Part B

Geology and Geotechnical

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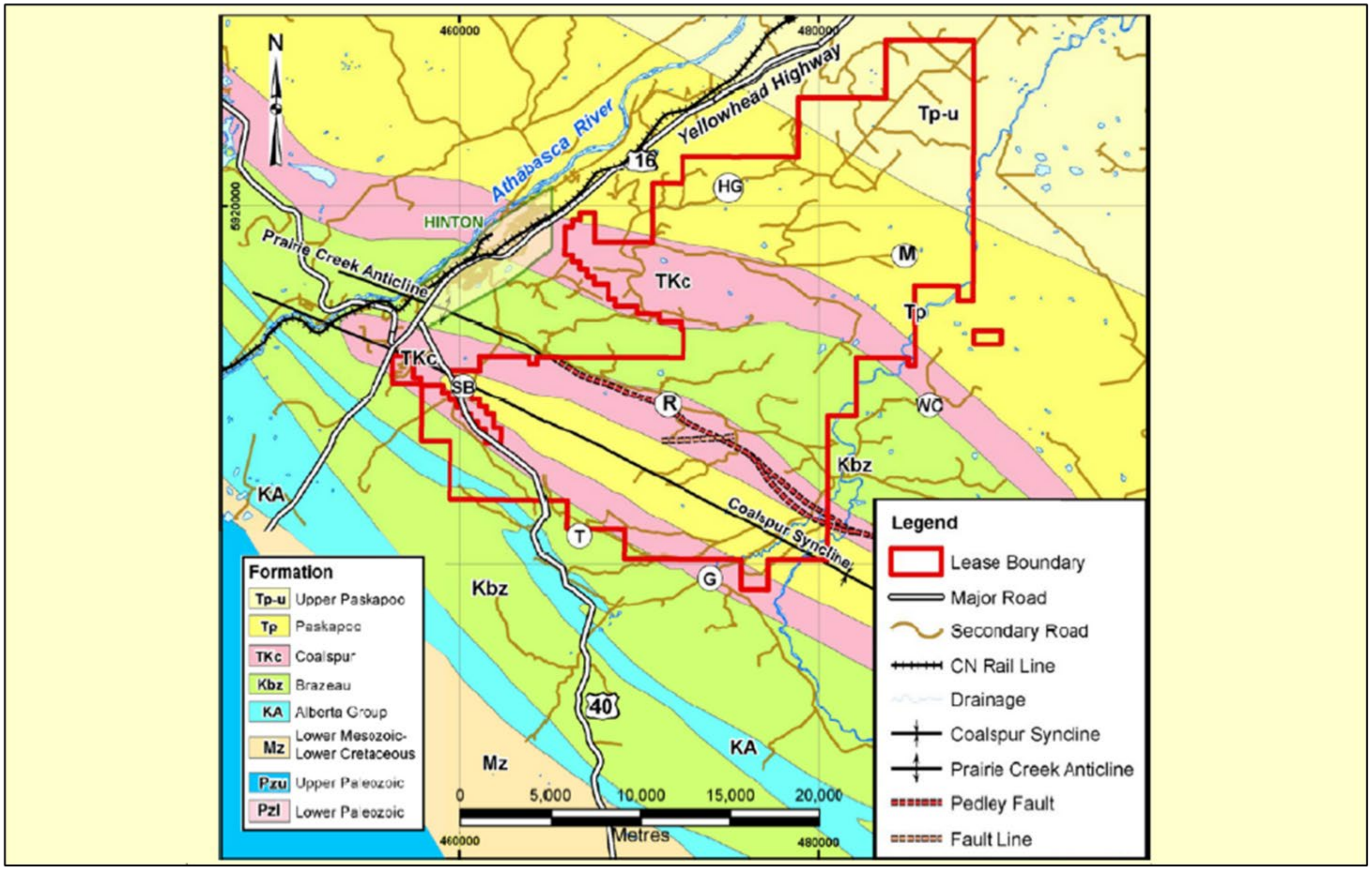
B Geology and Geotechnical

The Coalspur Mines Vista Phase II site is located adjacent to the current Phase I operations just outside of Hinton, AB; active mining has been ongoing for several years. The geological and geotechnical characteristics of the Phase II project area were previously reported in (Coalspur, 2014), which included a comprehensive summary of information from the original mine permit application the local and regional geology and stratigraphy of the formations present on site as well as the primary coal-bearing units targeted for extraction. Six coal zones within the upper Saunders Group from the Coalspur Formation have been identified on the property: the Val d'Or, Arbour, McLeod, McPherson, Silkstone, and Mynheer Zones. However, four major mineable seams targeted in the mine plan are the Val D'Or, Arbour, McLeod, and McPherson units. Coalspur conducted exploration activities in 2010 and 2011 to confirm the stratigraphy and extents of the units. Additional exploration and geotechnical drilling have been conducted in other areas of the currently operating mine as extraction has proceeded as well as drilling in support of the design of tailings containment structures.

B.1 Regional Geology

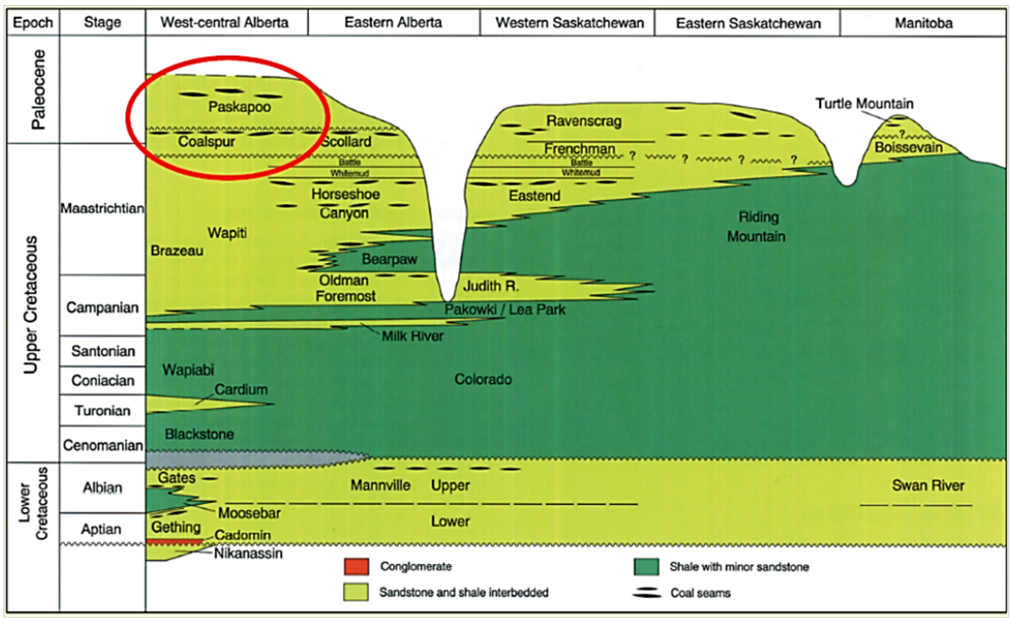
The coal deposits for the Vista Phase II project are in the Western Canada Sedimentary Basin on the eastern edge of the Rocky Mountain Foothills. This regional geology is shown in Figure B-1. The area was formed during the Laramide orogeny and developed primarily from eroded and transported Canadian Cordillera sediments deposited in fluvial and floodplain environments during the Upper Cretaceous period and Paleocene epoch. Three major geological formations exist at the mine site: the Paskapoo, Coalspur, and Brazeau Formations. The Paskapoo overlies the Coalspur Formation, which overlies the Brazeau Formation (Figure B-2). The Coalspur beds contain the major coal bearing sections. The Pedley Fault trends northwest/southeast along the southwestern boundary along the boundary of the coal deposits and separates the gently dipping coal beds from the steeper, faulted stratigraphy to the west. The surficial geology of the site consists of an upper layer of muskeg underlain by silty, sandy, and clayey glacial till. Although significant granular materials can be near the surface, previous hydrology investigations did not identify any major surficial aquifers.

Figure B-1 Coalspur Regional Geology

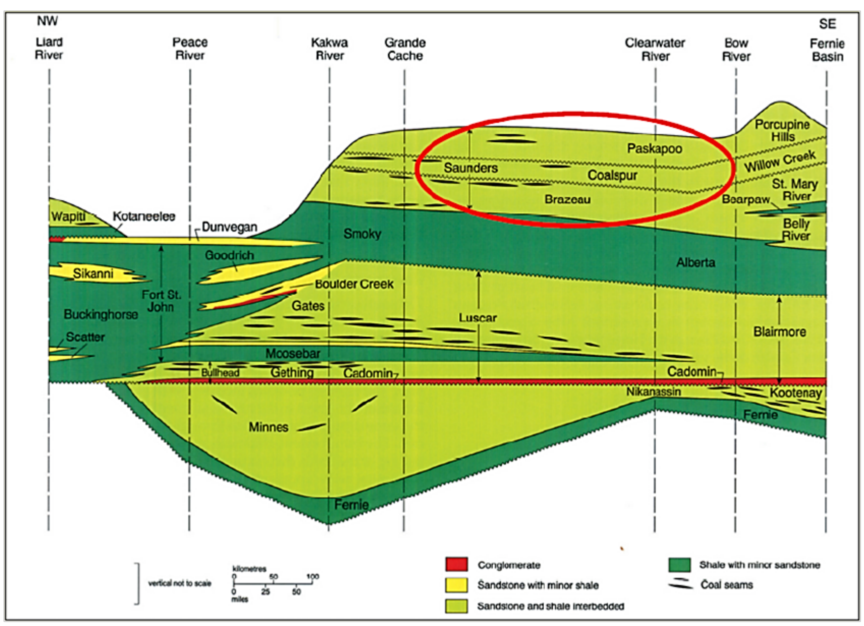


Coalspur Phase 2 EIA
Regional Geology
(Ref: Coalspur, 2014 as developed by Golder, 2012))

Figure B-2 Regional Stratigraphic Geology (Mossop and Shetsen, 1994)



Regional stratigraphic correlation cross-section across the Interior Plans.



Regional schematic cross-section.

BARR

Coalspur Phase 2 EIA
Regional Geology
Cross-sections
(Ref: Alberta Geological Survey)

B.1.1 Brazeau Formation

The Brazeau formation can be up to 2,000 m thick and consists of nonmarine mudstones, siltstones, and sandstones, with a chert-pebble conglomerate in the lower part of the formation. There are some thin coal beds interbedded with thin bentonites and coaly shales in the upper part of the formation that have been exploited by other mining operations but are not targeted by Coalspur.

B.1.2 Coalspur Formation

The Coalspur Formation consists of fluvially derived sediments that formed varying massive-to-thin interbedding of sandstone, siltstone, mudstone, and coal with some bentonite. The sandstone is largely grey and fine to coarse grained, with greenish-grey mudstone and siltstones. It was deposited in the end of Mesozoic and beginning of Cenozoic. The primary coal sources for the Phase II project are the Val d'Or, Arbour, McLeod, and McPherson zones. In many of these zones, the coal is interbedded with bentonite.

B.1.3 Paskapoo Formation

The Paskapoo Formation is the youngest strata in the area, and it is characterized as a Paleogene to earliest Eocene fluvial deposit (Hamblin, 2004; Leberkmo et al., 2008) dominated by siltstone and mudstone and interbedded with high-permeability, coarse-grained channel sands (Grasby et al., 2008). The exposed area of Paskapoo Formation is over 65,000 km^2, encompassing most of the southwestern Alberta and represents the uppermost bedrock unit over its area of occurrence (Chen et al., 2007; Hamblin, 2004).

The deposition of the Paskapoo Formation is unconformably over the Scollard Formation, which ages goes from the Upper Cretaceous to the lower Paleocene-aged (Jerzykiewicz, 1997). It's characterized as a high-energy alluvial fan and floodplain deposits sourced from the eroding Rocky Mountains to the west (Hamblin, 2004). The Paskapoo's deposition into the subsiding foreland basin formed an asymmetrical clastic wedge with a present-day maximum thickness of up to 850 m in the foothills, pinching out to a few tens of meters towards the plains (Hamblin, 2004). Demchuck and Hills (1991) divided the Paskapoo Formation into three members, named as Haynes, Lacombe, and Dalehurst.

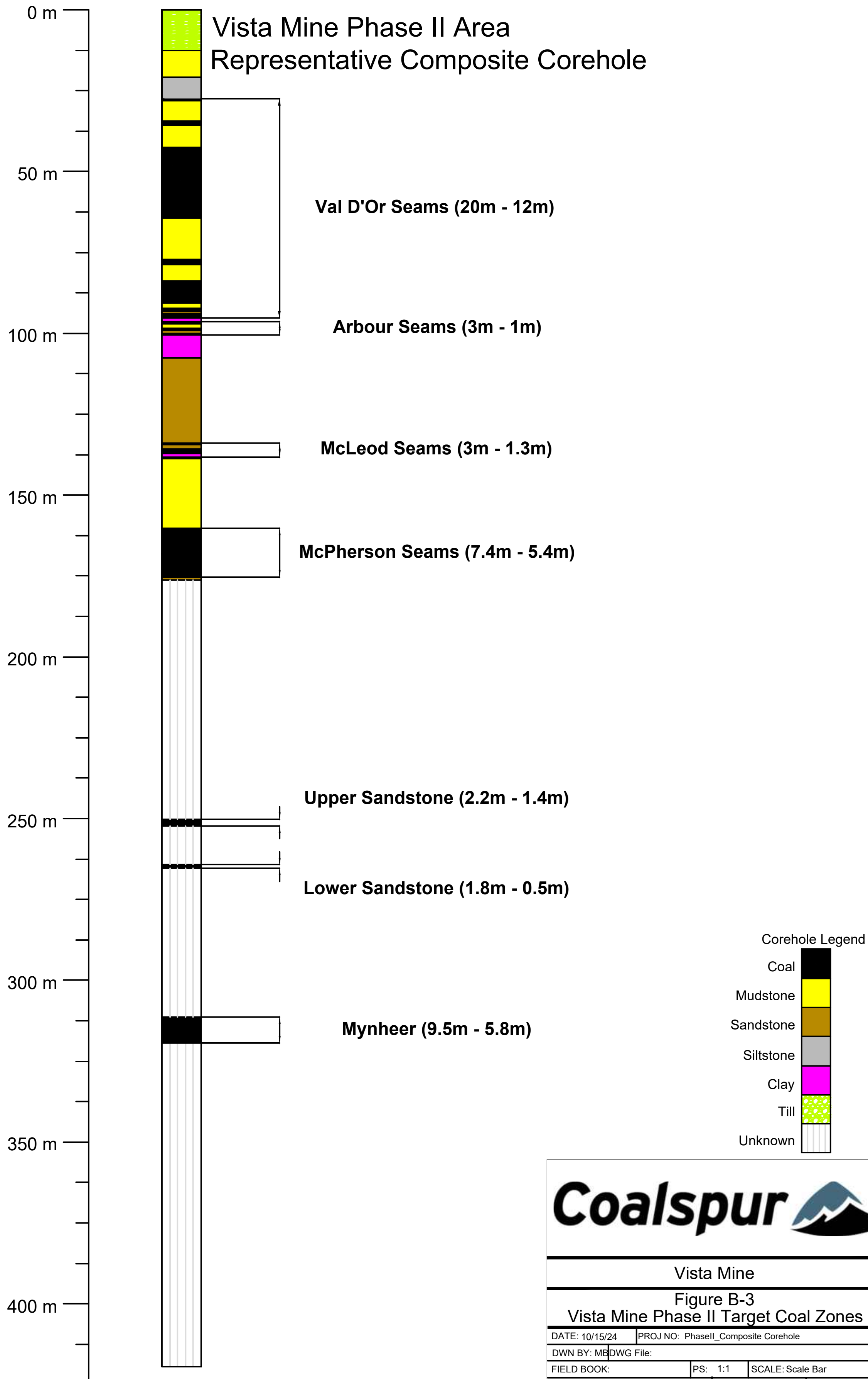
The lowermost Haynes Member is dominated by sandstones, characterized by its thick, massive, coarse-grained distribution, being the main geological characteristic observed at Paskapoo Formation. Outcrops are often biased towards these massive, cliff-forming, basal sandstones, leading early interpretations to suggest a sandstone-dominated system (Lyster and Andriashek, 2012).

The Lacombe Member consists of interbedded siltstone, mudstone, shale, and coal with minor fine to medium-grained sandstone and conglomeratic lag-deposits (Demchuk and Hills, 1991). Despite being a dominant component, this member is rarely exposed in outcrops due to its recessive nature (Lyster and Andriashek, 2012).

According to Demchuk and Hills (1991), the Dalehurst Member is overlying, and it is present only in the foothills of Alberta and displays interbedded sandstone, siltstone, mudstone, and shale with at least five thick (1.3 m to 6.1 m) coal seams.

B.2 Deposit Geology

Six coal zones have been identified within the lease boundary in descending order as the Val d'Or, Arbour, McLeod, McPherson, Silkstone, and Mynheer Zones. A representative stratigraphic column is provided in Figure B-3. Each zone consists of multiple coal plies separated by clastic bentonitic parting material of variable thickness. The total coal thickness of the combined zones averages 28.3 m over the 200 m stratigraphic interval. Overall, the structure of the deposit consists of a monocline trending at 300° (N 60 W), dipping from between 6° at the northern boundary of the property to as steep as 15° at the southern boundary by the McLeod River. Several of the coal seams are correlatable to other properties in the region, including the Val d'Or, Arbour, Silkstone, and Mynheer seams at Coal Valley.



The exploration of the Phase II area did not include below the McPherson seam. The information from the surface to McPherson is a composite of site specific information, below the McPherson the information presented is an extrapolation of regional geology. Ref: ARC, 1990 as developed by Jerzykiewicz, 1985

B.2.1 Lithology

Regionally and within the mine lease boundary, the Coalspur Formation consists of interbedded sandstones, siltstones, coal, bentonitic to carbonaceous mudstones, true bentonites, scattered bentonitic lacustrine rhythmite and tuff layers. The sandstones range from coarse cross-bedded units with local pebble zones to fine grained, massive, bedded units, and can be up to 70 m thick, though more commonly are found between 10-30 m thick. The siltstones and mudstones consist of thin bedded to laminated layers with varying silt content, with some plant remains found disseminated throughout. The coal thicknesses range from less than 5 cm to more than 2 m thick, with ash contents of individual layers ranging from 8% to 40%. The coal seams are often found with numerous thin interbeds of mudstones, bentonites, and tuffs, with occasional siltstone or sandstone layers. Pyrite and other sulphur forms were found to be rare but can occur along bedding planes.

B.2.2 Structure

The geological units within the mine lease boundary dip in a generally northeastern direction, varying from 5° to 26°. The mean structural dip of the Val d'Or seam for the north block is 10.5° and for the south block is 14°. The McPherson seam has the highest dip, whereas the McLeod seam normally has a dip intermediate between the Val d'or and McPherson seams.

Some faults were found in drill cores throughout the mine lease boundary, including normal faults and an apparent swarm of low angle thrust faults; however, the drill holes spacing used in exploration did not define faulted zones over much of the area. Other faulting was found in drill holes that may have been caused by glacial action due to low-angle fault traves near the bedrock-till interface.

B.2.3 Coal Seam Characteristics

B.2.3.1 Val d'Or Seam

The Val d'Or seam is characterized by seven individual subseams over a 15 to 70 m interval. Subseams 1 through 5 consists of continuous coal units with thin bentonite or carbonaceous parting intervals. Subseam 6 consists of two coal layers separated by a thin carbonaceous mudstone at the north end of the mine lease boundary, that split and thin towards the south. Subseam 7 also consists of coal layers that split and thin towards the south. The interval thicknesses range from 0.8 to 5 m, and generally thin from north to south.

The Val d'Or seam lies on an upward-fining sandstone-siltstone-mudstone sequence, that is hard, non-bentonitic, and laterally consistent. Overlying the Val d'Or seam is a moderately thick continues sandstone-siltstone sequence.

B.2.3.2 Arbour Seam

The Arbour seam is characterized by multiple thin coal layers interbedded with carbonaceous mudstones and bentonites. The interval between the Arbour and the Val d'Or seam consists of a bentonitic mudstone to siltstone layer range from 0.5 to 3.7 m. Directly below the Arbour is a 2 m siltstone-sandstone layer followed by a bentonite zone ranging in thickness from 2 to 5 m.

B.2.3.3 McLeod Seam

The McLeod seam is characterized by three subseams that are generally thin, dirty, and split by bentonitic to carbonaceous mudstone and siltstones.

B.2.3.4 McPherson Seam

The McPherson seam is characterized by four correlatable subseams. Subseam 2 splits into three parts separated by sandstone units of variable thickness. Subseam 4 has been eroded by a thick sandstone unit. The McPherson seam lies on a siltstone-sandstone sequence and is overlain by mainly interbedded sandstone and siltstone layers.

B.2.3.5 Silkstone Seam

The Silkstone seam is characterized by a single coal interval and is approximately 60 to 80 m below the McPherson seam and was only penetrated along the western edge of the mine lease boundary. It does not contain any identifiable partings and can range from 1.5 to 1.7 m in thickness.

B.2.3.6 Mynheer Seam

The Mynheer Seam is characterized by two coaly zones separated by a sandstone unit and was only penetrated in one location within the mine lease boundary.

B.3 Surficial Geology

The surficial geology of the site consists of an upper layer of muskeg underlain by silty, sandy, and clayey glacial till and alluvium, ranging in thickness from 5 to 30 m. The material is primarily very dense with some boding and cementation and has a matrix of silty to sandy glacially deposited soils with some gravel, cobbles, and boulders. Clasts consist of limestones, sandstones, and quartzites generated from the front range of the Rocky Mountains immediately west of McLeod River.

