14th October 2021

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Hawsons Iron
(by email)

Updated Resource Estimates for the Hawsons Magnetite Project, Western NSW

H&S Consultants Pty Ltd ("H&SC") completed updated Mineral Resource estimates ("MRE") at a 10% DTR cut off for Hawsons Iron's ("HIO" formerly Carpentaria Exploration ("CAP")) namesake Hawsons Magnetite Project in western New South Wales in March 2017. Based on subsequent work completed by CAP for its then PFS, 9.5% DTR was identified as a suitable cut off grade for the resource and new estimates were reported in June 2017. Recent pit optimisation studies by independent consultants KPS have now identified that 6% DTR represents a suitable cut off grade. As a result of this work the MRE are now re-reported for that cut off grade.

The new Mineral Resources are reported from the June 2017 model for a 6% DTR cut off grade, with no constraints for oxidation level. There has been no new drilling since that date.

Category	Mt	DTR %	DTR Concentrate Mt	Density t/m^3	Fe Head %
Indicated	960	13.7	132	3.03	17.3
Inferred	2,100	12.9	268	3.02	16.6
Total	**3,060**	**13.1**	**400**	**3.02**	**16.8**

Concentrate Grades

Category	Fe %	SiO_2 %	Al_2O_3 %	S %	P %	LOI %
Indicated	69.9	2.6	0.19	0.002	0.003	-3.0
Inferred	69.7	2.8	0.20	0.003	0.004	-3.1
Total	**69.8**	**2.8**	**0.20**	**0.003**	**0.004**	**-3.0**

The estimates have been reported using the 2012 JORC Code and Guidelines and the author has the requisite experience to act as a Competent Person under the code.

In addition an Exploration Target has be identified based on a nominal 150m down dip and across strike extrapolation of the existing drilling results and is immediately peripheral to the MRE.

Exploration Target:

1,200Mt to 1,800Mt at 12.5 to 13.5% DTR for 150 to 250Mt of DTR concentrate (6% DTR cut off)

The potential quantity and grade of the Exploration Target is conceptual in nature, that there has been insufficient exploration to estimate a Mineral Resource and that it is uncertain if further exploration will result in the estimation of a Mineral Resource.

More details are supplied in Appendix 1, which comprises extracts of the original MRE report published in June 2017.

Simon Tear
Director and Consulting Geologist
H&S Consultants Pty Ltd

The data in this report that relates to Exploration Results, Mineral Resource Estimates and Exploration Targets is based on information evaluated by Mr Simon Tear who is a Member of The Australasian Institute of Mining and Metallurgy (MAusIMM). Mr Tear has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the "JORC Code"). Mr Tear is a Director of H&S Consultants Pty Ltd and he consents to the inclusion in the report of the Mineral Resources in the form and context in which they appear.

Appendix 1

Figure 1 shows the location of the Hawsons Iron Ore Project.

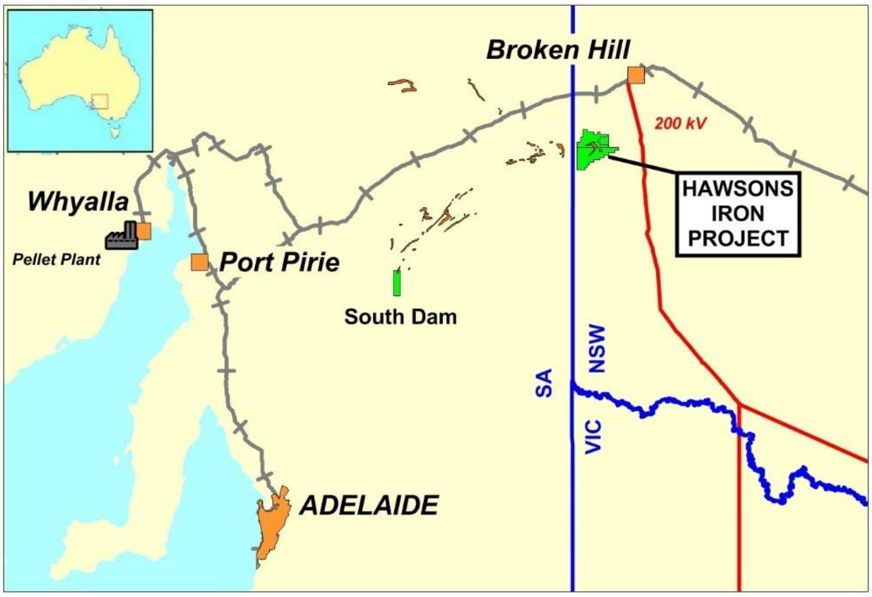

Figure 1 Location Map

Figure 2 is a polished section of the mineralisation showing the inclusion-free idioblastic nature of the magnetite grains within the siliciclastic host sediment that has been subjected to lower greenschist metmorphism.

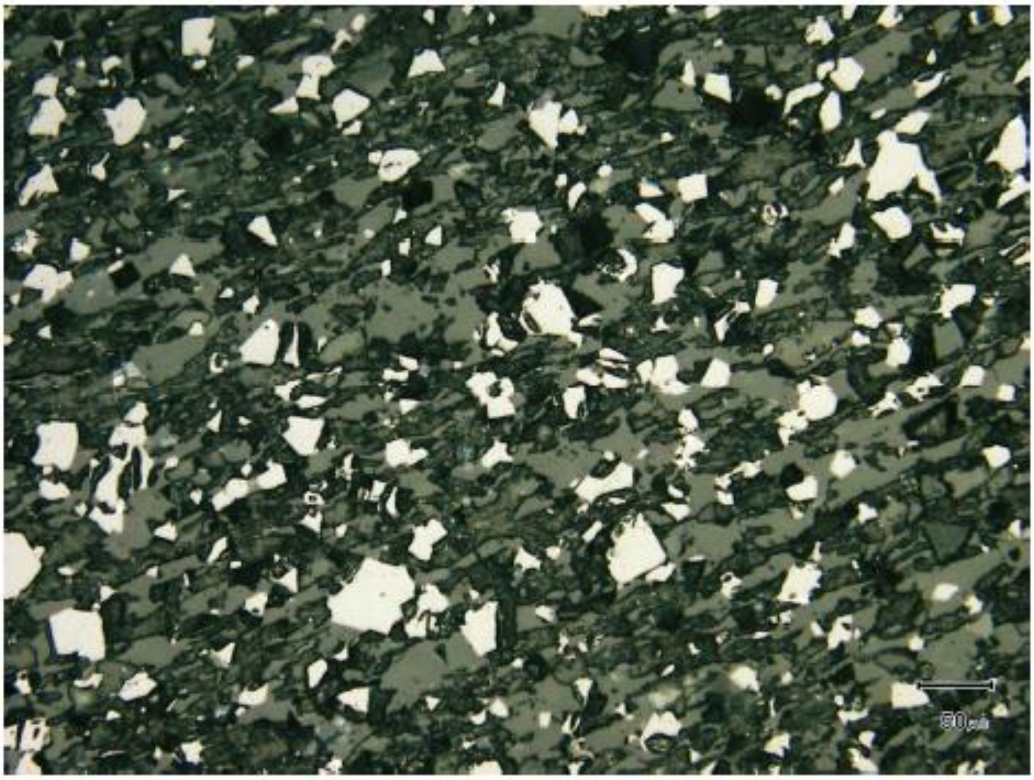

Figure 2 Polished Section Micrograph of Magnetite Mineralisation for the Hawsons Deposit

The resource estimates were produced from 73 drillholes for 21,429.5m, predominantly surface RC holes and a lessor amount of diamond drillholes (mixed HQ and NQ core sizes). Drillhole spacing ranges between 150m and 300m in both section and plan (Figure 3). RC drilling encountered predominantly dry samples; some samples were slightly damp but there were no reports of any groundwater inflows (<5% wet samples). Table 1 provides information of the drilling used in the resource estimation.

Table 1 Drillhole Information

Company	Year	Hole Type	No of Holes	Metres	DTR Analysis	DH Geophys
CRAE	1986	Perc	4	634.6	No	2 holes
	1988	DD	1	100.0	No	None
HIO	2009	RC	3	761.1	Yes	99% of drilling
	2010	DD	3	761.3	Yes	65% of drilling
	2010	RC	42	10,141.0	Yes	68% of drilling
	2010	DD Tails	17	3,068.5	Yes	40% of drilling
	2016	RC	20	5,963.0	Yes	88% of drilling
		Total	**73**	**21,429.5**		

A plan of the drillholes in national grid GDA94 Zone 54 projection is included as Figure 3.

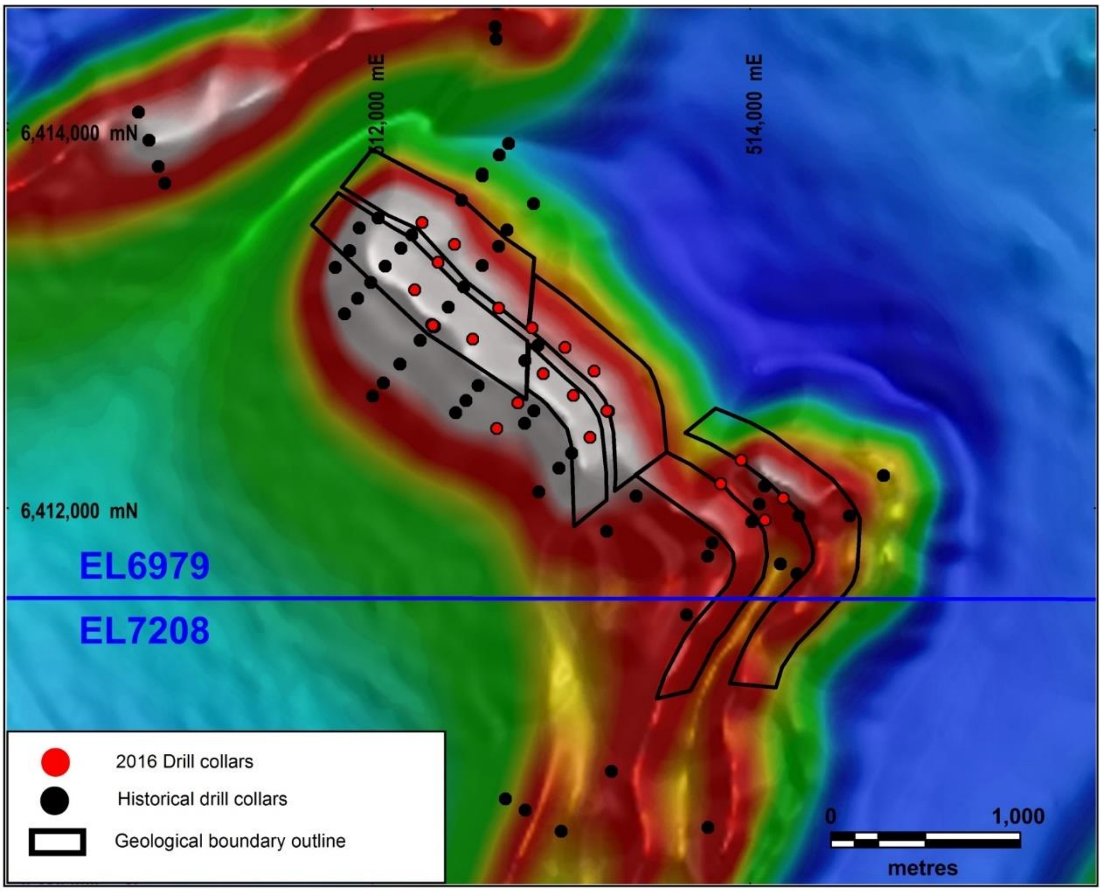

Figure 3 Core & Fold Targets Geology & Drillhole Location Map

(extent of resource over reduced to the pole magnetics)

From drilling intersections the magnetite mineralisation is interpreted to extend to a vertical depth of 400m below surface over a 4km strike length. A schematic cross section interpretation of the drilling from an earlier report is included as Figure 4. It shows the two substantial bodies of magnetite mineralisation (Units 2 and 3) with an interstitial lower grade zone known as the Interbed Unit. It is HIO's intention to mine the complete package of magnetite material, interstitial zone included. The magnetite mineralisation is considered open at depth. Additional low grade mineralisation occurs in the hanging wall which will be mined as part of any pit development.

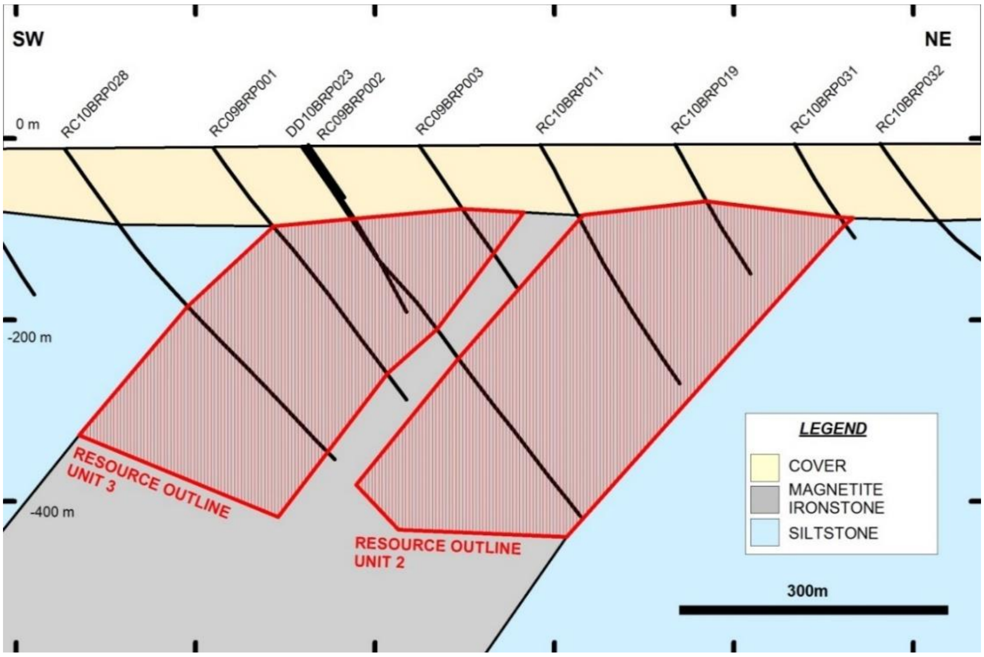

Figure 4 Core Target Schematic Cross Section

Additional interpretation has delineated several stratigraphic units generally based on their magnetite content within the overall magnetic package (Figure 5). As a result there is a now series of units with variable average magnetite grade, which are listed below from footwall to hangingwall. These units have been subdivided into 3 modelling/structural domains, Core West, Core East & Fold. Unit 1 is a narrow band of variably magnetic siltstone including some high grade material, in the footwall to the main two magnetic units.

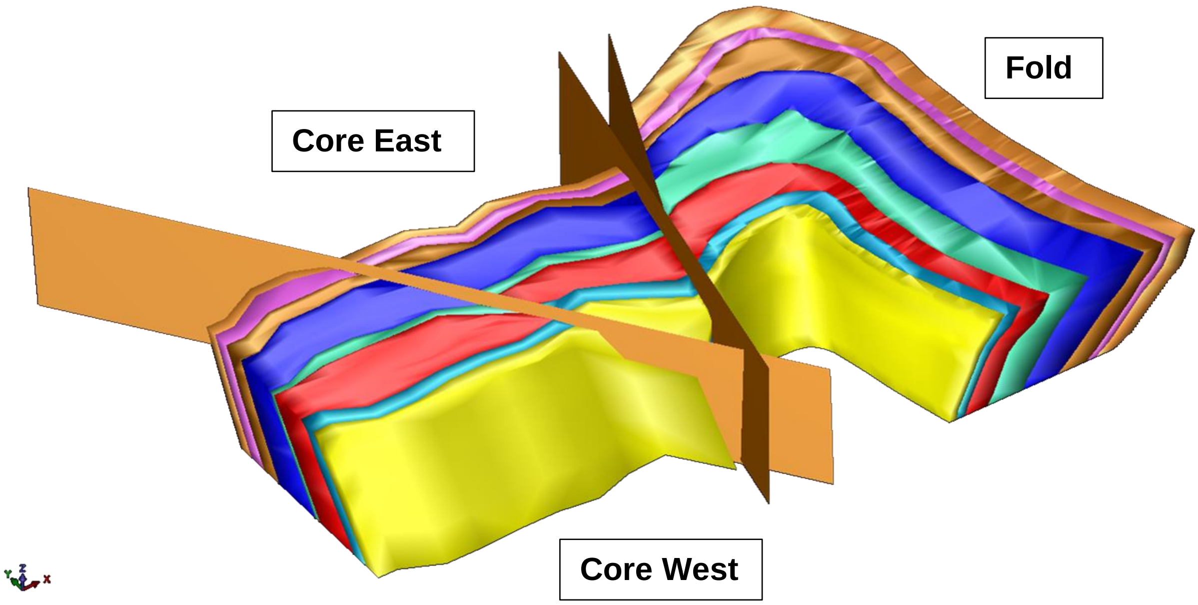

Figure 5 Core & Fold Target Areas 3D Geology

(view : looking down & to grid NE; pale brown = FW unit, purple = Unit 1, brown = interbed 1, blue = Unit 2, green = Interbed unit, red = Unit 3, cyan = Upper HW unit 1, yellow = upper HW unit 2; brown planes = fault surfaces)

In order to provide greater geological control to mineralisation a chronostratigraphic interpretation was undertaken using the gamma logging from the downhole geophysics. This established stratigraphic markers that were used to ascertain magnetite grade continuity between the drillholes and which resulted in very low coefficients of variation within the different units. This information was used subsequently to assist with the resource classification.

Unconstrained 5m downhole composites were generated from the drillhole database for the downhole mag_sus, short spaced density, hand held magnetic susceptibility, DTR analysis and concentrate grades (iron, alumina, phosphorous, sulphur, silica, titanium and loss on ignition). Where there were no DTR values the downhole mag_sus or hand-held mag sus data was used, via a regression equation, to populate the peripheral low grade and barren areas to the main magnetite mineralisation with DTR values.

A total of 3,924 5m DTR composites were generated with 2,732 in the fresh rock zone and 1,161 in the transition zone of which 209 were from direct DTR measurement. 74 of the fresh rock composites were generated from the downhole mag_sus data with 55 from the hand-held mag_sus data.

Figure 6 shows a plan (in local grid) of the DTR composites.

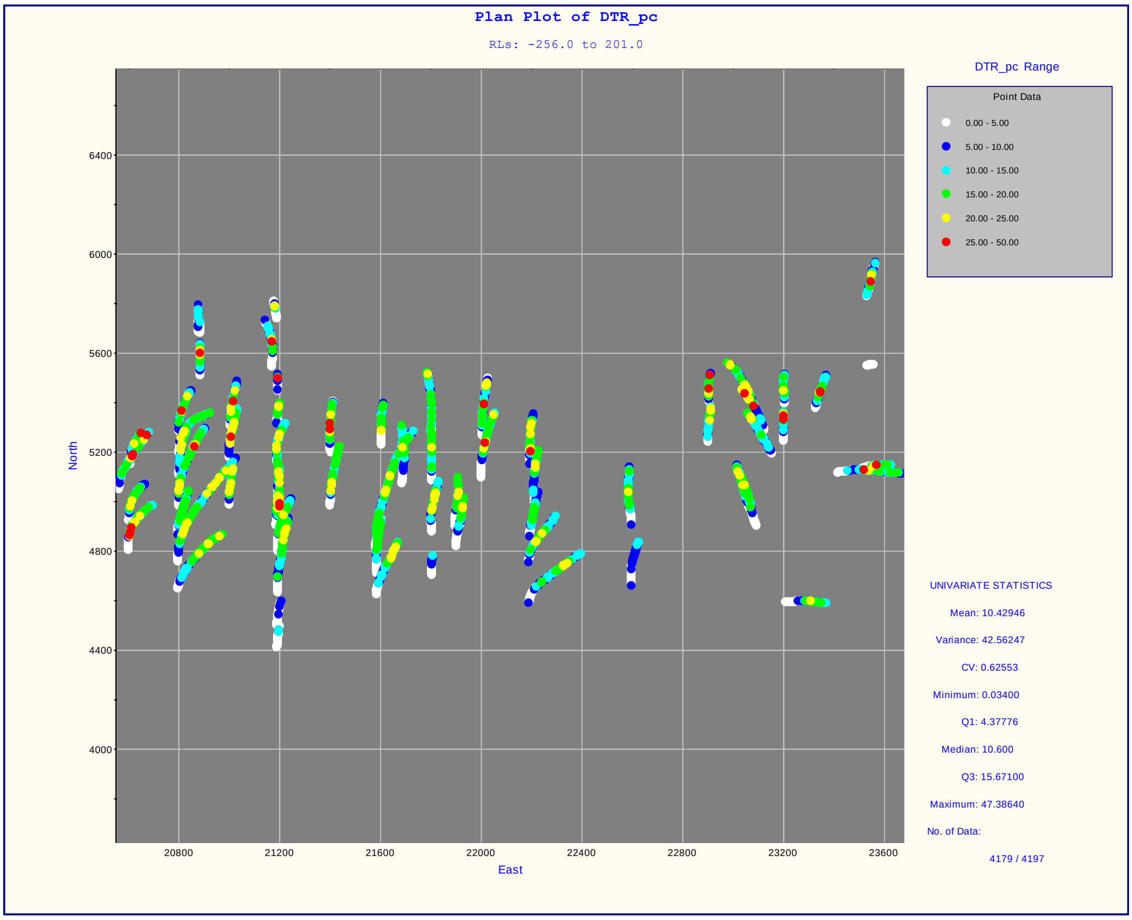

Figure 6 DTR Composites Plan View

The coefficients of variation for the DTR grades and the concentrate products were relatively low (0.1 to <1) and so allow for Ordinary Kriging ("OK") with dynamic interpolation as a valid modelling method.

No top cut was applied to the data.
Variogram models were produced for all estimated parameters from data below the top of fresh rock surface from a relatively consistent part of the Core West structural domain. These variogram

models showed longest ranges in the strike orientation, moderately long ranges in the down dip orientation and short ranges in the orientation perpendicular to strike and dip ie downhole.

The OK modelling used a 4 pass search strategy with the composites. A Pass 5 search was used to provide information on the Exploration Potential. Details of the search parameters are included in Table 2.

Table 2 Search Ellipse Parameters

Axis	Pass 1	Pass 2	Pass 3	Pass 4	Pass 5
Along Strike	250m	300m	450m	450m	900m
Down Dip	150m	150m	225m	225m	450m
Across Strike	40m	50m	75m	75m	75m
Composite Data Requirements					
Min Data	16	16	16	8	8
Max points per sector	8	8	8	16	16
Sectors	4	4	4	2	2
Hole Count	3	2	2	1	1

Contact plot analysis of the estimated elements were conducted in order to investigate how the Base of partial oxidation ("BOPO") and the top of fresh rock ("TOFR") surfaces should be treated in resource estimation. The TOFR surface was found to coincide with a marked difference in density and DTR and was therefore used as a hard boundary. The structural domain surfaces were used as hard boundaries, but the lithological subdivisions were used as soft boundaries.

Block dimensions are 100m by 50m by 15m (X, Y & Z directions).

The classification of the resource estimates is based primarily on the data distribution which is a function of the drillhole spacing. Other factors involved in the classification include the style of mineralisation, the geological model, the QAQC programme and results and comparison with previous resource estimates. HIO has informed H&SC that the mining method will be a bulk mining method via an open pit operation and the resources have been classified according to this assumption. The allocation of Indicated and Inferred in the block model is detailed in Table 3 and shown in Figure 7.

Table 3 Resource Classification

Search Pass	Classification	Map Colour
1	Indicated	Red
2	Indicated	Red
3	Inferred	Green
4	Inferred	Green
5	Exploration Potential	Blue

A review of the Indicated Resource block distribution resulted in a decision to create a Defined Shape to modify the Indicated Resource block categorisation i.e. remove the 'spotted dog' effect. Pass 2 blocks outside this shape reverted to Inferred Resource, as they were mainly individual blocks or small numbers of blocks in relative isolation.

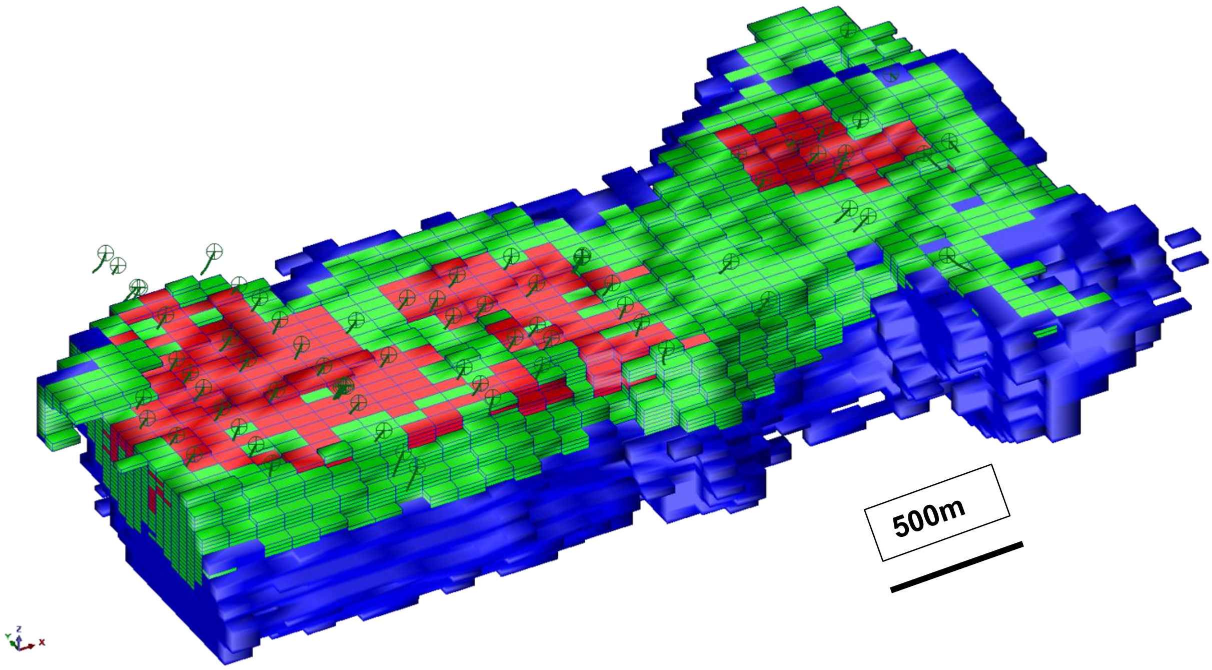

Figure 7 Resource Classification

(view looking down to grid north east)(green circles & lines = drillhole traces)

Figure 8 shows the Indicated resources at a 6% DTR cut off, irrespective of oxidation level.

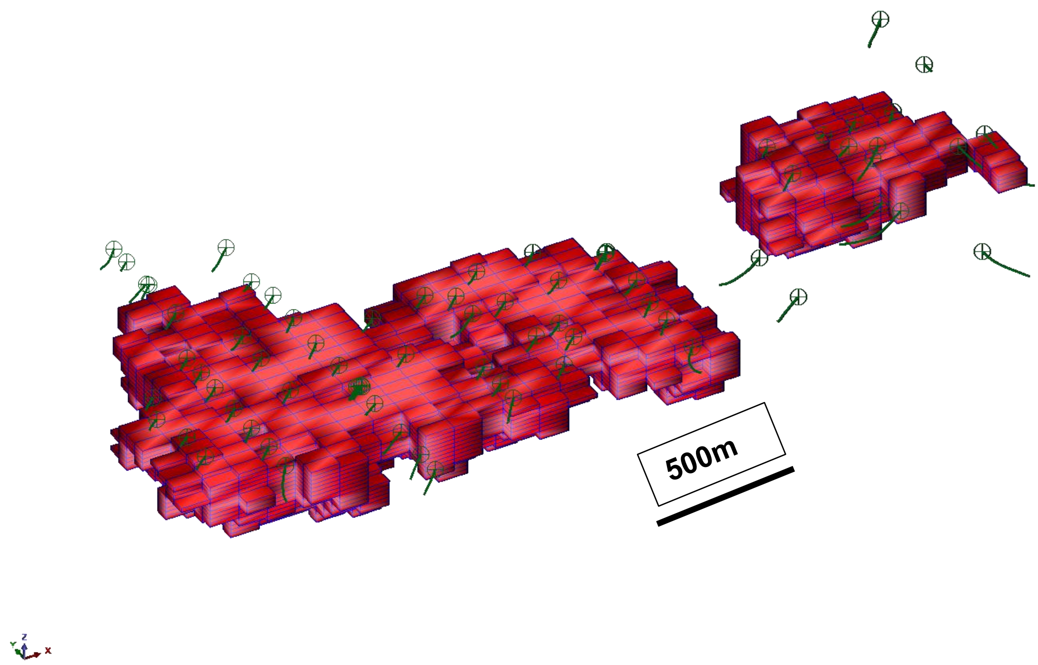

Figure 8 Indicated Resources

(view looking down to grid north east)(green circles & lines = drillhole traces)

An example of the global DTR block grade distribution is included as Figure 9. An example of the MRE DTR block grade distribution is included as Figure 10. An example of the Exploration Target block grade distribution is included as Figure 11. (The undefined term in the legend refers to DTR grades from 0 to 6%).

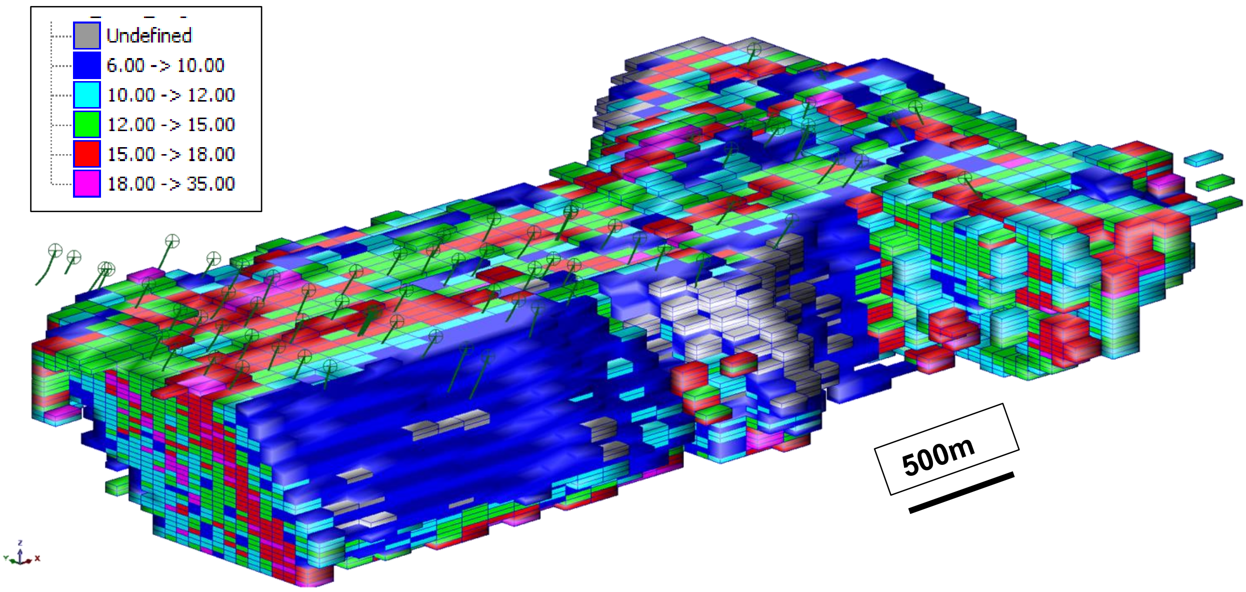

Figure 9 Global DTR Block Grade Distribution

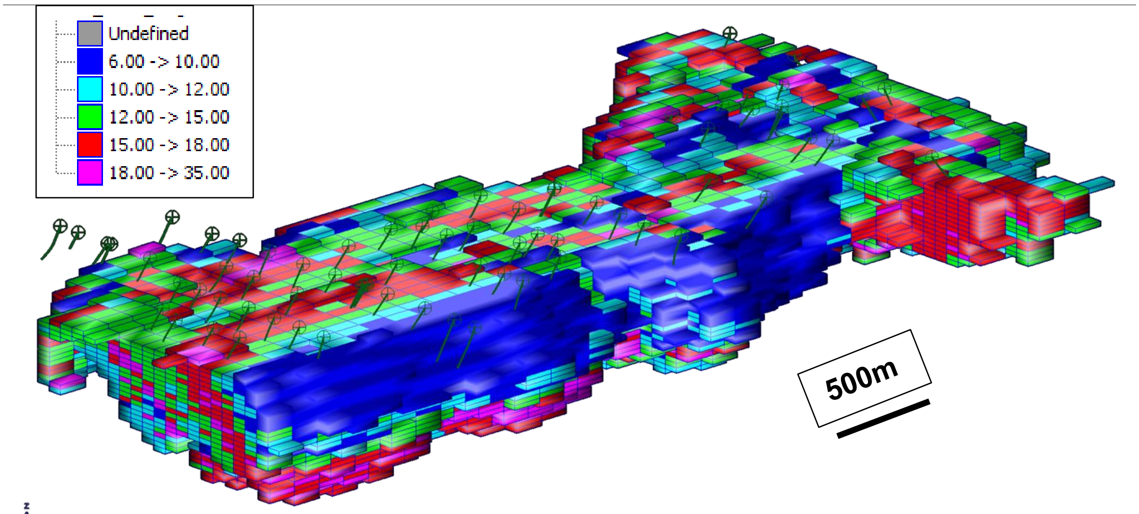

Figure 10 Mineral Resource DTR Block Grade Distribution

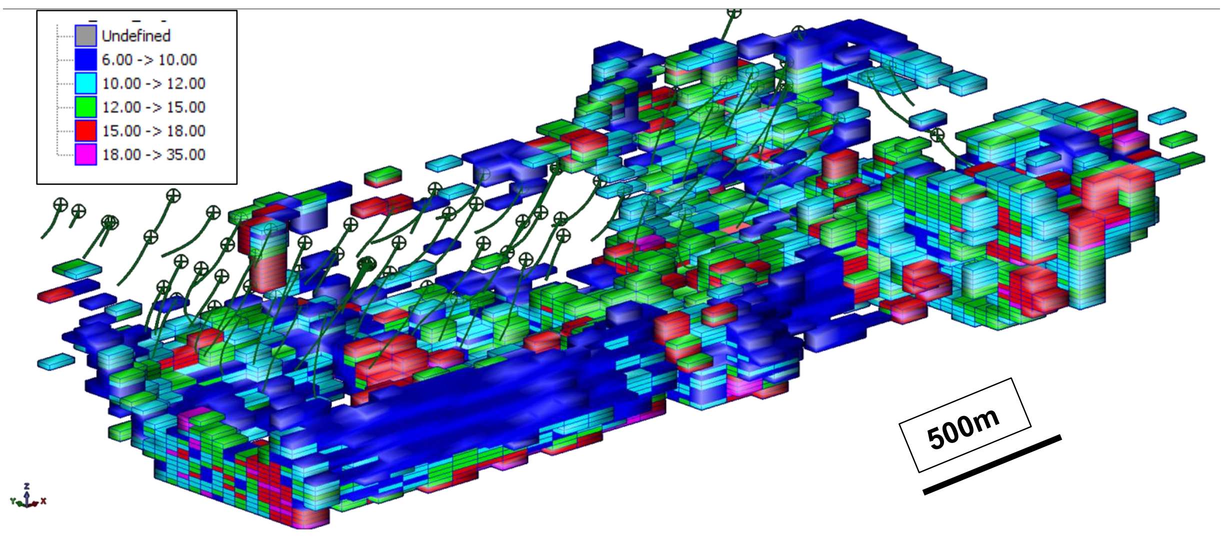

Figure 11 Exploration Target DTR Block Grade Distribution

JORC Code, 2012 Edition – Table 1   Hawsons Magnetite Project

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)(All work was completed by Carpentaria Exploration ("CAP"), predecessor to Hawsons Iron ("HIO")

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases, more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	• Sampling consisted of drillholes with a mixture of reverse circulation (RC) from surface, diamond tails to RC precollars (PD) and diamond from surface (DD). • A total of 73 drillholes for 21,429.5m, were drilled by CAP in two main phases i.e. 2010 (RC & DD) and 2016 (RC). • RC drillholes were drilled to obtain 1m bulk samples with sample compositing (various lengths under geological control) via spear sampling applied in order to obtain manageable sample sizes for laboratory sample prep and assaying • For the 2010 RC drilling, sampling comprised 2m to 10m 3kg composite samples. The 2016 sampling comprised 5m composites generating 6kg of sample. All samples were pulverized to produce 150g aliquot for X-Ray Fluorescence (XRF) and Davis Tube Recovery (DTR) analysis • Diamond core sampling involved sawing half core samples to produce an 8m composite sample (predominantly NQ core) which was pulverized to produce a 150g aliquot for XRF and DTR analysis. • Geophysical logging was completed for a majority of holes and consisted of natural gamma, magnetic susceptibility, density and calliper readings. • Mineralisation comprises bands of variable thickness of disseminated, idioblastic magnetite in low metamorphic grade fine grained siliciclastics and diamictites. Siliciclastic grain size tends to provide a strong control to mineralisation. Substantial regional deformation has occurred but locally the main mineral units are relatively straightforward moderately dipping units. • Consistency of sampling method was maintained. • The sampling technique is considered appropriate for deposit type with

Criteria	JORC Code explanation	Commentary
		all sampling to industry standard practices.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• The RC drilling for 2010 was carried out using a truck mounted Schramm and truck mounted KWL 1600H. Both rigs used 4.5" rods and 5.5" face bits. • PD and DD drilling was carried out using a truck mounted UDR650 using NQ2 and standard HQ diameters. Core orientation used the Ace Core orientation tool. • For the 2016 drilling (all RC drilling) truck-mounted Sandvik DE 840 (UDR1200), UDR1000 and Metzke rigs were used. All rigs used 4.5" rods with 5.5" face bits.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• The 2010 RC sampling was on 1m intervals into green plastic bags. Sample recoveries for RC were visually estimated by the geologist at the time of drilling and recorded. • Because no numerical RC chip recovery data existed it was not possible to conclude if there was a relationship between sample recovery and mineral grade • The 2016 RC drilling recorded sample weights for 272 1m samples with recoveries of 80-90% for dry samples and 40 to 50% for wet samples. Plotting of recoveries versus DTR grade indicated no sampling bias. • Core recoveries were recorded by measuring the length of core recovered in each drill run divided by the drilled length of the individual core runs; average recovery >97%. • A handheld XRF orientation study by CAP for the 2010 RC drilling concluded that there was no sample bias with loss or gain of fine/coarse material with the RC drilling. • A very modest number of wet samples were recorded in the 2010 RC drilling and for the 2016 drilling, <5% of samples were logged as wet. • A study by Keith Hannan of Geochem Pacific Pty Ltd, an independent geochemist/consultant determined, "at the deposit scale, average magnetite recoveries of complete intercepts of ore units 2 and 3, corresponding to 180-245 m lengths of continuous data, are indistinguishable by drill sample type (*i.e.,* RC versus NQ core samples). By implication, the magnetite recoveries for the composited intervals of individual samples are not systematically influenced (biased) by method of drilling and type of recovered sample".
Logging	• *Whether core and chip samples have been geologically and*	• Every RC, PD and DD drillhole was logged by a geologist &

Criteria	JORC Code explanation	Commentary
	geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	entered into Excel spreadsheets recording; Recovery, Moisture content, Magnetic susceptibility, Oxidation state, Colour, % of Magnetite, Gangue Min, Sulphide Min, Veins and Structure. Data was uploaded to a customised Access database. • Handheld magnetic susceptibility measurements and geological logging was completed for every metre of every drillhole. • Logging used a mixture of qualitative and quantitative codes. • All RC sample metres were sub-sampled, sieved, washed and stored in a labelled plastic chip tray. All remaining drill core after sampling was stored in labelled plastic core trays and subsequently stored at the company's offices in Broken Hill. • Processing of drillcore included core orientation, metre marking, magnetic susceptibility measurements (every 0.5m), core recoveries, rock quality designation (RQD). All drill core was photographed wet and dry after logging and before cutting. • All relevant intersections were logged. • Geological logging was of sufficient detail to allow the creation of a geological model.

Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.*
	• *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.*
	• *For all sample types, the nature, quality and appropriateness of the sample preparation technique.*
	• *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.*
	• *Measures taken to ensure that the sampling is representative of the in-situ material collected, including for instance results for field duplicate/second-half sampling.*
	• *Whether sample sizes are appropriate to the grain size of the material being sampled.*
		• The 2010 RC samples were composited using geological control via the spear sampling method of the 1m bulk sample bags. The spear method was concluded by CAP to be adequate based on the results of a handheld XRF orientation exercise. The green plastic bags were speared from a range of angles to the bottom of the bag to ensure a representative sample. The compositing produced a 2m to 10m 3kg sample for laboratory analysis at ALS Labs in Perth.
		• The 2016 RC samples were split using a riffle splitter (no details of type used) that produced a 1/16th split taken from the rig every metre and then composited to 5m intervals by splitting again using a 50/50 splitter to give a 6-7kg sample.
		• DD core was cut into half core using a brick saw and diamond blade. The core was cut using the orientation line or perpendicular to bedding. to produce an 8m composite sample (predominantly NQ core). Half core was sent to ALS Perth for analysis, whilst remaining half core was retained for reference.
		• <u>Sample Prep was completed at ALS Laboratories Perth</u>
		  ○ Crush the sample to 100% below 3.35 mm.
		  ○ A 150 g sub-sample for pulverizing in a C125 ring pulveriser (record weight) – DTR SAMPLE.

