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1.1 OVERVIEW

The Galilee Coal Project (Northern Export Facility) (also known as the China First Project), (hereafter referred to as the project) comprises a new coal mine located in the Galilee Basin, Queensland, approximately 30 km to the north of Alpha; a new rail line connecting the mine to coal terminal facilities; and use of coal terminal facilities in the Abbot Point State Development Area (APSDA) and port loading facilities at the Port of Abbot Point.

Figure 1 shows the overall project concept.

Waratah Coal proposes to mine 1.4 billion tonnes of raw coal from its existing tenements, Exploration Permit for Coal (EPC) 1040 and EPC 1079. The mine development involves the construction of four nine Million Tonnes Per Annum (Mtpa) underground long-wall coal mines, two 10 Mtpa open cut pits, two coal preparation plants with raw washing capacity of 28 Mtpa (see Figure 2) .

The annual Run-of-Mine (ROM) coal production will be 56 Mtpa to produce 40 Mtpa of saleable export highly volatile, low sulphur, steaming coal to international markets. At this scale of operation, the capital expense of constructing the required rail and port infrastructure is economically viable over the life of the project.

For the Environmental Impact Statement (EIS), the mine development is defined as the underground and open cut mines, Mine Industrial Area (MIA) and two coal handling and preparation plants (CHPP) and the supporting coal-handling infrastructure through to the train loading facility. The rail component commences at the balloon loop at the mine and ends at the balloon loop adjoining the T4 – T7 coal handling facility at the Abbot Point State Development Area, and includes the 447 km single gauge rail line. Marshalling and maintenance facilities for the rail and rolling stock are included as part of the rail component. The T4 – T7 coal terminal and coal handling facilities are located adjacent to the train unloading facility and includes infrastructure to convey the coal through to the ship loaders. Each of the three components includes numerous auxiliary and administrative infrastructure and these are included in the discussion for each component.

The assessment of the mining construction and operation is detailed throughout Volume 2 of this EIS. This chapter provides a description of the key components comprising the mine development and discusses the construction, operational and decommissioning phases associated with the mine.

1.1.1 LOCATION

The mine development is located approximately 30 km to the northwest of the township of Alpha in central Queensland, and falls within the Barcaldine Regional Shire Council administrative area. Figure 1 shows the location of the mine in the regional context and Figure 2 shows the mine infrastructure arrangement.

1.1.2 TENURE DESCRIPTION

The tenures incorporated into the project are Exploration Permit-Coal (EPC) 1040 and EPC 1079 both which are held by Waratah Coal. Waratah Coal has held a 100% interest in these tenements since 22 June 2006 and 2 November 2007 respectively. These tenures been granted for a five-year conditional term.

EPC 1040 covers 241 sub-blocks (which equates to approximately 725 km^2) adjoining the southern boundary of Mineral Development License (MDL) 285 (held by Hancock Prospecting P/L). EPC 1079 adjoins the western boundary of EPC 1040 as well as MDL 285 and MDL 333 (both held by Hancock Prospecting P/L). Additionally, Waratah Coal has been granted permits EPC 1039 and EPC 1053, which adjoin the northern boundary of MDL 333. The southeastern corner of EPC 1040 is located approximately 7 km to the west of the township of Alpha in central Queensland.

EPC 1079 covers 223 sub-blocks (which equates to approximately 704 km^2) and adjoins the boundaries of other Waratah EPC's 1039, 1040, 1080, 1105, 1155, 1156, 1157, in addition to MDL 285 and MDL 333 (both held by Hancock Prospecting P/L).

Waratah Coal is in the process of preparing a Mining Lease Application (MLA) for the Project. The area within the MLA consists of the northern part of EPC 1040 and part of the southern section of EPC 1079. The MLA area is shown at Figure 3. The MDL and MLA application areas are shown in Figure 3.

1.1.3 STUDY AREA

The study area for the mine is depicted in Figure 1 and comprises all of EPC 1040 and part of EPC 1079.

1.1.4 EXPLORATION HISTORY

Prior to the recent drilling programs conducted by Waratah Coal, there had been no exploration activity of significance in EPC 1040 or EPC 1079. The Geological Survey of Queensland (GSQ) conducted the only previous drilling in 1974. This comprised two boreholes, drilled alongside the railway line between Jericho and Alpha. These holes were designated Jericho 1 and 2, with the eastern most being Jericho 2. The cored boreholes were part of a petroleum stratigraphic drilling campaign of the eastern part of the Galilee Basin. The aim was to establish a fully cored and wireline logged section of the Upper Paleozoic strata, in order to correlate with fully cored sections of similar age on the Springsure Shelf and in the Denison Trough.

Since the granting of EPC 1040 in 2006, Waratah Coal has carried out an extensive exploration program within the project area. As of December 2009, Waratah Coal developed 295 chip holes with approximately 41,000 m drilled and 122 core holes with approximately 21,000 m drilled. Prior to any mining activities occurring further exploration drilling will occur to better define the coal resource in accordance with Joint Ore Reserves Committee (JORC) requirements for definition of coal reserves.

1.1.5 RESOURCE DESCRIPTION

The Galilee Basin covers an area estimated at 247,000 km^2 in central Queensland. This basin is entirely intracratonic and is naturally filled with Late Carboniferous to Middle Triassic sediments. These rocks are dominantly fluvial in origin with minor glacial material developed at the base of the succession. The aerial extent of the Galilee Basin is shown in Figure 4.

The Galilee Basin contains extensive coal deposits, however these are largely very deep, except for the eastern margin where the project lies. The Jurassic – Cretaceous Eromanga Basin, almost entirely unconformably overlies the Galilee Basin. The eastern margin of the Galilee Basin is the only exposed component of the Permo – Triassic sequence.

The principal tectonic elements of the Galilee Basin include:

- the east-west trending Barcaldine Ridge, which subdivides the basin into the northern and southern components. The Maneroo Platform and the Beryl Ridge, which results in the development of the western depression termed the Lovelle Depression and the eastern depression termed the Koburra Trough, subdivide the northern component of the basin. The Pleasant Creek Arch. divides the southern part of the basin into the western Powell Depression and the Springsure Shelf.
- The project area lies on the northern side of the Barcaldine Ridge. These features are shown in Figure 5.
- The project area is primarily overlaid by Quaternary alluvial; however, there is no outcrop of coal seams in the region.

Figure 1. Project Regional Concept

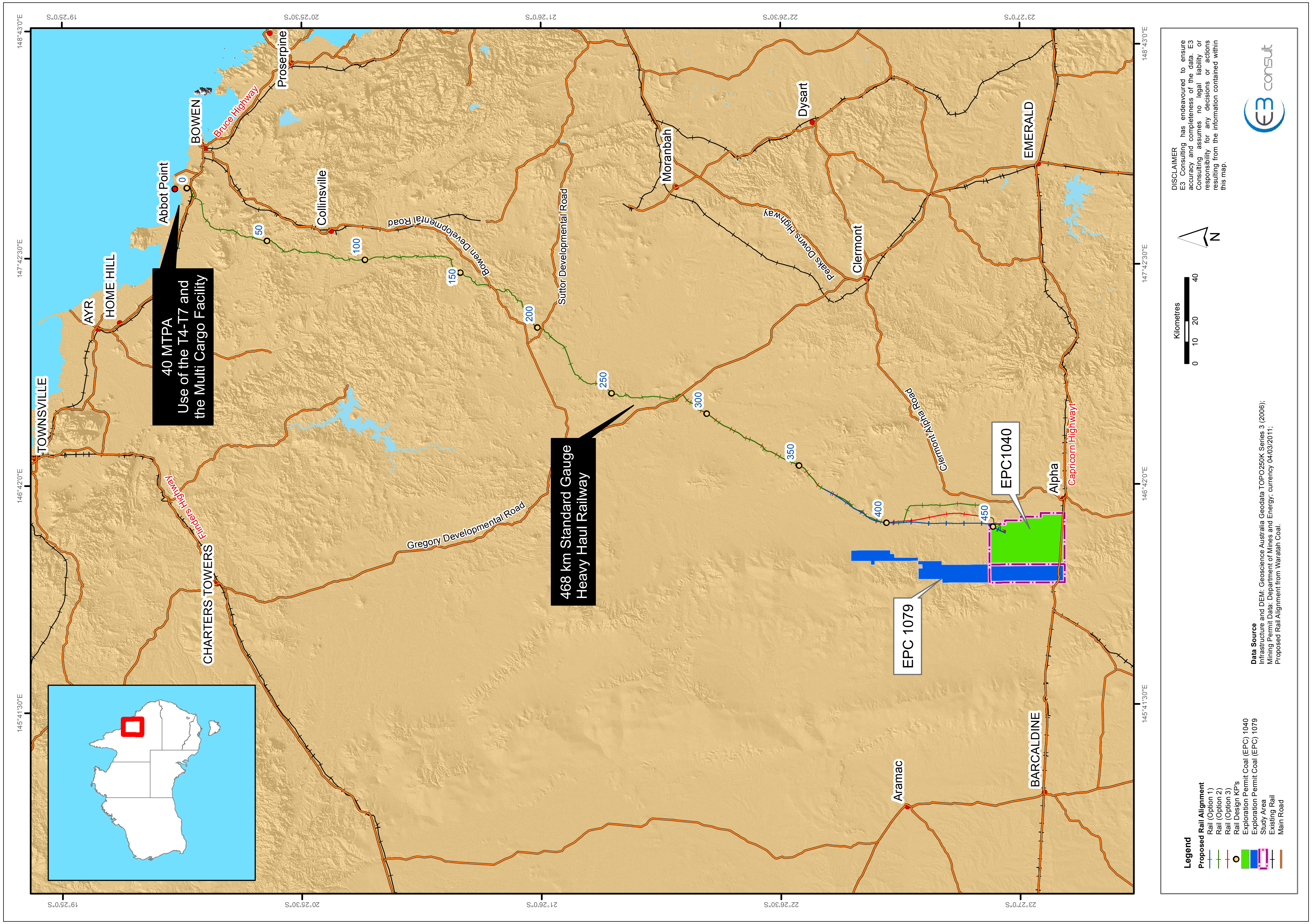


Figure 2. Mine Infrastructure Arrangement

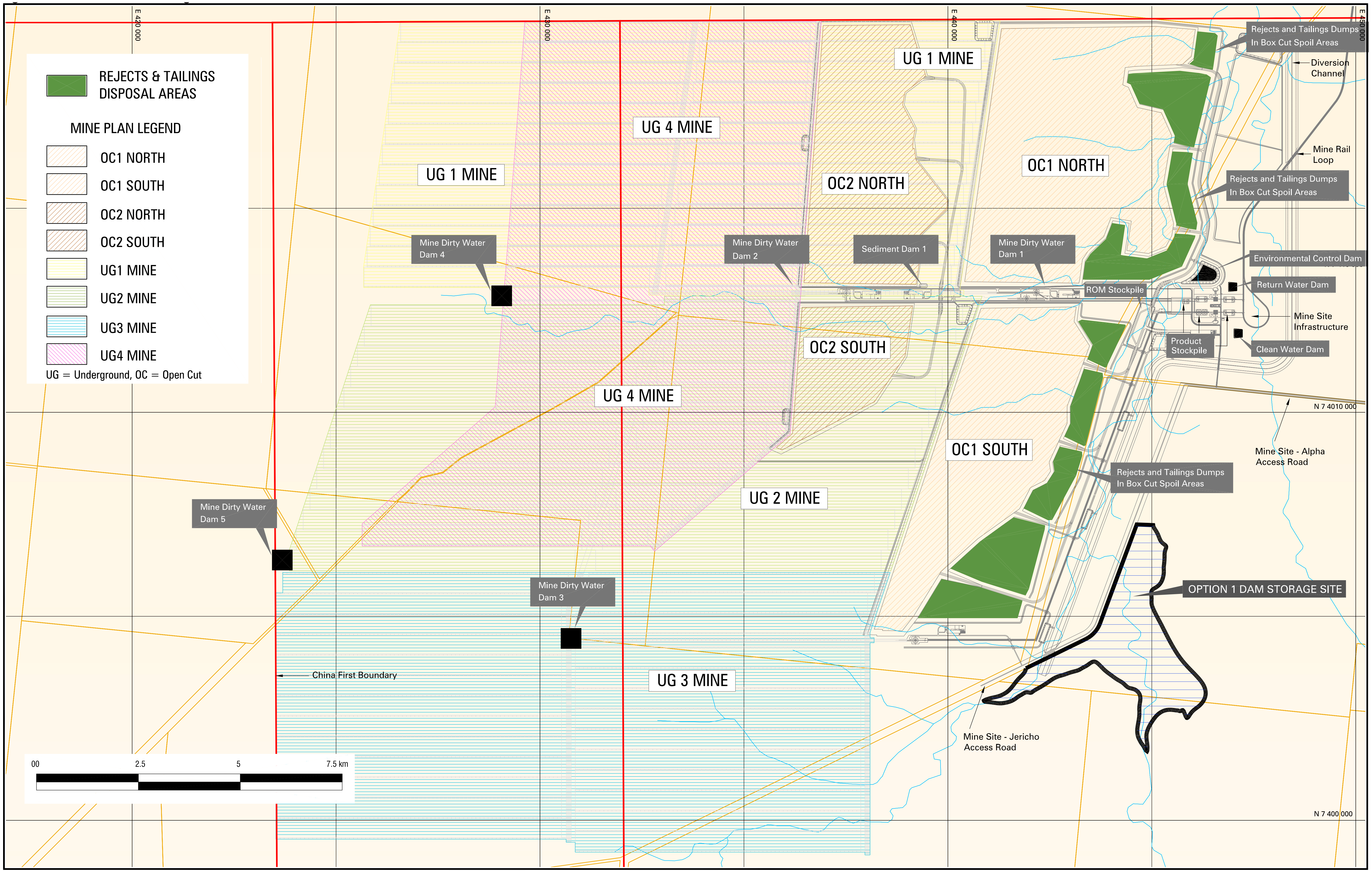


Figure 3. Mining Lease Application Area

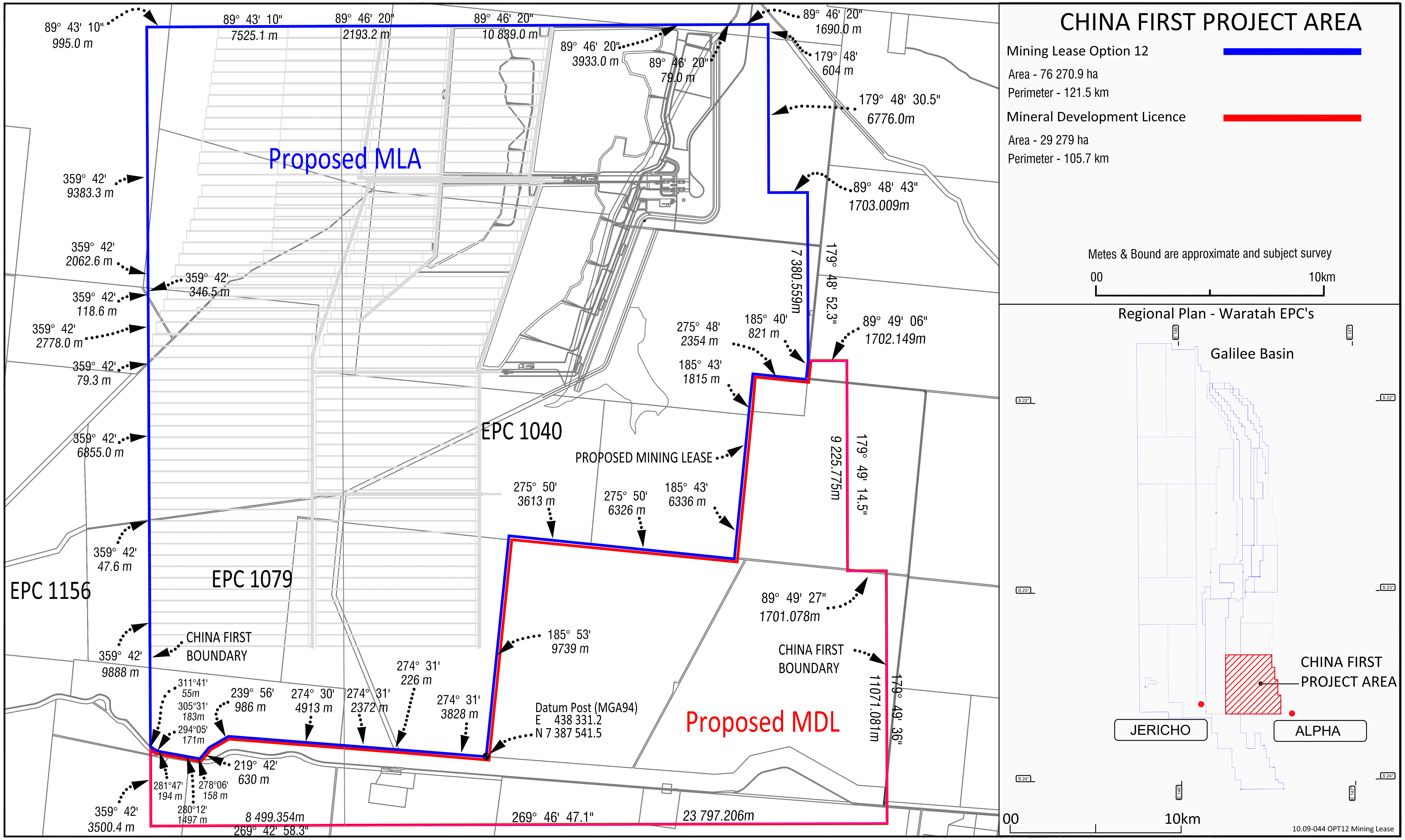
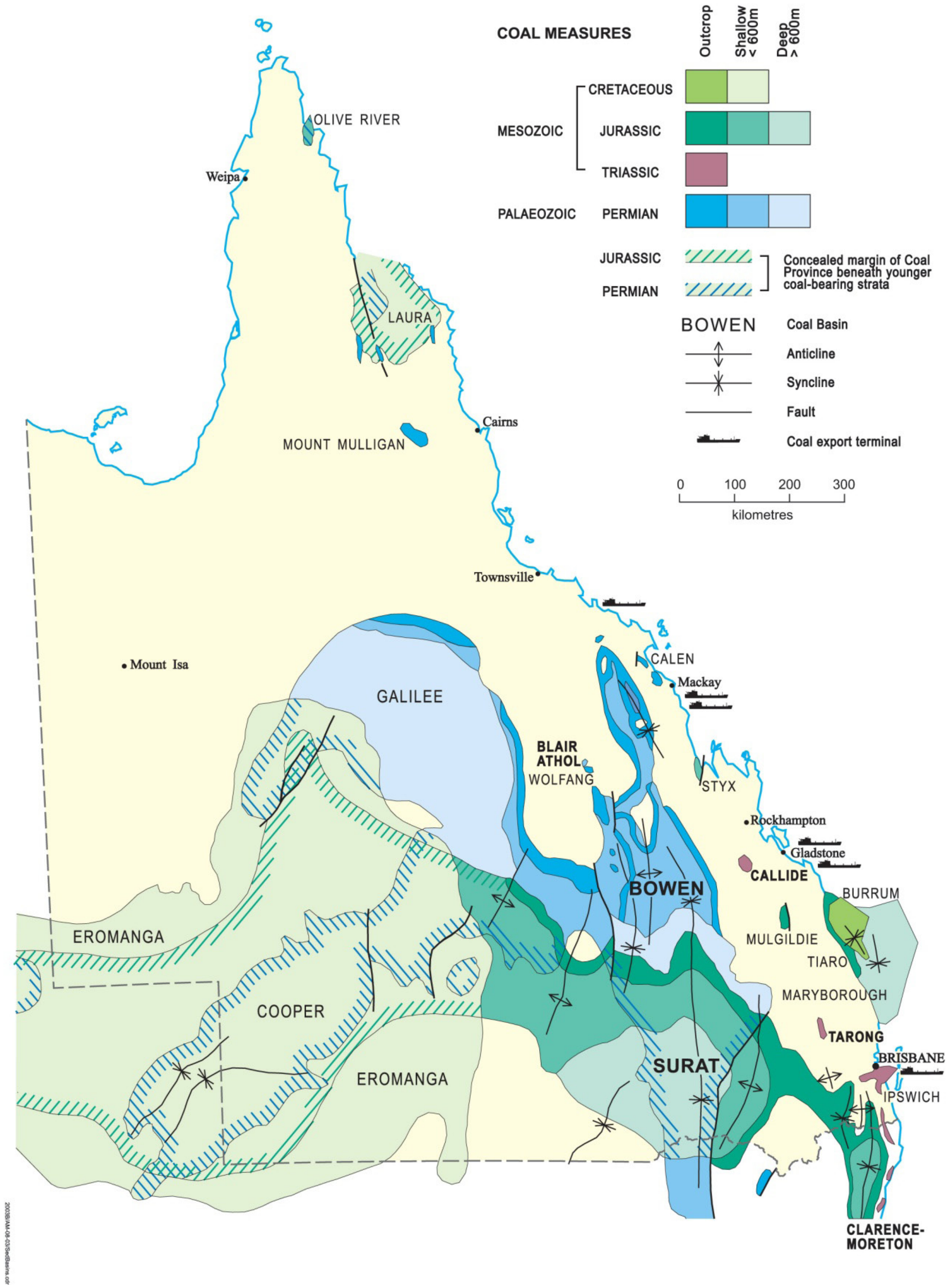
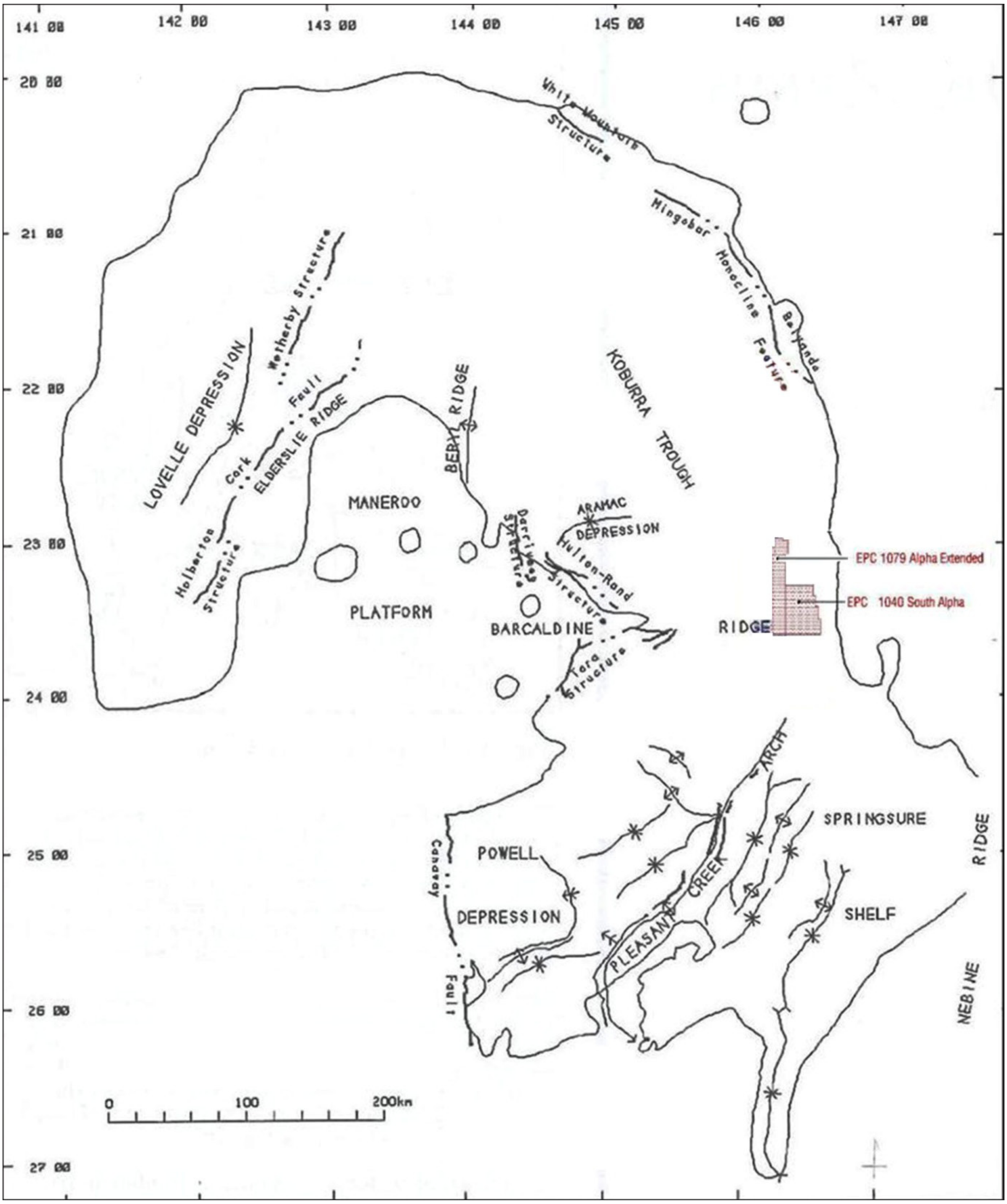


Figure 4. Queensland Coal Measures



Source: Queensland Coals, Physical and Chemical Properties Colliery and Company Information 14th Edition 2003 Ed A J Mutton.

Figure 5. Structural Elements of the Galilee Basin



Source: Scott *et al.*, Galilee Basin in Geology of Australian Coal Basins Geol. Soc. Special Publication No 1, 1995

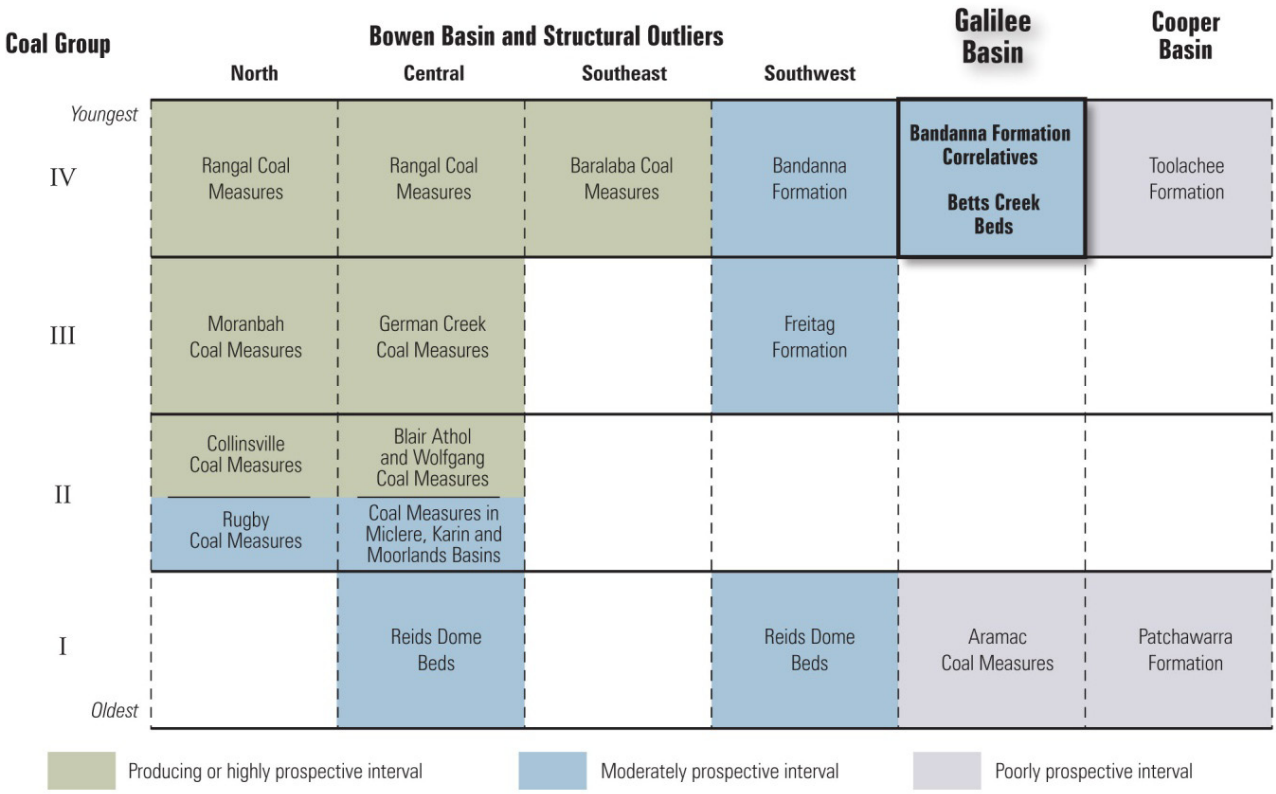
1.1.6 RELATIONSHIP TO OTHER MAJOR COAL BASINS IN QUEENSLAND

The stratigraphic succession of the Galilee Basin is partly related to the sedimentary successions of the Cooper and Bowen Basin's. Major coal deposition occurred in the Galilee during the Early Permian in the Aramac Coal Measures and in the late Permian in the Colinlea Sandstone and Bandanna Formation (and their correlatives the Betts Creek Beds) in the north of the Galilee Basin.

The stratigraphic table for the Galilee, Cooper and Bowen Basins showing the relationship between the major coal units and foundations is shown in Figure 6.

Coal development that has been defined to date is concentrated in the northern part of the basin, as south of the Barcaldine Ridge the identified seams identified to date are thin and sporadic. The coals in the project area occur in the Betts Creek Beds on the northern slope of the Barcaldine Ridge.

Figure 6. Interpreted coal group stratigraphic basin correlations



Source: Queensland Coals, Physical and Chemical Properties Colliery and Company Information 14th Edition 2003 Ed A J Mutton.

1.1.7 STRATIGRAPHY OF THE GALILEE BASIN

The generalised local Galilee Basin Stratigraphy is shown in Figure 7.

