1. TITLE PAGE

TECHNICAL REPORT
MINERAL RESOURCES AND RESERVES FOR THE PAULSENS GOLD MINE

Pilbara,

Western Australia

FOR

INTREPID MINES LTD

AUSTRALIA

MARCH, 2009

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3. SUMMARY

The Paulsens gold project is located in the Pilbara region of Western Australia and is owned and operated by Intrepid Mines Limited (IAU). After completion of a feasibility study in 2004, construction of the 250,000 tonne per year mine commenced in July of 2004 and concluded in early May of 2005. The first gold pour occurred in June 2005. Since then the mine and mill have been expanded to 330,000t per year and is currently producing at an average of 75,000 ounces of gold per year.

Paulsen's is located 7km off the Nanutarra Munjina road, 190km West of Paraburdoo, in Western Australia, Australia. The Paulsens project is wholly owned by Intrepid Mines Limited. Gold is found in structurally controlled quartz veining, hosted by a folded, sedimentary sequence. The mine is a follow up on the historic Melrose mine which had been worked in the 1930's.

Significant extensional work has been undertaken from underground drill positions. Surface exploration is at a pre-drilling stage for near mine (within 30km) targets. The operation is currently mining about 330,000t per year from underground, which is processed on site, with around 80 people on site at any time.

This technical report has been prepared in accordance with the requirements of National Instrument 43-101 as well as the Australasian Code for Reporting of Mineral Resources and Ore Reserves by appropriately qualified independent consultants and Intrepid staff.

The people involved in the preparation of this report are Mr Jonathon Abbott (Hellman and Schofield), Mr Per Scrimshaw (Creative Mined), Mr Craig Jones (Kadgie Mining), Don Russell (Intrepid Mines) and Mr Brook Ekers (Intrepid Mines).

As at 31 Dec2008 the following resources and reserves were estimated to remain at Paulsens:

Type	Measured		Indicated		Total Meas. & Indicated			Inferred		
	Tonnes T	Au g/t	Tonnes T	Au g/t	Tonnes T	Au g/t	Au Ounces	Tonnes T	Au g/t	Au Ounces
Upper			183,000	9.0	183,000	9.0	53,000	70,000	10	22,500
Lower			86,000	9.7	86,000	9.7	22,300	160,000	9	46,300
Other										
Total			269,000	9.2	269,000	9.2	79,800	230,000	9	68,800
ResEval December 2007 Resource Estimate					811,000	11.2	292,900	122,000	9.3	36,000

Table 1: Paulsens Deposit December 2008 Resource Estimate 4g/t Cut-off Grade 1200mRL – 670mRL, Undiluted Resource Remaining at 31 December 2008

The Ore Reserve is inclusive of the Mineral Resource shown above.

1200mRL – 670mRL, Diluted Reserve Remaining at 31 December 2008

Type	Proven		Probable		Total Proven & Probable			
	Tonnes T	Au g/t	Tonnes T	Au g/t	Tonnes T	Au g/t	Au Ounces	
Upper			166,600	6.9	166,600	6.9	36,900	
Lower			46,000	7.3	46,000	7.3	10,900	
Stockpiles	6,900	6.9			6,900	6.9	1,500	
Total					219,500	7.0	49,300	
Creative Mined December 2006 Reserve Estimate					591,000	8.6	163,000	

Table 2 Paulsens Deposit December 2008 Reserve Estimate 4.4 to 6.3 g/t Cut-off Grade

The 79,800 ounces of indicated resource, does not compare favourably with results estimated for 31 Dec 2007 (292,000 oz of indicated and measured) minus 2008 production (81,238). A drop of 130,000 ounces. In part this is due to a more conservative approach to classification (detailed in section 19), and also due to using site based interpretations and vastly more available data.

The previous resource estimate prepared by Resource Evaluations in May 2008 (31st December 2007 report) was a mining depleted resource. It is significant to note that the general geometry of the mineralisation defined in this previous resource is generally unchanged from the mineralisation defined in the original 2004 resource estimate and thus reserves and resources were potentially overstated by not remodelling the mineralisation on updated site experience. In 2006 the resource was produced offsite based on what was a legitimate interpretation for 2004, but not as valid in 2006, extended with new drill results. In 2007 the resource was estimated purely by excluding mined areas from the 2006 model.

The reserve conversion however has been much better and Reserve numbers are more closely aligned. At 31 Dec 2007 there were 163,000 ounces in Proven and Probable, compared to 49,300 at 31 Dec 2008. Minus production of 81,238 this leaves a deficit of 30,000 ounces.

Source of differences in Resource and Reserve numbers:

Interpretation

Early interpretations were often dependant on snapping to grade in drill holes. Through mining experience it has become apparent that structure, not grade are the most continuous. Focussing predominantly on grade led to unrealistic, narrow, high grade zones, especially to the west of Upper Zone.

Structural complexity

The Paulsens orebody has become structurally more complex below approximately the 900mRL, which has invalidated the reliance of earlier resource on extending grade down plunge.

Cutoff

Basing a sample cut-off grade on data that was skewed to higher grades, led to an unnaturally high cutoff. Higher cut offs were then smeared further.

Classification

The Hellman and Scofield approach to resource classification is a lot more structured and based on actual data rather then the holistic approach used before. This has downgraded all measured to indicated or inferred, and a lot of indicated to inferred (which has increased)

Accessibility

Mining to updated production models (which are estimated using detailed infill drilling), has sterilised areas that might have been economic in the older resource. These areas would not have been removed from the 2007 resource as it was purely a depletion exercise.

Resource Reserve Conversion.

An upside to the more conservative classification is that the relative resource to reserve conversion is much higher.

4. INTRODUCTION

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This report details the methodology and results of the March 2009 mineral resource and mineral reserve estimates for IAU's Paulsens project prepared for release in conjunction with the AIF. Resources were estimated by a Mr. Jonathon Abbott from Hellman and Schofield based on site work supplied by Mr. Brook Ekers. Reserves were estimated by Mr. Per Scrimshaw from Creative Mined, again based on site work. Both Mr. Abbott and Mr. Scrimshaw have visited the site (Nov 2008 and Mar 2009 respectively).

5. RELIANCE ON OTHER EXPERTS

Not deemed applicable due to both resource and reserve reports submitted by independent experts.

6. PROPERTY DESCRIPTION AND LOCATION

The mine site covers 134 hectares, inside a cattle proof fence and containing the mine, processing plant, tails storage area, waste rock storage areas, explosive compound, laydown area, crushing facility, core yard, water and effluent treatment, all offices, workshops and camp. An exploration coreyard is located 3km from site and is still to be moved, or buried at end of mine life

Coordinates of the portal to underground workings are 22.579205°S 116.242403°E
The mine and infra-structure are wholly contained on mining leases M08/099 and M08/196 both of which are in good standing. Expiry dates for these tenements are13/2/2011 and 2/3/2020 respectively.

Annual expenditure requirement for the 2 relevant mining leases is A$18,400 and A$80,000, and annual rents are A$2,752.64 and A$11,986.00

Total annual rents for all tenements (predominately for exploration) is A$84,700 and an expenditure commitment of A$922,320

The historic Paulsens/Melrose, Belvedere and Monster Lode gold mines, and Tombstone (historically known as Belfry or Blacks) copper mine is located within IAU's tenement holdings though outside mining leases M08/099 and M08/196. Notice of intent to mine on M08/099 and M08/196 is NOI number 3161.

The PPKP (local indigenous group the Puutu Kunti Kurrama and Pinikura people) are due payments of A$2 per ounce poured. The West Australian government realises a gold royalty of 2.5% on metal value.

The Paulsens Gold Mine is some distance from the coast, National Parks, major river systems, wetlands, DEP Environmental Protection Policy areas and the occupied town sites. Both tenements and associated Miscellaneous Licences are granted and are wholly owned by Intrepid Mines Ltd.

In the Wyloo - Mt Stuart District, cattle raising or activities associated with mining are currently the primary land uses. Mine closure criteria are being established to support a pastoral end use. The Paulsen operations are located on the Mt Stuart Pastoral Lease and the Project Area has been fenced to exclude stock.

7. ACCESS, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Paulsens project is located in the Pilbara region of Western Australia, approximately 180 km west-northwest of Paraburdoo and 6km north of the sealed highway between Paraburdoo and Nanutarra at an elevation approximately 200m ABSL(Figure 1). Access to the mine site is via a well maintained gravel road which connects with the highway. Regular daily flights from Perth to Paraburdoo provide access to the region and mine site personnel use this or a charter flight to nearby Wyloo Station air strip. Mining personnel are generally sourced from Perth.

The climate is hot and generally dry and experiences hot summers and mild winters. The area is on the fringe of the cyclone belt and can receive significant rainfall as a result of cyclone activity. However, the project is far enough inland to generally not be impacted by cyclonic winds. The local topography is gently rolling with some prominent mesa-type hills and a small but steep hill in the immediate vicinity of the project marking the outcrop of the Paulsens quartz vein. Vegetation comprises low shrubs, spinifex and occasional trees and the principal agricultural activity is low density cattle ranching.

Figure 1 Paulsens Location

The mine operates year round. Power is generated on site with diesel powered generators operated by Power West. Diesel is supplied by road from Karratha, Western Australia, 400km away on sealed roads. Water is pumped from near surface aquifers with relevant licenses in place.

Waste rock is stored on site for future use as armouring on the nearby tailing storage area once mining ceases.

8. HISTORY

Historical underground mining was carried out at the project area in the 1930's and modern exploration of the area commenced in the early 1980's. The Paulsens property changed hands between different companies until 2004 when NuStar Mining Corporation (NuStar) was created, with the objective of implementing the underground development of Paulsens. During 2006, NuStar merged with Canadian company Intrepid Minerals Corporation to form the dual listed entity Intrepid Mines Limited (IAU). In 2008, further corporate activity resulted in the merger of Intrepid and Emperor Mines Limited. The Paulsens Mine is wholly owned by Intrepid.

The current operation at Paulsens commenced in late 2004 with the development of an underground mine. A conventional CIL treatment plant with nominal capacity of 250,000TPA was commissioned in 2005. Since commencement of operations up to and including 31st December 2008, approximately 1,144,000t of ore has been processed with a mill recovery of 93.8% and a grade of 7.26g/t for 266,982 ounces produced.

The previous resource estimate was carried out in December 2007. Since that time further exploration drilling has been carried out from underground primarily targeting down plunge extensions. Ongoing grade control sampling and stope definition drilling has occurred as a

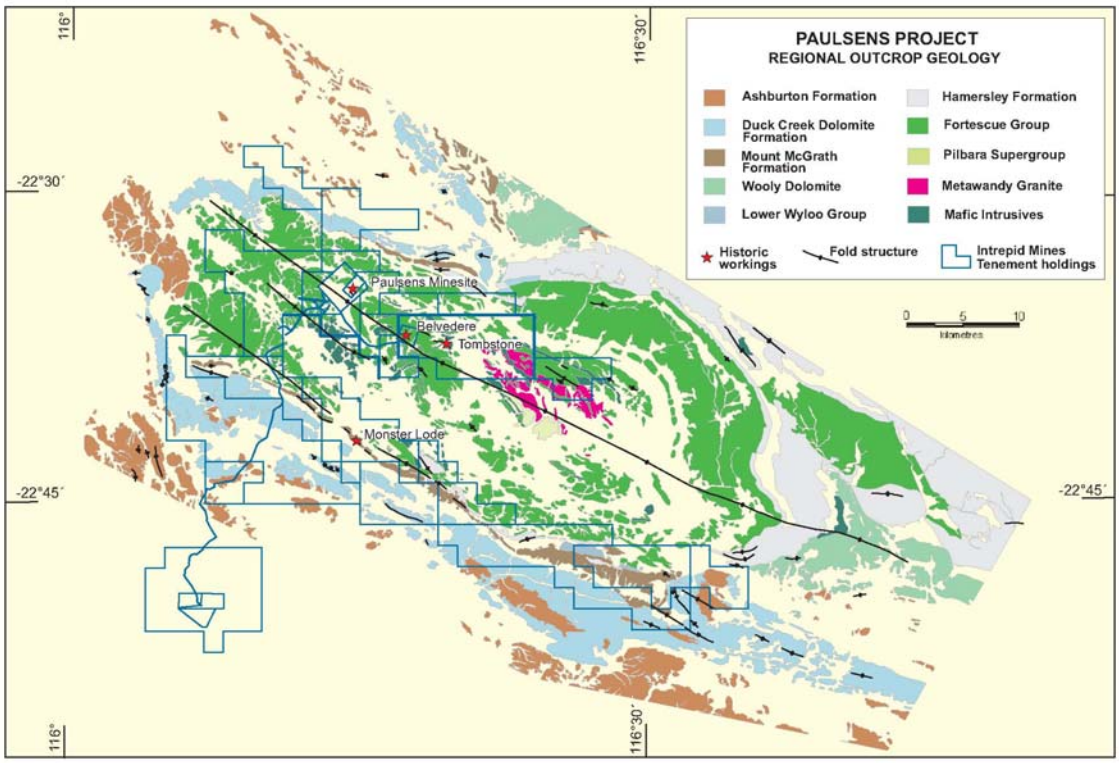
routine function in the mine and this data is also used in the resource update. The May 2008 resource statement took the December 2006 model and depleted it for those areas where mining has occurred up until December 31, 2007. No re-estimation of blocks was carried out at this time.

9. GEOLOGICAL SETTING

Regional Geology

The Paulsens project is located within the Ashburton geological province, an arcuate belt of Proterozoic Wyloo Group volcanics and sediments lying along the southern and western fringe of the Achaean Hamersley Basin iron ore province. The Wyloo Group is overlain stratigraphically by the Proterozoic sediments of the Bangemall Basin to the south.

Figure 2 Paulsens General Regional Geology



Local Geology

The Paulsens deposit lies within the Wyloo Dome, a west-northwest trending regional anticlinal and domal structure. Archaean Fortescue Group metasediments and metavolcanics of the Hamersley Basin are exposed in the core of the dome and form the host rocks of the Paulsens deposit. The sequence is cut by a series of steeply dipping gabbro and dolerite dykes.

The principal regional structure is the Wyloo dome, with the principal fold axis in the Paulsens area plunging to the northwest at approximately 35°. The host argillite sediments of the Fortescue Group are exposed in the core of the dome, and in the Paulsens area, sub-outcrop and dip to the northwest. The Melrose Fault, (predominantly bedding plane slip in

response to folding) displaces the Paulsens gabbro, showing an apparent dextral displacement of around 250m. The sediments responded to the faulting in a ductile fashion and show a relatively minor zone of clay in shearing along the fault zone, whereas the faulting of the relatively tough and brittle gabbro provided the structural displacement and brecciation for emplacement of the Paulsens quartz vein and gold mineralisation.

The ore body and surrounding sediments show a regional axial plane cleavage, striking northwest and dipping steeply to the southwest, related to the Wyloo folding, which forms a gentle anticlinal fold in the sediments and the quartz body. The later Billeroo dyke intrusions are mainly oriented along the cleavage direction.

Deposit Geology

The Paulsens mineralisation is hosted by a massive quartz vein, which occupies the Melrose Fault, a bedding slip fault zone. The vein and sediments are exposed within the core of the Wyloo anticlinorium. The quartz vein outcrops and forms a prominent 30m high hill at the Paulsens prospect.

The argillites dip to the northwest at 30-40°. The Melrose Fault is parallel to bedding, dipping at a similar angle to the northwest. The immediate footwall of the fault can be marked by a graphitic shale unit, the result of local shearing.

The argillite is cut by a northwest-trending 60 m wide gabbro body, which itself is cut and displaced by the Melrose Fault, showing an apparent 250m dextral displacement. The massive quartz vein occurs principally within the portion of the fault adjacent to the gabbro body. Whereas the argillites have deformed in a ductile fashion, it is assumed that the more rigid gabbro has undergone brittle failure, leading to a zone of structural disruption, brecciation and dilation which has facilitated the penetration of silica-rich fluids resulting in the emplacement of the quartz-carbonate vein. The bulk of the vein is barren, with the mineralisation focused along the hanging wall (Upper zone) and footwall (Lower zone) contacts. It is assumed that the late stage movement resulted in further brecciation along the contact zone of the quartz vein with the argillites and provided channel ways for the introduction of late-phase quartz-carbonate-sulphide-gold rich fluids. While the principal ore zones occur along the upper and lower contacts of the quartz body, occasional less continuous mineralisation is intersected elsewhere within the quartz vein or along splay faults intersecting the upper argillite.

The ore zones have a northeast-southwest strike length of 100-200m and plunge to the northwest at an average dip of around 30-35°, sub-parallel to bedding. Drilling has defined a down-plunge extent in excess of 1000m and to a depth of approximately 500m. Step-out drilling has shown that the mineralisation is still open down plunge, though with variable grade, with intersections to around 500m depth.

Post emplacement of the quartz vein, numerous steeply-dipping dolerite dykes have intruded, cutting the argillite, the quartz vein and mineralisation along the trend of the Wyloo cleavage. The dykes are from 1-10m in thickness and are estimated to cut out perhaps 10% of the mineralisation.

10. DEPOSIT TYPES

Paulsens is a mesothermal, orogenic lodestyle gold deposit. Mineralisation is hosted by vein quartz filling of voids, created in response to compression and subsequent uplift of the Wyloo Dome. Future down dip exploration will focus on lode repetitions offset by thrust faults.

11. MINERALISATION

Paulsen's is a structurally controlled deposit. Gold mineralisation is hosted within a folded, bedding parallel quartz vein up to 40m thick within an offset of gabbro. It is coincident with the Melrose Fault (bedding slip fault in response to folding) and with the Wyloo anticlinal fold hinge. The location of the quartz vein is interpreted to be influenced by the rigid gabbro dyke located within a fold hinge.

The Paulsens mineralisation comprises an Upper and Lower Zone, located along the hanging wall and footwall contacts respectively of a massive quartz carbonate vein (Figure 4). Gold mineralisation has also been intersected within the hanging wall sediments and within the body of the quartz vein, but these intersections tend to be isolated with little demonstrated continuity. Voyager, faulted off the main quartz body by the Apollo fault structure, has the same Upper and Lower Zone positions

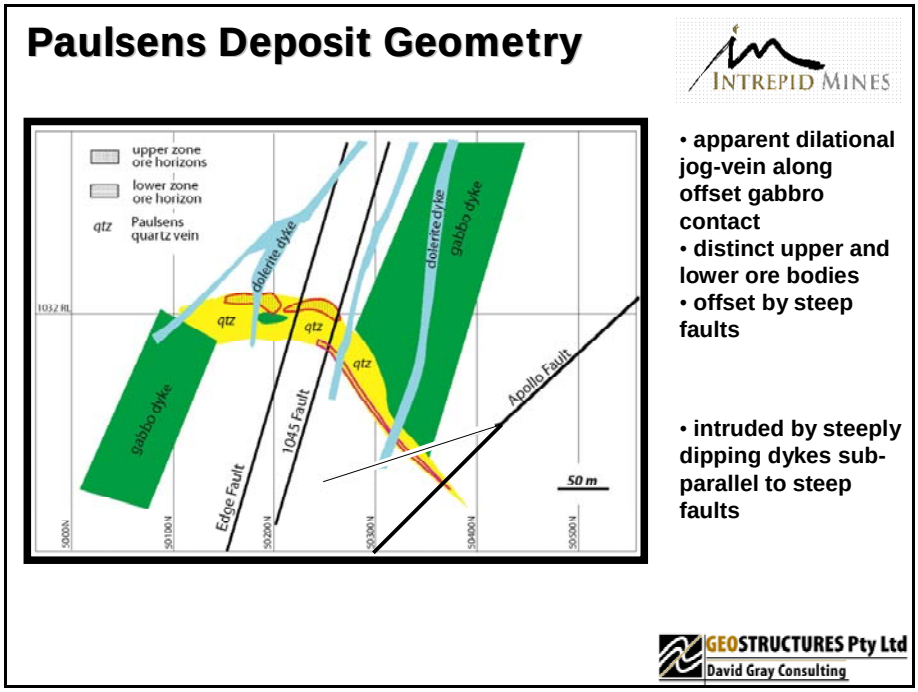


Figure 3 Cross-Sectional View of Paulsens Mineralisation (modified slightly from Gray D. 2007), Updip of the Voyager split.

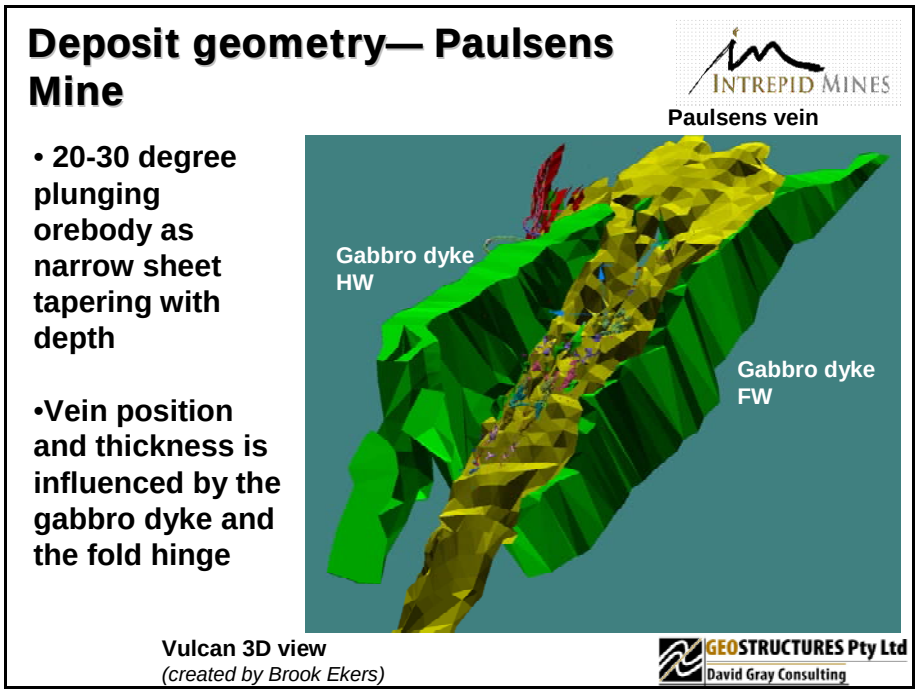


Figure 4 Three dimensional over view. Deposit geometry looking mine grid east

The Upper and Lower Zones are relatively continuous, with plan dimensions of approximately 100-200m along strike (northeast-southwest) and up to 1000m down plunge (northwest) to a depth of 500m. Upper Zone ranges in thickness from 1-10m but typically averages around 5m. It displays rapid changes in thickness giving a complex geometry in places. The hanging wall contact against the Melrose Argillite is relatively sharp, with a more gradational footwall cut off within the quartz vein. A number of drill hole intersections show the Upper Zone within the quartz vein rather than on the contact, with a barren quartz hanging wall.

