18 December 2012

LANGLOH DEPOSIT, HAMILTON, TASMANIA

CONCEPT MINING STUDY

Submitted to:
Black Rock Energy Australia Pty Ltd

1.0 REPORT

Report Number. 117621029-001-R-Rev0

Golder Associates

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APPENDIX A
Coal Quality Data

1.0 INTRODUCTION

Black Rock Energy Pty Ltd (BRE) is the holder of EL 27/2008, which includes the Langloh coal deposit, located near Hamilton, Tasmania. The project consists of a thermal coal resource that could be supplied to both domestic and export consumers. A small open cut mine, the Kimbolton Mine, which is owned by the Cornwall Coal Company (CCC), currently operates adjacent to the Langloh project area.

BRE has requested that Golder Associates Pty Ltd (Golder) undertake a study of this deposit addressing geological, mining and infrastructure plans in order to support a mining lease application. It should be noted that a mining lease application in Tasmania requires, at a minimum, a financial and technical capabilities report, a resource statement, a mining study and an environmental study. This report should meet the requirements for a mining study and resource statement. BRE will need to collate the relevant financial documents. A separate environmental study will need to be completed in order to lodge a mining lease application. BRE has not commenced an environmental study at the time of this report. The requirements listed above for a mining lease are believed to be current at the time of this report. All requirements should be directly verified with the relevant Tasmanian Authorities before proceeding.

This report also details the regional and local geology of the deposit including any present and potential geological hazards. Previous exploration programmes within and surrounding the area of interest have been investigated.

A bore hole database has been compiled using both historic and recent data, allowing for the construction of a geological model of the deposit using Maptek Vulcan software. Bore hole coverage across the area appears to be good, but detailed analysis of the bore hole data showed that a significant portion of the exploration data could not be used as it did not meet the criteria for use in calculating a JORC compliant resource. From this model a coal quality analysis has been conducted, and a potential export quality product has been identified.

Based on this model, an estimate of in situ resources has been calculated and reported in accordance with the “Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, The JORC Code” 2004 edition (JORC) guidelines. A mine plan to economically exploit the resource as an export product is also detailed within this report.

Run-of-mine (ROM) coal tonnage has also been estimated, but this is not JORC compliant and has been only used to calculate conceptual production schedules, equipment requirements and financial costings.

A unit based cost analysis consistent with a conceptual level study has been conducted in order to estimate the economic validity of this mine plan. Due to the largely conceptual nature of the study, an accuracy of ± 35% is to be expected.

Furthermore, this report has addressed risk management and mitigation strategies that are appropriate for this level of study. In conjunction with an environmental study, this report will form the basis to proceed to a mining lease application.

All costs within this report are quoted in Australian dollars.

2.0 EXECUTIVE SUMMARY

BRE has requested that Golder Associates Pty Ltd (Golder) undertake a study of this deposit addressing geological, mining and infrastructure plans in order to support a mining lease application. It should be noted that a mining lease application in Tasmania requires, at a minimum, a financial and technical capabilities report, a resource statement, a mining study and an environmental study. This report should meet the requirements for a mining study and resource statement. BRE will need to collate the relevant financial documents. A separate environmental study will need to be completed in order to lodge a mining lease application. BRE has not commenced an environmental study at the time of this report. The requirements listed above for a mining lease are believed to be current at the time of this report. All requirements should be directly verified with the relevant Tasmanian Authorities before proceeding.

2.1 Study Results

Estimated Coal Resources

Based on current drilling information, Golder has estimated that the Langloh deposit contains an estimated 8.1 million tonnes (Mt) of in situ coal resources. Table 2-1 Langloh Estimated Resources, summarises the estimated JORC compliant coal resources.

Table 2-1 Langloh Estimated Resources

Langloh Estimated Coal Resources							
Description	**Coal Resource Tonnes**	**Moisture (% adb)**	**Ash (% adb)**	**Volatile Matter (% adb)**	**Fixed Carbon (% adb)**	**Calorific Value (MJ/kg adb)**	**Sulphur (% adb)**
Measured	5,500,000	4.6	25.7	17.3	52.5	23.8	0.31
Indicated	1,200,000	5.2	28.9	19.2	46.7	24.1	0.32
Inferred	1,400,000	4.9	27.7	18.3	49.0	24.8	0.30
Total	**8,100,000**	**4.7**	**26.5**	**17.7**	**51.0**	**24.0**	**0.31**

Note: Coal resources are estimated in accordance with JORC Code.

As can be seen in Table 2-1, nearly 68% of the resources are within a Measured confidence level. Nearly 83% of the total is of a Measured plus Indicated status.

Life-of-Mine Production Schedule

Within the limits of the coal resources, Golder designed a conceptual pit shell for use in designing a life-of-mine production schedule. Based on this design Golder determined that the Langloh project could have a mine life of eight years and produce a total of 6.7 Mt as shown in Table 2-2 Life-of-Mine Plan Production Schedule. Average annual ROM production is estimated at 910,000 tonnes.

This production may have to be reduced depending on logistics and transport options, as it is not known exactly how much capacity is available on the Tasmanian rail network. Compared to other Australian rail systems, the capacity of the network is fairly low.

Table 2-2 Life-of-Mine Plan Production Schedule

Year	Coal Tonnage (ROMt)	Waste Volume (bcm)	Stripping Ratio (bcm/ROMt)
1	400,000	1,900,000	4.8
2	850,000	4,500,000	5.3
3	910,000	5,500,000	6.0
4	910,000	5,500,000	6.0
5	910,000	5,600,000	6.2
6	910,000	5,600,000	6.2
7	910,000	5,600,000	6.2
8	900,000	5,100,000	5.7
Total	**6,700,000**	**39,300,000**	**5.9**

Equipment Selection

Given the size and shape of the Langloh project combined with annual waste and coal production requirements and resulting stripping ratios, Golder determined that the mining method would be an open cut excavator/truck fleet operation supported by production dozers.

Golder selected an excavator/truck fleet that could accommodate waste removal and coal mining activities. The primary production equipment selected is:

1 – Hitachi EX1900- Hydraulic Backhoe, 12 cu.m. bucket capacity

4 – Caterpillar 777F End Dump Haul Trucks, 91-tonne capacity

3 – Caterpillar D11T Dozers, 611 kW

1 – Driltech D45KS Drill, 152 mm bit diameter

Golder has also estimated capital expenditures for support equipment typically found and required in a mining project of this size. The above indicated equipment manufacturers are listed only to aid in identifying a class of equipment and are not a recommendation on Golder's part as to the selected supplier.

Labour & Mine Operating Schedule

To operate, maintain and supervise the mine, Golder has estimated that approximately 18 operations personnel, 7 maintenance personnel and 10 salaried personnel for a total of 5 employees would be required to sustain operations. For the concept study, it is assumed that the mine would operate 1 shift per day, 12 hours per shift, 6 days per week for a total of 313 days per year.

Transportation Logistics

At this time, the transportation logistics of hauling the product coal from the mine to a port has not been determined. As part of the concept study, Golder has identified three potential alternatives: truck haulage only, truck haulage combined with rail haulage, or truck haulage combined with barging. Most options would eventually transport the coal to the port at Bell Bay on the north-central coast approximately 250 km from the project location. The other option would involve barging the coal down the river to a ship loading point off the coast near Hobart.

Mining Cost Estimates

Given the uncertainty of transportation modes and the resulting transportation costs, Golder chose to present the costs of the project and the coal price to be FOB Mine. That is, costs have been estimated up to placing product coal into trucks at the mine. As discussed below, an estimated coal price FOB Mine was determined given an assumed 12% internal rate of return. All costs within this report are quoted in Australian dollars.

Golder estimated direct mining costs on a unit cost basis for 12 functional cost centres. Total direct mining costs average $23.90/ROM tonne or $4.07/effective bcm. Excavator/truck waste stripping constitutes approximately 31.5% of total direct mining costs; waste drilling and blasting comprises about 20.6%; and coal drilling and blasting, loading and stockpiling comprises 12.0% of mining costs.

Mining indirect costs average $1.80 per ROM tonne over the project life.

Capital and Depreciation Estimates

Golder has estimated total life-of-mine capital expenditures to be $29.9M. Golder has estimated total mining equipment capital of $20.0M; infrastructure capital of $3.3M; $0.53M for EPCM and commissioning of infrastructure; $2.6M for sustaining or replacement capital; and, $3.4M for contingency.

Non-cash depreciation costs total $26.57M over the eight-year project life.

Production Costs

Production costs total $199.8M over the project life, of which 86.7% represent cash costs and 13.3% represent non-cash costs. On a unit cost basis, production costs average $37.27/ROM tonne.

The single largest cash cost is direct mine operating costs, which average $29.87/ROM tonne. Final rehabilitation expenditure is assumed to occur after mining ceases and is estimated at $1.1M.

FOB Mine Coal Price

Golder utilised a 30% income tax rate for the analysis. Golder included working capital at the rate of 45 days of cash operating costs. For the concept study, no salvage value was assessed.

Golder estimated the required product coal revenue (FOB Mine) by back solving based on project DCFROR on a net after-tax basis to be zero or break-even. This results in a required FOB mine coal price of $43.22/product tonne.

2.2 Recommendations

Golder offers the following recommendations:

- BRE should investigate the potential to acquire additional coal resources that are held by CCC, immediately adjacent to the Langloh project in order to increase the total project life.
- The exploration license that includes the Langloh Project is relatively large. The potential of the total area should be investigated and explored. Golder has not been provided with any data to assess the rest of the exploration lease.
- Based on the current amount of drilling information and to further improve the overall geological confidence in the resource base, Golder recommends that BRE conduct additional exploration programmes. The exploration should be designed to identify potential resources that are in areas of limited or no drilling that cannot currently be estimated under the JORC code. Additional exploration is also required in current resource areas to improve coal quality information, accurately define fault locations, and re-drill unreliable historic holes.
- An open hole LOX-line drill program is also recommended as a significant portion of the resource is adjacent to seam outcrop. Therefore a more accurate definition of resources and efficient pit design is dependent on an accurate weathering model which could be obtained from LOX drilling.
- The results of the concept study should enable BRE to begin the process of applying for and acquiring a mining license should BRE desire to do so.
- In conjunction with the mining lease application, BRE should also begin the development of an Environmental Impact Assessment (EIA).

- Assuming BRE is interested in advancing the project Golder recommends that BRE move the evaluation of the Langloh project to the next level of study, pre-feasibility.
- The initial steps of the pre-feasibility study should include an update of the geological model to include all new drilling information acquired from the above recommended programme. As such, the pre-feasibility study should start after the drilling information has been collected.
- Golder recommends that the transportation logistics be further investigated so that the costs of transporting the product from the mine to Bell Bay is estimated and included in the pre-feasibility study.
- Golder recommends that BRE investigate the potential to utilise an existing wash plant located in Fingal if it is determined that the ROM product should be washed. This would be in lieu of capitalising and operating a wash plant on the Langloh site.
- Golder recommends that a market study be evaluated either as part of the pre-feasibility study or as a precursor to the pre-feasibility study.
- Obtain more detailed topography survey as the current model is accurate to 10 m only.

3.0 GEOLOGY

3.1 Regional Geology

Economically viable Tasmanian coal deposits, lie predominantly within the late Triassic coal measures. A lithic sandstone sequence interbedded with mudstone, claystone, coal and minor tuff exists across the central area of the state. This sequence was intruded by dolerite during the mid-Jurassic.

Similarly, the Langloh deposit consists of a dominantly lithic sandstone sequence, interbedded with minor mudstone bands and coal seams, forming part of the Upper Parmeener Super Group. The sequence is said to be of fluviatile origin and Carnian in age. This lithic sandstone sequence is underlain by a quartz sandstone sequence which is devoid of coal. The location of Langloh and other producing coalfields in Tasmania are shown in Figure 3-1 and Figure 3-2.

The coal measures at the Langloh project are contained in a wedge shaped fault block, bounded on the west by a Tertiary graben fault, to the east by Jurassic dolerite, to the north by Tertiary basalt and by outcrops to the south.

Three major seams exist throughout the Langloh deposit, namely Seam A, Seam B and Seam C, each on average about 1 m thick. The coal geology throughout the deposit is lensoidal in nature, hence the seams tend to thin and pinch out in areas. Seam D exists below these three major seams, however, due to inferior coal quality, it has been identified as uneconomical to extract. This is largely due to the limited exploration data showing Seam D to be relatively thin with a thick interburden between Seam C and D. The three major seams are separated by silty mudstone units, with a typical thickness of 0.2 m between seams A and B and approximately 1 m between seams B and C. Small quantities of shale also exist in amongst the seams in various areas. Several other very thin (<0.1 m) seams exist below Seams A, B and C with thick interburdens between these minor seams.

Seams A and B remain relatively close together throughout the deposit, whilst seam C splits away to the north-west, thins to the west and appears to degrade to a carbonaceous shale to the northeast.

Faulting has been identified in areas surrounding drill holes H30 and H31 which may be related to the major graben fault or igneous intrusions. Other geological structures may also be present in the area surrounding drill hole H05, but more drilling is required to confirm this. The exact location of the faulting is unknown and more drilling is required to identify fault locations as it appears that no coal exists on the outside of the faults indicating a definite boundary to the resource area.

3.2 Stratigraphy

Figure 3-3 Stratigraphic Column shows the typical stratigraphy over the Langloh Deposit. Generally speaking, Seams A and B are relatively close together and are often only 0.5 m apart. Seam C is also usually close to A and B, but can split away suddenly in parts of the deposit. Seam D is not considered to be economic due to thick interburden and the seam being thin. Several other thin seams have been logged, especially in the older exploration programs, but all are considered to be too thin to be economic.

Figure 3-1 Project Location Map

Figure 3-2 Project Location – Local Area

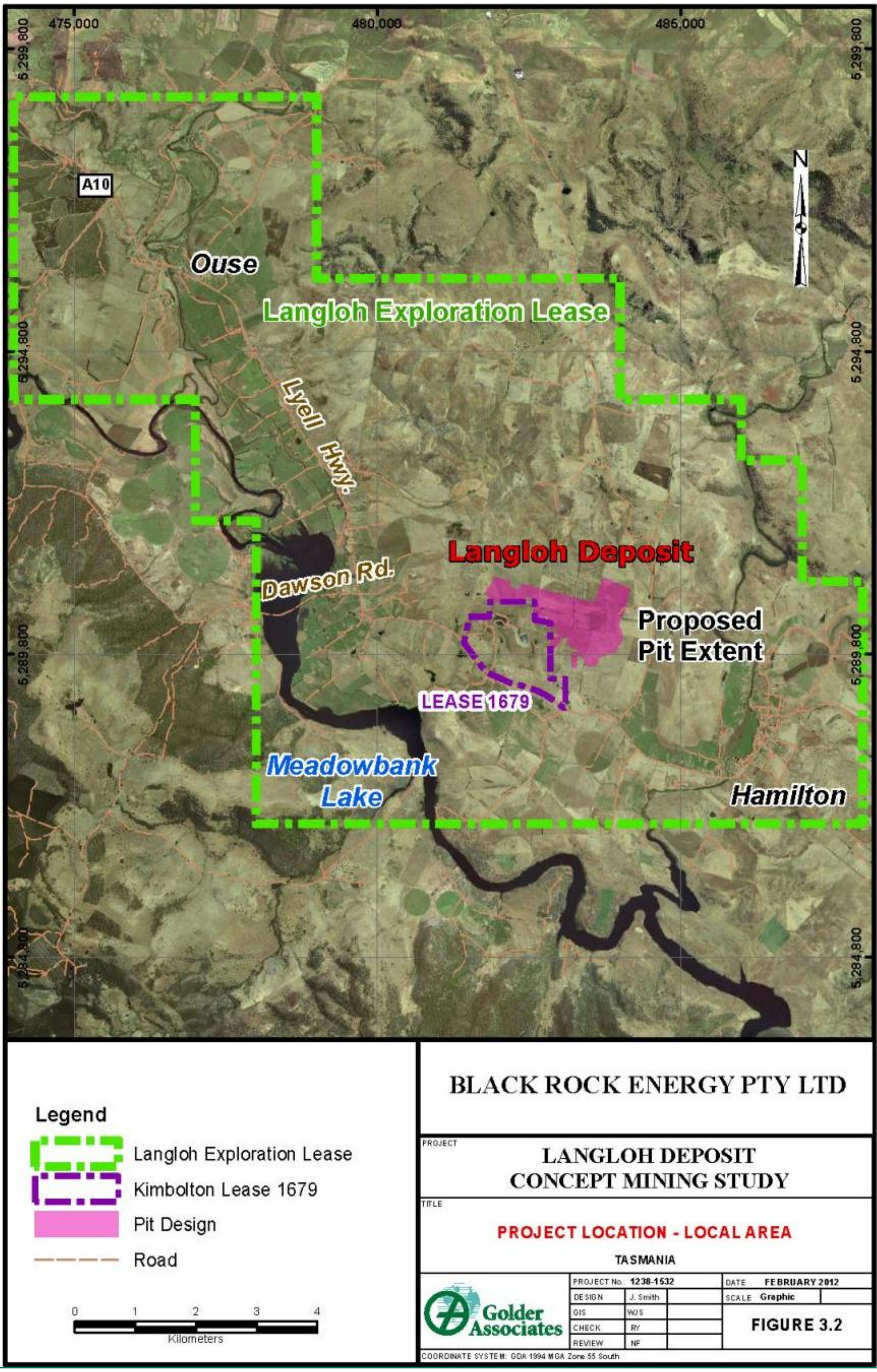
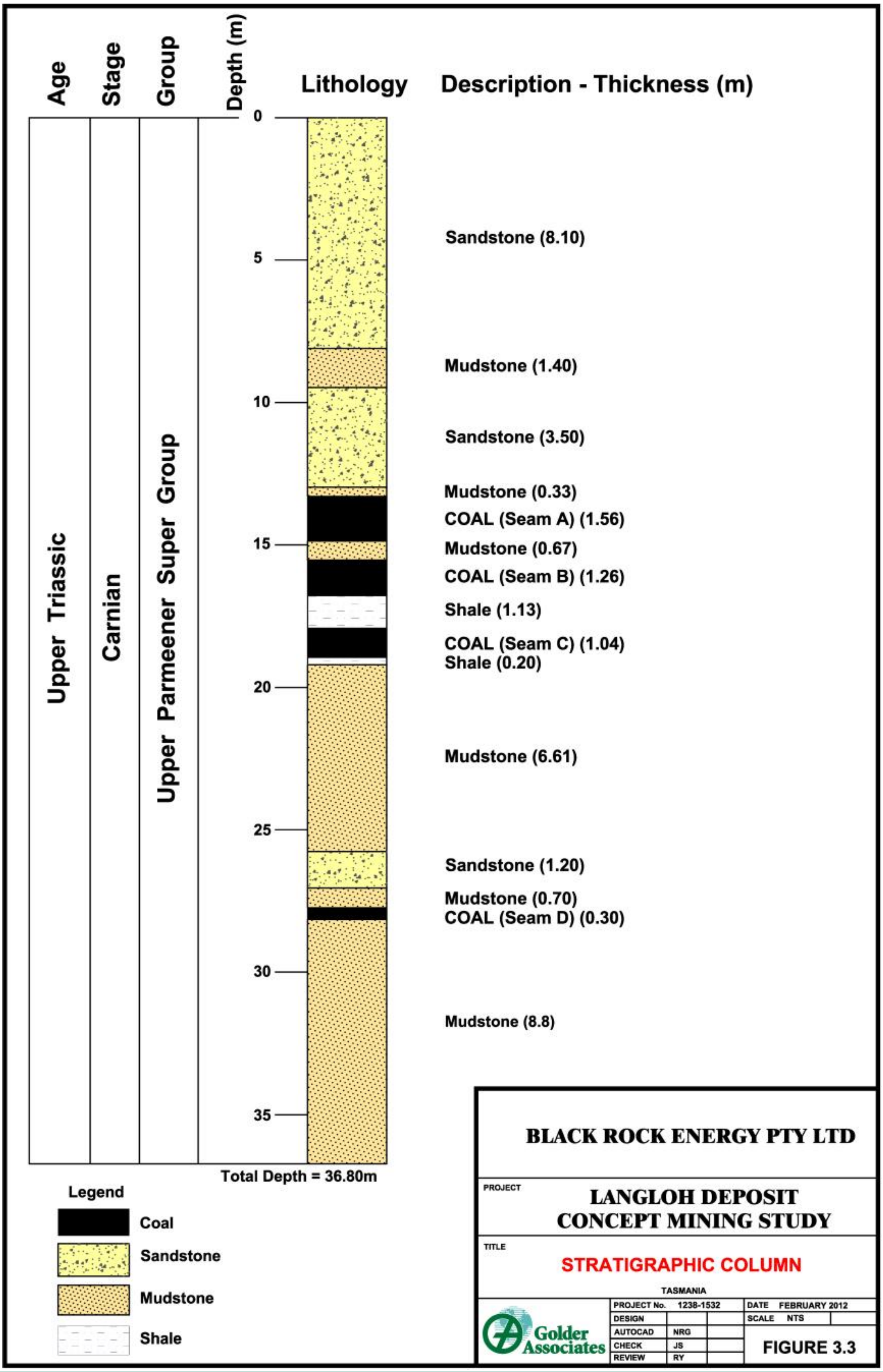


Figure 3-3 Stratigraphic Column



3.3 Previous Exploration and Drilling

Coal was known from outcrops in the Hamilton area from around 1855. Coal was subsequently located in a well sunk at the Langloh homestead, which was then enlarged into a small mine shaft. The first Government report on the area was prepared by Thureau in 1883. He recommended the area be drilled to determine the quality and quantity of coal present. At the time of Thureau's visit, small scale mining was conducted from shallow shafts and the coal was used for domestic and other purposes. Montgomery (1894) prepared a Government report on the coalfield and reported on four exploration holes that had been drilled in the area. He commented favourably on the results of trials of the coal in steam-raising for industrial use. Hills (1922) reported in detail on the geology, coal quality, resources and economics of coal mining in the Lawrenny (Langloh) area. The above reports all record the three main seams present; Seams A, B and C.

3.3.1 Langloh (Lawrenny) Mine

The Langloh Coal Mining Company commenced operations at the Langloh (Lawrenny) Colliery in 1938. The lease was transferred to the Hamilton Coal Company in 1942 and mining continued until 1963. The operation was a shallow underground mine accessed initially by shaft and subsequently by a dip drive. Coal was mined from the upper two seams present in the area. Mining was conducted via the bord and pillar method and only first workings were developed. Coal was supplied to local industry, the railways, and domestic users. The mine was a small operation employing between four and twelve workers, producing between 2,000 and 8,000 tonnes of coal per year. Ground and surface water entering the mine reacted with montmorillonite rich mudstone in the roof and floor of the seams producing difficult mining conditions for most of the mine's life.

3.3.2 Capricorn Mining

Exploration Licence EL 27/79 covering 870 square kilometres including the Langloh deposit was acquired by Capricorn Mining Ltd (Capricorn) on 16th October 1979. Capricorn was a subsidiary of Zanex Ltd and exploration was conducted by General Geological Services (GGS) and Petrecon Australia Pty Ltd (Petrecon). Capricorn conducted a detailed evaluation of the resource with the objective of establishing a moderate sized operation supplying the Tasmanian domestic coal market. The work conducted included regional mapping, exploration drilling, coal quality testing, float-sink testing, maceral analysis, palynology studies, environmental impact study, hydrology testing, mining study, bulk sampling and coal combustion testing.

Capricorn applied for a retention lease RL1/1989 covering 31 km^2 over the Langloh area in May 1989 and subsequently applied for RL2/1990 covering 14 km^2. The application stated Capricorn had been unable to find a coal market large enough to justify mine development and attempts to reach agreement with landowners at Langloh had been unsuccessful. The retention lease was revoked in November 1991.

An exploration programme was planned by McElroy Bryan Consultants in 1981. Three separate programmes were conducted in April/May 1981, May 1982 and January/February 1983, and 28 drill holes were completed. The location of the drill holes are shown in Figure 3-4.

Drill holes were a combination of open hole and NQ coring conducted by Fox 40L and Fox B80 truck mounted drill rigs. Most drill collars were surveyed and are shown in Table 3-1 Drill Hole Locations and Collars. A more detailed explanation on the treatment of drill hole collars and locations can be found in Section 3.5. Coal maceral studies and a palynology study were conducted. In situ coal resources were estimated and a saleable coal reserve estimate was prepared. The drill programme demonstrated the continuity of the three known coal seams, A, B and C, across much of the area and identified some faulting within the deposit.

Figure 3-4 Langloh Drill Hole Location Map

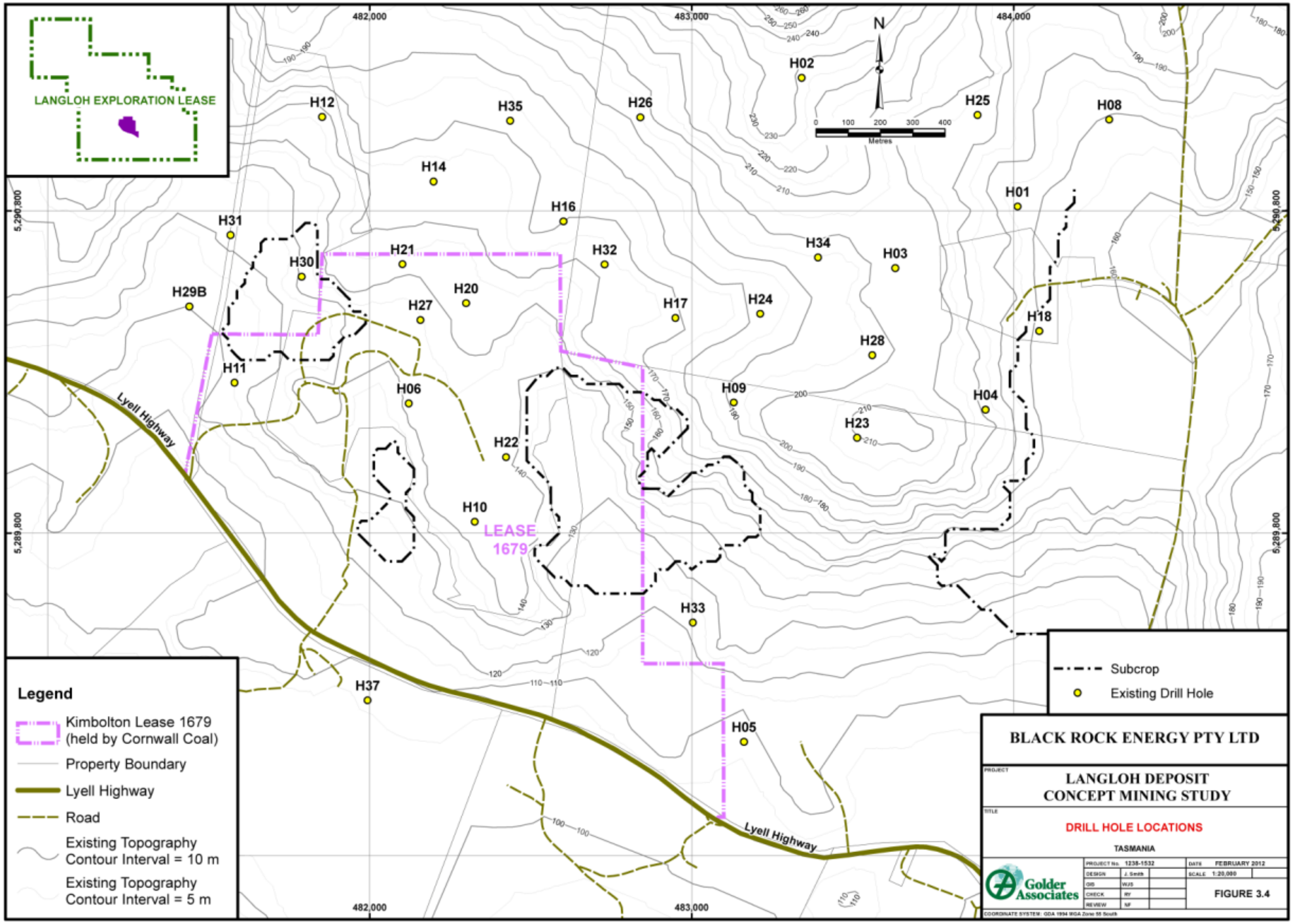


Table 3-1 Drill Hole Locations and Collars

Hole	Easting	Northing	RL	Depth	Notes
H01	484,550.0	5,290,550.0	-	44.4	See section 3.5.2, converted from AMG
H02	483,360.0	5,290,740.0	-	78.4	See section 3.5.2, converted from AMG
H03	483,860.0	5,290,410.0	-	84.4	See section 3.5.2, converted from AMG
H04	483,950.0	5,290,060.0	-	45.1	See section 3.5.2, converted from AMG
H05	482,820.0	5,289,170.0	-	135.5	See section 3.5.2, converted from AMG
H06	482,000.0	5,290,060.0	-	157.1	See section 3.5.2, converted from AMG
H08	484,214.0	5,290,833.0	165.5	21.0	Converted from AMG
H09	483,131.0	5,290,204.3	191.9	30.1	Converted from AMG
H10	482,327.0	5,289,834.3	145.9	27.0	Converted from AMG
H11	481,581.0	5,290,266.3	124.5	27.0	Converted from AMG
H12	481,853.0	5,291,090.3	156.2	69.0	Converted from AMG
H14	482,199.0	5,290,890.3	158.0	57.4	Converted from AMG
H16	482,602.0	5,290,766.3	168.4	57.5	Converted from AMG
H17	482,950.0	5,290,466.3	166.4	26.1	Converted from AMG
H18	484,079.0	5,290,426.3	165.2	21.0	Converted from AMG
H19	481,890.0	5,290,255.3	138.8	60.0	Converted from AMG
H20	482,300.0	5,290,512.3	151.4	30.0	Converted from AMG
H21	482,102.0	5,290,633.3	150.9	27.0	Converted from AMG
H22	482,424.0	5,290,035.3	139.1	20.9	Converted from AMG
H23	483,514.0	5,290,095.3	210.4	36.0	Converted from AMG
H24	483,213.0	5,290,479.3	180.3	27.0	Converted from AMG
H25	483,888.0	5,291,096.3	195.4	45.0	Converted from AMG
H26	482,841.0	5,291,089.3	192.3	89.0	Converted from AMG
H27	482,158.0	5,290,460.3	141.7	25.0	Converted from AMG
H28	483,561.0	5,290,351.3	200.1	39.0	Converted from AMG
H29B	481,439.9	5,290,502.4	120.0	43.8	Converted from AMG
H30	481,788.9	5,290,594.6	140.4	38.0	Survey ok
H31	481,568.2	5,290,724.0	133.7	59.0	Survey ok
H32	482,729.6	5,290,633.4	162.6	42.6	Survey ok
H33	483,004.1	5,289,521.9	120.9	50.0	Survey ok
H34	483,393.3	5,290,654.8	194.1	47.1	Survey ok
H35	482,436.7	5,291,079.1	171.2	77.1	Survey ok
H36	481,170.6	5,289,954.5	107.8	69.9	Survey ok
H37	481,994.2	5,289,280.9	105.5	102.7	Survey ok
H38	482,890.4	5,288,873.2	95.2	70.0	Survey ok

3.3.3 Spitfire Resources

Spitfire Resources acquired the Langloh exploration licence in 2008. Spitfire Resources engaged Marston International Pty Ltd (Marston) to complete a drilling program in 2010 for drill holes H29 to H38.