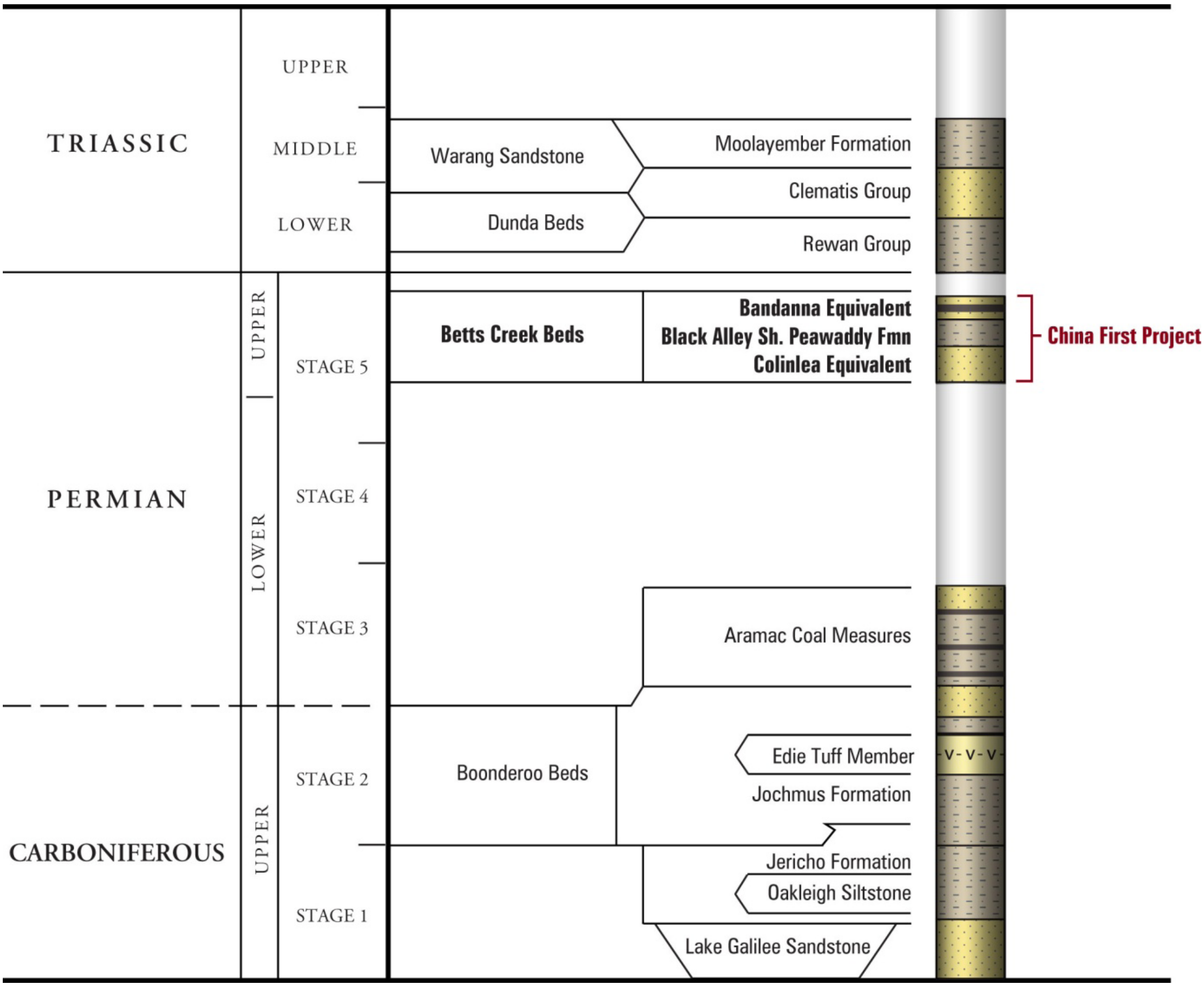
Within the project area, Quaternary alluvials and Tertiary sands, clays and laterites unconformably overlay the distinctive grey-greenish Triassic mudstones and claystones of the Rewan Formation. The Rewan Formation, in turn, unconformably overlays the Late Permian shales, siltstones, sandstones and coal seams of the Bandanna Formation.

The Quaternary sediments comprise of unconsolidated alluvial sands ranging in thickness from 0 metres below ground surface (mbgs) to 30 mbgs. The Tertiary sediments are unconsolidated to semi-consolidated ranging in thickness from 30 mgbs to 125 mgbs. Within the project area, the Quaternary and Tertiary combine to form a thick cover of overburden ranging from 95 mgbs to 125 mgbs over the Bandanna Formation. The Rewan Formation, consisting of Triassic competent claystones and siltstones, is situated unconformably between the overlying Tertiary and the underlying Late Permian Bandanna Formation. The Bandanna Formation and the Colinlea Sandstone comprises of lithic sandstone, siltstone, claystone, carbonaceous mudstone and coal seams.

1.1.8 MINERALISATION

The principal coal seams in the project area contain sub-bituminous high volatile perhydrous coals suitable for use as thermal coal and potentially for liquefaction, gasification and other petrochemical applications. The principal seams have defined continuity and significant resources. The seams dip gently (one to two degrees) to the west, and appear to be structurally continuous with little, if any, faulting. A schematic section is outlined in Figure 8.

Figure 7. Galilee Basin Stratigraphy



Source: Scott *et al*, Galilee Basin in Geology of Australian Coal Basins Geol. Soc. Special Publication No 1, 1995.

1.1.8.1 Mesozoic-Cainozoic Cover

Unconsolidated Cainozoic sediments dominate surface geology of the project area. Unconsolidated sands, silts and clay, lateritised in part, form an extensive blanket over the project area, with thickness of up to 90 m in eastern and central sections. The Permian does not outcrop in the project tenements. There is an assortment of Recent-Quaternary and Tertiary within the Cainozoic blanket but no attempt at demarcation has been established. In the east of tenements, the Cainozoic sits directly on the Permian. This contact is unconformable and represents an extensive time gap; the contact is erosional at least in part.

The Tertiary flood basalts that feature in the cover sequence in parts of the Bowen Basin are absent from the project area.

The Cainozoic tends to thin in the west and Waratah's drilling and previous exploration show the Triassic Rewan Formation rarely at outcrop or shallow near surface in this region. The Rewan Formation is unconformable on the Permian and consists of the greenish sandstones and siltstones well known in association with on the Rangal Coal Measures in the Bowen Basin to the east. Where not removed by the Cainozoic, the contact between the Rewan and Permian sits 20-40 m above the A seam.

1.1.8.2 Permian

The Permian consists of liable sandstones, siltstones, mudstones and claystones with intercollated coal seams. The Permian dips gently to the west at <1°dip and appears to be free of significant structure. The coal seams are currently allocated from the selection process of alphabetical sequence used by previous explorers on the area. The A and B seams are allocated membership of the Bandanna Formation and the sequence for C down the Colinlea Sandstone. It is acknowledged that the E and F seams may belong to a lower formation again. These allocations are tentative and if a definitive relationship can be proven, it will be readily adopted. The provision of Formation / Group membership has no material impact on the resource geology of the deposit.

The combination of a very gentle westerly dip and subdued topography creates relatively broad subcrop zones for each seam. Additionally, the B and C intervals are separated by a 90 m sandstone (vertical thickness); this separation and the dip / surface geometry cause two north-south orientated bands of seam subcrop; the A and B in the west and the C to DL in the east. The E and F Seams sit below the D splits and subcrop further east again, the seam limits often influenced by deeply incised alluvium channels associated with drainage along Sandy Creek. The full C-F sequence continues unbroken under the A and B subcrop zone and all seams continue down dip.

Weathering / oxidation is variable but tends to be deep for a coal Project. The weathering surface is commonly 30-50 m down into the Rewan / Permian rocks. It is noted that this limit to coal occurrence is in addition to the Cainozoic cover discussed above.

1.1.9 COAL SEAMS

Tertiary sediments vary in thickness across the coal deposit ranging from less than 20 meters below ground level (mbgs) in the North of the proposed MLA, but then increasing in thickness to the south to greater than 100 mbgs limiting the open cut potential in this area. The tertiary thickness is displayed in Figure 9. Results from the geological model for the average coal seam thicknesses for each of the seams included in the Resource Estimate are shown Table 1.

Within the B seam, three stone bands (B3, B5 and B7) are planned to be selectively removed as waste during open cut coal mining. Within the DL seam, two stone bands (DLX and DLY) are planned to be selectively removed.

The total coal thickness in each of the open cut mining pits is displayed in Figure 10. Coal thickness ranges from three m to seven m in each mining pit.

Total waste thickness ranges from 20-120 m and is shown in Figure 11. The in-situ strip ratio in each of the open cut mining pits is shown in Figure 12.

Table 1. Average seam thickness results from model

COAL SEAM	AVERAGE THICKNESS (M)	COAL SEAM	AVERAGE THICKNESS (M)
B2	1.26	DU	2.03
B3	0.32	DL1	0.62
B4	0.72	DLX	0.62
B5	0.46	DL2	1.21
B6	0.44	DLY	0.14
B7	0.36	DL3	0.71
B8	2.59	DL Total	3.30
B Total	6.15	Total of all Seams	12.85
C5	1.37		

Figure 8. Stratigraphic Cross-Section of the Project Area

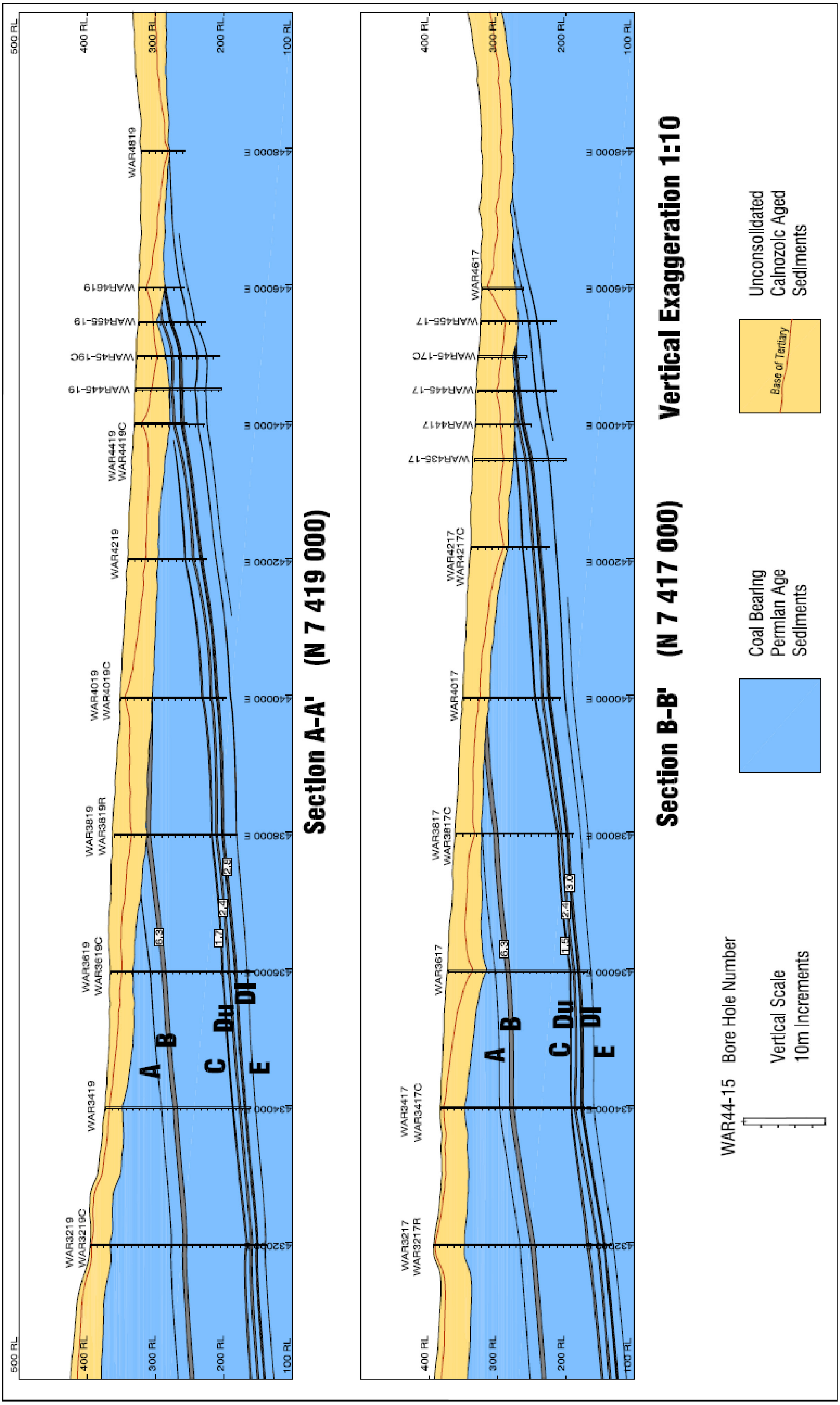


Figure 9. Tertiary Horizon Thickness

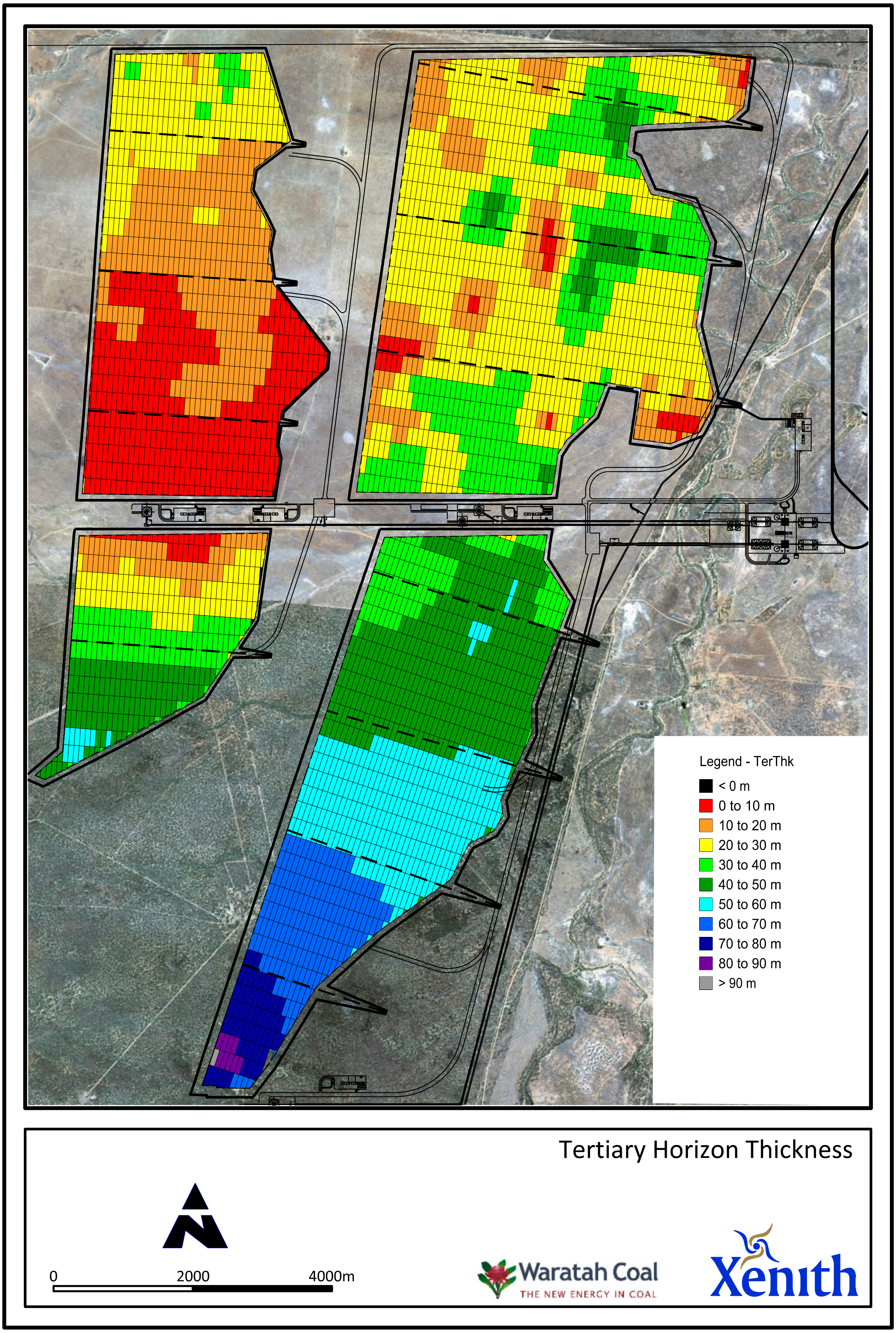


Figure 10. Total Coal Thickness

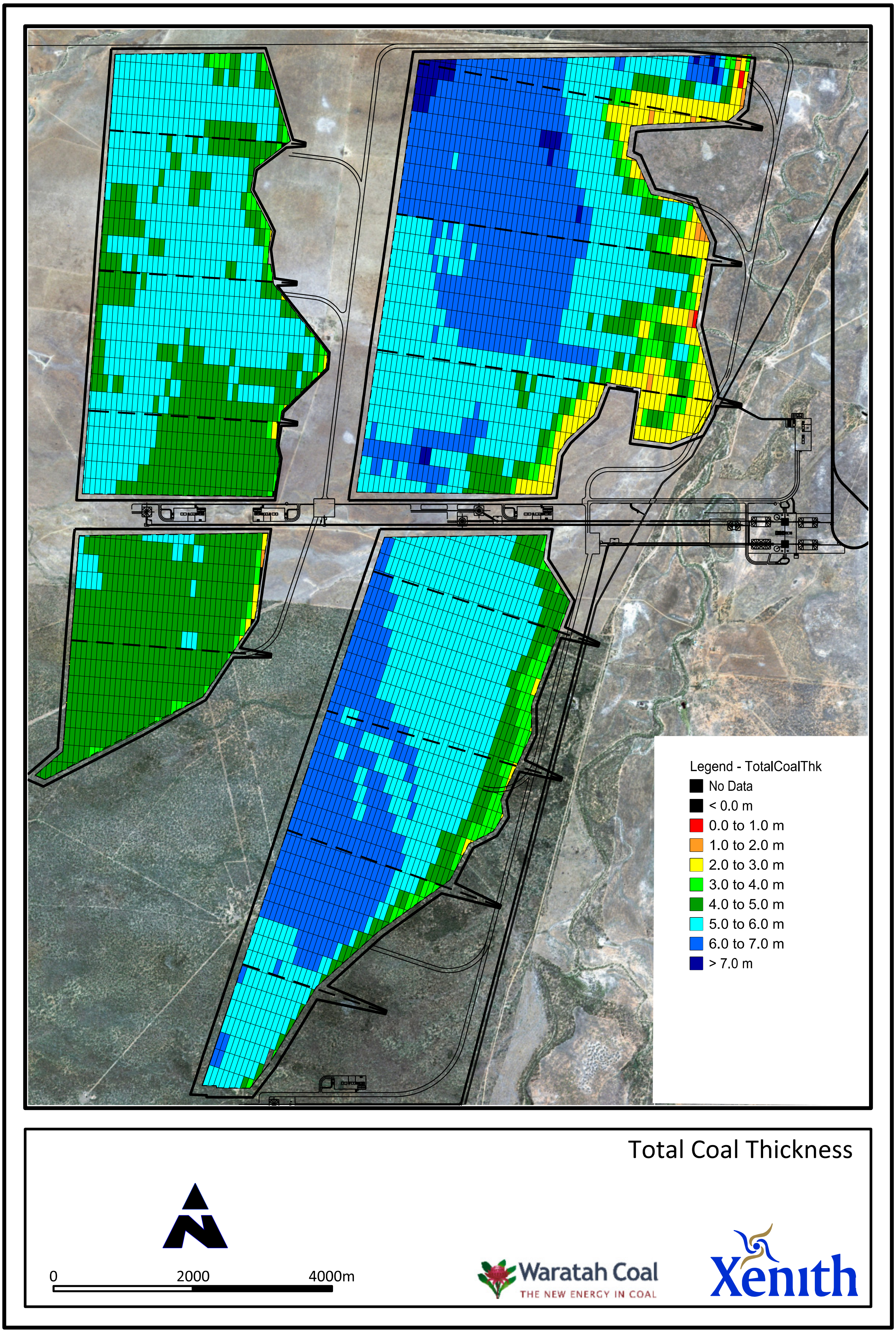


Figure 11. Total Waste Thickness

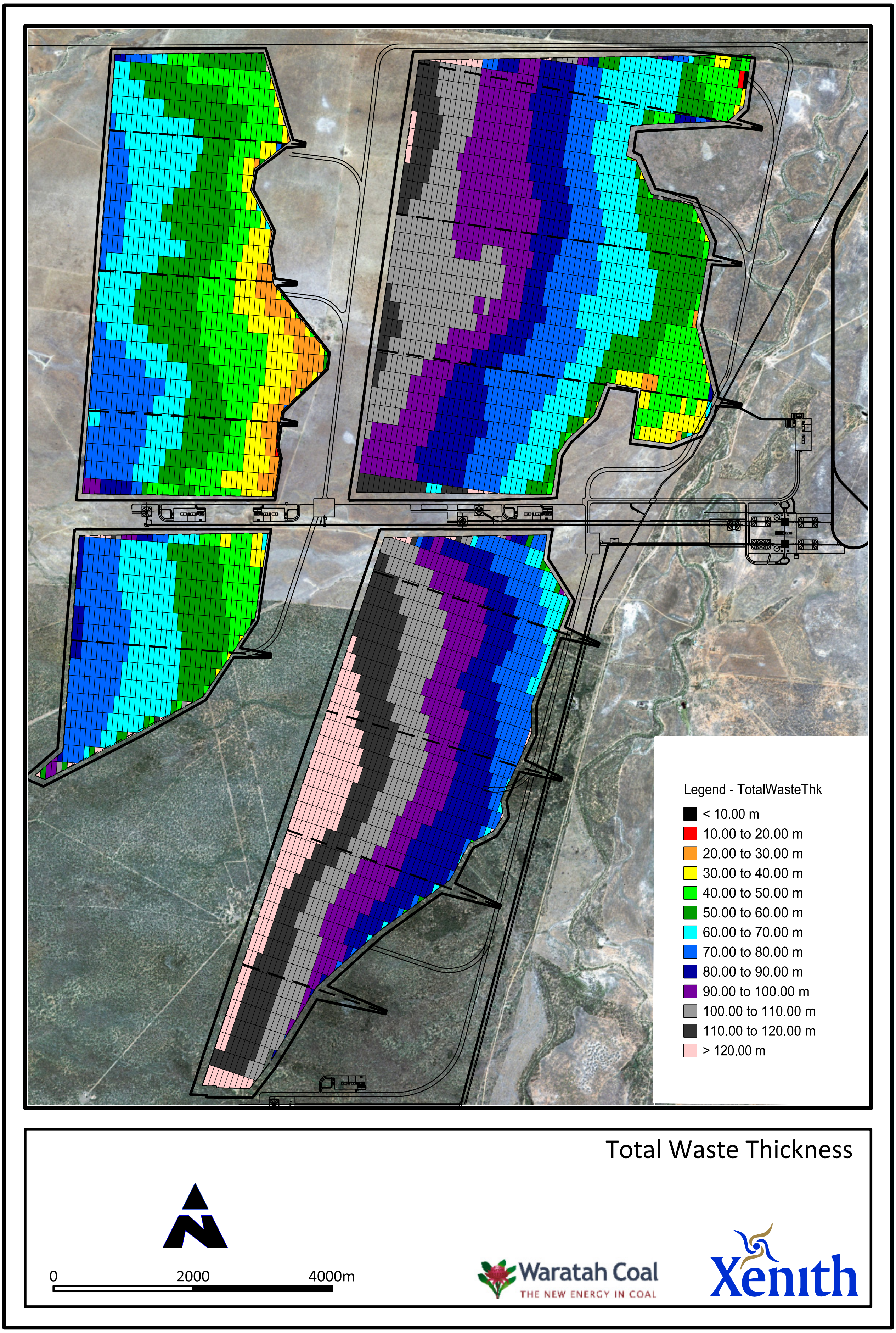
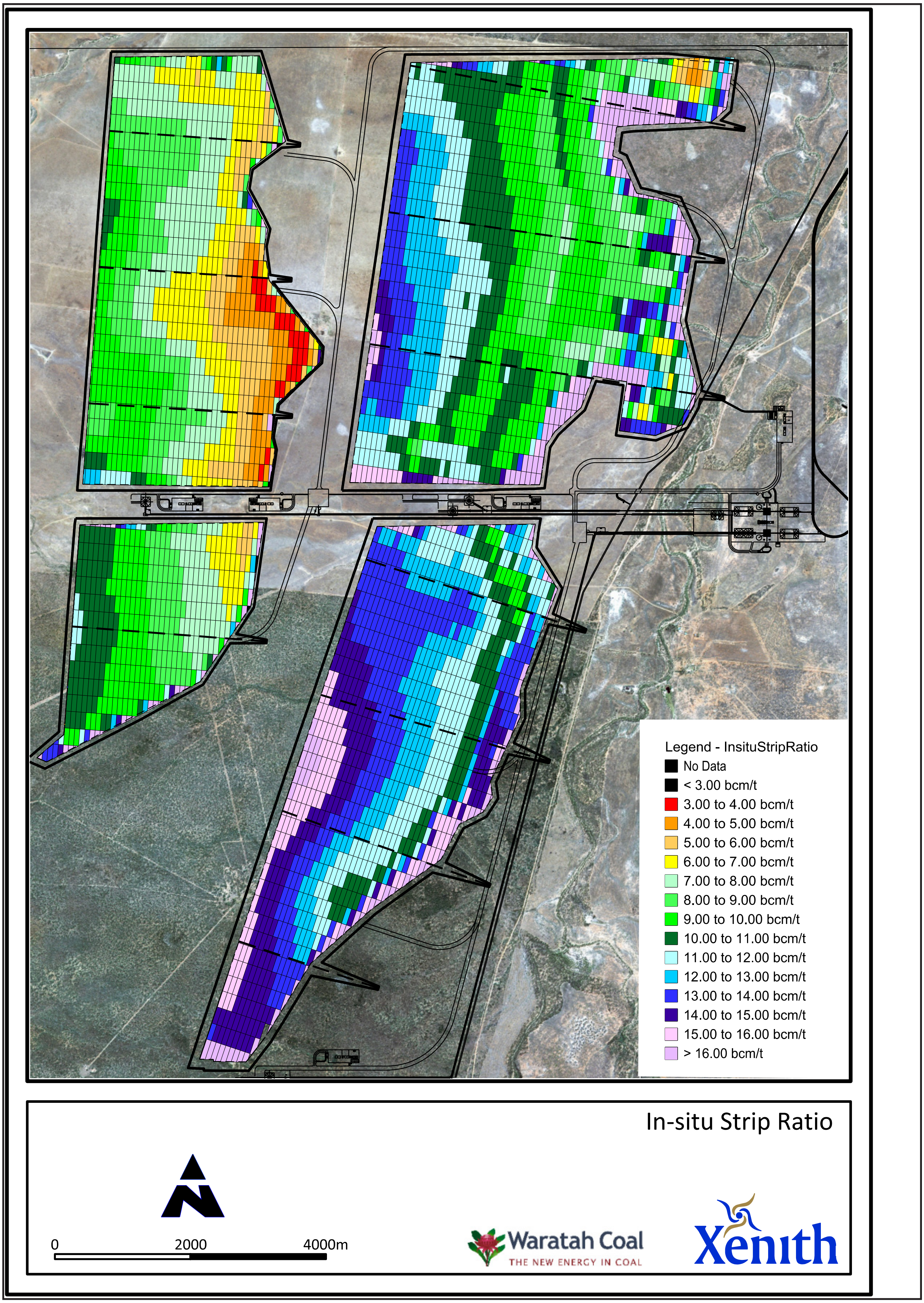


Figure 12. In-situ Strip Ratio



A brief summary of each coal seam is included below and is based on data obtained during the exploration program.

A Seam. The A seam is typically developed to one m thick, with the thickest intersection recognised so far being at around two m and located in the weathered zone in the southern region of the project area. Because of the dip and subcrop geometry, the A Seam only occurs in the far west and is not commonly intersected in drilling to date which has focused the subcrops of the B and C-D seam sets. The A seam tends to be poorly developed and contains considerable carbonaceous shale / mudstone partings.

B Seam. The B seam is the thickest in the set in the project area, typically reaching five m. The B Seam is richly banded with tuffaceous carbonaceous mudstones, especially in the top three m. This banding does influence raw ash of the overall seam and degrade its overall appeal. A distinctive, clean section of 2.0 to 2.8 m dull and bright-banded coal exists at the base of the seam. Selectively, various opportunities exist to mine the seam within this five m section.

C Seam. Thickness range of one to three m arises for C seam at the project area. This is typically developed at two m. A further two m of thinly banded stony coal and carbonaceous mudstone is often developed on the immediate roof of the C seam but is not considered to be of resource potential. The C seam profile is generally clean of bands with a trend of increasing frequency of non-coal weakness planes (penny bands) at the top of the seam near the C Upper (CU) interface.

DU Seam. The D Upper seam lies about 10 to 15 m below the C seam. It has uniform thickness in the order of 1.8 to 2.2 m. The DU seam carries some thin stone bands in the mid section but is generally clean. The DU seam has very sharp roof and floor definition and has a distinctive sharp, square-shouldered roof and floor trace on downhole geophysical logs.

This contrasts for example, with the C seam where increasing frequency of banding towards the roof causes an upwards, step-wise gradation in the geophysical logs at the roof. A variable parting of 1 to 10 m splits the DU seam away from the DL seam. All of the D seam splits are high quality and provide the lowest ash and highest energy, raw or washed, of the Project.