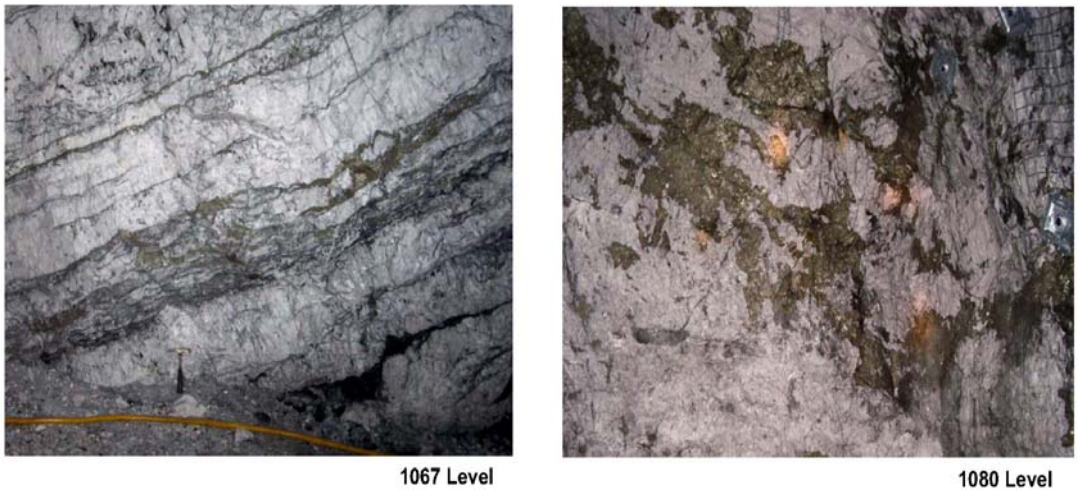
Lower Zone is somewhat thinner, typically 1-3 m averaging around 2m, but in general has a higher average grade, although typically containing a lower concentration of sulphide mineralisation. Lower Zone shows good continuity and generally more predictable geometry than the Upper Zone; the hanging wall is barren quartz but the footwall of the Lower lode is sharply defined by a graphitic shale, interpreted as representing the Melrose fault plane.

Gold is associated with quartz, carbonate (ankerite and dolomite) and pyrite, but principally with pyrite. Very minor arsenopyrite is also present. High grade gold intervals are commonly associated with massive sulphide zones, but significant grades also occur where the pyrite is more disseminated. Gold is distributed along crystal planes and micro-fractures within the pyrite grains and is readily recoverable by conventional cyanidation. Native gold has been identified but is reported to represent only a minor percentage of the total contained gold.

Three main host settings for the gold mineralisation have been recognized in the Paulsens deposit. These are sulphides, quartz/carbonates and country rock. The gold is associated predominantly with the sulphides (Figure 6). The sulphides can be present as massive material or laminated bands in quartz or as disseminated material in the quartz carbonates. There is an imperfect correlation between the gold grade and the content of sulphide as samples with relatively low sulphide content can yield a high gold grade.

Copper sulphide has been detected in the mineralisation ranging from chalcopyrite which is relatively cyanide-insoluble to chalcocite, which is readily soluble in cyanide solution.

The host rock is an argillite and carbonaceous argillites are present particularly along the footwall contact of the Lower zone. This carbonaceous argillite is a source of the high cyanide consumption properties of parts of the ore in treatment at Paulsen's.



(a) (b)

Figure 5 Typical Ore Mineralisation at Paulsens
(a) Laminated Stringers predominantly seen in Lower Zone and (b) Poddy Sulphide Distribution in Upper Zone

12. EXPLORATION

IAU holds the right to a controlling interest in 670km^2 of prospective territory in the vicinity of the Paulsens Mine and is exploring a number of nearby prospects to identify satellite deposits which could be mined and transported to Paulsens for processing. The prospects include the satellite deposits of Paulsens East, Paulsens West, Belvedere, Three Corner Bore, Paddys Well, and Billeroo Bore. IAU also holds a free carried interest in a further 76km^2 of tenements at Mt Clement, to the south of Paulsens Minesite.

Rock chip sampling of outcrop and soil geochemistry programs are used to provide a quick assessment of prospective areas and have generated several potential targets. Follow up work includes infill surface geochemistry and drilling. Large areas of the company's tenement holdings remain untested to any significant degree.

Detailed 1:1000 geological mapping within a 4km radius of the mine has aided in underground geological interpretation and indicated prospective drill targets based on stratigraphic positions.

IAU has joint venture agreements with several other companies including Pelican Resources NL (Wyloo Joint Venture) and Cullen Resources Limited (Hardey Junction Joint Venture) whereby IAU is conducting exploration in order to earn a majority interest in the joint venture properties. IAU also holds a twenty percent free carried interest to a decision to mine in the Mt Clement tenements.

Drilling of prospective surface targets is planned for commencement mid 2009

13. DRILLING

Diamond drilling is undertaken from underground drill platforms along with sludge sampling for grade control using a long hole top hammer percussion drill rig.

Barminco diamond drilling contractors provide all diamond drill services, with 2 LM drill rigs on site. For the year over 17,700 diamond drill meters were completed.

Sludge samples are drilled by Barminco Mining contractors and 3,700m was completed for the year.

Specific drilling details are in Section 19.

14. SAMPLING METHOD AND APPROACH

Sample numbers and details are covered Section 19 of this report in the H&S Resource report. A discussion on sample limitations and sample composites are also included within the same report. .

Core and face sample lengths are selected on the basis or rocktype (mineralised quartz) while aiming to keep sample length between 30 and 100cm. Where visible gold is present it will be sampled selectively. Barren rock types (dolerite, host sediments away from the veins and barren quartz) are often not sampled and coded GNS (geologically not sampled)

Sludge samples are taken at set lengths of 0.9 or 1.8m, being 1 or 2 samples per rod. Drill spoil from designated holes, drilled by long hole rig, is caught in a large funnel and transferred to sample bags.

15. SAMPLING PREPARATION, ANALYSES AND SECURITY

Sample Analyses – Core samples. All core is logged and whole core samples (if LTK48 size, NQ2 sized core is cut and half cored) are marked and prepared for shipping at the Paulsens Mine Property and sent to an independent Laboratory (ALS) for assay. The remaining half core is stored on site.

ALS Chemex – Australian Laboratory Services Pty. Limited in Karratha, Western Australia. Samples are weighed and crushed to 70% passing -6mm mesh. The crushed material is split and a portion is pulverised. A 100g pulp is sent to ALS Perth, Western Australia for assay. A 30 gram portion of the pulp is treated by Fire Assay method with atomic absorption finish (Au-AA25). A second pulp sample split (150-200g) is kept in Karratha. Sample rejects are discarded after 90 days. Over limit samples (>100ppm Au) are re-analysed using ALS' dilution method (Au-DIL). Intrepid inserts one standard in each hole, and one blank, though this practice has only recently been adopted. Lab standards and blanks are inserted by ALS and several pulp duplicates are also assayed as a determinant of mineralization variability.

ALS QAQC reports have been reviewed and found to be acceptable

ALS Chemex in Perth have AS/NZS ISO 9001:2000 certification. This does not cover the sample prep facilities in Karratha; however these prep lab's follow the same quality management system. They are not audited by NCSI but are audited internally.

Sample Analyses – Face and sludge samples. Both face and sludge samples are processed on site, in a lab originally set up by SGS but now run by Intrepid. Sludge samples are dried, split, chipped (for logging) and pulverised by geology field staff. Site lab personnel perform the sample prep of face samples. Both are analysed on site by the following procedure:

Sample Delivered By Geology and placed into drying oven

Sample dried overnight or minimum 2 hours

Dry sample crushed to minus 5mm (though gap may become more than that through wear)

Sample split to reduce to 500g

500g sample in pulverised (LM 1) - 5 - 10 mins

Sample is then bagged

50 g weighed from bagged sample

50g added to bottle roll bottle

1 Tablet leachwell added

100 ml fresh water added

Sample is bottle rolled for 1 hour

Sample is removed from bottle roll and allowed to settle

10 ml liquor removed from roll bottle dispensed to test tube

1 ml Cyanide Matrix added to test tube

5 ml of 2% DIBK added to test tube

Tube Agitated to mix

Analysed by AAS

Result is direct read.

Every 20th sample is sent to ALS external audit by fire assay. The assays compare well despite using different analytical techniques. Site average of the data is 11.4g/t compared to 11.0g/t for ALS, over 258 samples for 2008

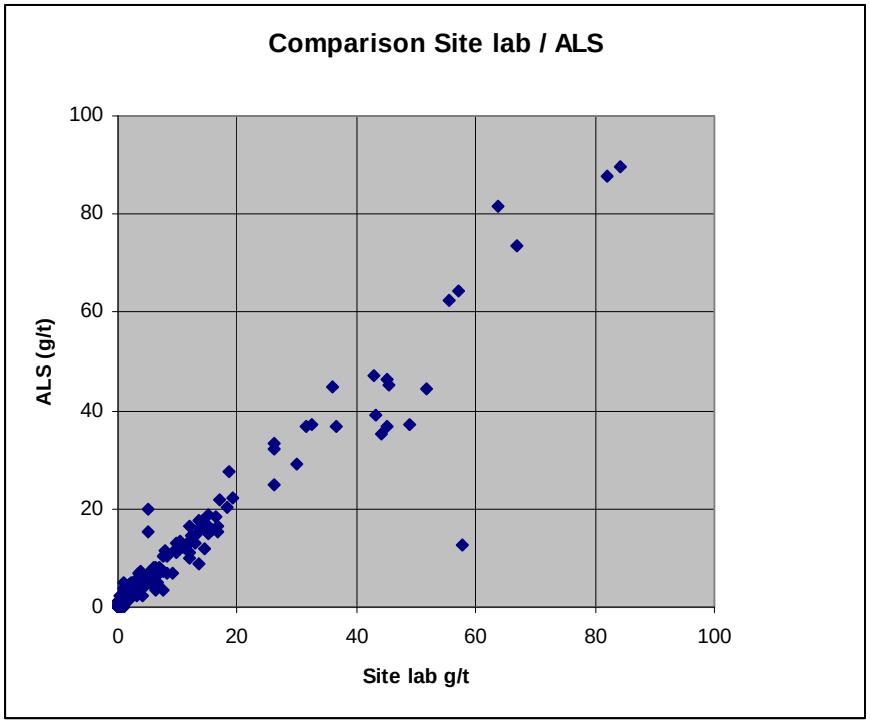


Figure 6 Plot of site laboratory geology sample, data against external lab (ALS) checks for 2008 (0-100g/t). Site lab uses leachwell, ALS uses fire assay.

The adequacy of sampling, sample preparation, security and the analytical procedures employed throughout the process is verified and deemed appropriate by site personnel.

16. DATA VERIFICATION

The database of information used on site is constantly being used and verified. Assays are directly imported into site database minimising data entry mistakes. The QP (Mr. Brook Ekers) for this section has verified the data as much as practicable.

Data was also reviewed by H&S and some problems were identified and fixed prior to the resource estimation, while the remaining inconsistencies are considered unlikely to have a significant impact on resource estimates.

Duplicate samples are routinely taken while face sampling. One sample will be selected, generally aiming to duplicate the best looking intersection. Resulting scatter plot shows very high variation (pseudo nugget effect in that the “nuggets” are blobs of pyrite) in the more visible ore. Average of all face samples is 9g/t, average of duplicated samples is 17g/t. This highlights the variability in face sampling and hence local variability in resource models and production models in particular.

A program of duplicating core samples has not yet been undertaken.

Further data verification is discussed in section 19.1 .

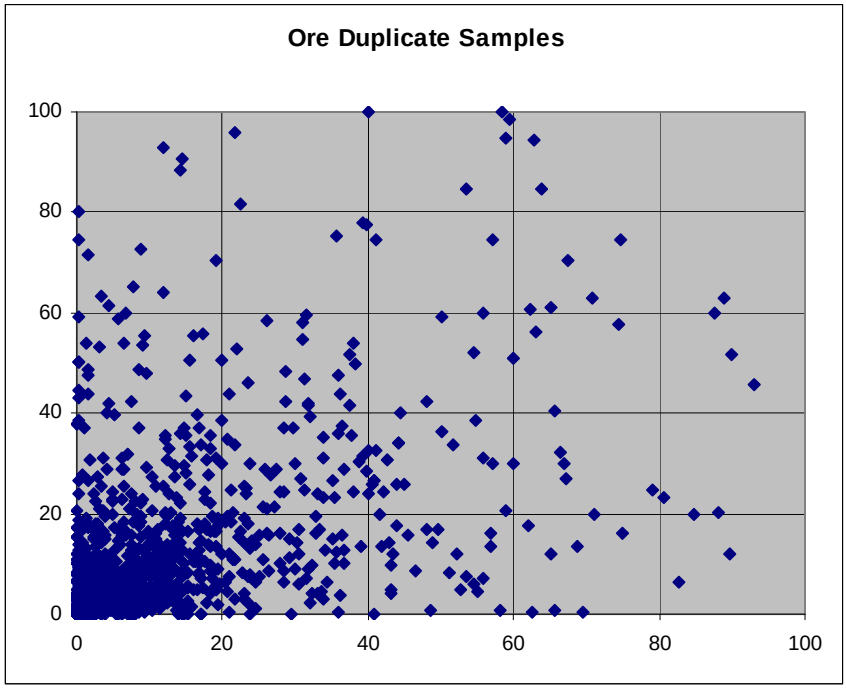


Figure 7 Scatter plot of duplicate face samples

17. ADJACENT PROPERTIES

This section is deemed not applicable as no adjacent properties exist in the area relative to the Paulsens Gold Mine.

18. MINERAL PROCESSING

General

The gold processing facility at Paulsens consists of the following unit operations:

Crushing and screening

Ore storage and reclaim

Grinding and classification

Leach feed thickening

Leaching and adsorption (Carbon-In-Leach)

Elution and gold recovery

Tailings disposal

Water and air services

Crushing

The crushing plant is a three-stage facility that treats up to 150 t/h of minus 550 mm Run-Of-Mine (ROM) ore to produce a crushed product P80 of 8 mm. The crushing plant operates 365 days a year 12 hours a day. Ore, on the ROM pad is taken, by front-end loader and tipped into the crusher feed bin. The ore is withdrawn at a controlled rate by apron feeder that discharges directly into the primary jaw crusher. A jaw crusher crushes the ore to a nominal P_{80} of 80 mm. There is a provision a rock breaker to handle the odd large rock in the primary crusher feed arrangement. (Large oversize is picked out by the loader and placed on an oversize pile). Jaw crusher product discharges onto the crusher discharge conveyor (CV01) which is fitted with a belt magnet at the head end to remove any tramp metal in the ore stream and discharges onto CV 02 which transports the ore to the inclined triple deck. A static magnet is located midway on conveyor CV02. The two top deck screen cloths are constructed of rubber with 40 mm and 20 mm apertures. The third deck screen cloth is made from woven wire mesh with an 11.5 mm aperture. The two top deck oversize product reports to the secondary crusher feed conveyor CV03, and is conveyed to the secondary crusher. . The lower third deck oversize product reports to the tertiary crusher feed conveyor CV04, and is conveyed to the tertiary crusher. The minus 11.5 mm product discharges onto CV 06 which discharges onto CV 07. CV 07 has a Weightometer installed mid length, CV 07 discharges into the CV 11 / CV 10 feed Box. CV 11 transports the ore to the Fine ore bin and when stopped the feed box overflows onto the CV 10 (Stacker) belt to the fine ore stock pile.

Ore Storage and Reclaim

Fine ore product from the crushing plant discharges into the fine ore bin which provides 12 hours surge capacity to the milling circuit. The ore is reclaimed from the fine ore bin via a slot feeder and variable speed belt feeder conveyor CV06. CV06 speed is controlled via the Weightometer located on the mill feed conveyor CV06. Dust generated in the fine ore bin is controlled by the use of appropriate skirting an insertable bag type dust collector located on top of the fine ore bin.

Crushed ore, stockpiled can be fed onto the mill feed conveyor CV11 via the emergency feed hopper using a FEL.

Quicklime is used to control the pH in the leach circuit and is stored in a lime silo adjacent to CV06. It is added to the process via a variable speed screw feeder that removes the lime from the silo and places it onto the tail end of the mill feed conveyor CV06.

Grinding and Classification

The grinding and classification facility operates 24 hours a day, 365 days a year to mill 335,000 t/a at a rate of 40 t/h. Ore discharges from CV06 and into the single stage mill via the mill feed chute. The rubber lined overflow ball mill (ML01) is 3.51 m in diameter by 5.63 m long (inside shell), equipped with a 1,000 kW motor. The mill operates in closed circuit with a single cluster of four, 250 mm diameter cyclones CY01 (three operating, one standby). Slurry discharges from the mill via the mill trommel. Trommel oversize is washed and directed in the scats bunker. The scats bunker is periodically cleared via front end loader and fed back through the mill via the emergency (Reclaim) feeder.

Trommel undersize flows directly into the mill discharge hopper. From the hopper it is pumped, using one of the two cyclone feed pumps, through a Tech-Taylor valve and pipeline to the cyclone cluster distributor. The cyclone feed line is fitted with a nucleonic density gauge.

Cyclone underflow gravitates back to the ball mill feed chute for further grinding. Cyclone overflow, with a P_{80} of 75um, gravitates to the trash screen via the trash screen feed box. The trash screen is fitted with 650 micron apertures polythene panel deck. Trash caught on the screen is washed and discharges by chute and pipe arrangement to a tails hopper. Trash screen undersize reports to the leach feed thickener feed box.

A ball charging hoist and kibble is used for charging balls into the mill via a purpose built impingement box to the mill feed chute.

Leach Feed Thickening

Trash screen underflow gravitates to the 7.0 m diameter 'High Rate' thickener fitted with an auto-dilution feed well. Diluted flocculent is added to the feed well through sparges. Thickener overflow flows by gravity back to the process water tank. Thickener underflow, at 53% solids w/w, gravitates into the leach feed pump hopper and is pumped to CIL tank No. 1 via the leach feed line

Leaching and Adsorption

The CIL circuit consists of 7 x 150 m^3 tanks operating in series to give a total residence time of 19.3 hrs. All seven tanks have dual open impellor agitators, tank No 1 is dedicated pre-oxidation with a MMS oxygen system and recirculation pump and all other six tanks are fitted with an Intertank screen for carbon retention. Cyanide is not added until tank No 2. A CIL circuit was chosen to minimise the pre-robbing characteristics of the ore.

The oxygen demand of the circuit is high at 0.2-0.5 $mgt^{-}'min^{-1}$ for some ore, and a cryogenic vessel was installed to inject oxygen into the CIL Tank No 1 recirculation pump and down the agitator shaft in tank No's 2 and 3 with a sparge retro fitted to tank No 4.

A valve manifold on the leach feed line enables CIL tank No 1 to be bypassed for maintenance purposes. Discharge from each tank flows by gravity through a launder into the next CIL tank. Each launder is provided with the facility to by-pass individual tanks using isolating gates.

The inter stage screens are vertically mounted, cylindrical, mechanically wiped, top discharge screens. These screens are of mild steel construction with 800 micron aperture stainless steel woven wire mesh. A mobile crane is used to remove the screens for maintenance.

The carbon concentration in the tanks averages 10 - 25 g/I to give a total carbon inventory of 18 -20 tonnes. Carbon is advanced throughout the day via airlifts. Carbon is recovered from the first Carbon contact tank via a recessed impeller pump which discharges over the loaded carbon screen. The screen is fitted with an 800 micron aperture woven wire screen deck. Undersize from the loaded carbon screen gravitates back to the CIL circuit. Loaded carbon, caught on the screen deck gravitates directly into the 2.0 tonne capacity cyanide wash column located directly below the loaded carbon screen.

Cyanide solution can be dosed to several CIL tanks down the CIL train via a manual dosing valve arrangement off the cyanide ring main although predominately Tank 2 and Tank 4 as required. PH probes in the leach feed hopper and CIL tank are used to monitor the pH.

Tailings Disposal

Slurry from the last CIL tank gravity flows via the screen feed box to the carbon safety screen. The carbon safety screen is fitted with 800 micron aperture woven wire mesh.

Carbon safety screen oversize reports, via a chute, to a fine carbon waste bin. Screen undersize reports to the tailings pump hopper at 50 % solids and is pumped to the tailings storage facility via the tailings disposal pumps.

The tailings are pumped through a polyethylene pipeline to the tailings storage facility (TSF). The line pressure is monitored via the control system. Monitoring is used to detect abnormal pressures, resulting from line obstructions or sanding and possible pipe failures. Flow meters are installed at the tailing pumps and TSF to detect any pipe leakage. A discrepancy between the flow meters initiates an alarm through the plant control system.

Carbon Stripping and Regeneration, Gold room

General

The elution and gold recovery circuit has the capacity to operate 7 days per week, treating 2.0 tonne batches of carbon. Carbon, loaded to 3,500 g/t (Min) gold is recovered from CIL tanks, screened and washed to remove the slurry and directed into the cyanide wash column.

The elution circuit comprises of two columns. In the first column a cold, cyanide solution wash is carried out to remove copper adsorbed onto the carbon followed by a hot acid wash to remove nickel and other deleterious elements. The carbon is then pneumatically transferred to the second column where elutriation takes place. The first column is constructed of mild steel and is butyl rubber lined. The second column is constructed from 304 stainless steel.

The total strip cycle will take 13 hours, Wash cycles 7.5 hours using 36 m3 of raw water whilst the elution cycle comprises uses 18 m3 of fresh water and takes 5.5 hours.

Cold Cyanide Wash

Concentrated Cyanide Solution NaCN (25% w/v) and Caustic Soda Solution NaOH (50% w/v) is added via the dosing pump and diluted in-line with raw water to a concentration of 3.0 % w/v.

The solution is pumped into the column for 15 minutes at 8 m^3/h. After which the pump is stopped and the solution is left to soak the carbon for 30 minutes prior to rinsing the carbon with water for 2 hours. The rinse solution discharging from the top of the column gravitates into the tailings disposal hopper. After rinsing the column is drained.

Acid Wash

After draining the wash column a 3 % hydrochloric acid solution, at 0-80°C, is pumped into the column over a period of 20 minutes. Concentrated hydrochloric acid is added via the acid metering pump at the base of the column and diluted in-line with raw water to a concentration of 3.0 % w/v.

After pumping the acid into the column it is left to soak the carbon for 30 minutes to effect removal of nickel, calcium and other acid soluble salts from the carbon surface. After soaking, raw water is pumped through the column for a 2 hour period to remove the acid and dissolved salts. The overflow from the column discharges to the tails hopper and is disposed in the tails dam. The column is then drained and the carbon is transferred to the Elution Column

Elution

A split AARL elution process is used to remove the gold from the carbon surface. Initially, the carbon is preheated to 80 deg C then soaked in a pre-treatment solution of 3% w/v NaCN

and 3% w/v NaOH. (Concentrated Cyanide Solution NaCN (25% w/v) and Caustic Soda Solution NaOH (50% w/v) is added via the dosing pump and diluted in-line with fresh water to a concentration of 3.0 % w/v. for 20 minutes at 8 m^3/h.)

Prior to entering the column the solution is heated to a temperature of 100-120°C. Once the addition is complete the cycle is stopped and the solution is left to soak the carbon for 40 minutes.

Following on the solution from the elution recirculation tank is pumped into the column via the heater and heat exchangers. Column overflow is directed to the electrolyte tank. This cycle continues for 100 minutes running at a temperature of 110 - 120°C. After the recirculation stage, fresh water from the fresh water tank is pumped by the fresh water pump into the base of the elution column. The column overflow is then directed into the empty elution recirculation tank. This cycle continues for 100 minutes running at a temperature of 110 - 120°C. After this cycle time the heater is switches off and the solution cools the column down. The elution is designed to operate at a temperature of 120°C and a column operating pressure of up to 450 kPa.

Heating of the solution is affected using a 1,000 kW gas fired indirect heat boiler. Mobiltherm 603 heating fluid is pumped from the heater into the primary heat exchanger to exchange heat with the solution that is pumped into the base of the column. Prior to passing through the primary heat exchanger this solution passes through the recovery heat exchanger to exchange heat with the solution discharge from the top of the column.