This drilling program consisted of 7 cored holes and 3 open holes with the following objectives:

- Obtain samples for confirmation coal quality testing
- Test for extensions of the deposit to the west and south.

Although Marston did not produce a geological model of the deposit the following conclusions can be made regarding the extent of resources:

- The coal seams do not extend to the south across the Lyell highway
- The coal seams extend further to the west but deteriorate in quality and thickness
- There are three seams present Seams A, B and C varying from 0.2 – 3.7 m thick
- Seam B has the lowest average ash content
- Most of the thicker coal bearing sections with 3 – 4 m total coal lie within the Cornwall Coal Company Lease
- Overall the program did not identify any significant new resources.

Coal quality testing confirmed the historically reported coal quality data for Langloh.

Holes H29, H30 and H31 were drilled along the western margin of the resource to test for extensions to the resource. While the holes demonstrated the coal extends to the west below the ferricrete, they indicate that the seams are deteriorating in both thickness and quality. Only Seam C is present as coal while the other seams have deteriorated into carbonaceous shale and mudstone. Hole H33 in the southeast of the area indicates the coal measures subcrop in this area and the lack of coal in holes H36, H37 and H38 confirms the seams do not extend south of the Lyell Highway.

The locations and depths of the drill holes are summarised in Table 3-1.

3.3.4 Geophysics

Most holes were logged by either: Mitre Geophysics Pty Ltd (H01-H07), BPB Instruments Pty Ltd (H08-H28) or Groundsearch (H29 to H38). Logged holes are listed in Table 3-3. The sondes used by the companies were generally; resistivity, calliper, long spaced density, short spaced density and natural gamma.

3.4 Coal Quality Analysis

The Coal quality analysis over the Langloh deposit is described in detail in Section 3.6 as geological modelling allows for a more accurate estimation. Consequently information provided in this section should be regarded as a first pass analysis of quality.

Coal samples were tested at SGS Laboratories Pty Ltd (SGS) in Sydney and Newcastle. Test work included proximate analysis, total sulphur content, calorific value, relative density, and float-sink testing. Unfortunately much of the results of the test work conducted by Capricorn Mining cannot now be used as the coal bearing intervals were either incompletely sampled and/or the sample depths were not reported. The coal quality testing results are summarized in Appendix A, Coal Quality Data.

SGS commented on the test results in December 1981: “The coal seams intersected to-date are typically high ash resulting mainly from finely disseminated mineral matter, which after washing, could be expected to yield coal with about 20% to 25% ash. Volatile matter appears variable from over 30%, calculated dry ash free (daf), to less than 10% (daf). This variation results from the effect of dolerite intrusions (or flows) on the coal seams.”

“In general your coal is similar to other known Tasmanian coal. It’s likely use is a fuel coal for either power generation of cement kilns. Clearly poorer quality (high ash) coals are better suited to any local markets, but as these are limited at present in Tasmania, the next likely areas to pursue are cement plants in South-East Asia and Korea.”

An analysis was conducted to determine the as received energy content of product coal, allowing for a more accurate cost analysis. As Total Moisture (TM) values were not analysed in either of the Langloh laboratory programs, a benchmarked average value of TM was used in order to perform change of basis calculations for ash content from an air dried basis to an as received basis.

A TM value of 5.6% taken from a similar Tasmanian Project was used as a default moisture. This value is considered appropropriate for the following reasons:

- Sourced from a Tasmanian coal exploration project to the northeast of the Langloh deposit
- Similar geology characteristics to the Langloh Deposit
- Similar coal quality characteristics to the Langloh Deposit

Fundamental data tests were also performed such as plotting Relative Density versus Ash Content and Calorific Value versus Ash Content regression curves. These curves allowed for the verification of data correlation and for the detection of any outliers.

The Calorific Value vs Ash content regression analysis provided a formula that was used to approximate CV (arb) from the previously calculated Ash (arb) values as not all laboratory analysis included CV (arb).

From these basic analyses a first pass coal quality estimation of the deposit was made. Table 3-2, shows the approximate average in-situ coal quality, measured on an air dried basis, across the Langloh deposit.

Table 3-2 Average In Situ Coal Quality

Average In Situ Quality (adb)	
Moisture (%)	4.7
Ash (%)	26.5
Vol Matter (%)	17.7
CV (kcal/kg)	5,727
Sulphur (%)	0.3

Blending and selective mining are indicated to produce a potentially marketable product. This will require further more detailed coal quality and mining studies.

3.5 Geological Modelling Methodology

3.5.1 Historical Data

As mentioned in Section 3.3 all relevant data used during compilation of the geological model was gathered over a range of drilling programs. Consequently, validation of historical data was crucial to the project's success. Golder has concluded that a number of data points be amended or excluded, due to unreliable geophysics, quality data and technical reasons. This is discussed in greater detail in the following sections of the report.

In construction of the borehole data base, Golder has utilised a variety of historical resources including:

- Capricorn Mining Ltd Annual and quarterly reports
- Coal quality analysis reports produced by the Australian Mineral Development Laboratories (AMDEL) and SGS Australia Pty Ltd
- Float-Sink data by SGS Australia Pty Ltd
- Geophysical logging data and reports by BPB Ltd
- Geophysical logging data and reports by Mitre Geophysics Pty Ltd
- Geophysical logging data and reports by Groundsearch Australia Pty Ltd
- Lithological logging data by Petrecon Australia
- Lithological logging data by Marston International Pty Ltd.

Historical data from these sources was collated, cross referenced and verified, as described in the sections below, in order to produce the projects final geological model.

3.5.2 Drill hole Database

As a prerequisite to building the final geological model for the Langloh deposit a borehole database was constructed to compile all available data. Since BRE requested that a JORC compliant resource be estimated, the borehole database had to be constructed in line with JORC requirements.

The quality and reliability of borehole location data is highly variable for each drilling program. The two drilling programs covering drill holes H19 to H38 are the only programmes that have a surveyors report for location and collars. Locations for drill holes from H9 to H18 are sourced from an exploration report, but there is some variation between reported locations and plotted locations on plans. The level of accuracy for drill holes H09 to H18 was accepted as satisfactory for a conceptual study with the collars of the holes matching reasonably well to the topography model.

The reported locations of drill holes H01 to H08 were initially deemed questionable. The drill holes did not match the predicted geology (especially drill holes that should have been within crop zones) or the plotted locations on exploration plans. Instead of discarding H01 to H08, Golder digitised the locations of the drill holes as shown on the exploration plans. This moved the drill holes to locations that agreed with the predicted geology quite well. Collar elevations were not provided for drill holes H01 to H08 and have been estimated from the topography model. For JORC purposes, H01 to H08 have been excluded from resource category boundary calculations (i.e., they do not influence the Measured, Indicated or Inferred boundaries) due to uncertainty over drill hole locations. However, these drill holes have still been left in the model as they increase the level of geological confidence for Langloh and do not cause the resource to significantly increase.

Seam data was sourced from original laboratory reports, and was provided in both ply-by-ply and composited formats. Most structural data, including geophysical and lithological logs, and quality data, including proximate, ultimate and ash analysis as well as calorific value (CV) and total sulphur (TS) data was readily available.

However, certain quality samples needed to be composited into a whole seam sample, especially in regards to Seam C in drill holes H29B - H38. This was done to enable the data from H29 to H38 to be directly used with the data from H01 to H28 which was mostly provided on a full seam basis. This allowed for creation of a complete composited database detailing all available coal qualities in each drill hole and seam. A small number of roof and floor quality samples were also available. Due to the relatively thin nature of partings within the seams, it is reasonable to assume at this stage that the seams would be mined as a whole seam composite. Due to the variable and sometimes quite high ash of Seam A, ply-by-ply analysis of Seam A may assist in identifying individual high ash plys that could be wasted to reduce the overall ash of the seam rather than potentially wasting the entire seam in some areas to lower the overall ash at Langloh. It would appear that there is currently no ply-by-ply samples for Seam A so the potential for selectively mining the seam to reduce overall ash is unknown.

The constructed database was then directly imported into Maptek Vulcan software allowing for 3D geological modelling of the Langloh project. All further project analysis, including mine planning, scheduling and resource estimates were based upon this model. A summary of which boreholes were used in the model is shown in Table 3-3 and Table 3-4.

Table 3-3 Summary of Drill Hole Analysis for Model - Structure

Hole Number	Location	Model Data	XYZ Co-ords	Lithology	Geophysics
H-01	Langloh	Modelled	Yes	Yes	Yes
H-02	Langloh	Modelled	Yes	Yes	Yes
H-03	Langloh	Modelled	Yes	Yes	Yes
H-04	Langloh	Modelled	Yes	Yes	Yes
H-05	Langloh	Modelled	Yes	Yes	Yes
H-06	Langloh	Modelled	Yes	Yes	Yes
H-07	Langloh		Yes	Yes	No
H-08	Langloh	Modelled	Yes	Yes	No
H-09	Langloh	Modelled	Yes	Yes	Yes
H-10	Langloh	Modelled	Yes	Yes	Yes
H-11	Langloh	Modelled	Yes	Yes	Yes
H-12	Langloh	Modelled	Yes	Yes	Yes
H-13	Langloh		No	Yes	No
H-14	Langloh	Modelled	Yes	Yes	Yes
H-15	Langloh		Yes	Yes	No
H-16	Langloh	Modelled	Yes	Yes	Yes
H-17	Langloh	Modelled	Yes	Yes	Yes
H-18	Langloh	Modelled	Yes	Yes	No
H-19	Langloh		Yes	Yes	Yes
H-20	Langloh	Modelled	Yes	Yes	Yes
H-21	Langloh	Modelled	Yes	Yes	Yes
H-22	Langloh	Modelled	Yes	Yes	Yes
H-23	Langloh	Modelled	Yes	Yes	Yes
H-24	Langloh	Modelled	Yes	Yes	Yes
H-25	Langloh	Modelled	Yes	Yes	No
H-26	Langloh	Modelled	Yes	Yes	Yes
H-27	Langloh	Modelled	Yes	Yes	Yes
H-28	Langloh	Modelled	Yes	Yes	No
H-29B	Langloh	Modelled	Yes	Yes	Yes
H-30	Langloh	Modelled	Yes	Yes	Yes
H-31	Langloh	Modelled	Yes	Yes	Yes
H-32	Langloh	Modelled	Yes	Yes	Yes
H-33	Langloh	Modelled	Yes	Yes	Yes
H-34	Langloh	Modelled	Yes	Yes	Yes
H-35	Langloh	Modelled	Yes	Yes	Yes
H-36	Langloh		Yes	No	Yes
H-37	Langloh	Modelled	Yes	No	Yes
H-38	Langloh	Modelled	Yes	No	Yes

Table 3-4 Summary of Drill Hole Analysis for Model – Quality

Hole Number	Quality Data	Float/Sink Data	Quality Modelled	Comments
H-01	prox	Yes	Yes	Collar adjusted to topography
H-02	No	No	No	No coal, collar adjusted to topography
H-03	prox	Yes	Yes	Collar adjusted to topography, quality data amended
H-04	prox	Yes	Yes	Collar adjusted to topography
H-05	prox	No	Yes	Collar adjusted to topography, fault
H-06	prox	Yes	Yes	Collar adjusted to topography
H-07	prox	Yes	No	No geophysics
H-08	No	No	No	No coal, no geophysics, dolerite @17m
H-09	prox, CV, TS	Yes	Yes	
H-10	prox, CV,TS	Yes	Yes	
H-11	No	No	No	
H-12	prox, CV,TS	Yes	Yes	
H-13	No	No	Yes	No coal, not surveyed, no geophysics
H-14	prox, CV,TS	Yes	Yes	
H-15	No	No	No	No geophysics, no quality data
H-16	prox, CV,TS	Yes	Yes	No geophysics for seam C
H-17	prox, CV,TS	Yes	Yes	
H-18	No	No	No	No geophysics - most likely outside crop zone
H-19	No	No	No	No quality data, unreliable geophysics
H-20	prox, CV,TS	No	No	No sample depths recorded
H-21	prox, CV,TS	No	No	No sample depths recorded
H-22	prox, CV,TS	No	No	No sample depths recorded
H-23	prox, CV,TS	No	No	No sample depths recorded
H-24	prox, CV,TS	No	No	No sample depths recorded, no geophysics for seam C
H-25	No	No	No	No sample depths recorded, no logs, no coal
H-26	No	No	No	No sample depths recorded
H-27	No	No	No	No sample depths recorded
H-28	prox, CV,TS	No	No	No sample depths recorded, no geophysics
H-29B	prox, ult, ash,CV,TS	Yes	Yes	
H-30	prox, ult, ash,CV,TS	Yes	Yes	Seam C weathered, quality removed from model
H-31	prox, ult, ash,CV,TS	Yes	Yes	
H-32	prox, ult, ash,CV,TS	Yes	Yes	
H-33	prox, ult, ash,CV,TS	Yes	Yes	
H-34	prox, ult, ash,CV,TS	Yes	Yes	
H-35	prox, ult, ash,CV,TS	Yes	Yes	
H-36	None	No	No	Unreliable geophysics
H-37	None	No	No	
H-38	None	No	No	No Coal - most likely outside crop zone

3.5.3 Data Validation

In order to produce reliable results, Golder has ensured that all data has been verified and validated, in instances where this was not possible, under Golder's discretion data has been amended or removed entirely from the geological model. This is outlined in the following sections of the report. The following steps were taken to ensure validation of data:

- Correlation of original geophysical logs and lithological logs. Where required, the lithological log depths were corrected by manual picking of the geophysical logs, taking into account short spaced density, long spaced density, natural gamma and calliper data where possible
- Verification of drill hole coordinates by reference to historical and current maps of the Langloh site

- Coal quality was validated thoroughly via cross checking the database with historical lab reports before data was entered into Vulcan software
- Digital topographic data was sourced from the Department of Primary Industries, Parks, Water and Environment (DPIPWE), Tasmanian Government. This included 10 m contours, transport segments, hydrogrpahy and cadastral boundaries over the area of interest.
- While data from the Marston exploration program could be verified by reference to orginal reports, this could not be done for all of the Capricorn Mining data which was taken from statutory reports.

3.5.4 Amended Data

A number of bore holes or data points were amended due to the variation in reliability of the first historical drilling program (H01 to H08). Section 3.5.2 discussed the treatment of H1 to H8 in detail. Golder has undertaken care in the alteration of any historical data to ensure no adverse effects to the project's outcome. The following changes have been made:

- Historical drill hole co-ordinates were provided in ISG/AMG format. These were converted to current MGA 94 coordinates before importing into Vulcan software
- Drill holes H01 – H08 did not have an identified collar elevation. Consequently, the collar elevation was adjusted to topographic data obtained from the DPIPWE
- The reported locations for drill holes H01 to H08 do not match the predicted geology or the plotted locations on exploration report plans. The modelled locations for drill holes H01 to H08 have been derived from exploration report location plans
- Raw relative density (RD) was calculated based on a ash content vs RD regression curve for holes H01, H03, H06, H09, H10, H12, H14, H16, H17. This was deemed acceptable due to the high correlation of ash and RD as seen in all other holes
- Golder has determined reported coal qualities for composite Seam A in H03 were incorrect due to the composite quality being unrealistically variable from the ply-by-ply analysis. Original ply-by-ply data was used to calculate the approximate percentage of parting bands within the seam. Conservative assumptions of 100% ash content, 2.00 RD, and 5.2% moisture content (adb) were adopted for these partings, before re-composition of the seam
- Available quality data from roof and floor samples were averaged, and applied across the entire deposit on a seam by seam basis. It should be noted that relatively few samples were available, so conservative estimates were used.

3.5.5 Excluded Data

Certain aspects of the drill hole data were removed due to unreliable historical data. Excluded data is listed below:

- Drill hole H07 was not surveyed and after an inspection of exploration plans, it was found that this hole is in an area that is currently deemed uneconomic due to sparse drilling and apparently deep seams. If more drilling is completed in the southeast corner, H07 should be included in the assessment of this area
- Drill hole H13 was not surveyed and has been approximately located within the western Tertiary ferricrete area. The model has been limited in the west by the tertiary ferricrete boundary
- Drill hole H15 had no geophysics or coal quality data and was removed from the model
- Drill holes H19 and H36 had unreliable geophysics hence were removed from the model
- Drill holes H19 to H28 did not have sample depths specified for quality data, hence this quality data was removed from the model.

3.6 Geological Modelling Results

3.6.1 Summary

Golder constructed a geological model for the deposit that was used to evaluate the structure and coal quality of the deposit, to estimate resources and to prepare mine designs and production schedules. The model was constructed using Maptek Vulcan software. As outlined in previous sections of this report, the majority of data was sourced from historical exploration reports with varying degrees of reliability.

3.6.2 Structure Model

Golder constructed a relatively simple structure model for the deposit as there were only four coal seams, and the majority of the lithology and quality data was not sufficiently detailed for a ply-by-ply analysis. Golder adopted the existing Seam A, B, C and D correlation which appears to work well as seams A, B and C appear to be relatively consistent and have reasonably distinctive geophysical characteristics.

The bore hole data base was checked via the Vulcan validation function for:

- Unique locations
- Collar RL to topography model
- Total depths match total sample depth
- Intervals do not overlap
- To and from fields are always increasing.

The validated database was used to create the Vulcan model. The drill hole intercepts and thickness data for each seam are shown in Table 3-5 Summary of Drill Intercepts and Modelled Thickness.

Table 3-5 Summary of Drill Intercepts and Modelled Thickness

Seam Name	Number of Intercepts	Average Thickness (m)	Minimum Thickness (m)	Maximum Thickness (m)
A	25	0.9	0.3	2.7
B	25	0.6	0.2	1.6
C	23	0.5	0.1	1.8
D	9	0.2	0.02	0.7

The data shows that the majority of drill holes intersect Seam A, B and C.

Due to seams A, B and C having the same relatively good coverage across the deposit, these seams were modelled using the normal interpolation method in Vulcan. Due to poor data coverage, Seam D was modelled to pinch out and to be stacked from the base of Seam C in areas of poor data. This method was chosen so Seam D would follow the structure of Seam C to prevent it from being unrealistically flat from only 9 data points.

Each model grid was constructed using 25 x 25 m grid cell spacing. Each grid was modelled using minor smoothing and trending to best estimate the structural geology. Base of weathering was chosen from limited and variable data to be 9 m from surface. More detailed base of weathering data is required to accurately model or estimate the weathering profile.

Modelled seam floor structure can be seen in the following figures:

- Figure 3-6, Seam A Floor Structure
- Figure 3-7, Seam B Floor Structure
- Figure 3-8, Seam C Floor Structure

Modelled seam thickness isopachs can be seen in the following figures:

- Figure 3-9, Seam A In Situ Coal Thickness Isopachs
- Figure 3-10, Seam B In Situ Coal Thickness Isopachs
- Figure 3-11, Seam C In Situ Coal Thickness Isopachs

Cross sections of the Langloh deposit can be seen in Figure 3-12.

