Appendix A

Revised mine plan - technical and financial considerations

Cobbora Holding Company Pty Limited

Revised Mine Plan
Technical and Financial Considerations

OWNERSHIP AND PERMITTED USE

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1. INTRODUCTION

Recommendation 4 of the Planning Assessment Commission (PAC) Review report requests that the Cobbora Coal Project mine plan be refined with the goal of:

- Reducing the impacts on threatened species and endangered ecological communities, particularly by: relocating the B-OOP E overburden dump and tailings emplacement areas, and avoiding or minimising intrusion of mining into the main remnant vegetation corridor (on the north eastern portion of the site)
- Minimising dust, particularly by reducing the land area that would be exposed at each stage of mining
- Maximising the land capability and productivity of the rehabilitated final landform
- Minimising the extent of any final void.

Within the PAC Review report, the PAC also notes that for similar reasons to the above, that efforts should be made to limit the northern extent of mining area A and the northern portion of mining area B.

This report describes CHC's revised mine plan and addresses the issues raised by the PAC Review report which reviewed with the mine plan associated with the Preferred Project Report and Response to Submissions (PPR&RTS).

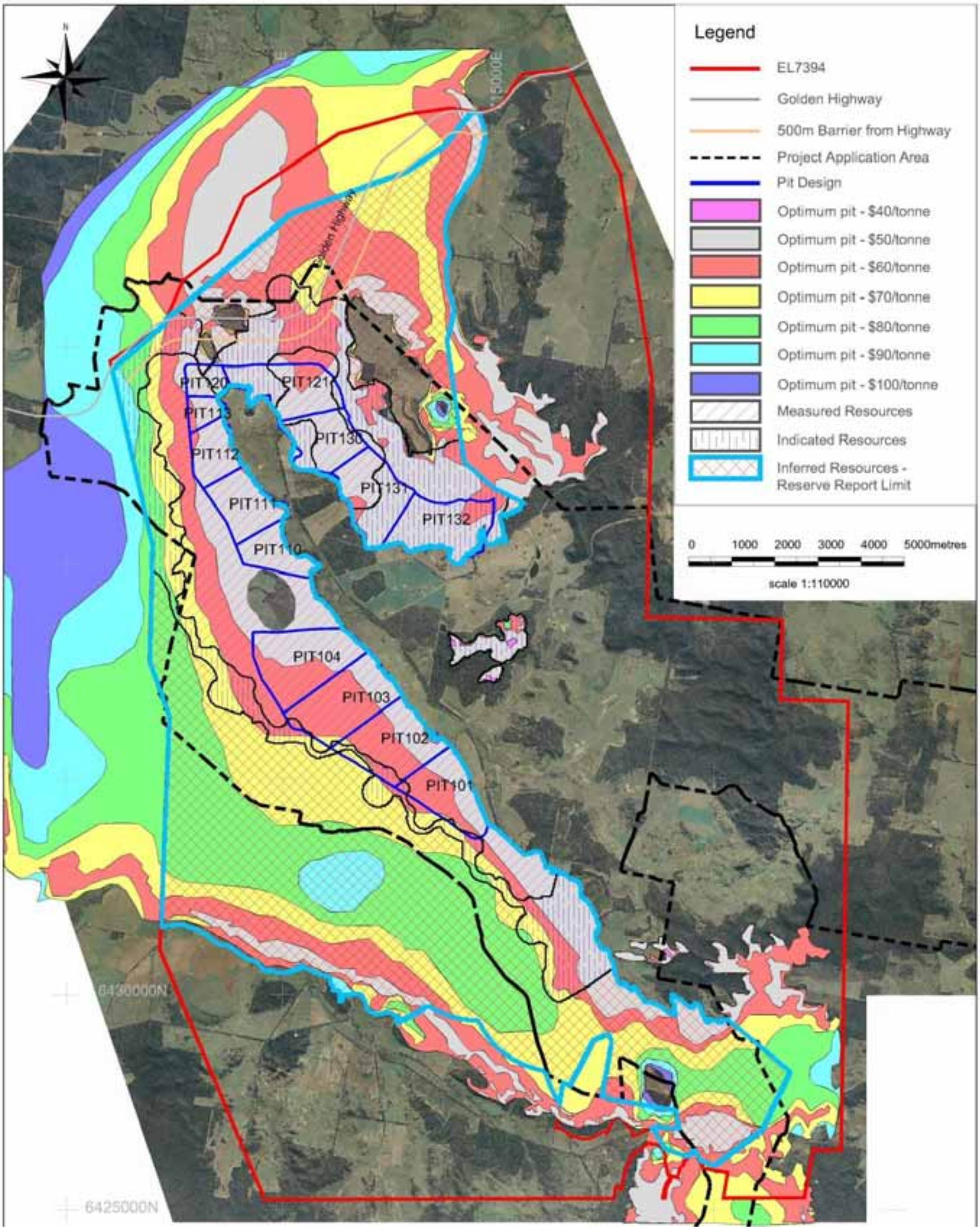
2. BASIS OF DESIGN - MINING

The Project envisages mining up to 20 Mtpa of Run of Mine (ROM) coal for 21 years. The revised mine plan has been developed to achieve this objective with the following key objectives in mind:

- Targeting the most cost effective areas to mine
- Reducing potential environmental impacts
- Avoiding sensitive environmental areas, including riparian areas and agriculturally valuable land
- Mining as efficiently as feasible using modern open cut mining techniques.

A mine optimisation study was conducted to determine the most cost effective areas to develop. The optimisation model results are shown in Figure 2.1. It can be seen that mining area C and the northern extent of mining area A (Pits 120 - 132) are some of the lowest cost mining reserves available (grey shading) within the Project Application Area and the Exploration Lease. The mine optimisation study is based on CHC's June 2013 geological model and September 2012 Reserve statement.

Figure 2.1 Pit Optimisation Results



The mining area shells were designed with the following general constraints:

- Northern Limit

  Golden Highway: Generally this highway denotes the most northerly limit of mining. Due to the mines proximately to the Golden Highway the pit crest has been offset at least 1,000 m from the road to ensure no road closures are required for blasting

  - Talbragar River: Avoid connectivity with the Talbragar River
  - Wooded areas: Northern constraints to ensure a remnant vegetation corridor of ~500 m is maintained.
- Southern Limit
  - Water management: A large catchment area exists further south therefore the mining area is limited by increasing the mine inundation risk
  - Creeks: The Sandy Creek and Laheys Creek are in confluence in this area
  - Haulage distance: The further south the mine extends the greater the haul distance and therefore more truck movements are required
  - Minimum mining area: The southern limit constraint was to ensure a practical mining area was available (strip width) whilst avoiding the ecologically endangered communities (EEC) and flora species as well as the Yarrobil National Park
  - Noise impacts: Constrained by proximity to residential blocks.
- Eastern Limit
  - Water management: Investigation into creek impacts to the east had not been completed, therefore a minimum of 1,000 m offset was applied
  - Remnant vegetation corridor: A 500 m wooded area corridor was to be maintained.
- Western Limit
  - Sandy Creek and Laheys Creek: A 200 m offset from the creek line was to be maintained to avoid sensitive heritage areas
  - Strip Ratio: As mining progresses further west strip ratio generally increases
  - Coal Quality: Lower confidence in the coal quality further to the west.

Pit shells were then developed within the proposed mining areas around the most cost effective blocks, containing sufficient reserves to allow mining at the 20 Mtpa mining rate for a 21 year period (393 Mt ROM). The pit shells developed overlaid onto the topography are shown in Figure 2.2.

The annual estimated mining quantities required to meet the 20 Mtpa target is shown Table 2.1.

Figure 2.2 Pit Shell Locations

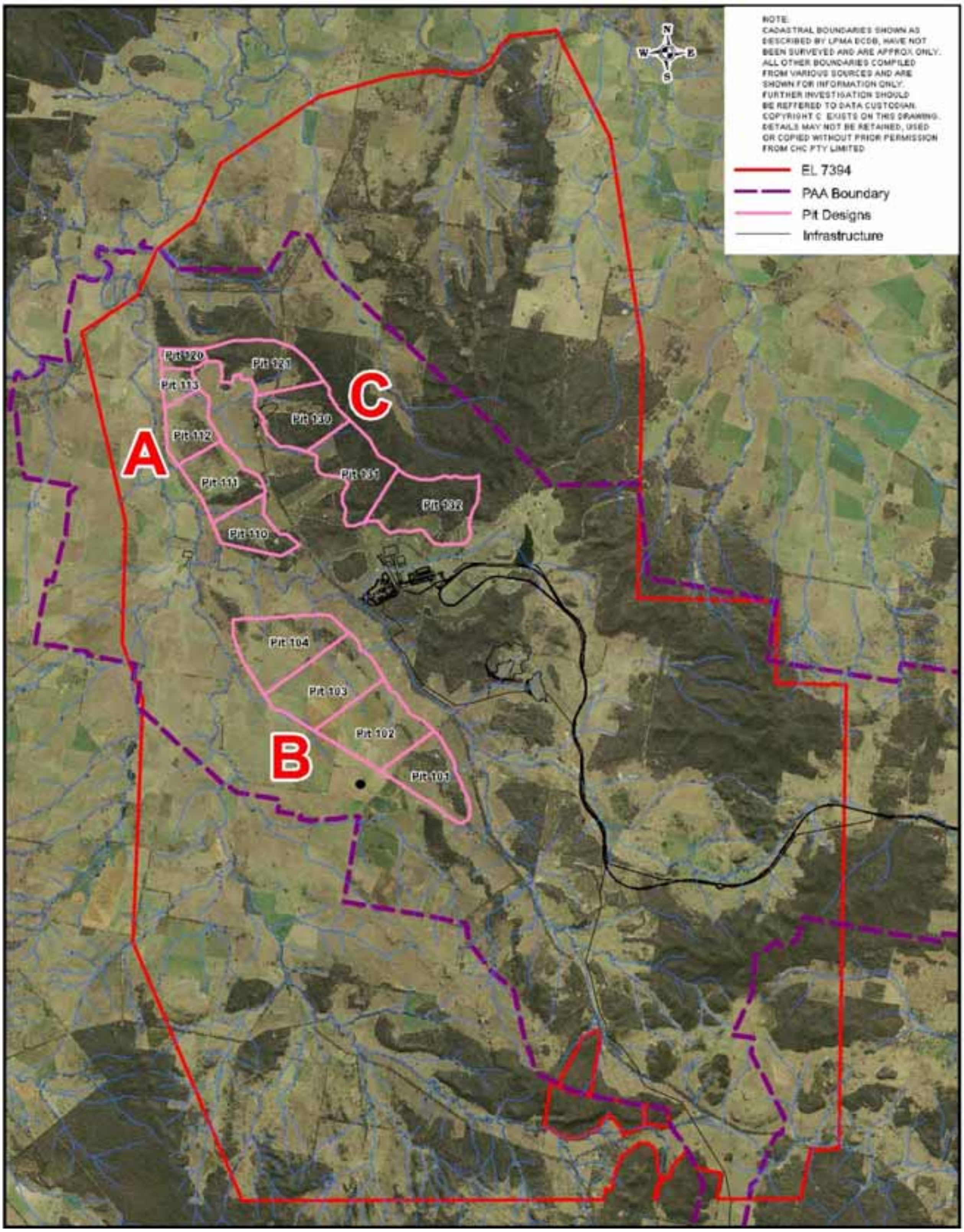


Table 2.1 Mining Quantities

Year	ROM (Mt)	Prod (Mt)	Yield (%)
1	11	8	70%
2	13	8	59%
3	19	11	61%
4	19	11	59%
5	20	11	55%
6	20	11	56%
7	19	10	54%
8	20	11	56%
9	20	11	56%
10	20	11	57%
11	20	11	56%
12	20	12	58%
13	19	11	56%
14	20	12	58%
15	20	11	53%
16	20	11	57%
17	19	10	54%
18	19	10	54%
19	20	11	54%
20	20	11	55%
21	14	7	54%
TOTAL	**392**	**22**	**56%**

These mining quantities partially deplete the available Reserves of the mine as shown in Table 2.2. Table 2.2 also shows the higher strip ratio and hence mining costs associated with mining area B when compared to mining areas A and C.

The reduction in mining Reserve from the PPR&RTS is due to a reduction in mine surface area to maintain an ecological corridor in northern area of the exploration licence, refer Section 5.

Table 2.2 Mining Reserves

Pit	Waste (Mbcm)	ROM Coal (Mt)	ROM Strip Ratio	Product (Mt)	Product Strip Ratio
Pit 110	27.2	16.2	1.68	11.0	2.47
Pit 111	46.3	27.1	1.71	18.8	2.47
Pit 112	38.1	17.7	2.15	11.6	3.28
Pit 113	14.8	5.9	2.51	4.0	3.68
Pit 120	8.9	5.4	1.65	3.5	2.54
Pit 121	61.3	29.9	2.05	18.8	3.25
Pit 130	50.7	28.2	1.80	15.5	3.26
Pit 131	61.9	28.6	2.17	14.4	4.30
Pit 132	82.5	35.0	2.36	18.8	4.39
Pit 101	134.7	38.5	3.50	21.3	6.33
Pit 102	348.8	84.0	4.15	48.7	7.17
Pit 103	305.9	77.3	3.96	45.7	6.69
Pit 104	132.9	50.9	2.61	31.3	4.24
Total	**1,314.1**	**444.7**	**2.96**	**263.4**	**4.99**

3. DESCRIPTION OF THE REVISED MINE PLAN

3.1 Description

Rather than three active mining areas as proposed in the PPR&RTS, the revised mine plan has an initial focus on mining in areas A and C and delaying mining in area B. This mine plan reduces the number of active mining areas which in turn reduces:

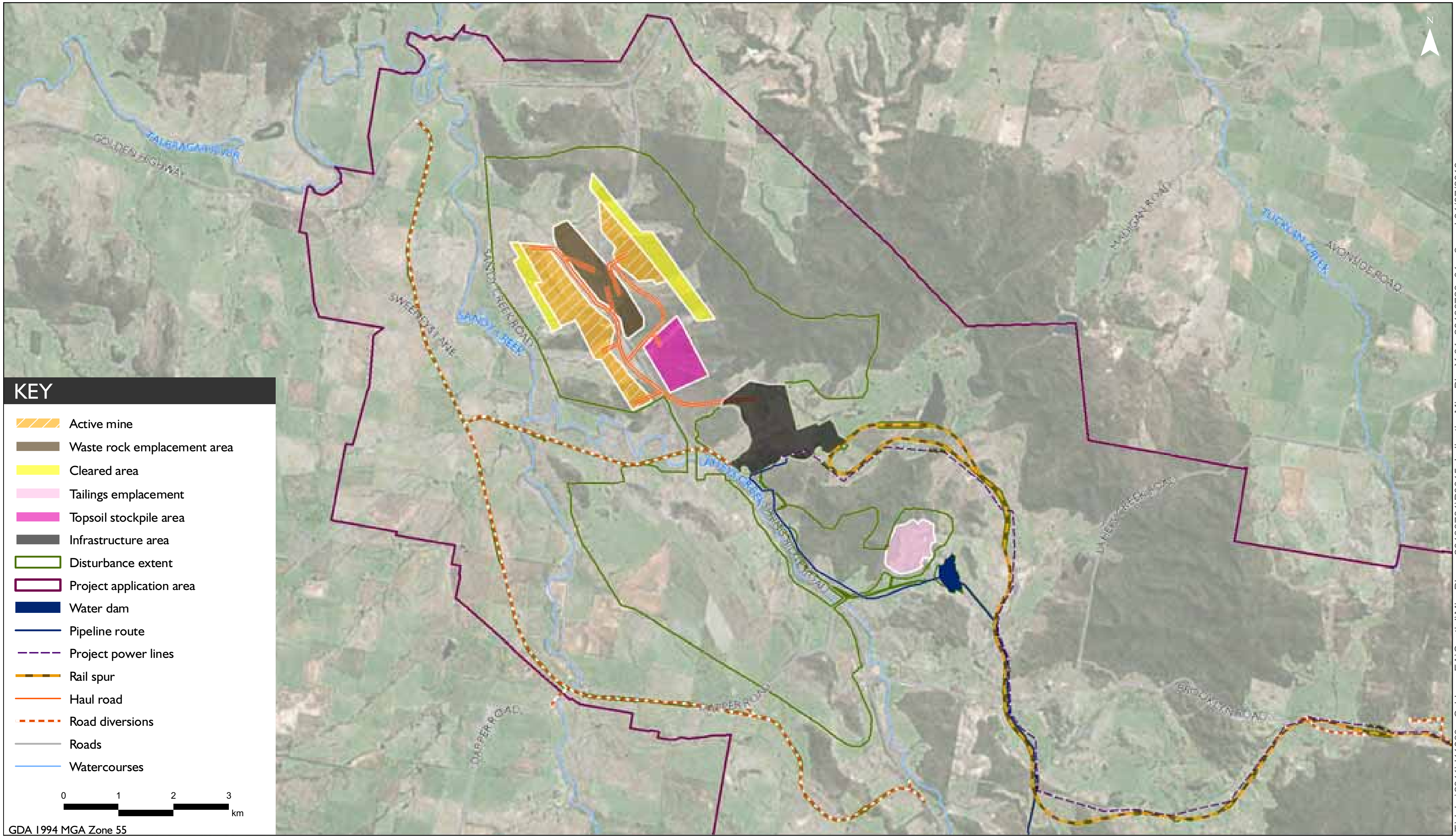
- The number and length of haul roads open at any one time
- The amount of exposed waste rock emplacement faces.

However, this mine plan does increase the in-pit and out-of-pit waste rock emplacement elevations to a maximum height of RL 450 mas shown below in Figure 3.8 (Year 21), an increase of 20 m.

The revised mine plan is based on production ramping up from year 1 to 20 Mtpa ROM in year 5 with an average strip ratio of 2.1. The strip ratio increases to an average of 3.3 when mining commences in mining area B. Mine progression is shown in the stage plans Figure 3.1 to Figure 3.6 and mining occurs in:

- Mining area A from Year 1 to year 10
- Mining area C from Year 1 to year 13
- Mining area B from Year 11 to year 21.

When the mining sequence exhausts mining areas A and C in year 13, mining area B will become a standalone operation.



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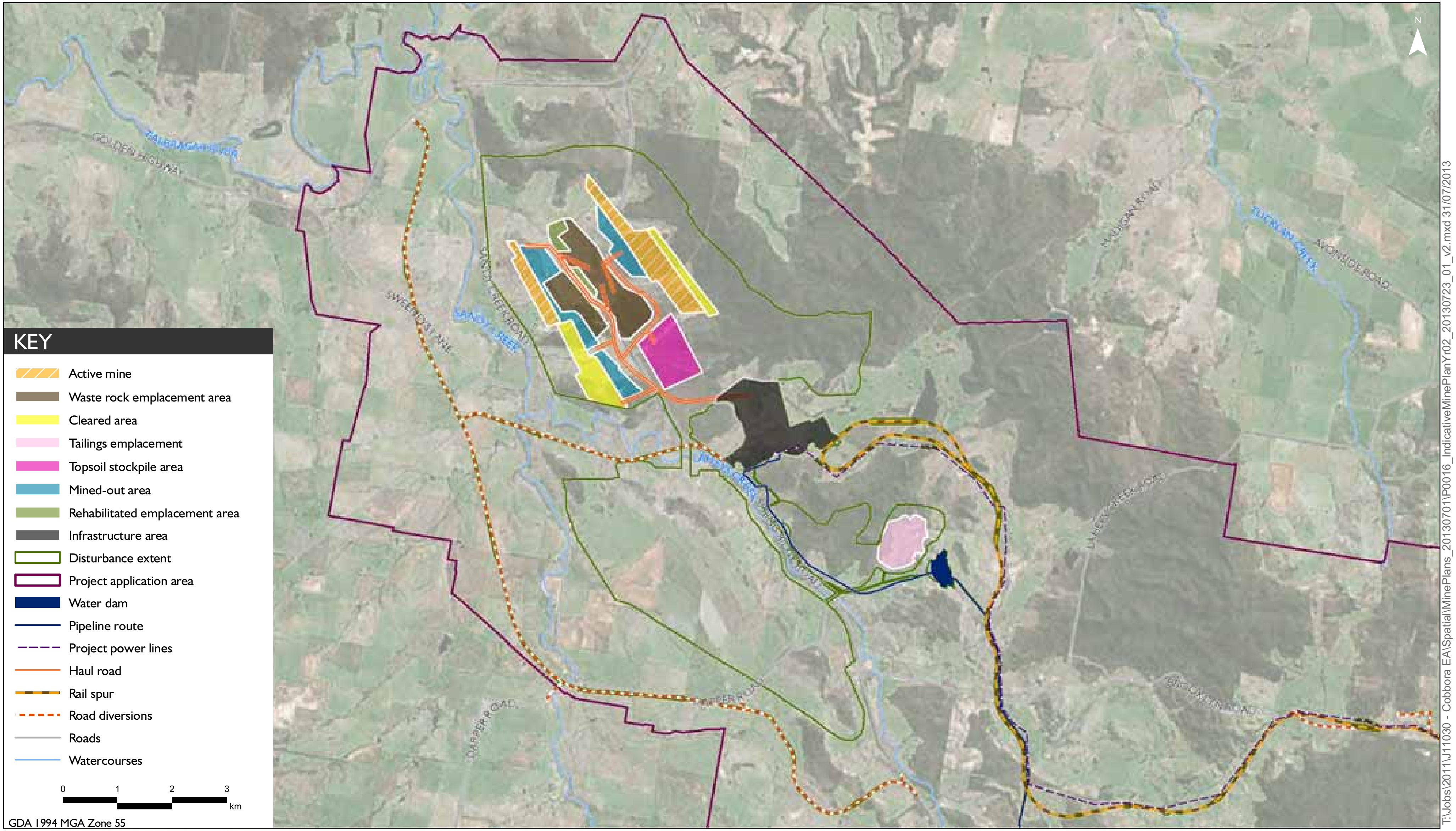
EMM

Indicative mine plan - Year 1

Cobbora Coal Project

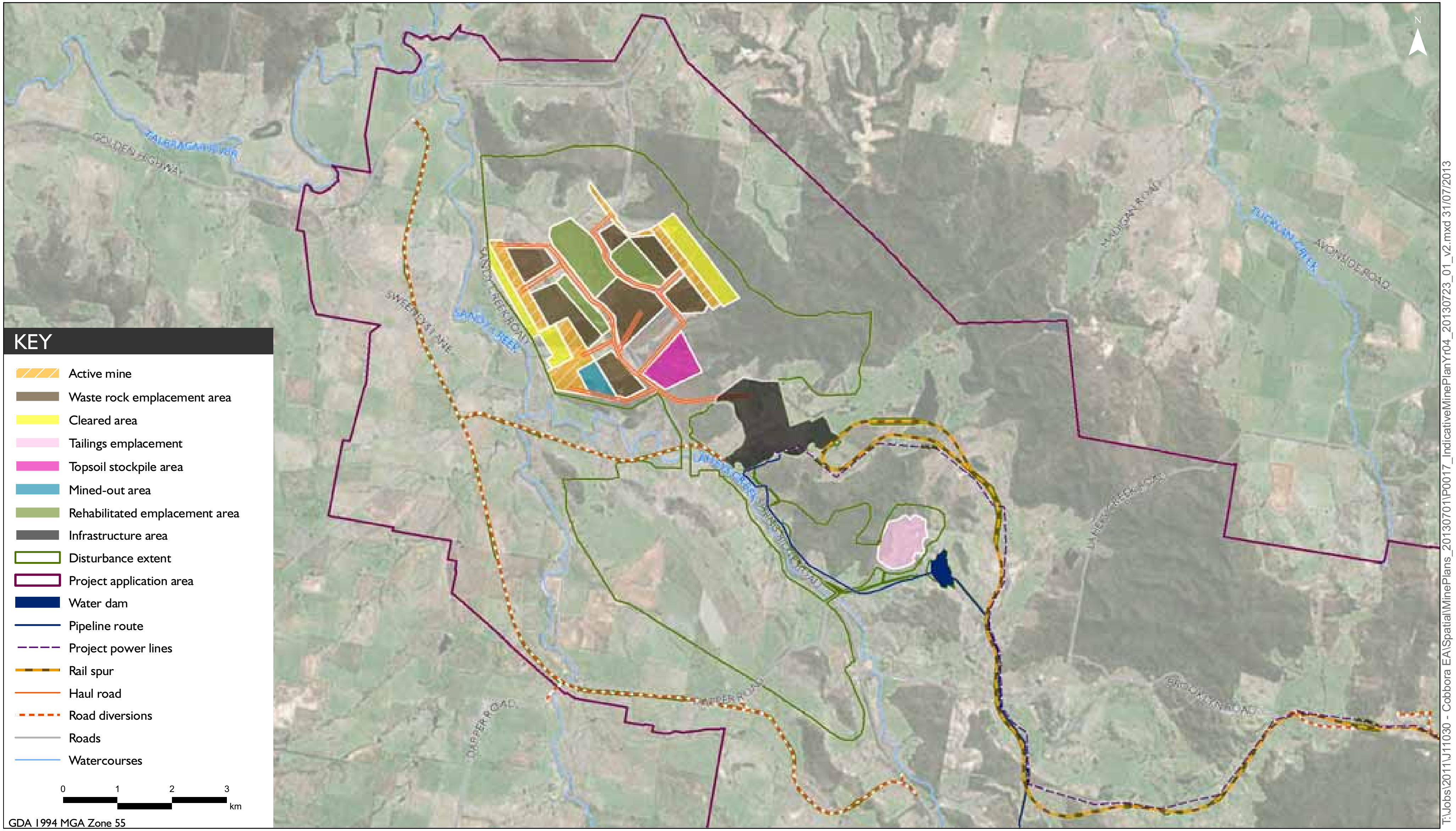
Responses to PAC Review

Figure 3.1



Indicative mine plan - Year 2
Cobbora Coal Project
Responses to PAC Review

Figure 3.2



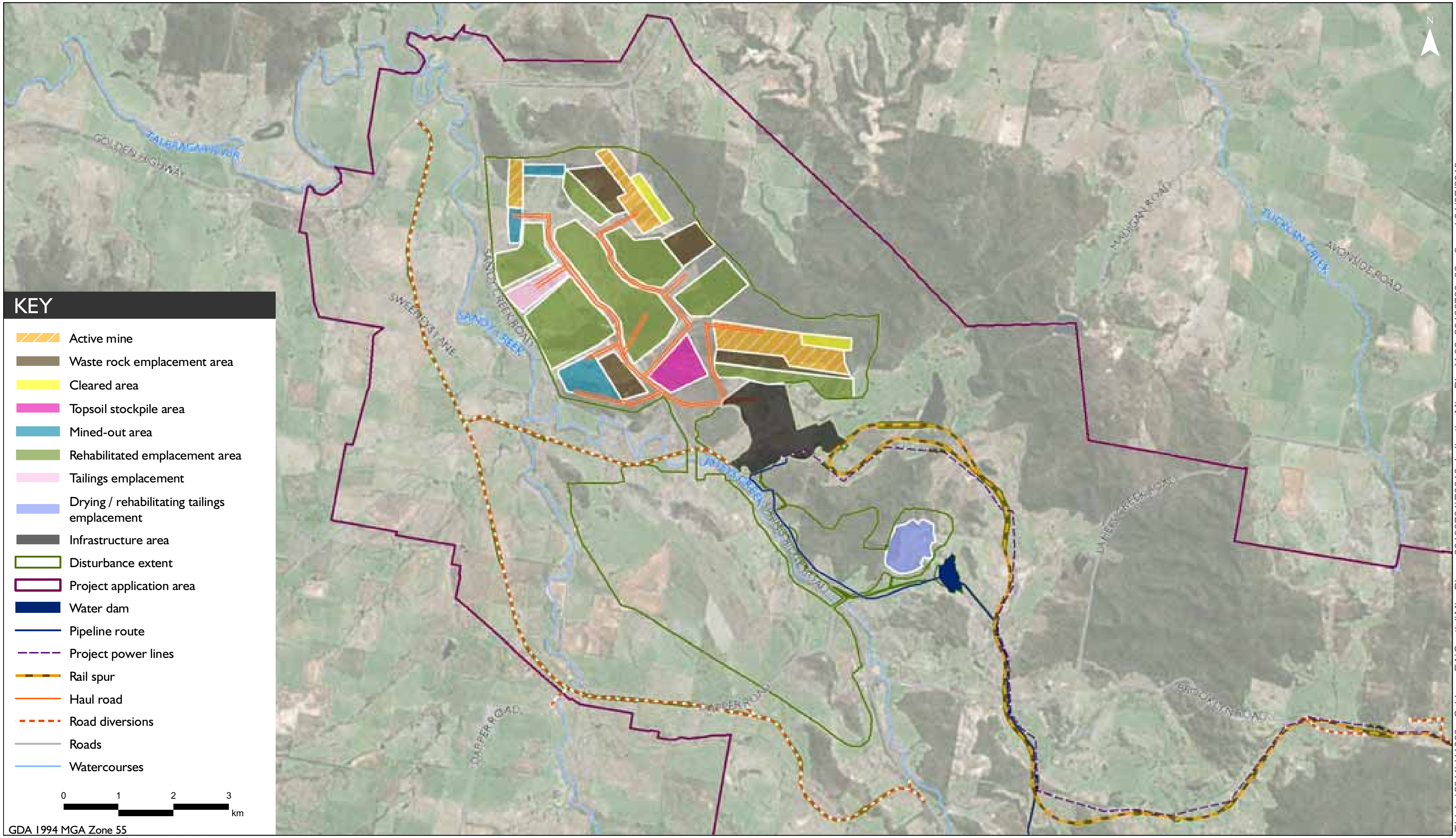
EMM

Indicative mine plan - Year 4

Cobbora Coal Project

Responses to PAC Review

Figure 3.3



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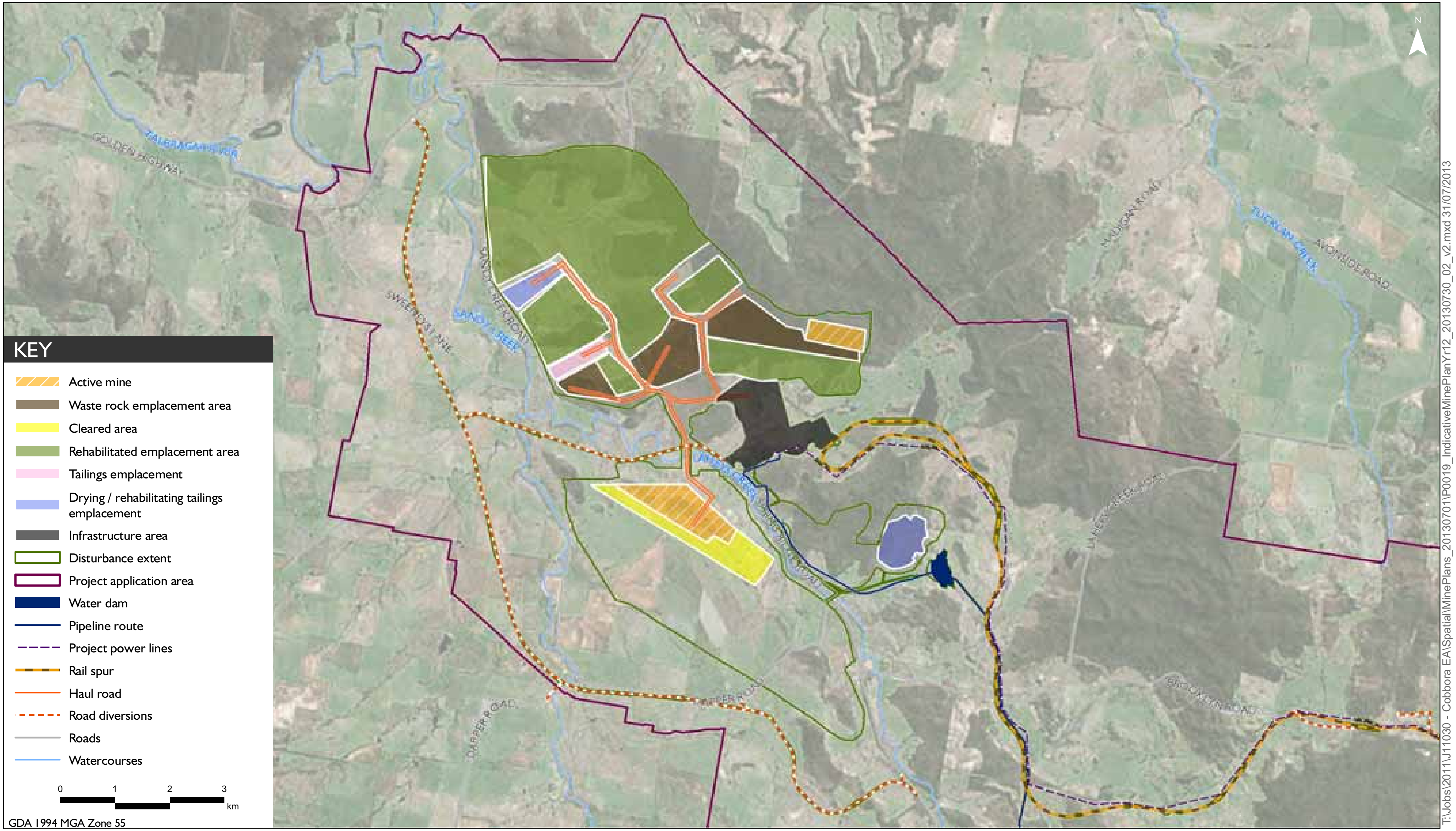
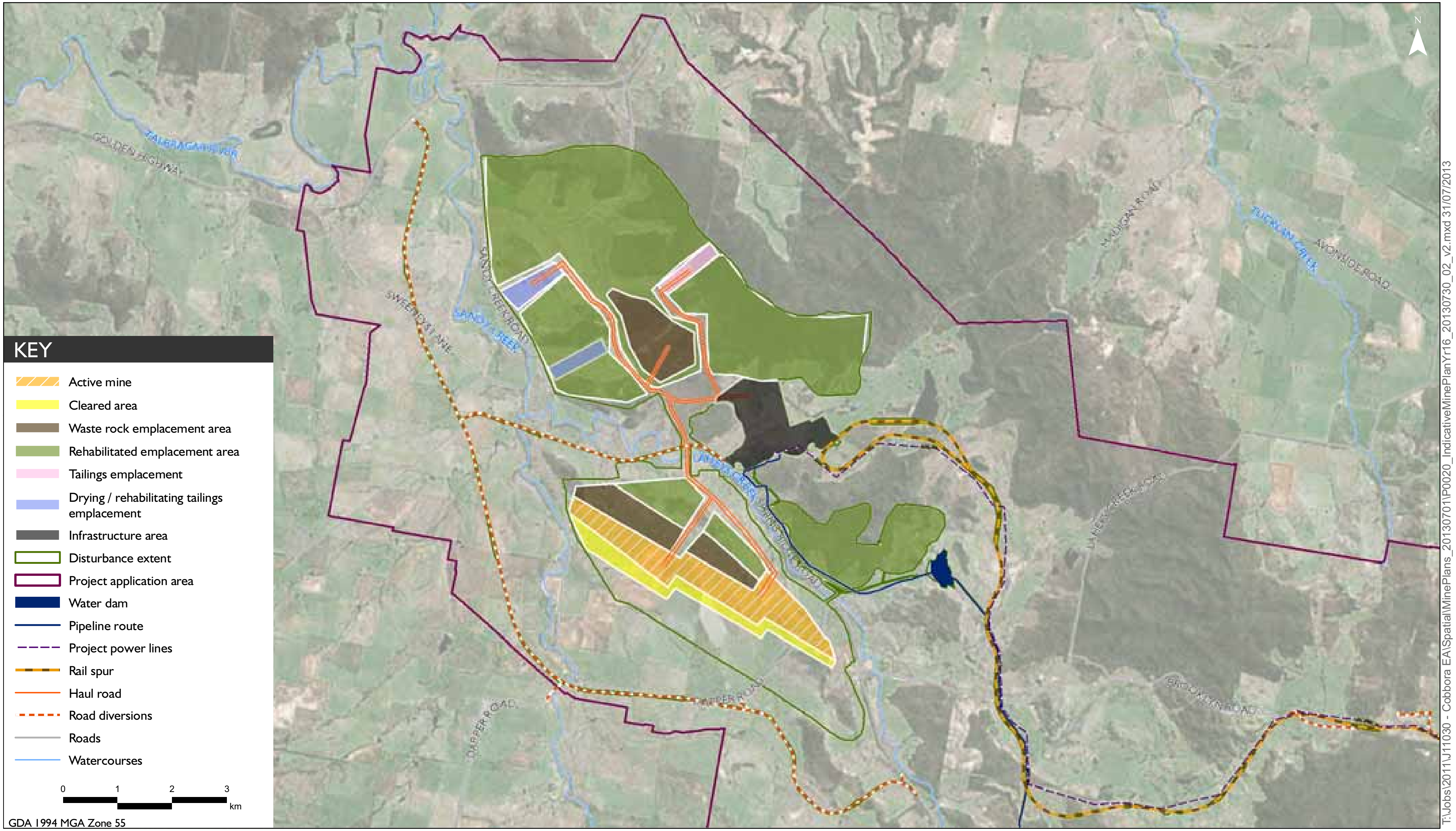
Indicative mine plan - Year 8
Cobbora Coal Project
Responses to PAC Review
Figure 3.4



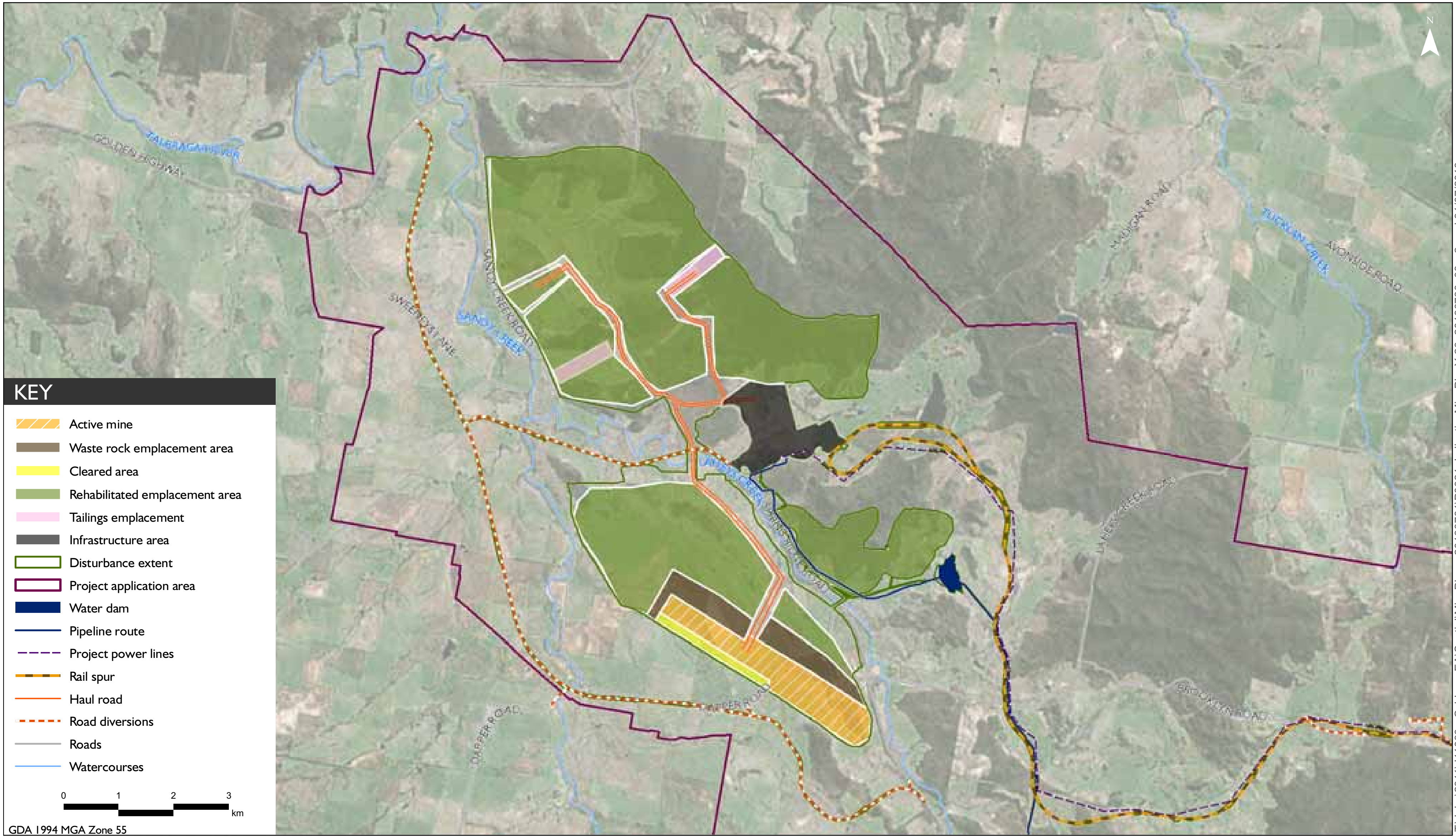
EMM

Indicative mine plan - Year 12
Cobbora Coal Project
Responses to PAC Review
Figure 3.5



EMM EMGA Mitchell McLennan

Indicative mine plan - Year 16
Cobbora Coal Project
Responses to PAC Review
Figure 3.6



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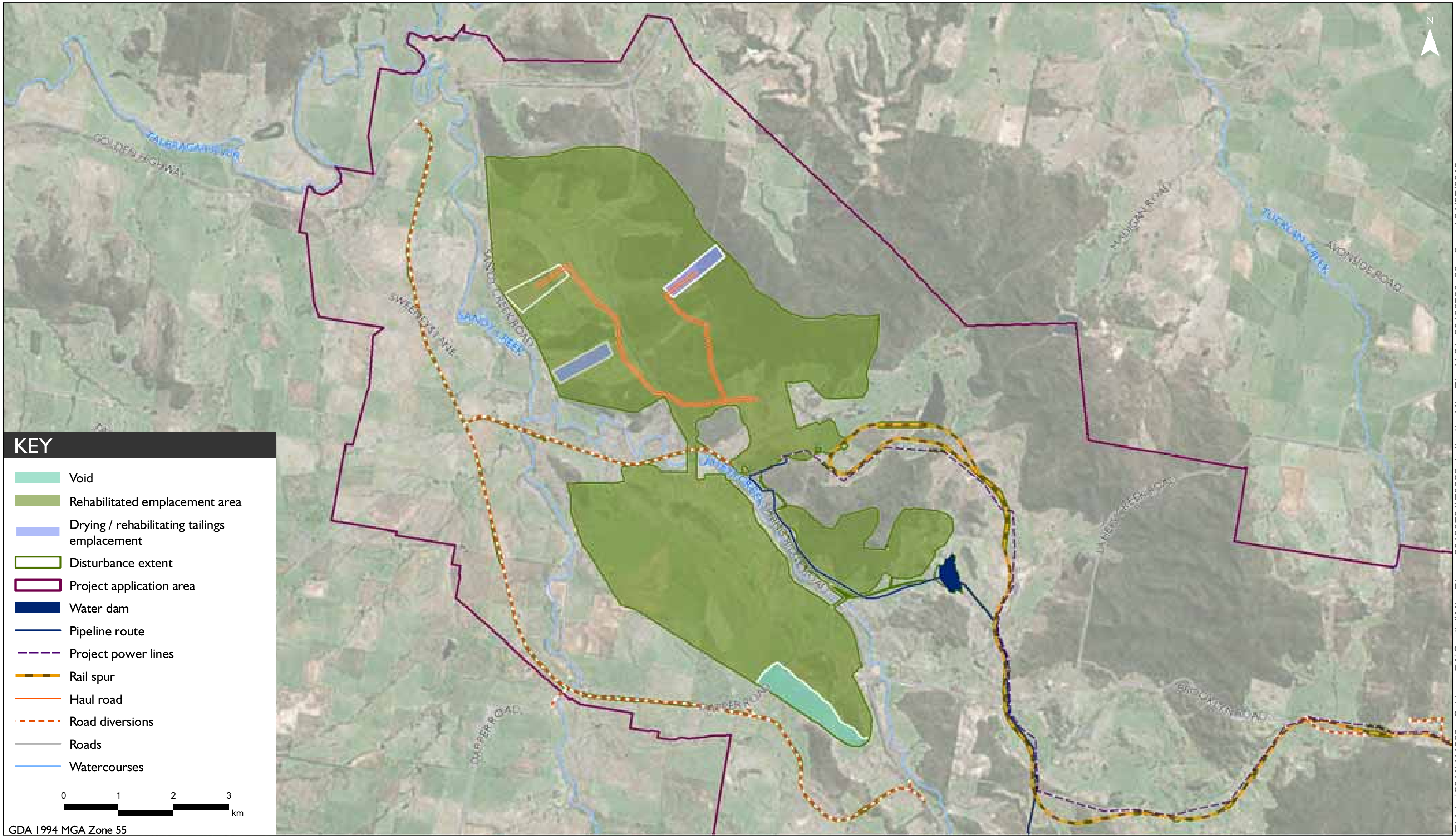
EMM

Indicative mine plan - Year 20
Cobbora Coal Project
Responses to PAC Review
Figure 3.7



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Indicative mine plan - Year 21
Cobbora Coal Project
Responses to PAC Review
Figure 3.8

3.2 Mining Method

The waste and coal will be uncovered and extracted using established open cut mining methods. The equipment utilised will include 600 t and 400 t hydraulic excavators with a large front end loader as a back-up loading unit.

