATE 4/3/91 CHECKED BY DATE	Job No.

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: 4/3/91 | Drill rig | Drilling Fluid
Hole completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
SANDSTONE, light brown, lithic, clay matrix, MW strength, medium to coarse grained.							65			
							67			
SANDSTONE, as above, fine grained				11				95	100	
SANDSTONE, as above, medium grained				12	11.99 JOINT, R3,P2, clean	71	71			
					12.02 JOINT, R3,P2, clean 12.83-12.86 CW zone, sand 12.96-13.00 CW zone, sand	75				12.61
				13				83	100	
				14						14.11
SANDSTONE, as above, fine to medium grained. MW strength							75 78	100	100	Sample DM01 14.11 to 14.32m Sample DM02 15.72 to 15.99m
				15			75			15.61
				16				100	102	
SILTSTONE, Dark brown, sandy carbonaceous, W strength.										16.66
SANDSTONE, as above				17	17.34 JOINT, R3,P3, very soft clay infill, disturbed	75				
					17.43 JOINT, R3,P2,clean	85		90	93	
SANDSTONE, as above, coarse grained, MWstrength				18						18.61
SANDSTONE, light brownish grey, as above, medium grained							71			
				19	19.47 JOINT, R3,P2, clean 1mm width, Black and yellow, Fe staining	36		88	100	
SANDSTONE, as above, MSstrength				20						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 128 Sheet 2 of 4 Figure:

BOREHOLE CORE LOG

LOGGED BY: SAM WALKER DATE 5/3/91

CHECKED BY DATE

Job No

Borehole dip (from horizontal) 90° Borehole dip direction Location AMG E N EL Location Mine E N

Hole commenced: 4/3/91 Hole completed

Drill rig Mounted on

Drilling Fluid Barrel Type

DESCRIPTION LITHOLOGICAL	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
SANDSTONE, Light brownish grey, lithic, clay matrix, medium grained, MS strength	21			73	88	100	Sample DM03 20.63 to 20.89m
SILTSTONE, light grey, sideritic, MS strength		21.10 JOINT, R3,P2, clean sub parallel to bedding	60				
		21.60 JOINT, R3,P2, clean B=249	22	80			21.61
SANDSTONE, as above, MW strength	22			50			foresets
	23				80	97	
SANDSTONE, grey, fine grained, very clayey, MW strength. Cross bedding outlined by heavy minerals.		23.20 JOINT, R3,P3, clean	72	72 77			
	24			57			foresets
							24.61
SILTSTONE, light grey, coarse grained, sandy. MW strength.	25			68			
SANDSTONE, light brown, silty, medium to coarse grained, MS strength.	26			NO BEDDING MEASURABLE	93	707	
SANDSTONE, as above, MS strength	27	27.17 JOINT, R3,P1,clean B=277	24				27.51 Extremely low strength rock is almost impossible to sample. It breaks up in hands while wrapping, etc.
SANDSTONE, very light brown, silty, med grained MW strength, HW	28						
	29				95	100	
	30	29.70 JOINT, R3,P1, clean (no B angle available)					

HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 128 Sheet 3 of 4 Figure

OREHOLE CORE LOG	LOGGED BY : SAM WALKER DATE ·	
	CHECKED BY DATE	Job No .

orehole dip from horizontal : 90°	Borehole dip direction	Location AMG E N / Location Mine E N	EL
ole commenced: / ole completed	Drill rig / Mounted on	Drilling Fluid · / Barrel Type	

DESCRIPTION LITHOLOGICAL
SANDSTONE, as above.
CORE LOSS
SILTSTONE, grey, carbonaceous, L strength
MUDSTONE, light grey, very silty, carbonaceous, MW strength
COAL, black, vitreous, highly cleated, MW strength
MUDSTONE, grey, carbonaceous
COAL, as above
SILTSTONE, dark brown, clayey, MW strength
COAL, as above
MUDSTONE, light grey, silty, carbonaceous, MW to W strength, puggy

DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
						30.61
31			71	88	90	
32			71			
	32.88 to 33.29 FRACTURED ZONE, fractures 5 to 20m apart, sub-parallel to bedding					32.96
33			77			
34				0	104	
	34.86 JOINT, R1,P2, clean slickensided	40				
35						35.01
				0	99	
36						36.61
				50	104	
37						37.11
38				10	63	
39	39.25 JOINT, R1,P2 clean	54				
	39.60 JOINT, R2,P3, clean	68				
	39.61 END OF HOLE					
40						

MLO

Borehole abandoned at 39.61m depth, after collapse of coal seam, and loss of core-lifter casing in hole.

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 128 Sheet 4 of 4 Figure

BOREHOLE CORE LOG

LOGGED BY: SAM WALKER DATE: 24/3/91

CHECKED BY DATE

Job No.

Borehole dip (from horizontal) 90° Borehole dip direction

Location AMG E N EL

Location Mine E N

Hole commenced: Drill rig Drilling Fluid

Hole completed Mounted on Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
				31	NON-CORED TO 33.17m DEPTH					
				32						
MUDSTONE, light grey, silty to very silty, MW strength, abundant small carbonaceous fragments and minor thin coaly zones.				33			65	80	98	
				34						
COAL, black, vitreous, MW strength				35			68			
MUDSTONE, as above										
COAL: black, vitreous, MW strength, brittle, common fractures, occ. clay/silty bands				36	CORES FROM 35.35m TO 40.13m DEPTH HAD BEEN REMOVED FROM THE BOXES AND SENT FOR COAL QUALITY ANALYSIS.					
				37					97	
		MLD		38						
				39						
				40						

HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

NOTE 1: Cores had been highly disturbed due to boxing, drying and transport to store.

2: BH129 is a redrill of BH128 which was abandoned due to hole collapse.

JOINT COAL BOARD

Borehole No: 129 Sheet 1 of 3 Figure

BOREHOLE CORE LOG	LOGGED BY: SAM WALKER	DATE 24/3/91	
	CHECKED BY	DATE	Job No.

Borehole dip from horizontal: 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: | Drill rig | Drilling Fluid
Hole completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
		CORES REMOVED					
MUDSTONE, light grey, silty to very silty, MW strength, Fr	41						
		41.57 JOINT, R4, P3, poorly slickensided	72	75	>95	100	B=302
		41.84 JOINT, R4, P3, poorly slickensided	73				B=283
	42	41.86 JOINT, R4, P3, poorly slickensided	74				B=283
SANDSTONE, light grey, fine to medium grained, lithic, clay matrix, MW strength, Fr							42.51
		41.99 JOINT, R2, P4, slickensided	67				B=225
	43	42.12 JOINT, R3, P4, poorly slickensided	65				B=177
					>98	100	
SILTSTONE, light grey, clayey to very clayey, MW strength, Fr	44						
	45						
COAL, black, very silty, MS strength, Fr							45.51
SILTSTONE, as above							
SANDSTONE, light grey, fine grained, MW strength, Fr	46			65			
				65			
	47				>98	100	
SILTSTONE, light grey, MW strength, Fr		47.88 JOINT, R2, P3, clean	52				B=261
	48			70			
				75			48.51
	49				>95	100	
	50						

NOTE: Cores had been highly disturbed due to boxing, drying and transport to the store. This has introduced uncertainty in the estimates of both rock strength and RQD.

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 129 Sheet 2 of 3 Figure

BOREHOLE CORE LOG

LOGGED BY: SAM WALKER DATE: 24/3/91

CHECKED BY DATE

Job No.

Borehole dip (from horizontal): 90° Borehole dip direction Location AMG E N EL

Location Mine E N

Date commenced: Drill rig Drilling Fluid:

Date completed Mounted on Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
SILTSTONE, light grey, MW strength Fr								>95	100	
SILTSTONE, dark brownish, grey, highly caronaceous, (coaly in parts), MW strength, Fr				51						51.51
COAL SAMPLE OF LOWER LIDDELL SEAM REMOVED FOR QUALITY TESTING		LLD		52	CORES FROM 52.20m to 53.44m DEPTH HAD BEEN REMOVED FOR COAL QUALITY ANALYSIS					
				53	53.44-53.79 ZONE OF VERY CLOSELY SPACED POLISHED SURFACES PARALLEL TO BEDDING.		73 75		100	
"Laminate" MUDSTONE, highly carbonaceous, extremely thinly bedded, W strength										
SILTSTONE, light grey, MW strength, Fr				54						
SANDSTONE, light grey, very silty, fine to very fine grained, MW strength, Fr										54.51
SILTSTONE, light grey, MW strength, Fr				55						
SIDERITIC SILTSTONE, mid grey, MS strength, Fr					55.95 JOINT, R4, P2, very thin pyrite coating	11		>95	100	No B available
SILTSTONE, light grey, MW strength, Fr				56	56.10 JOINT, R5, P3, clean	15				No B available
				57	END OF HOLE AT 57.51m DEPTH					
57.51	END OF BORE AT 57.51									

NOTE: Cores had been highly disturbed due to boxing, drying and transport to the store. This has introduced uncertainty in strength estimates and RQD values.

HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 129 Sheet 3 of 3 Figure

OREHOLE CORE LOG	LOGGED BY : SAM WALKER DATE 24/03/91 CHECKED BY DATE	Job No .

ehole dip om horizontal 90° | Borehole dip direction - | Location AMG E N EL | Location Mine E N

e commenced: | Drill rig | Drilling Fluid
e completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
				11	NON-CORED TO 13.14m DEPTH					
				12						
				13						
STONE, light brown, medium ned, lithic, W strength, clay matrix. ighly weathered.				14						
				15						
				16						
				17						
SANDSTONE, as above, MW strength.				18						
MUDSTONE, Light brown to light rey, silty, MW strength. Highly weathered.				19	18.88 JOINT, R2,P3, slickensided.	66	69			B=322
					19.10 JOINT, R3,P3, clean	42	72			B=0
					19.24 JOINT, R3,P3, clean	55	65			B=300
					19.29 JOINT, R4,P3, clean	64	75			B=300
				20						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

NOTE: Cores had been transferred to core boxes and transported days prior to logging. Rock strengths consequently very difficult to estimate. RQD not estimable.

JOINT COAL BOARD

Borehole No: 176 Sheet 1 of 3 Figure

BOREHOLE CORE LOG

LOGGED BY : SAM WALKER DATE 24/03/91

CHECKED BY DATE

Job No

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: | Drill rig | Drilling Fluid
Hole completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
COAL, Black, vitreous, L strength, highly cleated and fractured.		MLD		21 22 23 24	CORE SAMPLES FROM 19.89m TO 25.75m DEPTH HAD BEEN REMOVED PRIOR TO LOGGING. (COAL SAMPLE)					
SILTSTONE, light grey, MW strength. Moderately weathered.				25 26 27					97	
MUDSTONE, Light grey, silty to very silty, MW strength. Moderately weathered.				28	28.48 JOINT, R2,P2, slickensided	45				B=47
					28.56 JOINT, R2,P3, slickensided	55			83	B=47
				29	28.59 JOINT, R2,P3, slickensided	60	72			B=187
					28.86 JOINT, R3,P4, clean	68	69			B=281
				30	28.89 JOINT, R2,P4, slickensided	64				B=263

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

SEE NOTE ON PREVIOUS SHEET

JOINT COAL BOARD

Borehole No: 176 Sheet 2 of 3 Figure

BOREHOLE CORE LOG	LOGGED BY : SAM WALKER DATE 24/03/91	
	CHECKED BY DATE	Job No

Borehole dip (from horizontal) 90° Borehole dip direction Location AMG E N EL
Location Mine E N

Hole commenced: Drill rig Drilling Fluid
Hole completed Mounted on Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
MUDSTONE, as above							66			
				31					118	
				32						
SILTSTONE, light grey, clayey, MW strength					32.35 JOINT, R2,P1, clean	25	75			B=216
				33	END OF HOLE AT 32.61m DEPTH					

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 176 Sheet 3 of 3 Figure

BOREHOLE CORE LOG	LOGGED BY : SAM WALKER	DATE 23/3/91	
	CHECKED BY	DATE	Job No

Bo hole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Ho commenced: | Drill rig | Drilling Fluid
Ho completed | Mounted on | Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
NO SAMPLE RECOVERY				1	NON-CORED TO 5.79m DEPTH					
AY				2						
				3						
				4						
NDSTONE, light brown, fine ained, MW strength, EW				5						
NDSTONE, Light brown, fine ained, MW strength, HW				6			68	98		6.61
SANDSTONE is inly interbedded with MUDSTONE										
NDSTONE, light brown, very fine to fine grained, MW strength,				7			60			
IDERITIC SILTSTONE, brown, strength										
SANDSTONE, light brown, fine ained, MW strength, HW				8	8.30 JOINT,R3,P3, clean	2	68		95	B=90
UDSTONE, light grey and light rown, silty, MW strength, HW				9						9.61
				10			57			

NOTE: Cores were logged some days after drilling. Cores had been boxed and transported to store. Significant drying and disturbance had occurred.

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 160 Sheet 1 of 6 Figure

BOREHOLE CORE LOG	LOGGED BY : SAM WALKER	DATE 23/3/91	
	CHECKED BY	DATE	Job No

Borehole dip 90° (from horizontal)	Borehole dip direction	Location AMG E N EL	Location Mine E N
Hole commenced:	Drill rig	Drilling Fluid	
Hole completed	Mounted on	Barrel Type	

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	F C A	B C A	R Q D	RECOVERY	COMMENTS
MUDSTONE, light grey and light brown, silty, MW strength, HW							65 60			
				11	11.13 JOINT,R1,P1,clean	15			101	B=0
SANDSTONE, light brown, medium grained, sideritic, HW							70			
MUDSTONE, light brown, W strength, MW				12						
SANDSTONE, light grey, medium to coarse grained, MW strength, MW					12.40 JOINT,R2,P2,clean no B available	5				
					12.73 JOINT, R2,P1,clean	25				B=272
MUDSTONE, grey, MW strength MW				13	13.20 JOINT, R4,P2,clean 2mm calcite infill	20				B=270
									96	
							60 65			
SANDSTONE, light grey, fine grained,MW strength, MW				14						
MUDSTONE, slight grey, silty, minor carbonaceous zones and small coal inclusions,MW strength, SW-MW					14.56 JOINT, R3,P1,clean	40				B=215
					14.88 JOINT,R3,P2 clean					
				15						
							60			
				16	16.33 JOINT,R5,P2,clean	18			102	
					16.95 JOINT, R3,P1,clean	38				
				17						
				18						
							70 75		100	
				19						
SANDSTONE, grey,medium to coarse grained, lithic, L-M strength, SW-Fr							59 66			
				20						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

NOTE: Cores logged some days after drilling. Cores had been boxed and transported to store causing significant drying and disturbance. Consequently rock strength and RQD's are very difficult to estimate.

JOINT COAL BOARD

Borehole No: 160 Sheet 2 of 6 Figure

BOREHOLE CORE LOG	LOGGED BY: SAM WALKER	DATE: 23/3/91	
	CHECKED BY	DATE	Job No.

rehole dip (from horizontal) 90°	Borehole dip direction	Location AMG E N EL Location Mine E N
le commenced: le completed	Drill rig Mounted on	Drilling Fluid: Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
SANDSTONE, grey, medium to coarse grained, lithic, MW strength, SW-Fr					20.05 JOINT,R2,P4, clean 20.25 JOINT,R4,P2, clean	19 21	60			B=192 B=184
							61			20.98
SANDSTONE, grey, fine to medium grained, lithic, MW strength, finely laminated (2mm to 50mm)				21	21.60 JOINT,R2,P1,clean 21.66 JOINT,R2,P1,clean 21.73 JOINT,R2,P1,clean	25 31 22				B=178 B=178 B=178
SANDSTONE, light grey, coarse to very coarse grained, lithic, MS strength, Fr-SW				22			66		100	B=224
				23	22.98 JOINT,R4,P2,clean	23				
				24			60			24.07
				25	24.85 JOINT, R5,P1 crystalline calcite infill 2mm thick	20				B=164
							74		100	
				26						
				27			68 61			27.07
				28					102	
				29	29.38 JOINT,R4,P2,clean	21				B=182
SANDSTONE, see below				30						

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

NOTE: Cores logged some days after drilling. Cores had been boxed and transported to store. Significant drying and disturbance had occurred. Consequently rock strengths and RQD's are very difficult to estimate.

JOINT COAL BOARD

Borehole No: 160 Sheet 3 of 6 Figure:

[B]OREHOLE CORE LOG

LOGGED BY: SAM WALKER DATE 23/3/91

CHECKED BY DATE

Job No.

[Bo]rehole dip [fr]om horizontal) 90° Borehole dip direction Location AMG E N EL Location Mine E N

[Da]te commenced: [Da]te completed

Drill rig Mounted on

Drilling Fluid: Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
										30.07
[S]ANDSTONE, light grey, very coarse [t]o coarse grained, massive, strength, FR										
				31			67			B=228
					31.04 JOINT,R3,P1, clean	35				
					31.09 JOINT, R3,P1, clean	35				B=228
					31.22 JOINT,R3,P1, clean	28		100	100	B=224
				32						
				33						33.14
				34				100	100	
				35						
				36						36.14
				37				100	100	
				38						
				39						39.14
				40						

HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

NOTE: Cores were logged some days after drilling. Cores had been boxed and transported ot store causing significant drying and disturbance. Consequently rock strengths are difficult to estimate.