Figure 3-5 Seam A Floor Structure

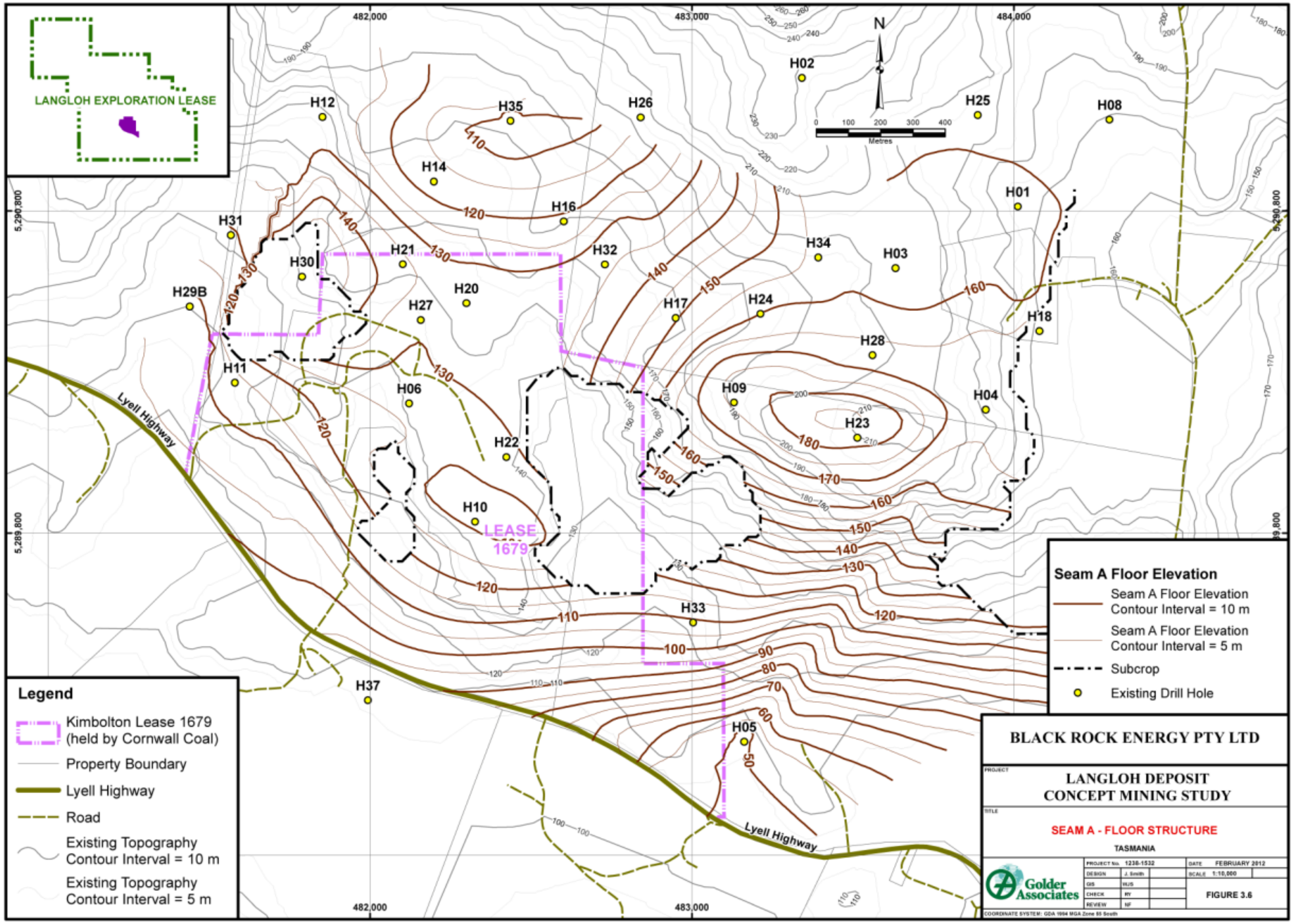


Figure 3-6 Seam B Floor Structure

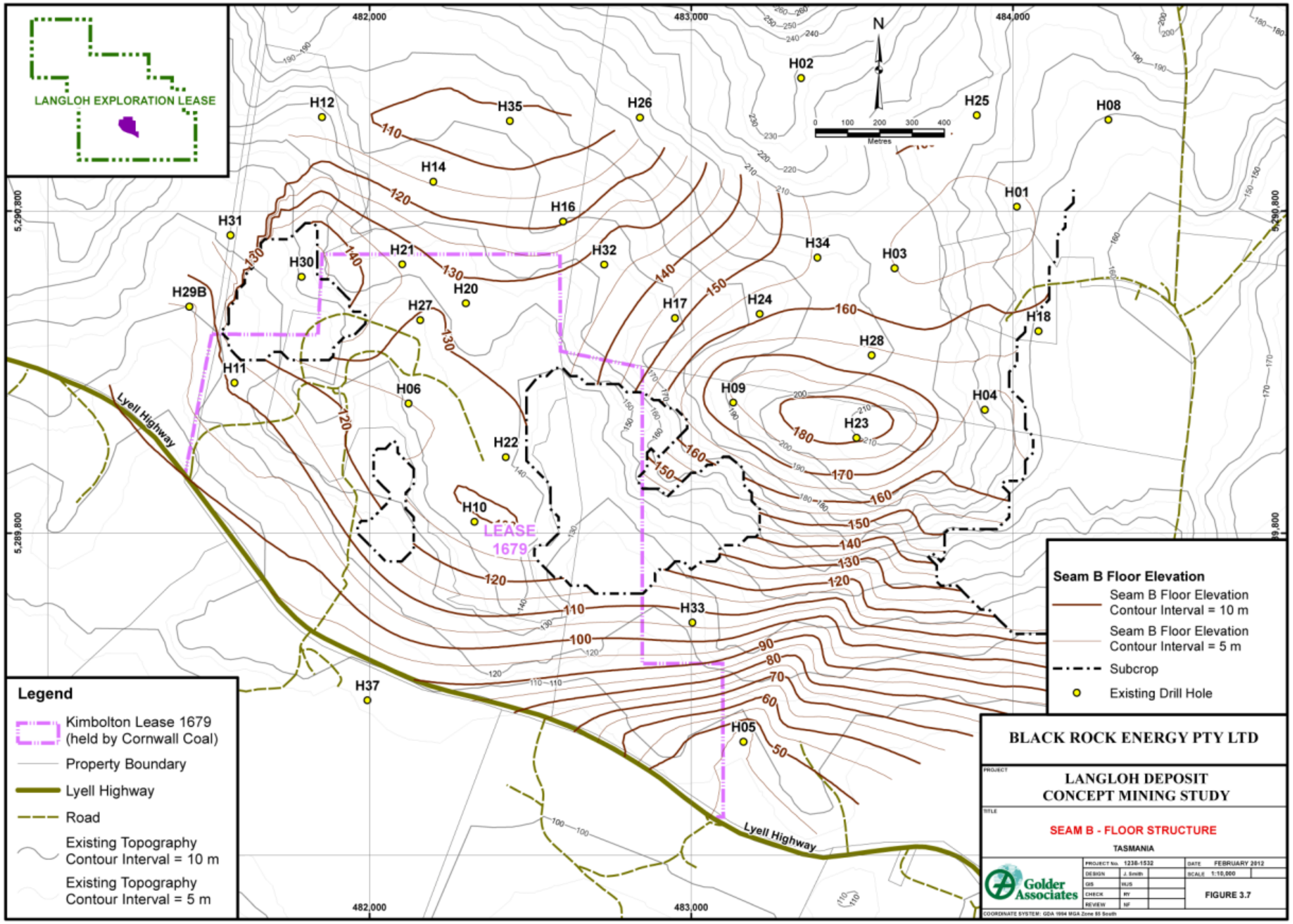


Figure 3-7 Seam C Floor Structure

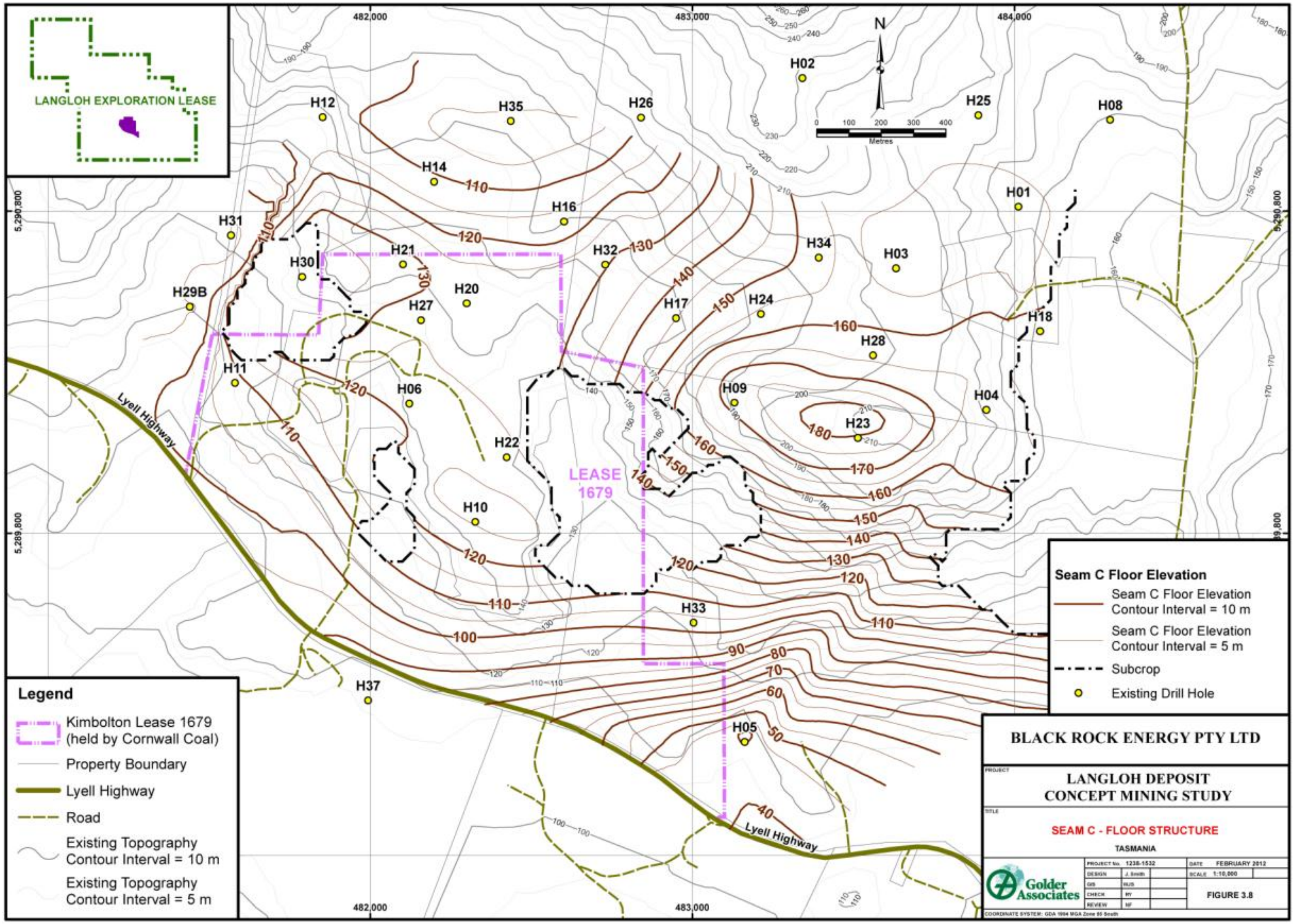


Figure 3-8 Seam A In Situ Coal Thickness Isopachs

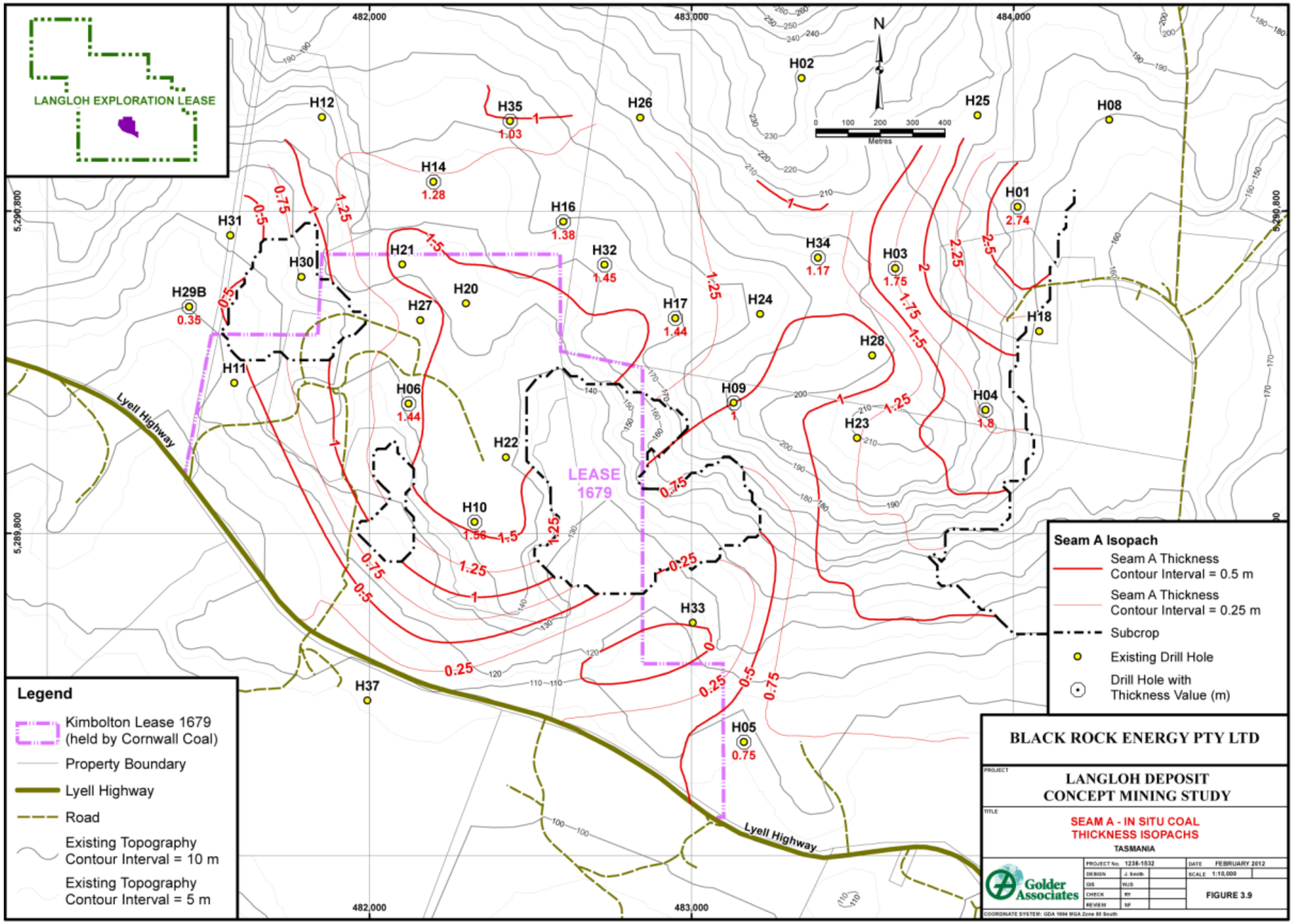


Figure 3-9 Seam B In Situ Coal Thickness Isopachs

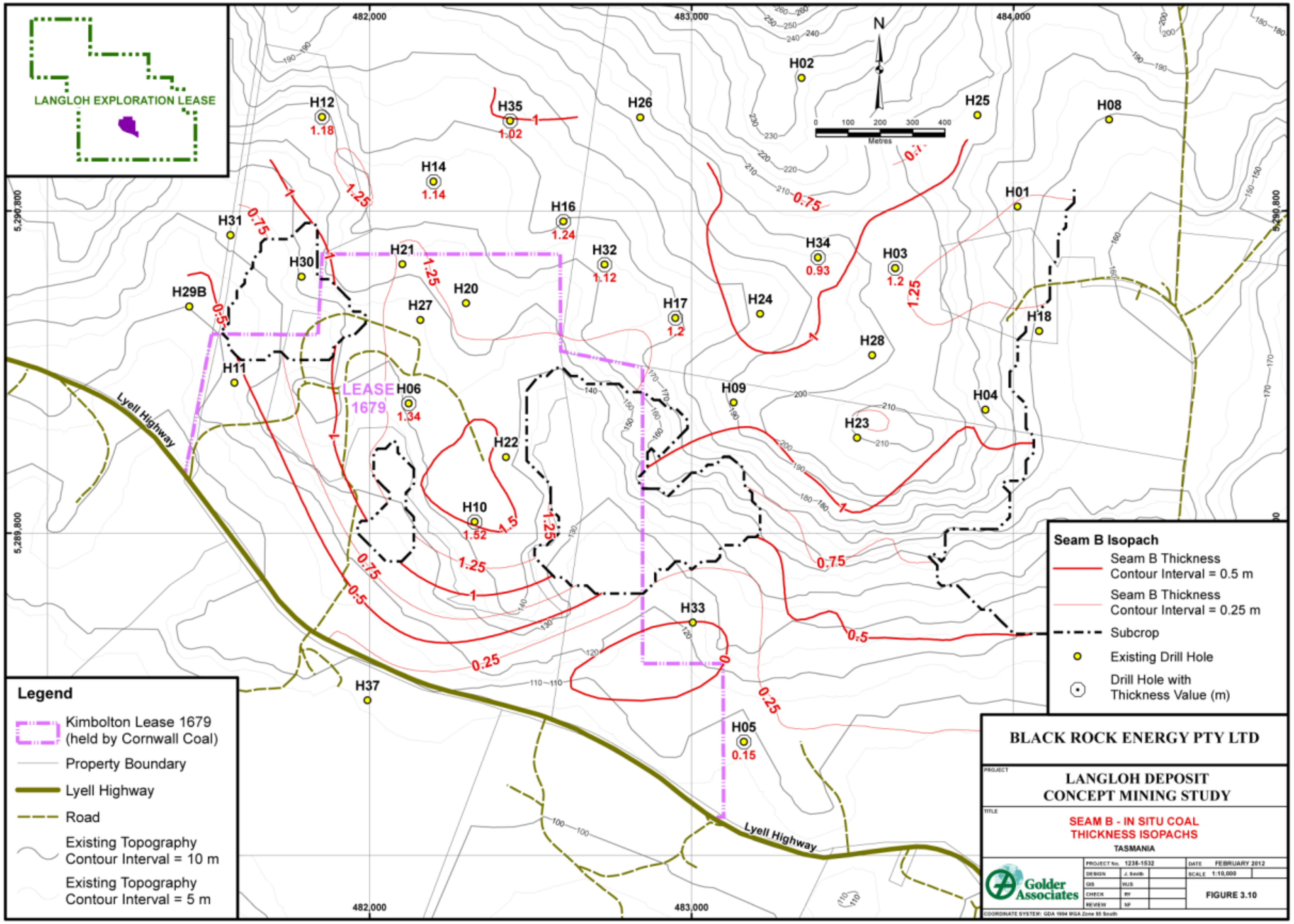


Figure 3-10 Seam C In Situ Coal Thickness Isopachs

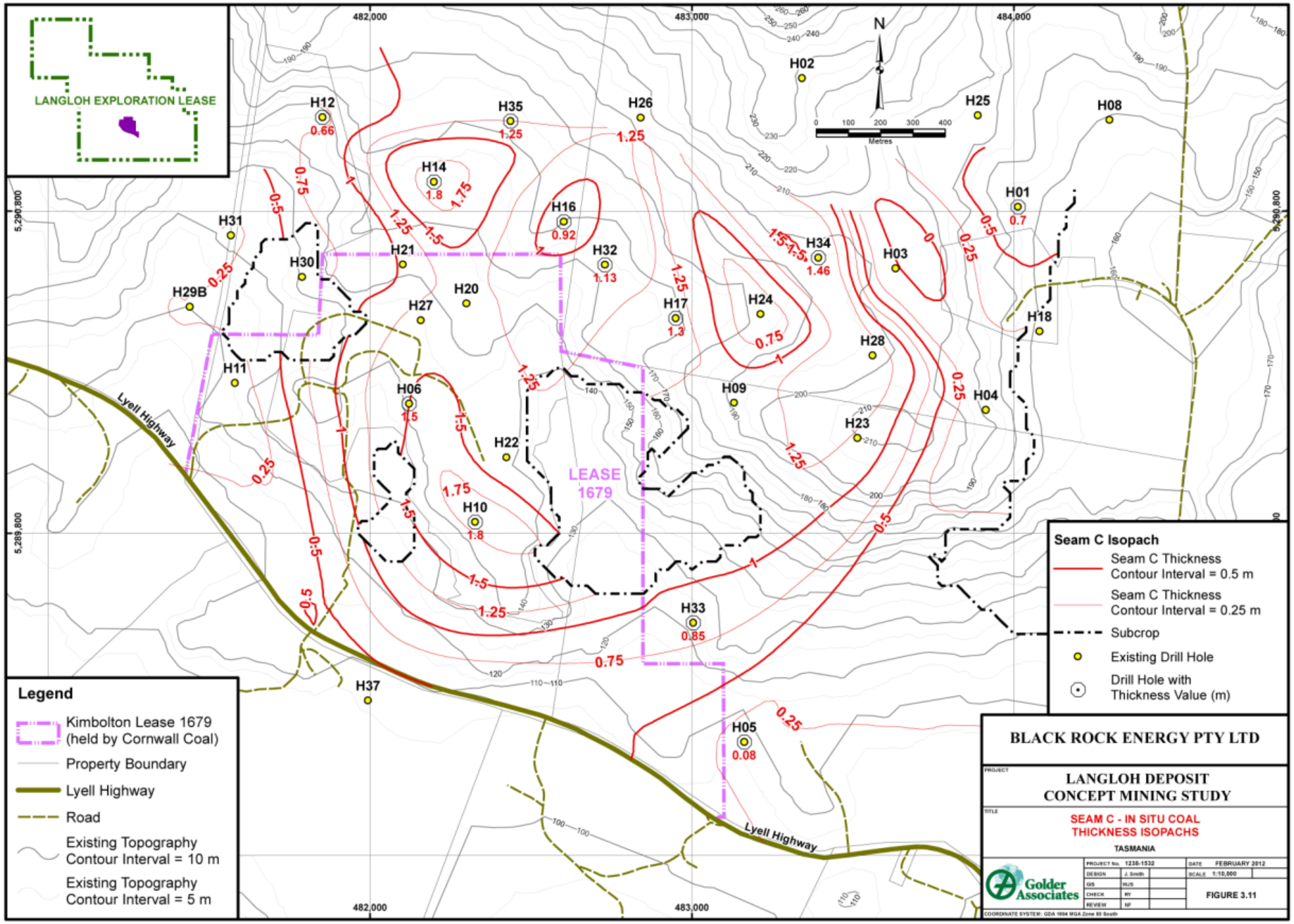
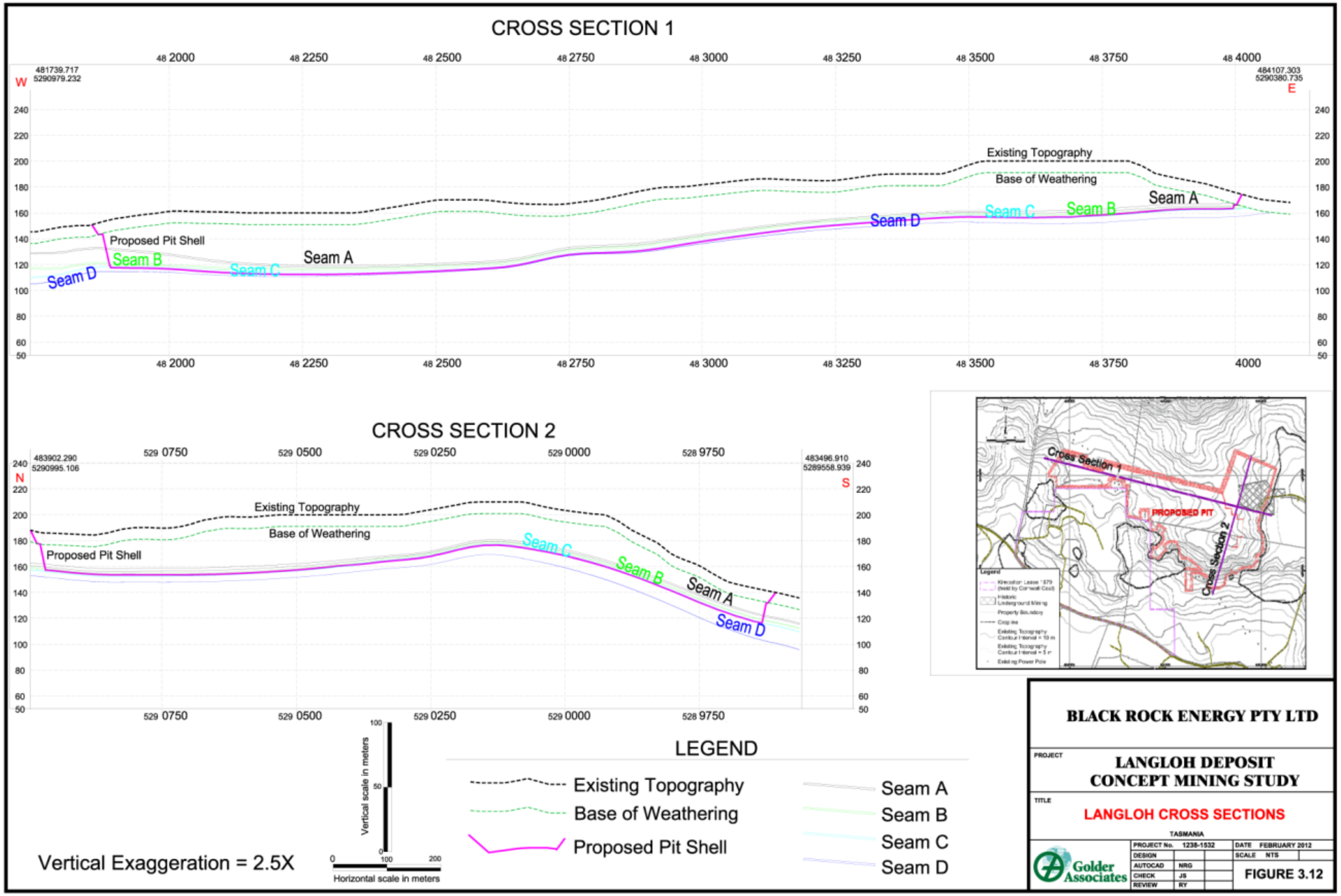


Figure 3-11 Cross Sections



3.6.3 Coal Quality

As discussed in previous sections, almost half of the historic coal quality samples had to be discarded which reduced the accuracy of the coal quality model compared to the structural model.

Most coal quality variables were modelled directly from the bore hole database using a straight point to grid method to honour the original data as close as possible. Sulphur content was modelled using an inverse distance squared method.

Quality isopleths were then created for each seam and each quality. These maps were cross referenced with the bore hole database to ensure no discrepancies were present.

3.6.4 Ash Content

Each seam at the Langloh deposit shows a unique ash trend as shown in the following figures:

- Figure 3-13, Seam A In Situ Ash Isopleths (adb)
- Figure 3-14, Seam B In Situ Ash Isopleths (adb)
- Figure 3-15, Seam C In Situ Ash Isopleths (adb)

The ash in Seam A is quite variable as it can range from 25% to 40% (adb) across a relatively short distance. The trend towards higher ash is not consistent, nor does it trend in any particular direction.

Due to the variable and sometimes quite high ash of Seam A, ply-by-ply analyses of the seam may assist in identifying individual high ash plys that could be wasted to reduce the overall ROM ash content of the seam rather than potentially wasting the entire seam in some areas to lower the overall ash at Langloh. It would appear that there are currently no ply-by-ply samples for Seam A so the potential for selectively mining Seam A to reduce overall ash is unknown.

However, there does appear to be a relatively consistent mud band of approximately 0.2 m throughout Seam A that has been identified through geophysical logs. Consequently, there may be potential for selective mining of this seam, again however, further in-depth study including ply-by-ply analyses are necessary to confirm this.

The ash content of Seam B appears to be more consistent. Depending on product specifications, it is most likely that Seam B will always be mined. Seam B in situ ash ranges from 14% to almost 30%.

The ash content of Seam C appears to be fairly constant across most of the deposit except toward the eastern boundary. Golder has drawn the conclusion that the seam degrades to carbonaceous shale in this area. This may be due to incorrect seam picks in some drill holes, as well as high ash contents (over 70% adb) in seam picks logged as carbonaceous shale. Drill holes surrounding this area are deep enough to have intersected the seam.

Figure 3-12 Seam A In Situ Ash Isopleths (adb)

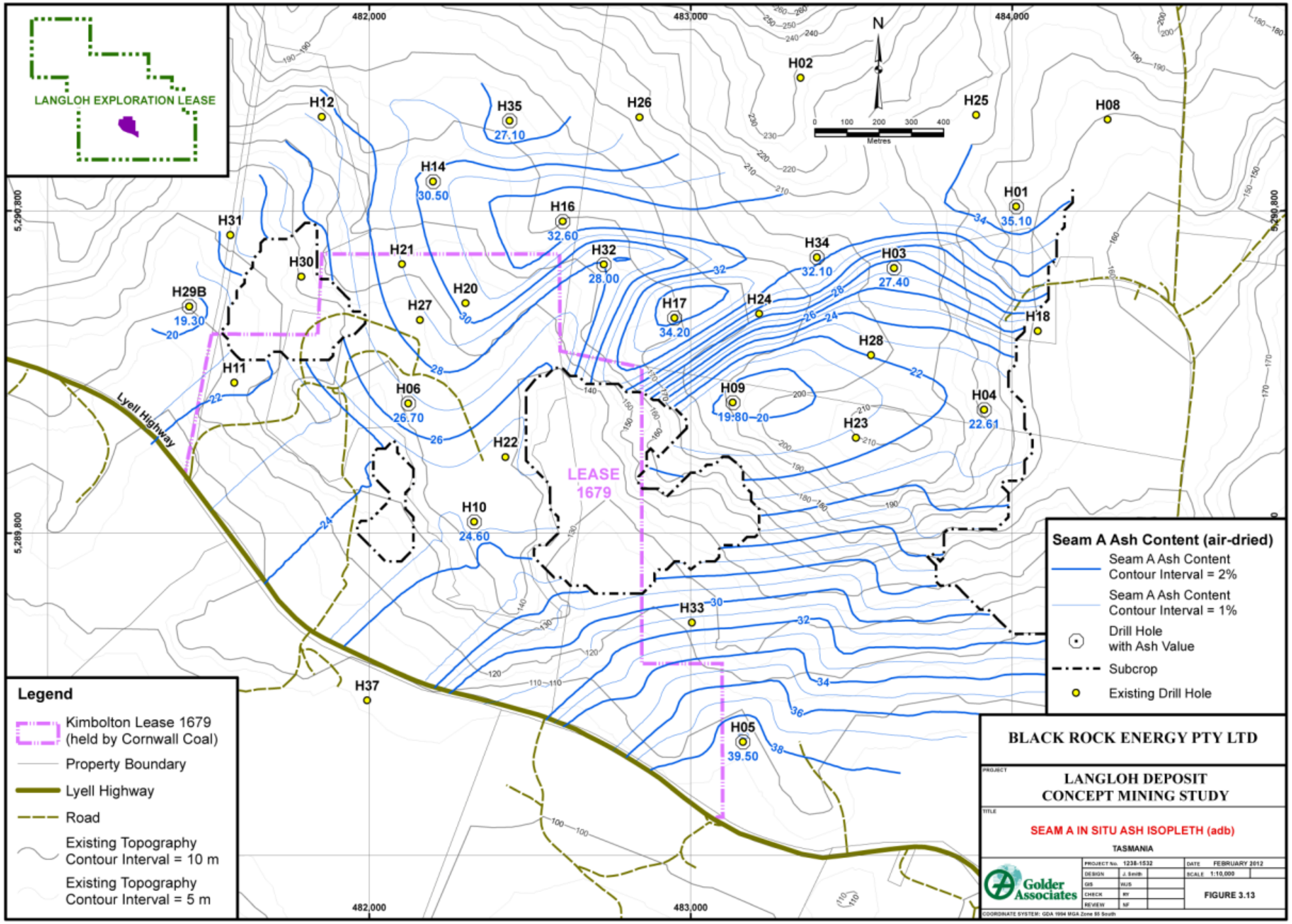


Figure 3-13 Seam B In Situ Ash Isopleths (adb)

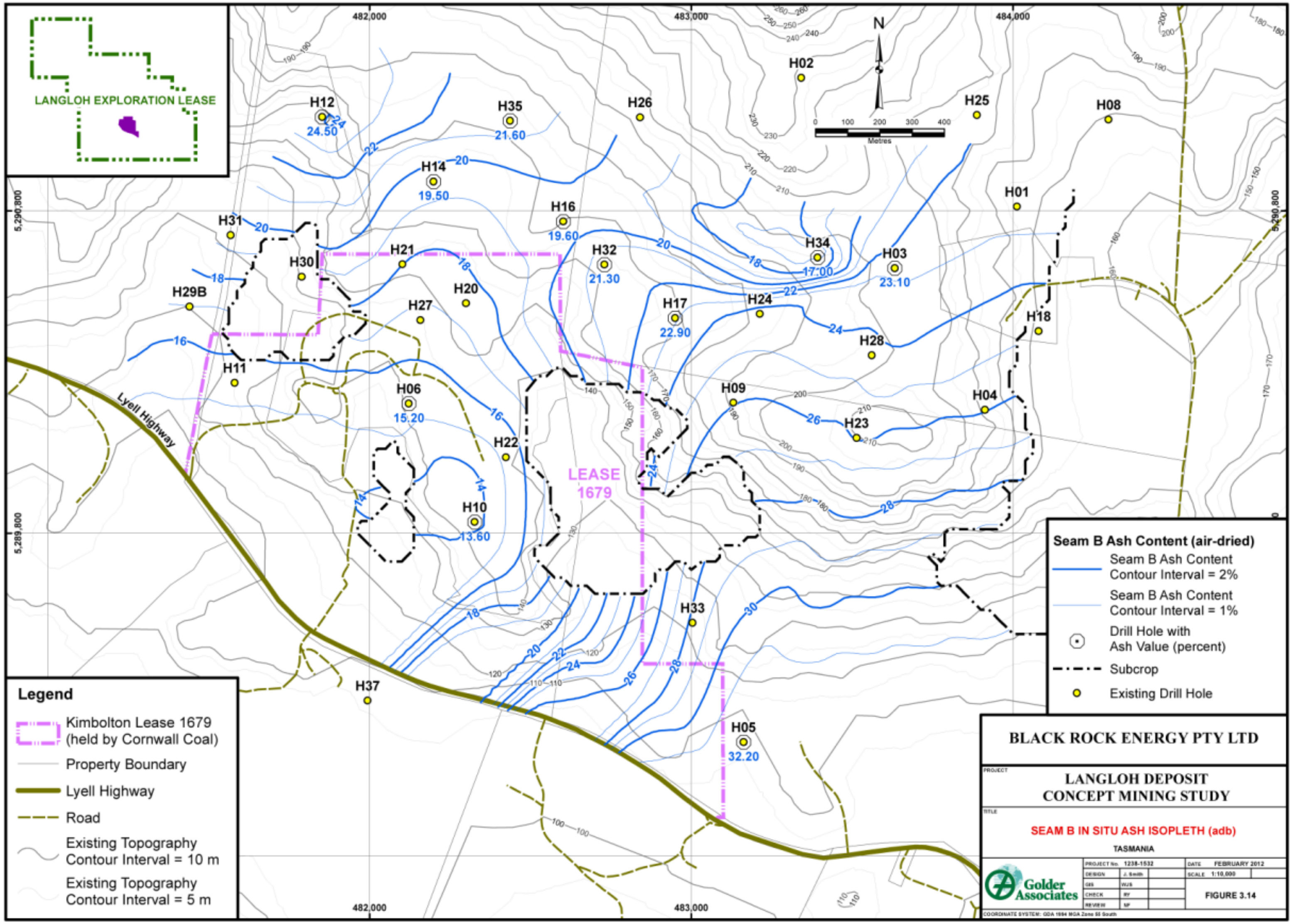
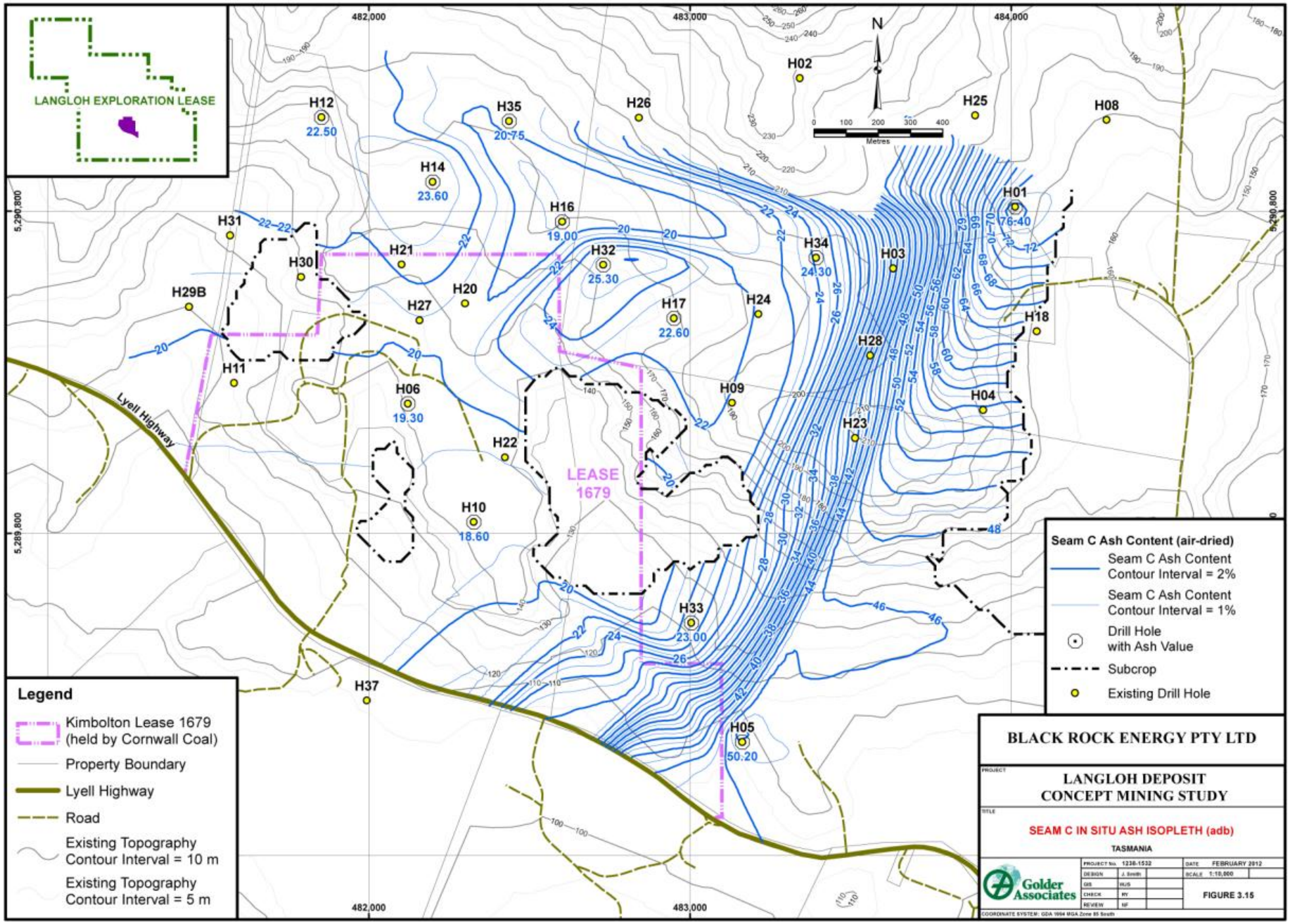


Figure 3-14 Seam C In Situ Ash Isopleths (adb)



3.6.5 Specific Energy Content

The specific energy or in situ calorific value across the deposit appears to be fairly consistent and it would appear that energy has not been affected by heating from igneous intrusions. Seam A appears to range from 19 to 26 MJ/kg (adb) with Seams B and C ranging from 22 to 24 MJ/kg (adb). This may be due to only moderate or mild heating of the coal which is enough to drive off volatiles and moisture but not burn energy, but this cannot be confirmed at this stage. Seam A shows the most variation, but has areas that have some of the highest energy content across the deposit. Seam B shows consistent energy and the Seam C shows consistent but lower energy compare to the other seams.