Generally the mining process will be as follows:

- Waste and coal will be drilled and blasted utilising 15 t and 10 t pull down capacity drill rigs
- Loading equipment used for waste removal will consist of 600 t hydraulic excavators for major waste material and 400 t for partings
- 220 t trucks will be utilised for waste and coal haulage
- Initially all waste will go to out-of-pit rock emplacement (dump) areas located between areas A and C where there is no coal. Once in-pit voids are available, dumping will occur in-pit
- The prestrip mining area has an overall layback of 14° while the dump area is at 19° in order to accommodate haulage ramps and safety berms
- Coal will be mined with a 400 t excavator, a excavation machine that reduces the reliance on dozers. Approximately 95% of all coal is greater than 1.0 m in thickness and will be mined with minimal dozer preparation
- Coal will be hauled from the active pits to the ROM stockpile or direct tipped into the dump hopper of the Coal Handling and Processing Plant (CHPP)
- Coal will then be reclaimed, crushed and conveyed to a Coal Processing Plant for washing
- After washing, the coal will be stacked on a product coal pad prior to being loaded onto a train for transport. The product coal pad will be large enough to stack different coal quality products separately
- A front end loader and small fleet of trucks will remove topsoil and another front end loader will be located at the ROM stockpile for coal rehandle and as a back-up machine.

Rehabilitation is scheduled to begin on the north out-of-pit waste emplacement area in year 2. The average rehabilitation rate is 150 ha per annum which includes a mixture of cropping, pastoral and woodland areas. Topsoil will be hauled to the topsoil emplacement areas for the first eight years for placement over rehabilitated surfaces and in the remaining years direct placement of top soil will take priority over stockpiling.

3.3 Mining Assessment

The following factors that were assessed in determining the revised mine plan that forms the basis of this report:

- Yield and product ash assessment for single stage washing at 1.60, 1.65 and 1.70 density that included coal properties including sulphur, specific energy, moisture and volatile matter analysis
- Identification of potential working sections (ie combination of coal plies) that will provide the best compromise between yield and mining costs to comply with the product coal quality requirements
- The stratigraphy identified has resulted in improved identification of internal parting bands within coal seams

- Refinement of the limits to the pit shell due to improved understanding of the marginal economic seams
- Analysis of coal size distribution used for the coal preparation plant specifications
- Assessment of potential groundwater inflows
- The open cut workings do not intersect the Laheys or Spring Ridge creeks
- Geotechnical information to avoid known stability issues and the use of end-wall benches used as roadways are of sufficient width to provide toe-of-wall catch berms
- Reducing the number of final voids.

3.4 Revisions since Submission of the Preferred Project Report/Response to Submissions

There have been changes to the mine plan submitted with the PPR&RTS which focus on the issues raised by the PAC with regards to the northern remnant vegetation corridor, mining sequence and waste rock emplacement areas but otherwise the plan is similar to that presented in the PPR&RTS.

The revised mining sequence has been optimised to minimise the amount of waste being hauled to mining area B's Out-of-Pit waste emplacement East (B-OOP E). Previously the mine plan had commenced mining in the southern end of mining area A. This revised mine plan delays mining in this area so waste can be hauled from the northern end of mining area B to area A to both in-pit and out-of-pit emplacement areas. Waste is hauled north from mining area B from year 13 to year 15.

In the previous mine plan, B-OOP E had a 75 Mbcm capacity and a maximum height of RL 450 m. As a result of the optimisation process described above, CHC have maintained the same maximum elevation and reduced B-OOP E waste emplacement area capacity to 40 Mbcm.

Another revision is the increase in the northern waste rock emplacement area dump height (between area A and C) from RL 410 m in the north to RL 435 to maintain sufficient volume for material from mining area B being hauled north once the topsoil dump has been reclaimed. Due to the higher strip ratio in mining area B and the size reduction in B-OOP E, the in-pit waste emplacement height must increase from RL 430 m to RL 450 m.

There has been a decrease in overall fleet numbers; the unit size has increased with the 250 t excavators on coal and partings being replaced with 400 t excavators. This is due to the coal seams being mined as a working section, whereas the previous mine plan had a more selective mining approach. The peak waste truck fleet has increased due to the higher overall strip ratio but this is offset by the reduction in the number of excavators and other support equipment.

The revised mine plan has reduced the in-pit tailings emplacement areas which are consistent with the reduction in fines being generated; refer Tailings Management Review M01-CHC-351-RP-ENV-001. The revised mine plan requires one out-of-pit tailings storage facility and three in-pit tailing emplacement areas.

Final rehabilitation, with the exception of the tailings emplacements, is due to be completed in the northern areas by year 18. Rehabilitation will commence in mining area B in year 16. The out-of-pit tailings facility will be rehabilitated in year 16 as shown in Figure 3.6. In-pit tailings storage facility number 1 will be rehabilitated in year 19.

Another revision is that there is no voids in either mining area A or C but the final void in mining area B is the approximately the same. The revised mine plan still includes blasting of the highwall in mining area B to reduce this void.

The revised mine plan moves the mine infrastructure area from east of the existing Spring Ridge Road to be adjacent to the coal handling and preparation plant. This revision also includes a re-alignment of the mine access road at its eastern end. This change:

- Reduces the duplication of services and hence disturbance area footprint
- Reduces the noise impacts associated with the mine support infrastructure maintenance activities (short and sharp sounds) as the new location is more hidden from the nearest receptors.

3.5 Risks

This revised mine plan has determined that it is possible to maintain two mining areas concurrently.

Risks associated with the changes to the mine plan include:

- If future exploration indicates degradation in the coal quality and the proponent is limited to the two mining faces, coal quality requirements might not be met
- Currently, the mine plan meets projected coal quality requirements however towards the end of the mine life the ash content increases and yield decreases which could result in a difficult product to market
- Difficulty in selling a higher ash product to a domestic market
- Reducing to two mining areas could impact the mining schedule by the mining faces being too congested with equipment and therefore lowering productivity and/or mine output and hence higher costs
- Unable to produce coal if flooding occurs – the mine is limited to the two pits instead of being able to mine three pits (water can be pumped to a pit void)
- Final dump heights are higher with an associated potential increase in dust generation and visual impacts
- Blast impacts being more concentrated in the south as only one pit will be operational in the latter years
- Increase in capital expenditure in the last eight years of mining
- Increase in capital and operation expenditure (ie more equipment required, infrastructure expansion (stockpiles), etc) in the last 8 years due to higher strip ratio in the south compared to having three mining areas open over the life of the mine.

4. PAC RECOMMENDATION – OUT-OF-PIT DUMP EAST

4.1 Alternative Locations

As part of the mine plan that formed the basis of both the EA and the PPR&RTS, an out-of-pit waste emplacement area/dump is required to accommodate the mining area B box cut material. An area east of mining area B and south of the CHPP plant was selected for its compact design which minimised disturbances, minimal haul distances and its proximity to the CHPP.

The PAC recommended that CHC:

- Do further work to find an alternative location for the waste dump and out of pit tailings; or
- Demonstrate that all reasonable measures have been taken to minimise the footprint, the surface area of exposed material and impacts on threatened species.

A number of alternative locations were investigated in response to the PAC Review report and these are shown in Figure 4.1 and include:

- Option 1: An area due south of the PPR&RTS base case, in the next valley
- Option 2: An area to east of the base option, over the hill and in the next valley
- Option 3: An area to the north-east of the CHPP plant, at the head of Blackheath Creek.

Areas to the south and west were eliminated as the areas are predominately Class 1 and Class 2 agricultural land. Areas to the south were also eliminated as they are constrained by existing and proposed road re-alignments and begin to encroach on land not owned by CHC.

Option 1, 2 and 3 are discussed in the following sections.

4.2 Option 1

The waste emplacement in this southern area, as shown in Figure 4.1, is constrained:

- To the east, by the rail spur
- To the south, by land not owned by CHC and an EEC (woodland)
- To the west, by the EEC along the Spring Ridge Road
- To the north, by an EEC (grassland).

To utilise a waste emplacement in this area, requires the re-alignment of the natural water course flowing through the valley at the centre of this emplacement. Given the RLs of the water course and the surrounding countryside, further investigation would be required to confirm that this realignment is possible. If it was it would require major civil works outside of the dump area shown with its consequential impact on the environment.

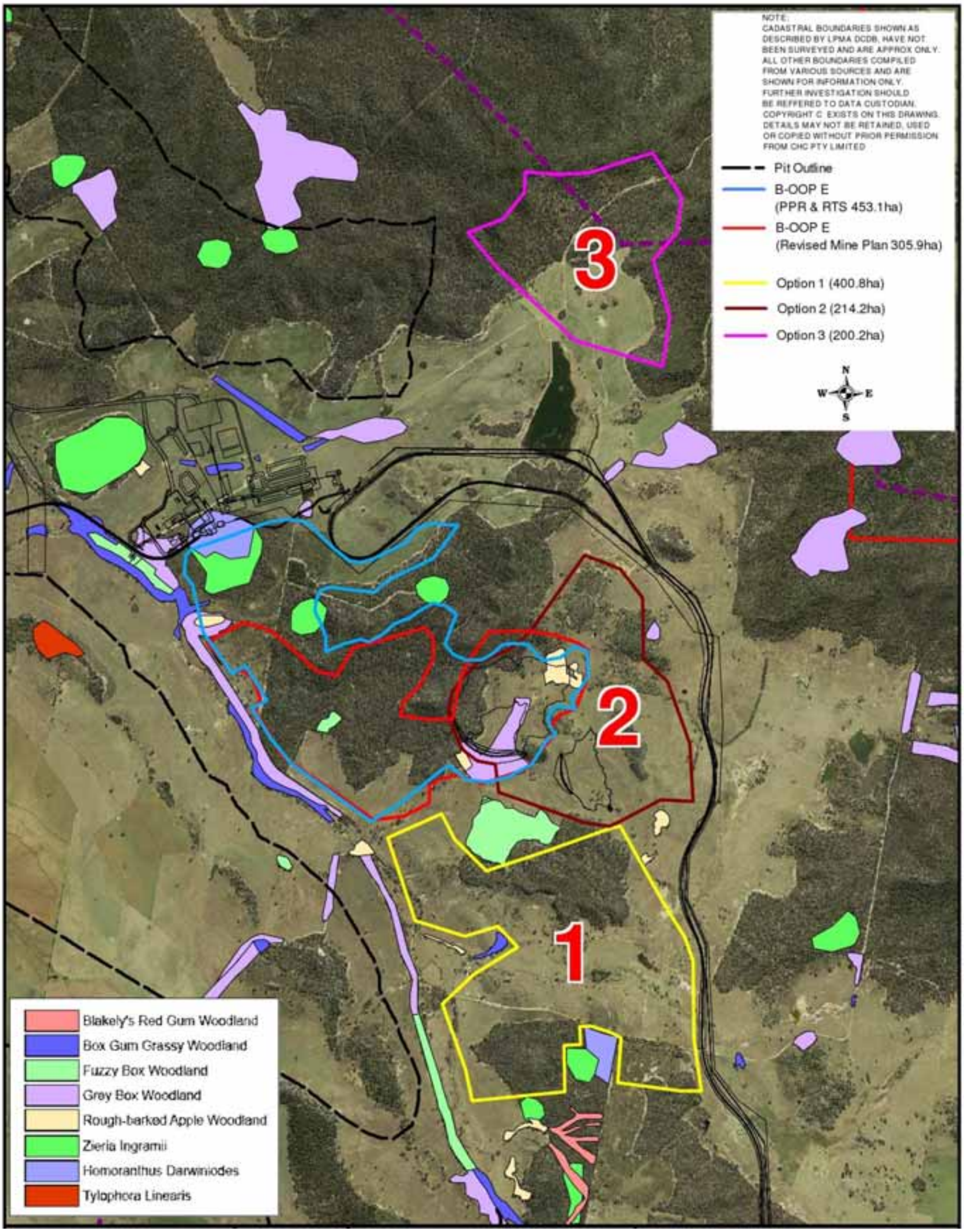
There are properties to the south along the rail corridor that are not owned by CHC that adjoin or are near this potential emplacement area. The combined impact of rail (visual and noise) and the potential dust and noise from mining waste movements make this option unviable.

The potential waste emplacement area disturbs the largest amount of surface area of all the alternative options.

The south west portion of this option, adjacent to and including the wooded area, is part of CHC biodiversity offset package and hence if selected would require additional offset land to be found.

This option also increases the haul distances of the waste compared to the PPR&RTS design. This then increases capital and operating costs, ie additional trucks, fuel, haul roads and greenhouse gas emissions. The increased haul distances also introduces an increased risk of overheating the haul truck tyres that would be need to be managed when travelling to the southern end of the waste emplacement area.

Figure 4.1 Mining Area B Waste Emplacement Options



4.3 Option 2

The waste emplacement in this eastern area, as shown in Figure 4.1, is constrained:

- To the east, by the cutting for the rail spur and EEC (woodland)
- To the south, by EEC (woodland)
- To the west, by the tailings emplacement (covered by waste rock) and EEC (woodland and grassland)

As shown in Figure 4.1, if this option was to be implemented, then an alternative location of the water storage dam would be required. Possible locations are in an adjoining valley to the west of the tailings emplacement or in a similar location to Option 3. Both of these locations would involve encroaching on woodland areas.

This waste emplacement area does not impact any EECs other than those associated with the tailings emplacement but is adjacent to land selected for the biodiversity offset package adjoining the Tuckland State Forest. This offset area would potentially be affected by dust and noise whilst the waste emplacement is being utilised.

To avoid EEC areas, the final height of the waste emplacement would need to be RL 500 m. At RL 500 m its peak will be approximately the same height as the highest hills to the east. This is higher visually than any other waste emplacement area option.

At this RL, the average elevation of a truck haul increases by 70 m compared to the PPR&RTS design. In mining terms, it is more efficient to have longer, flatter hauls than short, steep hauls and this waste emplacement area creates long and steep waste hauls. Thus, to design a waste emplacement area of this height (and distance) also increases the capital and operating costs and greenhouse gas emissions than the previous PPR&RTS design.

4.4 Option 3

The waste emplacement in this north-eastern area, as shown in Figure 4.1, is essentially unconstrained as the Woolandra irrigation dam is to be decommissioned shortly after the CHPP becomes operational.

Whilst not as significant a watercourse as Option 1, the waste emplacement area in this location would block a natural water course which would need to be re-aligned around the emplacement. Thus similar issues to Option 1 would be expected.

The waste emplacement area would have a final height of RL 475 m. This design increases the average elevation of a truck haul by 25 m. Similar to Option 2, this increase in the waste haul distance and height in turn increases capital and operating costs and greenhouse gas emissions than the PPR&RTS design.

Similar to Option 1, the haul distances increases the risk of overheating haul truck tyres when travelling to this location.

In addition, a portion of the proposed emplacement is currently outside the Project Application Area boundary and hence is in an area where no environmental surveys have been undertaken.

4.5 Summary Comparison

The alternative options for locating the waste emplacement are compared in Table 4.1.

Table 4.1 Waste Emplacement Location Comparison

Item	Revised Mine Plan	Option 1 South	Option 2 East	Option 3 North East
Disturbed area (est)	306 ha	401 ha	214 ha	200 ha
Wooded land impacted (est)	184 ha	113 ha	95 ha	138 ha
Woodlands: linked or isolated	Isolated	Linked	Isolated	Linked
Water course impact	No	Yes	No	Yes
EEC impacted	Yes	No	Yes	No
Biodiversity offset impacted	No	Yes	Potentially	No
Agricultural land impacted, Class	3, 4 &5	3	3 & 4	3 & 5
Height	RL450	RL440	RL500	RL475
Visual impact, resultant	Low	Low	High	High
Local community impacted	No	Yes	Potentially	No
Within Project Application Area?	Yes	Yes	Yes	Partly
Haul distance	Short	Long	Medium	Long
Impact on water usage	Least	High	Medium	High
Capital cost	Least	High	Medium	High
Operating cost	Least	High	Medium	High
Greenhouse gas emissions	Least	High	Medium	High
Haul truck tyre overheating issues	No	Yes	Potentially	Yes
Ranking	**1**	**3**	**2**	**3**

Option 1 can be eliminated due to water course and/or environmental amenity issues as well as haul distance issues and their associated cost impacts.

Option 3 can be eliminated due to water course and Project Application Area concerns as well as haul distance and height issues and their associated cost impacts.

Option 2 can be eliminated due the impact on existing use of agricultural land, its high visual impact, need for an alternative location for the water storage dam and the potential impacts on an biodiversity offset area as well as distance and height issues and their associated cost impacts.

4.6 Revised Mine Plan

The revised mine plan does not relocate the B-OOP E overburden dump and tailings emplacement areas but reduces the volume and therefore when compared to the PPR&RTS the revised footprint:

- Reduces the size of the impacted area from 453 ha to 306 ha, a 33% reduction (blue line vs red line in Figure 4.2)
- Increases the main remnant vegetation area by 109 ha
- Reduces the area of EEC species impacted by 4 ha, refer Table 4.2.

There is no change to the height of B-OOP E. However, the in-pit dump height needs to increase to RL 450 m, otherwise an expansion of B-OOP E is required and this will potentially affect an additional 25 ha of woodland area.

Figure 4.2 Revised Out-of-Pit Dump East

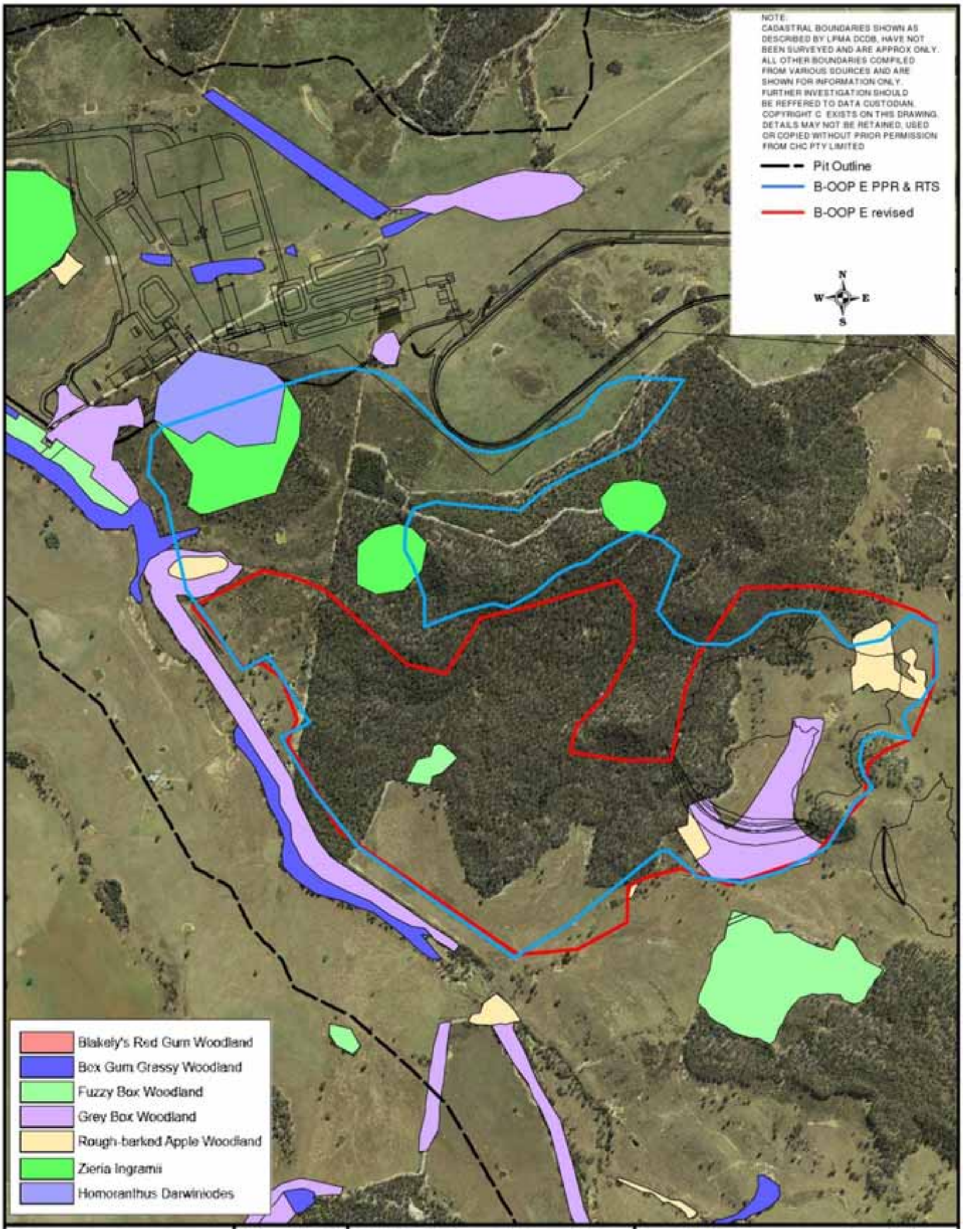


Table 4.2 EEC Preserved and Disturbed Areas

Species	PPR&RTS Areas (ha)	Revised Disturbed (ha)	Area Preserved (ha)
Box Gum Grassy Woodland	0.3		0.3
Fuzzy Box Woodland	2.1	2.1	0.0
Grey Box Woodland	2.4		2.4
Box Gum Woodland	9.8	8.3	1.4
Grey Box Woodland DNG	18.7	18.5	0.2
TOTAL	**33.0**	**28.9**	**4.4**

This reduction in the intrusion into remnant vegetated areas is possible by the delayed start in mining area B and moving the shortfall of available dump space waste from mining area B into the northern pits and/or out-of-pit dumps and raising the height of the in-pit dumps.

CHC believes that the reduced footprint of out-of-pit emplacement B-OOP E, which includes the tailings emplacement area, is an acceptable compromise that minimises the footprint, impacts on threatened species, the surface area of exposed material, length of haul roads requiring dust suppression and greenhouse gas emissions.

5. PAC RECOMMENDATION – NORTHERN REMNANT VEGETATION CORRIDOR

5.1 Scope

The impacts on woodland north and north-east of mining areas A and C that form an existing corridor of remnant vegetation will be minimised by:

- Regeneration of native vegetation (including assisted regeneration)
- Revision of the mine plan.

These are described in the following sections.

5.2 Regeneration of Native Vegetation

A vegetation corridor will be established on 185 ha of low quality agricultural land (generally Rural Land Capability Class IV and V) between the northern end of the mining area and the Golden Highway (diagonal hatching in Figure 5.1), in which native vegetation will be regenerated. This will improve the connectivity between the biodiversity conservation areas generally south-east of the mine (Yarrobil National Park (NP), Goodiman State Conservation Area (SCA), Tuckland State Forest (SF), the Southern NPWS addition offset, and the Eastern link areas offset) and the conservation areas generally north-west of the mine (Goonoo SCA, Cobbora SCA, Goonoo SCA addition offset and Cobbora SCA offset).

The biodiversity offsets are described in the biodiversity offsets package. The relationship between the existing reserves, the offsets and the vegetation corridor are shown in Figure 5.2. There is a gap of about 3 km between the western extent of the corridor and the southern end of the Cobbora SCA addition offset. This land is not suitable for regeneration or inclusion in the biodiversity offsets package, as it is higher quality agricultural land on alluvial flats either side of the Talbragar River. This prevents further extension of the biodiversity corridor in this direction.

The existing area of native vegetation will be expanded through natural and assisted regeneration. Native pasture areas will naturally regenerate over time through the management of threatening processes that inhibit natural regeneration. Throughout the Project area, natural regeneration is evident with shrubs and trees regrowing in native pasture where stock has been removed.

There are eight different native vegetation communities in the proposed regeneration area, two of which are the threatened ecological communities; Box Gum Woodland and Inland Grey Box Woodland. The native pasture areas to be regenerated are likely to have contained these threatened ecological communities prior to clearing for agriculture and grazing. The landscape is considered to be reasonably resilient and likely to respond well to management actions. Regeneration of similar environments, including those containing Box Gum Woodland and Inland Grey Box Woodland, has been shown to be successful elsewhere (Spooner et al 2002, Tiver and Andrew 1997). The regeneration program has a high likelihood of success, particularly as native pasture in this area exists adjacent to patches of good quality native vegetation.

The regeneration area will be managed for biodiversity conservation outcomes in accordance with the integrated land management plan. Natural regeneration actions will include the removal of stock and control of feral animals and weeds. In selected areas (see Figure 5.1), these actions will be supplemented by assisted regeneration including planting representative tree species tailored toward threatened species requirements, such as planting feed trees for the Glossy Black-cockatoo.

The regeneration in combination with the revised mine plan will create a vegetation corridor north of mining areas A and C that is 185 ha in area and between 900 m and 1,170 m wide (see Figure 5.1). This exceeds the requirement in the Planning Assessment Commission Review Report for the Boggabri Coal Project (February 2012, p25):

> *The Office of Environment and Heritage has noted the importance of any regional corridors maintaining a minimum width of 500 m. The Office of Environment and Heritage cited a number of studies which support the need for corridors of at least 500 m in order to ensure they are not dominated by edge effects and aggressive species that specialise in edge habitat. Wider corridors are also able to provide greater species diversity. While some studies and strategies have recommended regional corridors should have a minimum width of 1000 m, the Commission notes that the rehabilitation of the proposed mine sites will eventually provide for a much wider corridor. The Commission is satisfied that a 500 m wide corridor is appropriate and that as a consequence, both the Boggabri and Maules Creek Coal Mines would need to set aside a 250 m wide exclusion zone along their common boundary.*

Figure 5.1 Regeneration Area North of Mining Areas A and C

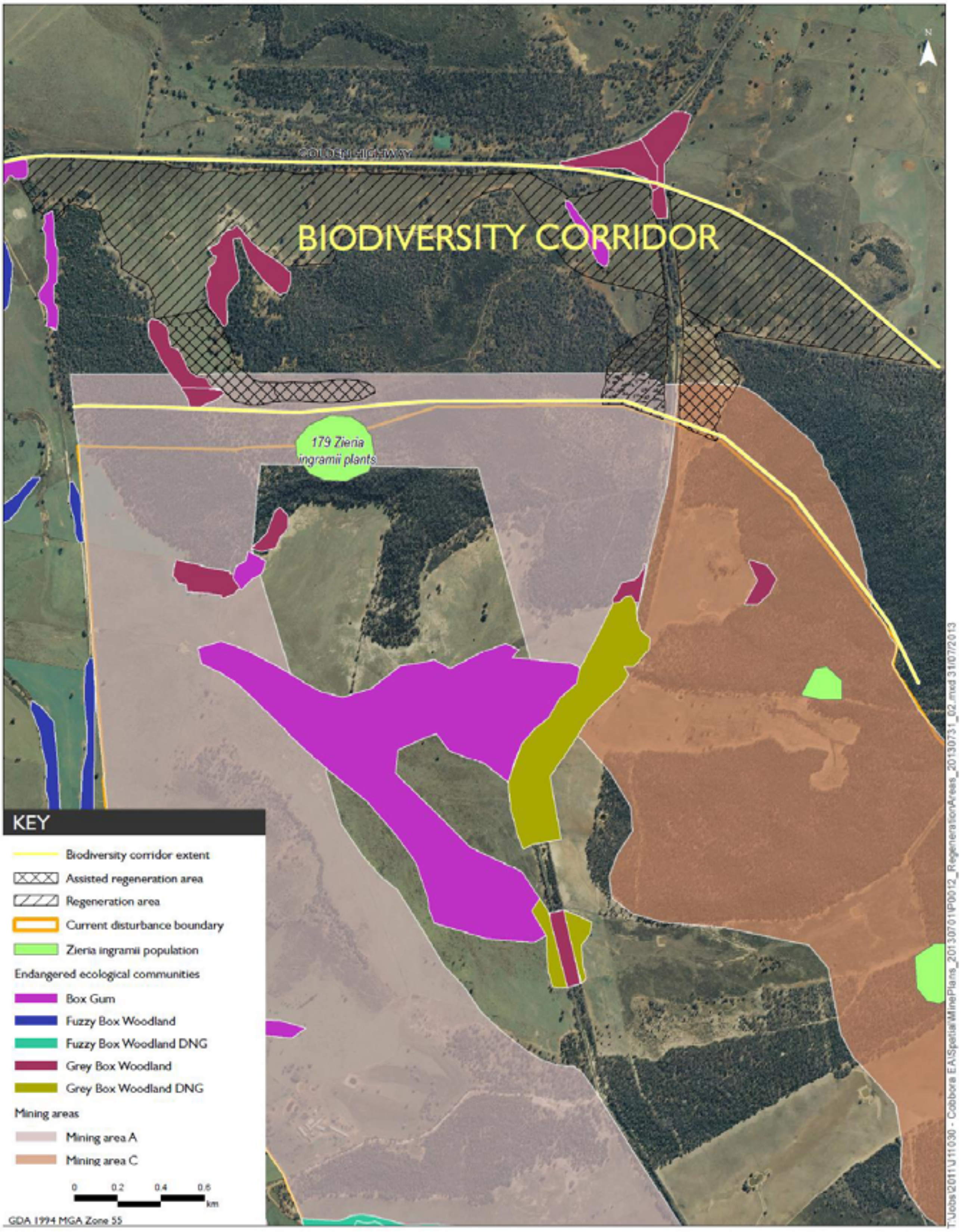
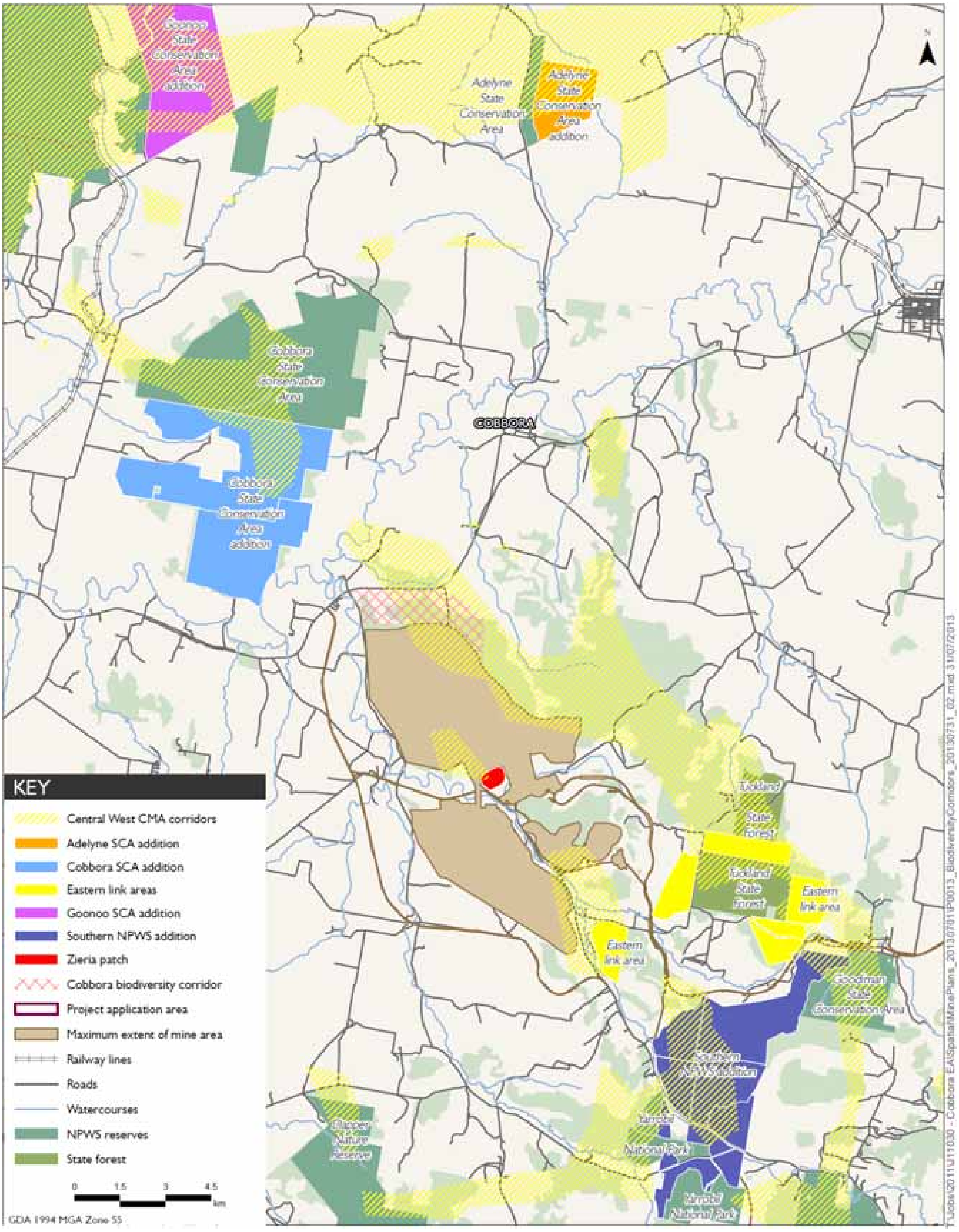


Figure 5.2 Biodiversity Offset Areas and Corridors



5.3 Revision of the Mine Plan

A series of mine plan revision options (A to F) were considered to address the PAC recommendation to reduce the impacts on the existing remnant vegetation corridor or to delay clearing in this corridor after demonstrating the biodiversity value of regenerated offset areas. This clearing was to start in Year 8 in the mine plan presented in the PPR&RTS. The mining and environmental advantages and disadvantages of these options are compared in Table 5.1.

The mine plan has been revised to increase the minimum width of the vegetation corridor by not mining about 4 Mt high quality ROM coal in the north (Area 1 in Figure 5.3). This can be done with or without replacing the 4 Mt coal resource. Option B and C consider no replacement mining areas. Options D, E and F include replacement mining areas.

Option C (Area 1 extension) avoids further areas of woodland, including part of a patch of *Zieria ingramii*, in the north-west corner of the mine (Figure 5.1). This option would remove a total of 6.6 Mt high quality/low strip ratio ROM coal from the mine plan (including the 4 Mt in Area 1). Removal of this high quality/low strip ratio coal from the mine plan will reduce the value of the Project.

The environmental benefits of avoiding this area would be to reduce the number of *Zieria ingramii* individuals impacted. The Area 1 extension will increase the width of the west end of the vegetation corridor width by 180 m compared to Area 1 alone (1,170 m).

The coal resource extends north of the most northerly extent of the mine plan. As well as the options compared in Table 5.1, it should be noted that the mine plan presented in the EA was restricted from extending further north to avoid impacts on woodland and to minimise visual impacts on the users of the Golden Highway and to residences north of the mine.

From a purely mining perspective, Option A would be CHC's preference and when environmental considerations are included, Option B is CHC's preference.

While Option C is less preferable, this option is proposed to ensure that the impacts of the mine on the patch of *Zieria ingramii* are minimised and that the benefits of the vegetation corridor are maximised. Without mining Areas 2, 3 or 4, Option C will remove about 6.6 Mt ROM coal from the proposal (1.5% of the total resource to be mined). The revised mine plan presented in Section 3 is based on this option.

Revision of the mine plan to address the PAC's recommendation to reduce the number of simultaneous mining areas required earlier clearing within the remnant vegetation corridor north-west of mining area C. This will not allow rehabilitation success to be demonstrated prior to clearing in this area. However, CHC believes that reducing the extent of the northern mining area and revegetation and enhancing 185 ha to the vegetation corridor north of mining areas A and C meets the intent of the PAC's recommendation to minimise the impacts of the Project on the remnant vegetation corridor and enhances the value of the corridor's connection between conservation areas south-east and north-west of the mine.