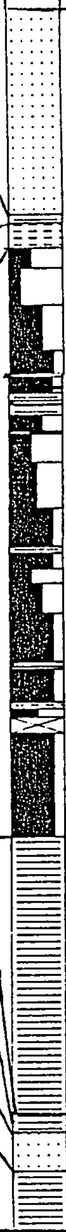
JOINT COAL BOARD

Borehole No: 160 Sheet 4 of 6 Figure

BOREHOLE CORE LOG	LOGGED BY : SAM WALKER DATE 23/3/91 CHECKED BY DATE	Job No

Borehole dip (from horizontal) 90° | Borehole dip direction | Location AMG E N EL | Location Mine E N

Hole commenced: | Drill rig | Drilling Fluid
Hole completed | Mounted on | Barrel Type



DESCRIPTION LITHOLOGICAL	CONSTRUCTION LOG	DEPTH	DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
SANDSTONE, light grey, very coarse to coarse grained, massive, MS strength, FR		41			58			
MUDSTONE, brown, sideritic, MS strength, Fr								
CLAYSTONE, dark grey, carbonaceous, MW strength, Fr		42						
COAL, black, vitreous, highly cleated, MS strength, Fr	MLD	43–46	CORES FROM 41.71m TO 46.85m DEPTH HAVE BEEN REMOVED FOR COAL QUALITY ANALYSIS PRIOR TO GEOTECHNICAL LOGGING.					
MUDSTONE, grey, MW strength, Fr		47				100	97	
SILTSTONE, light grey, MW strength Fr		48						
SIDERITIC SILTSTONE, dark brown, strength, Fr (Fractured)								
SANDSTONE, light grey, clayey, finely grained, MW strength, Fr		49	49.24 JOINT, R2, P2, clean	61		100	100	B=10
MUDSTONE, light grey, silty, MW strength, Fr		50			70			

HUNTER VALLEY COAL CORP
MT. OWEN PROJECT

NOTE: Cores were logged some days after drilling. Cores had been boxed and transported to store causing significant drying and disturbance. Consequently rock strengths are difficult to estimate.

JOINT COAL BOARD

Borehole No: 160 Sheet 5 of 6 Figure

BOREHOLE CORE LOG

LOGGED BY: SAM WALKER DATE: 23/3/91

CHECKED BY DATE

Job No.

Borehole dip (from horizontal) 90º

Borehole dip direction

Location AMG E N EL

Location Mine E N

Hole commenced:

Hole completed

Drill rig

Mounted on

Drilling Fluid

Barrel Type

DESCRIPTION LITHOLOGICAL	LITHOLOGICAL LOG	CONSTRUCTION LOG	STRUCTURE LOG	DEPTH	DESCRIPTION STRUCTURAL	FCA	BCA	RQD	RECOVERY	COMMENTS
MUDSTONE, light grey, silty, MW strength, Fr							69		100	
COAL, black, MS strength										
SANDSTONE, grey, medium to coarse grained, lithic, MS strength 51.14				51	END OF HOLE AT 51.14m DEPTH		68			
END OF BORE AT 51.14										
				52						

HUNTER VALLEY COAL CORP

MT. OWEN PROJECT

JOINT COAL BOARD

Borehole No: 160 Sheet 6 of 6 Figure

APPENDIX B

PIEZOMETERS AND FIELD PERMEABILITY TESTS

1.0 DETAILS OF PIEZOMETERS

Nine of the boreholes were instrumented with standpipe piezometers by officers of the Joint Coal Board. The completion of each of these is shown individually on Figure B1. At the completion of drilling 50mm class 18 PVC casing was installed in the hole. The bottom section had been slotted to provide hydraulic connection with the test interval. Gravel pack was inserted. Each hole was fully sealed from above the gravel pack to the surface using a cement grout.

2.0 FIELD PERMEABILITY TESTS

Rising head tests were performed by Joint Coal Board personnel on each of the instrumented boreholes. The following procedure was used:

i) Depth to standing water level was measured.

ii) 19mm diameter airline was inserted so the outlet was about 1m from the base of the casing.

iii) Compressed air was activated to blow as much as possible of the standing water column out of the casing.

iv) Recovery of the water level in the casing was measured with time.

Raw results of the monitoring of these recoveries are included in this Appendix. The data for the test on Borehole 171 is plotted, as an example, on Figure B2.

2.1 Results of Permeability Tests

The data from the rising head tests were analysed by two different methods. The first method is outlined in USBM, 1977, pp. 94-98. The second method was the Modified Non-steady Flow equation.

2.1.1 Analysis by USBM Method

This method involves plotting the head deficit at time t divided by the initial head deficit at time t = 0 (ie H/Ho) on a log scale versus time, and a straight line fitted. The basic time lag (T) is scaled off at H/Ho = 0.37 (correcting for the straight line not passing through the origin). A shape factor (F) is calculated from the physical dimensions of the hole, the casing and the slotted section.

The average permeability (k) of the test section is calculated using the formula.

$$k = \frac{A}{FT}$$

where A = cross-sectional area of stand-pipe

F = shape factor

T = basic time lag.

The resulting **permeabilities** are shown in Table B1.

2.1.2 Modified Non-Steady Flow Method

For analysis of the data by this method drawdown (in metres) is plotted versus time (on a log scale). The change in drawdown (Δs) is measured over one log cycle of time for a straight line fitted to the early time data. The flow rate (Q) during this time is estimated by multiplying the cross sectional area of the inside of the standpipe by the change in drawdown between two representative readings. The average transmissivity (T) of the test section is then calculated using the Modified Non-Steady Flow equation.

$$T = \frac{2.3\,Q}{4\pi\ \Delta s}$$

The resulting **transmissivities** are shown in Table B1.

2.1.3 Interpretation of Results

Most of the test sections spanned multiple aquifers. Permeabilities and transmissivities for individual seams and rock units have been interpreted from these values and are summarized in Table B2. These values must be considered to be approximate only and to be subject to confirmation by more rigorous methods of aquifer testing.

TABLE B1

Apparent Hydraulic Properties For Borehole Test Intervals

Borehole Number	Apparent **permeability** from USBM method m/day	Apparent **transmissivity** from Modified Non-steady Flow method m^2/day
129	0.35	0.31
170	0.23	0.40
171	0.036	0.51
172	0.10	1.6
173	0.015	0.11
174	0.12	1.7
175	0.19	1.3
178	0.085	0.18
179	0.16	0.10

TABLE B2

Interpreted Aquifer Hydraulic Properties

Stratigraphic Interval/seam	Estimated Permeability (k) [m/day]	Average Thickness (b) [m]	Estimated Transmissivity (T) [m^2/day]
Overburden of MLD seam	0.02	20	0.4
Middle Liddell coal seam	0.3	5	1.5
Interburden from MLD to LLD	0.05	14	0.7
Lower Liddell coal seam	0.3	1.5	0.45
Interburden from MLD to Barrett	0.2	15	3.0
Barrett coal seam	1.0	3	3.0

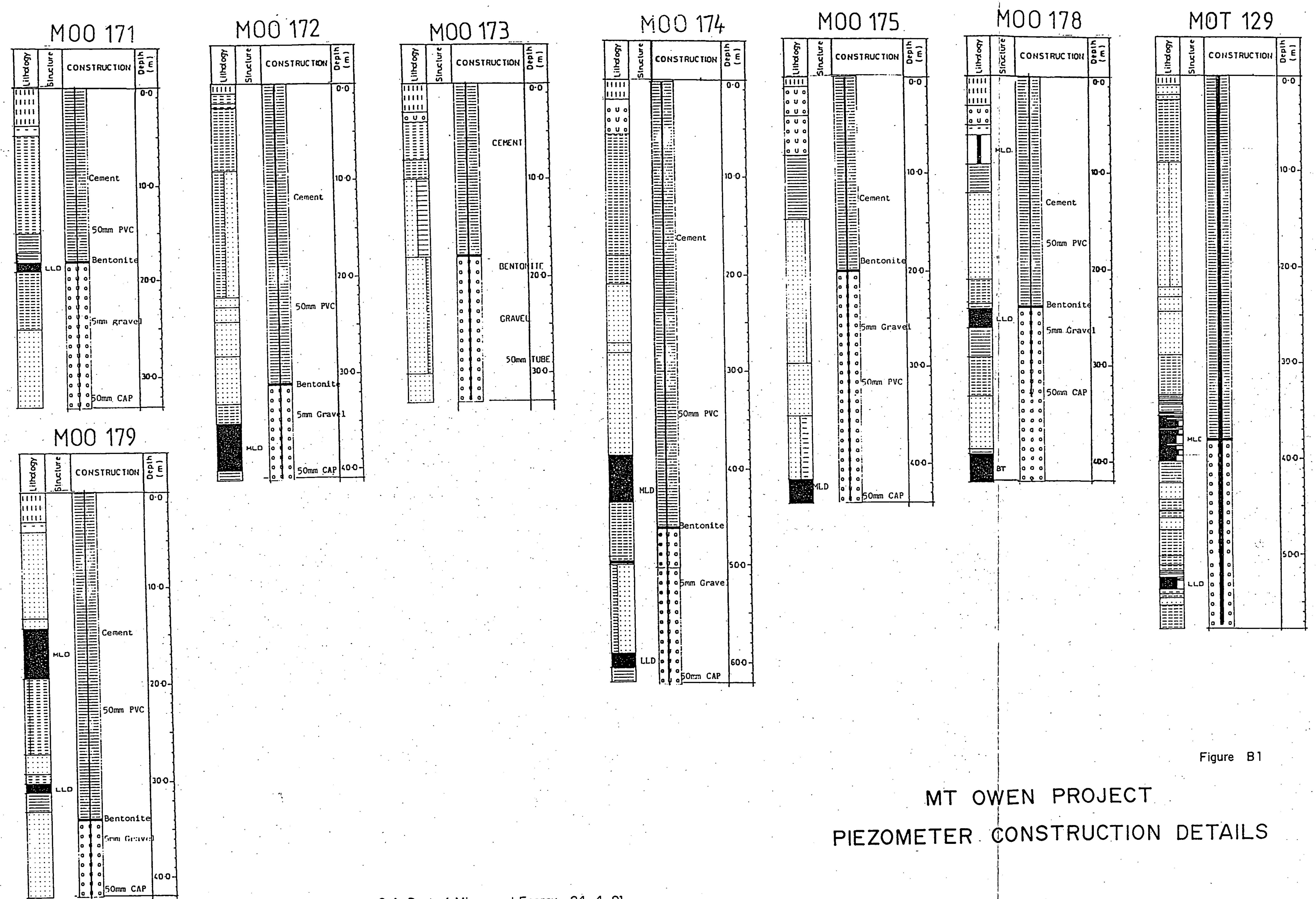


Figure B1

MT OWEN PROJECT
PIEZOMETER CONSTRUCTION DETAILS

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91-341

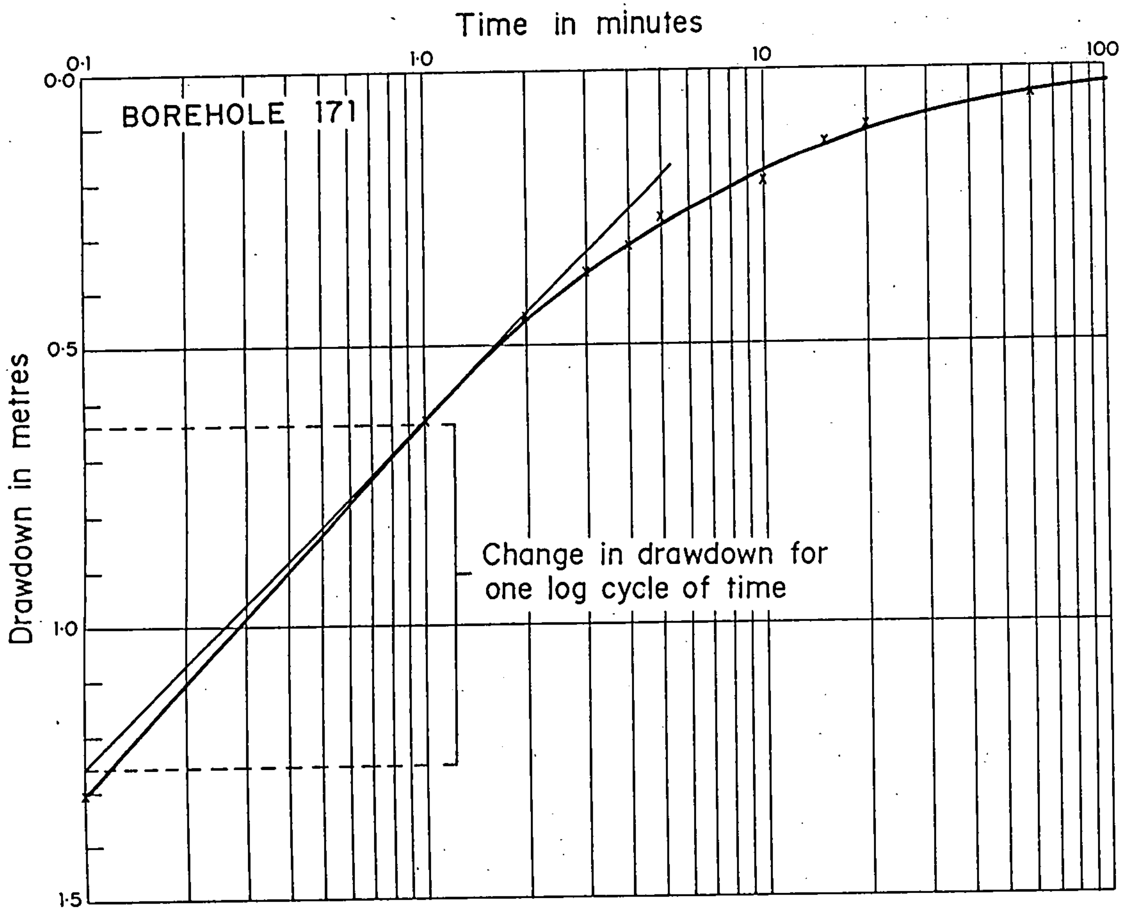


Fig. B2.a : ANALYSIS BY MODIFIED NON-STEADY FLOW METHOD

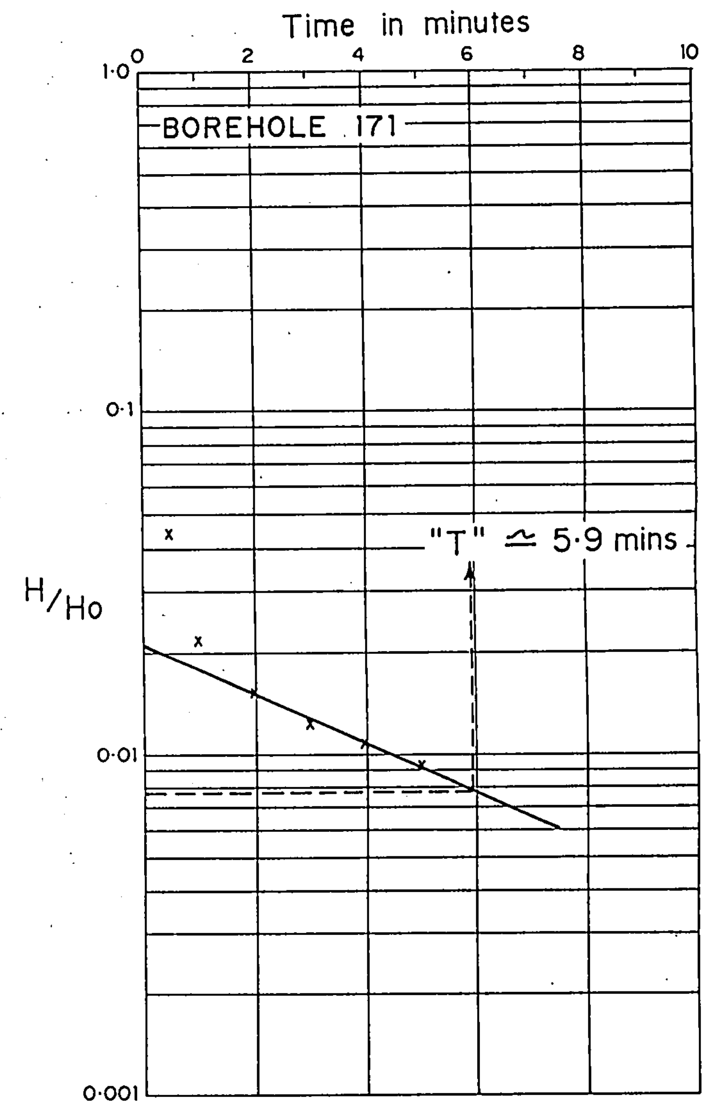


Fig. B2.b. : ANALYSIS BY METHOD OF USBM (1977)

MT OWEN PROJECT

TABULATED DATA FROM RISING HEAD PERMEABILITY TESTS

TABLE OF STATIC WATER LEVELS

BH No	SWL
129	5.98
170	2.91
171	3.27
172	6.17
173	2.94
174	6.81
175	7.09
178	9.90
179	6.86

NOTE: WL of Start (initial water level) is measured from the top of the piezometer standpipe. Static Water Level (SWL), measured from ground level (ie collar EL) is derived by subtracting the difference between top of casing and ground level.