It should be noted that there are less data points for specific energy so any trend that is shown in other qualities may not be seen in the model due to fewer data points.

The in situ calorific value isopleths (adb) for each seam can be seen on the following figures:

- Figure 3-16, Seam A In Situ Calorific Value Isopleths (adb)
- Figure 3-17, Seam B In Situ Calorific Value Isopleths (adb)
- Figure 3-18, Seam C In Situ Calorific Value Isopleths (adb)

3.6.6 Volatile Matter Content

There is a pronounced trend towards low volatile matter (VM) content towards the east of the Langloh deposit. This may have been produced by a dolerite sill that has intruded the area, heating the overlying coal and driving off volatiles. The sill outcrops east of the deposit and dips at about 5 degrees to the west. The degree of de-volatilisation decreases as the depth of the sill below the coal seams increases. In situ VM content varies from 4% – 29% (adb). Minimum acceptable VM content for utilities is usually around 20% on an as-received basis (arb).

Some lower VM coal, minimum 15% (arb) is exported from Australia and is blended with higher VM coal by customers. Some of the lower VM coal at Langloh could be blended with higher VM coal to maximise resource utilisation. No detailed studies have been conducted to determine the extent to which blending may be possible but it can be reasonably assumed that some coal will will have to be wasted.

The in situ volatile matter content on an air-dried basis for each seam can be seen in the following figures:

- Figure 3-19, Seam A In Situ Volatile Matter Content Isopleths (adb)
- Figure 3-20, Seam B In Situ Volatile Matter Content Isopleths (adb)
- Figure 3-21, Seam C In Situ Volatile Matter Content Isopleths (adb)

3.6.7 Sulphur Content

Seam A shows consistent sulphur of about 0.25% (adb) across most drill holes except hole H29B which shows 0.65% (adb). Seam B is a consistent 0.3% - 0.35% (adb) across the deposit and Seam C shows 0.4% - 0.45% (adb) range.

The in situ sulphur content on an air-dried basis for each seam is shown on the following figures:

- Figure 3-22 Seam A In Situ Sulphur Content Isopleths (adb)
- Figure 3-23 Seam B In Situ Sulphur Content Isopleths (adb)
- Figure 3-24 Seam C In Situ Sulphur Content Isopleths (adb)

Figure 3-15 Seam A In Situ Calorific Value Isopleths (adb)

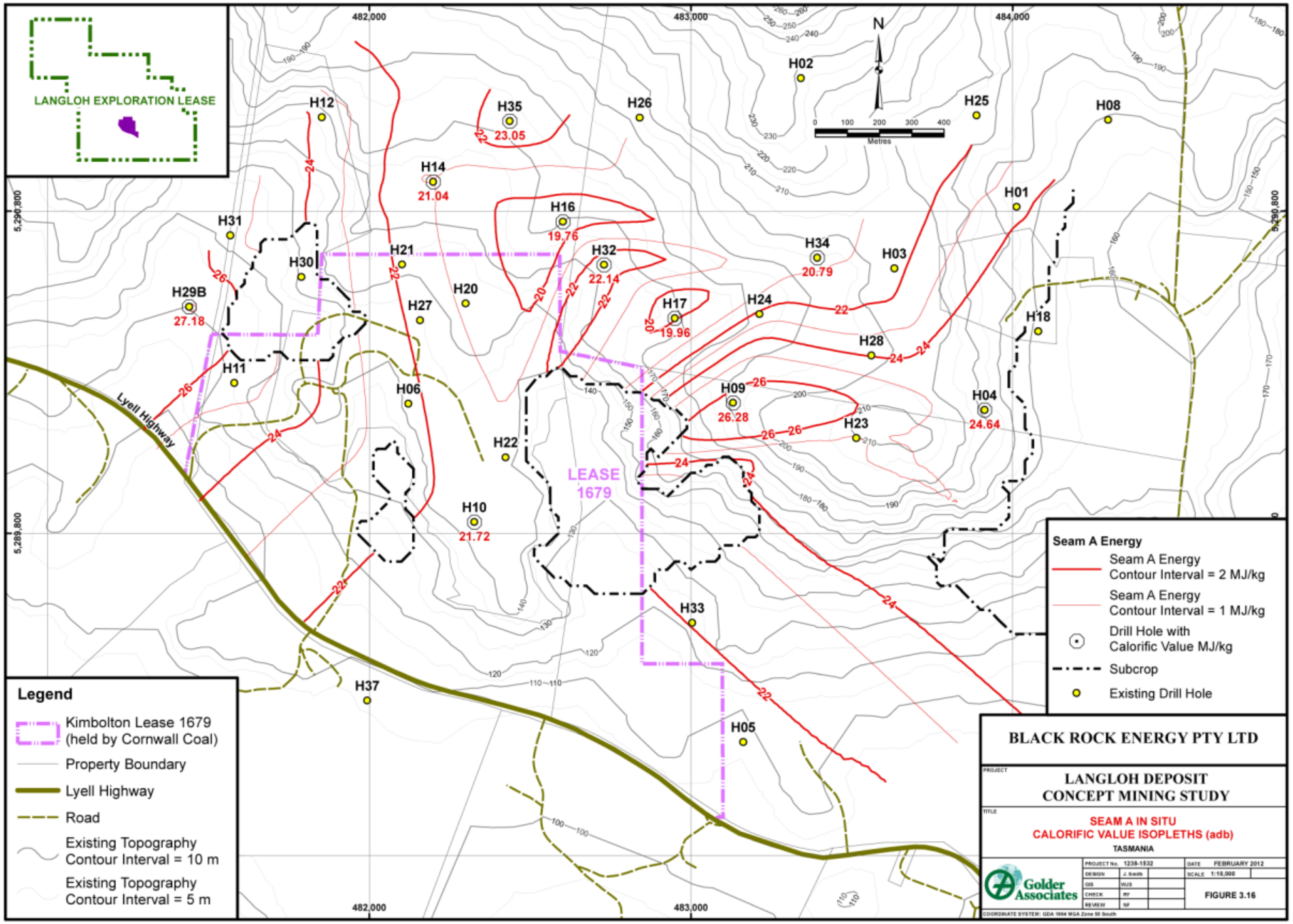


Figure 3-16 Seam B In Situ Calorific Value Isopleths (adb)

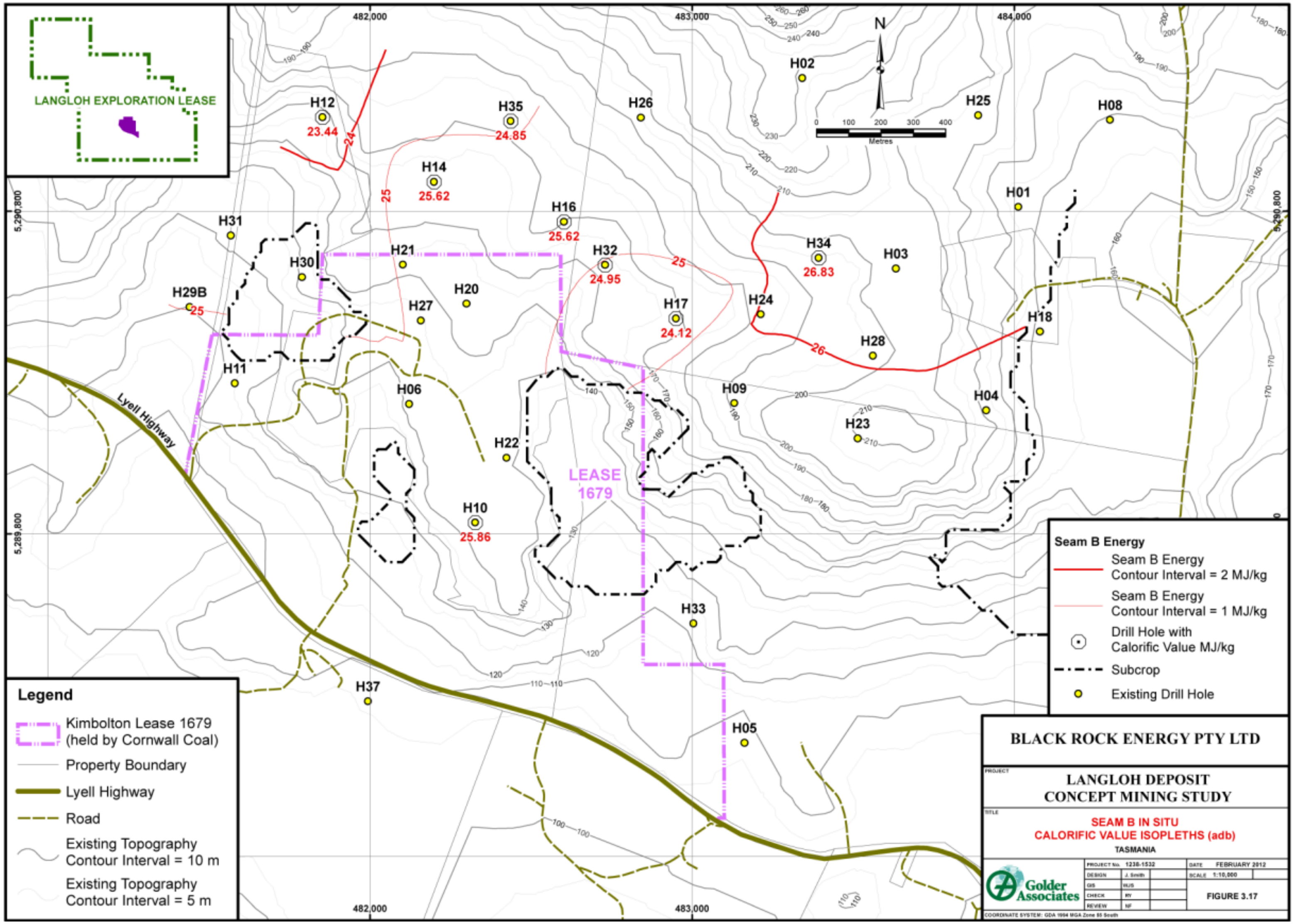


Figure 3-17 Seam C In Situ Calorific Value Isopleths (adb)

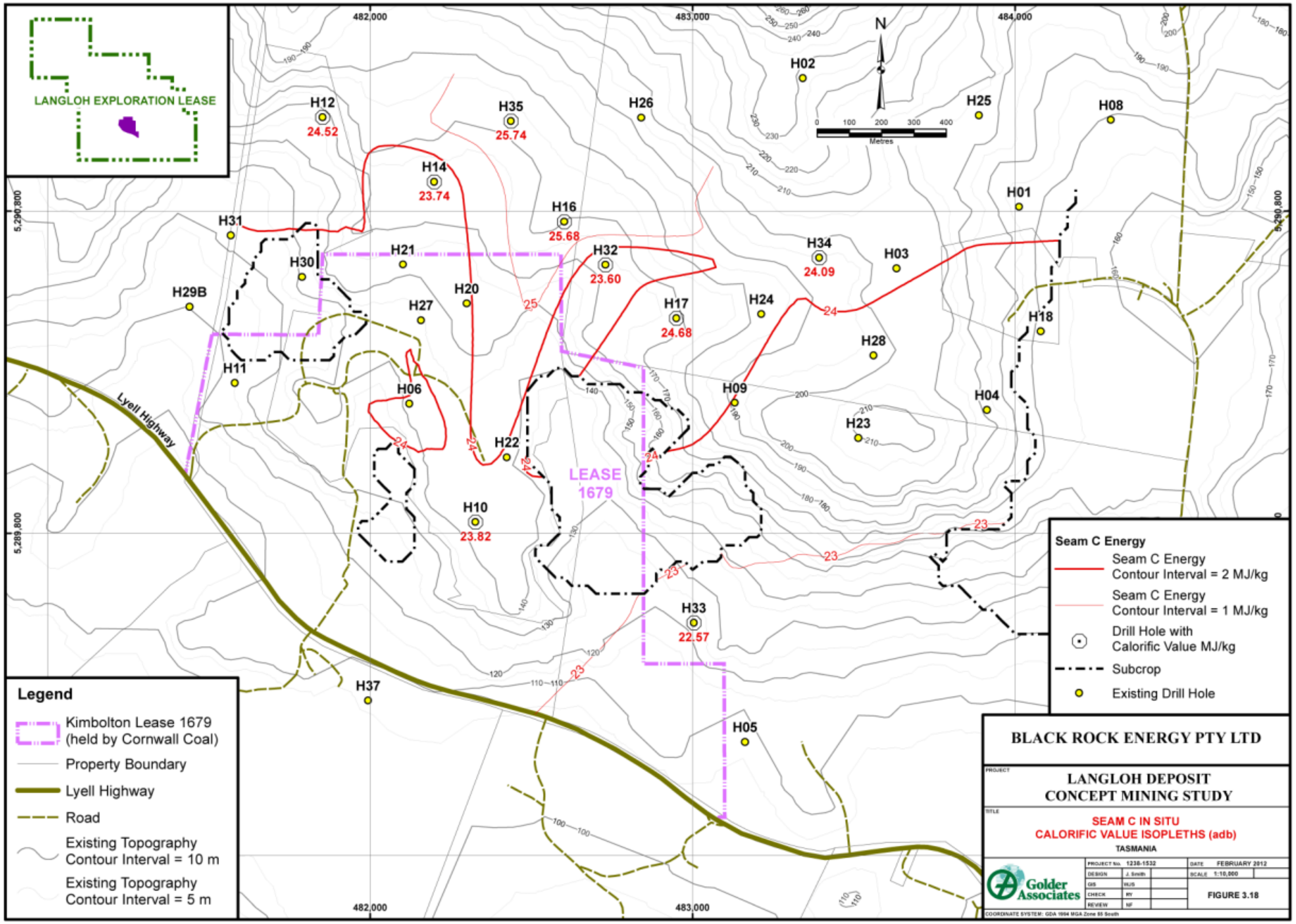
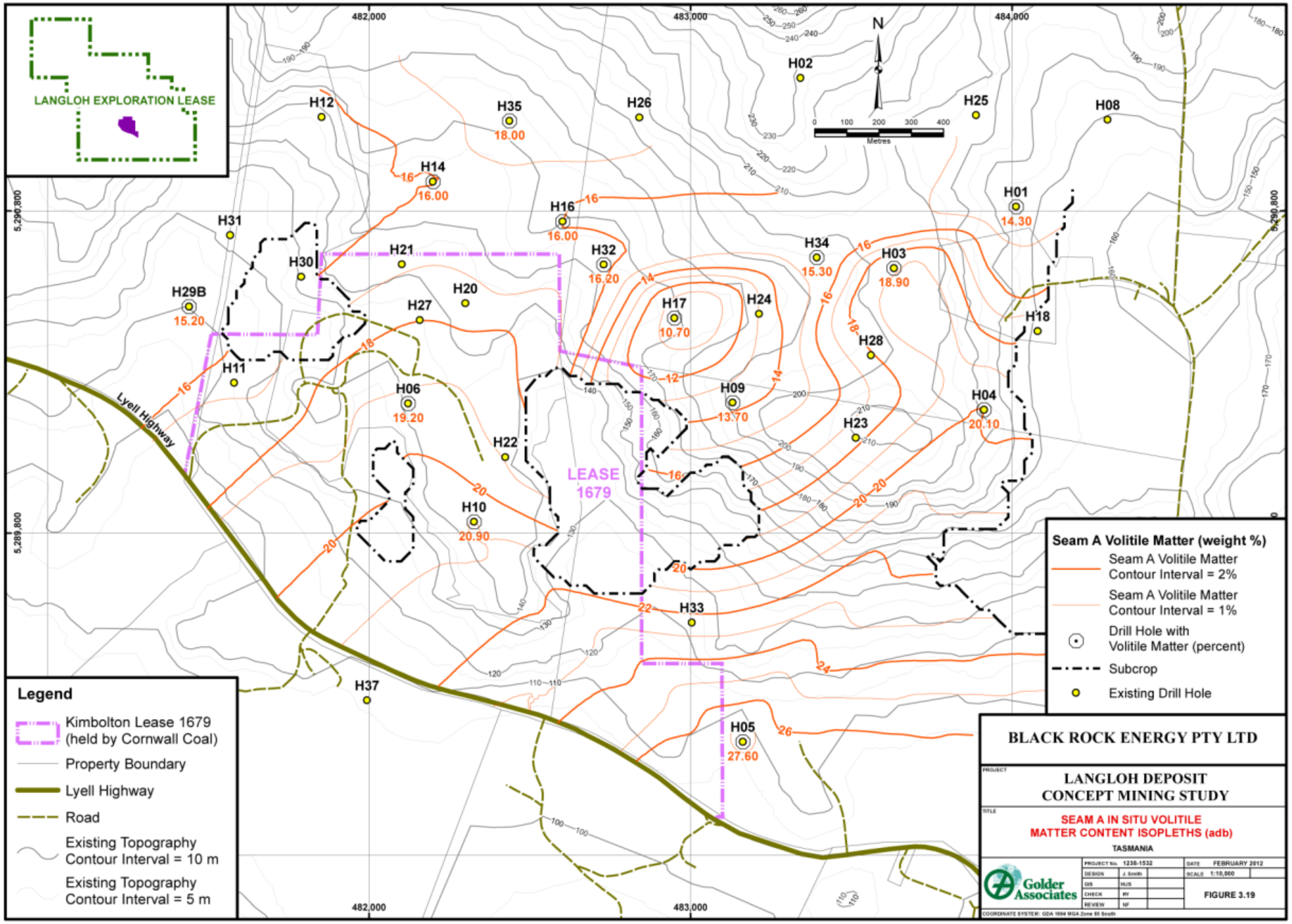


Figure 3-18 Seam A In Situ Volatile Matter Content Isopleths (adb)



Golder Associates

Figure 3-19 Seam B In Situ Volatile Matter Content Isopleths (adb)

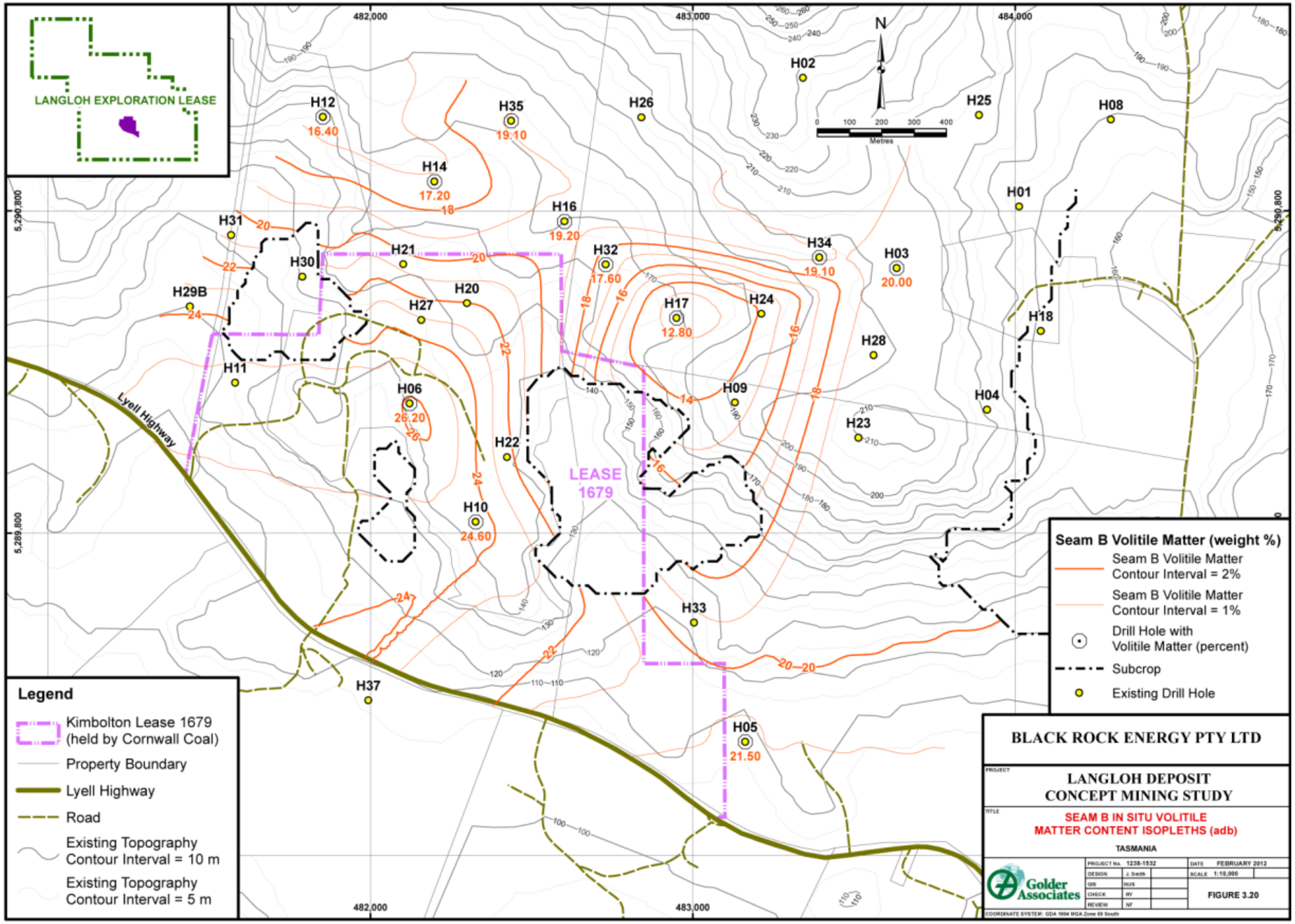


Figure 3-20 Seam C In Situ Volatile Matter Content Isopleths (adb)

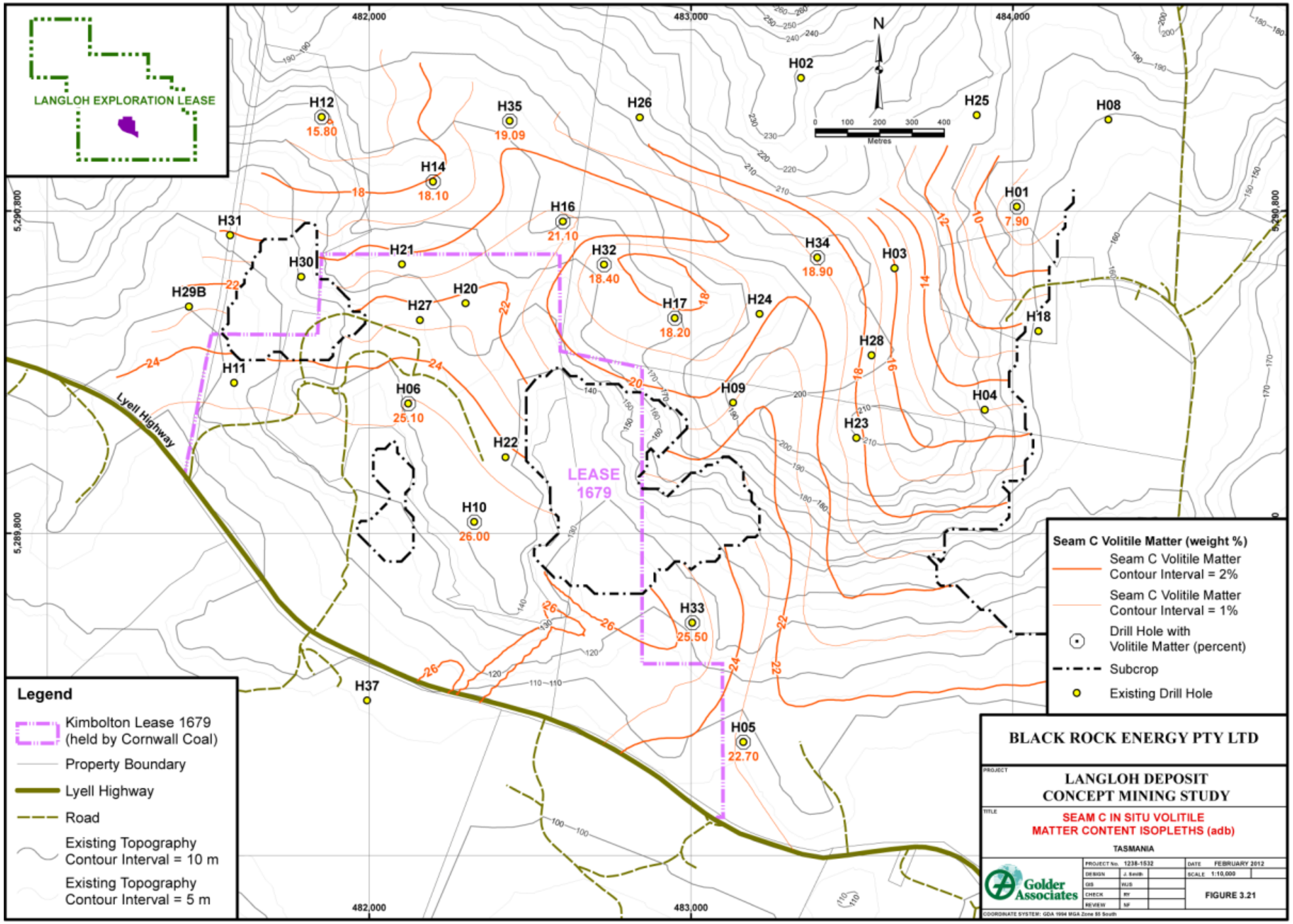


Figure 3-21 Seam A In Situ Sulphur Content Isopleths (adb)