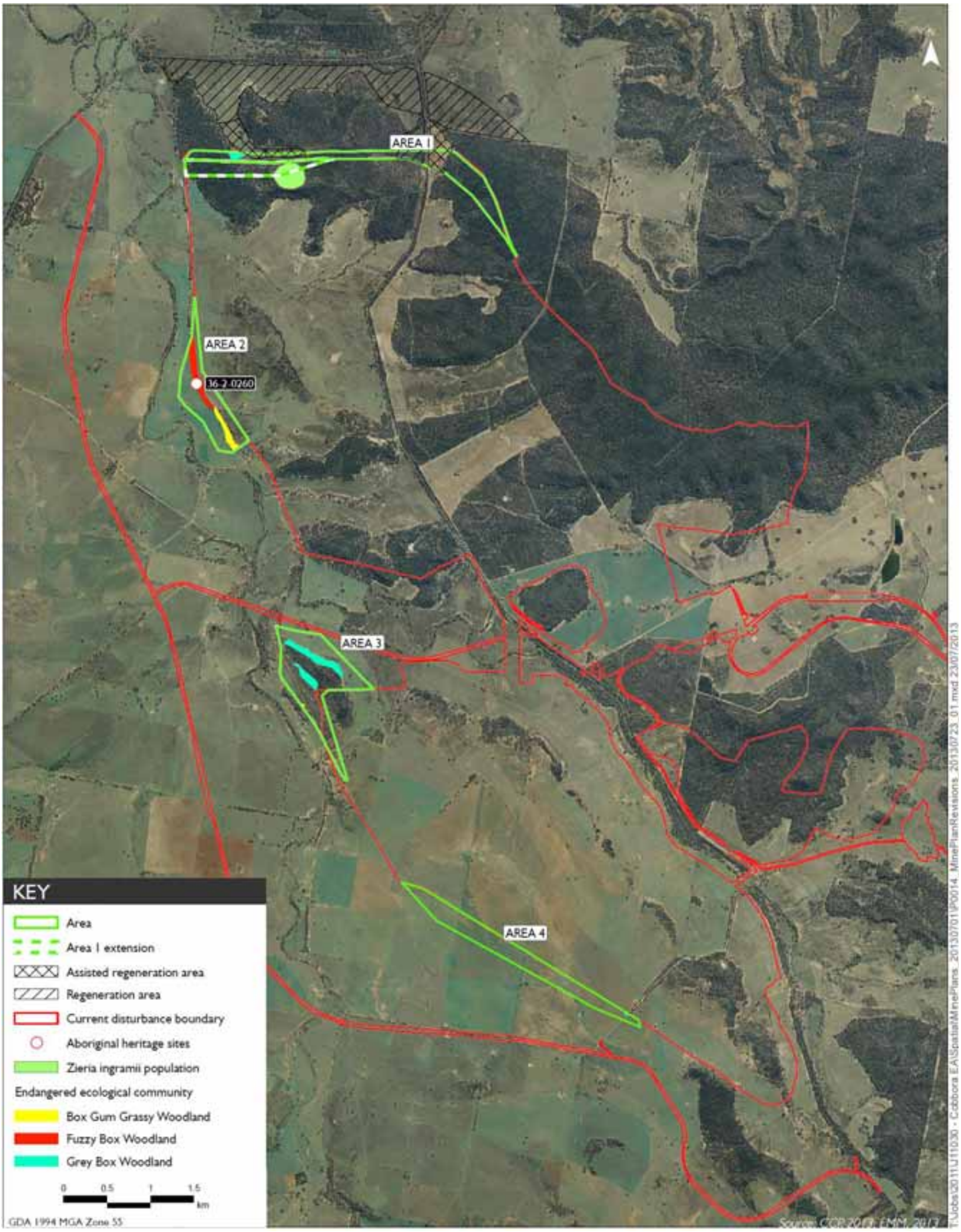
Table 5.1 Mine Revisions Considered in Response to PAC Review Report

Option	Advantages (compared to Option A)	Disadvantages (compared to Option A)
A. No change to Revised Mine Plan	**Mining** Maximises coal resource recovery Coal in Area 1 is a 'measured resource' Strip ratio (waste:ROM coal) of 1:2.53 (Pit 21 strip ratio)	
B. Remove north-east section of mining area (Area 1) - without replacement	**Mining** None	**Mining** Removes ~4 Mt ROM coal from the proposal. This reduces the coal volume by 1%. Coal quality in Area 1 is some of the best coal in the deposit. The mining area has a low strip ratio, the coal has low ash and high yielding properties Refer Appendix 1 for indicative evaluation
	Environmental 1.2 ha of Grey Box Woodland EEC and 60 ha of other woodland in Area 1 will not be removed Impacts lower quality agricultural land: 43.6 ha of Rural Land Capability Class IV	**Environmental** None
C. Remove extended north-east section of mining area (Area 1 and Area 1 extension) - without replacement	**Mining** None	**Mining** Removes an additional ~2.6 Mt ROM coal (ie a total of ~6.6 Mt ROM coal from Area 1 and Area 1 extension) from the proposal. This reduces the coal volume by 1.5%. Coal quality in Area 1 and Area 1 extension is some of the best coal in the deposit
	Environmental Avoids 32% (by area) of a patch of 179 *Zieria ingramii* plants 84 ha of woodland in Area 1 and Area 1 extension will not be removed including 2.13 ha of Grey Box Woodland Impacts lower quality agricultural land: 43.9 ha of Rural Land Capability Class IV and 22.51 ha of class VI land	**Environmental** None

Option	Advantages (compared to Option A)	Disadvantages (compared to Option A)
D. Remove north-east section of mining area (Area 1) – replace with Area 2	**Mining** Area 2 has the same strip ratio as in Area 1 (1:3.67) Coal in Area 2 is a 'measured resource' Access to ~6 Mt ROM coal **Environmental** 1.2 ha of Grey Box Woodland EEC and 60 ha of other woodland in Area 1 will not be removed Impacts lower quality agricultural land: 68 ha of Rural Land Capability Class IV land	**Mining** Removes ~4 Mt ROM coal in Area 1 from the proposal Potential geotechnical risks, investigation required due to the high wall angles near creek line **Environmental** 5.7 ha of Fuzzy Box Woodland EEC and 3.1 ha of Box Gum Woodland EEC will be removed No other woodland will be removed One Aboriginal scarred trees of low significance will be impacted
E. Remove north-east section of mining area (Area 1) – replace with Area 3	**Mining** Strip ratio of 1:2.7 Access to ~8 Mt ROM coal Coal in Area 3 is a 'measured resource' **Environmental** 1.2 ha of Grey Box Woodland EEC and 60 ha of other woodland in Area 1 will not be removed	**Mining** Removes ~4 Mt ROM coal in Area 1 from the proposal Close to the coal seam loxline therefore coal quality degrades Area 3 has higher strip ratio and hence is less economic than Area 1 and 2 Exploration intensity has been lower in Area 3 compared to Area 1 or 2 **Environmental** Close to the creek - water modeling has not been conducted in this area 0.2 ha of Box Gum Woodland EEC and 7.4 ha of Grey Box Woodland EEC will be removed 13 ha of other woodland will be removed

Option	Advantages (compared to Option A)	Disadvantages (compared to Option A)
F. Remove north-east section of mining area (Area 1) – replace with Area 4	**Mining** Access to ~8.9 Mt ROM coal	**Mining** Removes ~4 Mt ROM coal in Area 1 from the proposal Coal in Area 4 is an 'indicated/inferred resource' Strip ratio of 1:5.32 Mining in Area 4 is the least preferential as it is high strip ratio, lower quality and longer hauls to the coal preparation plant than any other area therefore the coal trucks will not be able to fully loaded due to tkph (tonne kilometre per hour) problems (tyre heating)
	Environmental 1.2 ha of Grey Box Woodland EEC and 60 ha of other woodland in Area 1 will not be removed	**Environmental** 0.1 ha of Grey Box Woodland EEC will be removed Impacts higher quality agricultural land: 53 ha of Rural Land Capability Class III land

Figure 5.3 Mine Plan Revisions to Minimise Impacts on Remnant Vegetation Corridor



6. PAC RECOMMENDATION – NORTHERN PORTION OF MINING AREA B

This PAC recommendation relates to nine *Tylophora linearis* located in the centre of the northern portion of mining area B in two small sub-populations, refer Figure 6.1.

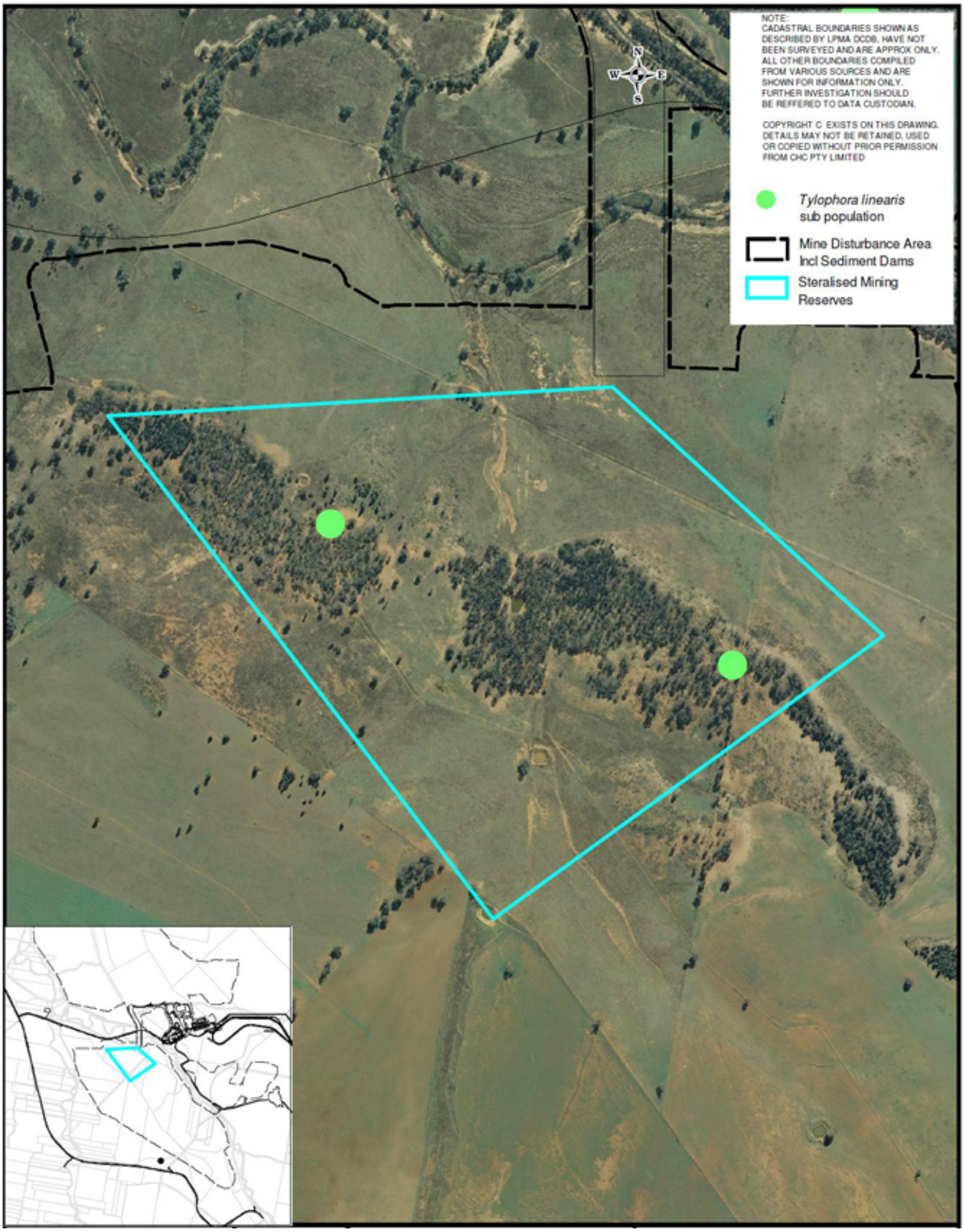
The PAC recommended that additional offsets for the impact of *Tylophora linearis* or mining be set back in this area.

This area contains 11.6 Mt of ROM coal which would be sterilised as a result of this setback and, as indicated in Appendix 1, replacement coal would be mined down dip to the west, an area with higher stripping ratios. Thus this would place an additional cost burden on CHC as well as increase in greenhouse gas emissions and size/height of waste emplacements.

However, since the compilation of the PPR&RTS targeted surveys to identify *Tylophora linearis* in the biodiversity offset areas were undertaken. A total of 45 individual *Tylophora linearis* were located and will be protected. This will provide an offset ratio (impacted:protected) of 1:5 which meets or exceeds the OEH and SEWPAC offset guidelines.

See also CHC response to Recommendation 17.

Figure 6.1 Tylophora Linearis Locations in Mining Area B



7. OTHER PAC RECOMMENDATIONS

7.1 Minimising Dust

The revised mine plan has the following characteristics that will reduce dust generation:

- Lower total length of haul roads open at any one time by not commencing mining area B until mining area A is completed. This also has dust suppression water savings
- Smaller footprint of open waste rock emplacement areas therefore reducing the areas of non-rehabilitated land due to mining the northern area first rather than three areas simultaneously.

The assessment of dust and noise impacts for the revised mine plan is addressed separately in the Response to Recommendations of the Planning Assessment Commission Review Incorporating a Revised Preferred Project Report.

7.2 Maximising Land Capability

The revised mine plan has the following characteristics that will maximise land utilisation:

- Maximising the land capability and productivity of the rehabilitated final landform by mining area A and C first and commencing rehabilitation progressively and finalising by year 17
- B-OOP E including cover over the out-of-pit tailings dam is rehabilitated by year 15 and could be used for agricultural purposes by year 17.

Further synergies between land capability on CHC-owned land, including the revisions to the mine plan will be considered in the development of the integrated land management plan.

7.3 Final Void

The revised mine plan has reduced the number and extent of the final voids. Previously, it was proposed that two of the three mining areas, areas A and C, would be back-filled to above the final water table with a void lake left in the third mining area A. The revised mine plan eliminates the voids and remaining highwalls in mining areas A and C and keeps the final void in mining area B.

Numerous configurations of the final void in mining area B have been examined to further reduce the size of the final void from that presented in the PPR&RTS while ensuring that the void lake remains a groundwater sink and does not spill into Laheys Creek. It has not been possible to reduce the size of the lake (based on surface area of the lake). The final void will contain a saline lake with an equilibrium lake level of RL 377.5 m that will have 6.5 m of free board (based on a conservative scenario).

Thus the total mine final void area including high walls has been reduced from 143 ha to 118 ha ie an 18% reduction.

7.4 Exposed Area Rehabilitation

The mine staging diagrams, Figure 3.1 to Figure 3.6, demonstrate that waste emplacement areas are designed to ensure rehabilitation begins in Year 2. Final landform shaping will begin as soon as the waste rock emplacement has reached its second height lift (15 m above topography).

The mine's waste emplacement strategy was developed so that areas would not require temporary rehabilitation and direct topsoil placement would occur as soon as areas were available to maintain the integrity of the material.

8. CONCLUSIONS

The recommendations by PAC have been assessed and implemented where possible in this revised mine plan. The revised mine plan:

- Reduces the impacts on threatened species and endangered ecological communities by reducing the size of the B-OOP E overburden dump
- Expands and enhances 185 ha of remnant wooded corridor to the north and north east through natural and assisted revegetation
- Reduces the land area that would be exposed at each stage of mining
- Reduces the final voids and high walls to mining area B only and reduces the final void area significantly
- Reduces the length of operational haul roads required and hence reduces the dust suppression water requirements
- Reduces the land area requirements for mine support infrastructure and the coal handling and preparation plant
- Reduces the noise and air quality impacts generally and specifically reduces the noise impacts associated with the mine support infrastructure maintenance activities (short and sharp sounds) as the new location is further away and more hidden from the nearest receptors

CHC has considered the PAC recommendations in formulating the revised mine plan. Whilst it is possible to maintain two active mining areas as opposed to three, CHC is making significant optimisation concessions. The revised mine plan reduces flexibility if the geological model is inaccurate or has a greater margin of error than allowed for. As a result, this will increase in coal blending and stockpiling activities and hence cost to ensure that projected coal quality specifications are satisfied.

CHC believes that the concessions and additional commitments associated with the revised mine plan are an acceptable compromise to meet the PAC Review report recommendations.

APPENDIX 1

Typical Coal Sterilisation Impacts

Coal Reserves

If all of the remnant vegetation in mining area C and in the north of mining area A (Pits 120-132) are avoided, it would result in 131.3 Mt ROM of coal being sterilised, refer Table A1. This coal would have to be replaced by additional coal from westward extensions to mining area B (ie extensions to Pits 101-104).

There is limited geological drilling and modelling to the south of the mining area B boundary and hence no detailed mine Reserves calculations have been conducted in this area.

Based on the Reserves and coal quality information available at the southern boundary of mining area B, it can be expected that the strip ratio will be significantly higher than mining area C. Coal quality is also questionable, with some thinning of coal plies and reduction of quality to the south.

Table A1 Impact of Avoiding North Eastern Area

Pit	Waste (Mbcm)	ROM Coal (Mt)	ROM Strip Ratio	Product Strip Ratio (bcm/t)
PIT 120	17.3	8.7	1.99	3.12
PIT 121	77.8	30.8	2.53	4.27
PIT 130	50.7	28.2	1.80	3.26
PIT 131	61.9	28.6	2.16	4.30
PIT 132	82.5	35.0	2.36	4.39
TOTAL	**290.2**	**131.5**	**2.21**	**3.95**

Mining Cost Impact

The impact of replacing the 131.3 Mt ROM lost from mining area C and the northern extent of mining area A:

- Strip ratio would increase from 3.1:1 in mining area C to approximately 6:1 in extensions of mining area B
- Waste removal requirement rises from 394 Mbcm to 788 Mbcm, an increase of 394 Mbcm
- Coal haulage to the ROM from the extended mining area B is double the distance required from mining area C.

Using an average waste cost of $4.00 (based on modelled costs) this represents an additional project cost of $1,576M. For coal haulage, the increased distance would also increase the coaling cost from $2.50/t ROM to $4.00/t ROM or result in an additional $197M in coal haulage costs.

Therefore, the financial impact for the replacement 149.5 Mt sourced from extensions to mining area B is estimated to be an extra $1,773M or ~$13.5/t product.

Truck Numbers/Greenhouse Gases

Table A2 shows the increase in truck numbers for the current mine plan relative to the revised waste haulage requirements by extending mining area B and relocating the out-of-pit dump further south. Apart from increased capital to purchase the additional trucks, there are also increased operational cost increases in labour and maintenance.

Table A2 Revised Truck Numbers Impacts

Number of Trucks	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Current mine plan	9	7	8	4	9	
Southern mining area extensions	11	10	11	4	10	
Delta	**2**	**3**	**3**	**-**	**1**	
Estimated extra GHG emissions (t)	**4,025**	**6,038**	**6,038**	**-**	**2,013**	**18,114**

Assumptions		
GHG Emissions	(Fuel Qty*Energy Content*Emission factor)/1000	
Truck diesel consumption	137	L/h
Hours per year	5,500	Hr
Energy content of diesel	39	GJ/kL
Emission factor of CO_2	69	Kg CO_2-e/Gj
ie GHG emissions/truck	2,013	t CO_2-e

Waste Dump Impacts

Associated with the increase in waste of 394 Mbcm (or 493 Mbcm with swell factor applied) is the issue of where to place it. Some can be placed in-pit, but the majority will have to be out-of-pit. If it is assumed that half can be accommodated in-pit, an additional ~256 Mbcm of out-of-pit placement or an area approximately 3.5 times the previous B-OOP E will be required. This has significant ecological and land management impacts.

Summary

Mining area C and the northern extent of mining area A contains some 114.5 Mt of coal that if sterilised would require CHC to extend mining area B westwards. An assessment of this change indicates the following impacts:

- Additional cost burden of $1,773M or $13.5/t product
- Additional 18,000 t of greenhouse gas emissions released over five years
- Footprint and land management issues associated with trying to locate an out-of-pit dump approximately 3.5 times the size of the existing B-OOP E, in an economic location, whilst avoiding ecologically sensitive areas.

Appendix B

Tailings management review

Cobbora Holding Company Pty Limited

Tailings Management Review Report

OWNERSHIP AND PERMITTED USE

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FIGURES

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1. INTRODUCTION

Recommendation 19 of the Planning Assessment Commission (PAC) Review report requests that the water management plan includes the principle that tailings be treated mechanically in order to minimise water requirements for the Project unless the Proponent can demonstrate that an alternative would satisfy best practice standards. Further the PAC report states:

"Based on the analysis provided by the Proponent…..solid bowl centrifuge technology appears to be a promising option worthy of detailed consideration" (Section 8.1.1)

This report reviews the basis for the PAC concerns that CHC were not targeting to minimise water usage ie Dr Perrens findings and presents a discussion of how best practice standards have been met to derive a preferred dewatering technology.

2. ADDITIONAL INFORMATION

Since the Preferred Project Report/Response to Submissions (PPR&RTS) was issued in February 2013, CHC has completed additional sizing, thickener and rheology testwork. The impact of the testwork affects some of the conclusions drawn from the PPR&RTS Appendix C Dewatering Options Report.

The results of the sizing testwork has allowed CHC to adjust the block model, which forms the basis of the mine plan, to include the ultra-fine rejects percentage ie tailings, as an attribute. As a result, actual tailings tonnages can be more accurately predicted on a yearly basis and the uncertainty about tailings quantities is largely removed. This analysis indicates a lower average tailings disposal requirement.

The thickener and rheology testwork relate specifically to Tailings Storage Facilities (TSF) based options. Thickener testwork is aimed at determining the density of the slurry that is achievable coming out of the bottom of a thickener ie its underflow. Rheology testwork is aimed at determining the resistivity of pumping slurries in pipelines at different densities to determine the optimum density for pumping.

The thickener testwork has indicated that an underflow density of up to 48% by mass is achievable, but based on the heterogeneous nature of the resource, an average thickener underflow density of 40% by mass is anticipated. This figure of 40% density is also confirmed as the acceptable density from the rheology testwork for the tailings pipeline. Note the results in some seams indicated a rheology of up to 42% is achievable. The application of the 40% slurry density helps reduce the overall water demand.

As a result of Recommendation 4 of the PAC review report, CHC have developed a revised mine plan (refer CHC's Revised Mine Plan Report M01-CHC-100-RP-ENV-0001). The calculations in this report are based on this revised mine plan.

3. DR PERRRENS' FINDINGS

3.1 Main Findings

Section 8.1.1 of PAC review report suggests that Dr Perrens has made a 'compelling case' for the introduction of mechanical dewatering. The foundations of the case against traditional tailings disposal by means of slurry are:

- An apparent high risk of water shortage during dry climate conditions attributed to uncertainty identified by Dr Perrens concerning individual demand components of the water balance
- The assertion that base costs are 'in line' with mechanical dewatering options costs.

3.2 Water Balance Assessment

3.2.1 Water Demands

Dr Perrens suggests the quantity of water imported during a 'dry climate' scenario is optimistically low. Uncertainty in water balance components is the reason cited. Each component identified by Dr Perrens is discussed below with respect to the revised mine plan, recently obtained testwork data and/or operational changes:

1. Water to Tailings

   Based on historical coal analysis data from CHC's 12 large diameter boreholes (LD01 to LD12) and the latest testwork (LD22A, LD23A and LD24A), the life of mine averages of ROM mass by seam reporting to tailings (ie <125 µm size fraction) in the revised mine plan are:

   - Fly Blowers – 4.9% (Range: 3.8 – 5.6%)
   - Ulan Upper – 4.1% (Range: 3.0 – 5.7%)
   - Ulan Lower – 4.5% (Range: 3.8 – 3.8%)

   The average life of mine across all seams is 4.5% of ROM mass reporting to tailings. It should be noted these tailings percentages are not affected by yield. However the total ROM feed is reduced as yield increases.

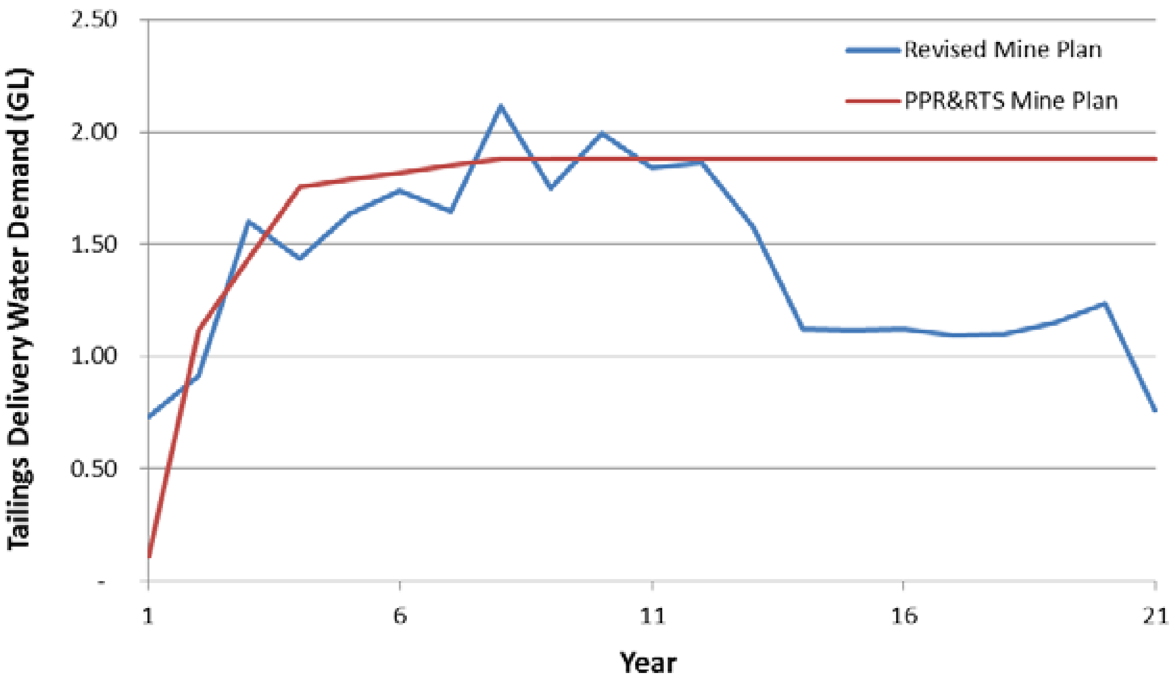
   The washery water balance undertaken by QCC Resources for the PPR&RTS adopted a higher value, an average 5.5% of ROM mass reporting to tailings.

   The revised mine plan, combined with the planned increase of operational thickener underflow density to 40% by mass, have significant impacts on water demand for pumping of tailings. Life of mine water demand decreases from 36.2 GL reported in the PPR&RTS to 29.6 GL based on the proposed operational changes. Figure 3.1 shows that water savings occur in 11 of the 21 years of mine life, with a peak saving of 1.1 GL/y in year 21. The peak increase in demand is 0.23 GL/y in year 8.

2. Dust Suppression

   The PPR&RTS water balance assessment estimated dust suppression demand on a long-term 'average climate' basis. Dr Perrens determined that demand would increase by up to 15% during a 'dry climate' scenario. The increase is considered by Dr Perrens to be of a magnitude that could be offset by the introduction of chemical dust suppressants (water extenders).

Figure 3.1 Comparison of Water Demand to Pump Tailings



It is accepted that the use of dust suppressants will reduce dependence on imported water during 'dry climate' year(s). CHC understands that water demand reductions of 40% are readily achievable, which equates to a saving of up to 515 ML/y during the peak dust suppression demand. This provides a net demand reduction of 325 ML/y after accounting for uncertainty associated with the application of long-term 'average climate' data for 'dry climate' scenario.

In addition, due to the reduction from three mining areas to two or one in the revised mine plan, the haul distances and number of trucks has been reduced. Further CHC has gained an improved understanding of truck haulage patterns and unsealed road lengths which it has applied to the dust suppression water usage calculations.

As haul road dust suppression requires the most water, the demand for dust suppression water has been significantly reduced in the revised mine plan.

3. Seepage

Dr Perrens stated that seepage losses from dams are not accounted for in the water balance. CHC advises that:

- Sedimentation Dams: Approximately 50% of runoff captured in sedimentation dams during a 'dry climate' is harvested to mine water dams for re-use onsite (year 4 to year 20). The remaining 50% is susceptible to loss by seepage. This volume peaks at 144 ML in year 12. Assuming very high seepage rates of 10% of stored volume, a maximum of 15 ML might be lost annually
- Out-of-pit Tailings Storage Facility and Mine Water Dams: These facilities are lined with 900 mm of clay having a permeability of <1 x 10^{-9} m/s (EPA requirement). Seepage through a liner of this specification is considered negligible
- In-pit Tailings Storage Facilities: In-pit tailings storage facilities will be unlined and the majority of seepage reports to the mine water dam via pit dewatering systems

- Clean Water Dams: Clean water dams contribute less than 10 ML annually to site re-use during periods of low rainfall. The impact of clean water dam seepage losses on the 'dry climate' water balance has been assessed as negligible
- Raw Water Dam: The 'dry climate' scenario is characterised by higher demand for imported river water. During these periods the raw water dam will generally operate at less than 50% capacity. At this level, the maximum areal seepage interface would be 2.1 ha. Assuming a high infiltration rate of 4 mm/day, a maximum of 30 ML might be lost annually.

As shown in the above analysis, seepage losses may be as high as 45 ML/y or less than 1% of the peak water demand and are therefore considered negligible and within the accuracy of the water modelling.

4. Washery Losses

For completeness, the washery water balance inputs consist collectively of tailings return water, mine water, raw water and water included with material feed (inherent and free moisture). Washery water balance output consists collectively of coarse rejects water, product coal water and water to tailings.

An average of 42% of input water is lost to coarse rejects water and product coal water. These losses have been accounted for in the site water balance as net water quantities were used.

3.2.2 Water Sources

Dr Perrens also suggests uncertainty in water source components of the water balance model. Each component identified by Dr Perrens is discussed below with respect to the revised mine plan (refer CHC's response to PAC Recommendation 4):

1. Groundwater Inflow

CHC agrees with Dr Perrens finding that a large proportion (up to 50%) of groundwater inflow to mining areas would be lost to evaporative processes. CHC has implemented a plan to counter these water losses through the introduction of a mine de-watering borefield. It is anticipated the direct extraction of groundwater would compare favourably to the volumes of groundwater considered available in the water balance assessment presented in the PPR&RTS.

Note: A secondary benefit of the introduction of a mine dewatering borefield will be a reduction in the occurrence and extent of fugitive NO_2 fume clouds during blasting.

2. Imported River Water

There are two extraction zones in the Macquarie and Cudgegong Regulated Rivers Water Source - upstream of Lake Burrendong on the Cudgegong River and downstream of the lake on the Macquarie River.

CHC purchased a 1 GL/y high security Water Access Licence from an existing user in the zone upstream of Lake Burrendong.

In accordance with requirements of the *Water Management Act 2000 NSW*, CHC has purchased and transferred 2.31 GL/y of high security Water Access Licences from Macquarie River downstream of Lake Burrendong, to the upstream Cudgegong River.

A change of extraction zone is permitted in the *Water Management Act 2000 NSW* if the environmental and third-party user impacts are not significant and if the total extraction potential in the upstream zone does not exceed 40 GL/y across all users. The potential extraction in this case was less than 27 GL/y.

The environmental and third-party user impacts involves an assessment by NSW Office of Water and includes hydrological changes. For an approval to be granted, the Minister needs to be satisfied that the transfer of entitlement will not have adverse consequences.

Applications to change the authorised extraction zone were submitted in June 2010. Following NSW Office of Water's determination that the impact will not be significant, the change of extraction zone for Water Access Licences associated with 2.31 GL/y was approved by the Minister under Section 71S of *Water Management Act 2000 NSW* in June 2011.

3.2.3 Summary

The apparent uncertainties/inaccuracies identified by Dr Perrens concerning individual demand components of the water balance have been addressed and as a result of the revised mine plan and additional testwork:

- The life of mine water to tailings reduces from 36.2 GL to 29.6 GL but its peak, over a nominal period of two years, is estimated to increase by 0.23 GL/y
- Flexibility offered by dust suppressants provides an estimated net water demand decrease of 325 ML/y during 'dry climate' conditions
- Seepage from all storage types has been demonstrated to be negligible or minor.

With the introduction of mine dewatering borefield the uncertainty around water sources is also reduced.

Therefore CHC considers that potential for water shortages resulting from water balance uncertainty has been resolved, even in 'dry climate' years, within the constraints of the licensed water entitlements held by CHC.

3.3 Economic Assessment

The PPR&RTS included a section whereby the relative merits of differing dewatering technologies were assessed. Part of that assessment was an economic evaluation based on estimated capital and operating costs and using Net Present Value (NPV) techniques.

The basis of those evaluations was the assumption by CHC that the mechanical dewatering options require an out-of-pit tailings storage facilities for use in the event of breakdown. Dr Perrens identified the provision for increasing the capacity of this storage in mine years 9 and 13 as unnecessary due to the availability of in-pit tailings disposal options.

Table 3.1 shows the amended NPV for mechanical dewatering with calculations based on in-pit disposal of tailings for mine years 9 and 13 and the deletion of the associated capital.

The NPV ranking indicates the base case option of tailings emplacement remains the most economic option. The NPV differential to the solid bowl centrifuge option exceeds $4M.

Note: This economic justification is revised in Section 4.5 based on the revised mine plan and impact of testwork.

Table 3.1 NPV without Tailings Out-of-pit Emplacement Lifts

Tailings Option	NPV ($M)	Ranking	Variance from Base Case ($M)
Base Case - Tailings Emplacements	-$179.8	1	0
Solid Bowl Centrifuge	-$184.3	2	-$4.5
Paste Thickener	-$185.0	3	-$5.2
Secondary Flocculation	-$198.0	4	-$18.2
Belt Press Filter	-$216.0	5	-$36.2
Pressure Filter	-$229.9	6	-$49.1

Based on Nominally 10% ROM Feed, with the raising of the emergency out-of-pit TSF removed and no replacement in-pit cost

3.4 Water Quality Impacts

CHC confirms that the Water Management Plan currently being developed has included as part of its basis of design that:

- All contaminated water (runoff from pit, plant and all mine active areas) will be captured and retained on-site within the mine water dams and recycled back to the CHPP
- Where water from sedimentation dams does not comply with discharge criteria then it will be transferred to the mine water dams.

3.5 Other Issues

A number of other water modelling and balance issues were raised by Dr Perrens which are addressed separately by Parsons Brinckerhoff in Appendices F and G of the Response to Recommendations of the Planning Assessment Commission Review incorporating a Revised Preferred Project Report. These specifically address the following issues raised in the PAC review report:

- Life of mine impacts on surface water and groundwater
- Final void water balance
- Design criteria of flood protection of the overburden dump on the western side of Laheys Creek.

4. BEST PRACTICE STANDARDS

4.1 General

The term "best practice standards" when applied to tailings management encompasses not just water conservation but also other factors including environmental, economic, risks, etc. CHC assessment used the following to determine "best practice standards":

- Water conservation
  - Water resources
  - Tailings water requirements.

- Greenhouse gas emissions
- Disturbance footprints including flora and fauna impacts
- Economic
- Technology risks including safety.

These factors are used to compare the options for tailings management to derive "best practice standard" for use on this Project. The options considered in the tailings management plan are:

- Tailings Storage Facilities (TSF) based options
  - Tailings emplacements
  - Secondary flocculation.
- Mechanical dewatering based options
  - Solid bowl centrifuge
  - Paste thickener
  - Belt press filter
  - Pressure filter.

These are the same options as those presented in the PPR&RTS and therefore the descriptions of the options and associated technologies are not repeated in this report. However it is important to note that the mechanical dewatering options involve tailings being trucked to dump locations as opposed to the first two which are pumped to the TSF.

The analysis presented in this report is Project specific. The amount of tailings, its variability, the location of various facilities in both distance and height and environmental constraints all affect the analysis.

4.2 Water Conservation

4.2.1 Resources

Surface water imported from off-site, surface water captured on-site and groundwater captured onsite are each made available for mining purposes through provisions of the *Water Management Act 2000 NSW*. Water access and trading rules for these water sources are provided in respective Water Sharing Plans.

CHC has implemented the following range of "best practice standards" to the procurement of water entitlement in accordance with the Water Sharing Plans:

- Prior to the commencement of mining, entitlement held in affected water sources will be equivalent to the peak life of mine utilisation rate
- There has been no creation of 'new' water as existing entitlement has been sourced entirely from water markets comprised of willing sellers.

CHC recognises the transfer of 2.31 GL/y of high security Water Access Licences between extraction zones in the Macquarie and Cudgegong Regulated Rivers Water Source received particular scrutiny from local council and community groups. In accordance with the legal requirements the change of extraction zone was assessed by NSW Office of Water as not having significant environmental or third party impacts, including hydrological changes.

The Water Access Licences will be used efficiently and as much water as practical will be recycled on-site to reduce the amount of water make-up from the river. Further, CHC has entered into an extraction strategy agreement with State Water Corporation to help reduce transmission losses in the Cudgegong River and maximise the use of excess flows in the lower reaches of the river.

High capacity pumps have been included in the design to allow up to 70% of the water to be extracted from unregulated high flow events. Such high flow events occur without releases from Windamere Dam and modelling shows that even in dry years, up to 58% of the licenced water can still be captured from these events.

4.2.2 Tailings Water Usage

The difference between the tailings options outlined in the PPR&RTS was the water recovery of mechanical dewatering when compared to slurry based TSF based options. These were assessed and ranked in order of water recovery as part of the PPR&RTS, refer QCC report Comparison of Dewatering Options and the ranking from that report is shown in Table 4.1.

Table 4.1 Dewatering Capabilities Comparison

Tailings Option	Ranking	Average Net Usage ML/y	Average Net Savings ML/y
Solid Bowl Centrifuge	1	452	699
Pressure Filter	2	452	699
Belt Press Filter	3	625	525
Paste Thickener	4	849	302
Secondary Flocculation	5	995	156
Base Case - Tailings Emplacements	6	1,151	-

The water required to transfer tailings (via slurry) to the tailings emplacements is minimised by increasing the operational thickener underflow density ie the percent solids of the slurry.

As indicated in Section 2, an increase has been achieved through new understanding gained from recent testwork on Cobbora samples by thickener manufacturers. It is now apparent that a pumping density of up to 48% by mass is achievable, but based on the heterogeneous nature of the resource, an average thickener underflow density of 40% by mass is anticipated. This compares to the average 35% by mass benchmark used for PPR&RTS calculations.

Modifying the thickener underflow density to mine specific results is considered by CHC as “best practice standard” for tailings management.

Also included in Table 4.1 is the average annual water usage over the life of mine based on the revised mine plan and testwork and the potential water consumption savings per annum using alternative technologies. As indicated, water usage savings in excess of 50% are possible using solid bowl centrifuge or pressure filter dewatering.

4.3 Greenhouse Gas Emissions

Greenhouse gas emission rates generated by the various dewatering options considered for the Project are ranked for comparison in Table 4.2. Emissions are based on tailings quantities for the revised mine plan and include operational electricity demand and on-site combustion of diesel fuel for transporting dewatered tailings.

Table 4.2 Life of Mine Equivalent Greenhouse Gases Produced Comparison

Tailings Option	LOM t CO_2-e (kt)	Ranking	Var Base Case (kt)
Base Case - Tailings Emplacements	50.4	1	0
Pressure Filter	54.6	2	4.3
Secondary Flocculation	55.8	3	5.4
Paste Thickener	71.1	4	20.7
Belt Press Filter	82.3	5	32.0
Solid Bowl Centrifuge	205.7	6	155.4

Note: Emissions from the production of flocculants and their transport to site have not been included

The calculations inherent in the table are site specific and the conclusion could be different for different projects as power demand (and hence greenhouse gases emissions) is affected by the quantity of material to be treated, densities, slurry rheology and location of the TSF height as well as distance.

The emissions which the Project can influence ie Scope 1 and 2 type greenhouse gas emissions, is approximately 7.7 Mt CO_2-e and therefore the impact of moving to more energy intensive mechanical dewatering technologies is minimal (<1%) except in the case of solid bowl centrifuges.

The solid bowl centrifuges suggested by Dr Perrens, generate a four-fold increase in emissions over the base case. For this option, more than 205,000 t CO_2-e is produced by the off-site power generation required to operate equipment or 2.7% overall direct project greenhouse gas emission.

The storing of tailings in specifically designed impoundments results in the lowest generation of greenhouse gas emissions. This contributes to the rationale that TSF are currently the best practice standard for coal mines in NSW.

4.4 Disturbance Footprints

None of the options presented have a significant difference in disturbance footprint as all of the tailings for the Cobbora Coal Project are co-disposed with the waste or covered by waste. The TSF options may incur a slight penalty in that the final bulk density of the tailings may be lower and hence require a larger footprint.

If, as on some other Hunter Valley mines, the TSF were to be separate facilities and not located in-pit or covered by waste then the disturbance footprint would be larger with potentially greater environmental impacts.

The reduction in tailings generation associated with the revised mine plan and recent coal seam testwork data, has allowed for rationalisation of tailings management resulting in a single out-of-pit storage. The key design factor affecting this decision relates to confirmation of an average 5.0% of ROM mass (which includes 10% contingency) reporting to tailings.

To take advantage of in-situ clay reserves, the east out-of-pit TSF will be retained in preference to the west out-of-pit TSF. The out-of-pit waste emplacement will occupy the existing footprint of the west out-of-pit TSF. The volume gain at this location provides an opportunity to limit the spatial extent of the out-of-pit waste emplacement elsewhere.

Tailoring the TSF capacity to mine specific resource data and having the TSFs located in-pit and only one out-of-pit covered by waste is considered “best practice standard” for tailings management if a TSF based option was selected.

None of the options presented have a significant difference in impact footprints as all of the tailings are co-disposed with the waste or covered by waste. As such the impact on flora and fauna is associated with the location of waste dumps and not the selection of a specific tailings management option.

The impact of waste dumps is discussed in CHC's Revised Mine Plan Report, M01-CHC-100-RP-ENV-0001.