Electrowinning

Pregnant solution is discharged from the elution column, via the eluate filters and recovery heat exchanger to the electrolyte tank. The solution is pumped from this tank via the electrolyte pump to two 600 mm by 600 mm, electrowinning cells operating in parallel. Cell discharge gravitates back to the electrolyte tank. Electrical current is supplied to the electrowinning cells by two dedicated rectifiers (6V, 1,000 A).

Carbon Regeneration

Barren carbon is educted from the elution column to the regeneration kiln dewatering screen and the dewater carbon discharges into the kiln feed hopper. The carbon is fed into the regeneration kiln where it spends 15 minutes at a temperature of 700°C. Carbon emerging form the kiln discharges into a quench hopper to cool the carbon. The carbon can then be educted back to the CIL circuit via the barren carbon dewatering screen.

The regeneration kiln is a horizontal rotary type unit, LPG fired and capable of a nominal 200 kg/h throughput. Addition of new carbon to the circuit is via the site based mobile crane into CIL tank No 7.

Gold Recovery

The mild steel wool cathodes containing the electro won doré are removed from the electro winning cells placed on trays and calcined at 700°C. The sludge in the cells is removed via the drains in the bottom of the cells and collected in buckets. It is then decanted to remove solution prior to calcining. The calcine is mixed with fluxes and smelted in the gas fired tilting bullion furnace with A100 crucible capacity. The bullion is stored in the gold room safe.

Water Services

General

The Paulsens ore processing plant water supply system will consist of the following water types and distribution circuits:

Process water

Raw water

Potable water

Underground dewatering

Process Water

Process water requirements are met by thickener overflow and raw water flows. Tailings decant water is negligible and is not used in the process due to the detrimental effects of metals, acids and cyanide contained in the water. Top up water to the process water tank is supplied via the raw water pump. Process water is distributed by the process water pump to the plant, predominantly in the milling circuit.

Raw Water

Raw water is provided from mine dewatering and local bore fields. Clear water overflows from the mine dewatering settling pond into the raw water pond. A suction manifold from the raw water pond supplies a raw water distribution pump and fire water pump. The raw water distribution supplies the Flocculant make-up, general elution and barren carbon transfer water requirements.

Potable Water

Fresh water is supplied from the water bore field via the two dedicated reverse osmosis plants. The incoming raw water is treated to remove impurities prior to discharge into the potable water tanks. The RO plants have a capacity of 60 tonnes per day. Potable water is withdrawn from the potable water tank using a potable water pumps. These pumps operate on a pressure demand basis, supplying the plant, stores and offices and camp. The elution circuit and the gold room fresh water is provided via fresh water pump drawing from a dedicated 150 m^3 fresh water tank.

Air Services

General

The Paulsens ore processing plant air supply system will consist of the following:

High pressure air

Low pressure air

Instrument air

High Pressure Air

A single compressor is used to provide plant high pressure air. This air is reticulated to all processing plant areas and the workshop for general use. Air is stored in a 1.0 m^3 capacity receiver prior to reticulation.

Low Pressure Air

A roots style low pressure blower provides air for running the air lifts (CIL tanks No's 1 to 7) was originally used but has been put into standby as the HP air compressor has sufficient capacity to supply all demands.

Instrument Air

Feed from the high pressure air circuit is filtered and dried in a refrigerated air drier to provide instrument air. A small air receiver provides storage capacity downstream of the air drier.

19. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

19.1. MINERAL RESOURCE ESTIMATE

The following description of resource estimation for the Paulsens deposit is derived from Abbott, 2009.

Resource Estimation for the Paulsens Gold Mine Western Australia

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Prepared for Intrepid Mining Ltd.
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Introduction and Summary

Hellman & Schofield Pty Ltd (H&S) was commissioned by Intrepid Mines Ltd (Intrepid) to estimate Mineral Resources for the Paulsens underground gold mine in the Pilbara region of Western Australia. The estimates include all of the mineralized zones currently interpreted at Paulsens, with the exception of some shallower mineralization that has been depleted by mining to date with no significance to estimation of remaining Mineral Resources.

The current Mineral Resources estimates are derived resource modelling conducted in December 2008 and January 2009. The December 2008 study included estimates for the remnant Paulsens lodes, with estimates for the Voyager zones updated in January 2009 after receipt of additional analytical results from diamond drilling in that area.

The resource estimates are based on sampling data, and mineralization interpretations provided by Intrepid. For the current study, the report author was not required to review the validity, or adequacy of the sampling data or the appropriateness of the mineralization interpretation, as Intrepid are taking responsibility for these aspects of the resource estimates.

Since 1998, the Paulsens deposit has been sampled by surface and underground drilling by Intrepid and previous project owners. Intrepid supplied H&S with sampling databases for Paulsens including collar location, orientations, geological logs, and analytical data for 399 reverse circulation (RC) drill holes, 145 generally RC pre-collared surface diamond holes and 562 underground diamond holes, for a total of 129,060 metres of drilling. The supplied sampling data also included 1,411 sludge holes drilled with an underground production drill rig, and 3,131 face samples. Surface trench samples and rotary air blast (RAB) holes provided by Intrepid were not compiled.

Although face and sludge sampling locally helps define lode positions, gold grades from these samples tend to be higher grade than shown by samples from diamond drilling. Further investigations are required to determine the reasons for this trend. In addition to differences in sampling technique, and the commonly incomplete coverage of the lodes by face and sludge sampling, the grade differences may partially reflect differing analytical methods used for each sampling phase.

Vulcan and Gemcom software were used for data compilation, wire-framing and composite calculation. GS3, the resource estimation software developed by H&S, was used for grade estimation. Sub-blocked Vulcan and Surpac format models were created for use by Intrepid, with resource estimates derived from the Vulcan format model.

Mineral Resources were estimated by Ordinary Kriging of nominally one meter downhole composited gold grades within the mineralized wireframes, incorporating upper cuts of 45 to 80 g/t selected for each lode. For each estimated block, the search ellipsoid was aligned with

the local mineralization orientation. The estimates assume bulk densities of 2.8 to 3.2 t/m^3 as specified by Intrepid

Estimates for portions of the mineralization tested by closely spaced RC and diamond drilling, and portions of the mineralization accessed by development with moderately spaced RC and diamond drilling are classified as Indicated. All other estimates are classified as Inferred.

The estimated resources cover a combined strike length of approximately 1.2 kilometers and extend to approximately 500 meters below surface.

Table 1 shows estimated Mineral Resources for the Paulsens deposit as of the 31st of December 2008. These estimates are reported outside a set of wireframes defining the limits of mined or inaccessible mineralization supplied by Intrepid and are reported above a block cut off grade of 4.0 g/t gold as stipulated by Intrepid. The figures in this table are rounded to reflect the accuracy of estimates and exhibit rounding errors.

Table 1: Paulsens Mineral Resource Estimates as of 31st December 2008

	Indicated			Inferred		
Zone	**Tonnes**	**Au g/t**	**Au Ounces**	**Tonnes**	**Au g/t**	**Au Ounces**
Upper	183,000	9.0	53,000	70,000	10	22,500
Lower	86,000	9.7	26,800	160,000	9	46,300
Total	**269,000**	**9.2**	**79,800**	**230,000**	**9**	**68,800**

H&S is a group of consulting geologists providing expert services to the hard-rock minerals industry in the fields of exploration, evaluation, resource estimation and optimisation of grade control. The group specialises in application of advanced geostatistical methods to resource estimation and grade control, and due diligence investigations.

The work reported herein was undertaken by Jonathon Abbott, MAusIMM, who is a full-time employee of H&S and a Member of the Australian Institute of Mining and Metallurgy. Mr Abbott has sufficient experience which is relevant to the style of mineralization and type of deposit under consideration to qualify as a Qualified Person in terms of NI43-101 standards for resource estimation. Mr Abbott visited the Paulsens site from the 11th to 13th of November 2008.

Mr Abbott accepts responsibility for classifying the current estimates as Indicated and Inferred, providing Intrepid nominate a Qualified Person, or Persons to accept responsibility for the data on which it is based, the mineralization interpretation and to attest to the reasonable prospect of eventual economic extraction of the mineral resources.

Supplied data

Paulsens sampling databases

Paulsens surface and underground sampling data are stored in separate databases. The surface drill hole database contains mostly older data, and is not routinely updated by Intrepid.

Intrepid currently maintain two databases for the underground drilling and sampling. One version is administered on site in Microsoft Access format, and a Datashed version is maintained in Perth.

Preliminary review of initially supplied versions of both underground sampling databases showed numerous errors and inconsistencies between the databases. These errors and inconsistencies which were summarized in several emails to Intrepid range from inconsistent descriptive field entries with limited impact on estimates, to drill holes with incorrect locations and grades.

As specified by Intrepid, for the December 2008 and January 2009 studies, underground sampling data were sourced from the site database which appears more complete and valid than the Datashed version. Although Intrepid corrected many of the errors noted in H&S's review of the initially supplied site database, the version supplied for resource estimation included several errors and inconsistencies.

Intrepid specified that the current study was not required to include review of the validity of sampling databases, or adequacy of sampling data such as collar and downhole surveying. Intrepid are accepting responsibility for these aspects of the resource estimate.

Data sources

Table 2 and Table 3 list the data files supplied by Intrepid which form the basis of the current resource estimates.

The resource estimates are based on sampling data supplied by Intrepid in two Microsoft Access databases representing the underground and surface sampling databases. The surface data (*PaulsensSurfaceDatabase.mdb*) was supplied on the 11th of August 2008, and the underground drill data (*NMC - DRILLING.mdb*) was initially supplied on the 5th of December 2008, with an updated file of the same name was supplied on the 19th of January 2009.

For the January 2009 Voyager resource update, the working database compiled for the December 2008 study was updated for only those holes included in the supplied Vulcan selection file that lists drill holes intersecting the interpreted Voyager mineralized domains.

In addition to numerous new assay results, the database revisions included updating of the collar and initial orientation for drill hole PDU591 with surveyed rather than assumed measurements.

Table 2: Data supplied for December 2008 study

	Description	File	Date Supplied
Vulcan Wireframes	Gemini	GEM_0606_01.00t	10th Dec. '08
	Soyuz	SOY_0811_01.00t	9th Dec. '08
	Cassini	CAS_0807_01.00t	9th Dec. '08
	Apollo	AP_0810_01.00t	9th Dec. '08
	Lower Zone production area	LZPRODX_0710_01.00t	9th Dec. '08
	Lower Zone West	LZW_0812_01.00t	9th Dec. '08
	Lower Zone	LZ_0807_02.00t	9th Dec. '08
	Upper Zone	UZ_0812_01.00t	9th Dec. '08
	Upper Zone Splay	UZSP_0811.00t	10th Dec. '08
	Voyager Upper Zone	VOY_0810_02.00t	9th Dec. '08
	Voyager Lower Zone	0812Voy_LZ_folded.00t	9th Dec. '08
	Gabbro	GABBRO_0809_02.00t	12th Dec. '08
	Voyager Dyke	VOY_dykes_0810_02.00t	11th Dec. '08
	Decline	all_dec081211.00t	12th Dec. '08
	Development	all_dev_deep_081211.00t	12th Dec. '08
	Development	all_dev_top_081211.00t	12th Dec. '08
	Misc development	all_misc_081211.00t	12th Dec. '08
	Stopes	all_stope_deep_081211.00t	12th Dec. '08
	Stopes	all_stope_top_081211.00t	12th Dec. '08
Sampling databases	Surface	PaulsensSurfaceDatabase.mdb	11th Aug. '08
	Underground	NMC - DRILLING.mdb	5th Dec '08

Table 3: Data supplied for January 2009 study

	Description	File	Date Supplied
Vulcan Wireframes	Voyager Upper Zone	VOY_0901_01.00t	19th Jan. '09
	Voyager Lower Zone	VOY_LZ_0901.00t	19th Jan. '09
	Voyager Dyke	VOY_DYKES_0901_01.00t	19th Jan. '09
Vulcan Selection	Selection file for Voyager drilling	p_0901_VOY_02.sel	19th Jan. '09
Vulcan Composite	One meter down hole composited gold grades within Voyager domains.	p_0901_voy_02su.map	19th Jan. '09
Sampling databases	Underground	NMC - DRILLING.mdb	19th Jan. '09

Available sampling data

The supplied sampling data was compiled into a working database as summarized in Table 4. The compiled database excludes trench samples and rotary air blast (RAB) holes in the supplied surface. The summary of surface diamond drilling presented in Table 4 includes the RC pre-collar component of pre-collared diamond holes. For the 136 diamond holes with pre-collar depths entries in the supplied database pre-collar depths range from 4 to 330 metres and average 198 metres.

Modifications to the supplied data were generally limited to adjusting overlapping assay and geological logging intervals, removing repeated downhole survey entries and replacing the variably entered below detection assay entries with a value of 0.005 g/t.

The supplied assay data includes gold grade entries of -555 g/t representing intervals considered by the logging geologist as barren. For the working database these intervals were assigned gold grades of 0.001 g/t. Database entries of -50 g/t represent potentially mineralized intervals without assay results, such as pending results or core loss and were assigned a null value and excluded from resource estimation.

Sampling data was supplied with Australian Map Grid (AMG) and local grid coordinates. For consistency with the supplied wireframes, and previous estimates the current model was developed in local grid coordinates.

Figure 1 shows a plan view of drill hole traces colored by sampling type, and **Figure 2** shows a long section of drill hole traces relative to the decline and elevation (1,115 mRL) specified by Intrepid as representing the base of partial oxidation..

Figure 3 shows a long sectional view of the mineralized composites compiled for resource estimation colored by sampling type.

Table 4: Summary of compiled sampling dataset

	Number of holes	Meters
Surface RC	399	51,647
Surface diamond (includes RC pre-collar)	145	40,055
Underground diamond	562	37,358
Subtotal RC and diamond	**1,106**	**129,060**
Sludge holes	1,411	15,263
Face samples	3,131	15,172
Subtotal face/sludge	**4,542**	**30,436**
Total	**5,648**	**159,496**

Mineralization wireframes

Intrepid supplied the mineralization wireframes used for the current study in Vulcan format as listed in Table 2. Wireframes supplied for Soyuz, Lower Zone production area and Upper Zone are inclusive of barren gabbro units, and were clipped to the gabbro wireframe as supplied by Intrepid. Similarly the wireframe representing the Voyager Upper Zone was clipped to the barren cross cutting dyke. All other wireframes were used without modification.

Lower Zone mineralization is subdivided into three separate domains, designated as Lower Zone, Lower Zone West and the Lower Zone production area, which represents the portion of the lode with substantial development.

Figure 6 presents a long section of the mineralized domains relative to the decline, and elevation (1,115 mRL) specified by Intrepid as representing the base of partial oxidation.

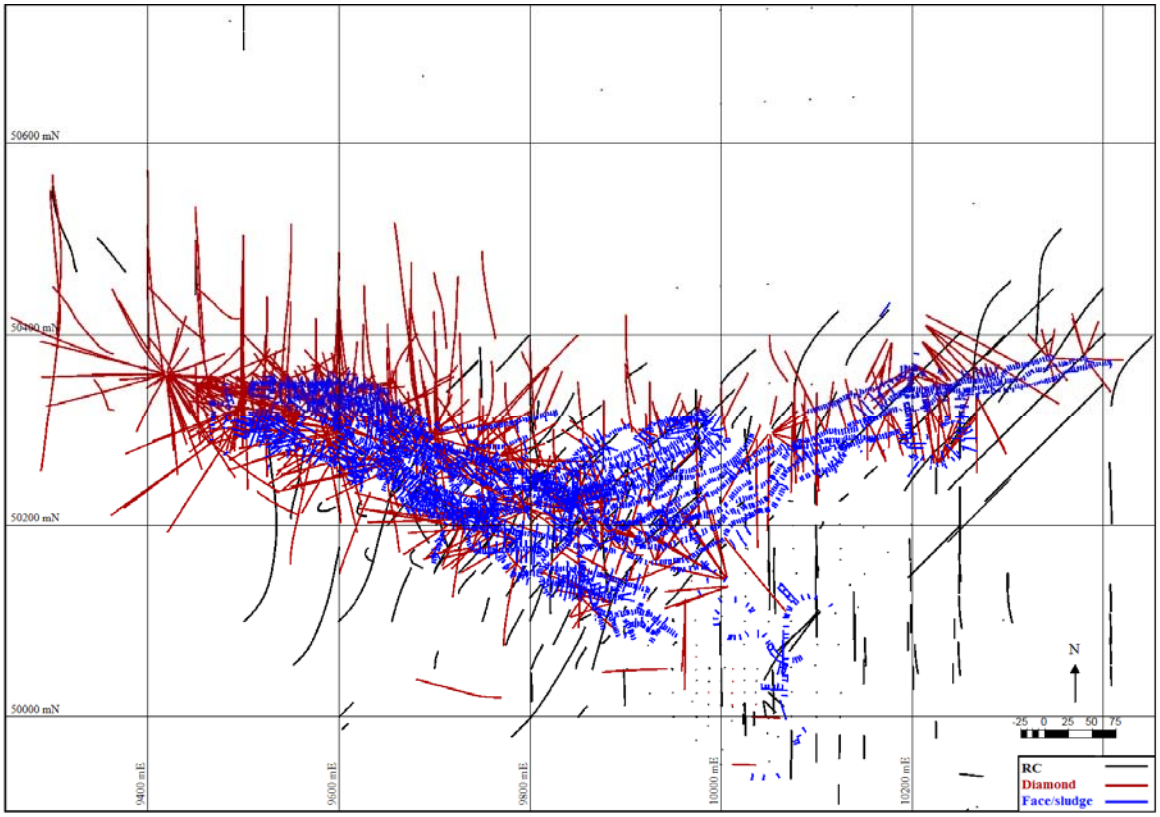


Figure 1: Plan view of drill hole and underground sampling traces

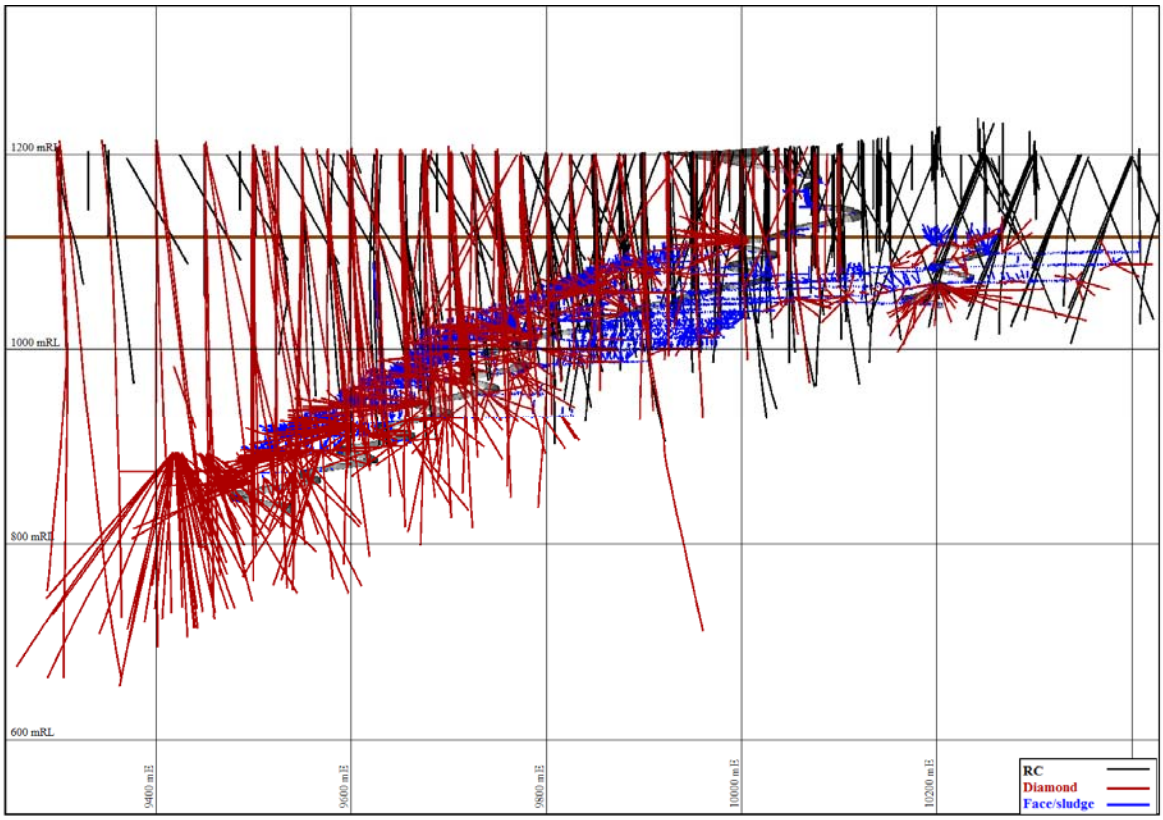


Figure 2: Long section of drill hole and underground sampling traces

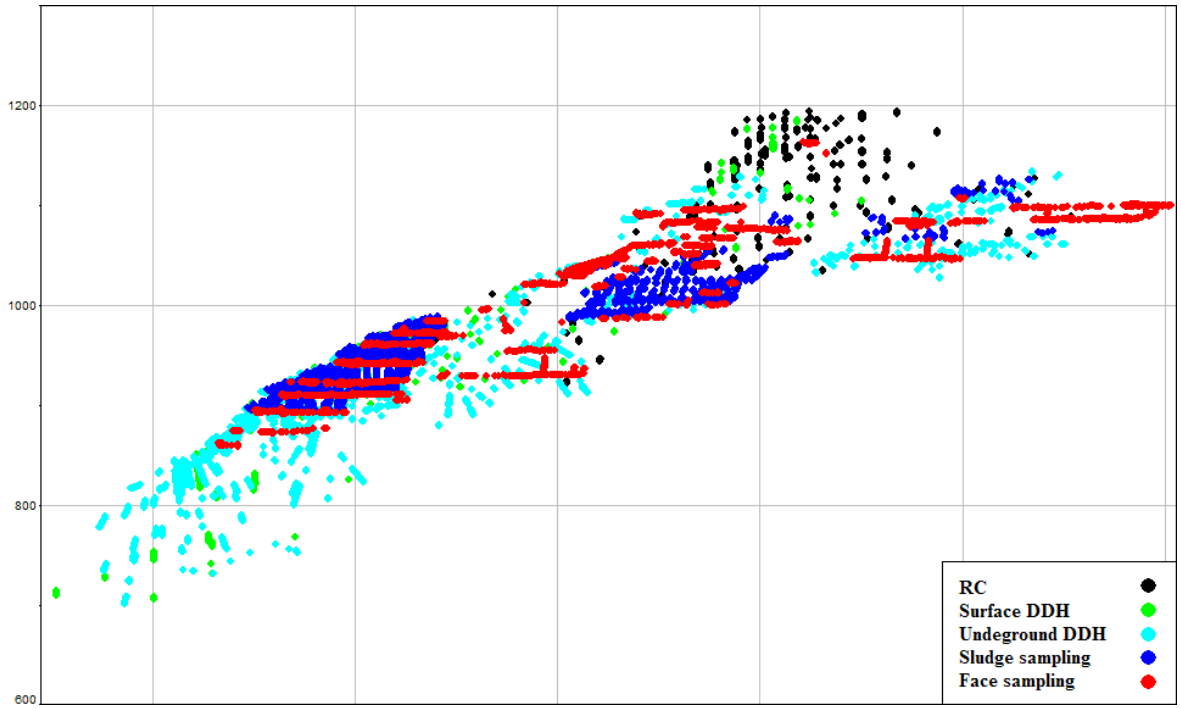


Figure 3: Long section of mineralized composites by sampling type

Bulk density

Bulk densities specified by Intrepid for use in the current estimate are listed in Table 5.