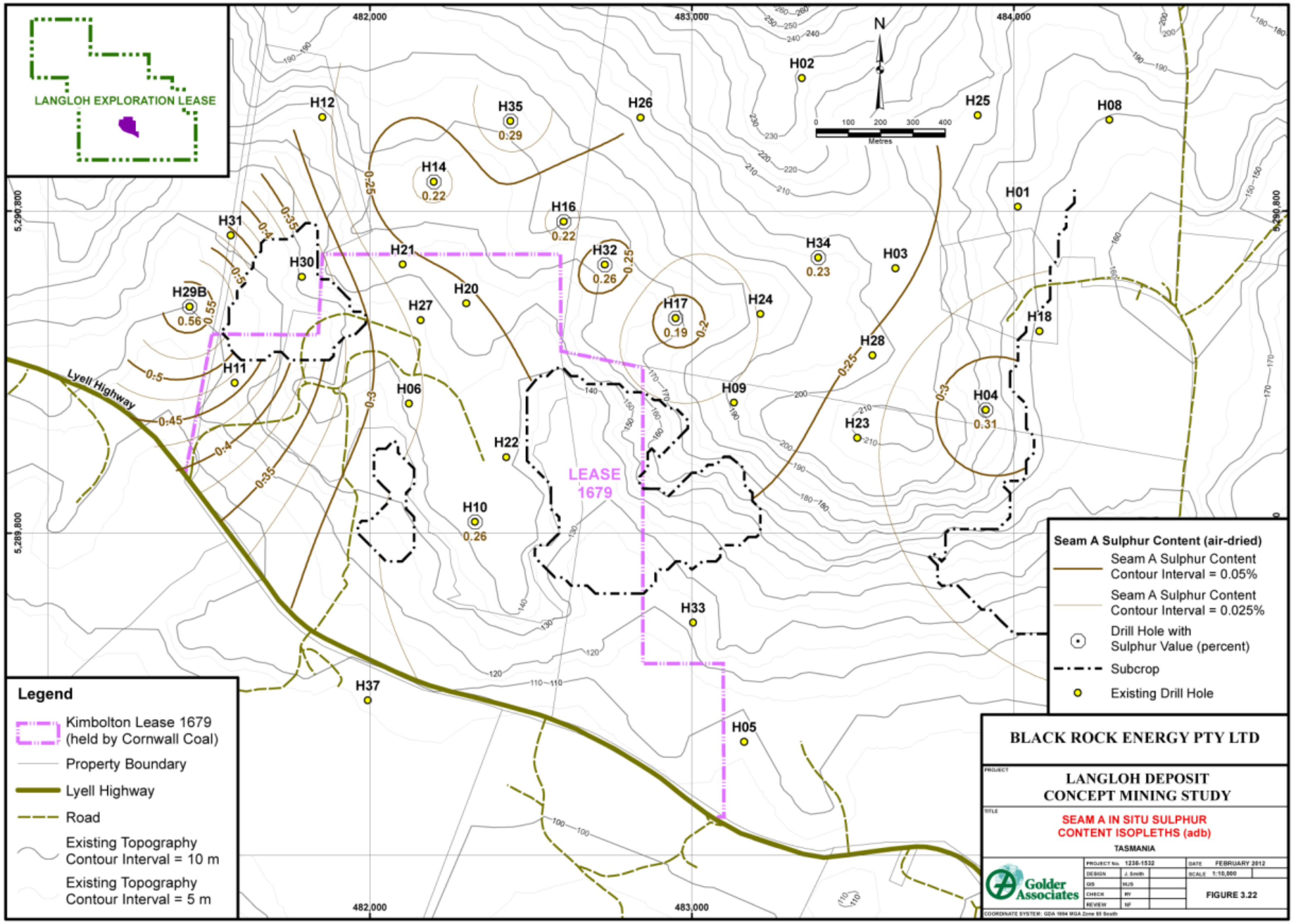


Figure 3-22 Seam B In Situ Sulphur Content Isopleths (adb)

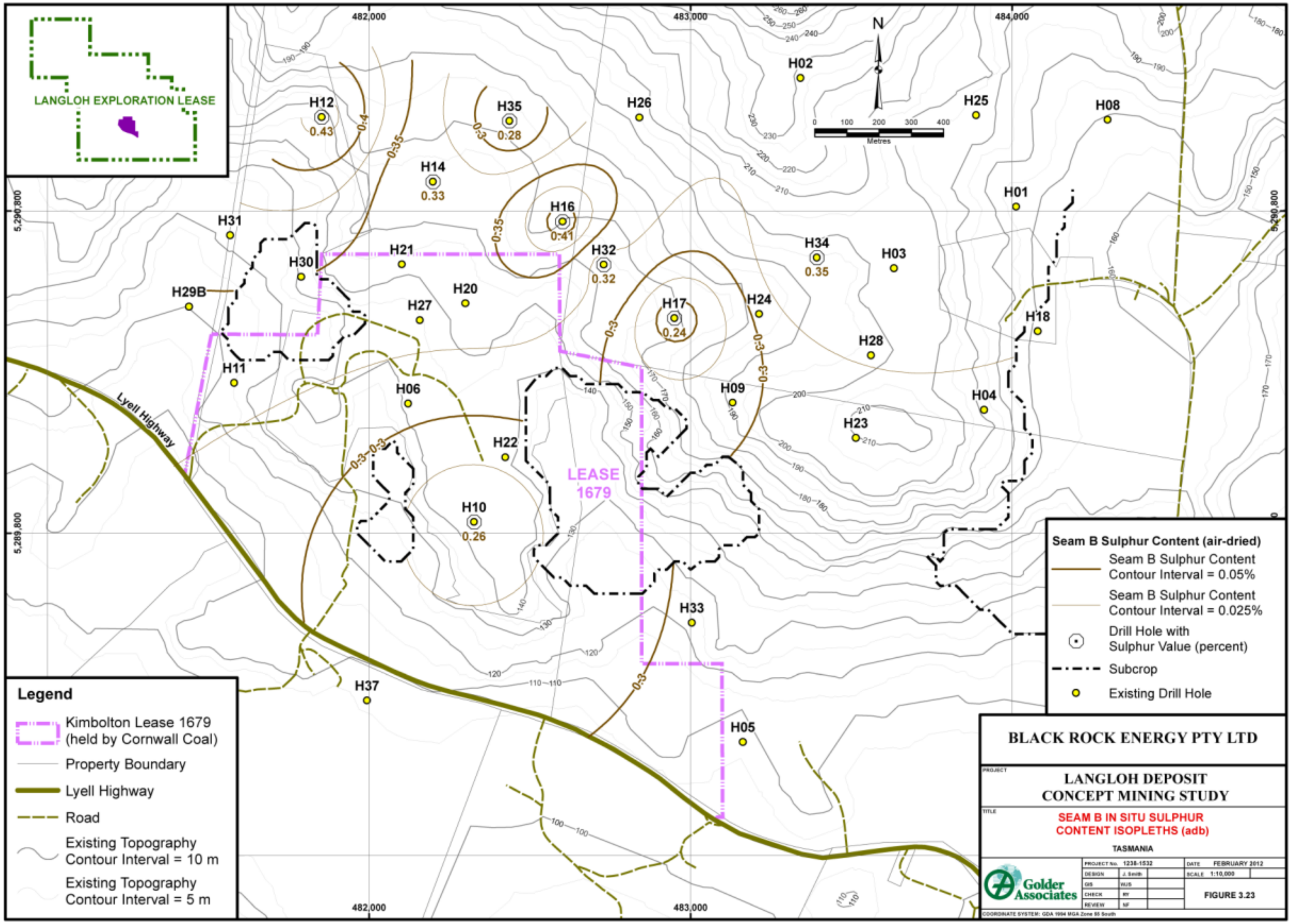
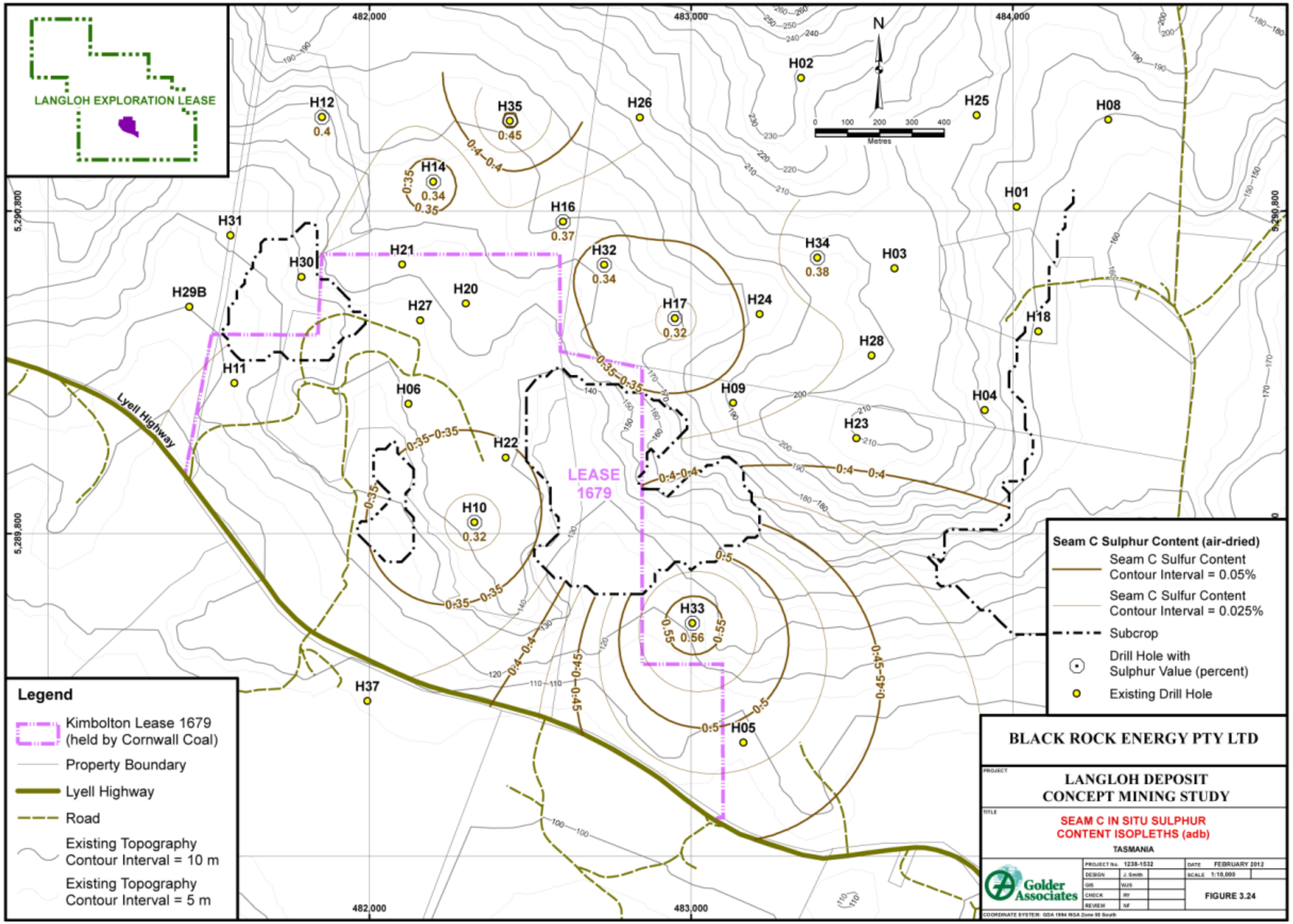


Figure 3-23 Seam C In Situ Sulphur Content Isopleths (adb)



4.0 COAL RESOURCE ESTIMATION

Mining One Consultants estimated an in situ coal resource of 9.96 Mt within EL 28/2008 in 2008. They also estimated an in situ coal resource of 3.2 Mt within the CCC lease ML 1679. CCC reported a mineable coal reserve of 1.8 Mt for the ML. These estimates do not take into account coal mined to date. Mineral Resources Tasmania (MRT) reports indicate that 239,497 tonnes of coal has been produced from the CCC Kimbolton Mine between 1998 and 2008.

Acquisition of the CCC lease may represent an opportunity to increase resources. This resource does not appear to have VM content affected by heat from the underlying dolerite sill.

Golder has estimated the in situ coal resources to be 8.1 Mt. A breakdown of these coal resources into JORC categories is shown in Table 4-1, Summary of Estimated Langloh Coal Resources and Table 4-2, Estimate Langloh Resources Quality.

Golder has restricted these resources to the following:

- Stripping ratio of less than 10:1 bcm/in situ tonnes
- Limited to lease boundaries
- Limited to known geological structures (generally taken into account in stripping ratios)
- Limited to reasonable coal quality (area of very high ash Seam C coal excluded).

Measured status required as a minimum:

- Structure holes overlapping with at least one other hole within 250 m (cored or open with geophysics)
- Quality holes within 250 m
- Geology appears to be consistent and predictable with no anomalous areas

Indicated status required as a minimum:

- Structure holes overlapping with at least one other hole within 500 m (cored or open with geophysics)
- Quality holes within 500 m
- Geology appears to be reasonably consistent and predictable with no anomalous areas

Inferred status required as a minimum:

- Structure holes overlapping with at least one other hole within 1,000 m (cored or open with geophysics)

All drill hole data had to meet criteria as outlined in Section 3.5.2 of this report. The criteria in Section 3.5.2 was developed to ensure the model could be used to produce a JORC compliant resource statement. Previous sections of this report detail how the model was built and should be read in conjunction with this section.

A plan of the Resource Polygons is shown in Figure 4-1 JORC Resource Areas.

Table 4-1 Summary of Estimated Langloh Coal Resources

Estimated Resources Langloh	Insitu Tonnes
Estimated Measured	5,500,000
Estimated Indicated	1,200,000
Estimated Inferred	1,400,000
Total	8,100,000

Table 4-2 Estimated Langloh Resources Quality

Langloh Estimated Coal Resources							
Description	**Coal Resource Tonnes**	**Moisture (% adb)**	**Ash (% adb)**	**Volatile Matter (% adb)**	**Fixed Carbon (% adb)**	**Calorific Value (MJ/kg adb)**	**Sulphur (% adb)**
Measured	5,500,000	4.6	25.7	17.3	52.5	23.8	0.31
Indicated	1,200,000	5.2	28.9	19.2	46.7	24.1	0.32
Inferred	1,400,000	4.9	27.7	18.3	49.0	24.8	0.30
Total	**8,100,000**	**4.7**	**26.5**	**17.7**	**51.0**	**24.0**	**0.31**

Note: Coal resources are estimated in accordance with JORC Code.

Golder has not prepared a JORC compliant estimate of reserves as part of this study. An estimate of mineable coal is provided in Section 5.0 of this report which lists mine planning and scheduling volumes in more detail. These mine volumes have been summarised in another section of the report as they are not JORC compliant and Golder wishes to remove any confusion caused by including these mine volumes with the JORC resources.

JORC Statement

The information compiled in this report relating to resources is based on information compiled by Neil Fraser, who is a member of the Australian Institute of Mining and Metallurgy and is a full time employee of Golder Associates Pty Ltd. Neil Fraser has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity he is undertaking to qualify as a Competent Person as defined in the 2004 edition of the “Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves”. Neil Fraser consents to the inclusion in the report of the matters based on his information in the form and context in which it appears. Golder Associates is an independent consultant for BRE. Remuneration for the preparation of this report is on a time and materials basis only.

4.1 Impact of Historical Underground Workings on Resources

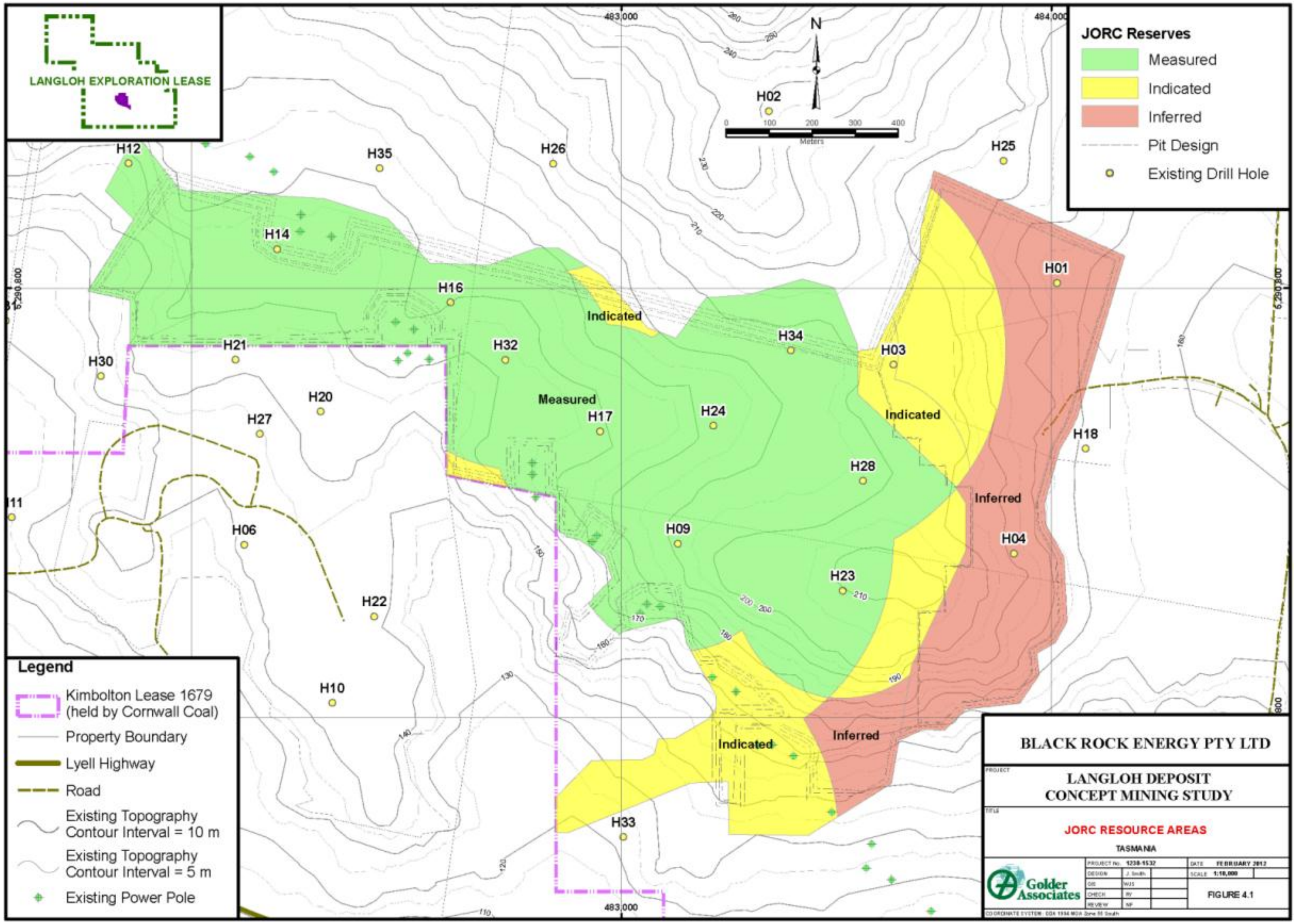
The impact of the historical underground workings is very small. It has been estimated that approximately 30 kt was mined using underground methods and this equals approximately 0.4% of the total calculated resources. MRT commissioned an Archaeological Survey Report in 2001 which may be an indication of some heritage significance at the site. It is unclear if these historical surface facilities will be able to be demolished to enable open cut mining.

It has been assumed for the purposes of resource calculation that open cut mining will be able to be conducted in the area as there may be potential to do some or all of the following to allow open cut mining:

- Detailed archaeology survey prior to mining to record the site (a survey was completed by MRT in 2001)
- Salvage of small items such as coal wagons
- Relocation of some or all of the buildings
- Considerable benefits from rehabilitating the site by removing the mullock dump and removal of open shafts.

It is currently very difficult to quantify what the impact of leaving the historic underground facilities in place will be on resources until the issue is addressed more formally with relevant government departments as any buffer zones and which buildings would be preserved is unknown. The location of the workings is estimated and has largely been inferred from the location of the remaining surface infrastructure. The approximate location of the workings can be seen on Figure 5-1 Proposed Pit Layout and Facilities.

Figure 4-1 JORC Resource Areas



5.0 MINE PLANNING

5.1 Open Cut Design

The Langloh project is located within EL 27/2008 which has an approximately area of 870 ha. Only a relatively small portion of the total EL area has been explored and considered in this study. Land is mostly owned by private landholders.

BRE is seeking a mining lease over the area. The application for the lease is expected to be lodged during Q1 2012. For conceptual mine planning purposes, it has assumed that any environmental issues that are identified should be able to be addressed and do not present a constraint or barrier to the mine plan proposed in this report. It is critical that BRE conduct relevant environmental studies to understand the environmental issues at Langloh and if there is any impact on mining.

5.1.1 Open Cut Objectives and Assumptions

The main objective of the mine plan developed for this study was to maximise the extraction of the coal resources with consideration given to the identified constraints to mining including lease boundaries and electricity infrastructure.

EL 27/2008 directly adjoins CCC's Kimbolton project's (ML1679). It has been assumed that an agreement could be reached with Kimbolton to at least allow the Langloh project to mine overburden right up to the limit of the lease so the top of the highwall is on the lease boundary. The potential for over-stripping overburden into the Kimbolton lease to enable coal extraction right up to the lease boundary should be explored at an appropriate time with the owners of the Kimbolton project. This approach would be beneficial for both projects.

The Langloh project is planned to be a conventional open cut mine utilising diesel powered hydraulic excavators matched with suitably sized dump haul trucks.

As a stand alone project, Langloh is planned to be a run-of-mine (ROM) project that wil produce up to 900,000 ROM tonne per year.

5.1.2 Geotechnical Analysis

Only a basic desktop geotechnical assessment has been completed from the photos of a site visit conducted by Golder staff to the Kimbolton Mine. The rock appears to be strongly jointed with at least 2 sets of joints which can create unstable and potentially hazardous walls. This depends greatly on the orientation of wall and jointing angles. Several sections of small rock falls and overhangs can be identified in the photos where the walls have been cut at certain angles to the joints. These hazards will be important considerations for the potentially significantly deeper Langloh project. Other forms of control such as pre-splitting should also be considered in future studies for effectiveness in controlling this issue. In addition, the duration of which any wall is left exposed will acerbate the potential for wall hazards. The mine plan will serve to mitigate any long-term exposure but certain areas may be required to stand for longer periods of time.

The angles of the joints should be calculated in future studies to determine the best angle for wall and strip orientation. It is recommended that an in depth geotechnical assessment should be part of future studies and drilling programs.

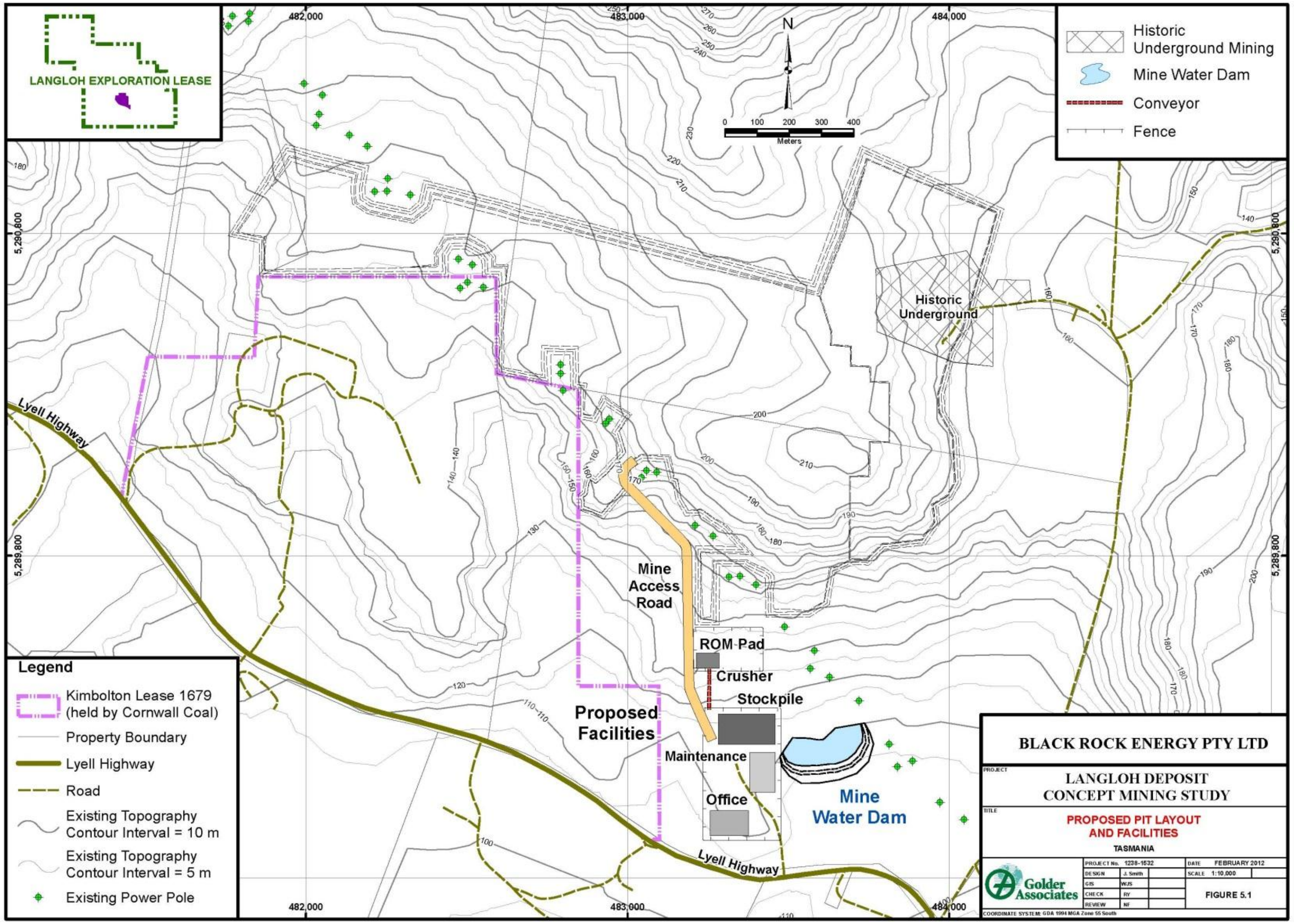
Due to the lack of geotechnical data, Golder has chosen reasonable parameters for wall design and has chosen these parameters to be conservative based on expereince at other Australian coal mines.

5.2 Mining Method Selection

The Langloh project is planned to be a conventional open cut mine utilising diesel powered hydraulic excavators matched with suitably sized dump haul trucks. This is due to a number of factors:

- Dips on seams vary from 2-8%
- Pit width is relatively narrow
- Small size of the deposit

Figure 5-1 Proposed Pit Layout and Facilities



- Flexibility of equipment
- Able to mine curved and variable shapes around outcrop zones
- Lower equipment capital costs compared to other mining equipment (dragline)
- Open cut excavators and trucks are the most common open cut mining method used in Tasmania, so skilled operators may be able to be sourced locally
- Excavators and trucks are more amenable to contractors for mobilisation and potential changes in annual production requirements

There is considerable potential for dozer push to assist overburden removal at Langloh which would offer significant cost savings compared to 100% overburden removal by an excavator. It is proposed that the excavator would strip the upper portion of the waste in advance with the dozers removing the deeper portion of the waste just above the coal. The excavator would then remove any remaining waste on top of the coal and any parting material between the coal seams. Dozer push would be a good option for Langloh for the following reasons:

- Pit is relatively shallow
- Seams are located at bottom of pit and are relatively close together
- At least 10-20 m of continuous overburden available for dozer push in most parts of pit
- Excavators should be available for partings and clean-up work
- Seam dips at Langloh are workable for dozer push but higher dips may increase the amount of cleanup work for excavators
- Relatively low production rates

5.3 Mine Layout and Design Parameters

Mine layout and design were evaluated and carried out giving consideration to the following:

- Kimbolton mine lease boundary
- Minimise box cut length
- Seam subcrop lines
- Stripping ratio
- Proximity to ex-pit dumps
- Ex-pit dump storage limitations
- Location of mine infrastructure area
- Location of coal haul road
- Location of power lines
- Variations in coal quality
- Fault locations

In conjunction with these considerations, Golder also compiled a parameters document outlining certain assumptions used in the conceptual design for Langloh. Table 5-1, summarises these parameters.

The majority of the pit mines down to C seam, but due to the deterioration of Seam C to the east, there is a significant portion of the pit in the east that only mines to Seam B. Additionally, due to the topography in the Seam B area, the mining strips would be best suited to being orientated in a different direction compared to the Seam C area. The Seam B area also mines through a ridge that screens the operation from neighbours towards the east and would probably be mined last to keep the operation screened for as long as possible, especially given that the strip ratios in the Seam B area are slightly higher that the Seam C area. Therefore, the pit has been split into two areas, the C pit and the B pit.

As shown in Figure 5-1, the main constraints on the pit are subcrop lines, the Kimbolton Mine lease and the power line towers.

Figure 5-2, Cumulative ROM Stripping Ratio, shows the combination of cumulative strip ratios for Seam B and for Seam C.

Due to the pit being only about 60 m deep at its deepest point, only two benches have been designed into the walls, one at the base of weathering and one approximately half way down the wall between base of weathering and the Seam C floor were the wall height between base of weathering and the Seam C floor is greater than 25 m. This gives a maximum wall height between benches of 25 m and a maximum overall wall angle of approximately 51 degrees at the deepest point.

The thickness of the overburden across the site is shown on Figure 5-3, Overburden Thickness Isopachs.