DL Seam. The D lower seam exists as the DL1 and DL2 splits, residing within 0.2 to 0.4 m of each other. The septum is occupied by a carbonaceous mudstone. The DL1 seam is around 0.7 to 0.9 m thick and the DL2 seam is 1.6 to 2.1 m thick. With the split included, the entire DL1 to DL2 interval has a cumulative consideration of around three to four m. The DL splits are also relatively clean intervals; three small penny bands persist in the DL2 dividing it into roughly equal intervals. Coal lithotypes are even mixtures of bright and dull coal for the D seams.

E and F Seams. Both E and F seams are one m thick. The E seam sits 10 to 20 m below the DL seam and the F seam a further 20 m lower again. They are slightly erratic in development tending to split and degrade. They have variable profiles reflecting differing levels of included stone bands. These seams sit outside limits for economic inclusion with any D seam operation, are too thin to support stand-alone development (they are not thick enough to support targeting mining; exist below thick Cainozoic associated with drainage), and so are without real potential.

1.1.10 COAL QUALITY

Product Air Dried Moisture results show a range from 7-9 %. Model results show that the B seams have much higher product ash values than the underlying C and D seams. The B seams have a product ash range from 15-20 %, while the C seam averages 8.5 %, the DU 8.5 % and the DL 8 %.

The B seams also have much lower laboratory yield (i.e. the percentage of coal extracted from a coal section) results ranging from 37 % for the B2 ply (i.e. the section of coal and bonds coupled), to 74 % for the B8 ply. If the B seam was considered as a total seam section (with stone bands included) the yield value is very low at 42 %. The C and D seam laboratory yields are within a tight range of 74 % to 84 %.

Product total sulphur values founded to be less than the raw total sulphur results, indicating the sulphur types are amenable to washing to reduce their levels. Average product sulphur across all seams in the deposit is 0.52 %.

Product coal energy for the B seams are in the 22-24 Mj/Kg range, while for the C and D seams is 26-27 Mj/Kg at a 9% moisture basis. Product coal qualities are displayed in Table 2.

Table 2. Average product quality results

COAL SEAM	PRODUCT AIR DRIED MOISTURE %	LABORATORY PRODUCT YIELD (F1.50) ADB	PRODUCT ASH % @ 9 % MOIST.	PRODUCT TOTAL SULPHUR %@9 % MOIST.	PRODUCT SPECIFIC ENERGY (MJ.KG) @ 9 % MOIST.	APPLICABLE AREA
B2	7.8	36.6	20.6	0.92	22.40	Opencut
B4	7.8	71.4	17.7	0.81	23.52	Opencut
B6	7.6	43.8	19.6	0.40	22.81	Opencut
B8	8.3	74.0	15.7	0.38	24.15	Opencut
B8	6.6	62.5	16.8	0.36	23.53	UG Working Section
B total (B2 – B8 inclusive of stone bands)	6.9	41.6	17.6	0.39	23.26	Total Deposit
C5	9.4	84.7	8.7	0.63	26.42	Opencut
Du	8.5	74.4	9.0	0.62	26.22	Opencut
DU	7.3	82.3	7.5	0.52	27.08	Underground
DL1	7.1	83.6	8.9	0.52	26.49	Opencut
DL2	7.4	79.6	7.3	0.52	27.00	Opencut
DL3	8.1	81.4	7.1	0.53	26.97	Opencut
DL	6.7	75.8	7.3	0.44	27.21	UG Working Section

1.2 KEY COMPONENTS

1.2.1 OVERVIEW AND SCHEDULE

The proposed mine consists of two open cut mines and four longwall underground mines delivering 56 Million tones per annum (Mtpa) Run of Mine (ROM) coal annually. The CHPPs are capable of producing 40 Mtpa of export coal. This will be commissioned for the mine operations. Open cut operations will involve dragline, truck and shovel operations whilst the underground operations will operate via continuous miners and longwall shearers. It is expected that the open cut and underground longwall operations will produce 20 and 36 ROM Mtpa, respectively.

The key components of the mine area are:

- two open cut mines;
- four underground longwall mines;
- two CHPPs;
- associated overland conveyors and transfer stations from mine sites to ROM and CHPP;
- ROM, primary, secondary and tertiary crushers, hoppers, apron feeders and belt and underground feeder conveyors supporting pre-preparation activities;
- four pre-preparation and two product coal storage yards;
- a mine infrastructure area that includes:
  - administration buildings and staff parking;
  - Petrol Oil Lubricant (POL) storage and handling facilities;
  - vehicle and equipment wash down facilities;
  - workshop and stores facilities;
  - laydown areas; and
  - electrical Power Substations and associated facilities.
- raw water supply for potable water production, fire fighting, coal dust suppression and coal washing;
- dragline construction facilities, including workshop, store and maintenance facility to service dragline erections and maintenance;
- a 2,000 person accommodation village including an appropriate scale wastewater treatment plant and irrigation system;
- upgrade of existing Alpha airstrip or construction of new airstrip;
- connections to the proposed 275 kV transmission line and supporting substations;
- internal road network including light-vehicle access roads, heavy-vehicle haul roads and a site access road;

- a water pipeline from a proposed dam site on the Tallarenha Creek to the mine and on-site water retention dams; and
- co-disposal and rejects storage facilities.

The proposed schedule for the development of the mine and associated infrastructure is provided in Figure 13.

1.2.2 MINING METHODS AND SUPPORTING INFRASTRUCTURE

The assessment of possible mining options has confirmed that the coal deposits are suitable for both open cut mining and underground longwall mining. The overall mine plan is to extract 56 Mtpa from two open cut and four underground longwall mining operations over a 25-year period.

The proposed mine arrangement (Figure 2) shows the key components of the selected mining methods, namely:

- topsoil stockpiles;
- water management structures (including sediment dams, levee banks, creek diversion);
- ROM and product stockpiles;
- coal rail loadout facilities;
- coal preparation plant;
- co-disposal dams and reject retention areas;
- overburden dumps;
- waste water treatment facilities;
- refueling and maintenance facilities;
- access and haul roads;
- power lines; and
- mine office, communications, and associated amenities.

The mining operations will commence with the in-parallel development of the open cut pits and the four underground mine portals.

The following sections describe in detail the selected methods for the open cut and underground mines.

1.2.2.1 Open Cut Mining Method

The Project open cut limits are defined by the following:

- eastern boundary is the relevant coal seam sub-crop line and box-cut overburden footprint;
- the extreme northern boundary allows a 50 m surface corridor adjacent to the lease boundary in B pit and a 50 m clearance from the boundary haul road in D pit;
- the southern boundary has been determined by the economic limit, mostly due to the deeper tertiary sediments and weathering profile;
- the western boundary has a 50 m stand-off at coal level from the proposed underground operations;
- a central corridor also exists and divides the open cut into North and South pits. The corridor is excluded to allow for surface infrastructure for the underground mines and conveyors;
- the mining blocks have been designed with a 20 m bench in the advancing highwall at the base of Tertiary level to act as a catch bench for any of the soft tertiary material slumping; and
- batter angle of 45 degrees in Tertiary horizon and 63 degrees in the Permian horizon.

Coal ramps are designed for the open cut mining pits that are spaced along each pit at nominal two km spacing (see Figure 14). Out of pit spoil, dumps are designed for the initial boxcut spoil volumes as well as the tertiary offset volume of the advancing strip. Out of pit spoil, dumps have a maximum height of 40 m above ground level.

Figure 13. Proposed Mine Development Schedule

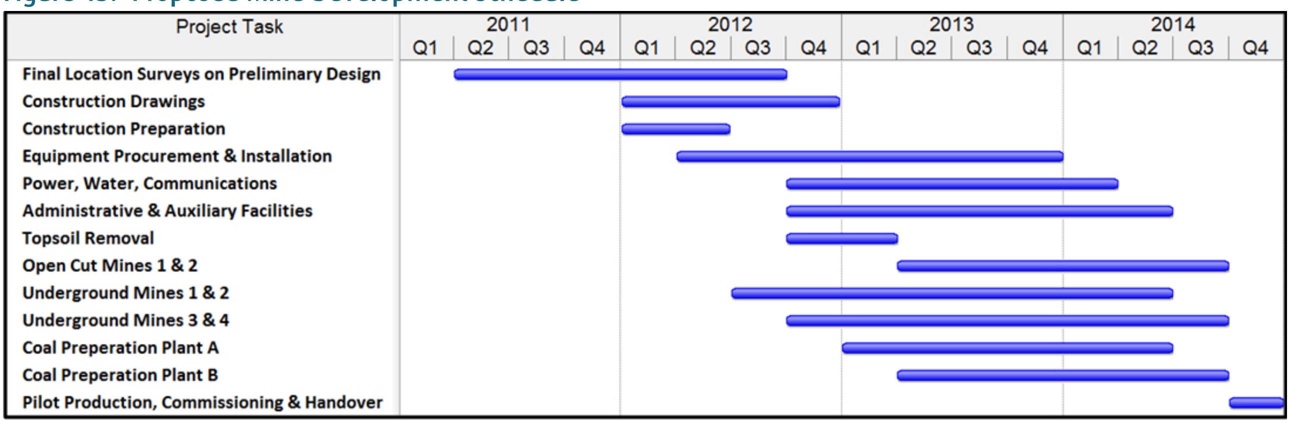
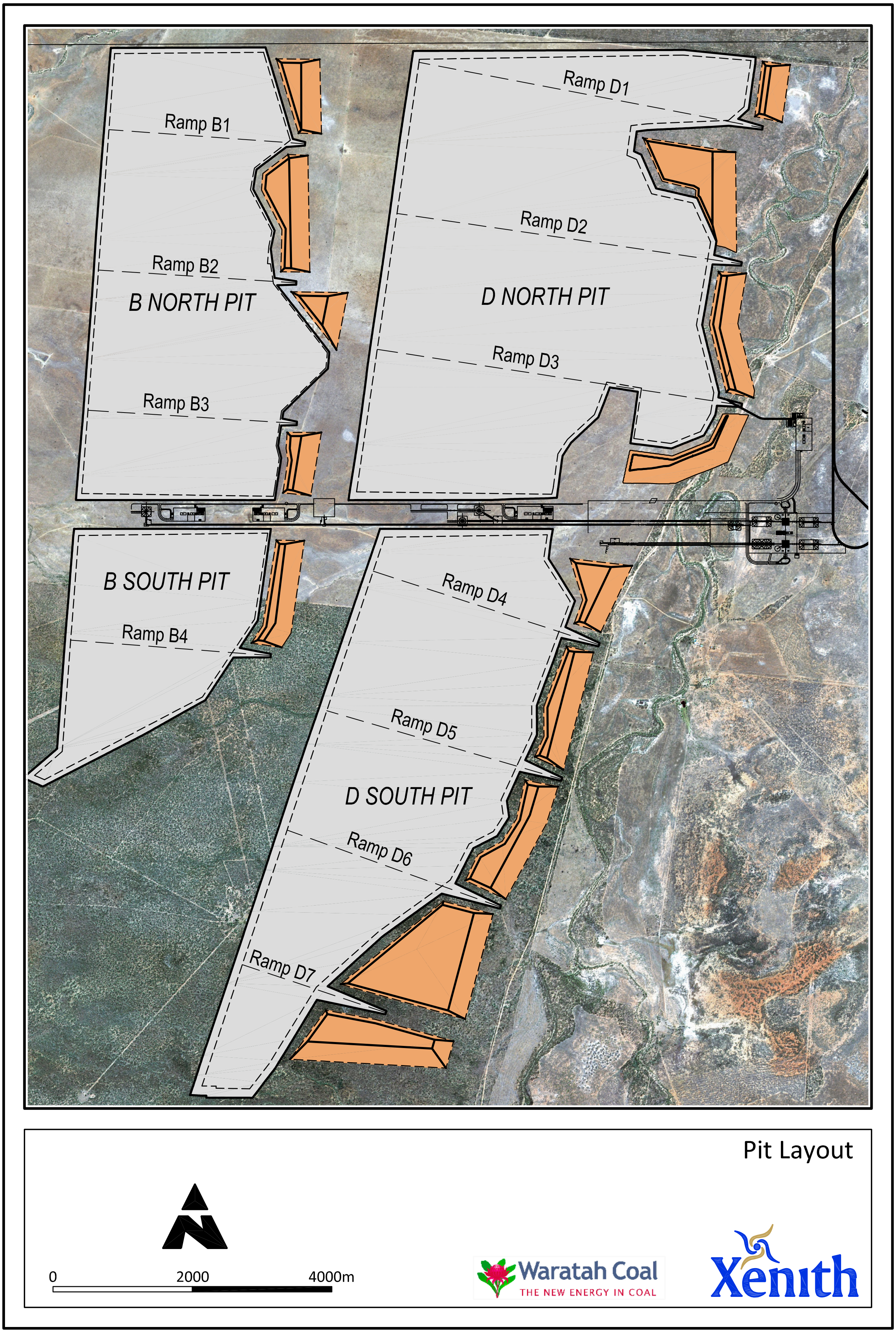


Figure 14. Opencut Pit Layout



The mining method adopted for this conceptual evaluation is a combined:

- topsoil removal and placement by scrapers;
- drill and blast operations to fracture overburden and interburden;
- large draglines removing overburden and uncovering the coal seams (see Plate 1);
- truck shovel fleets handling the overburden material not removed by the dragline including most of the tertiary material (see Plate 2); and
- truck excavator fleets handling the inter-burden between seams and to mine the coal seams.

The tertiary material is assumed to be excavated without blasting. All other overburden is assumed to be drilled and blasted prior to removal.

The dragline operation initially removes the hard blasted Tertiary and Permian material immediately above the coal seams as well as a proportion of the tertiary material. This tertiary material has to be selectively handled by the dragline in an offset strip operation resulting in significant rehandle. As the deposit deepens the proportion of this tertiary material handled by the draglines reduces, which results in less dragline rehandle and therefore more prime material is moved by the draglines. The depth of material allocated to the dragline horizon varies during the schedule with an average of approximately 45 m.

The excavator truck fleets handle the parting material between seams C and DU and between DU and DL1 that are both approximately five to ten m thick. The parting between the C and DU seams is assumed to be hauled out of pit and short dumped to regrade the coal haulage ramps. The parting between the DU and DL1 is will be be dumped in-pit to reduce the trucking requirements. The very thin DLX and DLY partings (i.e. stone bands) have also been allocated to the excavator truck fleets at a decreased productivity.

Coal is will be mined with hydraulic excavators and hauled to the ROM crushing facility for each open cut area.

Plate 1. Typical dragline

Source:photo courtesy of Bucyrus

Plate 2. Typical truck and hydraulic excavator in operation

Source: photo courtesy of Bucyrus

1.2.2.2 Open Cut Mining Development Sequence

The first stage of the mining process is for the vegetation to be cleared and the topsoil to be removed using scrapers and placed on dedicated topsoil stockpiles dumps or placed directly onto reshaped final landform if available.

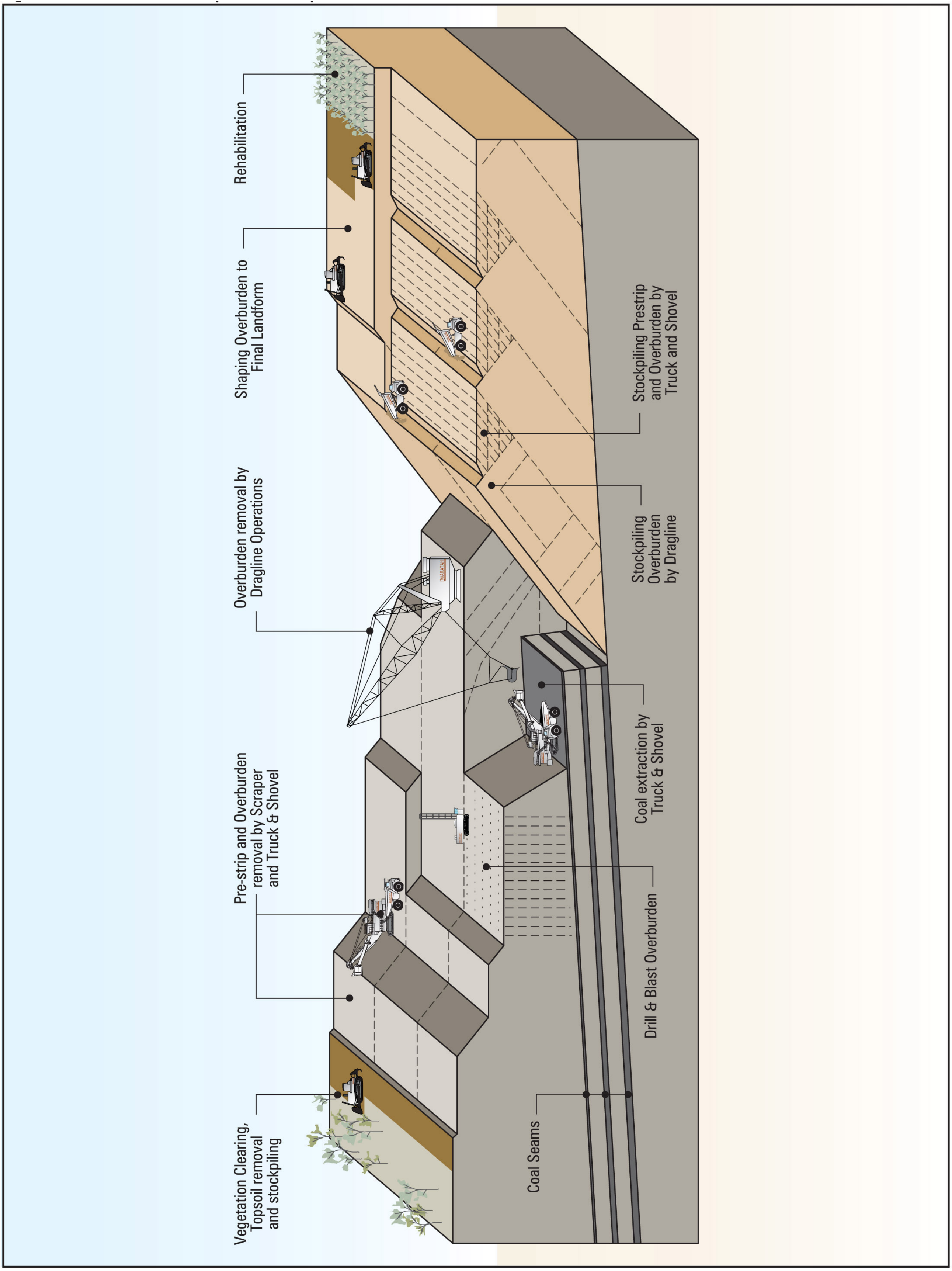
The upper portion of the Tertiary overburden where available is free dug and removed with a scraper and dozer and a truck and shovel fleet as shown at Figure 15. Where Tertiary capping rock and Permian materials become competent and digging operations cease, a drill and blast operation is utilized to fracture strata. The blast operation optimizes overburden removal by throw blasting prime material into the previous open cut void. The blasted Permian material thrown into the previous open cut void provides a substantial founding base for overburden spoil to be safely sited and anchored.

The dragline then enters the strip and the material is used to extend the initial dragline bench. Note that any tertiary material is kept high in the bench and therefore will not result in a weak spoil pile floor. The dragline then begins to remove the main Permian waste from above the coal seams. The remainder of the material above the top coal seam is then removed and used to build the spoil pile. The final material to be removed from the dragline block is from the low wall and coal seam edge, as is shown at Figure 15. The dragline will then move back to the high wall area to begin excavation of the next mining block.

The next step is for the coal mining fleet consisting of excavators, front end loaders and trucks to mine the coal seams, with the coal hauled to the CHPP for washing. Inter-burden waste between the main coal seams is then blasted and this waste is mined by the excavators and hauled by trucks to spoil dumps in the previous strips. The next coal seam is mined in the block, with the coal mining and parting operation planned to be performed in a series of sections up to 200 m in length along the pit.

The completed pit is then available for the next strip's overburden activities to begin the mining sequence again as described above. Progressive rehabilitation can be undertaken once the overburden stockpiles are reshaped by bulldozers and scrapers and the topsoil has been spread.

Figure 15. Initial Mine Concept Plan for the Open Cut Activities



1.2.2.3 Opencut Mine Development Schedule

A 25-year production schedule has been developed to produce 20 Mtpa ROM. Initially this is achieved by allocating two draglines to the D North pit, one dragline in the D South pit and one in the B North pit. Each dragline is scheduled to uncover five Mtpa. In the latter years, the draglines are moved around to balance the ratio of coal from the D and B pits.

Not all the mining blocks are extracted in the B north and B south pits during the 25 year mine plan. Coal access ramps are opened up as required, with the two most southerly ramps in the D south pit not required until year 14 and 15.

The mining sequence is shown in Figure 16.

Open cut stage plans have been developed to show the progress of the mine and the spoil dumps for milestone years – 1, 5, 10 and 20. Stage plans are shown in Figure 17 to Figure 20.

Out of pit spoil, dumps have sufficient capacity for the initial ramp, boxcut strips and the tertiary unit of the second strip after the boxcut. The spoil dumps have a maximum height of 40 m above ground level. After the out of pit spoil dumps are filled up, the spoil then progresses into mined out strips with a maximum height of 40 m above ground level. It is envisaged that most progressing spoil dumps will be at heights between natural ground level and the 40 m above ground, depending on the split of dragline spoil or truck shovel spoil.

The main coal access ramps are regraded regularly with the inter-burden spoil between the coal seams. It is anticipated that final voids with depths up to 120 m will remain in each of the four open cut pits at the completion of mining.