Criteria	JORC Code explanation	Commentary
		- Initially pulverize the 150 g sample for nominal 30 seconds – the sample is unusually soft for a ferro-silicate rock. - Wet screen the DTR sample at 38 micron pressure filter and dry, screen at 1 mm to de-clump and re-homogenize. - Record the oversize weights – if less than approximately 20 g is oversize, stop the procedure – failure. - If failure - select another 150 g DTR Sample and reduce the initial pulverization time by 5 secs, repeat until initial grind pass returns greater than approximately 20 g oversize. Once achieved retain the – 38 micron undersize. - Regrind only the oversize for 4 seconds of every 5 g weight of oversize. - Repeat the wet screening, drying, de-clumping & weighing stages until less than 5g above 38micron remains. - Ensure the remaining < 5 g oversize is returned back into the previously retained -38 micron product. - Report the times and weights for each grind pass phase. - Combine and homogenize all retained -38 micron aliquots and <5 g oversize –final pulverized product. Sub-sample the final pulverized product to give a 20 g feed sample for DTR work and a ~10 g sample for HEAD analysis via XRF fusion. • The 2010 work employed field duplicates (23 5m samples) using the spear sampling technique which on analysis produced acceptable results. • The 2016 work had a much more comprehensive QAQC programme which included 87 field pairs (not actual duplicates unfortunately) at an insertion rate of 1 in 10, 111 lab duplicates and 39 blanks (river sand) at an insertion rate of 1 in 20, 58 2nd lab checks (Intertek Labs in Perth), pulp duplicates for XRF analysis and sample prep checks. • For the 2016 work the field pair results produced a slightly sub-optimal outcome, but were still acceptable for the current resource classification and seemed to be less precise than the spear sampling method used in 2010. The lab duplicates (a second 150g split) produced good results indicating acceptable sample preparation procedures. The 2nd lab checks on 150g sub-samples produced results indistinguishable from the original lab results. Pulp duplicates demonstrated chemically homogeneity with the XRF analysis. • 30 Primary crush and sub-sample checks were completed by Aussam Geotechnical Services (Broken Hill) which concluded that no

Criteria	JORC Code explanation	Commentary
		evidence of bias with the oversize mineralogy. • Blank samples comprising river sand produced results that indicated no contamination of the samples during the sample prep process. • An additional check on the field sub-sampling and compositing procedure used a Jones 3 tier riffle splitter (1/8) and a free-standing 1:1 splitter to match the 1/16 rig splitter. A total of 30 5m composite intervals were utilised. Noting that all samples were dry, slightly better results were achieved than the original field pair process. However under full field conditions it was thought that there was likely to be no difference between the riffle splitting and spear sub-sampling methods. Both are at risk to human errors, which perhaps can be better managed with the riffle splitting. • All sampling methods and samples sizes are deemed appropriate.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	• <u>Davis Tube Recovery (DTR) Analysis</u> ○ Pulveriser bowl 150 ml ○ Stroke Frequency - 60/minute ○ Stroke length – 38mm ○ Magnetic field strength – 3000 gauss ○ Tube Angle – 45 degrees ○ Tube Diameter – 40mm ○ Water flow rate – 540-590 ml/min ○ Washing time 20 minutes ○ Collect the concentrate in small collector (magnetic fraction) and discard tails. ○ Dry the DTR concentrate and report the weight of the concentrate as a percentage of measured feed and report – DTR Mass Recovery. • <u>X-Ray Fluorescence (XRF) Assaying</u> ○ Using the Head Sample, analyse by <u>XRF fusion method</u> for the following elements: Al2O3%, As%, Ba%, CaO%, Cl%, Co%, Cr%, Cu%, Fe%, K2O%, MgO%, Mn% Na2O%, Ni%, P%, Pb%, S %, SiO2%, Sn%, Sr%, TiO2%, V%, Zn%, Zr% & LOI. ○ Using the DTR concentrate sample analyse by XRF fusion method for the following elements: Al2O3%, As%, Ba%, CaO%, Cl%, Co%, Cr%, Cu%, Fe%, K2O%, MgO%, Mn% Na2O%, Ni%, P%, Pb%, S %, SiO2%, Sn%, Sr%, TiO2%, V%, Zn%, Zr% & LOI • JH8 and KT5 magnetic susceptibility meters were used to record

Criteria	JORC Code explanation	Commentary
		magnetic susceptibility. A laboratory standard was used each day to calibrate each metre. A Niton XL3T Gold handheld XRF machine was used. A laboratory analysed sample was used to calibrate for Fe. • QAQC procedures consisted of the use of 3 certified reference materials for DTR (head and high grades) and XRF analysis at a frequency of 1 per 15 for the 2016 drilling. The reported results for the standards meet industry accepted criteria for accuracy, both for DTR magnetite recoveries and XRF analyses of the critical elements (Fe, Si, Al, and P). • It is uncertain if certified reference materials were used for the 2010 drilling. In CAP's documented drilling procedures it was indicated that a standard insertion rate of 1 in 30 should be used. In a QAQC review of procedures Keith Hannan noted that CAP utilises a 'monitor' standard consisting of crushed magnetite-rich rock derived from local outcrops but without commenting on any results. • Keith Hannan of Geochem Pacific Pty Ltd, an independent geochemist/consultant reviewed the QAQC results for both the 2010 and 2016 drilling and expressed satisfaction with precision, accuracy and any lack of bias in the data, making it fit for purpose for resource estimation. • All assay methods are deemed appropriate.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• Data was stored in a customised Access database. • Database checks were completed by S. Tear of H&SC on 5 randomly selected drillholes. Checks included comparing database values with original collar survey reports, downhole survey reports and assay certificates. • Two DD holes were used as twin holes to verify the results for 2 pairs of RC holes and the DTR performance. • The results are reasonable but there is some potential ambiguity mainly due to a fundamental lack of assay data (mainly with the diamond drilling) and the separation distance of the relative mineral intercepts. It was concluded by Keith Hannan that "the 'twin hole' site data that, although there is demonstrable variation in average magnetite grades within several metres along-strike, there is no evidence of a consistent positive bias in the magnetite levels

Criteria	JORC Code explanation	Commentary
		determined for RC samples". • No details are available for any documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • CAP used a suite of documented procedures for the 2016 drilling-related activities drawn as a flowsheet. • No adjustments were made to raw assay data except for the resource estimation where below detection results were recorded as half below detection value. • Density data from the downhole geophysics was adjusted upwards by 5.2% based on check density measurements using drillcore and the immersion in water, weight in air/weight in water (Archimedes) method.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• Drillhole collars were located by a local surveyor using a Differential GPS with accuracy to less than one metre. • Coordinates were supplied in GDA 94 – MGA Zone 54. H&SC used a local grid conversion which involved rotating the drilling data 320º in a clockwise direction to give an orthogonal E-W strike to the mineralisation. • Down hole surveys for the 2010 drilling were initially recorded as single shot digital displays and were then recorded using a gyroscope due to the highly magnetic nature of the deposit. All the 2016 drillholes had downhole surveys measured using a gyroscope. • It is noted that the downhole surveys in the database for the 2010 drilling consisted of 30 to 60m spaced single shot camera surveys and not the gyro data due to limitations with the gyro data as result of hole collapse and reluctance of the contractor to send the probe to the full hole depths. A 3D check plot of five holes indicated minimal deviation for the common downhole lengths between the single shot and gyro data. Hole deviation appeared to increase to significant distances but is associated with a 'run over' projection of the gyro data and therefore not necessarily accurate. • Topographic control was collected using a high-resolution Differential GPS by a local surveyor. • Location methods used to determine accuracy of drillhole collars are considered appropriate.

Criteria	JORC Code explanation	Commentary
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• The deposit is drilled at a nominal spacing of 150m to 200m in section and plan extending to 400m on the periphery of the deposit. • The drill spacing was deemed adequate for the interpretation of geological and grade continuity noting the along strike stratigraphic homogeneity associated with the style of mineralisation. • The 2010 drill samples were composited under geological control with an interval range of 2 to 10m with an average length of 8m. The 2016 RC drill samples were composited to 5m.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• Drilling was generally angled at -60º dip, and at right angles to geological strike to generally ensure sub-perpendicularity to the bedding, which is the primary control to the magnetite mineralisation. • Different azimuths were used to reflect the changing strike of the beds associated with folding of the sediments and were designed to maintain the steep angle to the bedding. • Locally holes suffered significant deviation to the right (east) with depth. This affected the lower Unit 2 more than the upper Unit 3. • Drilling orientations are considered appropriate with no bias. • The drilling dip and azimuths made it very difficult to intersect the cross cutting fault structures as the drilling was often sub-parallel to these features. Therefore information on the nature and impact on metal grade of the structures particularly with any potentially associated penetrative oxidation is relatively unknown.
Sample security	• *The measures taken to ensure sample security.*	• All samples were stored on site under CAP personnel supervision until transporting to the CAP Broken Hill office. • No details are available on the transportation of samples to the laboratory.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Sample procedures and results were systematically reviewed by CAP personnel. • The QAQC data was reviewed by CAP staff • The 2010 QAQC data was also reviewed by Keith Hannan of Geochem Pacific Pty Ltd, an independent Geochemist/consultant who concluded: 1. The duplication procedure for composite RC samples, by careful spearing, is demonstrably effective; 2. An absence of mismatches between duplicates and the consistency of analytical results for CAP blanks and the CAP

Criteria	JORC Code explanation	Commentary
		certified standards indicate that sample handling procedures in the field for this complex program are well executed 3. Based on the laboratory chemical analyses and derived parameters such as magnetite content, the CAP monitor standard is chemically and mineralogically uniform and therefore 'fit-for-purpose'. 4. The high degree of correlation between the averaged field portable (FP) XRF readings for Fe on primary bags of RC spoil and the laboratory analyses of Fe on the much smaller composite samples derived thereof, indicates that downhole Fe distributions are successfully mapped by FP XRF and that the compositing procedure is effective. • Keith Hannan completed an exhaustive review of the sampling and assaying for the 2016 drilling which concluded "*The investigation of multiple sources of QAQC data finds the magnetite recoveries and chemical analyses obtained for the sample composites of the Hawsons Iron Project 2016 RC Infill Drilling Programme to be fit for the intended purpose of ore resource estimation and planning. Sampling and laboratory preparation and analytical errors are well within industry standard tolerances, and without demonstrable bias*".

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The Hawsons Magnetite project is located in Western NSW, 60 km southwest of Broken Hill. The deposit is 30km from the Adelaide-Sydney railway line, a main highway and a power supply. • The project is under a Joint Venture between Hawsons Iron Ltd (HIO) and Starlight Investment Company Pty Ltd where HIO holds 94% and Starlight 6% equity in the project. HIO currently manage the project. • The project area is wholly within Exploration Licences (ELs) 6979,

Criteria	JORC Code explanation	Commentary
		7208 & 7504 which are 100% owned by HIO. • Licence conditions for all ELs have been met and are in good standing. • An application for a Mining Lease (ML) was lodged with the NSW Trade & Investment Department in October 2013 and HIO is not aware of any impediments to obtaining a mining lease.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• In 1960 Enterprise Exploration Company (the exploration arm of Consolidated Zinc) outlined a number of track-like exposures of Neoproterozoic magnetite ironstone (+/- hematite) which returned a maximum result of 6m at 49.1% Fe from a cross- strike channel sample. No drilling was undertaken by Enterprise. • CRAE completed in 1984, five holes within EL 6979 seeking gold mineralisation in a second-order linear magnetic low interpreted to be a concealed faulted iron formation within the hinge of the curvilinear Hawsons' aeromagnetic anomaly. CRAE's program failed to locate significant gold or base metal mineralisation but the drilling intersected concealed broad magnetite ironstone units interbedded with diamictite adjacent to the then untested peak of the highest amplitude segment of the Hawsons aeromagnetic anomaly.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Hawsons Magnetite Project is situated within folded, upper greenschist facies Neoproterozoic rocks of the Adelaide Fold Belt. The Braemar Facies magnetite ironstone is the host stratigraphy and comprises a series of strike extensive magnetite-bearing siltstones generally with a moderate dip (circa -55^{o}), primarily to the south west. The airborne magnetic data clearly indicates the magnetite siltstones as a series of parallel, high amplitude magnetic anomalies. Large areas of the Hawsons prospective stratigraphy are concealed by transported ferricrete and other younger cover. The base of oxidation due to weathering over the prospective horizons is estimated to average 80m from surface. • The Hawsons project comprises a number of prospects including the Core, Fold, T-Limb, South Limb and Wonga deposits. Mineral Resources have been generated for the Core and Fold areas which are contiguous. • The depositional environment for the Braemar Iron Formation is

Criteria	JORC Code explanation	Commentary
		believed to be a subsiding basin, with initial rapid subsidence related to rifting possibly in a graben setting as indicated by the occurrence of diamictites in the lower part of the sequence (Unit 2). A possible sag phase of cyclical subsidence followed with deposition of finer grained sediments with more consistent, as compared to the diamictite units, bed thicknesses, style and clast composition (Unit 3). The top of the Interbed Unit marks the transition from high (Unit 2) to lower (Unit 3) energy sediment deposition • The distribution of disseminated, inclusion-free magnetite in the Braemar Iron Formation at Hawsons is related to the composition and nature of the sedimentary beds. The idioblastic nature of the of the magnetite is believed due to one or more of a range of possible processes including in situ recrystallisation of primary detrital grains, chemical precipitation from seawater, permeation of iron-rich metamorphic fluids associated with regional greenschist metamorphism. Grain size generally ranges from 10microns to 0.2mm but tends to average around the 40microns. The sediment composition and grain size appear to provide the main control on the mineralisation. There is no evidence for structural control in the form of veins or veinlets coupled with the lack of a strong structural fabric • In the majority of the Core and Fold deposits the units strike southeast and dip between 45 and 65º to the south west. The eastern part of the Fold deposit comprises a relatively tight synclinal fold structure resulting in a 90º strike rotation.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* ○ *easting and northing of the drill hole collar* ○ *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* ○ *dip and azimuth of the hole* ○ *down hole length and interception depth* ○ *hole length.* • *If the exclusion of this information is justified on the basis that the*	• Exploration results not being reported

Criteria	JORC Code explanation	Commentary
	information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• Exploration results not being reported
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true width not known').*	• Drilling has tended to be at a steep angle to the dip angle of the sedimentary beds.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• Exploration results not being reported
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Exploration results not being reported
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• A substantial amount of polished and thin section work has been completed on both RC chips and diamond core. This work has confirmed the nature and style of both the original sediment and the iron minerals including magnetite, hematite, chlorite and ferroan dolomite. • Downhole geophysics comprises magnetic susceptibility, gamma and density and has been completed for a majority of the holes. This has resulted in the definition of a magnetic (and density- related) stratigraphy that is coincident with a chronostratigraphic interpretation.
Further work	• *The nature and scale of planned further work (e.g. tests for lateral*	• Infill drilling is planned to upgrade the current Mineral Resources to

Criteria	JORC Code explanation	Commentary
	extensions or depth extensions or large-scale step-out drilling). • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	Measured and Indicated, upgrade a portion of the Exploration Target to Inferred, and to provide geotechnical and hydrogeological data.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* • *Data validation procedures used.*	• Independently customised Access database by GR-FX Pty Ltd • Validation of database undertaken by Keith Hannan of Geochem Pacific Pty Ltd, an independent consultant. • Database validation was conducted by H&S Consultants (H&SC) to ensure the drill hole database is internally consistent. Validation included checking that no assays, density measurements or geological logs occur beyond the end of hole and that all drilled intervals have been geologically logged. The minimum and maximum values of assays and density measurements were checked to ensure values are within expected ranges. Further checks include testing for duplicate samples and overlapping sampling or logging intervals • H&SC takes responsibility for the accuracy and reliability of the data used to estimate the Mineral Resources. • H&SC created a local E-W orthogonal grid for all interpretation and modelling work.
Site visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	• Regular site visits were completed by CAP management for the period 2009 to 2017. • A site visit has been undertaken in 2012 by Simon Tear of H&SC, Competent Person for the Exploration Results and the reporting of the Mineral Resources. The visit including geological logging of diamond drillhole DD10BRP023 covering over 500m of stratigraphy and an inspection of drill sites and outcropping mineralisation.

Criteria	JORC Code explanation	Commentary
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• The broad geological interpretation of the Hawsons deposit is relatively straightforward and reasonably well constrained by drilling and the high amplitude airborne and ground magnetic anomalies. • The mineralisation is stratabound as disseminated grains of magnetite with no obvious structural remobilisation or overprint. Mineralisation exhibits relatively poor downhole continuity with zones of variable magnetite grade (a function of the clastic grain size and composition) but in most instance the contacts between higher and lower grade mineralisation are gradational and precludes the use of hard boundaries as stratigraphic control to mineral grade interpolation. • The downhole geophysical data, gamma and magnetic susceptibility, has been used in conjunction with DTR recovered magnetic fraction grades to produce a detailed geological interpretation and to the generation of a set of 3D wireframes representing variously mineralised units and provide a stratigraphic framework. • The consistency of the geophysical patterns for the sediments provides for a high level of confidence in the stratigraphic interpretation. • Two main cross faults, possibly a conjugate pair, have been interpreted and are believed to have caused small offsets in the mineral-bearing stratigraphy. The faults have been used to delineate three structural domains. • H&SC used the geological logs of the drill holes to create a wireframe surface representing the base of colluvium. • H&SC also used the geological logs of the drill holes to create wireframe surfaces representing the base of complete oxidation (BOCO) and the top of fresh rock (TOFR). Contact plot analysis of the estimated elements were conducted in order to investigate how these surfaces should be treated in the resource estimation. • Any additional faulting in the deposit is assumed to be insignificant relative to the resource estimation. • H&SC is aware that alternative interpretations of the mineralised zones and faults are possible but consider the wireframes to adequately approximate the locations of the mineralised zones for the

Criteria	JORC Code explanation	Commentary
		purposes of resource estimation. Alternative interpretations may have a limited impact on the resource estimates.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The Mineral Resources have a strike length of around 3.3km in a south easterly direction. The plan width of the resource varies from 700m to 1.9km with an average of around 1.1km (noting the relatively modest dip angle of the beds). The upper limit of the mineralisation occurs between 25 and 80m below surface (average 65m) and the lower limit of the Mineral Resource extends to a depth of 440m below surface. The lower limit to the Mineral Resource is a direct function of the depth limitations to the drilling in conjunction with the search parameters. The mineralisation is open at depth.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	• Ordinary Kriging with dynamic interpolation was used to complete the estimation in the Micromine software. H&SC considers Ordinary Kriging to be an appropriate estimation technique for the type of mineralisation and extent of data available from the Core and Fold prospects. All data have low coefficients of variation generally <1. • A total of 3,924 unconstrained 5m composites were generated from the drillhole database and modelled for Davis Tube recovered magnetic fraction ("DTR"), iron head grade and the concentrate elements of Al_2O_3, P, S, SiO_2, TiO_2 and LOI, • 2,862 composites were in fresh rock and 1,161 in the transition zone of which 209 were from direct DTR measurement. 74 of the fresh rock composites were generated from the downhole mag_sus data with 55 from the hand-held mag_sus data via regression equations, particularly peripheral to the main mineralisation and the transition zone. • A regression based on downhole magnetic susceptibility was used to calculate likely DTR values for untested intervals. A regression based on the handheld magnetic susceptibility data was used to estimate the DTR values where downhole magnetic susceptibility was not available. Missing Fe concentrate grades were calculated using a regression based on the DTR grades and the remaining concentrate elements were calculated using a regression based on the iron concentrate grade. Most of the missing DTR grades were on the

Criteria	JORC Code explanation	Commentary
		periphery of the mineralisation (often unsampled areas) and the missing concentrate grades the result of insufficient sample being available for XRF analysis mainly from the Interbed Unit. • The base of colluvium was used to control the upper limit of the resource estimation. Drill hole data from above the colluvium surface were not used in the resource estimates. • Two main cross faults have been delineated and have caused small offsets in the mineral-bearing stratigraphy. These faults were treated as hard boundaries during estimation so that data from within a particular fault block were only used to estimate blocks in that fault block. • H&SC created nine surfaces representing the margins of eight conformable lithological units based on drill hole data. These surfaces were combined to produce eight wireframe solids, the outer boundary of which was used to constrain the Mineral Resource Estimate. In order to reflect local variations of dip and strike, the orientation of the triangles that make up the nine surfaces were used to locally control the orientation of the search ellipse and variogram axes – the dynamic interpolation method. • The TOFR surface was found to coincide with a marked difference in density and DTR and was therefore used as a hard boundary. The density and DTR values in the volume above the TOFR surface were estimated using a flattened search ellipse. All other parameters did not take account of the top of fresh rock surface and the orientation of the search ellipse and variogram axes are controlled by the orientation of the lithological unit surfaces. • No recovery of any by-products has been considered in the resource estimates as no products beyond iron are considered to exist in economic concentrations. • No top-cutting was applied as extreme values were not present and top-cutting was considered by H&SC to be unnecessary • No check estimate was carried out though the estimates were in line with previous estimates. Hellman & Schofield, the predecessor to H&SC, estimated the Mineral Resources for Hawsons in 2010 and

Criteria	JORC Code explanation	Commentary
		updated in 2010. The resource estimates were further updated in 2013 by H&SC following an in-depth analysis and interpretation of downhole geophysical data resulting in the delineation of Indicated Resources. The 2017 Mineral Resources showed a modest increase in size at the same grade. but contain considerably more Indicated Resource which was the aim of the infill drilling. The extra Mineral Resources are primarily from peripheral areas in the Core and the Fold areas. The marked lowering of the cut off grade used for reporting the 2021 Mineral Resources has resulted in a substantial increase in size with a nominal 10% drop in DTR grade. • Block dimensions are 100m x 50m x 15m (Local E, N, RL respectively). The east and north dimensions were chosen as they are around half to a third of the nominal drillhole distances. The vertical dimension was chosen to reflect the sample spacing and possible mining bench heights. • Each element was estimated separately. Four search passes were employed with progressively larger radii or decreasing search criteria. The first pass used radii of 250x150x40m, the second pass used 300x150x50m, the third and fourth used 450x225x75m (along strike, down dip and across mineralisation respectively). All passes used a four-sector search with a maximum number of data points per sector of 8 (total 32). The first pass required a minimum of 20 data points from at least three different drill holes whereas the second and third passes required a minimum of 16 data points from at least two different drill holes. The fourth pass required a minimum of eight data points and had no restriction on the number of drill holes required. • The new block model was reviewed visually by H&SC and CAP geologists and it was concluded that the block model fairly represents the grades observed in the drill holes. H&SC also validated the block model using a variety of summary statistics and statistical plots.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• Tonnages of the Mineral Resources are estimated on a dry weight basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• The resources are reported at a cut-off of 6% DTR based on the outcome of a recently completed pit optimisation study by

Criteria	JORC Code explanation	Commentary
		independent consultants KPS Innovation of Brisbane. • The cut-off grade at which the resource is quoted reflects the intended bulk-mining approach.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• The Mineral Resources were estimated on the assumption that the material is to be mined by open pit using a bulk mining method. • Minimum mining dimensions are envisioned to be around 25m x 10m x 10m (strike, across strike, vertical respectively). The block size is significantly larger than the likely minimum mining dimensions. • The resource estimation includes internal mining dilution. • A 2017 PFS completed by GHD developed a mine plan to produce 10Mtpa of magnetite concentrates via on site processing. • The proposed mining method would use a combination of In-Pit Crushing and Conveying as well as truck and shovel opertions.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• The idioblastic nature of the magnetite lends itself to relatively easy liberation. • The ROM material is relatively soft for a magnetite deposit with a bond work index much lower than typical Banded Iron Formation deposits. • Initial laboratory testwork by the CSIRO in Brisbane identified that the ROM material could readily be reduced to a particle size less than 1mm in an impact crusher. • hrlTesting completed metallurgical testwork that showed better than 50% rejection can be achieved in the rougher stages. The ball mill operational power is lower than expected and at a P_{100} of 38µm a concentrate of ~69% Fe can be achieved.
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	• The deposit lies within flat, open country typical of Western NSW. • Predominantly scrub vegetation that allows for sheep grazing. • There are large flat areas for waste and tailings disposal. • Small number of creeks with only seasonal flows. • Baseline data collection of a variety of environmental parameters is in progress e.g. dust monitoring, surface water, weather records • Preliminary Ecology Assessments have led to field ecology studies under the guidance of the Office of Environment and Heritage in NSW. • A Water Optimisation Study identified ways to reduce water

Criteria	JORC Code explanation	Commentary
		consumption in the plant and has led to a new process design considering paste thickening in the metallurgical plant instead of the original conventional thickeners.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	• The short-spaced density (SSD) data from the downhole geophysics was used for the density of the Mineral Resources. The SSD data was collected using a FDS50 down hole tool containing a 3500CO radioactive source. • This data had a correction factor of +5.2% applied based on testwork completed on 194 NQ core samples using the immersion-in-water weight in air/weight in water (Archimedes) method. • The data was composited to 5m prior to modelling. • The density at Hawsons was estimated using Ordinary Kriging for search passes 1 to 3 and the remaining blocks were populated from values estimated from the Fe head grade of each block using a regression created from blocks where both variables had been estimated.
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• The classification of the resource estimates is based on the data distribution which is a function of the drillhole spacing • Other aspects have been considered including, the style of mineralisation, the geological model, sampling method and recovery, coherency of the downhole geophysics including density, the QAQC programme and results and comparison with previous resource estimates. • The resources were initially classified on the search criteria with blocks populated by Passes 1 and 2 being Indicated and Passes 3 and 4 being classed as Inferred. • Upon review of the Indicated resources a defined shape was delineated which reverted individual or small numbers of isolated blocks from Indicated to Inferred. • A detailed sedimentological review using gamma and magnetic susceptibility downhole data demonstrated strong stratigraphic continuity of the DTR grades with the sediment packages. • H&SC believes the confidence in tonnage and grade estimates, the continuity of geology and grade, and the distribution of the data reflect

Criteria	JORC Code explanation	Commentary
		Indicated and Inferred categorisation. The estimates appropriately reflect the Competent Person's view of the deposit. H&SC has assessed the reliability of the input data and takes responsibility for the accuracy and reliability of the data used to estimate the Mineral Resources.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• The estimation procedure was reviewed as part of an internal H&S Consultants peer review and the block model was reviewed visually by CAP geologists. • Mining Associates Limited ("MA") completed a technical review in 2016 on the 2014 Inferred and Indicated Resources. MA concluded that the model is a good global representation of the magnetite resource and considers Ordinary Kriging to be an appropriate estimating technique for the type of mineralisation with very low coefficients of variation. • In a follow up report in 2020 MA concluded that for the 2017 Mineral Resources: "Following [a] review of the geology, MRE and Reserve, MA does not consider the current approach to the geology model and MRE suitable. A much higher level of detail needs to be incorporated into the Geological Model and MRE" and strongly proposed its own methodology of using implicit modelling "with much smaller blocks" incorporating upwards of 20+ stratigraphic boundaries, as being more suitable. • Behre Dolbear Australia ("BDA") completed a technical review for CAP in 2010 based on a GHD study. BDA considered that the broad geology and geological controls on mineralisation, the sampling methodology and the geological database were generally adequately defined for estimation of Inferred [2010] Resources.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.*	• No statistical or geostatistical procedures were used to quantify the relative accuracy of the resource. The global Mineral Resource estimates of the Hawsons deposit are moderately sensitive to higher cut-off grades but does not vary significantly at lower cut-offs. • The relative accuracy and confidence level in the Mineral Resource estimates are considered to be in line with the generally accepted accuracy and confidence of the nominated Mineral Resource

Criteria	JORC Code explanation	Commentary
	• *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	categories. This has been determined on a qualitative, rather than quantitative, basis, and is based on the Competent Person's experience with similar deposits and geology • The Mineral Resource estimates are considered to be accurate globally, but there is some uncertainty in the local estimates due to the current drillhole spacing, a lack of geological definition in certain places and some ambiguity with the QAQC procedures and outcomes. • No mining of the deposit has taken place, so no production data is available for comparison.

PIT OPTIMISATION STUDY

KPS INNOVATION

OCTOBER 2021

Hawsons IRON

Disclaimer

This report has been prepared for Hawsons Iron and is based on the information currently available. Several assumptions have been made in this report all of which have been disclosed and agreed to by Hawsons Iron. These assumptions can have a material impact on outcomes presented in this report. Much of the analysis has been done on exploration targets and are not adequate to be classified as JORC reserves or resources, the analysis is to be used to identify value adding areas of the potential deposit for the upcoming drill campaign of Hawsons. Financial outcomes in this report are for illustrative purposes only and are not to be relied upon for investment decisions.

SUMMARY

KPS was engaged by Hawsons Iron to complete a preliminary pit optimisation on both it's JORC'd and potential resource as defined by its current block model. KPS calculated a cut-off grade of 4% Davis Tube Recovery (DTR) which enabled the first pass of pit optimisations to be run. This is similar to other feasibility studies on magnetite projects that have recently been completed using similar iron ore pricing assumptions.

Following the initial analysis of pit optimisations, the models were then rerun to at a variety of cut-off grades from 2.5% to 14.5% to model the effect on material movement and DCF of the pits. Utilising a discount rate of 8% it was decided that 6% be used as the optimum cut-off grade for the pit shells. Further work around scheduling and stockpiling of ore will likely further increase the NPV of this project. With current defined resources ore above 6% will produce 392MT of concentrate at a grade of 69.8% Fe, extending this across to potential resource increases the tonnage of concentrate to 576MT at a grade of 69.7% Fe.

To produce a 20 year mine life at 10Mtpa of concentrate in a model that includes indicated, inferred and potential resource only factors in 6-7% of potential resource with the rest of the ore feed being made up of an equal proportion of indicated and inferred resources. Thus, roughly 100Mt of ore is needed to be brought from inferred to measured (or conversely 100Mt from indicated to measured and 100Mt from inferred to measured). There is no shortage of ore however investigations should be made into potentially outcropping and shallow ore to feed the mill.

The project has enormous potential to produce a low carbon feedstock for the steel market and should be actively investigating new technologies which can develop this concept. Many of the concepts investigated in this report have the ability not only to reduce CO_2 emissions but also costs, most retain the flexibility of a traditional truck and shovel operation apart from continuous miners and conveyors. A high-level assessment of this technology should be carried out to see if it has any impact on the upcoming drill campaign.

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1 INTRODUCTION

Hawsons Iron Limited (ASX:HIO) (**"HIO"**) has completed a ~$35M capital raise to complete the Bankable Feasibility Study (BFS) on its Hawsons Iron Project (HIP). The HIP was subject to a PFS by GHD in 2017, and HIO is currently completing a gap analysis on the PFS and defining the BFS plan. One key element identified is the cut-off grade used in the resource calculations, which was not rerun post the PFS outcomes. This may impact the potential location of the waste rock dump which is currently located on sub-economic ores. The HIP is nominally to produce 10 Mtpa of a magnetite iron ore product from c. 70 Mtpa of ROM ore. Small incremental improvement in the load and haul system will have a major impact on the project outcomes, so a review of potential haulage systems is proposed.

1.1 PIT OPTIMISATION

A pit optimisation to take account of all key project parameters identified in the 2017 PFS to define a pit shell that will maximise value for these assumptions. Define possible staging – starting with a small higher-value shell and then pushing back to successively lower value shells, in order to maximise DCF.

1.2 CUTOFF GRADE OPTIMISATION

The project parameters defined in 1.1 will define the marginal economic cutoff that will maximise the undiscounted cash value. However, where the mine life is more than six to eight years, as is the case at Hawsons, an elevated cutoff grade is likely to give a higher DCF and project NPV.

1.3 COMMENTARY ON METHODS TO OPTIMISE THE LONG TERM PIT

Provide a brief discussion on each of the following alternatives to conventional load and haul by excavator/shovel and haul truck. For each option note the pros and cons, key considerations and qualitative assessment of potential to add value:

1.4 PIT AREA DEFINITION

Throughout this report the ore body is referenced to have 3 sections for ease of explaining locations within pit shells. These conventions have been developed independently and are not universal to other reports. In the northwest section of the main magnetic anomaly, we have the "Core" portion which is the thickest part of the ore body. Moving to the southwest and separated by a fault is the "Link" portion, which is slightly narrower, and continuing southeast the "Fold" portion which is outcropping at the surface. The fold section is poorly defined due to a low density of drilling and mineralised modelling appears to end abruptly at the edge of model. This however is yet to be verified by assessing the sections of these sections with the drillhole data displayed as Figure 1 shows sterilisation drilling may be present.

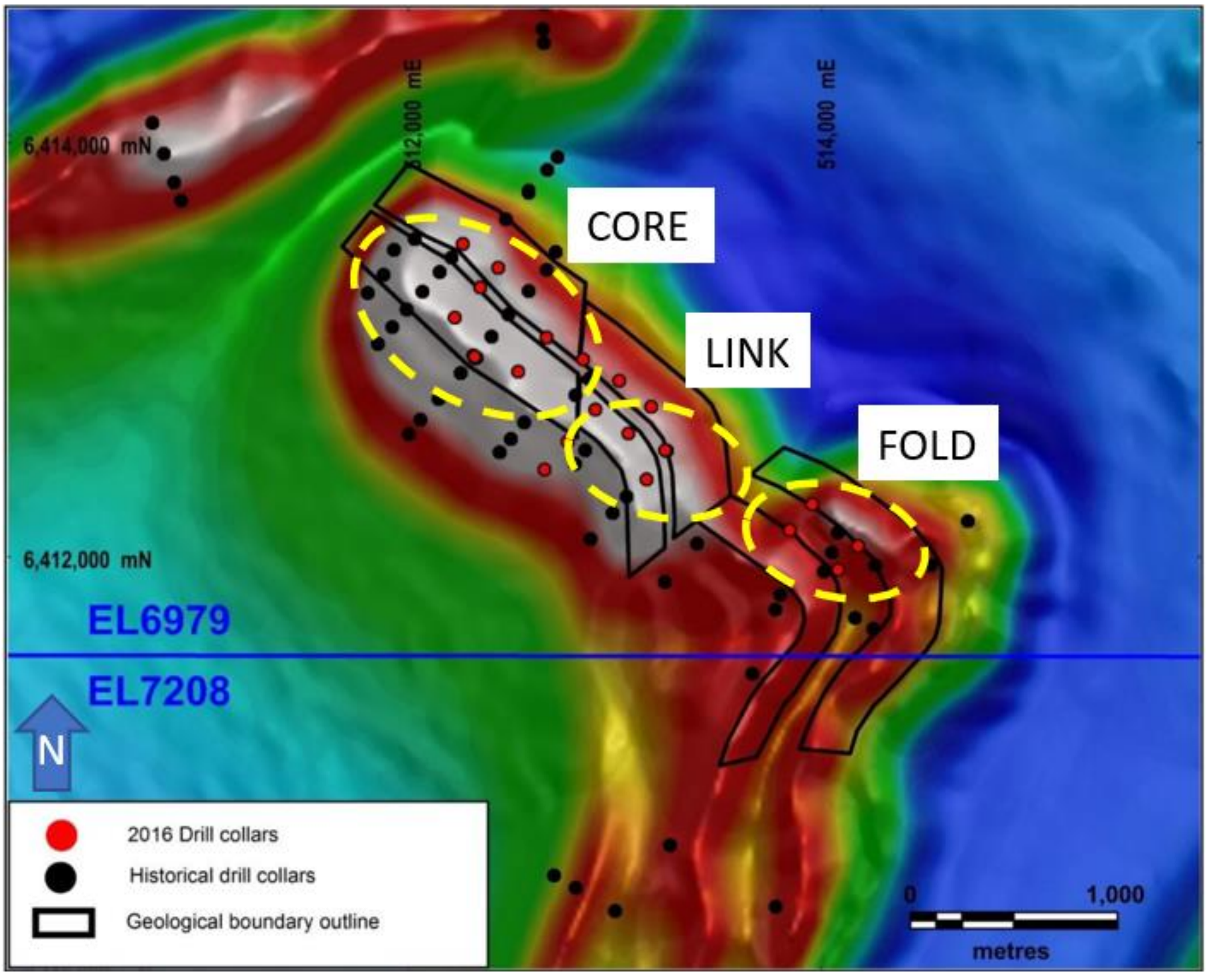

Figure 1. Resource modelling overlayed with the pole magnetics of the deposit.

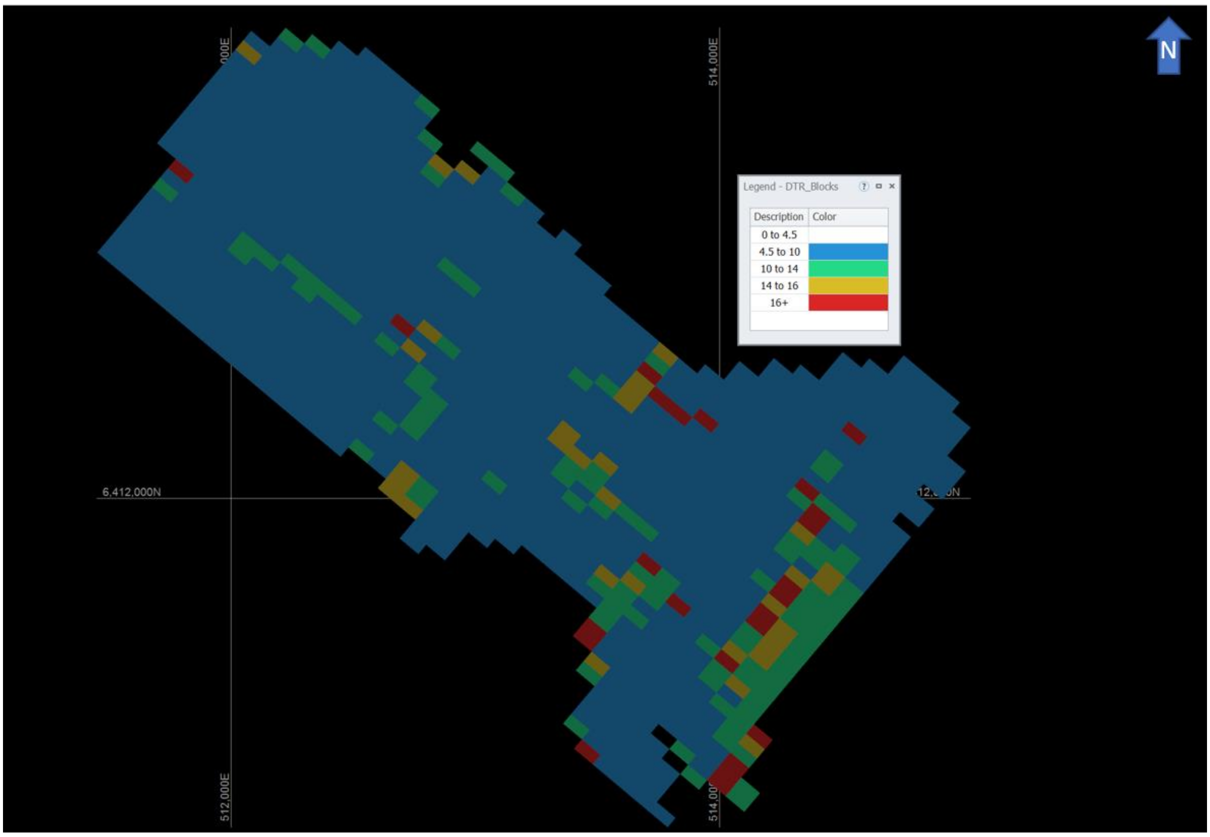

Figure 2. Showing HIP ore body with filter of DTR >4.5%

An aerial view of the ore body is shown in Figure 2 with the waste blocks being removed from view, this shows that there is typically transitional lower grade ore that sits above and around the high-grade core. This can be contrasted with the aerial view in Figure 3 which shows a smaller footprint of material that is considered ore. The difference between the depth of ore is highlighted in the angled views of Figure 4, Figure 5 and Figure 6. This shows that with the lower cut off grade, ore is reached sooner and may provide the flexibility for earlier ore feed for commissioning for the mill. With a larger footprint this will also reduce the strip ratio, particularly in early years (reducing the pre-strip required). In Figure 6 it is shown that the ore body has high grade cores which are separated by lower grade material. These high grade cores are not as well defined in the fold area of the pit and with better definition may prove to provide early mine life ore feed.