Table 5-1 Langloh Project Design Parameters

Material Parameters	Value	Units
Maximum Bench Height	25	m
Bench Width Base of Weathering	8	m
Bench Width Fresh Rock	5	m
Highwall/Endwall Angle	63	°
Highwall Angle Weathered Material	45	°
Minimum Coal Thickness	0.3	m
Waste Material Swell Factor	25	%
Coal Swell Factor	40	%
OB/IB Material Density	2.4	t/m^3
Coal Material Density	1.45	t/m^3
Spoil Dump Angle	37	°
Loss/Gain Parameters	**Value**	**Units**
Gain Material Density	2.4	t/m^3
Gain Ash	80	% adb
Gain Sulphur	0.1	% adb
Gain CV	0.1	MJ/kg adb
Gain Moisture	3.5	%
Gain Thickness	0.05	m
Loss Thickness	0.05	m
Mining Recovery Factor	98	%

5.3.1 Power Lines

The power lines are a constraint on the pit and there is probably little chance of justifying relocation of the lines due to the high cost of removing a minimum of 22 towers. Additionally, the diverted line would be approximately double the length of the original line, as it would be re-routed around the Langloh and/or the Kimbolton project (irrespective whether lines were diverted east or west). This potentially results in the reconstruction of approximately 44 towers. Golder has therefore assumed the power lines will stay in place. If BRE wishes to move the lines, this should be examined in detail for future studies. Costing for relocating the lines has not been examined in this study. It has been estimated that leaving the power lines in place will reduce mineable coal by approximately 8%.

Preliminary discussions with the Tasmanian electricity infrastructure manager Transend have been positive. Transend have indicated through their Transmission Lines Engineering Officer that it may be possible to mine under the lines depending on the controls implemented and as long as a standoff is maintained from the tower bases. Other mining operations in Tasmania mine underneath power lines. Transend has provided Golder with the locations of the towers with no worse than 10 m accuracy. Golder has therefore designed the pit to standoff the tower bases 30 m, but mine underneath the lines.

Figure 5-2 Cumulative ROM Stripping Ratio

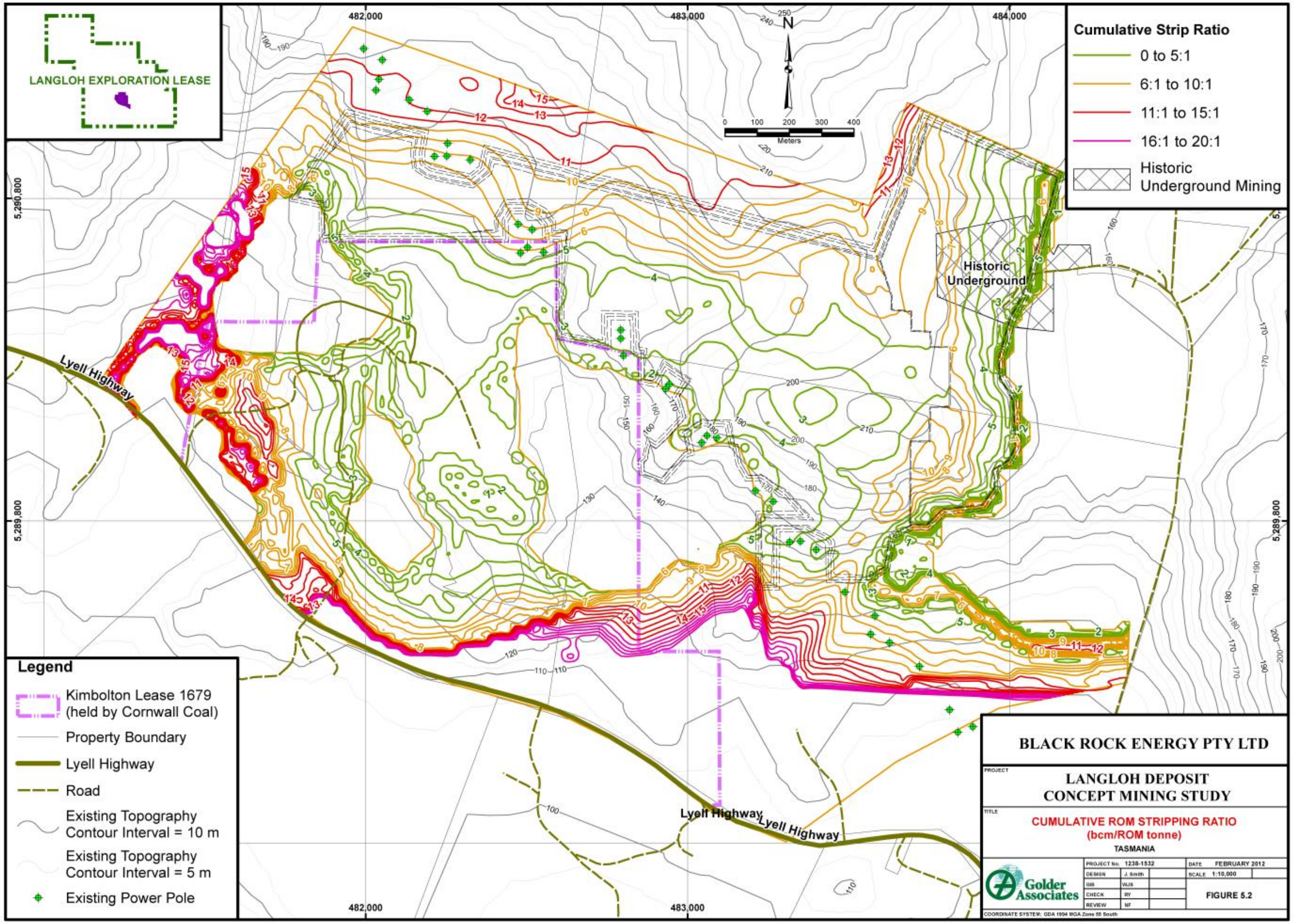
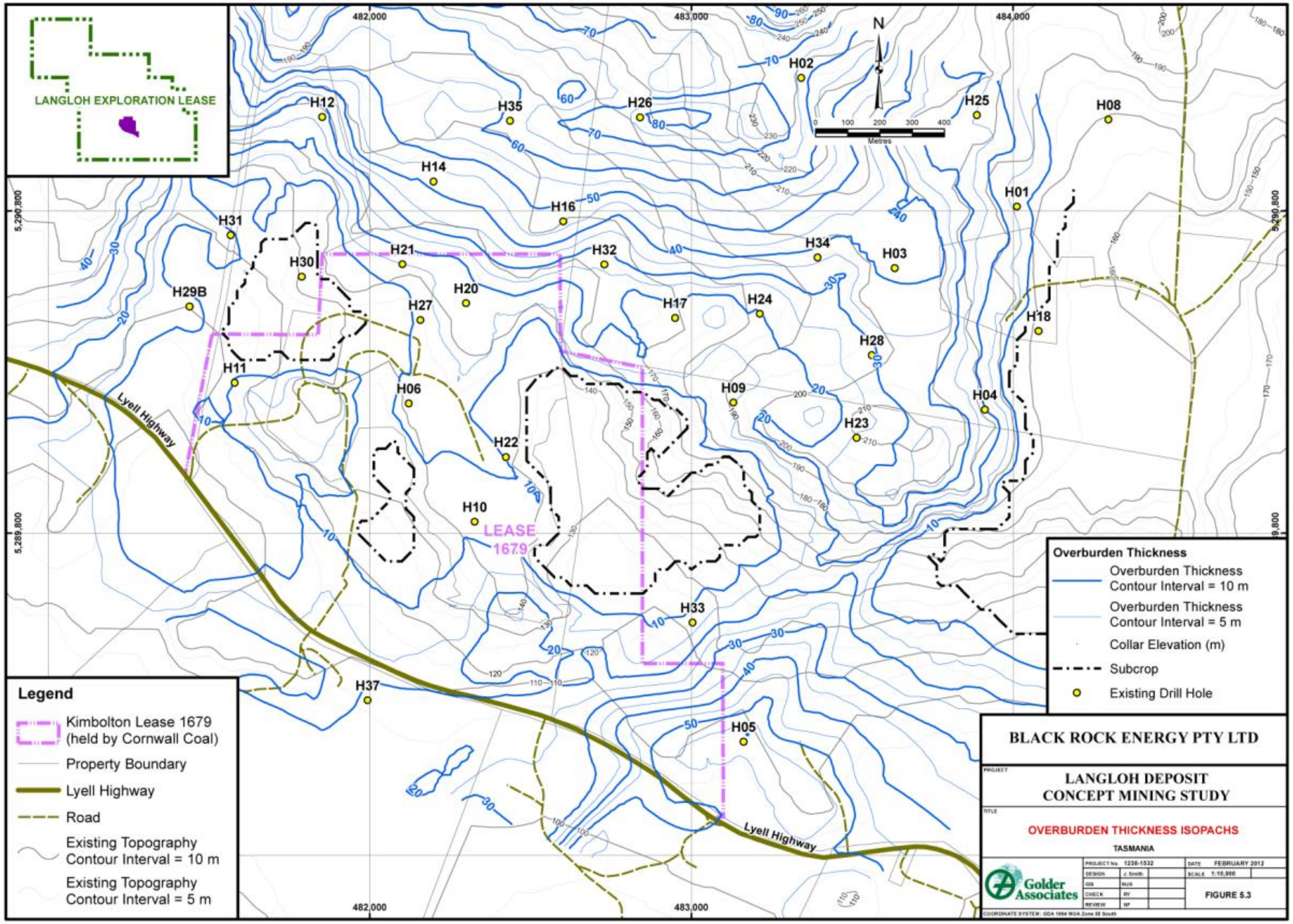


Figure 5-3 Overburden Thickness Isopachs



Most of the towers are located around the modelled crop zones of the coal seams, so stability of most of the towers should not be a major issue were the excavation is planned to be relatively shallow and the dip of the strata is low, especially given the 30 m standoff from the towers. Some towers however are located in deeper parts of the pit where seam dips are up to 8 degrees. Care was taken to minimise the chance of instability by not mining on the down dip side of the towers. In other words, the rock is left in place on the down dip side of the towers to reduce the major risk of the rock mass underneath the towers sliding down dip into an open excavation. All towers are currently located in areas that are allocated to be backfilled, so at the end of operations all towers are surrounded by original rock or fill. A detailed assessment of the towers stability should be part of future geotechnical studies as the assumptions used in this conceptual study are based on very limited data.

A detailed assessment of drill and blast procedures to reduce the risk of fly rock hitting the towers should also be examined in future studies.

It should be noted that the approach that has been adopted in this report is most likely a best case scenario as it is probable that the towers will not be moved. The true impact of the power lines will not be understood until formal talks are conducted with Transend.

The location of the power poles can be seen in Figure 5-1.

5.3.2 Coal Quality Impacts on Pit Design

As mentioned above, Seam C degrades in the east of the deposit. It is believed that the seam degrades to a carbonaceous mudstone or siltstone. This conclusion has been drawn from the very high ash (76% adb) Seam C sample in drill hole H01 and the logging of carbonaceous stone in drill holes H03 and H04 where Seam C should be located (no other coal was located below seams A and B in these holes). Seam C is therefore not a mining target in the very eastern parts of the pit.

Some of the low volatile matter (VM) coal resource in the east of the deposit may be too low to be blended with higher VM resource. Golder has not conducted any mining studies to confirm this as product marketing strategies are preliminary and the market for low VM coal has not been assessed in detail. Golder has also addressed this by working the east and west sides of the pit at the same time to increase the opportunities for blending.

Seam A also appears to exhibit variable ash across the deposit. Depending on belnding and selective mining strategy, some Seam A coal may have to be wasted as ROM ash for Seam A can reach an estimated 40% (adb) in some places. Higher ash Seam A coal could be stockpiled and blended when an opportunity arises depending on what the target quality specifications are for the product. Working the east and west sides of the pit at the same time may increase the opportunities for blending.

Due to the variable and sometimes quite high ash of SeamA, ply-by-ply analysis of Seam A may assist in identifying individual high ash plies that could be wasted to reduce the overall ash of the seam rather than potentially wasting the entire seam in some areas to lower the overall ash at Langloh. It would appear that there are currently no ply-by-ply samples for Seam A so the potential for selectively mining Seam A to reduce overall ash is unknown.

If limits on lower VM and higher ash are applied to product coal, this would reduce the size of the designed pit.

5.4 Development Schedule

Due to the overall size of the project it is not anticipated that much lead time will be necessary to purchase, erect, install or commission any of the fixed and mobile equipment. Development of haul roads, topsoil removal, drainage control, etc. also is not envisioned to take much lead time.

The more critical lead time items will be related to securing the necessary environmental and mining permit consents to begin development and mining activities. Golder has not developed an estimate of time required to complete the activities at this level of study. However, if the project advances, Golder advises that these efforts be investigated as soon as possible to ensure timely approvals.

5.5 Production Schedule

Due to the relatively small total resource size, the life-of-mine is only about eight years at 900,000 ROMt per annum. The original target was approximately 1 Mtpa and still may still be possible with a less consistent and slightly shorter schedule. As a stand alone operation, it is recommended that the production rate is not significantly increased beyond this level for the following reasons:

- Acceptable initial box cut and final void sizes
- Community and government would expect a reasonable timeframe for continuous employment
- Increased environmental impacts
- Larger equipment would be required
- More equipment would be required
- Larger capital investment for equipment, facilities and infrastructure
- Enough time to payback investment
- Train capacities of 750 t are a major constraint. The total haulage capacity of the train line is likely to be the absolute limiting factor on maximum production

Due to the size of the deposit a production rate of approximately 500,000 to 750,000 ROMt may provide a better balance between production and life–of-mine. This recommendation has not been tested to date.

Strip by strip volumes were calculated by dividing the pit into 60 m wide strips and then cutting the strips vertically to get reasonable estimates on waste and coal volumes contained in each strip. A basic schedule was constructed using these strip by strip volumes assuming that a box cut would be mined then approximately 900,000 ROMt would be mined each year after approximately two years ramp up in production. Each strip was mined in a sequence from the box cut until the pit was finished. Once each 900,000 ROMt was allocated to a year, the corresponding waste volume was added up to get a total waste volume for that year. This process was repeated until all coal and waste were allocated to a year and the pit was finished.

This simple approach has been deemed acceptable for a conceptual study due to the simple geology, flexible mining method and the single product type at Langloh.

Table 5-2, summarises the results of the conceptual level production schedule. Figire 5-4, graphically depicts the production schedule mine advance.

Table 5-2 Summary of Life-of-Mine Plan Production Statistics

Year	Coal Tonnage (ROMt)	Waste Volume (bcm)	Stripping Ratio (bcm/ROMt)
1	400,000	1,900,000	4.8
2	850,000	4,500,000	5.3
3	910,000	5,500,000	6.0
4	910,000	5,500,000	6.0
5	910,000	5,600,000	6.2
6	910,000	5,600,000	6.2
7	910,000	5,600,000	6.2
8	900,000	5,100,000	5.7
Total	**6,700,000**	**39,300,000**	**5.9**

As can be seen from Table 5-2, the life-of-mine ROM tonnes are estimated to be 6.7 Mt mined over a total mine life including ramp up production of 8 years. This tonnage has been calculated by applying the following factors to the coal resource calculated in Section 4:

- Mining factors and mining losses including dilution and depletion

- Losses due to power lines
- Pit walls must be laid back, leaving coal between the lease boundary and the final pit wall
- Sterilisation of resources due to infrastructure and expit dump location

Of the 6.7 ROM tonnes, it is estimated that 5.36 Mt of product coal will be realised (80% yield).

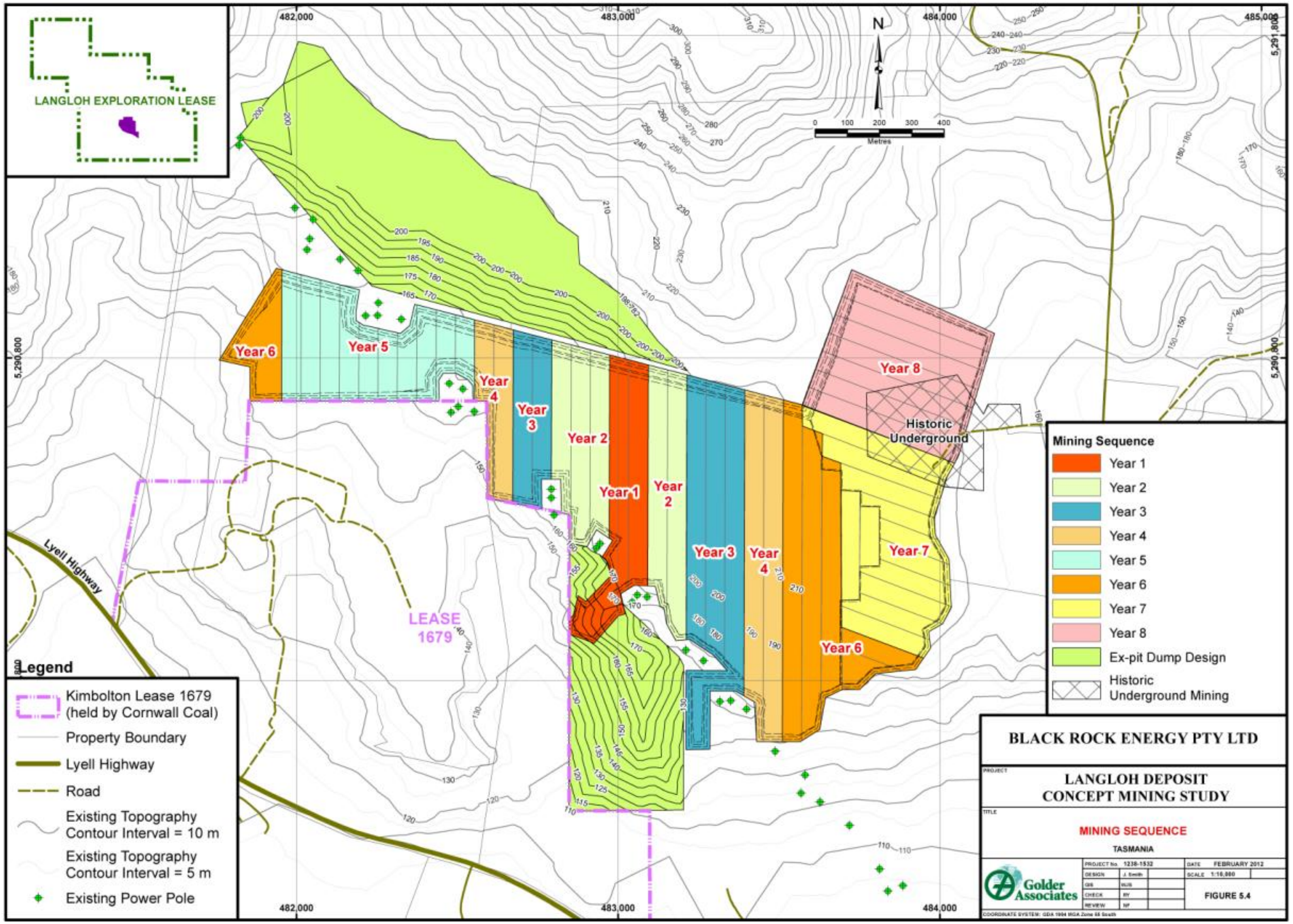
Of the total waste volume of 39.3 million bank cubic metres (Mbcm), approximately 72.5% is estimated to require blasting. The balance is assumed to be freely dug and consists of soils above the weathered horizon. Nearly 60% of the waste material is proposed to be moved by an excavator/truck fleet with the balance moved by dozers.

Average stripping ratios over the mine life are 5.9 bcm/ROM tonne or 7.3 bcm/product tonne.

Golder has also nominated and allocated sufficient ex-pit waste dump storage to place box cut waste material. As shown on Figure 5-4, the southern ex-pit dump area contains approximately 2.5 million cubic metres which is sufficient to handle most of the waste removal requirements of the initial start-up of the mine. The northern dump space as shown can accommodate approximately 5.0 million cubic metres of waste material. However, it is not envisioned that this total amount will need to be placed ex-pit. It should also be noted that the requirement for final rehabilitation has not been determined. If required, waste material from these dumps will be rehandled to assist with rehabilitating the final pit zones which are shown to occur in Year 6 and Year 8 of the plan. For this study, Golder has assumed that the final pits can be designed to remain as water impoundments with the perimeter sideslopes graded to accommodate stable slopes, revegetation and access.

Using this schedule, appropriate indicative mining equipment was chosen to achieve these targets. The selected equipment is discussed in detail in Section 6.

Figure 5-4 Mining Sequence



6.0 EQUIPMENT AND INFRASTRUCTURE

6.1 Open Cut Equipment

Regardless of the production level chosen at Langloh (500,000 t to 1 Mtpa), relatively small equipment will most likely be used to mine the deposit. It is proposed that at least one excavator and several dozers and trucks will be used for the project, with ancillary equipment including drills, a water cart, a fuel cart, maintenance vehicles, pumps and other supporting equipment.

Apart from yearly production rates, one of the biggest impacts on equipment selection will be the roster arrangement at Langloh, generally speaking an operation that operates 24 hours per day can achieve the same production using smaller equipment, compared to an operation that only works one or two shifts per day. For this study, Golder has assumed that Langloh will operate on a day/afternoon shift basis or one 12.5 hour shift per day with maintenance occurring during the night. It may be possible to mine during night shift, and it must be noted that Golder has not dismissed this option. Due to the small number of larger open cut operations in Tasmania it is difficult to establish precedence for roster arrangements. Therefore the equipment suggested in this report will be described by productivities, hours and capacities instead of specific machines.

At full production approximately 3.4 Mbcm of waste will need to be moved by an excavator and approximately 2.2 Mbcm will need to be moved by dozer push (calculated using the assumptions from Section 5.0.2) with 910,000 t of ROM coal being mined. The following indicative equipment list shown in Table 6-1 is technically capable of meeting those targets.

Table 6-1 Indicative Equipment List

Main Equipment		
Unit	**Number**	**Description**
Hitachi EX1900	1	Hydraulic Backhoe
Caterpillar 777F	4	Dump Haul Trucks
Caterpillar D11T	3	Dozers
Caterpillar D10T	1	Dozer
Driltech D45KS	1	152mm bit Diameter
Caterpillar 16M	1	Motor Grader
Caterpillar CS-56	1	Soil Compactor
Ancillary Equipment		
Unit	**Number**	**Description**
Caterpillar 992K	1	Wheel loader
Komatsu PC-200-8	1	Excavator
Caterpillar D9T	1	Dozer
Caterpillar 246C	1	Skid Steer Loader
Crew Vehicle	1	Pit vehicle
Supply Truck	1	Miscellaneous functions

After each piece of equipment was examined for actual yearly productivities, it was found that the dozer push production should be approximately 2.2 Mbcm per year with excavator waste production approximately 3.4 Mbcm per year. It is proposed that the excavator be used for both coal and waste, which would bring the total material movement for the excavator to approximately 4 Mbcm per year of waste and coal. At this stage there is approximately 0.5 Mbcm of spare capacity in the indicative excavator fleet. There are several options for reducing this spare capacity including working 1 less day per week (Sunday) or shortening each shift length.

It should be noted once again that it is difficult to estimate exactly what size and payload of equipment should be used until the roster arrangements are confirmed.

6.2 Equipment Productivity

6.2.1 Excavators

Given the average annual production statistics resulting from the schedule, Golder has selected a class of equipment similar to the Hitachi EX1900-6 hydraulic excavator in backhoe configuration matched with Caterpillar 777F 91-tonne end dump haul trucks. Golder is not recommending a specific equipment manufacturer but has referenced the specific equipment that would be suitable for the planned production requirements in order to estimate the number of units of equipment and for capital estimating purposes. Several equipment manufacturers produce similar class equipment.

Estimated production rates for the selected hydraulic excavators for planned stripping and coal loading applications are summarised in Table 6-2 Summary of Estimated Excavator Production Rate Calculations. The presented rates reflect effective annual productivities given estimated equipment operating parameters (e.g., material swell factors, material densities, bucket fill factors, cycle times, mechanical availabilities, etc.), machine usage, truck saturation and loading configurations. Productivity estimates for stripping and coal loading equipment were based on the assumption of a six-day per week, one-shift per day, 313 days per year. All shift schedules reflect 12 hours per shift. Equipment productivities include allowances for mechanical availability and utilisation.

The bucket fill factors appearing in Table 6-2 reflect the effectiveness of hydraulic excavator backhoe bucket filling. The fill factor is a function of the characteristics of the material being excavated, machine application and operator skill, and is expressed as a percentage of the rated (heaped) bucket capacity. Bucket fill factors used in excavator production derivations were based on Golder's experience.

The swell factor is defined as the percentage volumetric change of a material as a result of mining from an in situ (bank) condition to a mined condition (loose). In lieu of definitive swell characteristics, Golder has assumed a 25% swell factor for waste material and a 40% swell factor for coal.

Estimated mechanical availabilities, operating delays and other performance factors were derived for each equipment type. Truck saturation factors (i.e., the percentage of time that a truck is available for loading at the excavator) were set based on past experiences at other mining operations and adjusted for potential conditions to be experienced.

Mechanical availability is a measure of time that a piece of equipment is physically (mechanically) capable of operating. Mechanical availability is a function of the intensity of equipment usage and machine application. Equipment availabilities utilised in this conceptual study reflect Golder's experience, engineering estimates and file data. Estimated availabilities are intended to reflect average levels of mechanical availability over the effective life of a particular piece of equipment for the level of utilisation stipulated by the plan.

As seen in Table 6-2, estimated Hitachi EX1900-6 productivity is estimated at 6,615 bcm per scheduled 12-hour shift, or 4.1 Mbcm annually per machine. This equates to an average digging index of 345,000 bcm per m3 of bucket capacity, which should be achievable for an excavator of the selected size operating in pit conditions anticipated for Langloh.

Given the estimated productivities, Golder has assumed that one equipment fleet would be able to maintain and sustain waste and coal removal requirements over the mine life. On average, each excavator would work about 82% of its time in waste removal and 14% of its time in coal loading with about 4% time available.

Golder also examined the potential for utilising a slightly larger class excavator as shown on Table 6-2. However, it was determined that an excavator with a 12 cubic metre bucket capacity was a better match to planned production needs.

Table 6-2 Summary of Estimated Excavator Production Rate Calculations

% Matl. Swell: Waste =	25.0%	In Situ Matl. Weights:	Overburden (tonnes/cu.m.)	2.50	Stripping Hours Per Shift = 12.0
Coal =	40.0%		Interburden (tonnes/cu.m.)	2.50	Coal Loading Hours Per Shift = 12.0
			Coal (tonnes/cu.m.) =	1.63	Days Per Year = 313

MACHINE	TRUCK FLEET Capacity (tonnes)	EFF. TRUCK PAYLOAD CAPACITY (tonnes)	N0. OF PASSES	RATED BUCKET CAPACITY (cu. m.) [B]	BUCKET FILL FACTOR [FF]	SWELL FACTOR [S]	EFFECTIVE BUCKET CAPACITY (1) (bcm) [EB1]	(tonne) [EB2]	CYCLE TIME (secs.) [CT]	NOMINAL SHIFT SCHED.	MECH. AVAIL. (2)	OPER. USAGE (3)	ESTIMATED PROD. RATE PER SHIFT (4)	ESTIMATED ANNUAL PRODUCTION CAPACITY (5)
STRIPPING MACHINES														
Liebherr R9250	91	100.8	4	14.0	0.900	0.800	10.1	25.2	36	6 x 1	87.0%	73.3%	7,720 bcm	4,832,500 bcm
Hitachi EX1900-6 - Backhoe	91	86.4	4	12.0	0.900	0.800	8.6	21.6	36	6 x 1	87.0%	73.3%	6,615 bcm	4,141,000 bcm
COAL LOADING MACHINES														
Liebherr R9250	91	92.9	6	14.0	0.950	0.714	9.5	15.5	36	6 x 1	87.0%	73.3%	11,855 tonnes	7,421,000 tonnes
Hitachi EX1900-6 - Backhoe	91	92.9	7	12.0	0.950	0.714	8.1	13.3	36	6 x 1	87.0%	73.3%	10,165 tonnes	6,363,500 tonnes

Notes: (1) Effective Bucket Capacity In Bank Cubic Metres ("bcm") = EB1 = B x FF x S
Effective Bucket Capacity In Tonnes = EB1 x Material Weight
(2) Mechanical Availability = Avail. Hours / Sched. Hours
(3) Operational Usage = Working Hours / Avail. Hours
(4) Rate At Given Mech. Avail. and 75% to 95% Truck Saturation
= EB1 or EB2 x (3600 / CT) x Hours Per Shift x Mech. Avail. x Oper. Utilization
(5) Based On 1,080 (8-hour) Shifts Per Year for 7 x 3 Shift Schedule

6.2.2 Trucks

For the conceptual study, Golder did not perform waste or coal haulage profiles to estimate cycle time and productivity capabilities for the truck fleets. However, Golder utilised past work experience in combination with anticipated waste and coal haul distances at Langloh to estimate the number of waste and coal haul trucks needed,

Based on this information, Golder has assumed that each excavator would be matched with three end dump trucks. Given typical mechanical availabilities of the end dump trucks, Golder has assumed that the total fleet requirements would result in 4 end dump trucks required on the site. This will account for downtime and the added length of coal haul as the mine develops.

Dozers

In addition to the primary excavator and truck fleet, Golder has also assigned approximately 40% of the waste volume to be moved by production dozers. As noted above, this equates to about 2.2 Mbcm per year.

Based on these needs in combination with estimated dozer productivities, Golder has selected a class of dozer similar to a Cat D11T, 611 kW machine. Utilising production curve derivations from the Cat Handbook Edition 34, Golder has estimated the production capability assuming a 120 m average push distance over a 0% average working grade. This results in an unadjusted maximum production of 640 loose cubic metres per hour (lcm/hr). Given the anticipated shift schedule, mechanical availability and operational utilisation of the dozer, Golder has estimated that a dozer of this class can achieve 2,355 bcm/scheduled shift.