Figure 16. Opencut Mining Sequence

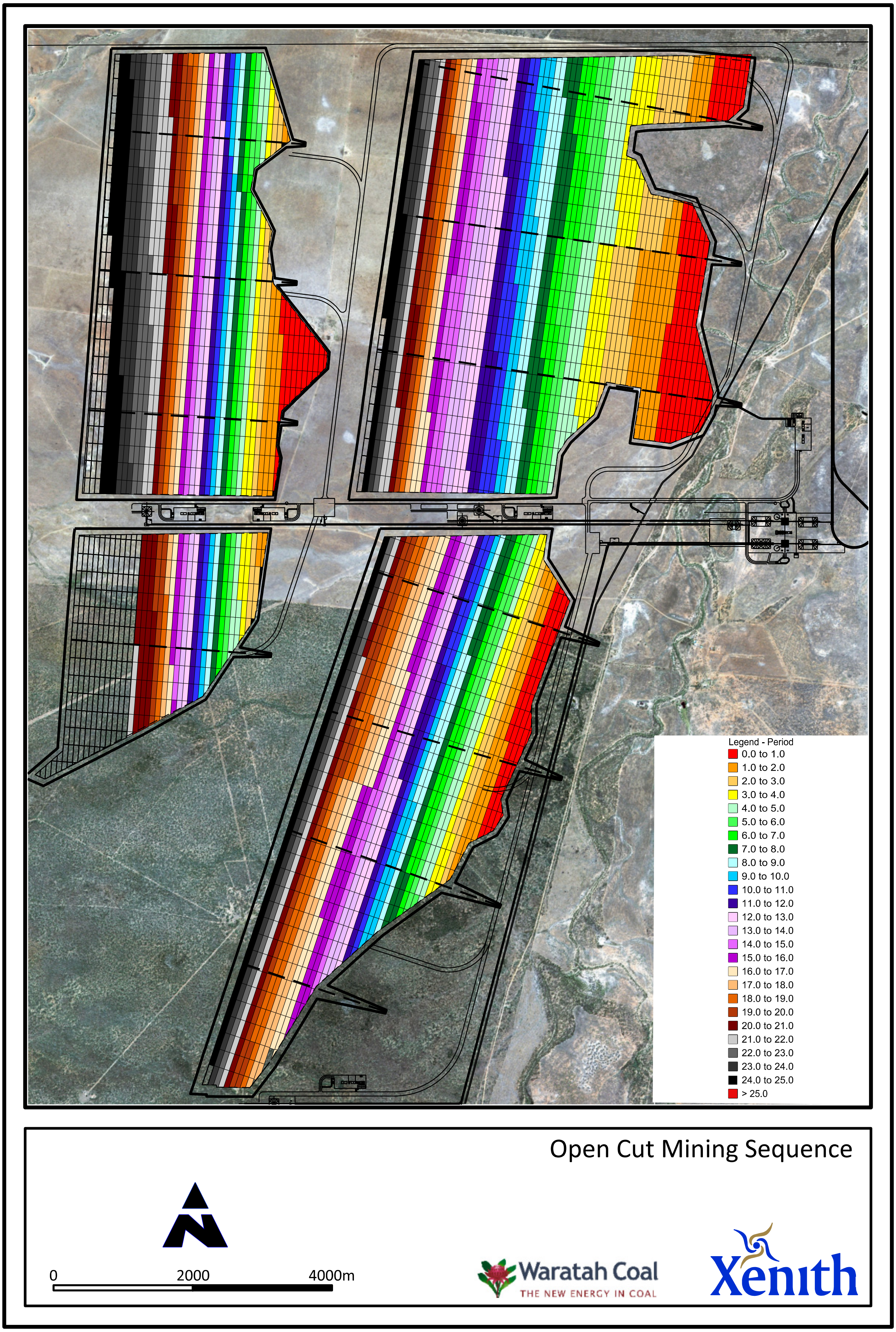


Figure 17. Opencut Year 1 Stage Plan

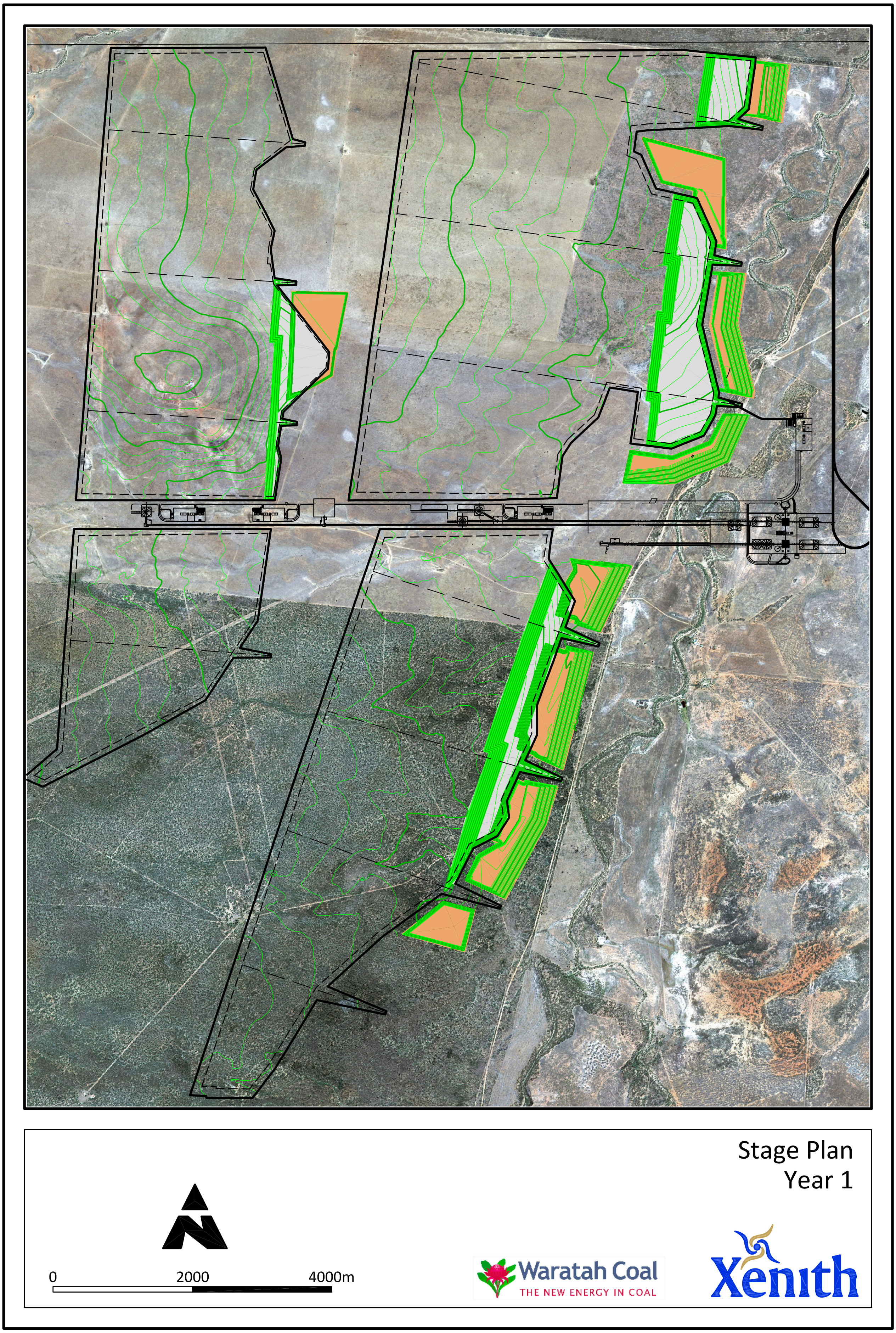


Figure 18. Opencut Year 5 Stage Plan

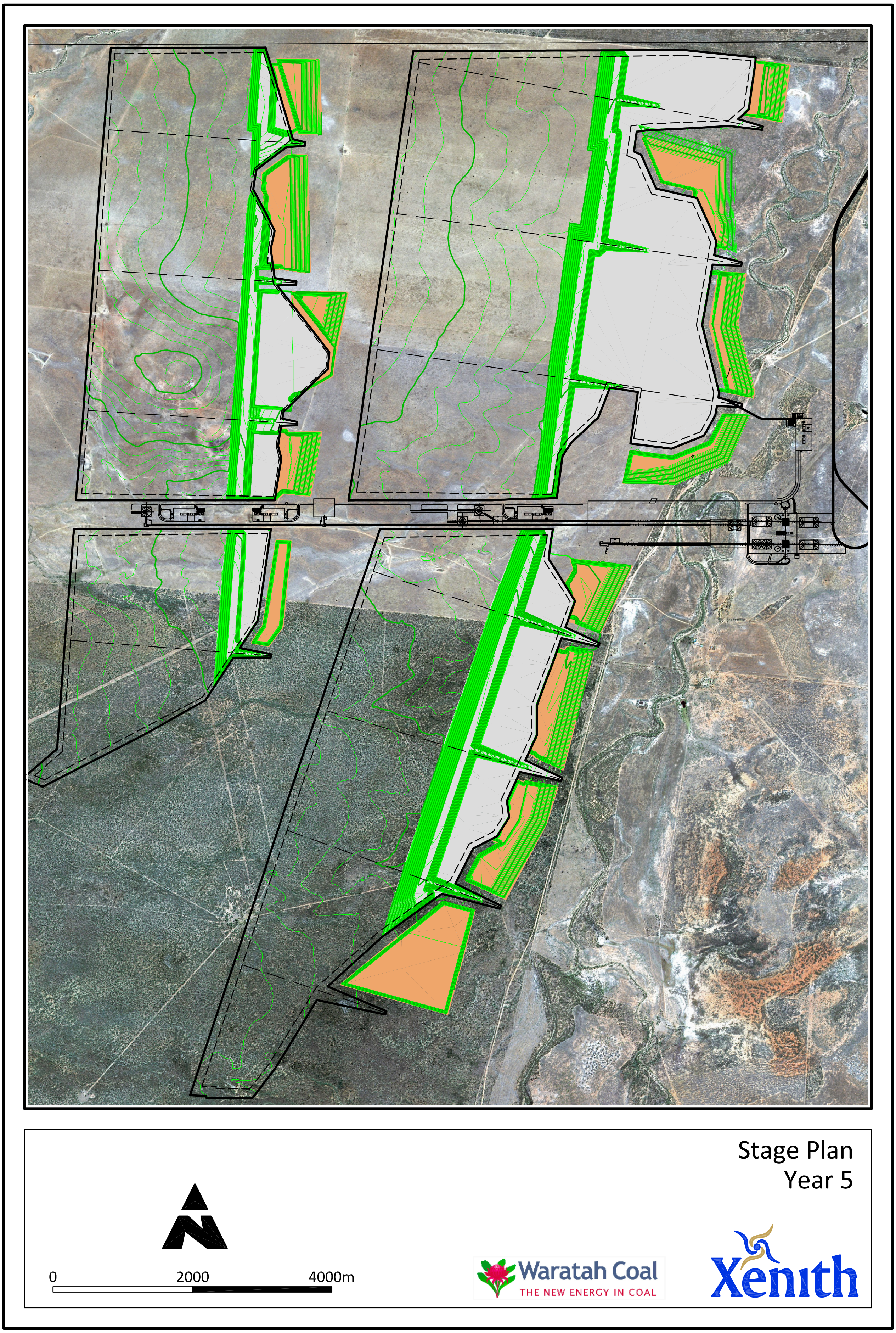


Figure 19. Opencut Year 10 Stage Plan

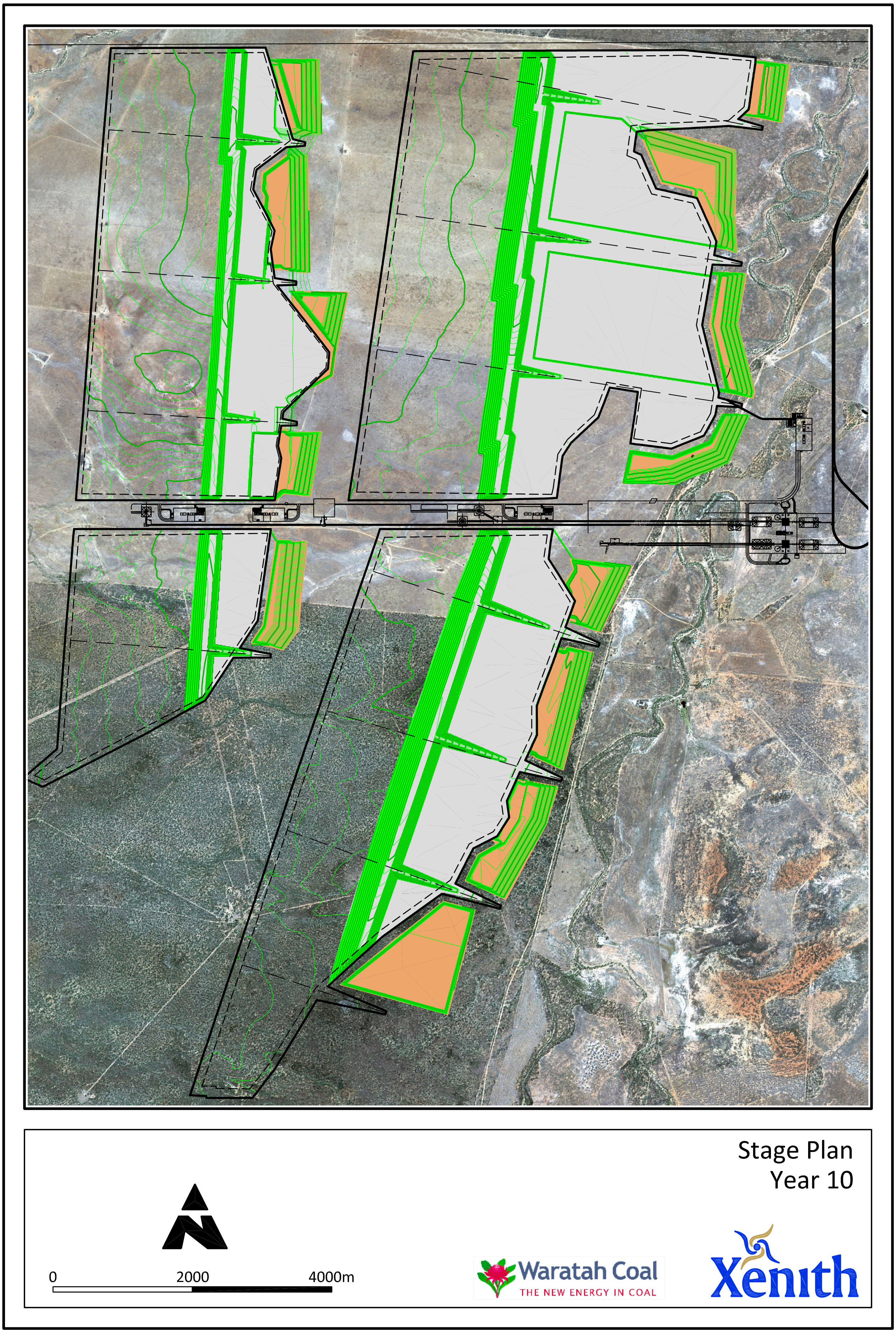
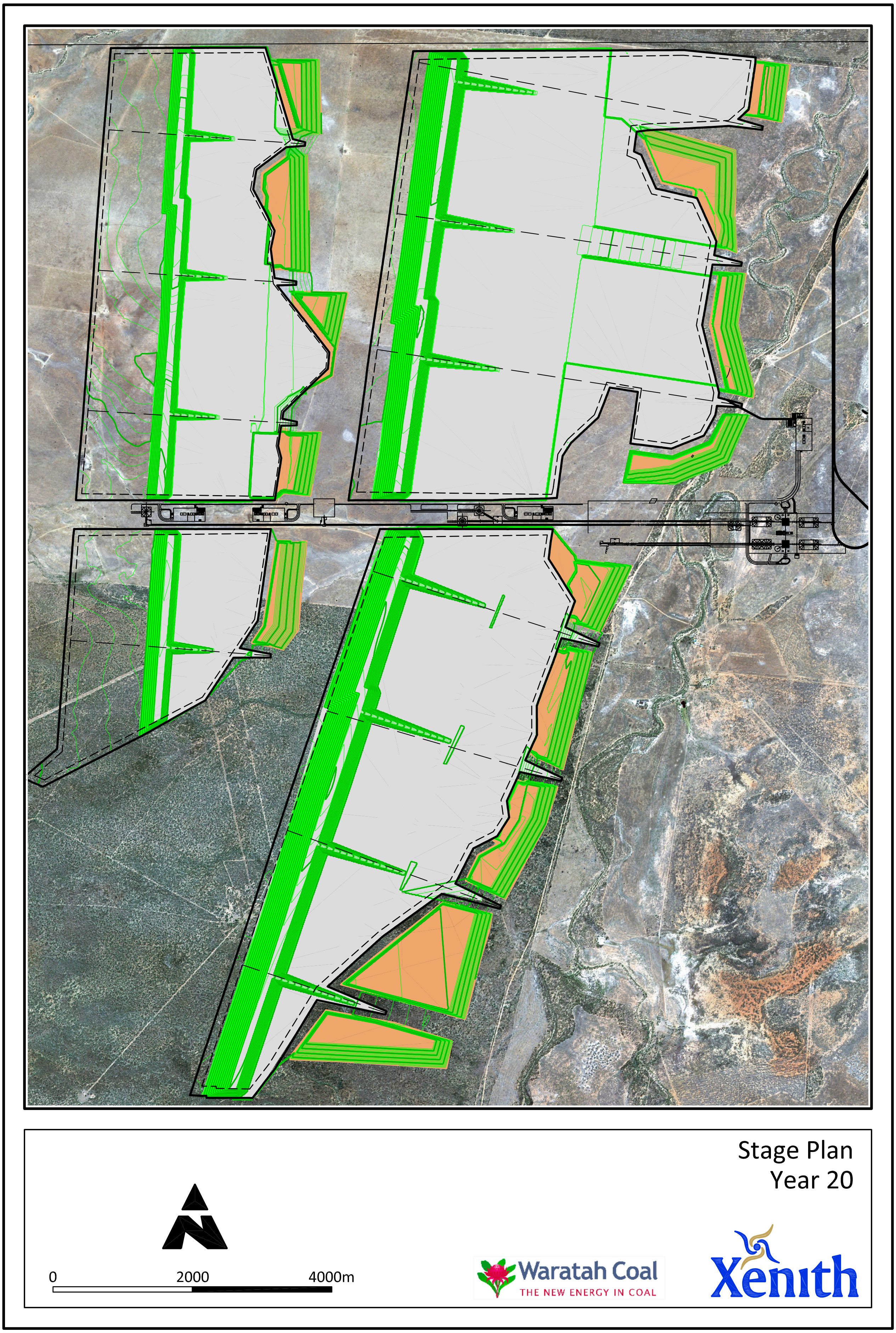


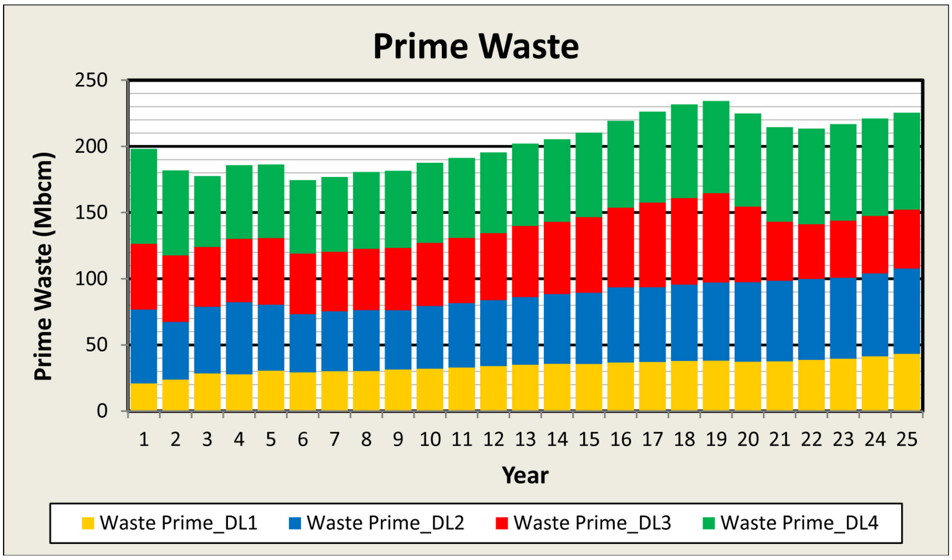
Figure 20. Opencut Year 20 Stage Plan



1.2.2.4 Opencut Waste Volumes

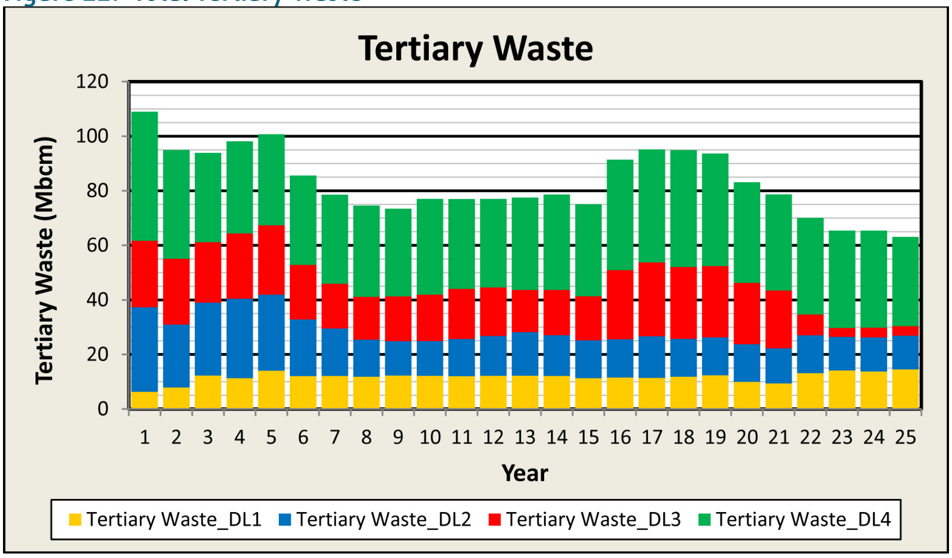
Based on the 20 Mtpa ROM coal schedule, total prime waste steadily increases from approximately 180 Million bank cubic metres per annum (Mbcmpa) in the early years up to 220 Mbcmpa in the latter years as the ROM strip ratio increases. Each dragline system (Dragline, Truck Shovel and Truck Excavator) shows variation in prime waste volumes depending on the ROM strip ratio in each of the mining pits. The potential total generation of prime waste is shown in Figure 21.

Figure 21. Total Prime Waste



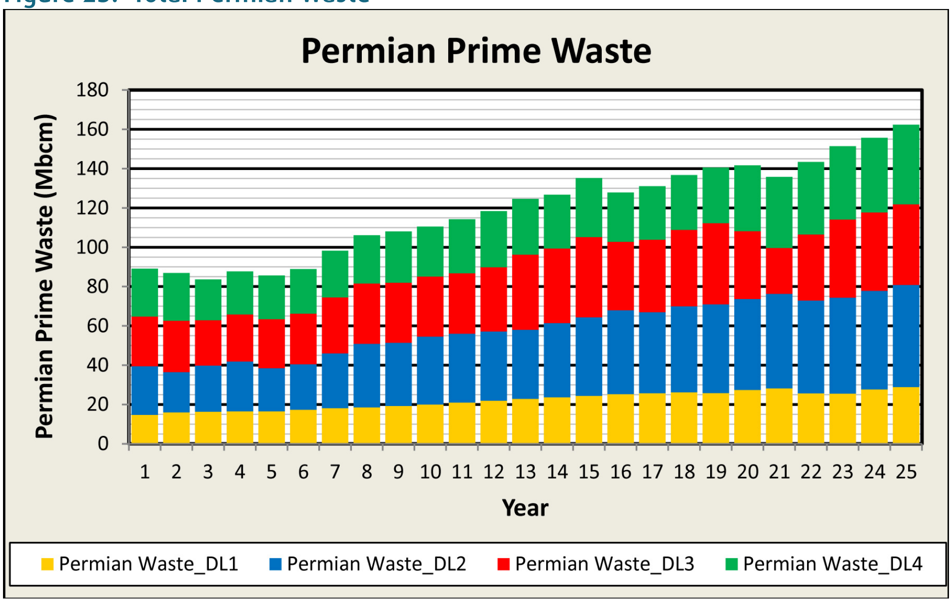
The Tertiary waste is the free-dig waste predominantly mined by the truck shovel fleets with smaller amounts handled by the draglines in offset mode. The Tertiary waste averages approximately 80 Mbcmpa over the 25 years (refer Figure 22).

Figure 22. Total Tertiary Waste



The Permian waste includes the overburden waste above the first coal seam and the interburden waste between the coal seams. The Permian waste increases over the life of the mine as the depth to the first coal seams increases as mining moves down dip. The Permian waste ranges from approximately 90 Mbcmpa in the early years to over 140 Mbcmpa from year 18 onwards (refer Figure 23).

Figure 23. Total Permian Waste



Both the Tertiary and Permian waste is scheduled to be mined by different machine combinations dependant on the dragline capacity versus the overburden requirement for each system to uncover 5 Mtpa of ROM coal. If the dragline has sufficient capacity then it is moved up into the tertiary horizon to maintain its total 28 million m3 capacity. The truck shovel system then removes any overburden waste not handled by the draglines.

A staged ramp up has also been scheduled to allow sufficient time for machine purchase and erection. The estimated life-of-project dragline and truck-shovel is shown at Figure 24 to Figure 27.

1.2.2.5 Run of Mine Strip Ratio

Average ROM strip ratio for the life of the mine is 10:1. Generally, steady increases are observed; however, this can change depending on the final dragline system implemented in each pit. The estimated ROM strip ratio is shown at Figure 28.

1.2.2.6 Blasting

Blasting will be required for the Permian overburden and inter-burden horizons in each of the four mining pits. Blasting will not be required for the coal as generally the coal seams are less than 2.5 m thick.

Figure 24. Dragline Permian Waste

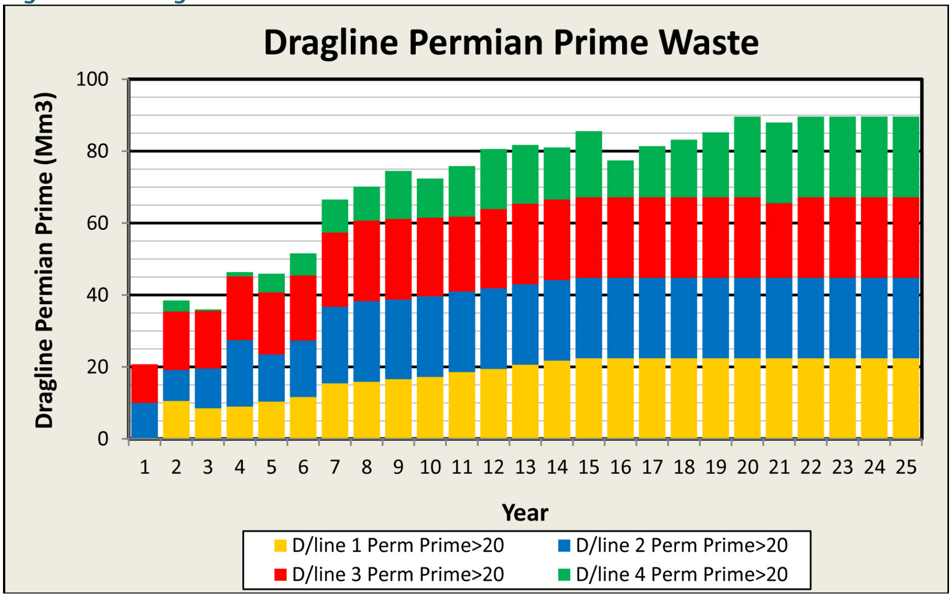


Figure 25. Dragline Tertiary Waste

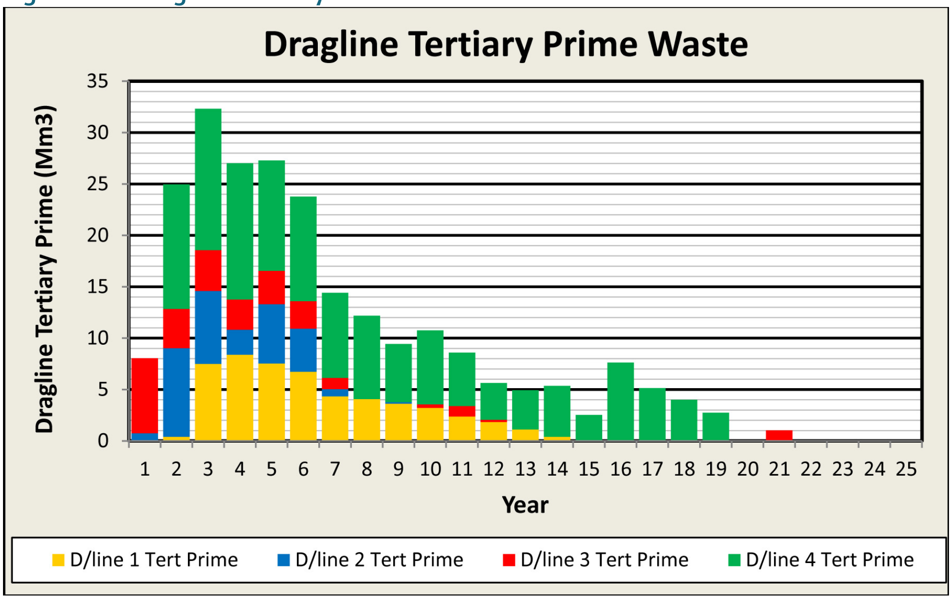
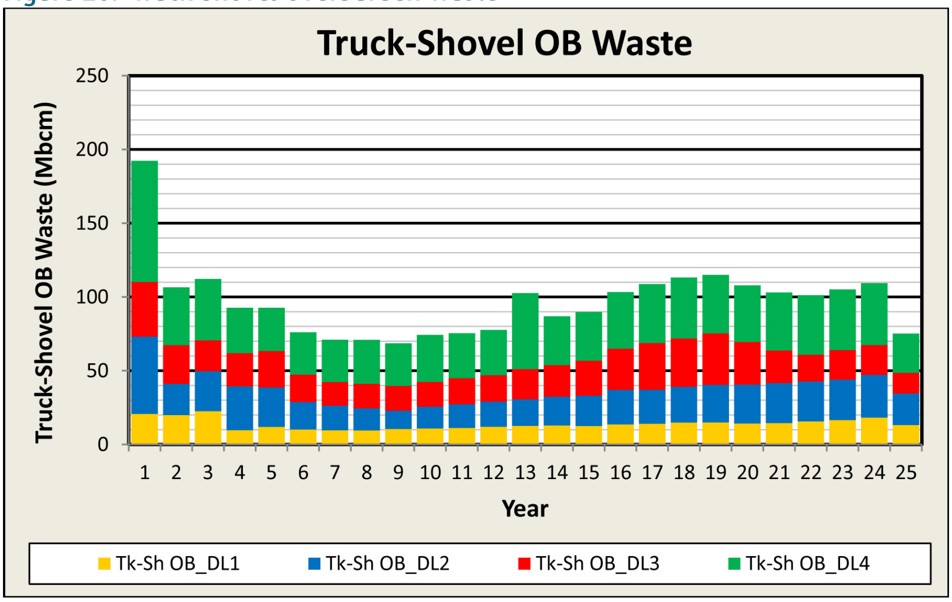


Figure 26. Truck-Shovel Overburden Waste



The range of individual blast sizes will generally be one – two Mbcm for the overburden blasts and 0.1 to 0.2 Mbcm for the interburden blasts. The total number of blasts per week is estimated to be four, with an average weekly blasted volume of 2.4 Mbcm. Table 3 provides a summary of the indicative weekly blasting requirements.

Stemming depths for blasts will typically be five m and initiation delays will most likely be around 50 milliseconds. Blasting design changes may be required when blasting approximately the infrastructure corridors as in some cases they may be inside the typical 500 m buffer zone.

It is envisaged that an explosives contractor will provide the explosives for the site. The preferred option for storage and supply of bulk explosives is for the contractor to store the unmixed chemicals at an approved facility just outside the mining lease boundary, and then transport them to site in specially designed trucks for loading into the blast holes.

Over the life of the mine the amount of bulk explosives used per annum will typically be in the 40,000 - 60,000 tonne range. Overburden and inter-burden blasted quantities are shown in Figure 29 and Figure 30.

Figure 27. Truck-Shovel Inter-Burden Waste

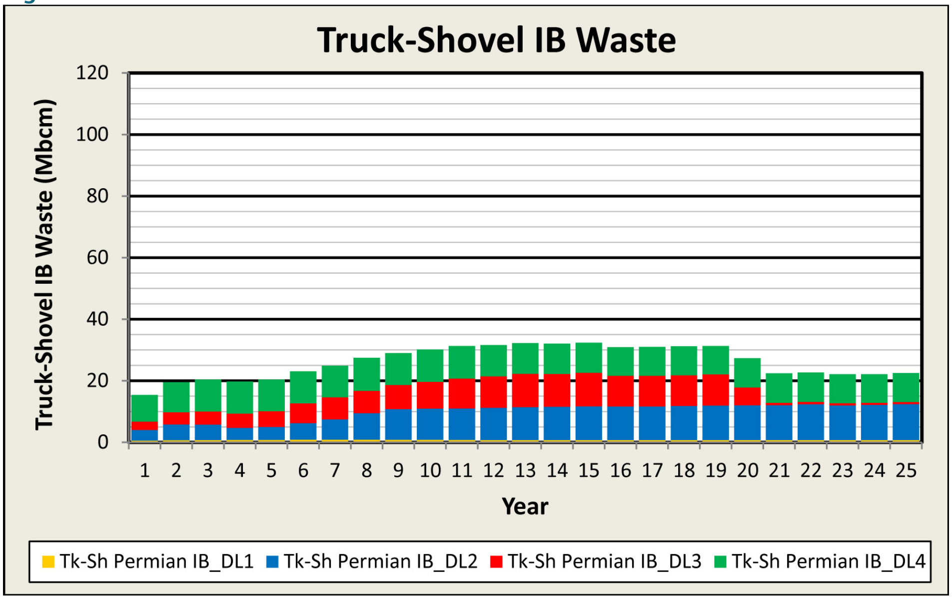


Figure 28. ROM Strip Ratio

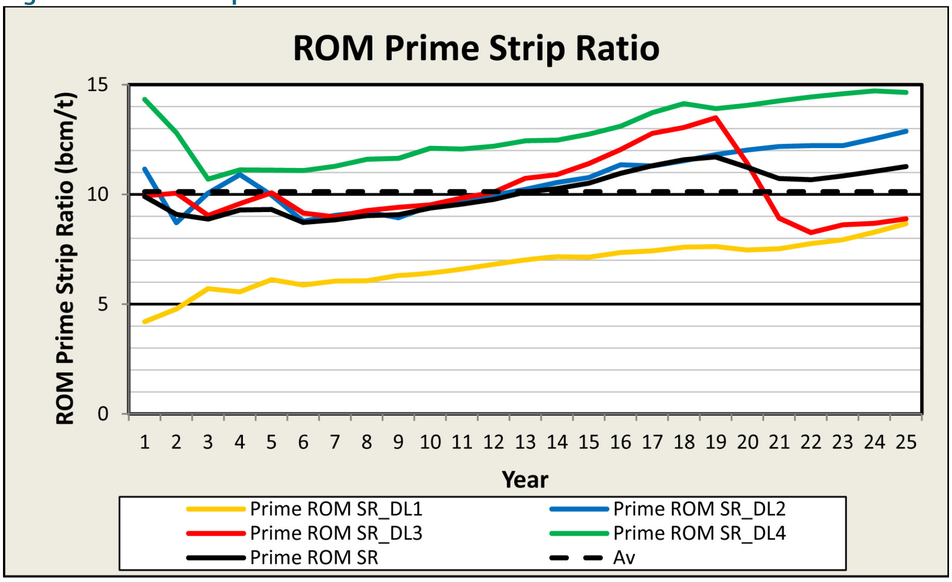


Table 3. Blasting Summary

ITEM	UNITS	ANNUAL	WEEKLY	TYPICAL BLAST SIZE	TYPICAL NUMBER OF BLASTS PER WEEK
Average Blasted OB Volume	Mbcm	93.5	1.9	1.5	1
Average Blasted IB Volume	Mbcm	26.1	0.5	0.18	3
Total Blasted Volume	Mbcm	119.6	2.4		4
Average Explosive Usage for OB	t	37,400	748	600	
Average Explosive Usage for IB	t	9,100	182	63	
Average Total Explosive Usage	t	46,500	930		

1.2.2.7 Underground Mining Method

The underground mines will produce coal by a modern, mechanized, retreating longwall mining system. This mining method is well established, and is used widely in Australia and overseas. Use of the longwall mining method will enable an annual production rate of approximately nine Mtpa ROM from each mining area. Four mining areas are planned to be mined in parallel (Mines 1 to 4), with three mines in the D-Seam, and one mine (Mine 4) in the B-Seam.

The proposed longwall mining blocks are approximately 470 m wide, rib-to-rib. Once extracted, and including the development roadways on either side of the longwall block, the total extracted width is 480 m. The lengths of the longwall blocks will be up to 7,000 m.