B.3.1 Exploration Activities

Previous exploration of the area was conducted between 1971 and 1974 and between 1980 and 1985. Associated Porcupine Mines Ltd carried out initial exploration between 1971 and 1974, in a total of 15 drillholes, completed with downhole geophysical logging and minor sampling. Density, gamma ray and neutron logs were run on all holes and coal samples were taken from two holes. From 1981 to 1985, Manalta carried-out exploration campaigns on the Hinton properties, consisting of drillholes over nine cross section lines spaced roughly 1 km apart from the Mcleod River east to the boundary of Esso East Block. Their work included the drilling of 94 drill holes on the property for a total of 14,145.3 m, with 182 core samples taken. Drill holes were geophysically logged with a full suite of geophysical logs, including gamma ray, caliper, long-spaced density, bed resolution density, focused beam electric, and sonic.

Coalspur conducted a drilling program within the mine lease boundary in February 2010 consisting of seven cross section drill lines in the Mcleod North Block to infill between pre-existing Manalta lines with both rotary drilling and coring to collect samples for coal thickness and coal quality verification and validation. The drillholes were geophysically logged with gamma, density, single point resistance and caliper. Five holes were drilled on Hinton West and seven holes were drilled on Hinton East. In the 2011/2012 season, Coalspur drilled a further four drill holes (three cored and one rotary) totaling 1,126 m. In total, Coalspur drilled 1,978.2 m. The drillholes were geophysically logged with gamma, density, single point resistance and caliper. An additional ten closely space cores were collected from a single drill site from the Val d'Or seam to provide approximately 3 tonnes of material for bulk sample product washability testing and combustion tests. A map of drill hole locations is shown on Figure B-4.

B.3.2 Sampling Approach

The core logging and sampling procedures was performed by others following the ASTM Standard (D5192). The collection of coal samples from recovered core was handled according to the following procedures as documented in the Phase I EIA report:

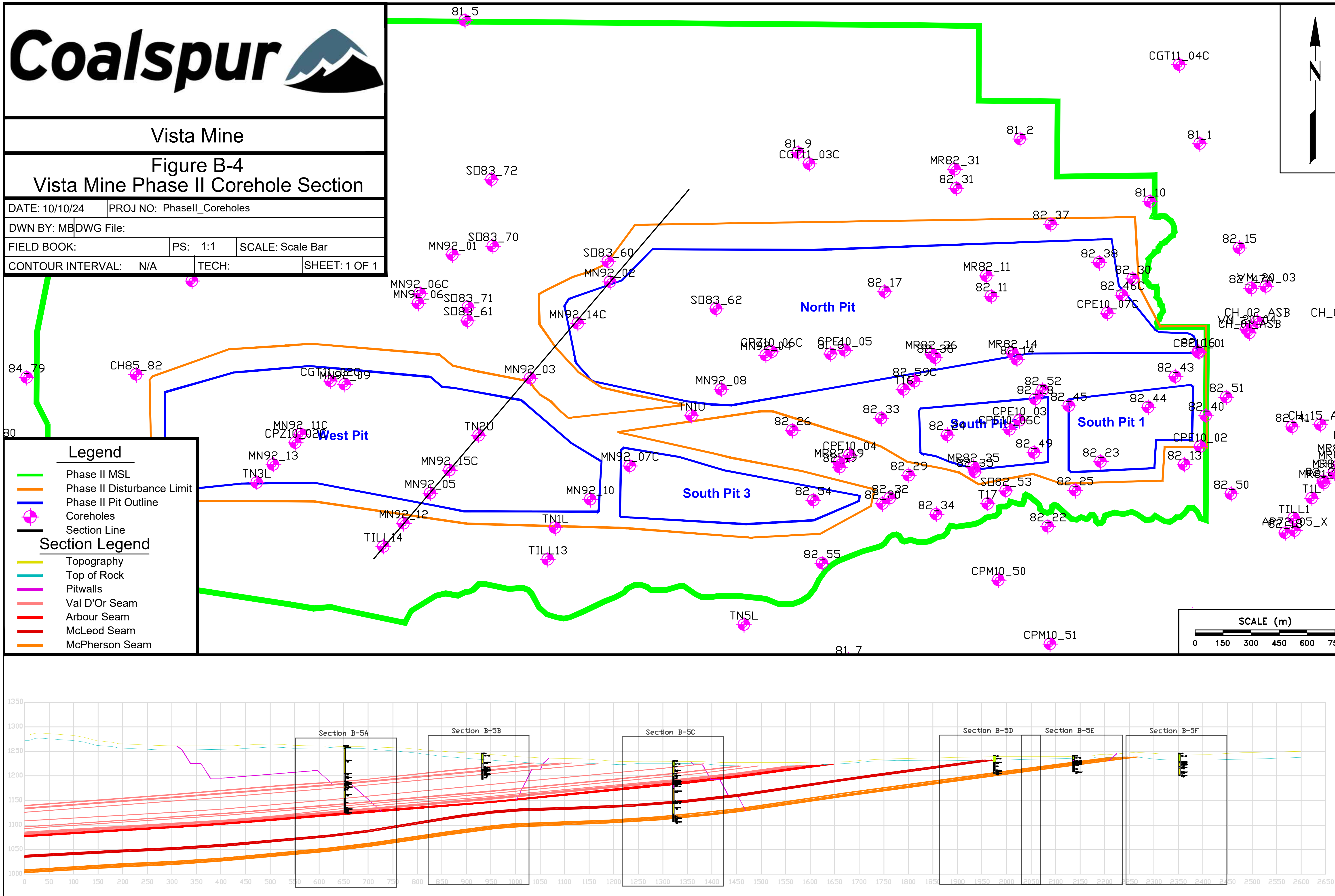
- To identify the coal intervals and their host rock material, each drilled hole was geophysically logged using a four-function downhole tool recording borehole diameter, bulk rock density, natural gamma, and resistivity of the formation.
- The coal cores of 3-meter-long run, were cleaned of any mud or contaminants, marked with the top and bottom run intervals, and then photographed for permanent visual identification.
- Recovery for each core run was recorded to determine overall recovery. Using the geophysical log record, the recovered coal intervals were also compared to the true in-situ coal thickness. Any recovered coal core thicknesses less than 85% of in-situ thickness were re-cored to improve recovery. If after several attempts the recovery remained less than 85%, the recovered coal core with the best recovery was used for sample analysis.
- Using the best-recovered coal core interval, the core was then subdivided into separate lithologic units. These were then measured and described using standard geological terms to identify lithology, colour, hardness, grain size, contacts, contamination, etc. as well as core loss and any coal sample intervals extracted for analysis.

Samples taken for analysis were extracted according to the following procedures:

- The minimum thickness for a coal sample interval was 60 cm (2.0 ft.).
- Inter-seam partings, up to a maximum thickness of 15 cm (6 in.), were included in any sampled coal intervals.
- Where the inter-seam parting was less than the maximum parting thickness of 15 cm, the adjacent coal beds must individually be at least two times the parting thickness to allow the coal and parting material to be sampled together. The total sample thickness must be greater than the minimum thickness for a coal sample interval.

- Carbonaceous shale, bone, and rock partings greater than 15 cm were sampled separately to determine their dilution effect. If the carbonaceous material, when combined with the coal, meets the minimum requirements for coal quality, they may be included with the overall coal sample interval.
- A 15 cm roof and floor sample were taken for the top and bottom of each major coal zone.

The samples collected from core were then placed in individual plastic bags marked on the outside with the core hole number and sample number and then carefully sealed to prevent excessive moisture loss. They were then placed together in one larger collecting bag and marked on the outside with the core hole number.

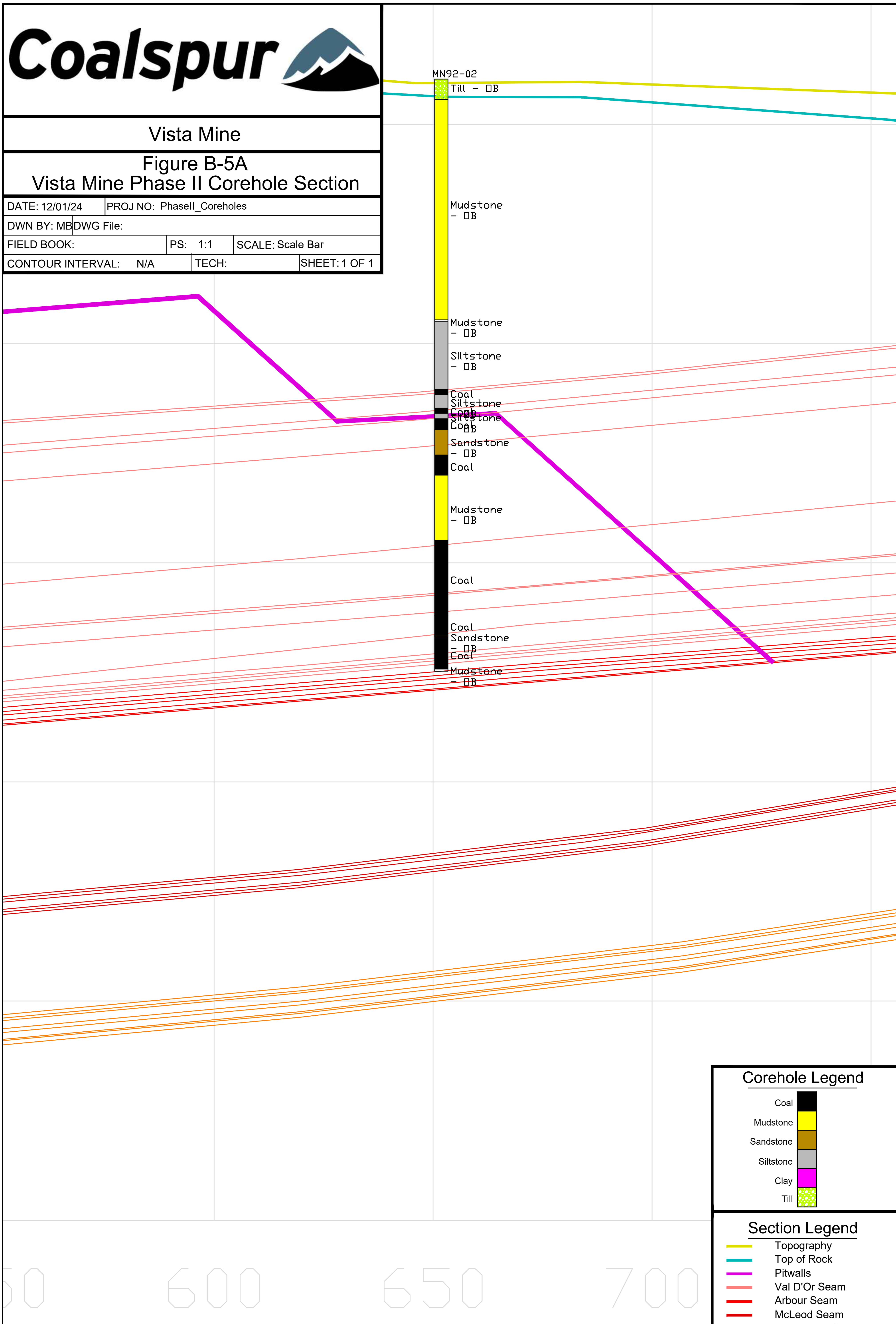

Coalspur
Vista Mine
Figure B-4
Vista Mine Phase II Corehole Section
DATE: 10/10/24
PROJ NO: PhaseII_Coreholes
DWN BY: MB
DWG File:
FIELD BOOK:
PS: 1:1
SCALE: Scale Bar
CONTOUR INTERVAL: N/A
TECH:
SHEET: 1 OF 1
N
North Pit
West Pit
South Pit 3
South Pit 2
South Pit 1
Legend
Phase II MSL
Phase II Disturbance Limit
Phase II Pit Outline
Coreholes
Section Line
Section Legend
Topography
Top of Rock
Pitwalls
Val D'Or Seam
Arbour Seam
McLeod Seam
McPherson Seam
SCALE (m)
0 150 300 450 600 750
Section B-5A
Section B-5B
Section B-5C
Section B-5D
Section B-5E
Section B-5F

Coalspur

Vista Mine

Figure B-5A
Vista Mine Phase II Corehole Section

DATE: 12/01/24	PROJ NO: PhaseII_Coreholes	
DWN BY: MB	DWG File:	
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1

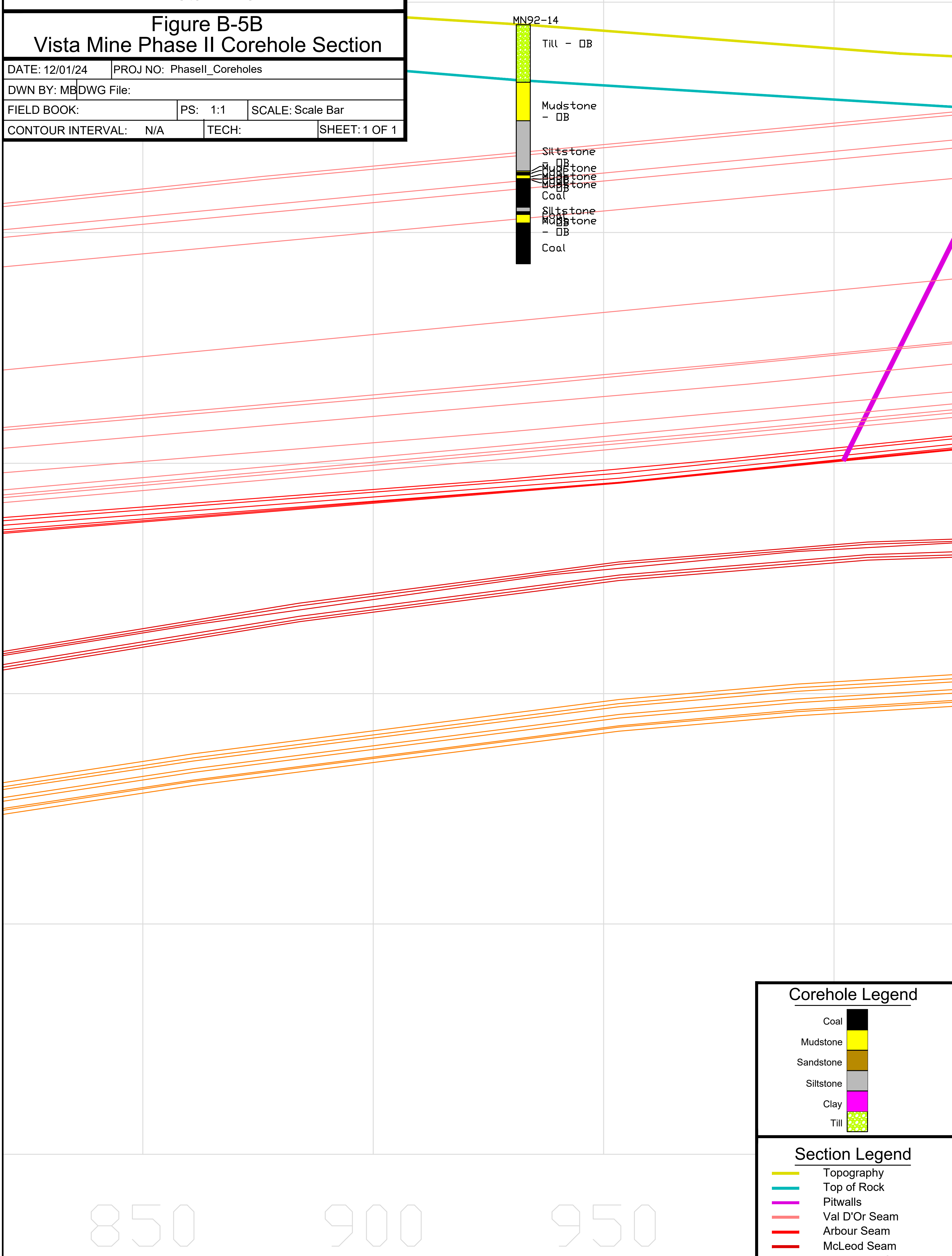


Coalspur

Vista Mine

Figure B-5B
Vista Mine Phase II Corehole Section

DATE: 12/01/24	PROJ NO: PhaseII_Coreholes	
DWN BY: MB	DWG File:	
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1

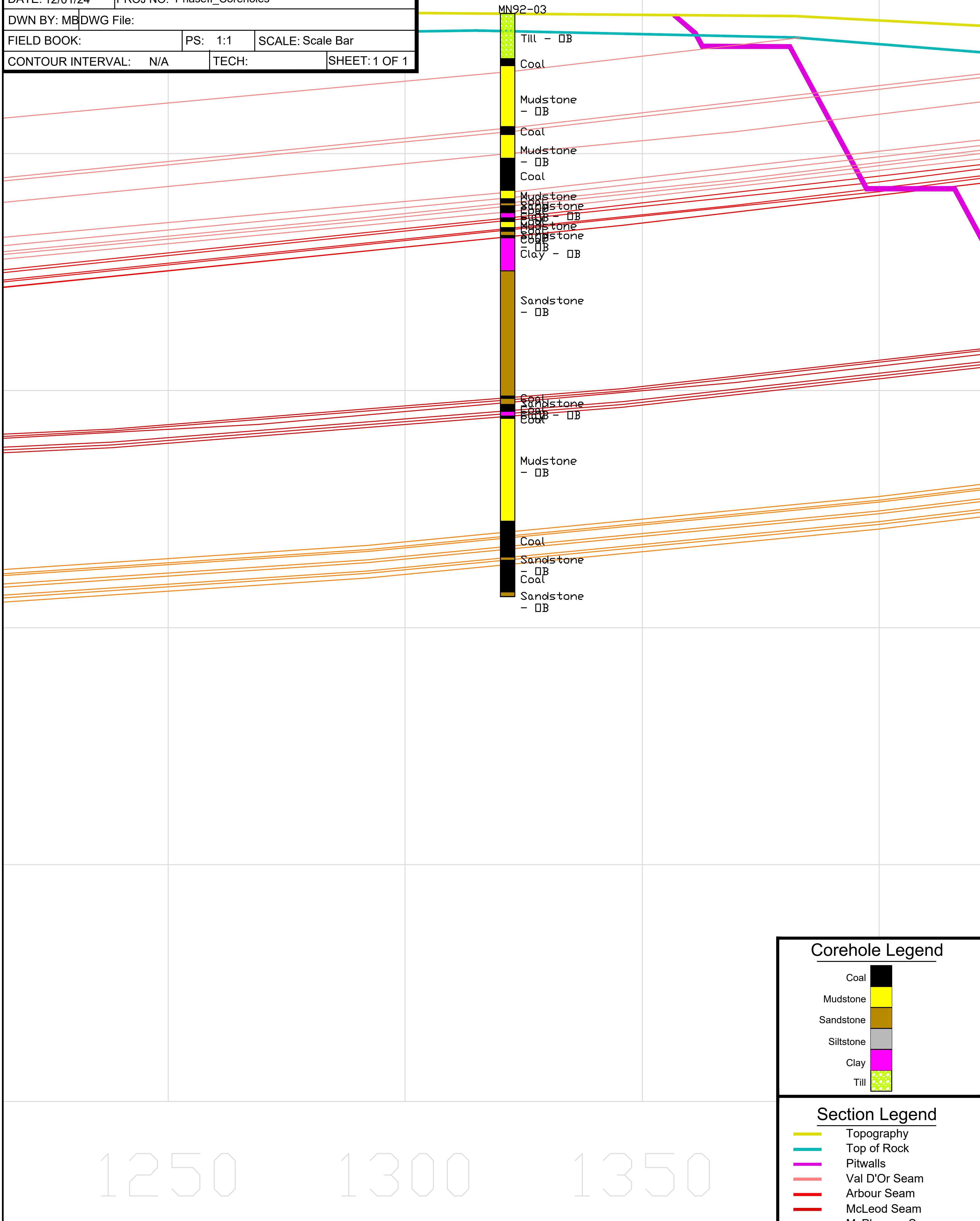


Coalspur

Vista Mine

Figure B-5C
Vista Mine Phase II Corehole Section

DATE: 12/01/24	PROJ NO: PhaseII_Coreholes	
DWN BY: MB	DWG File:	
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1