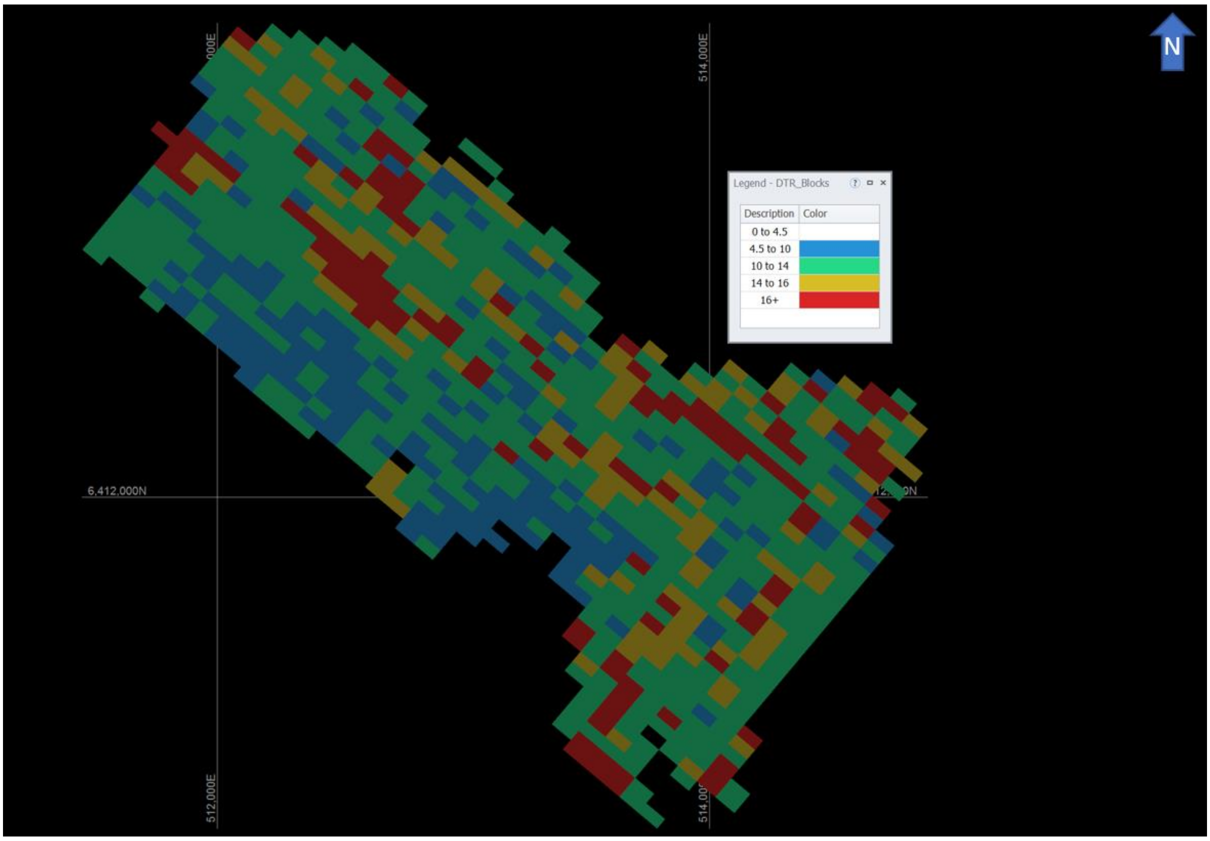

Figure 3. Showing HIP ore body with filter of DTR >9.5%

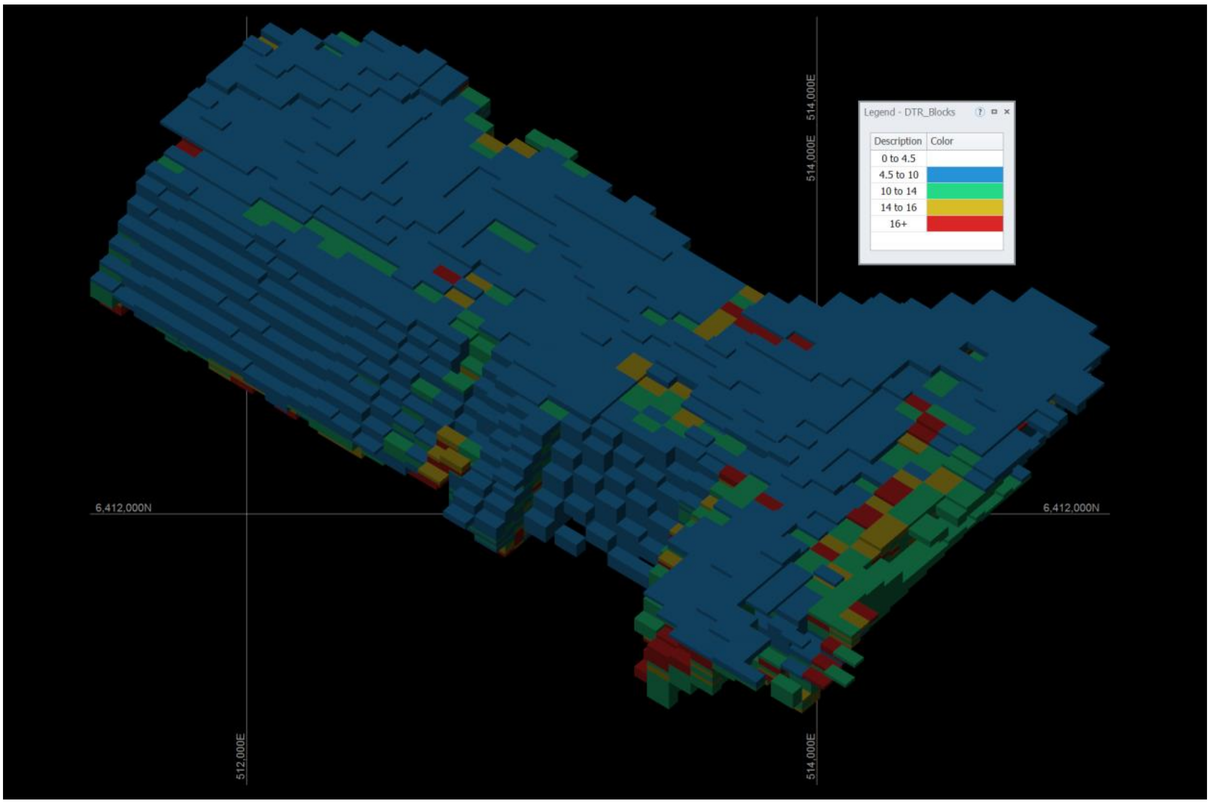

Figure 4 HIP Ore body on a 45º angle with filter of DTR > 4.5%

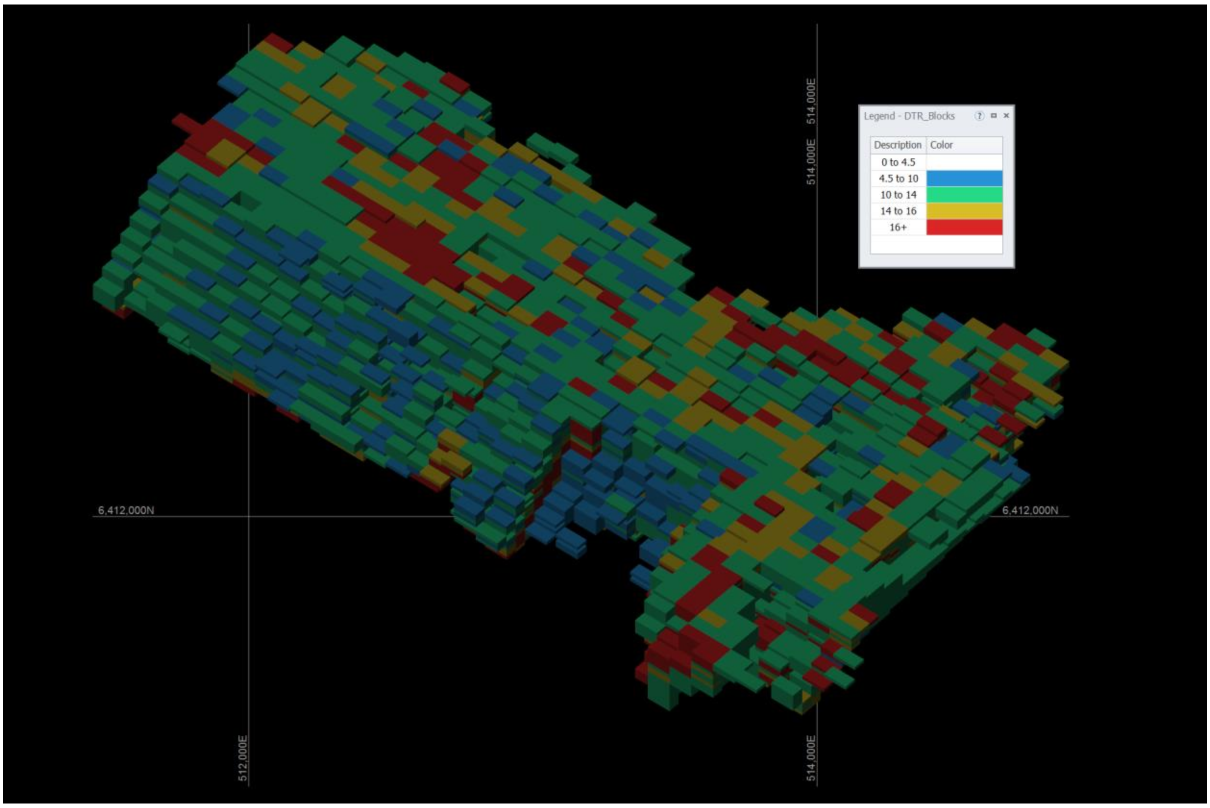

Figure 5 HIP Ore body on a 45º angle with filter of DTR > 9.5%

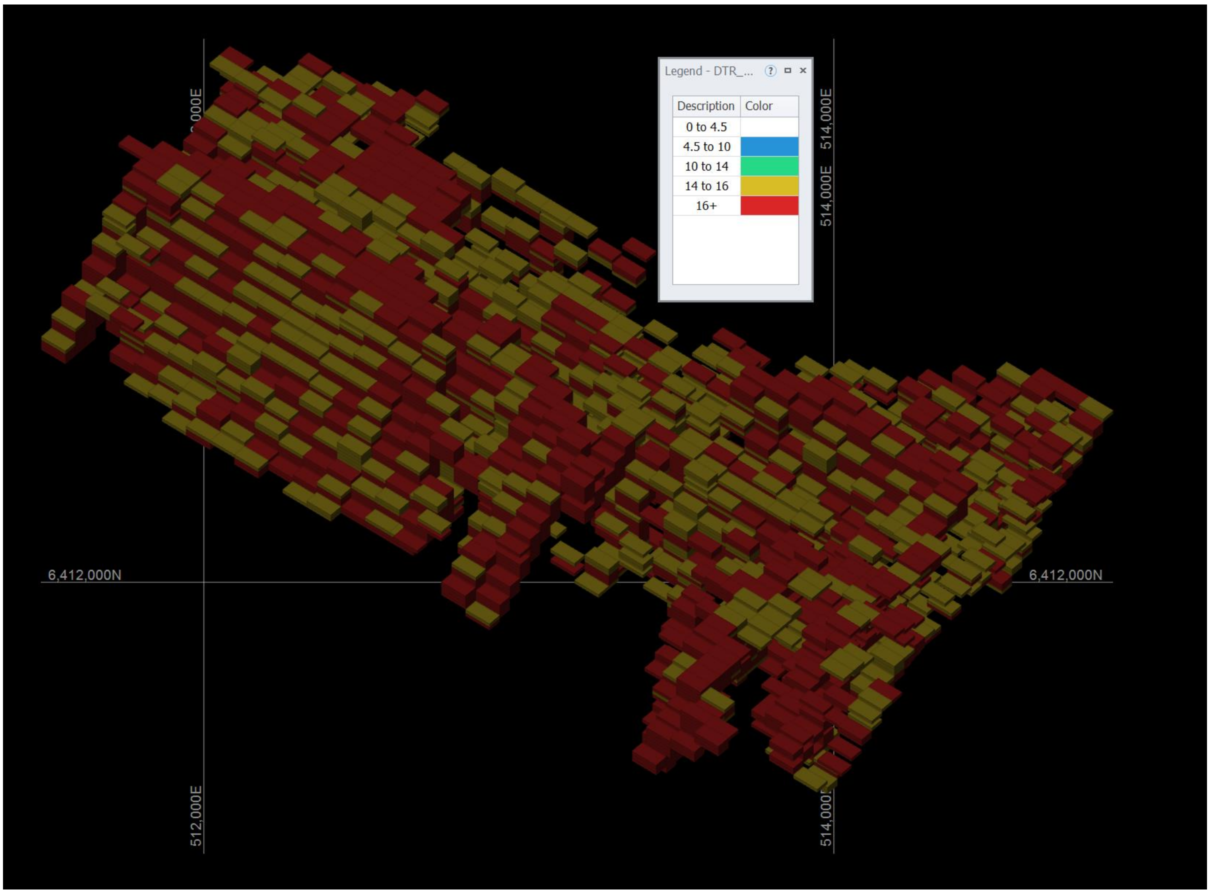

Figure 6. HIP Ore body on a 45º angle with filter of DTR > 15%

2 PIT OPTIMISATION

2.1 INTRODUCTION

Broadly before a block model can be optimised, cost and revenue variables must be assigned to each block after which they can be fed into a network flow algorithm. Whilst the Lerchs-Grossmann algorithm is typically used, for this project we have applied Deswik's Pseudoflow algorithm which is computationally more efficient whilst delivering the same result. This has enabled many scenarios to be run in a far shorter timeframe, whilst also allowing for a greater degree of control over the variables. Industry standards have been used for the definition of these variables which will save time going forward once finer optimisation is applied.

2.2 CLASSIFICATION OF PARAMETERS

Before the algorithm can be run the software must assign an ore or waste property to each block. This can be done several ways however in this model we have assumed a mandatory 10% margin is required on each block. This is done on a twofold basis, the first determining if an ore block is economic (via a mining Lerchs Grossman 3-d algorithm) and then once a pit envelope has been determined off the raw economics defining if it is more economic for a block to be treated as ore or as waste.

$$Cutoff\ LG\ Shell\ \left(\frac{\$}{t_{ore}}\right) = \frac{C+M+P+I+T+A+R}{DTR} \qquad (1)$$

Where all costs are on a per tonne of rock processed basis:

- C = Capital Cost (\$0.62/t ore) (Source: Calculated from page 8 of PFS using assumptions from PFS*)
- M = Mining Cost (\$2.29/t ore) (Source: Calculated from page 8 of PFS using assumptions from PFS*)
- P = Processing Cost (\$1.56/t ore) (Source: Calculated from page 8 of PFS using assumptions from PFS*)
- I = Infrastructure Cost (\$0.18/t ore) (Source: Calculated from page 8 of PFS using assumptions from PFS*)
- T = Transport Cost (\$1.66/t ore) (Source: Calculated from page 8 of PFS using assumptions from PFS*)

- A = General and Administration Costs ($0.66/t ore) (Source: Calculated from page 327 of PFS using assumptions from PFS*)

- R = Rehabilitation (($1.00/t ore) (Source: Discussion with HIO with use as a factor of conservatism)

- DTR = Davis Tube Recovery

* Assumptions are a 10Mtpa Concentrate production, a DTR of 14% and Strip ratio of 0.58 from (GHD, August, 2017)

Table 1.
DTR *Cut-off* grade for mining and processing against 62% Fe Benchmark pricing

DTR	All in Cost/t con (AUD)	Process Cost/t Con (AUD)	Iron Ore Price required 62% Fe (USD) (All In)	Iron Ore Price required 62% Fe (USD) (Process)
1.0%	$ 797.4	$ 568.1	$ 537.3	$ 382.8
1.5%	$ 531.6	$ 378.7	$ 358.2	$ 255.2
2.0%	$ 398.7	$ 284.1	$ 268.7	$ 191.4
2.5%	$ 319.0	$ 227.2	$ 214.9	$ 153.1
3.0%	$ 265.8	$ 189.4	$ 179.1	$ 127.6
3.5%	$ 227.8	$ 162.3	$ 153.5	$ 109.4
4.0%	$ 199.4	$ 142.0	$ 134.3	$ 95.7
4.5%	$ 177.2	$ 126.2	$ 119.4	$ 85.1
5.0%	$ 159.5	$ 113.6	$ 107.5	$ 76.6
5.5%	$ 145.0	$ 103.3	$ 97.7	$ 69.6
6.0%	$ 132.9	$ 94.7	$ 89.6	$ 63.8
6.5%	$ 122.7	$ 87.4	$ 82.7	$ 58.9
7.0%	$ 113.9	$ 81.2	$ 76.8	$ 54.7
7.5%	$ 106.3	$ 75.7	$ 71.6	$ 51.0
8.0%	$ 99.7	$ 71.0	$ 67.2	$ 47.9
8.5%	$ 93.8	$ 66.8	$ 63.2	$ 45.0
9.0%	$ 88.6	$ 63.1	$ 59.7	$ 42.5
9.5%	$ 83.9	$ 59.8	$ 56.6	$ 40.3
10.0%	$ 79.7	$ 56.8	$ 53.7	$ 38.3
10.5%	$ 75.9	$ 54.1	$ 51.2	$ 36.5
11.0%	$ 72.5	$ 51.6	$ 48.8	$ 34.8
11.5%	$ 69.3	$ 49.4	$ 46.7	$ 33.3
12.0%	$ 66.5	$ 47.3	$ 44.8	$ 31.9
12.5%	$ 63.8	$ 45.4	$ 43.0	$ 30.6
13.0%	$ 61.3	$ 43.7	$ 41.3	$ 29.4
13.5%	$ 59.1	$ 42.1	$ 39.8	$ 28.4
14.0%	$ 57.0	$ 40.6	$ 38.4	$ 27.3
14.5%	$ 55.0	$ 39.2	$ 37.1	$ 26.4
15.0%	$ 53.2	$ 37.9	$ 35.8	$ 25.5
15.5%	$ 51.4	$ 36.7	$ 34.7	$ 24.7
16.0%	$ 49.8	$ 35.5	$ 33.6	$ 23.9
16.5%	$ 48.3	$ 34.4	$ 32.6	$ 23.2
17.0%	$ 46.9	$ 33.4	$ 31.6	$ 22.5
17.5%	$ 45.6	$ 32.5	$ 30.7	$ 21.9
18.0%	$ 44.3	$ 31.6	$ 29.9	$ 21.3
18.5%	$ 43.1	$ 30.7	$ 29.0	$ 20.7
19.0%	$ 42.0	$ 29.9	$ 28.3	$ 20.1
19.5%	$ 40.9	$ 29.1	$ 27.6	$ 19.6
20.0%	$ 39.9	$ 28.4	$ 26.9	$ 19.1

All variables were confirmed with HIO with the scope to optimise these costs through calculation from first principles and vendor quotes to bring these more in line on a unit basis. If we calculate this formula out over a range of Davis Tube Recoveries we get the corresponding price required for the iron ore Table 1. HIO has selected to use a benchmark price of US$100/t for 62% Fe content which is based on recent iron ore feasibility studies including that of Magnetite Mines (ASX:MGT) (**"MGT"**) which used a base case price of US$110/t and upside case of US$150/t for 62% Fe benchmark. Open file iron marketing outlook and historic iron contract price average, US$100/t for 62% Fe benchmark is considered reasonably conservative in the current market and provides additional comfort within the 10% margin required for the block to be considered ore. This price is then applied to formula 2 below results in a product price of A$172/7/t of super grade (69.9% Fe).

$$Value\ of\ Supergrade = \frac{B*(1+\mathrm{X})*\left(1+\frac{\mathrm{G}}{\mathrm{D}}\right)*(1-\mathrm{R})}{E} \qquad (2)$$

Where parameters are as follows:

- B = Benchmark Price (US$100)
- X = Super grade Premium (15%)
- G = Grade of super grade product (69.9%) (to calculate intrinsic premium)
- D = Default Benchmark Grade (62%)
- R = Royalty (4% NSW + 1.5% Perilya)
- E = Exchange rate AUD:USD (0.75)

The driving force of the cut-off grade is the grade of the material. For simplicity in equation 1 all costs have been driven to a 10Mtpa scenario, in line with the 2017 PFS, and based on a per unit basis. This is a simplification which may not hold true however due potential changes in initial and final throughputs being confirmed and to the relatively consistent grade of the ore feed it is fit for purpose. The relationship between DTR and cost is shown in Figure 7.

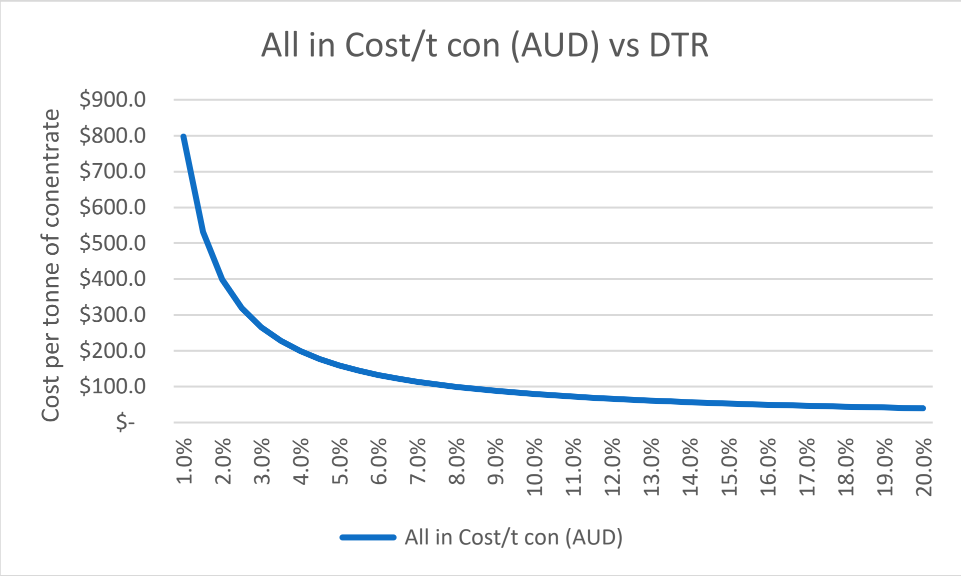

Figure 7. Production cost per tonne of Super grade concentrate driven by DTR.

Pit design constraints were next assessed, with the first being the slope angle. Whilst some initial reports have been provided on the geotechnical competency of material, these are based off limited data and contradicting rulesets were found in the PFS. For this high-level assessment a flat 45^{o} has been used for all walls which is considered a conservative approach. A bench height of 15m was used which matched that of the block model (with each block being 100m x 50m x 15m).

2.3 Global Optimisation

With these parameters defined the pit optimisation could be run, for the first pass or "global" scenario all ore in the block model was used, included indicated, inferred and potential blocks. For the first pass optimisation all ore above the cut-off grade was considered which resulted in a cut-off-grade of ~4%, it is noteworthy that MGT has recently used a 5.8% estimated Davis Tube Recovery in their Maiden Ore Reserve and subsequent Pre-Feasibility Study. A linear cost model was applied to the blocks except for mining cost which increased at a rate of 2% per bench (set at 15m high). This resulted in almost all ore that had been defined being extracted. The scope of this project is to assess 20 years of plant feed producing 10Mtpa of concentrate. As a result, the revenue factors used were maximised at 0.41, to prevent all material being allocated in 3 large shells, the shells were kept artificially small by placing a maximum 45m depth increase per shell (excluding the initial shell) which resulted in 7 shells to give the result. The first shell is in the fold region which gives a total concentrate tonnage of just under 3Mt. This could prove to be a good ore source for commissioning of the processing plant.

To meet the 20 year, mine life at a production rate of 10Mtpa concentrate a revenue factor of 0.41 was used. This resulted in 224 Mt of concentrate with a grade of 69.77%, requiring the mining of 515.6Mt of waste and 1,692Mt of ore for a total material movement of 2,208Mt, which results in a strip ratio of 0.3. The outcomes are shown below in Figure 8.

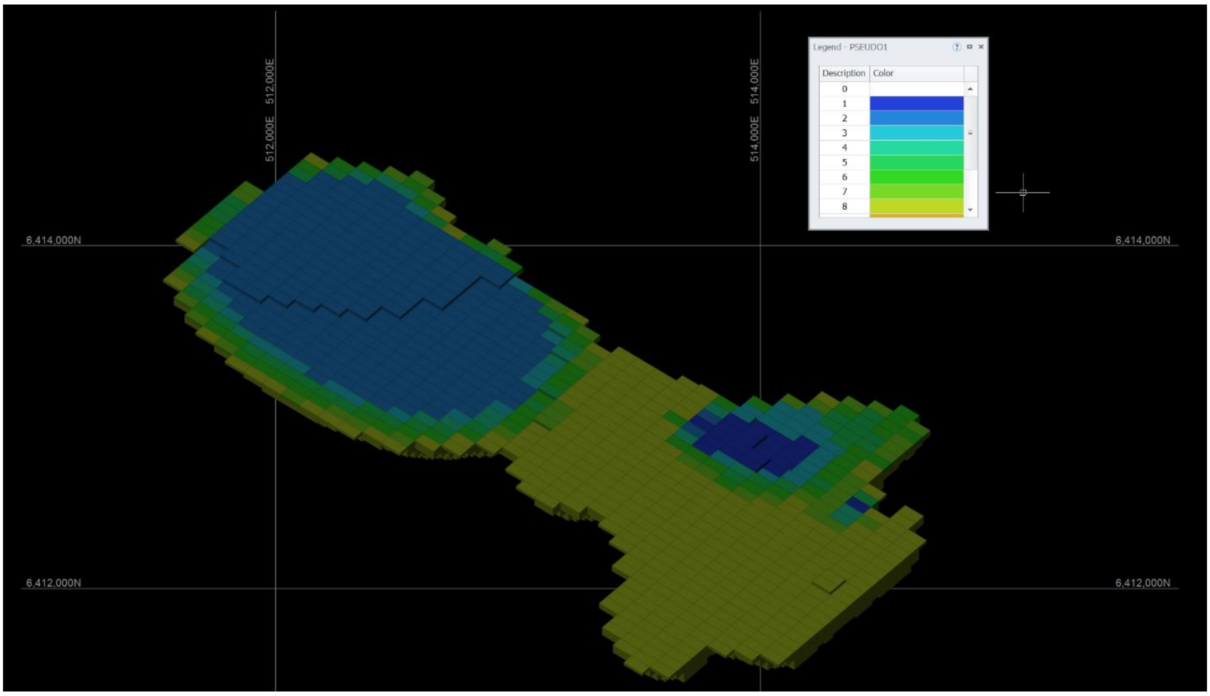

Figure 8. Pit Optimisation by revenue factor (Pit number) for entire deposit

There is a minor amount of material which is sourced from a starter pit in the Fold region in the top right corner. Following this a larger pit is then sunk (Pit shell number 2) in the Core area of the ore body which is progressively developed to a total depth of 510m through to pit shell number 6. The next significant development is in pit shell number 7 where the two separate pits are linked (this is indicated by the light grey pit in Figure 8). The amount of ore and waste is detailed in Figure 9, it should be noted that whilst the optimiser has shown two major pushbacks (being that of pit 2 and 7) in reality these can be done incrementally such that excessive waste won't have to be moved in single years. This will be able to be optimised under a scheduling with real world constraints scenario, following mine design.

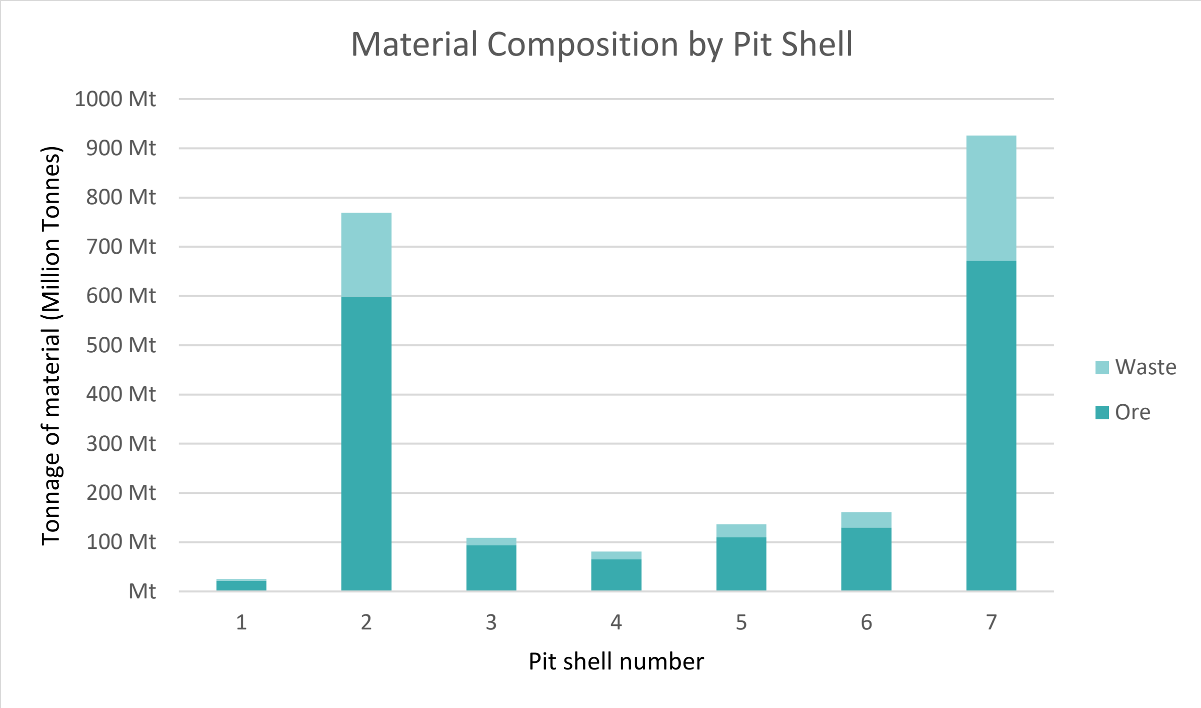

Figure 9. Ore and Waste profile by Pit shell for entire deposit

The concentrate tonnage by year in Figure 10 is consistent with the amount of ore mined per year in Figure 9 with the major amount of concentrate produced in pit shells 2 and 7. The grade of the concentrate is consistently between 69-70% however it is at the lower end of the spectrum in the starter pit (pit shell 1) and when the two pits are linked (pit shell 7), the desire for a higher grade product and a lack of work done on cut-off grade may have driven the initial work around the core area as opposed to the shallower Fold area material. It is recommended that the impacts of developing the resource certainty in the fold area be completed through drilling to allow a complete assessment of what that may mean for the overall pit design as well as earlier year mill feed.

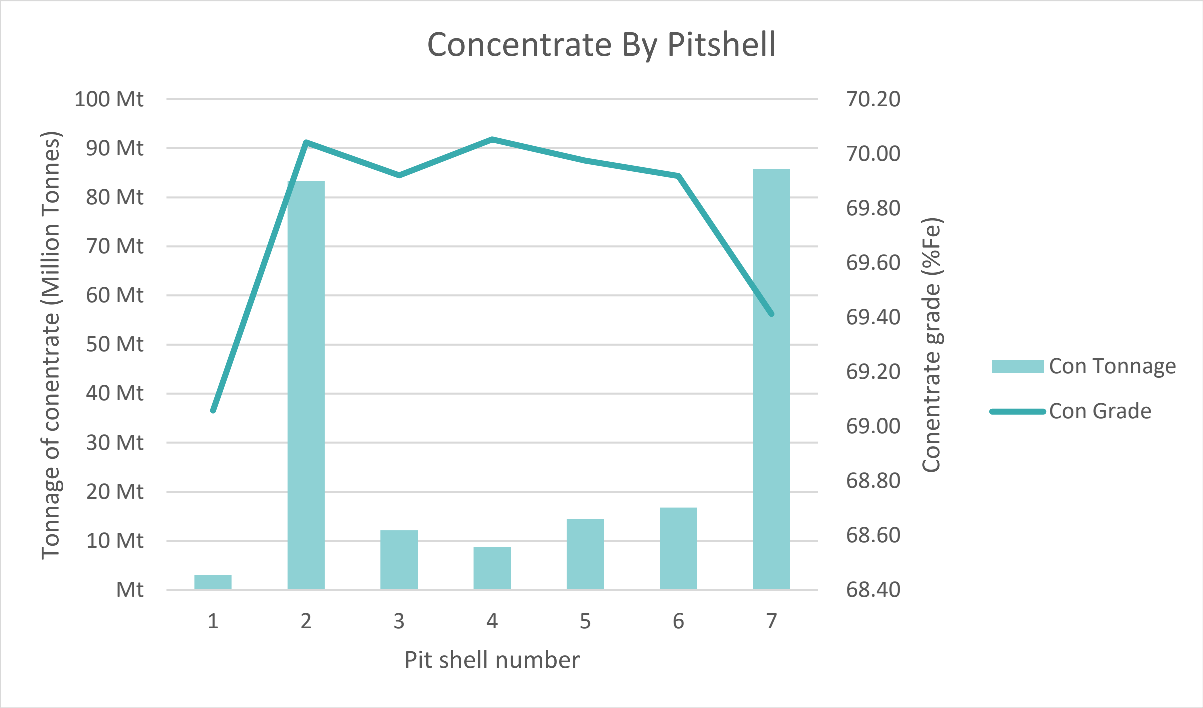

Figure 10. Concentrate tonnage and grade by pit shell for entire deposit

Net Present Values were calculated using a discount rate of 8% and a mining rate of 70 Mt of ore per year with the processing plant being the bottleneck of operations, producing a best case NPV_8 of A$9.99 Bn and a worst case NPV_8 of A$8.18Bn, with an average of NPV_8 of A$9.08Bn. This is shown in Figure 11 with minimal divergence until later stages where larger amounts of inefficiencies can be created through excessive pre-strip.

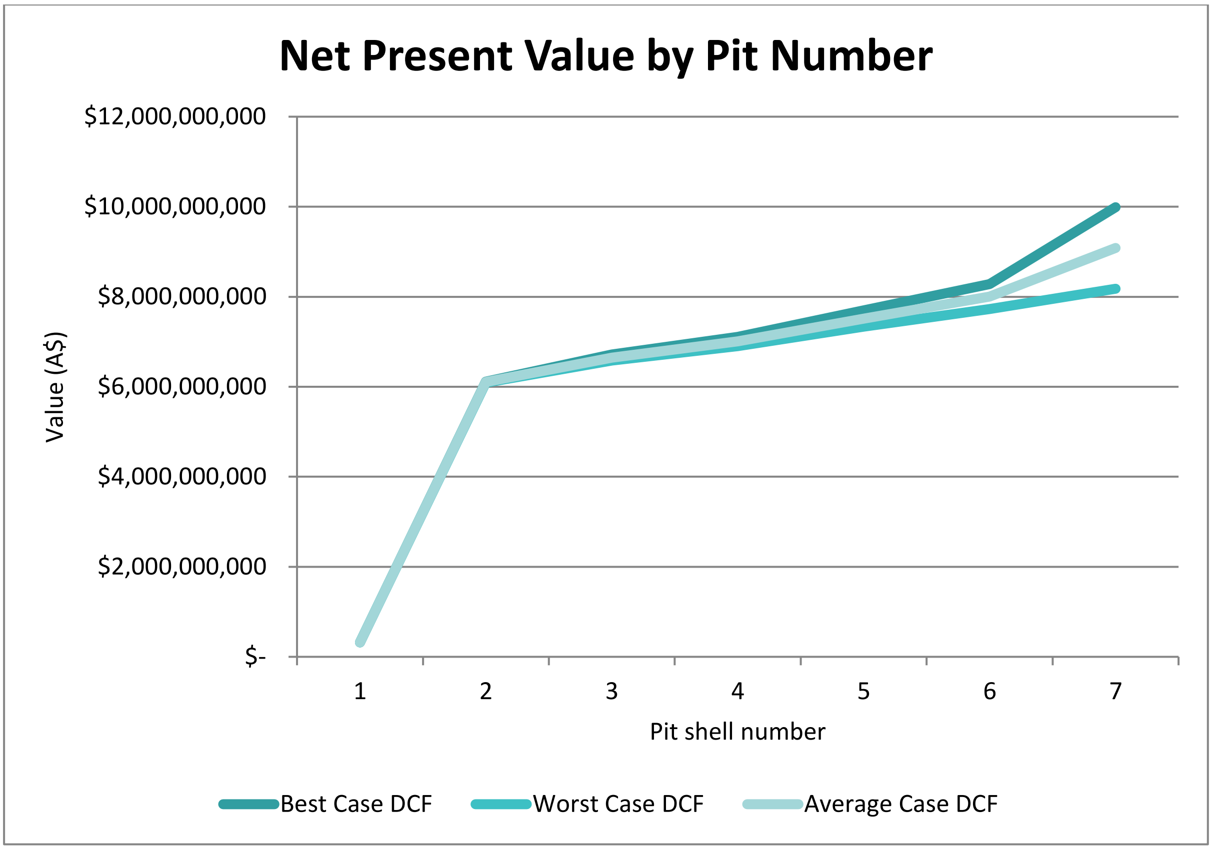

Figure 11. Net Present Value by Pit number (AUD) for non JORC constrained

2.4 JORC Only Optimisation

The steps of 2.3 were repeated to produce a set of similar results with the condition that only Inferred and Indicated JORC resources could be included in the optimisation. This reduces a vast amount of near surface tonnage which as a result requires a revenue factor of 0.44 to reach the required tonnes. The same constraints around inter pits were used with maximum incremental depth at 45m which helped to provide a higher degree of resolution around the pit shells. The resulting pit produced 238Mt of concentrate at a grade of 69.86%, requiring the mining of 566.7Mt of waste and 1,803Mt of ore for a total material movement of 2,370Mt, which results in a strip ratio of 0.31. The outcomes are shown in Figure 12. Note how there is less development around the top right section of the pit. This is because the shallow ore material has not been included in the pit due to its non JORC definition and as a result in this model has been treated as waste. Some is exploited in later years so that the JORC defined material below it can be targeted.

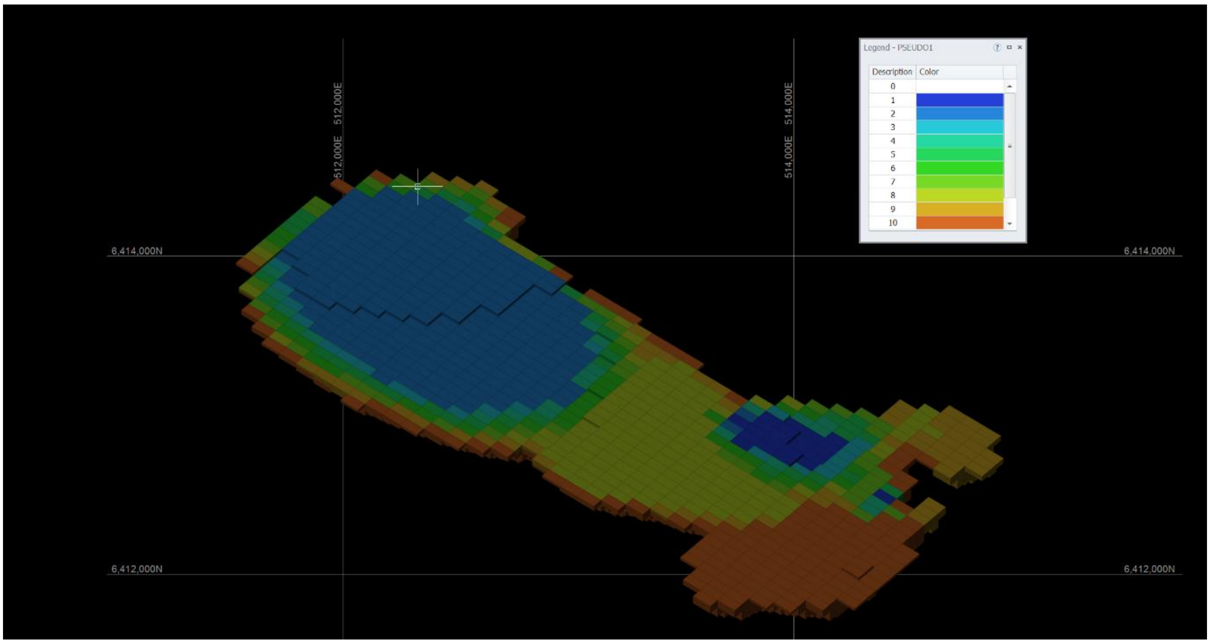

Figure 12. Pit Optimisation by revenue factor (Pit Number) for JORC portion of deposit

As a result the starter pit, shown in green in Figure 12 has a smaller footprint and does not extend as far as the pit in Figure 8 due to the exclusion of exploration target. This also has the consequence of a smaller tonnage leading to a larger revenue factor having to be used for pitshell 2 in Figure 13 with a higher proportion of waste.