Table 5: Densities assigned to current estimate

Domain	Density (t/m^3)	
	Fresh	Oxidized (>1115 mRL)
Gemini	3.2	2.5
Soyuz	3.2	2.5
Cassini	2.8	2.5
Apollo	2.8	2.5
Lower Zone production area	2.8	
Lower Zone West	2.8	-
Lower Zone	2.8	-
Upper Zone	3.2	-
Upper Zone Splay	3.2	-
Voyager Upper Zone	2.8	-
Voyager Lower Zone	2.8	-

Comparison of sampling types

Summary

The gold grades shown by face and sludge samples were compared to diamond drilling results by comparing composite grades within an area of Upper Zone mineralization comprehensively sampled by these methods, and by a nearest neighbor comparison. Both comparisons show a tendency for the face and sludge samples show to higher average gold grades than the diamond drilling, with face samples tending to show the greatest grade difference.

The reasons for the grade difference between sampling types is not clear and require further investigation. In addition to differences in sampling technique, and the commonly incomplete coverage of the lodes by face and sludge sampling, the grade differences may partially reflect differing analytical methods used for each sampling phase.

Representative area comparisons

As shown in **Figure 3** a portion of the Upper Zone mineralization centered at approximately 9,600 mE, 930 mRL is comprehensively sampled by diamond drilling, sludge samples and face sampling. Table 6 and **Figure 4** summarize composite grades by drilling type from this

area. This table and figure demonstrate that within this representative area the face and sludge samples show higher average gold grades than the diamond drilling, with face samples tending to show the greatest grade difference.

Table 6: Upper Zone representative area composite statistics

	All composites				Subset 0.1 to 50 g/t			
	No.	Minimum Au g/t	Average Au g/t	Maximum Au g/t	No. Au g/t	Minimum Au g/t	Average Au g/t	Maximum Au g/t
DDH	1,504	0.001	5.73	259.9	1,004	0.10	4.72	49.7
Sludge samples	2,189	0.001	6.91	461.7	1,877	0.10	5.21	48.9
Face samples	1,115	0.001	6.98	293.6	910	0.10	6.07	49.6

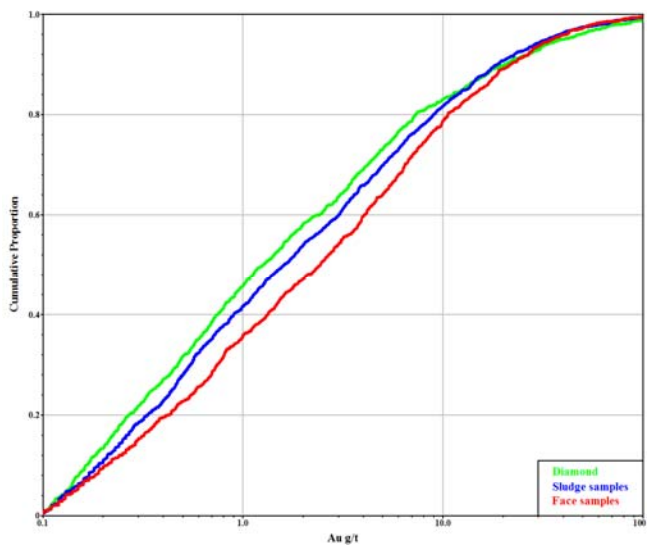


Figure 4: Cumulative distribution of Upper Zone representative area composites

Nearest neighbor comparisons

Table 7 and Table 8 compare composite grades from diamond drilling with the closest composite from sludge and face sampling respectively. These tables show summaries for pairs separated by less than 10 by 5 by 10 meters (East, North, Elevation) and 5 by 2 by 5 meters. Table 9 shows a similar comparison between nearby sludge and face samples.

Figure 5 shows quantile-quantile plots for diamond versus sludge and face samples separated by less than 10 by 5 by 10 meters restricted to pairs where both gold grades are between 0.1 and 50 g/t.

These nearest neighbor comparisons show a tendency for composites from sludge and face sampling to show higher grades than nearby diamond composites, with face samples tending to show a greater grade discrepancy.

Table 7: Sludge sampling versus diamond drilling nearest neighbor comparison

	<10,5,10 m separation All grades		**<10,5,10 m separation 0.1 to 50 g/t**		**5,2,5 m separation 0.1 to 50 g/t**	
	DDH Au g/t	**Sludge Au g/t**	**DDH Au g/t**	**Sludge Au g/t**	**DDH Au g/t**	**Sludge Au g/t**
Number	1,033		566		226	
Mean	**4.52**	**8.71**	**4.10**	**5.54**	**4.53**	**5.23**
Mean difference		**93%**		**35%**		**15%**
Variance	190.8	835.3	55.2	75.5	67.0	91.1
Coefficient of variation	3.06	3.32	1.81	1.57	1.81	1.83
Minimum	0.00	0.00	0.10	0.10	0.10	0.10
1st Quartile	0.05	0.27	0.27	0.47	0.25	0.37
Median	0.28	1.16	0.87	1.64	0.85	1.10
3rd Quartile	2.18	5.98	4.09	6.66	4.81	4.96
Maximum:	161	565	47.8	48.4	47.8	48.4

Table 8: Face sample versus diamond drilling nearest neighbor comparison

	<10,5,10 m separation	**<10,5,10 m separation**	**5,2,5 m separation**

	All grades		0.1 to 50 g/t		0.1 to 50 g/t	
	DDH Au g/t	**Face Au g/t**	**DDH Au g/t**	**Face Au g/t**	**DDH Au g/t**	**Face Au g/t**
Number	918		421		213	
Mean	**3.26**	**7.81**	**3.61**	**6.33**	**3.54**	**5.85**
Mean difference		**139%**		**75%**		**65%**
Variance	133.4	387.6	47.4	90.3	39.4	76.9
Coefficient of variation	3.54	2.52	1.91	1.50	1.78	1.50
Minimum	0.00	0.00	0.10	0.10	0.10	0.10
1st Quartile	0.03	0.22	0.22	0.59	0.21	0.47
Median	0.14	1.48	0.72	2.11	0.70	2.04
3rd Quartile	1.04	6.73	3.40	6.85	3.10	6.60
Maximum:	139.9	375.4	46.1	47.5	32.3	49.6

Table 9: Face sample versus sludge sampling nearest neighbor comparison

	<10,5,10 m separation All grades		<10,5,10 m separation 0.1 to 50 g/t		5,2,5 m separation 0.1 to 50 g/t	
	Sludge Au g/t	**Face Au g/t**	**Sludge Au g/t**	**Face Au g/t**	**Sludge Au g/t**	**Face Au g/t**
Number	1,159		715		339	
Mean	**4.40**	**7.82**	**2.92**	**6.53**	**4.30**	**5.60**
Mean difference		**78%**		**124%**		**30%**
Variance	223.0	393.4	31.1	87.0	56.3	75.4
Coefficient of variation	3.40	2.54	1.91	1.43	1.75	1.55
Minimum	0.00	0.00	0.10	0.10	0.10	0.10
1st Quartile	0.13	0.18	0.26	0.64	0.37	0.48
Median	0.49	1.38	0.68	2.61	0.93	1.72
3rd Quartile	2.39	6.75	2.90	8.27	4.52	6.67
Maximum:	223	305	38.4	50.0	47.4	50.0

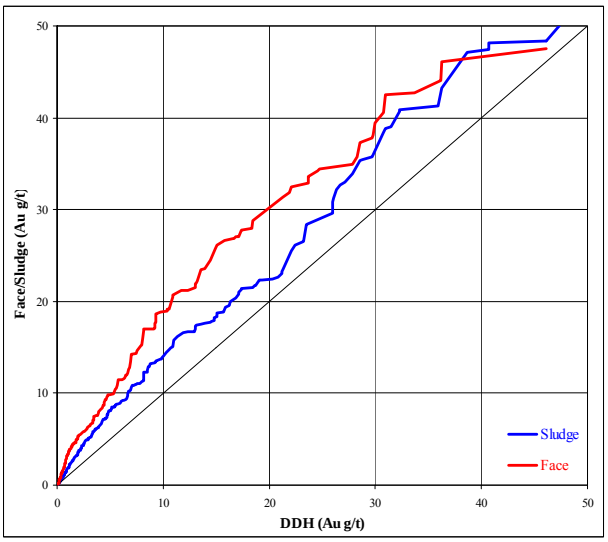


Figure 5: DDH versus sludge and face sample nearest neighbor QQ plots

Mineralization

Mineralization interpretation

Intrepid supplied the mineralization wireframes used for the current study in Vulcan format as listed in Table 2. Wireframes supplied for Soyuz, Lower Zone production area and Upper Zone are inclusive of barren gabbro units, and were clipped to the gabbro wireframe as supplied by Intrepid. Similarly the wireframe representing the Voyager Upper Zone was clipped to the barren cross cutting dyke. All other wireframes were used without modification.

Lower Zone mineralization is subdivided into three separate domains, designated as Lower Zone, Lower Zone West and the Lower Zone production area, which represents the portion of the lode with substantial development.

Intrepid's mineralization interpretation does not routinely include "snapping" outlines to sampling intervals. Due to the generally narrow and irregular lodes and high proportion of obliquely oriented sampling this leads to the wireframes incorrectly excluding high grade assay intervals and including lower grade material which was not intended to form part of the mineralized domain. This will tend to lower the average grade of sampling identified as lying within the mineralized domains and H&S generally recommends that for such deposits the mineralization wireframes be "snapped" to drill hole traces. The degree to which the lack of interval snapping affects the current estimates is unclear.

Figure 6 presents a long section of the mineralized domains relative to the decline, and elevation (1,115 mRL) specified by Intrepid as representing the base of partial oxidation.

Mineralization thicknesses

The true widths of drill hole intersections with the mineralized domains have not been compiled for the current study. Table 10 provides a summary of downhole intercept lengths for RC and diamond drilling. Although generated from down-hole rather than true thicknesses, this table demonstrates that for most lodes, the majority of mineralized drill intercepts are greater than the three meters in length, and are therefore likely to be greater than the minimum mining widths at Paulsens where the mining scenarios include hand-held stoping.

Table 10: Summary of drilling intersections

	Intersection lengths				Proportion of drilling		
	Number	**Min (m)**	**Avg (m)**	**Max (m)**	**>1m**	**>2m**	**>3m**
Gemini	42	1.03	3.59	8.69	100%	89%	75%
Soyuz	43	0.57	3.73	19.7	98%	92%	75%
Cassini	38	0.84	2.38	4.21	99%	75%	17%
Apollo	117	0.59	2.56	9.08	98%	70%	54%
Lower Zone Production	53	0.65	3.32	8.14	99%	88%	65%
Lower Zone West	121	0.51	2.34	10.3	95%	70%	38%
Lower Zone	49	0.89	4.51	22.9	99%	85%	73%
Upper Zone	217	0.56	9.30	29.7	100%	98%	96%
Upper Zone Splay	18	1.28	4.25	15.6	100%	94%	77%
Voyager Upper Zone	55	1.16	6.85	30.73	100%	93%	85%
Voyager Lower Zone	43	0.29	2.07	7.39	70%	44%	23%
Total	**796**	**0.29**	**4.95**	**30.73**	**99%**	**90%**	**79%**

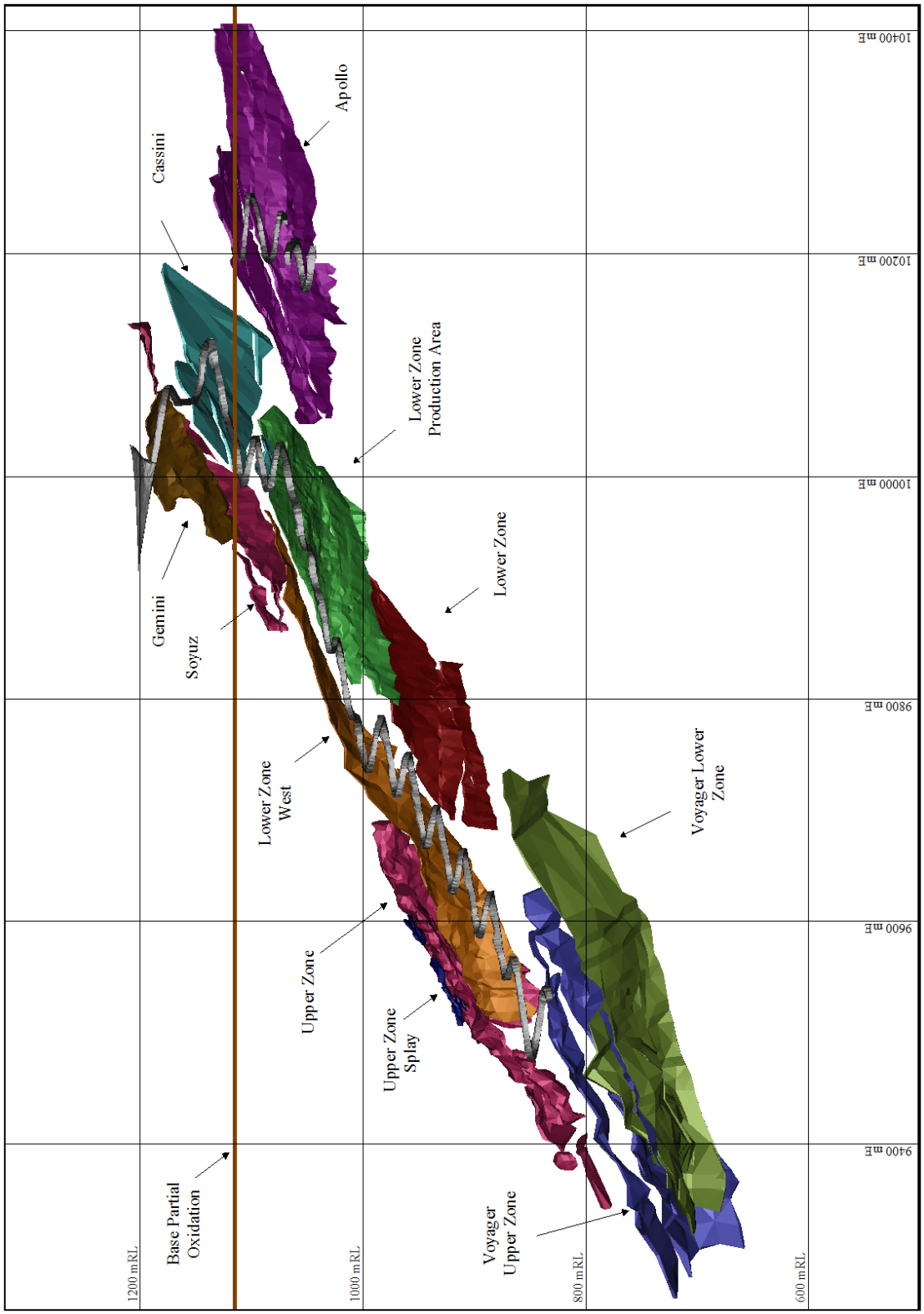


Figure 6: Long section of mineralized domains

Voyager mineralization domain interpretation

Since Voyager mineralization has not yet been accessed by development, or explicitly included in previous resource estimates, the following section provides additional details of the Voyager sampling and mineralized domain interpretation.

Figure 7 presents a long sectional view of the Voyager mineralization domains relative to the traces of surface and underground diamond drill holes intersecting the domains.

Figure 8 and **Figure 9** present inclined sections of drill hole intersection points with the Voyager Upper and Voyager Lower mineralized domains colored by gold grade. In these figures the mineralized domain outlines and cross section pierce points are projected onto a 40 degree north dipping plane approximating the average orientation of the domains.

The colored circles in **Figure 8** and **Figure 9** provide an indication of data spacing within the plane of the mineralization. The pink circles represent a radius of 10 meters from assayed drill hole intercepts. Where these circles coalesce, the data spacing is 20 by 20 meters or tighter. The blue circles represent a radius of 20 meters, and where these circles coalesce, the data spacing is 40 by 40 meters or tighter. The green circles represent a radius of 40 meters, and where these circles coalesce, the data spacing averages 80 by 80 meters or tighter.

Figure 8 shows that the Voyager Upper zone is sampled on an irregular pattern. A small area in the central upper area is sampled by approximately 20 by 20 meter spaced drilling, and most of the upper portion above approximately 750 mRL is sampled by 40 by 40 meter spaced drilling. Below approximately 700 mRL and in the western extents of the domain the sampling is broader than 40 by 40 meter spacing.

Figure 9 demonstrates that the upper and western portions of the Voyager Lower Zone above approximately 750 mRL and to the west of approximately 9,540 mE the domain is sampled at approximately 40 by 40 meter spacing. The rest of the domain is sampled at approximately 80 by 80 meter spacing.

Broadly sampled portions of the upper Voyager mineralization include a region around 9,370 mE where, both domains are interpreted to form an anitform, varying from gently north dipping to steeply south dipping. **Figure 10** shows the mineralized domain interpretation at this easting and drill hole traces colored by mineralized domain intersection and annotated by intersection length and gold grade. The apparent offset between drill traces and the mineralization interpretation reflects is due to the interpretation being presented at 9,370 mE and the obliquely oriented drill holes being shown within 40 meters of the section line.

Figure 10 shows that the northern limb of the antiform interpreted for the Voyager Upper zone around 9,370 mE is not intersected by any drill holes for approximately 100 meters

down dip, and more than 80 meters along strike. In this area there are no drill intersections with the lode between 750 mRL where it is intersected by drill hole PDU499 (3.3 m @ 0.09 g/t Au) and 670 mRL where it is intersected by drill hole PDU500 (3.4 m @ 0.005 g/t Au). Although confidently estimating the grade of this broadly sampled and variably folded portion of the lode is currently impossible, the available sampling in the area shows only low gold grades.