BOREHOLE NUMBER	:	129
STANDPIPE STICKUP	:	1.00
WL OF START	:	6.98
AIRLIFT LEVEL	:	58.00
DATE	:	27/3/91
TIME (start)	:	10:50am
TIME (finish)	:	11:50am

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	58.00	51.02	1.0000
0.50	18.87	11.89	0.2330
1.00	11.18	4.20	0.0823
2.00	7.81	0.83	0.0163
3.00	7.39	0.41	0.0080
4.00	7.28	0.30	0.0059
5.00	7.22	0.24	0.0047
10.00	7.17	0.19	0.0037
15.00	7.14	0.16	0.0031
20.00	7.10	0.12	0.0024
25.00	7.08	0.10	0.0020
30.00	7.05	0.07	0.0014
40.00	7.03	0.05	0.0010
60.00	7.01	0.03	0.0006

PERMEABILITY TEST- RAWDATA

BOREHOLE NUMBER	:	170
STANDPIPE STICKUP	:	1.00
WL OF START	:	3.91
AIRLIFT LEVEL	:	43.00
		4.00
DATE	:	27/3/91
TIME (start)	:	9:00am
TIME (finish)	:	11:00am

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	43.00	39.09	1.0000
0.50	8.94	5.03	0.1287
1.00	6.15	2.24	0.0573
2.00	5.32	1.41	0.0361
3.00	4.58	0.67	0.0171
4.00	4.14	0.23	0.0059
5.00	4.10	0.19	0.0049
10.00	4.01	0.10	0.0026
15.00	4.00	0.09	0.0023
20.00	3.97	0.06	0.0015
30.00	3.97	0.06	0.0015
40.00	3.96	0.05	0.0013
60.00	3.95	0.04	0.0010
120.00	3.95	0.04	0.0010

PERMEABILITY TEST -RAW DATA

BOREHOLE NUMBER	:	171
STANDPIPE STICKUP	:	1.00
WL OF START	:	4.27
AIRLIFT LEVEL	:	34.00
DATE	:	27/03/91
TIME (start)	:	9:30am
TIME (finish)	:	11:30am

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	34.00	29.73	1.0000
0.50	5.58	1.31	0.0441
1.00	4.91	0.64	0.0215
2.00	4.72	0.45	0.0151
3.00	4.64	0.37	0.0124
4.00	4.59	0.32	0.0108
5.00	4.54	0.27	0.0091
10.00	4.47	0.20	0.0067
15.00	4.40	0.13	0.0044
20.00	4.37	0.10	0.0034
25.00	4.35	0.08	0.0027
35.00	4.34	0.07	0.0024
60.00	4.32	0.05	0.0017
120.00	4.29	0.02	0.0007

PERMEABILITY TEST-RAW DATA

BOREHOLE NUMBER	:	172
STANDPIPE STICKUP	:	1.00
WL OF START	:	7.17
AIRLIFT LEVEL	:	42.00
DATE	:	27/03/91
TIME (start)	:	10:25am
TIME (finish)	:	11:25am

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	42.00	34.83	1.0000
0.50	18.38	11.21	0.3218
1.00	10.01	2.84	0.0815
2.00	9.44	2.27	0.0652
3.00	8.86	1.69	0.0485
4.00	7.73	0.56	0.0161
5.00	7.65	0.48	0.0138
10.00	7.47	0.30	0.0086
15.00	7.37	0.20	0.0057
20.00	7.30	0.13	0.0037
30.00	7.24	0.07	0.0020
40.00	7.21	0.04	0.0011
60.00	7.19	0.02	0.0006

PERMEABILITY TEST - RAW DATA

BOREHOLE NUMBER	:	173
STANDPIPE STICKUP	:	1.00
WL OF START	:	3.94
AIRLIFT LEVEL	:	34.00
DATE	:	27/3/91
TIME (start)	:	11:30am
TIME (finish)	:	1:30pm

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	34.00	30.06	1.0000
0.50	31.33	27.39	0.9112
1.00	30.81	26.87	0.8939
2.00	28.55	24.61	0.8187
3.00	26.96	23.02	0.7658
4.00	25.47	21.53	0.7162
5.00	24.01	20.07	0.6677
10.00	18.04	14.10	0.4691
15.00	13.05	9.11	0.3031
20.00	12.72	8.78	0.2921
25.00	12.26	8.32	0.2768
30.00	11.96	8.02	0.2668
60.00	11.52	7.58	0.2522
120.00	11.07	7.13	0.2372

PERMEABILITY TEST-RAW DATA

BOREHOLE NUMBER	:	174
STANDPIPE STICKUP	:	0.62
WL OF START	:	7.43
AIRLIFT LEVEL	:	42.00
DATE	:	27/03/91
TIME (start)	:	12:00 noo
TIME (finish)	:	1:00pm

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	42.00	34.57	1.0000
0.50	15.09	7.66	0.2216
1.00	8.68	1.25	0.0362
2.00	8.11	0.68	0.0197
3.00	7.86	0.43	0.0124
4.00	7.81	0.38	0.0110
5.00	7.72	0.29	0.0084
10.00	7.64	0.21	0.0061
15.00	7.60	0.17	0.0049
20.00	7.57	0.14	0.0040
30.00	7.55	0.12	0.0035
40.00	7.54	0.11	0.0032
60.00	7.54	0.11	0.0032

PERMEABILITY TEST-RAW DATA

BOREHOLE NUMBER :	175
STANDPIPE STICKUP :	1.10
WL OF START :	8.19
AIRLIFT LEVEL :	62.00
DATE :	27/03/91
TIME (start) :	12:30pm
TIME (finish) :	1:30pm

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	62.00	53.81	1.0000
0.50	19.20	11.01	0.2046
1.00	14.32	6.13	0.1139
2.00	10.02	1.83	0.0340
3.00	8.31	0.12	0.0022
4.00	8.28	0.09	0.0017
5.00	8.27	0.08	0.0015
10.00	8.26	0.07	0.0013
15.00	8.25	0.06	0.0011
20.00	8.25	0.06	0.0011
30.00	8.24	0.05	0.0009
60.00	8.24	0.05	0.0009

PERMEABILITY TEST-RAW DATA

BOREHOLE NUMBER	:	178
STANDPIPE STICKUP	:	1.00
WL OF START	:	10.90
AIRLIFT LEVEL	:	43.00
DATE	:	27/03/91
TIME (start)	:	8:00am
TIME (finish)	:	10:00am

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	43.00	32.10	1.0000
0.50	13.10	2.20	0.0685
1.00	12.03	1.13	0.0352
2.00	11.68	0.78	0.0243
3.00	11.52	0.62	0.0193
4.00	11.39	0.49	0.0153
5.00	11.30	0.40	0.0125
10.00	11.26	0.36	0.0112
15.00	11.22	0.32	0.0100
20.00	11.20	0.30	0.0093
30.00	11.18	0.28	0.0087
40.00	11.16	0.26	0.0081
60.00	11.10	0.20	0.0062
120.00	11.07	0.17	0.0053

PERMEABILITY TEST-RAW DATA

BOREHOLE NUMBER	:	179
STANDPIPE STICKUP	:	1.00
WL OF START	:	7.86
AIRLIFT LEVEL	:	43.00
DATE	:	27/03/91
TIME (start)	:	1:30pm
TIME (finish)	:	2:30pm

TIME	DEPTH TO WATER LEVEL	EXCESS DRAW-DOWN	H/Ho
0.00	42.00	34.14	1.0000
0.50	15.01	7.15	0.2094
1.00	13.48	5.62	0.1646
2.00	11.89	4.03	0.1180
3.00	10.40	2.54	0.0744
4.00	9.62	1.76	0.0516
5.00	8.97	1.11	0.0325
10.00	8.51	0.65	0.0190
15.00	8.38	0.52	0.0152
20.00	8.29	0.43	0.0126
25.00	8.26	0.40	0.0117
30.00	8.23	0.37	0.0108
40.00	8.21	0.35	0.0103
60.00	8.18	0.32	0.0094

PERMEABILITY TEST-RAW DATA

APPENDIX C

POINT LOAD STRENGTH TESTS AND
SONIC LOG STRENGTH CORRELATION

1.0 POINT LOAD STRENGTH TESTS

Representative samples of the rock materials from boreholes 128, 161, 176 and 206 were tested using a Point Load Test frame, belonging to the Joint Coal Board. A total of 35 tests were performed on 14 samples according to the ISRM, Suggested Method for Determining Point Load Strength (ISRM, 1985). An explanatory sheet is included with this Appendix.

Point Load Tensile strength is obtained by loading core samples either diametrically or axially between two steel points. The average point load tensile strength across the plane of loading is then given by the load at failure divided by the square of the diameter of the sample. This tensile strength is then corrected to equivalent tensile strength measured over 50mm diameter core ($Is_{(50)}$).

The ten (10) test sheets which follow show details of each of the tests performed. Details of calibration of test frame are also included.

Table C1 summarizes the results of the tests.

2.0 CORRELATION OF SONIC VELOCITY WITH ROCK STRENGTH

Downhole sonic velocity logging was performed in each of the cored holes. The velocity of sound in rock is dependent on many and varied characteristics which include:

- rock substance strength
- rock type (mineralogy/texture/etc)
- in-situ stress state
- moisture content
- weathering
- degree of fracturing etc.

A general and approximate relationship between sonic log transit time (t - in microseconds per foot) and Unconfined Compressive Strength (UCS - in MPa) is given by McNally, 1987, p55 as:

$$UCS = 1000e^{(-0.035\,t)} \quad (MPa)$$

This relationship has been plotted relative to strength data measured during the present investigation on Figure C1.

For a number of reasons great care should be taken if any attempt is made to use this relationship to interpret rock strengths from downhole sonic logs:

- there are only a small number of strength values measured.

- the strength values show a wide scatter when plotted against transit time.

- the relationship between Point Load Strength ($Is_{(50)}$) and Unconfined Compressive Strength (q_u) has not been established for this site.

- the "noisy" nature of the sonic logs makes it difficult to relate reliable values of transit time to corresponding rock strength measurements.

A useful relationship will probably be able to be developed in the future, provided the following considerations are taken into account during further work. A significant number of additional strength tests should be done and should include enough Unconfined Compressive Strength (UCS) tests to establish the relationship to Point Load Tests. Sufficient strength tests of both types should be carried out for each of the major rock types on the site to allow reliable individual correlations to be calculated.

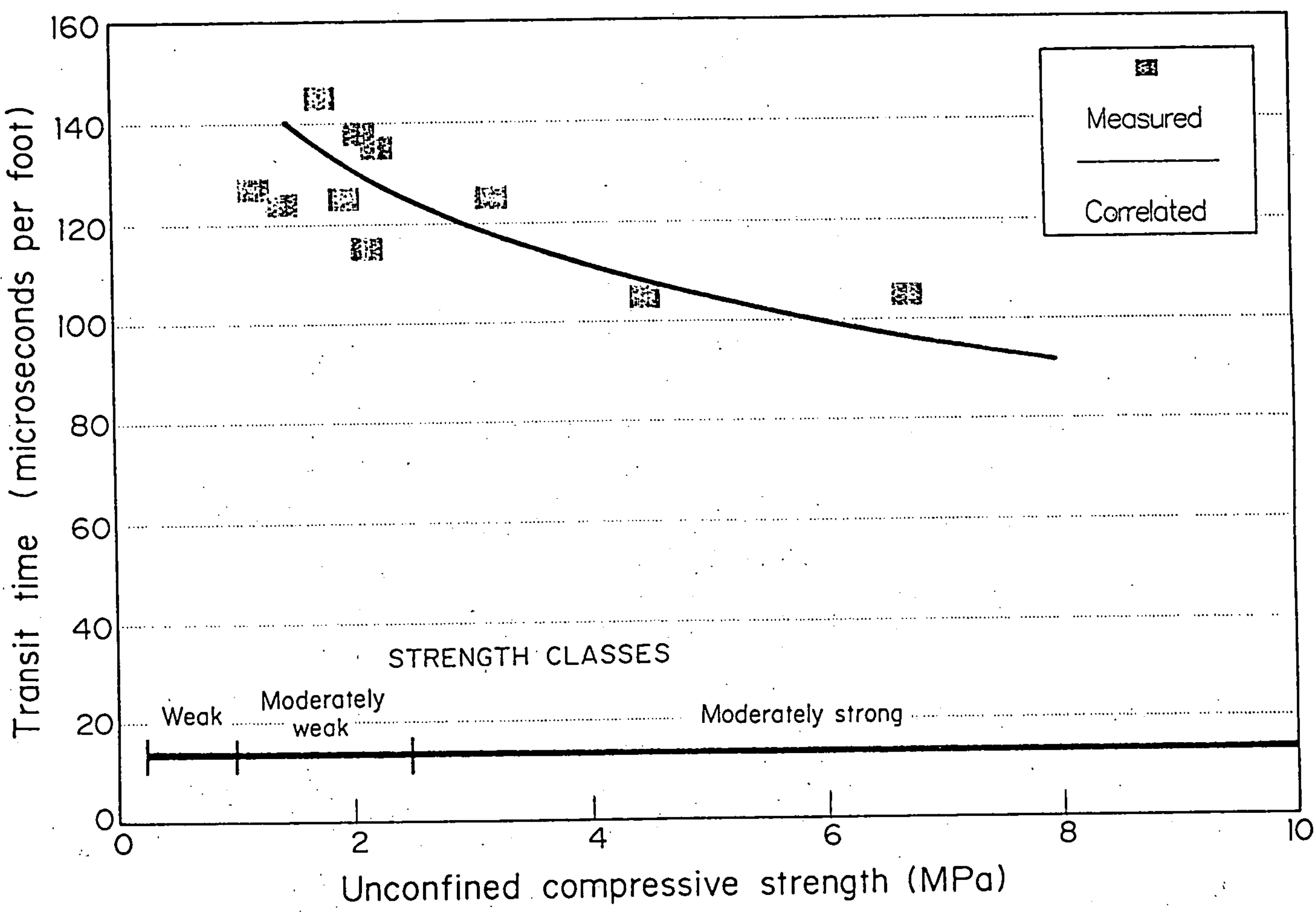


Figure C1

MT OWEN PROJECT

CORRELATION OF UCS WITH TRANSIT TIME

S.A. Dept of Mines and Energy 24-4-91

S22189

TABLE C1

Summary of Point Load Results

Hole Number	Sample Depth (m)	Rock Type	Interval[1]	Test[2] Type (A/D)	$Is_{(50)}$ MPa	Sample Condition
128	29.1	sandstone	M	A	0.32	No drying.
	29.1	sandstone	M	D	0.19	No drying.
	30.2	sandstone	M	D	0.27	Not wrapped 24 hours.
	30.2	sandstone	M	A	0.34	Not wrapped 24 hours.
	30.2	sandstone	M	D	0.24	Not wrapped 24 hours.
	30.2	sandstone	M	A	0.35	Not wrapped 24 hours.
	39.2	mudstone	L	D	0.29	No drying.
	39.2	mudstone	L	D	0.33	No drying.
	39.25	mudstone	L	A	0.43	No drying.
	39.3	mudstone	L	A	0.45	No drying.
161	47.2	siltstone	B	D	0.47	No drying, 1 week wrapped.
	47.2	siltstone	B	D	0.38	No drying, 1 week wrapped.
	47.2	siltstone	B	A	0.36	No drying, 1 week wrapped.
	47.2	siltstone	B	A	0.42	No drying, 1 week wrapped.
176	14.3	sandstone	M	D	0.50	No drying, 1 week wrapped.
	14.3	sandstone	M	D	0.59	No drying, 1 week wrapped.
	14.3	sandstone	M	A	0.57	No drying, 1 week wrapped.
	15.7	sandstone	M	D	0.45	No drying, 1 week wrapped.
	16.5	sandstone	M	D	0.99	One week drying in box.[3]
	16.5	sandstone	M	A	0.89	One week drying in box.[3]
	26.8	mudstone	L	D	0.45	No drying, 1 week wrapped.
	26.8	mudstone	L	A	0.42	No drying, 1 week wrapped.
	31.8	mudstone	L	D	0.61	No drying, 1 week wrapped.
	31.8	mudstone	L	A	0.97	No drying, 1 week wrapped.
206	33.5	mudstone	L	D	0.45	No drying, 3 days wrapped.
	33.5	mudstone	L	A	0.48	No drying, 3 days wrapped.
	32.83	siltstone	L	A	1.01	3 days drying in box.[3]
	38.2	mudstone	L	D	0.70	No drying, 3 days wrapped.
	38.2	mudstone	L	A	0.64	No drying, 3 days wrapped.
	40.2	siltstone	L	A	1.23	No drying, 3 days wrapped.
	40.2	sandstone	L	D	1.56	No drying, 3 days wrapped.
	40.2	sandstone	L	D	>1.9	No drying, 3 days wrapped.
	40.2	sandstone	L	D	1.38	No drying, 3 days wrapped.
	45.4	sandstone	L	D	0.90	No drying, 3 days wrapped.
	45.4	sandstone	L	A	0.98	No drying, 3 days wrapped.