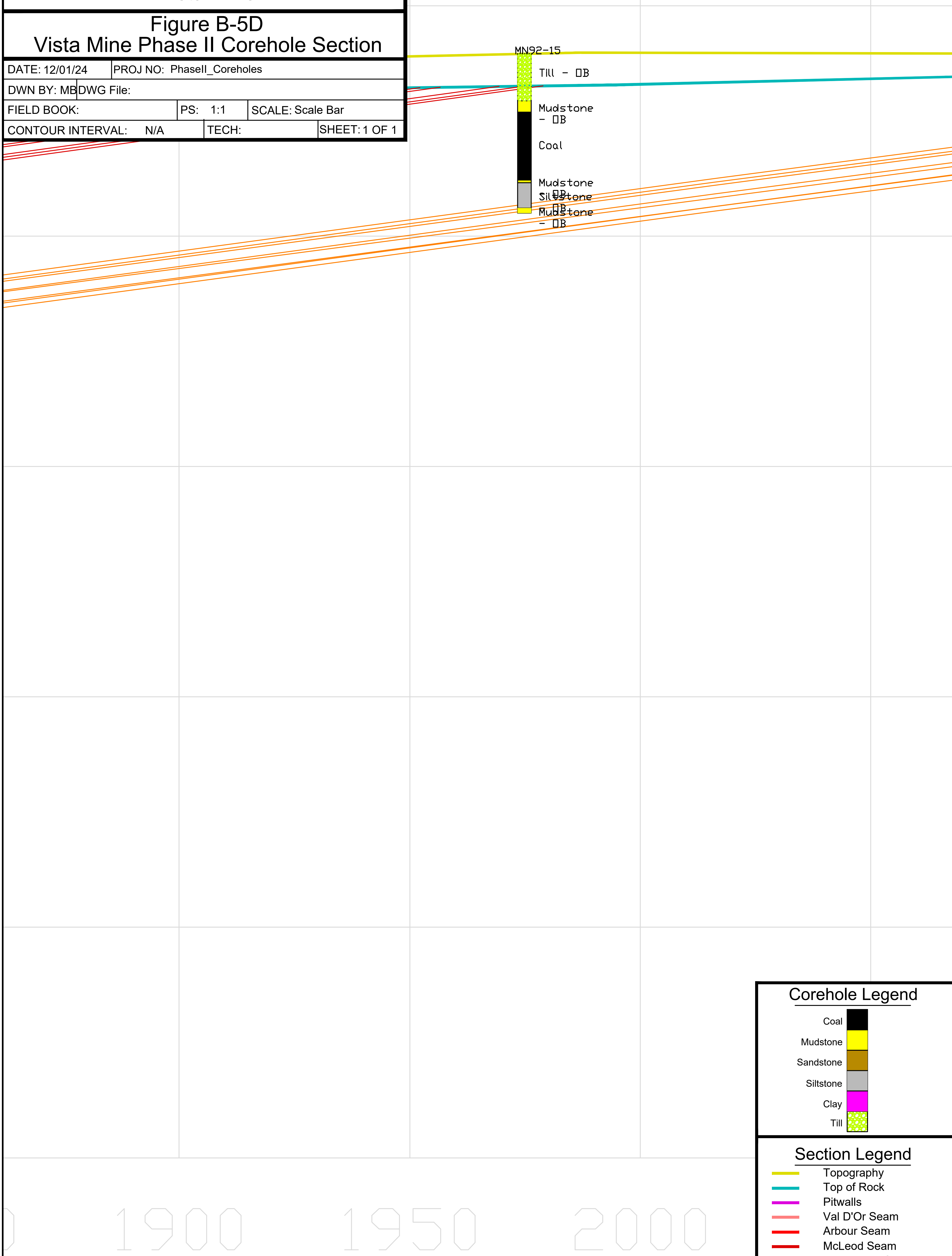


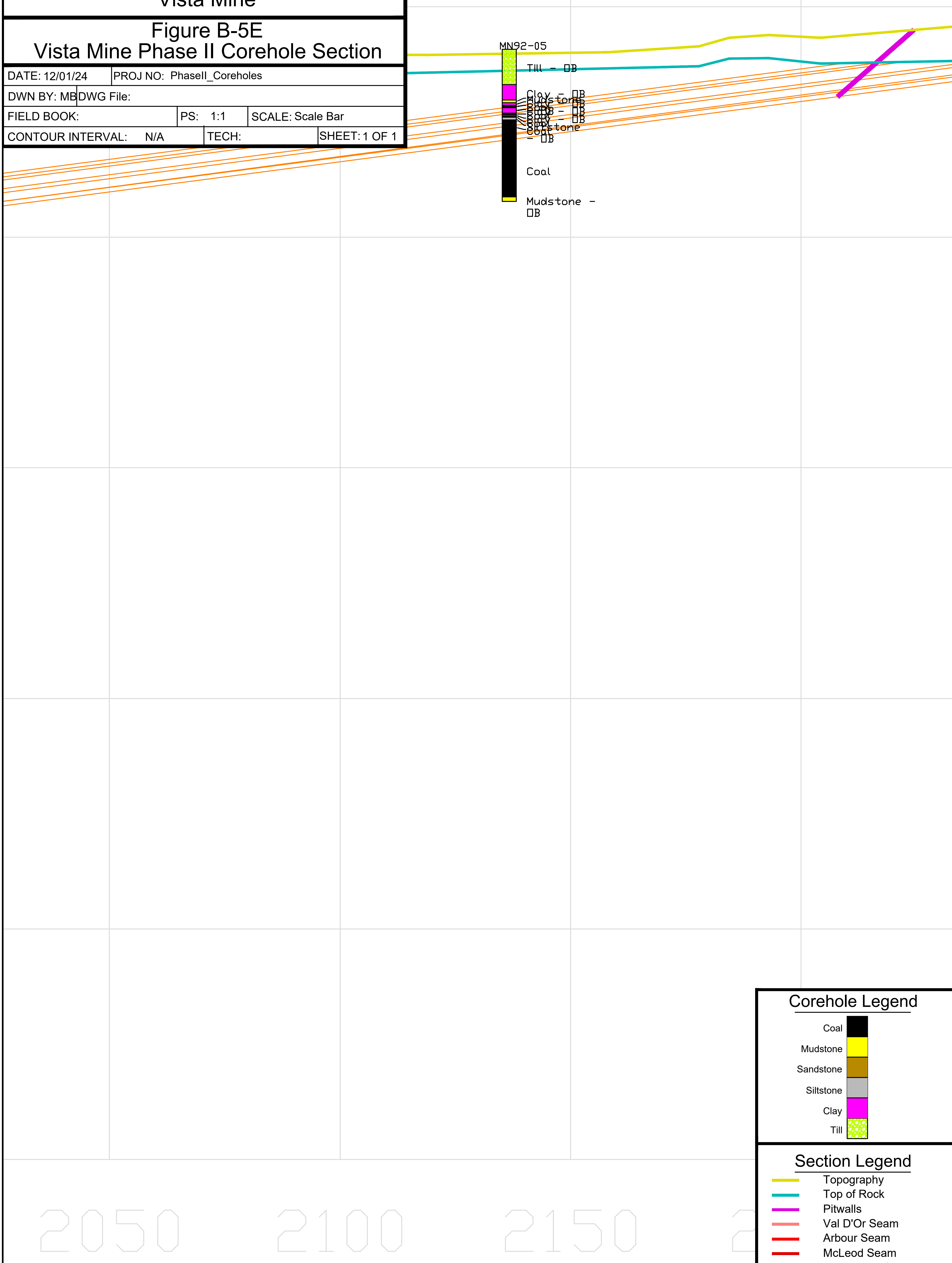
Coalspur

Vista Mine

Figure B-5D
Vista Mine Phase II Corehole Section

DATE: 12/01/24	PROJ NO: PhaseII_Coreholes		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	





Coalspur

Vista Mine

Figure B-5E
Vista Mine Phase II Corehole Section

DATE: 12/01/24	PROJ NO: PhaseII_Coreholes		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	

Corehole Legend

- Coal
- Mudstone
- Sandstone
- Siltstone
- Clay
- Till

Section Legend

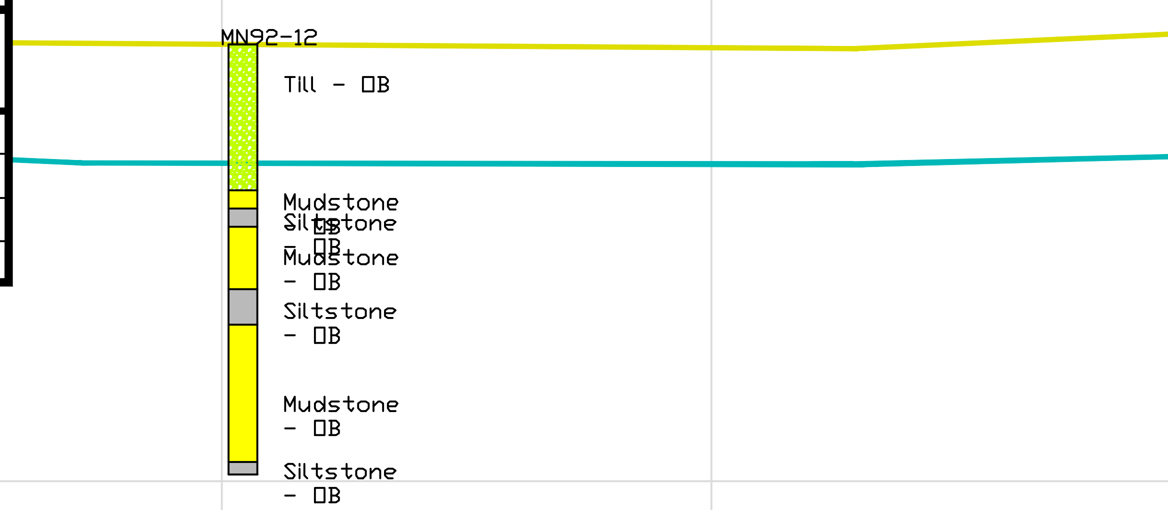
- Topography
- Top of Rock
- Pitwalls
- Val D'Or Seam
- Arbour Seam
- McLeod Seam
- McPherson Seam

Coalspur

Vista Mine

Figure B-5F
Vista Mine Phase II Corehole Section

DATE: 12/01/24	PROJ NO: PhaseII_Coreholes	
DWN BY: MB	DWG File:	
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1



Corehole Legend

- Coal
- Mudstone
- Sandstone
- Siltstone
- Clay
- Till

Section Legend

- Topography
- Top of Rock
- Pitwalls
- Val D'Or Seam
- Arbour Seam
- McLeod Seam
- McPherson Seam

B.3.3 2011 Exploration Sampling and Analysis

Individual coal seam and rock ply core samples were selected as outlined previously and shipped to ALS Laboratories out of Vancouver, BC.

- For the NQ size core (7.6 diameter) samples the following protocol was followed:
  - Each sample was weighed, and Apparent Relative Density Tests were undertaken prior to sample crushing. Instructions were provided to composite ply samples into logical mining units (coal and non-removable parting material). Each ply was crushed to minus 19 mm and combined based on ARD and thickness.
  - One quarter of the combined sample was tested for Proximate Analysis, Calorific Value, Total Sulphur, Chlorine, and Specific Gravity.
  - The remaining three quarters of the composite samples were screened at plus/minus 0.5 mm. The Minus 0.5 mm fraction was analyzed for Proximate Analysis and Calorific Value.
  - The plus 0.5 mm material was subjected to Float/Sink at 1.40.1.50, 1.60, 1.70, 1.80 and 2.00 Specific Gravity. Proximate Analysis and Calorific Value were performed on all increments.
  - Instructions were provided to create further clean coal composites.
- For the 15.2 cm large diameter core, the following protocol was followed to generate attrition data for wash plant design.
  - Each sample was weighed, and Apparent Relative Density Tests were undertaken prior to sample crushing. Instructions were provided to composite ply samples into logical mining units (coal and non-removable parting material).
  - The combined sample was subjected to a Drop/Shatter test. The sample was dropped twenty times from 2 m and screened at minus 50 mm. Any oversize was hand-knapped to pass 50 mm. The broken sample was dry sized at 32, 16, 8, 4, and 2 mm. The dry size distribution and any coal losses were calculated for material reporting below 2 mm.
  - A wet tumble sample was constructed according to instructions. The sample was wet tumbled for 5 minutes with cubes. Wet sizing was performed at 32, 16, 8, 2-, 1-, 0.25- and 0.125-mm fractions.
  - Float /sink samples of +16 mm, 16 mm x 4mm, 4 mm x 2 mm, and 2 mm x 0.25mm were constructed. Each increment was float/sank at 1.30, 1.35, 1.40, 1.45, 1.50, 1.60, 1.70, 1.80 and 2.0 specific gravity. Each fraction was analyzed for Proximate Analysis and Calorific Value.
  - The 0.25 mm x 0.125 mm and minus 0.125 mm fractions were analyzed for Proximate Analysis.

  - Clean composite samples from both sets of core data were further analyzed for Ash Chemistry, Ash Fusion, and Petrographic Analysis.

The bulk sample was taken as a simulated product, and so only the coal core was sampled, leaving the roof and floor rock.

The drill and coring intersections from the Coalspur 2010 and 2011 programs for all of Vista were previously reported in Coalspur (April 2012 and November 2014) reports referenced in their entirety. Since Coalspur has not conducted additional exploration and testing on the Phase II area since these previous drilling programs, there are no new or additional results that require inclusion and interpretation.

B.4 Coal Reserves and Resources

Marston (Golder) prepared stratigraphic and coal quality models for the project as part of the overall Vista Mine's feasibility study in 2011. Information provided by Coalspur at the time of the initial model included data from both Phase I and Phase II coal areas. In 2019, the stratigraphic model was updated. This model utilized a smaller grid size. No coal quality information has been collected in the Phase II area since the Spring 2011 drilling program, however the geological grid was updated to create a model of higher resolution and to ensure Phase II data was consistent with the Phase I model. The coal quality model is still relevant to date and no changes within the quality of the seams is anticipated. The data discussed in this Section includes the results of modelling efforts completed during previous studies, and represents the totality of information collected to date within the Phase II Project area.

B.4.1 Calculations

The data were in digital format, with planimetric data contained in AutoCAD dwg or dxf format and log records in formatted ASCII or Microsoft Excel files.

Topography data was originally obtained in 2011 for the overall property including both Phase I and Phase II areas. The Phase II coal, top of rock and topographic models were updated in 2019 to include all drilling completed. The phase II coal model included all geologic information obtained since the early stages of the Vista project in 1983 all the way up to the most recent drilling in 2011. The early information regarding the project came from e-logs, the more modern drilling was coreholes. Each of the sample locations were adjusted so the collar matched the topography for the location and then used to create coal models.

B.4.2 Stratigraphic Modelling Assumptions

Marston prepared stratigraphic and coal quality models for Coalspur based on the survey, lithology and coal quality data sets in tandem with the topographic digital terrain model (DTM.) Base data included digital topographic triangulation data, drillhole survey and lithology records and coal quality ply sample data. The models generated consisted of regular arrays of data, or grids, distributed over the project area. Initially one model was built for the entire Coalspur property, in 2019 the modelling was updated to create separate higher resolution models for Phase I and Phase II.

Base data for the generation of the topography model consisted of digital triangulation segments exported from a previous model of the area. The grid surface for the topography was created using a regular 5-meter grid cell interval.

Table B-1 Coalspur Stratigraphic Model Horizons

Elemental Seam	Compound Seam	Seam Group	Included in Mine Plan?
V7	V7	Val d'Or	Yes
V6U	V6		Yes
V6L			Yes
V5Ub	V5		Yes
V5Ua			Yes
V5L			Yes
V4	V4		Yes
V3T	V3		Yes
V3Ub			Yes
V3Ua			Yes
V3L			Yes
V2U	V2		Yes
V2L			Yes
V1	V1		Yes
A3	A3	Arbour	Yes
A2	A2		Yes
A1	A1		Yes
L3	L3	McLeod	Yes
L2U	L2		Yes
L2L			Yes
L1	L1		Yes
P4	P4	McPherson	Yes
P3U	P3		Yes
P3L			Yes
P2U	P2		Yes
P2L			Yes
P1	P1		Yes

Base data for the full Vista model included 315 drill holes (66 core holes) along 17 major drill lines, containing data for 23 component seam splits, as well as 2 parent seam intervals. Two modelling horizons (TILL and TREND) were included to establish a till floor boundary and to control structural trends, respectively. The primary seam groups include the Val d'Or, McLeod and McPherson plies. Coalspur requested that 4 additional minor seams (A, SLK2, SLK1 and MYN) be included with the models. Table B-2 includes information regarding the inclusion of estimated tonnages from each seam or ply in the mine development plan.

Table B-2 Surface Mining Reserves

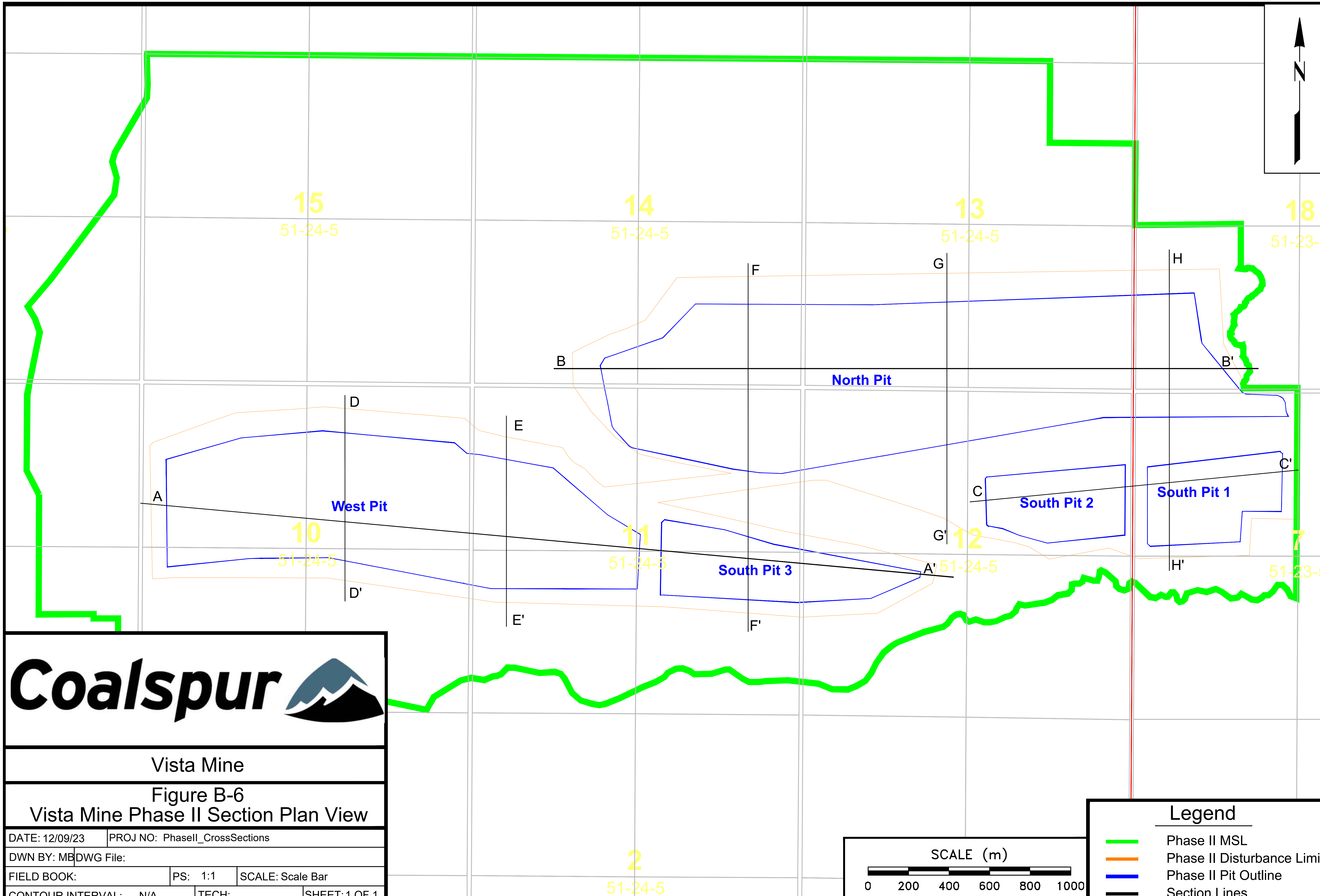
Seam	Raw Metric Coal ('000 tonnes)	Clean Metric Coal ('000 tonnes)	Calorific Value (kcal/kg)
Val D'Or	37,040	20,418	5,810
Arbour	6,358	3,119	5,835
McLeod	3,224	1,562	5,518
McPherson	16,213	8,959	5,816
Total	62,836	34,057	5,799

Coal structure contours and isopach maps were created for each coal seam defined in the schema. A series of figures are included in Appendix B1 that show the coal seam isopach that depict the thickness of the piles.

The final phase II stratigraphic model contains 27 coal seams. The stratigraphic model is based on a regular 5-meter grid cell interval, the geometric base point, grid extents and rotation angle are:

- Coordinates of lower left corner of grid: E 467,541.309 N 5,914,639.522
- Extents along longitudinal and lateral axes: 7,570 m 3,050 m
- Number of columns and rows in grids: 1,514 610
- Rotation of grids: 0°

The locations of the Coalspur drill holes are shown in Figure B-4. Figure B-6 shows the locations of typical cross-sections through the Project area. Typical geological cross-sections are shown in Figure B-7 through Figure B-9. Estimated in situ stripping ratios for the project area are illustrated in Figure B-10.