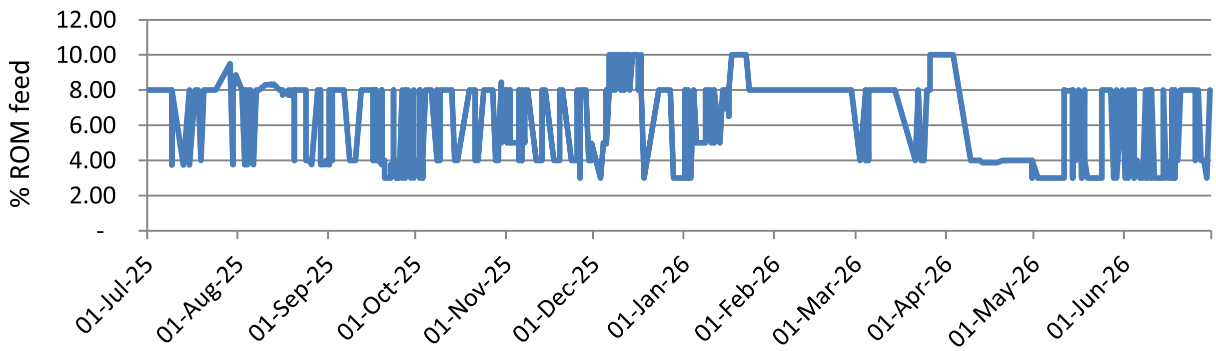
4.5 Economic Assessment

4.5.1 Net Present Value

The economic assessment calculations (NPVs) have been re-calculated for the revised mine plan and the recent coal seam testwork for all tailings management options.

Whilst the average fine rejects is expected to be 5.0% including a 10% contingency and is used to determine operating costs, the equipment selection and hence capital costs for dewatering are based on 10%. This is required to manage periods where up to 10% of ROM mass is reporting to tailings for extended periods due to the variability with the resource. The variability for a typical year (year 10) is shown in Figure 4.1.

Figure 4.1 Percentage of ROM Mass Reporting to Tailings for Year 10



The revised mine plan which incorporates the reduction in the amount of tailings generated has allowed rationalisation of the tailings management plan resulting in a single out-of-pit storage (down from two) and three in-pit storages (down from six).

The revised NPV ranking for the options under consideration is shown in Table 4.3 and this indicates the base case tailings emplacement option remains the most economic. The table also includes the average annual operating cost of each of the options on a $/t product basis.

In terms of average annual operating costs, there is an order of magnitude difference in operating costs which is small on $/t basis (up to $0.44/t) but equates to substantial sums on a cost per year basis. For example solid bowl centrifuge costs are $4.1M/y and the belt press filter cost are $5.9M/y vs the base case of $0.5M/y.

In terms of the total processing (non-mining) costs of $4/t these operating costs represent an increase from 1% to 8% for the solid bowl centrifuge costs and to 11% belt press filter.

Table 4.3 NPV for Revised Mine Plan Comparison

Tailings Option	NPV ($M)	Ranking	Variance Base Case ($M)	Operating Cost $/t
Base Case - Tailings Emplacements	-$116.3	1	0	$0.05
Secondary Flocculation	-$121.6	2	-$5.2	$0.18
Solid Bowl Centrifuge	-$134.8	3	-$18.5	$0.34
Paste Thickener	-$135.5	4	-$19.2	$0.49
Belt Press Filter	-$149.1	5	-$32.7	$0.30
Pressure Filter	-$184.0	6	-$67.7	$0.34

Based on Nominally 5.0% ROM Feed per New Mine Plan, with the Raising of the Emergency Out-of-pit TSF Removed and no Replacement In-Pit Cost (revised mine)

Also in relation to the solid bowl centrifuges, the start-up capital of this option is double that of the TSF option and power use is over four times higher.

Table 4.3 when compared to Table 3.1:

- Demonstrates the reduction in life of mine costs for all options as a result of reducing the quantity of tailings generated using the updated coal testwork data
- Shows the second ranked option changes from solid bowl centrifuge to secondary flocculation
- Indicates that the NPV differential to the solid bowl centrifuge option has increased to in excess of $16M which is a function of its capital cost to handle the variability up to 10%.

Significantly, it also shows that the difference between the mechanical dewatering options and TSF based options has widened, as the number of TSF's have been reduced.

The above NPV calculation excludes the costs associated with purchasing the water entitlements referred to in Section 3.2.2. As these entitlements have been purchased in an open market, CHC believe it should be able to use them to support the Project and not have further covenants placed on their use, as this would be contrary to the intent of the *Water Management Act 2000 NSW*.

4.5.2 Water Savings Value

An alternative economic assessment is to value the water saved for the different options compared to the cost of realising these savings.

One method of determining the value of water is to assume that all of the water used is made up from the Cudgegong River. Note this only occurs in 'dry climate' years. Recent market purchases of water entitlements from the Cudgegong River put the cost of obtaining the ~3.3 GL/y at $300,000/y and this combined with pumping station operating cost gives an estimate cost of water at $480/ML.

The water savings identified in Table 4.1 and the cost of water derived above is then used to derive the monetary value of the water savings and these are shown in Table 4.4.

Table 4.4 Water Savings vs Operating Cost Comparison

Tailings Option	Average Water Saving (ML/y)	Value of Resultant Water Savings ($M/y)	Increase in Average Operating Cost ($M/y)	Multiplier or OPEX/Savings	Equivalent Cost of Water Saved ($/ML)
Solid Bowl Centrifuge	699	$0.34	$3.6	10.5	$5,070
Pressure Filter	699	$0.34	$3.1	9.1	$4,390
Belt Press Filter	525	$0.25	$5.3	21.0	$10,160
Paste Thickener	302	$0.15	$3.5	24.2	$11,730
Secondary Flocculation	156	$0.08	$1.7	22.0	$10,650
Base Case - Tailings Emplacements	-		$0		

Table 4.4 shows the increase in average annual operating costs over the base case of tailings emplacement and indicates multipliers and equivalent cost of water that would be required to offset the increase in operating costs. For example, the increase in the annual operating cost of using solid bowl centrifuges would be $3.6M more than the base case tailings management. However, the water saved would only be worth $0.34M.

Table 4.4 re-enforces the NPV based assessment that indicates that tailings emplacement is preferred economic option and that the cost of water would have to increase by an order of magnitude to offset the increase in operating costs of the different options.

4.6 Technology Risk

4.6.1 High Level Assessment

The technology risks associated with each of the options was not addressed in the PPR&RTS review of dewatering options and as such is discussed in this section.

Of the technologies offered, all have been used in similar applications ie fine dewatering with varying degrees of success depending on the product being dewatered and the level of operator and maintenance support. The exception is high capacity solid bowl centrifuge technology which is a recent improvement of a 1980’s technology and its application to any industry in Australia and its use on coal specifically is unproven.

The technologies respond differently to changing process parameters such as the presence of clays and the increasing percentage of slimes (<38 µm size fraction). Thickener based applications accommodate these changes by varying the flocculant addition rates. However the more complex technologies ie solid bowl centrifuge, belt press filter and pressure filter encounter binding issues in the presence of clay. Similarly, if the slimes content increases then filter cloth selection needs to be finer which results in reduced capacity and hence the requirement for more and/or larger machines to be installed.

The more complex technologies ie solid bowl centrifuge, belt press filter and pressure filter also require a higher level of operator interaction and incur higher maintenance costs. The maintenance tasks are also correspondingly more complex. This becomes a risk, especially in coal preparation plants, as this type of equipment is an order of magnitude more complex than for example a coal sizer with its auxiliary drives. The risk is not only with respect to operator and/or maintainer error but also the safety implications of an error on the process.

Secondary flocculation and paste thickeners, whilst simpler technologies, do carry risks associated with over flocculation and resultant "bogging" of equipment and pipelines due to operator error. "Debogging" of equipment and pipelines also introduces its own risks with respect to undertaking these tasks safely.

There are operational risks associated with TSF related to wall failures, pipeline failures, etc. There are also operational risks associated with transporting the mechanical dewatered product by truck and placing the mixed product in waste dumps eg carry back on trucks, dust generated by dry carry back, sloppy cake leading to dump stability issues, etc. These are considered to be equivalent in terms of risks.

A risk assessment and the resultant ranking of the technological risk of the options is indicated Table 4.5. This shows that the simpler TSF based options are preferable to mechanical dewatering options.

Table 4.5 Risk Comparison

Risk Description	Impact on Value	Base Case - Tailings Emplacem'ts	Secondary Flocculation	Belt Press Filter	High Capacity Solid Bowl Centrifuge	Paste Thickener	Pressure Filter
Technology proven	Positive						
Technology proven on coal tailings	Positive						
Process impacts eg clays, high portion of slimes, variability, etc	Negative						
Engineering complexity and hence reliability	Negative						
Operator interface/control complexity	Negative						
Maintenance complexity	Negative						
Operating cost creep eg consumables, personnel numbers, etc	Negative						
Capital cost creep eg growth due to variability in tailings, consumables, etc	Negative						
Safety - failure mode of equipment	Negative						
Safety - failure mode of tailings placement if dewatering target not met	Negative						
Power outage impact	Negative						
Escalation exposure eg labour, power, consumable price increases	Negative						
Average Score		**3.0**	**2.8**	**1.8**	**1.4**	**2.6**	**2.0**
Ranking		**1**	**2**	**5**	**6**	**3**	**4**

4.6.2 *Additional Testwork*

As the Project develops, the knowledge bank of the resource and its coal qualities through additional drilling and monitoring of actual operational outputs will also grow. This knowledge growth will also involve an ongoing testwork program in order to mitigate risk and maximise operational efficiency. In particular, the coal quality testing and further understanding of the rheology will contribute to a greater understanding of the suitability of tailings management options.

In parallel, CHC will be developing a better understanding of groundwater and surface water conditions and interactions through its monitoring of water bores and installation of addition bores for construction water, mine dewatering, stock water bores, etc.

This will contribute to the refinement of the water management plan as the Project develops.

CHC is committed to monitoring the development of dewatering technologies including undertaking testwork and piloting if a technology appears to be environmentally and economically promising. As a minimum, a feasibility study will be undertaken to determine the preferred dewatering option before in-pit tailings placement is required.

4.7 Summary

The discussions above about the issues around "best practice standards" identified a number of issues to assess and where the options differed they were ranked. For ease of discussion the previous comparison tables have been combined in Table 4.6 and an unweighted total added.

Table 4.6 Summary Comparison Matrix

Tailings Option	Water Recovery Ranking	GHG Emission Ranking	Economic Ranking	Risk Ranking	Unweighted Total
Base Case - Tailings Emplacements	6	1	1	1	9
Secondary Flocculation	5	3	2	2	12
Belt Press Filter	3	2	5	5	15
Solid Bowl Centrifuge	1	6	3	6	16
Paste Thickener	4	4	4	3	15
Pressure Filter	2	5	6	4	17

Table 4.6 shows that:

- TSF based options are lower ranked than the mechanical dewatering options
- Tailings emplacement is the lowest ranked option
- In terms of water recovery, tailings emplacement is the least efficient but, as indicated in Section 4.5.2, the value of water recovered from other mechanical dewatering technologies exceeds the cost of operating those facilities
- Solid bowl centrifuges, whilst the most efficient for water recovery, carries the highest greenhouse gas emission levels, operating cost and technology risk ie it remains unproven.

Therefore, in light of CHC's purchase of Water Access Licences and their prescribed use, CHC believe that the use of tailings emplacement meets the requirement of "best practice standards".

5. RECENT ENVIRONMENTAL ASSESSMENTS

It is noted that recent environmental assessments, namely Watermark and Boggabri, have committed to mechanical dewatering.

The selection of tailings dewatering technologies on a project should take into account a number of issues including:

- Process: eg presence of clays, the amount of tailings, its variability, % of slimes, etc
- Physical: eg the location of various facilities in both distance and height
- Environmental constraints: eg areas available and the presence of sensitive flora and fauna, water availability, greenhouse gas impacts, dust, etc
- Economic: eg NPV
- Risks: eg each company's attitude and/or ability to accept risks.

CHC have reviewed the Environmental Assessments for Boggabri and Watermark. It is difficult to gain a full appreciation of the above factors and hence the decision making that led to the preferential selection of belt press filters for tailings management.

The Boggabri coal resources are generally low ash, high volatile, high energy thermal coal with some seams exhibiting high volatile metallurgical coal characteristics. The Watermark coal is PCI or semi-soft coking coal with low ash 10% and secondary product with 18% ash. These coals would typically sell for 20% to 30% more than the Project coal at 24% to 26% ash. This allows more freedom to explore and implement alternatives to conventional tailings storage facilities.

In addition, Boggabri seems to have significant ecological constraints relative to the Cobbora Coal Project, as the mine is wholly located within a wooded area and hence would need to limit disturbance areas. The Cobbora Coal Project, whilst similar, does not have the same magnitude of ecological constraints, as the project application area has a comparatively greater proportion of historically cleared farm land.

6. CONCLUSION

The PAC review report, suggests that Dr Perrens has made a 'compelling case' for the introduction of mechanical dewatering based on:

- An apparent high risk of water shortage during dry climate conditions attributed to uncertainties in individual components of the water balance
- The assertion that base costs are 'in line' with mechanical dewatering options costs.

As a result of the revised mine plan and additional testwork:

- The life of mine water to tailings reduces from 36.2 GL to 29.6 GL but its peak, over a nominal period of two years, is estimated to increase by 0.23 GL/y
- Flexibility offered by dust suppressants provides an estimated net water demand decrease of 0.33 GL/y during 'dry climate' conditions
- Seepage from all storage types has been demonstrated to be negligible or minor
- With the introduction of a mine dewatering borefield, the uncertainty around water sources is also reduced.

Therefore CHC considers that potential for water shortages resulting from water balance uncertainty has been resolved even in 'dry climate' years within the constraints of the licensed water entitlements for the Project and as such mechanical dewatering is not required.

As indicated in the economic assessment Table 4.3, based on the Project's site specific parameters, the lower risk mechanical dewatering alternatives are at least 15% more expensive in NPV terms than tailings emplacement. The belt press filter alternative being proposed at Boggabri and Watermark is 22% more expensive than tailings placement. Thus mechanical dewatering costs on this Project are not 'in line' with tailings emplacement but significantly more expensive.

In the assessment, based on the revised mine plan, to determine "best practice standards" as applied to dewatering, CHC has reviewed:

- Water conservation
- Greenhouse gas emissions
- Disturbance footprints
- Economic
- Technology risks.

The assessment is location specific as the technology selection is influenced by:

- Process: eg the amount of tailings, its variability, presence of clays, etc

- Physical: eg the location of various facilities in both distance and height
- Environmental constraints: eg areas available and the presence of sensitive flora and fauna, water availably, greenhouse gas impacts, dust, etc
- Economic: eg NPV and cost of water vs operating costs
- Risks: eg each company's attitude and/or ability to accept risks.

Whilst the CHC proposed option of tailings emplacement is the least preferred in terms of water recovery, it generates the lowest amount of greenhouse gases (eg 40% less than belt press filters), is the most economic and offers the least technology risk to the Project. The multiple tailings emplacements (one out-of-pit in a waste rock emplacement and three in-pit) minimise the disturbance area. This is in contrast to many older mines that have developed tailings emplacements remote from the mining areas in otherwise undisturbed land.

CHC has also demonstrated the cost to implement solid bowl centrifuge tailings processing for the revised mine plan scenario is significantly higher than traditional tailings disposal eg:

- Annual operating costs are 7.5 times that of tailings emplacement
- The equivalent cost of water to offset increase in operating cost would be $5,708/ML vs an actual cost of $480/ML.

In addition, the application of high capacity solid bowl centrifuge is unproven in coal mining in Australia and therefore high risk.

Therefore, in light of CHC's purchase of Water Access Licences in the water market and their prescribed use, CHC believe that the use of tailings emplacement meets the requirement of "best practice standards" for tailings management.

CHC is also committed to minimising and recycling water usage on-site, to limit the necessity to draw on licenced entitlements. This is further supported by the extraction strategy agreement with State Water Corporation which will maximise the amount of water taken from operational surplus flows rather than releases from Windamere Dam.

As the mine develops, knowledge of the resource will increase, as will the understanding of the suitability of alternative tailings management technologies. CHC is committed to monitoring the development of dewatering technologies including undertaking testwork and piloting if a technology appears to be environmentally and economically promising. As a minimum before CHC begins planning for the move to in-pit tailings placement a feasibility study will be undertaken to determine the preferred dewatering option going forward.

Appendix C

Tailings storage facilities management plan

Cobbora Holding Company Pty Limited

Tailings Storage Facilities Management Plan

Document Control						
			Signatures:	(sign here)	(sign here)	(sign here)
0	31/01/2013	Submission	Approved for submission	G.O.	D.N.	E.H.
1	29/07/2013	Revision	Revised to reflect mine plan issued to DP&I 17 June 2013	G.O.	D.N.	E.H.
Rev No	**Date**	**Status**	**Revision Details**	**Originator**	**Verifier**	**Approver**
This document is not approved for use unless signed above by the Approver						

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1. INTRODUCTION

Tailings, are fine particles that generally contain a high proportion of non-coal material, including clays and other minerals. This document details how tailings will be stored in sites called emplacements for the life of the Cobbora Coal Project. It also describes how the tailings emplacement areas will be rehabilitated.

2. PROJECT BACKGROUND

Cobbora Holding Company Pty Limited (CHC) will design, construct and operate a series of tailings emplacements collectively known as Tailing Storage Facilities (TSF) over the life of the Project to store fine-grained (< 125 µm) slurry waste from the Coal Handling and Preparation Plant (CHPP). Each emplacement will have a design life incorporating a final decommissioning and rehabilitation phase. The overriding strategy will be to use a combination of multiple in-pit and out-of-pit storage emplacements to manage rise rates and hence settled densities. Rehabilitation will be an ongoing process throughout the life of the Project.

3. OBJECTIVES AND REGULATORY COMPLIANCE

3.1 Objective and Regulatory Bodies

The objective of the TSF Management Plan is to describe the development of tailings emplacement areas and their eventual rehabilitation to a final landform.

The design, construction, operation and rehabilitation of the tailings emplacements are regulated by the NSW Trade and Investment, Division of Resources and Energy (DRE) and the NSW Dams Safety Committee (DSC). They are the primary authorities that endorse the dam design. The NSW Environment Protection Authority (EPA) will also be a stakeholder in the performance of the emplacements and their impact on the environment.

The requirements of these regulatory authorities are outlined in the followings ections.

3.2 NSW Trade and Investment, Division of Resources and Energy

The establishment of tailings emplacements on coal leases are governed under Section 100 of the *Coal Mine Health and Safety Act*, 2002. The construction and use of the emplacement is covered in Section 102 of the Act, which requires that the emplacements must be:

- Constructed in accordance with sound engineering practice
- Compatible with the environment
- Kept secure.

An emplacement is considered to be ‘secure’ for the purposes of the Act if it is not unstable, it is not on fire and no noxious water is escaping from it. As a general rule, where an emplacement uses a dam to retain the tailings, DRE relies on the DSC to provide the lead regulatory role in determining whether it is stable or not.

TSF’s will need to be approved by DRE as part of the Project as a whole and any future modifications to the Project will also need to be approved.

Note the *Coal Mine Health and Safety Act,* 2002 is superseded by the *Work Health and Safety (Mines) Act* 2013 but the revisions to the section of the Act have not be promulgated at the time of writing this plan.

3.3 NSW Dams Safety Committee

The DSC regulates tailings emplacements if they include a dam as a means of retaining the tailings. A “prescribed” dam is one that is listed in the Appendix to the NSW Dams Safety Act 1978 and subject to continuing oversight by the DSC. The DSC takes an initial interest in a prescribed dam project when a final decision is made by the owner to proceed to construct a dam following approvals by planning and regulatory authorities.

3.4 NSW Environment Protection Authority

The EPA will issue the Scheduled Development Works Environment Protection Licence prior to scheduled development works proceeding (i.e. construction works) and the Scheduled Activity Environment Protection Licence prior to scheduled activities proceeding (i.e. mining related activities generally). These licences will regulate discharges to the environment, including surface water and groundwater.

4. SCOPE

CHC has set performance requirements for the design, construction, operation and rehabilitation of the TSF to meet regulatory and business objectives. These requirements consider any known restrictions or needs of the affected stakeholders. The general requirements are listed below:

- Comply with the DSC regulatory requirements
- Be secure, as defined by DRE
- Meet environmental standards
- Provide tailings storage of approximately 36 million m^3 (based on 55 % solids by volume) over the life of mine
- Maintain seepage rates at acceptable levels
- Design that balances constructability, economic and environmental considerations
- Maximise the volume of collected decant and seepage water returned to the mine water system.

5. TAILINGS MANAGEMENT

Tailings from the CHPP will be pumped through a pipeline to an out-of-pit TSF and in later years to the three in-pit emplacements. Refer to Appendix 1.

For each of these emplacements, the following will be applied:

i) A 2 m to 3 m deep crust will be established by transferring part of the deposition to the next TSF

ii) This remaining freeboard will be filled at a rise rate of 1 m to 2 m per year by routinely cycling deposition so that each deposited layer of tailings is fully exposed to the effects of evaporative beach drying. This is to make the upper few metres denser to create the desired crust strength.

Tailings will be mainly deposited from the back of the out-of-pit emplacement with the decant pond forming at the embankment wall. For the in-pit emplacements, the reverse will apply, beaching at the embankment with decant at the back. The appointed CHPP tailings coordinator will relocate the discharge point as necessary to optimise the tailings beach.

The decant pond water level will be minimised using a floating pontoon pump.

Return water (decant water) will be returned to the CHPP via the mine water system.

The recovery of water from the TSF is a priority of the mine operation. Ground conditioning of floor and embankment area within the out-of-pit TSF will be undertaken to limit seepage rates, and maximise water available for decant. Conditioning would typically take the form of a clay liner. Seepage that presents to the sand filter will be captured in a pond downstream of the embankment.

In-pit TSFs will be formed by mining plant using material from redundant mine access ramps. The walls will comprise emplaced mine waste of varying permeability and the embankments will utilise selected bulk waste to enable minimum DSC design criteria to be complied with. The storages will be generally underlain by a low permeability rock base. The high permeability walls will act as a preferred lateral pathway for seepage, which will be captured downslope by construction of collection drains and a sump in the pit floor rock base. Captured seepage will be transferred to the mine water system for dust suppression and re-use in the CHPP.

6. PROJECTED VOLUMES

The annual estimate of tailings (dry mass) was generated as part of the mine schedule. An interpolated grid was applied for each coal working section based on the large diameter washability test work (Refer Appendix 2, for results on each working section). The values from these grids were then assigned to each coal reserve block in the mine schedule.

The tailings pumped from the CHPP will be approximately 40 % solids (by weight) and within two days the tailings will consolidate to around 55 % solids. The density based on 0.55 dry solids tonnes (t) per cubic metre (m^3) has been adopted for the initial sizing purposes even though the top 2 m to 3 m of each emplacement is targeted to reach a density 0.9 dry solids t/m^3. This equates to an overall insitu density of 1.2 t/m^3 to 1.47 t/m^3 respectively.

The expected tailings placement rate is summarised in Table 7.1.

7. LIFE OF MINE EMPLACEMENT

7.1 Projected Emplacement Plan

The life of the mine plan will be developed to allow the TSF's to be filled progressively to the level required for capping. The plan provides sufficient storage capacity to allow each emplacement to reduce its filling rate to 1 m/y to 2 m/y as the maximum capacity is approached, by bringing the next emplacement online concurrently, as shown in Table 7.1, refer to colour coding for the different stages.

Table 7.1 Projected Emplacement Plan

Year	Percentage ROM Feed	Tailings Placement (Mm^3)		Tailings Emplacement Areas			
				Out-of-pit	In-pit		
	Annual	Annual	Cumulative	East	1	2	3
1	4.04%	0.94	0.94				
2	4.28%	1.11	2.06				
3	5.18%	1.94	4.00				
4	4.59%	1.74	5.74				
5	4.96%	1.98	7.72				
6	5.26%	2.10	9.83				
7	5.33%	1.99	11.82				
8	6.40%	2.56	14.38				
9	5.31%	2.12	16.51				
10	6.06%	2.42	18.93				
11	5.66%	2.23	21.15				
12	5.65%	2.26	23.41				
13	5.08%	1.91	25.33				
14	3.40%	1.36	26.68				
15	3.39%	1.35	28.04				
16	3.39%	1.36	29.40				
17	3.45%	1.33	30.72				
18	3.43%	1.33	32.06				
19	3.39%	1.39	33.45				
20	3.34%	1.49	34.94				
21	3.35%	0.93	35.87				
22							
23							
24							
25							
26							
27							
28							
29							
30							
31							

Deposition
Reduced filling rate for approx. 1–2 m/year rise
Drying years before capping based on eight years
Rehabilitation period

There will be 7.4 million cubic metres (Mm^3) of out-of-pit storage and 27.9 Mm^3 of in-pit storage for a total 35.3 Mm^3. The shortfall to estimated projected volumes will be accounted for by reaching tailings densities approaching 0.9 t/m^3 for the top 2 m to 3 m of each dam.

To allow the final dam to be filled at this reduced rate the second last dam will remain open so deposition can be shared at the mines final period of operation.

7.2 Out-of-pit Storage

One out-of-pit tailings emplacement will be constructed in the development phase and filled for the first six years of production at the predicted coal production rate and settled tailings density.

The out-of-pit TSF will be sufficient for about the first four years at full production leaving the upper 2 m of tailings to be placed at a lower rate by starting deposition into the in-pit 1 emplacement.

The out-of-pit TSF embankment would be constructed to RL 445 m. It would provide 7.4 Mm^3 storage, with an embankment volume of about 1.25 Mm^3. The layout of the proposed TSF's are presented in Appendix 1.

The storage has been modelled using down valley discharge, with a single discharge point and a beach with a 0.3 % slope.

7.3 In-Pit Storage

Three in-pit tailings emplacements will be progressively built when the out-of-pit emplacement is nearing capacity, as shown in Appendix 1 with indicative capacities shown in Table 7.2.

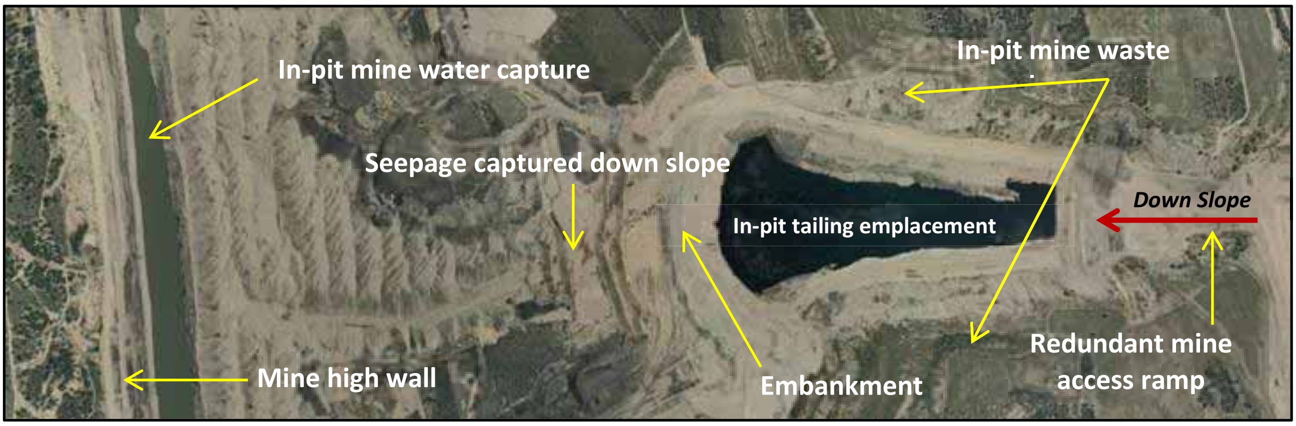
Table 7.2 Estimated In-Pit Emplacement Capacities

In-Pit Tailings Emplacement	Capacity (Mm^3)
1	9.3
2	9.3
3	9.3
Total	**27.9**

These emplacements will be created in the voids left by the redundant mine access ramps and will constructed as part of the mining operation. A typical example is shown in Table 7.1.

Allowing multiple emplacements to be operated simultaneously increases the flexibility in the deposition rates of tailings and maximises ability to dewater the material.

Figure 7.1 Example of In-Pit Tailings Dam Emplacement



8. DESIGN

8.1 Basis

The storage strategy for the life of the mine is to utilise out-of-pit tailings storage until sufficient capacity can be gained within the mined pits to store tailings.

The design for the TSF takes into consideration the following:

- Environmental limitations
- Geotechnical and chemical properties of the tailings
- Production rates and delivery conditions of the tailings, including the likely changes to the delivery schedule
- Design cycle life and the storage capacities of out-of-pit, in-pit 1, in-pit 2 and in-pit 3
- Availability and physical properties of construction materials
- Geology, hydrogeology, groundwater quality and strength of the existing foundation and the mine waste that will form the walls and foundation of embankments
- Geometry and topography of the emplacement site
- Local hydrological and climatic conditions, including average rainfall and evaporation, and extreme storm data
- Rainfall run-off conditions, both of the tailings and the surrounding spoil material
- Rehabilitation to achieve the final landform.

8.2 Tailings Storage Facility Embankments

The out-of-pit embankment will need to be built to retain water. General fill and weathered rock material and suitable clay material will be used to build the embankments.

The out-of-pit embankment will generally be built as follows:

- Strip top soil and excavate under the entire embankment footprint, typically between 0.3 m and 1 m deep (the top soil will be stockpiled and used in rehabilitation)

- Excavate a trench beneath the low permeability zone to form a key between the wall and natural material. This is estimated to be about 2 m to 3 m deep
- Incorporate a low permeability zone 3 m to 5 m wide (wall cross-sectional) on the upstream face, depending upon the permeability of the available borrow material
- Construct the rest of the embankment from rock fill, which may be zoned according to particle size depending upon the available material
- Construct a crest 10 m to 20 m wide
- Construct embankments with batters of 2:1 to 2.5:1 (H:V) upstream and downstream slopes, depending upon the height of the embankment and the available material
- Construct a seepage collection dam at filter discharge.

In-pit embankments will be built as follows:

- Spoil placed by the mine fleet as part of ongoing operations in conjunction with earthworks contractors levelling and compacting each layer
- Crest typically, up to 50 m wide
- The batter slopes will be controlled to typically 2.5:1 to 3:1 (H:V)
- The upstream embankment face will be compacted to decrease the permeability of the embankment.

A sump and diversion drains will be excavated on the downstream side of the embankment to collect seepage.

8.3 Diversion Drains

Diversion drains will be built to minimise stormwater runoff entering the TSFs.

8.4 TSF Water/Flood Storage Capacity

The diversion drains effectively limit upslope catchment area to the TSF itself. With these upslope diversion drains the Probable Maximum Precipitation (PMP) capacity is equivalent to the Probable Maximum Flood (PMF). The storage capacity and freeboard is designed in accordance with DSC guideline (DSC3RF) of the 1 in 1,000 year 72 hour design storm.

The out-of-pit emplacement spillway will allow excess stormwater to spill from the storage during a storm exceeding the design storm without jeopardising the integrity of the dam wall.

Mine waste will be placed around the in-pit emplacements to limit the up-slope catchment; therefore the probable maximum precipitation freeboard for the out-of-pit storages will apply to the in-pit storages. Spillways will not be included in the in-pit emplacements as they will be below the surface of surrounding mine waste emplacements. Emplacements will be designed such that the design flood volume can be safely contained.

Only when the storages are nearing capacity does the flood storage volume becomes an operational consideration.

8.5 Decant Water System

The tailings decant water will be one of the main sources of water for the CHPP. Decant water will be pumped from the decant pond into the mine water system, which then goes to the CHPP. The pumps operate manually and are to be operated to minimise the stored volume in the decant pond, including after rainfall.

8.6 Seepage Control

For the out-of-pit emplacement, seepage will be collected in the filter zone within the embankment wall and drain to a collection dam at the base of the valley. Minor bunds will divert uncontaminated runoff around the seepage collection dam. A return water pump will be installed in the seepage collection dam to return collected water to the mine water system.

The floor of the out-of-pit emplacement will incorporate a clay liner with a 900 mm thickness that achieves permeability 1×10^{-9} m/s or less (or alternative geosyntheic liner of equivalence).

The in-pit emplacements will generally comprise a rock floor, and a perimeter of permeable emplaced mine waste. The embankments will generally be built as part of mining operations. Decant water is expected to flow freely through the embankments and surrounding mine waste and continue following along the mine floor. Over time the tailings will seal the embankment, reducing the effective permeability hence there will be less seepage. While decant ponds may form on the tailings surface, the greatest volume of water is expected to be recovered in the collection sump down gradient of the tailings emplacement.

In the event the mining area downslope of a tailings emplacement is backfilled during active receipt of tailings, seepage capture will be facilitated by a bore recovery system. The bore will be screened within those dams previously excavated in the pit floor to receive seepage through side walls. The bore shaft will be constructed on the downslope embankment for protection during backfilling.

In-pit seepage water will be put into the mine water system and pumped to the CHPP.

8.7 Recoverable Process Water

For the out-of-pit emplacement, it is estimated the recoverable process water will be 25 – 30 % of the water contained in the tailings received. This includes an allowance for evaporative and seepage losses.

Evaporative losses will be reduced by managing the decant pond such that the volume of surface water is minimised. This will also help dry the tailings and increase the bulk density.

For in-pit emplacements, the higher expected seepage losses mean that achievable decant recovery will be less (10 – 15 %) than for an engineered out-of-pit storage with controlled low permeable elements.

9. INSPECTION AND MONITORING

9.1 Inspection Regime

The tailings emplacements will be prescribed by the DSC that requires a dam safety management program with inspections, monitoring and reporting refer Table 9.1. CHC will prepare a Dam Safety Management Plan that includes:

Table 9.1 Inspections and Reporting

Inspection	Frequency	By	Reported by	Reported to
Dam routine	Daily, weekly, monthly	CHPP Manager or delegate	CHC	CHC
Annual intermediate	Annual	Dam Engineer	CHC	DSC
Comprehensive	Every five years	Dam Engineer	CHC	DSC
Independent	Every three years	Independent party	Independent party	CHC
Special/emergency	As required	Dam Engineer	CHC	DSC and DRE
Rehabilitation monitoring	Annual	CHPP Manager	CHC	CHC
Surface waters	Annual	Environment	CHC	DP&I and EPA

- Daily inspections – visual inspections by the CHPP Manager or a qualified delegate designed to highlight any deficiencies in the dams that may lead to an emergency condition. Daily activities will be managed by the CHPP Tailings Coordinator
- Weekly routine inspections – to consider the following:
  - Evidence of distress (this may include cracking, slumping, seepage, excessive erosion, settlement and sinkholes) in the crest, upstream and downstream embankment faces
  - Flow, quantity and clarity of the down slope seepage collection drains
  - Downstream area of embankments, for example, seepage, wet patches; if deficiencies are observed, the area will be pegged and the extent and flow rate will be noted
  - Tailings discharge and deposition characteristics.
- Every three years inspections – by an independent engineer in accordance with DSC requirements to inspect:
  - Embankments
  - Tailings emplacement areas.
- Every five years inspections – by experienced dam engineers/specialists to confirm the safety of the dams. Deficiencies will be identified through:
  - Onsite inspection
  - An evaluation of available data
  - Any applied criteria and up-to-date best practices.
- Special inspections – in the event of any of the following:
  - Earthquake – any recordable earthquake where the mine site is in the affected area
  - Blasting – within 300 m of any embankments or when required by the DSC conditions.

Tailings surface levels will also be monitored, along with seepage, settlement and effects of blasting.

Under the instruction of the CHPP Tailings Coordinator the levels of the top surface of the tailings will be surveyed to assess fill rate. As the final level of each emplacement is approached, the frequency of these surveys will increase to ensure that 2.0 m of flood freeboard is maintained.

The frequency and items the dam inspection regime will address will be presented in the operations and maintenance manual for the respective emplacements.

9.2 Groundwater Monitoring

9.2.1 Baseline Hydrogeological Geotechnical Assessment

Baseline groundwater status and underlying material properties will be assessed prior to construction of emplacements. At least one down gradient nested monitoring bore will be installed as part of the baseline assessment to obtain site specific hydraulic parameters for shallow aquifers at that specific location for use in subsequent modelling and calculations. Geotechnical investigations up-gradient of the emplacement, as well as down-gradient of the downstream toe of the embankment, will be performed to confirm:

- Underlying strata
- Strata permeability
- Identification of soil and rock layers potentially confining to groundwater flow
- Evidence of historical groundwater levels above the current water table.

9.2.2 Predictive Modelling of Groundwater Seepage

Based on the hydrogeological characteristics of underlying strata, an emplacement specific predictive groundwater model will be developed to assess the migration of leachate seepage potentially generated by the tailings emplacements. The model predictions will be used to establish a groundwater monitoring bore network which allows for early identification of groundwater mounding and leachate seepage.

9.2.3 Baseline Groundwater Monitoring

Nested groundwater monitoring bores will be installed up-gradient of the emplacements to allow monitoring of background groundwater properties. Guided by the predictive groundwater modelling, additional nested monitoring bores will be established at selected locations down-gradient of the downstream toe of the embankments. Bores will be screened to provide the ability to sample specific groundwater bearing strata to allow for the following:

- Logging of groundwater to establish baseline piezometric head
- Sampling to establish baseline groundwater quality. The monitoring suite will include:
    - Field parameters - pH, electrical conductivity, redox, temperature, total dissolved solids, dissolved oxygen
    - Laboratory analytes - major ions (calcium, potassium, sodium, magnesium, chloride, total alkalinity and sulfate) and dissolved metals (aluminium, arsenic, barium, beryllium, cobalt, iron, manganese, lead, nickel and zinc).

An electromagnetic survey at the down-gradient side of the embankments will also be performed to establish the baseline spatial representation of soil moisture status.

Monitoring will be performed after establishment of the emplacement facilities to determine the spatial extent and quality of leachate seepage. At this time, the monitoring bore network would be expanded to also include locations on the embankments.

Ongoing monitoring and investigations will include:

- Electromagnetic survey within 12 months of emplacement of tailings to determine spatial changes to the moisture content of underlying strata

- Monthly water level and quarterly water quality monitoring for the suite of parameters outlined above
- Annual reporting of monthly water level and quarterly water quality analyses.

The chemical signature of leachate generated from a tailings emplacement is a function of the tailings characteristics. During operation of the emplacement, the site specific characteristics of the generated leachate will be established. The improved understanding regarding leachate quality may enable rationalisation of the chemical suite initially selected for monitoring purposes.

9.3 Response to Groundwater Impacts

Should any tested parameters exceed agreed trigger levels, regulatory authorities will be notified within one month of receipt of water quality analysis results. Further investigations will take place to establish:

- The spatial extent of contaminated seepage
- Whether environmental harm has occurred.

If it is established that environmental harm has the potential to occur or has occurred, an assessment will be undertaken to establish:

- Measures required to prevent the migration of the contaminant plume
- Measures required to remediate any contamination
- Measure to prevent recurrence of contamination, which may include:
  - Installation of a low-permeability cut-off wall
  - Installation of a high permeability interception trench and collection sump system to collect leachate from underlying strata and allow transfer of leachate to the on-site process water stream.

Trigger levels will be established based on the baseline ground water quality (and natural variations), and in conjunction with OEH and NOW.

10. REHABILITATION

The tailings emplacement will be allowed to dry until it is suitable to be capped. After drying, it will be covered with waste rock that has been stockpiled close by for this purpose. As for the waste rock emplacements, a layer of soil or suitable top dressing will be placed at the top of the profile to facilitate revegetation.

The principal rehabilitation measures will be as follows:

- Tailings will be allowed to dry to form a stable surface that can support capping and covering layers. This is expected to take about five to eight years from the last placement
- Tailings beaches will be managed to maximise the removal of surface water to decant ponds
- Decommissioned tailings emplacements will be covered with:
  - A cap of at least 1 m low permeability material
  - A ‘capillary break’ (with wide interstitial spacing) at least 1.2 m thick preventing water moving upwards to the soil layer (mine waste)

  - Topsoil or suitable top dressing at least 0.3 m thick.
- The final cover design will be developed in consultation with DRE
- The covered emplacements will be shaped re-vegetated and monitored in the same way as the waste rock emplacements.

11. EMERGENCY PREPAREDNESS

The primary risk specific to the TSF's are an embankment failure. CHC will prepare a Dam Safety Emergency Plan (DSEP) that prescribes specific actions to be followed in the event of a dam emergency, including a tailings emplacement. The DSEP clearly identifies stakeholders and emergency personnel to notify and nominates the responsibilities of key personnel.

In the case of an emergency, the DSEP identifies two actions that require individual notification procedures and involve different resources and personnel. These actions are the 'Emergency Action', which is initiated for a potential dam failure and a 'Significant Incident', which is for an incident that may pose immediate danger.

The Emergency Action is used to advise appropriate authorities as soon as possible. When the Mine Manager decides dam failure is imminent, the following notifications will occur:

- CHPP Manager (CHC)
- State Emergency Services
- DRE (Inspector of Coal Mines)
- Local Area Commander (Police)
- Executive Engineer NSW Dam Safety Committee
- Warrumbungle Shire Council (Directors of Operations).

Where a Significant Incident occurs but immediate dam failure is deemed unlikely, the aim is to closely monitor the condition of the TSF and apply immediate measures to return to a safe condition as soon as possible. In the case of a Significant Incident, the following notification will occur:

- The Mine Manager is to notify the CHPP Manager and DRE (Inspector of Coal Mines) with details of the incident and potential risk to the dam
- The Mine Manager will assess the situation and arrange any necessary investigations or remedial action. The Mine Manager may be directed to initiate the Emergency Action.

Full details of Emergency Actions are in the DSEP.