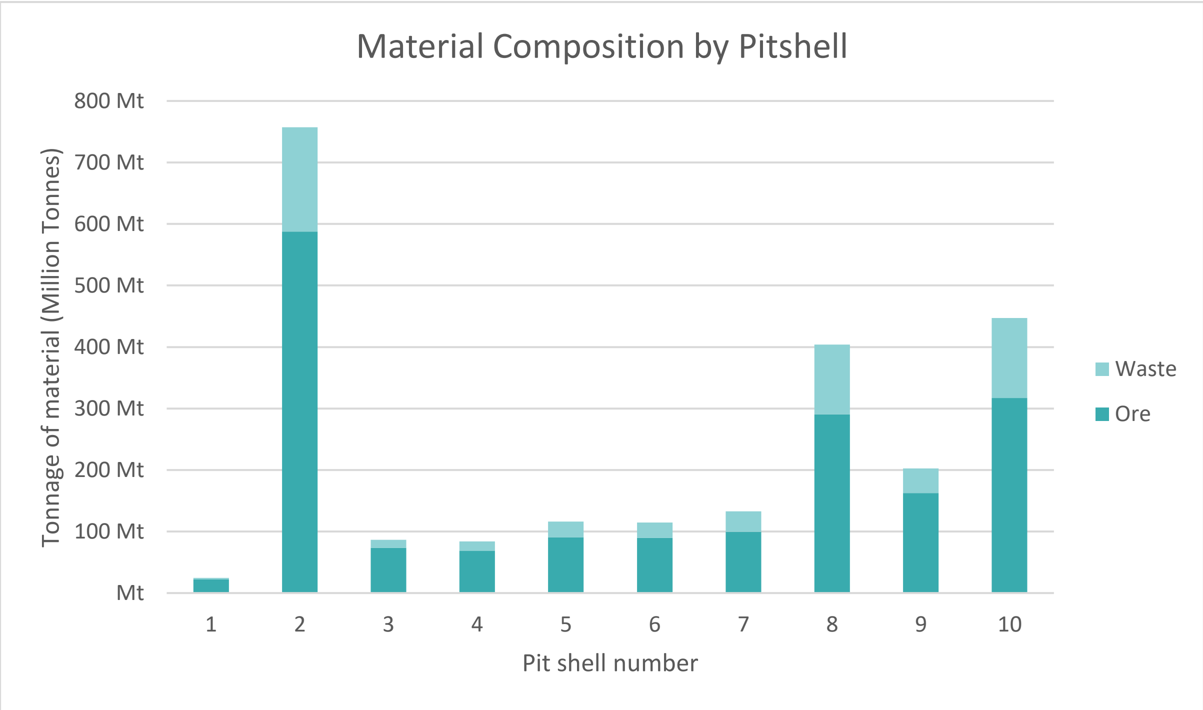

Figure 13. Ore and Waste profile by Pit shell for JORC component of deposit

The concentrate tonnage by year in Figure 14 is consistent with the amount of ore mined per year in Figure 13 with the major amount of concentrate produced in pit shells 2, 8 and 10. The grade of the concentrate is consistently between 69-70% however is at the lower end of the spectrum in the starter pit (pit shell 1, green), when the two pits are linked (pit shell 8, dark red) and when the Fold area material is fed in (pit shell 10, light red), this higher grade product may have driven the initial work around the Core body as opposed to the shallower Fold area material.

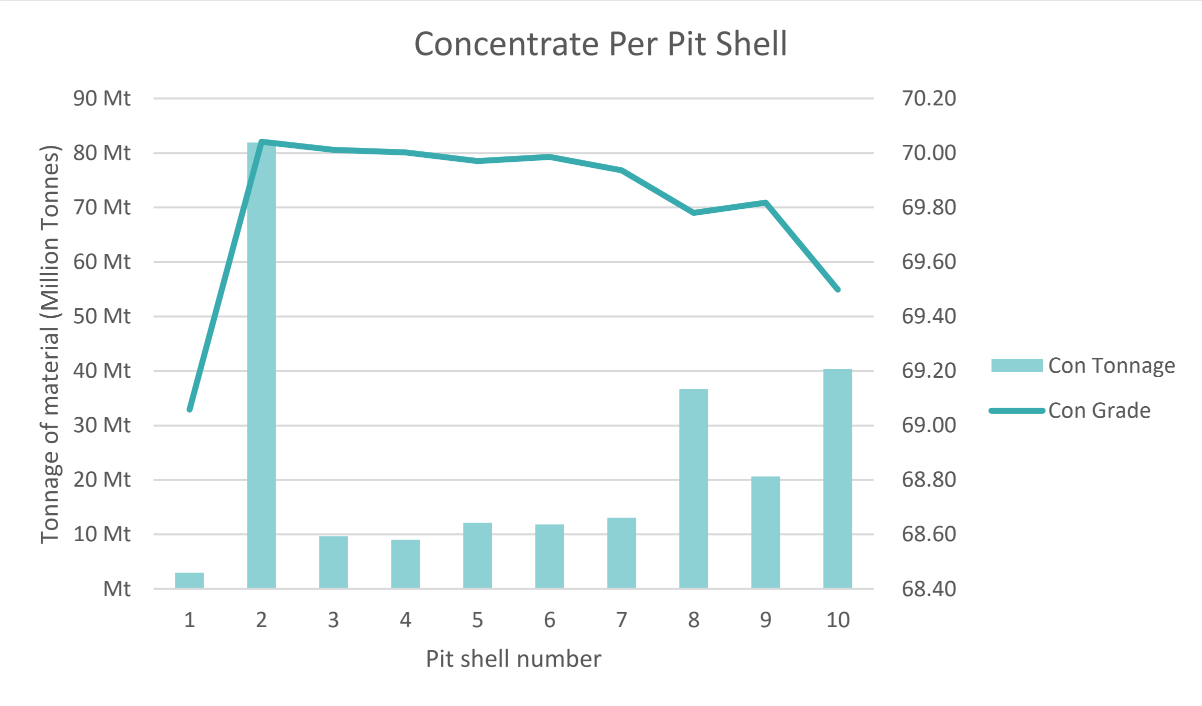

Figure 14. Concentrate tonnage and grade by pit shell for JORC component of deposit

Net Present Values were calculated using a discount rate of 8% and a mining rate of 70 Mt of ore per year with the processing plant being the bottleneck of operations, producing a best case NPV_8 of A$10.3 Bn and a worst case NPV_8 of A$8.26Bn, with an average of NPV_8 of A$9.298Bn. This is shown in Figure 15 with minimal divergence until later stages where larger amounts of inefficiencies can be created through excessive pre-strip.

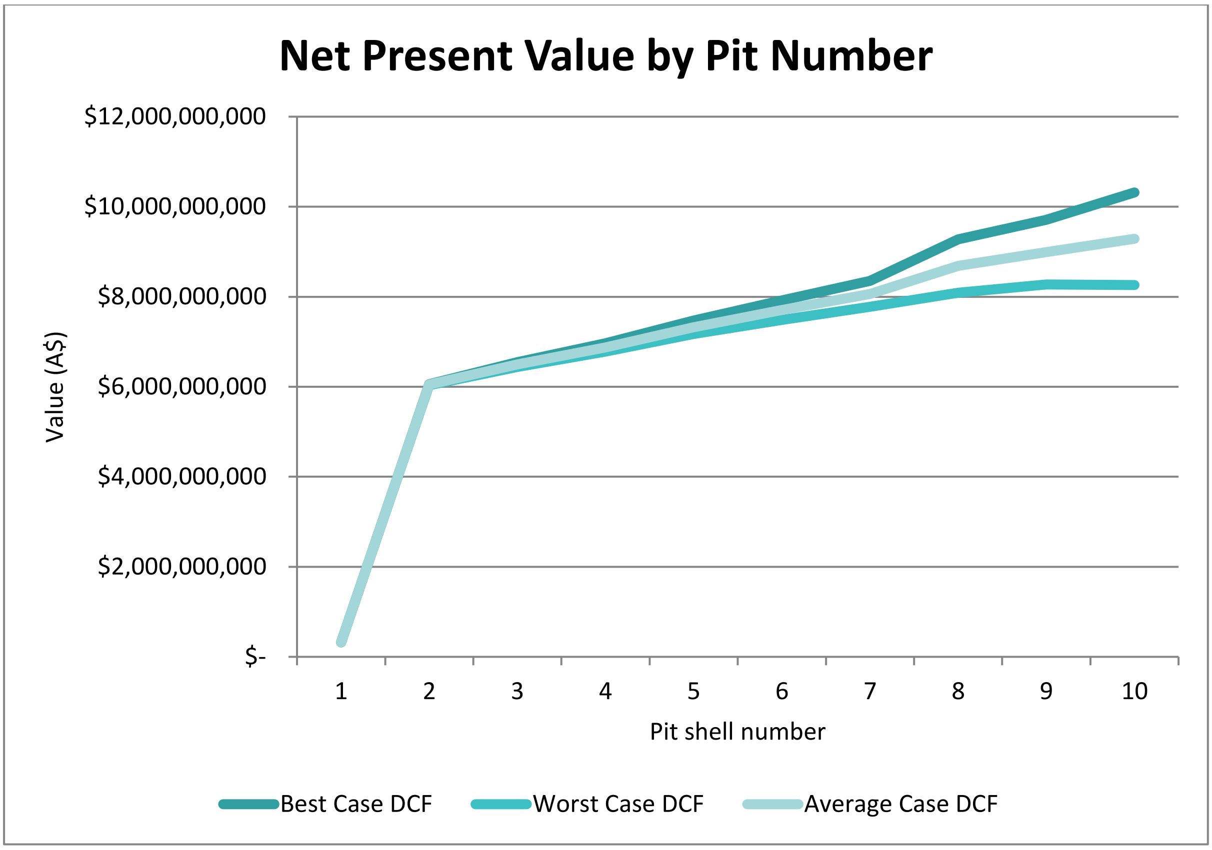

Figure 15. Net Present Value by Pit number (AUD) for JORC component of deposit

3 CUT-OFF ANALYSIS

3.1 INTRODUCTION

With the block model and parameters now set, the next steps were to redefine the ore and the waste around different cut-off grades. This was achieved by changing the definition of ore to waste by strictly using the DTR, rather than calculating it by having the revenue 10% greater than that of the cost.

The pit optimiser was then run 25 separate times between cut-off grades between 14.5% DTR and 2.5% DTR at 0.5% steps each of these producing a result optimised around a pit shell to produce approximately 100Mt of concentrate and 200Mt of concentrate, correlating to a 10-year and 20-year mine life (assuming no ramp up period and mine life measured from commissioned plant (i.e. at 10Mtpa of concentrate).

It was also assumed that the constraining factor in these scenarios was 70Mtpa feed in the mill, at higher cut-off grades this means >10Mtpa would be produced however the mining rate would also be significantly higher due to the amount of overburden having to be removed. Conversely the lower cut-off grades the lower the mining requirement however this results in a lower head grade and subsequent lower number of concentrate tonnes.

The characteristics of the ore deposit are as below in Figure 16 which show that there is a negligible amount of material below 4% DTR with most ore having >12% DTR. It's important to know that each block represents roughly 225kt.

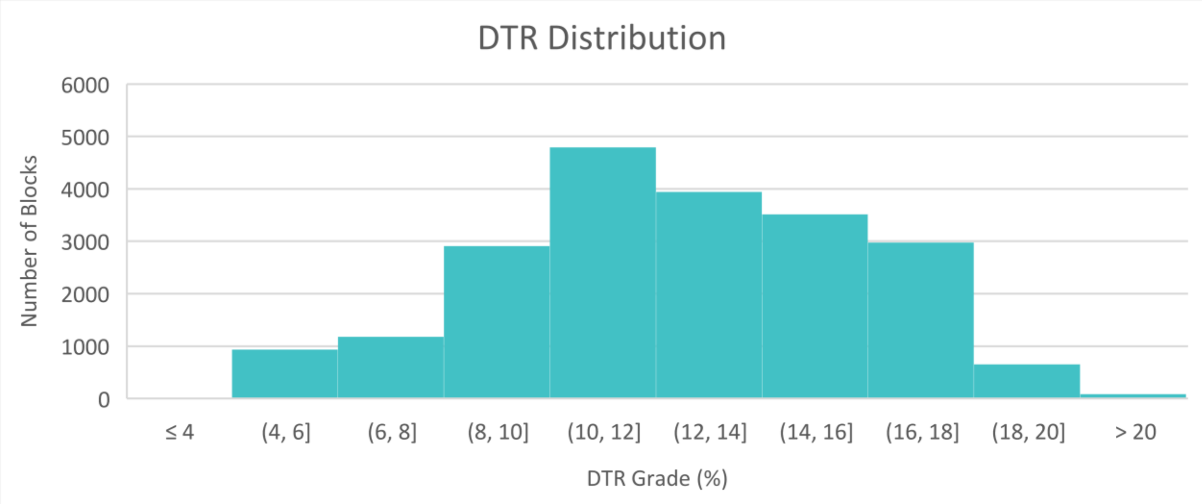

Figure 16. Distribution of DTR values in the block model

3.2 100Mt Concentrate Pits

All pits were optimised for 100Mt using unconstrained ore (such that it incorporated non-JORC "potential" ore) however due to the way the pit optimiser works the revenue factors that generate the pits often overshoot the required tonnage. This is due to the large orebody below the overburden which automatically drives the pit deep which is enabled by the wide dimensions of the orebody. This can make the figures not exactly an apples-to-apples comparison, the most extreme example of this is at 11% DTR where pit 8 (Revenue factor 0.42) was at 96Mt of concentrate whereas pit 9 (Revenue factor 0.43) was at 129Mt. However, trends in the data are clearly visible which are fit for the purpose of this analysis which is to identify areas for future drilling to help drive a process for future pit design and optimisation. A summary of results is shown in Table 2, it is important to note that the mine life is driven by 70Mtpa feed of the plant and in reality, this may not hold true.

Table 2.
Summary of results from 100Mt Concentrate Scenarios

Cutoff	Average Depth	Mine Life	Cash Flow	DCF	Total Material Movement	Ore	Waste	Strip Ratio	DTR	Concentrate
2.5%	171	11.1	$ 10,507,070,918	$ 6,750,648,330	899,572,284	779,633,635	119,938,649	0.15	13%	100,504,870
3.0%	176	12.0	$ 11,460,603,003	$ 6,820,824,954	991,698,085	837,248,936	154,449,149	0.18	13%	109,694,663
3.5%	175	11.8	$ 11,506,903,267	$ 6,418,898,845	1,000,249,360	825,295,736	174,953,624	0.21	13%	109,936,140
4.0%	174	11.4	$ 11,337,525,001	$ 5,931,761,833	984,427,585	797,518,586	186,908,999	0.23	14%	107,972,055
4.5%	174	11.2	$ 11,277,787,362	$ 5,597,130,815	984,427,585	781,488,911	202,938,674	0.26	14%	107,289,440
5.0%	174	11.0	$ 11,255,215,430	$ 5,301,113,484	988,593,460	770,123,936	218,469,524	0.28	14%	107,014,531
5.5%	174	10.8	$ 11,169,779,508	$ 5,015,344,611	986,833,960	756,084,911	230,749,049	0.31	14%	106,172,047
6.0%	174	10.6	$ 11,085,218,228	$ 7,106,279,357	986,467,210	743,701,136	242,766,074	0.33	14%	105,403,527
6.5%	174	10.5	$ 11,017,946,558	$ 7,112,845,202	986,033,860	734,377,886	251,655,974	0.34	14%	104,796,519
7.0%	173	10.1	$ 10,700,397,427	$ 7,028,581,264	964,576,210	706,990,211	257,585,999	0.36	14%	101,802,117
7.5%	173	10.0	$ 10,634,409,575	$ 7,026,585,036	964,576,210	699,282,386	265,293,824	0.38	14%	101,244,435
8.0%	179	11.0	$ 11,702,314,726	$ 7,480,977,460	1,073,305,510	772,169,112	301,136,399	0.39	14%	111,846,683
8.5%	174	9.7	$ 10,376,199,009	$ 6,990,537,873	960,786,310	678,951,686	281,834,624	0.42	15%	99,168,011
9.0%	177	10.3	$ 11,028,042,762	$ 7,291,706,512	1,035,523,585	722,155,212	313,368,374	0.43	15%	105,820,828
9.5%	175	9.7	$ 10,431,260,551	$ 7,064,768,791	991,643,860	679,260,911	312,382,949	0.46	15%	100,260,321
10.0%	175	9.9	$ 10,738,475,044	$ 7,159,182,734	1,036,251,461	695,936,187	340,315,274	0.49	15%	103,533,169
10.5%	177	10.3	$ 11,029,695,407	$ 7,331,145,152	1,109,522,336	719,872,287	389,650,049	0.54	15%	107,532,617
11.0%	171	12.3	$ 13,000,775,175	$ 8,023,837,280	1,416,755,713	863,029,740	553,725,973	0.64	15%	129,406,957
11.5%	177	9.9	$ 10,710,486,117	$ 7,195,891,513	1,148,764,211	693,647,937	455,116,273	0.66	15%	105,961,406
12.0%	177	9.4	$ 10,136,731,626	$ 6,963,927,763	1,132,847,860	654,583,737	478,264,124	0.73	15%	101,310,193
12.5%	169	10.3	$ 11,024,108,868	$ 7,266,747,603	1,352,174,636	723,073,063	629,101,573	0.87	16%	113,065,551
13.0%	162	10.0	$ 10,524,912,028	$ 7,015,185,284	1,405,406,860	698,110,661	707,296,199	1.01	16%	110,758,283
13.5%	175	9.3	$ 9,807,010,748	$ 6,693,571,924	1,371,179,858	651,693,309	719,486,549	1.10	16%	105,035,549
14.0%	149	9.7	$ 9,690,614,917	$ 6,463,477,991	1,654,279,880	680,544,907	973,734,973	1.43	16%	110,913,991
14.5%	153	8.8	$ 8,510,577,986	$ 5,873,724,694	1,635,415,204	617,046,756	1,018,368,448	1.65	17%	102,175,645

Cutoff	Maximum Depth	Mine Life	Cash Flow	DCF	Total Material Movement	Ore	Waste	Strip Ratio	DTR	Concentrate
2.5%	495	11.1	$ 10,507,070,918	$ 6,750,648,330	899,572,284	779,633,635	119,938,649	0.15	13%	100,504,870
3.0%	525	12.0	$ 11,460,603,003	$ 6,820,824,954	991,698,085	837,248,936	154,449,149	0.18	13%	109,694,663
3.5%	525	11.8	$ 11,506,903,267	$ 6,418,898,845	1,000,249,360	825,295,736	174,953,624	0.21	13%	109,936,140
4.0%	510	11.4	$ 11,337,525,001	$ 5,931,761,833	984,427,585	797,518,586	186,908,999	0.23	14%	107,972,055
4.5%	510	11.2	$ 11,277,787,362	$ 5,597,130,815	984,427,585	781,488,911	202,938,674	0.26	14%	107,289,440
5.0%	510	11.0	$ 11,255,215,430	$ 5,301,113,484	988,593,460	770,123,936	218,469,524	0.28	14%	107,014,531
5.5%	510	10.8	$ 11,169,779,508	$ 5,015,344,611	986,833,960	756,084,911	230,749,049	0.31	14%	106,172,047
6.0%	510	10.6	$ 11,085,218,228	$ 7,106,279,357	986,467,210	743,701,136	242,766,074	0.33	14%	105,403,527
6.5%	510	10.5	$ 11,017,946,558	$ 7,112,845,202	986,033,860	734,377,886	251,655,974	0.34	14%	104,796,519
7.0%	510	10.1	$ 10,700,397,427	$ 7,028,581,264	964,576,210	706,990,211	257,585,999	0.36	14%	101,802,117
7.5%	510	10.0	$ 10,634,409,575	$ 7,026,585,036	964,576,210	699,282,386	265,293,824	0.38	14%	101,244,435
8.0%	540	11.0	$ 11,702,314,726	$ 7,480,977,460	1,073,305,510	772,169,112	301,136,399	0.39	14%	111,846,683
8.5%	510	9.7	$ 10,376,199,009	$ 6,990,537,873	960,786,310	678,951,686	281,834,624	0.42	15%	99,168,011
9.0%	525	10.3	$ 11,028,042,762	$ 7,291,706,512	1,035,523,585	722,155,212	313,368,374	0.43	15%	105,820,828
9.5%	510	9.7	$ 10,431,260,551	$ 7,064,768,791	991,643,860	679,260,911	312,382,949	0.46	15%	100,260,321
10.0%	510	9.9	$ 10,738,475,044	$ 7,159,182,734	1,036,251,461	695,936,187	340,315,274	0.49	15%	103,533,169
10.5%	555	10.3	$ 11,029,695,407	$ 7,331,145,152	1,109,522,336	719,872,287	389,650,049	0.54	15%	107,532,617
11.0%	555	12.3	$ 13,000,775,175	$ 8,023,837,280	1,416,755,713	863,029,740	553,725,973	0.64	15%	129,406,957
11.5%	540	9.9	$ 10,710,486,117	$ 7,195,891,513	1,148,764,211	693,647,937	455,116,273	0.66	15%	105,961,406
12.0%	540	9.4	$ 10,136,731,626	$ 6,963,927,763	1,132,847,860	654,583,737	478,264,124	0.73	15%	101,310,193
12.5%	540	10.3	$ 11,024,108,868	$ 7,266,747,603	1,352,174,636	723,073,063	629,101,573	0.87	16%	113,065,551
13.0%	540	10.0	$ 10,524,912,028	$ 7,015,185,284	1,405,406,860	698,110,661	707,296,199	1.01	16%	110,758,283
13.5%	540	9.3	$ 9,807,010,748	$ 6,693,571,924	1,371,179,858	651,693,309	719,486,549	1.10	16%	105,035,549
14.0%	465	9.7	$ 9,690,614,917	$ 6,463,477,991	1,654,279,880	680,544,907	973,734,973	1.43	16%	110,913,991
14.5%	510	8.8	$ 8,510,577,986	$ 5,873,724,694	1,635,415,204	617,046,756	1,018,368,448	1.65	17%	102,175,645

One of the biggest drivers the cut-off grade has is moving the average grade of the material up and increasing the strip ratio (and therefore the total amount of material movement required). This is shown in Figure 17 which holds true despite the varying total concentrate tonnage. The average grade varies from 12.9% to 16.6% from a DTR cut-off of 2.5% to 14.5% respectfully.

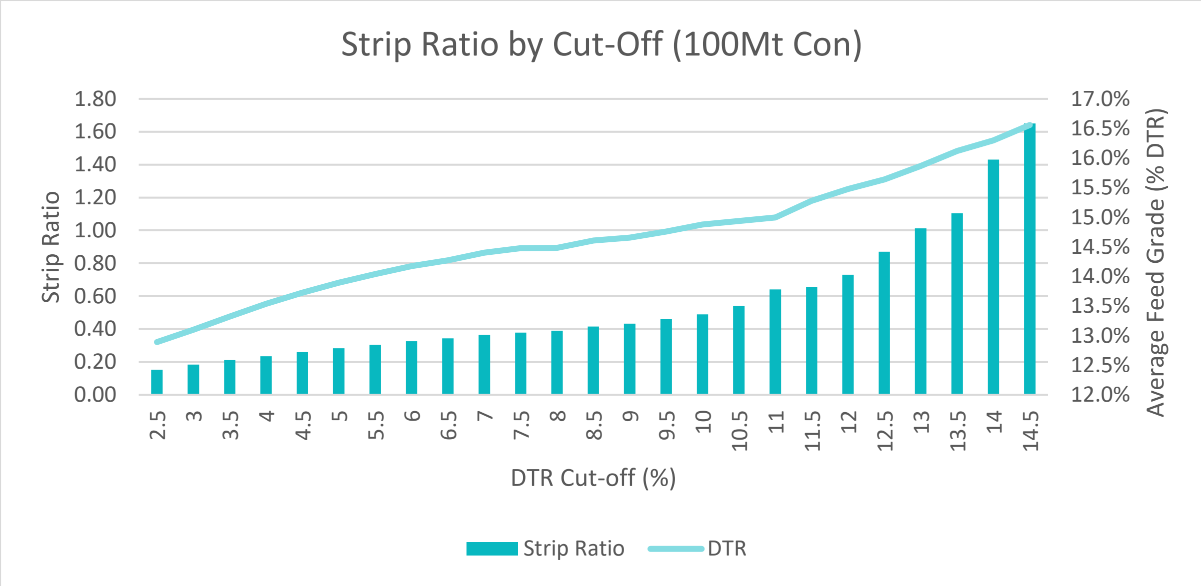

Figure 17. Strip ratio and average feed grade against the DTR Cut-off used 100Mt Con

The most important outcome of this exercise was assessing the discounted cashflow to assess if there is a major change in NPV and what the best Cut-off grade is. This produced some interesting results which are shown in Figure 18 and Figure 19. Most noticeably that there was minimal effect on the NPV between a cut-off of 6.5% to 12.5% (the variations being driven by larger or smaller pit shells). At 5.5% there is a sharp drop in NPV however as the breakeven price for an ore block is approached (at 3.5% DTR) there is a boost in NPV. This is likely a trade-off between early year feeding and not incurring the prestrip without revenue and reduced revenue/opportunity cost of not feeding the mill with higher quality material. This trade-off will be better understood with scheduling although it clearly shows the benefit of a dynamic cut-off grade depending on the commodity price, and when combined with strategic stockpiling will enable a boosted NPV.

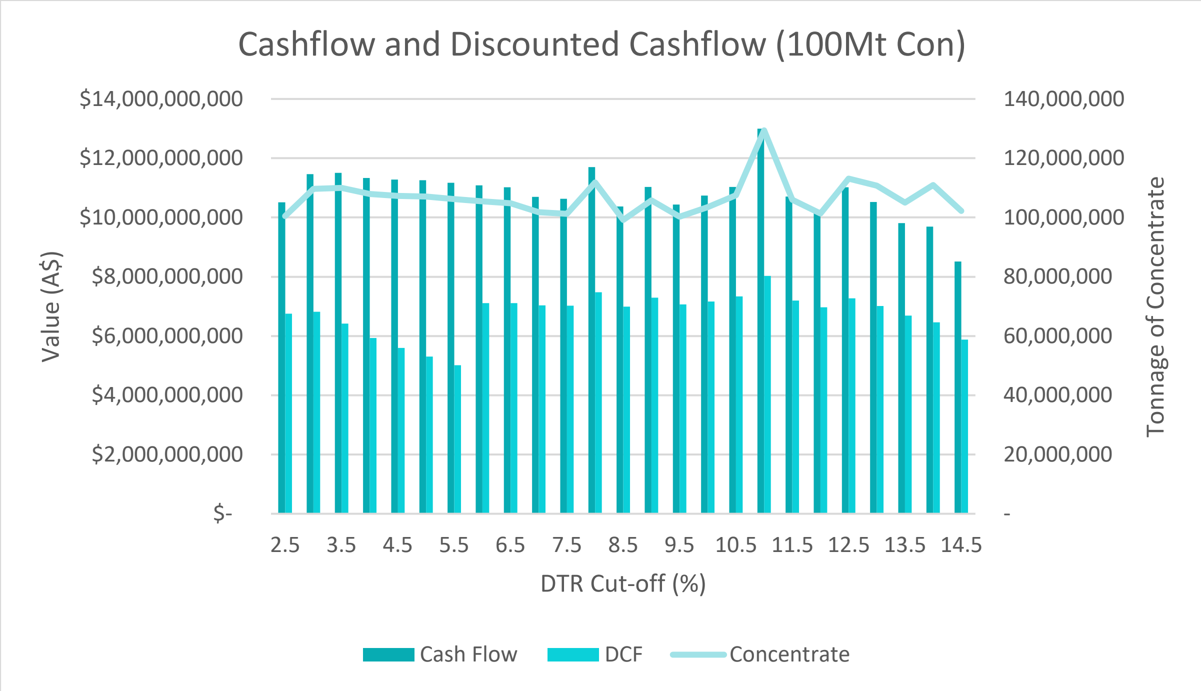

Figure 18. Cashflow, discounted cashflow and total tonnage in pit shell vs DTR Cut-off 100Mt Con

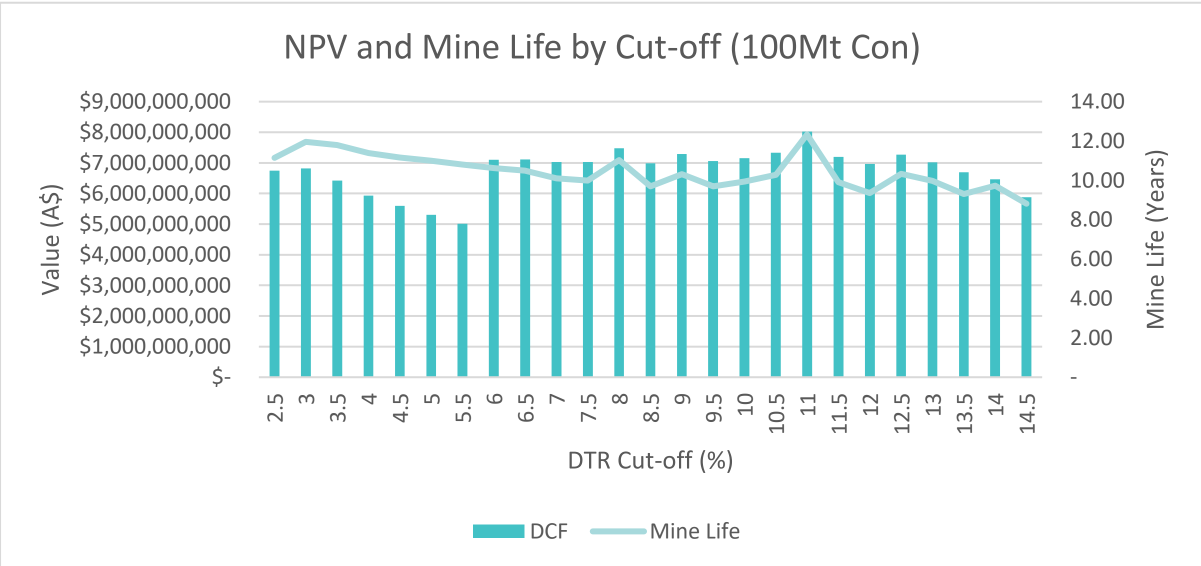

Figure 19. Discounted Cashflow and mine life (70Mtpa ore feed) vs DTR Cut-Off 100Mt Con

The next parameter assessed was the average depth of the material and total tonnage of ore mined vs the DTR Cut-off. In Figure 20 we can see the average depth of the pit is largely unaffected by the DTR cut-off with the exception of the high-grade cut-off which is driven by leaving some of the deeper lower grade material behind.

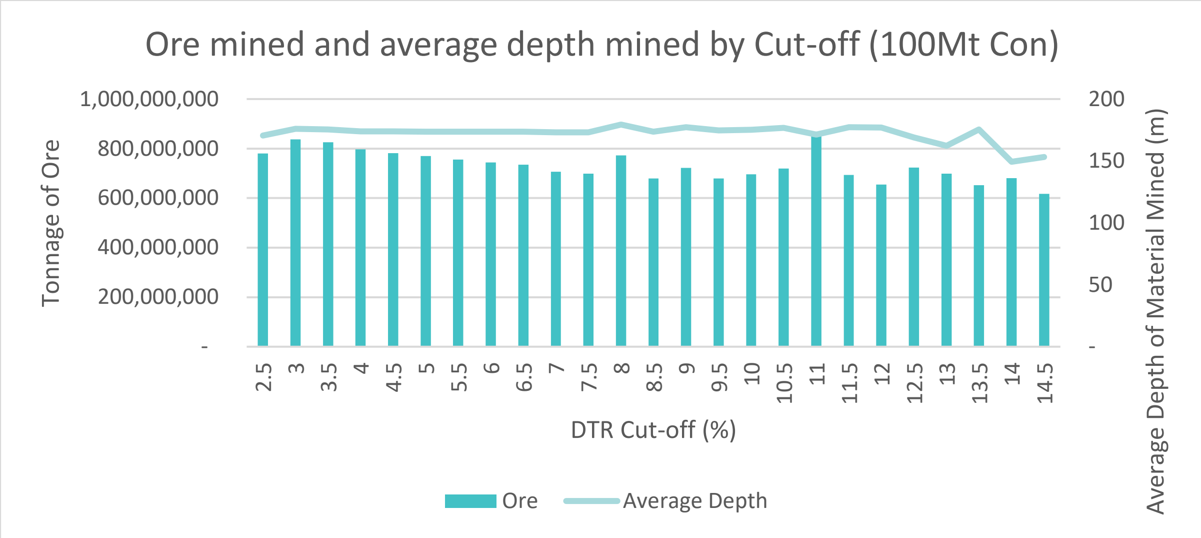

Figure 20. Ore mined and average depth against DTR Cut-off 100Mt Con

The costs and proportion of costs are relatively constant due to the definition of these variables in the block model. The main variable is the mining cost which is driven by the depth of the material but also the total material moved. It is noteworthy that as the strip ratio increases so does the total mining cost. Therefore, outcropping low grade shallow ore can be profitable and boost the NPV of the project. The lowest DTR cut-off produced the pit with the highest proportion of indicated material, however there is no significant change until DTR 11%, these results are shown in Table 3 and outlined in Figure 21.

Table 3.
Summary of costs and sources of ore from 100Mt Concentrate Scenarios

	Costs						Ore Source		
Cutoff	Mining	Processing	Transport + Downstream	G+A + Rehab + Sustaining Capital	Total	Revenue	Indicated	Inferred	Potential
2.5%	$ 2,028,068,214	$ 1,216,228,470	$ 1,294,191,834	$ 1,394,337,041	$ 5,932,825,559	$ 16,439,896,477	73%	26%	1%
3.0%	$ 2,251,600,173	$ 1,306,108,340	$ 1,389,833,233	$ 1,537,132,031	$ 6,484,673,777	$ 17,945,276,780	71%	29%	1%
3.5%	$ 2,269,523,174	$ 1,287,461,348	$ 1,369,990,922	$ 1,550,386,508	$ 6,477,361,951	$ 17,984,265,218	70%	29%	1%
4.0%	$ 2,229,088,043	$ 1,244,128,994	$ 1,323,880,853	$ 1,525,862,757	$ 6,322,960,646	$ 17,660,485,647	72%	27%	1%
4.5%	$ 2,229,088,043	$ 1,219,122,701	$ 1,297,271,592	$ 1,525,862,757	$ 6,271,345,093	$ 17,549,132,455	72%	27%	1%
5.0%	$ 2,237,987,220	$ 1,201,393,340	$ 1,278,405,734	$ 1,532,319,863	$ 6,250,106,157	$ 17,505,321,587	72%	27%	1%
5.5%	$ 2,234,008,072	$ 1,179,492,461	$ 1,255,100,952	$ 1,529,592,638	$ 6,198,194,124	$ 17,367,973,631	72%	27%	1%
6.0%	$ 2,232,906,439	$ 1,160,173,772	$ 1,234,543,886	$ 1,529,024,175	$ 6,156,648,272	$ 17,241,866,500	72%	27%	1%
6.5%	$ 2,232,042,634	$ 1,145,629,502	$ 1,219,067,291	$ 1,528,352,483	$ 6,125,091,910	$ 17,143,038,467	72%	27%	1%
7.0%	$ 2,182,401,751	$ 1,102,904,729	$ 1,173,603,750	$ 1,495,093,125	$ 5,954,003,355	$ 16,654,400,782	72%	27%	1%
7.5%	$ 2,182,401,751	$ 1,090,880,522	$ 1,160,808,760	$ 1,495,093,125	$ 5,929,184,158	$ 16,563,593,733	72%	27%	1%
8.0%	$ 2,448,186,165	$ 1,204,583,814	$ 1,281,800,726	$ 1,663,623,541	$ 6,598,194,246	$ 18,300,508,971	70%	29%	1%
8.5%	$ 2,175,284,737	$ 1,059,164,630	$ 1,127,059,799	$ 1,489,218,780	$ 5,850,727,946	$ 16,226,926,955	72%	28%	0%
9.0%	$ 2,355,007,063	$ 1,126,562,130	$ 1,198,777,651	$ 1,605,061,557	$ 6,285,408,401	$ 17,313,451,164	70%	29%	1%
9.5%	$ 2,247,998,671	$ 1,059,647,021	$ 1,127,573,113	$ 1,537,047,983	$ 5,972,266,788	$ 16,403,527,339	71%	29%	0%
10.0%	$ 2,350,944,965	$ 1,085,660,452	$ 1,155,254,070	$ 1,606,189,764	$ 6,198,049,252	$ 16,936,524,295	71%	28%	1%
10.5%	$ 2,522,373,988	$ 1,123,000,768	$ 1,194,987,997	$ 1,719,759,621	$ 6,560,122,374	$ 17,589,817,781	68%	31%	1%
11.0%	$ 3,197,695,511	$ 1,346,326,395	$ 1,432,629,369	$ 2,195,971,356	$ 8,172,622,631	$ 21,173,397,806	65%	33%	2%
11.5%	$ 2,613,213,906	$ 1,082,090,782	$ 1,151,455,576	$ 1,780,584,527	$ 6,627,344,791	$ 17,337,830,908	67%	31%	2%
12.0%	$ 2,575,442,302	$ 1,021,150,629	$ 1,086,609,003	$ 1,755,914,183	$ 6,439,116,118	$ 16,575,847,744	68%	31%	2%
12.5%	$ 3,042,556,628	$ 1,127,993,978	$ 1,200,301,284	$ 2,095,870,686	$ 7,466,722,577	$ 18,490,831,444	67%	31%	2%
13.0%	$ 3,133,975,628	$ 1,089,052,631	$ 1,158,863,697	$ 2,178,380,633	$ 7,560,272,589	$ 18,085,184,617	65%	34%	1%
13.5%	$ 3,111,123,061	$ 1,016,641,563	$ 1,081,810,893	$ 2,125,328,780	$ 7,334,904,297	$ 17,141,915,045	63%	34%	3%
14.0%	$ 3,626,607,430	$ 1,061,650,055	$ 1,129,704,545	$ 2,564,133,813	$ 8,382,095,843	$ 18,072,710,760	63%	37%	1%
14.5%	$ 3,603,824,135	$ 962,592,939	$ 1,024,297,614	$ 2,534,893,566	$ 8,125,608,254	$ 16,636,186,240	62%	37%	1%

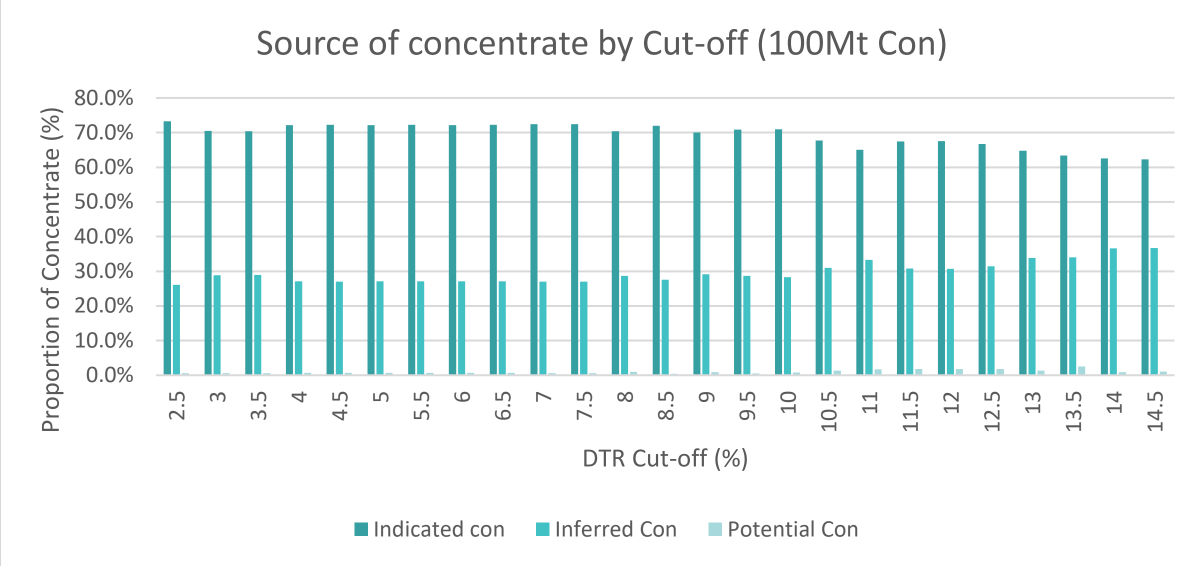

Figure 21. Proportion of feedstock by ore classification 100Mt Con

3.3 200Mt Concentrate Pits

All pits were optimised for 200Mt using unconstrained ore (such that it incorporated non-JORC "potential" ore) however due to the way the pit optimiser works the revenue factors that generate the pits often overshoot the required tonnage. This is due to the large orebody below the overburden which automatically drives the pit deep which is enabled by the wide dimensions of the orebody. This can make figures not exactly an apples-to-apples comparison, however this effect is minimised comparatively to the 100Mt scenario due to the time value of money and the longer mine life mitigating some of these effects. the most extreme example of this is at 4% DTR where pit 7 (Revenue factor 0.41) was at 155Mt of concentrate whereas pit 8 (Revenue factor 0.42) was at 243Mt. However, trends in the data are clearly visible which are fit for the purpose of this analysis which is to identify areas for future drilling to help drive a process for future pit design and optimisation. A summary of results is shown in Table 4, it is important to note that the mine life is driven by 70Mtpa feed of the plant and, this may not hold true.