Vista Mine

Figure B-6
Vista Mine Phase II Section Plan View

DATE: 12/09/23	PROJ NO: PhaseII_CrossSections		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	

Legend

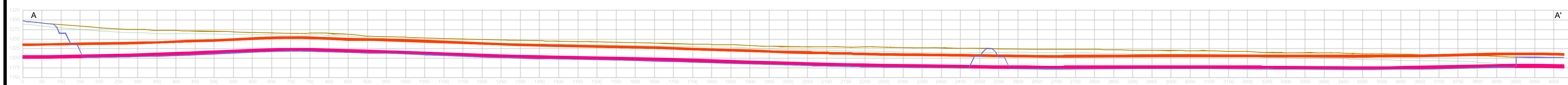
- Original Topo
- Top of Rock
- Final Pitwalls
- Valdor Seam
- Arbour Seam
- McLeod Seam
- McPherson Seam

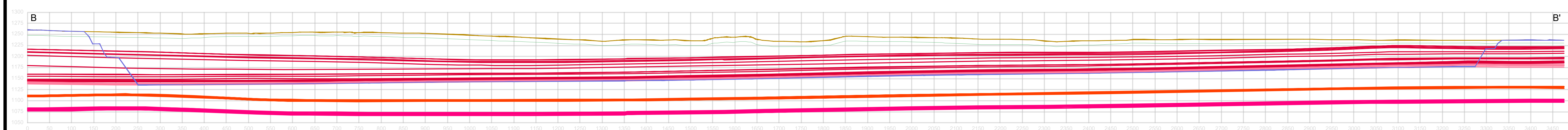
Coalspur

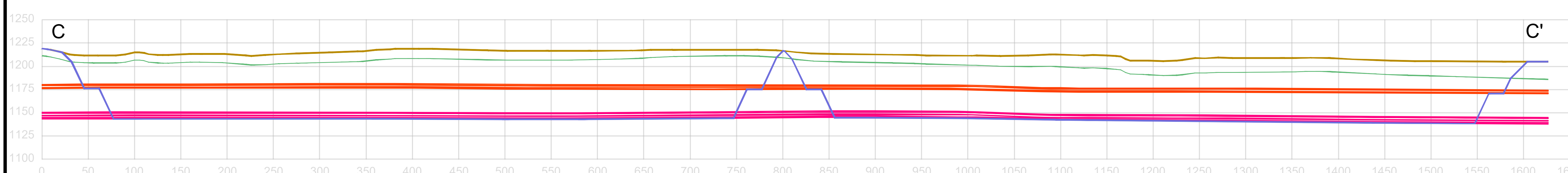
Vista Mine

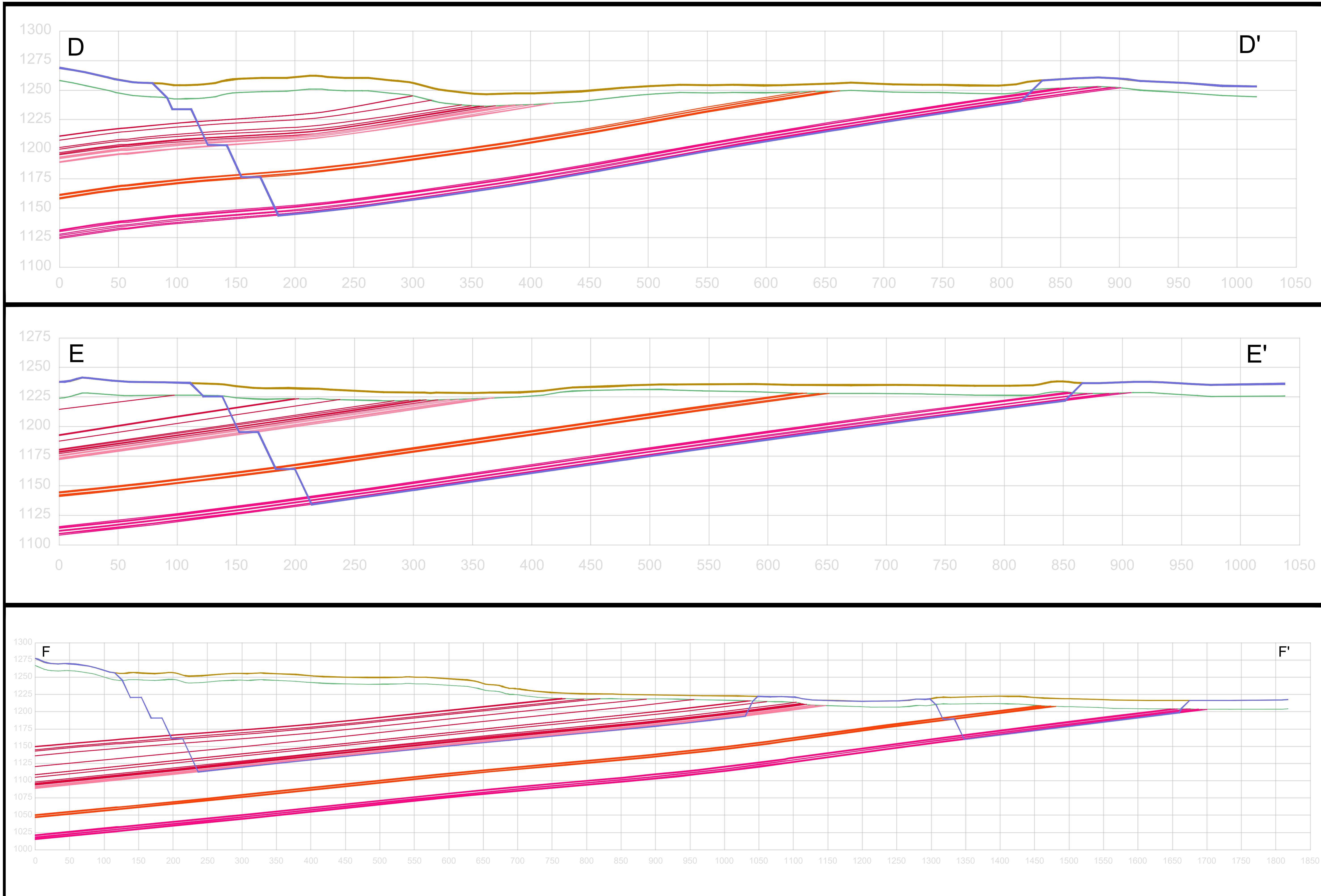
Figure B-7
Vista Mine Phase II Section A,B&C

DATE: 12/09/23	PROJ NO: PhaseII_CrossSections	
DWN BY: MB	DWG File:	
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1








D
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F
F'

Legend

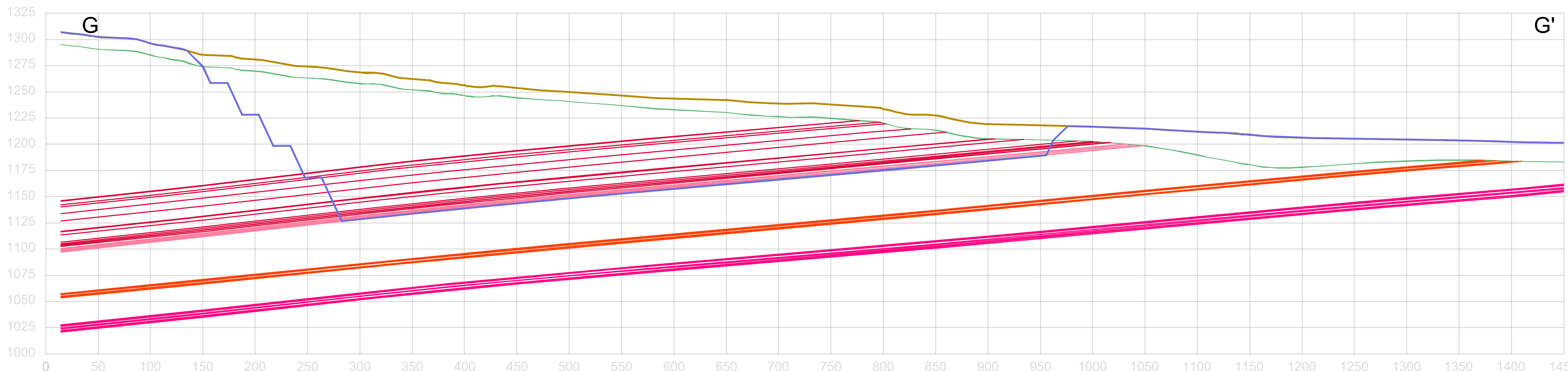
- Original Topo
- Top of Rock
- Final Pitwalls
- Valdor Seam
- Arbour Seam
- McLeod Seam
- McPherson Seam

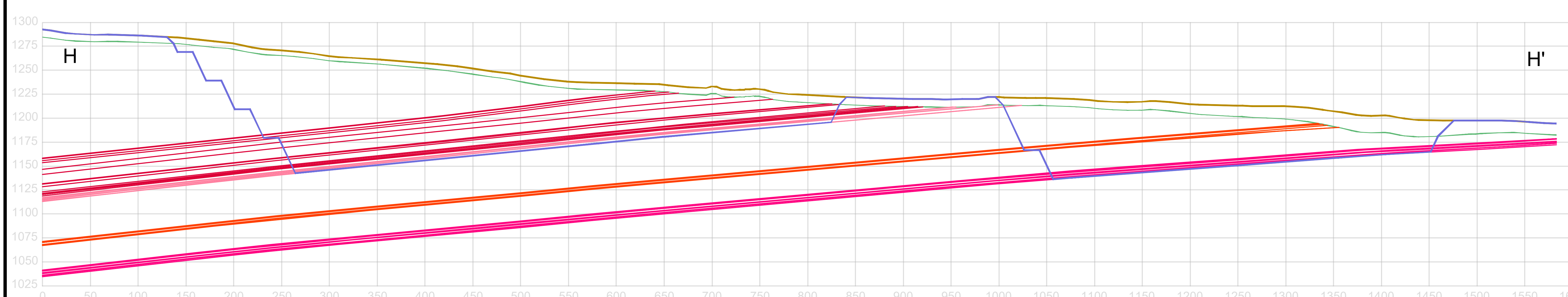
Coalspur

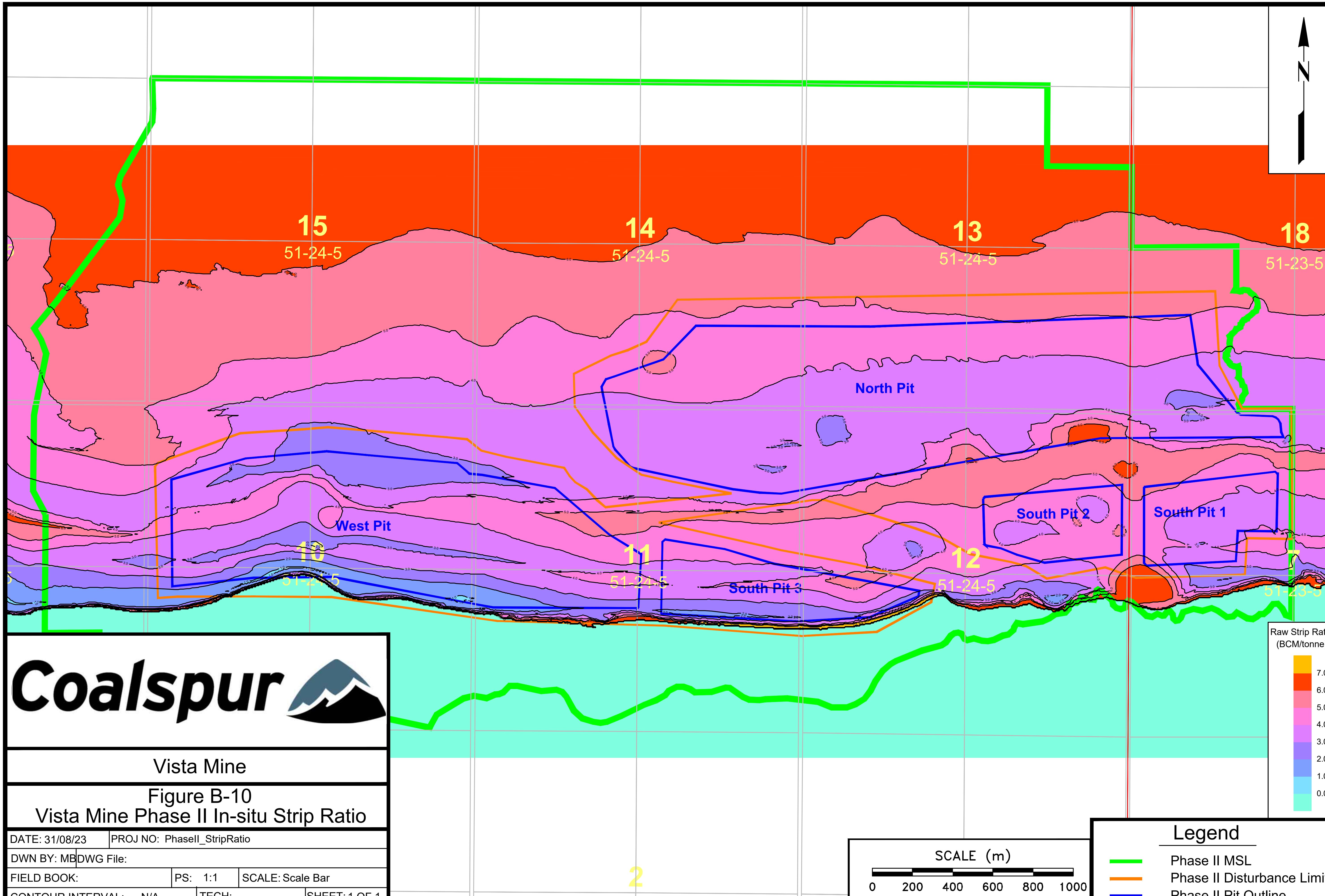
Vista Mine

Figure B-9
Vista Mine Phase II Sections G&H

DATE: 12/09/23	PROJ NO: PhaseII_CrossSections		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	







Coalspur

Vista Mine

Figure B-10
Vista Mine Phase II In-situ Strip Ratio

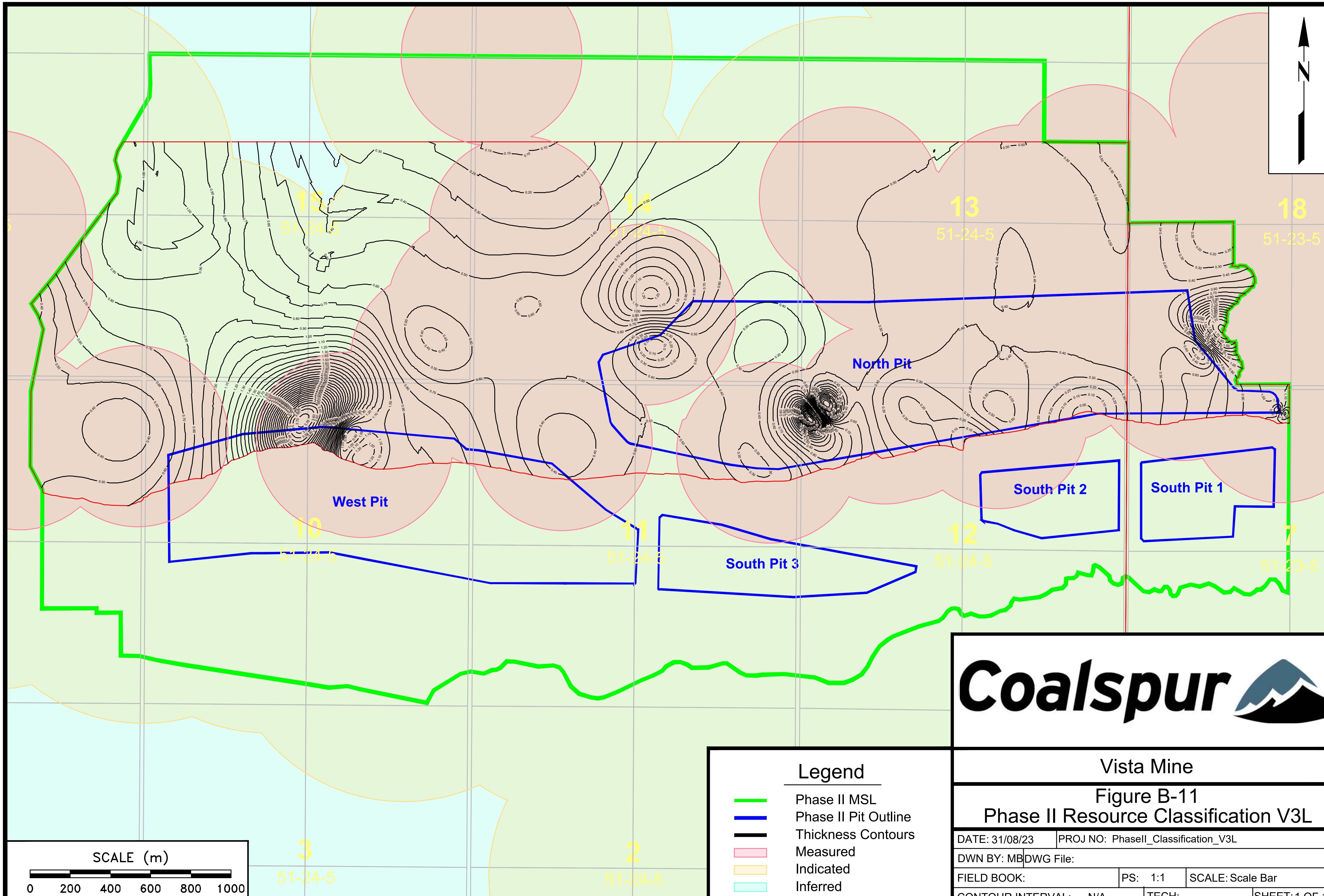
DATE: 31/08/23	PROJ NO: PhaseII_StripRatio		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	

B.4.3 Estimated Coal Resources

The Geological Survey of Canada Paper 88-21 "A Standardized Coal Resource/Reserve Reporting System for Canada" (GSC Paper 88-21) outlines definitions, concepts and parameters used to determine coal resource and reserve quantities and has historically provided the framework of categorizing coal quantities. The Vista property is classified as a moderate, potentially surface mineable deposit. GSC Paper 88-21 suggests that 0.6 m be used as the minimum mineable thickness assumption. Resource assumptions for Vista have been based on a slightly lower thickness cut-off of 0.5 m.

Resource classifications for this type of coal deposit, as prescribed by GSC 88-21, are based on the distance from a sampling data point. The search radii for the classifications are 450 m (measured), 900 m (indicated) and 2400 m (inferred). For the planned phase II pits, the entire reserve is classified as measured or indicated due to the proximity of core holes.

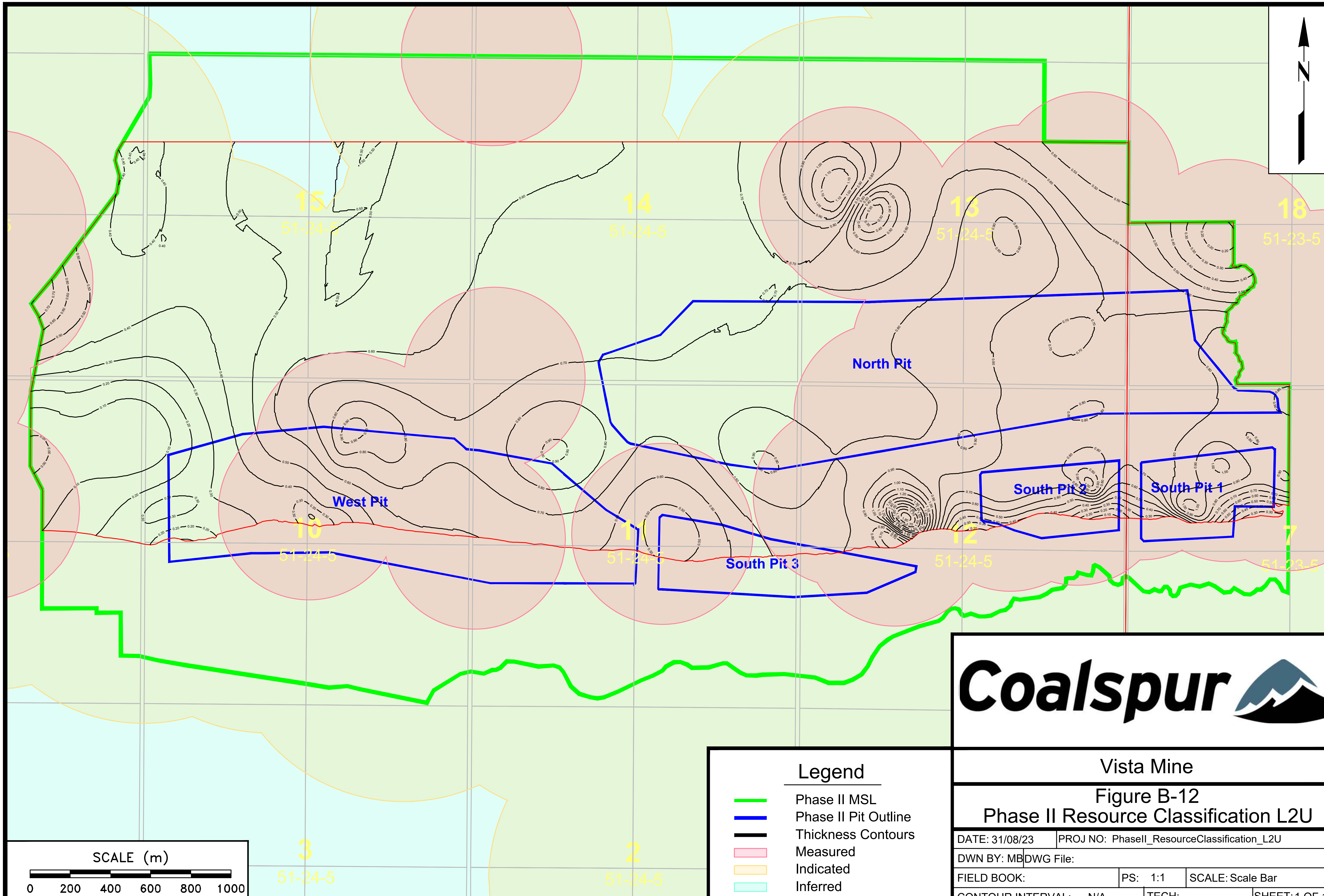
Figures illustrating the resource classifications which are based on the search radii prescribed by GSC 88-21 for various seam plies have been included for review. Included are the Val d'Or 3 Lower (V3L) (Figure B-11), McLeod 2 Upper (L2U) (Figure B-12), McPherson 4 (P4) (Figure B-13), and Arbour 3 (A3) (Figure B-14).