12. RISK ASSESSMENT

CHC will undertake risk assessments for the following stages during the life of the mine and during rehabilitation:

- Construction and operation of the out-of-pit emplacement
- Construction and operation of in-pit 1

- Construction and operation of in-pit 2
- Construction and operation of in-pit 3
- Capping operation of each dam
- Rehabilitation of each dam.

This TSF Management Plan and associated operating procedures will be updated as required to address the risks identified from the risk management process.

13. REVIEW OF MANAGEMENT PLAN

The Mine Manager or their delegate will review this management plan annually, where material changes to the CHPP operation life of mine, Life of Mine (LOM) volumes occur or where an alternative rejects disposal proposal is adopted.

14. RESPONSIBILITIES

Table 14.1 CHC Representative Responsibilities

CHC Representative or Equivalent	Responsibility
Mine Manager	Provide resources to implement this plan Provide adequate resources to manage the tailings at CHC Review this document
CHPP Manager	Manage and monitor CHC emplacement area in accordance with this plan
CHPP Tailings Coordinator	Execute the day to day activities of filling the tailings dam and returning decant water
Technical Services Manager	Consider this plan during mine planning Account for adequate tailings and reject capacity for LOM
Environment and Community Coordinator	Assist with the review and implementation of this plan

15. REFERENCES

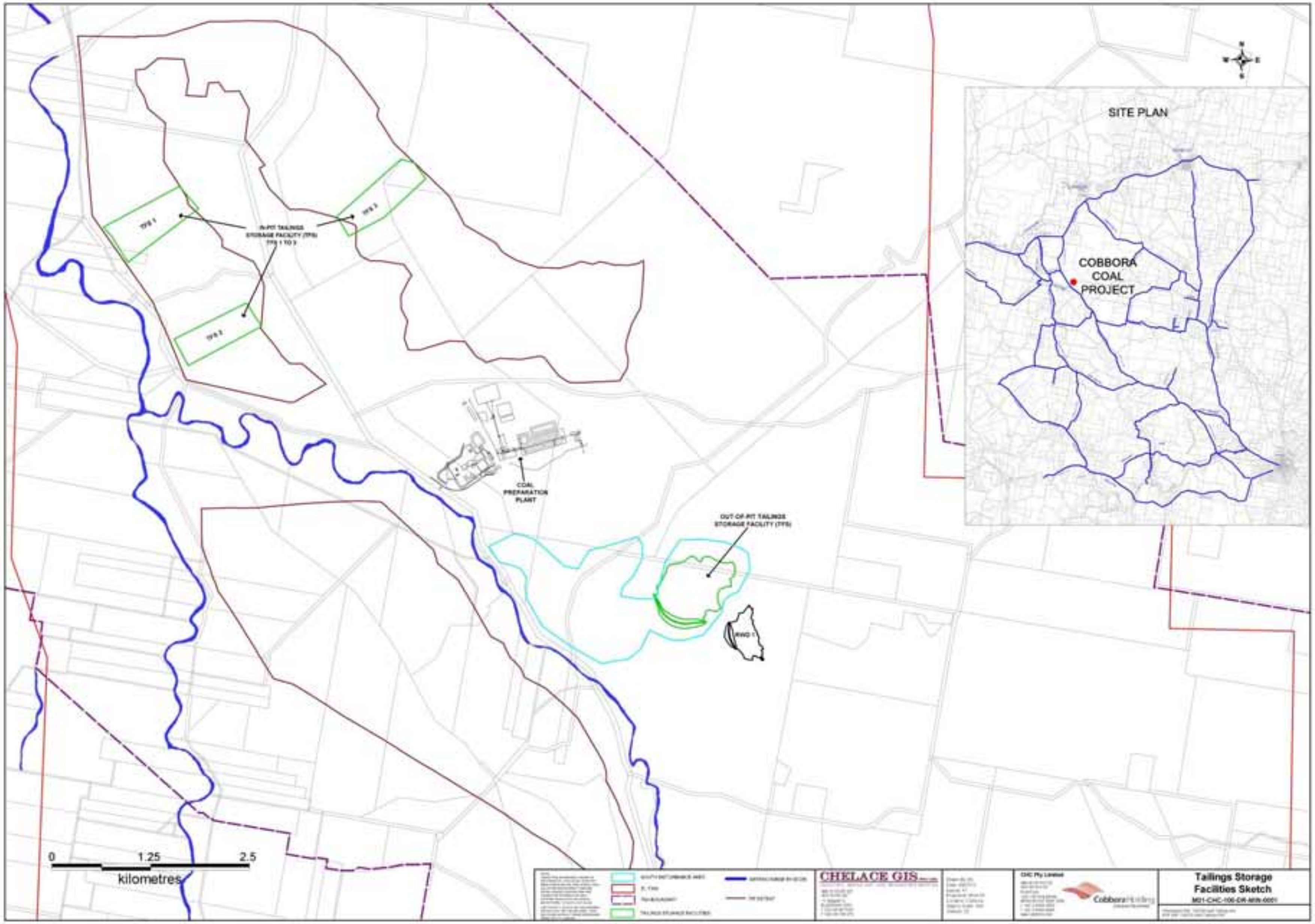
Cobbora Holding Company 2012, *Life of Mine Tailings Options Study*, December 2012 (112120R01)

Nathan, RJ and Weinmann, E 1999, 'Estimation of Large to Extreme Floods', Book VI in *Australian Rainfall and Runoff – A Guide to Flood Estimation*, The Institution of Engineers, Australia, Barton, ACT.

New South Wales Department of Primary Industries 1999, *Synoptic Plan, Integrated Landscapes for Coal Mine Rehabilitation in the Hunter Valley of NSW*, Sydney.

APPENDIX 1

Tailings Emplacement Location Sketch

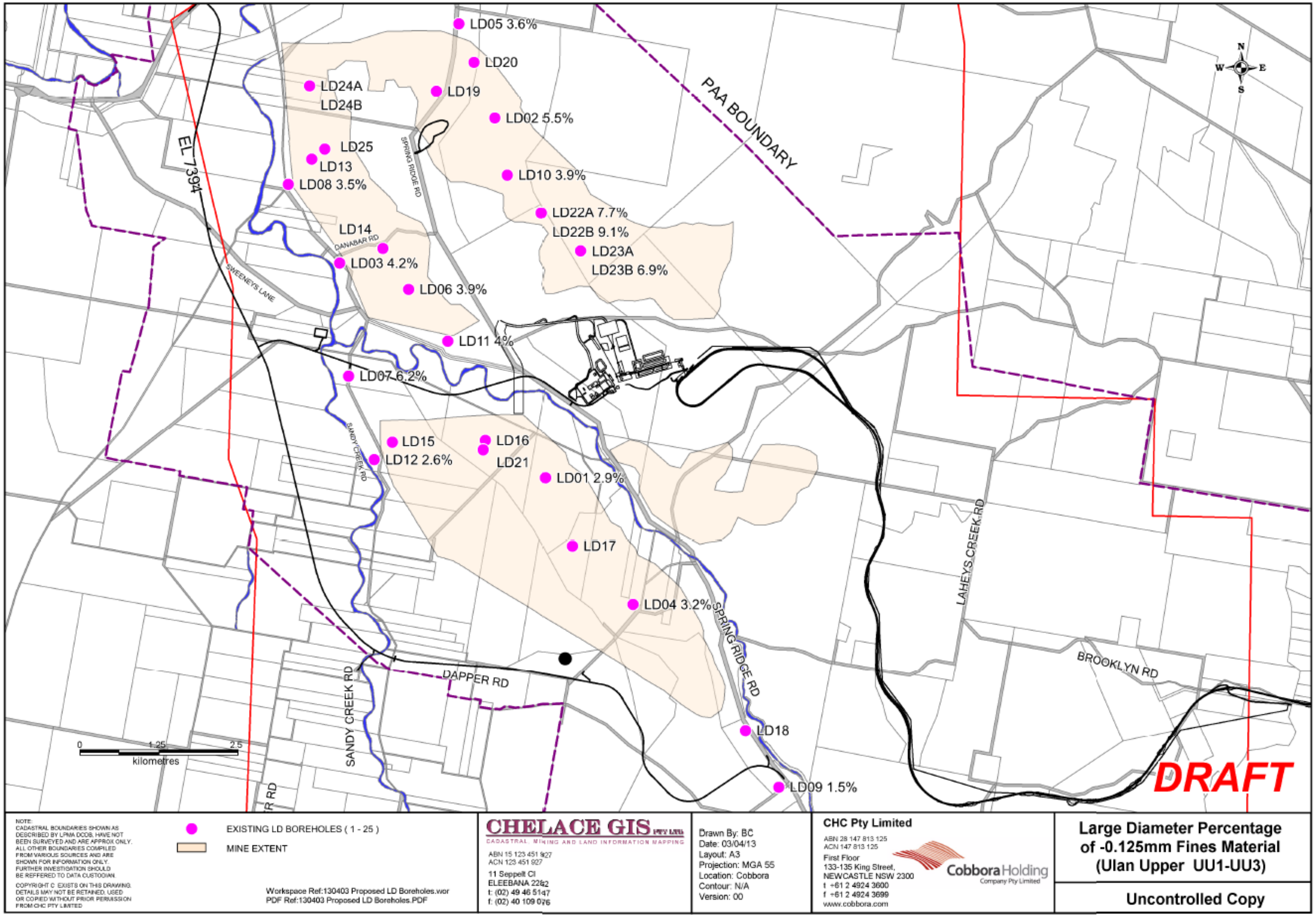

SITE PLAN
COBBORA
COAL
PROJECT
COAL
PREPARATION
PLANT
OUT OF PIT TAILINGS
STORAGE FACILITY (TFS)
0
1.25
2.5
kilometres
CHELACE GIS
Tailings Storage
Facilities Sketch

APPENDIX 2

Large Diameter Percentage
of
-0.125 mm Fines Material

APPENDIX 2a

Ulan Upper UU1 – UU3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LD24A
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LD02 5.5%
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LD08 3.5%
LD10 3.9%
LD22A 7.7%
LD22B 9.1%
LD23A
LD23B 6.9%
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LD03 4.2%
LD06 3.9%
LD11 4%
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LD12 2.6%
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LD01 2.9%
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LD18
LD09 1.5%
PAA BOUNDARY
EL 7394
SPRING RIDGE RD
DANABAR RD
SWEENEYS LANE
SANDY CREEK RD
LAHEYS CREEK RD
BROOKLYN RD
DAPPER RD
N
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E
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1.25
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kilometres
DRAFT
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EXISTING LD BOREHOLES (1 - 25)
MINE EXTENT
Workspace Ref:130403 Proposed LD Boreholes.wor
PDF Ref:130403 Proposed LD Boreholes.PDF
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CADASTRAL, MINING AND LAND INFORMATION MAPPING
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ACN 123 451 927
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Drawn By: BC
Date: 03/04/13
Layout: A3
Projection: MGA 55
Location: Cobbora
Contour: N/A
Version: 00
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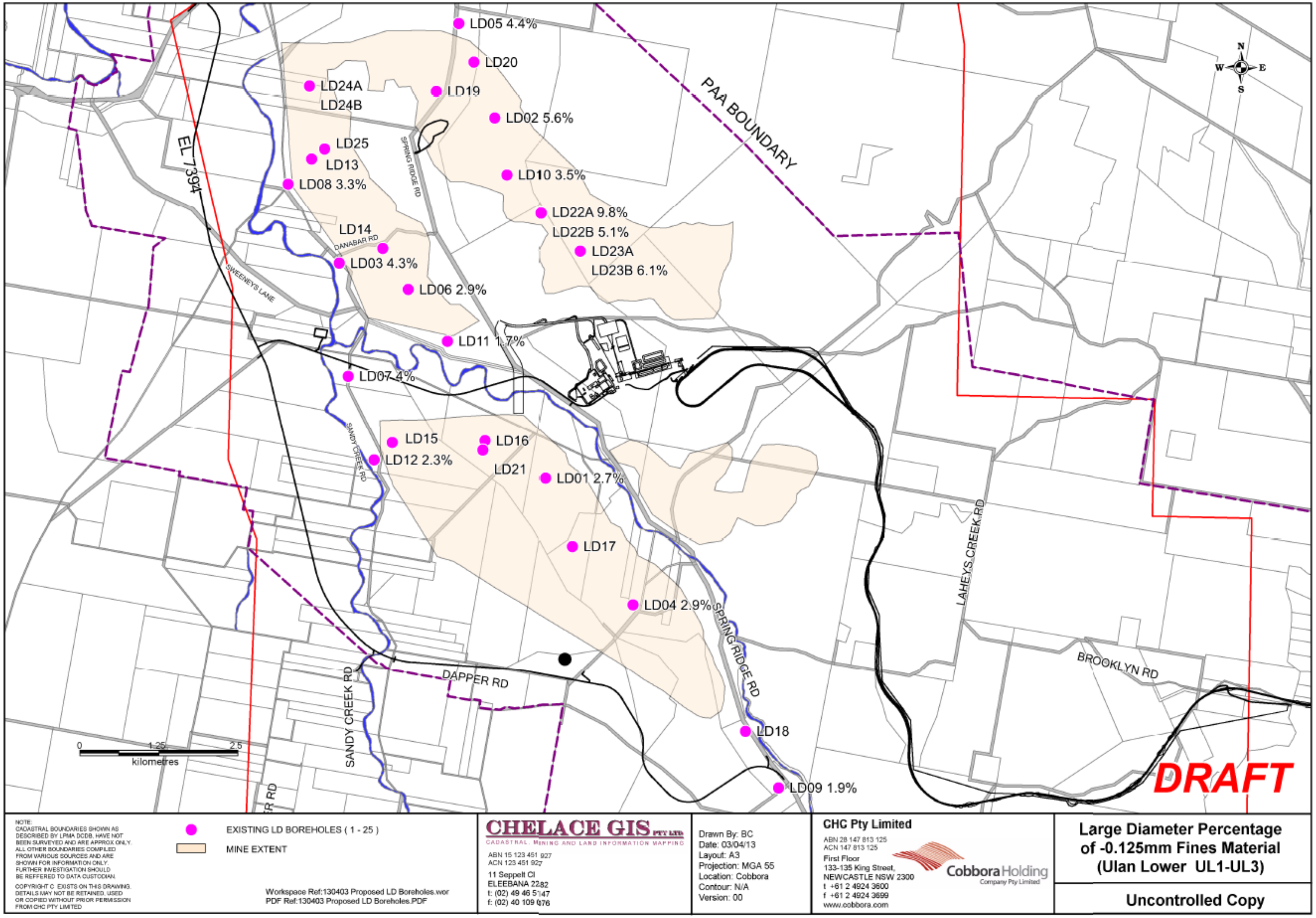
Company Pty Limited
Large Diameter Percentage of -0.125mm Fines Material (Ulan Upper UU1-UU3)
Uncontrolled Copy

APPENDIX 2b

Ulan Lower UL1 – UL3


LD05 4.4%
LD20
LD24A
LD24B
LD19
LD02 5.6%
PAA BOUNDARY
EL 7394
LD25
LD13
SPRING RIDGE RD
LD08 3.3%
LD10 3.5%
LD22A 9.8%
LD22B 5.1%
LD14
DANABAR RD
LD23A
LD23B 6.1%
LD03 4.3%
SWEENEYS LANE
LD06 2.9%
LD11 1.7%
LD07 4%
SANDY CREEK RD
LD15
LD16
LD12 2.3%
LD21
LD01 2.7%
LAHEYS CREEK RD
LD17
LD04 2.9%
SPRING RIDGE RD
BROOKLYN RD
DAPPER RD
SANDY CREEK RD
LD18
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1.25
2.5
kilometres
LD09 1.9%
DRAFT
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NOTE:
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EXISTING LD BOREHOLES (1 - 25)
MINE EXTENT
Workspace Ref:130403 Proposed LD Boreholes.wor
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ELEEBANA 2282
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Large Diameter Percentage of -0.125mm Fines Material (Ulan Lower UL1-UL3)
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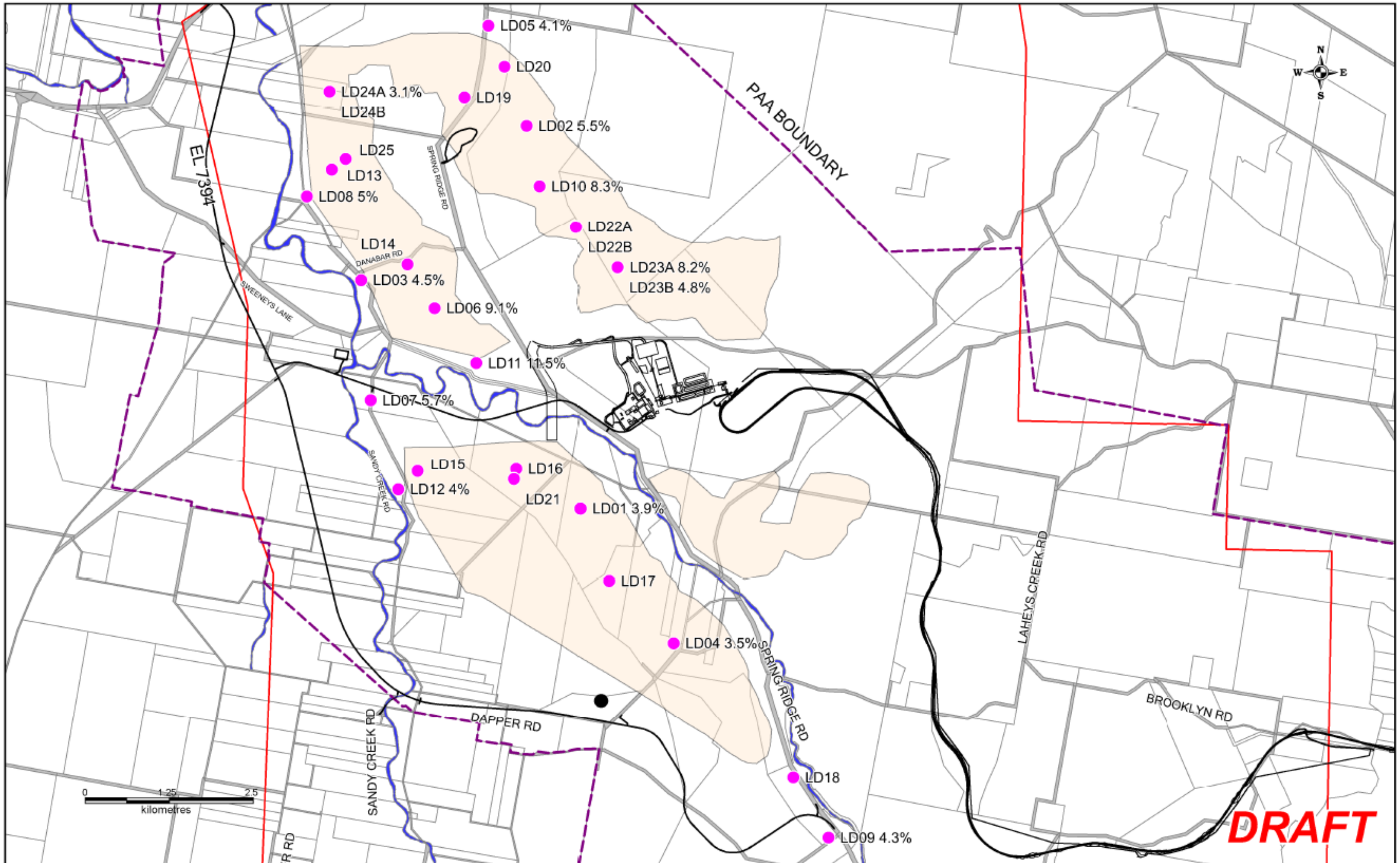
APPENDIX 2c

Flyblowers Creek FB1 – FB6



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Large Diameter Percentage of -0.125mm Fines Material (Flyblowers Creek FB1-FB6)

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Appendix D

Response to PAC review of water modelling

Parsons Brinckerhoff Australia Pty Limited

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A GRI Rating: Sustainability Report 2011

Memo

Date	14 May 2013
To	Andrew Krause, Trish McDonald, CHC Phil Towler, EMM
From	Rob Leslie
Ref	2162570C-DMS-WAT-007 RevG
Subject	Cobbora Coal Project - Responses to PAC Review of Water Modelling

Dear Andrew and Trish

1. Introduction

This memo provides Parson Brinkerhoff's (PB's) responses to Dr Steve Perren's Review of Potential Water Impacts for the Cobbora Coal Project as provided in Appendix 7 of the Planning Assessment Commission (PAC) Review Report dated April 2013.

2. General

The majority of Dr Perrens' comments relate to the impact on flows in the creek systems downstream of the mining areas due to a combination of changes to the surface water catchments predicted by the site water balance model (which includes a hydrological model) and losses to the river systems due to water table drawdown predicted by the groundwater model.

Dr Perrens concluded his review by noting:

"This review has identified a range of uncertainties associated with the estimated water requirements for the operation of the Cobbora Mine and the relative contributions from different sources of supply. Notwithstanding these uncertainties, none of them individually or collectively would be 'show stoppers'. A range of options are available that would allow the mine to adapt its water use and manage the various sources to allow the mine to operate within the constraints of the available water resources."

The review pointed to a number of specific issues on water supply and consumption but ultimately, most of them can only be resolved once detailed design is completed. The present submission responds to each of the questions raised by the Department of Planning and Infrastructure following its receipt of the review report, while recognising the need for detailed design to answer them in more specific terms. Nevertheless, PB concurs with Dr Perrens' assessment that none of the issues raised could not be resolved during detailed design stage and that none would compromise the ability of the mine to appropriately manage surface and ground water resources.

This section combines information from the Surface Water and Groundwater Assessments to respond to the PAC Review Report.

2.1 Impacts on downstream flows

The main impacts on downstream flows will be experienced in the following reaches:

- Lower 3km of Blackheath Creek between mining areas A/C and mining area B
- Lower 8km of Laheys Creek adjacent to mining area B
- Lower 11km of Sandy Creek adjacent to mining areas B and A

The flow impacts will also extend downstream to the Talbragar River below its confluence with Sandy Creek; however, the impacts will be greatly diminished due to the influence of the larger Talbragar catchment. These impacts would also diminish further downstream as the contributing catchment of the Talbragar River increases. The Sandy Creek sub-catchment is approximately 8% of the Talbragar River catchment to the Sandy Creek and Talbragar River confluence; and approximately 1% of the Macquarie River catchment to the Talbragar River and Macquarie River confluence. Therefore, the Sandy Creek sub-catchment is a very small portion of the broader Macquarie River catchment system. In terms of magnitude of flows, the Surface Water Assessment found that the mean daily flow in Sandy Creek is approximately 11% of the mean daily flow in the Talbragar River at Elong Elong, 0.9% of the mean daily flow in the Macquarie River at Warren Weir and 0.2% of the mean daily flow in the Macquarie River at Oxley Station.

In order to quantitatively assess the potential changes in annual flow in Sandy Creek and the resulting impact on flows in the Talbragar River, estimates of annual stream flow from surface water modelling were combined with estimates of potential seepage losses from the creeks from the groundwater model. It should be noted that the two systems (surface and groundwater) were modelled using different approaches and software, and have different spatial and temporal resolutions. Groundwater impacts were modelled in 3D using MODFLOW with yearly time steps and a relatively coarse grid structure to assess regional impacts, whereas surface water models used a 1D model to assess changes in catchment runoff and stream flow for each tributary with daily time steps under a number of rainfall scenarios. The groundwater model in particular includes a number of conservative assumptions to allow for adequate licensing of surface water and groundwater take and to provide conservative predictions of groundwater drawdown impacts to the environment and bore users. As a result, a simple combination of model outputs has limitations because the conservative assumptions of the groundwater model, when combined with the surface water predictions can be overly conservative to the point of being unrealistic. Therefore, in the approach outlined below an effort has been made to compensate for this and align the model outputs.

The groundwater model tends to overestimate losses from the creeks for the following reasons: First, the groundwater model represented the creeks using the MODFLOW ‘river cell’ module which assumes that the creeks supply a continuous source of water that can be impacted by drawdown of the water table. This is an appropriate conservative assumption for the purpose of water licencing as it overestimates the river losses due to drawdown caused by mining. However, it assumes that the creeks are flowing continuously, whereas in reality the Sandy Creek system is ephemeral and flow only occurs 60% of the time (see Surface Water Report Appendix C Figures 5-1 to 5-3). Secondly, the elevation of the river cells in the model are defined using a regional DTM based on 10 m contour data. In many cases this may also result in a slight overestimate of the increase in seepage to the groundwater system as a result of drawdown. For these reasons, the base case total stream flow estimates have been presented, followed by potential reductions in flow expected in response to typical climate conditions. Potential reductions have been estimated by adjusting the stream losses down by a factor related to the amount of time the stream actually flows during average and dry years.

Tables 1 and 2 give a highly conservative (base case) assessment of impacts on annual flows at the downstream reach of Sandy Creek and at Elong Elong on the Talbragar River downstream of its confluence with Sandy Creek for the representative dry, median and wet years in the historical climate sequence. The tables combine the changes in surface water flow regime predicted by the water balance model with river losses predicted by the groundwater model.

Tables 3 and 4 present the same stream flow estimates but apply a seepage loss that is factored down by 40% to account for the observation that, in a median rainfall year, the creek flows only 60% of the time and can therefore only lose water during periods of flow. Tables 5 and 6 apply a seepage loss that is factored down by 80% to account for the amount of time (20%) that the creek is estimated to flow in dryer conditions.

Tables 1 to 6 demonstrate the following:

- For the conservative scenario (Tables 1 and 2) in which the full estimate of modelled river losses are applied, calculations suggest that there may be reductions in annual flow in Sandy Creek compared with baseline (pre-mining) conditions of approximately 21% in the median rainfall case. There would be an almost negligible reduction (2%) in annual flow in the Talbragar River at Elong Elong in a median year. In a dry year the calculated reduction in annual flow would be up to 86% in Sandy Creek and 9% in the Talbragar River under this scenario. However for reasons outlined above, while useful as a check on modelling consistency, this combination of scenarios is not considered to be a realistic estimate of total system impacts.
- When the groundwater model river losses are applied only 60% of the time (reduced by 40%; Tables 3 and 4), the maximum reduction in flow is 11% in the Sandy Creek system in a median year and 1% (negligible) for the Talbragar River at Elong Elong. In a very dry year the losses may be up to 54% and 11% respectively. This is assessed as the most likely upper bound or most severe impact on the river system.
- When the groundwater model river losses are applied only 20% of the time (reduced by 80%; Tables 5 and 6), the maximum reduction in flow in the Sandy Creek system and the Talbragar River at Elong Elong is negligible in a median year. In a dry year, the reductions in flow under this scenario may be up to 21% and 2% respectively. This is the most likely lower bound impact on the rivers.

In all cases, estimated flow reductions are much less for the reference median year and slight flow increases are predicted for the reference wet year due to increased runoff from the modified surface water catchments, which compensates for the river losses to groundwater.

As noted above, the Sandy Creek system is ephemeral, flowing 60% of the time, and therefore baseflow conditions in the system involve filling and drying of isolated pools and discharge from temporary storage in the alluvium, rather than perennial low flow fed by groundwater discharge (as assumed in the conservative groundwater model). Given that the majority of the groundwater model river loss occurs in the Sandy Creek system and this system provides minimal baseflow to the Talbragar River, it can be concluded that the predicted flow impacts to the Talbragar River are likely to be relatively minor.

2.2 Impacts on refuge pools

In the absence of detailed survey and water balance modelling for each individual refuge pool, the impacts on the pools were assessed based on the water table drawdown predicted by the groundwater model. The assessment aimed to determine potential impacts on refuge pools that persist during extended dry conditions by focussing on groundwater drawdown impacts rather than flow impacts on the refuge pools, given that pools that persist during extended dry periods are likely to be sustained by groundwater.

Table 7 repeats the results of the refuge pool assessment from the Surface Water Report. The results of a more recent sensitivity analysis on drawdown undertaken using the groundwater model (assuming no constant head in the creeks) are also presented in the table. The base case assessment predicted that pools 3, 6 and 10 would lose groundwater supply due to drawdown and pool 5 is also at risk. The conservative case modelled in the sensitivity analysis predicts that pool 5 would lose supply and pool 3 would be affected earlier by year 12 rather than by year 16 in the base case.

CHC is committed to providing contingency measures for pools that could be affected by drawdown. An aquatic monitoring strategy will be developed to detect changes to the quality and quantity of water in semi-permanent pools. A river monitoring committee will be formed to review the results from this strategy and to assist formulate adaptive management measures. These measures may include releasing water from the mine site to fill pools. It may be necessary to transport the water by pipe or water truck if gravity feed is not possible for upstream locations or where water transmissions losses are unacceptable for downstream locations.

3. Responses to PAC Review Report comments

The Department of Planning & Infrastructure summarised the key PAC Review Report comments on the water assessments into 19 key points. These are repeated in this section with PB's responses below:

1. p17 impacts of assumptions of dry creeks and rivers (additional groundwater sensitivity run assuming no constant head) on dewatering flows to the mine, baseflow losses to Talbgragar River and pools on Sandy and Laheys Creek have not been assessed.

PB response:

Additional model sensitivity runs were carried out to assess the potential groundwater drawdown impacts in the vicinity of nearby refuge pools on Sandy and Laheys Creeks, in line with the recommendations of Dr Kalf. The results of that sensitivity analysis in terms of impacts on refuge pools are described in Section 2.2. Under the highly conservative assumption that the creeks provide no recharge to the groundwater systems when they are flowing, the model predicts that between 5 m and 22 m of drawdown will occur within the coal measures underlying the refuge pools. This amounts to between 4 m and 14 m greater drawdown than the original EA modelling that assumed that creeks provide recharge to groundwater. Baseline monitoring since 2010 has shown that significant recharge to groundwater does occur during high rainfall and runoff events which will mitigate mine related drawdown to some extent.

It is expected that in a case where no recharge to the aquifers occurs via the streams during high flow events then the predicted inflow to the mine pits will be slightly less in the long term. However most inflow will be initially derived from aquifer storage and therefore mine inflows in the first several years should not differ under such a scenario. The numerical groundwater model estimates that a maximum of 480 ML/a may be lost from the surface water systems towards the end of mining, and in the years following the end of mining. This amounts to between 10% and 15% of maximum predicted groundwater inflows to the mine. Therefore the uncertainty in groundwater inflows to the mine as a result of uncertainty in the assumptions associated with recharge from the streams will be of the same order (10 to 15%).

As discussed in Section 2.1, there is no impact on baseflow in the Talbragar River as the water table drawdown impacts are concentrated in the Sandy Creek catchment which does not contribute baseflow to the Talbragar River.

2. p17 Baseflow % - larger impacts during lower flow periods in Talbragar River – further discussion needed

PB response:

Section 2.1 presents further results of downstream flow impacts and combines the predicted flow changes from the water balance model with the groundwater model predictions of river losses. This shows that for the highly conservative case adopted in the EA groundwater modelling, the maximum reduction in annual flow at Elong Elong on the Talbragar River is 9% for the reference dry year. However, taking into account the conservative nature of the groundwater model, this reduction is more likely to be in the range of 2 to 6%.

Also, Sandy Creek does not contribute baseflow to the Talbragar River system; therefore, there are no impacts on baseflow in the Talbragar River.

3. p18 - loss of base-flow in Sandy and Laheys Creek is not clear – figures are provided for entire Talbragar River water source within the g/w model domain – not clear what reduction in baseflow impacts will have on tributary creek flows

PB response:

The EA groundwater model reported the estimated total loss of surface water due to partial loss of baseflow and increased infiltration during high flow. This total loss will relate mainly to losses from the streams adjacent to the mining areas and within the cone of drawdown (Sandy Creek and Laheys Creek), with relatively little direct loss from the Talbragar River. Tables 3 and 5 provide upper and lower bound estimates of potential reductions in based flows to Sandy Creek.

The majority of the loss is attributed to the streams of the Sandy Creek system adjacent to the mining areas, i.e. the lower 3km of Blackheath Creek, the lower 8km of Laheys Creek and the lower 11km of Sandy Creek. The combined loss in these reaches is then transferred to the Talbragar River downstream of its confluence with Sandy Creek. The loss transferred to the Talbragar River is loss due to increased infiltration during high flow conditions. There is no loss of baseflow in the Talbragar River due to baseflow reductions in the Sandy Creek system.

4. p19 Clarify the status and capacity of Clean Water Dams 9 and 10 – discrepancy in volumes

PB response:

The revised mine plan has reduced the in-pit tailings emplacement areas, which is consistent with the reduction in fines being generated, refer Tailings Management Review M01-CHC-351-RP-ENV-001. The revised mine plan requires one out-of-pit tailings storage facility (Out-Of-Pit East) and three in-pit tailings emplacement areas. Clean water dam 10 will be located upslope of Out-Of-Pit East and have a volumetric capacity of 357 ML.

CHC intends to licence clean water dam 10 under its harvest right provision in accordance with part 1 of Chapter 3 of the *Water Management Act 2000*.

5. p21 - Clarification on potential contingency to maximise utilisation of captured water from sediment dams to reduce raw water demand from Cudgegong – however will also affect assumptions regarding provision of contingent flows to Laheys/ Sandy Creek system.

PB response:

The mine water system has been designed to harvest sedimentation dam water up to a point, but balanced so as to avoid significant impacts on the flow regime in the creeks downstream. A range of rules were tested during development of the water balance model for harvesting from sedimentation dams; (1) no sedimentation dam water harvesting; (2) pump from sedimentation dams to mine water dams when mine water dams fall below 25% full; and (3) pump from sedimentation dams to mine water dams when mine water dams fall below 50% full. The results of these early tests are summarised below for mining year 20 in Table 8:

Table 8 – Annual flow impacts at Sandy Creek outlet

Climate condition	Impact on creek flow – mining year 20		
	0% rule	25% rule	50% rule
10%ile (dry) year	+11%	-5%	-5%
50%ile (median) year	+12%	+5%	+3%
90%ile (wet) year	+2%	+2%	+1%

The 25% rule was chosen as the preferred operating procedure as it was found to provide a moderate volume of water to reduce reliance on the Cudgegong River entitlement while keeping the reductions in dry year flow within 5% assuming that the resulting impacts will be acceptable.

Harvested (from sedimentation dams) and imported (from the Cudgegong River) water volumes are presented in the Surface Water Report Appendix E Addendum Tables 2-3 to 2-5. For mining year 20 under the reference dry year conditions the harvested volume is 123 ML and the imported volume is 2,400 ML, approximately 900 ML below the full entitlement from the Cudgegong River. For mining year 20 under the reference wet year conditions the harvested volume is 104 ML and the imported volume is 400 ML, approximately 2,900 ML below the full entitlement from the Cudgegong. This demonstrates that there is a low probability that the full entitlement from the Cudgegong River will be required and that moderate volumes of water can be harvested from the sedimentation dams under the 25% pumping rule under a range of climate conditions.

6. p22 - additional freeboard capacity of 1m on mine water dams recommended

PB response:

The mine water dams were initially sized using the Blue Book procedure to retain the 100 year 72 hour rainfall volume. The dams were then tested in the water balance model under the historical climate and upsized as required to avoid spilling under prolonged wet sequences in the climate record. A standard 500mm freeboard was then adopted on top of the upsized dams. This is considered to be a sufficient basis of design to avoid spilling of the dams.

7. p22. Incidental take of base-flow (480ML peak) in Talbragar River – how administered within water sharing rules – prohibits pumping when there is no visible flow into and out of pools.

CHC has received advice from NSW Office of Water that surrendering of surface water access entitlement in the Lower Talbragar River Water Source is an acceptable offset of baseflow loss to the Talbragar River. CHC has not been advised of any temporal constraint associated with the offset. Licensed dams in the area of the mine have a reliability factor of 1.17, requiring at least 562,000 m^3 or 562 ML (480 ML x 1.17) of licensed dam capacity to be surrendered to offset baseflow loss.

8. p25 – Clarification of total combined induced loss of flows in Talbragar River Water Source of 799ML/year – inclusive of groundwater baseflow loss of 480ML/year – it is not clear what the additional flow loss is from?

PB response:

The EA groundwater model predicted that a maximum of 480 ML per year would be lost from all surface water drainages as a result of groundwater drawdown related to mining. That maximum would occur in the year following the end of mining and reduce thereafter. The additional loss component (319 ML) reflects the capture of the enhanced recharge in the spoils area by the remaining void. This was accounted for as a relative loss to the surface water system.

9. p31 – anomalous data for evaporation losses from water storages in the revised water balance tables

PB response:

The apparent anomaly is in the higher evaporation loss in the reference dry year than in the reference median year. This is explained by the preceding climate of the reference dry year, which was wetter than that of the reference median year. This results in considerably larger volumes of water in storage at the start of the dry year than for the median year, and therefore more volume is evaporated throughout the dry year than in the median year.

10. Provide further advice as to potential contingency and strategy to manage water in dry and wet years – linked to PAC recommendation that mechanical dewatering be implemented to reduce raw water demand.

Dr Perrens has indicated that 'based on the assumptions that underpin the water balance analysis, it appears water shortage is more likely than excess'. For responses regarding potential contingency and strategy to manage water in dry years, refer to Cobbora Holding Company report M01-CHC-350 -RP-ENV-0001 (see Section 3.2). *11. p36 – further clarification on discharging to creek systems using operational rules – eg. where pump to mine dams if capacity of MWD is less than 25% - to specifically meet flow objectives*

PB response:

Refer to response to comment 5 above.

12. p36 – Baseflow losses of up to 480ML/year need to be considered in assessment of impacts on Sandy Creek/ Laheys Creek and how this may affect flow regime and impacts on creek system.

PB response:

Refer to Section 2.1 and 2.2. The groundwater model is conservative in estimation of river losses for the purposes of licencing. When impacts on creeks predicted by the water balance model and groundwater model are combined, significant flow reductions are seen in dry and median years in the Sandy Creek system, in the range of 10 to 20% for median years. However as discussed above, due to the conservative nature of the stream loss estimates, it is appropriate to adjust the estimated stream losses down to account for the times when the stream is not flowing (40% in a median year and 80% in a dry year). With these adjustments, predicted reductions in total flow in Sandy Creek are moderate to minor (Tables 3 to 6).

13. p37 - Groundwater drawdown on semi-permanent pools – underestimation of groundwater drawdown due to running River Package assuming constant head in creek system – additional information of drawdown predictions on semi-permanent pools is needed.

PB response:

Refer to Section 2.2. The sensitivity test shows that an additional pool would be affected by drawdown under this scenario.

14. p38- Further analysis of how releases from sediment dams could positively affect semi-permanent pools is required.

PB response:

Refer to the response to comment 5.

Table 8 demonstrates that if all of the water collected in sedimentation dams were to be released (see 0% rule column) to the creeks there will be increase in flows in Sandy Creek for all reported climate scenarios. This is expected to compensate for potential reduction in base flow contributions to the pools in Sandy Creek. It is expected that controlled releases would be made directly to the creek via an appropriate dedicated pipeline during very low flow conditions.

The water management system has been balanced to avoid significant impacts on the flow regime in downstream creeks. Harvesting of sedimentation dam water could be further reduced to provide more releases to the downstream systems without significantly increasing demand from the Cudgegong. The imported water requirement from the Cudgegong remains well below the full entitlement under the reference dry year climate condition.

15. p38 - Combined impact of reduced surface runoff and baseflow loss on semi-permanent pools not assessed (link to 3, 12 above)

PB response:

Refer to responses to comments 3 and 12 above.

16. p38 - Clarification regarding offsetting of cumulative baseflow losses in Talbragar River from Ulan Coal Mine by discharge from treated mine water from Ulan post mining. Further assessment is recommended.

PB response:

Refer to Section 2.1. Due to the ephemeral nature of Sandy Creek, it does not contribute significant baseflow to the Talbragar River, and therefore the Project impacts on the Sandy Creek system have minimal to negligible impact on flows downstream in the Talbragar River.

17. p41 - Flood impacts of increased channel and flood velocities in the vicinity of crossings 4, 5 and 7 and proposed mitigation measures to prevent scouring; protection of the toe of overburden dump within 1 in 100 year flood in Laheys Creek

The impacts of the crossing structures on local channel and floodplain velocities are available from the flood model. Model predictions of the changed velocity profiles will be used to inform design of: scour protection measures such as energy dissipation aprons and basins at the inlets and outlets of culverts; assessment of extreme flood event scour potential at abutments and piers of bridges and design of measures to protect the structures under these events; assessment of extreme flood event scour potential along the areas of the flood protection levees and overburden dumps and dams that are at risk of flooding and design of measures to protect these structures from erosion during extreme events.

Existing and future case flood velocities at Crossings 4, 5 and 7 are provided below in Table 9.

Table 9 – Existing and future case flood velocities at key crossing structures

Crossing no.	Flood velocities (m/s)			
	100 year ARI		2000 year ARI	
	Existing	Future	Existing	Future
4	1.1	1.4	0.9	1.0
5	0.5	0.5	0.9	0.6

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7	1.0	1.2	1.3	3.8

Velocities at crossings 4 and 5 are relatively low for extreme events and standard scour protection measures will be adopted for these crossings. Crossing 7 has significantly higher velocities for the 2000 year ARI event and will require significant scour protection for this permanent structure.

2000 year ARI velocities will be used to design the scour protection around the toe of flood protection levees, overburden dumps and sedimentation dams that are located at the edge of the creek floodplains.