Table 4.
Summary of results from 200Mt Concentrate Scenarios

Cutoff	Average Depth	Mine Life	Cash Flow	DCF	Total Material Movement	Ore	Waste	Strip Ratio	DTR	Concentrate
2.5%	174	27.8	$ 24,343,689,050	$ 9,586,928,307	2,348,103,313	1,946,128,966	401,974,348	0.21	12%	242,722,955
3.0%	169	25.6	$ 22,851,124,747	$ 8,374,056,105	2,199,341,339	1,793,713,441	405,627,898	0.23	13%	226,903,724
3.5%	169	25.1	$ 22,857,492,742	$ 7,521,515,699	2,205,629,114	1,758,423,767	447,205,347	0.25	13%	226,390,482
4.0%	174	26.6	$ 24,513,773,881	$ 6,789,643,790	2,402,660,188	1,863,304,216	539,355,972	0.29	13%	243,462,406
4.5%	172	24.2	$ 22,728,672,227	$ 6,173,998,261	2,208,085,589	1,695,133,141	512,952,447	0.30	13%	224,523,529
5.0%	172	23.8	$ 22,603,968,280	$ 5,645,664,839	2,208,934,289	1,666,345,291	542,588,997	0.33	13%	223,203,618
5.5%	172	23.4	$ 22,410,826,760	$ 5,190,224,187	2,203,745,039	1,634,985,392	568,759,647	0.35	14%	221,279,804
6.0%	174	25.0	$ 23,983,607,755	$ 9,556,786,816	2,413,460,038	1,746,580,441	666,879,597	0.38	14%	238,074,703
6.5%	174	24.6	$ 23,799,536,779	$ 9,624,193,234	2,410,776,238	1,721,143,741	689,632,497	0.40	14%	236,371,008
7.0%	173	24.3	$ 23,565,024,246	$ 9,669,346,747	2,402,136,538	1,697,909,867	704,226,672	0.41	14%	234,219,173
7.5%	174	23.4	$ 22,905,365,755	$ 9,661,317,801	2,343,905,189	1,640,453,042	703,452,147	0.43	14%	227,715,628
8.0%	172	22.4	$ 22,107,403,124	$ 9,606,725,261	2,256,906,538	1,565,610,916	691,295,622	0.44	14%	219,301,713
8.5%	169	21.4	$ 21,201,032,175	$ 9,514,945,983	2,201,352,089	1,496,063,417	705,288,672	0.47	14%	210,977,234
9.0%	171	20.9	$ 20,857,914,626	$ 9,541,793,159	2,215,551,463	1,466,086,291	749,465,172	0.51	14%	208,738,711
9.5%	170	20.7	$ 20,672,125,546	$ 9,564,506,454	2,240,724,013	1,447,946,191	792,777,822	0.55	14%	207,778,600
10.0%	170	19.8	$ 20,028,206,989	$ 9,493,231,221	2,204,930,713	1,387,588,217	817,342,497	0.59	15%	201,779,791
10.5%	170	19.7	$ 19,945,471,043	$ 9,502,070,043	2,273,429,713	1,379,322,991	894,106,722	0.65	15%	202,665,313
11.0%	175	20.6	$ 20,950,266,827	$ 9,740,493,410	2,444,549,939	1,443,580,743	1,000,969,196	0.69	15%	214,281,199
11.5%	179	20.0	$ 20,385,795,945	$ 9,637,594,461	2,477,638,138	1,396,896,317	1,080,741,821	0.77	15%	210,768,022
12.0%	183	19.4	$ 19,795,940,390	$ 9,533,805,003	2,546,907,387	1,359,273,841	1,187,633,546	0.87	15%	208,202,328
12.5%	187	18.6	$ 18,757,924,721	$ 9,276,232,618	2,601,712,436	1,299,786,615	1,301,925,821	1.00	16%	202,107,705
13.0%	186	18.4	$ 18,154,851,128	$ 8,976,149,108	2,815,667,260	1,288,203,164	1,527,464,096	1.19	16%	203,378,520
13.5%	190	18.1	$ 17,314,075,264	$ 8,578,052,045	3,018,470,631	1,266,488,111	1,751,982,520	1.38	16%	202,963,233
14.0%	197	17.7	$ 15,843,522,155	$ 7,901,991,342	3,343,254,904	1,236,796,360	2,106,458,543	1.70	16%	201,608,312
14.5%	206	17.2	$ 13,661,792,963	$ 6,830,161,223	3,830,421,300	1,205,494,658	2,624,926,641	2.18	17%	200,003,458

Cutoff	Maximum Depth	Mine Life	Cash Flow	DCF	Total Material Movement	Ore	Waste	Strip Ratio	DTR	Concentrate
2.5%	555	27.8	$ 24,343,689,050	$ 9,586,928,307	2,348,103,313	1,946,128,966	401,974,348	0.21	12%	242,722,955
3.0%	555	25.6	$ 22,851,124,747	$ 8,374,056,105	2,199,341,339	1,793,713,441	405,627,898	0.23	13%	226,903,724
3.5%	555	25.1	$ 22,857,492,742	$ 7,521,515,699	2,205,629,114	1,758,423,767	447,205,347	0.25	13%	226,390,482
4.0%	555	26.6	$ 24,513,773,881	$ 6,789,643,790	2,402,660,188	1,863,304,216	539,355,972	0.29	13%	243,462,406
4.5%	555	24.2	$ 22,728,672,227	$ 6,173,998,261	2,208,085,589	1,695,133,141	512,952,447	0.30	13%	224,523,529
5.0%	555	23.8	$ 22,603,968,280	$ 5,645,664,839	2,208,934,289	1,666,345,291	542,588,997	0.33	13%	223,203,618
5.5%	555	23.4	$ 22,410,826,760	$ 5,190,224,187	2,203,745,039	1,634,985,392	568,759,647	0.35	14%	221,279,804
6.0%	555	25.0	$ 23,983,607,755	$ 9,556,786,816	2,413,460,038	1,746,580,441	666,879,597	0.38	14%	238,074,703
6.5%	555	24.6	$ 23,799,536,779	$ 9,624,193,234	2,410,776,238	1,721,143,741	689,632,497	0.40	14%	236,371,008
7.0%	555	24.3	$ 23,565,024,246	$ 9,669,346,747	2,402,136,538	1,697,909,867	704,226,672	0.41	14%	234,219,173
7.5%	555	23.4	$ 22,905,365,755	$ 9,661,317,801	2,343,905,189	1,640,453,042	703,452,147	0.43	14%	227,715,628
8.0%	555	22.4	$ 22,107,403,124	$ 9,606,725,261	2,256,906,538	1,565,610,916	691,295,622	0.44	14%	219,301,713
8.5%	540	21.4	$ 21,201,032,175	$ 9,514,945,983	2,201,352,089	1,496,063,417	705,288,672	0.47	14%	210,977,234
9.0%	555	20.9	$ 20,857,914,626	$ 9,541,793,159	2,215,551,463	1,466,086,291	749,465,172	0.51	14%	208,738,711
9.5%	555	20.7	$ 20,672,125,546	$ 9,564,506,454	2,240,724,013	1,447,946,191	792,777,822	0.55	14%	207,778,600
10.0%	555	19.8	$ 20,028,206,989	$ 9,493,231,221	2,204,930,713	1,387,588,217	817,342,497	0.59	15%	201,779,791
10.5%	555	19.7	$ 19,945,471,043	$ 9,502,070,043	2,273,429,713	1,379,322,991	894,106,722	0.65	15%	202,665,313
11.0%	555	20.6	$ 20,950,266,827	$ 9,740,493,410	2,444,549,939	1,443,580,743	1,000,969,196	0.69	15%	214,281,199
11.5%	555	20.0	$ 20,385,795,945	$ 9,637,594,461	2,477,638,138	1,396,896,317	1,080,741,821	0.77	15%	210,768,022
12.0%	555	19.4	$ 19,795,940,390	$ 9,533,805,003	2,546,907,387	1,359,273,841	1,187,633,546	0.87	15%	208,202,328
12.5%	555	18.6	$ 18,757,924,721	$ 9,276,232,618	2,601,712,436	1,299,786,615	1,301,925,821	1.00	16%	202,107,705
13.0%	555	18.4	$ 18,154,851,128	$ 8,976,149,108	2,815,667,260	1,288,203,164	1,527,464,096	1.19	16%	203,378,520
13.5%	555	18.1	$ 17,314,075,264	$ 8,578,052,045	3,018,470,631	1,266,488,111	1,751,982,520	1.38	16%	202,963,233
14.0%	555	17.7	$ 15,843,522,155	$ 7,901,991,342	3,343,254,904	1,236,796,360	2,106,458,543	1.70	16%	201,608,312
14.5%	545	17.2	$ 13,661,792,963	$ 6,830,161,223	3,830,421,300	1,205,494,658	2,624,926,641	2.18	17%	200,003,458

Here max depth is static.

Again, the biggest drivers the cut-off grade has is moving the average grade of the material up and increasing the strip ratio (and therefore the total amount of material movement required). This is shown in Figure 22 which holds true despite the varying total concentrate tonnage. The average grade varies from 12.5% to 16.6% from a DTR cut-off of 2.5% to 14.5% respectfully. The average grade between the 100Mt of concentrate and 200Mt concentrate scenarios is quite similar due to the abundance of ore, which is relatively homogeneous, at the lower DTR Cut-off grades there is a slight reduction in grades due to a larger surface footprint and incorporating more lower grade shallow ore.

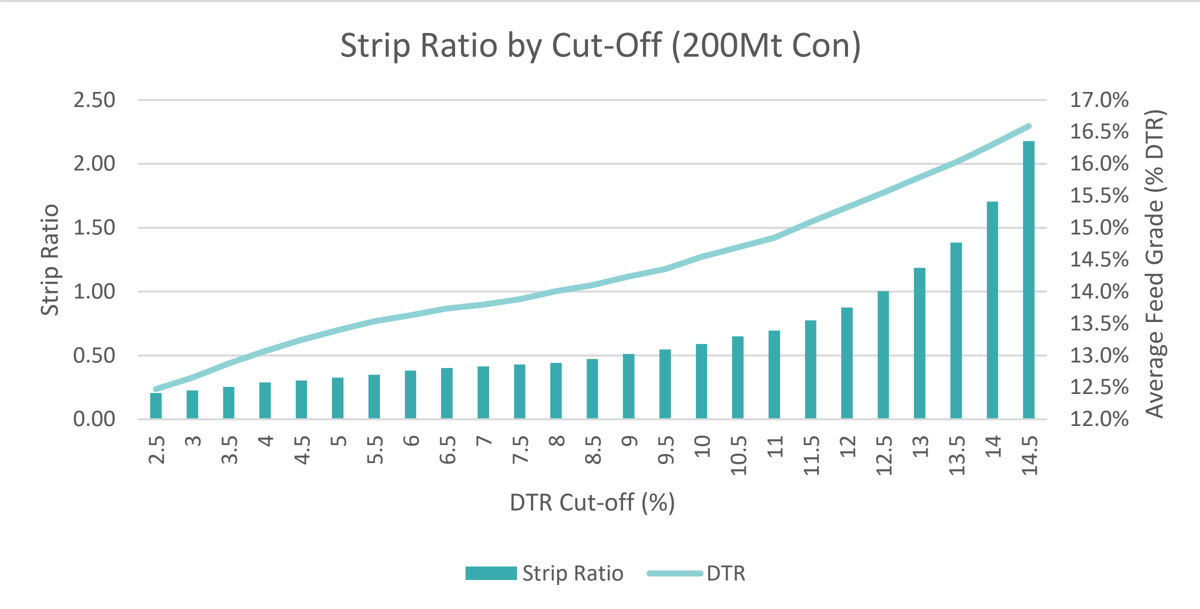

Figure 22. Strip ratio and average feed grade against the DTR Cut-off used 200Mt Con

The NPV analysis for the 200Mt scenario was largely like that of the 100Mt scenario. The 200Mt scenario produced results which were less sensitive to cut-off grade, this is likely driven by the longer mine life and reduced impact of variations in total concentrate, this is illustrated in Figure 23 and Figure 24. Most noticeably that there was minimal effect on the NPV between a cut-off of 6.5% to 13.5% (the variations being driven by larger or smaller pit shells). Like the 100Mt concentrate scenario at 5.5% there is a sharp drop in NPV however as the breakeven price for an ore block is approached (at 3.5% DTR) there is a boost in NPV. This is likely a trade-off between early year feeding and not incurring the prestrip without revenue and reduced revenue/opportunity cost of not feeding the mill with higher quality material. This trade-off will be better understood with scheduling although it clearly shows the benefit of a dynamic cut-off grade depending on the commodity price, and when combined with strategic stockpiling will enable a boosted NPV.

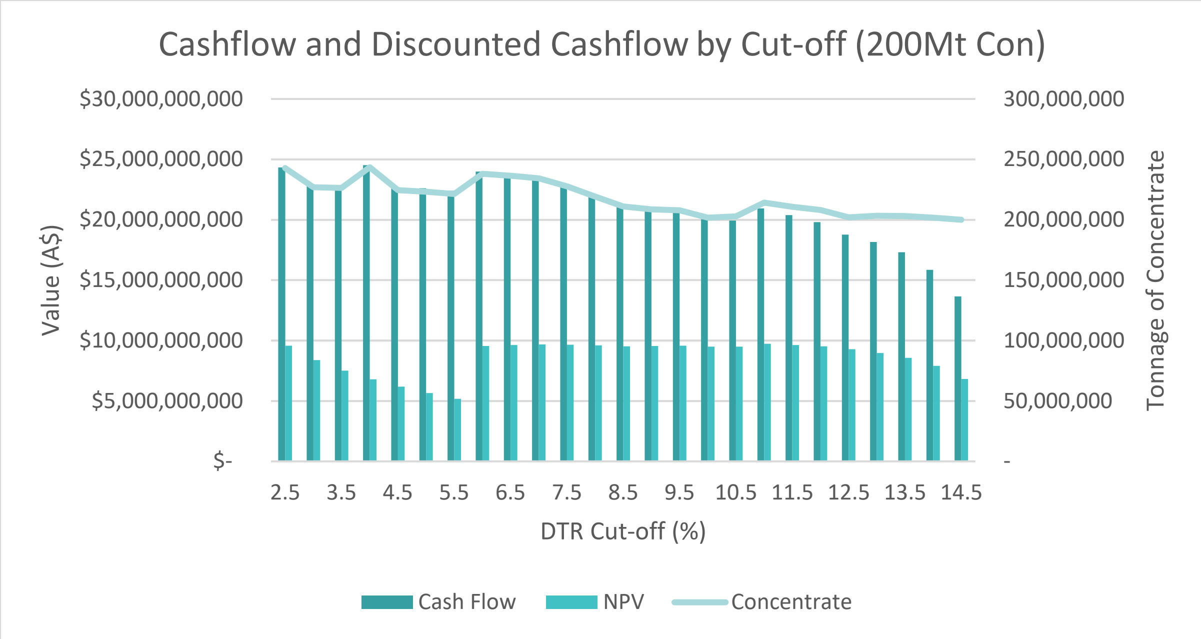

Figure 23. Cashflow, discounted cashflow and total tonnage in pitshell vs DTR Cut-off 200Mt Con

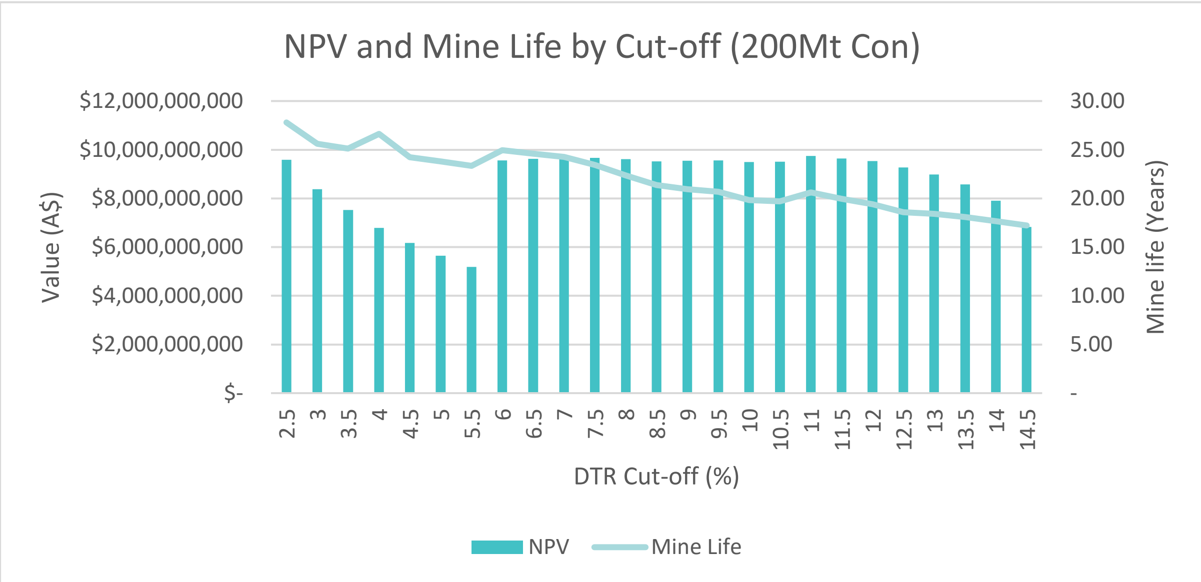

Figure 24. Discounted Cashflow and mine life (70Mtpa ore feed) vs DTR Cut-Off 200Mt Con

The next parameter assessed was the average depth of the material and total tonnage of ore mined vs the DTR Cut-off. In Figure 25 we can see the average depth of the pit is largely unaffected by the DTR cut-off except for cut-off grade above 10.5%. Interestingly this is the opposite relationship shown to the 100Mt scenario. This is likely due to the ore being restricted to the two high grade shoots which to get the required 200Mt of concentrate must be followed deep.

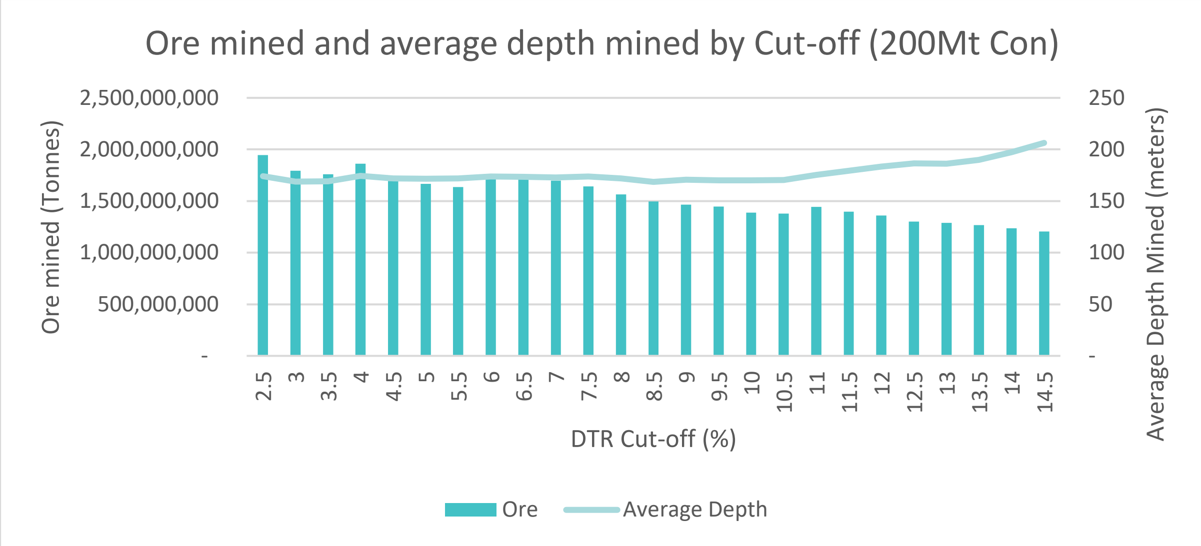

Figure 25. Ore mined and average depth against DTR Cut-off 200Mt Con

The costs and proportion of costs are relatively constant due to the definition of these variables in the block model. The main variable is the mining cost which is driven by the depth of the material but also the total material moved. It is noteworthy that as the strip ratio increases so does the total mining cost, this effected to a higher degree in the 200Mt concentrate scenario where the strip ratio reaches as high as 2.2. Therefore, outcropping low grade shallow ore can be profitable and boost the NPV of the project. The lowest DTR cut-off produced the pit with the highest proportion of indicated material, however there is no significant change until DTR 12.5%, these results are shown in Table 5 and outlined in Figure 26.

Table 5.
Summary of costs and sources of concentrate from 200Mt Concentrate Scenarios

	Costs						Ore Source		
Cutoff	Mining	Processing	Transport + Downstream	G+A + Rehab + Sustaining Capital	Total	Revenue	Indicated	Inferred	Potential
2.5%	$ 5,318,189,786	$ 3,035,961,186	$ 3,230,574,083	$ 3,639,560,135	$ 15,224,285,191	$ 39,567,974,241	46%	48%	6%
3.0%	$ 4,948,218,237	$ 2,798,192,969	$ 2,977,564,313	$ 3,408,979,076	$ 14,132,954,594	$ 36,984,079,341	48%	46%	6%
3.5%	$ 4,963,675,137	$ 2,743,141,076	$ 2,918,983,453	$ 3,418,725,127	$ 14,044,524,794	$ 36,902,017,536	48%	46%	6%
4.0%	$ 5,444,310,601	$ 2,906,754,577	$ 3,093,084,998	$ 3,724,123,291	$ 15,168,273,466	$ 39,682,047,347	45%	48%	7%
4.5%	$ 4,986,345,232	$ 2,644,407,701	$ 2,813,921,015	$ 3,422,532,662	$ 13,867,206,609	$ 36,595,878,836	48%	47%	6%
5.0%	$ 4,988,061,724	$ 2,599,498,655	$ 2,766,133,184	$ 3,423,848,147	$ 13,777,541,710	$ 36,381,509,989	48%	47%	6%
5.5%	$ 4,977,359,553	$ 2,550,577,211	$ 2,714,075,750	$ 3,415,804,810	$ 13,657,817,323	$ 36,068,644,083	48%	46%	6%
6.0%	$ 5,463,645,993	$ 2,724,665,488	$ 2,899,323,532	$ 3,740,863,058	$ 14,828,498,071	$ 38,812,105,826	45%	48%	7%
6.5%	$ 5,457,149,795	$ 2,684,984,236	$ 2,857,098,610	$ 3,736,703,168	$ 14,735,935,810	$ 38,535,472,588	45%	48%	7%
7.0%	$ 5,432,371,094	$ 2,648,739,392	$ 2,818,530,379	$ 3,723,311,635	$ 14,622,952,500	$ 38,187,976,746	46%	48%	7%
7.5%	$ 5,307,538,912	$ 2,559,106,745	$ 2,723,152,049	$ 3,633,053,043	$ 14,222,850,749	$ 37,128,216,505	47%	47%	7%
8.0%	$ 5,098,096,570	$ 2,442,353,029	$ 2,598,914,121	$ 3,498,205,134	$ 13,637,568,853	$ 35,744,971,977	45%	48%	6%
8.5%	$ 4,950,107,783	$ 2,333,858,930	$ 2,483,465,272	$ 3,412,095,737	$ 13,179,527,722	$ 34,380,559,897	47%	47%	6%
9.0%	$ 4,996,703,047	$ 2,287,094,614	$ 2,433,703,243	$ 3,434,104,768	$ 13,151,605,672	$ 34,009,520,298	47%	47%	6%
9.5%	$ 5,048,256,062	$ 2,258,796,058	$ 2,403,590,677	$ 3,473,122,220	$ 13,183,765,016	$ 33,855,890,562	47%	47%	6%
10.0%	$ 4,967,473,441	$ 2,164,637,618	$ 2,303,396,440	$ 3,417,642,606	$ 12,853,150,104	$ 32,881,357,093	48%	46%	5%
10.5%	$ 5,123,777,421	$ 2,151,743,866	$ 2,289,676,165	$ 3,523,816,055	$ 13,089,013,508	$ 33,034,484,551	47%	48%	5%
11.0%	$ 5,545,422,295	$ 2,251,985,959	$ 2,396,344,033	$ 3,789,052,406	$ 13,982,804,692	$ 34,933,071,519	47%	48%	5%
11.5%	$ 5,651,285,202	$ 2,179,158,254	$ 2,318,847,886	$ 3,840,339,113	$ 13,989,630,455	$ 34,375,426,400	47%	48%	5%
12.0%	$ 5,840,662,915	$ 2,120,467,192	$ 2,256,394,576	$ 3,947,706,450	$ 14,165,231,134	$ 33,961,171,524	46%	49%	5%
12.5%	$ 5,991,469,858	$ 2,027,667,120	$ 2,157,645,782	$ 4,032,654,276	$ 14,209,437,035	$ 32,967,361,756	45%	49%	6%
13.0%	$ 6,480,049,864	$ 2,009,596,935	$ 2,138,417,251	$ 4,364,284,253	$ 14,992,348,304	$ 33,147,199,432	44%	50%	7%
13.5%	$ 6,981,264,626	$ 1,975,721,454	$ 2,102,370,265	$ 4,678,629,479	$ 15,737,985,823	$ 33,052,061,087	41%	51%	8%
14.0%	$ 7,810,052,950	$ 1,929,402,322	$ 2,053,081,958	$ 5,182,045,100	$ 16,974,582,330	$ 32,818,104,484	38%	52%	10%
14.5%	$ 9,054,012,198	$ 1,880,571,667	$ 2,001,121,132	$ 5,937,153,014	$ 18,872,858,011	$ 32,534,650,974	35%	49%	16%

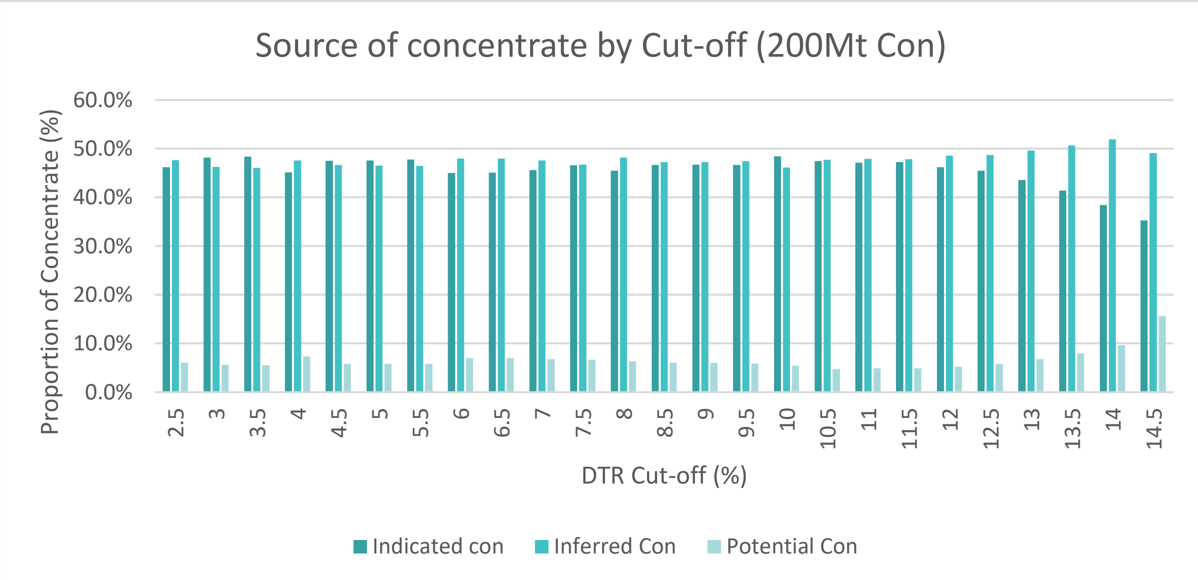

Figure 26. Proportion of feedstock by ore classification 200Mt Con

4 FUTURE OPTIMISATION

4.1 INTRODUCTION

As this is a new Greenfields project it is important to assess all methodologies and technologies in the execution of this project such that it remains competitive with other tier 1 assets around the world. To this end a list of considerations are made below which should be more thoroughly investigated throughout the Bankable Feasibility Study.

The ability to reduce greenhouse gases is becoming an ever more important aspect to the obtaining of project financing. The project has a unique value proposition in that its magnetite (Fe_3O_4) the project produces has a lower energy intensity compared to conventionally mined hematite (Fe_2O_3) to create steel giving it a competitive advantage (Sparrow, 1983). This advantage is exacerbated by the extremely high grade of the product as well as the relative energy intensity to liberate the magnetite when compared to other ores (GHD, August, 2017).

The total material movement in the Pre-Feasibility report by GHD was 2,140 Mt to produce 202Mt of concentrate (p.110) (GHD, August, 2017). The unit costs for these mining parameters are shown in Table 6. These costs have been simplified in Figure 27 where grade control, fragmented ore and fragmented waste have been rounded into Drill and Blast costs comprising 17% of the costs, T&S ore and T&S waste rounded into Load and Haul costs comprising 58% of the costs and the remaining costs to be rounded into pit maintenance comprising 25% of the costs. This shows that reducing the load and haul costs will deliver the most leverage to driving down the mining cost on a proportionate basis.

Table 6.
Breakdown of mining costs as per (GHD, August, 2017)

Raw	A$/t con	A$/t rock	%
Wall control	$ 0.21	$ 0.02	1%
Ore grade control	$ 0.70	$ 0.07	5%
Frag ore	$ 1.19	$ 0.11	8%
Frag waste	$ 0.60	$ 0.06	4%
T&S ore	$ 4.81	$ 0.45	33%
T&S waste	$ 3.61	$ 0.34	25%
Ore Stockpiling	$ 0.29	$ 0.03	2%
Waste dump	$ 0.70	$ 0.07	5%
Roads	$ 2.51	$ 0.24	17%
Total	$ 14.62	$ 1.38	100%

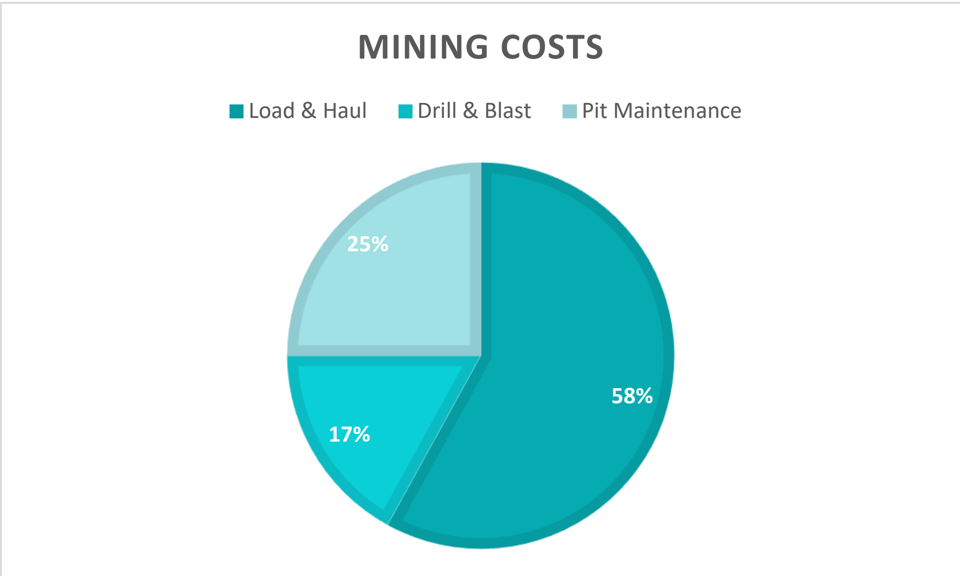

Figure 27. Consolidated mining costs by sub section (extrapolated)

4.2 Available Proven Technologies

An operation like the one being described is a major earth moving exercise contemplating an average of 94 million tonnes of material per annum for the first 10 years and an average of 101Mt per annum averaged over 20 years. This would put it on par with the largest magnetite mines in the world (CITIC, 2012) in terms of material movement.

4.2.1 *Autonomous Haulage*

Autonomous haulage is used widely across bulk tonnage projects in Australia, having been used extensively in Pilbara Iron ore projects for the past 10+ years (Rio Tinto, 2012) and more recently in more complex coal pits in the Bowen Basin (International Mining, 2021). The projects pit shape and geological complexity is likely amendable to the implementation of autonomous haulage which has the capability to perform in more complex settings.

Autonomous haulage has the capacity to significantly improve the economics of a project through the cost reduction of; labour, increased capacity, and reduced maintenance, however the larger benefits are driven by pit optimisation which can take advantage of reliable and less error prone automated driving (Whittle Consulting, 2018). This comes at a minor initial capital cost as shown in Figure 28.

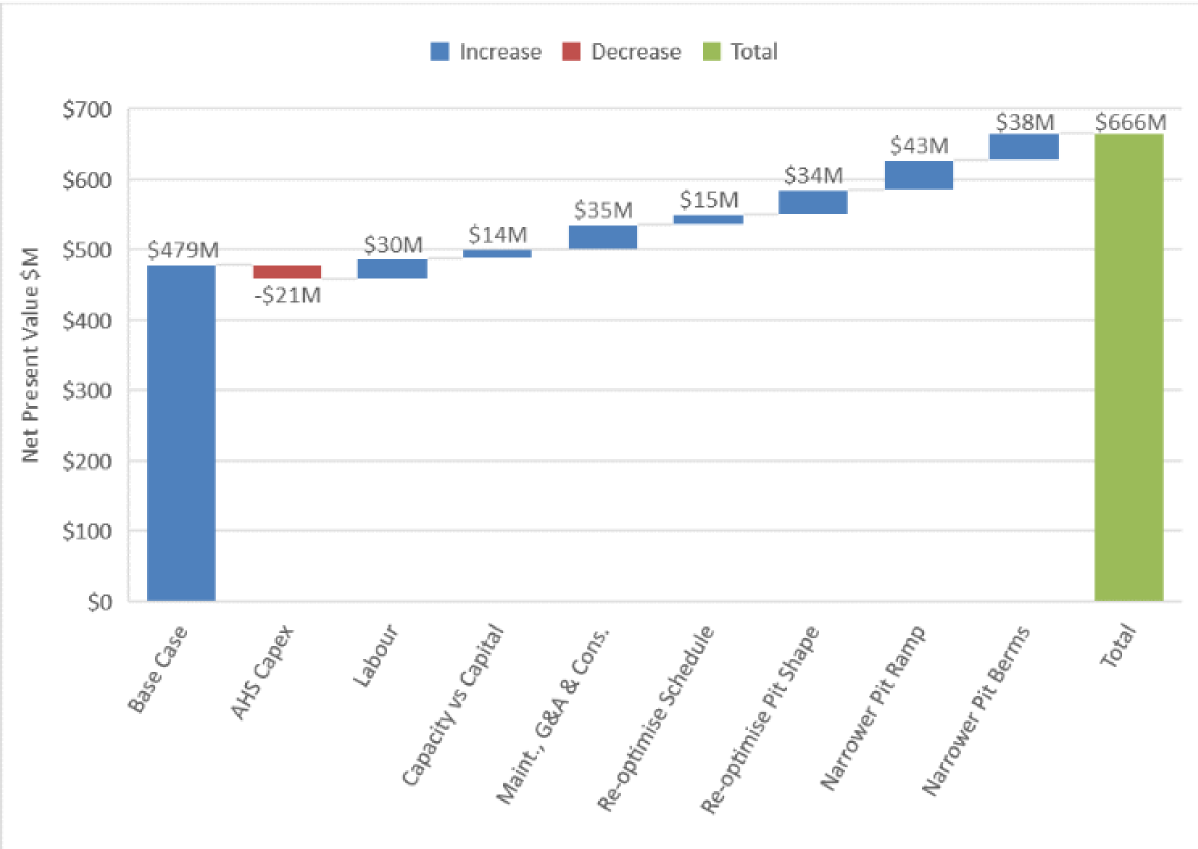

Figure 28. Cost-benefit analysis of implementation of an Autonomous Haulage System on a Copper/Gold ore body (Whittle Consulting, 2018)

The benefits of pit design are likely to be less important for the Hawsons project due to large thick ore body and low strip ratio. The ability to work with narrower ramps will enable quicker access to ore in initial years which will benefit the mine with earlier ore feed. Autonomous haulage is becoming more common place in conjunction with Ultra Class equipment and is beginning to be used with trolley assist (International Mining, 2021). Whilst not common place using an aggregation of these technologies, they should be strongly considered for the Hawson project as it's development timeline will coincide with the maturity of this technology.

4.2.2 Ultra-class Equipment.

In the original PFS a 220-240t dump truck size was selected with the Komatsu 830E used as an indicative haulage vehicle (GHD, August, 2017). When the preliminary work for this project was done for this project ultra-class haul trucks (with >300t payload) were still in the process of becoming more widely adopted in Australia outside of a select few mine sites (Australian Mining, 2019). With the amount of material being moved each year the economies of scale of ultra-class gear should be investigated, particularly if used in conjunction with other capital items such as autonomous haulage.

It is noteworthy that in the similar deposit of Magnetite Mines, that they have elected to use to use 150-190t medium class haul trucks. This was primarily driven by the need to achieve 1m selectivity which led to the decision of a 350t excavator (Magnetite Mines Limited, 2021). The Alderon Iron Ore Corp used CAT 794 with a nominal payload of 292t in their full feasibility study (43-101) which was used to deliver 23 Mtpa of ore to their processing plant on the Kamistiatusset iron ore project in Labrador (BBA, 2018). Black Iron opted for the CAT 793 with a nominal payload of 228t in their preliminary economic assessment (43-101) which was used to deliver up to 28.7 Mtpa of ore to their processing plant on the Shymanivske Iron ore project in Ukraine (BBA, 2017). Experienced magnetite miner Champion Iron opted for a similar middle class of truck with a payload of 218t in their preliminary economic assessment (43-101) which was used to deliver up to 41.9 Mtpa of ore to their processing plant on the Bloom Lake Mine project in Quebec (BBA, 2019). These smaller haul trucks can be contrasted with CITIC which uses the Bucyrus MT6300AC which has a 360t payload and is used at the Sino Mine in Western Australia (CITIC, 2021). s

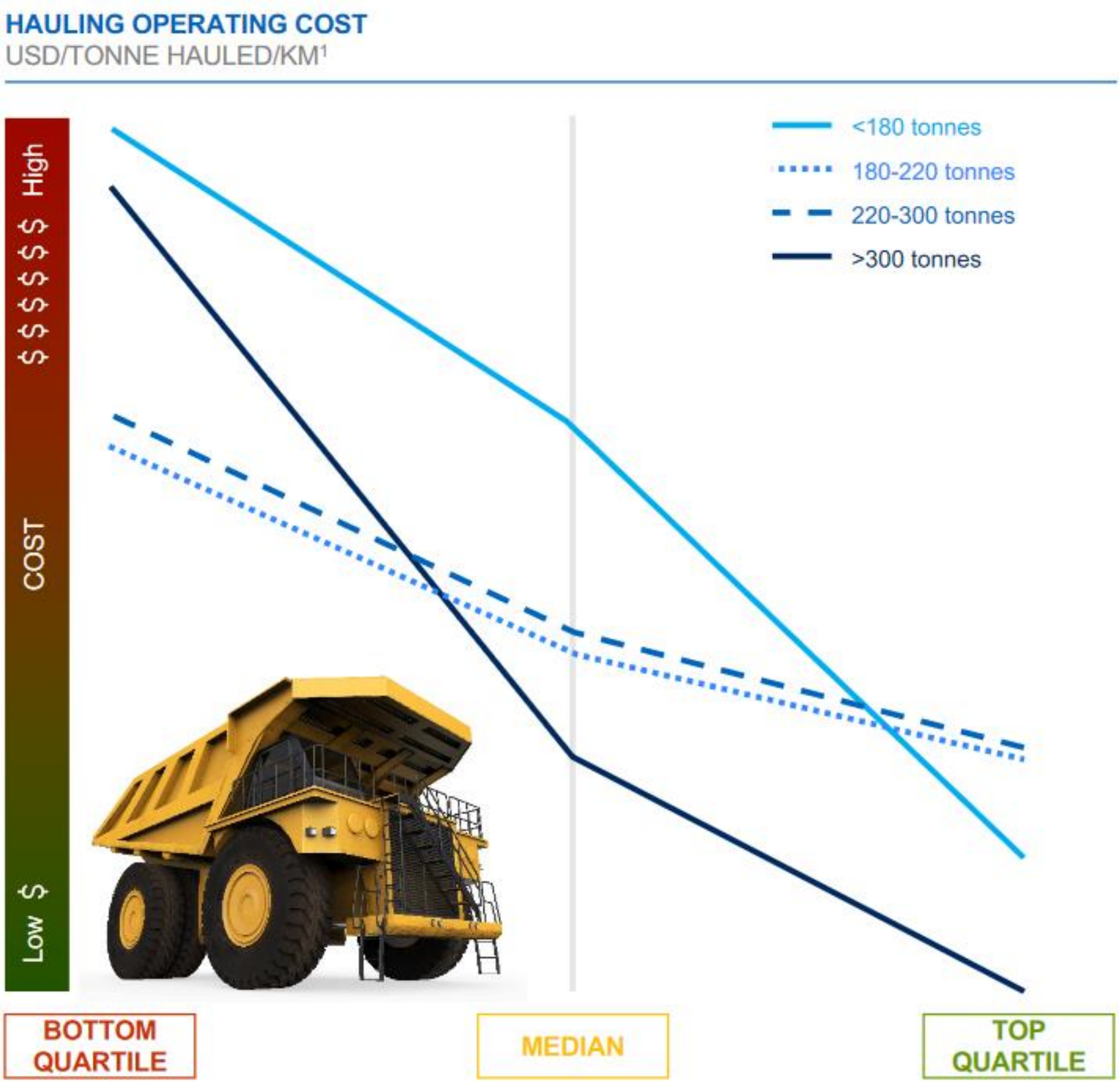

Figure 29. Haulage cost by class of truck (McKinsey, 2019)

Middle class equipment has a more consistent operating cost curve compared to ultra-class haulage which perform worse in the lower quartile as in Figure 29. However, when implemented correctly Ultra class equipment can readily deliver over 15% in cost savings when compared to a similarly well-run medium class fleet (Shea, 2020). The important considerations for the implementation of this technology are the competency of the running material, pit design constraints and selectivity of mining (by matching with appropriately sized excavators). This technology is readily available with synergistic technologies such as trolley assist (Caterpillar, 2021) and automation (Komatsu, 2021).