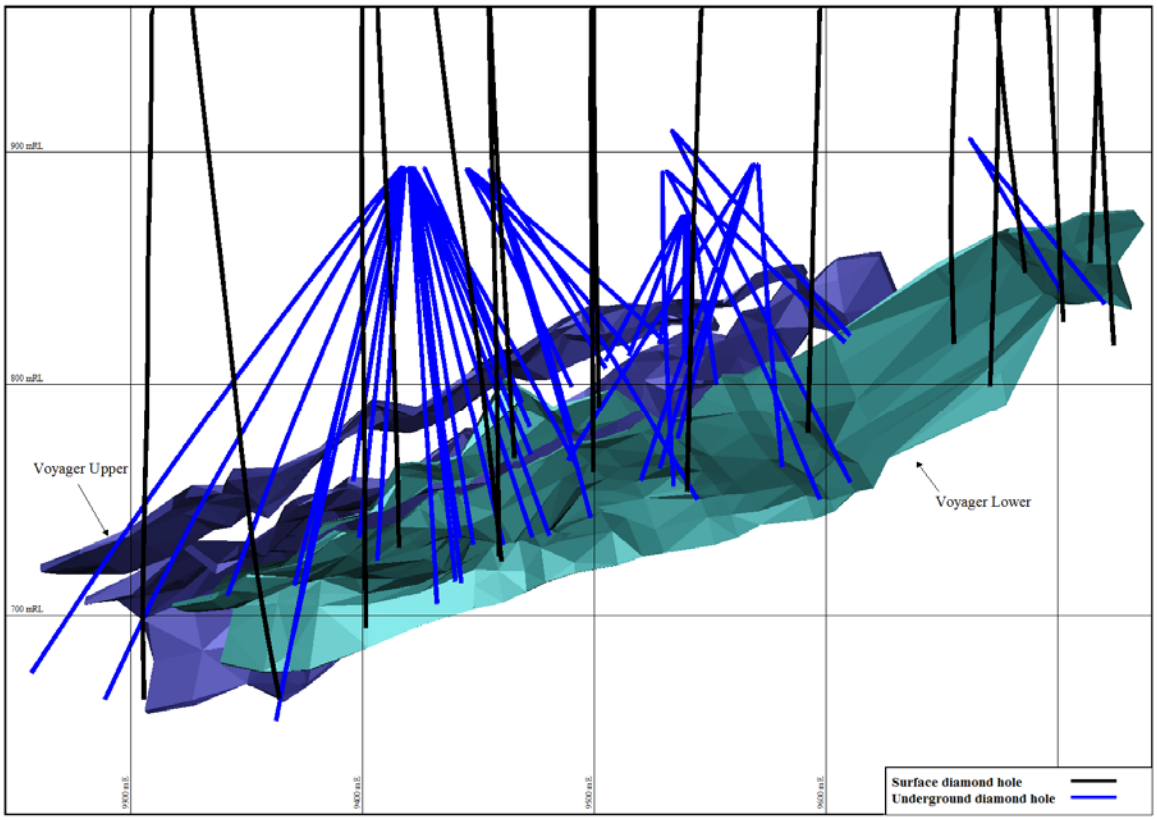


Figure 7: Voyager mineralized domains and drill traces

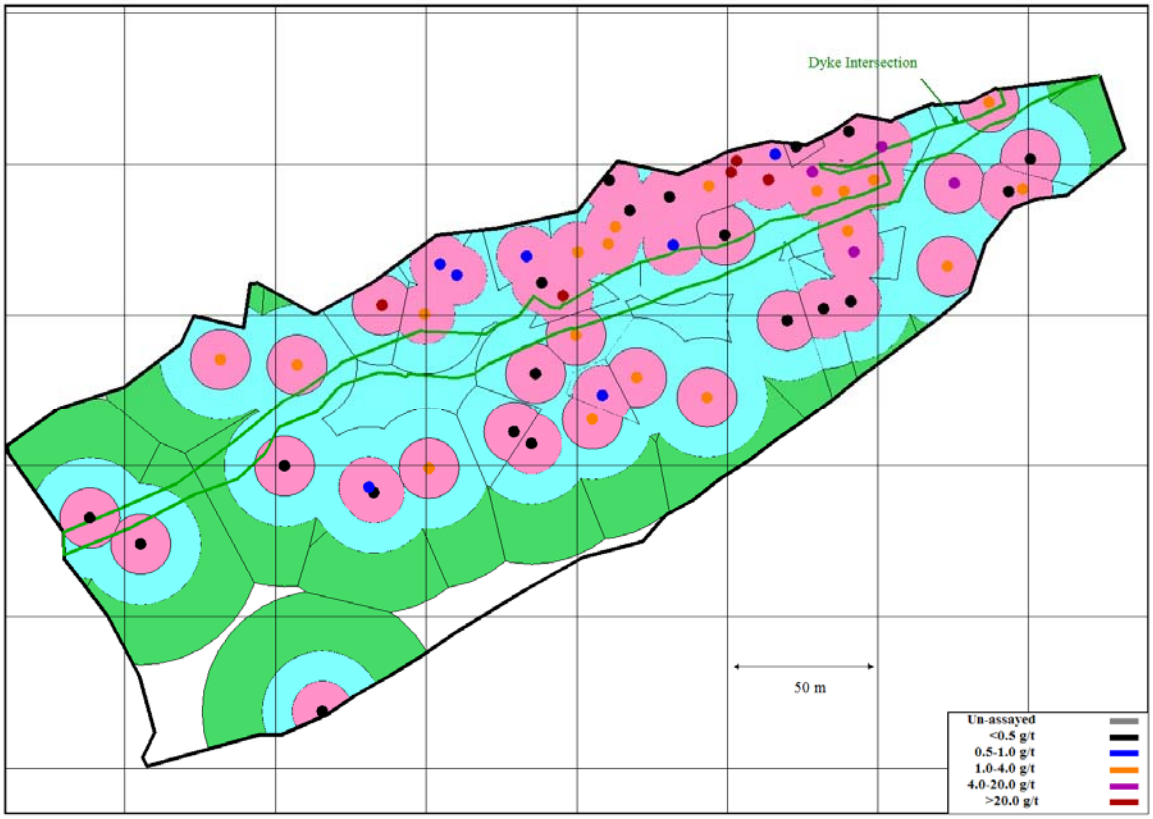


Figure 8: Voyager Upper Zone Inclined section showing intercept spacing

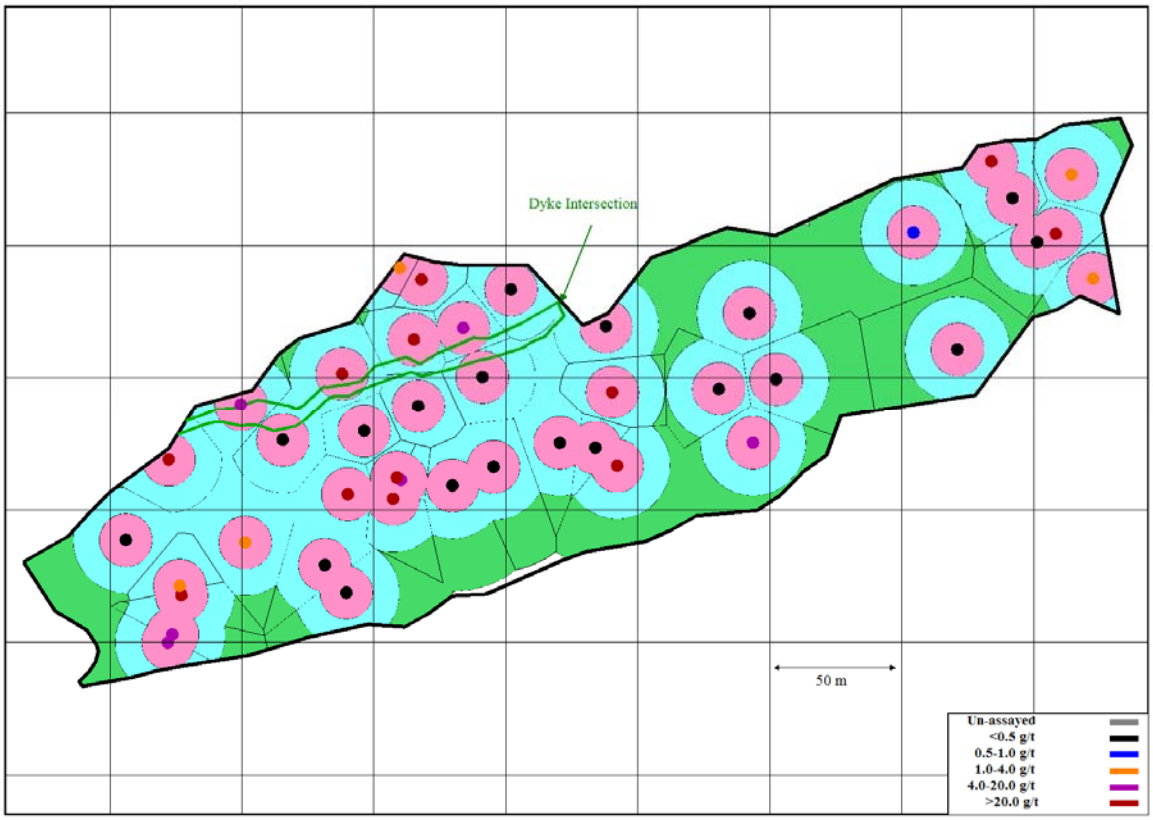


Figure 9: Voyager Lower Zone Inclined section showing intercept spacing

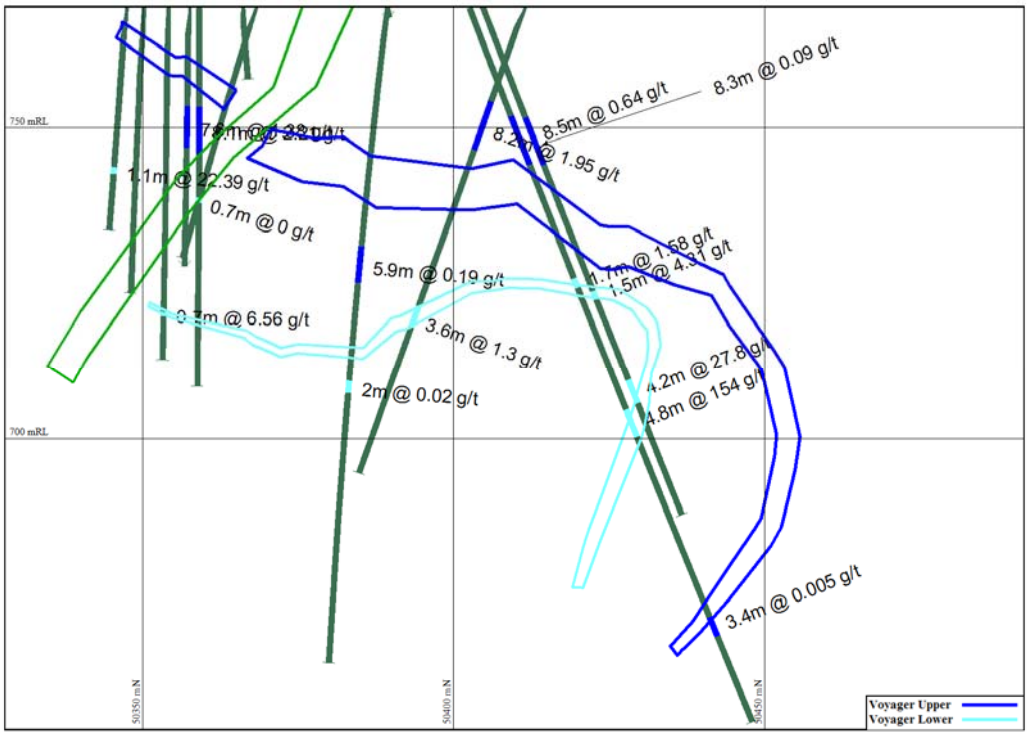
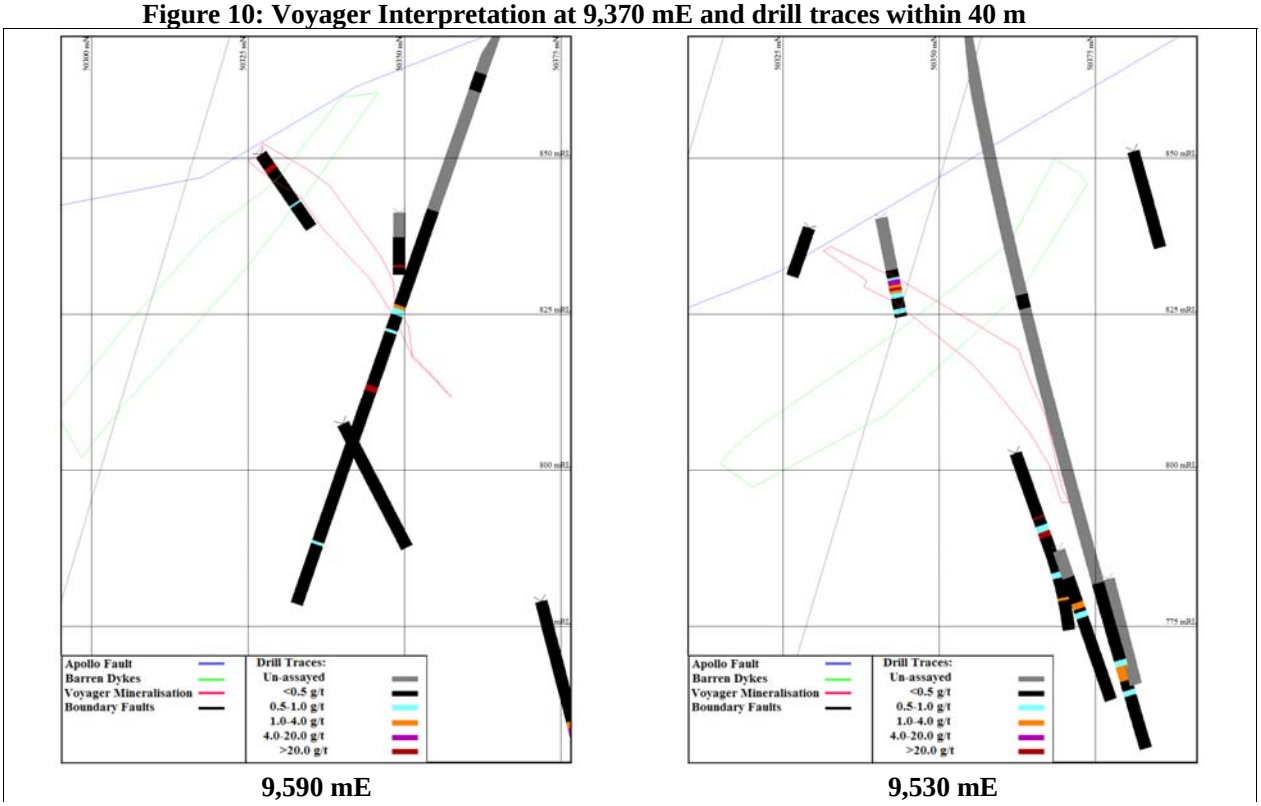


Figure 10: Voyager Interpretation at 9,370 mE and drill traces within 40 m



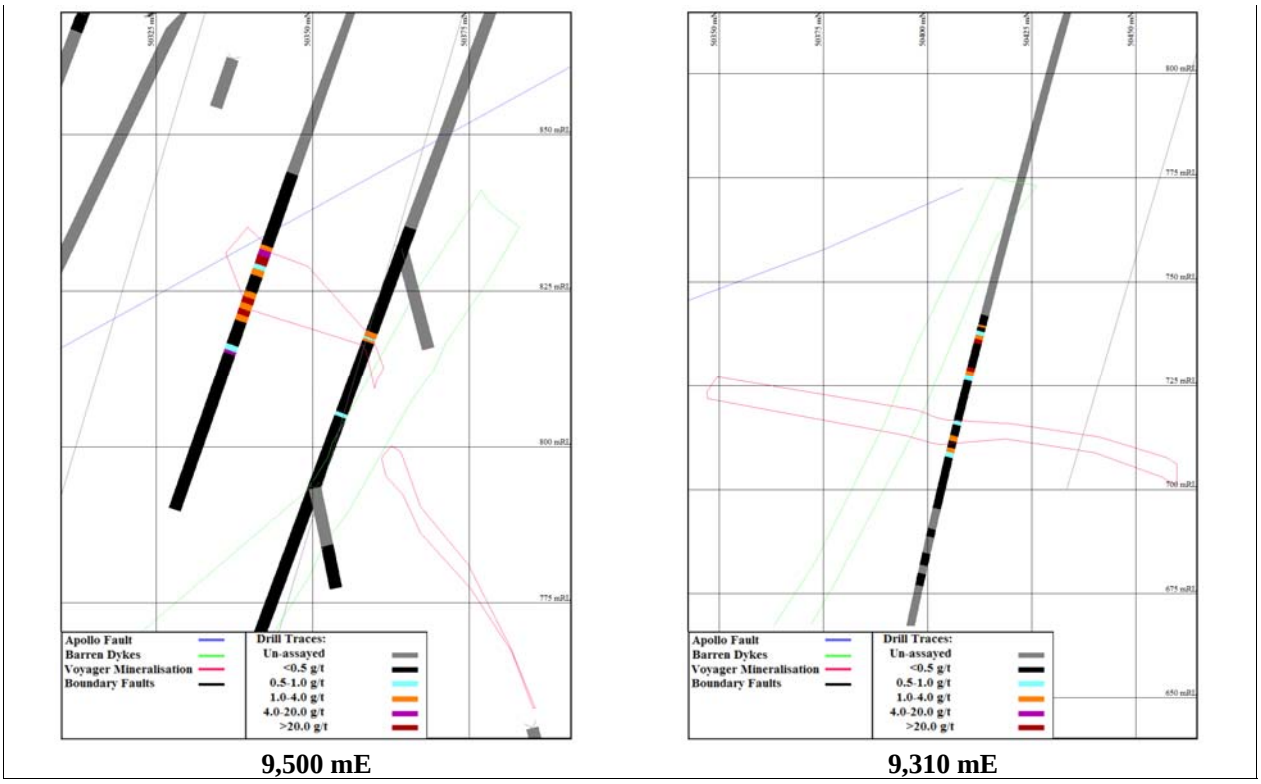


Figure 11: Cross sections of Voyager mineralization interpretation and drilling

Resource Estimation

General

Mineral Resources were estimated by Ordinary Kriging of nominally one meter down-hole composited gold grades within the mineralized wireframes, incorporating upper cuts of 45 to 80 g/t selected for each lode. For each estimated block, the search ellipsoid was aligned with the local mineralization orientation.

Although the interpreted dip of Gemini mineralization is relatively consistent, this domain shows considerable strike variation, so the dip was assumed to be constant and the strike estimated for each block. Upper Zone, mineralization shows considerable variation in dip and strike, so for this domain both dip and strike were estimated. Since the Upper Zone Splay domain is relatively small, and has been largely mined out this domain was estimated with a single set of search passes and orientations. All other domains have relatively consistent strikes but variable dips, so only the dips were estimated for each block within these domains.

Gold grades were Kriged into parent blocks of 10 by 4 by 10 meters (east, north, elevation) which were sub-blocked to a minimum of 1.0 by 0.25 by 0.25 meters for accurate domain boundary resolution. For each domain between one and 13 Kriging runs were undertaken with variably oriented search ellipsoids and variogram models. The appropriate estimate was assigned to each parent block on the basis of local domain orientation.

Each Kriging run included estimates with and without sludge and face samples. The grade estimates including sludge and face samples were used only in areas which have insufficient RC and diamond sampling to provide estimates.

The main steps of the resource estimation are described below:

- For most mineralized domains, centre lines were digitised for ten metre spaced north-south sections. For Gemini centre lines were digitised on ten metre spaced plan vies, and for Upper Zone both sectional and plan view mid point strings were digitised.
- The location and dip or strike as appropriate were calculated for points along each centre line at intervals of generally two metres.
- Dip and or strike values were estimated for each domain into parent cell blocks with dimensions of 20 metres east west by 4 metres north-south by 10 metres vertical. Orientations were estimated using a simple inverse distance squared weighting method with a search ellipsoid aligned with each domain's average mineralization orientation.
- Gold grades were estimated for the parent cells by Ordinary Kriging of the mineralized composites. For each mineralized domain, between one and thirteen Ordinary Kriging estimates were undertaken with search ellipsoid and variogram models variably aligned to cover the range of domain orientations.

- Each Kriging estimate included three or four progressively more relaxed search criteria, and two passes. The first pass included only composites from RC and diamond drilling, and the second pass included estimates also included composites from face and sludge sampling.
- The appropriate Kriged gold grade was assigned to each parent cell on the basis of the estimated orientation.. For example blocks with an estimated dip of 45 degrees were assigned a Kriged grade from the grade model that utilized a 45 degree dipping search ellipsoid and corresponding variogram model.
- Importing the resultant estimates into Vulcan software, and sub-blocking the 20 by 4 by 20 metre parent blocks to a minimum of 2.0 by 0.5 by 1.0 metres for accurate volumetric representation.

Model extents and panel sizes

Model dimensions and block sizes are shown in Table 11. Grades were estimated into 20 by 4 by 20 metre blocks which were sub-blocked in Vulcan to a minimum size of 2.0 by 0.5 by 1.0 metres for precise domain boundary resolution.

Table 11: Block model extents and block sizes

	Easting	Northing	Elevation
Minimum	9,200 mE	50,280 mN	640 mRL
Maximum	9,700 mE	50,540 mN	900 mRL
Extents	500 m	260 m	260 m
Parent block size	20 m	4.0 m	20 m
Number of parent blocks	25	65	13
Minimum sub-block size	2.0 m	0.5 m	1.0 m

Estimation of domain orientation

The Paulsens mineralized domains are generally interpreted with relatively consistent strikes but variable dips, so for most domains the dips were estimated for each block, and the strike assumed to be constant, with the following exceptions:

- Although the interpreted dip of Gemini mineralization is relatively consistent, this domain shows considerable strike variation, so the dip was assumed to be constant and the strike estimated for each block.
- Upper Zone, mineralization shows considerable variation in dip and strike, so for this domain both dip and strike were estimated.

- The Upper Zone Splay domain is relatively small, and has been largely mined out. Estimation of this domain was therefore simplified by using a single set of search passes and variogram models.
- Voyager Lower Zone is insufficiently sampled to justify estimation with variable search passes, so an isotropic search ellipsoid was used for this domain.

For the domains for which orientations were estimated sectional strings were digitized at lode mid points for north-south cross sections or plan views dependent on the direction which showed greatest variability. For upper zone both sectional and plan views mid point strings were digitized. The location and dip or strike as appropriate was calculated for points along each centre line at intervals of generally two metres.

Dip and or strike values were estimated for each domain into parent cell blocks with dimensions of 20 meters east west by 4 meters north-south by 10 meters vertical. Orientations were estimated using a simple inverse distance squared weighting method with a search ellipsoid aligned with each domain's average mineralization orientation. The search parameters applied to each domain were selected on the basis of the domain's orientation variability and data spacing.

As an example of the criteria used for estimation of domain orientation Table 12 shows the search criteria used for estimation of the Lower Zone dips.

Table 12: Estimation criteria for Lower Zone dips

Orientation	50° towards 075
Rotation	Z+15,X-50
Radii	Strike: 20 m, dip 30 m, cross strike 15 m
Minimum data	2
Maximum data	8

Composited gold grades used for estimation

The current estimates are based on wireframes generated by Intrepid using Vulcan software. Gemcom software was used to generate de-surveyed composites within the mineralized domain which form the basis of resource estimates.

Vulcan software uses a differing de-surveying algorithm to other resource software packages which, dependent on the configuration of down hole surveying positions, can give small variations in the calculated de-surveyed drill hole traces. This can lead to inconsistencies where different software packages are used for calculating intercepts and producing wireframes. All those discrepancies are typically very small, however, for narrow, complex mineralized zones sampled by long, variably surveyed drill holes such as is commonly the case at Paulsens, they can cause significant differences in locating the drill hole intercepts.

The current estimates were generated from nominally one meter down-hole composites which honor domain boundaries. To ensure consistency with Intrepid's interpretation of mineralized domains, for the December 2008 model Vulcan software was used to intersect drill hole intervals with domain boundaries. For the January 2009 (Voyager) model, consistency between Intrepid's wireframe interpretation and mineralized intervals a Vulcan composite file supplied by Intrepid (Table 3) was used to define drill hole intersections with domain boundaries.

Sampling intersections with the mineralized domain include a number of intersections of less than 0.5 meters in length. Rather than narrow zones of mineralization, these short intervals represent intervals at the start or end of holes, or where drilling sub-parallel to the lode has a glancing intersection with the domain and does not penetrate the full lode thickness. These short intersections were excluded from the dataset used for resource estimation.

Composites at domain boundaries less than 0.5 meters in length were merged with the previous up-hole composite, giving a dataset with composite lengths ranging from 0.5 to 1.5 meters in length. As shown by the summary of composite lengths for RC and diamond drilling presented in Table 13 the majority of mineralized composites are one meter in length.

Table 14 shows the number and mean grade of composites by mineralization domain and sampling type. This table demonstrates that the Gemini drilling is dominated by RC sampling, and Soyuz and the Lower Zone production area have significant proportions of RC drilling of 19% and 8% respectively. The other domains have only small proportions of RC drilling.

Table 13: Composite length summary for RC and diamond drilling

Domain	Composite Length (m)				< 1.0 Meter		>1.0 <Meter	
	Number	Min.	Avg	Max.	No.	Prop'n	No.	Prop'n
Gemini	148	0.53	1.02	1.47	17	11%	26	18%
Soyuz	160	0.51	1.00	1.48	20	13%	19	12%
Cassini	93	0.52	0.96	1.43	25	27%	12	13%
Apollo	315	0.51	1.01	1.49	56	18%	68	22%
LZ Prod'n	183	0.51	0.98	1.49	36	20%	20	11%
LZ West	283	0.51	1.01	1.49	61	22%	66	23%
Lower Zone	231	0.53	1.00	1.49	32	14%	24	10%
Upper Zone	2,241	0.51	1.00	1.49	151	7%	118	5%
UZ Splay	97	0.52	0.98	1.44	10	10%	9	9%
Voyager UZ	377	0.51	1.00	1.47	30	8%	25	7%
Voyager LZ	91	0.29	0.98	1.48	22	24%	19	21%
Total	**4,219**	**0.29**	**1.00**	**1.49**	**460**	**11%**	**406**	**10%**

Table 14: Composites by domain and sampling type

Domain		RC	Surface DDH	UG DDH	Sludge	Face	Total
Gemini	Number	118	30	-	-	19	167
	Average	7.01	10.4	-	-	0.57	6.89
Soyuz	Number	62	2	96	-	161	321
	Average	7.36	3.18	4.43	-	5.28	5.42
Cassini	Number	74	11	8	23	5	121
	Average	3.15	0.77	1.18	13.1	0.88	4.60
Apollo	Number	35	-	280	94	371	780
	Average	2.74	-	9.47	5.25	13.04	10.4
Lower Zone Prod'n	Number	96	22	65	775	288	1,246
	Average	15.0	21.0	6.34	14.1	21.2	15.5
Lower Zone West	Number	14	35	234	36	357	676
	Average	7.05	5.47	6.69	17.7	10.6	9.26
Lower Zone	Number	10	12	209	1	124	356
	Average	2.97	4.53	5.16	5.24	11.1	7.14
Upper Zone	Number	12	297	1,932	2,189	1,152	5,582
	Average	1.64	6.40	7.34	6.91	6.92	7.02
Upper Zone Splay	Number	-	15	82	229	4	330
	Average	-	11.1	4.81	7.80	0.001	7.11
Voyager Upper Zone	Number	-	66	311	-	-	377
	Average	-	2.20	5.55	-	-	4.96

Voyager Lower Zone	Number	-	33	58	-	-	91
	Average	-	2.84	20.3	-	-	14.0

Composite statistics and upper cut selection

Table 15 shows composite statistics for composited gold grades for RC and diamond sampling within the mineralized domains. For each domain the composite grades are highly variable as reflected by the high coefficients of variation, with mean grades strongly influenced by a few high grade outliers which are commonly from only one or two drill holes. Treatment of these high grade outliers in resource estimates represents a significant source of potential model inaccuracy.

The less closely drilled domains, such as Voyager Lower Zone, contain too few composites for accurate selection of an appropriate upper cut value. For these domains, the upper cut selected for the current estimate can be only regarded as an approximation and is likely to change after completion of further drilling. Given the sensitivity of estimates to treatment of high grade composites, the uncertainty in upper cut selection represents a significant source of potential model inaccuracy.

For each mineralized domain, upper cuts were selected from ranked lists of composites from RC and diamond drilling. Sludge and face samples were not included in the dataset used for selection of upper cuts. As shown in Table 15 the selected upper cuts generally approximate the 98th percentile of the drilling datasets.