18. p42 - Consideration of 3m cover in backfilling of voids to minimise soil salinisation

The 3 m backfill cover is sufficient to avoid soil salinisation (refer to Attachment 1)

19. p55 – issues regarding base-flow and surface runoff losses require further clarification as to magnitude, location and progression over time – and proposed mechanism for offsetting the losses needs clarification. Refer also items 1-3, 12 above.

PB response:

Refer to responses to comments 5, 11, 15 and 16. In Tables 1 to 6, potential losses from the stream are applied to the total flow which includes the baseflow component and runoff. The estimates of total flow in Sandy Creek and the Talbragar River therefore include a conservative (high) estimate of potential losses to those systems. In the case of the Talbragar River it is noted that losses transferred downstream to the Talbragar River equate to between 1% and 6% decrease in total flow and would diminish downstream as the Talbragar catchment increases.

The mechanism for offsetting the losses to the system during and post mining is yet to be determined, but a management plan will be developed based on the ongoing surface water monitoring program that will continue through the operational life of the mine (see also the response to comment 7).

Yours sincerely

Rob Leslie

Rob Leslie
Team Manager, Water Resources NSW
Parsons Brinckerhoff

Table 1 – Annual flow impacts at Sandy Creek outlet – no reduction in groundwater model river losses

Year	Annual flow (ML/yr)			Change in annual flow from SW model (ML/yr)			River losses from GW Model (ML/yr)			Change in annual flow (ML/yr)			Change in annual flow (%)		
	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)
0 (baseline)	575	1,852	26,088	-	-	-	-	-	-	-	-	-	-	-	-
1	618	1,960	27,241	43	108	1,154	-4	-4	-4	39	104	1,150	7%	6%	4%
4	559	2,014	27,355	-16	161	1,267	-125	-125	-125	-141	36	1,142	-25%	2%	4%
12	538	2,046	27,301	-37	193	1,214	-341	-341	-341	-378	-148	873	-66%	-8%	3%
16	540	2,043	27,462	-35	191	1,374	-416	-416	-416	-451	-225	958	-78%	-12%	4%
20	548	1,930	27,439	-27	78	1,351	-469	-469	-469	-496	-391	882	-86%	-21%	3%
2035 (Post mining)	642	1,933	28,830	67	81	2,742	-474	-474	-474	-407	-393	2,268	-71%	-21%	9%
Post recovery	642	1,933	28,830	67	81	2,742	0	0	0	67	81	2,742	12%	4%	11%

Table 2 – Annual flow impacts at Elong Elong on Talbragar River – no reduction in groundwater model river losses

Year	Annual flow (ML/yr)			Change in annual flow from SW model (ML/yr)			River losses from GW Model (ML/yr)			Change in annual flow (ML/yr)			Change in annual flow (%)		
	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)
0 (baseline)	5,227	16,836	237,164	-	-	-	-	-	-	-	-	-	-	-	-
1	5,618	17,818	247,645	43	108	1,154	-4	-4	-4	39	104	1,150	1%	1%	0%
4	5,082	18,309	248,682	-16	161	1,267	-125	-125	-125	-141	36	1,142	-3%	0%	0%
12	4,891	18,600	248,191	-37	193	1,214	-341	-341	-341	-378	-148	873	-7%	-1%	0%
16	4,909	18,573	249,655	-35	191	1,374	-416	-416	-416	-451	-225	958	-9%	-1%	0%
20	4,982	17,545	249,445	-27	78	1,351	-469	-469	-469	-496	-391	882	-9%	-2%	0%
2035 (Post mining)	5,836	17,573	262,091	67	81	2,742	-474	-474	-474	-407	-393	2,268	-8%	-2%	1%
Post recovery	5,836	17,573	262,091	67	81	2,742	0	0	0	67	81	2,742	1%	0%	1%

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Table 3 – Annual flow impacts at Sandy Creek outlet – 40% reduction in groundwater model river losses

Year	Annual flow (ML/yr)			Change in annual flow from SW model (ML/yr)			River losses from GW Model (ML/yr)			Change in annual flow (ML/yr)			Change in annual flow (%)		
	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)
0 (baseline)	575	1,852	26,088	-	-	-	-	-	-	-	-	-	-	-	-
1	618	1,960	27,241	43	108	1,154	-2	-2	-2	41	106	1,152	7%	6%	4%
4	559	2,014	27,355	-16	161	1,267	-75	-75	-75	-91	86	1,192	-16%	5%	5%
12	538	2,046	27,301	-37	193	1,214	-205	-205	-205	-242	-12	1,009	-42%	-1%	4%
16	540	2,043	27,462	-35	191	1,374	-250	-250	-250	-285	-59	1,124	-49%	-3%	4%
20	548	1,930	27,439	-27	78	1,351	-281	-281	-281	-308	-203	1,070	-54%	-11%	4%
2035 (Post mining)	642	1,933	28,830	67	81	2,742	-284	-284	-284	-217	-203	2,458	-38%	-11%	9%
Post recovery	642	1,933	28,830	67	81	2,742	0	0	0	67	81	2,742	12%	4%	11%

Table 4 – Annual flow impacts at Elong Elong on Talbragar River – 40% reduction in groundwater model river losses

Year	Annual flow (ML/yr)			Change in annual flow from SW model (ML/yr)			River losses from GW Model (ML/yr)			Change in annual flow (ML/yr)			Change in annual flow (%)		
	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)
0 (baseline)	5,227	16,836	237,164	-	-	-	-	-	-	-	-	-	-	-	-
1	5,618	17,818	247,645	43	108	1,154	-2	-2	-2	41	106	1,152	1%	1%	0%
4	5,082	18,309	248,682	-16	161	1,267	-75	-75	-75	-91	86	1,192	-2%	1%	1%
12	4,891	18,600	248,191	-37	193	1,214	-205	-205	-205	-242	-12	1,009	-5%	0%	0%
16	4,909	18,573	249,655	-35	191	1,374	-250	-250	-250	-285	-59	1,124	-5%	0%	0%
20	4,982	17,545	249,445	-27	78	1,351	-281	-281	-281	-308	-203	1,070	-6%	-1%	0%
2035 (Post mining)	5,836	17,573	262,091	67	81	2,742	-284	-284	-284	-217	-203	2,458	-4%	-1%	1%
Post recovery	5,836	17,573	262,091	67	81	2,742	0	0	0	67	81	2,742	1%	0%	1%

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Table 5 – Annual flow impacts at Sandy Creek outlet – 80% reduction in groundwater model river losses

Year	Annual flow (ML/yr)			Change in annual flow from SW model (ML/yr)			River losses from GW Model (ML/yr)			Change in annual flow (ML/yr)			Change in annual flow (%)		
	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)
0 (baseline)	575	1,852	26,088	-	-	-	-	-	-	-	-	-	-	-	-
1	618	1,960	27,241	43	108	1,154	-1	-1	-1	42	107	1,153	7%	6%	4%
4	559	2,014	27,355	-16	161	1,267	-25	-25	-25	-41	136	1,242	-7%	7%	5%
12	538	2,046	27,301	-37	193	1,214	-68	-68	-68	-105	125	1,146	-18%	7%	4%
16	540	2,043	27,462	-35	191	1,374	-83	-83	-83	-118	108	1,291	-21%	6%	5%
20	548	1,930	27,439	-27	78	1,351	-94	-94	-94	-121	-16	1,257	-21%	-1%	5%
2035 (Post mining)	642	1,933	28,830	67	81	2,742	-95	-95	-95	-28	-14	2,647	-5%	-1%	10%
Post recovery	642	1,933	28,830	67	81	2,742	0	0	0	67	81	2,742	12%	4%	11%

Table 6 – Annual flow impacts at Elong Elong on Talbragar River – 80% reduction in groundwater model river losses

Year	Annual flow (ML/yr)			Change in annual flow from SW model (ML/yr)			River losses from GW Model (ML/yr)			Change in annual flow (ML/yr)			Change in annual flow (%)		
	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)	10th %ile dry year (1967)	50th %ile median year (1906)	90th %ile wet year (1990)
0 (baseline)	5,227	16,836	237,164	-	-	-	-	-	-	-	-	-	-	-	-
1	5,618	17,818	247,645	43	108	1,154	-1	-1	-1	42	107	1,153	1%	1%	0%
4	5,082	18,309	248,682	-16	161	1,267	-25	-25	-25	-41	136	1,242	-1%	1%	1%
12	4,891	18,600	248,191	-37	193	1,214	-68	-68	-68	-105	125	1,146	-2%	1%	0%
16	4,909	18,573	249,655	-35	191	1,374	-83	-83	-83	-118	108	1,291	-2%	1%	1%
20	4,982	17,545	249,445	-27	78	1,351	-94	-94	-94	-121	-16	1,257	-2%	0%	1%
2035 (Post mining)	5,836	17,573	262,091	67	81	2,742	-95	-95	-95	-28	-14	2,647	-1%	0%	1%
Post recovery	5,836	17,573	262,091	67	81	2,742	0	0	0	67	81	2,742	1%	0%	1%

Table 7 – Annual flow impacts at Sandy Creek outlet
*(adverse impacts highlighted in **bold text**, additional sensitivity run impacts highlighted in **red text**)*

Site	Location	Groundwater fed under baseline conditions?	Groundwater fed at Year 4?	Groundwater fed at Year 12?	Groundwater fed at Year 16?	Groundwater fed at Year 20?
1	Talbragar River upstream of Sandy Creek confluence	No	No	No	No	No
2	Sandy Creek downstream of Laheys Creek confluence	No	No	No	No	No
3	Sandy Creek downstream of Laheys Creek confluence	Potentially	Potentially	Potentially **No**	**No**	**No**
4	Sandy Creek downstream of Laheys Creek confluence	No	No	No	No	No
5	Sandy Creek downstream of Laheys Creek confluence	Yes	Yes	**Potentially** **No**	**Potentially** **No**	**Potentially** **No**
6	Sandy Creek upstream of Laheys Creek confluence	Potentially	**No**	**No**	**No**	**No**
7	Laheys Creek downstream of Blackheath Creek confluence	No	No	No	No	No
8	Laheys Creek at Blackheath Creek confluence	No	No	No	No	No
9	Laheys Creek at Blackheath Creek confluence	No	No	No	No	No
10	Laheys Creek upstream of Blackheath Creek confluence	Potentially	Potentially	**No**	**No**	**No**
11	Laheys Creek upstream of Blackheath Creek confluence	No	No	No	No	No
12	Laheys Creek upstream of Blackheath Creek confluence	No	No	No	No	No
13	Laheys Creek upstream of Blackheath Creek confluence	Potentially	Potentially	Potentially	Potentially	Potentially
14	Fords Creek	Yes	Yes	Yes	Yes	Yes

Attachment 1: Capillary rise in overburden

Memorandum

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13 May 2013

To Trish McDonald
Cobbora Holding Company Pty Limited
From Timothy Rohde

Subject Capillary rise in overburben

Dear Trish,

The purpose of this memorandum is to explain the concept of capillary rise, how it is calculated and whether the proposed rehabilitation strategy for final voids at the Cobbora mine is appropriate. The closure strategy is to backfill the voids to 3 m above the permanent water table post-mining. Capillary rise in the reinstated profile could interfere with the successful revegetation of the site if excess soluble salt from backfilled overburden or groundwater inhibits plant growth.

1 What is capillary rise?

Water in soil or rock (this includes overburden) is held to the surface of the particles by adhesive and cohesive forces. The downward movement of water occurs when gravitational forces exceed adhesive-cohesive forces. When atmospheric forces from evaporation and transpiration exceed both gravitation and adhesive-cohesive forces then it is possible for soil water to rise back towards the surface. This is capillary rise.

2 The calculation of capillary rise

Refer to Annexure A. The annexure also includes some basic details on how to interpret a soil water characteristic curve (SWCC).

The extent of capillary rise depends somewhat on soil texture. Capillary rise is usually greater with fine-textured (small pore size) soils then coarse-textures (large pore size) rock. This is illustrated in Figure 1 by the comparison of SWCCs for topsoil to overburden.

The success of the strategy is a question of whether there is a risk of capillary rise of groundwater, carrying salts that may inhibit vegetation growth.

Figure 1 presents two indicative SWCC for topsoil and overburden. The SWCC have been taken from a database of curves contained within the *Soilvision* program. Figure 1 illustrates that that the potential capillary rise potential of overburden is less than 0.1 m By comparison topsoil is much higher with a maximum potential capillary rise of 20 m.

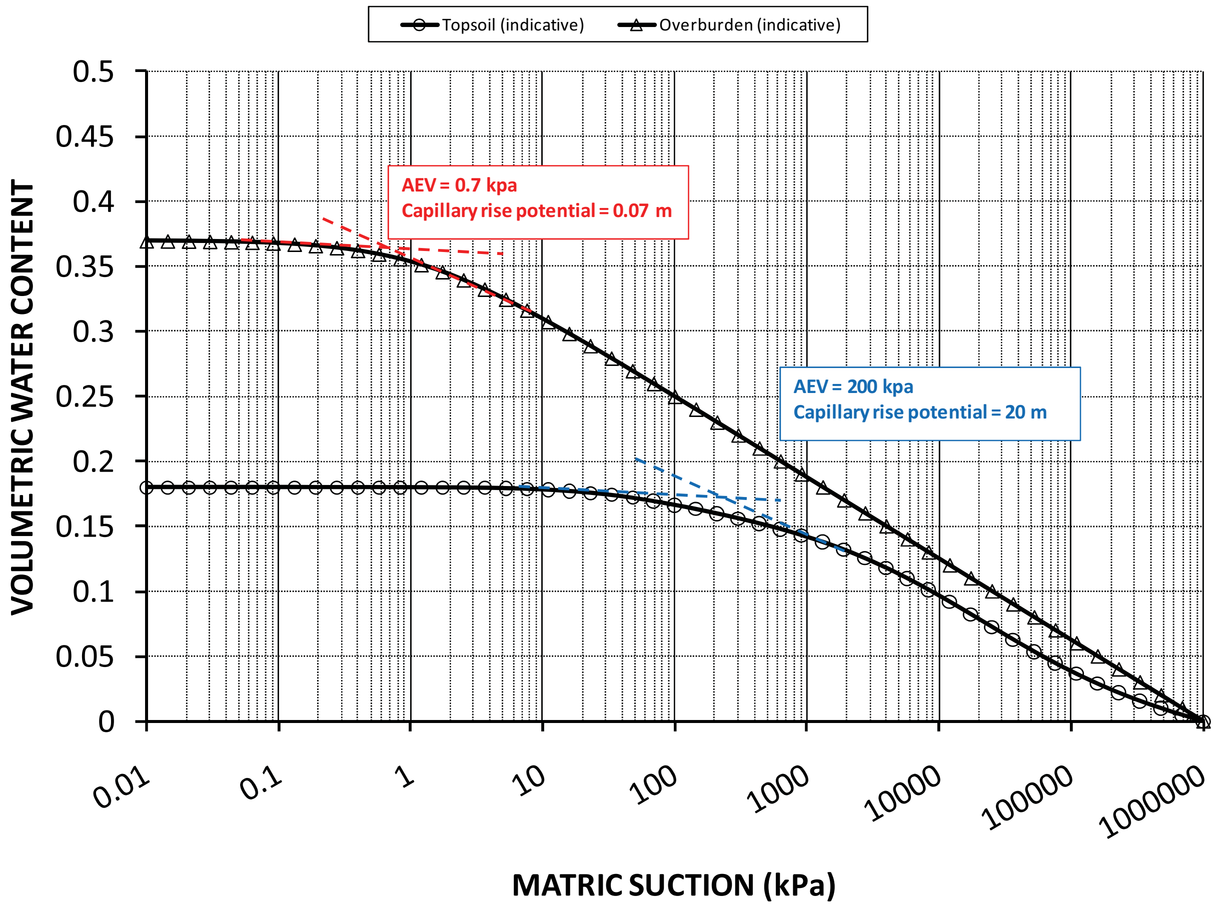

Figure 1 Indicative soil water characteristic curve for topsoil and overburden (Soilvision)

3 Cobbora mine rehabilitation strategy

Capillary rise in fresh and competent overburden is expected to be far less than 3 m. This allows for a layer of soil to support native vegetation.

Provided that the overburden does not contain considerable amounts of kalonite or illite, the weathering process will not result in a pore size distribution that is closer to soil over time. Weathering is not expected to greatly influence the capillary rise potential over time. However, if this does occur, capillary rise in weathered overburden is expected to be far less than 3 m.

The closure strategy for the placement of Cobbora mine overburden in voids is expected to be sufficient to prevent capillary rise.

4 Credentials

This appraisal of whether the conceptual design is suitable at Cobbora mine has been completed by Timothy Rohde. Timothy has a PhD (mining engineering), Graduate Diploma (mined land rehabilitation) and a Bachelor of environmental science (natural resource science). He has practice in the field of mine closure for the last 10 years, specialising in landform and cover design.

The appraisal was completed based on a review of *Mine Rehabilitation Strategy: Cobbora Coal Project* prepared by GSS Environmental. The potential for capillary rise has been determined from indicative air-entry values derived from the *Soilcover* program.

Inc. Annexure A

Appendix A

SWCC interpretation

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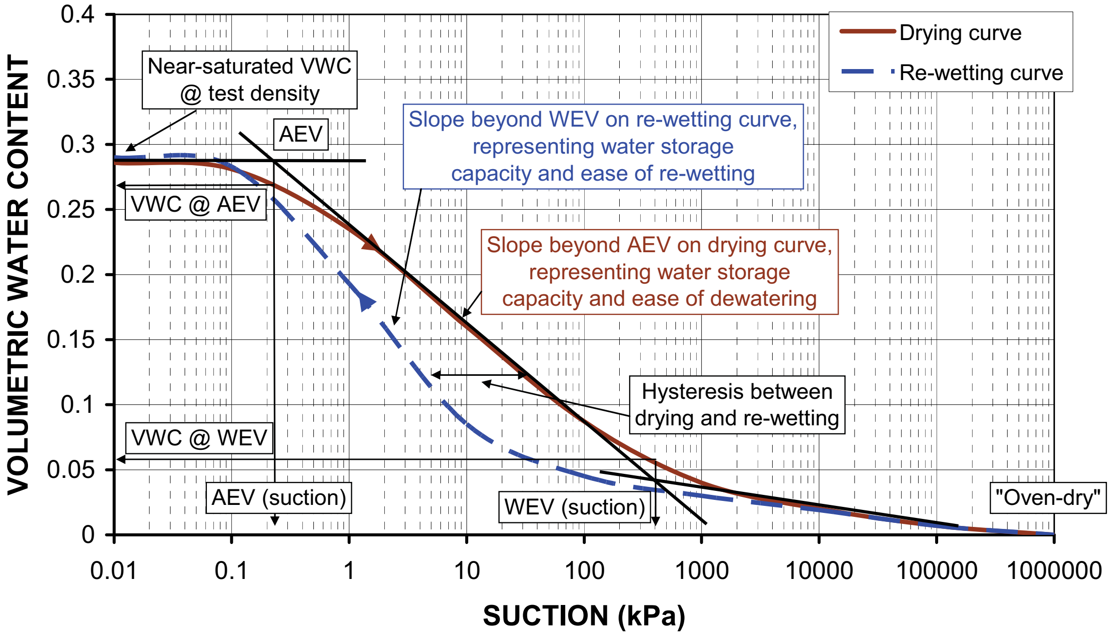
SWCC INTERPRETATION

The Soil Water Characteristic Curve (SWCC) is fundamental to understanding unsaturated soil mechanics. It is a plot of soil water (conventionally in terms of volumetric water content = volume of water/total volume; but it could also be in terms of degree of saturation = volume of water/volume of voids; or gravimetric moisture content = mass of water/mass of solids, expressed as a %.

The key elements of the SWCC are the following (Figure 1).

- The intercept on the vertical axis represents ***near-saturated conditions*** at the test density (the higher the density the lower the intercept, and increasing the density will induce drainage).
- The break in the curve at a high degree of saturation or high water content, referred to as the ***Air-Entry Value*** (AEV) on drying, beyond which the material is unable to remain saturated, and air starts to replace any further moisture lost from the pores of the material. Up to the AEV, the material is essentially saturated (degree of saturation S > 85%) and suction effects can be ignored. The capillary rise in metres at the AEV = AEV/9.81.
- The slope of the curve at matric suctions higher than the AEV. The flatter the curve, the more water the material is able to ***"store"***, and the harder it is to ***dewater*** (that is, the higher the applied pressure required to effect dewatering). Over this portion of the curve, matric (or capillary) suction and liquid water flow dominate.
- The break in the curve at a low degree of saturation or low water content, referred to as the ***Water-Entry Value*** (WEV) on re-wetting, beyond which osmotic suction and water vapour flow dominate. ***The WEV is the suction at which the material starts to rapidly wet up on re-wetting***. As the material dries beyond the WEV, the salt concentration of the diminishing pore water increases and so too does the osmotic suction. Beyond the WEV, further dewatering is more difficult to achieve, as evidenced by the flatter curve. Evaporation continues unabated to about 3,000 kPa suction, thereafter decreasing at an increasing rate and ceasing at a suction of about 100,000 kPa.
- The ***"oven-dry"*** (zero moisture) state corresponds to a suction of 1,000,000 kPa, for all materials.
- There is a ***hysteresis*** between drying and re-wetting cycles. As a soil desaturates, moisture is first lost from the largest pores, with residual moisture retreating to ever-finer pores, requiring ever-higher matric suctions to remove it. As a soil re-wets, the largest pores saturate first, with the finer pores saturating last, but at much lower matric suctions than were required to drain them during the drying cycle.
- Over the suction range up to about 1,000 kPa, matric (or capillary) suctions dominate, while above about 1,000 kPa the increasing concentration of salts in the pore water mean that osmotic (or solute) suctions come to dominate. Most soil-like materials exist at a suction of < 10,000 kPa, and hence matric suction usually dominates. An exception is salt pan deposits and hypersaline tailings.



SWCC data are conventionally measured in the laboratory using a Tempe cell and the curves are then fitted to the measured data using the method of Fredlund and Xing (1994)[1]. Field SWCC data may also be collected, and the curve fitted using the same method. The laboratory and field data may produce quite different SWCCs, the differences being greatest where there is significant structure or cementation in situ, which is destroyed on sampling and laboratory testing. The SWCC for a material with significant structure or cementation is shifted to the right, having a much higher AEV and steeper post-AEV slope. An estimate of the laboratory SWCC may also be obtained from the particle size distribution, density and specific gravity of the material using the Fredlund *et al*. (1997)[2] and the library of data contained within the program *SoilVision*.
In addition, the saturated hydraulic conductivity of the material at a representative density may be obtained by constant or falling head testing or calculated from consolidation testing.

From the SWCC and measured saturated hydraulic conductivity, the unsaturated hydraulic conductivity function of the material may be calculated using the method of Fredlund *et al.* (1994)[3].

---

[1] Fredlund, D.G. and Xing, A. (1994). Equations for the soil-water characteristic curve. *Canadian Geotechnical Journal*, **31**, 521-532.
[2] Fredlund, M.D., Fredlund, D.G. and Wilson, G.W. (1997). Prediction of the soil water characteristic curve from grain size distribution and volume mass properties. *Proceedings of 3rd Brazilian Symposium on Unsaturated Soils, Rio de Janeiro, Brazil, 22-25 April 1997*, 12 pp.
[3] Fredlund, D.G., Xing, A. and Huang, S. (1994). Predicting the permeability function for unsaturated soils using the soil water characteristic curve. *Canadian Geotechnical Journal*, **31**, 533-546.

Appendix E

Revised mine plan - groundwater and surface water assessment

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A GRI Rating: Sustainability Report 2011

Memo

Date	31 July 2013
To	Phil Towler, EMM Trish McDonald, CHC Andrew Krause, CHC
From	Robert Leslie
Ref	2162570C-WAT-MEM-009 RevC
Subject	Cobbora Coal Project - Assessment of revised mine plan in response to PAC recommendations

1. Introduction

In response to the Planning Assessment Commission (PAC) review recommendations for the Cobbora Coal Project (the Project), Cobbora Holding Company (CHC) has developed a revised mine plan in order to reduce ecological impacts, minimise dust, maximise land productivity of the final landform and reduce the number of pits active simultaneously.

The revised mine plan is presented in the CHC report Revised Mine Plan, Technical and Financial Consideration, doc. no. M01-CHC-100-RP-ENV-0001. The revised mine plan sequence differs from the mine plan that was assessed by Parsons Brinckerhoff (PB) for the Groundwater and Surface Water Assessments prepared in January 2013 and based on the mine plan developed in December 2012.

This report details findings of an assessment of the impacts of the modified mine plan on the groundwater and surface water environment by drawing on the considerable body of technical work already undertaken as part of the aforementioned groundwater and surface water assessments.

2. Methodologies

2.1 Overview

The Groundwater and Surface Water Assessments identify potential impacts of the Project on the water environment through a series of linked technical analyses summarised in the flow chart below:

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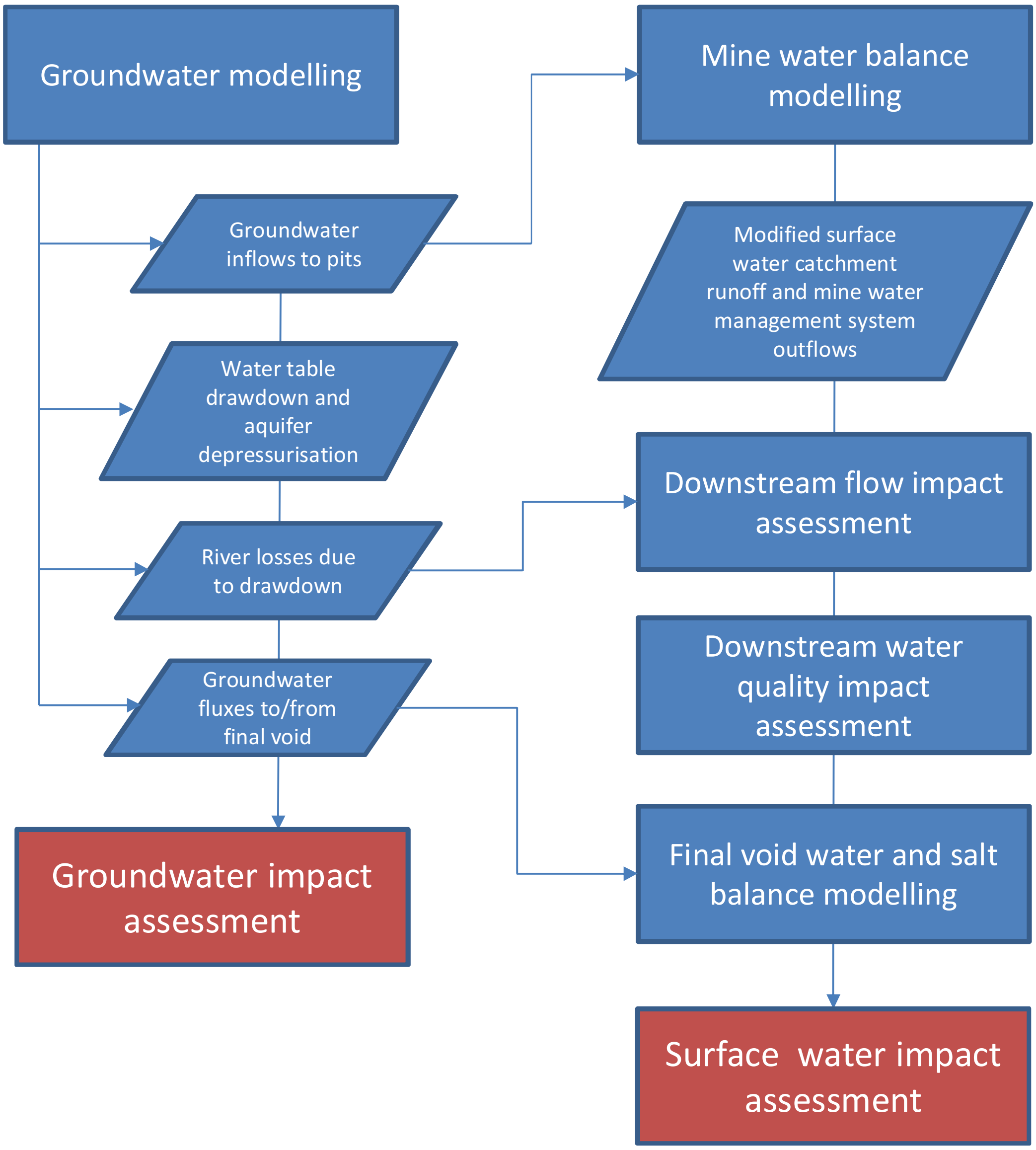

For the purpose of this assessment, the approach was to undertake a confirmation level of assessment which identifies changes to aspects of the mine plan that affect the key impact indicators of: water table drawdown and aquifer depressurisation; mine water demand; modified surface water catchment runoffs; and mine water management system outflows.

This approach was justified on the basis that:

- Mine voids are similar or less extensive than for the previous mine plan, and therefore the volumes of groundwater inflows to the pits will be similar or less, resulting in similar or less impacts on drawdown and river losses.
- Mine water demand is similar or less than for the previous mine plan, and therefore the amount of water required to be harvested from the Project area to help meet the demand will be similar or less.
- The extent of disturbance of the surface water catchments within the Project area is similar or less than for the previous mine plan, and therefore the number and volume of sedimentation and contaminated water dams will be similar or less; resulting in similar or less impacts on flow and water quality in the surface water catchments and creeks downstream of the Project area.

The detailed methodologies are described in the following sections.

2.2 Groundwater assessment

A high level groundwater assessment compared the two mine plans (December 2012 and May 2013) in terms of the likely groundwater drawdown and pit inflow. The basic premise for the analysis is that, to a first order approximation, the magnitude of groundwater impacts and groundwater inflow at a particular site is proportional to the area and depth of a mine pit below the water table. These parameters therefore can be considered as a broad proxy for groundwater impacts related to contrasting mine plans at the site (assuming the overall mining rate and pit locations are broadly similar). The following approach was adopted:

- Predicted saturated pit extents were determined for both December 2012 and May 2013 mine for mine years 1, 4, 12, 16 and 20 using an unconfined water table surface inferred from groundwater elevation data obtained as part of the original groundwater assessment program (Cobbora Coal Project Groundwater Assessment, January 2013).
- Using the saturated pit extents, the volume, pit perimeter and combined surface area of pit wall and floor below water table was calculated for each of mining areas A, B and C for both December 2012 and May 2013 mine plans.
- Using the proportion of pit shell surface below water table as a proxy for mine inflows, key changes likely to result from the modified May 2013 mine plan were determined for mine years 1, 4, 12, 16, 20 and post mining year 21.

2.3 Surface water assessment

The surface water assessment compared the December 2012 and May 2013 mine plans using three methods:

- Comparison of land use areas for the two mine plans for the mine years 1, 4, 12, 16 and 20.
- Comparison of project water demands for the two mine plans for the mine years 1, 4, 12, 16 and 20.
- Comparison of the final void lake water and salt balance for the two mine plans.

These methodologies are described in the following sections.

2.3.1 Land use area and water demand comparison

The impacts of the Project on the surface water environment are related to the extent of disturbance of the surface water catchments within the Project area, and the water demand for mining operations which affects the amount of water harvested from the surface water catchments within the Project area, as follows:

- Area of active mine: This contributes to the disturbance area in the surface water catchments and loss of natural catchment runoff to the downstream creeks (i.e. rainfall on the pit voids does not run off to local creeks but is instead captured in contaminated water dams and harvested for use in the mine water management system). These areas therefore reduce natural catchment flows.
- Area of tailings emplacements: As above.
- Area of active overburden emplacement: This contributes to the disturbance area in the surface water catchments and drains to sedimentation dams from which water is harvested for use in the mine water management system or is released to the environment from the dams.
- Area of rehabilitated emplacement: As above.

- Area of established rehabilitated emplacement: This area drains directly to the creeks but has different runoff characteristics to the undisturbed catchment it replaces.
- Infrastructure areas: Runoff from these areas is captured in contaminated water dams and harvested for use in the mine water management system. These areas therefore reduce natural catchment flows.
- Haul road areas: These areas drain to sedimentation dams from which water is harvested for use in the mine water management system or is released to the environment from the dams.
- Mine water demand: The mine water demand determines the amount of water harvested from the uncontaminated surface water catchments within the Project area, such as the area of active and rehabilitated waste rock emplacements and haul roads. If mine water demand reduces, then the amount of water harvested from the site surface water catchments should also reduce, allowing more water from the disturbed but uncontaminated catchments to be released to the environment.

If all elements above reduce or remain very similar to the previous mine plan for each year, and the rate of mining (determined as the rate of change of each land use type listed above between the milestone years) is less than or similar to that of the previous mine plan, then it can be inferred that the impacts of the Project will be less than or similar to those already determined for the previous mine plan.

The potential impacts of the revised mine plan were assessed by calculating all of the above elements and comparing these for both mine plans. In addition, the total mine water demand and significant components of the mine water demand were also compared for both plans. These comparisons were done for mine years 1, 4, 12, 16 and 20, which were the key milestone years analysed in the January 2013 Water Balance and Surface Water Management System Report (which formed Appendix E of the January 2013 Surface Water Assessment).

2.3.2 Final void lake water and salt balance modelling comparison

The Water Balance and Surface Water Management System Report (January 2013) predicted the final void lake water and salt balance following completion of mining.

The final void water balance model was developed using GoldSim software. The model was used to calculate the volume of water in the final void at the end of each day taking into account rainfall-runoff inflow, groundwater inflows/outflows and evaporation. The model was also used to calculate the salinity concentration in the final void at the end of each day. Instantaneous mixing of the various inflow types was assumed in the model, and no allowance was made for the stratification of the final void.

The final void water balance model was simulated at a daily time step for a period of 1,000 years. The model was simulated 100 times using 100 replicates (or sequences) of stochastic rainfall data. This method allows for climate variability and potential impacts of climate change.

This analysis was repeated with the new mine plan, which required input of a new level-area relationship for the final void in mining area B. The same groundwater inflow – level relationship was adopted as for the December 2012 mine plan. To address uncertainty in adopting this previous relationship, a sensitivity test of the water and salt balance model was also undertaken with an increased groundwater inflow of 50%.

3. Results

3.1 Groundwater assessment

The extent of pit shell surface area extending below water table (BWT) for each of pits A, B and C are shown in Figure 1 and Table 1 for both December 2012 and May 2013 mine plans.

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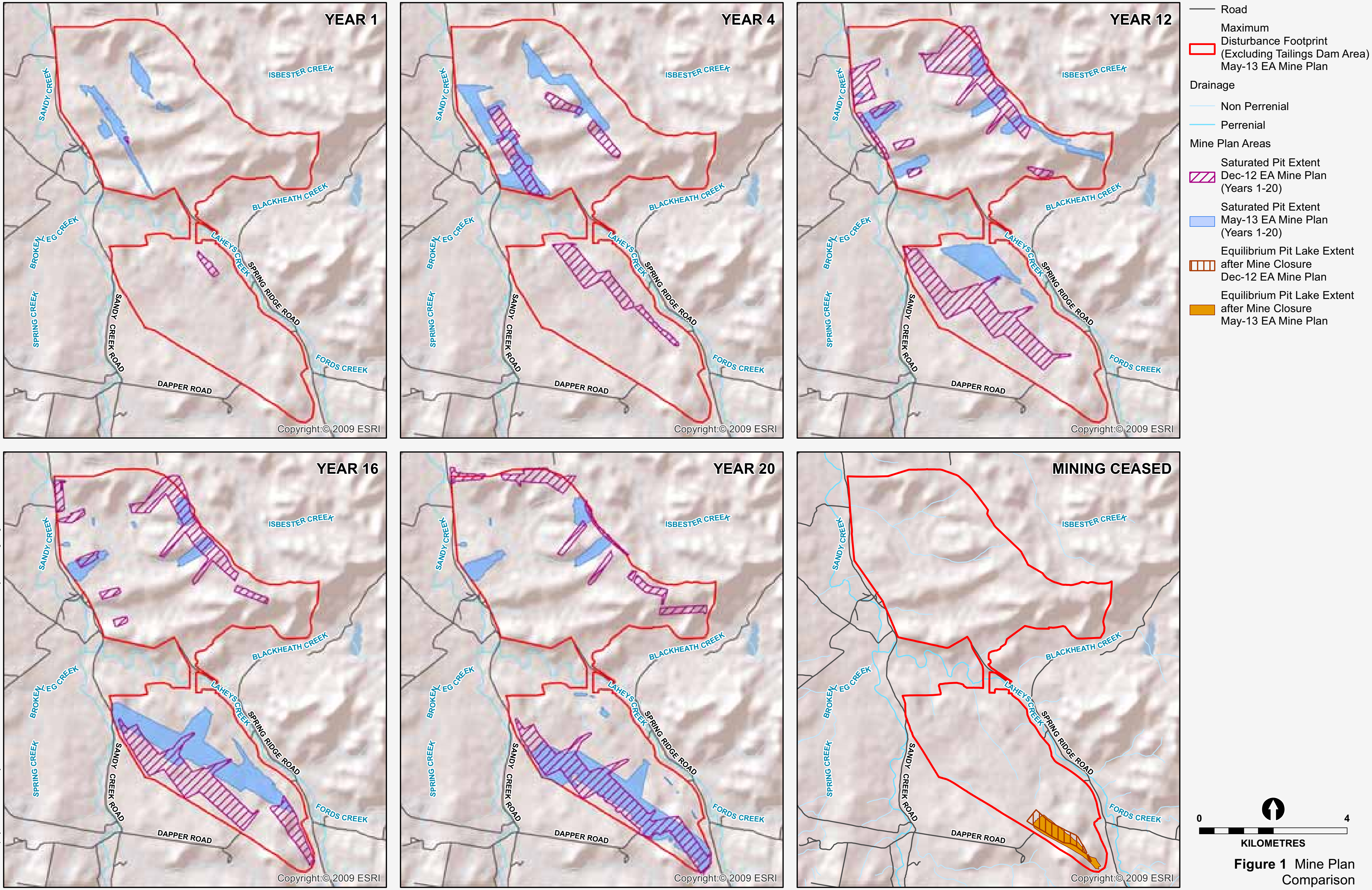


Figure 1 Mine Plan Comparison

Table 1 Pit shell surface area (m^2) below water table for December 2012 and May 2013 mine plans

Year	December 2012			May 2013			Relative % Difference		
	Pit A	Pit B	Pit C	Pit A	Pit B	Pit C	Pit A	Pit B	Pit C
1	12,240	126,828	-	640,319	-	324,013	+5,132%	↓	↑
4	897,591	1,189,267	516,043	1,639,047	-	939,636	+83%	↓	+82%
12	1,017,339	2,910,713	2,506,933	712,030	1,158,430	1,097,502	-30%	-60%	-56%
16	587,052	3,285,830	1,990,697	435,025	3,513,512	567,555	-26%	+7%	-71%
20	779,929	3,617,963	848,347	435,025	3,979,757	567,555	-44%	+10%	-33%
21	-	1,379,065	5,865	-	5,048,048	-	-	+266%	↓

Key:
↓ December 2012 pit shell is BWT, May 2013 pit shell is AWT
↑ December 2012 pit shell is AWT, May 2013 pit shell is BWT

Relative percentage difference is used as a basis to describe the differences between the May 2013 and December 2012 mine plans for prescribed mine years 1, 4, 12, 16, 20 and 21. These differences are further illustrated in Figures 2 and 3 which compare pit shell surface area and pit volume below water table for the two mine plans.