NOTES:
1. Stratigraphic interval the sample comes from
   M = interburden between MLD and AUL
   L = interburden between LLD and MLD
   B = interburden between BT and LLD
2. Type of test. A = axially loaded, D = diametrally loaded
3. Samples with significant history of moisture loss showed marked apparent increase in strength

POINT LOAD STRENGTH TEST

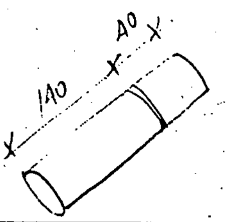
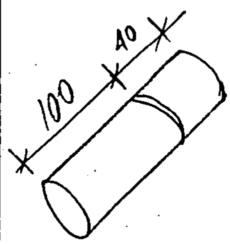
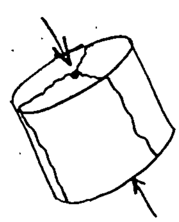
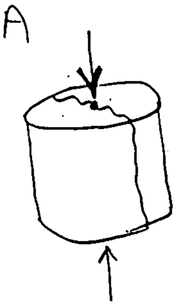
JOB NO.: DME 7/91
SHEET: 1 of 10

DEPARTMENT OF MINES AND ENERGY - SOUTH AUSTRALIA

CLIENT: HVCC | TEST LOCALITY: DRILL SITE | DATE: 6/3/91

PROJECT: MT OWEN COAL | TEST MACHINE: CSIRO MODEL P39 | TESTED BY: SW

LOCATION: BH 128 | DATE CALIBRATED: 15/4/91 | CHECKED BY:

SAMPLE ID and LOCATION		BH 128 39.2m	BH128 39.2m	BH 128 39.25m	BH 128 39.3m
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		MUDSTONE, light grey very silty; sl. carbonaceous	see left	see left	see left
MOISTURE / STORAGE HISTORY		FRESH FROM CORE BARREL	FRESH FROM CORE BARREL	FRESH FROM CORE BARREL	FRESH FROM CORE BARREL
DIAMETER 1	W_1 (mm)	58.0	58.0	58.0	58.0
DIAMETER 2	W_2 (mm)	58.0	58.0	58.0	58.0
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	58	58	58	58
PLATEN SEPARATION	D(mm)	58	58	37	28
LENGTH / DIAM. RATIO $0.3 < D/W < 1.0$	D/W	1.0	1.0	0.64	0.48
EQUIVALENT DIAM. $De = \sqrt{1.2732(W \times D)}$	De (mm)	58.0	58.0	52.3	45.5
FAILURE LOAD (or GAUGE PRESSURE)	kN(kPa)	1.0	1.2	1.4	1.1
CORRECTED LOAD	P(kN)	0.91	1.03	1.15	0.97
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/De^2) \times 1000$	I_s (MPa)	0.27	0.31	0.42	0.47
SIZE CORR. FACTOR $F = (De/50)^{0.45}$	F	1.07	1.07	1.02	0.96
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	0.29	0.33	0.43	0.45
APPROX. EQUIV. U.C.S. $Qu \triangleq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	7.0	7.9	10.3	10.8
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		D Perpendicular, clean fracture 	D 	A  Fractured into 3 pieces	A 

NOTES:
1. Testing in accordance with ISRM Point Load Test Method - see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

POINT LOAD STRENGTH TEST

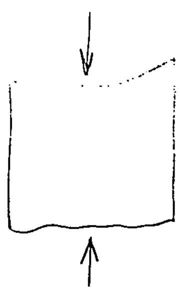
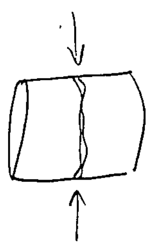
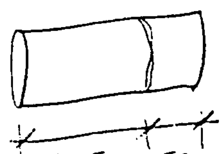
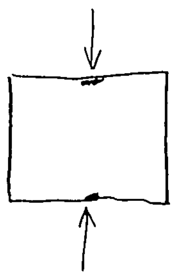
JOB NO.: DME 7/91
SHEET: 2 of 10

DEPARTMENT OF MINES AND ENERGY - SOUTH AUSTRALIA

CLIENT: HVCC | TEST LOCALITY: AT DRILL SITE | DATE: 6/3/91

PROJECT: MT OWEN COAL | TEST MACHINE: CSIRO MODEL P39 | TESTED BY: SW

LOCATION: BH128 | DATE CALIBRATED: 15/4/91 | CHECKED BY:

SAMPLE ID and LOCATION		BH 128 29.1 m	BH 128 29.1 m	BH128 30.2 m	BH128 30.2 m
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		SANDSTONE, lt. brownish grey, clayey, lithic, medium grained, HW.	see left	see left	see left
MOISTURE / STORAGE HISTORY		PLASTIC WRAPPED FOR 24 HRS	PLASTIC WRAPPED FOR 24 HRS	IN CORE BOX ONE DAY - NOT WRAPPED	IN CORE BOX ONE DAY - NOT WRAPPED
DIAMETER 1	W_1 (mm)	59.0	59.0	59.0	59.0
DIAMETER 2	W_2 (mm)	59.0	59.0	59.0	59.0
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	59.0	59.0	59.0	59.0
PLATEN SEPARATION	D(mm)	42	59	59.0	44
LENGTH / DIAM. RATIO $0{\cdot}3 < D/W < 1{\cdot}0$	D/W	0.71	1.0	1.0	0.75
EQUIVALENT DIAM. $D_e = \sqrt{1{\cdot}2732(W \times D)}$	D_e (mm)	56.2	59.0	59.0	57.5
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	1.1	0.5	0.95	1.25
CORRECTED LOAD	P(kN)	0.97	0.61	0.88	1.06
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.31	0.18	0.25	0.32
SIZE CORR. FACTOR $F = (D_e/50)^{0{\cdot}45}$	F	1.05	1.08	1.08	1.06
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	0.32	0.19	0.27	0.34
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	7.8	4.5	6.6	8.2
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		A 	D 	D 	A 

NOTES:
1. Testing in accordance with ISRM Point Load Test Method - see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

MF 195

POINT LOAD STRENGTH TEST

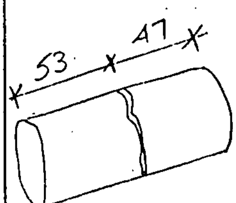
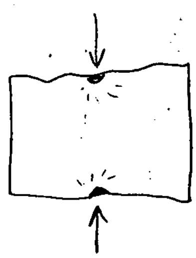
JOB NO.: DME 7/91
SHEET: 3 of 10

DEPARTMENT OF MINES AND ENERGY - SOUTH AUSTRALIA

CLIENT: HVCC — TEST LOCALITY: ~~AT~~ DRILL SITE — DATE: 6/3/91

PROJECT: MT OWEN COAL — TEST MACHINE: CSIRO MODEL P39 — TESTED BY: SW

LOCATION: BH 128 — DATE CALIBRATED: 15/4/91 — CHECKED BY:

SAMPLE ID and LOCATION		BH 128 30.2 m	BH 128 30.2 m		
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		SANDSTONE, light brownish grey, clayey, lithic, medium grained HW.	see left		
MOISTURE / STORAGE HISTORY		IN CORE BOX ONE DAY - NOT WRAPPED	see left.		
DIAMETER 1	W_1 (mm)	59.0	59.0		
DIAMETER 2	W_2 (mm)	59.0	59.0		
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	59	59		
PLATEN SEPARATION	D(mm)	59	48		
LENGTH / DIAM. RATIO $0.3 < D/W < 1.0$	D/W	1.0	0.81		
EQUIVALENT DIAM. $D_e = \sqrt{1.2732(W \times D)}$	D_e (mm)	59.0	60.0		
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	0.8	1.4		
CORRECTED LOAD	P(kN)	0.79	1.15		
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.23	0.32		
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.08	1.09		
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	0.24	0.35		
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	5.9	8.3		
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		D 	A 		

NOTES: 1. Testing in accordance with ISRM Point Load Test Method - see Explanatory Sheet 3.

2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

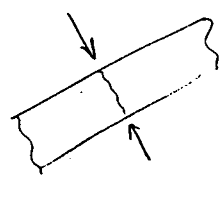
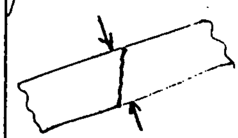
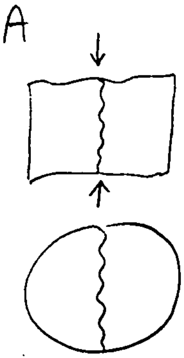
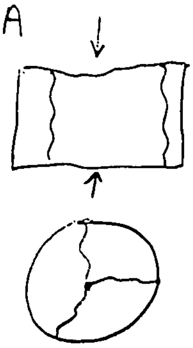
MF 195

JOB NO.	DME 7/91
SHEET	4 of 10

POINT LOAD STRENGTH TEST

DEPARTMENT OF MINES AND ENERGY - SOUTH AUSTRALIA

CLIENT	HVCC	TEST LOCALITY	CORE SHED	DATE	24/3/91
PROJECT	MT OWEN COAL	TEST MACHINE	CSIRO MODEL P39	TESTED BY	SW/LM
LOCATION	BOREHOLE 161	DATE CALIBRATED	15/4/91	CHECKED BY	

SAMPLE ID and LOCATION		BH 161 47.16m to 47.33m	see left	see left	see left
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		Light grey clayey siltstone. Fresh.	see left	see left	see left
MOISTURE/STORAGE HISTORY		PLASTIC WRAPPED FOR ONE WEEK	see left	see left	see left
DIAMETER 1	W_1 (mm)	60.5	60.5	60.5	60.5
DIAMETER 2	W_2 (mm)	60.5	60.5	60.5	60.5
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	60.5	60.5	60.5	60.5
PLATEN SEPARATION	D(mm)	60.5	60.5	45	42
LENGTH / DIAM. RATIO $0.3 < D/W < 1.0$	D/W	1.0	1.0	0.75	0.69
EQUIVALENT DIAM. $D_e = \sqrt{1.2732(W \times D)}$	D_e (mm)	60.5	60.5	58.9	56.9
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	2.1 MPa	1.6 MPa	1.4 MPa	1.6 MPa
CORRECTED LOAD	P(kN)	1.57 kN	1.27 kN	1.15 kN	1.27 kN
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.43	0.35	0.33	0.39
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.09	1.09	1.08	1.06
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	0.47	0.38	0.36	0.42
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	11.2	9.1	8.6	10.0
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		D 	D Fracture was along bedding at about 30° from normal. 	A 	A  BROKE IN THREE

NOTES:
1. Testing in accordance with ISRM Point Load Test Method - see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

MF 195

POINT LOAD STRENGTH TEST

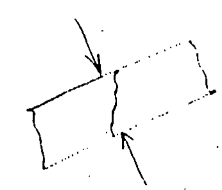
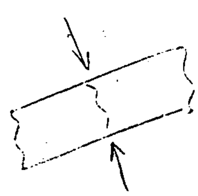
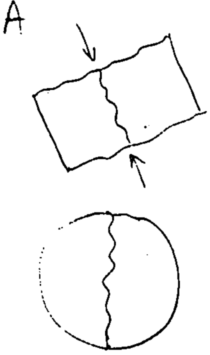
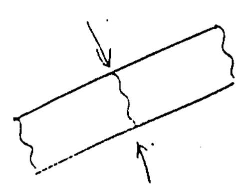
JOB NO.: DME 7/91
SHEET: 5 of 10

DEPARTMENT OF MINES AND ENERGY – SOUTH AUSTRALIA

CLIENT: HVCC TEST LOCALITY: CORE SHED DATE: 24/3/91

PROJECT: MT OWEN COAL TEST MACHINE: CSIRO MODEL P39 TESTED BY: SW/LM

LOCATION: BOREHOLE 176 DATE CALIBRATED: 15/4/91 CHECKED BY:

SAMPLE ID and LOCATION		BH 176 14.21 to 14.35m	BH 176 14.21 to 14.35m	BH 176 14.21m to 14.35m	BH 176 15.61 to 15.77m
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		Light brown, highly weathered, medium grained, lithic sandstone.	see left	see left	Light brown, highly weathered, medium grained, lithic SANDSTONE
MOISTURE / STORAGE HISTORY		PLASTIC WRAPPED FOR ONE WEEK	see left	see left	see left
DIAMETER 1	W_1 (mm)	60.5	60.5	60.5	60.5
DIAMETER 2	W_2 (mm)	60.5	60.5	60.5	60.5
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	60.5	60.5	60.5	60.5
PLATEN SEPARATION	D(mm)	60.5	60.5	39	60.5
LENGTH / DIAM. RATIO $0.3 < D/W < 1.0$	D/W	1.0	1.0	0.65	1.0
EQUIVALENT DIAM. $D_e = \sqrt{1.2732(W \times D)}$	D_e (mm)	60.5	60.5	54.8	60.5
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	2.3 MPa	2.8 MPa	2.2 MPa	2.0 MPa
CORRECTED LOAD	P(kN)	1.69 kN	2.00 kN	1.63 kN	1.51 kN
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.46	0.55	0.54	0.41
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.09	1.09	1.04	1.09
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	0.50	0.59	0.57	0.45
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	12.1	14.3	13.6	10.8
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		D  FRACTURED ALONG BEDDING AT APPROX 30° TO PERPENDICULAR	D 	A 	D 

NOTES:
1. Testing in accordance with ISRM Point Load Test Method – see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

MF 195

POINT LOAD STRENGTH TEST

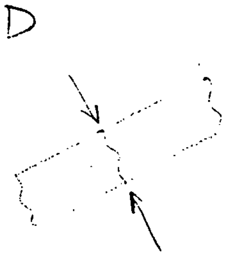
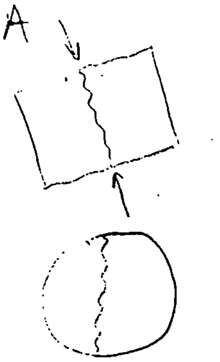
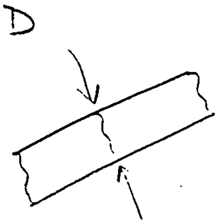
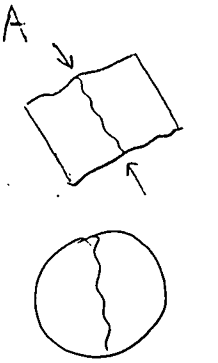
JOB NO. : DME 7/91
SHEET : 6 of 10

DEPARTMENT OF MINES AND ENERGY – SOUTH AUSTRALIA

CLIENT : HVCC TEST LOCALITY : CORE SHED DATE : 24/3/91

PROJECT : MT OWEN COAL TEST MACHINE : CSIRO MODEL P39. TESTED BY : SW/LM

LOCATION : BH 176 DATE CALIBRATED : 15/4/91 CHECKED BY :

SAMPLE ID and LOCATION		BH 176 16.46 to 16.60m	BH 176 16.46 to 16.60	BH 176 26.70 to 26.93	BH 176 26.70 to 26.93m
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		Light brown, medium grained lithic, clayey sandstone. Highly weathered	see left	Light grey silty mudstone. Slightly to moderately weathered.	see left
MOISTURE / STORAGE HISTORY		Significant drying over one week.	see left	Plastic wrapped for one week.	see left
DIAMETER 1	W_1 (mm)	60.5	60.5	60.5	60.5
DIAMETER 2	W_2 (mm)	60.5	60.5	60.5	60.5
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	60.5	60.5	60.5	60.5
PLATEN SEPARATION	D(mm)	60.5	52	60.5	57
LENGTH / DIAM. RATIO $0.3 < D/W < 1.0$	D/W	1.0	0.86	1.0	0.94
EQUIVALENT DIAM. $D_e = \sqrt{1.2732(W \times D)}$	D_e (mm)	60.5	63.3	60.5	66.3
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	5.0	4.8	2.0	2.2
CORRECTED LOAD	P(kN)	3.32	3.20	1.51	1.63
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.91	0.80	0.41	0.37
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.09	1.11	1.09	1.14
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	0.99	0.89	0.45	0.42
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	23.7	21.3	10.8	10.1
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		D 	A 	D 	A 

NOTES :
1. Testing in accordance with ISRM Point Load Test Method – see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

MF 195

POINT LOAD STRENGTH TEST

JOB NO.: DME 7/91
SHEET: 7 of 10

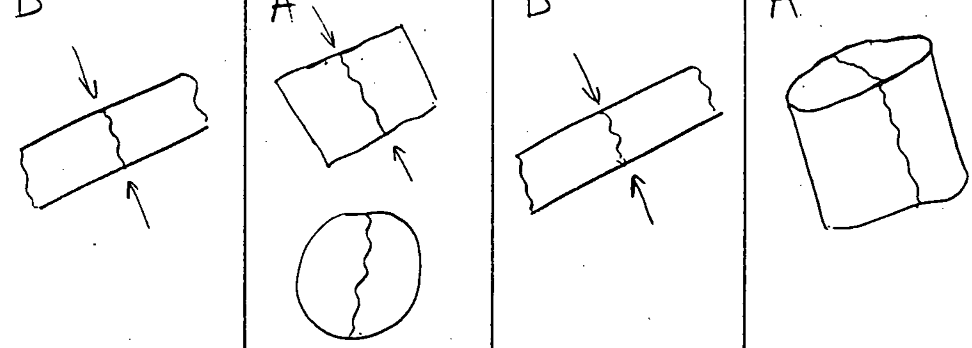
DEPARTMENT OF MINES AND ENERGY – SOUTH AUSTRALIA

CLIENT: HVCC TEST LOCALITY: CORE SHED DATE: 24/3/91

PROJECT: MT OWEN COAL TEST MACHINE: CSIRO MODEL P39. TESTED BY: SW/LM.