Table 15: Composites statistics and upper cuts

	Gemini	**Soyuz**	**Cassini**	**Apollo**	**Lower Zone Prod**	**Lower Zone West**
Number	148	160	93	315	183	283
Mean	**7.70**	**5.55**	**2.70**	**8.72**	**12.7**	**6.55**
Variance	233	121	132	931	598	204
Coef. var.	1.98	1.98	4.26	3.50	1.93	2.18
Minimum	0.050	0.001	0.005	0.001	0.003	0.001
25%ile	1.79	0.23	0.05	0.04	0.31	0.06
50%ile	3.85	1.08	0.52	0.88	3.08	0.86
75%ile	8.27	4.23	2.04	5.92	13.73	4.71
Maximum	143	63.3	111	296	167	94.1
Upper cut	**50**	**50**	**50**	**60**	**80**	**70**
%ile	98.0%	97.5%	98.9%	98.1%	97.3%	98.6%
No. Cut	3	4	1	6	5	4
Cut mean	**6.71**	**5.35**	**2.04**	**5.94**	**11.47**	**6.33**

	Lower Zone	**Upper Zone**	**Upper Zone Splay**	**Voyager Upper**	**Voyager Lower**	
Number	231	2,241	97	377	91	
Mean	**5.03**	**7.19**	**5.78**	**4.96**	**14.0**	
Variance	216	855	191	1,251	2,152	
Coef. var.	2.92	4.07	2.39	7.13	3.32	
Minimum	0.001	0.001	0.001	0.001	0.001	
25%ile	0.03	0.06	0.06	0.04	0.03	
50%ile	0.30	0.43	0.72	0.21	2.45	
75%ile	3.41	3.14	3.81	1.45	6.53	
Maximum	148	704	75.3	588	355	
Upper cut	**60**	**75**	**45**	**60**	**60**	
%ile	99.1%	98.0%	95.9%	98.9%	94.5%	
No. Cut	2	44	4	4	5	
Cut mean	**4.44**	**5.58**	**4.99**	**2.65**	**7.94**	

Estimation of gold grades

As listed in Table 17 for each domain between one and 13 Kriging runs were undertaken with variably oriented search ellipsoids and variogram models, with the appropriate estimate assigned to each parent block on the basis of local domain orientation. The apparent dips (i.e. to the north) derived from digitized section lines were converted to true dips at the local mineralization strike. **Figure 12** shows three dimensional plots of the four search ellipsoids used for Kriging of gold grades for the combined Lower Zone domain as examples of the ellipsoids used for estimation.

The ellipsoid rotations listed in Table 17 which align the x, y and z ellipsoid axes with the strike, dip and across strike directions respectively are defined in the anticlockwise positive convention used by many software packages including GS3 and may differ from the convention used in Vulcan.

Each Kriging run included estimates with and without sludge and face samples. The grade estimates including sludge and face samples were used only in areas with insufficient RC and diamond sampling to provide estimates.

One set of variograms (Table 16) modeled from a closely sampled portion of the upper zone mineralization was used for estimating all domains with appropriate rotations.

The search criteria used for estimation (Table 18) included three progressively relaxed search passes with a fourth, larger search pass included for the broadly sampled Soyuz, Cassini and Voyager Lower Zone domains. The same set of search ellipsoid radii were used for each domain with the exception of flatter portions of Upper Zone which used broader across strike searches, and the Voyager Lower Zone. Voyager Lower Zone is insufficiently sampled to justify estimation with variable search passes, so an isotropic search ellipsoid was used for this domain.

With the exception of the three Lower Zone domains which were combined for estimation, all domain boundaries were treated as hard. For compilation of the Kriged grades into a final model, a single grade was used for each 10 by 4 by 10 meter parent block on the basis of dominant mineralization domain. With the exception of adjacent Upper Zone Splay and Upper Zone domains, very few parent blocks are intersected by more than one domain which justifies this simplification.

Table 16: Variogram models

Nugget (C_0)	First Structure (C_1)	First Structure (C_2)
0.10	0.68 exp(8.0,9.5,3.5)	0.22 exp (21,14,4.0)

Table 17: Kriging runs

Domain	Krig	Strike	Dip (North)	Dip True	Rotation

	Run	Range	Average	Range	Average	Average	
Gemini	1	<077	064		40	43	z+26, x-43
	2	075-090	082		40	40	z+8, x-40
	3	>090	110		40	42	z-20, x-42
Soyuz	1		065	<25	6	6	z+25,x-6
	2		065	25-40	32	35	z+25,x-35
	3		065	>40	46	48	z+25,x-48
Cassini	1		044	<25	20	27	z+35,x-27
	2		044	>25	42	42	z+35,x-42
Apollo	1		075	10-40	31	32	z+15, x-32
	2		075	40-60	51	53	z+15, x-52
	3		075	60-80	66	71	z+15, x-67
Lower Zone	1		075	<30	22	23	z+15, x-23
Combined	2		075	30-45	37	38	z+15, x-38
	3		075	45-60	50	51	z+15, x-51
	4		075	>60	78	78	z+15, x-78
Upper Zone	1	<000	339	All	-14	33	z-76, x+33
	2	000-030	014	-30 to 0	-6	25	z+76, x+25
	3	000-030	013	0 to 30	11	40	z+77, x-40
	4	000-030	016	>30	43	74	z+74, x-74
	5	030-060	042	<-30	-44	55	z+48, x +55
	6	030-060	045	-30 to 0	-14	20	z+45 x+20
	7	030-060	043	0 to 30	18	27	z+49, x-27
	8	030-060	043	30 to 60	45	56	z+47, x-56
	9	>060	101	<0	-6	6	z-11, x+6
	10	>060	085	0 to 30	17	17	z+5, x-17
	11	>060	082	30 to 60	50	50	z+8, x -50
	12	>030	089	60 to 90	70	70	z+1, x-70
	13	>030	093	>90	71 S	71	z-3, x+19
Upper Zone Splay	1		025		23	35	z+65,x-35
Voyager Upper	1		085	0-15	7	7	z+5 x-7
	2		085	15-30	23	23	z+5 x-23
	3		085	30-45	36	36	z+5 x-36
	4		085	45-60	51	51	z+5 x-51
	5		085	60-90	72	72	z+5 x-72
	6		085	>90	107	107	z+5 x+73
Voyager Lower	1		075	<0	-13	-13	z+15 x+13
	2		075	0-25	15	15	z+15 x-15
	3		075	25-50	36	37	z+15 x-37

	4	075	50-70	58	59	z+15 x-59
	5	075	70-90	79	79	z+15 x-79
	6	075	90-110	102	-78	z+15 x+78
	7	075	>110	112	-68	z+15 x+68

Table 18: General search criteria

Search Pass	Radius (strike, dip, cross strike)	Minimum Data	Minimum Octants	Maximum Data
1	40,40,8	16	4	32
2	60,60,12	16	4	32
3	60,60,12	8	2	32
4	120,120,24	8	2	32

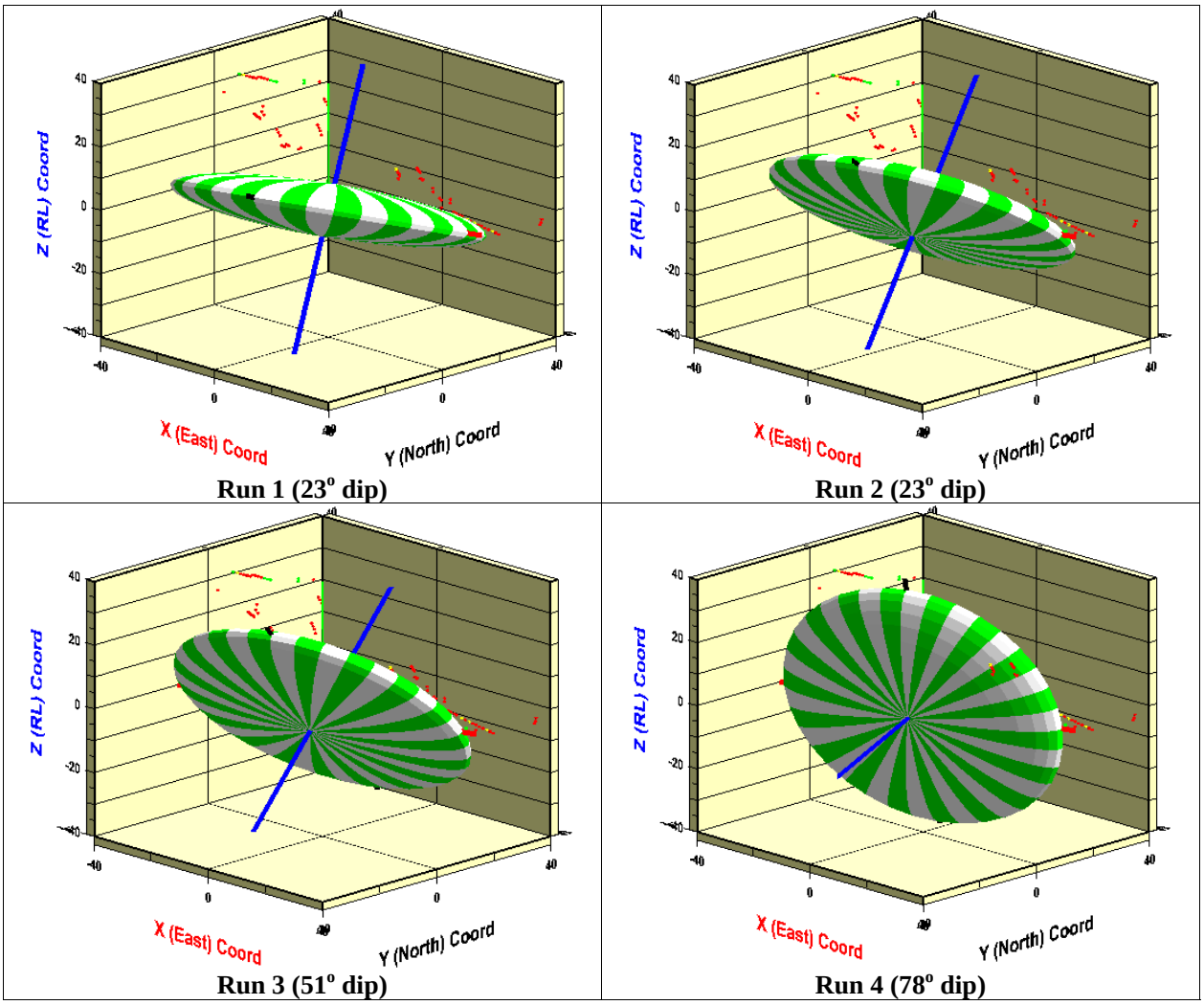


Figure 12: Lower Zone search ellipsoids

Resource classification

Attributes available for each block include the search pass, and estimated gold grade for Kriging passes with and without sludge and face samples. The resource classification criteria applied to the current estimates as shown in Table 19 assigns blocks estimated by closely spaced RC or diamond drilling, and blocks in areas of development with moderate drill coverage as Indicated. All other blocks are classified as Inferred. The strategy classifies blocks in areas which have been accessed by development but only broadly drilled as Inferred.

This classification strategy is based on the interpretation that although grades from the face and sludge sampling may be unreliable, this sampling defines the location and size of the lodes with sufficient accuracy to justify classifying the local estimate as Indicated in moderately drilled areas tested by face and sludge sampling.

Table 19: Resource classification

Search Pass	**Search Pass including all data**			
With RC & DDH Only	1	2	3	4
1	**Indicated**	Inferred	Inferred	Inferred
2	**Indicated**	Inferred	Inferred	Inferred
3	Inferred	Inferred	Inferred	Inferred
Un-estimated	Inferred	Inferred	Inferred	Inferred

4.8 Estimates versus composite grades

Figure 13 to **Figure 15** compare average estimated gold grade with average cut composite grade by elevation for the main mineralized domains included in the current estimate. These figures show that although the Kriged estimates are generally more smoothed than the average composite grades, they generally closely follow the trends shown by the composite mean grades with the exception of areas of very limited sampling at the upper and lower extents of domains.

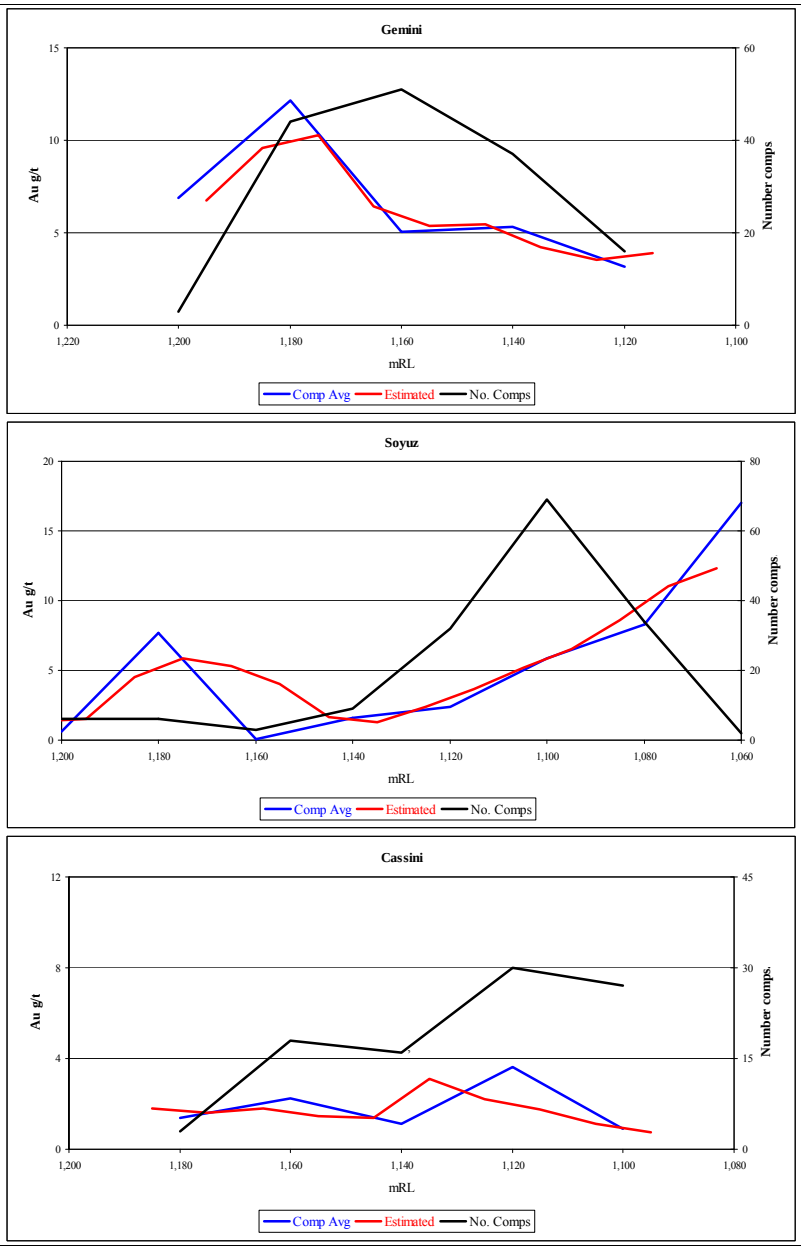


Figure 13: Model versus composites for Gemini, Soyuz and Cassini

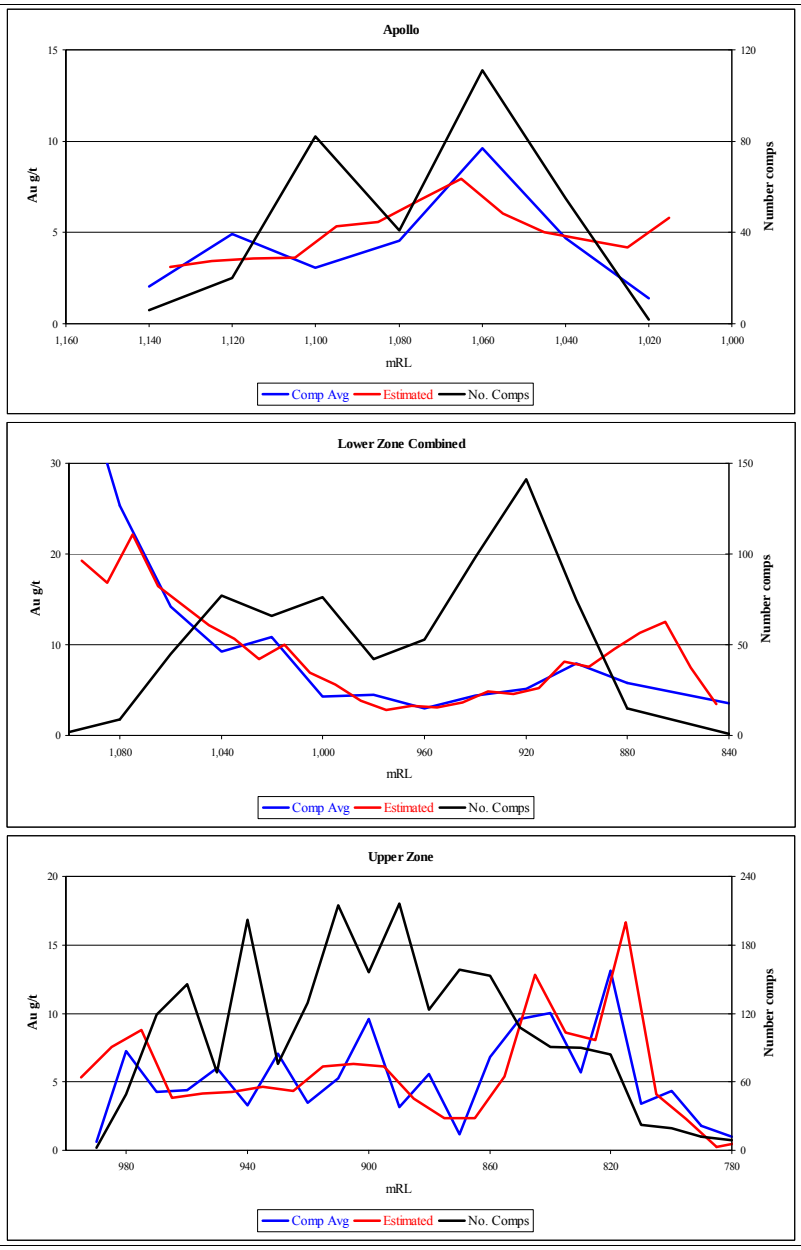


Figure 14: Model versus composites for Apollo, Lower Zone and Upper Zone

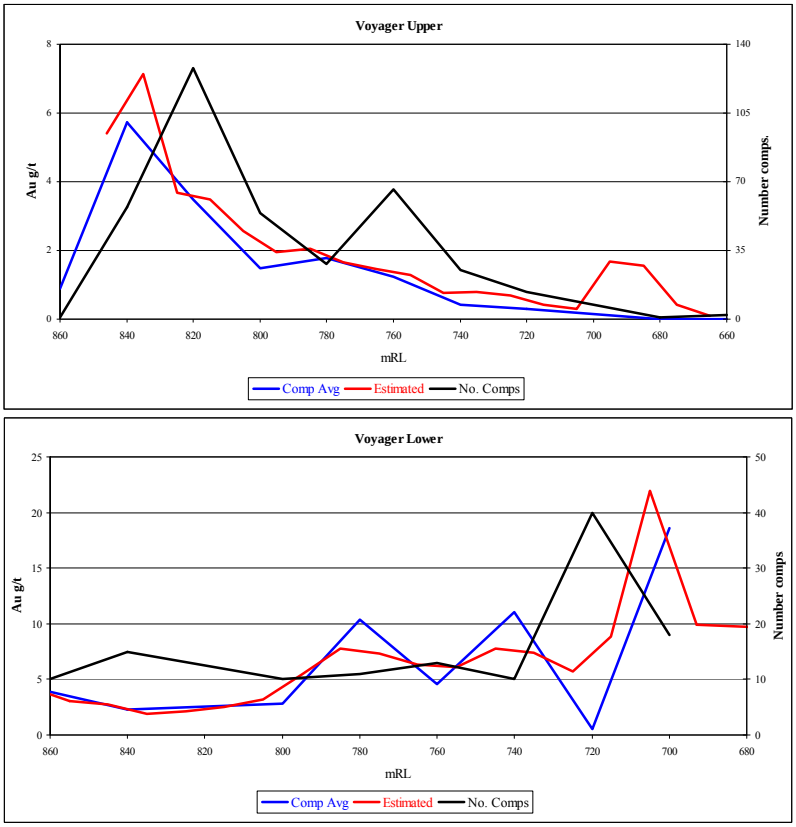


Figure 16: Model versus composites for Voyager Upper and Voyager Lower

Resource estimates

Pre-mining estimates

Table 20 shows in situ (pre-mining) model estimates for each domain subdivided by resource classification, and Table 21 shows in-situ estimates at the 4.0 g/t block cut off grade stipulated by Intrepid. These estimates exclude some shallower mineralization that has been depleted by mining to date with no significance to estimation of remaining Mineral Resources. The figures in these tables are for internal auditing and do not reflect the accuracy of estimates.

Publicly reporting resource estimates consistent with NI43-101 requires exclusion of estimates without potential economic viability such as small isolated zones. Reporting estimates above a block cut off grade includes isolated blocks which may not be viable in a realistic underground mining scenario.

Rather than aggregating individual blocks above a cut off grade, it is generally more appropriate to base publicly released estimates on continuous zones of mineralization above an appropriate grade, and or grade-thickness cut off. However, for most of the Paulsens domains, blocks estimated at above 4.0 g/t generally compromise reasonably continuous zones, so the effect of reporting above this block cut off grade does not appear to materially affect the reliability of estimates.

Table 20: Paulsens pre-mining resource estimate entire lodes

	Indicated		Inferred		Total		
	Tonnes	**Au g/t**	**Tonnes**	**Au g/t**	**Tonnes**	**Au g/t**	**Au oz**
Gemini	40,943	6.88	17,660	5.22	58,603	6.38	12,016
Soyuz	33,537	5.71	39,925	5.18	73,462	5.42	12,798
Cassini	4,881	2.62	103,680	1.98	108,560	2.01	7,023
Apollo	68,129	6.72	101,999	4.42	170,128	5.34	29,228
Lower Zone Prodx	117,337	12.01	53,756	10.62	171,093	11.57	63,639
Lower Zone West	36,096	6.90	46,961	8.98	83,057	8.07	21,557
Lower Zone	30,697	3.39	22,687	3.09	53,384	3.26	5,593
Upper Zone	568,203	5.50	46,029	8.26	614,231	5.71	112,745
Upper Zone Splay	18,383	5.48	5,698	3.47	24,081	5.00	3,874
Voyager Upper Zone	30,164	3.00	347,163	1.51	377,327	1.63	19,788
Voyager Lower Zone	-	-	151,438	7.14	151,438	7.14	34,772
Total	**948,370**	**6.35**	**936,994**	**4.29**	**1,885,364**	**5.33**	**323,034**

Table 21: Paulsens pre-mining resource estimate December 2008 4 g/t cut off

	Indicated	Inferred	Total

	Tonnes	Au g/t	Tonnes	Au g/t	Tonnes	Au g/t	Au oz
Gemini	34,561	7.52	11,196	6.44	45,757	7.26	10,675
Soyuz	22,464	7.30	22,037	8.12	44,501	7.71	11,024
Cassini	1,387	4.76	12,703	6.29	14,090	6.14	2,781
Apollo	50,133	8.26	48,479	6.99	98,612	7.64	24,214
Lower Zone Prodx	105,344	13.06	47,913	11.58	153,257	12.59	62,058
Lower Zone West	28,503	7.89	36,611	10.77	65,114	9.51	19,908
Lower Zone	9,102	6.43	6,729	6.52	15,831	6.47	3,292
Upper Zone	310,972	8.36	18,435	17.93	329,407	8.89	94,162
Upper Zone Splay	11,976	6.92	2,372	5.47	14,348	6.68	3,082
Voyager Upper - east	-	-	38,104	11.77	38,104	11.77	14,415
Voyager Lower	-	-	33,250	14.30	33,250	14.30	15,292
Total	**574,442**	**9.03**	**277,828**	**10.55**	**852,271**	**9.52**	**260,902**

Estimates of remaining resources

Table 22 lists a set of Vulcan format triangulations supplied by Intrepid representing the mineralized volumes mined by level development, and areas stoped out or rendered inaccessible by mining to date, or areas of peripheral lower grade mineralization that are not considered of economic interest. The supplied wireframes also include a set of triangulations representing designed stopes that Intrepid consider as potentially exploitable by hand-held mining within otherwise inaccessible areas.