Figure 29. Overburden Blast Quantities

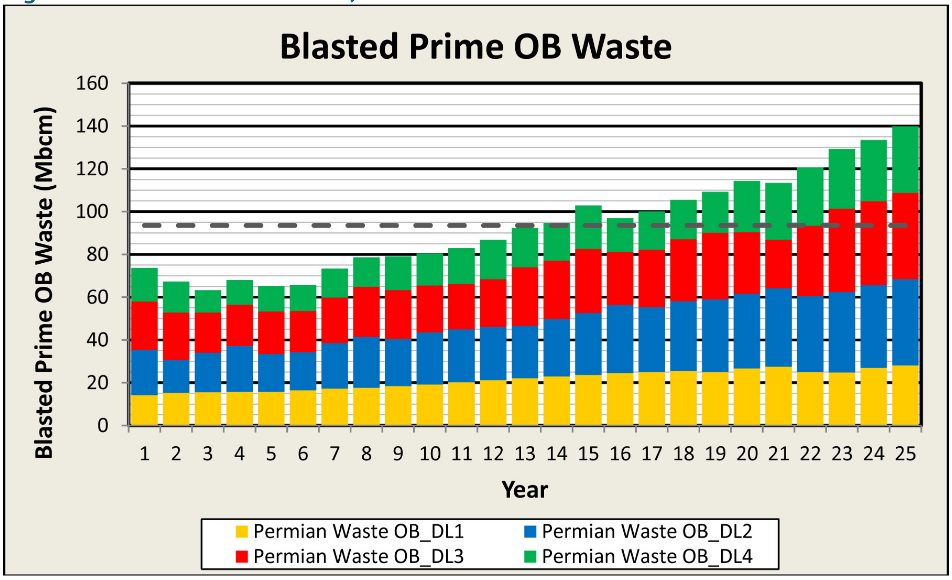
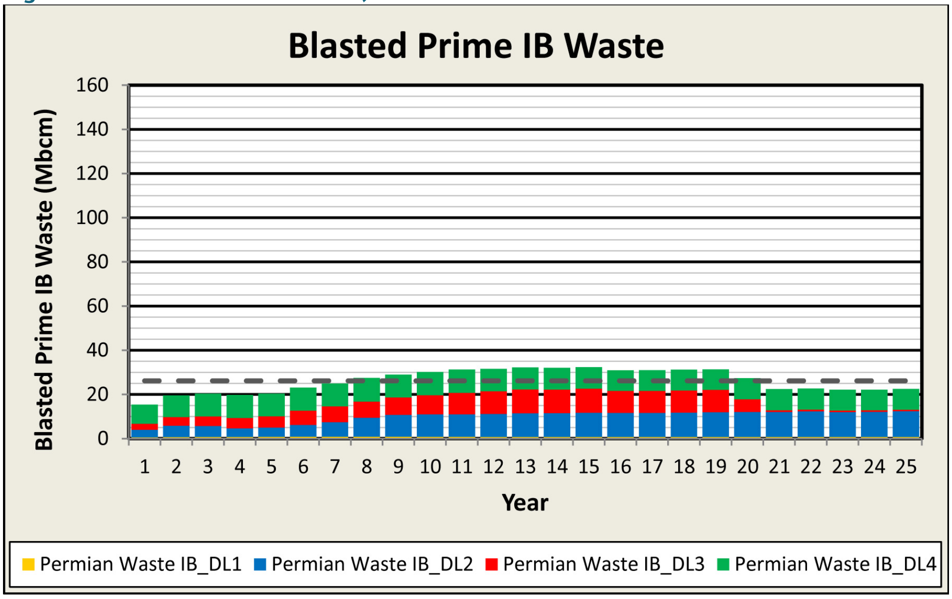


Figure 30. Inter-Burden Blast Quantities

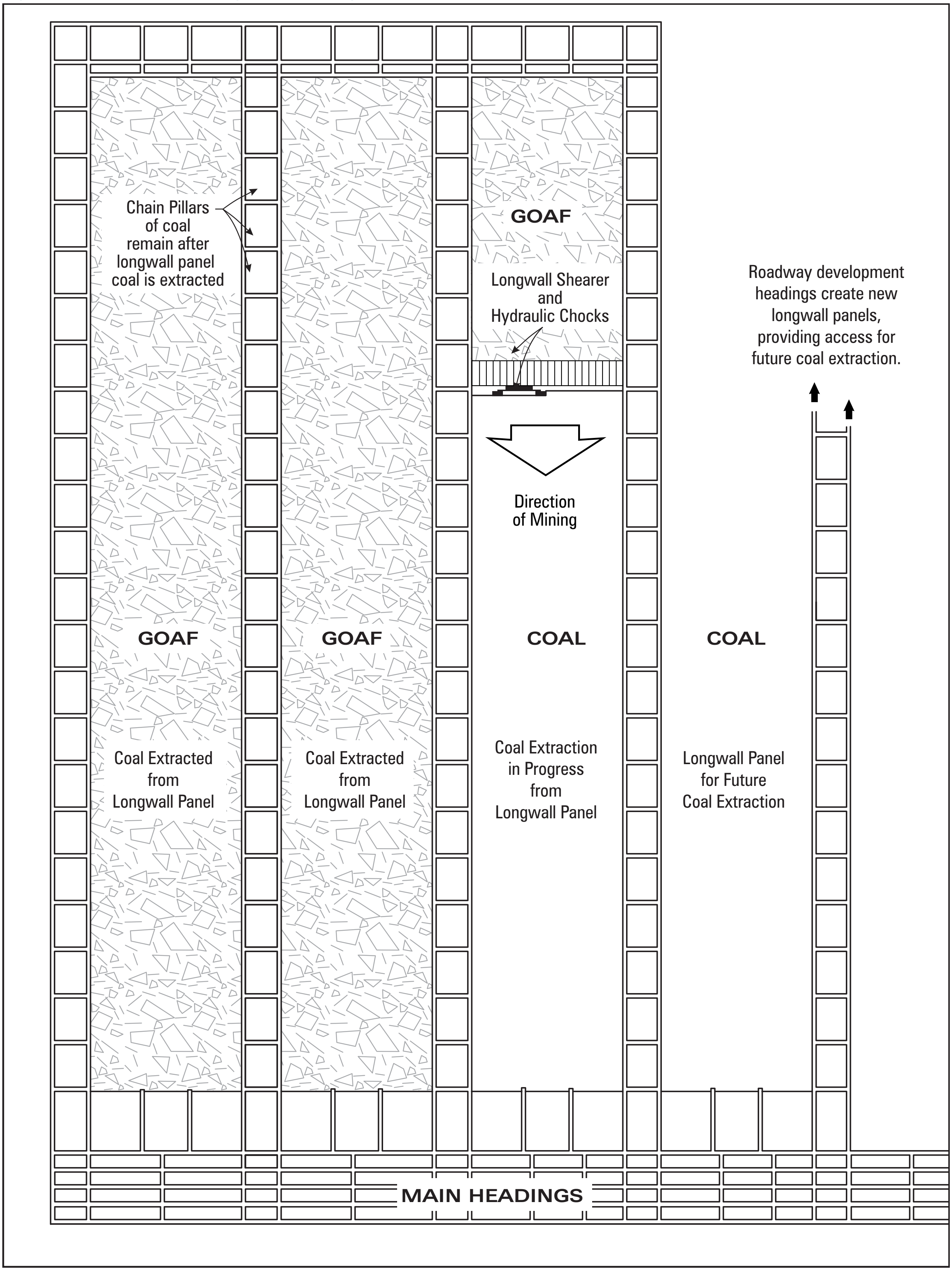


Between each longwall, extraction block and a coal pillar will be left with a total width of 20 m rib-to-rib and a length between cut-through of 95 m rib-to-rib.

The projected mine access roadways will be mined at a width of 5.0 m, and a minimum height of 2.5 m. The gateroads alongside the longwall blocks will be mined as two headings with a centre-to-centre distance of 25 m, and a distance between cut-through of 100 m (centre-to-centre). The main roads will consist of five headings running parallel, with a centre-to-centre distance of 28.75 m and 100 m spacing between cut-through (centre-to-centre).

Illustrated schematic of the proposed development is Figure 31.

Figure 31. Proposed Underground Mining Concept



1.2.2.8 Underground Mining Development Sequence

The underground mines development will initiate via the inclined drifts down from the surface. There will be three drifts per mine. These drifts will separately service personnel and materials, the conveyor and ventilation. The drifts will begin on the surface near the open cut mining areas, and develop in an east-to-west direction to meet the coal seams below ground.

Once the drifts have reached the coal seams, main development headings (consisting of five roadways running parallel to each other), develop in order to reach to mining areas for all the subsequent longwall blocks.

The initial production stage of longwall mining involves the development of roadways around the blocks of coal. This process will extract the coal via longwall mining. The roadways define the boundaries of the block, which known as “gateroads”. These roads are also required to provide employee access, machine access, ventilation, electrical supply, communications systems, services lines and coal transport.

The development roadways remove only a minor portion of the coal seam area, and are designed to maintain stability during both the development and longwall extraction phases. The roadways support mechanical strata control, which is not intended to fail or converge significantly during the life of the mine. Consequently, there are no subsidence impacts from development roadway workings (“first workings”).

The value of coal extracted with the associated development of roadways does not meet mining costs of extracting this coal. However, the economic returns from investing in roadway development result from the subsequent longwall extraction, utilising previously developed roadways.

Longwall face equipment installation at the end of the longwall block is furthest away from the main headings, where extracting the coal in a “retreating” method back towards the main headings. Upon completion of the mining of each block, the longwall equipment will locate back to the other end of the next block in the series, and the mining process repeats.

Longwall mining totally removes the blocks of coal between the developed roadways. Longwall shearing machinery travels back and forth across the coalface in each block. This machine (“shearer”) cuts the coal from the coalface on each pass and a face conveyor, running along the full length of the coalface, carries this away to discharge onto a belt conveyor. A series of belt conveyors then carry the coal out of the mine.

The section in front of the coalface is held up by a series of hydraulic roof supports. These temporarily hold up the roof strata, enable enough space for the shearer, and face conveyor. After each slice of coal is removed (typically one m in width), the face conveyor, hydraulic roof supports and the shearer are moved forward. As the hydraulic roof support moves forward the overlying strata (“roof”) behind the equipment collapses in the goaf. The extent of the overlying strata collapse and the associated shearing and cracking of the strata depends upon the strata geology, the longwall block width, the seam height extracted, and the depth of cover.

A cross-section through a typical longwall face is shown in Figure 32. An image of the machinery arrangement in operation on a typical longwall face is shown in Plate 3. The hydraulic roof supports are visible on the right hand side and the coalface on the left hand side of the image. The drum in the background is the rotating cutting head of the coal shearer, and the chain face conveyor can be seen fully loaded with coal in the foreground.

During the longwall mining process, the entire coal seam (or a selected section of it where applicable to the specific mining area), is removed from the ground. In areas where the coal seam has been extracted, the strata immediately above fails into the void, creating what are known as the goaf areas. Due to the breaking up and swelling of the rock mass into this void, the amount by which the overlying strata subsides is less than the height of the coal extracted, with the amount of subsidence movement decreasing with height above the coal seam.

Figure 32. Cross Section of a Typical Longwall Face

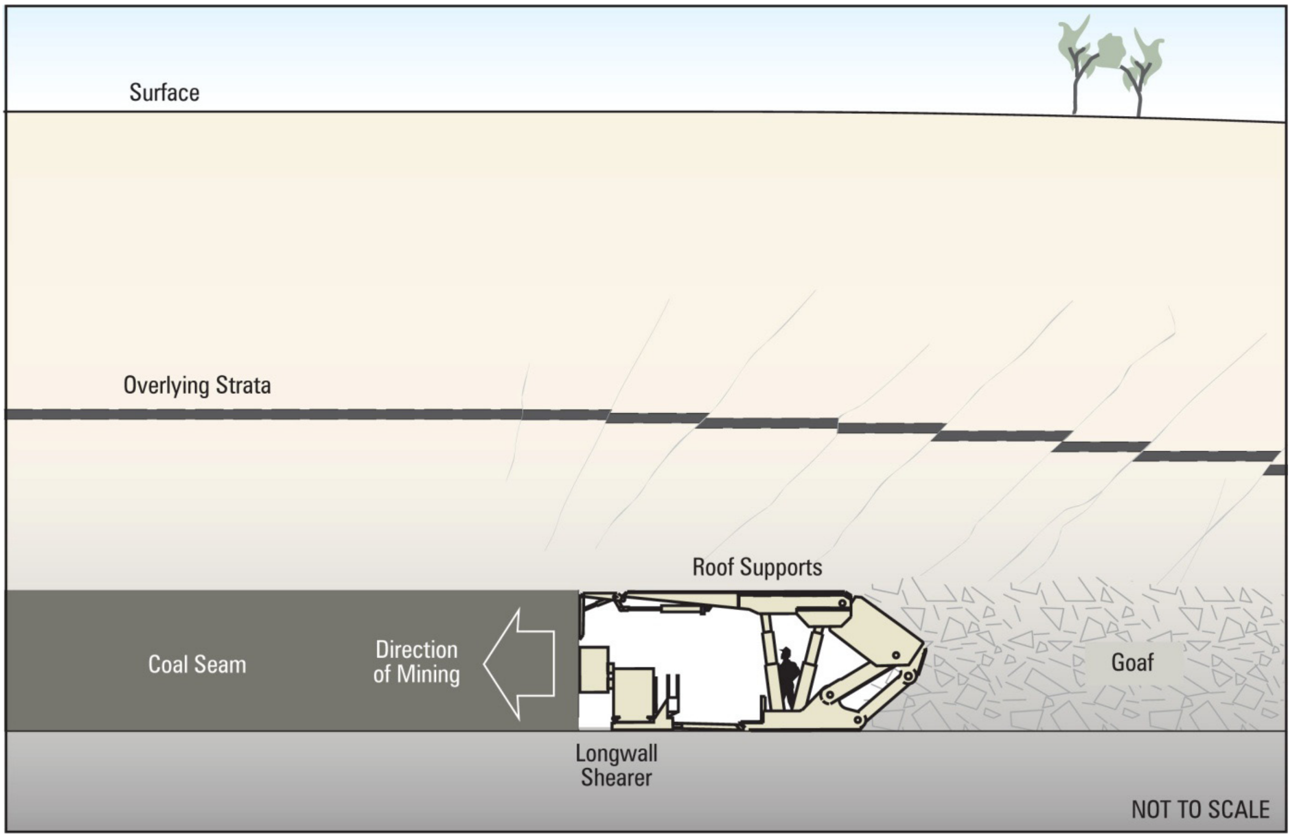


Plate 3. Typical Longwall Face Equipment Arrangement

The five year underground development sequence for the B and D seams are shown in Figure 33 and Figure 34 respectively.

Figure 33. B Seam Mine Development – 5 Year Intervals

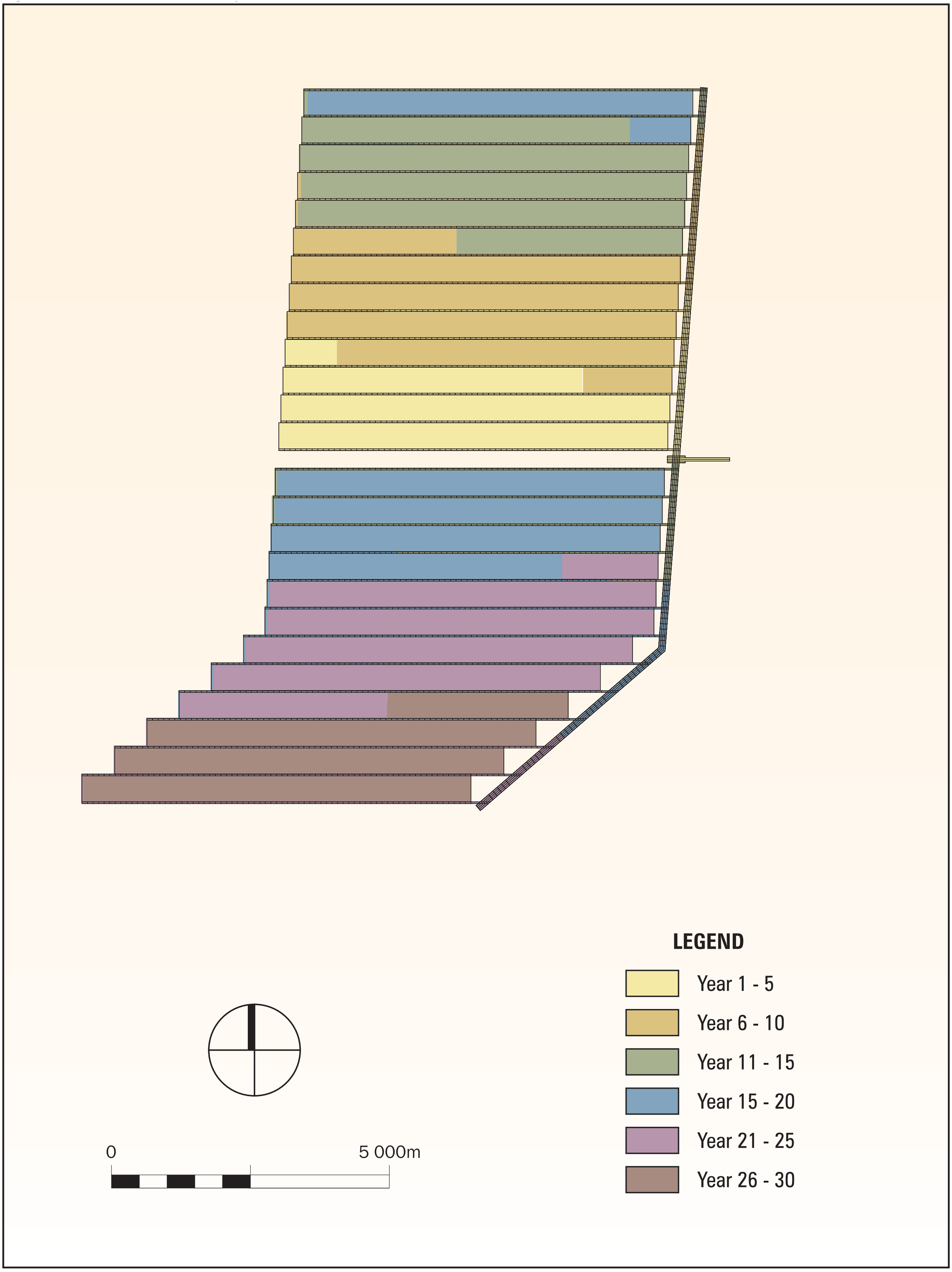


Figure 34. D Seam Mine Development – 5 Year Intervals

1.2.3 COAL HANDLING SYSTEM

The coal handling system consists of a raw coal system, a product coal system and a rejects coal system. This incorporates simultaneous coal feed from four underground mines and four open cut mines supplying two stand-alone CHPPs each capable of treating 4,000 tonnes per hour (tph). Materials handling capacity has been set at 56 Mtpa of raw coal. The product coal handling plant has a capacity of 40 Mtpa. A schematic showing the coal handling system is shown in Figure 35.

The underground longwall mines are designated:

- longwall Mine 1 in the northern area mining the D upper and D lower seams;
- longwall Mine 2 in the central area mining the D lower seam;
- longwall Mine 3 in the southern area mining the D lower seam; and
- longwall Mine 4 in the western area mining the B seam.

The open cut mines have been designated:

- OC1 North mining the C and D seams;
- OC1 South mining the C and D seams;
- OC2 North mining the B seam; and
- OC2 South mining the B seam.

The raw materials handling system provides for four streams feeding the raw coal stockpiles:

- LW1 and LW2 feeding Seam D at 18 Mtpa;
- OC2 and LW4 feeding Seam B at 19 Mtpa;
- OC1 feeding Seams C and D at 10 Mtpa; and
- LW3 feeding Seam D at nine Mtpa.

This effectively rationalises the conveyor systems to two basic feed rates for best design scale.

1.2.3.1 Raw Coal Plant Layout

1.2.3.1.1 ROM Coal – Open Cut

Raw coal from the open cut pits will be transferred to a ROM pad by truck at nominal 600 mm size. The B seam pits OC2 North and South will discharge to a common primary crushing station as will OC1 North and South for seams C and D. There will be one ROM pad, ROM bin and primary crusher arrangement at each of the open cut mines OC1 and OC2. Secondary and tertiary crushing stations will be located immediately after each of the primary crushing stations.

Coal dumped directly into a ROM bin when the CHPP is running at capacity or deposited into a stockpile to allow surge capacity.

Plate 4 shows a typical ROM dump station. Reclaim feed to the ROM bin from the stockpile will be by front end loader. An elevated ROM pad will be constructed using a reinforced concrete design around the crusher pocket. The top level will be nominally 20 m high to allow transfer chute layout within the crushing station.

Primary crushing takes place immediately under the ROM feed bin with the crusher set to 300 mm. The primary sizer is a low speed sizer, a combination of high torque and low roll speeds with a unique tooth profile. Plate 5 shows a typical open cut sizer.

The secondary and tertiary crushing stations are effectively identical to the configuration adopted for the underground ROM coal. That configuration replicates the longwall layout to provide a common CHPP raw coal feed at 50 mm throughout.

1.2.3.1.2 ROM Coal – Underground

Each longwall mining operations will deliver +300 mm coal to dedicated drift stockpiles. Each drift stockpile will be a single cone stockpile 60 m high, providing up to a 450,000 t capacity with additional storage capacity available from dozer push-out.

Each drift stockpile will incorporate a single reclaim tunnel with three reclaim chutes rated at 1,000 tph each to provide 3,000 tph feed capacity to the coal handling and preparation plant stockpile system. Feed from the stockpile is sized at +300 mm. Coal valves and belt feeders will control loading of the drift stockpile ROM reclaim conveyor.

The reclaim chambers and tunnel will be cast *in-situ* with reinforced concrete. The conveyor will be hung from the tunnel roof with access to both sides for personnel and for bobcat machine clean up. Escape tunnels in compliance with code requirements will extend to clear the stockpile footprint. The conveyor tunnel will have induced draft ventilation.

The reclaim conveyor from each drift stockpile will feed coal to a two stage crushing plant, comprising a secondary sizer, roller screen and tertiary sizer, sizing the coal to 50 mm from 300 mm.

In this process, any undersize coal from the reclaim conveyor reports directly to the tail end of a transfer conveyor via lined chutes (Plate 6). A magnet will be installed at the head pulley of the secondary sizer. The magnet will be placed to remove foreign objects from the process. The secondary sizer will size the product from 300 mm to approximately < 150 mm. The product will leave the secondary sizer and discharge onto a roller screen. The roller screen will filter out the product sized to 50 mm and transfer that product through the tertiary sizer directly to the outloading conveyor through chutes.

There will be a secondary and tertiary crushing station dedicated to each underground mining operation.

The conveyed "raw coal" transferred and loaded to an overland conveyor. This process continues to a transfer tower for transportation to raw coal stockpiles.

1.2.3.2 Raw Coal Conveyor Configuration

Conveyor transfers the B seam product to the B overland conveyor. The C and D seams report to the dedicated C and D overland conveyor. The raw coal stockpile configuration and feed to the CHPP shown in Figure 35.

The ROM conveyor configuration is shown in Table 4.

Table 4. ROM conveyor configuration specifications

DESCRIPTION	BELT SPEED (M/S)	BELT WIDTH (MM)	CAPACITY (TPH)
Drift Stockpile Reclaim Conveyors	4.0	1,600	3,000
ROM Reclaim Conveyor	4.0	1,600	3,000
Transfer Conveyor	4.0	1,600	3,000

Plate 4. Typical Open Cut ROM Dump Station

Figure 35. Schematic Representation of the Coal Handling System

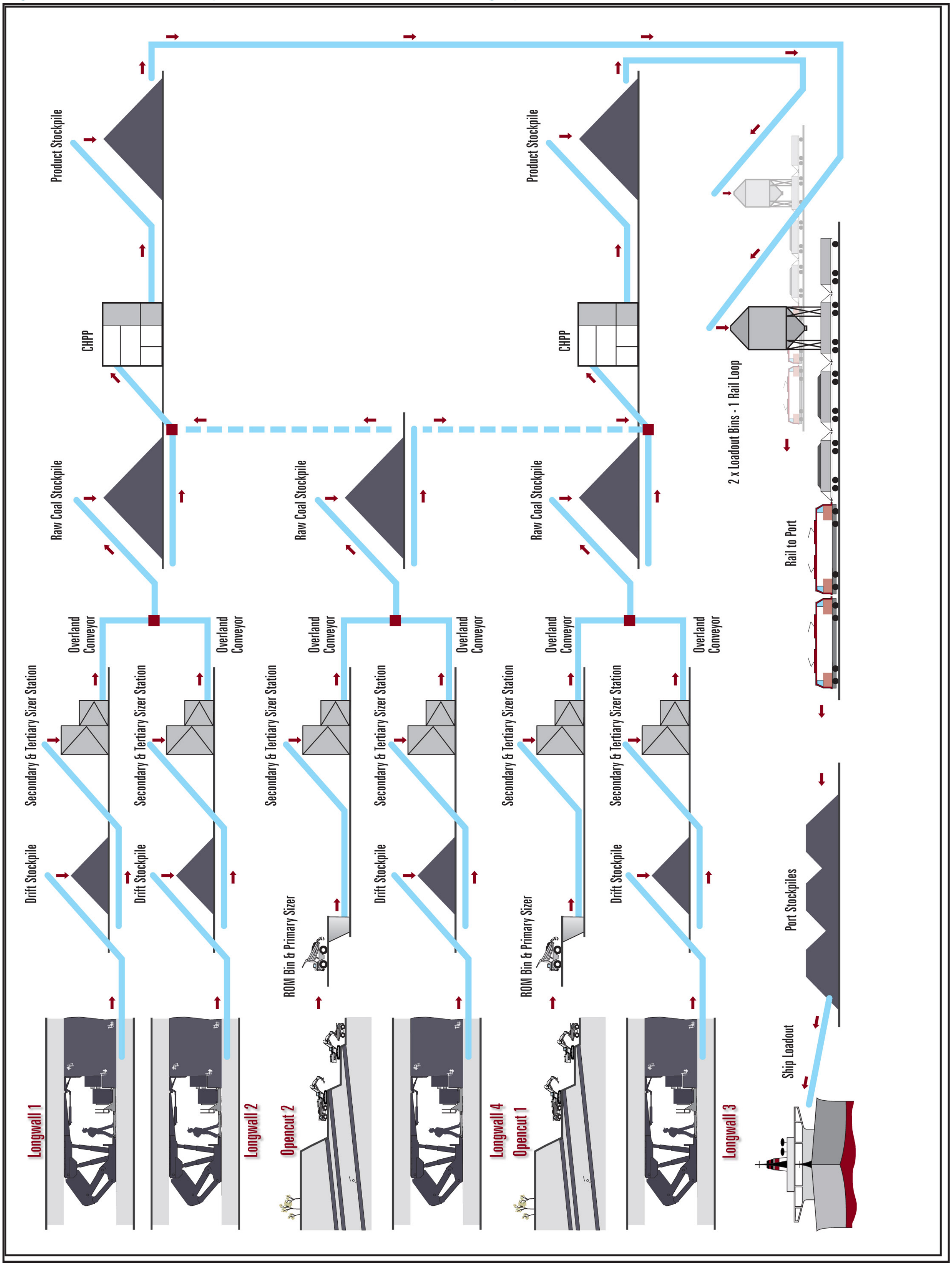


Plate 5. Typical Crusher / Sizer

Plate 6. Typical Trunk or Drift Conveyor

ROM coal conveyors will deliver sized (-50 mm) Raw Coal to one of four overland conveyor (OLC) streams. A separate OLC is dedicated to each of the four coal seams B, C and D and two D. The OLC system from seam D (underground Long Wall 3) will comprise of two separate overland conveyors linked by transfer stations.

The overland conveyors will transfer the raw coal to elevated stockpile tripper conveyors. These rising plant conveyors will discharge onto the Raw Coal Stockpiles via a standard elevated conveyor and tripper arrangement as shown on Figure 35.

The four overland conveyor streams will discharge onto three Raw Coal Stockpiles. The details of these are:

- 400,000 t stockpile – D seam from Long Wall Mines 1 and 2;
- 200,000 t stockpile – B seam from Long Wall Mine 4 and Open Cut Mine 2;
- 400,000 t stockpile – compromising:
  - 200,000 t – seam C and D from Open Cut Mine 1; and
  - 200,000 t – seam D from Long Wall Mine 3.

The 200,000 tonne (t) stockpile will be 140 m long and 35 m high, while the 400,000 t stockpiles will be 280 m long.

The B seam overland conveyor for mines OC2 and LW4 feeds a Raw Coal stockpile of 200,000 t capacity. This conveyor system first elevates the coal to a Transfer Bin fitted with two discharge feeders. Coal is then transferred to the tail end of the main reclaim conveyors feeding each of the CHPP's. This allows the B seam coal to be fed to either Coal Preparation Plant. It will also allow limited blending with the reclaimed coal from either of the D seam and C and D seam stockpiles. The transfer system for B seam coal is not intended to feed both CHPP's in tandem.

The D seam, C, and D raw coal stockpiles each have 400,000 t capacity with the D system dedicated to LW1 and LW2 supply.

Reclamation from the Raw Coal Stockpiles will be via a reclaim tunnel and coal valve arrangement. Two coal valves will be required for the 200,000 t stockpiles and four each for the 400,000 t stockpiles.

A single reclaim conveyor from each of the 400,000 t stockpiles will feed into a single CHPP. Reclaim from the 200,000 t stockpile (B seam) can be diverted to either CHPP via a transfer tower and conveyor discharging onto the head end of either 400,000 t stockpile reclaim conveyor. This provides a simplistic raw coal blending capability.

Each CHPP will have only one feed conveyor, being the feed from one 400,000 t raw coal stockpile. Each CHPP will be fitted with a bunkering system to ensure even coal flow to each of the four operating modules.

1.2.3.3 Product Coal and Train Load Out

Each CHPP will have only one product coal conveyor discharging washed coal to a 400,000 t product coal stockpile. The product stockpiles will be 280 m long and 35 m high. Product coal stacking will again be via conventional elevated gantry conveyor and tripper arrangement.

Product coal reclamation, for each CHPP, will be via bulldozer and coal valve operation discharging coal onto a single reclaim tunnel conveyor. Each product stockpile will be fitted with four reclaim valves. Reclaimed product coal will be conveyed to a train load-out (TLO) bin for loading into trains.