At the above rate, a production dozer can achieve nearly 740,000 bcm per year. This equates to the Langloh operation requiring 3 – Cat D11T class dozers.

6.2.3 Drills

Golder has estimated that approximately 4 to 4.4 Mbcm of waste will have to be blasted per year. It has been calculated that a Sandvik D45KS drill, drilling a 20m bench with a 6x6.5m pattern and using a 0.51 powder factor, could be capable of blasting approximately 4.1 Mbcm per year. For the purposes of this study this has been deemed acceptable, as there should be sufficient room in the two working areas of the pit to build up a blasted inventory.

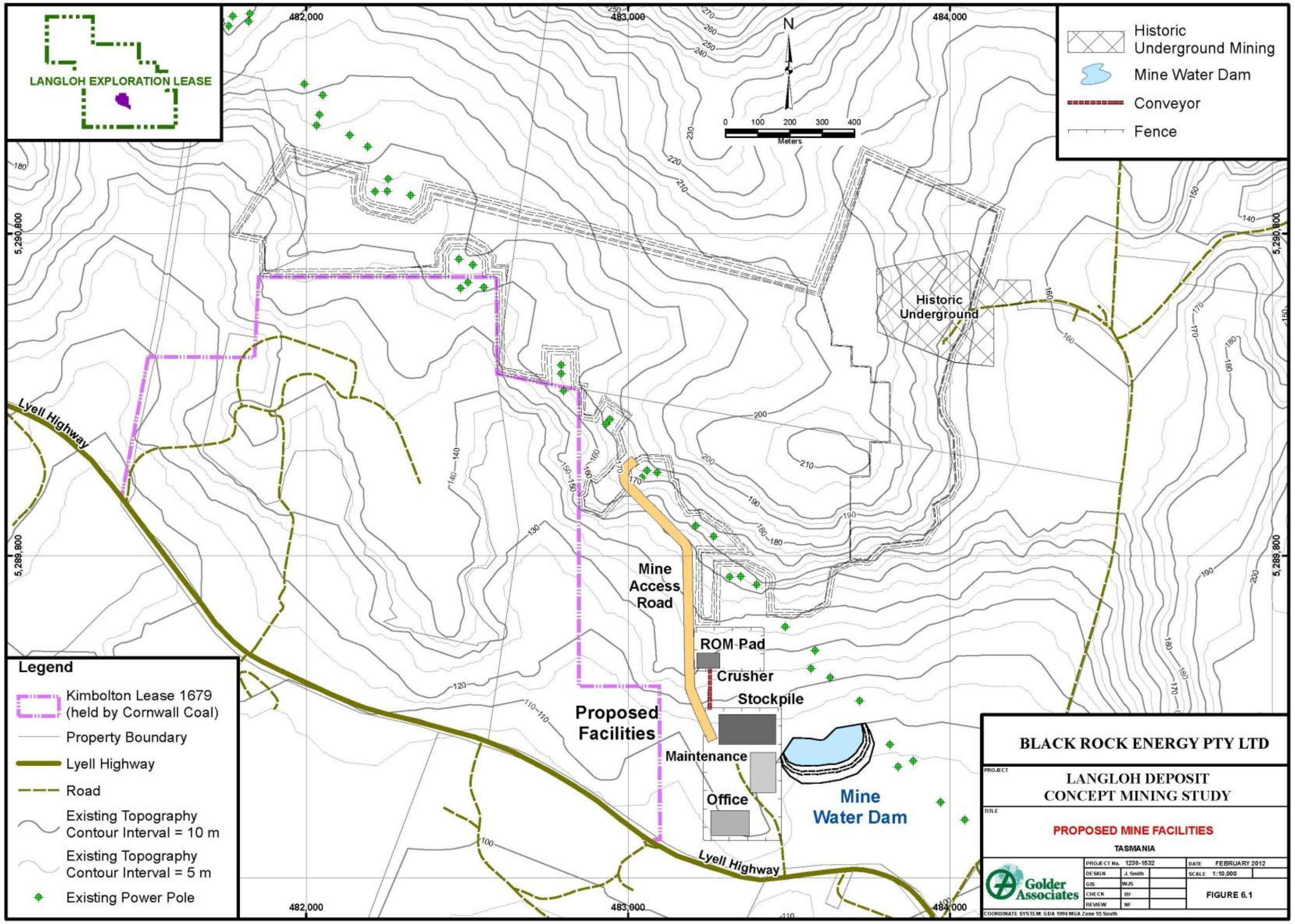
6.3 Infrastructure

The proposed infrastructure for Langloh should be kept to a minimum to control capital costs. Due to the short mine life, items should be temporary and non-permanent, with second hand items being an attractive cost saving alternative. It is proposed that the following structures and items are required for Langloh:

- Office building for administration (portable style)
- Maintenance area or shed (or a covered, free draining and compacted pad)
- ROM pad, conveyor to stockpile and crusher
- Park up area (could combine with ROM pad or add space around maintenance area for small amount of trucks)
- Mine road to allow access from pit to ROM and maintenance areas
- Water management structures including water storage dam for mine

The conceptual locations for these items can be seen in Figure 6-1. The ROM pad has been kept as close as possible to the pit to minimise haulage distances and to reduce trucks climbing up and down the hill. The majority of the infrastructure has been located towards the Lyell highway on a flatter, more open area with a conveyor linking the ROM crusher to the stockpile.

Figure 6-1 Proposed Mine Facilities



Water management at all mine sites is important and thought must be given to water management structures. Mine water and dirty runoff must be directed towards the pit or the mine water dam. Clean water runoff should be directed away from the mine water management system where possible. Drains will have to be moved as the mine progresses to separate clean and dirty water. Other minor sediment controlling structures such as traps and small dams may be required.

6.3.1 Services

Water could be sourced from the following:

- Water licences from purchased properties
- Onsite sources including pit water and water stored in onsite dams
- Meadowbank Lake

Golder has not verified that any of these water sources are available. It is common mining practice to have at least one sizeable dam onsite for water management and storage purposes.

Electricity could be sourced from the power lines running through the project. It is anticipated that a substation would have to be built somewhere near the project. Golder has not attempted to estimate a cost for a substation.

6.4 Logistics

6.4.1 Product Coal Logistics

Three options have been identified for transporting product coal from the proposed Langloh mine to export shipping facilities. These are road transport, rail transport and barging. All options would involve an element of trucking.

6.4.2 Road Transport

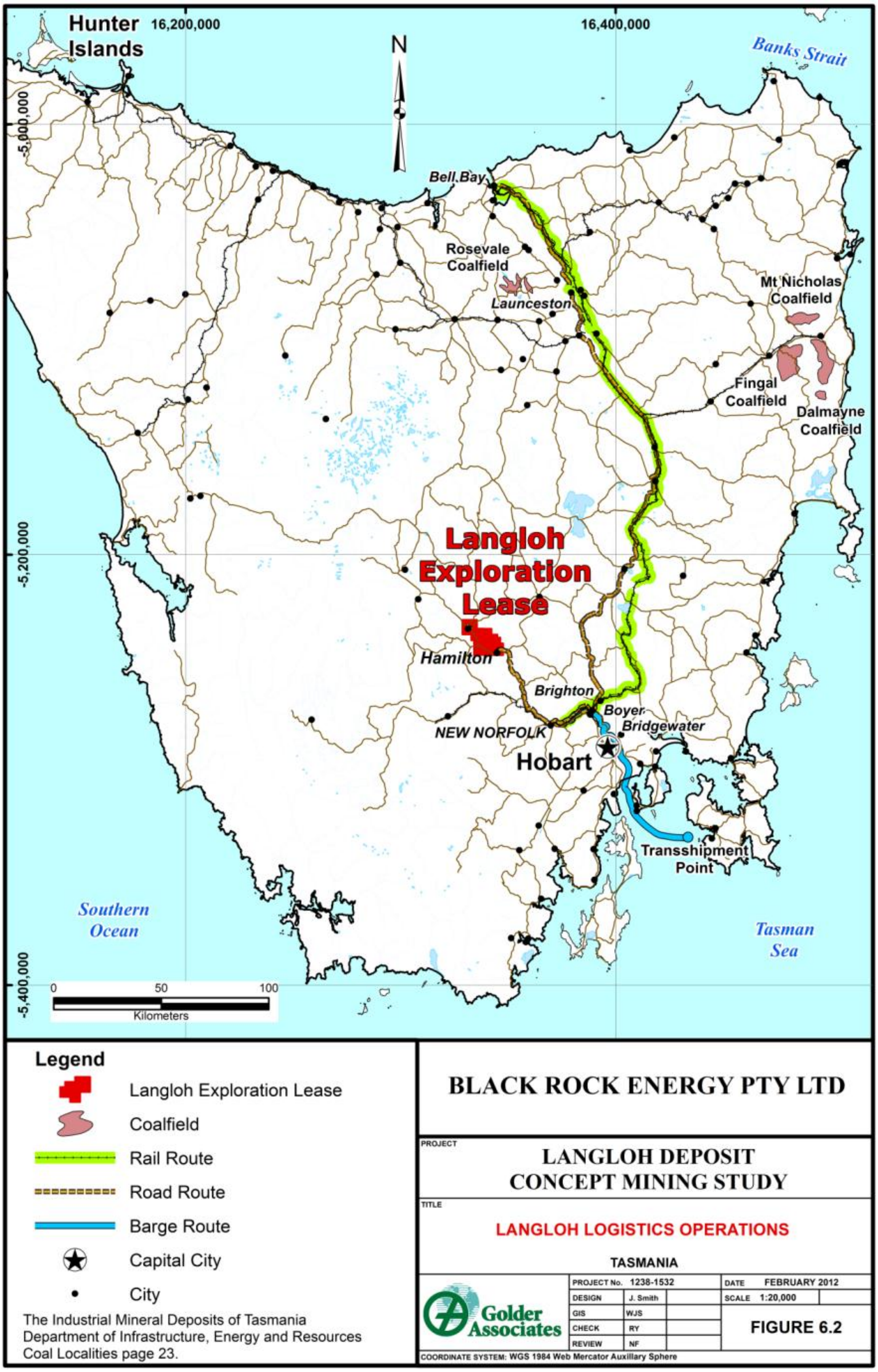
The road transport option would require product coal to be loaded from the mine stockpile onto trucks and transported to Bell Bay. The probable route would be from Langloh to New Norfolk via the Lyell Highway, from New Norfolk to Bridgewater via Boyer Road, from Bridgewater to Launceston via Midland Highway, and from Launceston to Bell Bay via East Tamar Highway. The proposed route is shown in Figure 6-2 Langloh Logistics Options. All roads along this route are sealed and in good condition. Truck loads would have to be covered to eliminate dust. The total travel distance is approximately 286 km. At this stage, it is not known what size or type of trucks would be permitted on this route, however truck loads would need to be covered to reduce dust. No discussions regarding trucking the product with relevant authorities have been initiated at this time.

This is probably the least attractive option due to the relatively low productivity of trucking and potentially higher unit cost than compared to alternate options. It is, however, noted that CCC is trucking coal from the Kimbolton Mine to a coal preparation plant at Fingal, a distance of 238 km.

6.4.3 Barge Transport

The barge transport option would require product coal to be loaded from the mine stockpile onto trucks and transported to a barge loading facility on the Derwent River near Boyer. The trucking distance is approximately 48 km. The barge loading facility would have to be sufficiently distant from the Norske Skog paper mill at Boyer to ensure that coal dust is not an issue. Coal would be loaded onto barges and towed to a sheltered location near the mouth of the Derwent River for transfer to Handymax vessels using ships gear. The proposed route is shown in Figure 6-2 Langloh Logistics Options.

Figure 6-2 Langloh Logistics Options



This route would require transit under three bridges over the Derwent River. The Bridgewater river crossing comprises a causeway and a low level steel bridge with a central vertical lift section that can be raised to allow passage of vessels. The bridge carries a two lane roadway and a single line railway. It is not known if the raising mechanism is still operable or the height to which it can be raised. The cables that support the concrete counterweights for raising the bridge are reported to have maintenance problems. Until 1984, the paper mill at Boyer moved all products by barge down river to Hobart for trans-shipment. The size of barge operated by the mill is unknown.

The Bowen Bridge has a central span of 46 m wide with a clearance of 14.5 m. It is a concrete bridge carrying a four lane roadway with no issues having been identified to date.

The Tasman Bridge has a central span of 45 m wide and is 95 m high. Water depth is 48 m. Vessels over 35 m length are required to have a pilot when passing under the bridge. Handysize bulk carriers pass regularly under the bridge transporting zinc concentrate to the refinery at Risdon.

The depth of the Derwent River near Boyer is unknown. Travel distance to a theoretical trans-shipment point south of Hobart would be approximately 45 km.

An alternate barge loading facility south of the Bridgewater Bridge does not appear possible due to river front residential development, steep terrain in some areas and the presence of Eastern Derwent Highway along the river front in places.

No discussions regarding barging have been initiated at this time and it is not known if the required permits could be obtained. This is potentially the most attractive option due to lowest cost but it would result in a longer sea transit distance and limit ship size to geared Handymax.

6.4.4 Rail Transport

TasRail operate a narrow gauge rail system that is currently undergoing redevelopment. The rail transport option would require product coal to be loaded from the mine stockpile onto trucks and transported to a rail loading facility near Boyer. Trucking distance is approximately 48 km. The rail loading facility would have to be sufficiently distant from the Norske Skog paper mill at Boyer to ensure coal dust is not an issue.

TasRail operates a narrow gauge rail system, capable of carrying a 16 tonne axle load, between ports in the north of the island and the Hobart region to the south and the dominant freight is containers. The main line from Hobart to Launceston and Bell Bay has a connecting line from Boyer that joins at Bridgewater. Rail distance from Boyer to Bell Bay is approximately 250 km. An alternate location for a rail loading facility is at the Intermodal Freight Hub currently under construction near Brighton. It is operated by TasRail and is due to be completed in mid-2012. This would require product coal to be trucked approximately 66 km and then hauled by rail 230 km to Bell Bay. The trucking route to Brighton would be via the Lyell Highway and Boyer Road. The proposed route is shown in Figure 6-2. This route is currentyl used by trucks with articlated trailers that haul raw timber.

An alternate short term option could utilise open top containers that would be loaded at the mine, trucked to Brighton and loaded to intermodal rail wagons for transport to Bell Bay. This option could be utilised during the mine ramp up phase while stockpile and loading facilities were being completed.

Initial discussions have been held with TasRail who advised that they currently have no bulk cargo wagons suitable for transporting coal. Investment in new rolling stock will require definite commitment from users. TasRail have recently ordered 17 new more powerful locomotives that will replace a mixture of older variably powered units. The new locomotives have a maximum haulage capacity of 750 tonnes and will be in service between 18 months and 2 years. TasRail advised that the southern 80 km of line from Hobart to Parattah was steep with tight curves and required additional locomotive assistance. A tunnel on this route also imposes constraints that should be overcome by the new locomotives. TasRail believe there is sufficient capacity on the existing line for the additional volumes generated by coal transport. This however is subject to confirmation by modelling.

TasRail were uncertain of the size of unit trains and wagons they would operate to haul coal and were currently conducting a study. They advised they had similar discussion with HardRock Coal Mining Pty Ltd

(HardRock) who is proposing to develop an export coal mine near Fingal. Further information will be provided by TasRail in the near future.

The combined truck and rail option using dedicated coal wagons appears the most probable long term transport option for the Langloh Mine at this time. Table 6-3 Estimated Train Movements, summarises the average daily and weekly tonnage hauled and the number of unit trains required.

Table 6-3 Estimated Train Movements

Working days	Product per annum (t)	Axle Load (t)	Track Gauge	Max Haulage Capacity (t)	Required Product Hauled (t)		Number of trains required	
					Daily	Weekly	Daily	Weekly
5	720,000	16	Narrow	750	2769	13,846	4	19
6	720,000	16	Narrow	750	2308	13,846	3	19

6.4.5 Bell Bay Port Facilities

The Bell Bay port facilities are operated by TasPorts and were inspected on 22 February 2012. Bell Bay is near the mouth of the Tamar River on the north coast of Tasmania. It has deep water and can be accessed by Handymax and Panamax vessels, although the latter cannot be fully loaded due to restrictions at the river mouth which has maximum depth 14.6 m. Rail lines at the port service six berths that handle a variety of bulk materials including wood chips, timber, ferro-manganese, coke, alumina, refined aluminium and petroleum products.

Berth 2, a former roll-on-roll-off container terminal is currently unused and has an open wharf area approximately 100 m x 200 m that could be used as a stockpile area. TasPorts advised this could accommodate 100,000 - 180,000 tonnes of product, depending on the number of users. To utilise this berth incremental capital expenditure will be required to extend the existing rail line approximately 500 m, construct a bottom dump unloading hopper and a short product stacking conveyor to the stockpile area. TasPorts propose using mobile equipment to manage and reclaim product on the stockpile. The stockpile would have to be partially covered to prevent coal dust contaminating nearby wood chip stockpiles and run-off controls would have to be constructed.

The current berth is not suitable for Handymax or larger vessels and a new wharf and ship loader would have to be constructed 50 m from the dock in water deep enough to accommodate Panamax vessels. TasPorts has proposed that a ship loading facility currently operated by Temco to load ferro-manganese products could be used temporarily while permanent loading facilities are constructed.

TasRail has also proposed that mobile unloading and tipping device could be leased and used as a temporary measure to unload coal in containers while permanent loading and unloading facilities are completed.

Initial discussions have been held with TasPorts and further information will be provided in the near future. Any development at Bell Bay will also have to take into consideration the requirements of other users such as Hard Rock.

7.0 RISK ASSESSMENT

Golder has conducted a high level risk assessment in order to evaluate risks that may be encountered and should be addressed in developing the project through to a pre-feasibility study level. Golder has prepared this assessment based on available information and therefore this cannot be relied upon as a complete risk assessment for the project. BRE should conduct their own assessment using all internal information in order to assess all risks.

A simple risk assessment matrix, shown in Figure 9.1, has been used to classify a number of identified hazards. The ranking and mitigation measures for these identified risks are presented in Table 9.1.

Figure 7-1 Risk Assessment Matrix

			Likelihood				
			1	2	3	4	5
			Rare	**Unlikely**	**Moderate**	**Likely**	**Almost Certain**
Consequence	5	**Catastrophic**	5	10	15	20	25
	4	**Significant**	4	8	12	16	20
	3	**Moderate**	3	6	9	12	15
	2	**Minor**	2	4	6	8	10
	1	**Insignificant**	1	2	3	4	5

The assessment identifies two high risk hazards and five moderate risk hazards. This reflects the current conceptual nature of the project. Any identified high risks should be mitigated to a lower risk level prior to commencement of the next phase of the project. Furthermore, any identified moderate risks must have appropriate management strategies defined.

The high risk items apparent in upgrading the Langloh project to a pre-feasibility study includes:

- Unable to obtain mining approvals – If the appropriate licenses and permits are not granted, the project will be unable to progress resulting in the inability to mine. This includes both mining and environmental approvals. Golder recommends BRE commence the approval process as soon as possible, including appropriate discussions with MRT and EPA.
- Poor coal quality data – If sufficient, reliable coal quality data is not collected the Coal Preparation Plant design may be adversely impacted. Golder has recommended an additional drilling plan be properly constructed, managed and executed, in accordance with clearly defined data requirements including laboratory reporting requirements.

Table 7-1 Risk Assessment

Hazard	Consequence	Consequence Description	L	C	Risk Ranking	Mitigation Measures
Unable to secure a product transport route	Unable to transport product to market	No agreements with TasRail, Tasports etc	2	5	10	Negotiate agreements with port, rail and road authorities. Potential sites to be identified. Loading facilities at transfer points to be designed and costed
Unable to obtain mining approvals	Unable to mine	MRT, EPA do not grant necessary permits/approvals	3	5	15	Commence approval process as soon as practicable. Ensure scope of work covers all permitting requirments. Commence discussions with MRT, EPA, council and other stakeholders
Poor project management	Cost and timing overrun	Timelines, deadlines etc underestimated	3	4	12	Develop detailed project management plan. Appoint qualified project manager
Poor quality of technical data (Drilling, geotechnical data etc)	Poor geological model, pit design and other technical designs	Innaccurate assessment of resoucrses, product quality and financial model	3	4	12	Clearly define data requirements and laboratory reporting requirements. Identify additional drilling and testing plan.
Poor coal quality data (Materials handling, quality etc)	Unsuitable blending plans and materials handling design	Poor product quality and unsuitable handling characteristics	3	5	15	Clearly define data requirements and laboratory reporting requirements for materials handling equipment design
Unable to access sufficient water supply	Insufficient water for mine operations, high dust levels	Unable to draw water from Meadowbank Lake, insufficient onsite dam capacity	2	5	10	Confirm water supply with local/state governments. Design appropriate onsite water storage. Purchase water licences from local irregators
Unable to mine under powerlines	Loss of coal resources	Not obtaining approval/developing a management plan with Transend	2	3	6	Confirm restrictions with Transend and develop a management plan. Obtain approval for plan.

Key:
L = Likelihood
C = Consequence

8.0 REFERENCES

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Report Signature Page

Neil Fraser
Principal Mining Consultant

Robert Yarkosky, P.E., RPEQ MAusIMM, CP (Min)
Associate - Mining Practice Leader

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APPENDIX A

Coal Quality Data

FLOAT SINK DATA														
Marston Float Sink Data														
Drill Hole	Sample No.	Seam	From Depth (m)	To Depth (m)	Sample Thick (m)	Raw Ash	RD	Mass (kg, adb)	RD x thick	Sink-Float Fraction	Fraction Mass %	Fraction Ash %	Cumulative Mass %	Cumulative Ash %
LANG_H29B	004	C	14.070	14.430	0.360	19.300	1.490	1.045	0.536	F1.30	0.47	10.80	0.47	10.80
										S1.30 - F1.40	62.63	10.80	63.10	10.80
										S1.40 - F1.50	22.52	17.70	85.62	12.61
										S1.50 - F1.60	4.07	26.50	89.69	13.24
										S1.60 - F1.70	1.89	40.20	91.58	13.80
										S1.70 - F1.80	0.95	40.20	92.53	14.07
										S1.80 - F1.90	1.61	53.20	94.13	14.74
										S1.90 - F2.00	0.85	53.20	94.99	15.08
										S2.00 - F2.20	2.55	63.50	97.54	16.35
										S2.00	2.46	77.20	100.00	17.85
LANG_H30	001	C	3.400	3.700	0.300	30.100	1.720	0.507	0.516	F1.30	0.00	0.00	0.00	0.00
										S1.30 - F1.40	0.00	0.00	0.00	0.00
										S1.40 - F1.50	35.91	21.10	35.91	21.10
										S1.50 - F1.60	38.89	25.90	74.80	23.60
										S1.60 - F1.70	10.71	32.50	85.52	24.71
										S1.70 - F1.80	4.96	33.80	90.48	25.21
										S1.80 - F1.90	1.39	39.30	91.87	25.42
										S1.90 - F2.00	0.79	39.30	92.66	25.54
										S2.00 - F2.20	0.00	0.00	92.66	25.54
										S2.00	7.34	69.10	100.00	28.74
LANG_H31	007	C	49.480	49.820	0.340	57.500	1.910	1.414	0.649	F1.30	0.00	0.00	0.00	0.00
										S1.30 - F1.40	0.00	0.00	0.00	0.00
										S1.40 - F1.50	1.58	27.10	1.58	27.10
										S1.50 - F1.60	8.78	27.80	10.37	27.69
										S1.60 - F1.70	25.20	42.60	35.57	38.25
										S1.70 - F1.80	16.27	50.50	51.84	42.10
										S1.80 - F1.90	8.71	56.00	60.55	44.10
										S1.90 - F2.00	4.10	62.40	64.65	45.26
										S2.00 - F2.20	19.80	73.00	84.45	51.76
										S2.00	15.55	78.80	100.00	55.97
LANG_H32	011-013	A+B	27.500	29.850	2.350	27.105	1.584	6.824	3.722	F1.30	1.93	9.80	1.93	9.80
										S1.30 - F1.40	12.83	10.60	14.77	10.50
										S1.40 - F1.50	43.55	17.60	58.31	15.80
										S1.50 - F1.60	19.50	25.10	77.81	18.13
										S1.60 - F1.70	5.56	34.80	83.37	19.24
										S1.70 - F1.80	3.02	41.80	86.39	20.03
										S1.80 - F1.90	1.62	51.30	88.01	20.60
										S1.90 - F2.00	1.76	56.60	89.77	21.31
										S2.00 - F2.20	1.96	62.20	91.73	22.19
										S2.00	8.27	85.10	100.00	27.39
LANG_H32	015-016	C	31.580	32.760	1.180	25.279	1.545	2.666	1.823	F1.30	7.93	10.90	7.93	10.90
										S1.30 - F1.40	33.25	12.20	41.18	11.95
										S1.40 - F1.50	29.91	19.30	71.09	15.04
										S1.50 - F1.60	9.23	27.30	80.32	16.45
										S1.60 - F1.70	5.67	36.90	85.99	17.80
										S1.70 - F1.80	2.71	43.50	88.70	18.58
										S1.80 - F1.90	2.48	51.00	91.18	19.47
										S1.90 - F2.00	1.56	59.40	92.74	20.14
										S2.00 - F2.20	2.85	64.20	95.59	21.45
										S2.00	4.41	75.40	100.00	23.83
LANG_H33	009	C	17.000	17.960	0.960	23.000	1.520	2.693	1.459	F1.30	17.17	8.90	17.17	8.90
										S1.30 - F1.40	20.64	11.50	37.81	10.32
										S1.40 - F1.50	29.99	21.40	67.80	15.22
										S1.50 - F1.60	17.10	28.80	84.90	17.95
										S1.60 - F1.70	5.39	35.40	90.29	19.00
										S1.70 - F1.80	0.96	42.90	91.25	19.25
										S1.80 - F1.90	0.85	52.80	92.10	19.56
										S1.90 - F2.00	0.59	61.10	92.69	19.82
										S2.00 - F2.20	1.26	68.30	93.94	20.47
										S2.00	6.06	74.70	100.00	23.75
LANG_H34	018-019	A+B	36.040	38.210	2.170	26.116	1.585	1.738	3.439	F1.30	0.57	10.89	0.57	10.89
										S1.30 - F1.40	17.71	10.89	18.28	10.89
										S1.40 - F1.50	31.32	14.30	49.60	13.04
										S1.50 - F1.60	20.62	22.04	70.22	15.68
										S1.60 - F1.70	10.99	31.46	81.21	17.82
										S1.70 - F1.80	5.01	40.08	86.22	19.11
										S1.80 - F1.90	3.47	51.08	89.69	20.35
										S1.90 - F2.00	2.79	54.44	92.48	21.38
										S2.00 - F2.20	4.90	63.29	97.38	23.49
										S2.00	2.62	67.05	100.00	24.63
LANG_H34	020	C	39.320	40.590	1.270	24.300	1.550	1.310	1.969	F1.30	2.94	8.20	2.94	8.20
										S1.30 - F1.40	33.16	10.80	36.10	10.59
										S1.40 - F1.50	27.73	14.39	63.83	12.24
										S1.50 - F1.60	11.30	23.80	75.13	13.98
										S1.60 - F1.70	7.99	33.79	83.12	15.88
										S1.70 - F1.80	3.92	41.54	87.04	17.04
										S1.80 - F1.90	2.34	47.73	89.37	17.84
										S1.90 - F2.00	1.88	54.86	91.26	18.60
										S2.00 - F2.20	3.84	63.42	95.10	20.41
										S2.00	4.90	74.53	100.00	23.07
LANG_H35	021	A	61.770	62.940	1.170	27.100	1.570	3.549	1.837	F1.30	0.28	12.40	0.28	12.40
										S1.30 - F1.40	11.03	12.40	11.31	12.40
										S1.40 - F1.50	41.24	18.40	52.55	17.11
										S1.50 - F1.60	16.86	26.10	69.41	19.29
										S1.60 - F1.70	11.62	36.20	81.03	21.72
										S1.70 - F1.80	5.13	45.80	86.15	23.15
										S1.80 - F1.90	5.15	51.50	91.31	24.75
										S1.90 - F2.00	2.06	57.80	93.37	25.48
										S2.00 - F2.20	4.68	62.30	98.05	27.24
										S2.00	1.95	64.60	100.00	27.97
LANG_H35	023	B	63.900	65.160	1.260	21.600	1.540	3.183	1.940	F1.30	0.90	9.00	0.90	9.00
										S1.30 - F1.40	20.43	9.20	21.34	9.19
										S1.40 - F1.50	34.44	15.20	55.78	12.90
										S1.50 - F1.60	22.17	23.00	77.95	15.77
										S1.60 - F1.70	8.57	31.90	86.52	17.37
										S1.70 - F1.80	3.63	40.30	90.16	18.30
										S1.80 - F1.90	2.67	44.40	92.83	19.05
										S1.90 - F2.00	1.96	47.00	94.78	19.62
										S2.00 - F2.20	3.04	58.30	97.83	20.83
										S2.00	2.17	63.00	100.00	21.74
LANG_H35	024-025	C	68.250	69.610	1.360	20.875	1.498	3.617	2.037	F1.30	0.71	8.70	0.71	8.70
										S1.30 - F1.40	36.04	8.80	36.75	8.80
										S1.40 - F1.50	31.66	15.10	68.42	11.71
										S1.50 - F1.60	9.79	24.30	78.21	13.29
										S1.60 - F1.70	5.71	34.10	83.92	14.71
										S1.70 - F1.80	2.86	43.80	86.78	15.66
										S1.80 - F1.90	2.45	50.90	89.23	16.63
										S1.90 - F2.00	1.47	56.60	90.70	17.28
										S2.00 - F2.20	5.88	72.10	96.57	20.61
										S2.00	3.43	75.00	100.00	22.48
LANG_H35	026	D	70.330	70.990	0.660	27.300	1.560	1.825	1.030	F1.30	8.29	11.20	8.29	11.20
										S1.30 - F1.40	21.13	12.50	29.41	12.13
										S1.40 - F1.50	27.68	18.90	57.10	15.41
										S1.50 - F1.60	13.11	27.80	70.21	17.73
										S1.60 - F1.70	8.18	34.50	78.39	19.48
										S1.70 - F1.80	3.85	47.40	82.23	20.78
										S1.80 - F1.90	2.44	52.40	84.67	21.69
										S1.90 - F2.00	1.84	57.50	86.51	22.46
										S2.00 - F2.20	4.44	69.40	90.95	24.75
										S2.00	9.05	81.40	100.00	29.87
Composite	All seams Does not include H30 (weathered) or H31 (high ash)									F1.30	3.38	9.75	3.38	9.75
										S1.30 - F1.40	22.91	10.71	26.29	10.59
										S1.40 - F1.50	33.87	16.86	60.16	14.12
										S1.50 - F1.60	16.02	24.62	76.18	16.33
										S1.60 - F1.70	7.65	33.88	83.83	17.93
										S1.70 - F1.80	3.50	42.35	87.33	18.91
										S1.80 - F1.90	2.59	50.33	89.93	19.82
										S1.90 - F2.00	1.84	55.31	91.77	20.53
										S2.00 - F2.20	3.60	64.73	95.37	22.20
										S2.00	4.63	77.29	100.00	24.75