The level wireframes are restricted to the mineralized areas, and exclude some development outside mineralization.

The validity of these wireframes has not been reviewed for the current study. Intrepid are accepting responsibility for this aspect of the resource estimate.

Figure 16 presents a long section view of the depletion drive and stoped/inaccessible wireframes relative to the mineralization wireframes used for the current estimates. For clarity, this figure is trimmed to exclude some lower Voyager mineralization beyond the limits of the depletion wireframes.

The triangulation supplied for the 1,089 mRL level development (cut _0812_lev1089.00t) included two areas of invalid triangles. To create a valid triangulation, these portions of the drive were cut from the triangulation. The combined 14 meters of development removed in this process do not fall within mineralized domains, and therefore have no influence on estimates of remaining resources.

The mining depletion wireframes were merged into a single triangulation (20090317_Mined.00t), and the potentially mineable lower zone stopes within otherwise inaccessible areas were merged into a single triangulation (20090319_LZ_Stopes.00t).

Table 22: Supplied mining depletion wireframes

Levels	cut _0812_lev808lzw.00t cut _0812_lev842.00t cut _0812_lev873.00t cut _0812_lev895uz.00t cut _0812_lev904lzw.00t cut _0812_lev919lzw.00t cut _0812_lev928.00t cut _0812_lev943LZW.00t cut _0812_lev943LZW_AL.00t cut _0812_lev986LZ.00t cut _0812_lev986LZW_A.00t cut _0812_lev986LZW_B.00t cut _0812_lev995LZW.00t	cut _0812_lev1007LZ.00t cut _0812_lev1019LZ.00t cut _0812_lev1028lz.00t cut _0812_lev1028LZW.00t cut _0812_lev1060.00t cut _0812_lev1067W.00t cut _0812_lev1089.00t cut _0812_lev1161.00t cut _0812_levA1046.00t cut _0812_levA1068.00t cut _0812_levA1081.00t cut _0812_levA1095.00t
Mined and inaccessible areas	cutmined_0812_ap2.00t cutmined_0812_AP3.00t cutmined_0812_ap.00t cutmined_0812_lz2x.00t	cutmined_0812_lz.00t cutmined_0812_lzw.00t cutmined_0812_uz22x.00t cutmined_0812_UZ.00t
Lower Zone stopes within otherwise inaccessible areas	0901_lz_1010_200_b6.00t 0901_lz_1019_200_b6.00t 0901_lz_1045_200_1.00t	0901_lz_1052_200_1.00t 0901_lz_1060_100_1.00t

Estimates of remaining resources

To estimate remaining resources, the Vulcan resource model was evaluated for estimates greater than 4 g/t gold outside the merged mining depletion triangulations excluding potentially mineable lower zone stopes within generally inaccessible areas. The block cut off grade was stipulated by Intrepid. Table 23 lists the estimated remaining resources by lode and resource category with estimates are subdivided into upper and lower zones for consistency with Intrepid's standard resource reporting format.

Table 23: Remaining resources by domain at 4.0 g/t cut off

		Indicated		Inferred		Combined	
		Tonnes	**Au g/t**	**Tonnes**	**Au g/t**	**Tonnes**	**Au g/t**
Upper Zone	Gemini	34,561	7.52	11,196	6.44	45,757	7.26
	Soyuz	18,168	7.03	21,520	8.02	39,688	7.57
	Upper Zone Splay	381	5.00	82	4.95	463	4.99
	Upper Zone	120,642	9.86	9,953	23.14	130,595	10.87
	Voyager UZ	9,301	7.12	28,918	7.17	38,219	7.16
	Subtotal	**183,053**	**8.99**	**71,669**	**9.53**	**254,723**	**9.14**
	Rounded	**183,000**	**9.0**	**70,000**	**10**	**253,000**	**9.3**
Lower Zone	Cassini	1,387	4.76	8,768	5.21	10,155	5.15
	Apollo	36,206	8.38	22,253	6.23	58,459	7.57
	Lower Zone Prod	18,500	16.93	**3,837**	**10.21**	22,337	15.78
	Lower Zone West	22,076	7.27	31,085	10.76	53,161	9.31
	Lower Zone	7,603	6.50	4,686	6.35	12,288	6.45
	Voyager LZ	-	-	92,180	10.17	92,180	10.17
	Subtotal	**85,772**	**9.72**	**162,809**	**9.37**	**248,581**	**9.49**
	Rounded	**86,000**	**9.7**	**160,000**	**9**	**246,000**	**9.2**
Total unrounded		**268,825**	**9.22**	**234,478**	**9.42**	**503,304**	**9.31**
Total of rounded		**269,000**	**9.2**	**230,000**	**9**	**499,000**	**9.3**

19.2. MINERAL RESERVE ESTIMATE

The following description of reserve estimation for the Paulsens deposit is derived from Scrimshaw, 2009.

PAULSENS GOLD MINE ORE RESERVES, 31ST DECEMBER 2008

Intrepid Mines Limited
Paulsens Operation

TR_intrepid_paulsens_final_090325

25th March 2009

Mintech Services Pty Ltd T/A Creative Mined Enterprises
Ground Floor, 13 Phillimore St, Fremantle WA 6160

EXECUTIVE SUMMARY

Creative Mined Enterprises was commissioned by Intrepid Mines Limited to provide an independent Ore Reserve Estimation for the Paulsens Gold Mine in the Pilbara region of Western Australia. The work has been carried out under the supervision of Per Scrimshaw, a Competent Person as defined in the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (The JORC Code). Mr Scrimshaw visited the Paulsens site from the 12th to 13th March 2009.

Hellman & Schofield Pty Ltd (H&S) was commissioned by Intrepid Mines Ltd (Intrepid) to estimate Mineral Resources for the Paulsens underground gold mine in the Pilbara region of Western Australia as at the 31st December 2008. This Resource estimate forms the basis for the 31st December 2008 Ore Reserve estimate. All stopes and stockpiles were subjected to an economic evaluation based on historical site fixed costs and contractor/site variable costs. Revenue was based on a gold price of USD750 and USD to AUD exchange rate conversion of 0.7. Costs were based on a mill throughput of 338,000tpa and process recovery of 93.5%. Paulsens is an operational site. All government permits and licenses and statutory approvals have been granted. No risk factors have been applied to the mining rate.

A dilution factor equivalent to 0.75m was applied to both the foot wall and hanging wall of stope boundaries within the Upper Zone regions, while a 0.5m dilution skin was applied to all other regions. A stope mass recovery factor of 91% was applied to the Upper Zone stopes and 84% was applied to all other stopes. In addition, Pillar factors between 6.3% and 11.3% have been applied to the stope shapes.

Stockpiles have been classified as Proven Ore Reserves and Indicated Resources have been converted to Probable Ore Reserves subject to modifying factors, detailed mine design and an economic evaluation. There is no Measured category in the current Resource Model.

The following table outlines the Ore Reserve estimate for Paulsens Gold Mine as of the 31st December 2008. The Ore Reserves are inclusive of the Mineral Resource estimate as at 31st December 2008.

	Proven			Probable			Total		
Zone	Tonnes t	Au g/t	Au Ounces	Tonnes t	Au g/t	Au Ounces	Tonnes t	Au g/t	Au Ounces
Upper				166,600	6.9	36,900	166,600	6.9	36,900
Lower				46,000	7.3	10,900	46,000	7.3	10,900
RoM S/pile	6,900	6.9	1,500				6,900	6.9	1,500
Total	**6,900**	**6.9**	**1,500**	**212,700**	**7.0**	**47,800**	**219,500**	**7.0**	**49,300**

Introduction

Intrepid Mines Limited (Intrepid) commissioned Creative Mined Enterprises (CME) to conduct an Ore Reserve estimate for the Paulsens Gold Mine, located west of Paraburdoo in Western Australia. The work entailed estimating Ore Reserves in conformance with the 2004 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (The JORC Code) and may also be used as reference in compilation of the Ore Reserve section of a Technical Report in compliance with National Instrument 43-101.

The work represents a decrease in the Ore Reserves estimate of the Paulsens Gold Mine since the last disclosure on this deposit. This Ore Reserve estimate represents a depleted Mineral Resource, as well as modified cost and mining factors based on the increased historical performance and changed economic conditions. The Ore Reserve estimate (Proven and Probable) as at 31st December 2008 is 219,500 tonnes grading 7.0 grams per tonne for 49,300 ounces of gold.

Information and data for the independent Ore Reserve Estimate was obtained from the Paulsens Mine Site and Intrepid personnel. The information obtained was not validated in detail, however KPI's and costs are considered to be in line with current industry performances.

The work reported herein was undertaken by Per Scrimshaw who is a full-time employee of CME and a Member of the Australasian Institute of Mining and Metallurgy. Mr Scrimshaw has sufficient experience which is relevant to the style of mineralization and type of deposit under consideration to qualify as a Qualified Person in terms of NI43-101 standards for ore reserve estimation. Mr Scrimshaw visited the Paulsens site from the 12th to 13th of March 2009.

Terms of Reference

The Paulsens Gold Mine consists of gold mineralization within a massive quartz vein. Mineralization is often associated with the footwall and hanging wall portion of the vein and subsequently referred to as the Upper and Lower Zones. Faulting and post mineralized dykes further separate these zones into what is referred to as mining blocks or satellites.

All units of measurement used in this report are metric except contained metal quantities, which are expressed in troy ounces.

Project History

The Paulsens Mine is located 180 km west of Paraburdoo in Western Australia's Ashburton Mineral Field. The topography is generally flat in the area of the deposit, with low surrounding hills. The project is located on the Mt. Stuart Pastoral Station. It is in an area where cyclones dissipate occasionally experiencing heavy rains and elevated wind speeds.

The Property was first explored in 1976 by CRA Limited and eventually acquired by NuStar Mining Corporation Ltd (NuStar) in 1997. In December 2003, a new Board and management raised equity, extinguished debt and increased the Resource by 60%. The underground mine development and mill facility was debt financed based on 1.44 million tonnes of indicated and inferred Resources (541,300 ounces) for average gold production of 80,000 oz per year.

In July 2006 NuStar merged with Intrepid Minerals Corporation, creating Intrepid Mines Limited (Intrepid) listed in both Australia (ASX) and Canada (TSX), under the code, IAU.

In March 2008 Intrepid merged with Emperor Mines Limited.

In December 2008 Intrepid delivered the final ounces associated with its hedge commitments in relation to the Paulsens project.

266,982 oz of gold has been produced from the Paulsens Gold Mine from commencement of operations to the 31st December 2008.

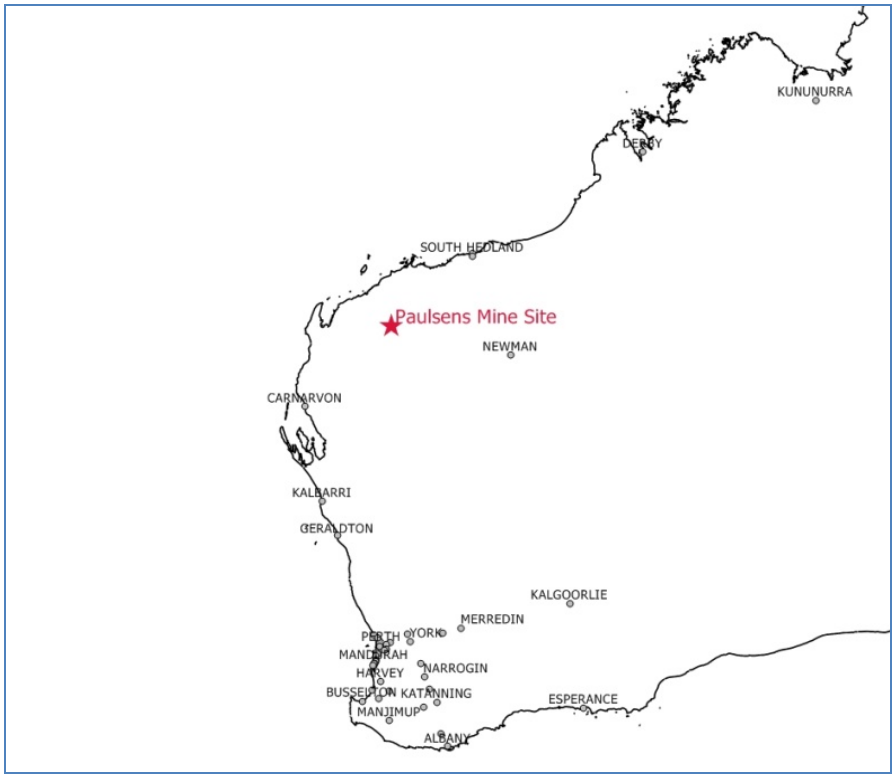


Figure 8 Paulsens Mine Site Location

Economic Parameters

Exchange Rate/Commodity Price

Economic parameters used in this estimation were provided by Intrepid. A gold price of US $750 and a USD to AUD exchange rate of 0.7 was used, providing an Australian dollar gold price of AU $1071.

All dollar values referred to in this report are in Australian dollars unless explicitly stated otherwise.

Secondary Credits

There are minor secondary silver credits associated with the mineralisation at the Paulsens deposit. The silver credits typically offset the majority of gold refining and bullion transportation costs. The cost of refining and the revenue from silver has been considered minimal, and as silver grade has not been estimated in the block model, is therefore disregarded in the economic evaluation.

Costs

Fixed costs for the Paulsens operation are divided into six categories: Processing, Mining Services, Geology Services, Administration, PKKP Royalty and Government Royalty. Predicted Processing, Mining Services, Geology Services and Administration fixed costs for 2009 were based on the Key Performance Indicators achieved for the calendar year 2008 as recorded by Paulsens site personnel. The PKKP Royalty costs are calculated as a flat rate per ounce, and the Government Royalty is applied as a percentage of metal value.

The mining operation utilises the services of a mining contractor and a crushing contractor. The mining contract Schedule of Rates (SOR) is based on a fixed and variable component. The fixed component is primarily mining equipment and contractor staff salaries and is paid as a lump sum per month of the contract. The variable component of the SOR represents operating wages, consumables and maintenance costs and payment is based on a monthly measure and value model.

For the purposes of this report, all costs both fixed and variable incurred from the mining contractor have been removed from the Mining Services cost and applied as separate unit rates in the economic evaluation.

Fixed Costs

Historical cost data has been tracked since the commencement of mining operations. The fixed costs have been converted to KPI's based on monthly mill throughput. The average of these fixed cost KPI's over the 2008 calendar year are used in the economic evaluation. No corporate costs have been included in the analysis. The crushing contract was incorporated into the milling fixed costs KPI. Mining contractor fixed costs have been applied as a standard unit rate.

Mill process throughput has continued to increase on an annualised basis for the Paulsen Site, in line with mill expansion works and optimisation studies. A budgeted Mill process rate of 338,000 tonnes has been used for the 2009 calendar year and subsequent economic evaluation and cut of grade calculations discussed in this study. The increase from the previous Reserve Estimation rate of 325,000 tonnes is considered appropriate given the historical increases achieved following mill commissioning and together with recent monthly throughputs. Figure 9 shows historical annual mill process throughput for years where full year data exists.

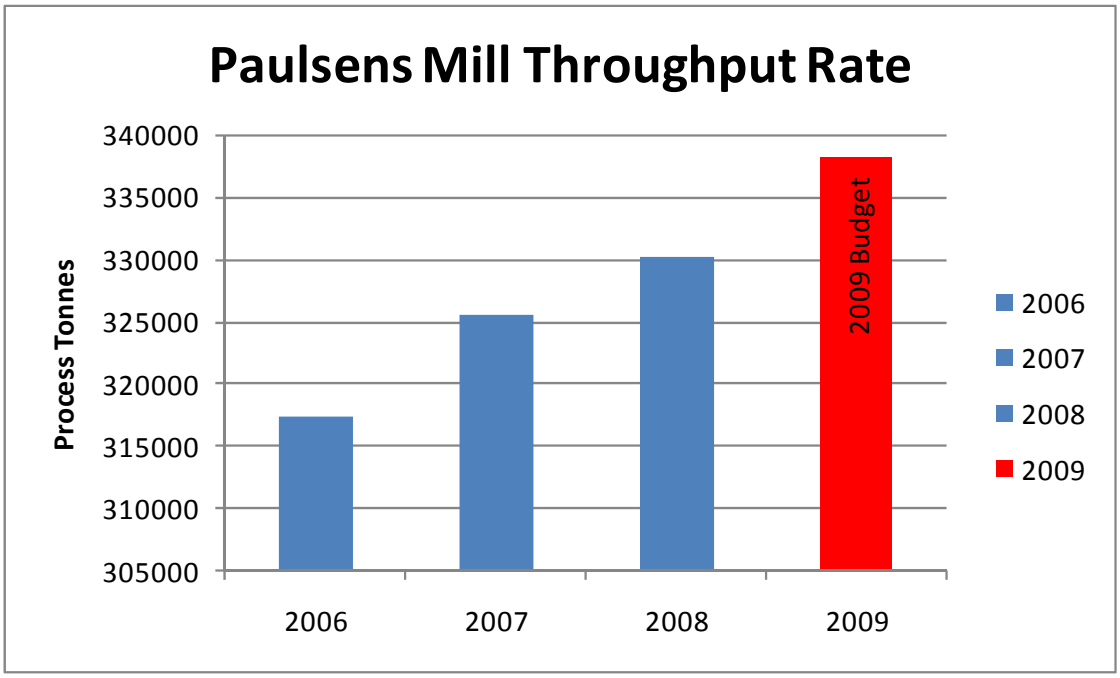


Figure 9 - Paulsens Historical Mill Throughput

Although treated as a fixed cost for the determination of cut off grade, Intrepid advise that approximately 50% of the processing costs vary in proportion to ore throughput.

Table 3 summarises the fixed costs used in the economic evaluation and the source from which they were derived. It also provides an indication of the variances in costs compared to the previous year.

	Unit	***2007 Cost***	***2008 Cost***	***Reference***
Processing	$/t	37.27	45.7	2009 Budget
Mining Services (excluding contractor costs)	$/t	5.54	9.8	Mining Performance 2008.xls
Mining Contractor	$/t	15.4	14.9	Schedule Of Rates
Geology Services	$/t	6.32	7.0	Dec08 Monthly_KPI's.xls
Administration	$/t	9.17	9.2	Dec08 Monthly_KPI's.xls

Table 3 Fixed Costs

The increase in mining services costs can be attributed to an increase in numbers of professional staff employed within the mining services department and the associated travel, accommodation, messing and software costs incurred by such. The increase in the processing cost per tonne is related to increasing consumables, power and crushing costs. There were no significant variances in geology and admin costs. Mining Contractor fixed costs are composed of management, administration, plant and equipment charges and are based on the SOR values together with manning and fleet levels as at the 31st December 2008. There have been no significant changes to these fixed costs since the previous Ore Reserve Estimate.

Variable Costs

Variable costs have been based on mining KPI's in association with the mining contractor SOR and Progress Claims for the calendar year 2008. The variable costs and / or cost multipliers used in the economic evaluation are provided in Table 4.

Type	***Unit***	***Cost***	***Reference***
Jumbo Development (5x5.5)	$/m	1632	Schedule Of Rates
Jumbo Ground Support (5x5.5)	$/m	808	Ground Support Mining Instructions
Jumbo Development (5x5)	$/m	1632	Schedule Of Rates
Jumbo Development (5x5)	$/m	750	Ground Support Mining Instructions
Jumbo Development (4.5x4.8)	$/m	1484	Schedule Of Rates
Jumbo Ground Support (4.5x4.8)	$/m	427	Ground Support Mining Instructions
Jumbo Development (4.1x3.8)	$/m	1351	Schedule Of Rates
Jumbo Ground Support (4.1x3.8)	$/m	427	Ground Support Mining Instructions
Air Leg Development (2.5x2.5m)	$/m	1050	Schedule Of Rates
Air Leg Ground Support (2.5x2.5m)	$/m	180	Ground Support Mining Instructions
Development Wet Blasting Factor	% of cost/m advance	2.54	2008 Progress Claims
Development stripping	m^3/m advance	1.9	2008 Progress Claims
Sludge Drilling	m/ stope t	0.015	Mining Performance 2008
Diamond Drilling	m/stope t	0.104	Advice from Intrepid Geo Dept
Stope Production Drilling UZ	Stope t/drill m	5.39	2008 Progress Claims Dec08 Monthly_KPI's.xls
Stope Production Charging UZ	Stope t/charge m	6.47	2008 Progress Claims Dec08 Monthly_KPI's.xls
Stope Production Drilling non-UZ	Stope t/drill m	3.53	2008 Progress Claims Dec08 Monthly_KPI's.xls
Stope Production Charging non-UZ	Stope t/charge m	4.24	2008 Progress Claims Dec08 Monthly_KPI's.xls
Stope Ground Support	Cable bolt m/stope t	0.006	2008 Progress Claims
Airleg development for non-airleg stoping	m/stope t	0.0028	2008 Progress Claims Dec08 Monthly_KPI's.xls
Airleg stoping	$/m3	118.5	Schedule of Rates
Airleg Bolting	bolts / m3	0.197	2008 Progress Claims
Airleg Meshing	m2 / m3	0.013	2008 Progress Claims
Airleg Scraping	hours / stope t	0.054	2008 Progress Claims
Remote Bogging	% of stope t	62.3	2008 Progress Claims
Long Tram UZ	% of total t	16.9	2008 Progress Claims
Long Tram non-UZ	% of total t	55.9	2006 Progress Claims
Development Over Break on Wall	% development t	17	2008 Monthly reports

Table 4 Variable Costs

Mine Development

Ground support development costs were calculated on a cost per meter advance basis, based on ground support mining instructions (Appendix C). Development stripping costs were based on stripping tonnes claimed per development meter for the 2008 progress claims. Wet blasting costs have also been calculated as a percentage of the development advance cost from 2008 Progress claims.

Stope Definition

Underground diamond drilling and sludge drilling is used to define stope boundaries. The sludge drilling cost has been calculated based on historical drill meters per stope tonne, while the cost of diamond drilling is based on a recommendation from the Intrepid Senior Mine Site Geologist.

Stope Production

Stope production drilling and charging costs were calculated based on a drill or charge meter per stope tonne, derived from historical mining performance for the 2008 calendar year. Airleg development for non-airleg stoping (e.g. remote cuddies, escapeways) has been determined as a metre per non-airleg stope tonne from 2008 Progress claims. Remote bogging is charged on a per tonnage basis, and is derived from determining a percentage split of remote to conventional bogging for the year from progress claims, and applying the remote surcharge accordingly. A similar method was used to determine the percentage of long haul (>160m) bogged tonnes, on which a surcharge applies. The stope tonnes per drill and charge metre have been separated due to the significant differences in stope geometries between the Upper Zone stopes and all others. Likewise, the long tram figures have been separated in the same manner due to the decline being closer to the Upper Zone stopes. Stope ground support costs (predominantly brow cable bolting) are calculated based on 2008 calendar year progress claims and where deemed necessary allocated on a meter per stope tonnage basis. Airleg ground support costs (bolting and meshing) are similarly calculated and assigned on a units per volume basis. Provision for scraping of airleg stopes has been assigned on a hours per stope tonne basis based on a more limited dataset of 4 months of airleg scraping data available (November 2008 to February 2009 inclusive).