4.2.3 In Pit Crushing and Conveying (IPCC)

In Pit Crushing and Conveying (IPCC) has long been known to significantly reduce costs (20-60%) and emissions of a mine site, this however does come at a significant cost to operational flexibility and redundancy (McEwing, 2019). Using IPCC is effective in dynamic weather environments working continuously, however due to the low number of rain days in Broken Hill this benefit is unlikely to be realised. IPCC problems typically stem from inconsistent material feed which can result in a stoppage in the system, with a single point of failure stopping production from the mine.

One of the benefits of IPCC is that the machinery used to transport the material from the mine is far more efficient then using haul trucks. This is because haul trucks must apply energy to transport the mass of the truck along with the material to final destination and then come back to the source of the material using energy to transport the mass of the truck back empty (Scales, 2017). Utilising solar powered in pit crushing and conveying systems can dramatically reduce energy costs, currently crushing, milling, and grinding account for 2.7% of all electrical energy consumed (FLSmidth, 2020).

As the transport cost for material can be significantly less than that of conventionally hauled material the cut-off grade can be lowered and mine design further optimised (Johnson, 2015). The reliance on a fixed conveyor reduces the operational flexibility but also creates a single bottleneck in the operation which if fails stops the entire production. The other important consideration is with the friability of the material (GHD, August, 2017) there could be excessive dust creation in the pit which must be controlled or could have other operational impacts. Overall IPCC will have diminishing returns when combined with other haulage benefits, the opex savings are significant and should be further investigated. The benefits of IPCC are stronger with deeper pits and the capital could be incurred once a deep pit and final wall are established. This has the added benefit of reducing the impact on NPV through delayed capital expenditure.

4.2.4 Truck Trolley Assist

Truck and Trolley systems have the ability to lower fuel consumption by 90% whilst connected to the grid on diesel electric hybrid systems (Hitachi Construction, 2018) if this grid power is competitively priced it can significantly reduce operating costs. This system again works best when infrastructure can be put in a fixed position such as a final wall which does reduce some operational flexibility.

Whilst reduced fuel costs provide the biggest savings there is also the increased speed on uphill slopes with the example used delivering (+10km/hr, on average) which resulted in significant productivity increases in the fleet, requiring less trucks (capex) and less operators (opex), there is an additional benefit in trucks needed less maintenance due to lower engine loading factors. These benefits are partially offset by the infrastructure costs, power costs and pit design costs (optimising the pit with wider ramps) (St-Onge, 2014). An example of what these costs look like is shown in Figure 30 for an existing Canadian mine.

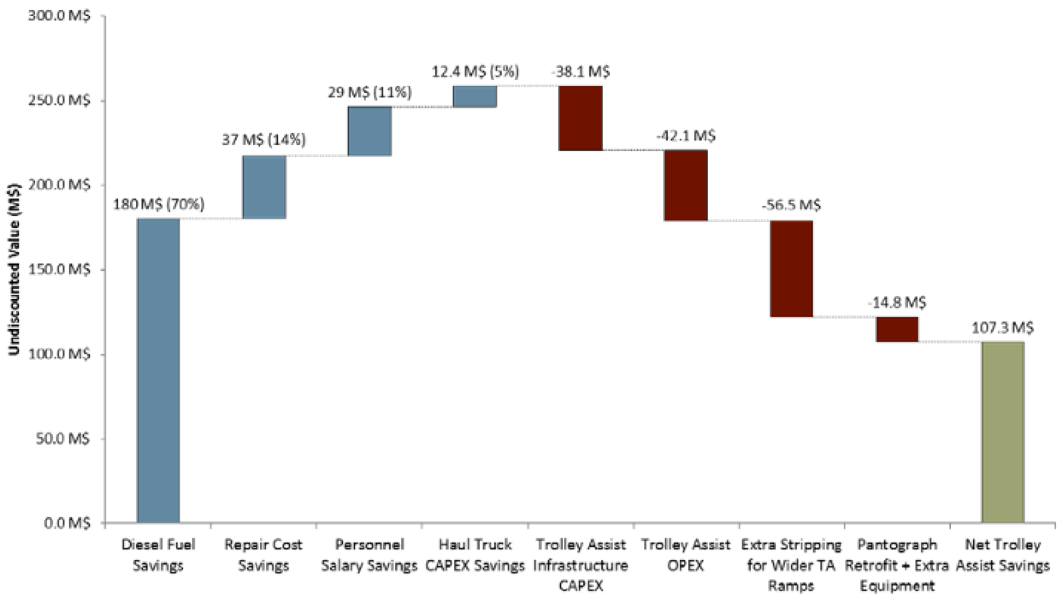

Figure 30. Economic benefit for a Truck and Trolley installation on a mining operation in Canada. Source: (St-Onge, 2014)

There are several benefits not directly related to costs also associated with the truck trolley assist, the reduction in fuel lessens greenhouse gases, NO_x and particle matter emissions, lower noise and decreased waste (oils/tyres/parts). The reduction of 20-60% of CO_2 emissions also allows room for carbon tax credits depending on the operation jurisdiction (Caterpillar, 2021).

4.2.5 Continuous Surface Miners

Continuous surface miners enable the production of homogeneous particle size feed through to trucking or conveyor systems. When fed onto a conveyor system they have the operating cost benefit of IPCC however, they have the added benefit in feeding the conveyor with a consistent product which is less prone to causing issues. One of the big issues with IPCC is the cyclical nature of traditional mining drill, blast and load not providing a consistent feed to the in-pit crusher due to heterogeneity in the rock mass. The comparison of different mining methodologies is shown in Figure 31.

Figure 31. Comparison of process chains in open pits

When looking at the implementation of continuous surface miners at FMG they have been stated to reduce the production costs by 40% and the capital requirements by 50% (Wirtgen, 2021). This could provide significant mining cost savings to the Hawsons project as well as potentially reduce processing costs through a more homogenous feed through to the mill.

Continuous miners do have limitations to pit shapes due to their minimum turning angle and ultimate width of mining. An optimisation study should be completed with these constraints in mind to assess the impact of these design constraints on the ultimate economics of the project. The application of this technology to the soft ore mass of Hawsons should be further investigated. Many papers on the subject have been written by Wirtgen who appear to be the leader in the Australian mining space. The benefits of a Wirtgen system compared to a traditional mining fleet are shown in Figure 32.

PROJECT SCORE CARD

	PERFORMANCE AREA	EXAMPLE	RESULT
Production Requirements	Coal Productivity (per op hr)	up to 2375 t/hr	☑
	Parting Productivity (per op hr)	up to 1450 bcm/hr	
	Machine availability	> 90%	
Health safety and Environment	Dust	Contained in mill housing + reduced by dust supression system	☑
	Carbon emissions	79% less fuel per bcm	
	Risk management	Reduced fleet, so red. interaction	
	Ergonomics	Vibration measured as "low", according to the Australian Standard AS 2670-2001	
Impact on CHP	Coal sizing	14% reduced fines < 2 mm and reduced oversize	☑
	Workability at CHP	Reduced power draw at CHP	
	Manage Siderite intrusions	Zero CHP crusher blockage	
	Coal loss & Dilution	Visual improvement in pit	
Mining unit cost	Labour	Reduced by 60%	☑
	Fuel	Reduced by 79%	
	Cost per ton	Reduced by 60%	

Figure 32. Wirtgen 4200SM when compared to coal mining fleet utilising 130-190t trucks

4.3 New Possible Technologies

Whilst new technologies should not be relied upon to deliver a profitable project and should not be relied upon to make the project viable, rather, they should be considered as they represent a paradigm shift from current hydrocarbon powered equipment.

4.3.1 Hydrogen Powered Trucks

The economies of hydrogen ultimately come down to the cost of hydrogen vs conventional sources of combustion like hydrocarbons as the biggest driver for the adoption of the technology. Australian companies are actively assessing the viability of hydrogen at different price points, the largest driver for green hydrogen (hydrogen derived from the hydrolysis of water and not hydrocarbons) is power price (CSIRO, 2021). Abundant cheap solar power particularly when the duck curve is peaking provide opportunity for hydrogen creation at competitive rates. The adoption of hydrogen in different industries is shown in Figure 33.

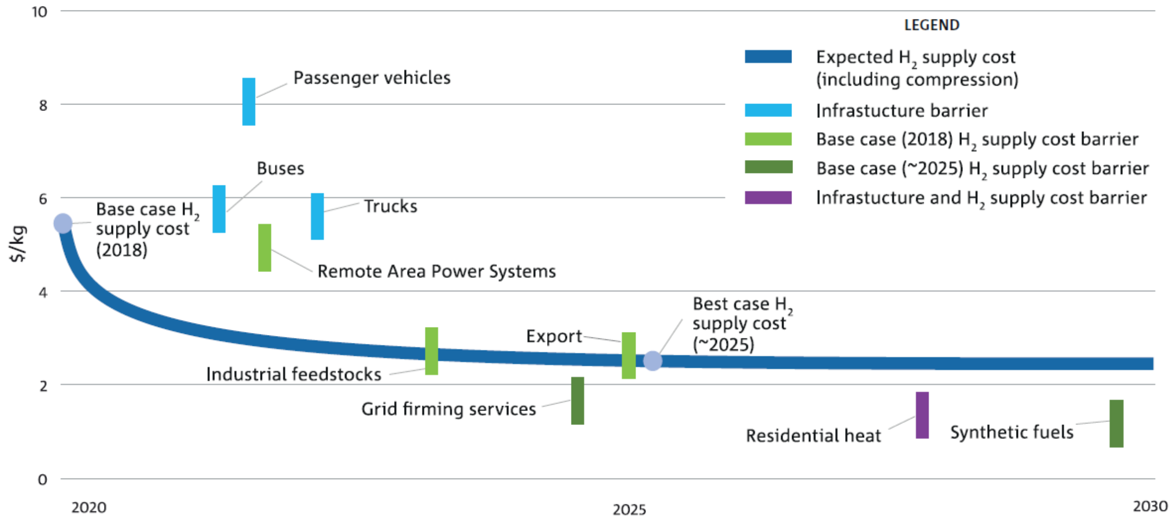

Figure 33. Hydrogen competitiveness in targeted application (CSIRO, 2021)

Another barrier to hydrogen adoption in mining is currently getting the required energy density vehicles such that refuelling is not prohibitive to productivity of vehicles. This is currently being researched by a range of mining companies with the current fuel cell technologies showing promise, Anglo American says large haul trucks account for over 70% of diesel consumption on site and has committed to carbon neutrality by 2040. Large miners across the world have committed to carbon neutrality by 2050 and supply chain logistics may demand this be the case for all steel production (NPROXX, 2021). Given the long life of the project and the encroaching deadlines many organisations face, hydrogen or other green technologies should be strongly considered by HAW. Early adoption could have the duel benefit of government incentives and additional product premium.

4.3.2 Battery Powered Trucks

Electric Vehicle (EV) Haul trucks offer an alternative for utilising solar energy to provide the haulage of material on a mine site. Trucks using battery systems are currently only viable is a select number of locations where the location of the ore has a significant height advantage compared to its destination. In these environments the trucks can use the gravitational potential energy of the ore to recharge the battery system on decent of the hill and use that energy to carry the truck back up the hill empty using batteries. As the geography of the HIP is very flat it is unlikely that a pure battery powered truck would be viable without a step change in battery energy density and cost.

This technology however could be combined with trolley assist infrastructure providing the advantage of cheap power delivered to the vehicle with the benefit of “last mile” flexibility to the trucks. This is currently being investigated as it removes many moving parts from traditional diesel electric trucks. Currently electric battery equipment is two to three times more expensive compared to diesel equipment, however the price of this equipment is quickly falling and is expected to meet cost parity (King-Abadi, 2020). This adoption may reach parity faster with many ultra-class mining equipment already employing the use of large electric drive motors.

Whilst stand-alone battery powered trucks may not be ready for the mine by start of construction, a system of battery powered, and trolley assisted trucks are already being adopted around the world (Guthrie, 2021). Their performance should be closely monitored to observe what degree of cost saving is possible.

5 CONCLUSIONS

From a resource perspective the project is in a good position. If inferred and potential resources can be increased in size and confidence whilst maintaining similar characteristics to that which has been used then the project will easily meet the requirement of having 20 years of ore feed.

It is noteworthy that current modelling only factors in 6-7% of potential resource with the rest of the ore feed being made up of an equal proportion of indicated and inferred resources. Thus, roughly 100Mt of ore is needed to be brought from inferred to measured (or conversely 100Mt from indicated to measured and 100Mt from inferred to indicated). There is no shortage of ore however investigations should be made into potentially outcropping and shallow ore to feed the mill.

The most interesting area is around the Fold section of the ore body which is currently poorly defined. Although this material produces a slightly lower grade concentrate (69% vs 70% of the main zone) it is the most economic part of the entire ore body and under all scenarios this area was targeted first with the early ore driving NPV north. With better definition around these ore bodies (particularly in areas on the edge of current modelling) there exists opportunity for economic ore close to surface which may play a role in reducing the projects NPV. Sterilising this area may also prove beneficial for placement of critical infrastructure such as processing plants, tailings, waste dumps or stockpiles.

The project has enormous potential to produce a low carbon feedstock for the steel market and should be actively investigating new technologies which can develop this concept. Many of the concepts investigated in this report have the ability not only to reduce CO_2 emissions but alsocosts, most retain the flexibility of a traditional truck and shovel operation apart from continuous miners and conveyors. A high-level assessment of this technology should be carried out to see if it has any impact on the upcoming drill campaign.

6 RECOMMENDATIONS

6.1 COST DEFINITION

KPS Innovations strongly recommends building up a model from first principals. To ensure accuracy this should be benchmarked against similar projects of scale around Australia. To add validity these costs and operating metrics should be compared to contractor rates if possible and firsthand quotes. This should culminate in a market appropriate variable which will allow a more accurate pit optimisation study to be completed.

6.2 FUTURE DRILLING

The Fold area of the pit should be delineated with further drilling and any potential area which is proposed to be used for significant infrastructure should be sterilised with drilling, as should and prospective dump areas. In Figure 34 we can see a high surface expression of iron from XRF analysis. This is correlated with siltstone with iron interbeds, this material should be checked for Davis Tube Recovery numbers however if consistent with the Core ore body, then could prove to be advantageous with early mine life exploitation.

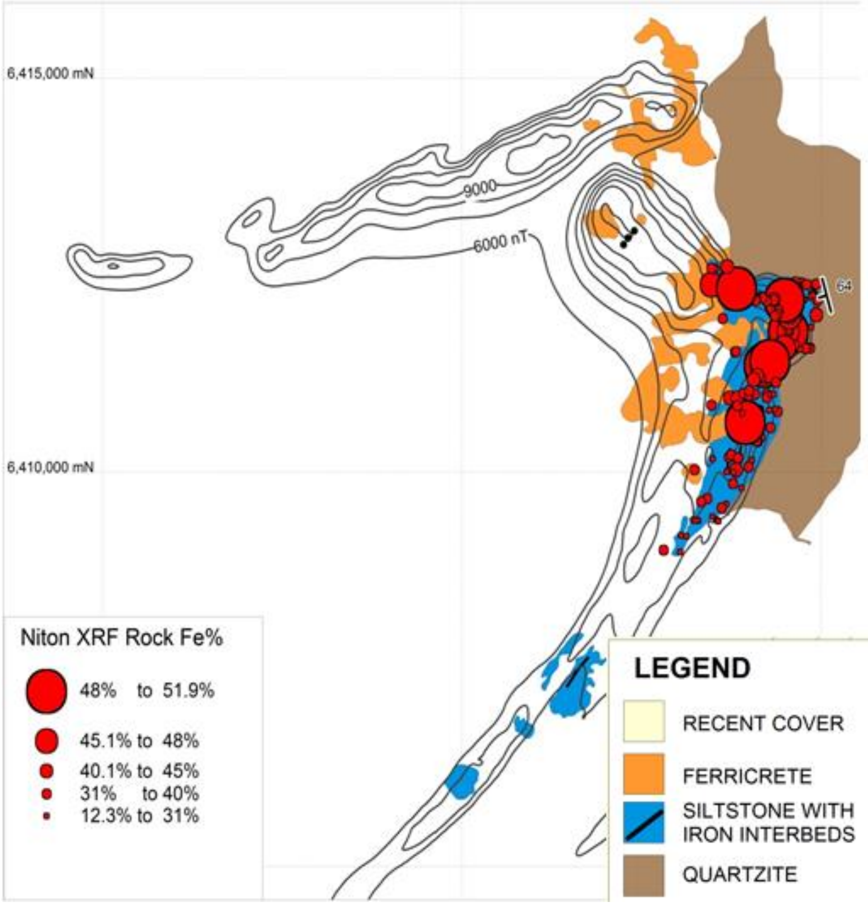

Figure 34. Photo showing the ore body and high density of surface expression of iron in Fold area

The company should seek to plot drillholes along the current modelled ore body and if not already validated find areas without mineralisation for placement of waste dumps, tailings, and infrastructure. The company should also potentially seek to setup its own software licenses and take ownership of the models and naming conventions ensuring they are well documented.

6.3 TAILINGS STRATEGY

Tailings are likely going to prove incredibly difficult for this project, the juggling of competent materials to build the dam, obtaining insurance and bond financing may be the highest project risk. This is without considering the ability to get such a dam approved. Using the fold area to create smaller pits may also prove useful for tailings deposition. As shown in Figure 35 which is the optimised pit for a 6% cut-off grade at 100Mt of concentrate ~ 10 year mine life.

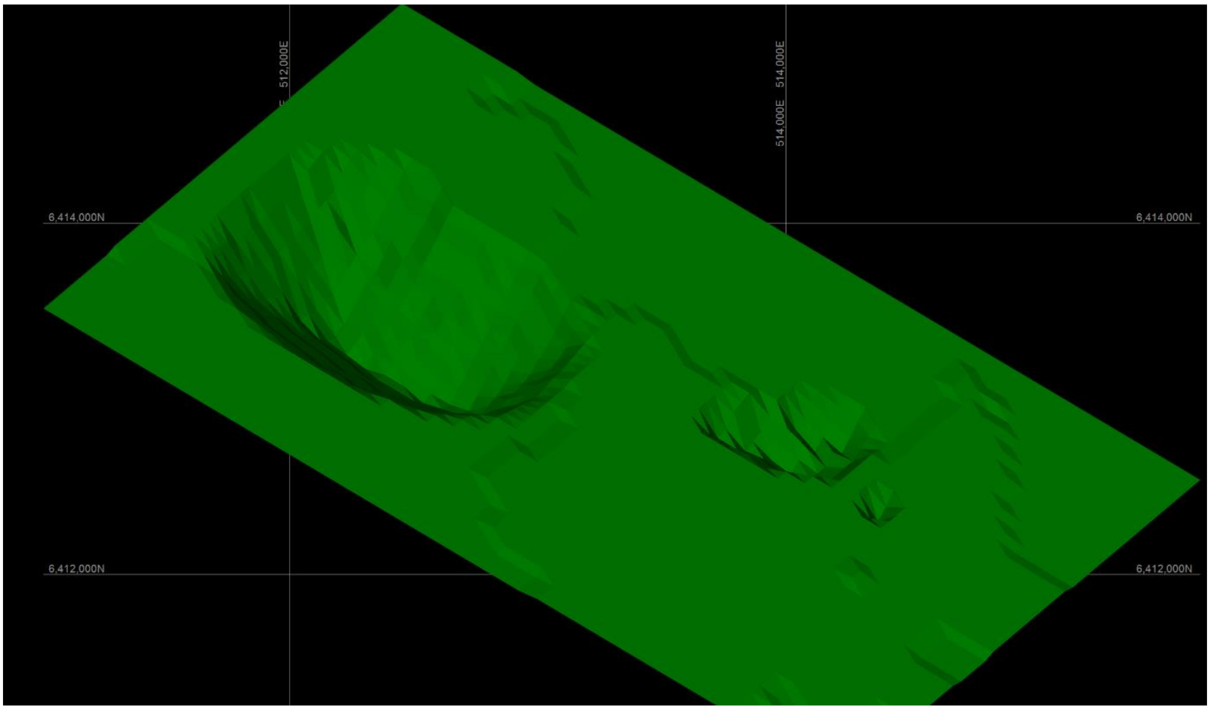

Figure 35. 100Mt of concentrate pit for a DTR cut-off of 6%

There may also exist the possibility for a “saddle” like structure in the main pit to provide progressive backfilling and storage of tailings. This would require significant understanding of the ore and a well-defined mine plan as it could result in sterilisation of other ore as well as create potential hydrological and geotechnical risks. An example of what this kind of pit shell may look like is in Figure 36 which is the optimised pit shell for a 6% cut-off grade at 200Mt of concentrate ~ 20 year mine life.

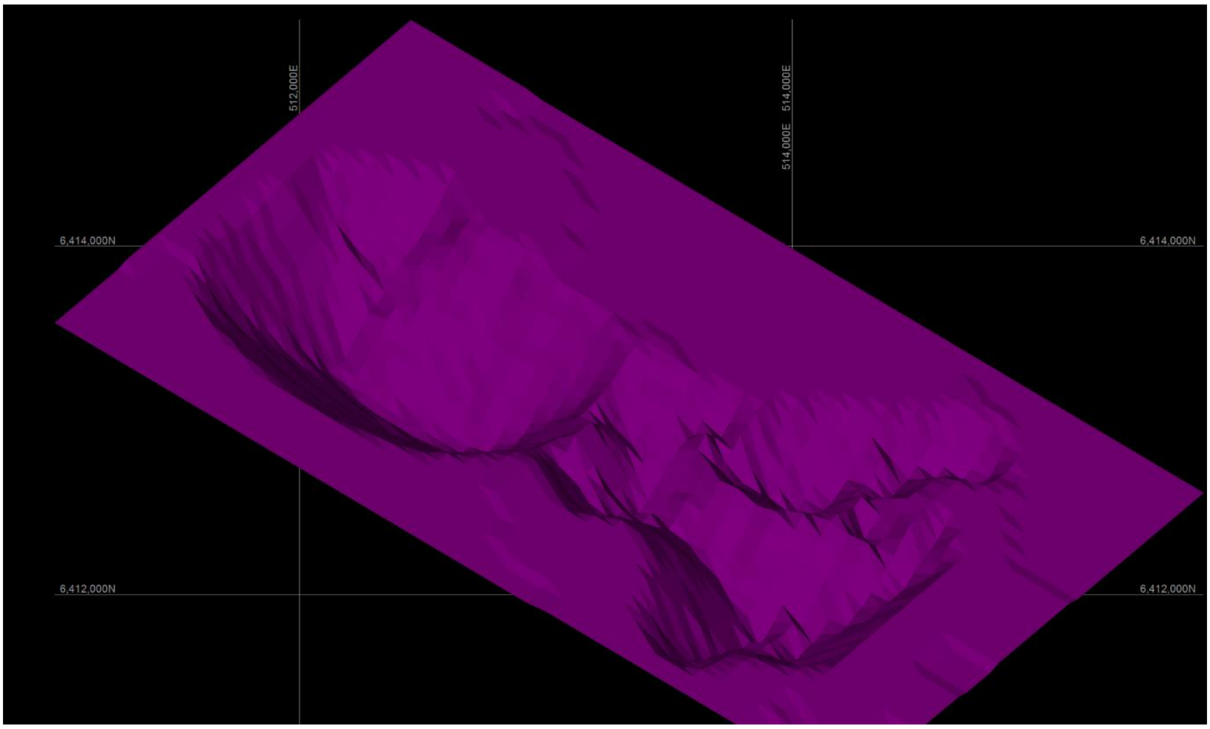

Figure 36. 200Mt of concentrate pit for a DTR cut-off of 6%

6.4 Pit Scheduling

Currently all models have been focused around a 10Mtpa concentrate product rate. A sensitivity analysis should be done around the effect mining rate has on the costs of the project, subsequent cut-off grade and ultimate viability of the project. This will enable a life of mine schedule to be developed which can assess the opportunity to have a saddled pit approach as well as in-pit dumping and tailings deposition.

The scheduling of the pit will also dictate the feed provided to the mill and the capex required by the project. If processing plant units are modular there may also be opportunity to do a ramp up approach allowing for a lower initial capex.

6.5 Surface Miner

All methodologies discussed in section 4 (Future Optimisation) of the report are variations on the traditional mining methodology of drill, blast, load, haul, beneficiate and ship. The one exception to this is the use of surface miners which provide a unique approach to mining. This methodology is not without its limitations particularly in respect to the dimensionality of the pit and whilst on paper may appear superior to other mining methodologies in terms of cost, restrictions around pit shape may have an overall adverse effect on the NPV of the project. It is recommended rough costs and design constraints using equipment from Wirtgen and/or Vermeer are obtained, and another pit optimisation study is run to assess the viability and impact on the drill campaign.

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8 APPENDIX

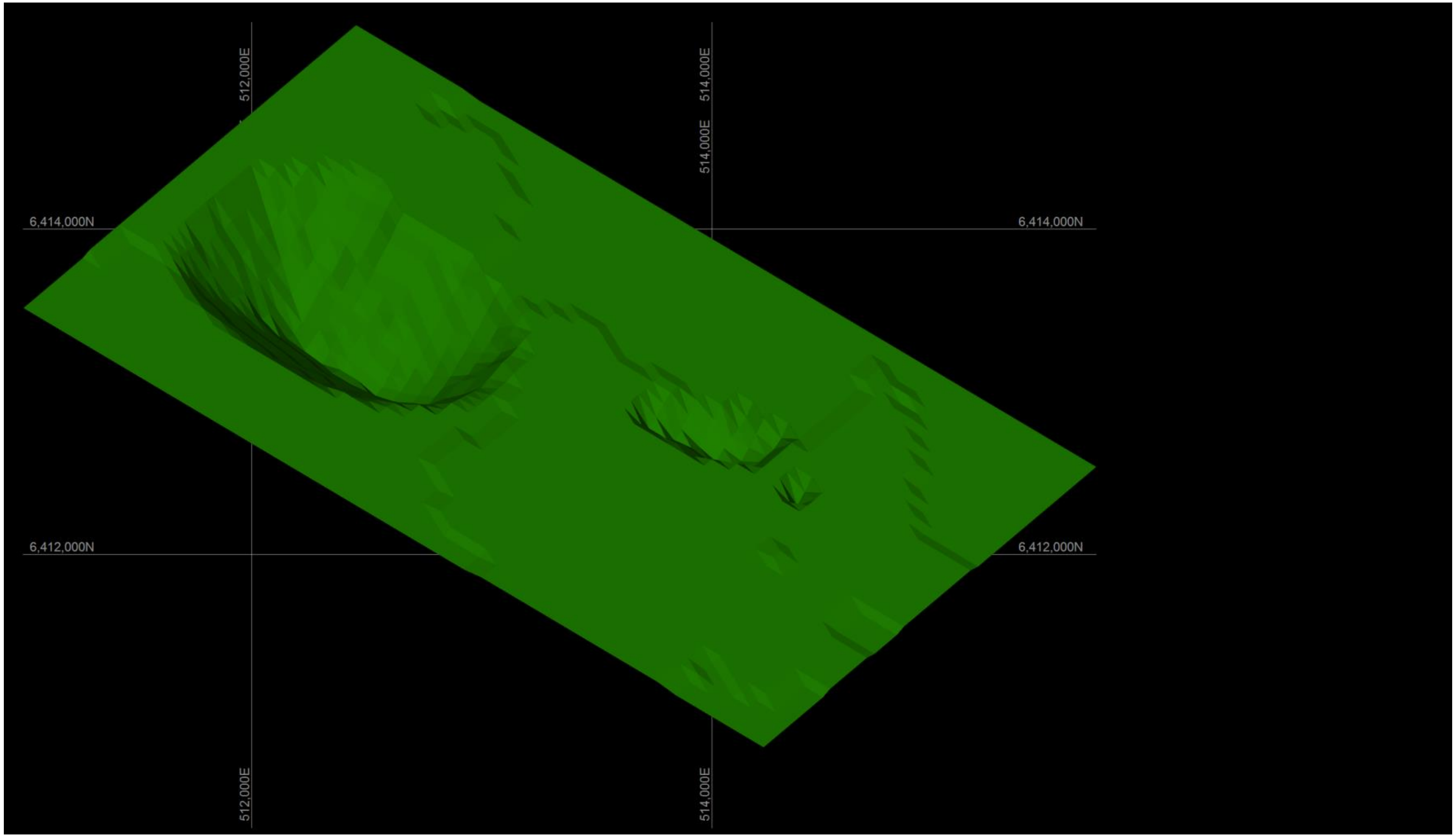

Figure 37. 100MT Concentrate pit shell for JORC Resource

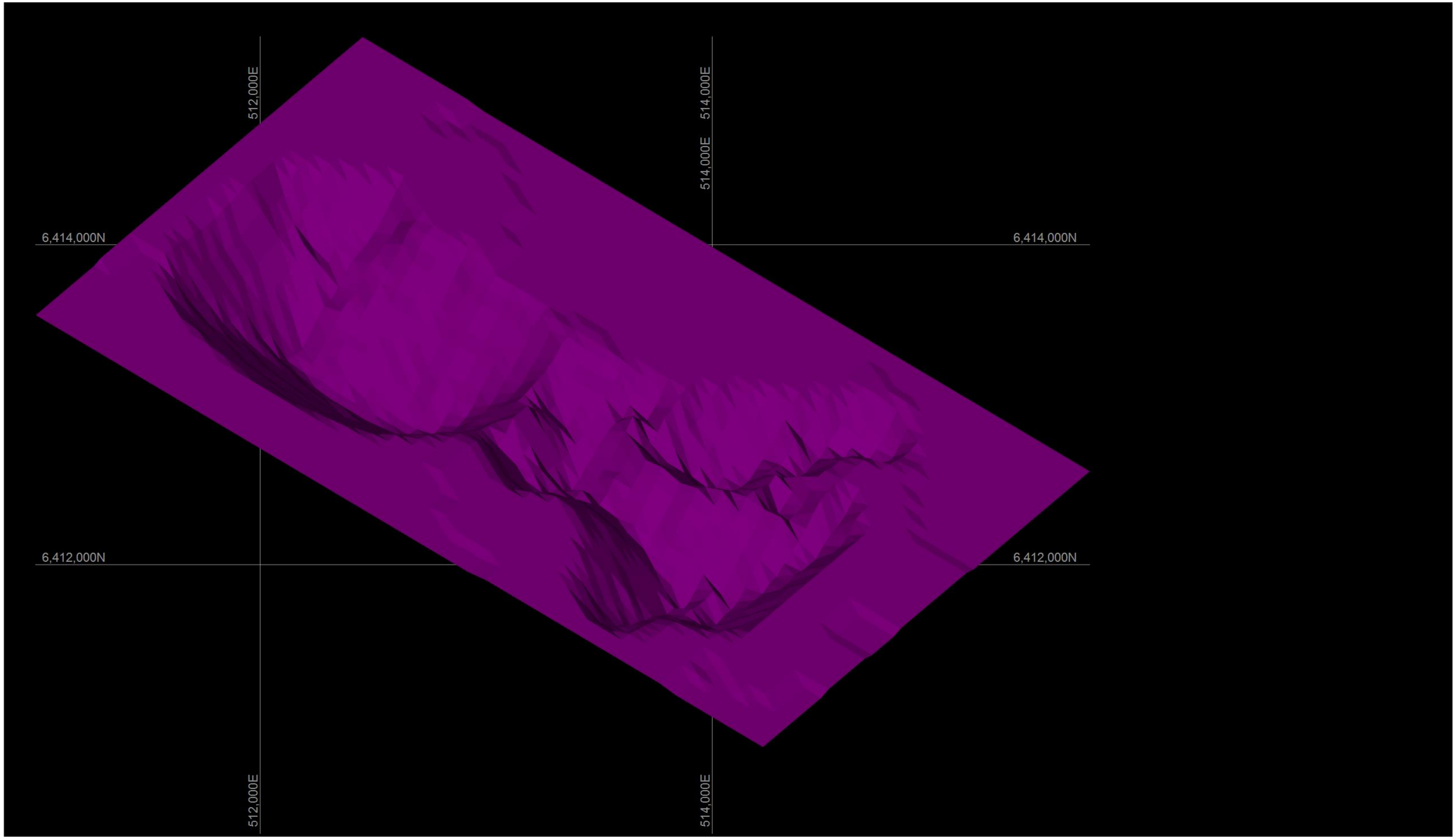

Figure 38. 200MT Concentrate pit shell for JORC Resource

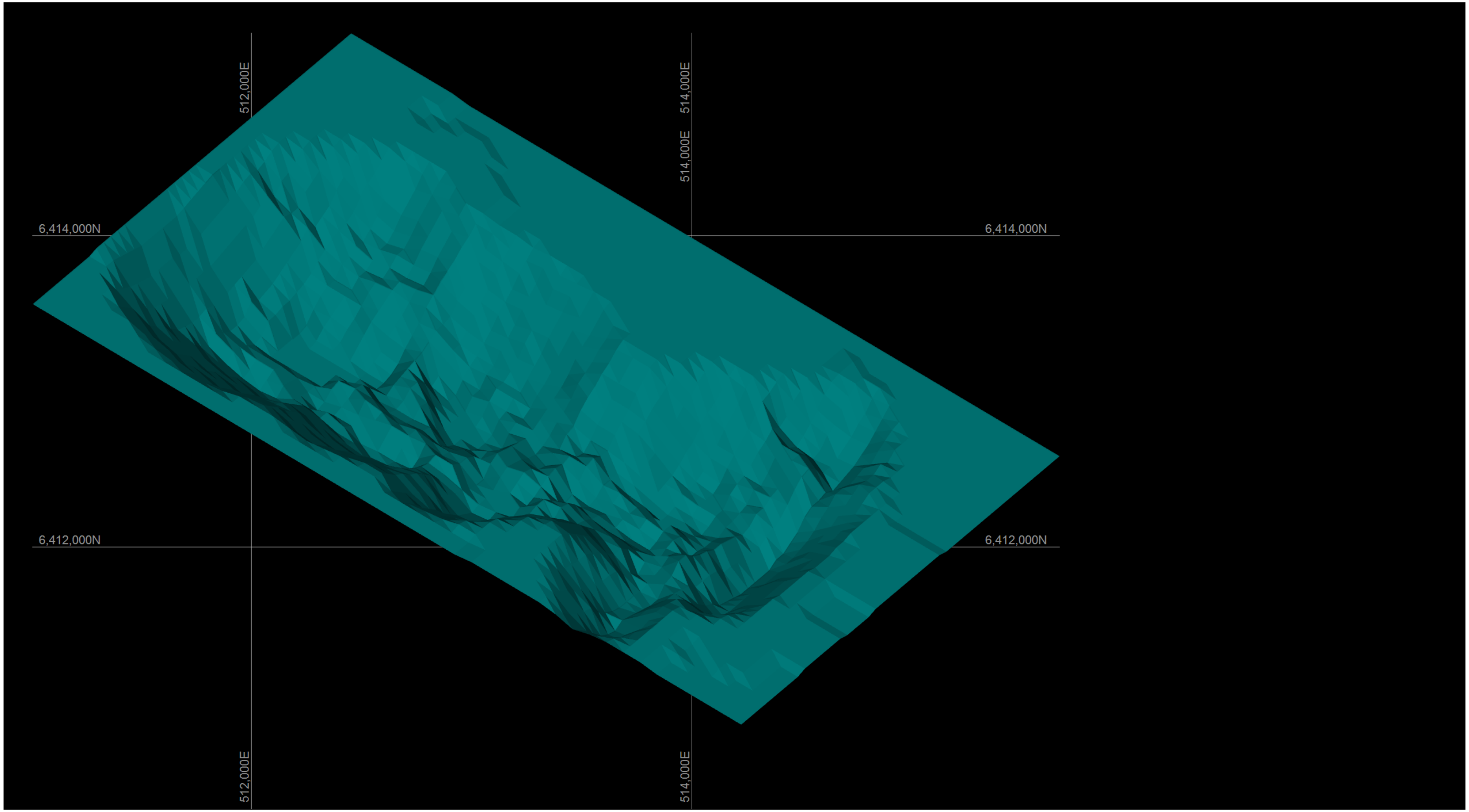

Figure 39. Ultimate Pit limit for JORC resource (372MT)