Vista Mine

Figure B-11
Phase II Resource Classification V3L

DATE: 31/08/23	PROJ NO: PhaseII_Classification_V3L		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	



Legend

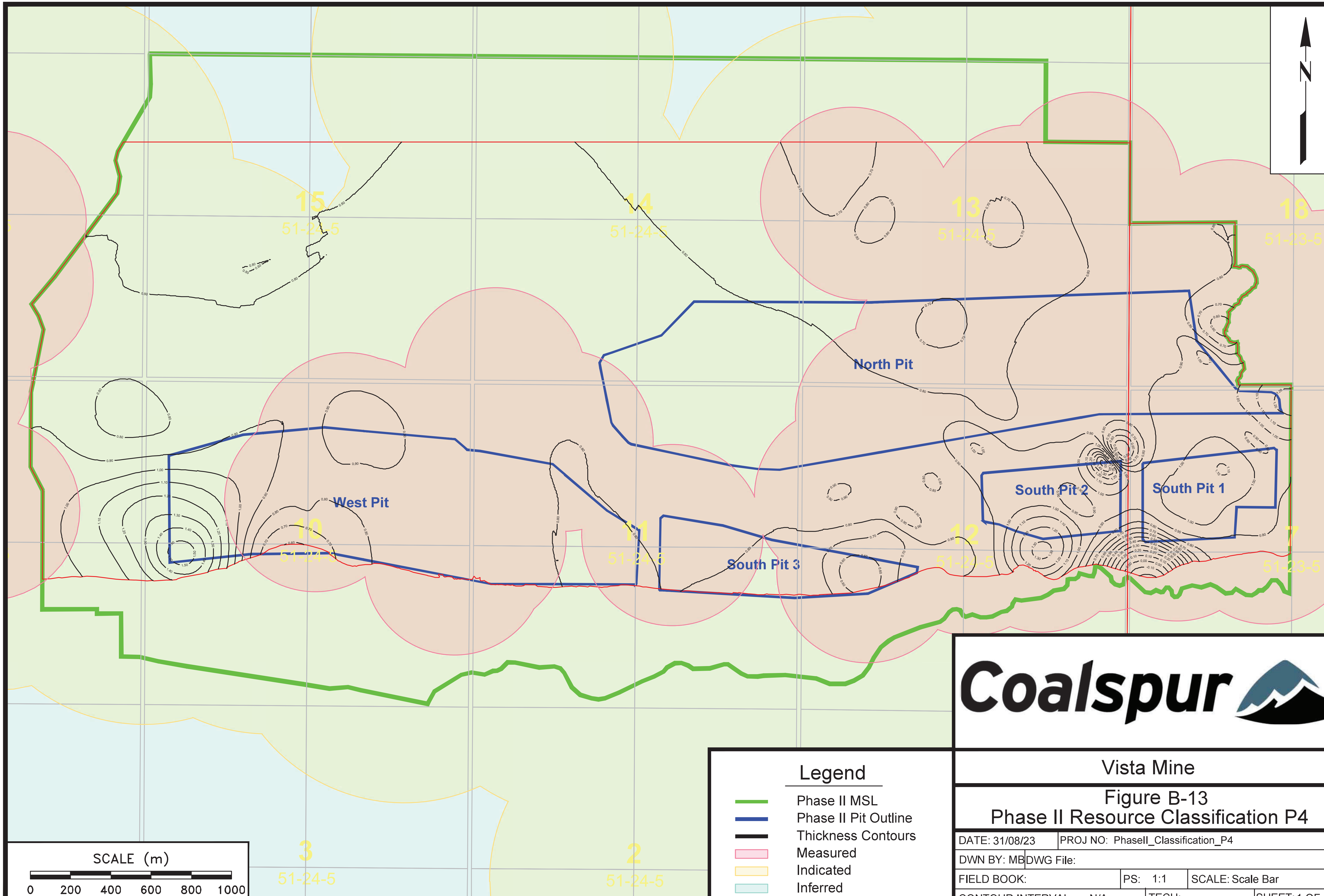
- Phase II MSL
- Phase II Pit Outline
- Thickness Contours
- Measured
- Indicated
- Inferred

Coalspur

Vista Mine

Figure B-12
Phase II Resource Classification L2U

DATE: 31/08/23	PROJ NO: PhaseII_ResourceClassification_L2U		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	



Legend

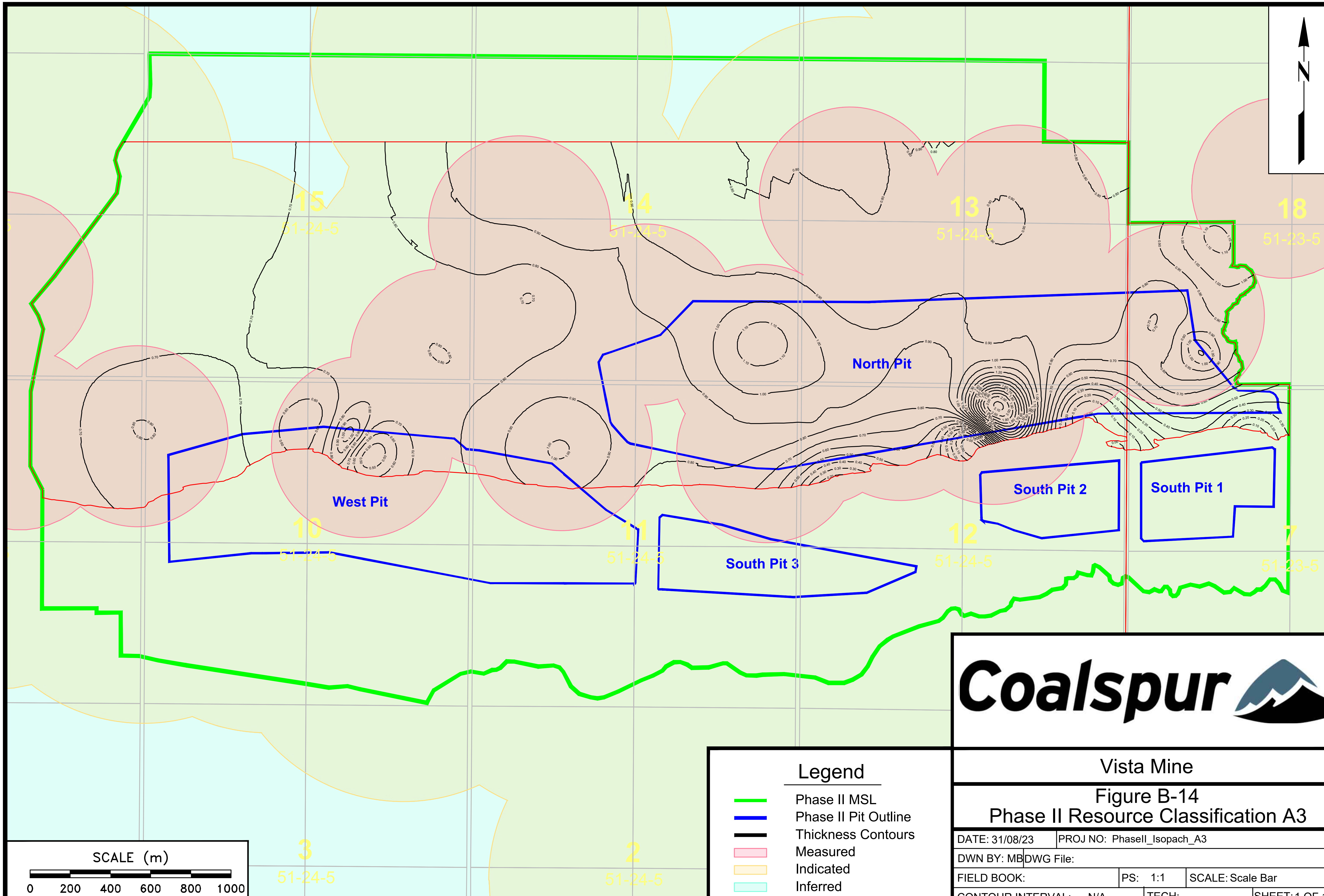
- Phase II MSL
- Phase II Pit Outline
- Thickness Contours
- Measured
- Indicated
- Inferred

Coalspur

Vista Mine

Figure B-13
Phase II Resource Classification P4

DATE: 31/08/23	PROJ NO: PhaseII_Classification_P4		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	



Coalspur

Vista Mine

Figure B-14
Phase II Resource Classification A3

DATE: 31/08/23	PROJ NO: PhaseII_Isopach_A3		
DWN BY: MB	DWG File:		
FIELD BOOK:	PS: 1:1	SCALE: Scale Bar	
CONTOUR INTERVAL: N/A	TECH:	SHEET: 1 OF 1	

B.4.4 Estimated Coal Reserves

Estimation of run-of-mine (ROM) coal qualities were completed using the same modifying factors as were applied in the mine scheduling database to estimate ROM coal quantities. These factors are based on the blasting, coal cleaning and coal mining techniques and equipment selected.

- Minimum mineable coal thickness 0.5 m
- Minimum removable parting thickness 0.3 m

B.4.5 Unrecovered Coal

Below the McPherson seam occurs a uniform 75 to 80 m of barren strata before the main Silkstone resource occurs. This is 1.5 to 2 m thick and called the SLK1 seam, or S1, and is very consistent across the Vista property. It occurs 5 to 15 m below the SLK2 carbonaceous zone which is highly variable in both position and character; sometimes verging on coal, but more often absent entirely. Neither of the two Silkstone zones are surface mineable below the current economic pit, but the SLK1 might support underground exploitation.

Below the Silkstone, the Mynheer is potentially present, but appears non-depositional within the entire Vista area. There is a 0.5 to 1 m coaly zone within approximately 30 m below the Silkstone that may be linked to the Silkstone zone, but is modelled as Mynheer. It occurs in fewer than 10 drill holes across the Vista Project, largely located to the west of the Vista Project area.

B.4.6 Coal Recovery

For recovery estimates, Table B-3 provides a summary of coal recovery averages per ply thickness:

Table B-3 Raw Coal Thickness and Recovery

Phase II Seam Averages		
Ply	**Thickness (m)**	**Recovery**
V7	0.6	50.9%
V6U	0.5	63.3%
V6L	1.1	63.3%
V5Ub	1.0	59.6%
V5Ua	0.7	59.6%
V5L	1.6	59.6%
V4	0.5	63.0%
V3T	0.8	54.9%
V3Ub	1.0	54.9%
V3Ua	2.0	54.9%
V3L	0.6	54.9%
V2U	0.5	36.9%
V2L	0.4	36.9%

V1	1.0	54.5%
A3	0.8	49.0%
A2	0.7	49.0%
A1	0.4	49.0%
L3	0.6	48.4%
L2U	0.7	48.4%
L2L	0.6	48.4%
L1	0.6	48.4%
P4	0.9	55.3%
P3U	0.4	55.3%
P3L	1.2	55.3%
P2U	0.6	55.3%
P2L	0.8	55.3%
P1	0.9	55.3%

B.5 Coal Quality

As previously described, coal quality data were provided by Coalspur on 1,247 proximate analysis results from 66 uniquely named core holes. Each record included "As Analyzed" and "Raw" values for various analytic and calculated parameters. The coal quality data for the overall Vista area (Phase I and Phase II inclusive) were utilized as base data for subsequent compositing and modelling. The parameters in the coal quality model include specific gravity, equilibrium and total moisture contents, ash content, sulfur content, volatile matter content, fixed carbon content and calorific value. The coal quality model was based on the "Raw" parameters in the supplied spreadsheets. The coal quality model has been largely validated by the phase I operation.

The initial step in the creation of the coal quality models was to compare the positional data in the quality ply database to the seam positions in the stratigraphic reserve model. In this fashion, the ply samples are combined to determine composite quality values for each seam in drill holes containing coal quality data. Because the major seam groups contain multiple plies, and because the thickness of waste between these plies varies from non-separable to separable values, this exercise had several goals, namely:

- Adjust the seam positional data (depth from: depth to) values in the quality database to conform to the values in the lithology database. This included pro-rata adjustment of ply thicknesses within a given seam intercept, as required.
- Identify removable and non-removable parting horizons through appropriate lithology codes. This has two effects; first, to eliminate the use of quality values for removable parting horizons in seam composite quality estimates and second, to enable the estimation of removable parting quantities within a given seam.

- Establish the quantity of removable partings in any given seam. This value is required to enable accurate accounting of coal loss and dilution values for estimates of run-of-mine (ROM) quantities and qualities.

A summary listing of the available data for modelling for each seam resulting from the compositing exercise is shown in Table B-4.

Table B-4 Modelling Data by Quality Parameter

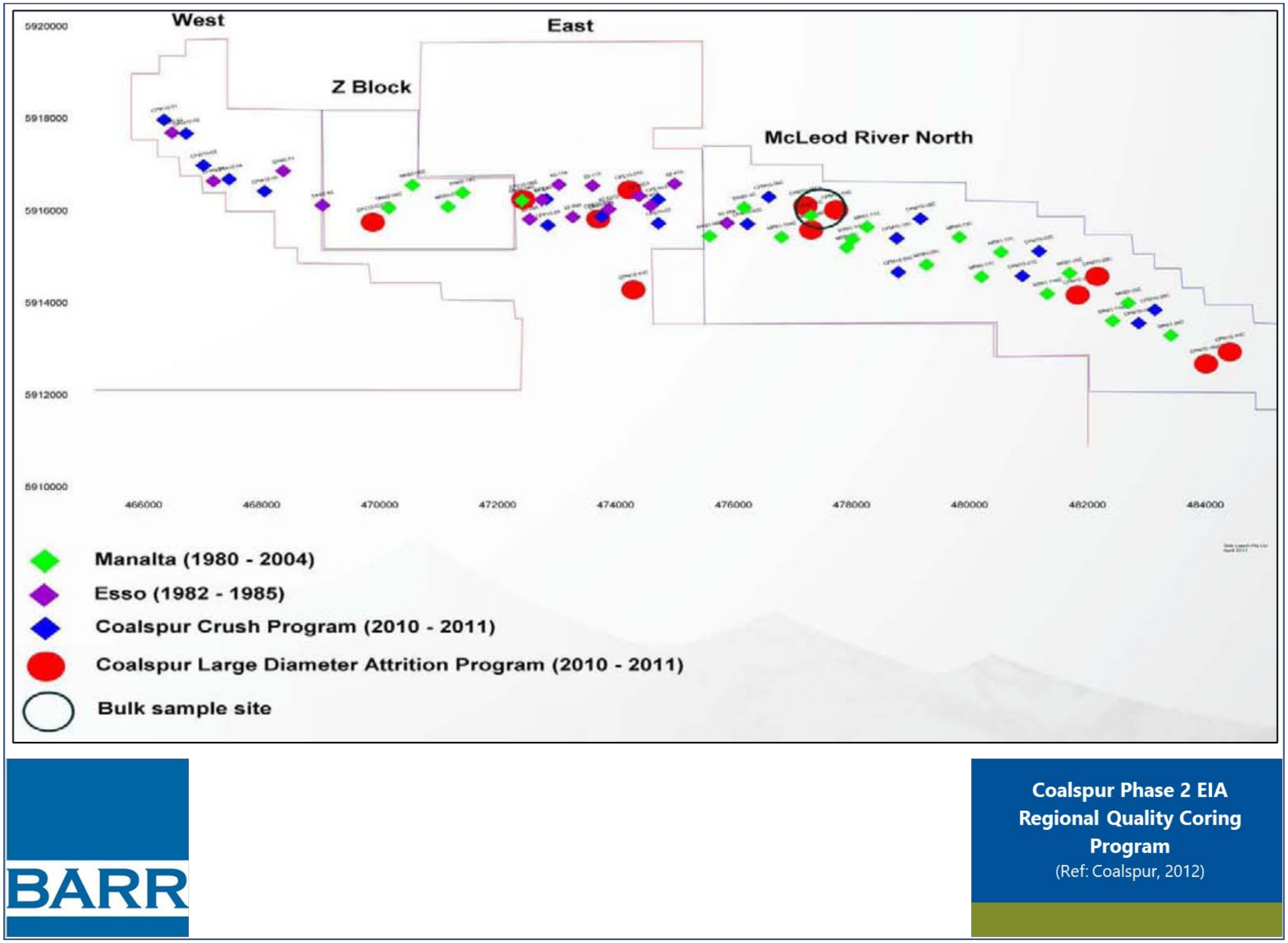
Item	Description	Units	Qty	Comment
A	Equilibrium Moisture Content	(Wt. %)	156	
B	Total Moisture	(Wt. %)	256	
C	Air Dried Moisture Content	(Wt. %)	491	
D	Relative Density	(g/cc)	416	Air dried moisture basis
E	Ash Content	(Wt. %)	491	Air dried moisture basis
F	Volatile Matter Content	(Wt. %)	314	Air dried moisture basis
G	Total Sulfur Content	(Wt. %)	314	Air dried moisture basis
H	Calorific Value	(Wt. %)	371	Air dried moisture basis
Raw Values (In Situ Moisture Content Basis)				
I	Equilibrium Moisture Content	(Wt. %)	491	Assigned value
J	In Situ Moisture Content	(Wt. %)	491	Eq. Moisture + 1 (I + 1)
K	In Situ Density	(g/cc)		From ash: density regression curves
L	Ash Content	(Wt. %)	491	Ash adjusted to In Situ Moisture E * (100-K)/(100-C)
M	Volatile Matter Content	(Wt. %)	314	VM adjusted to In Situ Moisture F * (100-K)/(100-C)
N	Total Sulfur Content	(Wt. %)	314	Sulfur adjusted to In Situ Moisture G * (100-K)/(100-C)
O	Calorific Value	(kcal/kg)	371	CV adjusted to In Situ Moisture and to kcal/kg H * 238.8 * (100-K)/(100-C)
P	Moisture and Ash Free Calorific Value	(kcal/kg)	371	Dry, ash free calorific value O * 100 / (100 - J - L)

* Quantity of sample points

The modelling technique for the coal quality parameters consisted of the creation of grid based surfaces representing projected in-place quality values on a pre-determined in situ moisture basis using the composited ply quality values as base data.

Grid surfaces created from data sets with less than four data points generally do not provide reasonable projections of quality over an area the size of Coalspur. Typically, the spatial distribution of sample populations of this size localizes the effects of known data points, and can produce misleading extrapolations. Marston does not typically model values for coal seams containing fewer than four data points; this was not an issue for Coalspur, as all seams included in mine planning had a sufficient quantity of data for preparation of grid-based surfaces. The locations of drill holes with associated proximate data used for quality modelling are shown Figure B-15.

Figure B-15 Regional Quality Coring Program Drill Hold Data Locations



B.5.1 Plant Yield and Clean Coal Quality Model

The proximate quality composite data was further utilized for the projection of clean coal qualities by seam through the use of plant simulation software and validated by actual Phase I mining. Essentially, the in-situ composites were modified by coal loss and waste dilution factors to estimate likely ROM coal qualities by seam in each drill hole. This approach starts with an estimate of the likely preparation plant coal feed quality by seam, with subsequent process simulation or analysis to estimate clean coal yield and qualities, these values were again validated by Phase I actual mining.

The in-situ coal composite qualities were each adjusted for coal losses and rock dilution additions, resulting in projected diluted coal feed qualities to the proposed preparation plant facilities. Clean coal quality parameters included in the model are yield, total moisture content, ash content and calorific value. These were available for all seams in drill holes that had proximate analysis data.

Mapping of the average calorific value of clean coal by seam group (Val d'Or, McLeod, and McPherson) illustrates an interesting trend of approximately 200 kcal/kg from low to high values as locations vary from the southeast of the project area to the northwest. These trends are based on the 1.55 gravity cut point cut-off. The Val d'Or group clean coal calorific value ranges from the 5600s (kcal/kg) to the 5800s, the McLeod from the 5200s to the 5400s, and the McPherson from the 5500s to the 5700s. This is unusual, and impacts mine development sequencing as a result of the need to blend the seam group coals to achieve target product quality characteristics.

B.5.1.1 Core Quality Testing Program

The coal bearing strata in the project is part of the upper Saunders Group from the Coalspur Formation. The two major mineable seams present are the Val d'Or and the McPherson though two other seams, Arbour & McLeod will also be mined. The coal is moderately low rank bituminous suited to thermal coal production targeting calorific value in the range of 5700 kcal/kg to 5900 kcal/kg on a gross, as received basis.

The feasibility study assessment on coal quality has been completed on a series of exploration programs undertaken by the project from 2010 - 2011, in addition to historical information retrieved from Esso's programs in the West and East blocks, and Mancal's work in the Z and McLeod River North blocks. The results have been transcribed into raw coal, washability, clean coal and yield databases for use in the feasibility study. The values estimated by the feasibility study have been largely validated by actual phase I mining. The Arbour seam sampled in the Phase II area shows the seam to be similar to the McLeod seam in both Phase I and Phase II, the Vista mine has been successful in processing surface mined McLeod coal during Phase I operations which provides additional confidence in the CPP's ability to process Arbour coal. If actuals and recoverable individual seams varies significantly from the model throughout the course of Phase II mining, additional coal quality testing will be completed, however it is not anticipated at this time.

B.5.1.2 In Situ Coal Quality

In Situ Moisture

Based on equilibrium moisture results which range from 9% in the west to 11% in the far easterly extent of the McLeod River North block, the in-situ moisture will range from 10% in the west to 12% in the east.

Inherent Calorific Value

Calorific Value dry, ash free, ranges from near 7600 kcal/kg in McPherson seam in the west, to slightly lower than 7400 kcal/kg in the east. Combined, the equilibrium moisture and calorific value results confirm there is a rank decrease west to east in the deposit and stratigraphically from the upper Val d'Or seam to the lower Silkstone seam.

B.5.1.3 Raw Ash and Sulphur

In Situ Moisture

The seams in Vista are generally variable in raw ash due to the presence of occasional thin stone bands. All seams are low to moderate in raw total sulphur content.

Val* d'Or *Seam

Val d'Or seam varies in thickness from approximately 8 m in the west to in excess of 16 m in the east. The seam generally presents as seven plies (upper V7 to lower V1) of quite variable thickness. Often, the plies contain interburdens splitting the plies into upper and lower sub plies. In the west, the upper plies V7 to V4 tend to thin out, reducing the total seam thickness. Most of the plies are moderate in raw ash though some sub plies (V6U, V5L, V4) are quite low in ash. All plies would be classed as low to moderate in total sulphur though there is a tendency for the upper plies to have moderate results.