The product coal reclaims and TLO conveyors bins will be rated to 6,000 tph.

1.2.3.4 Rejects

Each CHPP will have a single reject conveyor discharging into a rejects bin. The reject bin will be used to fill mine trucks, which will return the reject coal back to the open cut mine sites for disposal.

The basic design characteristics of the CHPP conveyor are shown at Table 5.

Table 5. CHPP basic design characteristics

DESCRIPTION	BELT SPEED (M/S)	BELT WIDTH (MM)	CAPACITY (TPH)
Overland Conveyors	5.4	1,600	4,500
Raw Coal Conveyors	5.0	1,600	4,000
Product Coal Conveyors Stacking	5.0	1,600	4,000
Reject Coal Conveyors	4.0	1,600	4,000
Product Coal Reclaim Conveyors	6.6	1,600	6,000
Train Load Out Conveyor	6.6	1,600	6,000

1.2.3.5 Coal Handling Preparation Plant

The CHPP facility will operate at a nominal plant feed rate of 8,000 tph as received (ar) to target the required annual plant feed rate of 56 Mtpa ar with a full plant operating hours design allowance of 7,000 hours (h). To maximise modular throughput for the proposed CHPP a desliming screen aperture of two mm chosen and (at this aperture), a capacity of approximately 1,000 tph / module should be achievable for the range of likely feed types to the plant. This modular capacity and the requirement for dual rail load out loops dictated the arrangement for the CHPP facility would be two plants each consisting of four 1,000 tph modules.

A single conveyor will feed each of the two plants and this will require a suitable feed distribution system to be installed to evenly distribute the feed tonnage across the four modules in each plant. The feed will become slurry at this point through addition of water to transport and optimise feed conditions to the desliming screens (Plate 7).

The function of the desliming screen is to remove sub-sized particles (-2.0 mm material) from, and dewater, the dense medium cyclone feed (+2.0 mm material). Screening is achieved by presenting particles to the screen deck surface and moving particles smaller than the aperture through the sieve surface. Vibration of the screen assists this process by stratifying the bed, giving particles more opportunity to present to the screen surface.

Plate 7. Desliming Screen

Plate 8. Dense Medium Cyclone

The CHPP will be based on conventional wet beneficiation processes using proven technology that is used extensively throughout the Australian coal industry. The 2 mm coarse coal fraction will be beneficiated in dense medium cyclones (Plate 8). In this process the 2 mm material from the desliming screens is mixed with a magnetite / water medium and pumped to a single large diameter dense medium cyclone in each module. Dense medium cyclones separate based on density with the high-density non-coal material reporting to coarse rejects and the lower density coal reporting to product after dewatering in coarse coal centrifuges.

The 2.0 mm raw coal slurry from the desliming screens is pumped to classifying cyclones in each module that remove the 0.125 mm material and the bulk of the water from this stream. The <-2 to +0.125> mm fine coal fraction will be beneficiated using spirals in a water based separation. Spirals product is dewatered in fine coal centrifuges (Plate 9) and reports with the dense medium cyclone product to the plant product conveyor. Spirals reject is dewatered on high frequency screens with the coarse spirals reject particles reporting with the dense medium cyclone reject on the plant reject conveyor and the fine spiral reject particles reporting to the tailings thickener.

The 0.125 mm material will be discarded to tailings due to the high operating / capital costs and low marginal value typically associated with coal in this size fraction. The proposed tailings system will be a simple "high-rate" thickener (Plate 10) and tailings dam process. Four 48 m diameter tailings thickeners will be installed as part of the CHPP. Once thickened, the tailings are pumped to the tailing storage facility.

The two proposed tailing systems being reviewed are the traditional co-disposal system and the capital intensive filter press system. Both systems require the sub <0.125 mm particles to be conditioned with flocculants, a process carried out within thickening tanks. The thickening process forms an aqueous tailings slurry allowing tailings to either be transported via a pipe network to a co-disposal or filter press system. Four 48 m diameter tailings thickeners will be installed as part of the Project. The traditional co-disposal system has the tailings slurry being pumped to a sealed specifically created tailings storage containment structure. The tailings are deposited into various cells where excess water is decanted and recycled to the CHPP.

The later filter press method is expensive to setup and utilizes either belt or filter presses to dewater tailings forming a dry paste. The water is recycled to CHPP while the tailings paste is conveyed to the rejects surge bin for disposal in rejects containment structures. Excess water from rejects containment structures is also recycled.

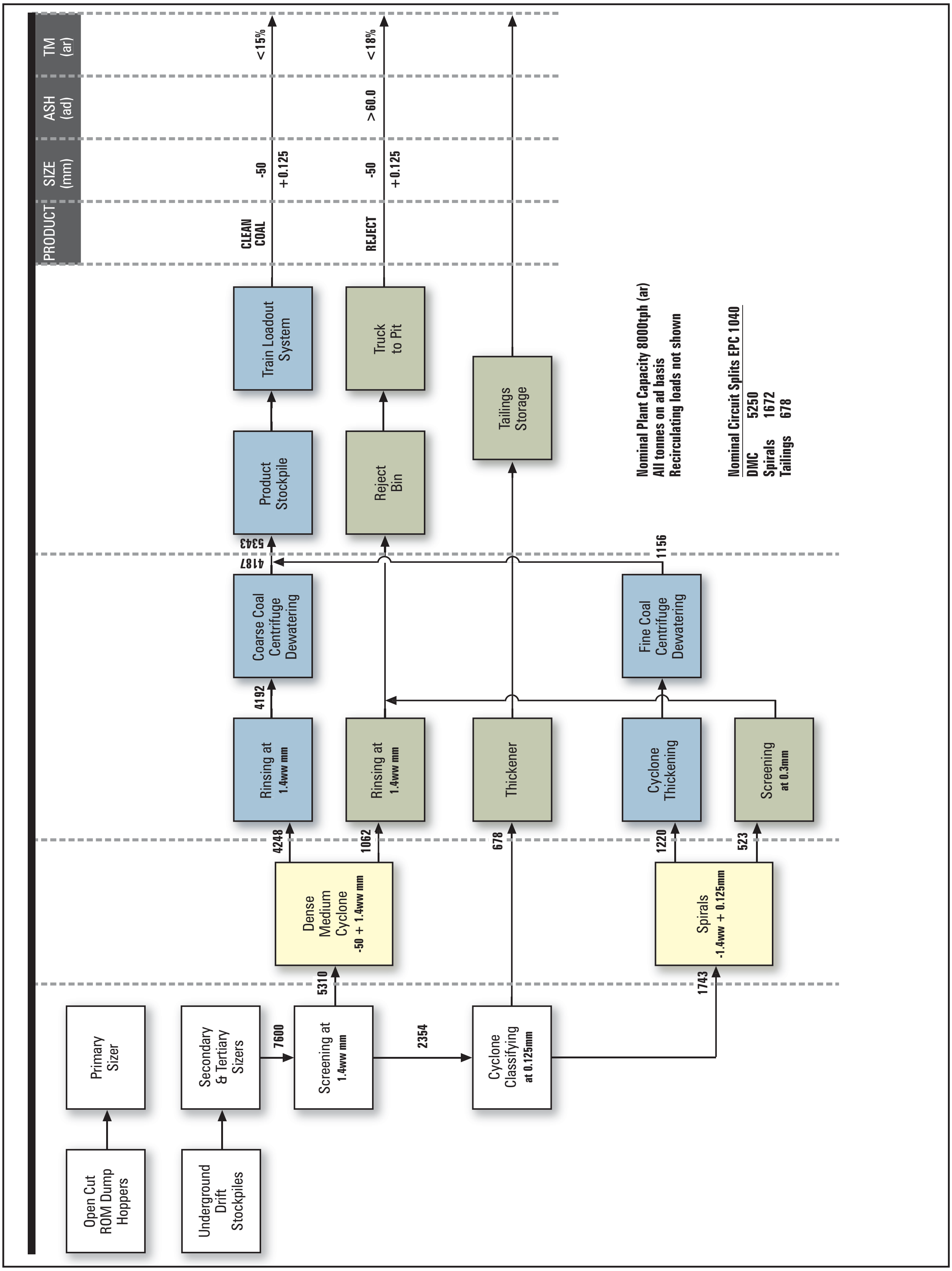
The plant will be controlled from a single computerised control room. The control room is part of a building separated from the CHPP, but adjacent to the CHPP, which also houses all the power supply and motor control panels and PLC hardware.

The nominal CHPP process is shown in Figure 36.

Plate 9. Fine Coal Centrifuge

Plate 10. Tailings Thickener

Figure 36. Block Flow Diagram



1.2.4 SITE WATER MANAGEMENT SYSTEM

1.2.4.1 Water Demands and Sources

The estimated required annual quantity of clean water is 4,550 megalitres per annum (ML/a) of which 2,400 ML/a is needed for the four longwall mines, 2,000 ML/a is required for the CHPP vacuum pumps and potable and fire water usage will be approximately 150 ML/a. Clean water for the mine will be sourced from a proposed dam to be constructed on Tallarenha Creek.

Potable water demand is estimated to range from 1 ML/a to 290 ML/a during mine development and from 100 ML/a to 150 ML/a during operations. Potable water supplies during early construction will come from contracted potable water suppliers carting from an offsite source. Once major construction activities have commenced a package potable water treatment plant will be installed to cater for potable water demands during the remaining construction and operating phases of the mine. This water will be sourced from the Tallarenha Creek Dam.

Raw water will be required for coal washing and dust suppression in the open cut mines. The estimated annual water requirements for these uses are:

- Open cut mine dust suppression: 2,000 ML/a;
- CHPP (coal washing): 11,200 ML/a.

Excess water in the CHPP will be recycled to the Return Water Dam and be available to meet the raw water demands. The quantity of water that can be returned from the CHPP to the Return Water Demand will depend on the method used to dispose of rejects and tailings. Two rejects/tailings disposal options have been identified for the mine:

- Pumping rejects and tailings to disposal cells as a slurry (co-disposal);
- Trucking rejects and filter pressed tailings to disposal cells.

The co-disposal method requires significantly more water and involves higher water losses in the disposal cells. Accordingly, there will be less water returned from the CHPP to the Return Water Dam using the co-disposal method. Preliminary mass flow calculations for the CHPP and disposal cells have identified the following return flows from the CHPP to the Return Water Dam:

- Co-disposal: 9,360 ML/a;
- Trucking rejects and filter pressed tailings: 12,351 ML/a.

The estimated net raw water requirement for the mine (allowing for water returned from the CHPP) will be:

- Co-disposal: 3,840 ML/a;
- Trucking rejects and filter pressed tailings: 849 ML/a.

Preliminary hydrogeological and water balance modelling investigations (AMEC, July 2010) have identified the following raw water sources for the mine (suitable for coal washing and dust suppression):

- Aquifer inflows from open cut pits and underground mines: 4,045 ML/a;
- Rainfall inflows to the open cut pits: 305 ML/a to 863 ML/a depending on stage of mining;
- Catchment inflows to the CHPP environmental control dam: 39 ML/a.

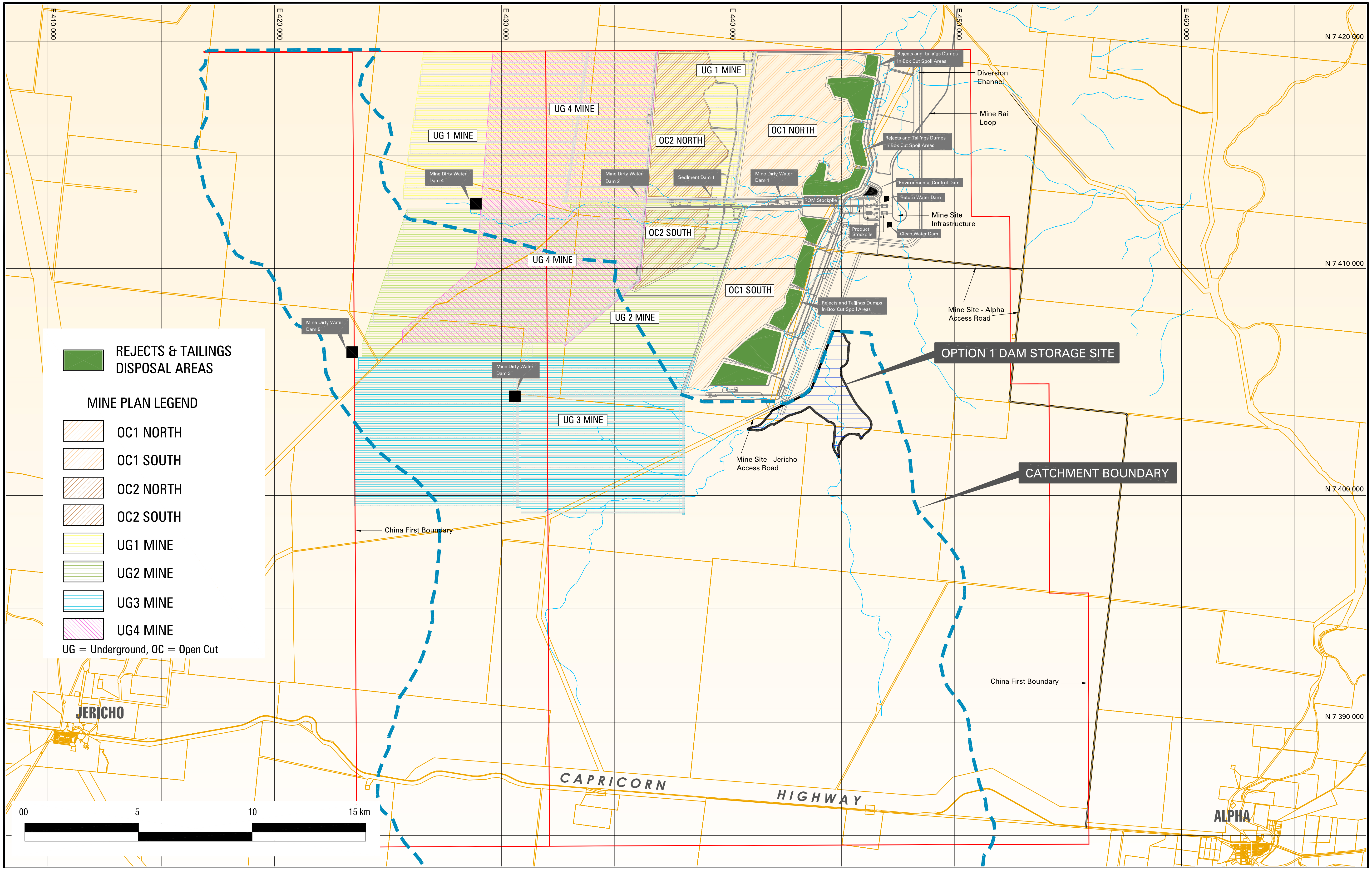
There will be an excess of raw water to meet the operational mine demands.

1.2.4.2 Tallarenha Creek Dam

The clean water supply for the mine (4,550 ML/a) will be sourced from a proposed new dam constructed on Tallarenha Creek (Monklands Dam) at the junction with Beta Creek.

The proposed dam site (see Figure 37), is on Tallarenha Creek at Zone 55, E 444 499 and N 7 404 737 (GDA 94 Datum). The watershed basin is Burdekin, Drainage Division 1. The catchment area is 866 km2 comprising the catchment areas of Beta Creek and Tallarenha Creek. Preliminary investigations (AMEC, November 2010) identified a reservoir storage volume of 18,098 ML corresponding to a full supply level of 345 m AHD and a maximum dam embankment height of 7 m. Tallarenha Creek extends 48 km upstream of the dam site and the Belyando River is 70 km downstream. A detailed engineering investigation is required to determine the suitability and type of impoundment structure required.

Figure 37. Proposed Tallarenha Dam Site Location



A preliminary yield analysis for the storage was undertaken using a computer based water balance model for the historical period 1900 to 2008. This model uses daily 'inputs' and 'outputs' into the storage structure and determines the resultant storage volume, overflows and actual reclaim from the structure for a nominated demand. The actual reclaim is the amount obtained after all other inputs and outputs have been accounted for. The reliability of the supply is therefore a measure of the number of times the required demand is achieved.

Inputs:

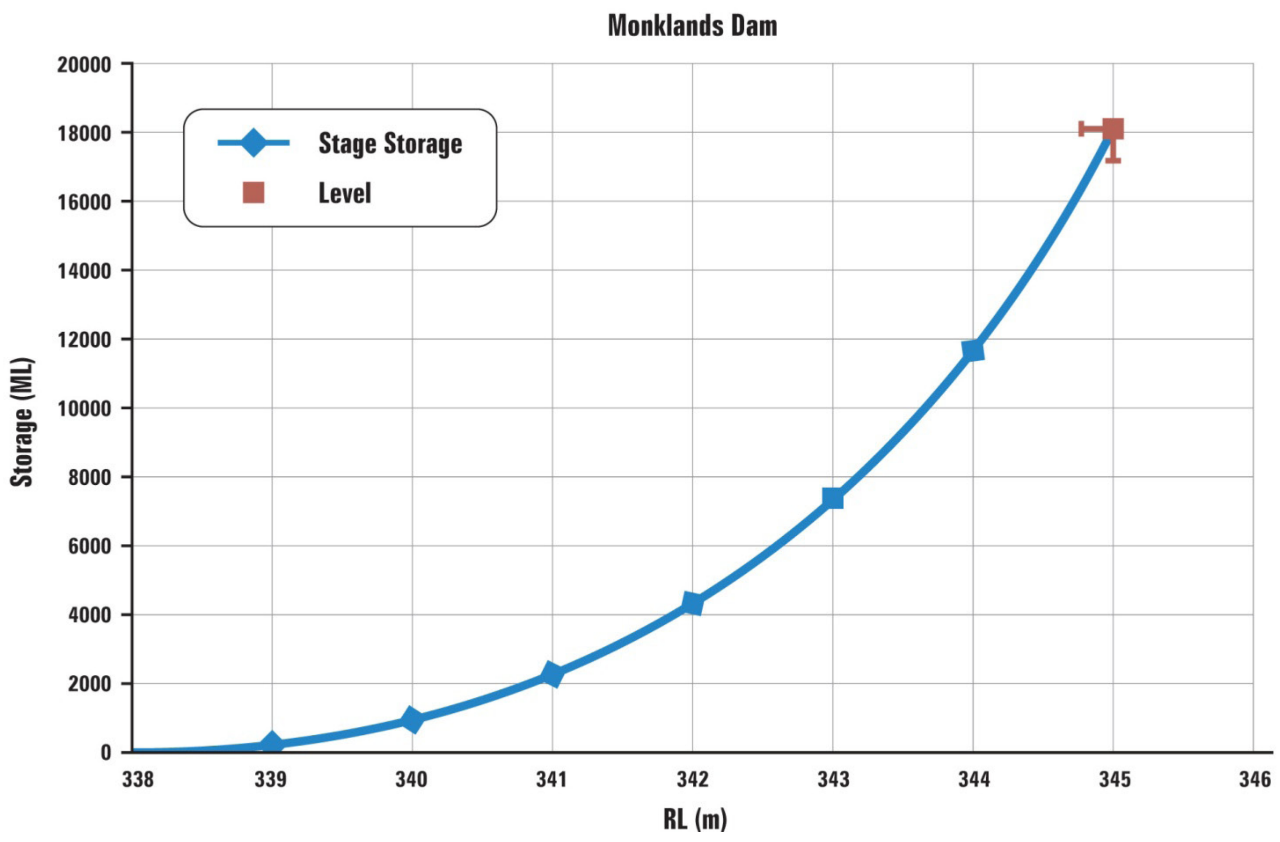
- Daily rainfall falling directly on the storage surface. SILO Data Drill applicable to the site location used.
- Daily runoff from rainfall falling on the catchment that reports to the storage. Determined using AWBM runoff generation model.
- Other daily inflows such as water harvesting – nil.

Outputs:

- Daily evaporation from structure. SILO Data Drill applicable to the site location used.
- Water reclaimed from the structure to meet demand.
- Spillway discharge.
- Seepage losses.

The analyses have been carried out for a range of annual demands ranging from 500 ML to 10,000 ML. A dam site stage storage curve has been generated using available topographic data for the impoundment area. Details of the storage curve used with the water balance model are provided at Figure 38. Full supply volume is 18,098 ML.

Figure 38. Proposed dam site storage curve



The results of the dam yield assessment are shown at Table 7. The preliminary yield assessment indicates that the dam will be able to supply the mine clean water demand of 4,550 ML/a with a reliability of approximately 100 %. If the dam has a lower yield than that identified in the preliminary yield assessment, then additional clean water supplies will be obtained from the following sources:

- Desalination of excess groundwater pumped from the open cut pits and underground mines;
- Proposed SunWater pipeline from Moranbah to Galilee Basin coal mines as part of the Connors River Dam project (if this project proceeds).

The results of the assessment are shown at Table 6.

Under the provisions of the *Water Supply (Safety and Reliability) Act 2008* and *Water Act 2000*, a dam that would, in the event of failure, put a population of two or more people at risk is classified as 'referable'. The population at risk is determined by a dam failure impact assessment which assigns a failure impact rating for the dam as follows:

- Less than 2 people at risk – no failure impact rating.
- 2 to 100 people at risk – Category 1 failure impact rating.
- More than 100 people at risk – Category 2 failure impact rating.

Dams that are given a Category 1 or 2 failure impact rating are classified as 'referable'.

A failure impact assessment will be undertaken for the proposed Tallarenha Creek Dam as part of the engineering investigations and design for the mine. It is likely that the Department of Environment and Resource Management will classify the dam as referable because of the large storage capacity of the dam and the location of the mine industrial area, CHPP, open cut workings, access roads and rail loop in the downstream failure flow path for the dam.

Under the *Sustainable Planning Act 2009* a development permit is required for all new referable dams. The design and operation of the dam will comply with all dam safety conditions imposed by DERM as part of the development permit approving the dam construction, including:

- Submission of a certified Design Plan including Data Book, Design Report and as-constructed documentation;
- Development of Standard Operating Procedures and Operating and Maintenance Manuals;
- Development of an Emergency Action Plan;
- Development of a program for and undertaking dam safety inspections and reviews; and
- Development of a Decommissioning Plan.

Section 76G of the *Fisheries Act 1994* requires that new waterway barriers must adequately provide for fish passage. A development permit is required for the construction of a new waterway barrier under the *Sustainable Planning Act 2009*. A fishway will be incorporated into the proposed Tallarenha Creek Dam to facilitate fish passage. The type and arrangement of fishway will be determined as part of the detailed design of the dam.

Table 6. Water Yield and Reliability Assessment Results – Tallarenha Creek Dam

REQUIRED ANNUAL DEMAND (% AVE, ANNUAL CATCHMENT YIELD)	RELIABILITY	AVERAGE NO. OF DAYS IN A YEAR WITH ZERO YIELD	AVERAGE NO. OF DAYS IN A YEAR WITH YIELD < REQUIRED	RATION AVERAGE SPILL VOLUME/YIELD (AVERAGE ANNUAL SPILL)
	%	DAYS	DAYS	%(ML)
500 (1.1)	99.9	-	-	98 (47,000)
1,000 (2.1)	99.9	-	-	97 (46,800)
2,000 (4.2)	99.9	-	-	96 (46,000)
3,000 (6.3)	99.9	-	-	93 (44,800)
4,000 (8.4)	99.9	-	-	92 (44,200)
5,000 (10.5)	99.9	-	-	89 (43,500)
7,500 (15.8)	99.3	2.4	2.5	84 (41,700)
10,000 (21.0)	97.6	8.6	9.0	81 (40,500)

1.2.4.3 Water Management Flow Sheets

Water balance flow charts indicate that if rejects and filter pressed tailings are trucked to disposal rather than co-disposal pumping, there is annual water saving of 2,991 ML. The flow charts also show that after one year of mining, there is an excess of dirty water excluding evaporation and seepage losses.

Two flow sheets have been prepared for 40 Mtpa of coal production. Figure 39 is a flow chart where coarse rejects and tailings are co-disposed and Figure 40 is a flow chart in which coarse rejects and filter pressed tailings are trucked to dumps.

Evaporation losses have been included for aquifer water reclaimed from open cut pits. Runoff yield volumes are total volumes for 90% probability of exceedance, excluding evaporation and seepage losses.

Total water quantity fed into the CHPP in Figure 39 and Figure 40 is 18,240 ML/a, which includes 5,040 ML in raw coal, 11,200 ML/a from the return water dam and 2,000 ML/a for the vacuum pumps. Product moisture content accounts for 2,880 ML/a. Water is lost in the rejects and tailings disposal processes. Excess CHPP water is recycled to the return water dam.

In comparison, an additional 2,991 ML per year of water is required for co-disposal, compared to trucking coarse rejects and tailings. Further comparison shows that after one year of mining there are 749 ML and 3,740 ML of excess dirty water, excluding evaporation and seepage losses for the co-disposal and filter press options respectively. Excess water (primarily groundwater pumped from open cut pits and underground mines) will be disposed of using evaporation dams or will be desalinated and used to supplement clean water supplies from the Tallarenha Creek Dam.

1.2.4.4 Mine Dewatering

A mine dewatering system will be required to remove water from the open cut and underground workings prior to any mining operations. Sources of water will include groundwater inflows from the coal seam and overlying strata, overland flows and surface water runoff, gas drainage activities.

The dewatering system will consist of compressed air driven pumps that will pump accumulated water from each working face to an electric pod pump connected into a dewatering pipeline. The dewatering pipeline will then typically discharge into a central pumping station where the water will be pumped to the main dewatering dam. The anticipated volume of water able to be recovered through mine dewatering is estimated to be a minimum of 4,550 ML/a.

1.2.4.5 Water Storages

The site water balance model (AMEC, July 2010) indicates that the operations will have a surplus of water. To achieve this surplus, a number of water management dams are required, the location of which are shown at Figure 41.

Water from the Tallarenha Creek Dam will be pumped to the clean water dam which will be located upslope of the return water dam so that reservoir water can gravitate into the return water dam, or be released into creeks through a bywash, during intense rainfall events. The clean water dam will require a high-density polyethylene (HDPE) liner. The return water dam is located next to the CHPP and variable speed pumps will control flow rate in the plant. The environmental control dam is downslope of the CHPP and coal stockpile areas.

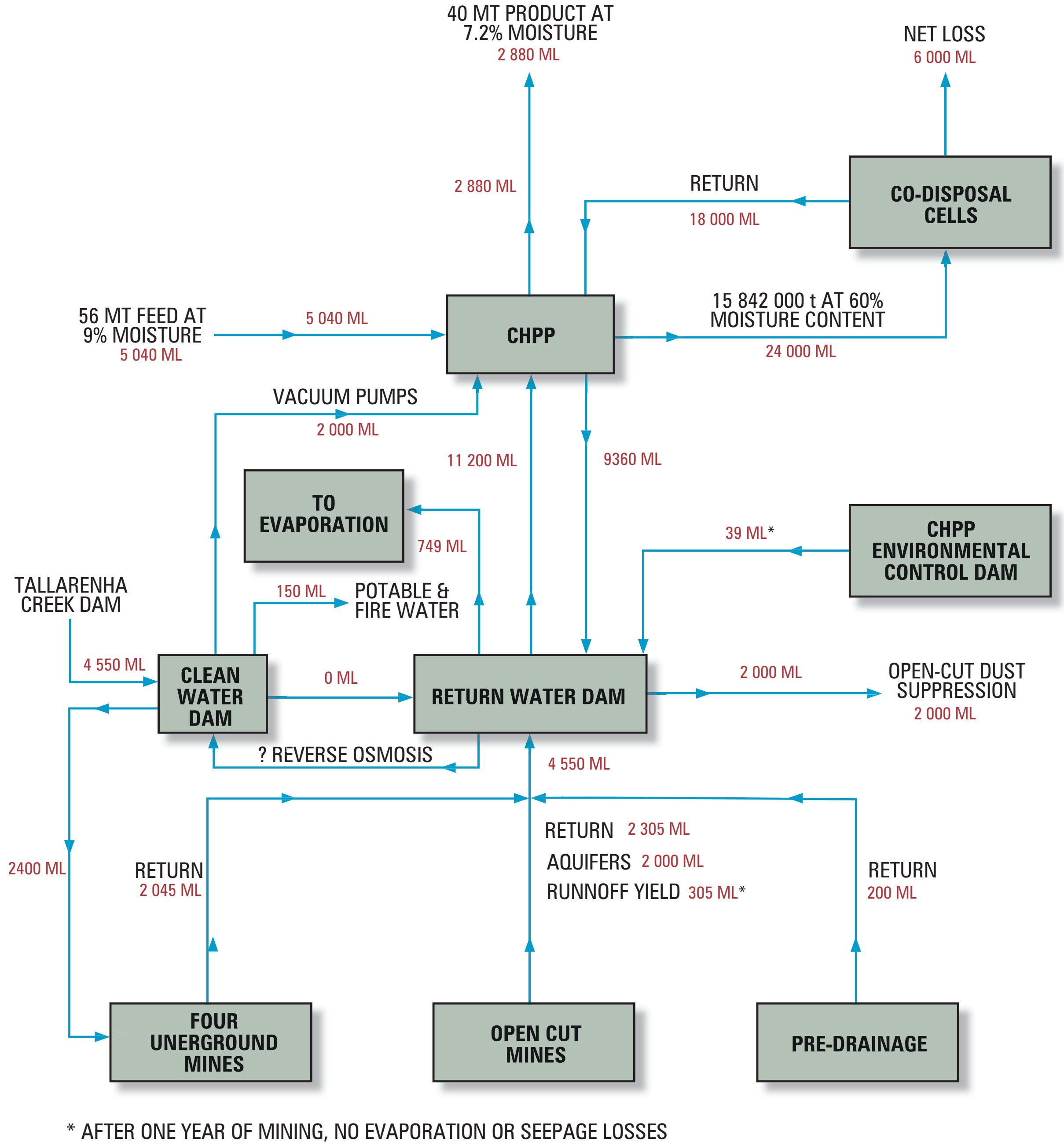
Five mine dirty water sites have been identified and these are shown at Figure 41. Mine water will be pumped from these sites to the return water dam. Additional, temporary dirty water dam sites could be required during mining.