LOCATION: TEST DATE CALIBRATED: 15/4/91 CHECKED BY:

SAMPLE ID and LOCATION		BH 176 31·70 – 31·96m	BH 176 31·70 – 31·96m	BH 206 33·45 to 33·70m	BH 206 33·45 to 33·70m
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		Light grey, silty mudstone. Slightly weathered to Fresh.	see left	Mid-grey, silty mudstone Fresh.	see left
MOISTURE / STORAGE HISTORY		Plastic wrapped for one week.	see left	Plastic wrapped for three days.	see left
DIAMETER 1	W_1 (mm)	60·5	60·5	60·5	60·5
DIAMETER 2	W_2 (mm)	60·5	60·5	60·5	60·5
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	60·5	60·5	60·5	60·5
PLATEN SEPARATION	D(mm)	60·5	43	60·5	65
LENGTH / DIAM. RATIO $0{\cdot}3 < D/W < 1{\cdot}0$	D/W	1.0	0.71	1.0	1.07
EQUIVALENT DIAM. $D_e = \sqrt{1{\cdot}2732(W \times D)}$	D_e (mm)	60·5	57.6	60·5	70·8
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	2·9	4·5	2·0	2·9
CORRECTED LOAD	P(kN)	2.06	3.02	1.51	2.06
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.56	0.91	0.41	0.41
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.09	1.07	1.09	1.17
POINT LOAD STRENGTH (SIZE CORRECTED) $I_{s(50)} = I_s \times F$	$I_{s(50)}$ (MPa)	0.61	0.97	0.45	0.48
APPROX. EQUIV. U.C.S. $Q_u \simeq 24 \times I_{s(50)}$	Qu (MPa) (see Note 2)	14.7	23.3	10.8	11.5
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		D	A	D	A



NOTES: 1. Testing in accordance with ISRM Point Load Test Method – see Explanatory Sheet 3.

2. Value for UCS is approximate only. Conversion from $I_{s(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

MF 195

POINT LOAD STRENGTH TEST

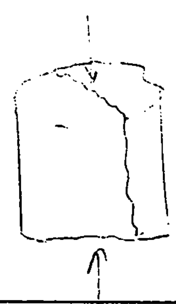
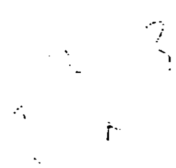
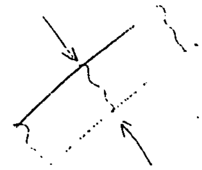
JOB NO.: DME 7/91
SHEET: 8 of 10

DEPARTMENT OF MINES AND ENERGY - SOUTH AUSTRALIA

CLIENT: HVCC
TEST LOCALITY: CORE SHED
DATE: 24/3/91

PROJECT: MT OWEN COAL
TEST MACHINE: CSIRO MODEL P39
TESTED BY: SW/LM

LOCATION: BH 206
DATE CALIBRATED: 15/4/91
CHECKED BY:

SAMPLE ID and LOCATION		BH 206 33.83 m	BH 206 40.13 – 40.34 m	BH 206 40.13 to 40.34	BH 206 40.13 to 40.34
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		SILTSTONE, light grey, very clayey.	Light grey, fine grained sandstone.	see left.	see left
MOISTURE / STORAGE HISTORY		not wrapped, significant drying	Plastic wrapped for three days	see left	see left
DIAMETER 1	W_1 (mm)	60.5	60.5	60.5	60.5
DIAMETER 2	W_2 (mm)	60.5	60.5	60.5	60.5
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	60.5	60.5	60.5	60.5
PLATEN SEPARATION	D (mm)	56	60.5	60.5	60.5
LENGTH / DIAM. RATIO $0.3 < D/W < 1.0$	D/W	0.93	1.0	1.0	1.0
EQUIVALENT DIAM. $D_e = \sqrt{1.2732(W \times D)}$	D_e (mm)	65.7	60.5	60.5	60.5
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	5.9	8.2	>10.0	7.2
CORRECTED LOAD	P (kN)	3.86	5.25	>6.33	4.64
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.90	1.43	>1.73	1.27
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.13	1.09	1.09	1.09
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	1.01	1.56	>1.88	1.38
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	24.3	37.5	>45.1	33.2
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		A 	D 	D 	D 

NOTES:
1. Testing in accordance with ISRM Point Load Test Method - see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

MF 195

POINT LOAD STRENGTH TEST

JOB NO. : DME 7/91
SHEET : 9 of 10

DEPARTMENT OF MINES AND ENERGY - SOUTH AUSTRALIA

CLIENT : HVCC — TEST LOCALITY : CORE SHED — DATE : 24/3/91

PROJECT : MT OWEN COAL — TEST MACHINE : CSIRO MODEL P39 — TESTED BY : SW/LM

LOCATION : BH 206 — DATE CALIBRATED : 15/4/91 — CHECKED BY :

SAMPLE ID and LOCATION		BH 206 45.27 - 45.61	see left	BH 206 38.12 - 38.39m	see left
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		Light grey, very fine grained, sandstone	see left	Light grey, slightly carbonaceous mudstone.	see left.
MOISTURE / STORAGE HISTORY		PLASTIC WRAPPED FOR 3 DAYS	see left	see left	see left
DIAMETER 1	W_1 (mm)	60.5	60.5	60.5	60.5
DIAMETER 2	W_2 (mm)	60.5	60.5	60.5	60.5
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	60.5	60.5	60.5	60.5
PLATEN SEPARATION	D(mm)	60.5	39	60.5	53
LENGTH / DIAM. RATIO $0.3 < D/W < 1.0$	D/W	1.0	0.64	1.0	0.88
EQUIVALENT DIAM. $D_e = \sqrt{1.2732(W \times D)}$	D_e (mm)	60.5	54.8	60.5	63.9
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	4.5	4.2	3.4	3.4
CORRECTED LOAD	P(kN)	3.02	2.84	2.36	2.36
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	0.82	0.94	0.64	0.58
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.09	1.04	1.09	1.12
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	0.90	0.98	0.70	0.64
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	21.6	23.6	16.8	15.5
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		D	A	D	A

NOTES:
1. Testing in accordance with ISRM Point Load Test Method - see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

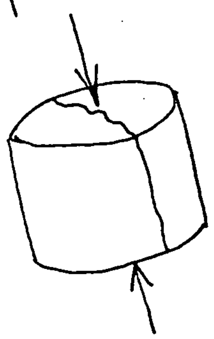
MF 195

POINT LOAD STRENGTH TEST

JOB NO. : DME 7/91
SHEET : 10 of 10

DEPARTMENT OF MINES AND ENERGY - SOUTH AUSTRALIA

CLIENT : HVCC TEST LOCALITY : CORE SHED DATE : 24/3/91
PROJECT : MT OWEN COAL TEST MACHINE : CSIRO MODEL P39 TESTED BY : SW/LM
LOCATION : BH 206 DATE CALIBRATED : 15/4/91 CHECKED BY :

SAMPLE ID and LOCATION		BH 206 40.13 to 40.34			
SAMPLE DESCRIPTION (NOTE PLANES OF WEAKNESS)		Light grey, fine grained sandstone.			
MOISTURE / STORAGE HISTORY		PLASTIC WRAPPED FOR 3 DAYS			
DIAMETER 1	W_1 (mm)	60.5			
DIAMETER 2	W_2 (mm)	60.5			
AVE. DIAMETER $W = \frac{W_1 + W_2}{2}$	W (mm)	60.5			
PLATEN SEPARATION	D(mm)	36			
LENGTH / DIAM. RATIO $0{\cdot}3 < D/W < 1{\cdot}0$	D/W	0.60			
EQUIVALENT DIAM. $D_e = \sqrt{1{\cdot}2732(W \times D)}$	D_e (mm)	52.7			
FAILURE LOAD (or GAUGE PRESSURE)	kN (kPa)	5.0			
CORRECTED LOAD	P(kN)	3.32			
UN-CORRECTED P.L. STRENGTH INDEX $I_s = (P/D_e^2) \times 1000$	I_s (MPa)	1.20			
SIZE CORR. FACTOR $F = (D_e/50)^{0.45}$	F	1.02			
POINT LOAD STRENGTH (SIZE CORRECTED) $Is_{(50)} = Is \times F$	$Is_{(50)}$ (MPa)	1.23			
APPROX. EQUIV. U.C.S. $Qu \simeq 24 \times Is_{(50)}$	Qu (MPa) (see Note 2)	29.4			
TEST TYPE, SKETCH AND NOTES A = AXIAL SAMPLE TEST D = DIAMETRAL SAMPLE TEST B = BLOCK SAMPLE TEST I = IRREGULAR LUMP TEST		A 			

NOTES : 1. Testing in accordance with ISRM Point Load Test Method - see Explanatory Sheet 3.
2. Value for UCS is approximate only. Conversion from $Is_{(50)}$ to Qu is only accurate if extensively calibrated for specific site materials.

MF 195

Precision Calibration Services
(Testing Division) Pty. Limited A.C.N. 003 356 189
SERVICE and CALIBRATION of MATERIALS TESTING SYSTEMS
Telephone: (02) 609 7400
Fax: (02) 604 9662
NATA Registered Laboratory 1710
Unit 6, Lot B, Daniel St.
Wetherill Park 2164

NAFDR02

CALIBRATION REPORT
on
A JACKING SYSTEM

REPORT No. N202JS0101 ISSUE DATE: 15-04-1991

FOR: JOINT COAL BOARD

1 CIVIC AVENUE
SINGLETON 2330

CALIBRATION DATE: 15-04-1991

DESCRIPTION:
Type: Point load tester
Maker: CSIRO Model: -
Serial Number: Gauge Nos. T20201A & T20201B Plant No. -
Range/s: Gauge T20201A 6 kN
Gauge T20201B 25 kN
Force indication: Bourdon pressure gauges

SPECIFICATION: Australian Standard 2193 - 1978

PRELIMINARY INSPECTION:
Prior to calibration, the system was serviced in order to reduce hydraulic leakage.

CALIBRATION:
Calibration procedures in compression as described in Clause 3.3.6.2 (a) of the standard were followed. The maximum load pointer was used during all series of tests. The calibration of standards used is traceable to National Standards. Temperature during these procedures was 24 deg. C.

ADJUSTMENT SETTINGS: Refer page 2 for scale factors.

USER CHECKS: Nil.

CONCLUSION:
The device was found to comply with the requirements of Australian Standard 2193 - 1978.

CALIBRATION EXPIRY: 15-10-1991
RECOMMENDED RECALIBRATION: 15-10-1991

ROWAN J. HIRD
Approved Signatory.

PRECISION CALIBRATION SERVICES (Testing Division) Pty. Limited.

The following table/s indicate the range over which particular grades apply. Results are calculated using the following formulae.

$$D = \frac{F - B}{A} \quad \text{OR} \quad F = (D \times A) + B$$

where: D = Device deflection (MPa)
F = Applied force (kN)
A = Refer columns 1 & 2 of tables
B = ...

In the following table one scale division = 0.2 (MPa)

Range COMPRESSION (kN)	Scale Readability (kN)	Applied Force (kN)	Calibrated Deflection (MPa)	Computed Deflection (MPa)	Mean Error %	Repeat'y %	Grade AS 2193
6	0.15	1.6	2.27	2.143	-5.5	52.9	NIL
LOW		2.4	3.22	3.472	7.9	49.7	NIL
CAPACITY		4	6.38	6.130	-4.0	4.7	NIL
GAUGE		5	7.73	7.791	0.7	11.0	NIL
		6	9.38	9.452	0.7	5.9	NIL
FACTOR A =	0.602						
FACTOR B =	0.31						

In the following table one scale division = 2 (MPa)

Range COMPRESSION (kN)	Scale Readability (kN)	Applied Force (kN)	Calibrated Deflection (MPa)	Computed Deflection (MPa)	Mean Error %	Repeat'y %	Grade AS 2193
25	0.14	2.5	4.7	4.99	6.8	8.6	NIL
LOW		5	8.6	8.53	-0.8	4.7	NIL
CAPACITY		10	15.8	15.61	-1.2	2.5	NIL
GAUGE		15	23.0	22.69	-1.3	1.7	NIL
		20	29.9	29.77	-0.5	2.0	NIL
FACTOR A =	0.706	25	36.5	36.86	0.9	3.3	NIL
FACTOR B =	-1.02						

In the above table/s error is the difference between computed deflection and calibrated deflection expressed as a percentage of calibrated deflection. e.g. a positive error indicates that computed deflection is numerically greater than calibrated deflection.

ROWAN J. HIRD
Approved Signatory

Suggested Method for Determining Point Load Strength ______ [in part]

PROCEDURE

Specimen selection and preparation

6.(a) A test sample is defined as a set of rock specimens of similar strength for which a single Point Load Strength value is to be determined.

(b) The test sample of rock core or fragments is to contain sufficient specimens conforming with the size and shape requirements for diametral, axial, block or irregular lump testing as specified below.[7]

(c) For routine testing and classification, specimens should be tested either fully water-saturated or at their natural water content.[8]

Calibration

7. The test equipment should be periodically calibrated using an independently certified load cell and set of displacement blocks, checking the P and D readings over the full range of loads and displacements pertinent to testing.

The diametral test[2]

8.(a) Core specimens with length/diameter ratio greater than 1.0 are suitable for diametral testing.

(b) There should preferably be at least 10 tests per sample, more if the sample is heterogeneous or anisotropic.[7]

(c) The specimen is inserted in the test machine and the platens closed to make contact along a core diameter, ensuring that the distance L between the contact points and the nearest free end is at least 0.5 times the core diameter (Fig. 3a).

(d) The distance D is recorded $\pm 2\%$.[4]

(e) The load is steadily increased such that failure occurs within 10–60 sec, and the failure load P is recorded. The test should be rejected as invalid if the fracture surface passes through only one loading point (Fig. 4d).

(f) The procedure (c) through (e) above is repeated for the remaining specimens in the sample.

The axial test[2]

9.(a) Core specimens with length/diameter ratio of 0.3–1.0 are suitable for axial testing (Fig. 3b). Long pieces of core can be tested diametrally to produce suitable lengths for subsequent axial testing (provided that they are not weakend by this initial testing); alternatively, suitable specimens can be obtained by saw-cutting or chisel-splitting.

(b) There should preferably be at least 10 tests per sample, more if the sample is heterogeneous or anisotropic.[7]

(c) The specimen is inserted in the test machine and the platens closed to make contact along a line perpendicular to the core end faces (in the case of isotropic rock, the core axis, but see paragraph 11 and Fig. 5).

(d) The distance D between platen contact points is recorded $\pm 2\%$.[4] The specimen width W perpendicular to the loading direction is recorded $\pm 5\%$.[4]

(e) The load is steadily increased such that failure occurs within 10–60 sec, and the failure load P is recorded. The test should be rejected as invalid if the fracture surface passes through only one loading point (Fig. 4c).

(f) The procedures (c) through (e) above are repeated for the remaining tests in the sample.

The block and irregular lump tests

10.(a) Rock blocks or lumps of size 50 ± 35 mm and of the shape shown in Fig. 3(c) and (d) are suitable for the block and the irregular lump tests. The ratio D/W should be between 0.3 and 1.0, preferably close to 1.0. The distance L (Fig. 3c and d) should be at least $0.5W$. Specimens of this size and shape may be selected if available or may be prepared by trimming larger pieces by saw-or chisel-cutting.

(b) There should preferably be at least 10 tests per sample, more if the rock is heterogeneous or anisotropic.[7]

(c) The specimen is inserted in the testing machine and the platens closed to make contact with the smallest dimension of the lump or block, away from edges and corners (Fig. 3c and d).

(d) The distance D between platen contact points is recorded $\pm 2\%$. The smallest specimen width W perpendicular to the loading direction is recorded $\pm 5\%$. If the sides are not parallel, then W is calculated as $(W_1 + W_2)/2$ as shown in Fig. 3d.[4] This smallest width W is used irrespective of the actual mode of failure (Figs 3 and 4)

(e) The load is steadily increased such that failure occurs within 10–60 sec, and the failure load P is recorded. The test should be rejected as invalid if the fracture surface passes through only one loading point (see examples for other shapes in Fig. 4d or e).

(f) The procedure (c) through (e) above is repeated for the remaining tests in the sample.

Anisotropic rock

11. (a) When a rock sample is shaly, bedded, schistose or otherwise observably anisotropic it should be tested in directions which give the greatest and least strength values, which are in general parallel and normal to the planes of anisotropy.

(b) If the sample consists of core drilled through the weakness planes, a set of diametral tests may be completed first, spaced at intervals which will yield pieces which can then be tested axially.

(c) Best results are obtained when the core axis is perpendicular to the planes of weakness, so that when possible the core should be drilled in this direction. The angle between the core axis and the normal to the weakness planes should preferably not exceed 30°.

(d) For measurement of the I_s value in the directions of least strength, care should be taken to ensure that load is applied along a single weakness plane. Similarly when testing for the I_s value in the direction of greatest strength, care should be taken to ensure that the load is applied perpendicularly to the weakness planes (Fig. 5).

(e) If the sample consists of blocks or irregular lumps, it should be tested as two sub-samples, with load applied firstly perpendicular to, then along the observable planes of weakness.[10] Again, the required minimum strength value is obtained when the platens make contact along a single plane of weakness.

CALCULATIONS

Uncorrected point load strength

12. The Uncorrected Point Load Strength I_s is calculated as P/D_e^2 where D_e, the "equivalent core diameter", is given by:

$D_e^2 = D^2$ for diametral tests;

$= 4A/\pi$ for axial, block and lump tests;

and

$A = WD$ = minimum cross sectional area of a plane through the platen contact points.[4]

Size correction

13.(a) I_s varies as a function of D in the diametral test, and as a function of D_e in axial, block and irregular lump tests, so that a size correction must be applied to obtain a unique Point Load Strength value for the rock sample, and one that can be used for purposes of rock strength classification.