Capricorn Float Sink Data														
Drill Hole	Sample No.	Seam	From Depth (m)	To Depth (m)	Sample Thick (m)	Raw Ash	RD	Mass (kg, adb)	RD x thick	Sink-Float Fraction	Fraction Mass %	Fraction Ash %	Cumulative Mass %	Cumulative Ash %
H-01		A + B	13.92	18.94	5.02	23.1	1.47		7.38	F1.40	49.2	10.2	49.2	10.2
										S 1.40-F 1.60	36.2	22.7	85.4	15.5
										S 1.60-F 1.80	7.9	42.8	93.3	17.8
										S 1.80	6.7	66.3	100.0	21.1
H-03		A	42.31	44.19	1.88	24.5	1.47		2.76	F1.40	33.8	11.4	33.8	11.4
										S 1.40-F 1.60	49.3	24.0	83.1	18.9
										S 1.60-F 1.80	11.4	40.7	94.5	21.5
										S 1.60	5.5	57.0	100.0	23.5
H-03		B	45.81	47.05	1.24	19.4	1.44		1.79	F1.40	55.8	10.1	55.8	10.1
										S 1.40-F 1.60	31.8	22.2	87.6	14.5
										S 1.60-F 1.80	8.5	40.2	96.1	16.8
										S 1.60	56.3	100.0	100.0	18.3
H-04		A + B	21.28	24.53	3.25	23.1	1.54		5.01	F1.40	43.6	10.2	43.6	10.2
										S 1.40-F 1.50	30.1	20.0	73.7	14.2
										S 1.50-F 1.60	11.2	28.0	84.9	16.0
										S 1.60-F 1.70	6.4	36.8	91.3	17.5
										S 1.70-F 1.80	5.3	43.4	96.6	18.9
										S 1.80	3.4	56.1	100.0	20.2
H-06		A + B	13.30	15.04	1.74	22.9	1.53		2.66	F1.40	44.3	13.8	44.3	13.8
										S 1.40-F 1.50	23.6	19.3	67.9	15.7
										S 1.50-F 1.60	15.0	28.0	82.9	17.9
										S 1.60-F 1.70	6.2	36.0	89.1	19.2
										S 1.70-F 1.80	4.1	43.9	93.2	20.3
										S 1.80	6.8	51.2	100.0	22.4
H-06		C	17.23	18.21	0.98	15.2	1.47		1.44	F1.40	44.3	13.8	44.3	13.8
										S 1.40-F 1.50	23.6	19.3	67.9	15.7
										S 1.50-F 1.60	15.0	28.0	82.9	17.9
										S 1.60-F 1.70	6.2	36.0	89.1	19.2
										S 1.70-F 1.80	4.1	43.9	93.2	20.3
										S 1.80	6.8	51.2	100.0	22.4
H-06			88.76	89.00	0.24	31.5	1.57		0.38	F1.40	44.3	13.8	44.3	13.8
										S 1.40-F 1.50	23.6	19.3	67.9	15.7
										S 1.50-F 1.60	15.0	28.0	82.9	17.9
										S 1.60-F 1.70	6.2	36.0	89.1	19.2
										S 1.70-F 1.80	4.1	43.9	93.2	20.3
										S 1.80	6.8	51.2	100.0	22.4
H-07		A	15.73	16.94	1.21	31.4	1.57		1.90	F1.40	22.8	13.6	22.8	13.6
										S 1.40-F 1.50	25.1	20.7	47.9	17.3
										S 1.50-F 1.60	14.9	29.6	62.8	20.2
										S 1.60-F 1.70	12.2	38.9	75.0	23.3
										S 1.70-F 1.80	12.6	49.3	87.6	27.0
										S 1.80	12.4	58.4	100.0	30.9
H-09	S2162-64	A + B	15.35	17.43	2.08	31.4	1.57		3.27	F1.40	12.9	13.2	12.9	13.2
										S 1.40-F 1.50	43.0	21.1	55.9	19.3
										S 1.50-F 1.60	18.1	29.3	74.0	21.7
										S 1.60-F 1.80	8.2	44.4	82.2	24.0
										S 1.80	17.8	78.6	100.0	33.7
H-09	S2166	C	19.13	20.18	1.05					F1.40	46.0	11.6	46.0	11.6
										S 1.40-F 1.50	34.9	19.1	80.9	14.8
										S 1.50-F 1.60	7.9	28.4	88.8	16.0
										S 1.60	11.2	46.3	100.0	19.4
H-10	S2168-69	A	12.93	14.40	1.47					F1.40	42.5	14.3	42.5	14.3
										S 1.40-F 1.50	20.8	21.2	63.3	16.6
										S 1.50-F 1.60	17.6	28.8	80.9	19.2
										S 1.60	19.1	46.7	100.0	24.5
H-10	S2171-72	B	14.77	16.31	1.54					F1.40	78.1	10.6	78.1	10.6
										S 1.40-F 1.50	13.8	19.4	91.9	11.9
										S 1.50-F 1.60	3.9	30.6	95.8	12.7
										S 1.60	4.2	43.0	100.0	14.0
H-10		C	17.90	19.80	1.90					F1.40	68.5	10.7	68.5	10.7
										S 1.40-F 1.50	15.3	20.5	83.8	12.5
										S 1.50-F 1.60	4.4	28.9	88.2	13.3
										S 1.60	11.8	56.2	100.0	18.4
H-12	S2177	B	42.45	43.66	1.21					F1.40	17.5	9.7	17.5	9.7
										S 1.40-F 1.50	43.1	16.8	60.6	14.7
										S 1.50-F 1.60	15.4	26.5	76.0	17.1
										S 1.60	24.0	46.7	100.0	24.2
H-12	S2179	C	44.24	44.84	0.60					F1.40	6.9	11.2	6.9	11.2
										S 1.40-F 1.50	51.1	16.3	58.0	15.7
										S 1.50-F 1.60	27.4	24.5	85.4	18.5
										S 1.60	14.6	45.0	100.0	22.4
H-14	S2181	A	39.30	40.55	1.25					F1.40	22.5	13.6	22.5	13.6
										S 1.40-F 1.50	25.7	20.8	48.2	17.4
										S 1.50-F 1.60	17.2	28.2	65.4	20.3
										S 1.60-F 1.80	22.8	43.1	88.2	26.2
										S 1.80	11.8	59.8	100.0	30.1
H-14	S2183	B	42.52	43.69	1.17					F1.40	42.1	11.6	42.1	11.6
										S 1.40-F 1.50	41.2	18.8	83.3	15.2
										S 1.50-F 1.60	7.0	30.2	90.3	16.3
										S 1.60	9.7	46.8	100.0	19.3
H-16	S2187-88	A	46.48	47.98	1.50					F1.40	48.9	21.9	48.9	21.9
										S 1.40-F 1.50	20.4	33.1	69.3	25.2
										S 1.50-F 1.60	21.8	40.6	91.1	28.9
										S 1.60	8.9	59.0	100.0	31.6
H-16	S2191-92	B	48.20	49.42	1.22					F1.40	41.2	10.9	41.2	10.9
										S 1.40-F 1.50	38.1	18.5	79.3	14.6
										S 1.50-F 1.60	9.3	28.2	88.6	16.0
										S 1.60	11.4	46.1	100.0	19.4
H-16	S2194	C	51.37	52.30	0.93					F1.40	38.5	10.2	38.5	10.2
										S 1.40-F 1.50	39.9	17.4	78.4	13.9
										S 1.50-F 1.60	9.7	26.8	88.1	15.3
										S 1.60	11.9	43.7	100.0	18.7
H-17	S2195-96	A	15.46	16.96	1.50					F1.40	48.9	22.3	48.9	22.3
										S 1.40-F 1.50	23.3	32.1	72.2	25.5
										S 1.50-F 1.60	12.2	43.1	84.4	28.0
										S 1.60	15.6	65.3	100.0	33.8
H-17	S2198	B	17.06	18.16	1.10					F1.40	12.6	10.4	12.6	10.4
										S 1.40-F 1.50	51.3	19.5	63.9	17.7
										S 1.50-F 1.60	23.2	28.5	87.1	20.6
										S 1.60	12.9	38.2	100.0	22.9
H-17	S2200	C	20.24	21.42	1.18					F1.40	32.6	11.1	32.6	11.1
										S 1.40-F 1.50	36.5	19.5	69.1	15.5
										S 1.50-F 1.60	12.4	30.2	81.5	17.8
										S 1.60	18.5	44.2	100.0	22.7

RAW QUALITY DATA

Capricorn Raw Ply Data

Drill Hole	Sample No.	Sample	Seam	From Depth (m)	To Depth (m)	Sample Thick (m)	Mass (g) received	RD (g/cc) Apparent	Moisture (%ad)	Ash (%ad)	VM (%ad)	FC (%ad)	Calorific Value (MJ/kg)		Sulfur (%ad)
													Uncorrected	Corrected	Total
H-01		Coal	A	13.92	14.48	0.56			3.4	24.5	15.6	56.5			
		Coal	A	14.52	14.69	0.17		1.58	5.1	32.0	15.5	47.4			
		Coal	A	14.80	15.07	0.27		1.45	3.9	21.6	19.0	55.5			
		Coal	A	15.11	15.43	0.32		1.56	4.9	35.9	14.2	45.0			
		Coal	A	15.61	16.23	0.62		1.46	3.4	19.1	16.3	61.2			
		Coal	B	17.57	17.64	0.07		1.38	2.0	19.0	15.2	63.8			
		Coal	B	17.80	18.94	1.14		1.44	3.7	18.2	16.7	61.4			
		Coal	C	19.63	19.85	0.22		1.46	2.9	23.6	16.4	57.1			
		Coal	C	19.87	20.14	0.27		1.45	5.1	26.6	13.9	54.4			
		Coal	C	20.16	20.19	0.03		1.48	4.5	30.0	14.6	50.9			
		Coal	C	34.39	34.71	0.32		1.77	5.4	55.2	7.5	31.9			
H-03		Coal	A	42.31	42.49	0.18		1.54	4.1	26.3	21.2	48.4			
		Coal	A	42.51	42.98	0.47		1.48	6.1	27.2	18.6	48.1			
		Coal	A	43.02	43.32	0.30		1.49	5.6	26.0	18.4	50.0			
		Coal	A	43.33	44.19	0.86		1.46	4.8	22.2	20.3	52.7			
		Coal	B	45.51	46.20	0.69		1.44	4.6	18.8	20.9	55.7			
		Coal	B	46.24	47.05	0.81		1.44	4.4	19.9	20.7	55.0			
		Coal		68.08	68.40	0.32		1.60	5.4	39.9	13.5	41.2			
H-04		Coal	A	21.28	21.46	0.18		1.48	5.9	23.5	19.7	50.9			
		Coal	A	21.49	21.82	0.33		1.47	5.1	25.6	20.6	48.7			
		Coal	A	21.84	22.98	1.14		1.45	5.3	21.6	20.0	53.1			
		Coal	B	23.03	23.53	0.50		1.41	5.2	17.5	20.3	57.0			
		Coal	B	23.96	24.16	0.20		1.43	4.0	28.1	21.5	46.4			
		Coal	B	24.18	24.53	0.35		1.49	4.9	26.1	26.1	42.9			
H-05		Coal	A	49.08	49.21	0.13		1.47	7.2	33.3	28.8	30.7			
		Coal	A	60.14	60.89	0.75		1.58	6.2	39.5	27.6	26.7			
		Coal	B	66.23	66.38	0.15		1.63	6.9	32.2	21.5	39.4			
		Coal	C	70.16	70.24	0.08		1.56	7.8	50.2	22.7	19.3			
		Coal		81.60	81.72	0.12		1.75	4.7	46.8	15.0	33.5			
		Coal		92.44	93.04	0.60		1.55	4.9	27.0	19.6	48.5			
		Coal		122.85	122.96	0.11		1.70	5.2	37.2	16.1	41.5			
H-06		Coal	A	13.30	13.74	0.44		1.54	8.0	24.4	18.7	48.9			
		Coal	A	13.77	14.59	0.82		1.53	8.3	23.8	21.2	46.7			
		Coal	A	14.64	15.04	0.40		1.50	7.1	19.2	18.7	55.0			
		Coal	B	17.23	18.21	0.98		1.47	7.7	15.2	26.2	50.9			
		Coal	C	19.32	20.49	1.17		1.47	6.9	19.3	25.1	48.7			
		Coal		71.84	72.17	0.33		1.89	9.6	51.0	13.4	26.0			
		Coal		88.76	89.00	0.24		1.57	7.0	31.5	21.8	39.7			
		Coal		106.24	106.40	0.16		1.52	5.8	33.7	25.4	35.1			
		Coal		133.39	133.54	0.15		1.80	7.7	55.7	10.5	26.1			
		Coal		146.89	147.38	0.49		1.68	6.5	36.1	3.4	54.0			
H-07		Coal	A	15.73	16.73	1.00		1.57	7.1	30.6	24.9	37.4			
		Coal	A	16.74	16.94	0.20		1.57	6.5	35.2	28.3	30.0			
		Coal	B	22.60	22.80	0.20		1.58	6.9	32.0	26.4	34.7			
		Coal		26.68	26.75	0.07		1.62	6.1	37.6	18.9	37.4			
		Coal		37.06	37.18	0.12		1.47	5.9	20.2	27.5	46.4			
H-09	S2161	Shale					0.540	2.61		93.1					
	S2162	Coal	A				1.804	1.76	3.2	40.4	10.1	46.3			
	S2163	Coal	A				1.592	1.57	3.3	23.7	11.6	61.4			
	S2164	Coal	B				1.648	1.66	3.8	31.7	9.6	54.9			
	S2165	Shale					0.430	2.58		91.8					
	S2166	Coal	C				2.264	1.55	2.7	22.5	13.7	61.1			
H-10	S2167	Shale					0.476	2.09		66.6					
	S2168	Coal	A				1.510	1.66	4.7	31.1	19.7	44.5			
	S2169	Coal	A				1.042	1.54	4.1	19.3	23.1	53.5			
	S2170	Shale					0.930	2.33		80.7					
	S2171	Coal	B				1.424	1.49	3.9	14.8	23.1	58.2			
	S2172	Coal	B				1.410	1.46	4.2	13.3	26.0	56.5			
	S2173	Shale					0.400	2.60		91.6					
	S2174	Coal	C				1.790	1.47	4.0	13.9	28.2	53.9			
	S2175	Shale	C				0.072	2.35		72.2					
	S2176	Coal	C				1.726	1.52	3.7	20.6	27.4	48.3			
H-12	S2177	Coal	B				2.626	1.56	2.8	24.3	16.4	56.5			
	S2178	Shale					0.584	2.62		92.7					
	S2179	Coal	C				1.204	1.55	3.2	24.0	15.8	57.0			
H-14	S2181	Coal	A				2.908	1.61	3.7	29.8	16.0	50.5			
	S2182	Shale					0.514	2.54		92.5					
	S2183	Coal	B				2.398	1.53	2.8	19.9	17.2	60.1			
	S2184	Shale					0.446	2.64		92.1					
	S2185	Coal	C				2.664	1.58	3.4	25.3	18.1	53.2			
H-16	S2186	Shale					0.136	1.98		60.5					
	S2187	Coal	A				1.418	1.69	4.8	37.6	15.1	42.5			
	S2188	Coal	A				1.664	1.60	3.8	28.1	16.7	51.4			
	S2190	Shale					0.828	2.52		88.3					
	S2191	Coal	B				1.310	1.57	2.9	22.1	18.2	56.8			
	S2192	Coal	B				1.348	1.53	2.0	20.8	20.2	57.0			
	S2193	Shale					0.380	2.68		92.8					
	S2194	Coal	C				1.588	1.50	1.7	18.4	21.1	58.8			
H-17	S2195	Coal	A				1.692	1.77	3.1	40.5	9.9	46.5			
	S2196	Coal	A				1.758	1.73	2.6	35.4	11.5	50.5			
	S2197	Shale					0.318	2.51		87.1					
	S2198	Coal	B				2.232	1.60	2.2	22.8	12.8	62.2			
	S2199	Shale					0.528	2.58		91.0					
	S2200	Coal	C				2.226	1.55	2.0	23.3	18.2	56.5			
H-20		Coal					2.047	1.62	5.7	36.3	17.4	40.6			
		Coal					1.615	1.53	4.2	26.6	20.0	49.2			
		Coal					1.283	1.52	3.8	24.3	19.9	52.0			
		Coal					1.184	1.44	3.0	17.6	24.3	55.1			
		Coal					1.222	1.49	3.6	22.5	22.5	51.4			
		Coal					1.316	1.50	3.6	24.1	21.6	50.7			
H-22		Coal					1.679	1.56	5.7	28.8	21.0	44.5			
		Coal					1.678	1.53	5.4	24.0	22.8	47.8			
		Coal					1.541	1.53	5.0	20.0	23.9	51.1			
		Coal					1.449	1.44	4.9	14.4	28.5	52.2			
		Coal					0.907	1.41	4.7	13.4	29.3	52.6			
		Coal					1.302	1.59	4.6	21.9	26.8	46.7			
H-23		Coal					2.423	1.60	4.2	32.5	16.8	46.5			
		Shale					0.222	2.39		82.2					
		Coal					1.334	1.55	3.3	26.0	17.4	53.3			
		Coal					1.565	1.47	2.7	17.4	19.8	60.1			
		Coal					1.292	1.52	2.6	14.2	19.3	63.9			
		Coal					1.438	1.55	3.0	22.4	19.1	55.5			
H-24		Coal					2.331	1.93	3.0	41.1	9.8	46.1			
		Shale					0.343	2.56		89.5					
		Coal					2.010	1.68	2.6	25.4	9.3	62.7			
H-28		Coal					2.241	1.58	3.8	28.6	18.8	48.8			
		Shale					0.160	2.21		76.5					
		Coal					2.691	1.51	2.6	19.6	20.4	57.4			
		Coal					3.192	1.50	2.7	19.2	20.6	57.5			

Capricorn Raw Composite Data															
Drill Hole	Sample No.	Sample	Seam	From Depth (m)	To Depth (m)	Sample Thick (m)	Mass (g) received	RD (g/cc) Apparent	Moisture (%ad)	Ash (%ad)	VM (%ad)	FC (%ad)	Calorific Value (MJ/kg) Uncorrected	Calorific Value (MJ/kg) Corrected	Sulfur (%ad) Total
H-01		Coal	A	13.92	16.23	2.31			4.4	35.1	14.3	46.2			
		Coal	C	19.63	20.29	0.66			8.9	76.4	7.9	6.8			
H-03		Coal	A	42.31	44.19	1.88			9.1	62.1	10.0	18.8			
		Coal	B	45.81	47.05	1.24			4.4	23.1	20.0	52.5			
H-04		Coal	A + B	21.28	24.53	3.25			5.3	26.7	20.1	47.9			
H-06		Coal	A	13.3	14.04	0.74			7.8	26.7	19.2	46.3			
		Coal	C	17.23	18.3	1.07			7.3	24.8	23.5	44.4			
				88.76	89.09	0.33			6.1	51.4	16.8	25.7			
H-07		Coal	A	15.73	16.94	1.21			7	31.9	25.3	35.8			
H-09	S2162-64	Coal	A + B	15.35	17.43	2.08			5.1	33.5	10.3	51.1	4905	4905	0.23
	S2166	Coal	C	19.13	20.18	1.05			4	19.8	13.7	62.5	6276	6276	0.29
H-10	S2168-69	Coal	A	12.93	14.4	1.47			6.9	24.6	20.9	47.6	5187	5187	0.26
	S2171-72	Coal	B	14.77	16.31	1.54			7.4	13.6	24.6	54.4	6175	6175	0.26
	S2174-76	Coal	C	17.9	19.8	1.9			6.8	18.6	26.0	48.6	5688	5688	0.32
H-12	S2177	Coal	B	42.45	43.66	1.21			4.3	24.5	16.4	54.8	5597	5597	0.43
	S2179	Coal	C	44.24	44.84	0.6			4.5	22.5	15.8	57.2	5855	5855	0.4
H-14	S2181	Coal	A	39.3	40.55	1.25			5.4	30.5	16.0	48.1	5024	5024	0.22
	S2183	Coal	B	42.52	43.69	1.17			4.4	19.5	17.2	58.9	6118	6118	0.33
	S2185	Coal	C	45.2	46.44	1.24			4.5	23.6	18.1	53.8	5669	5669	0.34
H-16	S2187-88	Coal	A	46.48	47.98	1.5			5.8	32.6	16.0	45.6	4719	4719	0.22
	S2191-92	Coal	B	48.2	49.42	1.22			3.9	19.6	19.2	57.3	6118	6118	0.41
	S2194	Coal	C	51.37	52.3	0.93			3.6	19	21.1	56.3	6132	6132	0.37
H-17	S2195-96	Coal	A	15.46	16.96	1.5			4.7	34.2	10.7	50.4	4766	4766	0.19
	S2198	Coal	B	17.06	18.16	1.1			3.9	22.9	12.8	60.4	5760	5760	0.24
	S2200	Coal	C	20.24	21.42	1.18			3.6	22.6	18.2	55.6	5894	5894	0.32
H-20			A			1.45			5.4	30.3	17.8	46.5	4895	4895	0.25
			B			1.2			3.7	20.4	21.6	54.3	5808	5808	0.36
			C			1.3			3.9	22.2	21.5	52.4	5750	5750	0.36
H-22			A			1.6			5.8	26.3	20.7	47.2	5000	5000	0.21
			B			1.47			5.2	17.4	24.8	52.6	5836	5836	0.33
			C			1.35			5	18.3	26.3	50.4	5789	5788	0.33
H-23			A + B			2.35			3.7	28.9	17.4	50.0	5258	5258	0.28
			C			1.21			3	18.4	18.7	59.9	6300	6300	0.38
H-24			A + B			1.99			3.8	38.8	8.7	48.7	4289	4289	0.33
H-28			A + B			2.08			3.5	24.9	18.9	52.7	5631	5631	0.36
			C			1.36			2.8	19.2	20.6	57.4	6309	6309	0.36

*Composite VM data for hole H09 - H17 estimated using sample intervals calcualted from sample weight and core dia

Marston Raw Ply Data														
Geophysically Corrected Depths						Raw Ply Analysis Results								
Hole No.	Seam	Depth From (m)	Depth To (m)	Thick (m)	Litho	Sample Mass kg (ar)	Sample Mass kg (ad)	RD	M % (ad)	A % (ad)	VM % (ad)	FC % (ad)	S % (ad)	CV MJ/kg (ad)
H29B		0.00	8.85	8.85	We									
		8.85	14.11	5.26	W									
		14.11	14.21	0.10	R	0.607	0.590	2.40	3.5	86.6	5.8	4.1	0.17	2.29
	A	14.21	14.56	0.35	CO	1.452	1.398	1.49	3.2	19.3	15.2	62.3	0.56	27.18
		14.56	14.66	0.10	F	0.598	0.570	2.44	4.1	87.7	5.6	2.6	0.10	0.06
		14.66	17.13	2.47	W									
	B	17.13	17.48	0.35	CO									
		17.48	27.28	9.80	W									
	C	27.28	27.53	0.25	CO									
		27.53	28.50	0.97	W									
	D	28.50	28.81	0.31	CO									
		28.81	41.00	12.19	W									
H30		0.00	3.31	3.31	We									
	C	3.31	3.61	0.30	CO (We)	0.738	0.670	1.72	17.8	30.1	20.8	31.3	0.26	12.69
		3.61	3.71	0.10	F	0.546	0.525	2.44	6.2	89.4	5.6	-1.2	0.04	0.13
		3.71	5.01	1.30	W									
	D	5.01	5.25	0.24	CO									
		5.25	38.00	32.75	W									
		0.00	14.41	14.41	We									
	A	14.41	14.66	0.25	CO									
		14.66	16.67	2.01	W									
	B	16.67	17.26	0.59	CO									
		17.26	40.96	23.70	W									
	C	40.96	41.15	0.19	CO									
		41.15	49.55	8.40	W									
		49.55	49.66	0.11	R	0.767	0.750	2.42	5.6	87.1	5.3	2.0	0.05	0.14
	D	49.66	49.90	0.24	CO	1.959	1.907	1.91	8.6	57.5	8.7	25.2	0.15	9.10
		49.90	50.00	0.10	F	0.456	0.443	2.45	5.3	89.2	4.9	0.6	0.05	0.14
H31		50.00	56.00	6.00	W									
H32		0.00	4.00	4.00	We									
		4.00	27.06	23.06	W									
	A	27.06	28.51	1.45	CO	4.046	4.015	1.58	4.6	28.0	16.2	51.2	0.26	22.14
		28.51	28.61	0.10	F	0.690	0.684	2.38	4.2	86.7	5.6	3.5	0.07	0.14
		28.61	28.74	0.13	W									
	B	28.74	29.86	1.12	CO	4.581	4.552	1.52	3.8	21.3	17.6	57.3	0.32	24.95
		29.86	29.96	0.10	F	0.352	0.348	2.38	4.4	84.5	5.8	5.3	0.11	1.56
		29.96	31.73	1.77	W									
	C	31.73	32.86	1.13	CO	1.005	0.999	1.47	3.1	17.2	19.2	60.5	0.42	27.21
	C				CO	2.651	2.631	1.58	4.3	29.0	18.0	48.7	0.30	21.94
		32.86	33.42	0.56	W									
	D	33.42	33.61	0.19	CO									
		33.61	42.00	8.39	W									
H33		0.00	8.00	8.00	We									
		8.00	17.06	9.06	W									
	C	17.06	17.91	0.85	CO	3.629	3.595	1.52	6.5	23.0	25.5	45.0	0.56	22.57
		17.91	18.01	0.10	F	0.639	0.631	2.33	4.9	85.1	6.4	3.6	0.18	0.07
		18.01	32.85	14.84	W									
	D	32.85	33.05	0.20	CO									
		33.05	49.00	15.95	W									
H34		0.00	5.00	5.00	We									
		5.00	36.07	31.07	W									
		36.07	36.17	0.10	R	0.674	0.673	2.66	1.2	94.5	4.5	-0.2	0.05	0.10
	A	36.17	37.34	1.17	CO	5.270	5.240	1.64	4.1	32.1	15.3	48.5	0.23	20.79
	B	37.34	38.27	0.93	CO	3.066	3.050	1.50	2.8	17.0	19.1	61.1	0.35	26.83
		38.27	39.28	1.01	W									
	C	39.28	40.74	1.46	CO	4.865	4.837	1.55	3.5	24.3	18.9	53.3	0.38	24.09
		40.74	46.00	5.26	W									
H35		0.00	10.00	10.00	We									
		10.00	61.89	51.89	W									
	A	61.89	62.92	1.03	CO	4.989	4.945	1.57	3.9	27.1	18.0	51.0	0.29	23.05
		62.92	63.97	1.05	W									
		63.97	64.07	0.10	R	0.589	0.580	2.39	3.2	86.8	5.7	4.3	0.05	1.63
	B	64.07	65.09	1.02	CO	4.424	4.386	1.54	3.2	21.6	19.1	56.1	0.28	24.85
		65.09	68.41	3.32	W									
	C	68.41	68.84	0.43	CO	1.476	1.467	1.43	2.4	13.9	21.1	62.6	0.60	28.65
	C	68.84	69.66	0.82	CO	3.535	3.510	1.53	3.7	24.1	18.1	54.1	0.38	24.31
		69.66	70.39	0.73	W									
	D	70.39	70.95	0.56	CO	2.605	2.577	1.56	3.6	27.3	17.3	51.8	0.41	23.21
		70.95	74.00	3.05	W									

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solutions@golder.com
www.golder.com

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