For the OC1 North and OC1 South pits, low wall surface runoff could be initially directed into the rejects and tailings cells prior to transfer to the return water dam. Once the boxcut spoil piles have been topsoiled and rehabilitated, clean runoff water would be directed into the Tallarenha/Lagoon Creek diversion channel away from the CHPP dirty water catchment.

The OC2 North and OC2 South pits require a low wall sediment dam until the boxcut spoil piles have been rehabilitated. The proposed location (as shown in Figure 41) is outside any longwall subsidence area. Additional, temporary low wall sediment dams can be constructed, as required.

A hazard assessment will be undertaken for all dams and levees proposed for the mine in accordance with the DERM Manual for Assessing Hazard Categories and Hydraulic Performance of Dams to determine the likely impacts on downstream waterways and lands in the event of failure of the dams and levees. Dams that are likely to contain contaminated water or solids will be designed with sufficient storage capacity to prevent

Figure 39. Water Management Flow Sheet for Co-Disposal Option



discharges of contaminated water in accordance with the DERM Manual. The design of these dams will ensure that the dams can withstand flow conditions experienced during extreme flood events (both local and regional flooding).

1.2.4.6 Proposed Tallarenha/Lagoon Creek Diversion

Beta Creek and Tallarenha Creek combine at the southern end of the mine site (near south-east corner of OC1 South pit) and discharge into Lagoon Creek which flows in a northerly direction through the main industrial part of the proposed mine area. It will be necessary to divert Tallarenha/Lagoon Creek around the eastern side of the mine industrial area. The proposed diversion channel alignment starts downstream of the Tallarenha Creek Dam spillway and passes around the eastern side of the mine workings, CHPP and rail loop before discharging into Lagoon Creek at the northern mine tenement boundary (refer Figure 37).

The diversion channel will be designed in accordance with relevant design standards and guidelines including:

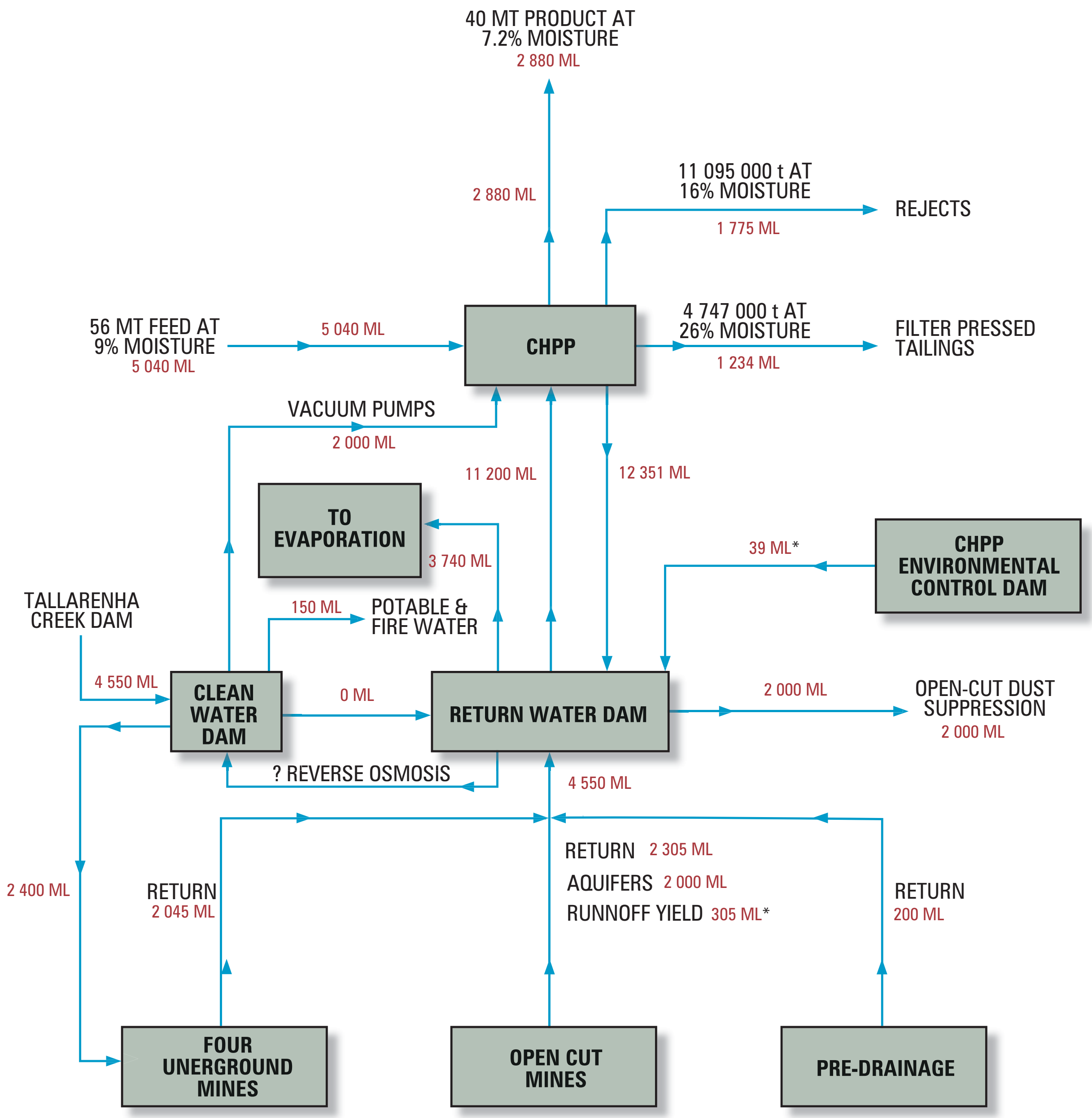
- DERM Manual for Assessing Hazard Categories and Hydraulic Performance of Dams (includes design criteria for levees);

- ACARP Report on Maintenance of Geomorphic Processes in Bowen Basin River Diversions;
- ACARP Report on Monitoring Geomorphic Processes in Bowen Basin River Diversions;

The creek diversion will include a main channel designed to convey the 1 in 100 Annual Exceedance Probability (AEP) catchment discharge. A system of pools and riffles will be constructed into the low flow section of the main diversion channel to provide habitat for aquatic ecosystems and to facilitate fish passage. A levee will be constructed along the western edge of the main diversion channel to protect the mine area (open cut pits, rejects/tailings disposal cells, CHPP, mine industrial area and rail loop) against flooding for flood events larger than the 1 in 100 AEP event.

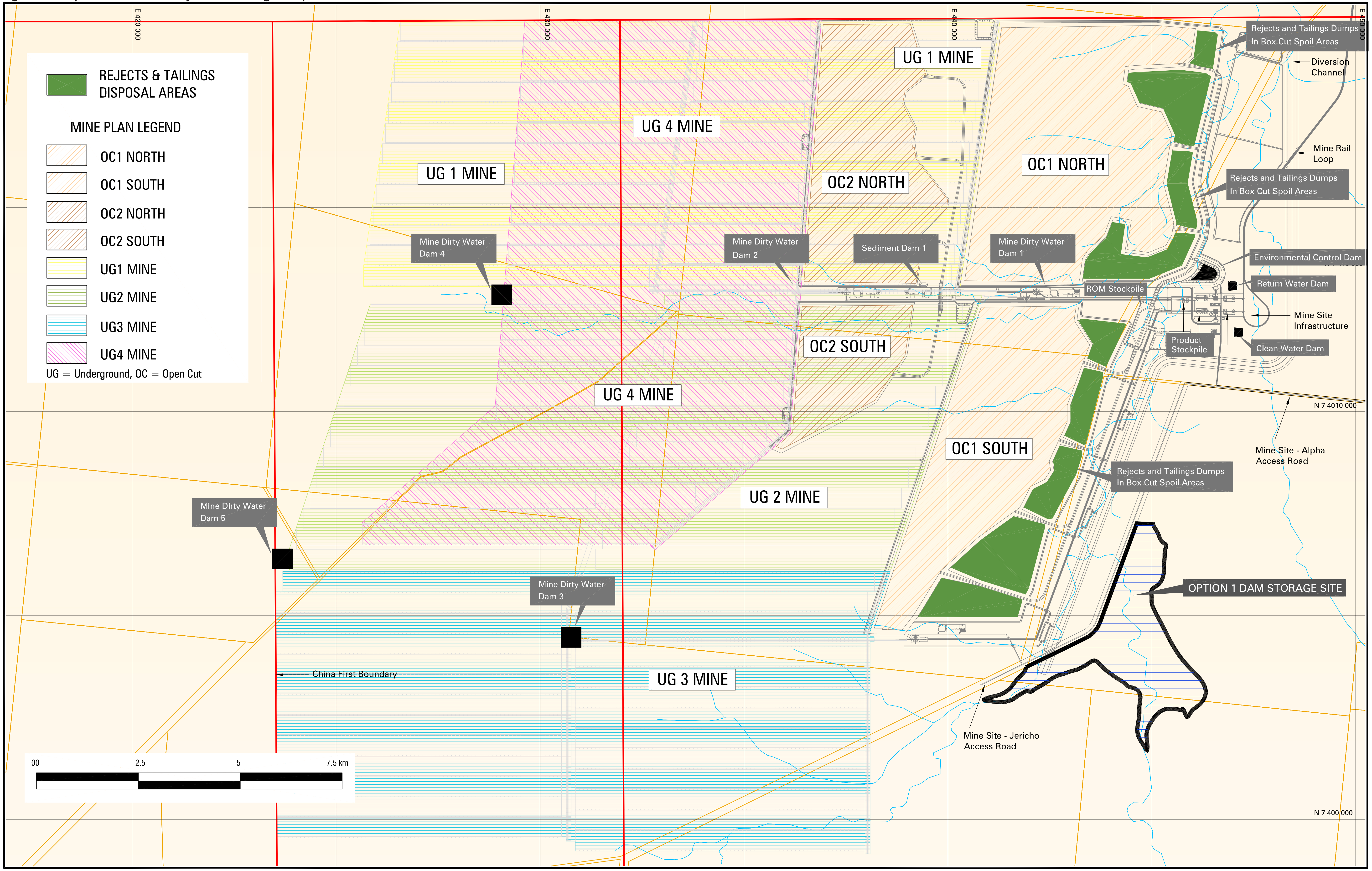
A hazard assessment will be undertaken for all dams and levees proposed for the mine in accordance with the DERM *Manual for Assessing Hazard Categories and Hydraulic Performance of Dams* to determine the likely impacts on downstream waterways and lands in the event of failure of the dams and levees. It is envisaged that the levee will be designed to protect the mine from flood events up to a 1 in 50,000 AEP event in accordance with the DERM Manual.

Figure 40. Water Management Flow Sheet for Filter Press Option



* AFTER ONE YEAR OF MINING, NO EVAPORATION OR SEEPAGE LOSSES

Figure 41. Proposed Locations of Rejects and Tailings Dumps



1.2.5 REJECTS AND TAILINGS DISPOSAL

1.2.5.1 Disposal Alternatives

Two disposal methods are described in this study. The preferred option is to truck rejects and filter pressed tailings to disposal cells. Filter pressing of tailings is a new technique in coal wash plants that is now successfully operating in Australia. Prior to implementation of this method, thorough testing will be undertaken to ensure that effective pressing of tailings occurs, particularly for coal from the open cut mines.

The alternative method is co-disposal of rejects and tailings, using gravel pumps and steel pipework.

1.2.5.2 Trucking Rejects and Filter Pressed Tailings

Coarse rejects from the underflow of the dense medium cyclone will be discharged onto a reject conveyor, as are fine rejects, which are the overflow from the fine coal reject dewatering screen. Coarse and fine rejects will then be conveyed to the reject bin for truck disposal.

The -2 + 0.125 mm fine coal fraction will be beneficiated using spirals with desliming cyclone overflow being pumped to the tailings thickener where flocculent will be added. The thickened tailings are then passed through a filter press where the moisture content is reduced to 26%. The pressed tailings are then discharged onto the rejects conveyor for disposal via the reject bin.

1.2.5.3 Co-disposal of Rejects and Tailings

Co-disposal involves pumping rejects and tailings to cells, using gravel pumps and steel pipework. For co-disposal of rejects and tailings, the total annual quantity of solids is approximately 15,842,000 t, which requires a moisture content of 60% for pumping. Water quantity needed is 24,000 ML of which 75% or 18,000 ML will be recycled. The net annual water loss from this process is estimated to be 6,000 ML.

Rejects and tailings dumps initially will be positioned in close proximity to the CHPP. These will be located in the boxcut spoil areas to allow the co-disposal pipework to be rotated every three months in the case of steel lined pipework or every 12 months if it is basalt lined. This process is to prevent invert abrasion failures.

1.2.5.4 Comparative Assessment of Disposal Methods

For an annual production of 40 Mtpa of washed coal, total rejects and tailings quantity are estimated to be 15,842,000 t. By constructing co-disposal cells, within the boxcut spoil piles using Tertiary Clay and weathered Permian spoil to seal them, final rehabilitation is facilitated. In addition, the floors of the cells comprise impervious, residual clay that prevents water seepage into the environment and downdip to the final voids.

Trucking rejects and filter pressed tailings is the preferred disposal method as these materials can be hauled as back loads to disposal areas using coal haulage trucks. Prior to implementing filter pressing, extensive testing will be undertaken to ensure that excessive quantities of reactive clays are not present. Such clays adversely affect moisture removal.

Co-disposal is labour intensive involving regular rotation of steel pipework, movement of discharge points and installation of decant water pipework. An additional 3,000 ML per year of water is required, compared to trucking rejects and filter pressed tailings.

Rehabilitation is the same for both disposal methods and involves capping with benign spoil, topsoiling and seeding.

1.2.5.5 Chemical Properties

The tailings are expected to have a low capacity to be potentially acid forming. No oxidisable pyrite has been detected in any logged coal samples. Sulphur content in coal samples ranges from 0.4 to 0.7 %, indicating low sulphur content for tailings.

The salinity of tailings is expected to be low. Interseam aquifers have total dissolved salts concentrations ranging from 260 to 1,750 parts per million (ppm). Surface salinity contents of exposed tailings surfaces can increase by oxidisation, capillary action and surface evaporation. Such surfaces will be progressively capped with benign spoil prior to topsoiling.

No deleterious metal concentrations have been detected in any tested coal samples.

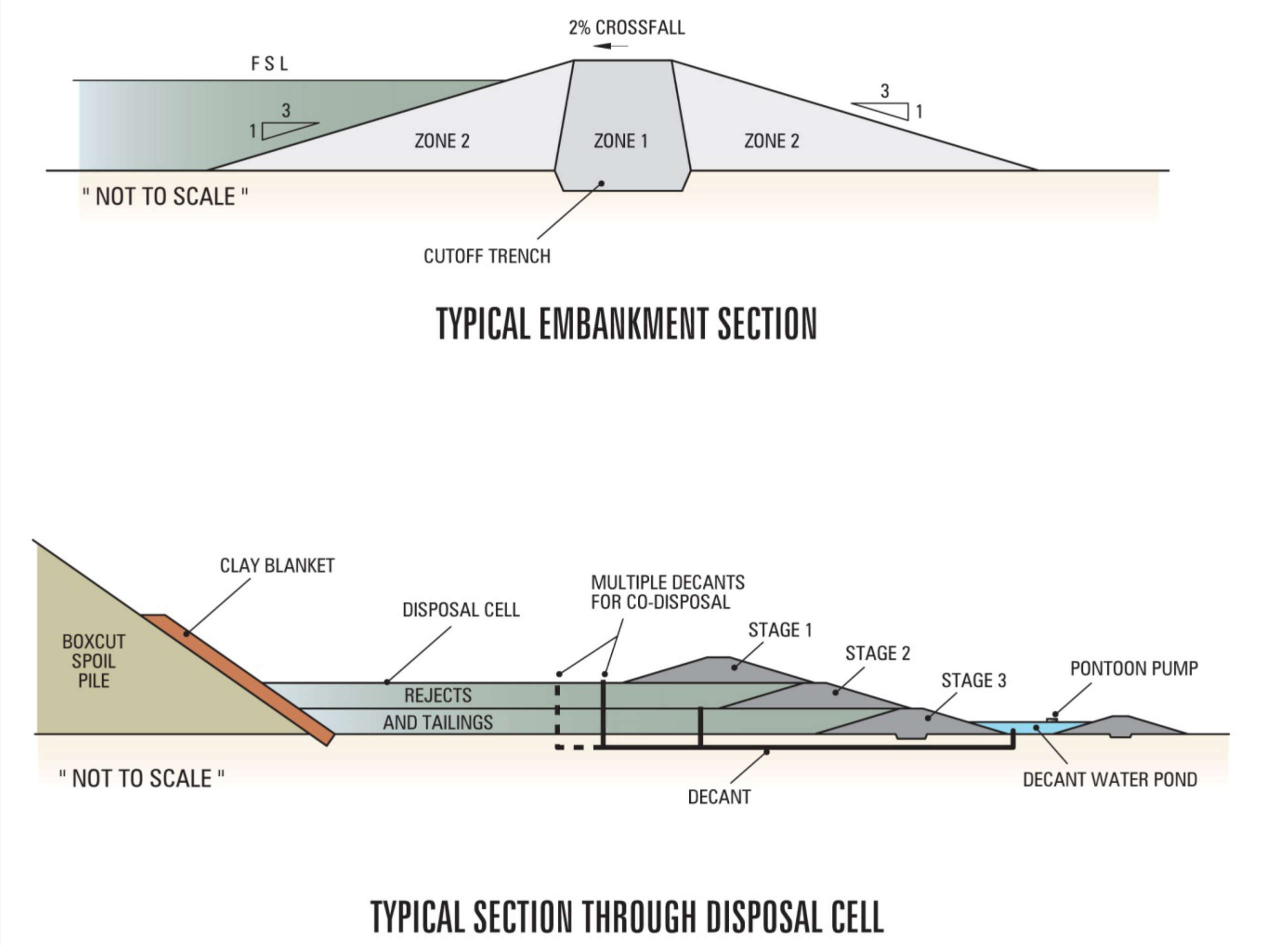
1.2.5.6 Design of Rejects and Tailings Cells

It is proposed to construct cells next to boxcut spoil areas using clayey boxcut spoil as embankment material. The proposed locations of the disposal cells are shown at Figure 42. Boxcut excavations and construction of cells would be completed as a truck and shovel operation. Topsoil removed from the boxcut spoil piles and disposal cells foundations will be stockpiled east of the tailings cells for future rehabilitation use.

The foundation material of the disposal cells generally comprises 15 m to 25 m of Tertiary Clay overlying 15 m to 20 m of weathered Permian strata, both of which are effectively impervious. Downward seepage of decant water is not possible in such materials.

The embankments for the disposal cells and decant water ponds will be constructed to Australian water dam standards. Figure 42 shows a typical embankment section with upstream and downstream batter angles of 1.0 (vertical) to 3.0 (horizontal). Dam height is 7.5 m and crest width is 5.0 m with a 2 % crossfall to the reservoir. The cutoff trench is excavated down to impervious clay. The embankment is zoned with a central core and upstream and downstream shell. Minimum required dry density ratio is 98 % standard compaction at optimum moisture content plus 2 % for cohesive soils and 70 % density index for cohesionless soils. The maximum dry density shall be determined in accordance with Test No. 5.1.1. (Standard Compaction) of AS 1289 for cohesive material and in accordance with Test No. 5.5.1 and 5.6.1 of AS 1289 for cohesionless materials.

Figure 42. General Arrangement for Rejects and Tailings Disposal



Good quality non-dispersive, impervious material termed Zone 1 Clay is required for the dam core. The Zone 1 Clay Core and cut-off trench backfill shall be well-graded sandy / silty clay with a liquid limit (LL) ranging from 30 % to 60 % and plasticity index (PI) ranging from 15 % to 45 %.

The Zone 2 Select Fill material used in the upstream and downstream shell has similar material requirements as for Zone 1 material except that the material classification may be gravelly / sandy / silty clay. Weathered rock may be used for Zone 2 Select Fill if it meets the following criteria. In general the select material shall be in accordance with the following requirements, which are a liquid limit ranging from 25 % to 60 % and a plasticity index of 10 % to 45 %.

1.2.5.7 Disposal Procedures

Rejects and tailings will be deposited in cells constructed between the boxcut spoil piles and dam embankments constructed to the east. The embankments will be raised in stages and clay blankets will be constructed against the boxcut spoil to prevent seepage through spoil piles. Decant structures and decant water ponds will be constructed to remove water from the disposal cells. Decant water will be pumped back to CHPP return water dam from the decant ponds.

The disposal cells and decant water ponds will be classified as Regulated Dams and will be designed with sufficient storage capacity to prevent discharges of contaminated water in accordance with the DERM *Manual for Assessing Hazard Categories and Hydraulic Performance of Dams.*

Haul trucks which offload coal at the ROM stockpile will be backloaded at the reject bin to transport rejects and tailings to disposal cells. Dumped material would be dozed and track compacted in layers, with gradients to the decant structures. The decant pipework will direct water to decant water ponds, where pontoon pumps recycle contaminated water to the CHPP return water dam. Decants will be raised as the disposal cells are infilled. Upstream raising of the cell embankments will be undertaken in stages in order to provide effective sealing of the disposal cells.

Water levels within the decant ponds will be undertaken as a controlled operation, supported with a backup monitoring systems. Water levels will be kept at minimal levels at the beginning of the wet season and during the wet season to prevent any overflow.

Bypass pipework to in-pit emergency storage will be considered as part of the final design of the return water management system.

Final surfaces in disposal cells will be graded and capped with benign spoil, prior to topsoiling and seeding.

1.2.5.8 Environmental Monitoring

It is proposed to install piezometers downstream of the decant water ponds embankments, to below the dry season groundwater levels. Regular monitoring will be completed to ensure that no groundwater contamination is occurring from the decant water ponds and disposal cells.

All embankment structures will be regularly inspected to ensure structural integrity and watertightness of embankment foundation material. Embankment batters will be topsoiled and seeded, to minimise erosion.

1.2.6 SUPPORTING INFRASTRUCTURE

1.2.6.1 275 kV Power Supply

During the initial phase of construction, portable diesel generators and existing single wire earth return (SWER) lines will be used to supply energy. When available, energy will be supplied to the mine site via a new 275 kilovolt (kV) line being developed by Powerlink. Powerlink is proposing to acquire a suitable site for a substation north of the proposed mine (to be known as Surbiton Hill Substation). An easement is also required for a proposed 275kV transmission line that will run between the Surbiton Hill Substation and Powerlink's existing Lilyvale Substation near Emerald. The transmission line will be approximately 200 km in length. The new line development will incorporate a 275 kV feed into a sub-station to the north of the mine, whereby the power supply will be reduced and reticulated throughout the mine site at various voltages including 66 kV, 22 kV and 11 kV. A Power Allocation (Power Enquiry) has been made to Powerlink by both Waratah Coal and AMCI (proponents of the South Galilee Coal Project located directly to the south of the Galilee Coal Project) seeking confirmation of a regulated or unregulated supply to both mines.

During the Project development, the annual energy consumption is estimated to be up to 20 – 100 Megawatts (MW)/year. This is expected to increase to 150 MW/year during operations. Waratah Coal will develop energy conservation strategies for the construction and operation of the mine. The strategies will be developed to minimise energy consumption throughout the duration of the project.

1.2.6.2 Telecommunications

Waratah Coal proposes to establish a fibre optic cable linking the mine, rail and the facilities at Abbot Point. Communications at the mine will be a combination of fibre optic and connection into the local telecommunication network.

1.3 MINE DECOMMISSIONING AND REHABILITATION

This section describes the broad strategies and methods for progressive and final rehabilitation of areas disturbed by mining and associated infrastructure activities, expected final landforms and the proposed final land uses. The section also describes the decommissioning plan and preferred rehabilitation strategy for the mine and the MIA.

Whilst general information regarding rehabilitation and decommissioning is provided in this section, specific rehabilitation and decommissioning measures to avoid or minimise any impacts will be identified in the Environmental Authority, the Environmental Management Plan (EMP) and the Mine Closure Plan.

It may be the case that the best beneficial use of some of the supporting infrastructure components (i.e. water supply infrastructure, roads, power transmission lines) would be to leave the infrastructure in place to support other local needs. This will be discussed with the relevant authorities and landholders prior to formalizing the decommissioning strategy. If the preferred plan is to leave some of the infrastructure components *in-situ as operating infrastructure, Waratah Coal that facilitates the transfer of operating licences and obligations to the relevant parties will prepare a transitional outcome.*

1.3.1 OBJECTIVES

The overriding mine closure objective is to successfully implement an economically feasible closure that incorporates community priorities, environmental aspects, sustainable rehabilitation and ongoing land uses.

Rehabilitation and decommissioning strategies will be prepared and implemented to ensure that the final landform is:

- returned in a safe manner, with public safety risks reduced to acceptable levels;
- stable and resistant to erosive processes;
- suitable for the post-mining land uses agreed with relevant government agencies;
- within the limits of appropriate and agreed levels of contamination;
- in a condition which satisfies community, agency and landowners expectations;
- in a condition that meets the agreed discharge licence conditions;
- where required, managed under a site specific Site Management Plan (SMP) in place; and
- in compliance with all EMP commitments.

In addition to the EMP, a mine clousure plan MCP will be prepared that establishes the specific operational activities required to be undertaken in order to complete rehabilitation and decommissioning of the Project.

1.3.2 DECOMMISSIONING

The following decommissioning strategies are proposed for various remaining structures post-mine closure.

All infrastructure will be removed unless agreed with the subsequent post-mining landowner. This includes:

- a contaminated land assessment of relevant locations;
- remediating land from any contamination;
- removal of all items of the mine infrastructure area, and any temporary buildings and facilities;
- ripping, topsoiling, and seeding of this land; and
- establishing safety bunds and fencing of final void areas.

1.3.2.1 Decommissioning Action Plans

The following action plans (based on the above strategies) will be undertaken.

1.3.2.1.1 Mine Industrial Area, Conveyors and Accommodation Facilities

All items of the infrastructure area and including conveyors and any temporary buildings and facilities will either be removed from site or, if agreed by the landholder, left operational on site. After all external structures, concrete bases and footings have been removed; these areas will be investigated for contamination and remediated where necessary, ripped, profiled, topsoiled and seeded. Protection of these areas from re-compaction (i.e. vehicles or grazing animals) after ripping is required to allow the soil structure to reform. Drainage control through ripping, profiling or the provision of erosion control structures will also be undertaken.

1.3.2.1.2 Mine Water Storages

The mine water storages will be removed including removal of dam embankments and contaminated sediments within the dam storage area.

The decommissioning strategy for the Tallarenha Creek Dam will be determined in consultation with relevant authorities and landholders. Potential decommissioning strategies include:

- Full decommissioning – removal of dam embankment and associated pumping facilities.
- Partial decommissioning – retention of a smaller dam structure as a water supply for landholders or other third parties.
- No decommissioning – sale or donation of the dam to landholders or other third parties to be used as a water supply.

1.3.2.1.3 Mine Water Supply Pipelines

The decommissioning strategy for the water supply pipeline will be either:

- abandonment – where the pipeline is purged, and physically disconnected from the point of supply, and sealed (capped) at both ends; or
- beneficial re-use – where sale or donation of the infrastructure to a third party occurs for other beneficial use.

Before deciding if abandonment (after capping) or beneficial re-use is the preferred option, Waratah Coal will liaise with relevant authorities and landholders in order to determine the most appropriate desired outcome. Once the relevant authorities agree the desired outcome, a decommissioning plan that takes into account the desired outcome will be prepared.

1.3.2.1.4 Power Supply and Transmission Lines

The power supply will be dismantled and removed off site unless a beneficial re-use can be identified. The transmission lines and poles may be retained for future use by local government.

1.3.2.1.5 Waste Management Facility

Any landfills established as part of the mine operations will be decommissioned at the conclusion of mining, and a contaminated land assessment (which will include mitigation measures) consistent with the requirements of the *Queensland Environmental Protection Act 1994* (EP Act) will be undertaken on the landfill site.

1.3.3 REHABILITATION

Waratah Coal supports the 'Enduring Value – the Australian Minerals Industry Framework for Sustainable Development' principles and desired outcomes. Waratah Coal has incorporated the intent of these principles, and in particular, Element 6.3 'Rehabilitate land disturbed or occupied by operations in accordance with appropriate post-mining land uses' in the preparation of its post mining rehabilitation strategies.

The following sections provide the general rehabilitation goals, objectives and strategies of the Project rehabilitation strategy, and have been developed with consideration given to DERM's *Guideline 18 Rehabilitation requirement for mining projects* (EPA,2007) (Guideline 18) and *Leading practice sustainable development program for the mining industry: Mine Rehabilitation* (Commonwealth Department of Industry, Tourism and Resources,2006).

1.3.3.1 Rehabilitation Hierarchy

The Department of Environment and Resource Management (DERM) has established a rehabilitation hierarchy to minimize environmental harm. The rehabilitation hierarchy, in order of decreasing capacity, is to:

- avoid disturbance that will require rehabilitation;
- reinstate a 'natural' ecosystem as similar as possible to the original ecosystem (where the Project is occurring on previously natural vegetated land);
- develop an alternative outcome with a higher economic value than the previous land use;
- reinstate the previous land use (e.g. grazing or cropping); and
- develop lower value land use.

1.3.3.2 Rehabilitation Goals

The four general rehabilitation goals of Guideline 18 are rehabilitation of areas disturbed by mining to result in sites that are:

- safe to humans and wildlife;
- non-polluting;
- stable; and
- able to sustain an agreed post mining land use.