(b) The size-corrected Point Load Strength Index $I_{s(50)}$ of a rock specimen or sample is defined as the value of I_s that would have been measured by a diametral test with $D = 50$ mm.

(c) The most reliable method of obtaining $I_{s(50)}$, preferred when a precise rock classification is essential, is to conduct diametral tests at or close to $D = 50$ mm. Size correction is then either unnecessary ($D = 50$ mm) or introduces a minimum of error. The latter is the case, for example, for diametral tests on NX core, $D = 54$ mm. This procedure is not mandatory. Most point load strength testing is in fact done using other sizes or shapes of specimen. In such cases, the size correction (d) or (e) below must be applied.

(d) The most reliable method of size correction is to test the sample over a range of D or D_e values and to plot graphically the relation between P and D_e^2. If a log–log plot is used the relation is generally a straight line (Fig. 6). Points that deviate substantially from the straight line may be disregarded (although they should not be deleted). The value of P_{50} corresponding to $D_e^2 = 2500$ mm^2 ($D_e = 50$ mm) can then be obtained by interpolation, if necessary by extrapolation, and the size-corrected Point Load Strength Index calculated as $P_{50}/50^2$.

(e) When neither (c) nor (d) is practical, for example when testing single sized core at a diameter other than 50 mm or if only a few small pieces are available, size correction may be accomplished by using the formula:

$$I_{s(50)} = F \times I_s$$

The "Size Correction Factor F" can be obtained from the chart in Fig. 7,[11] or from the expression:

$$F = (D_e/50)^{0.45}$$

For tests near the standard 50 mm size, very little error is introduced by using the approximate expression:

$$F = \sqrt{(D_e/50)}$$

(f) The size correction procedures specified in this paragraph have been found to be applicable irrespective of the degree of anisotropy I_a and the direction of loading with respect to planes of weakness, a result that greatly enhances the usefulness of this test.

Mean value calculation

14.(a) Mean values of $I_{s(50)}$ as defined in (b) below are to be used when classifying samples with regard to their Point Load Strength and Point Load Strength Anisotropy Indices.

(b) The mean value of $I_{s(50)}$ is to be calculated by deleting the two highest and lowest values from the 10 or more valid tests, and calculating the mean of the remaining values. If significantly fewer specimens are tested, only the highest and lowest values are to be deleted and the mean calculated from those remaining.[12]

Point load strength anisotropy index

15. The Strength Anistropy Index $I_{a(50)}$ is defined as the ratio of mean $I_{s(50)}$ values measured perpendicular and parallel to planes of weakness, i.e. the ratio of greatest to least Point Load Strength Indices. $I_{a(50)}$ assumes values close to 1.0 for quasi-isotropic rocks and higher values when the rock is anisotropic. On average, uniaxial compressive strength is 20–25 times point load strength, as shown in Fig. 9. However, in tests on many different rock types the ratio can vary between 15 and 50 especially for anisotropic rocks, so that errors of up to 100% are possible in using an arbitrary ratio value to predict compressive strength from point load strength.

$I_{s(50)}$ is approximately 0.80 times the uniaxial tensile or Brazilian tensile strength.

7. Because this test is intended primarily as a simple and practical one for field classification of rock materials, the requirements relating to sample size, shape, numbers of tests etc, can when necessary be relaxed to overcome practical limitations. Such modifications to procedure should however be clearly stated in the report.

11. The size correction factor chart (Fig. 7) is derived from data on cores tested diametrally and axially and from tests on blocks and irregular lumps, for rocks of various strengths, and gives an averaged factor. Some rocks do not conform to this behaviour, and size correction should therefore be considered an approximate method, although sufficient for most practical rock classification applications. When a large number of tests are to be run on the same type of rock it may be advantageous to first perform a series of tests at different sizes to obtain a graph of load vs D_e^2 as in Fig. 6. If the slope of such a log–log graph is determined as "n", the size correction factor is then $(D_e/50)^m$ where $m = 2(1 - n)$. This can either be calculated directly or a chart constructed.

*Paragraph and figure numbers are those of the original reference: ISRM Commission on Testing Methods 1985. Suggested method for determining Point Load Strength (revised version) Int. J. Rock Mech. Min. Sci. & Geomech. Abstr. 22, 51-60.

Important references for further reading:-

GUIFU, X., Hong, L. On the statistical analysis of data and strength expression in the rock point load tests. Proc. 5th Int. Cong. Int. Ass. of Engng. Geology, Beunos Aires 1986.

READ, J.R.L., THORNTON, P.N., REGAN, W.M. A rational approach to the Point Load Test. Third Australia New Zealand Conf. on Geomechanics, Wellington 1980. Vol. 2, pp2-35 to 2-39.

TURK, N. and DEARMAN, W.R. Improvements in the determination of point load strength. Bull. Int. Ass. Engng. Geol. No. 31. paris 1985 pp137-142.

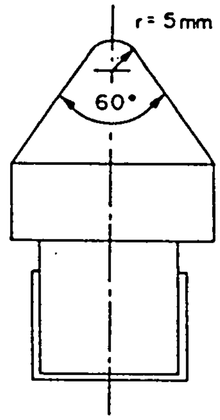


Fig. 2. Platen shape and tip radius.

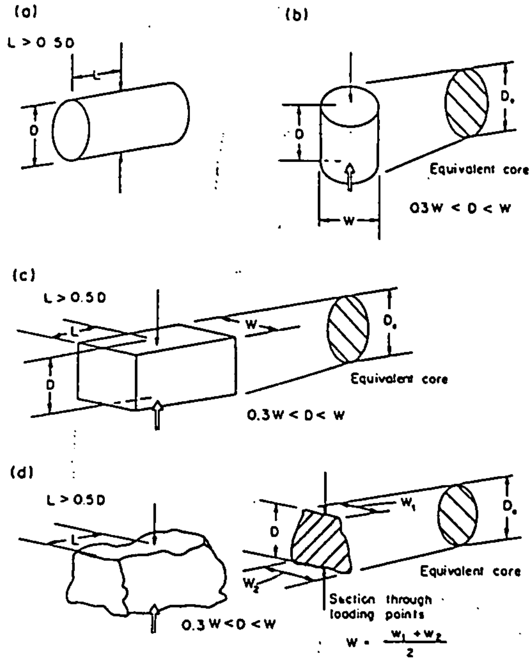


Fig. 3. Specimen shape requirements for (a) the diametral test, (b) the axial test, (c) the block test, and (d) the irregular lump test.

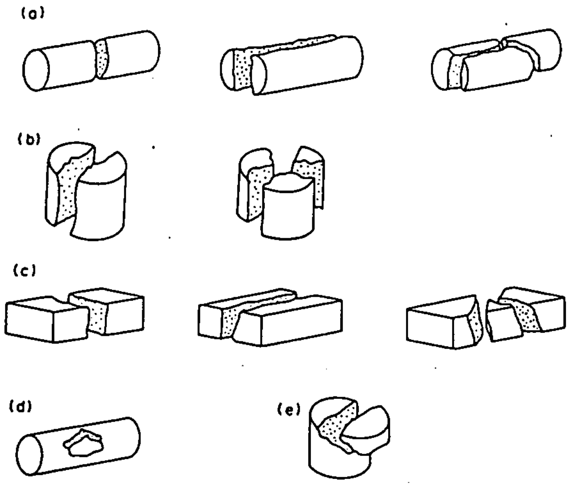


Fig. 4. Typical modes of failure for valid and invalid tests. (a) Valid diametral tests; (b) valid axial tests; (c) valid block tests; (d) invalid core test; (e) invalid axial test.

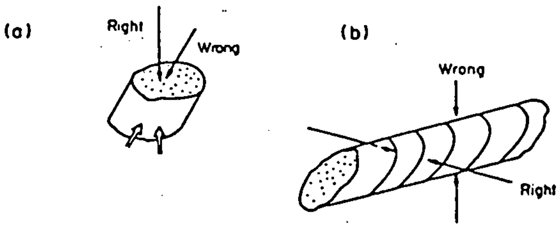


Fig. 5. Loading directions for tests on anisotropic rock.

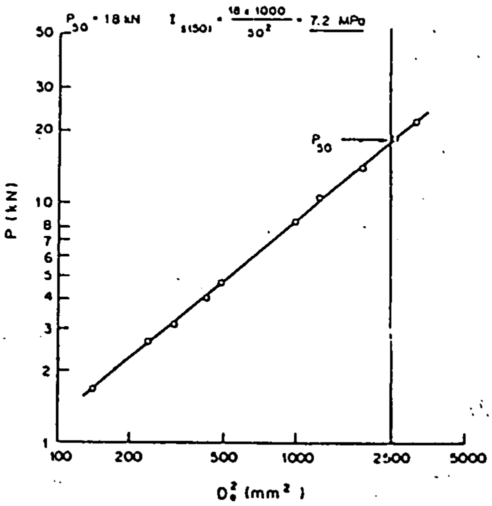


Fig. 6. Procedure for graphical determination of $I_{s(50)}$ from a set of results at D_e values other than 50 mm.

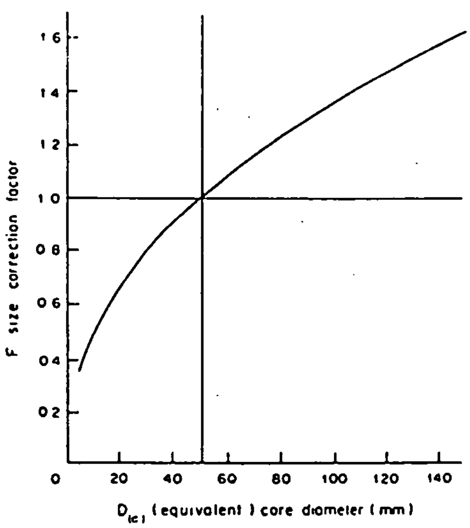


Fig. 7. Size correction factor chart.

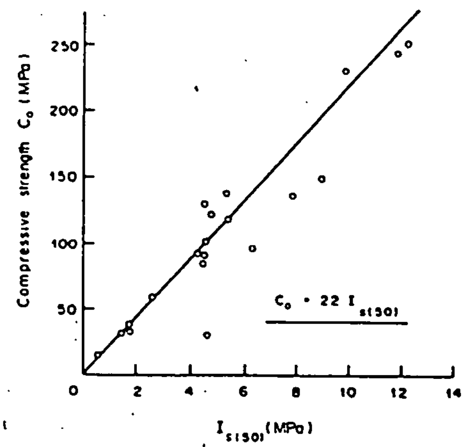


Fig. 9. Example of correlation between point load and uniaxial compressive strength results.

APPENDIX D

CORE DEFECT ORIENTATIONS

1.0 ORIENTATION OF DRILL CORE

Where it was possible to do so, rock defects in the cores were measured to provide orientations relative to the orientation of nearby bedding surfaces. Two measurements are required to orient each defect relative to bedding; FCA angle and B angle. The FCA (fracture to core axis) angle is the smallest angle between the fracture surface and a line parallel to the core axis. The angle B is measured as shown in Figure D1. For a vertical borehole B is the angle, in a clockwise direction looking towards the down-hole direction, between the dip direction of bedding (D_b) and the dip direction of each defect (D_d). This angle for each oriented defect is recorded on the geotechnical logs (Appendix A) as B in degrees. Details for each of the oriented defects are shown in Table D1.

An estimate can be made of the true orientation of each defect by adding this B angle to the average dip direction of bedding in the hole (calculated from three point intersections between boreholes).

Lower hemisphere, equal angle stereonet plots of all oriented joints have been made with the joints plotted in two different ways.

i) relative to north on the Australian Map Grid (AMG).
ii) relative to bedding dip direction.

No discernible pattern emerges on the plot of joint orientations relative to AMG north. This indicates that there is no regional regime of defect sets imprinted over the Mt Owen area. However, when the joint orientations are plotted relative to the dip direction of bedding (which changes throughout the planned pit location) they appear to be concentrated into a number of joint sets. Figure D2 shows contours of concentration of the defects plotted in this way. Six sets are indicated with maximum concentrations having the orientations:

SET 1	75° - 182 + bedding dip direction
SET 2	33° - 000 + bedding dip direction
SET 3	61° - 221 + bedding dip direction
SET 4	71° - 273 + bedding dip direction
SET 5	18° - 295 + bedding dip direction
SET 6	40° - 044 + bedding dip direction.

It should be noted that a number of factors detract from the accuracy of this interpretation.

a) Only a relatively small total number (74) of rock defects could be oriented.

b) Relative orientations of large scale cross bedding, indistinguishable from true bedding orientations in core samples, are unknown.

c) A directional bias has been introduced due to all the boreholes being vertical. The probability of vertical boreholes intersecting a defect of any given orientation is lowest for those with steepest dip and highest for those with shallowest dip.

A number of cored boreholes were drilled prior to this study, but no orientations were recorded in the earlier work. Defects from two of these boreholes were measured as described above, and the results included in Table D1. Limited time on site precluded further detailed examination of the old cores, but the exercise is considered worthwhile as a part of future studies.

2.0 BEDDING DIP OBSERVATIONS

It was felt necessary to compare bedding dip angles observed and measured in the drill core with the mean bedding dip angles obtained from the major stratigraphic units in adjacent boreholes which were used by JCB to produce their cross sections (eg Fig 6). Any major disagreement between these dips would require an explanation.

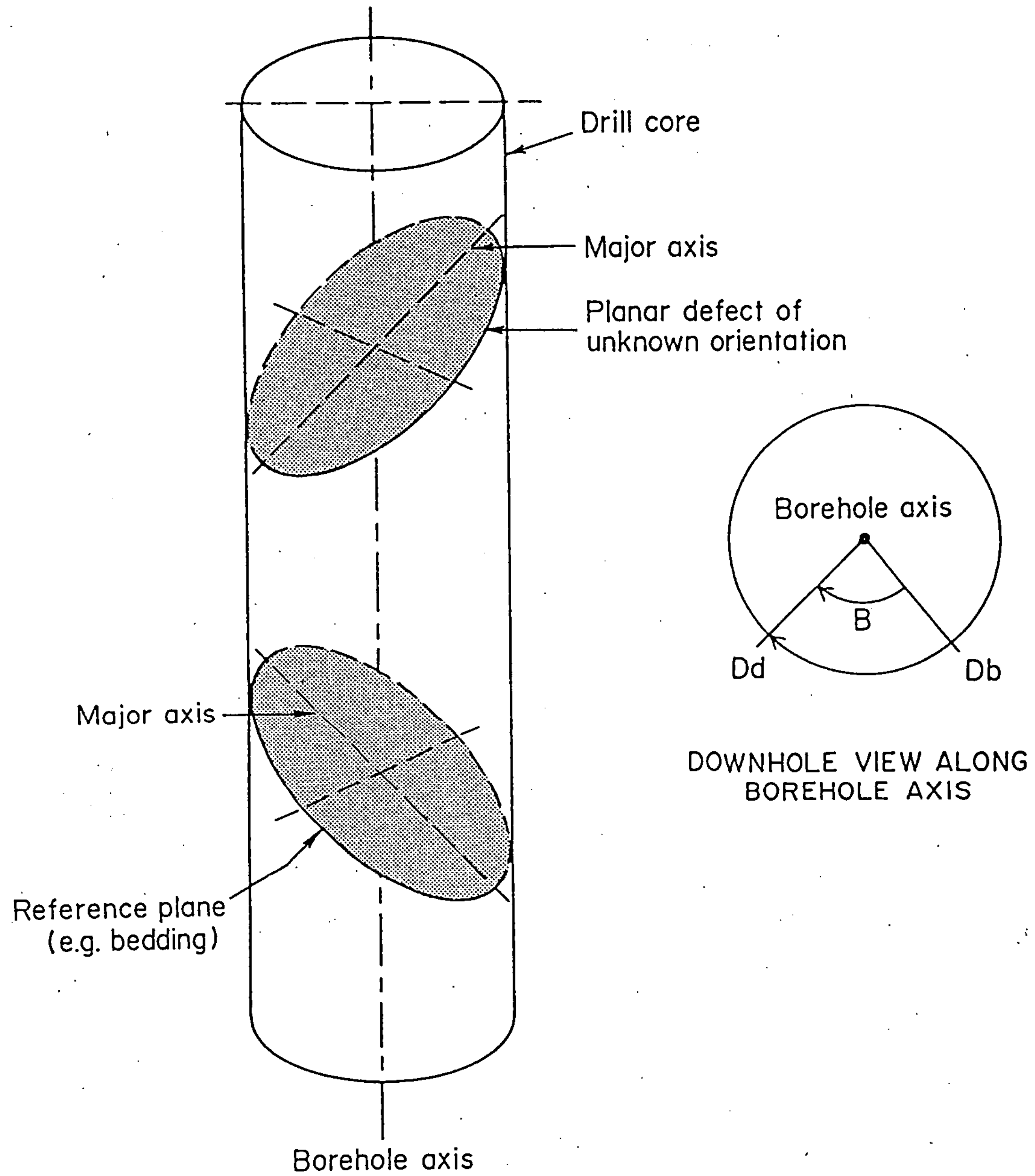
Figure D3A shows the frequency distribution of measured bedding dips which range from 10 to 42 degrees to have a mean of 23.5 degrees.

It was thought that a significant bias could have arisen from the inclusion of sandstone cross bedding with normal bedding plane dips. Figure D3B shows the result of plotting only bedding dips from the fine grained rocks where cross bedding would be unlikely to occur or be confused with normal bedding. The result leads to a mean dip of 22.9 degrees. Bedding dip figures are summarized in Table D2.