Figure 2 – Comparison of pit area below water table over years from mine inception

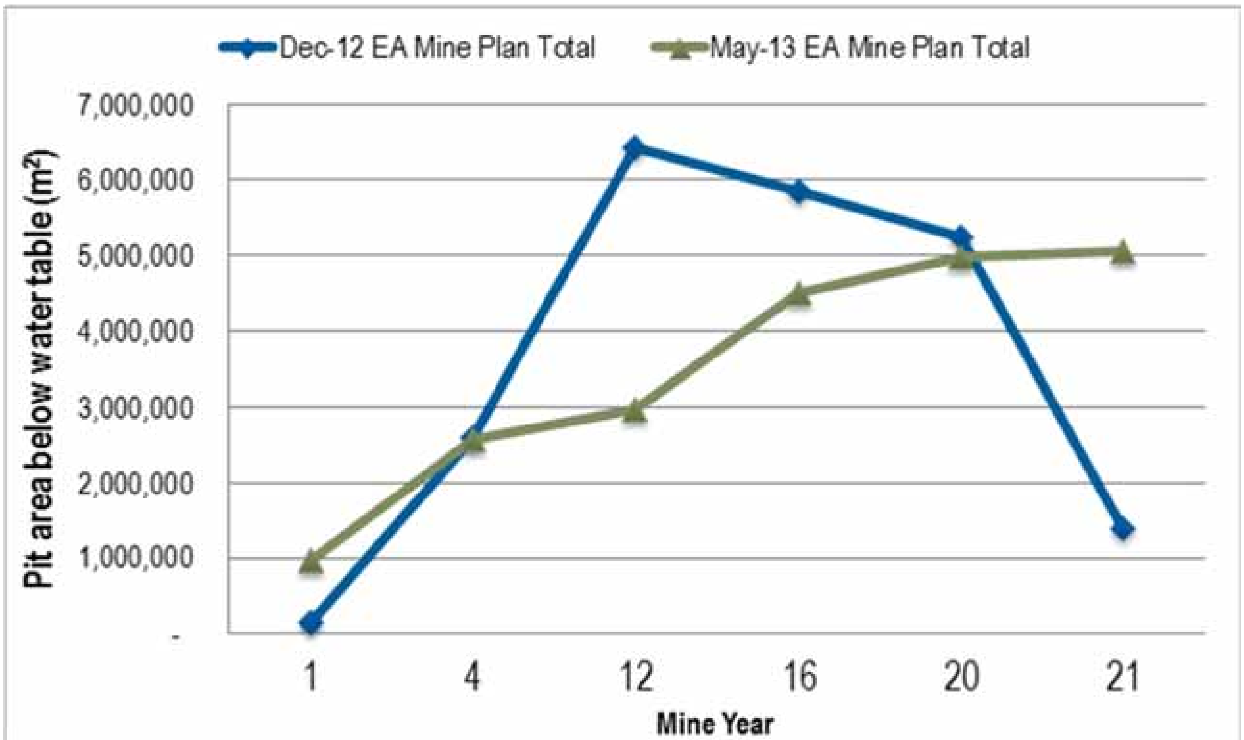
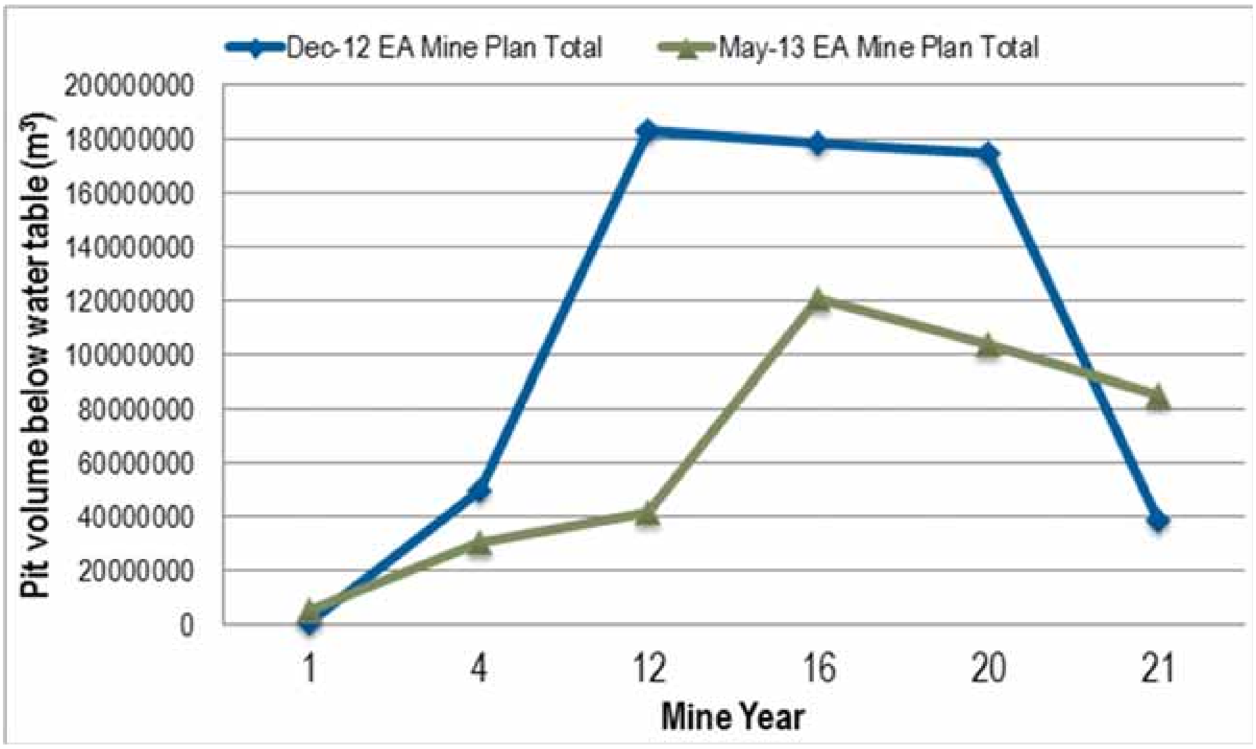


Figure 3 – Comparison of pit volume below water table over years from mine inception



Using the proportion of pit shell surface BWT as a proxy for mine inflows, changes likely from the December 2012 mine plan to the May 2013 mine plan are as follows:

- In general terms, the May 2013 mine plan has similar or somewhat less excavation below the water table in the majority of mine years (4, 12, 16 and 20), than the December 2012 mine plan. It is therefore reasonable to infer that the May 2013 mine plan is unlikely to result in a greater degree of groundwater drawdown and inflow than the assessed December 2012 mine plan.
- In detail however, differences in mine schedule are such that, on a yearly basis, predicted drawdown and inflow will differ from earlier predictions throughout the life of the mine. The following key differences are noted:
  - Year 1: Pits A and C extend BWT more in the May 2013 mine plan than for the December 2012 mine plan. In contrast Pit B is inactive and remains above water table (AWT) in year 1 (of the May 2013 plan) whereas previously it had extended BWT from year 1.
  - Year 4: Pits A and C remain BWT in the May 2013 mine plan. For the mine years being considered, year 4 Pit A has its greatest pit shell surface area BWT. Pit B remains inactive and AWT whereas previously it had extended BWT. The overall area and volume of all pits BWT is similar between the two plans.
  - Year 12: Pits A, B and C all extend BWT. The extent to which pits A, B and C occur BWT is less than for the December 2012 mine plan. In the December 2012 mine plan, Pits A, B and C were most active BWT in year 12. A reduction in BWT shell of Pit A occurs between Years 4 and 12. This can be attributed to backfilling between these mine years.
  - Year 16: Pits A, B and C remain BWT. Pit A and C are no longer active, with a significant portion of BWT extent attributed to active tailings dams. Between years 12 and 16, a reduction in the BWT shell of Pit A and Pit C occurs (backfilling). The Pit B BWT shell increases by more than 300%

between years 12 and 16. The total area and volume of pits below the water table is lower for the May 2013 mine plan than the December 2012 mine plan.

- Year 20: Pit B continues to extend BWT and moves further south, with 13% greater BWT shell than Year 16. However, the total area and volume of pits below the water table is lower for the May 2013 mine plan than the December 2012 mine plan.
- Year 21 (post mining): The residual void of Pit B extends below the pre-existing water table and will gradually fill to an equilibrium level that is dependent on a number of factors, as discussed below. The void lake will continue to act as a groundwater sink as long as the pit lake level is below the water table. Revised pit void modelling (below) indicates that the equilibrium level will be below the original water table at approximately 378 m AHD. At this elevation the area and elevation of the final void lake will be similar in the May 2013 mine plan to that predicted for the December 2012 mine plan. Again, based on these outcomes it is reasonable to infer that the ongoing groundwater impacts will be similar for the two mine plans.

3.2 Surface water assessment

3.2.1 Land use area comparison

The following tables summarise the results of the land use area comparison for both mine plans for each mine schedule milestone.

Table 2 Land use area comparison – Year 1

Land use type	Area (ha)			Difference (%)
	Dec 12	May 13	Difference	
Active Overburden Emplacement	363	144	-219	-60%
Active Mine	50	221	+171	+342%
Cleared Area	256	123	-133	-52%
Haul Road/ Infrastructure	173	245	+72	+42%
Tailings Emplacement Area	38	65	+27	+71%
Topsoil Stockpile	8	87	+79	+988%

Table 3 Land use area comparison – Year 4

Land use type	Area (ha)			Difference (%)
	Dec 12	May 13	Difference	
Active Overburden Emplacement	686	386	-300	-44%
Active Mine	292	190	-102	-35%
Cleared Area	135	123	-12	-9%
Haul Road/ Infrastructure	232	268	+36	+16%
Rehabilitated Emplacement Area	340	159	-181	-53%
Tailings Emplacement Area	132	65	-67	-51%
Topsoil Stockpile	4	58	+54	+1,350%

Table 4 Land use area comparison – Year 12

Land use type	Area (ha)			Difference (%)
	Dec 12	May 13	Difference	
Active Overburden Emplacement	970	409	-561	-58%
Active Mine	537	150	-387	-72%
Cleared Area	126	152	+26	+21%
Established Rehabilitated Area	391	158	-233	-60%
Haul Road/ Infrastructure	268	266	-2	-1%
Rehabilitated Emplacement Area	736	1,203	+467	+63%
Tailings Emplacement Area	227	121	-106	-47%

Table 5 Land use area comparison – Year 16

Land use type	Area (ha)			Difference (%)
	Dec 12	May 13	Difference	
Active Overburden Emplacement	987	399	-588	-60%
Active Mine	471	286	-185	-39%
Cleared Area	143	103	-40	-28%
Established Rehabilitated Area	912	1,287	+375	+41%
Haul Road/ Infrastructure	309	266	-43	-14%
Rehabilitated Emplacement Area	875	785	-90	-10%
Tailings Emplacement Area	113	91	-22	-19%

Table 6 Land use area comparison – Year 20

Land use type	Area (ha)			Difference (%)
	Dec 12	May 13	Difference	
Active Overburden Emplacement	958	252	-706	-74%
Active Mine	444	239	-205	-46%
Cleared Area	64	37	-27	-42%
Established Rehabilitated Area	1,350	2,384	+1,034	+77%
Haul Road/ Infrastructure	274	266	-8	-3%
Rehabilitated Emplacement Area	874	342	-532	-61%
Tailings Emplacement Area	130	92	-38	-29%

3.2.2 Water demand comparison

The following tables summarise the water demand comparison for both mine plans for each mine year. These water demand tables can be compared with Tables 5-10, 5-12 and 5-14 respectively in the Water Balance and Surface Water Management System Report (January 2013).

Dust suppression demands are based on the updated mine plan which requires a shorter haul road network to service mining areas. A more sophisticated understanding of coal and waste haulage routes has also been applied to refine truck passes per road length. A minimum 80% particulate control efficiency has been applied to determine watering rates.

Table 7 CHPP make-up water demand estimates

Yr	Product coal (Mt/a)		Raw water demand (ML/a)		Mine water demand (ML/a)		Tailings return (%)		Tailings return (ML/a)		Mine water demand net of tailings return (ML/a)		Total CHPP make-up water demand (ML/a)		Difference in total CHPP make-up water demand	
	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	ML/a	%
1	0.7	6.6	33	330	134	1,067	30	30	33	219	101	848	134	1,178	+1,044	+779
4	11.2	11.4	431	475	2,188	2,025	30	30	527	431	1,661	1,594	2,092	2,069	-23	-1
12	12.0	12.0	462	507	2,345	2,440	15	15	282	280	2,062	2,160	2,524	2,667	+143	+6
16	12.0	12.0	462	487	2,345	1,800	15	15	282	168	2,062	1,632	2,524	2,119	-405	-16
20	12.0	12.0	462	48	2,345	1,787	15	15	282	165	2,062	1,622	2,524	2,108	-416	-16

Table 8 Haul road dust suppression demand estimates

Year	Dust suppression demand		Difference (ML/a)	Difference (%)
	Dec 12	May 13		
1	376	761	+385	+102
4	968	907	-61	-6
12	1,651	1,098	-553	-33
16	1,603	1,053	-550	-34
20	1,371	1,290	-81	-6

Table 9 Water demand summary

Yr	Product coal (Mt/a)		CHPP make-up water demand (ML/a)		MIA demand (ML/a)		Haul road dust suppression demand (ML/a)		Potable water demand (ML/a)		Total site demand (ML/a)		Difference in total site demand	
	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	Dec 12	May 13	ML/a	%
1	0.7	6.6	134	1,178	9	9	376	761	5	5	524	1,953	+1,429	+273
4	11.2	11.4	2,092	2,069	140	140	968	907	10	10	3,210	3,126	-84	-3
12	12.0	12.0	2,524	2,667	150	150	1,651	1,098	15	15	4,340	3,930	-410	-9
16	12.0	12.0	2,524	2,119	150	150	1,603	1,053	15	15	4,292	3,337	-955	-22
20	12.0	12.0	2,524	2,108	150	150	1,371	1,290	10	10	4,055	3,558	-497	-12

The tables indicate that the Project water demand for the May 2013 mine plan has significantly reduced for peak operational years when compared to the December 2012 mine plan.

3.2.3 Final void lake water and salt balance comparison

The final void water and salt balance GOLDSIM model was run with the May 2013 mine plan final landform. Figures 4 to 7 present the final void lake water and salt balance results for the December 2012 and May 2013 mine plans.

Figure 4 – Final void lake water level estimates – December 12 mine plan (copy of Figure 8.8 from Water Balance and Surface Water Management System Report, January 2013)

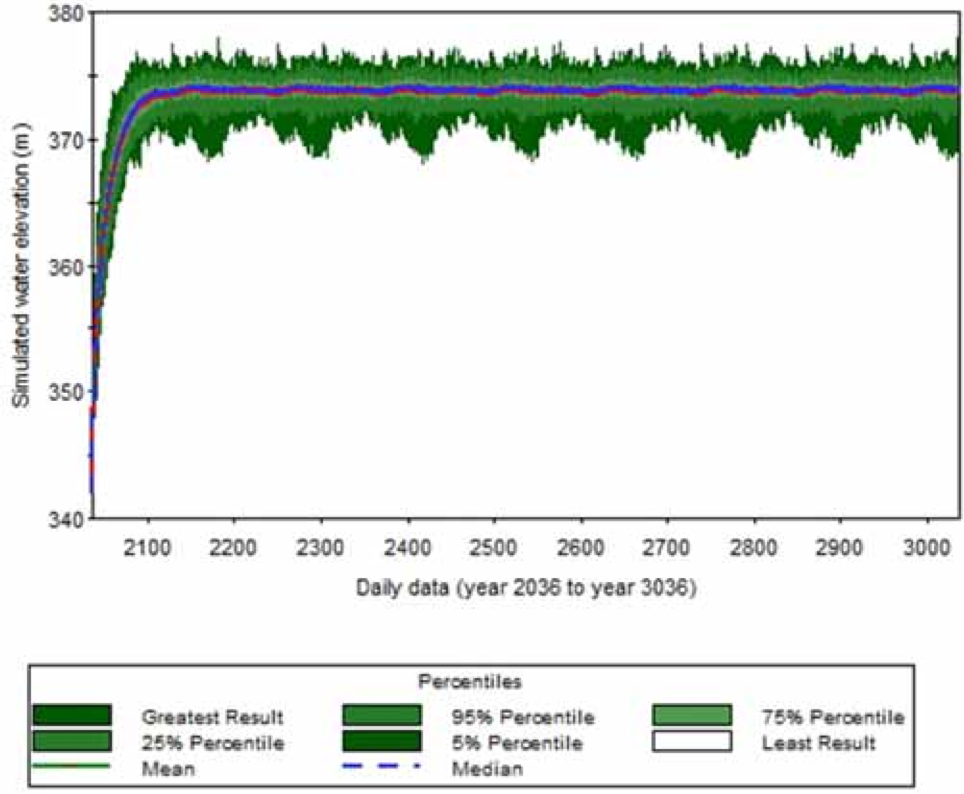


Figure 5 – Final void lake water level estimates – May 13 mine plan

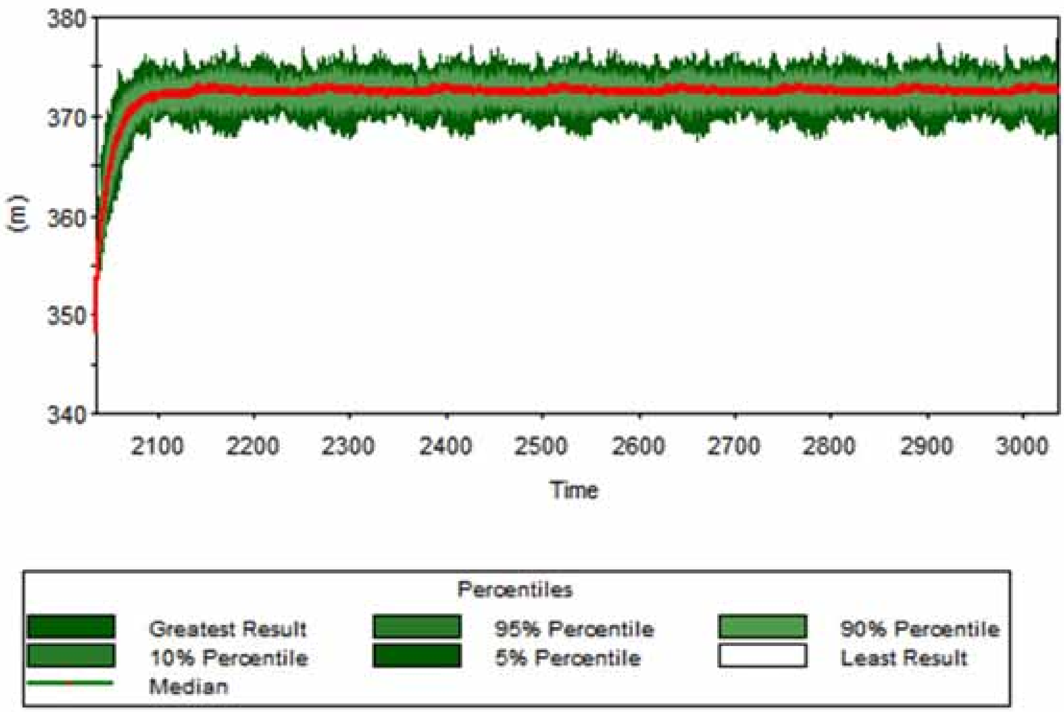


Figure 6 – Final void lake salinity estimates – December 12 mine plan (copy of Figure 8.9 from Water Balance and Surface Water Management System Report, January 2013)

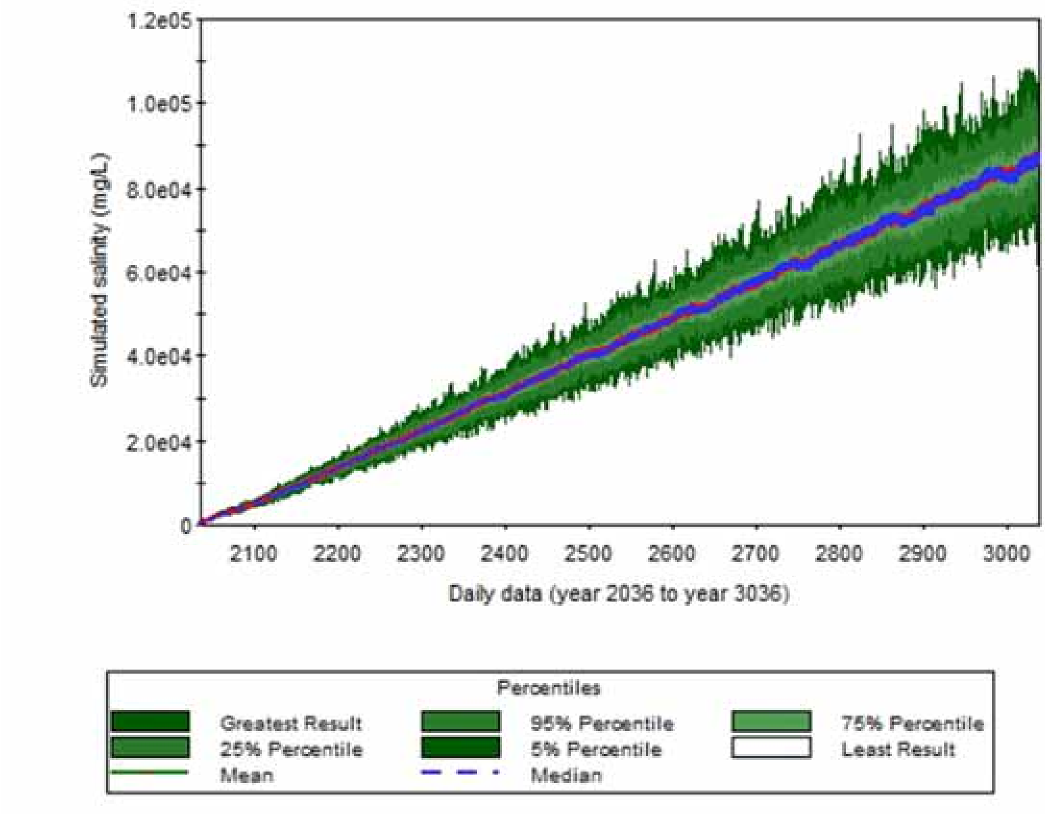


Figure 7 – Final void lake salinity estimates – May 13 mine plan

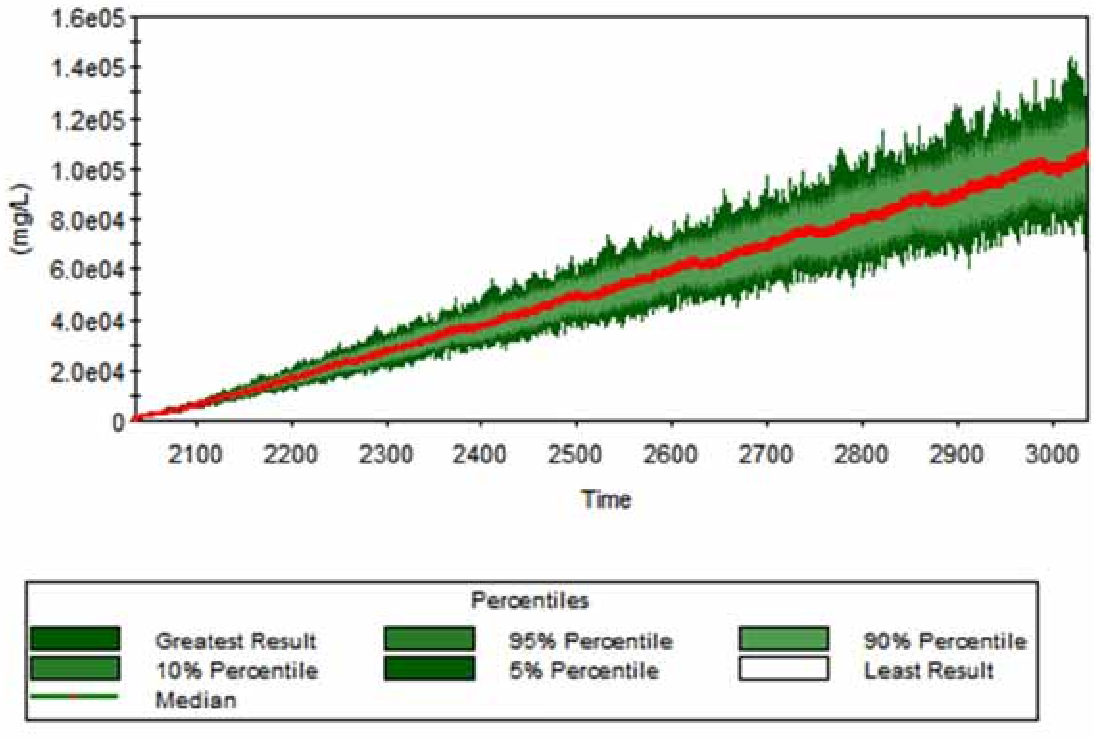


Table 10 compares key results from the final void water and salt balance modelling for both mine plans.

Table 10 Final void water and salt balance modelling key results

Result	Dec 12	May 13
Equilibrium lake level – greatest result* (m AHD)	378	378
Wetted surface area at equilibrium level (ha)	50.8	53.4
Spill level (m AHD)	407	384
Freeboard to spill level (m)	29	6.5
Peak salinity after 100 yrs (mg/L)	115,000	150,000
*greatest result is the highest value predicted by the simulation based on the stochastic rainfall data		

The table demonstrates that the new mine plan achieves a very similar result in terms of equilibrium lake level, wetted surface area and salinity. While there is less freeboard to the spill level due to the modified backfilling design, the equilibrium lake level greatest result remains approximately 6 metres below the spill level. The additional storage volume within the final void between the equilibrium lake level greatest result and the spill level is considerable at approximately 4,200 ML.

It should be noted that the salinity balance model assumes instantaneous mixing of the various inflow types and does not allow for stratification of the final void and establishment of a long term equilibrium in the lake salinity. The salinity estimates are therefore indicative only and the results provided above in Table 10, while differing by approximately 30%, can be considered similar in terms of the predicted hyper-saline nature of the lake.

As a sensitivity test, an additional model run was undertaken with groundwater inflows to the final void increased by 50%. These results are presented in Figures 8 and 9 below. Note that this test was run as a simple sensitivity test without the multiple sequences of stochastic rainfall data. The results produced a similar result with an equilibrium lake level of around 378 m AHD and peak salinity after 100 years of approximately 115,000 mg/L.

Figure 8 – Final void lake water level estimates – May 13 mine plan with 50% increase in groundwater inflows

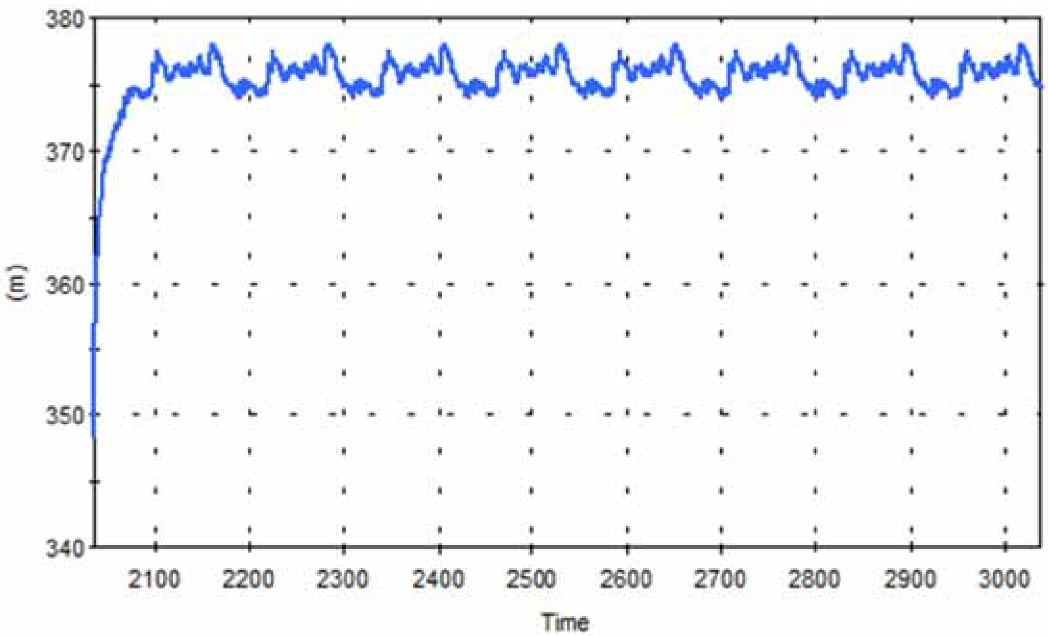
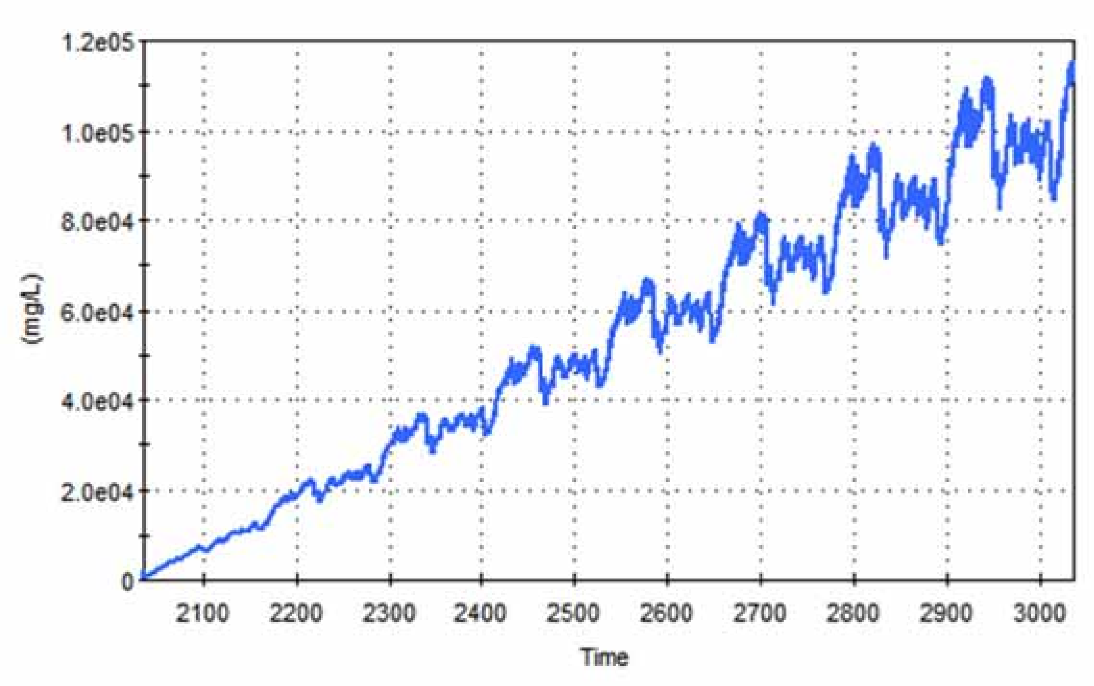


Figure 9 – Final void lake salinity estimates – May 13 mine plan with 50% increase in groundwater inflows



Further work would be required at the operational stage to determine a final void lake management solution as part of a mine closure plan. This work would be based on further monitoring data collected through the operational phase to develop a more detailed understanding of the long term behaviour of the final void lake.

4. Conclusions

The following conclusions are drawn from the assessments.

4.1 Groundwater assessment

- Overall the mine plans are broadly comparable in terms of mine pit area below the water table, and therefore likely impact footprint.
- Impacts to groundwater will likely occur earlier in mine schedule compared to the December 2012 mine plan.
- The geometry of the final void is similar in both mine plans, as is the equilibrium water level of the final void lake, as determined by surface water modelling. As for the December 2012 mine plan, the final void lake for the May 2013 mine plan will act as a groundwater sink.

4.2 Surface water assessment

- The land use area comparison demonstrates more rapid development of mining in the early years compared to the December 2012 mine plan, with higher areas of active mine, haul road/infrastructure, tailings emplacement and topsoil stockpiling.
- During peak mine years (year 4 onwards), the new mine plan is less extensive, with subsequent significant reductions in areas of active mine, active overburden emplacement and active tailings emplacement.

- During the peak mining years, the disturbed areas (active mine, active overburden emplacement, active tailings emplacement) are reduced and the rehabilitated areas (rehabilitated and established rehabilitated emplacement) are increased.
- The reduction in disturbed areas and increase in rehabilitated areas during peak mining indicates that surface water impacts of the new mine plan will not be greater than the December 12 mine plan in terms of volumes of runoff from contaminated/disturbed areas.
- The water demands are increased for the early years reflecting the more rapid development, but are significantly reduced for the peak mining years by up to 22%; therefore, the new mine plan will place significantly less demand on imported water from the Cudgegong River source, and will allow CHC greater flexibility in trading unused water allocation back into the market for agricultural use.
- The final void analysis demonstrates that the new mine plan will produce a similar final void lake in terms of peak lake water level, wetted surface area and salinity. As previously, this will act as a groundwater sink and will not produce a groundwater flow towards Sandy Creek.

4.3 Overall summary conclusions

- The new mine plan demonstrates more rapid development of the mine but lesser extent of mining at the peak operational years.
- The modelled final void impacts on the groundwater and surface water environments are similar, with similar equilibrium lake levels that will govern the long term groundwater drawdown and result in the final void acting as a groundwater sink with no saline flux towards Sandy Creek.
- Impacts on the groundwater and surface water environments will be within a similar range to those assessed in detail for the December 2012 mine plan, with some lesser impacts possible due to the lesser extent of disturbed area and lower demands for imported water.

Appendix F

Revised mine plan - mine rehabilitation strategy

GSS ENVIRONMENTAL
Environmental, Land and Project Management Consultants

SLR

Our Ref: Rp_EMG00-012_Rehab strategy_Letter Report_20130806

6th August 2013

Phil Towler
Associate Director
EMGA Mitchell McLennan
Ground Floor, Suite 01, 20 Chandos Street
St Leonards NSW 2065

Sent via: Email Transmission

Dear Phil,

RE: COBBORA COAL PROJECT - UPDATE TO MINE REHABILITATION STRATEGY

1.0 INTRODUCTION

EMGA Mitchell McLennan Pty Limited (EMM), on behalf of the proponent, Cobbora Holding Company Pty Limited (CHC), previously engaged GSS Environmental (GSSE) to prepare the Mine Rehabilitation Strategy. This supported an Environmental Assessment (EA) to accompany a major Project Application under Part 3A of the *Environmental Planning and Assessment Act 1979* (NSW) (the EP&A Act) for the proposed Cobbora Coal Project (the Project).

The EA was placed on public exhibition for six weeks between 5 October 2012 and 16 November 2012. Subsequently, a Preferred Project Report and Response to Submissions (PPR&RTS) was prepared (February 2013) that included an updated mine plan in response to submissions and an updated Mine Rehabilitation Strategy.

The Planning Assessment Commission (PAC) reviewed the EA and PPR&RTS and provided a series of recommendations in the Cobbora Coal Project PAC Review Report (April 2013). This included revising the mine plan to reduce the footprint, to reduce the number of mining areas operating and to minimise the final void. The mine infrastructure area (MIA) has been relocated to be adjacent to the coal handling and preparation plant (CHPP) as part of these changes. The final landform has changed as a result of these revisions.

The revised mine plan reduces:

- disturbance of the biodiversity corridor north and north-east of mining areas A and C;
- impacts on threatened species and endangered ecological communities by reducing the size of the B-OOP E waste rock emplacement;

- the area of the land that will be exposed at each stage of mining;
- the final voids to one location only and reduces the areas that will not be suitable for any agricultural enterprise (Class VIII).
- will not be rehabilitated to agricultural land or woodland;
- the length of operational haul roads and hence water requirements for dust suppression; and
- the land area requirements for mine support infrastructure and CHPP.

This letter addresses these changes and has been prepared as an amendment to the Mine Rehabilitation Strategy.

The Project's disturbance footprint is 4,232 ha. The Project Application Area (PAA) has been divided into six domains for the purposes of rehabilitation planning (**Figure 1**). This division is based on the level of disturbance and type if activity (**Table 1**). An overview of the change to each domain is provided below.

Table 1 – Project Application Area: Revised Disturbance Overview

Domain	Domain Name	Disturbance Impact	Land Area	
			ha	%
1	Mining Operations Domain	High	3,739	13.7
2	Mine Infrastructure Domain	Medium	194	0.7
3	Auxiliary Infrastructure Domain	Low	168	0.6
4	Road Network Domain	Low	112	0.4
5	Raw Water Dam Domain	High	19	0.1
Sub total			**4,232**	**15.5**
6	Nil Disturbance	Nil	23,154	84.5
Total			**27,386**	**100.0**

2.0 POST-MINING LANDFORM AND LAND USE

This section provides a summary of the Project's conceptual final landform design; the post-mining landform; Rural Land Capability and Agricultural Suitability classifications; and the proposed post-mining land use.

Conceptual Final Landform Shape

It is proposed that all the upgraded and realigned roads, as well as the raw water dam, will be retained at closure. All land covered by main and auxiliary infrastructure components, with the exception of the rail spur formation, will be returned to the pre-mining landform to be capable of supporting pre-mining land uses. The three mining areas will be largely backfilled with overburden and reshaped in accordance with the landform design. Reshaping will ensure that final slopes around the margin of the elevated landform, while generally 10° or less, will not exceed 18° (except regraded highwalls). This landform will differ from its pre-mining state. **Figure 2** shows the slopes of the Mining Operations Domain and further description is provided below.

The reshaped Northern Mining Operations area, which encompasses open-cut Mining Areas A and C and the out-of-pit waste rock emplacement AC-OOP, will be an elevated landform largely composed of flat to gently inclined land with some steeper fringing slopes on the northern and eastern perimeter. Some elevated land with steeper slopes where Mining Area A meets Mining Area C. The maximum design height for the elevated landform will be approximately 40 m above the pre-mining landform. The base of the original void areas in Mining Areas A and C will be filled with overburden to become level with the surrounding land. High walls will be regraded to a slope incline of 18-32%. Regrade methods may include throw blasting and or dozer push.

The reshaped Southern Mining Operations area, which encompasses Mining Area B and out-of-pit waste rock emplacement B-OOP W, will be largely composed of flat to gently inclined land with some steeper fringing slopes on the perimeter. Mining Operations area B also contains some steeper slopes centrally located with flat land on the crest. The maximum design height for the elevated landform will be approximately 60 m above the pre-mining landform. It will contain one void in the south that will become a lake on closure with a void lake with a surface level of 278 m.

The reshaped Eastern Mining Operations area, which covers the out-of-pit waste rock emplacement area B-OOP E, will be an elevated landform with no voids. This landform will be largely composed of flat to gently inclined land with some steeper fringing slopes on the western slopes. The eastern slopes abut steeper land to the east. The maximum design height for the elevated landform will be approximately 20 m above the pre-mining landform.

Other mine-related features forming part of the final landform within the PAA will be rock-drop structures, water storage dams and sediment basins used for surface-water management and erosion and sediment control.

Post-mining Rural Land Capability Classification

The Rural Land Capability classification for the rehabilitated disturbance footprint will range from Class III to Class VIII (**Figure 3**). This will provide a post-mining landform capable of both cropping (Class III) and grazing (Class IV) enterprises along with some land that is best revegetated with trees and shrubs for erosion control (Class VI). The post-mining landform will also contain areas not suitable for any agricultural enterprise (Class VIII).

Table 2 provides the quantity of each Rural Land Capability class in the disturbance footprint for both the pre-mining and post-mining landform. Overall, there will be limited impacts on the overall quantity of Rural Land Capability classes across the PAA (**Table 3**). In summary:

- The primary changes to the Rural Land Capability Class areas will be a 535 ha increase of Class IV land (12.7% of disturbance footprint, 2.0% of PAA), and a 217 ha increase of Class VIII land (5.1% of disturbance footprint, 0.8% of PAA).
- The increases in the Rural Land Capability Class IV and VIII areas will be accompanied by a 376 ha decrease of Class VI land (8.9% of disturbance footprint, 1.4% of PAA); a 230 ha decrease of Class VII land (5.4% of disturbance footprint, 0.9% of PAA) and a 143 ha decrease of Class V land (3.4% of disturbance footprint, 0.5% of PAA).
- There will be minor changes to the areas of Class II
- There is no change to Class III land.

Table 2: Post-mining Rural Land Capability – Disturbance Footprint

Rural Land Capability	Pre-mining		Post-mining		Change	
Class	ha	%	ha	%	ha	%
II	3	0.1	0	0	-3	-0.1
III	427	10.1	427	10.1	0	0.0
IV	1,991	47.0	2,526	59.7	535	12.7
V	148	3.5	5	0.1	-143	-3.4
VI	1,379	32.6	1,003	23.7	-376	-8.9
VII	284	6.7	54	1.3	-230	-5.4

Rural Land Capability	Pre-mining		Post-mining		Change	
VIII	0	0	217	5.1	217	5.1
Total	**4,232**	**100.0**	**4,232**	**100.0**		

Table 3: Post-mining Rural Land Capability – PAA

Rural Land Capability	Pre-mining		Post-mining		Change	
Class	ha	%	ha	%	ha	%
II	572	2.1	569	2.1	-3	0.0
III	5,691	20.8	5,691	20.8	0	0.0
IV	9,785	35.7	10,320	37.7	535	2.0
V	4,474	16.3	4,331	15.8	-143	-0.5
VI	5,311	19.4	4,935	18.0	-376	-1.4
VII	1,553	5.7	1,323	4.8	-230	-0.9
VIII	0	0.0	217	0.8	217	0.8
Total	**27,386**	**100.0**	**27,386**	**100.0**		

Post-mining Agricultural Suitability

The Agricultural Suitability classification for the post-mining landform will range from Class 2 through to Class 5 (**Figure 4**). This means that the post-mining landform will contain arable land that is well suited to regular cultivation (Class 2), good grazing land that is well suited to pasture improvement (Class 3) and land that is suitable for grazing using native or unimproved pastures only (Class 4). It will also contain land that is considered unsuitable for agricultural enterprises but may allow for light grazing (Class 5).

Overall there will be small impacts on the overall quantity of Agricultural Suitability classes across the disturbance footprint (**Table 4**) and the PAA (**Table 5**). In summary:

- The primary change to the Agricultural Suitability Class areas will be will be a 392 ha increase of Class 3 land (9.4% of disturbance footprint, 1.5% of PAA).
- This increase will be accompanied by a 376 ha decrease of Class 4 land (8.9% of disturbance footprint, 1.4% of PAA).
- There will be minor changes to Class 1 and 5 land.
- There are no changes to Class 2 Land.