Arbour Seam

Arbour seam is present in the west only. It has moderate to high raw ash and low sulphur. This seam is included in the mine plan reserves.

McLeod Seam

McLeod seam generally presents as three plies with a total coal thickness up to 4 m. All of the plies are moderate to high in raw ash. Total sulphur is low.

McPherson Seam

McPherson seam is the second seam of major interest and generally presents as a total seam thickness of 6 m to 7 m throughout the deposit. The four coal plies in the seam are moderate in raw ash and low in sulphur.

Silkstone Seam

Little is known of the quality in Silkstone seam. Based on scant results, it presents as three sub plies of which the upper two are high in ash. The seam has low to moderate total sulphur. This seam is not included in the mine plan reserves.

Clean Coal Yield, Product Ash, Moisture and Calorific Value

Val d'Or and McPherson seams represent approximately 80% of the mineable resource. Both will contribute to realizing an export quality product with gross calorific value in the range of 5700 kcal/kg to 5900 kcal/kg, depending upon the choice of cut-point density.

B.5.2 Coal Seams and Preparation Plant

The phase I operation as it currently functions is mining limited, meaning that the plant is capable of processing more coal than the mine is supplying. With the addition of Phase II Coalspur plans to construct one additional module, this additional module combined with the currently unused capacity will be enough to process the increased production. The new module will be the same as the existing modules. The material handling infrastructure for transporting raw coal, clean coal, course and fine refuse do not require any modification to accommodate phase II, the only change to infrastructure be the addition of the new modules.

No other changes or modifications to the wash plant are anticipated outside of increased water and chemical usage to match the increase in plant productivity. This wash plant has proven its ability to successfully process the Val D'Or, McPherson and McLeod coal seams. In 2022 the plant recovery was 54.2% which is the same as the anticipated 54.2% overall plant recovery. The Arbour seam is the only coal seam included in the phase II mine plan that is not currently mined or processed in Phase I. The Arbour seam makes up approximately 7.9% of the Phase II reserve, based on phase I processing experience and the Arbour seams similarity to the McLeod seam, Coalspur believes that the wash plant will be successful in handling the Arbour seam. The viability of the Phase II project does not rely on the successful processing of Arbour seam coal, it is however included in this mine plan because it can be successfully processed and in the interest of maximizing resource recovery from the project.

B.5.2.1 Tailings Generation Sensitivity Analysis

Coalspur has conducted a sensitivity analysis to compare the tailings storage capacity in both Phase I and Phase II against the combined tailings generation from Phase I and Phase II. For this sensitivity analysis, we have provided definitions for some key terms as they are used in the sensitivity analysis.

- Tailings Slurry
    - The total volume of tailings water and solids. This is the volume that comes out of the thickener underflow.
- Tailings Deposition (Tailings)
    - The volume of solids and unrecovered water deposited as tailings.
- Water Recovery
    - The volume of water that is recovered (recycled) from the tailings cell over the course of active deposition. Does not include water that is liberated from the tailings while the cell is inactive or undergoing capping.
- Percent Solids
    - The percentage of solids, by weight, that is present in the tailings slurry.

A sensitivity analysis was conducted for this based on the estimated coal resource for each coal member and under different extraction operation, and normal and upper range of variables affecting produced tailings. For this analysis the variables affecting the volume of tailings produced are:

1. The volume of slurry produced per raw tonne processed at the coal processing plant
2. The percentage of solid particles present in the slurry
3. The amount of water recycled from the tailings cell

Since the coal quality sampling completed in Phase II shows the coal to be very similar to the coal sampled and mined in Phase I, the ranges for each variable considered come from laboratory testing and actual McPherson cell measurements. This provides the best possible information for analysis of tailings produced by Phase II.

Table B-5 shows the estimated raw coal resource based on each coal member for Phase I and Phase II. Table B-6 provides information of available tailings storage capacity for each of the tailings storage cells. Based on Table B-6, a total of 26,540,630 m^3 of total tailings storage capacity will be available at the beginning of Phase II for both Phase I and Phase II operations.

Regarding the Arbour seam, the only seam not currently mined in the Phase I operation. The Arbour seam consist of the small portion of overall coal production for Phase II accounting for 8% of Phase II production and 5% of Phase I and Phase II production. Therefore, sensitivity of the operation to quality of Arbour seam is minimal. Based on the available testing data from Phase II, the quality of coal of McLeod, McPherson and Val d'Or seams are in same range as what has been produced in Phase I (as expected based on the proximity). Arbour seam testing showed that the coal quality is similar to the McLeod seam and therefore, it is expected to produce tailings at a similar rate.

Table B-5 Combined Raw Coal Reserves Beginning of Phase II

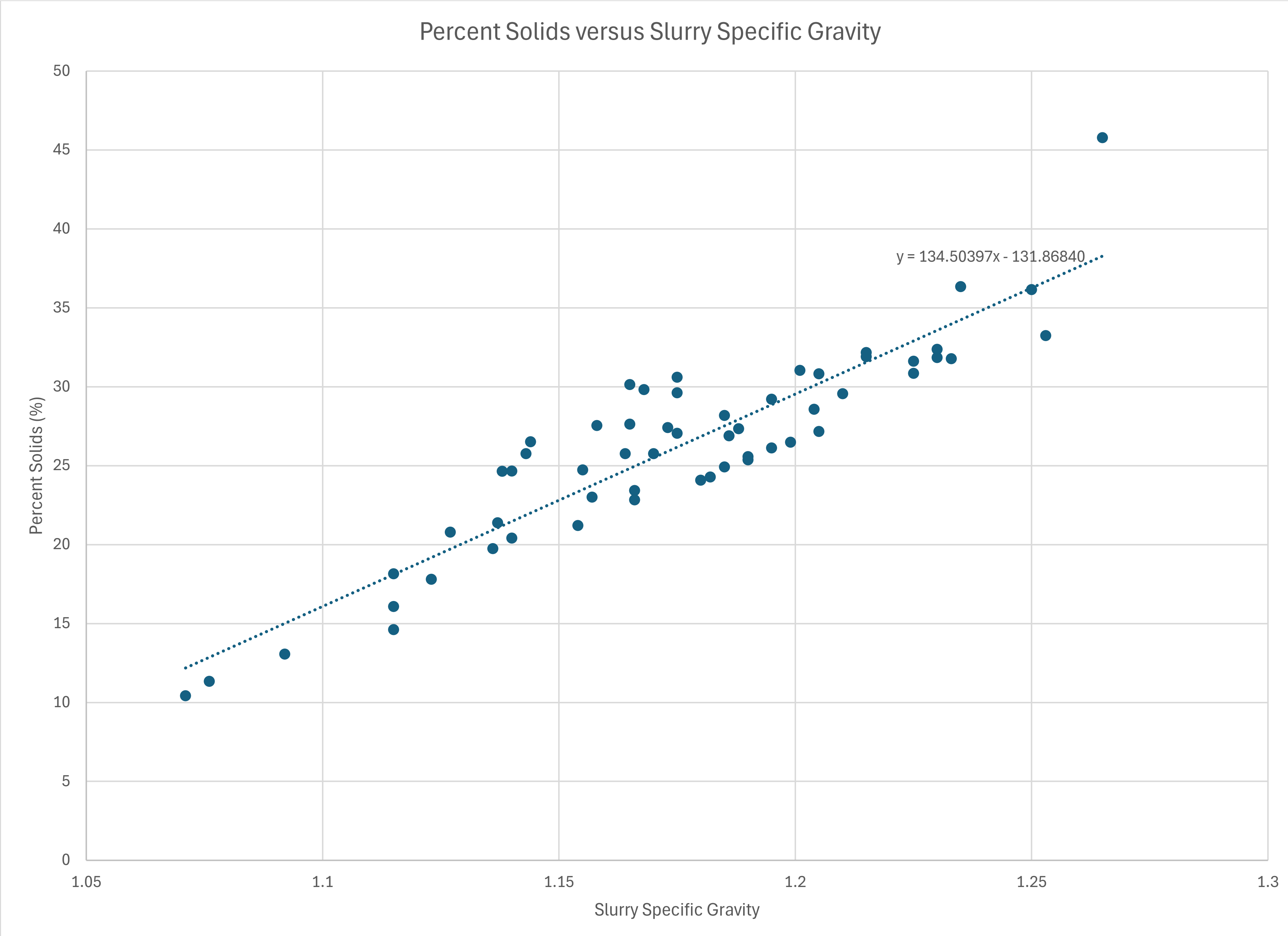
Operation	Tonnes				
	Val d'Or	Arbour	McLeod	McPherson	Total
Phase 1					
Surface Mined Raw Coal	24,010,418		520,762	3,391,192	27,922,372
Highwall Miner Raw Coal	8,212,210		1,559,368	1,831,172	11,602,750
Vista Test Underground Mine Raw Coal	3,379,999				3,379,999
Total Phase I Raw Coal	**35,602,627**	**-**	**2,080,130**	**5,222,364**	**42,905,121**
Phase 2					
Surface Mined Raw Coal	37,040,098	6,358,494	3,224,264	16,213,050	62,835,906
Highwall Miner Raw Coal	9,099,281	-	2,249,962	4,081,180	15,430,423
Total Phase II Raw Coal	**46,139,379**	**6,358,494**	**5,474,226**	**20,294,230**	**78,266,329**

Table B-6 Remaining Tailings Storage Capacity Beginning of Phase II

Beginning of Phase II - 2026	
Remaining Tailings Storage Capacity (m^3)	
Cell 6	1,269,119
Cell 7	2,296,497
Cell 8	3,928,372
Cell 9	5,175,328
SP1	6,595,807
SP2	5,438,847
SP3	1,836,660
Total (m^3)	**26,540,630**

From former testing programs Coalspur has found that the coal in the Phase II area is very similar to the coal in the Phase I area with respect to the raw coal ash and fine content. The interbedded rock volumes expected to be processed by the CPP is determined more by mining method than geologic data, however, boreholes show the partings between coal are similar to the coal that has been encountered in the Phase I mining area. The mining method is the same for Phase II as Phase I. Because the mining method is the same for Phase II as Phase I, testing of Phase II coal is so similar to Phase I and the availability of information of the actual tailings generation from the CPP, this information was selected to be the basis for this sensitivity analysis. Below is a list of assumptions.

- The found relationship between percent solid in the slurry and slurry density remains the same. This is presented in Figure B-16
- Water recovery is the water recovered from the cell during active deposition
- Tailings cells are not filled more than once
- There is no mining loss, which would result in a reduction in tailings production



Percent Solids versus Slurry Specific Gravity
y = 134.50397x - 131.86840
Percent Solids (%)
Slurry Specific Gravity
50
45
40
35
30
25
20
15
10
5
0
1.05
1.1
1.15
1.2
1.25
1.3

The volume of tailings produced is determined by finding the total volume of tailings slurry (total fines and water) produced, then determining the total volume of fines and water in the tailings slurry and finally subtracting the recycled water to find the total amount of fines and unrecovered water deposited. For each variable an upper, middle and lower bound was selected. Then each combination of potential outcomes is calculated.

For the volume of slurry produced per raw tonne of coal processed a lower bound of 10.09% and an upper bound of 13.95% was used. These were determined based on measurements from the CPP, the tailings volume based on flow meter measurements and the raw tonnes processed based on belt scale measurements. Outliers in the data were removed, and the upper and lower bounds were determined as being the average plus or minus the standard deviation in the data. The percentage of solids in the slurry was determined in the same manner as above, with a lower bound of 19.94% and an upper bound of 32.40%

For the water recovery a conservative approach was used. Laboratory testing on the tailings found that 70% of water is freed from the tailings within 24 hours of deposition. Based on this information and measurements from cell 1 actual deposition an upper bound of 72% water recovery was selected. Based on measurements from the McPherson tailings cells where the worst period of water recovery was experienced a lower bound of 58% was selected. The water recovery used in this analysis is the percentage of water deposited in the cell that is recovered while the cell is in active deposition. Total water recovery is expected to be higher as additional water is recovered while the cell is inactive and during capping. Table B-7 shows the values used in this analysis.

Table B-7 Sensitivity Analysis Input Variables

Scenario	Water Recovery	Slurry Volume per Raw Tonne	Percent solid
Low	58.0%	0.325	19.94%
Medium	65.0%	0.431	26.17%
High	72.0%	0.538	32.40%

In the McPherson tailings cell applications tailings generation has been referred to as a volume of slurry produced per clean tonne of coal processed. CPP production is commonly measured by the amount of clean coal, therefore the clean coal tonnage has been used for describing tailings generation rates. This can potentially be misleading, as it may imply that a higher clean recovery from the plant would result in an increase in tailings generation, when the opposite is true. Coalspur will continue to present tailings generation rates as a volume per clean tonne of coal processed as it is useful to compare with previous applications and measurements from the existing McPherson Tailings cells. In this sensitivity analysis the minimum rate of tailings generation was found to be 0.217m3/clean tonne and the maximum is 0.523m3/clean tonne.

Table B-8 shows the results of the sensitivity analysis, a total of 28 scenarios were considered for the sensitivity analysis. Of the 28 scenarios there is adequate or spare capacity in 19 (68%) of the scenarios and there is inadequate capacity in 9 (32%) of the scenarios.

Table B-8 Tailings Generation Sensitivity Analysis Results

Water Recovered from Tailings	Tailings m3 / Raw Tonne Processed	Percent Solid in Slurry	Tailings Generated	Remaining Storage
Low	Low	Low	19,241,054	7,299,576
Low	Low	Mid	20,045,874	6,494,756
Low	Low	High	20,979,311	5,561,319
Low	Mid	Low	25,538,712	1,001,918
Low	Mid	Mid	26,606,952	(66,322)
Low	Mid	High	27,845,906	(1,305,276)
Low	High	Low	31,836,370	(5,295,740)
Low	High	Mid	33,168,030	(6,627,400)
Low	High	High	34,712,500	(8,171,870)
Mid	Low	Low	16,811,428	9,729,202
Mid	Low	Mid	17,713,381	8,827,249
Mid	Low	High	18,759,474	7,781,156
Mid	Mid	Low	22,313,861	4,226,769
Mid	Mid	Mid	23,511,027	3,029,603
Mid	Mid	High	24,899,509	1,641,121
Mid	High	Low	27,816,294	(1,275,664)
Mid	High	Mid	29,308,672	(2,768,042)
Mid	High	High	31,039,545	(4,498,915)
High	Low	Low	14,381,801	12,158,829
High	Low	Mid	15,380,888	11,159,742
High	Low	High	16,539,638	10,000,992
High	Mid	Low	19,089,010	7,451,620
High	Mid	Mid	20,415,101	6,125,529
High	Mid	High	21,953,113	4,587,517
High	High	Low	23,796,219	2,744,411
High	High	Mid	25,449,315	1,091,315
High	High	High	27,366,589	(825,959)
Average			**23,574,799**	**2,653,852**

*Negative remaining storage

With respect to tailings generated by Phase II specifically, the expected volume of tailings generated from Phase II is ~14,126,000 m^3. From the sensitivity analysis, the minimum required Phase II tailings storage is 7,925,826m^3 and the maximum is 18,753,216m^3.

This sensitivity analysis provides a wide range of scenarios for total tailings generation, including rates of tailings generation that are much higher and lower than what has been indicated from laboratory testing and observed during McPherson Tailings cell operation. The tailings production at the Vista mine operation is closely monitored, if the actual tailings deposition is projecting to be above the total storage remaining then several mitigation methods can be employed to maximize deposition volumes. Mitigation methods include:

- Refilling Tailings Cells after initial deposition.
    - The McPherson tailings cells have demonstrated that significant consolidation of the solid material in the cell occurs after deposition has ended. After consolidation has occurred these cells can be refilled to maximize available storage capacity. Coalspur has started adopting this procedure as a best practice to maximize available storage capacity.
- Reducing the rate of CPP processing to improve consolidation and water recovery during active deposition.
- Treating and releasing liberated tailings water to provide more of the tailings storage space for tailings
    - Would be employed as an option if necessary, in the later years of operation, when the tailings affected water is no longer required for further coal processing.

In the majority of cases analyzed in this sensitivity analysis, including the expected case, there is adequate tailings storage available for the Phase I and Phase II projects. If actual tailings generation rates are found to be at a rate at which insufficient tailings storage is available, then mitigation measures will be employed to ensure adequate capacity.

B.6 Geotechnical Assessment

Geotechnical assessments were completed by Barr Engineering & Environmental Science Canada, Ltd. (Barr). The assessment uses both historical data sets, information collected during the field investigation programs in 2010-2011, 2012-2013 and geotechnical analyses completed between 2018-2022. Since then, no additional exploration or assessment has been completed for the Phase II area, although additional geotechnical and hydrogeological assessments have been completed in support of the construction and operation of tailings cells, waste rock dump, high walls and highwall mining within the Phase I area. The most recent data and analysis from ongoing Phase I operations have been used to compare and validate the findings from Phase II feasibility study. Barr's approach and rationale are provided in Appendix B2.

B.6.1 Geotechnical Characterization

Between 1981 and 1983 a total of approximately 80 boreholes (including core holes) were drilled to help with exploration activities and geotechnical characterization of the subsurface. Later, fifteen core holes were drilled in 2011 and logged by KCB to undertake geotechnical evaluation of the Phase II area. two core holes were drilled in March 2011 (CPM10-047 and CPM10-048), and 13 core holes (GT11-01-CH to GT11-09-CH and GT11-13-CH) were drilled between August 29 and November 7, 2011. This investigation included three inclined holes GT11-06B-CH and GT11-06D-CH undertaken adjacent to the vertical core hole GT11-06A-CH.

The geotechnical investigation found that the encountered expected units of the Coalspur Formation, including interbedded sandstones, siltstones, bentonitic to carbonaceous mudstones, and coal seams. The sandstone unit was found to be medium strong to strong, with very weak to medium strong siltstone and mudstone layers. Bentonite and bentonitic mudstones were found associated with coal seams and represent the weakest layer in the units of concern.

The stratigraphic units within the mine lease boundary are generally planar, dipping gently toward north to northeast at 6° increasing to 11° towards the southeast in the Hinton East lease area, with some steeper bedding angles up to 25° in the sandstone, siltstone, and mudstone layers in some boreholes in the southeast of the McLeod River property area. Bedding dips generally appear consistent across upper and lower seams within the Vista Coal property, indicating a parallel sequence of bedding with no significant faulting. The assessment of 3D geological model produced for the area indicated dips ranging between 4° to 10° for GT11-01-CH; 6° to 8° for GT11-03-CH; 7° to 8° for CPM10-47, and dips angles of 7° for CPM10-48. All these dips were orientated in northerly direction.

KCB performed a stereographic analysis of dipmeter data which indicated a dip/dip direction of 07°/12°. Acoustic televiewer data indicated a joint set with a dip/dip direction of 64°/064° and 78°/078°. Based on a review of the available data, the previously assumed bedding angle of 6° was considered appropriate for the stability analyses.

The drilling program indicated that bentonite layers are present below the Val d’Or seam. Bentonite layers occur frequently on the top of coal seams and as partings within the individual coal seams.

The findings of this study are consistent with the multiple geotechnical investigations conducted within ongoing Phase I operation between 2018-2022.

B.6.2 Rock Mass Interpretation

B.6.2.1 Rock Quality and Rock Fracture Frequency

KCB utilized the modified Hoek-Brown failure criterion to determine equivalent angles of friction and cohesive strengths for the rock mass and stress range based on the intact UCS strength, material constants, geological strength index, and disturbance factor. The comparison of rock quality and rock fracture frequency data from both historical and the 2011 investigation is summarized in Table B-9.

Table B-9 Comparison of Rock Quality Designations from KCB Data

Rock Type	Hinton West	Hinton East	McLeod River
Sandstone	Excellent	Excellent	Good to Excellent
Mudstone	Excellent	Good to Excellent	Fair to Excellent
Siltstone	Excellent	Excellent	Good to Excellent

B.6.2.2 Unconfined Compressive Strengths

Uniaxial compressive strength (UCS) tests were undertaken on 44 rock core samples representing each rock grade and for each rock type. The average UCS laboratory strength tests for each rock grade and rock type are summarized in Table B-10.

Table B-10 Summary of Laboratory UCS Strength Results

No of Tests	Description	Rock Grade	ISRM Intact Rock Strength Classification	Laboratory UCS Results (MPa)			
				Min	Max	Average	SD
1	Mudstone	R1	Very weak			4.6	
10	Mudstone	R2	Weak	6.7	24.5	14.6	5.5
4	Mudstone	R3	Medium strong	25.6	37.6	30.2	5.6
1	Mudstone	R4	Strong			54.2	
7	Sandstone	R2	Weak	14.4	19.0	16.8	1.5
9	Sandstone	R3	Medium strong	25.4	46.6	37.1	8.3
3	Sandstone	R4	Strong	50.7	61.3	57.2	5.7
4	Siltstone	R2	Weak	10.6	20.9	16.5	4.5
3	Siltstone	R3	Medium strong	27.3	46.4	39.4	10.5
2	Siltstone	R4	Strong	50.9	63.7	57.3	9.0

Based on the results of the 2011 investigation and previous laboratory data (a total of 809 UCS rock strength point load tests), the results indicate a good correlation between the average UCS laboratory results and the average UCS strengths from the point load tests results, for rock grades including R1 to R3 (medium strength rock), indicating that the test correction factor of 23 is reasonable. The average UCS laboratory results for rock grades above R4 (strong rock) are showing a lower average UCS laboratory strength, when compared with the average UCS correlated values from the point load test results, indicating that a test correction factor of about 20 is more suitable for strong rock.

The results of the UCS testing from this study are consistent with UCS testing on the same formations which was conducted between 2018-2022.