Waratah Coal's desired outcome of the rehabilitation strategy is to ensure that post mine land use outcomes meet regulatory and other stakeholder expectations.

1.3.3.3 Rehabilitation Objectives

The objectives for rehabilitation throughout the construction, operational and decommissioning phases of the Project are to:

- return the land to a post-mine land use that will be stable, self-sustaining and require minimal maintenance;
- create stable landforms with rates of soil erosion not exceeding the pre-mine conditions; and
- maintain downstream water quality, during the construction, operational and post operation phases of the Project.

1.3.4 REHABILITATION INDICATORS

To ensure that the objectives of mine closure, decommissioning and rehabilitation (both progressive and final) are achieved, Waratah Coal will establish criteria and performance indicators which, once achieved, demonstrate that decommissioning and rehabilitation strategies have been undertaken successfully and that desired outcomes have been achieved.

The EMP will establish in detail, performance indicators to demonstrate the successful completion of the closure process, and provide timeframes within which completion is to be achieved. Indicative performance indicators are included in Table 8.

Successful mine closure, decommissioning and rehabilitation will be considered completed when conditions within the Project area meet the pre-determined performance indicators to the satisfaction of regulatory authorities and tenement relinquishment is obtained.

1.3.5 COMPLETION CRITERIA

The ultimate aim of the defined objectives is to create sustainable landforms that require no more resources to maintain than a similar landuse in an area that has not been mined.

Rehabilitation success is defined as the achievement of objectives set out in Section 1.3.3.3, and performance indicators shown in Table 7. A completion criterion is used to define the successful rehabilitation, and relate specifically to the environmental, social and economic context of the Project site.

Completion criteria will be developed in consultation with landowners, indigenous groups, community groups and Government agencies closer to the time of mine closure and presented in a Final Rehabilitation Strategy. The completion criteria will be based on field trials and monitoring program findings, industry research and the standards of the day, which will be at least equitable to current completion standards.

1.3.5.1 Rehabilitation Action Plans

Final land uses proposed for each mine component have been based on a land suitability assessment in accordance with the *Technical Guidelines for Environmental Management of Exploration and Mining in Queensland* (DME, 1995).

Progressive rehabilitation of worked areas will be undertaken within two years of becoming available or as soon as practicable thereafter. Rehabilitation strategies will take into consideration physical and biophysical attributes such as the geology, groundwater and surface water hydrology and ecology of the site. Action plans will be prepared that support desired end land-use strategies to guide the rehabilitation activities.

An investigation into the rehabilitation of disturbed areas will be undertaken and a report will be submitted to the administering authority proposing acceptance criteria for landform design and final land use. The timing of the report will be agreed with the administering authority.

Table 7. Draft performance indicators for the decommissioning and rehabilitation program

MINE COMPONENT	ASPECT	
Mine voids	Landform	Benches and faces stable, minimal evidence of erosion, revegetation successful.
	Safety	Access controlled via fencing and protective barriers.
	Surface water quality	Water quality in local waterways not to be adversely affected by mining activities (if discharge evident from final voids). Monitoring program implemented.
	Groundwater quality	Local groundwater quality not to be adversely affected. Monitoring program established.
Overburden and waste rock dumps	Landform	Landform stable, minimal evidence of active erosion.
	Safety	Access controlled via fencing and protective barriers.
	Revegetation	Dumps successfully revegetated in accordance with agreed criteria and supported with ongoing monitoring and maintenance program.
Co-disposal Infrastructure	Landform	Landform stable, minimal evidence of erosion, revegetation successful.
	Safety	Access controlled via fencing and protective barriers.
	Surface water quality	Water quality in local waterways not to be adversely affected by mining activities (if discharge evident from final voids). Monitoring program implemented.
	Groundwater quality	Local groundwater quality not to be adversely affected. Monitoring program established.
	Revegetation	Dumps successfully revegetated in accordance with agreed criteria and supported with ongoing monitoring and maintenance program.
Mine Industrial Area	Removal	All mine related infrastructure dismantled and removed from the Project site.
	Revegetation	MIA successfully revegetated according to agreed criteria and supported with ongoing monitoring and maintenance program.
Water storage dams	Landform	Landform stable, minimal evidence of erosion, revegetation successful.
	Safety	Access controlled via fencing and protective barriers.
	Surface water quality	Water quality in local waterways not to be adversely affected by mining activities (if discharge evident from final voids). Monitoring program implemented.
Haul roads and access tracks	Landform	Landform stable, minimal evidence of erosion, revegetation successful and sediment control devices in place and monitored as per license conditions.
	Revegetation	Successful revegetated according to agreed criteria and supported with ongoing monitoring and maintenance program.

1.3.5.1.1 Final Voids

A single final void will remain after completion of mining for each pit. The banks of the final void (i.e. the high wall, low wall and end walls) will be reshaped to achieve long term geotechnical stability. Ramps will be levelled to similar grades as the surrounding wall slopes.

The final slope gradients of each void, including the outer boxcut spoil slopes, low wall of the final voids, and high wall slopes will be assessed and recommended by a suitably qualified person based on the risk of long term geotechnical instability.

The voids will be externally drained so that water from the overburden piles drains away from the voids. Final void modelling will be conducted to establish the required parameters for long term void stability and water quality. A Final Void Plan will be prepared prior to completion of mining in the first pit, based on the final void modelling and detailing the design parameters for each final void. The Final Void Plan will include assessment of groundwater hydrology and properties, surface water hydrology and pit wall stability.

These studies will be undertaken during the life of the mine, and will include detailed research and modelling. In the final five years of mine life, the capability of the void to support endemic flora and fauna will be ascertained.

Final voids are unlikely to be suitable for agricultural use, and will be investigated for alternative beneficial uses such as wetlands.

At the end of the mine life, the final voids remaining will be bunded and fenced to inhibit access to the area. The integrity of the bund will be the responsibility of the subsequent landowner.

Waratah Coal will conduct an investigation into residual voids and a report will be submitted to the administering authority proposing acceptance criteria for final voids. The timing will be agreed with the administering authority.

1.3.5.1.2 Mine Infrastructure Areas

Following decommissioning, infrastructure areas will be returned to the pre-mining landform, where practicable. Where this is not practicable, bench cuts will be removed, any steep grades reduced and the landform returned to a profile similar to that of landforms in the region.

Land used for infrastructure components will be returned to improved pasture grazing land or dry land cropping land, and will generally be able to be used for beef cattle grazing or potentially for fodder cropping if the water pipeline is left commissioned.

Building end use will be assessed at the time of closure, as alternative uses may be available.

1.3.5.1.3 Overburden Stockpiles

The following measures apply to both the in-pit overburden placed by dragline, and elevated out of pit overburden stockpiles.

Overburden stockpiles will be progressively rehabilitated over the life for the mine, and rehabilitation will commence within two years of the land becoming available for rehabilitation. Progressive rehabilitation will function to reduce erosion potential and improve the water quality runoff from overburden stockpiles. Runoff from overburden stockpiles will pass through sediment dams in the Water Management System.

Overburden stockpiles will be reshaped to stable landforms in accordance with agreed end outcomes. The stockpiles will be designed to reduce the catchment area and drainage ways through the overburden.

Low gradient sections of overburden stockpiles will be rehabilitated to grazing land, and generally be able to be used for low stock rates of beef cattle grazing, or alternatively for nature conservation in areas supporting agreed offset and / or connectivity outcomes.

Steeper gradient overburden stockpiles, and overburden stockpiles that trials show are unsustainable for cattle grazing, will be used for nature conservation outcomes.

1.3.5.1.4 Creek Diversions and Levee Banks

Creek diversions will be retained following mine closure, as they will have been designed to provide stable landforms and by time of mine closure, would be established with riparian vegetation and aquatic habitat. At the conclusion of mining, the creek diversions will be left in a stable and sustainable condition in line with the creek diversion rehabilitation plan. The levee banks of all constructed diversions will be maintained and the landforms merged in with overburden stockpiles.

Post-mining, the creek diversions will be retained in a nature conservation land use.

1.3.5.1.5 Water Storage Dams

Water storage dams will either be retained for the subsequent agricultural use or rehabilitated.

The rehabilitation process will entail dewatering, removal of any embankments, revegetation and monitoring. Rehabilitation will also vary depending on the storage history. Dams that have contained saline water may require remediation. The membrane liner of the dam and any saline material inside the dam will be removed during rehabilitation and will be disposed of by appropriate methods in accordance with the accepted management of saline overburden material.

If not retained as water storages, water storage dams will be rehabilitated to improved pasture grazing land and will generally be able to be used for beef cattle grazing.

1.3.5.1.6 Tailings Dam

Opportunities for coal recovery from tailings (reprocessing of the tailings to extract additional coal) will be investigated during the life of the mine. If recovery is not viable, the tailings dam will be rehabilitated.

Tailing dam rehabilitation will be undertaken after drying of the dam. The tailings surface will be covered and capped with benign overburden material to prevent further rainwater ingress into the tailings, and will be topsoiled and vegetated with native species.

The cover will be designed to provide a relatively flat low gradient final landform. The rehabilitated tailings dam will be vegetated with deep rooted grass species or alternate native vegetation and will be placed on the DERM Environmental Management Register (EMR). Preference will be given to using endemic flora during rehabilitation programs.

The post-mining land use of tailings dam areas is proposed to be beef cattle grazing, or for conservation purposes (i.e. habitat connectivity).

If coal recovery is undertaken, following the coal recovery, the tailings dams will be filled and then closed, capped and rehabilitated.

1.3.5.1.7 Haul Roads and Access Tracks

A number of the haul roads may be retained for use by future landowners post mine closure and rehabilitation. A number of additional haul roads will also be temporarily retained following rehabilitation as access roads for rehabilitation monitoring purposes. This will be determined in consultation with stakeholders and local council.

The majority of haul roads and access tracks across the Project area will be highly compacted. As such, rehabilitation will require a combination of deep ripping, profiling, topsoiling and seeding activities. Drainage construction will be applied where necessary.

Land used for roads that are not required by future landowners will be rehabilitated to improved pasture grazing land and will generally be able to be used for beef cattle grazing.

For those roads to be left operational, either permanently or temporarily, containment measures to minimize potential erosion and sediment entering into waterways will be installed.

1.3.5.2 Implementation of Rehabilitation Strategy

1.3.5.2.1 Program

A Plan of Operations will be developed for the mine to guide implementation of progressive rehabilitation.

The Plan of Operations will include a schedule of rehabilitation activities that are proposed within the life of the Plan of Operations. Based on the approved mine plan, detail will be provided regarding the types and areas of land that will be disturbed within the Project area for the term of the Plan of Operations, along with proposed rehabilitation activities.

1.3.5.2.2 Rehabilitation Monitoring

Monitoring and assessment of progressive rehabilitation processes will be undertaken throughout the planning, construction, operational and decommissioning phases of the Project. If monitoring and assessment results indicate that the rehabilitation objectives may not be achieved, then the rehabilitation strategy will be modified.

Non-compliance with the established objectives will trigger a review of processes such as planning and design, and / or repair and maintenance of failed rehabilitation work.

As rehabilitation technologies, strategies and monitoring techniques change and / or are improved over time, Waratah Coal will regularly review and update the Project's rehabilitation and monitoring procedures to include the most effective processes and strategies.

1.3.5.2.3 Rehabilitation maintenance

Two types of rehabilitation maintenance will be performed in rehabilitated areas:

- progressive maintenance (on a planned basis); and
- failure mitigation maintenance (conducted as ongoing required).

Progressive maintenance is planned as part of rehabilitation scheduling. It will comprise repairs that are necessary following the initial construction and adjustment of planning processes if needed.

Following initial rehabilitation, new processes such as erosion, soil formation, vegetation cover and infiltration rates will develop on the modified landform. These processes may be sustainable in the long term, or more likely they may represent an intermediate stage before final landforms / ecosystems are achieved.

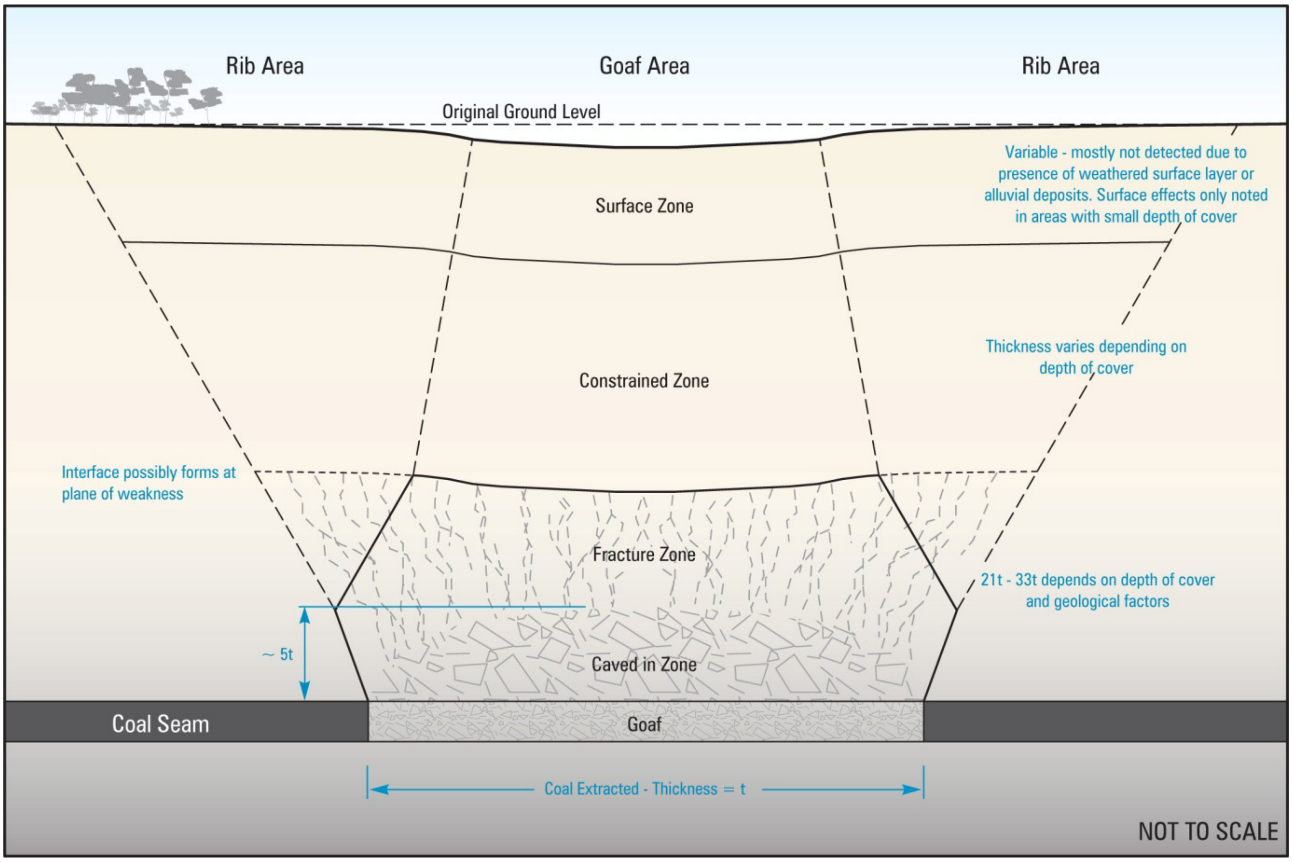
Progressive maintenance activities will be scheduled to transfer intermediate landforms into permanent, long term stable landforms. The type of construction maintenance activities that will achieve this outcome will include removal of graded banks, and repair of areas where excessive erosion has removed the protective capping and exposed spoil.

Rehabilitation failure mitigation will be carried out where the established landforms are not achieving the rehabilitation objectives. The aim of the monitoring and maintenance program will be to identify any systematic issues that may result in broad scale failure of rehabilitated areas. Failure in this sense is defined as non-achievement of the rehabilitation objectives as outlined above.

1.3.6 SUBSIDENCE MANAGEMENT AND REHABILITATION

The underground longwall mining activities will result in surface subsidence. A schematic drawing of the ground effects above the extracted blocks of coal in a longwall mining system is shown in Figure 43.

Figure 43. Schematic of Potential Ground Impacts Associated With Underground Mining



As coal seams are removed by the longwall mining method, a void remains, which is the thickness of the longwall seam, and covers the entire mining block area. Ground immediately above (called the "roof strata") collapses into this void. The overlying strata (or "overburden") then sags down onto the collapsed material, resulting in an elongated subsidence "bowl" developing on the surface.

The act of this strata failure into the void is integral to the success of the longwall mining method, as it relieves the stress that is being loaded onto surrounding mining blocks and development roadways.

The cavity, which remains behind the retreating longwall face and is subsequently filled with the collapsed overlying strata, is commonly called the "goaf" or "gob".

Above this goaf area the strata fails in a generally similar manner to that shown in Figure 44, with progressively less effects as the fracturing moves further above the coal seam.

The extent of the overlying strata collapse and the associated shearing and cracking of the strata depends upon the strata geology, the longwall block width, the seam height extracted, and the depth of cover.

The strata immediately above the longwall goaf collapses into the open void, and hence moves down by a height equal to the thickness of the seam, which was extracted. Due to the way the broken strata material "bulks" or "swells" as it breaks into the cavity, the cavity is eventually filled with broken material (shown as "caved zone" on the diagram above) and a physical cavity no longer exists. However, the vertical displacement in the strata continues to propagate upwards in the strata. Cracking and strata damage do not continue to move vertically beyond the "fractured zone", even though the ground strata all the way to the surface may be displaced vertically.

When the ground stratum moves downwards sufficiently that the vertical movement reaches the surface, the surface of the land may also move downwards over the extracted mining areas. This movement is called "subsidence".

The amount of subsidence witnesses at the surface is dependent on a large range of factors such as:

- thickness of coal seam extracted (mining height);
- depth of cover;
- properties and rock types of ground strata (i.e. overburden strength);
- stiffness and bulking characteristics of the collapsed strata;
- width and length of longwall block;
- dimensions of the gate road coal pillars; and
- the maximum subsidence usually occurs in the middle of the extracted longwall panel.

For the case of single seam mining, the maximum subsidence is expected to be 60 % of the mining height. This is a general average for longwall coal mines in the NSW and Qld coalfields of Australia.

Super-critical Mining Geometries

The combination of the physical properties of the mining situation, particularly panel width and depth of cover, determines whether a single longwall panel will be sub-critical, critical or supercritical. In the Australian coalfields, sub-critical or (spanning) behaviour generally occurs when the panel width (W) is <0.6 times the cover depth (H). If massive strata exist, then sub-critical spanning behaviour can occur for panel W/H ratios up to 1:4. The maximum subsidence for this scenario is usually significantly < 60 % of the extraction height and could range between 10 % and 50 %.

Beyond the sub-critical range, the overburden is unable to span and fails or sags down onto the collapsed or caved roof strata immediately above the extracted seam (i.e. the panel is critical or super-critical).

Critical panels refer to panels with widths where maximum possible subsidence starts to develop, and supercritical panels refer to panels with widths that cause complete collapse of the overburden.

In the case of super-critical panels, maximum panel subsidence does not usually continue to increase significantly with increasing panel width. A panel is considered supercritical when the ratio of panel width to depth of cover is greater than 1:2. The longwall associated with the project will primarily exhibit super-critical behaviour due to the panel widths being greater than the depth of cover for all blocks.

The surface subsidence 'bowl' extends outside the limits of extraction for a certain distance (i.e. the angle of draw). It is usually assumed equal to half the depth of cover in the Queensland coalfields.

Table 8. Longwall block details for each underground mine

UNDERGROUND MINE	NUMBER OF LONGWALL BLOCKS	TOTAL EXTRACTED PANEL WIDTH	PANEL LENGTH RANGE	DEPTH OF COVER RANGE	EXTRACTED THICKNESS RANGE
No. 1	26	480 m	7,000 m	150 – 330 m	1.8 – 4.2 m
No. 2	26	480 m	7,000 m	130 – 350 m	1.8 – 3.8 m
No. 3	26	480 m	7,000 m	100 – 300 m	1.8 – 2.8 m
No. 4	25	480 m	7,000 m	80 – 210 m	1.8 – 3.4 m

Subsidence Surface Impacts

The number of longwall blocks and the key dimension and parameters for each underground mine are shown in Table 8.

Subsidence Estimates

Surface subsidence will develop progressively within each longwall block and will present on the landform surface as a series of trough like depressions. An assumption has been made about the amount of subsidence that will occur on the land surface in comparison to the thickness of the coal seam removed underground. For the purposes of this study, this ratio has been set to 60 %. Assumed vertical movement of the surface will be 60 % thickness of the coal seam removed from underground.

The greatest (maximum) total subsidence will occur in the surface areas which are affected by the operations in both the B-seam and D-seam operations. Based on these assumptions, the maximum depth of subsidence impact from the mining operations will be in the areas where mining in the B-seam and D-seam overlap, and in the centre region of the longwall blocks in these area. This area occurs in the north western section of the underground mine foot print. The total cumulative subsidence in this area is predicted to reach a maximum depth of 3.27 m. Average subsidence across the bulk of the underground mine areas is expected to range between 1.3 m to 1.61 m.

It has been assumed that the coal pillars, which remain in the development gateroad areas, will undergo significant failure once goaf has formed on both sides of the gateroads. It is assumed that these pillars will go into a yield condition and that the floor and roof strata around the pillars will fail. Due to these factors, it has been assumed that the pillars will be compressed to 30 % of their pre-mining seam height.

As discussed previously, it is usual for the surface subsidence 'bowl' to extend outside the limits of extraction by a distance equal to half the depth of cover. This assumption has been utilised in the subsidence predictions for the underground mines. This assumption equates to an angle of draw of 26.5 degrees.

The area where subsidence will likely occur has little topographical relief, and consists of both cleared (chain pulled and blade ploughed) and remnant open woodland, both of which are currently used for cattle grazing. The area where maximum subsidence will occur consists of cleared, improved pasture, to the north-west of the study area.

Potential impacts resulting from subsidence in a rural location would usually result in a change of drainage patterns due to a depression in the ground which may have an effect on the existing hydraulics of surface waters near the mine. Surface waters located above the underground mine include unnamed tributaries of Tallarenha Creek that currently drain eastwards. Subsidence can also cause increased cracking in clays. The generally sandy soils identified over the underground mining are considered unlikely to be significantly impacted by any minor subsidence however the maximum predicted level of 3.27 m has the potential to result in some cracking.

Subsidence will potentially affect surface drainage and groundwater quality and carrying capacity in these areas. Each of these potentially affected aspects is discussed in detail below.

1.3.6.1 Surface Drainage

The creation of surface depressions associated with subsidence can affect surface drainage through the modification to the local drainage patterns. Monitoring of impacts associated with alterations to the drainage regime will be conducted on a regular basis and where necessary rectification works will be undertaken to mitigate affected areas. A range of techniques can be implemented to re-establish drainage patterns and these include the ripping, ploughing and reseeding of surface cracks and earthworks to redirect drainage and address erosion.

Progressive earthworks to re-establish drainage within the subsidence area will be undertaken and will typically involve cut-fill earthworks to address depression and ponding issues, and the excavation of drainage channels. Drainage channels will have sufficient capacity to cater for incoming catchment flows and will be connected to existing drains. There may be a requirement to harden drainage channels to cater for greater than predicted flows and the need for these earthworks will implement the outcomes of the regular subsidence trough monitoring.

Materials excavated will be stockpiled, this will ensure the separation of topsoil from the lower strata soils and stored outside of drainage lines. Where appropriate, use of excavated materials will address issues associated with subsidence and ponding.

Flood modeling undertaken at the mine site has concluded that the subsidence will have minimal impact to the upstream and downstream processes. As such, the low velocity flows are not likely to initiate significant erosion on subsided areas that maintain a vegetation cover. A detailed flood assessment is located at Volume 2, Appendix 17.

1.3.6.2 Groundwater

The groundwater assessment concluded that given the predicted level of subsidence, cracking of the overlying geology is likely to occur. This cracking may result in rapid infiltration of rainfall into the aquifers surrounding the mine, potentially leading to increased rates of flow into the goafs requiring increased dewatering

1.3.6.3 Land Use

Current land uses within the area that may potentially be affected by subsidence are cattle grazing and nature conservation. With the implementation of mitigation measures to address possible drainage issues, and with the ongoing presence of a stable vegetation cover, there is unlikely to be any significant impacts that prevent the continuance of the current grazing regime. The impacts to the natural values are discussed below.

1.3.6.4 Natural Values

Whilst the predicted levels of subsidence can be quantified, the impacts of those changes on natural features such as stream flow, groundwater regime, water discoloration, habitat alteration and vegetation die-back are less easily quantified. These changes can lead to alteration of species habitats and the ecological function of communities. Species and ecological communities dependent upon aquatic and semi-aquatic habitats are particularly susceptible to the impacts of subsidence. Effects can be temporary or long-term.

Given the lesser level of subsidence above the open woodland areas (i.e. expected to range between 1.3 m to 1.61 m as opposed to 3.27 m above the north-west corner of the study site) and sandy nature of the soils in this area there is not expected to be any substantial cracking. The surface above the underground mining area will not be cleared of vegetation, but it is acknowledged that there may be long-term impacts to the surface vegetation communities due to changes in hydrology and subsidence because of the underground operations.

A Subsidence Management Plan will be prepared prior to the commencement of underground mining operations. The plan will be risk based, flexible, responsive and capable of dealing with unexpected changes or uncertainties. The plan will consider and include if necessary the mitigation measures outlined above to re-establish drainage patterns and included the ripping, ploughing and reseeding of surface cracks and earthworks to redirect drainage and address erosion. In addition, Waratah Coal will provide compensation for unavoidable impacts of subsidence within the Bimblebox Nature Refuge.

1.4 MINE WORKFORCE

A construction workforce of approximately 2,500 contractors will be required at peak construction period. The workforce will be predominantly fly-in / fly-out (FIFO); however, expectation is there will be a portion of local workers in this project. Accommodation will be provided at a purpose built 2,000 person workers village adjacent to the site. The mine development is expected to operate on a two shift, seven day rotating roster.

A proposed workforce of 2,360 permanent employees / contractors will be required during the mine operations. This will comprise 2000 workers at the mine site of which 1978 will be FIFO, and 28 will be housed in Alpha.

The remaining 360 workers will be required for the rail (275) and the port operations (185).

As per the construction phase, the mine workforce is to be housed in the workers village and it is expected that external contractors will from time to time stay at the workers village whilst on site. The operational workforce will likely be structured on a two shift, seven day rotating roster.

Transportation of construction and operational workers between the accommodation village and the mine site will be by bus.

At this stage it is not possible to identify the likely workforce number for the decommissioning and rehabilitation phases, and these numbers are unknown at present, therefore final decisions will be made at the end of the Project around which infrastructure will remain commissioned.

1.4.1 WORKFORCE ACCOMMODATION

The majority of the workforce for the construction and operational phases will be FIFO. To cater for the estimated workforce levels during both phases, a temporary 2,000 person workers village will be established at the mine site (Figure 44). The workers village at the mine site is considered able to accommodate the rail line construction workers also; however, this will depend on the level of available accommodation.

The workers accommodation village will require potable and non-potable water supplies. Water for the workers accommodation village will be derived from a water treatment plant located at the mine site.

The Tallarenha Creek Dam will supply 4,550 ML of raw water reporting to a clean water dam located near the Mine Infrastructure Area (MIA) and the CHPP (refer Figure 41). A water treatment plant located at the MIA will process 150 ML of water from the clean water dam. Potable water produced from the water treatment plant will be piped to the workers accommodation village storage header tanks ready for consumption.

Raw water will be required at the workers accommodation village for uses such as dust suppression and toilet flushing. Raw water will be supplied via a pipeline connecting the clean water dam at the MIA to the raw water header storage tanks at the workers accommodation village. The raw water header storage facility will be of sufficient size and height to satisfy the village consumption requirements.

Power to the site will be sourced from the Powerlink grid system. Power will be supplied to the workers village from the mine site substation that will be located near the mine infrastructure area or the CHPP. The contractor will be required to obtain all required approvals relevant to the power supply.

Package sewage treatment plants (STP) suitable for 2,000 equivalent persons will be used at the workers village. Effluent from the STP will be fed to the dedicated STP waste disposal area. The dedicated waste disposal area will be determined in greater detail during the detailed design phase, but will consist of irrigated pastures (or similar vegetation) and will be located at sufficient distance from the camp to provide buffer from odour, and waterways to ensure adequate buffering of instream values. The irrigation areas will be of sufficient size that the treated effluent can be applied a suitable rate to prevent runoff into local waterways. No storage is of treated effluent is proposed other than the storage tank associated with the sewage treatment plant.

In order to minimise the amount of waste taken to landfill, a dedicated waste management area will be constructed to enable the separation of wastes in accordance with the adopted waste hierarchy. Where possible waste will be re-used on site; however, a registered waste disposal company will be engaged to remove waste to appropriate off-site treatment facilities.

The management of storm water will be considered as part of the design of the workers village. The design and intent of the storm water management system will be to avoid ponding and flooding from overland flows. Where storm water capture is able to be included in the design, storm water discharge points will be engineered to avoid affecting the natural flow system.

The actual footprint of the workers village and associated infrastructure is still being considered. Prior to finalizing the location of the accommodation village, Waratah Coal will liaise with the appropriate local authorities and landowner/s as well as take a range of operational, environmental and community factors into consideration. Preference will be given to locating the workers village on disturbed land; however, other factors that will be considered include:

- the proximity to the rail easement to minimise travel distances;
- minimizing the amount of vegetation clearance required;
- avoiding locations that are flood and bushfire prone;
- minimise impacts to local communities; and
- proximity to existing infrastructure (i.e. power and water supplies and waste treatment facilities).

Figure 44. Likely Mine Site Workers Camp Configuration