TABLE D2: ANOMALOUS BEDDING DIP OBSERVATIONS

SOURCE OF INFORMATION	RANGE OF DIP	NO OF OBSERVATIONS	MEAN DIP
JCB CROSS SECTIONS	16-22	3[1]	17[2]
CORE LOGS (all rocks)	10-42	124	23.5
CORE LOGS (siltstones and mudstones)	10-38	65	22.9

Note 1: From three cross sections judged to be representative of the subject area.
2: Bulk of subject area.



MT OWEN PROJECT

Fig. D1. ORIENTATION OF DEFECTS USING BEDDING AS A REFERENCE ORIENTATION
(Adapted from Priest (1985) Fig. 4.8)

MT OWEN PROJECT
DEFECT ORIENTATION CONTOURS

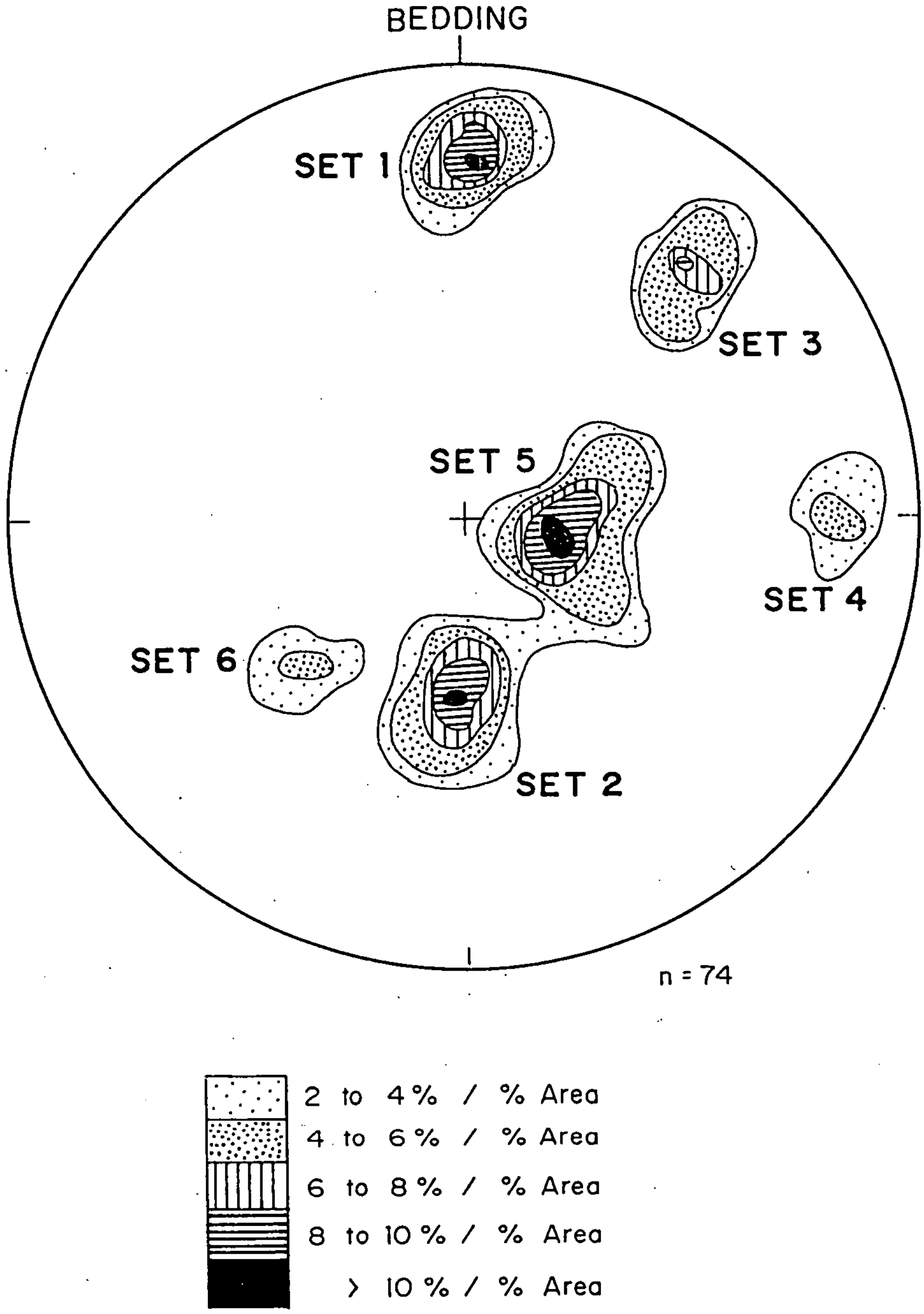


Figure D2

S.A. Dept of Mines and Energy 24-4-91

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DISTRIBUTION OF BEDDING DIPS

(all rock types)

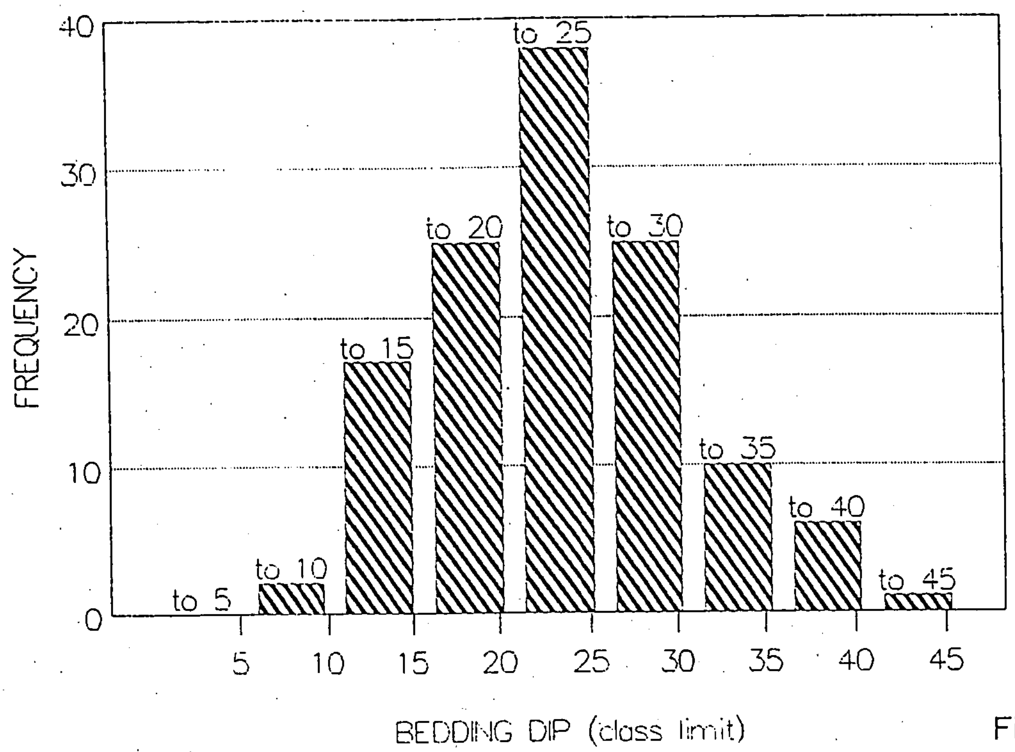


Figure D3.a.

DISTRIBUTION OF BEDDING DIPS

(Siltstones and Mudstones only)

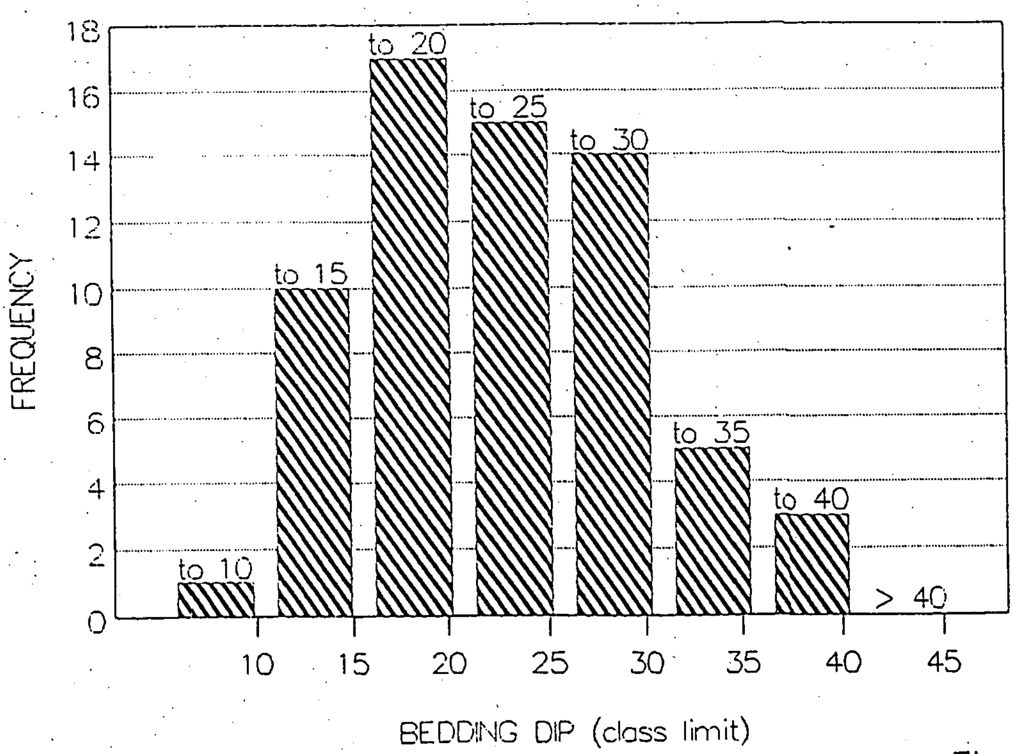


Figure D3.b.

TABLE D1

Details of Oriented Rock Defects

HOLE NUMBER	DEPTH (m)	DIP ANGLE	B ANGLE
128	21.60	68	249
128	27.17	66	277
129	41.57	18	301
129	41.84	17	283
129	41.86	16	283
129	41.99	23	225
129	42.12	25	177
129	47.88	38	261
160	8.30	88	90
160	11.13	75	0
160	12.73	65	272
160	13.20	70	144
160	14.56	50	214
160	14.88	0	177
160	20.05	71	192
160	20.25	69	184
160	21.60	65	178
160	21.66	59	178
160	21.73	68	178
160	22.98	67	224
160	24.85	70	164
160	29.38	69	182
160	31.04	55	227
160	31.09	55	227
160	31.22	62	224
160	49.24	29	9
161	37.76	72	269
161	37.90	75	269
161	38.00	78	269
161	39.60	76	278
161	39.80	42	93
176	18.88	24	322
176	19.10	48	0
176	19.24	35	300
176	19.29	26	300
176	28.48	45	47
176	28.56	35	47
176	28.59	30	187
176	28.86	22	281
176	28.89	26	263
176	32.35	65	216
206	34.79	28	261
206	34.82	43	75
206	35.24	32	0
206	35.29	32	0
206	37.28	41	315
206	37.93	42	18
206	38.49	43	37
206	38.65	32	341
206	40.15	23	0
206	40.35	57	180
206	40.58	49	55
206	41.69	69	187
206	42.51	28	28
206	43.14	12	308
206	43.19	28	308
206	43.20	80	134
206	43.36	22	278
206	43.69	31	0
206	44.70	56	65
206	44.80	42	9
206	44.81	42	9
206	47.02	43	0
206	47.39	15	297
206	47.42	15	297
206	47.46	58	37
206	48.57	37	0
206	51.91	58	330
206	52.99	41	0
14	21.50	65	180
14	24.12	53	126
14	24.56	45	234
14	26.18	66	225
14	26.34	66	298
14	26.97	27	253
14	27.07	68	155
24	23.12	57	267
24	24.17	70	84

APPENDIX E
GENERAL EXPLANATORY NOTES

G02783

NOTE (A) SOIL TYPE

42. Classification of soils

42.1 Use of the system. Use of the British Soil Classification System for Engineering Purposes (BSCS) described in 42.3 is discretionary; for many purposes, a full description of soils in accordance with clause 41 will suffice. The BSCS is recommended primarily for soils to be used as construction materials, when it is particularly useful. When the symbols for the BSCS are used, a full written description, including both the soil group name and supplementary descriptive terms as discussed in clause 41, is also required.

Not included in these notes

42.2 Nature and purpose of soil classification

42.2.1 *Distinction between soil description and soil classification.* A full description gives detailed information on the grading, plasticity, colour, and particle characteristics of a soil, as well as on the fabric, the state of bedding, nature of discontinuities and strength condition in which it occurs in a sample, borehole or exposure. Few, if any, soils will have identical descriptions. On the other hand, a soil classification places a soil in a limited number of groups on the basis of grading and plasticity of a disturbed sample. These characteristics are independent of the particular condition in which a soil occurs, and disregard the influence of the structure, including fabric, of the soil mass; but they can give a good guide to how the disturbed soil will behave when used as a construction material, under various conditions of moisture content.

41.3 Material characteristics of soils

41.3.1 *Range of application.* Material characteristics refer to those characteristics that can be described from visual and manual examination of either disturbed or undisturbed samples, and include soil name, colour, particle shape and particle composition.

In a soil description, the main characteristics should preferably be given in approximately the following order.

(a) *Mass characteristics* (see 41.2)

(1) Field strength or compactness (see table 6), and indication of moisture condition.

(2) Bedding.

(3) Discontinuities.

(4) State of weathering.

(b) *Material characteristics* (see 41.3)

(1) Colour.

(2) Particle shape and composition.

(3) Soil name (in capitals, e.g. SAND), grading and plasticity.

(c) *Geological formation, age and type of deposit* (see 41.4).

(d) *Classification* (optional) (see clause 42).

Soil group symbol.

Examples:

Firm closely-fissured yellowish-brown CLAY of high plasticity. London Clay.

41.3.2.5 *Colour.* Details are given in the extreme right hand column of table 6. For more detailed descriptions, colour charts based on the system of Munsell may be used [156, 157] †.

41.3.2.6 *Particle shape and composition.* Where appropriate, particle shape may be described by reference to the general form of the particles, their angularity which indicates the degree of rounding at edges and corners, and their surface characteristics. Some recommended terms are as follows.

Angularity	angular subangular subrounded rounded
Form	equidimensional flat elongated flat and elongated irregular
Surface texture	rough smooth

41.3.3 *Made ground.* It is rarely possible to carry out significant soil tests on made ground, and descriptions of the material are all that remains after the samples have been discarded or pits filled in. Good descriptions are, therefore, of even greater importance with this type of material and should include information on the following as well as on the soil constituents.

(a) Mode of origin of the material.

(b) Presence of large objects such as concrete, masonry or old motor cars.

(c) Presence of voids or collapsible hollow objects.

(d) Chemical waste, and dangerous or poisonous substances.

(e) Organic matter, with a note on the degree of decomposition.

(f) Odourous smell.

(g) Striking colour tints.

(h) Any dates readable on buried newspapers.

(i) Signs of heat or internal combustion under ground, i.e. steam emerging from borehole.

41.3.2 *Soil name*

41.3.2.1 *Introduction.* The soil name is based on particle size distribution and plastic properties. These characteristics are used because they can be measured readily with reasonable precision, and estimated with sufficient accuracy for descriptive purposes.

Table 7. Names and descriptive letters for grading and plasticity characteristics

		Descriptive name	Letter
Coarse components	Main terms	GRAVEL	G
		SAND	S
	Qualifying terms	Well graded	W
		Poorly graded	P
		Uniform	Pu
		Gap graded	Pg
Fine components	Main terms	FINE SOIL, FINES may be differentiated into M or C	F
		SILT (M-SOIL)* plots below A-line of plasticity chart of figure 31 (of restricted plastic range)	M
		CLAY plots above A-line (fully plastic)	C
	Qualifying terms	Of low plasticity	L
		Of intermediate plasticity	I
		Of high plasticity	H
		Of very high plasticity	V
		Of extremely high plasticity	E
		Of upper plasticity range† incorporating groups I, H, V and E	U
Organic components	Main term	PEAT	Pt
	Qualifying term	Organic may be suffixed to any group	O

*See note 5 following table 8.

† This term is a useful guide when it is not possible or not required to designate the range of liquid limit more closely, e.g. during the rapid assessment of soils.