B.6.2.3 Geological Strength Index

The geological strength index (GSI) has been based on the review of historical evidence and the review of the KCB rock cores. KCB recommended a GSI value of 60 based on "Good, Rough slightly weathered" joint surface conditions and "Blocky - well interbedded undisturbed rock mass consisting of cubical blocks formed by three intersecting discontinuity sets" for rock structure.

B.6.2.4 Rock Mass Shear Strength Parameters

KCB's rock mass shear strength parameters using the Hoek-Brown failure criterion are shown in Table B-11. The Material Index (mi) values and the modulus ratio (MR) were based on published information for similar rock material types. The effects of heavy blast damage as well as stress relief due to removal of overburden result in disturbance of the rock mass. KCB considered that the "disturbed" rock mass rating using D=1 is appropriate. The disturbance factor of 1.0 is recommended to be applied for the zones within a radius equal to heigh of the bench and beyond that, a disturbance factor of 0.2 seems appropriate.

Table B-11 Estimated Rock Mass Discontinuity Shear Strength Parameters

Description	Rock Grade	UCS (MPa)	Geological Strength Index (GSI)	Material Index (mi)	Disturb Factor [1] (D)	Modulus Ratio (MR)	Cohesion (kPa)	Friction angle, Ø (degrees)
Sandstone	R2	16.8	60	17	1	275	600	23
Sandstone	R3	37.1	60	17	1	275	1300	23
Sandstone	R4	57.2	60	17	1	275	2000	23
Siltstone	R2	16.5	60	7	1	375	400	17
Siltstone	R3	39.4	60	7	1	375	1000	17
Siltstone	R4	57.3	60	7	1	375	1500	17
Mudstone	R1	4.6	60	4	1	250	90	13
Mudstone	R2	14.6	60	4	1	250	300	13
Mudstone	R3	30.2	60	4	1	250	600	13
Mudstone	R4	54.2	60	4	1	250	1100	13

[1] For the zone of a radius less than the bench height

Table B-12 shows the design parameters KCB and Barr recommended to be used in the slope stability analysis of the highwall slope.

Table B-12 Design Parameters for Stability Analyses

<table>
<tr><th rowspan="3">Material</th><th rowspan="3">Rock Grade</th><th colspan="4">Strength Parameters</th></tr>
<tr><th colspan="2">Intact Rock</th><th colspan="2">Discontinuities</th></tr>
<tr><th>Friction (°)</th><th>Cohesion (kPa)</th><th>Friction (°)</th><th>Cohesion (kPa)</th></tr>
<tr><td>Glacial Till</td><td>n/a</td><td>37</td><td>0</td><td>n/a</td><td>n/a</td></tr>
<tr><td>Sandstone</td><td>R2</td><td>23</td><td>600</td><td rowspan="3">31</td><td rowspan="3">20</td></tr>
<tr><td>Sandstone</td><td>R3</td><td>23</td><td>1300</td></tr>
<tr><td>Sandstone</td><td>R4</td><td>23</td><td>2000</td></tr>
<tr><td>Siltstone</td><td>R2</td><td>17</td><td>400</td><td rowspan="3">20</td><td rowspan="3">50</td></tr>
<tr><td>Siltstone</td><td>R3</td><td>17</td><td>1000</td></tr>
<tr><td>Siltstone</td><td>R4</td><td>17</td><td>1500</td></tr>
<tr><td>Mudstone</td><td>R1</td><td>13</td><td>90</td><td rowspan="3">18</td><td rowspan="3">0</td></tr>
<tr><td>Mudstone</td><td>R2</td><td>13</td><td>300</td></tr>
<tr><td>Mudstone</td><td>R3</td><td>13</td><td>600</td></tr>
</table>

Material	Rock Grade	Strength Parameters			
		Intact Rock		Discontinuities	
		Friction (°)	Cohesion (kPa)	Friction (°)	Cohesion (kPa)
Mudstone	R4	13	1100		
Coal	n/a	26	3.5	n/a	n/a
Bentonite	n/a	13.1	0	n/a	n/a

B.6.2.5 Soil Overburden Interpretation

Laboratory testing of soil samples comprising visual classification, moisture contents, Atterberg limits and grain size distribution including hydrometer tests were undertaken on the overburden material. Grain size analyses on till samples indicated approximately 55% to 60% of silt and clay. Atterberg limit test results indicated that the tills range from non-plastic to medium plastic.

North Dump

KCB conducted three test holes in 2010 and 2011 in the north dump prior to construction: GT11-22-AG, GT11-27-AG, and CT 11-29-AG. The results of the investigation indicated that the site is underlain primarily by poorly graded silty sand till with trace to some clay, gravel, and cobbles. A clayey till layer was encountered at GT11-22-AG from 1.1 to 19.m below ground surface, and at GT11-29-AG from 17.7 to 19.8m below ground surface. Weathered sandstone was encountered at a depth of 0.8m below ground surface at GT1-27-AG.

This facility was constructed by Coalspur and has been in operation with start of Phase I mining activity. Coalspur will continue to utilize the facility during Phase II mining activity; therefore, a summary has been included in this report.

B.6.3 Hydrogeology

B.6.3.1 Methodology

This section provides a summary of the hydrogeology of the Vista Coal Mine site, which has previously been characterized and described in multiple reports. KCB (2012) prepared a geotechnical and hydrogeological investigation report for Phases I and II, Matrix (2022) prepared a baseline hydrogeology report for Phase II, Barr (2022) prepared the hydrogeology Environmental Impact Assessment (EIA) for Phase II, and Barr (2024) prepared an update the to the hydrogeology assessment. The hydrogeology assessments included the characterization of both the shallow overburden and the deeper, fractured rock groundwater systems. The objectives of the assessments were to determine the interactions between the proposed mine developments and the natural groundwater systems. Barr developed a 3D numerical groundwater flow model to assess the impacts of mining development on the aquifers and streamflow as well as groundwater implications for mine dewatering, seepage pathways, and water supply.

B.6.3.2 Surficial Hydrogeology

Additions to the historical data set were derived from the 2011 site investigation (KCB, 2012), which included the installation of 39 standpipe piezometers and in-situ hydraulic conductivity testing of 18 piezometers, and more recent work as compiled by Matrix (2022). Many of the piezometers were screened in various types of glacial till with some installed in bedrock units and isolated gravel units. The glacial till encountered across the site was heterogeneous and ranged in hydraulic conductivity from 3.8×10^{-9} to 2.4×10^{-4} m/s. Table B-13 contains the range of hydraulic conductivity values for the different soil units tested.

Table B-13 Summary of Hydraulic Conductivities for Each Major Soil Type

Soil Unit	Minimum Hydraulic Conductivity (m/s)	Maximum Hydraulic Conductivity (m/s)	Geometric Mean of Hydraulic Conductivity (m/s)
Gravel	8.6E-06	2.4E-04	5.0E-05
Sand Till	1.4E-07	2.1E-04	2.6E-06
Silt Till	1.4E-08	8.6E-07	2.1E-07
Clay Till	3.8E-09	1.6E-06	1.6E-07
Sandstone	7.2E-07	1.2E-06	1.0E-06

Glacial till is the predominant soil unit present across the Vista site, consisting of interbedded compact to hard sand, silt, and clay tills with layers of gravel, cobbles, and boulders. Bedrock depths vary across the site with rafted bedrock units common. The regional surficial geology consists mainly of an upper layer of muskeg, ranging from 0.1m to 0.5m thick underlain by glacial till and bedrock.

Groundwater level measurements were collected from each piezometer during the site investigation. A summary table of observed water levels is provided in Matrix (2022), Appendix A. Matrix (2022) also contains groundwater contour maps; these are reproduced in the Phase II Hydrogeology EIA Report (Barr, 2022).

No significant surficial aquifer systems were identified during the previous investigations. Sand and gravel units encountered were infrequent and discontinuous across the investigated areas. Groundwater in these units appears to be confined and the water levels measured are not indicative of the regional groundwater table. The major site-wide surficial water bearing unit was identified as the glacial till unit, with minor perched and confined units observed. The heterogeneous nature of glacial till unit creates an aquifer system that does not appear to be laterally extensive or continuous, leading to variability in the water table.

Surficial groundwater flow direction in the McLeod River Block area was determined to be towards the southeast. The flow direction follows the topography which slopes towards McPherson Creek, in the southeast. This flow direction is also consistent with the findings of the Manalta, 1981 investigation, which is based on the interpretation of resistivity logs. The water table is generally about 5m below ground level except at the upland areas in the northwest and the southeast areas of McLeod River Block where the water table is found at approximately 12 to 17 m bgs. The groundwater flow system commonly exhibits topographical control in the overburden.

B.6.3.3 Bedrock Hydrogeology

Wardrop (2011) indicated two distinct groundwater flow systems within the lease area. The first is a deep, regional flow system found at depths of greater than 150 m. The second system is of local extent and is characterized by shallow groundwater flow (usually less than 150 m deep) through fractured sandstones and coal. This appears to be confirmed by a pumping test carried out by KCB at GT11-04-PW, installed in the sandstone formation (likely the Paskapoo Formation) between 50 m and 80 m below ground level. KCB concluded that groundwater flow in the sandstone appeared to be mainly controlled by a fracture system, as indicated by extensive oxidation of fractures found in the core samples. Drawdown was not observed in any of the overburden wells during pumping of GT11-04- PW suggesting limited hydraulic connectivity of bedrock formation with the overburden. No drawdown was observed in the core holes located within a radius of 2 km. Drilling and hydraulic testing results of GT11-05-PW and GT11-09-MW indicated low permeability in the McPherson coal seam and the sandstone between the McLeod and McPherson coal seams.

The inferred groundwater flow in most of the formations – Paskapoo sandstone, Val d'Or coal seam, McLeod coal seam, sandstone unit between McLeod and McPherson and McPherson coal seam is generally towards the east and southeast in the project area, with some localized west and north components. The measured groundwater elevations in the above formations range from 1083 masl to 1330 masl. Based on the packer testing, the hydraulic conductivity of bedrock formations ranges between 1.10×10^{-8} to 1.60×10^{-6} m/sec. Transmissivity of sandstone aquifer in bedrock was calculated from pumping test and is in the order of 84 m^2/day. The Storage coefficient was measured as 8.3×10^{-5}. KCB concluded that the well yield of 320 L/min appears to be sustainable over a period of 50 years ignoring the potential effects of unknown boundary conditions. A table of hydrogeological parameters of bedrock formations in McLeod block and Hinton East block of the project area based on the Wardrop (2011) and KCB (2012) studies is given in Table B-14.

Table B-14 Hydrogeological parameters of bedrock formations in McLeod block and Hinton East block (Wardrop, 2011 and KCB, 2012)

Property	Hydraulic Conductivity (m/s)		Transmissivity	Storage Coefficient	Reference
	Range	Average	Range (m^2/d)	Range	
Hinton East	7.4×10^{-9} to 2.5×10^{-6}	5.0×10^{-7}	0.025 – 11.2	10^{-3} to 10^{-4}	Wardrop (2011)
McLeod River North Block	10^{-6} to 10^{-8}	2.1×10^{-6}	1.7 – 6.8*	3×10^{-2} to 7×10^{-4}	
McLeod Block	1.1×10^{-8} to 1.30×10^{-6} [a]	1.31×10^{-7} [a]	84 [b]	8.3×10^{-5} [c]	KCB (2012)

Note: * Data represents average transmissivity in McPherson (1.7) and Val d'Or (6.8) sites (KCB, 2012).
a. Hydraulic conductivity of bed rock formation based on packer tests data analyses of core holes (current study)
b. T value for Sandstone overlying Val d'Or formation (KCB, 2012)
c. S value for Sandstone overlying Val d'Or formation (KCB, 2012)

B.6.3.4 Groundwater Modelling Overview

In support of the geotechnical and hydrogeological design components, Barr developed a 3D numerical groundwater flow model for the Vista Project using MODFLOW 6 (Langevin et al., 2017). A complete description of the construction and calibration of the MODFLOW model is included as Appendix A to the hydrogeology EIA (Barr, 2024).

The modelling assessed a Base Case of the existing Phase I mining operation, as well as an Application Case which includes the planned Phase II mining operation both with and without additional McLeod River diversion. The Base Case assessment evaluated groundwater flow, quantity, and quality based on existing conditions and Phase I operations so that the Phase II Application Case could be compared in the Application Case. Numerical modelling was used to assess groundwater flow and quantity based on the existing hydrogeological conditions for the LSA and RSA. Groundwater quality was assessed by reviewing the Phase I EIA evaluation and by assessing groundwater monitoring data from 2018 to 2020. The Application Case describes the combined effect of the proposed Phase II expansion with the currently operating and approved planned expansion of the Base Case. For the Application Case, the following potential incremental impacts of the Phase II Project were evaluated:

- Potential for mine cell dewatering to affect groundwater quantity, groundwater flow, and streamflow
- Potential for tailings and general mine operations to affect groundwater quality
- Filling time for the EPL.

The groundwater model was used to estimate impacts on water quantity and streamflow from the proposed mine pit dewatering under the Phase II expansion. The data were also used to qualitatively assess potential impacts to water users in the LSA and RSA.

Results

Barr used the calibrated model to simulate concurrent Phase I and Phase II operations from 2026-2035, Phase II mining only from 2036-2037, regrading of the end pit lake in 2038-2039 and 100 years of post-mining recovery from 2040-2240 (Barr, 2024). Hydraulic conductivity values within the mine cell footprints were changed during the simulation to simulate mining, backfilling, and in-pit tailings disposal and model boundary conditions were used to simulate mine dewatering, production well pumping, and streamflow augmentation.

The potential impacts of the proposed Phase II Project on groundwater in the LSA and the RSA include:

- Drawdown from Phase II mine cell dewatering remains within the LSA in the unconsolidated (till) aquifer but would extend into the RSA for the Paskapoo/Upper Coalspur bedrock aquifer.

- Drawdown would not impact privately owned wells in the RSA due the limited areal extent of drawdown in the unconsolidated (till) aquifer and the Paskapoo/Upper Coalspur bedrock aquifer, which are the most commonly utilized aquifers in the region.

- The filling time of the Phase II end pit lake from natural surface water and groundwater inflow is estimated to be approximately 165 years. Diversion of water from the McLeod River to augment the natural inflow could reduce the filling time to approximately 30 years.

- Phase II mine dewatering would potentially reduce baseflow to zero in the uppermost reaches of McPherson Creek and McPherson Creek Tributary 13 during mining. Downstream reaches of McPherson Creek could experience baseflow reductions of 60-90% relative to modeled pre-mining conditions. Recovery to post-mining equilibrium was estimated to take in excess of 100 years; however, the diversion of McLeod River water to the Phase II end pit lake accelerated the baseflow recovery by 40-60 years.

- Tailings disposal in mined-out cells is not expected to impact groundwater or surface water quality due to very low seepage rates compared to background flows.

- Mine operations currently underway at the Vista Mine have not impacted groundwater quality and the proposed additional mine operations that would be part of the proposed Phase II expansion are not expected to impact groundwater quality.

B.6.4 Pit Slope Stability

B.6.4.1 Design Criteria

The minimum factor of safety for a mine slope is based on the accepted probability of failure, the characteristics and size of the potential instability, and the associated costs according to "Guidelines for Open Pit Slope Design" by John Read and Peter Stacey. For slope stability assessments, the factor of safety used for open pit mines slopes is dependent on the operating environment. The typical values for factors of safety range from 1.2 for non-critical slopes to 1.5 for slopes containing critical access ramps or infrastructure such as in-pit crushers. However, a minimum factor of safety of 1.2 for overall highwall slope has been accepted by the industry and was followed for the stability assessment.

In general, for the highwall stability under static loading, a target minimum factor of safety of 1.2 has been targeted. A minimum factor of safety of 1.0 was adopted for pseudo-static seismic analyses.

Using the stratigraphic profiles developed from the boreholes located on the proposed highwall (GT11-01-CH, GT11-03, CPM10-47 and CPM10-48), four slope stability models were developed using the highwall parameters shown in Table B-15. Following the review of the 3D geological model developed for the proposed mine, the bedding dips in northerly direction into the highwall slope as follows:

- CPM10-047 – dips ranged from 8.2° to 7.2° with an average true dip of 7.6°. The lowest apparent dip was 7.3°.

- CPM10-048 – dips ranged from 7.2° to 6.6° with an average true dip of 7°. The lowest apparent dip was 6.5°.

- GT11-01-CH - dips ranging from 10° to 4° with an average true dip of 6°. The lowest apparent dip was 3°.

- GT11-03-CH - dips ranged from 8° to 6° with an average true dip of 6.5°. The lowest apparent dip was 5.7°.

The design profile for the highwall is based on the pit design parameters shown below.

Table B-15 Summary of Pit Design Parameters

Description	Value
Overburden slope angle	2H:1V
Offset distance from top of highwall to toe of overburden slope (m)	7.5
Bench Heights (m)	30
Bench Width (m)	16
Bench slope angles (°)	65
Average overall pit slope angle (°)	45
Dip angle of all strata into highwall (°)	6

The proposed highwall comprises eight to nine 30 m high benches and includes the excavation of the lowest coal seam. As a strip is mined, the existing groundwater table in the highwall is drawn down by seepage into the pit. Seepage occurs through joints and fractures in the rock and coal, some of which occur naturally while others develop from stress relief due to excavation and subsequent expansion of the highwall toward the pit. The effect of the groundwater drawdown will increase the stability of the highwall by lowering pore pressures behind the excavated face and increasing the effective shear strength. The phreatic surface used in the stability models is derived from the groundwater assessment, described in Section 1.6.3, with additional natural dewatering near the face because described above. For each cross-section, the highwall stability model uses the surface at which the coal mine has reached its maximum depth.

B.6.4.2 Design Earthquake

The highwall slope has been evaluated for an earthquake with an annual exceedance probability of 1:5,000 return period. This equates to a peak ground acceleration (PGA) of 0.18g for this location. The horizontal seismic coefficient used in the pseudo-static slope stability analysis was 2/3 of the PGA, or 0.12g. This is a conservative assumption based on the consequence of failure and seismic coefficient based on maximum historical earthquake magnitude. In general, using half of the PGA (0.06 is recommended for the analysis based on the highest magnitude of earthquake less than 5).

B.6.4.3 Results of Global Stability Analyses

The following tables present the calculated global slope stability results using the estimated rock mass shear strength parameters shown in Table B-16 and Table B-17 completed by KCB, EBA and Thiess Pty. The difference in the stability cases is the inclusion of the low strength bentonite layers in the coal seam. Example cross-sections were included for the conditions summarized in Table B-16 (see Figure B-17) and Table B-17 (see Figure B-18) in Coalspur (2014); however, example cross-sections were not included in the same report for the results summarized in Table B-18 and Table B-19.