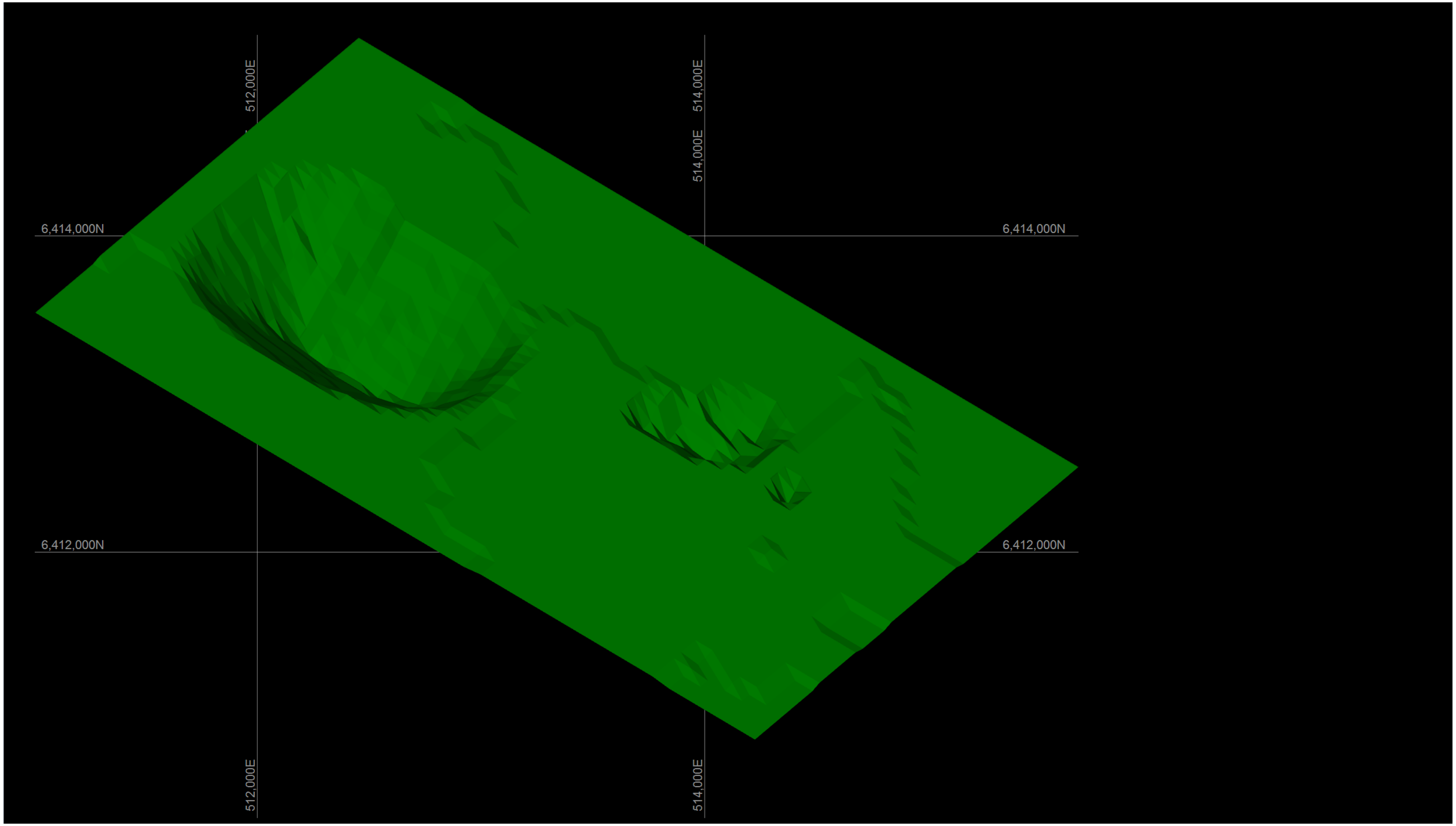

Figure 40. 100MT Concentrate pit shell for Global Resource

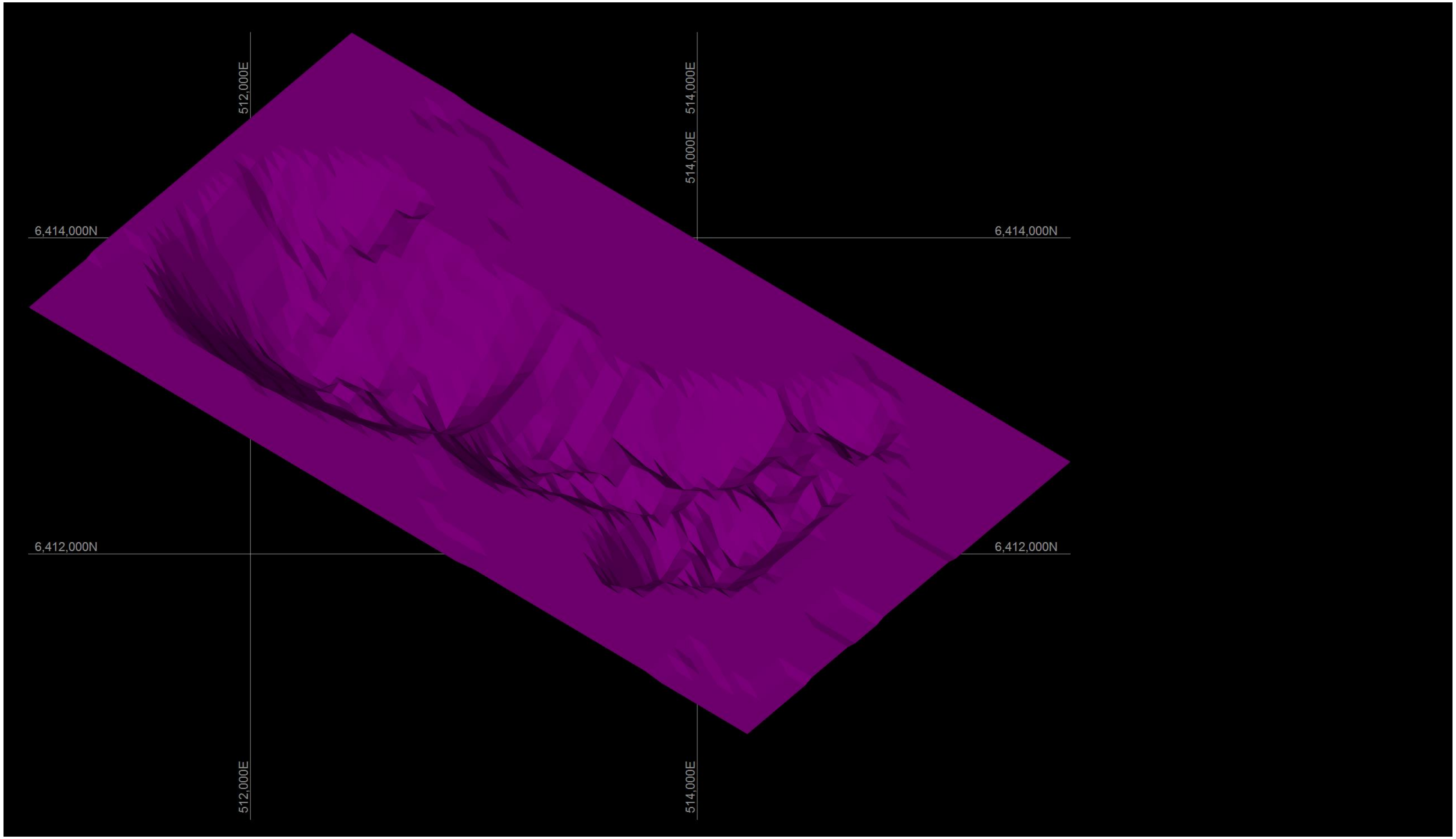

Figure 41. 200MT Concentrate pit shell for Global Resource

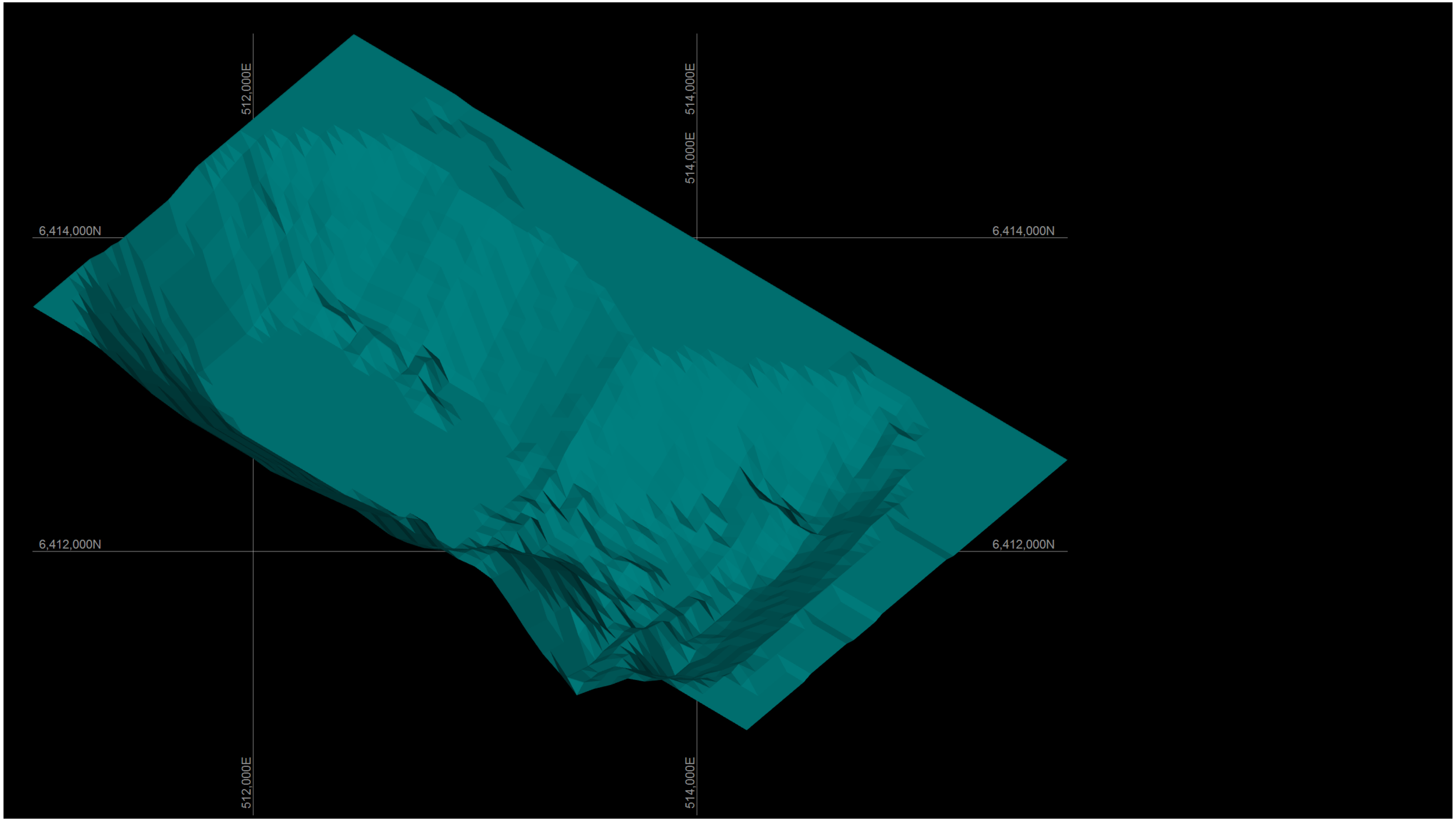

Figure 42. Ultimate Pit limit for Global resource (576MT)

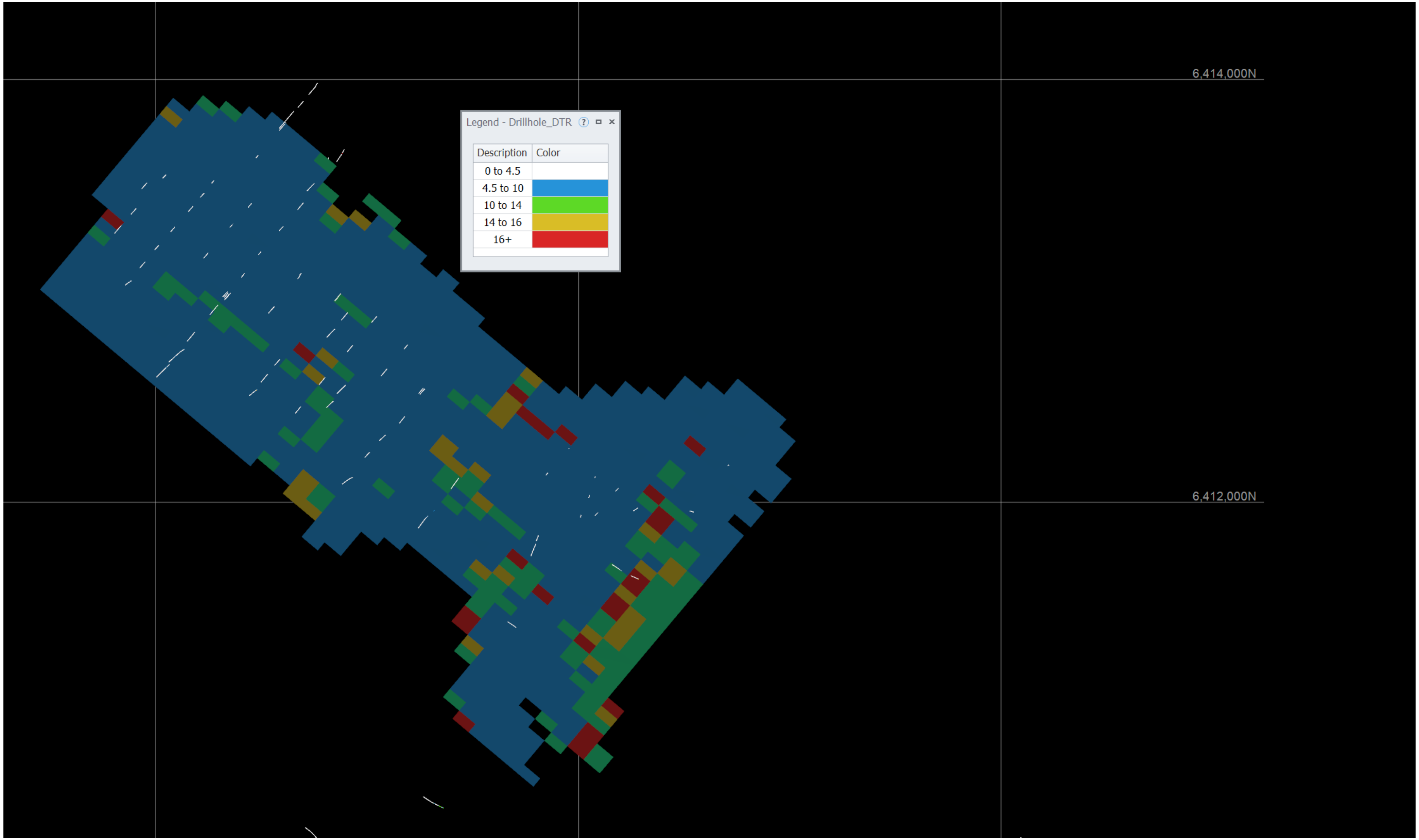

Figure 43. White lines indicate drilling (Visible only above ore body)

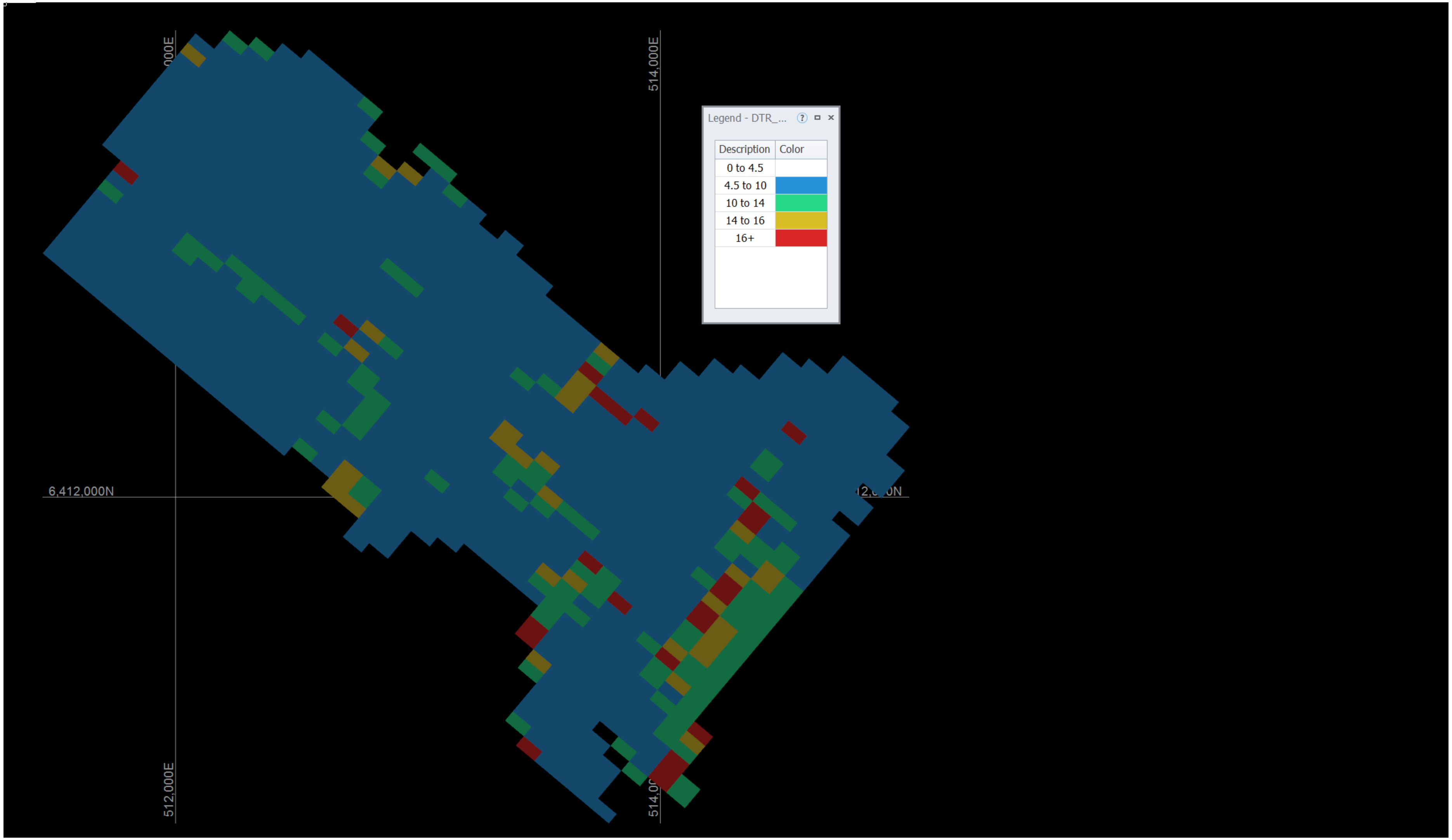

Figure 44. Ore body with overburden and <4.5% DTR set to transparent. To be used as overview window for cross sections.

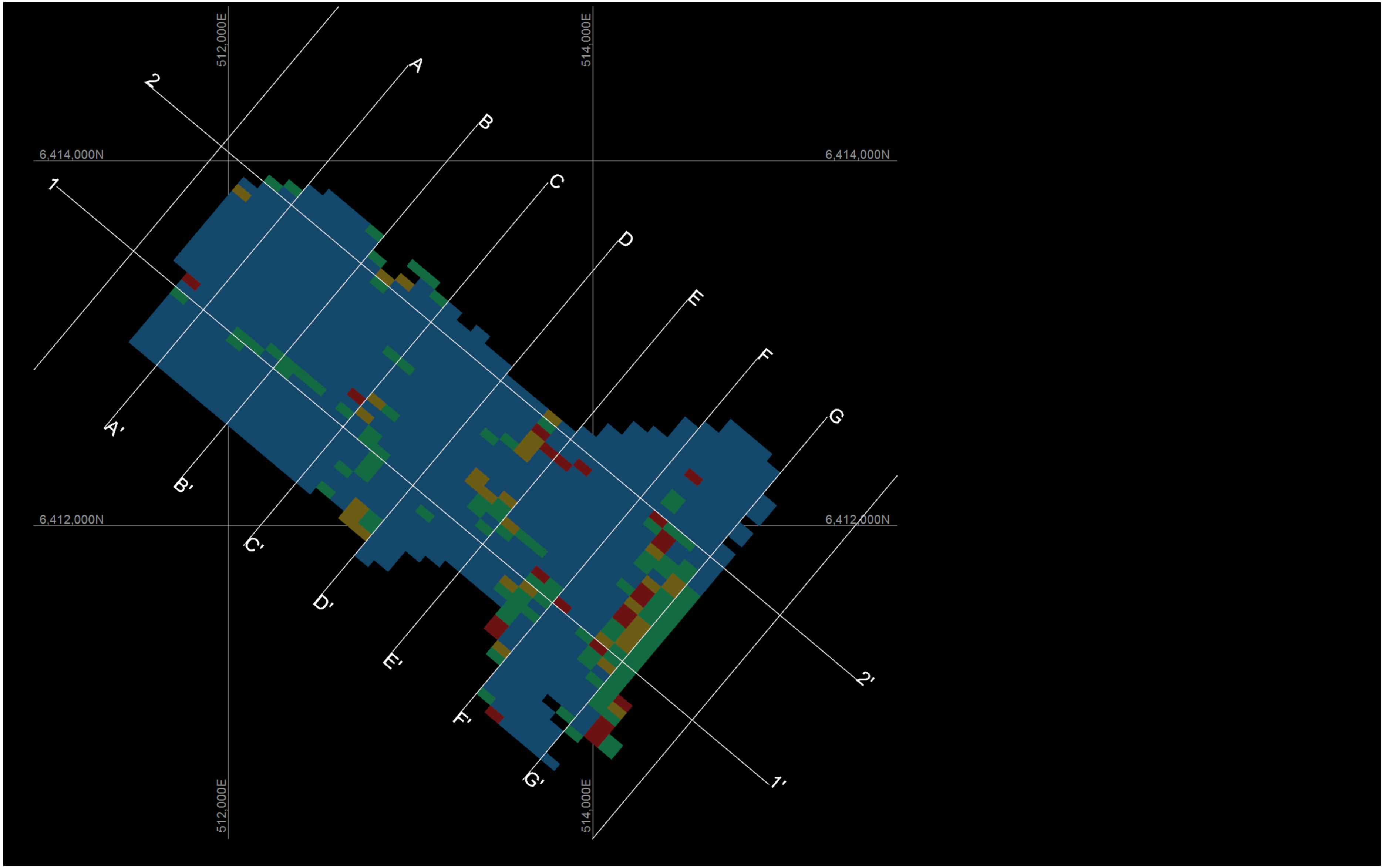

Figure 45. Grid System used for cross sections.

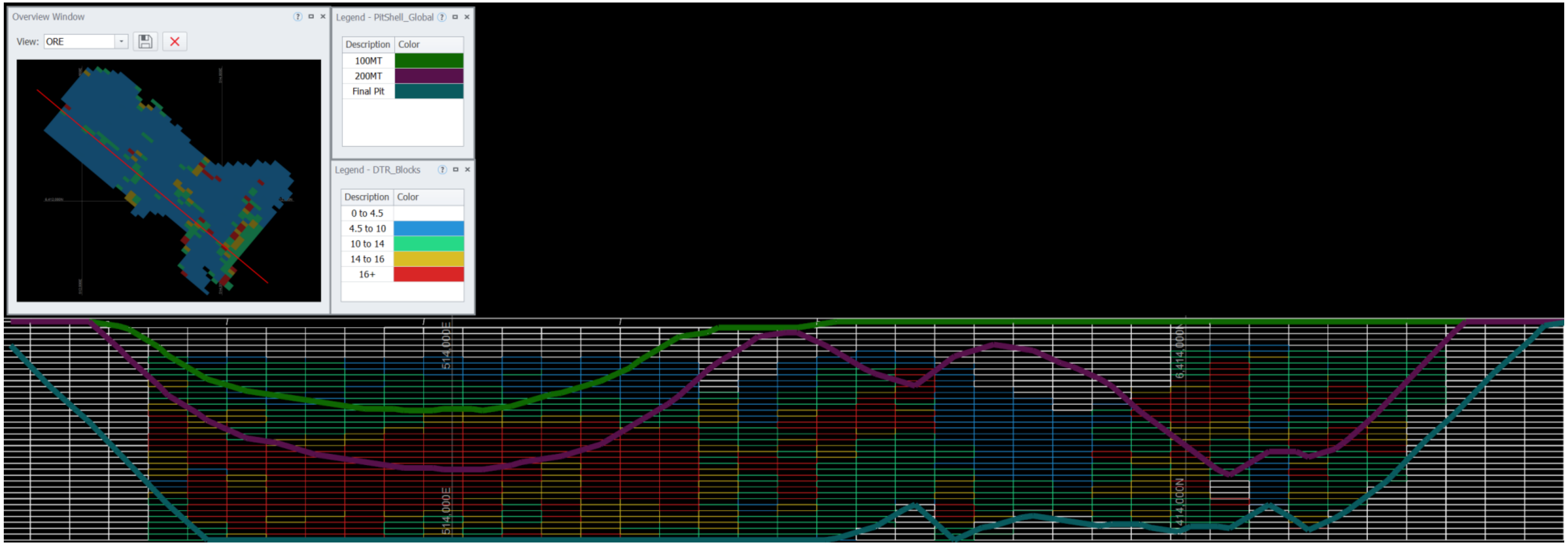

Figure 46. 1-1' Showing the pit shells for unconstrained ore (including potential).

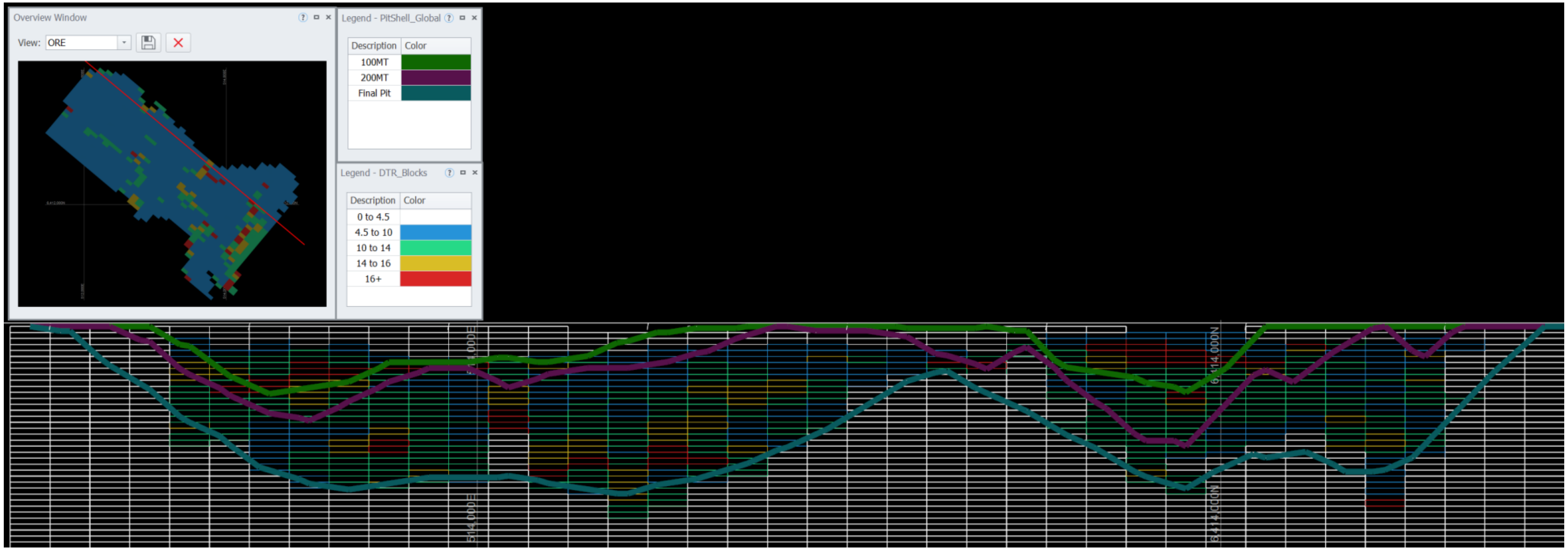

Figure 47. 2-2’ Showing the pit shells for unconstrained ore (including potential).

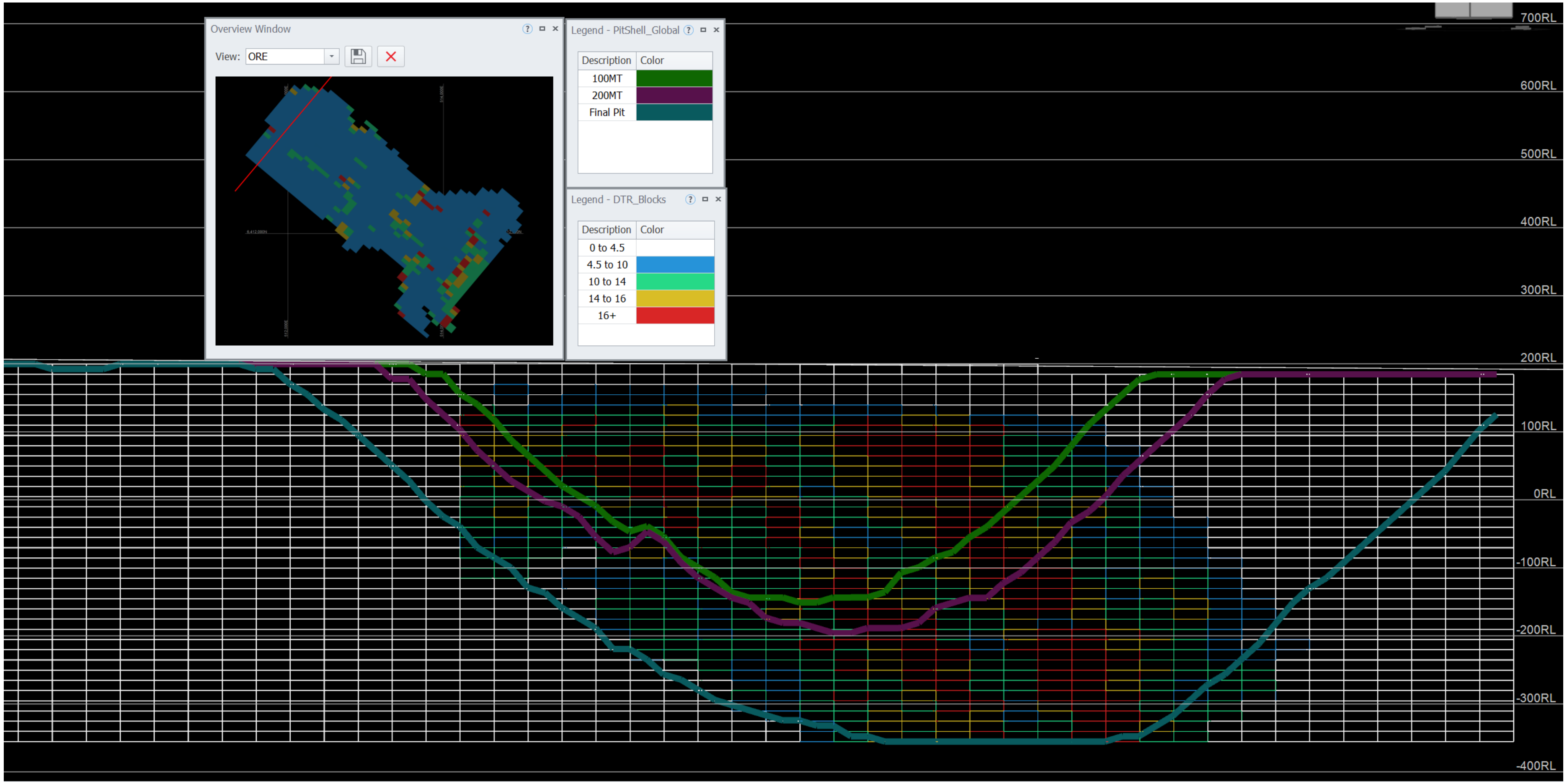

Figure 48. A-A’ Showing the pit shells for unconstrained ore (including potential).

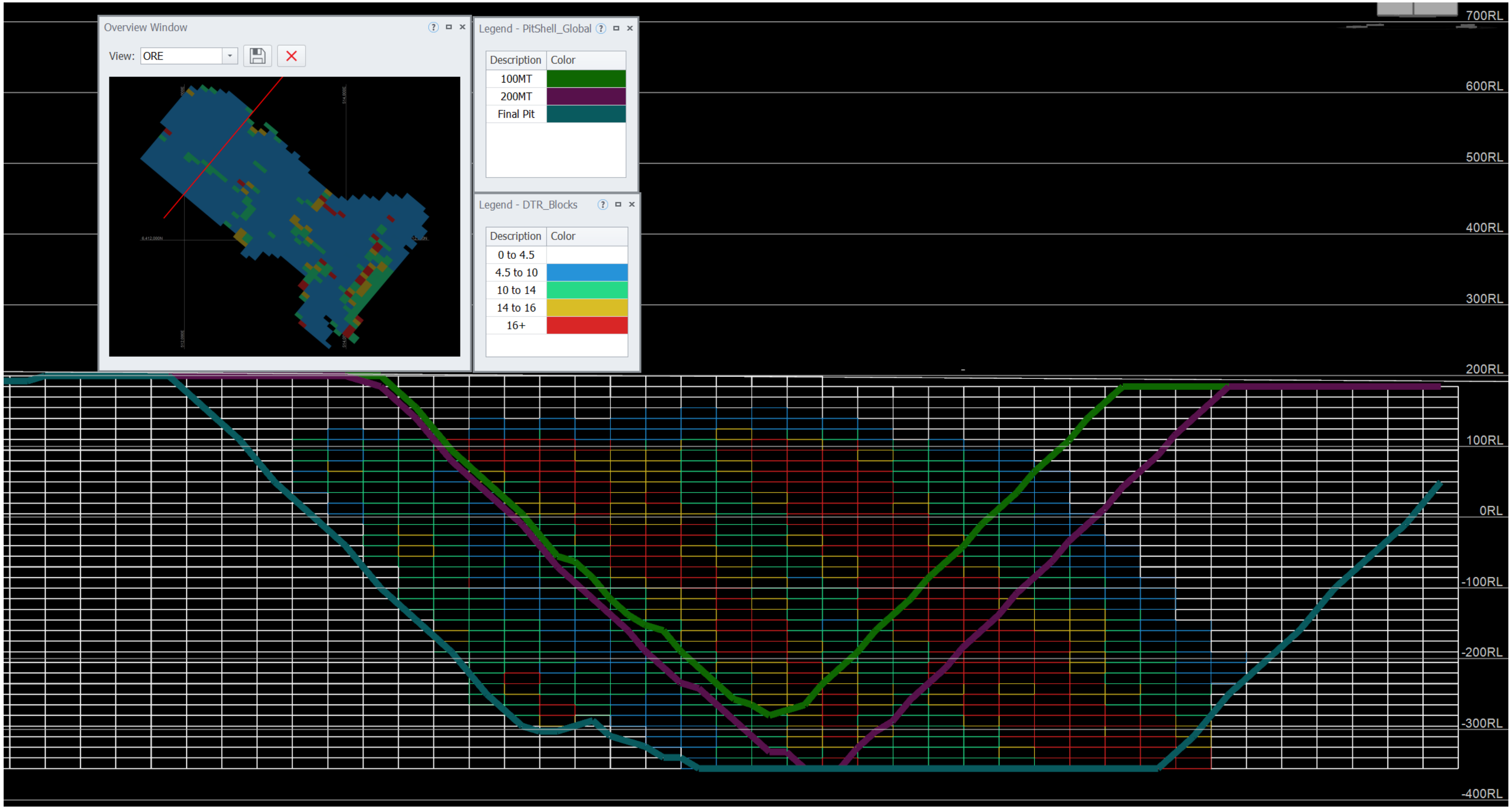

Figure 49. B-B' Showing the pit shells for unconstrained ore (including potential).

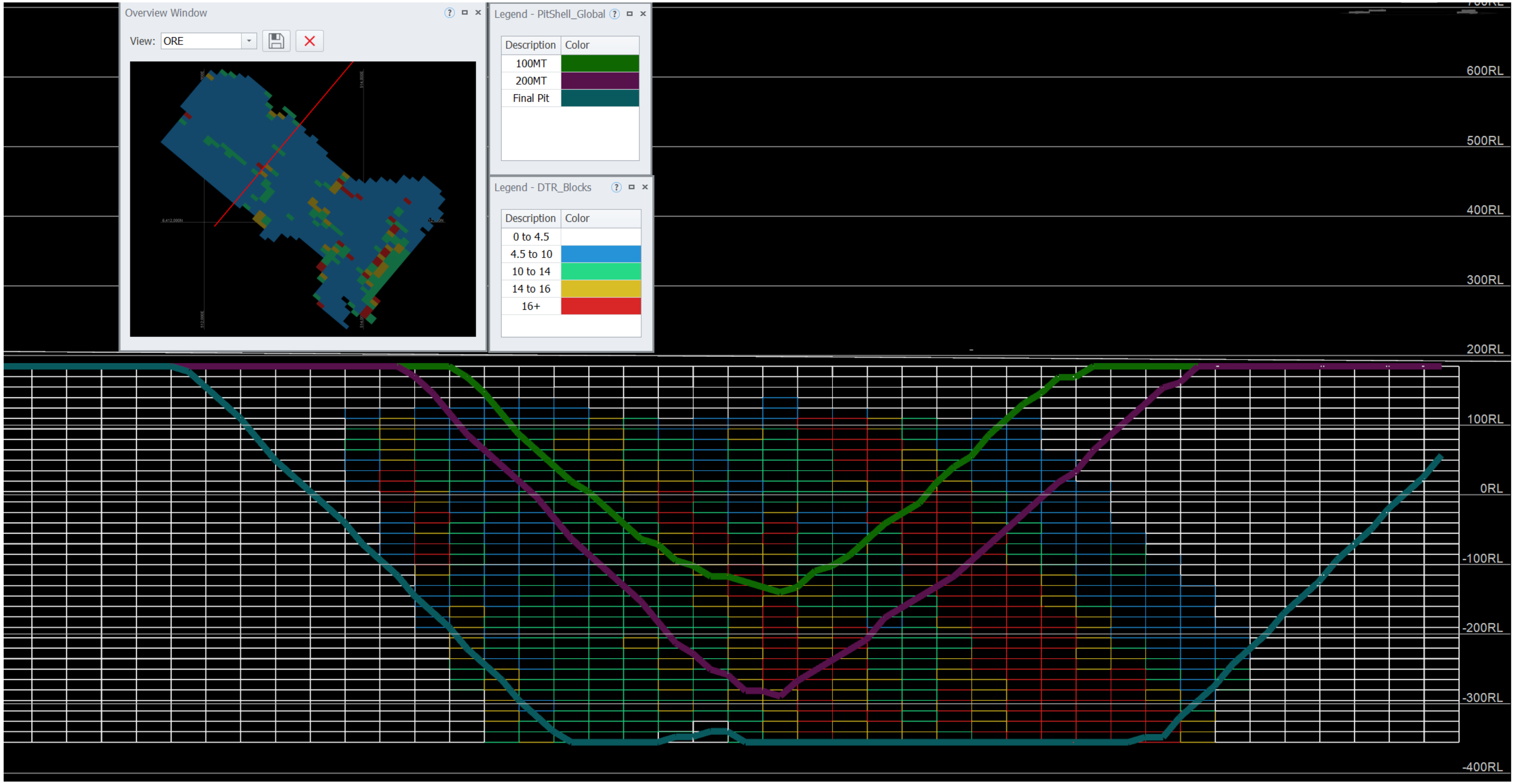

Figure 50. C-C’ Showing the pit shells for unconstrained ore (including potential).

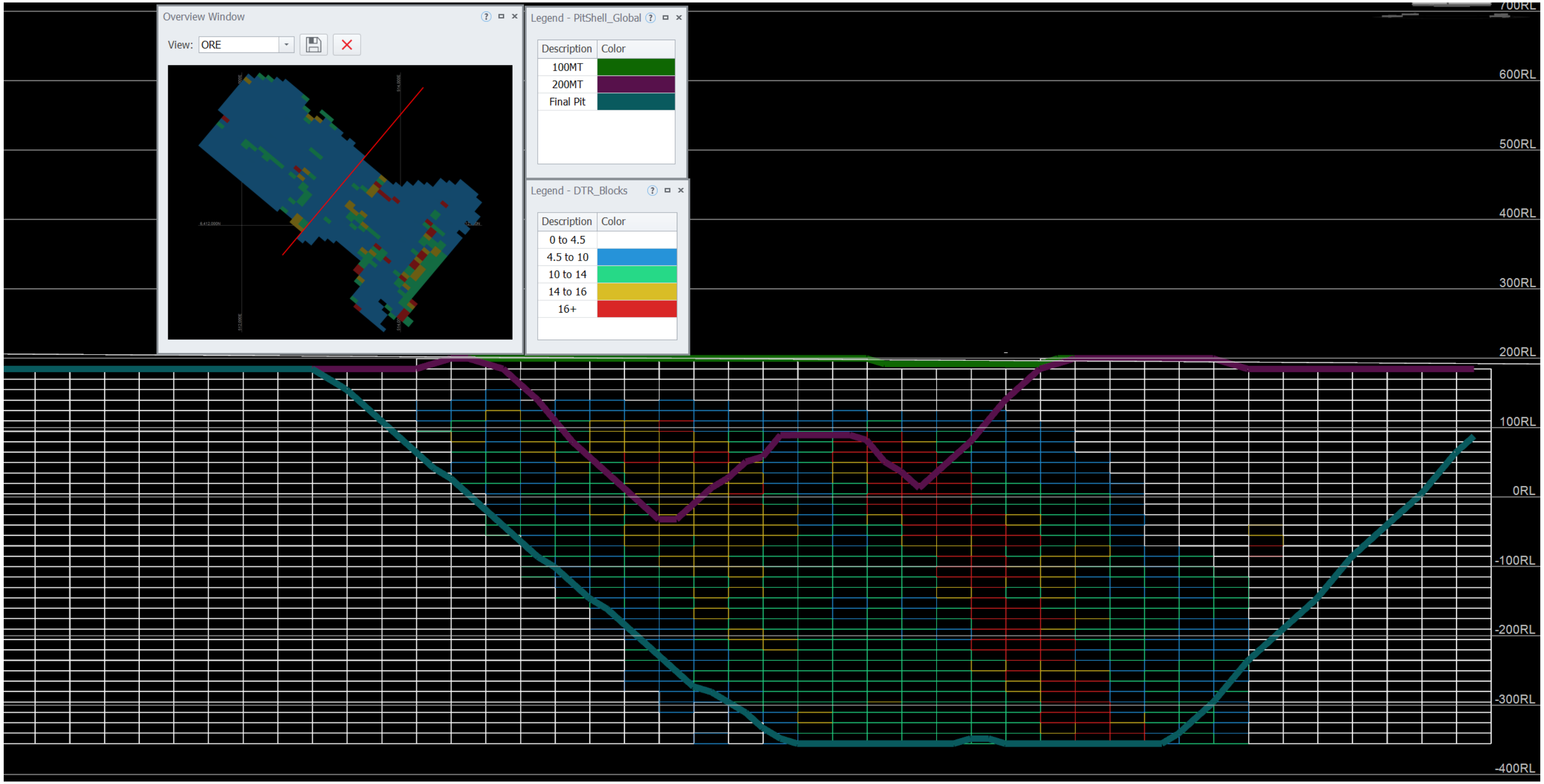

Figure 51. D-D' Showing the pit shells for unconstrained ore (including potential).

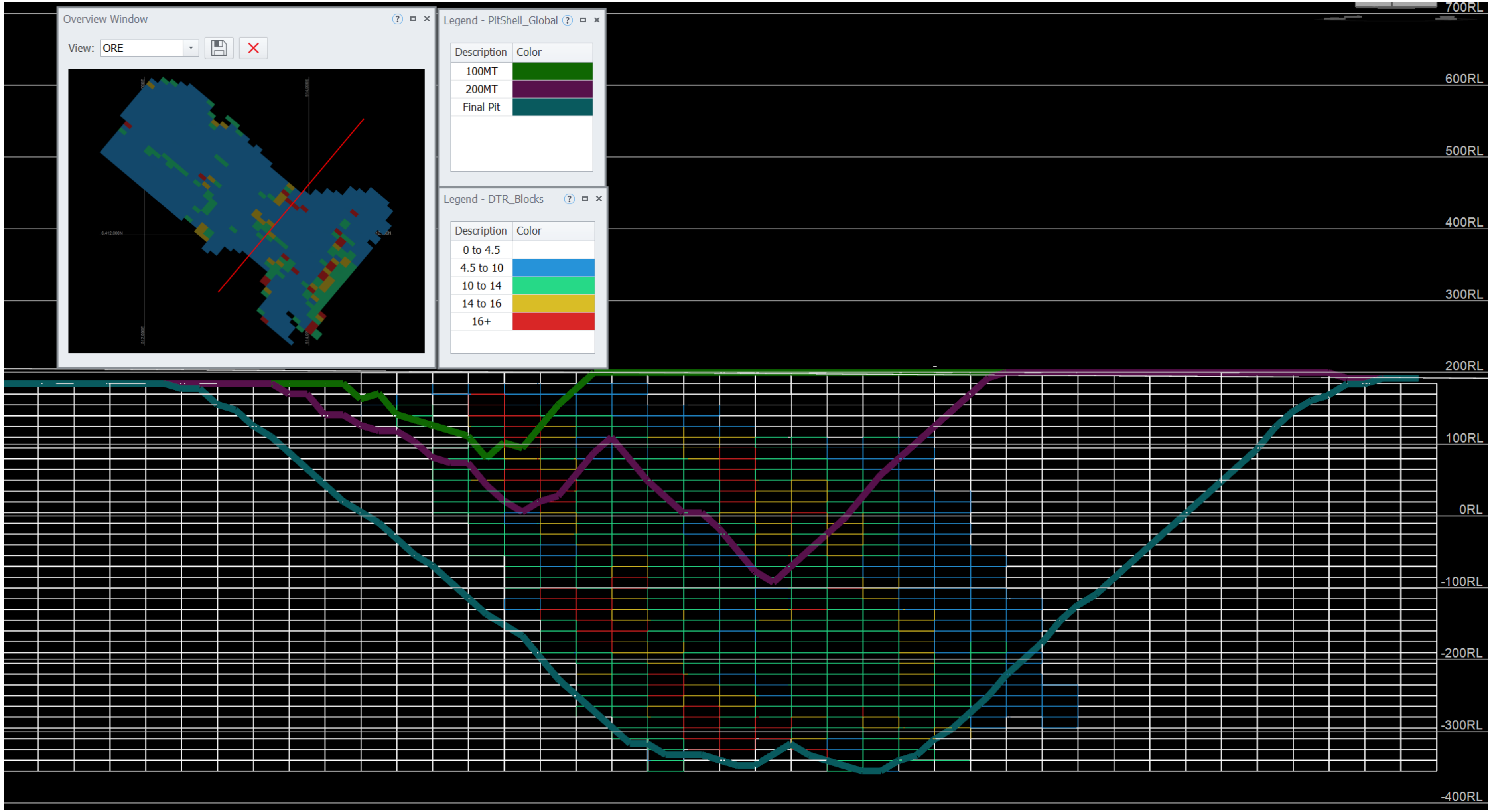

Figure 52. E-E' Showing the pit shells for unconstrained ore (including potential).