Table 4: Post-mining Agricultural Suitability – Disturbance Footprint

Agricultural Suitability	Pre-mining		Post-mining		Change	
Class	ha	%	ha	%	ha	%
1	3	0.1	0	0.0	-3	-0.1
2	427	10.1	427	10.1	0	0.0
3	2,139	50.5	2,531	59.8	392	9.3
4	1,379	32.6	1,003	23.7	-376	-8.9
5	284	6.7	271	6.4	-13	-0.3
Total	**4,232**	**100**	**4,232**	**100**		

Table 5: Post-mining Agricultural Suitability – PAA

Agricultural Suitability	Pre-mining		Post-mining		Change	
Class	ha	%	ha	%	ha	%
1	572	2.1	569	2.1	-3	0.0
2	5,691	20.8	5,691	20.8	0	0.0
3	14,259	52.0	14,651	53.5	392	1.5
4	5,311	19.4	4,935	18	-376	-1.4
5	1,553	5.7	1,540	5.6	-13	-0.1
Total	**27,386**	**100**	**27,386**	**100**		

Post-mining Land Use

The greatest area where land use will be altered is in the Mining Operations Domain (refer **Section 4.3**). This domain's post-mining land use is shown in **Figure 5**.

In summary, the post-mining land use in the Mining Operations Domain will be 51.1% woodland (Class IV, VI and VII), 11.4% cropping (Class III) and 34.3% pastoral land (Class IV). The remaining 3.2% of land will be associated with the final void and remaining high wall (**Table 6**).

Table 6: Post-mining Land Use – Mining Operations Domain

Land Use	Associated Rural Land Capability Class	Associated Landform	Area (ha)	Area (%)
Cropping	III	Flat to gently sloping land	427	11.4
Pastoral	IV	Flat to gently sloping land	1,278	34.3
Woodland	IV, VI, VII	Flat to steeply inclined land	1,916	51.1
Subtotal			3,621	96.8
Void & high wall	VIII	Steep to moderately inclined	118	3.2
Subtotal			118	3.2
Total			**3,739**	**100.0**

3.0 REHABILITATION MANAGEMENT STRATEGY

Progressive Rehabilitation Schedule

The Mining Operations Domain will be reshaped and progressively rehabilitated to be compatible with the proposed post-mining land use. The components of the Mine Infrastructure Domain and Auxiliary Infrastructure Domain (with the exception of the rail spur) will be decommissioned by removing all infrastructure, reducing slope angles of embankments and cuttings to blend with the surrounding landform, and revegetating with endemic species. The rail spur will have its infrastructure elements removed; however, the embankments and cuttings will be left. The upgraded and realigned roads will remain in place as an active network, whilst the haul roads will be rehabilitated. The raw water dam will also be left as it will provide water storage for post-mining agricultural activities

The Mining Operations Domain, excluding the final voids, contains 3,621 ha that will require rehabilitation works. A progressive approach to the rehabilitation of disturbed areas within the Mining Operations Domain will be adopted. It is proposed that disturbed areas be reshaped within one year of the final overburden placement and then rehabilitated to the target Rural Land Capability classes. This will ensure that areas where mining and waste rock placement is completed are promptly shaped, topsoiled and vegetated to provide a stable landform. The progressive formation of the post-mining landform and the establishment of a vegetative cover will reduce the amount of disturbed land at any one time and also reduce the visibility of mine-related activities from surrounding properties and roads. Early reprofiling and revegetation of the external embankments and cutting slopes of the emplacement areas is particularly important and will be targeted as a priority.

The progressive site rehabilitation schedule for the life of the mine is listed in **Table 7** and described below. **Figures 6.1a** and **6.1b** illustrate the progressive rehabilitation of the site for years 4, 8, 16 and closure (year 21).

Table 7: Progressive Rehabilitation – Mining Operations Domain

Rehabilitation	**Land Use**	**Rural Land Capability**	**Area**	
Year	Type	Class	ha	%
2–4	Cropping	III	0	0.0
	Pastoral	IV	53	1.5
	Woodland	IV, VI, VII	105	2.9
		Subtotal	158	4.4
4–8	Cropping	III	0	0.0
	Pastoral	IV	255	7.0
	Woodland	IV, VI, VII	268	7.4
		Subtotal	523	14.4
8–16	Cropping	III	0	0.0
	Pastoral	IV	381	10.5
	Woodland	IV, VI, VII	1,161	32.1
		Subtotal	1,542	42.6
16–21 (end of mine life)	Cropping	III	427	11.8
	Pastoral	IV	561	15.5
	Woodland	IV, VI, VII	350	9.6
		Subtotal	1,338	36.9
21-29	Cropping	III	0	0.0
	Pastoral	IV	28	0.8
	Woodland	IV, VI, VII	32	0.9
		Subtotal	60	1.7
		Total	**3,621**	**100.0**

In the Mining Operations Domain a total of 3,621 ha (**Table 6**) will be rehabilitated and 118 ha will be left as void or high wall areas. Annual rehabilitation sequencing will be as follows:

Year 1 No rehabilitation.

Years 2–4 Rehabilitation of 158 ha of land in the Northern Mining Operations area.

Years 4–8 Rehabilitation of 523 ha of land in the Northern Mining Operations area.

Years 8–16 Rehabilitation of 1,542 ha of land, which includes the entire Eastern Mining Operations area and nearly completes the rehabilitation of the Northern Operations area.

Years 16–21 Rehabilitation of 1,338 ha of land, which includes all remaining land in the Mining Operations Domain, predominately the Southern Mining Operations area, with the exception of the tailings emplacement areas.

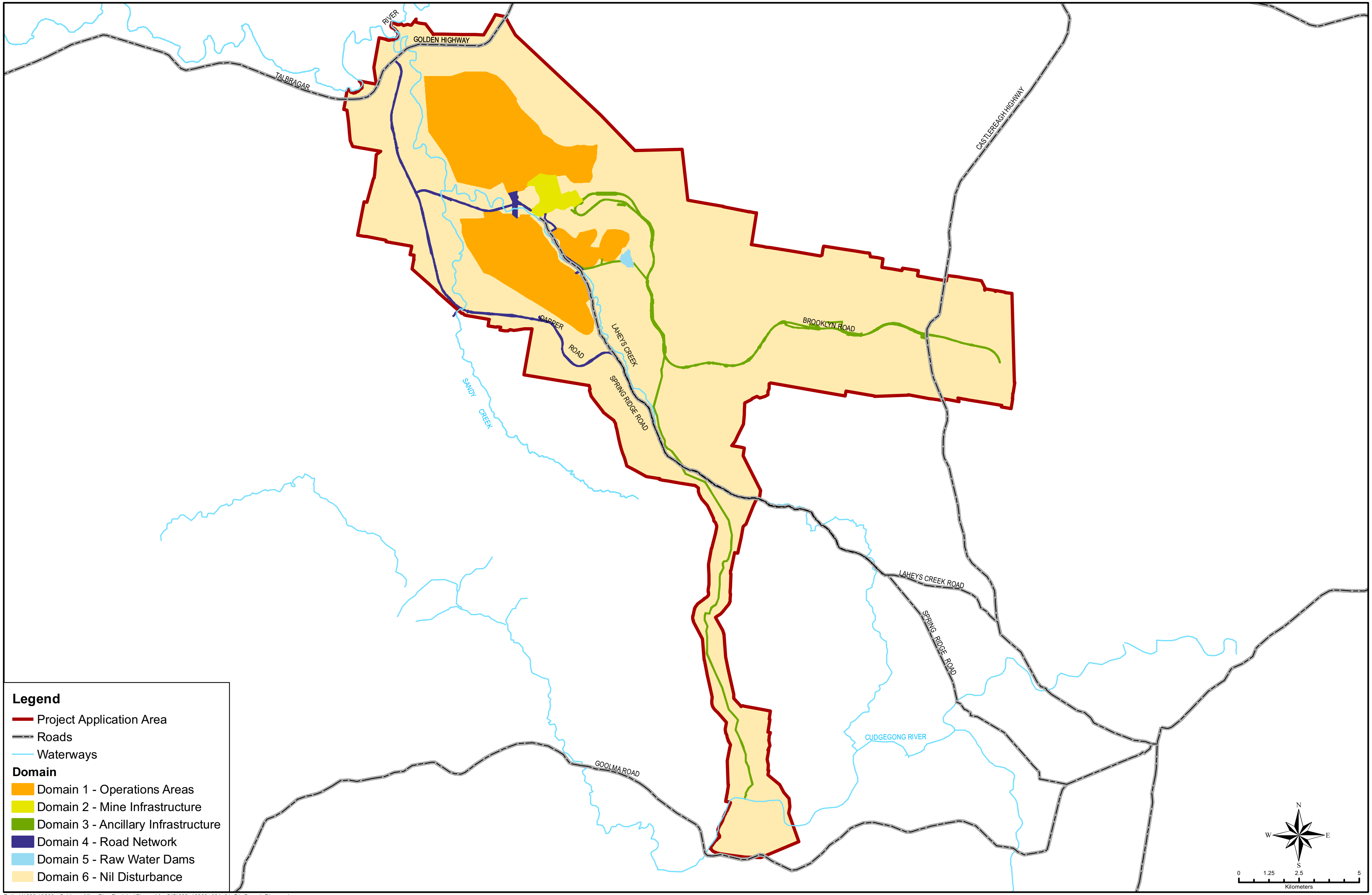
Years 21–29 Rehabilitation of 60 ha of land, which includes the stabilised and revegetated tailings emplacement areas.

4.0 CONCLUSION

This letter report has been prepared as an amendment to the Mine Rehabilitation Strategy due to changes to the mine plan to address the PAC review recommendations.

In summary:

- The existing dominant Rural Land Capability is Class IV (47.0% of disturbance footprint, 35.7% of PAA). This land is mainly used for cattle and sheep grazing.
- Other major Rural Land Capability classifications include: Class III land (10.1% of disturbance footprint, 20.8% of PAA) that can be used for the production of crops and Class VI land (32.6% of disturbance footprint, 19.4% of PAA) which, where timber has been removed, can be is used for low intensity cattle grazing along with some merino wool production. The PAA also contains a small quantity of Class II land (2.1%), which is good quality cropping land.
- The same quantity of Rural Land Capability Class III land will be reinstated following mining as is present pre-mining. There will also be a 535 ha increases in Class IV land.
- The post-mining landform has been designed to be compatible with the surrounding environment. This includes 1,916 ha of woodland.
- Rehabilitation will be progressive and disturbed areas will be reshaped within one year of the final overburden placement and then rehabilitated to the target Rural Land Capability Classes.
- This rehabilitation strategy will be continually reviewed and updated throughout the life of the Project. Five years prior to mine closure, a more detailed mine closure plan will be prepared.



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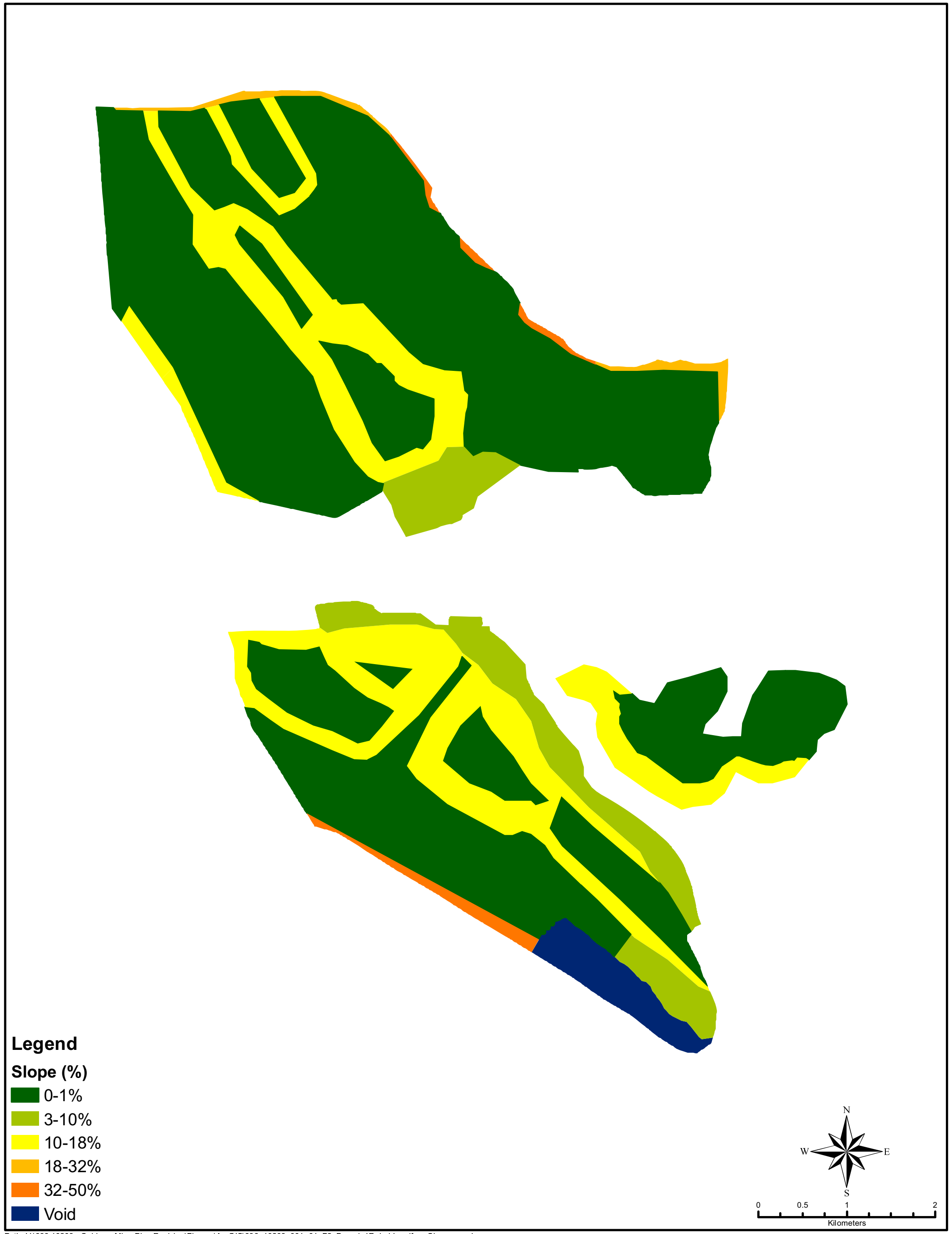
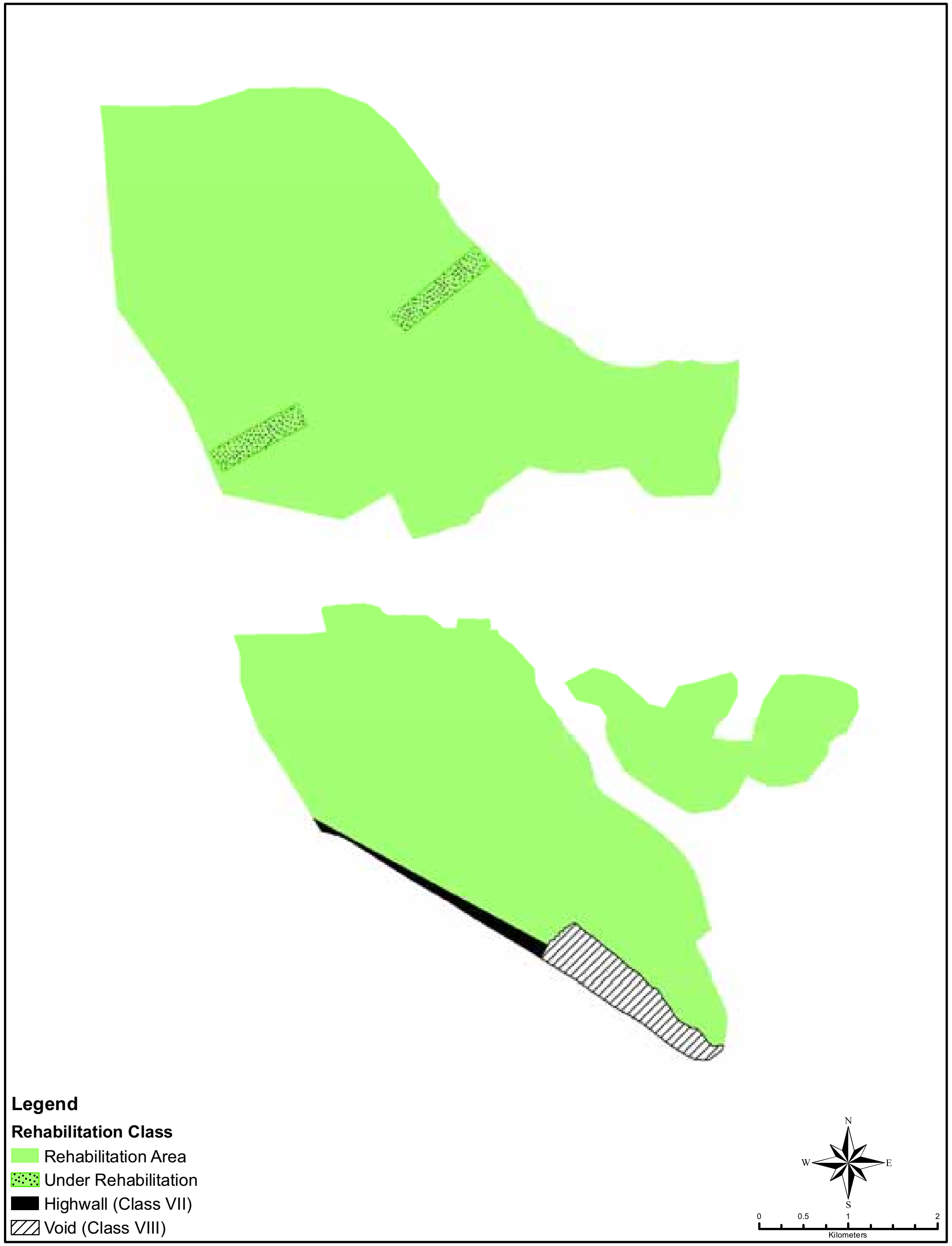
Revised Domain Plan
Cobbora Coal Project
Figure 1





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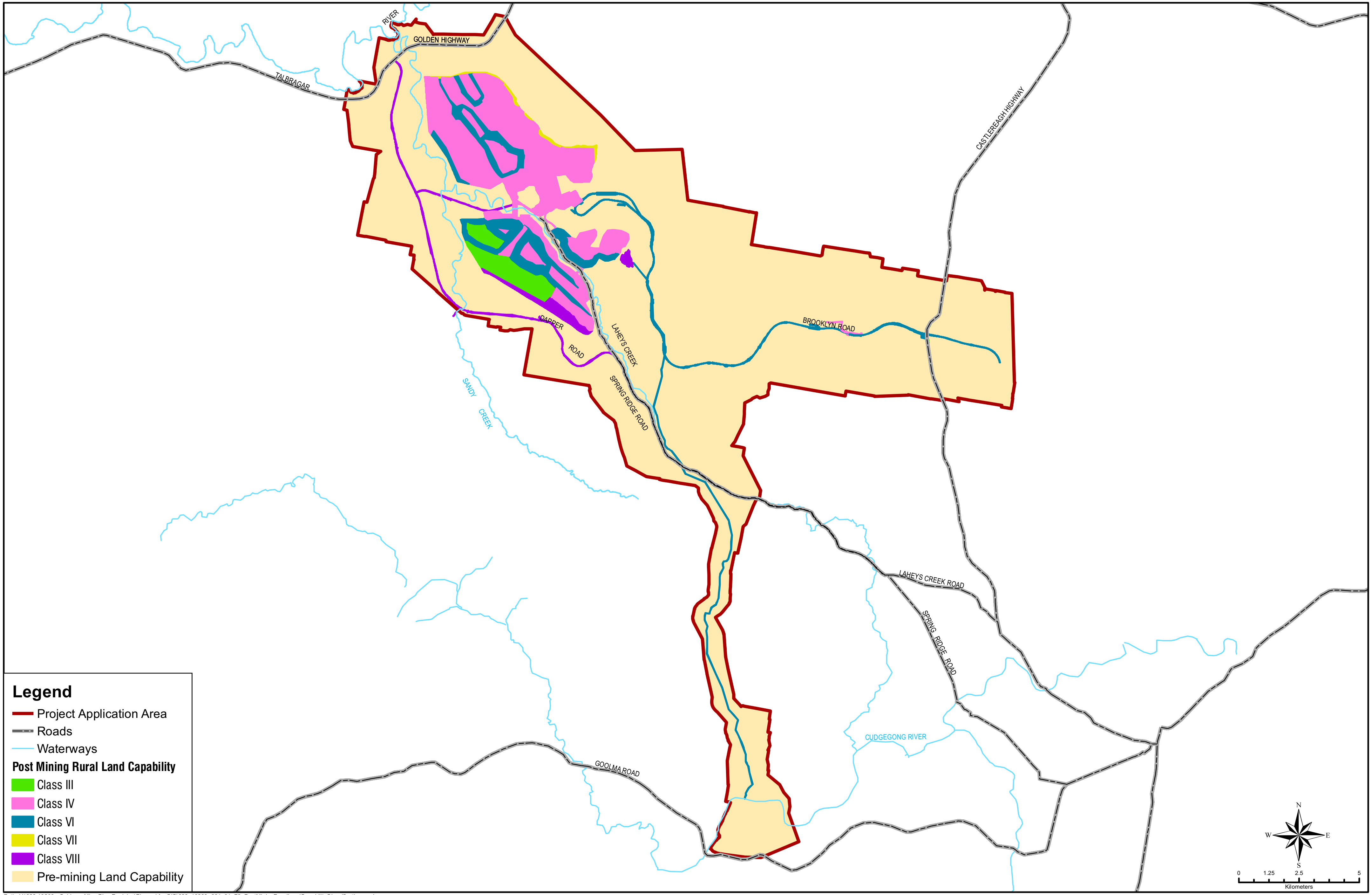
Domain 1: Rehabilitated Landform (Closure)

Cobbora Coal Project

Figure 2



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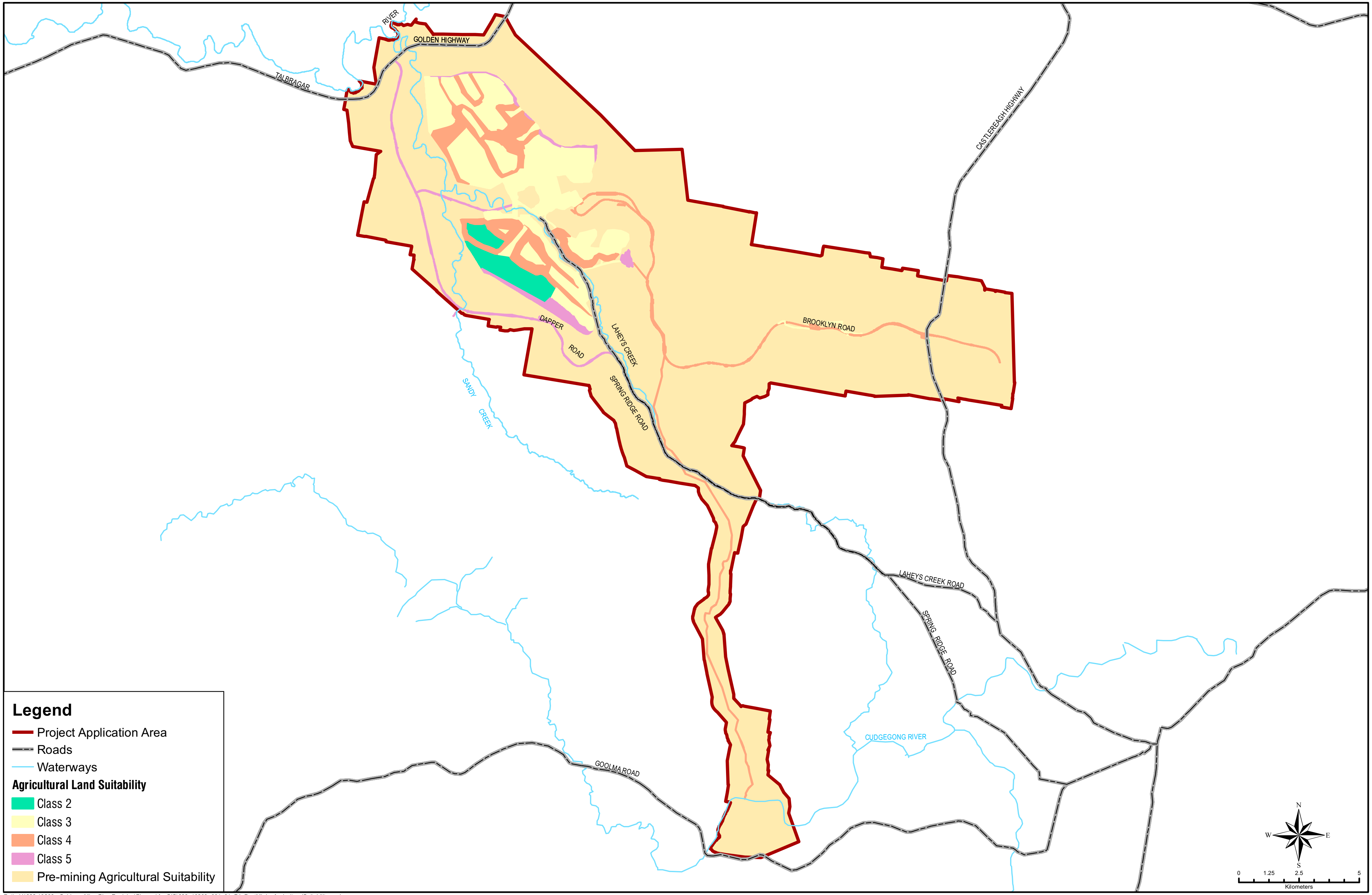
Post-mining Rural Land Capability Classification
Cobbora Coal Project
Figure 3



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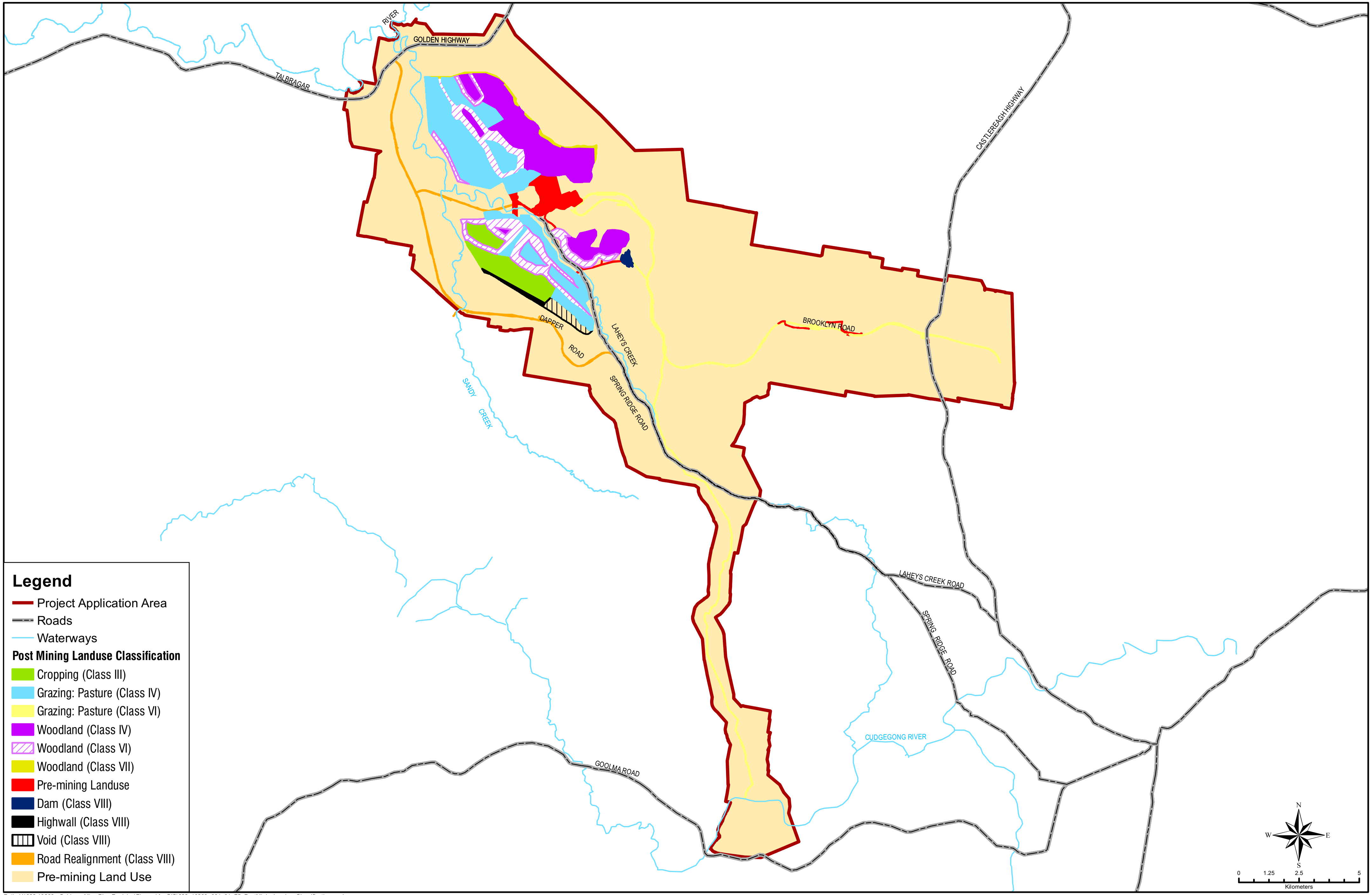
Post-mining Agricultural Land Suitability
Cobbora Coal Project
Figure 4



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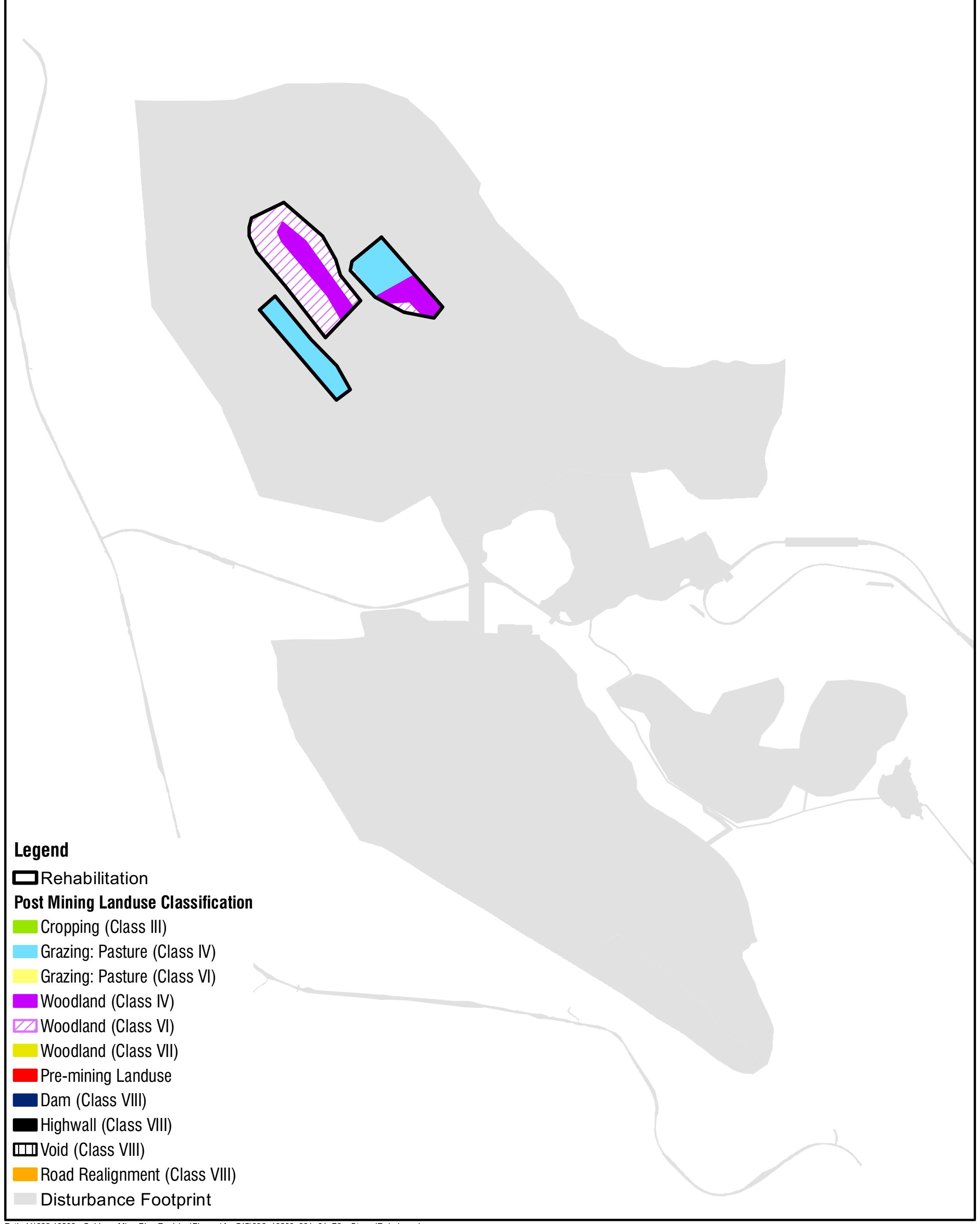
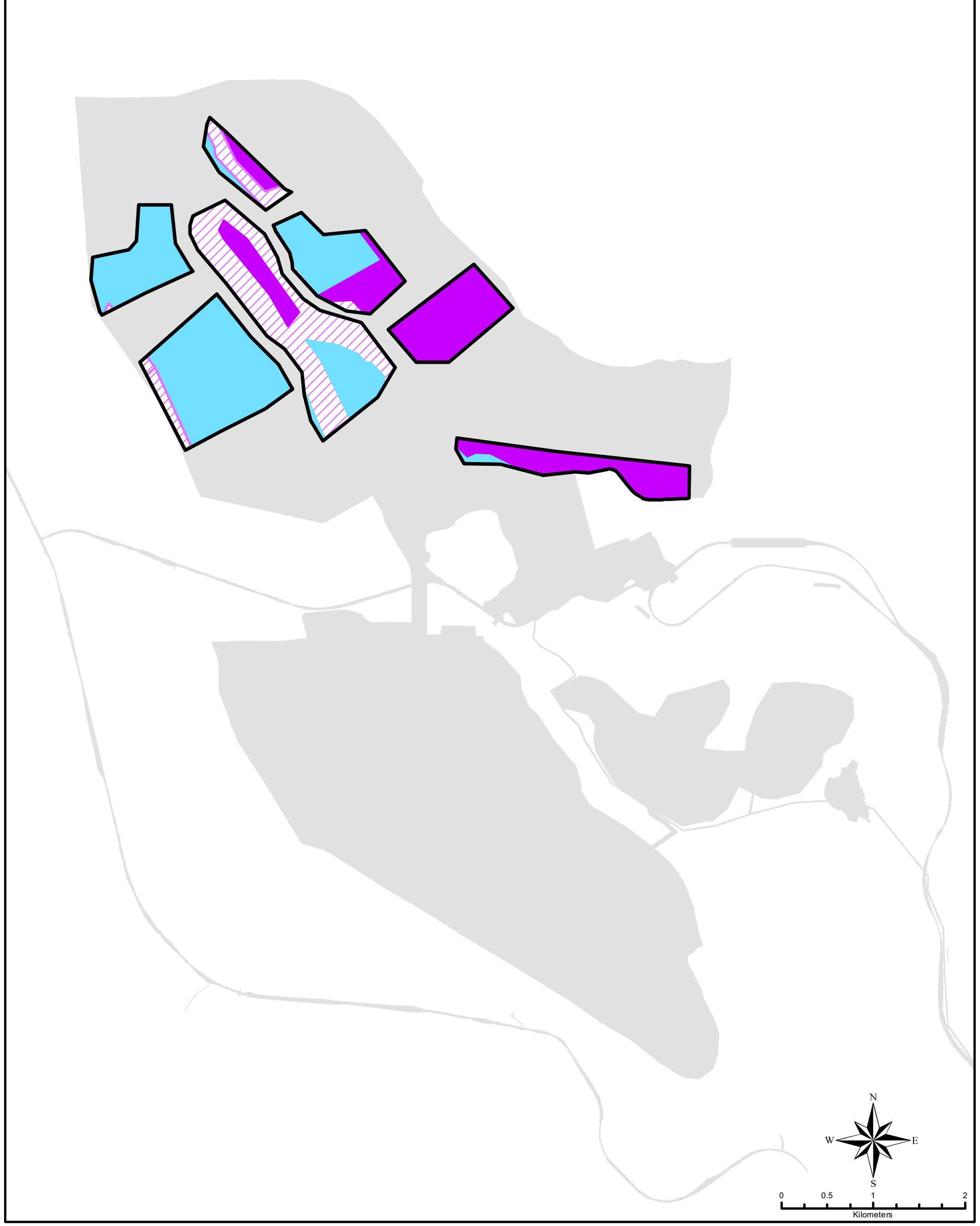
Post-mining Landuse Classification
Cobbora Coal Project
Figure 5

Year 4



Year 8



Legend

Rehabilitation

Post Mining Landuse Classification

- Cropping (Class III)
- Grazing: Pasture (Class IV)
- Grazing: Pasture (Class VI)
- Woodland (Class IV)
- Woodland (Class VI)
- Woodland (Class VII)
- Pre-mining Landuse
- Dam (Class VIII)
- Highwall (Class VIII)
- Void (Class VIII)
- Road Realignment (Class VIII)
- Disturbance Footprint

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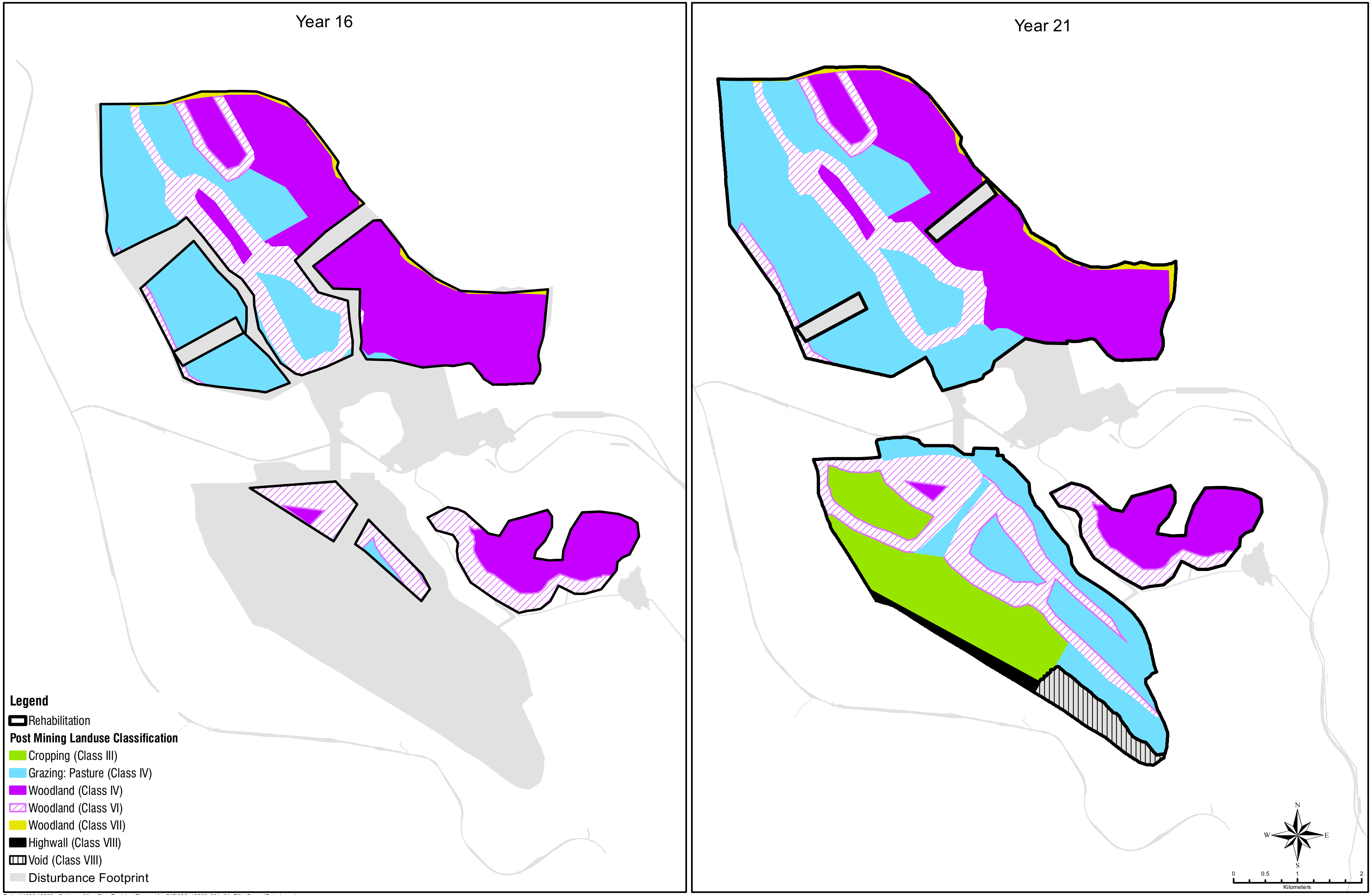
Mine Rehabilitation Strategy: Staged Rehabilitation Year 4 & 8

Cobbora Coal Project

Figure 6a



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Mine Rehabilitation Strategy: Staged Rehabilitation Year 16 & 21

Cobbora Coal Project

Figure 6b