Table B-16 Slope Stability Results for CPM10-47 Section

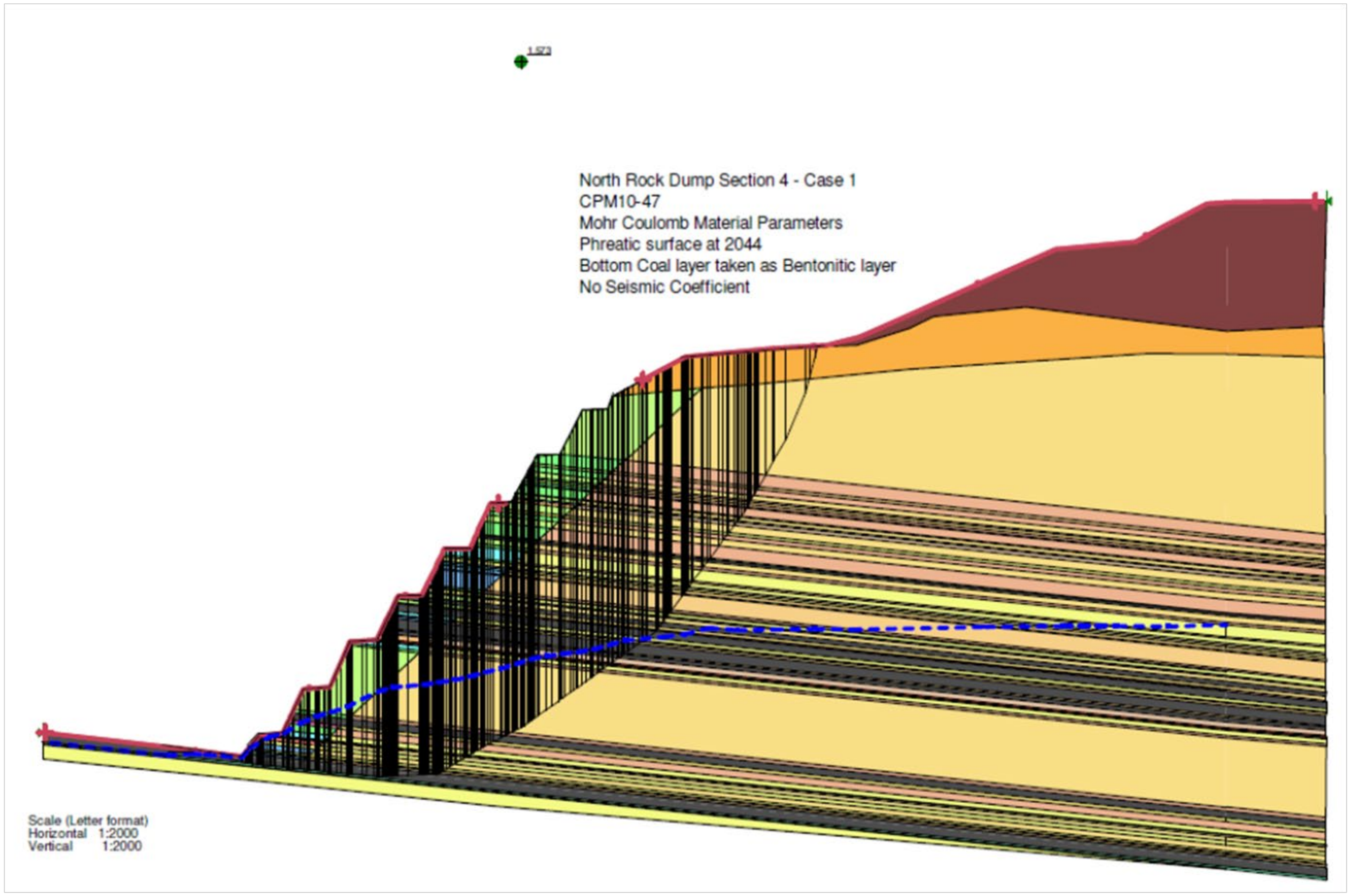
Case No	Seismic Coefficient (g)	Description	Factor of Safety
1	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite.	1.35
2	0.12		1.16
3	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite. Lowest coal seam assumed to have Bentonite shear strength parameters.	1.30
4	0.12		1.11
5 (1)	NA		3.96
6 (1)	NA		4.66
7 (1)	NA		4.05
8	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite. Lowest coal seam assumed to have Bentonite shear strength parameters. (Slip surfaces forced through bottom bentonite layer).	1.24
9	0.12		1.05

(1) Bench global slope stability assessments

Table B-17 Slope Stability Results for CPM10-48 Section

Case No	Seismic Coefficient (g)	Description	Factor of Safety
1	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite.	1.68
2	0.12		1.34
3	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite. Lowest coal seam assumed to have Bentonite shear strength parameters.	1.84
4	0.12		1.30
5	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite. Lowest coal seam assumed to have Bentonite shear strength parameters. (Slip surfaces forced through bottom bentonite layer)	1.63
6	0.12		1.29

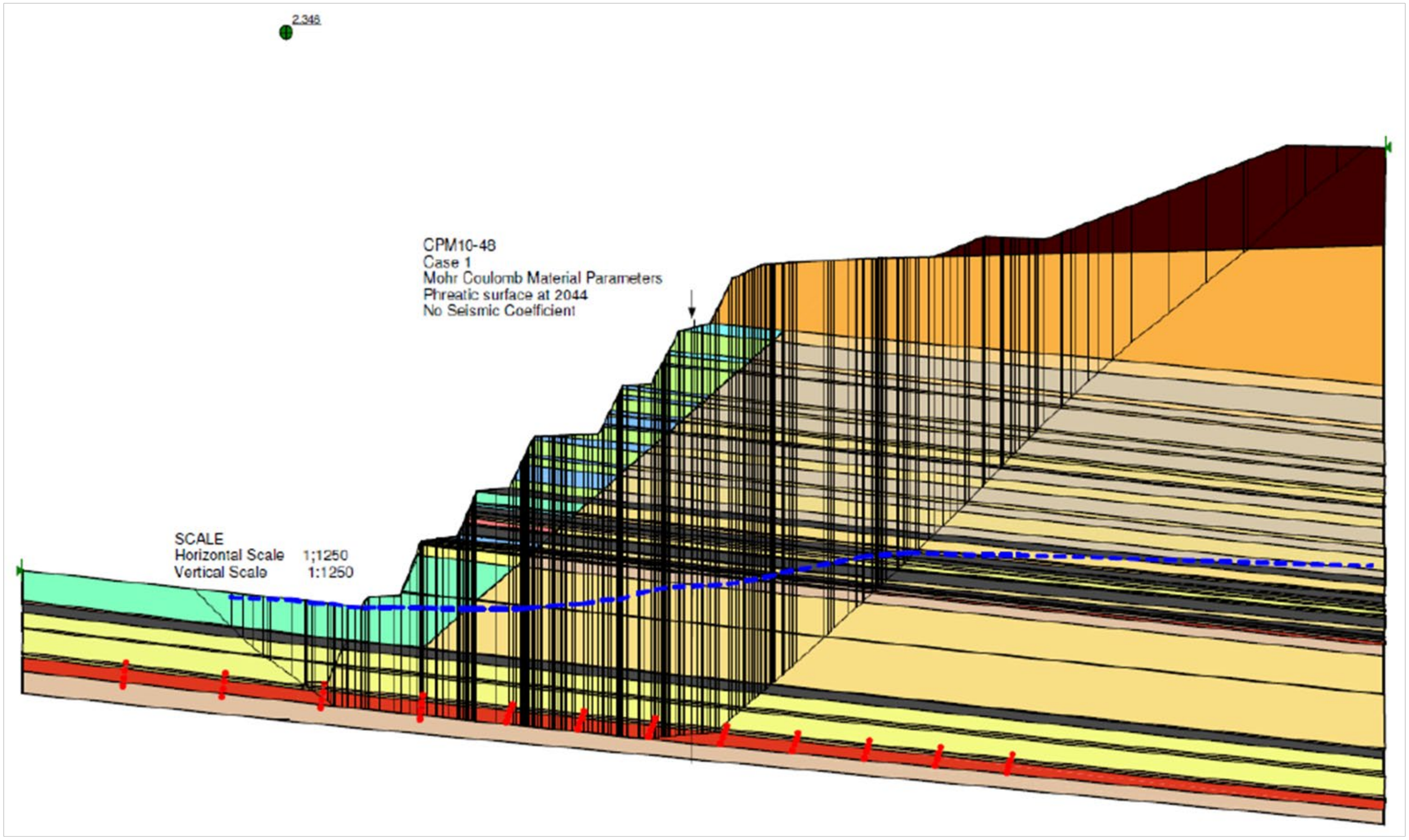
Figure B-17 Example Slope Stability Output for CPM10-47



BARR

Coalspur Phase 2 EIA
Example slope stability output for CPM10-47
(Ref: Coalspur, 2014 as developed by KCB)

Figure B-18 Example Slope Stability Output for CPM10-48



Coalspur Phase 2 EIA
Example slope stability output for CPM10-48
(Ref: Coalspur, 2014 as developed by KCB)

Table B-18 Slope Stability Results for GT11-01-CH Section

Case No	Seismic Coefficient (g)	Description	Factor of Safety
1	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite.	2.15
2	0.12		1.56
3	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite. Lowest coal seam assumed to have Bentonite shear strength parameters.	2.15
4	0.12		1.55
5	NA		6.08

Table B-19 Slope Stability Results for GT-03-CH Section

Case No	Seismic Coefficient (g)	Description	Factor of Safety
1	NA	Rock Mass Shear Strength Parameters for Sandstone, Siltstone, Mudstone, Coal, and Bentonite. Lowest coal seam assumed to have Bentonite shear strength parameters.	1.60
2	NA		1.49
3	NA		4.86

The slope stability analyses undertaken on the simplified stratigraphic cross sections at the core hole locations analyzed demonstrate that the pit slope highwall can be excavated as per the proposed design with an acceptable factor of safety.

B.6.4.4 Local Bench Stability

The main modes of bench failure were identified as bi-planar shear and wedge failure. The main joint set is a steep 67-degree southwest dipping structure. Another potential joint set is a northeast-southwest trending vertical set and is perpendicular to bedding. This set could provide the lateral release surfaces required for a planar shear or wedge failure mode to develop.

The proposed bench face angle is 65° which is about equal to the southwest dipping joint set and so parallel with the bench face angle. In forming a biplanar failure surface, a steep backplane and shallow basal surface are required. The steep joint set is almost parallel to bench face and therefore does not daylight out of the bench face. Therefore, biplanar failure mode is only possible if a weak basal failure surface, *i.e.,* bentonite layer is present within the stratigraphic sequence. In such a case, bench scale or multi-bench failure modes are potentially possible.

Results of the analysis performed previously indicated that because of orientations of the main joint set and orthogonal release surfaces, the requirement of a large scale (multi-bench) wedge failure does not result in a critical highwall instability. Assessment of individual wedges formed by pairing the joints showed that pit wall would be stable if joints have a minimum apparent cohesion of 1 kPa and a friction angle of about 30°.

Considering majority of bedrock above and between coal seam sequences is of a competent sandstone rock mass type, the overall stability of highwall can be maintained. The analysis assumed continuous joints with no intact rock bridges present along the joint surface. It should be noted that joint surface conditions (roughness and large-scale waviness) were not considered while developing discontinuity strength properties.

It is possible that small wedges may form within the pit wall benches, however, the performance indicator of Phase I has not shown such instability so far. Having said that, continuing current monitoring practice and scaling bench faces where required should be carried out to provide a safe work environment at the bottom of the pit.

B.6.4.5 Footwall Stability

The stability of the footwall is governed by bedding orientation, material shear strength properties, and groundwater conditions. The presence of the weak bentonite layers interbedded within the coal seam sequences, as with the highwall, is of significance for stability of the footwall and should be removed or mitigated as much as practically possible. Groundwater control is critical in stabilizing footwall slopes, and this is normally provided by pumping from in-pit sumps.

The main modes of footwall instability include shear and buckling. Due to the shallow dip angle of the bedding planes, a potential failure is expected to be in the form of sliding and shear towards the high wall toe. Manalta (1981) assessed the footwall stability against buckling and estimated the height of the footwall before failure can be initiated in the footwall. Manalta concluded that for the gentle footwall slopes at Coalspur, any failure due to buckling will be in the form of gradual sliding towards the toe resulting in operational issues rather than a safety concern.

Groundwater control is critical in stabilizing footwall slopes. In the presence of high pore pressure in the footwall, the factor of safety against potential failure modes could significantly decrease. Footwall dewatering is required to minimize the potential footwall instabilities. This is normally controlled by pumping from in-pit sumps.

Based on the current mine design, the footwall area is stable with the following assumptions:

- The pit floor slopes at a maximum of 14° into the highwall.
- All bentonitic material is removed from the pit floor prior to spoil placement (*i.e.,* no weak materials exist on the pit floor at the end of mining).
- The spoil material has a friction angle of 30° with a cohesion value of 10 kPa.
- A small degree of groundwater recharge has occurred in the spoil pile.

The analyses indicated that the calculated factor of safety was about 1.3 to 1.5 for this condition.

B.6.5 Highwall Mining

Coalspur has an existing Highwall Mining (HWM) plan that is approved by the Alberta Energy Regulator (AER) (Coalspur, 2018). The focus of Barr's most recent study (Barr, July 2020) was to understand the design basis of the highwall mining configuration for long-term pillar stability. Previous studies, such as

Agapito Associates' (Agapito, 2019) report dated September 6, 2019, provide a basis for HWM operational evaluations and Coalspur's decision to proceed with HWM operations at Coalspur Mine Operations.

Specifically, Barr's study reviewed the pillar stability and provides recommendations for pillar geometry considering anticipated overburden loads, desire to minimize long-term settlement, and current knowledge of material properties. The study included a review of the previously developed and approved (Coalspur, 2018) HWM design and completed assessments. Barr also reviewed available geology information from coreholes that intercepted the McPherson coal seam and floor material. Previous laboratory test data from the McPherson coal and floor material were assessed and new laboratory tests completed in support of the current HWM design study. The geological data was evaluated with respect to projected topography, HWM equipment geometries, and HWM depths provided by Coalspur.

Given the spatial availability of data, observed weaker material strengths for the McPherson coal seam and floor material, and requirement to mitigate future settlement, Barr recommended increasing minimum web and barrier pillar design stability factor to 2.3. This value may be optimized in Phase II as additional observation and material characteristic information is obtained for the McPherson coal seam and floor material with more core drilling and laboratory testing.

Based on Barr's observations and engineering judgement, the maximum depths and widths of potential web pillar widths for long-term stability are:

- 13.5 m for HWM penetration distance of 427 m with barrier pillar width of 36 m
- 12.5 m for HWM penetration distance of 366 m with barrier pillar width of 34 m
- 11.7 m for HWM penetration distance of 305 m with barrier pillar width of 32 m

Due to expected lower cover during active mining, web pillar stability factors during mining will exceed 2.7. Such high pillar stability factors should provide for adequate rib stability during highwall mining operations. Pillar stability factor of safety during mining near the highwall will be on the order of 10, due to anticipated reduced overburden stresses, in all optimized HWM designs.

B.6.6 Dump Stability

B.6.6.1 Overview

The waste rock material generated from the excavation of the mine will be placed as Phase I backfill and then backfill for Phase II. End dumping will result in 35° to 37° slopes which is the angle of repose of the fill and will be reclaimed to 22° (2.5H:1V slope).

In addition, the strength of the foundation soil should be considered in designing the maximum height of the subsequent lifts. Foundation soil failure could occur because of:

- Excessive high loading in one lift.
- Rapid loading between subsequent lifts without allowing sufficient time for excess pore pressure to dissipate.

- Lack of proper drainage of the fill material resulting in excess pore pressure build up in the foundation soil.
- Formation of freeze-thaw interface because of thawing of frozen ground on which waste rock has been dumped over wintertime.

Previous design studies suggested a maximum vertical height for any lift of 25 m and maximum total dump height of 120 m from crest of dump to toe (at soil foundation). Also, a minimum height of 15 m has been considered to ensure segregation of the basal coarse materials from the fill as a drainage blanket at the base of the dump.

Proper drainage and sufficient time between subsequent fills (minimum 6 months) must be implemented to allow dissipation of excess pore pressure from the fill and to ensure stability of the waste dump. No poor-quality waste rock or bentonite or bentonitic rock materials should be placed as the drainage blanket material at the base of the dump.

B.6.6.2 Design Criteria

A target minimum factor of safety of 1.2 has been used in this slope stability assessment. A target minimum factor of safety of 1.0 was adopted for pseudo-static seismic analyses.

B.6.6.3 North Refuse Dump

During construction of the north dump and prior to placement of material in the fines settling pond, the stability of the east side of the north dump was assessed; assuming groundwater at the original ground level (OGL). Following placement of the material in the fines tailings cells, the stability of the east side of the north refuse dump was assessed with groundwater at the original ground surface and at the highest elevation (HWL) in the fine tailings' cells. The design parameters are shown in Table B-20 and the results of the latest analyses including the additional of Phase II refuse material are shown in Table B-21. As shown, the calculated factors of safety are acceptable.

Table B-20 Summary of North Rock Dump Design Parameters

Description	Value
North refuse dump lower slope angle	2.5H:1V
North refuse dump higher slope angle	2.5H:1V
Offset distance from top of overburden slope for highwall to toe of north rock dump (m)	100

Table B-21 North Rock Dump Stability Analysis Results

Scenario	Design FOS	Calculated FOS
1.1 - ESSA - A to A' - Design Height	1.2	1.29
1.2 - ESSA - A to A' - Design Height - Seismic	1.0	1.06
1.3 - ESSA - A' to A - Design Height	1.2	1.57
1.4 - ESSA - A' to A - Design Height - Seismic	1.0	1.27
1.5 - USSA - A to A' - Design Height	1.2	1.22
1.6 - USSA - A to A' - Design Height - Seismic	1.0	1.03
1.7 - USSA - A' to A - Design Height	1.2	1.50
1.8 - USSA - A' to A - Design Height - Seismic	1.0	1.23

Currently, the North Dump has an overall and bench slope of 14°-16°. The stability of North dump was reassessed in 2021 (Barr, Jan 2021). Because of the planned Phase II mining activities a redesign of the North Dump was necessary to accommodate the Phase II refuse material, the stability of the larger North Dump was assessed in 2024 (Appendix B3). Based on the revised geometry, the coarse refuse pile will have a maximum height of approximately 90 meters and comprising a total of five benches. The North Dump will feature inter-bench side slopes of 2.5H:1V instead of the previous 3H:1V, with an overall slope of 2.8H:1V. Each bench will have a height of 15 meters and a width of 6 meters. The geotechnical properties and pore pressure profile previously defined (Barr, 2021) were utilized in the analysis.

The pore water pressure profile is conservative, as results from instrumentation monitoring indicate a significantly lower phreatic surface within the pile due to the presence of a substantial proportion of coarse particles (sand-sized and larger). Nevertheless, the results from the current monitoring system have been considered in defining the phreatic surface within the pile. Table B-21 presents the results of the updated stability analysis, wherein all factor of safety values exceed the minimum requirements. It should be noted that, based on ongoing refuse material testing, the strength parameters are significantly higher than the design values, indicating that the actual factor of safety will be higher than the values reported in Table B-21.

The stability of the north refuse dump has been assessed through instrumentation and monitoring as the north refuse dump progresses on a semi-annual basis at the minimum. Based on the ongoing stability analysis, the factor of safety of the dump slopes has been maintained above the minimum requirement.

B.6.7 Process Fines In-Pit Storage Design

Tailings generated from coal processing will be delivered as slurry to multiple in-pit tailings impoundments where it’s allowed to settle out of suspension and consolidate. For Phase I, the designed and planned McPherson Cells provide sufficient storage for the Vista Mine licensed pit. The same strategy will be used for Vista Phase II development where the tailings will be discharge in the mined pit where it will be separated by native plugs.

The concept includes leaving native plugs of intact material at predefined intervals in the South Pits. This approach will provide multiple excavated cells into which the slurry tailings is deposited. Each native plug performs like an external embankment while tailings are deposited on the upstream side of the plug. This continues to be the case until tailings are subsequently deposited in the next cell which results in the downstream slope being covered as well. At this point, the native plugs become an internal berm having no retaining function. The pond elevation would be maintained a minimum of 2 m below the lowest contact of the till and the bedrock; therefore, the stability of the till unit forming the uppermost layer of the native plugs will not pose any risk to the health and safety of the structure and the performance of the plugs in terms of retaining impounded materials and overtopping. The freeboard for each plug will be equal to the 2 m of till/rock contact plus the till thickness which can range between 10 and 20 m.

The factor of safety requirement for the design of the native plug was based on the Canadian Dam Association (CDA) technical bulletin (Application of Dam Safety Guidelines to Mining Dams, 2019). According to this bulletin, a minimum factor of safety of 1.5 is required for any potential failures resulting in the release of stored materials into the environment. A minimum factor of safety of 1.3 is required when the failure does not include loss of stored material to the environment but does result in structural damage to the dam. A minimum factor of safety of 1.0 is required for the seismic loading using the pseudo-static analysis method, and a factor of safety of 1.1 is required for post-peak strength.

Alberta Dam and Canal Safety Directive (ADCSD) (2018) and CDA (2013) guidelines require an evaluation of the consequence of failure for a particular retaining structure. Based on the determined consequence, a specific set of design criteria can be applied to the impoundment design. The location and geometry of each cell relative to their respective native plugs are key criteria that facilitate the classification. Several categories and factors were considered to determine the consequence classification: loss of life, damage to environmental and cultural resources, and damage to infrastructure, economics, and other property. The primary assumption related to the classification is that any potential uncontrolled release of the tailings will be fully contained within the adjacent pit. Based on the explanation provided above to determine the classification regarding the incremental consequences of failure, in-pit tailings Cells are deemed facilities of “significant” consequence for initial operation. However, as subsequent deposition and capping and reclamation advances the facilities are no longer impoundments.

Coalspur’s mine plan includes multiple fine coal refuse tailings cells in the Phase II South Pits. It is anticipated that the similar design and operation apply to Phase II tailings Cells. All previous geotechnical investigations have provided an understanding of the site geology and the typical material properties within the mine lease boundary. Furthermore, extensive geotechnical modelling has been completed to evaluate various potential slope failure scenarios, examining slightly variable subsurface geology and material strength conditions.

The native plugs design to impound tailings within the in-pit McPherson Pit tailings cells have been designed within the geometry ranges provided below and similar design principles will be applied to Phase II tailings cells.

- An upstream slope angle of 50°– 60°
- A downstream slope angle of 45°- 35°
- A typical crest width within the range of 7 – 9 m

- An appropriately sized toe buttress on the downstream toe to mitigate the potential for coal seam failure and uncontrolled seepage

B.6.8 Instrumentation and Operation

Pits and dumps slopes monitoring is required throughout the life of the mine. Phase I uses a combination of drone and radar slope monitoring for the dumps and pit slopes. The same monitoring approach is recommended for Phase II. The native plugs deformation is monitored using in place inclinometer (IPI) and it has proven efficient for the application. Phase II native plugs deformation will be monitored using IPI as well. For groundwater level monitoring, series of piezometers (standpipe or vibrating wire) will be installed at different locations around the pit and within the waste dumps. Nested vibrating wire (VW) piezometers will be coupled with IPI to monitor phreatic surface within the native plugs.

Like Phase I, Safety Management Plan including Operation, Surveillance and Monitoring Manual (OMS), Emergency response and preparedness plans will be developed for each structure based on the criticality of each. Regular safety inspection will be completed for the critical structures to complement the instrumentation data.

The information and analyses discussed in this report include preliminary study completed for Phase II and updated detailed analyses completed while Phase I mining has been ongoing. Additional geotechnical investigation, laboratory and in-situ testing to better characterize the materials as the mining progresses during Phase II should be performed like Phase I. If the phreatic surface and pore pressure control is found to be insufficient to provide a stable highwall and footwall, depressurization methods like horizontal drainage system or vertical pressure relief wells should be considered.

B.6.9 Refinements to the Phase I Val d'Or Pit

In addition to the refinements of the Phase I area in relation to the Phase II Project, Coalspur is proposing some refinements to the currently approved Phase I Mine Plan. These refinements are further described in Section 5.6 of Part C – Project Description. The proposed changes to the mine plan will economically extract additional volumes of resources available therefore improve resource recovery for the Phase I current operation. As part of the revised Phase I Mine plan, the Val d'Or pit will shift a portion of the final highwall and highwall mining further north. Highwall mining will be conducted similarly as approved however due to the geology of the new location, the coal model indicates a slightly thicker coal seam within the new area, slightly increasing the highwall mining reserves. A geotechnical assessment was completed on the new highwall mining location to ensure safety ratings are appropriate. The geotechnical assessment is provided in Appendix B4.

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