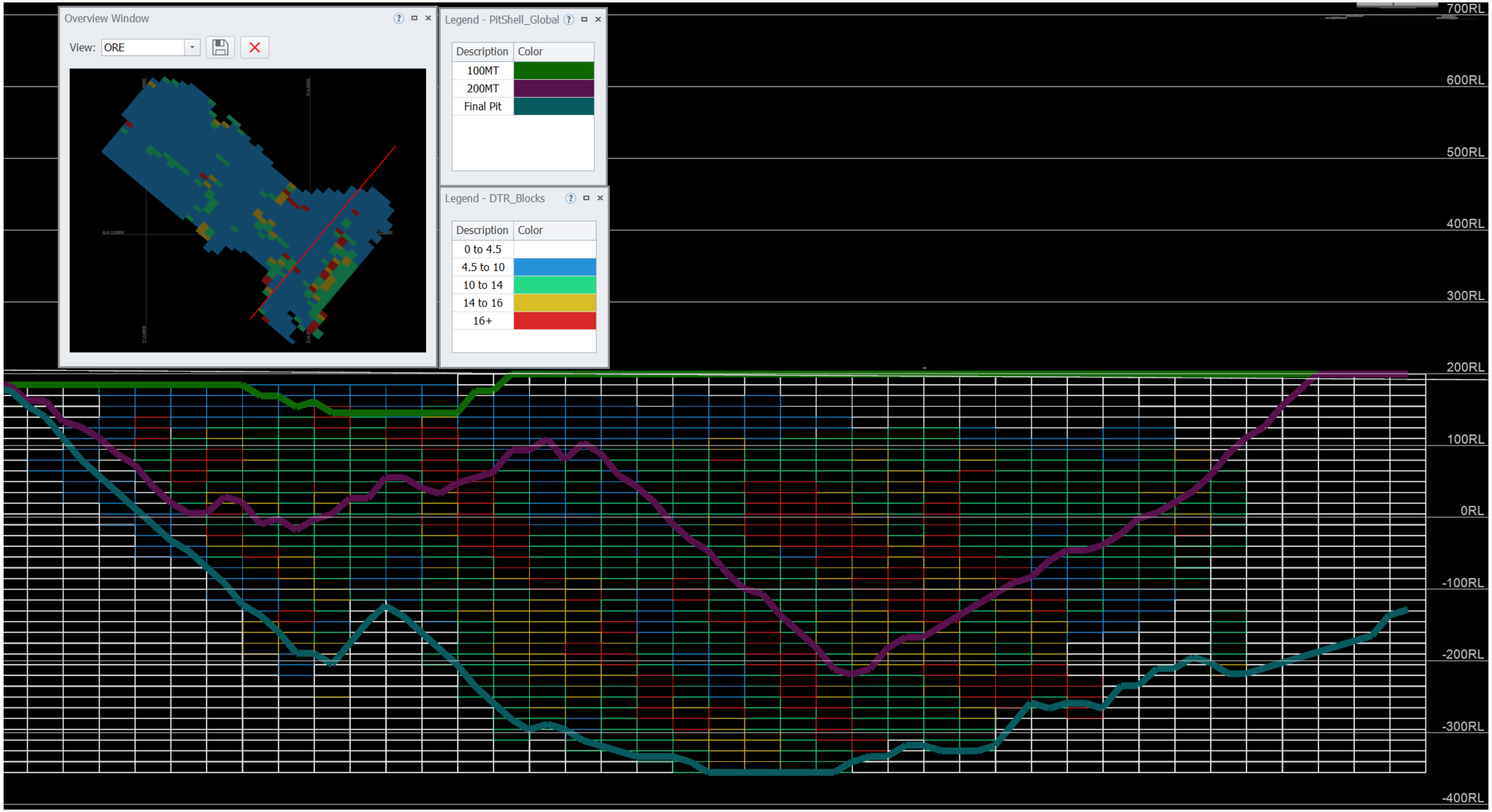

Figure 53. F-F' Showing the pit shells for unconstrained ore (including potential).

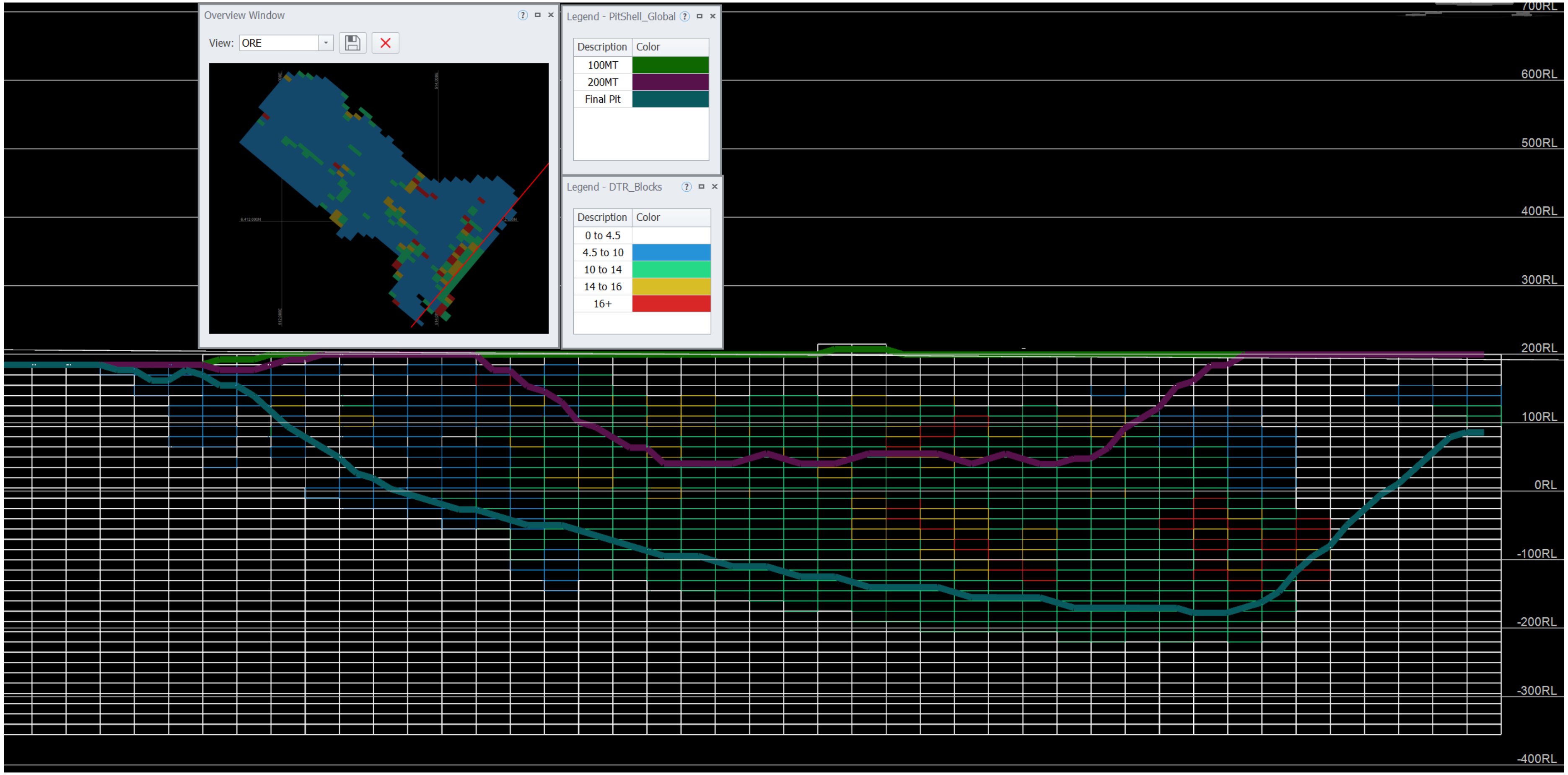

Figure 54. G-G' Showing the pit shells for unconstrained ore (including potential).

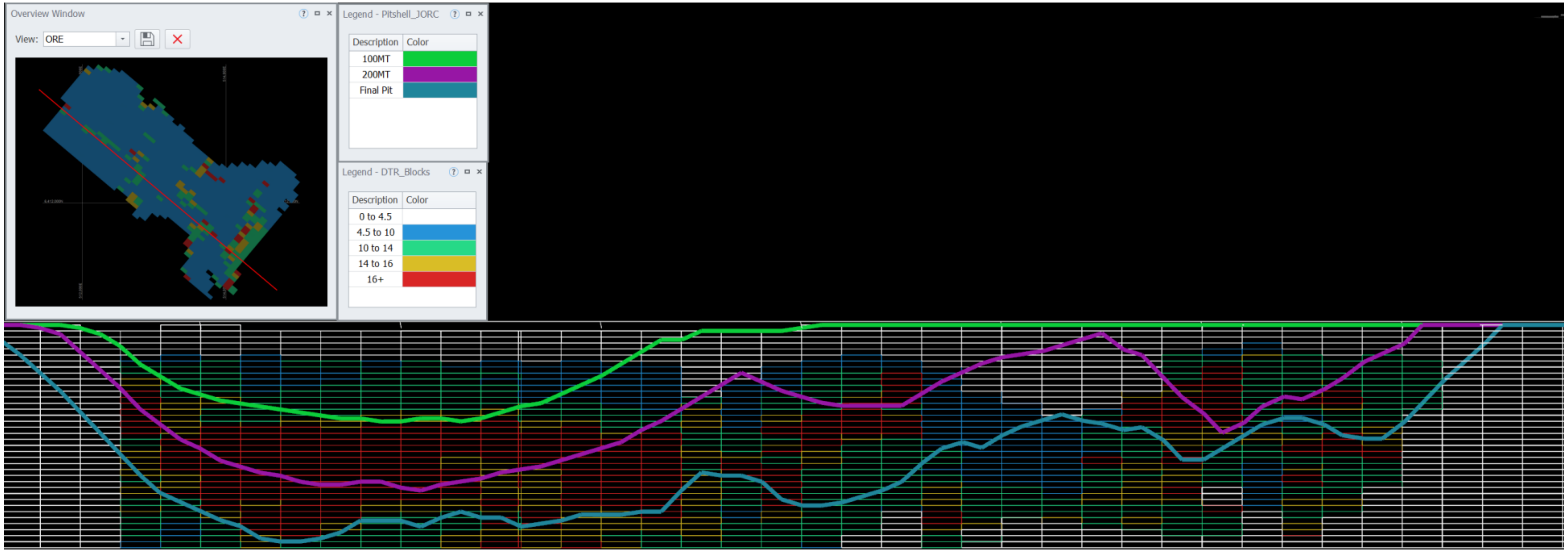

Figure 55. 1-1’ Showing the JORC pit shells for global ore.

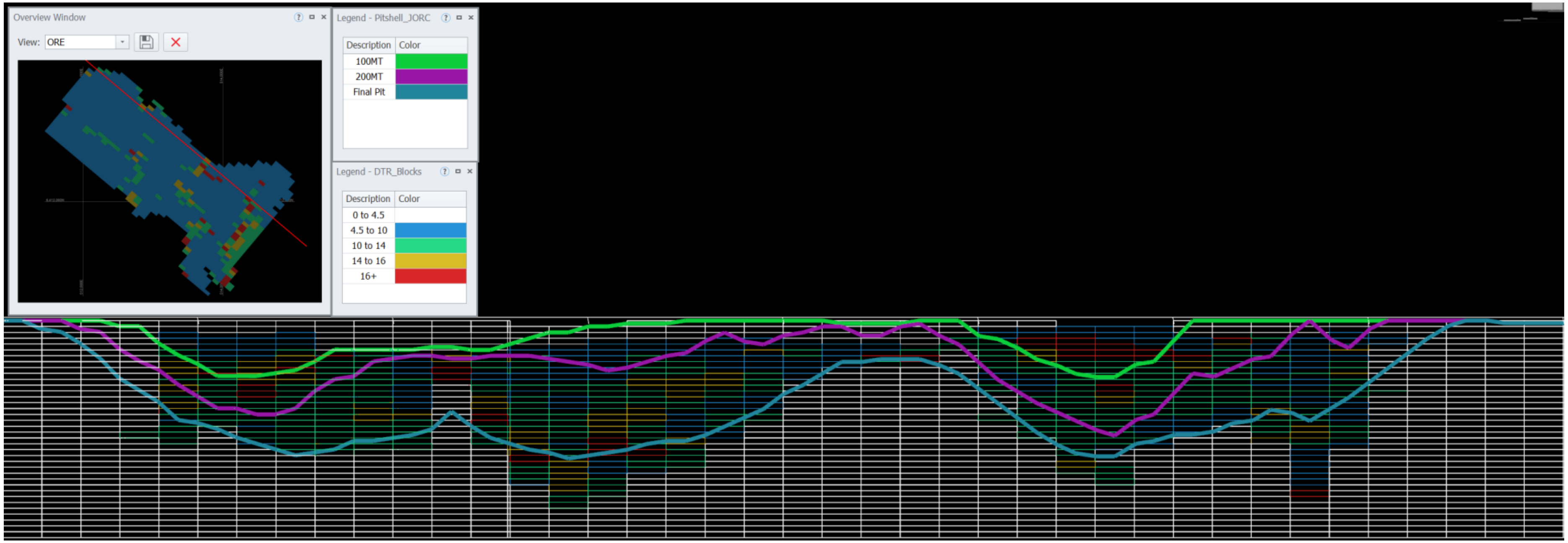

Figure 56. 2-2’ Showing the JORC pit shells for global ore.

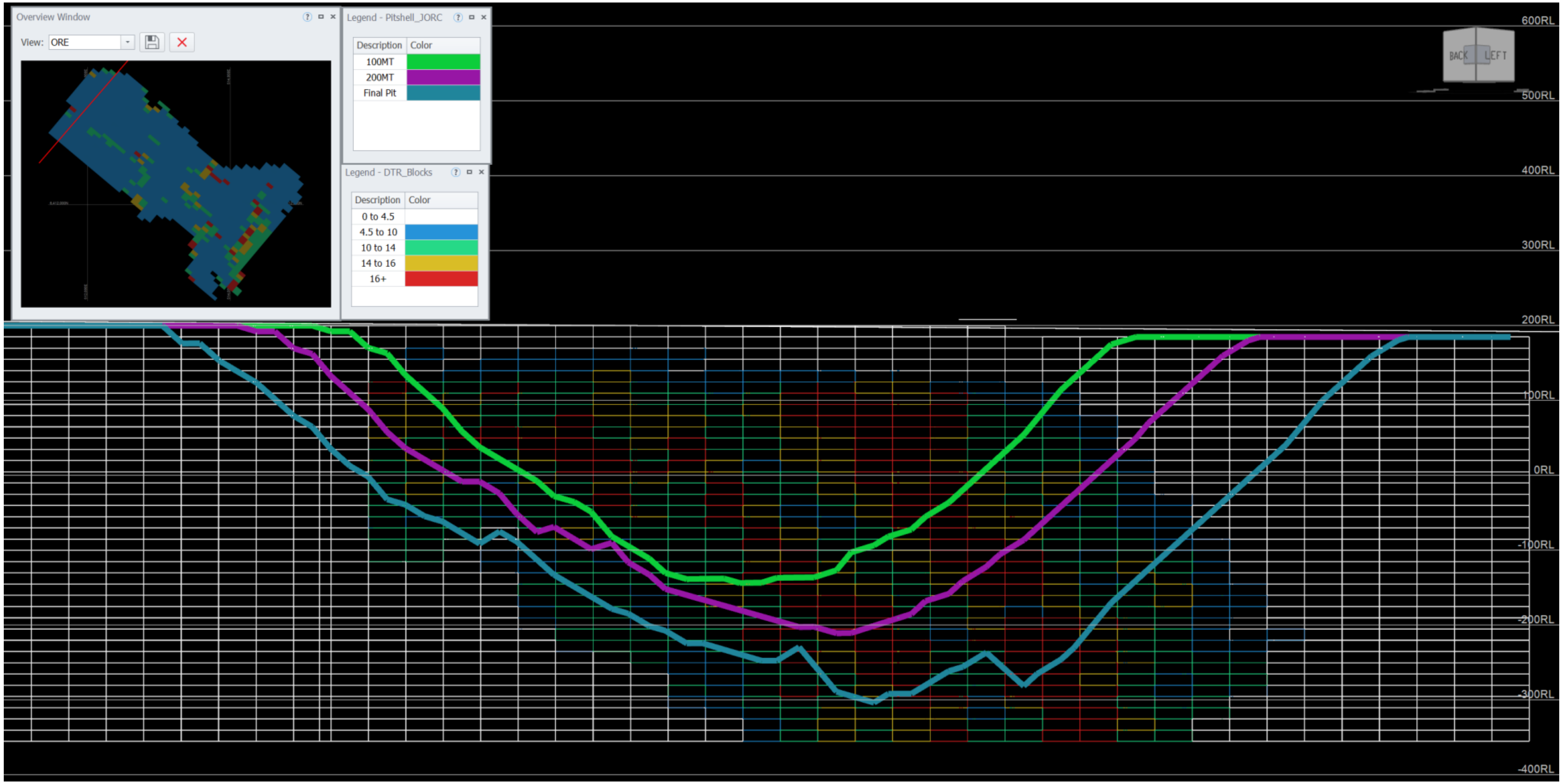

Figure 57. A-A' Showing the JORC pit shells for global ore.

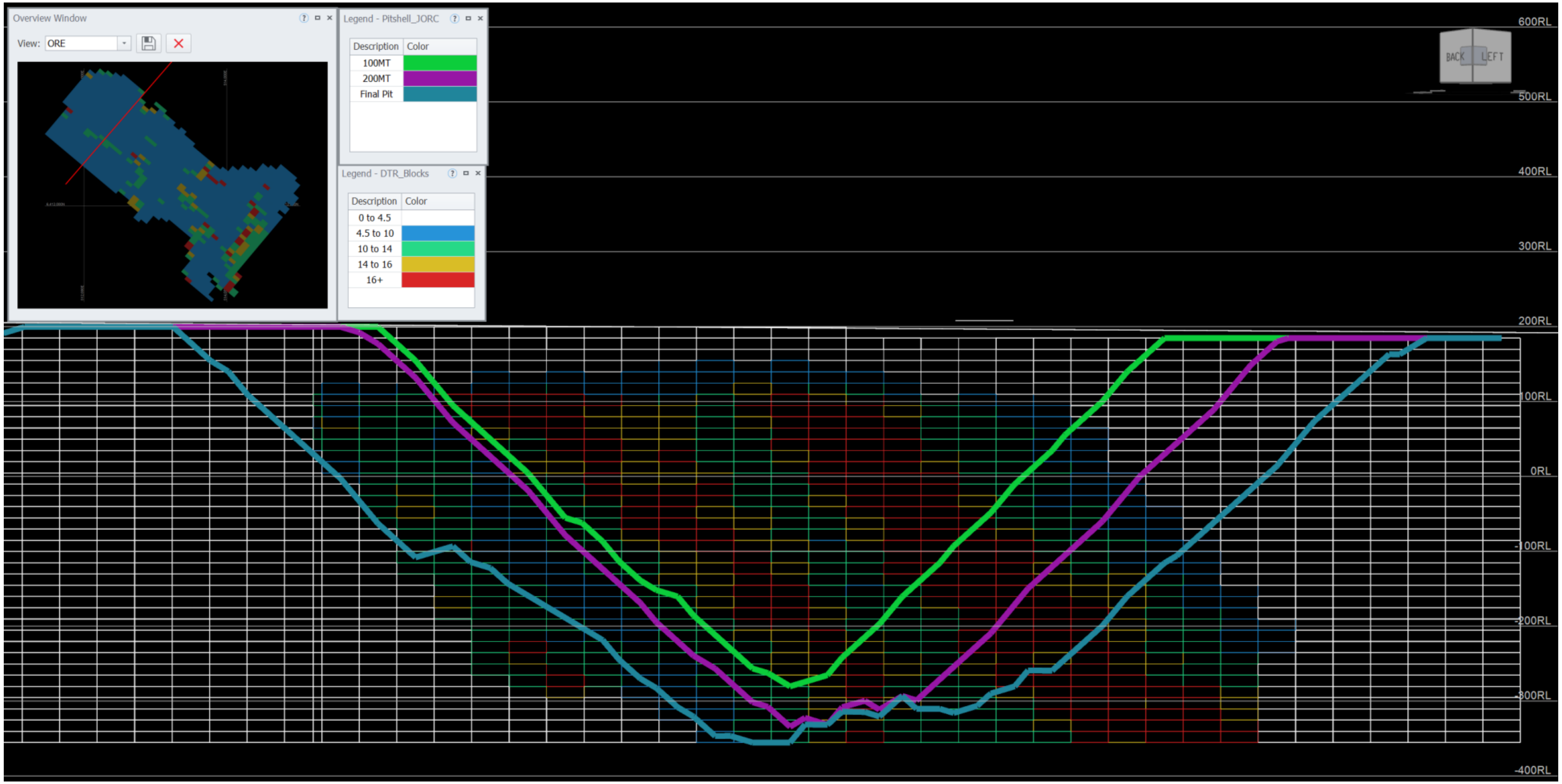

Figure 58. B-B' Showing the JORC pit shells for global ore.

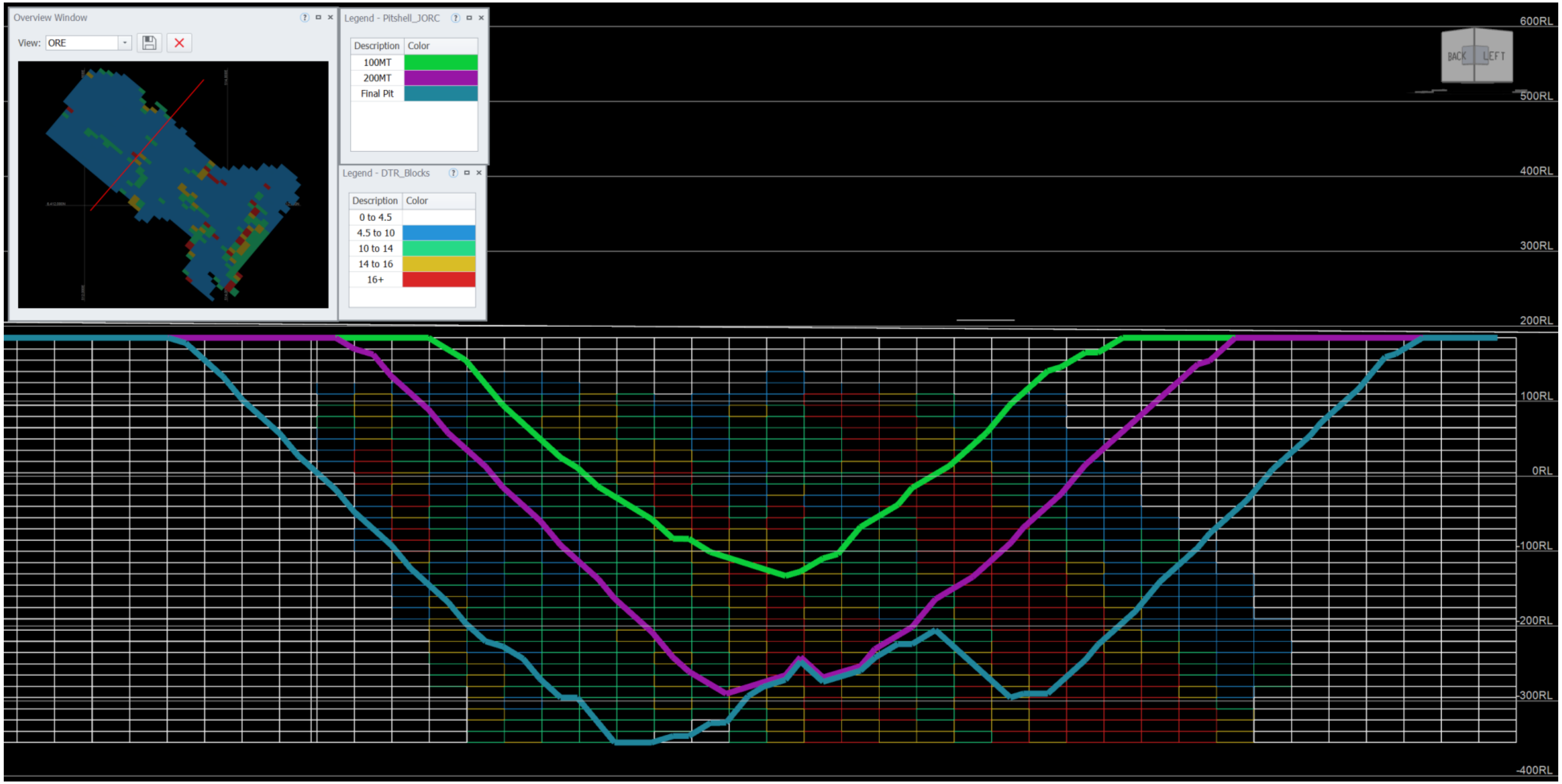

Figure 59. C-C' Showing the JORC pit shells for global ore.

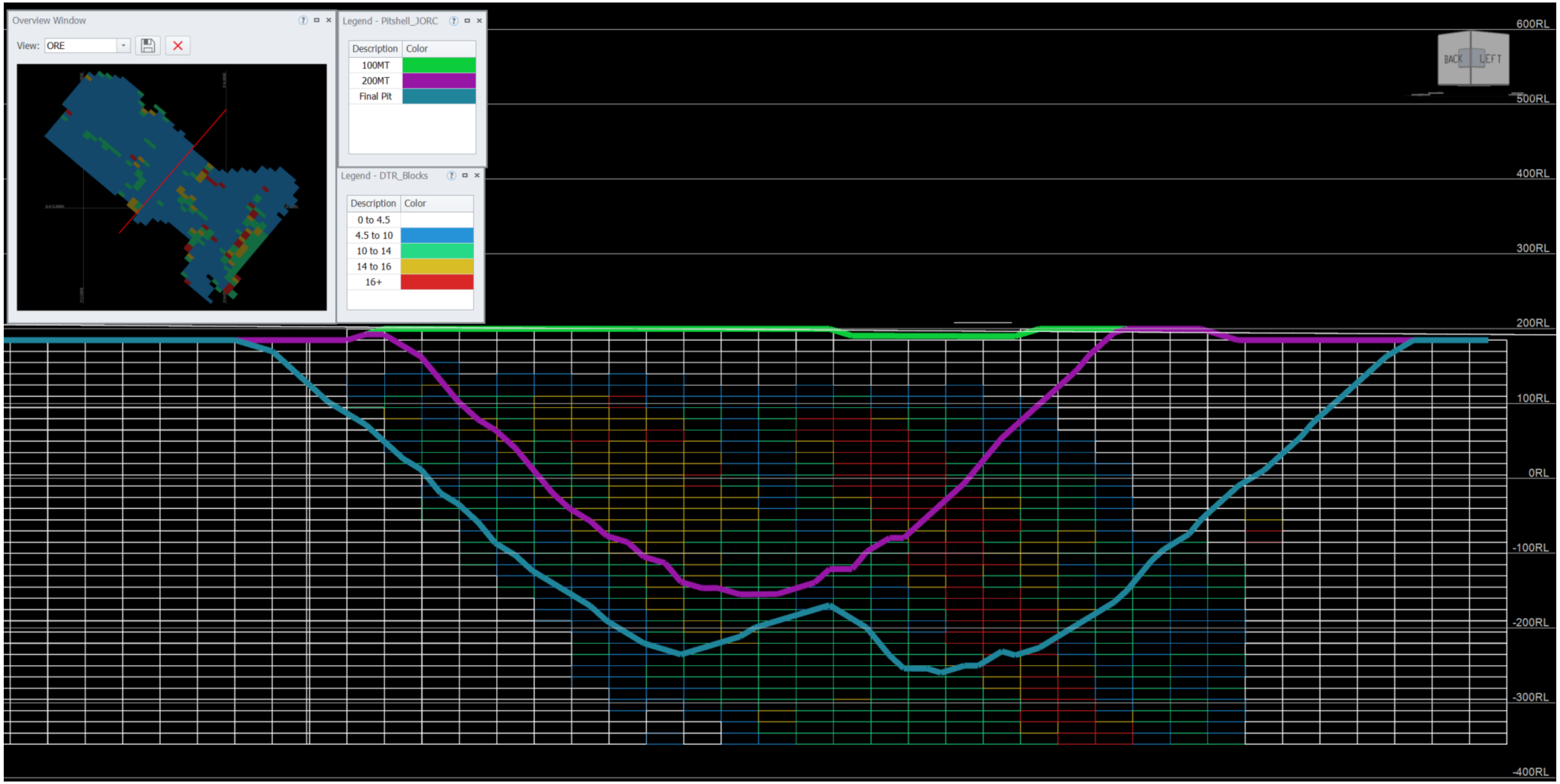

Figure 60. D-D’ Showing the JORC pit shells for global ore.

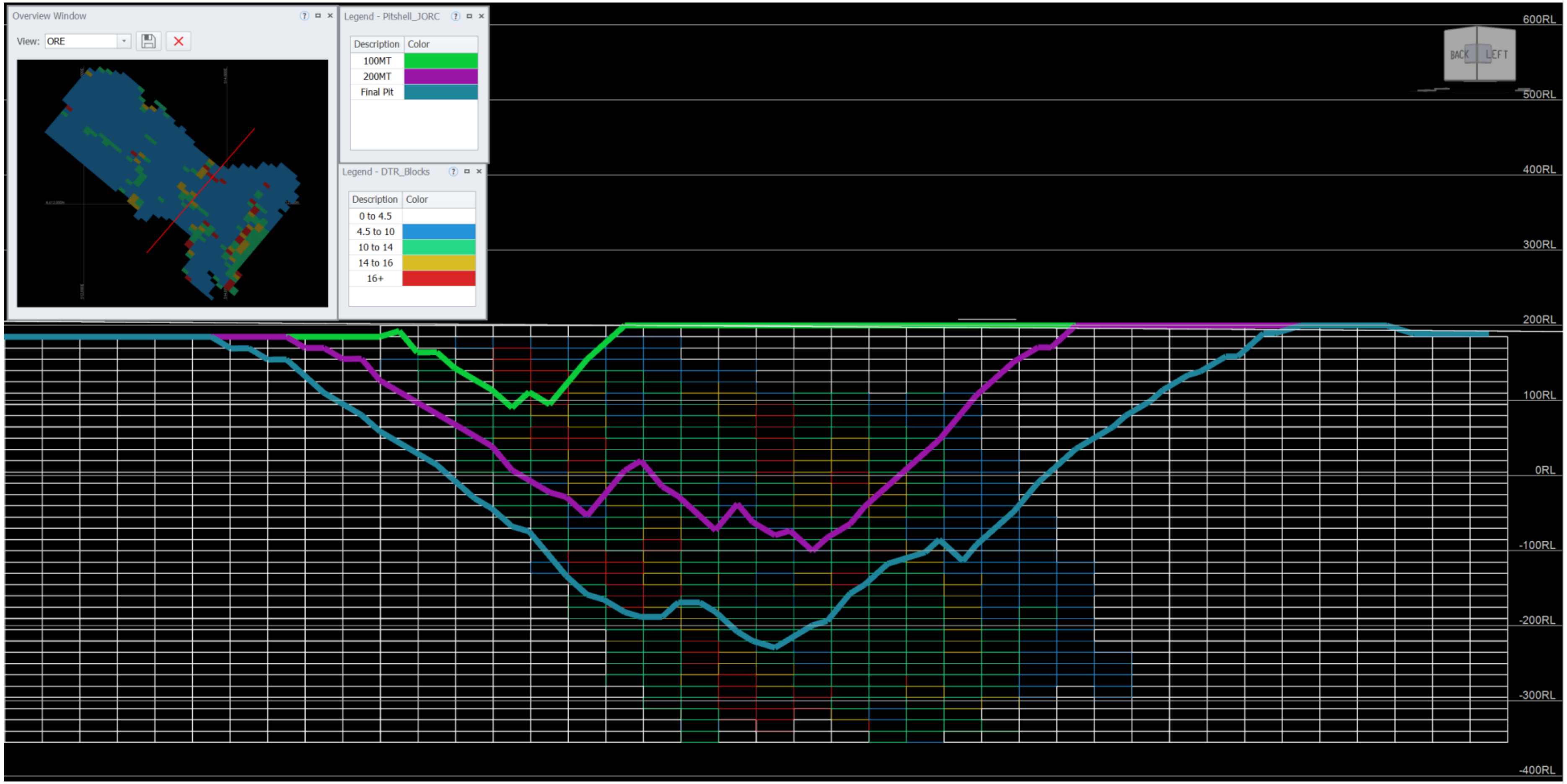

Figure 61. E-E' Showing the JORC pit shells for global ore.

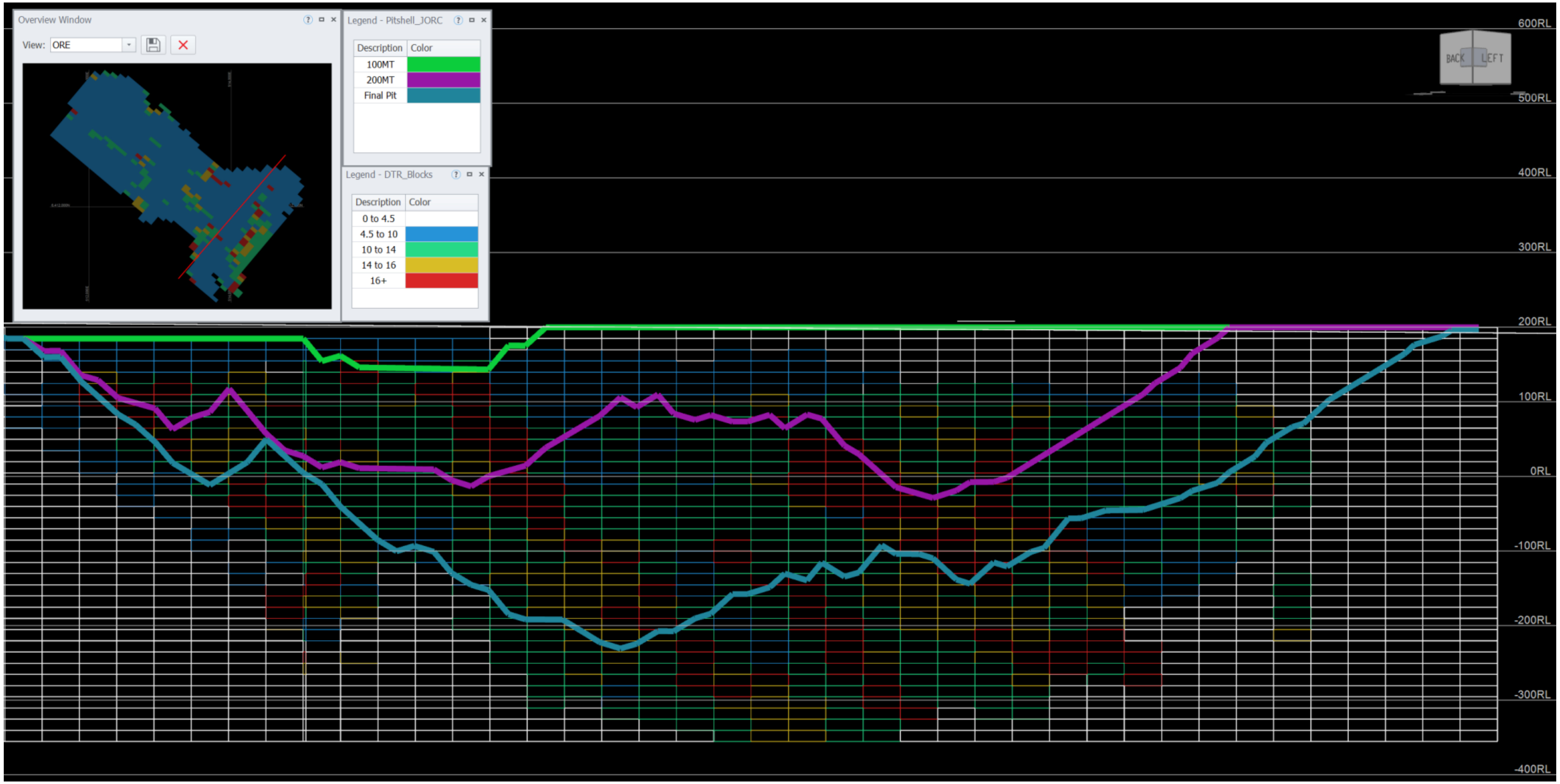

Figure 62. F-F’ Showing the JORC pit shells for global ore.

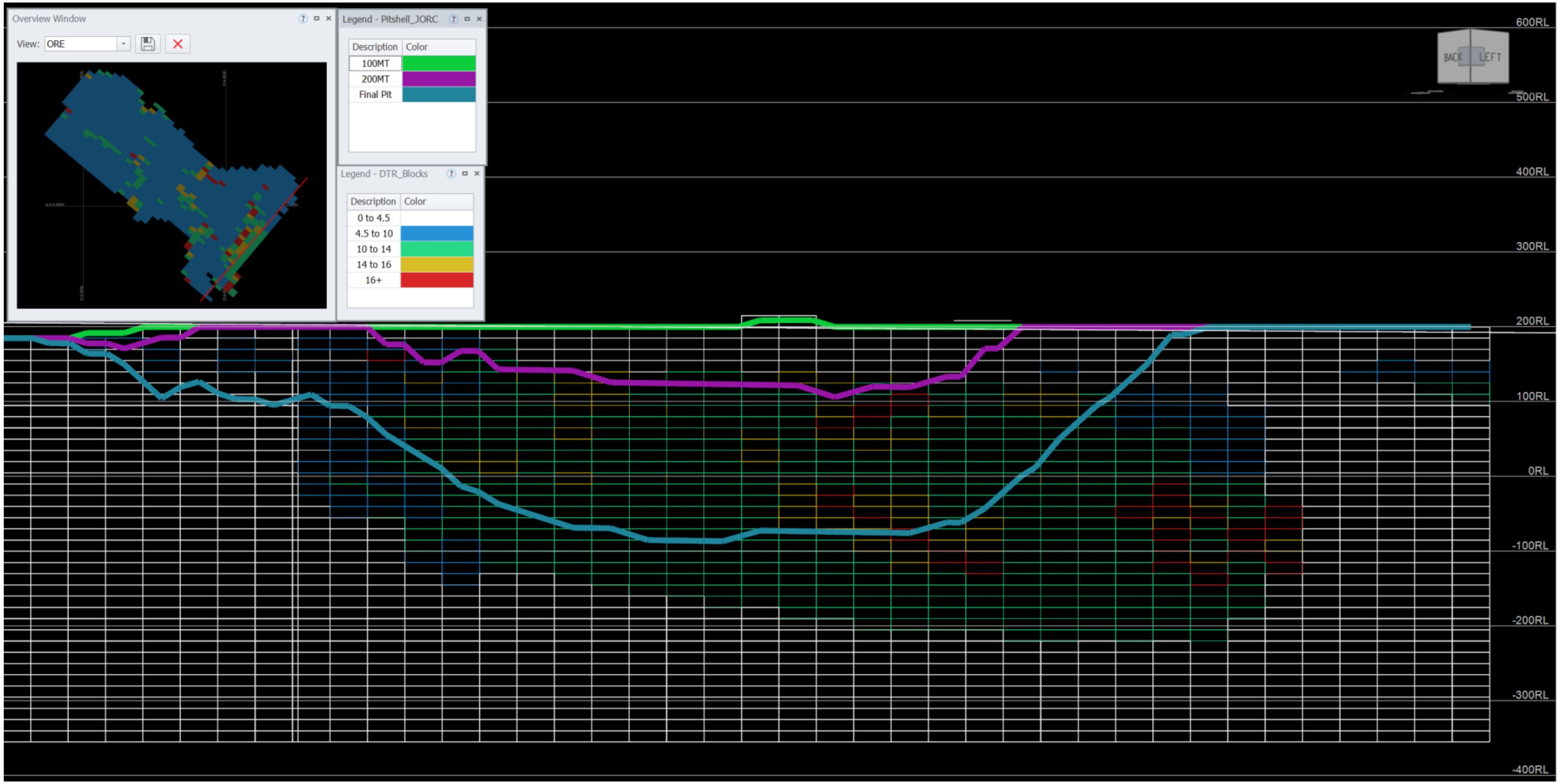

Figure 63. G-G' Showing the JORC pit shells for global ore.