41.3.2.3 *Plasticity.* Clay and silt, both alone and in mixtures with coarser material, may be classified as follows:

Term	Range of liquid limit
of low plasticity	under 35 %
of intermediate plasticity	35 % to 50 %
of high plasticity	50 % to 70 %
of very high plasticity	70 % to 90 %
of extremely high plasticity	over 90 %

Term	Composition of the coarse fraction
Slightly sandy GRAVEL	up to 5 % sand
Sandy GRAVEL	5 % to 20 % sand
Very sandy GRAVEL	over 20 % sand
GRAVEL/SAND	about equal proportions of gravel and sand
Very gravelly SAND	over 20 % gravel
Gravelly SAND	20 to 5 % gravel
Slightly gravelly SAND	up to 5 % gravel

41.3.2.4 *Deposits containing boulder-sized and cobble-sized particles.* Usually, very coarse deposits can be described only in excavations or exposures. They are described as follows:

	Main name	Estimated boulder or cobble content of very coarse fraction
Over 50 % of material is very coarse (over 60 mm)	BOULDERS or BOULDER GRAVEL	Over 50 % is of boulder size (over 200 mm)
	COBBLES or COBBLE GRAVEL	Over 50 % is of cobble size (200 mm to 60 mm)

BOULDERS			> 200mm
COBBLES			> 60 < 200mm
GRAVEL	Coarse		20 - 60mm
	Medium		6 - 20mm
	Fine		2 - 6mm
SAND	Coarse	0.6 - 2mm	
	Medium		0.2 - 0.6mm
	Fine		0.06 - 0.2
SILT	Coarse		0.02 - 2mm
	Medium		0.006 - 0.02
	Fine		0.002 - 0.006
CLAY			< 0.002

NOTE (B) GRAPHIC LOG

Made ground

GRAVEL (GW,GP)

SAND (SW,SP)

SILT (ML,MH)

CLAY (CL,CH)

ORGANIC SOILS (OL,OH,Pt)

THIN LAYER

COMPOSITE/INTERBEDDED - EXAMPLES

SILTY SAND (SM)

SANDY CLAY (CL)

SP CLAY (CL) WITH THIN SAND (SP) INTERBEDS

CLAY (CH) WITH SAND (SC)

DYKE AND FISSURE (BOTH VERTICAL)

NOTE: In all the above cases bedding would have been either observed as, or assumed to be horizontal, unless otherwise stated.

To be used for Peat exclusively

Rocks				
Sedimentary		Metamorphic	Igneous	
	Chalk			
	Limestone	Coarse-grained	Coarse-grained	
Rudaceous	Conglomerate			
Rudaceous	Breccia	Medium-grained	Medium-grained	
Arenaceous	Sandstone	Fine-grained	Fine-grained	
	Siltstone			
Argillaceous	Mudstone			
Argillaceous	Shale			
	Coal			
	Pyroclastic (volcanic ash)			
	Gypsum, Rocksalt etc			

Note that symbols for organic and composite/interbedded examples are modified BS5930 1981 by D. Stapledon. All other symbols are BS5930 1981.

Table 12. Special symbols for borehole record

Fault

Slip surface

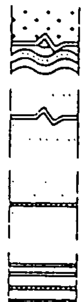
Organic layer*

Marine band

Shell band

(* Not within BS 5930 1981)

Examples



Medium-grained igneous faulted against coarse-grained metamorphic rock

Fault in sandstone

Slip surface in sandstone

Slip surface in shale

NOTE (C) MOISTURE CONTENT

2. DEGREE OF SATURATION OF SANDS — DESCRIPTIVE TERMS

Condition of sand	Criteria	Degree of saturation (%)
Dry	Oven dried. Not usually met in field.	0
Humid	Feels dry, grains "run" freely in bands.	1-25
Damp	Feels cool, slight darkening of colour, grains have slight tendency to adhere to one another.	25-50
Moist	Feels cool, darker colour, grains tend to adhere to one another	50-75
Wet	Feels cold, makes hands wet; should be close to water table.	75-99
Saturated	Below water table, or static water level in excavation or drill holes	100

7.8.4. MOISTURE CONTENT OF CLAY SOILS

Abbreviations	Meaning
MC ≃ LL	Moisture content near liquid limit
MC < LL	Moisture content less than liquid limit
MC > PL	Moisture content greater than plastic limit
MC ≃ PL	Moisture content near plastic limit
MC ≤ PL	Moisture content less or equal to plastic limit
MC < PL	Moisture content less than plastic limit
MC ≪ PL	Moisture content much less than plastic limit

(Taken from AIMM Field Geologists Manual 2nd edition)

NOTE (D) DENSITY

The relative density of sands and gravels may be determined by the standard penetration test. A scale in terms of *N*-values (see BS 1377) is as follows.

Term		SPT *N*-values blows/300 mm penetration	Relative density %
Very loose	(VL)	0 to 4	< 15
Loose	(L)	4 to 10	15 to 35
Medium dense	(MD)	10 to 30	35 to 65
Dense	(D)	30 to 50	65 to 85
Very dense	(VD)	over 50	> 85

Correct for effect of overburden pressure

NOTE (E) CONSISTENCY

Consistency*: For materials which have cohesive properties the following terms are used to describe consistency

		Term		qu unconfined compressive strength (KPa)	Undrained shear strength (KN/m²)
SILT		Soft or loose	Easily moulded or crushed in the fingers.		
SILT		Firm or dense	Can be moulded or crushed by strong pressure in the fingers.		
CLAY	(VS)	Very soft	Exudes between fingers when squeezed in hand.	< 25	less than 20
CLAY	(S)	Soft	Moulded by light finger pressure.	25 to 50	20 to 40
CLAY	(F)	Firm	Can be moulded by strong finger pressure.	50 to 100	40 to 75
CLAY	(St)	Stiff	Cannot be moulded by fingers. Can be indented by thumb.	100 to 200	75 to 150
CLAY	(VSt)	Very stiff	Can be indented by thumb nail.	200 to 400	greater than 150
		Hard*		> 400	

* From Australian Standard 1726-1981 (ref: D.3.2.1)

* Legend colum may be used to indicate pocket penetrometer values (p.p).

NOTE (F) IN-SITU TESTS

2.5 In situ tests

S Standard penetration test (SPT). A 50 mm diameter split spoon sampler is driven 450 mm into the soil using a 65 kg hammer with a 760 mm drop, and the penetration resistance is expressed as the number of blows required to obtain 300 mm penetration below an initial penetration of 150 mm through any disturbed ground at the bottom of the borehole.

In the borehole record, the depth of the test is that at the start of the normal 450 mm penetration.

The number of blows to achieve the standard penetration of 300 mm (the '*N*' value) is shown after the test index letter, but the seating blows through the initial 150 mm penetration are not reported unless the full penetration of 450 mm is not achieved. In the latter case, the symbols below are added to the test index letter:

S* Seating blows only.

S‡ Blow count includes seating blows.

S° No penetration.

S+ Split spoon sampler sank under its own weight.

The test is usually completed when the number of blows reaches 50. For tests achieving the full penetration of 450 mm, the depth at which the test procedure is commenced is given in the depth column on the borehole record, whilst for those tests not achieving full penetration, the depths of both the top and the bottom of the test drive are shown. If a sample is not recovered in the split spoon sampler, a disturbed sample is taken on completion of the test drive. Both are given the same depth as the top of the SP test drive.

C Dynamic Cone Penetration Test (CPT). A test conducted usually in coarse granular soils using the same procedure as for the SPT but with a 50 mm diameter, 60° apex solid cone fitted to the split spoon sampler. Variations in test results are indicated by the same symbols as for the SPT. The bulk disturbed sample taken, is given the same depth as the top of the CP test drive.

V Vane test.

J Borehole jack test. See text of report for full description.

K Permeability test. See text of report for full description.

; Legend column may be used to indicate drill water loss % if applicable.

NOTES (A) - (M) FOR CONDENSED REPORT LOG

NOTE (G) ROCK TYPE

44.1.2 *General description.* Rocks seen in natural outcrops, cores and excavations should generally be described in the following sequence:

- colour;
- grain size;
- texture and structure; (Fabric)
- state of weathering;
- rock name (in capitals, e.g. GRANITE);
- strength;
- other characteristics and properties.

Term	Description
Fresh	No visible sign of weathering of the rock material.
Discoloured	The colour of the original fresh rock material is changed and is evidence of weathering. The degree of change from the original colour should be indicated. If the colour change is confined to particular mineral constituents this should be mentioned.
Decomposed	The rock is weathered to the condition of a soil in which the original material fabric is still intact, but some or all of the mineral grains are decomposed.
Disintegrated	The rock is weathered to the condition of a soil in which the original material fabric is still intact. The rock is friable, but the mineral grains are not decomposed.

44.2.5 *Rock name.* An aid to the identification of rocks for engineering purposes is given in table 9. The table follows general geological practice, but is intended as a guide only; geological training is required for the satisfactory identification of rocks. Engineering properties cannot be inferred from the rock names in the table.

44.3 Description of rock masses

44.3.1 *Introduction.* The description of rock masses requires information additional to the description of the rock material. A rock mass should be described first as a rock material, followed by additional information about discontinuities and other features of engineering significance. Such information includes:

(a) the description of rock types in the mass, with reference to major geological structures;

(b) the dip magnitude and direction, nature, spacing and persistence, width of opening of discontinuities;

(c) details of the weathering profile.

44.3.2 *Structure.* The structure of the rock mass is concerned with the larger-scale inter-relationship of textural features. Common terms should be used where possible. Terms frequently used to describe sedimentary rocks include bedded, laminated; metamorphic rocks may be foliated, banded, cleaved; igneous rocks may be massive, flow banded.

Descriptive terms used for the spacing of these planar structures are as follows.

Term	Spacing
Very thick	greater than 2 m
Thick	600 mm to 2 m
Medium	200 mm to 600 mm
Thin	60 mm to 200 mm
Very thin	20 mm to 60 mm
Thickly laminated (Sedimentary) Narrow (Metamorphic and igneous)	6 mm to 20 mm
Thinly laminated (Sedimentary) Very narrow (Metamorphic and igneous)	less than 6 mm

Spacings can be shown graphically adjacent to "core" column if desired

44.2.2 *Grain size.* A descriptive classification scheme is given in table 9. Grain size refers to the average dimension of the mineral or rock fragments comprising the rock. It is usually sufficient to estimate the size by eye, which may be aided by a hand lens in the assessment of fine-grained or amorphous rocks. The limit of unaided vision is approximately 0.06 mm.

44.2.3 *Texture and fabric.* The texture of a rock refers to individual grains. The arrangement of grains, referred to as the rock fabric, may show a preferred orientation. Terms frequently used include: porphyritic, crystalline, cryptocrystalline, granular, amorphous and glassy.

Table 10. Scale of weathering grades of rock mass

Term	Description	Grade	Legend*
Fresh	No visible sign of rock material weathering; perhaps slight discoloration on major discontinuity surfaces.	I	(Fr)
Slightly weathered	Discoloration indicates weathering of rock material and discontinuity surfaces. All the rock material may be discoloured by weathering.	II	(SW)
Moderately weathered	Less than half of the rock material is decomposed or disintegrated to a soil. Fresh or discoloured rock is present either as a continuous framework or as corestones.	III	(MW)
Highly weathered	More than half of the rock material is decomposed or disintegrated to a soil. Fresh or discoloured rock is present either as a discontinuous framework or as corestones.	IV	(HW)
Completely weathered	All rock material is decomposed and/or disintegrated to soil. The original mass structure is still largely intact.	V	(CW)
Residual soil	All rock material is converted to soil. The mass structure and material fabric are destroyed. There is a large change in volume, but the soil has not been significantly transported.	VI	(RS)

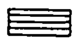
(*Choice of symbol at discretion of user and is not within BS 5930 1981)

44.2.8 *Examples.* The following descriptions are given for guidance in the use of appropriate descriptive terms.

For example, a metamorphic rock material might be described as a 'dark grey, medium grained, thinly foliated, fresh GNEISS, very strong'; a typical description of a sedimentary rock might be a 'yellowish brown, coarse grained, wholly discoloured, micaceous SANDSTONE, moderately weak'; an igneous rock might be described as 'dark greyish green, medium grained, partly discoloured, quartz DOLERITE, extremely strong'.

For strength term see note (H)

In CONDENSED REPORT LOG use the following symbols. Where rock mass is converted to residual soil use appropriate soil symbol.

Fresh to slightly weathered 

Moderately weathered 

Highly weathered 

Completly weathered 

NOTE (H) ROCK STRENGTH

44.2.6 *Strength of rock material.* A scale of strength, based on uniaxial compressive testing, is as follows.

Term	Symbol	Compressive strength, MN/m^2	Approx. Is 50 (MPa) *	Geol. Society (Ref 3) +	Approx kg/cm^2
Very weak	(VW)	less than 1.25	< 0·05	Very weak: Broken by hand with difficulty.	<15
Weak	(W)	1.25 to 5	0·05 – 0·2	Weak: Material crumbles under blows with the sharp end of a geological pick.	15 – 50
Moderately weak	(MW)	5 to 12.5	0·2 – 0·5	Moderately weak: Too hard to cut by hand into a triaxial specimen.	50 – 130
Moderately strong	(MS)	12.5 to 50	0·5 – 2·0	Moderately strong: 5 mm indentations with sharp end of pick.	130 – 500
Strong	(S)	50 to 100	2·0 – 4·0	Strong: Hand held specimen can be broken with single blow of geological hammer	500 – 1000
Very strong	(VS)	100 to 200	4·0 – 8·0	Very strong: More than one blow of geological hammer required to break specimen.	1000 – 2050
Extremely strong	(ES)	greater than 200	> 8·0		

(note · 1MPa ~145psi ~10kg/cm^2)

The strength of a rock material determined in the uniaxial compression test is dependent on the moisture content of the specimen, anisotropy and the test procedure adopted.

* Legend column may be used to indicate point load strength index, Is 50 (MN/m^2). (Correcting for any fabric anisotropy)

\+ Geological Society. Engineering Group Working Party (1977) The Description of Rock Masses for Engineering Purposes. Q Jnl Eng Geol. V-10. pp355 - 388.

Australian Standard AS 1726 - 1981

D5.2 Strength. The following terms are used to describe rock strength:

Rock strength class	*Abbreviation*	*Point load strength index, I_s (50)* (MPa)
Extremely low	EL	< 0.03
Very low	VL	0.03 to 0.1
Low	L	0.1 to 0.3
Medium	M	0.3 to 1
High	H	1 to 3
Very high	VH	3 to 10
Extremely high	EH	> 10

NOTE (I) CORE QUALITY

44.2.7 *Fracture state.*

Elsewhere [161], a determination of Rock Quality Description (RQD) has been proposed as a quantitative measure of the fracture state of rock. RQD is the percentage of rock recovered as sound lengths which are 100 mm or more in length. Only core lengths determined by geological fractures should be measured. Descriptive terms are as follows.

RQD	Term	Suggested symbol *
0 % to 25 %	Very poor	(VP)
25 % to 50 %	Poor	(P)
50 % to 75 %	Fair	(F)
75 % to 90 %	Good	(G)
90 % to 100 %	Excellent	(E)

(* Not within BS 5930 1981)

2.3 Rock core descriptions (Values may be shown graphically in "core" column)

TCR Total core recovery. The length of the total amount of core sample recovered expressed as a percentage of the length of core run

SCR Solid core recovery. The length of core recovered as solid cylinders, expressed as a percentage of the length of core run.

RQD Rock quality designation. The sum length of all core pieces that are 10 cm or longer, measured along the centre line of the core, expressed as a percentage of the core drilled.

NOTE (J) HYDROGEOLOGY

Water cut, standpipe and piezometer installation indicate if sample taken, date and by whom. Show base of standpipe and centre of piezometer in legend column. Show time and dates of water level observations.

NOTE (K) DISCONTINUITY DESCRIPTION

44.3.3.2 *Discontinuity spacing in one dimension.*
The following descriptive scheme should be used.

Term	Spacing	Suggested Symbol *
Very widely spaced	greater than 2 m	(VWS)
Widely spaced	600 mm to 2 m	(WS)
Medium spaced	200 mm to 600 mm	(MS)
Closely spaced	60 mm to 200 mm	(CS)
Very closely spaced	20 mm to 60 mm	(VCS)
Extremely closely spaced	less than 20 mm	(ECS)

(* Not within BS 5930 1981)

44.3.3.3 *Discontinuity spacing in three dimensions.*
The spacing of discontinuities may be described with reference to the size and shape of rock blocks bounded by the discontinuities. Rock blocks may be approximately equidimensional, tabular or columnar in shape. Descriptive terms may be used in accordance with the following.

First term	Maximum dimension
Very large	greater than 2 m
Large	600 mm to 2 m
Medium	200 mm to 600 mm
Small	60 mm to 200 mm
Very small	less than 60 mm

Second term	Nature of block
Blocky	Equidimensional
Tabular	Thickness much less than length or width
Columnar	Height much greater than cross section

The use of these terms requires an understanding of the distribution of discontinuities in three-dimensions; in consequence they cannot be used in the description of drill core.

NOTE (L)

41.4 Geological formation, age and type of deposit. The geological formation should be named where this can be done with confidence, but it may not be easy to tell to which formation a sample belongs, or to locate formation boundaries in a borehole or exposure; conjecture should be avoided.

The characteristic lithology is sometimes indicated in the formation name, e.g. London Clay, but it should be remembered that at a particular location or horizon the lithology may be completely different from that indicated in the formation name.

A term indicating the geological origin or type of the deposit may be given on the map legend, e.g. Made ground, Peat, Head, Alluvium, River terrace, Brickearth, Blown (aeclian) sand, Till. The term can indicate to the engineer some of the characteristics that the deposit may be expected to show.

NOTE (M)

41.5 Additional information. Any additional information on the composition, structure, behaviour or other characteristics of the soil that would be of value in assessing its nature and properties should be recorded. Special note should be made if the properties of the material are considered to be unusual in relation to the rest of its description. Note should also be made if there is doubt whether the sample described is representative of the material at the level from which it was sampled, due, for instance, to fracture of particles or loss of fines during sampling, or to the sample size or borehole diameter being too small in relation to the grading or structure of the material being sampled. Where relevant, it should be made clear whether the sample on which the description is based was disturbed or undisturbed. Where the strength of the soil is likely to vary because of seasonal variations in moisture content, this should be noted.