Rise and Fall

Rise and Fall is assigned on a percentage cost basis to all variable costs associated with the main project service contracts. Intrepid advise that current Rise and Fall of 25.3% is applicable to the mining contract variable rates and that provision relating to the crushing and camp service contracts are included for in their respective rates.

Cut Off Grade

A design cut off grade matrix was developed based on a mill throughput of 338,000tpa and is provided in Table 5. The fixed and variable costs described in Section 0 were used for the development of the cut off grade matrix. Capital development was based on decline development for 2008, while lateral development was based on physicals for 2008 level development quantities. Provision for Airleg mining block delineation is also included as this method is more commonly used for narrower and shallower dipping regions of the deposit not amenable to bulk mining methods.

	$/t	New Mining Block - Airleg	Block Developed - Airleg	New Mining Block - Longhole	Block Developed - Longhole	Stope Blasted	Ore in Surface SP
Geological Definition	12.1	12.1		12.1			
Capital Development	11.3	11.3		11.3			
Lateral Development	13.7	13.7		13.7			
Stope Drill & Blast & Support - AL	62.4	62.4	62.4				
Stope Drill & Blast & Support - LH	13.5			13.5	13.5		
Stope Load & Haul	16.3	16.3	16.3	16.3	16.3	16.3	
Mining Contractor Fixed	14.9	14.9	14.9	14.9	14.9	14.9	
Mining Services Fixed	9.8	9.8	9.8	9.8	9.8	9.8	
Geology Services Fixed	7.0	7.0	7.0	7.0	7.0	7.0	
Processing	45.7	45.7	45.7	45.7	45.7	45.7	45.7
Administration	9.2	9.2	9.2	9.2	9.2	9.2	9.2
Total Cost		202.3	165.3	153.4	116.4	102.9	54.9
Provision - Gov Royalty		5.1	4.1	3.8	2.9	2.6	1.4
Provision - Corporate Hurdle Rate		0.0	0.0	0.0	0.0	0.0	0.0
Total Cost (Incl Provisions)		207.4	169.5	157.2	119.3	105.5	56.3
Breakeven COG (No Provisions) g/t		6.28	5.13	4.76	3.61	3.20	1.71
Breakeven COG (Incl Provisions) g/t		6.44	5.26	4.88	3.70	3.28	1.75

Table 5 Cut Off Grade Matrix

The resultant cut-off-grades were used in subsequent stope design using Vulcan Mine Planning software, and interrogation of the Resource block model in Gemcom Surpac software.

Metallurgy

Method

The Paulsens gold mine utilises a CIL circuit for the extraction of gold. The gold processing facility at Paulsens consists of the following fixed plant circuit operations:

Crushing and Screening

The crushing plant is a three-stage facility that treats up to 150 t/h of minus 550 mm Run-Of-Mine (ROM) ore, to produce a crushed product P_{80} of 8 mm. The crushing plant consists of a primary jaw crusher and secondary and tertiary cone crushers. Oversize from an inclined triple deck vibration screen (apertures 40 mm, 20mm & 8mm) reports to the representative cone crusher, and undersize reports to the Fine Ore storage Bin (FOB), or the on-ground Fine Ore Stockpile (FOS).

Ore Storage and Reclaim

Fine ore product from the crushing plant discharges into the fine ore bin, which provides 12 hours surge capacity to the milling circuit. The ore is reclaimed from the fine ore bin via a slot feeder and variable speed belt feeder conveyor

Grinding and Classification

The grinding and classification facility operates 24 hours a day, 365 days a year, to mill 338 000 tpa. Grinding is performed by a 3.51m in diameter by 5.63m long (inside shell) rubber lined overflow ball mill, equipped with a 1,000 kW motor. A cluster of four hydro cyclones (three operation, 1 standby) are used to classify the slurry to P_{80} of 75 um.

Leach feed thickening

Underflow from the cyclones gravitates to a 7.0m diameter 'High Rate' thickener fitted with an auto-dilution feed well. Diluted flocculent is added to the feed well through sparges. Thickener overflow flows by gravity back to the process water tank. Thickener underflow, at 53 % solids w/w, gravitates into the leach feed pump hopper and is pumped to the first of seven CIL tanks.

Leaching and Adsorption (Carbon-In-Leach)

The CIL circuit consists of 7 x 150m^3 tanks operating in series to give a total residence time of 19.3 hours. Each tank has dual open impellor agitators with the first tank being dedicated to pre-oxidation and the other 6 remaining tanks have an Intertank screen.

Elution and gold recovery

The elution and gold recovery circuit has the capacity to operate 7 days per week, treating 2.0 tonne batches of carbon. Carbon, loaded to a minimum 3,500 g/t gold, is recovered from CIL tanks, screened and washed to remove the slurry, and directed into the cyanide wash column. The elution circuit comprises of two columns. In the first column a cold cyanide solution wash is carried out to remove copper adsorbed onto the carbon, followed by a hot acid wash to remove nickel and other deleterious elements. The carbon is then pneumatically transferred to the second column, where elutriation takes place. The first column is constructed of mild steel and is butyl rubber lined. The second column is constructed from 304 stainless steel. The total strip cycle will take 13 hours, Wash cycles 7.5 hours using 36 m3 of raw water, whilst the elution cycle uses 16.7 m3 of fresh water and takes 5.5 hours.

Pregnant solution is discharged from the elution column, via the eluate filters and recovery heat exchanger, to the electrolyte tank. The solution is pumped from this tank via the electrolyte pump to two 600 mm by 600 mm, electrowinning cells operating in parallel. Cell discharge gravitates back to the electrolyte tank. Electrical current is supplied to the electrowinning cells by two dedicated rectifiers (6V, 1,000 A). The mild steel wool cathodes containing the electrowon doré are removed from the electrowinning cells placed on trays and calcined at 700°C. The sludge in the cells is removed via the drains in the bottom of the cells, and collected in buckets. It is then decanted to remove solution prior to calcining. The calcine is mixed with fluxes, and smelted in the gas fired tilting bullion furnace with A100 crucible capacity. The bullion is stored in the goldroom safe.

Tailings Disposal

Slurry from the last CIL tank gravity flows via the screen feed box to the carbon safety screen. The carbon safety screen is fitted with 750 micron aperture woven wire mesh. Carbon safety screen oversize reports, via a chute, to a fine carbon waste bin. Screen undersize reports to the tailings pump hopper at 50% solids, and is pumped to the tailings storage facility via the tailings disposal pumps.

Recovery

Figure 10 shows the actual and budget mill gold recovery for 2008. The average mill recovery for 2008 was 93.6% (Dec08 Monthly Kpi's.xls). The budgeted mill recovery of 93.5% has been used in the Ore Reserve estimation for economic evaluation purposes.

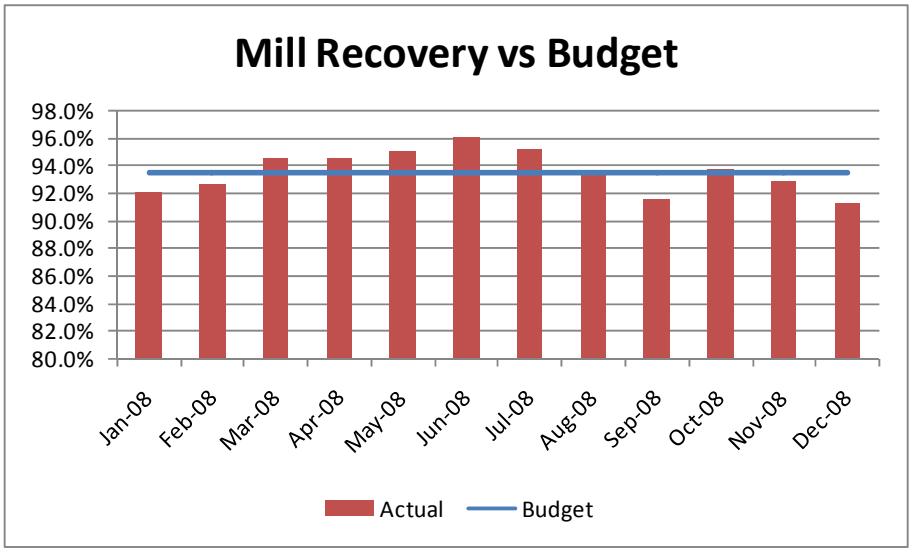


Figure 10 Mill Recovery Actual vs Budget 2008

Stockpiles

Closing stocks as of the 31st of December 2008 consisted of crushed stocks, run of mine ore and low grade run of mine ore.

- Crushed Ore Stocks – 1,607t @ 5.46g/t
- Oversize Ore Stocks – 2,890t @ 7.9g/t
- Run of Mine Stockpiled Ore – 2,360t @ 6.79g/t
- Low Grade Run of Mine Stockpiled Ore – 20,180t @ 1.62g/t

Mining

Mining Method

There are three mining methods in use at Paulsens dependent upon orebody geometry and grade. The majority of stope tonnes are extracted using uphole long hole open stoping. Jumbo stripping is occasionally used for stoping areas with a narrow width and height, such as can be found in some areas of the Lower Zone and Apollo. The Lower West Zone stopes will be extracted by an airleg room and pillar stoping method due to the shallow-dipping, thin nature of the orebody. Additionally, some remnant Apollo stopes may be extracted through Airleg stoping methods where the orebody is too thin to support the higher levels of planned dilution associated with long hole mining methods.

Orebody Geometry

The Paulsens deposit lies within the Wyloo Dome, a Northwest anticlinal and domal structure. Mineralisation occurs on the footwall and hanging wall contacts of a massive quartz vein hosted within Archaean argillites and sandstones. The hanging wall mineralisation is referred to as the Upper Zone while the footwall mineralisation is referred to as the Lower Zone. Post mineralisation faulting and dykes cross cut mineralisation resulting in what is referred to as mineralised blocks/lodes. There are typically three distinct orebody geometries at Paulsens:

Thick shallow dipping as can be found in the Upper Zone stopes;

Thin shallow dipping as found in the Lower Zone West and Upper Lower Zone stopes; and

Thin steeply dipping as found in the Apollo and lower Lower Zone areas.

The orientation of these orebodies is graphically represented in

Figure 11 Isometric View of Resource Domains

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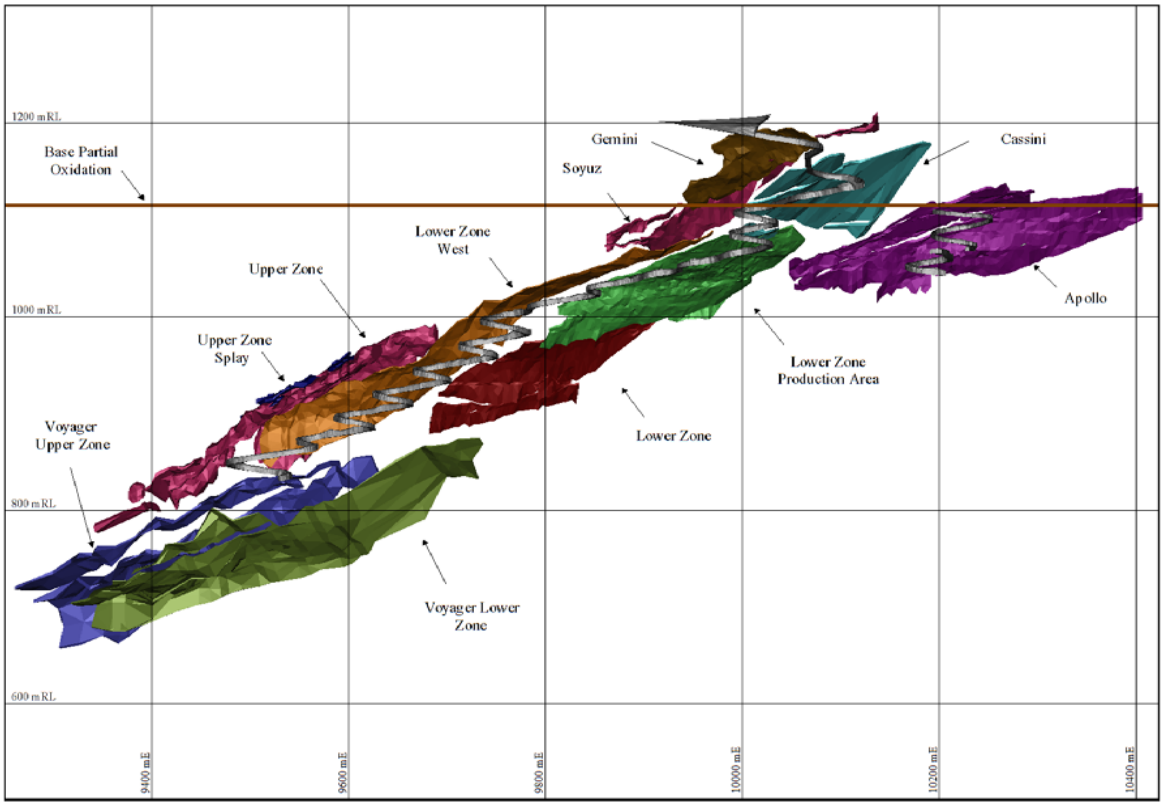


Figure 11 Isometric View of Resource Domains

Hellman & Schofield Pty Ltd (H&S) have prepared the Resource Estimate for the Paulsens Mine for the period ending December 31st 2008. The reader is directed to their report entitled "Resource Estimation for the Paulsens Gold Mine Western Australia" for further information regarding Geological setting and geometry.

Rock Mass Characteristics

There are three rock domains at Paulsens, weathered, quartz and sediments. All stoping and most development to date has been in fresh, unweathered rock. The transition between weathered and fresh rock ranges from 20m to 100m depth below surface. Generally this transition is shallower, when closer to the Melrose fault.

Quartz hosts the gold mineralisation, within the Lower Zone mineralisation extending from the footwall contact, and within the Upper Zone mineralisation extending from the hanging wall contact. The footwall contact is a graphitic shale/siltstone, and blasting practices in the Lower Zone stopes generally break smoothly to this contact. The hanging wall contact principally comprises mudstones and sandstones, therefore a lower hydraulic radius factor has been recommended for stope stability in the Upper Zone compared to the Lower Zone.

The rock strength of the quartz is described as very strong, with an average estimated strength of 125MPa and a range from 50MPa to 250MPa. The rock strength of sediments is described as strong, with an average strength of 35MPa and a range from 30MPa to 50MPa.

RQD from NuStar's 2004 drilling campaign is as follows:

- Quartz = 90 Standard deviation 15
- Sediment = 90 Standard deviation 26

Major structures at Paulsens include post mineralisation dykes, which have been utilised as pillars, and a 50m thick gabbro which has been displaced by the Melrose fault.

Defect surfaces within the quartz domain have been described by Snowdens (2004) as rough/irregular and undulating, with no or hard infill. Within the sediment domain there appears to be little difference between bedding, cleavage and joints, with respect to surface characteristics. All have been described as smooth to rough and undulating to planar, with no infill or slightly altered joint walls.

For the purpose of mine design, Snowdens (2004) assumed a minor principle vertical stress $б_3$, to a depth of 150m below the surface equivalent to overburden, $б_3$=4.0 (MPa), and major and intermediate principle stresses equivalent to two to three times the principle stress. No favourable azimuth was defined.

The Snowdens geotechnical report (2004) assumed groundwater to be minimum, mining has validated this assumption.

Using the above rock mass parameters, a Q system of classification has to date been used for the basis of design. A Q value of 16 with a minimum of 2 and maximum of 100 was calculated for quartz, while a Q value of 4 with a maximum of .3 and minimum of 18 was calculated for sediment. The Q system for support selection and Q values for excavation, in both Quartz and Sediments, are provided in Appendix A. The Q system is more typical of those used in civil engineering and therefore Intrepid adopted these based on Snowden's precedent experience.

Mill Process Rate

The mill process rate has been assumed at 338,000tpa. This is an increase on the previous Reserve estimate of 325,000 tpa. Ore drive development and the current open stoping mining method has been able to support an increasing mill feed over the last three years and as such this is considered a reasonable mill throughput rate.

Mining Method Selection

The mining method selection for the majority of this Ore Reserve estimate has been based on the current site mining method – mechanised long hole open stoping, sequenced top down, retreating from the extremities of the orebody to the cross cut access. The level intervals have previously been a function of orebody dip and thickness. In the estimation of Ore Reserves, the level intervals remained a function of orebody dip and thickness. An airleg room and pillar method has been selected for the Lower Zone West orebody and some remnant Apollo stopes.

Equipment

The equipment presently on site is predicted to remain unchanged for the foreseeable future, assuming mechanical reliability. A list of the Heavy Mechanised Mobile Equipment used underground is provided in Table 6. On site light vehicles include an ambulance, fire truck, coaster bus & 15 Toyota Landcruisers.

Name	***Function***
Tamrock Powerclass Jumbo	Decline, level and inclined development.
Solo 5-5V Long Hole Drill	Production & sludge long hole drilling

2 x Elphinstone R1700G Loaders	Production and development bogging
Toro 151D Loader	Airleg Stope Bogging
Cat AD45 dump truck	Production and development trucking
Cat AD55 dump truck	Production and development trucking
Normet charge up	Production and development charging
Cat IT 28G	Integrated tool carrier for UG mine services
Volvo L50 IT	Integrated tool carrier for surface workshop & stores
Cat 12G	Grader for underground and surface roads
Scania water truck	Surface dust suppression

Table 6 Heavy Mechanised Mobile Equipment

Mine Operating Strategy

Scheduling

Mine scheduling is based on production rates considered achievable with respect to the equipment that is presently on site. Development using one twin boom jumbo with two operators in a three-shift cycle is predicted to advance at no more than 54m/week (Figure 12). Production drilling using the Solo 5-5V long hole rig is expected to achieve 1500m/week (Figure 13). A total ore (development and stoping) trucking target of 6500t/week will provide the required 338,000t of mill feed per annum (Figure 14).

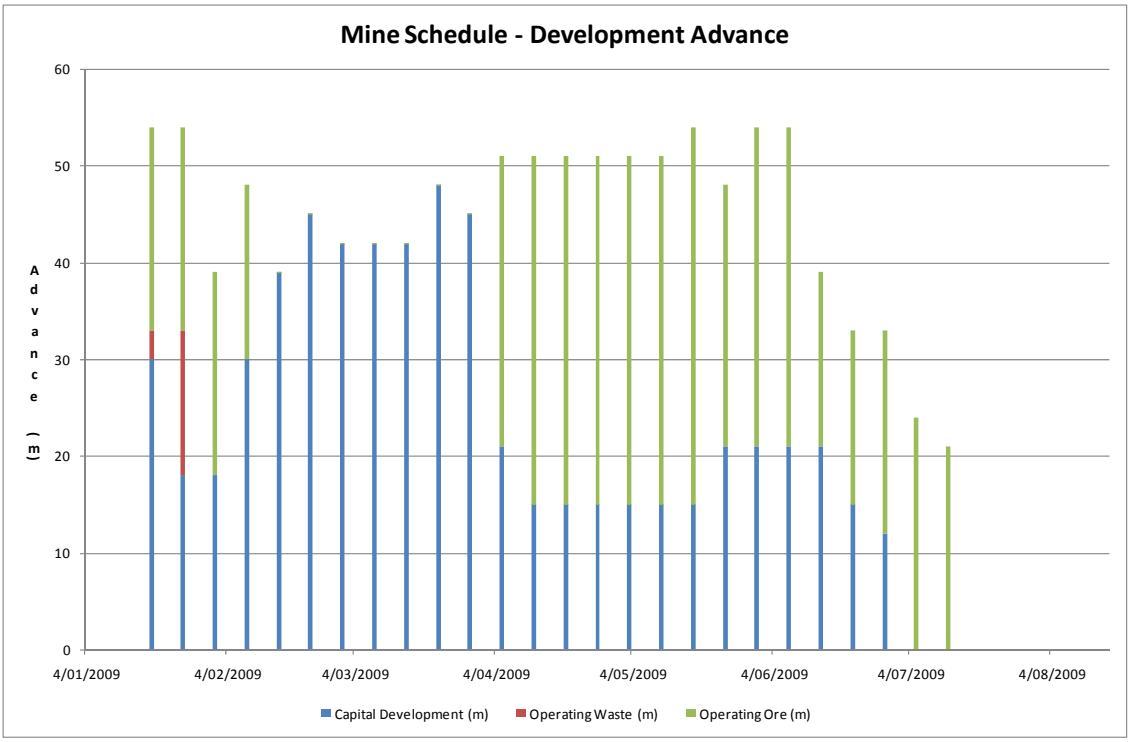


Figure 12 Weekly Development Advance Schedule

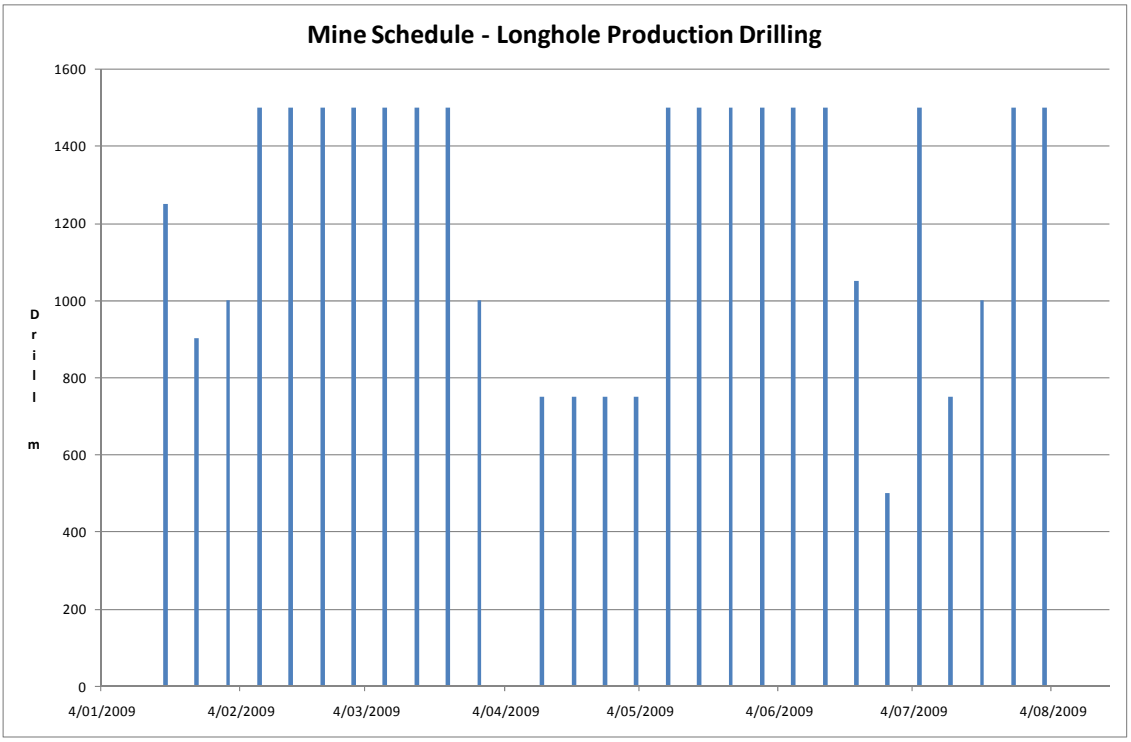


Figure 13 Weekly Production Drill Schedule

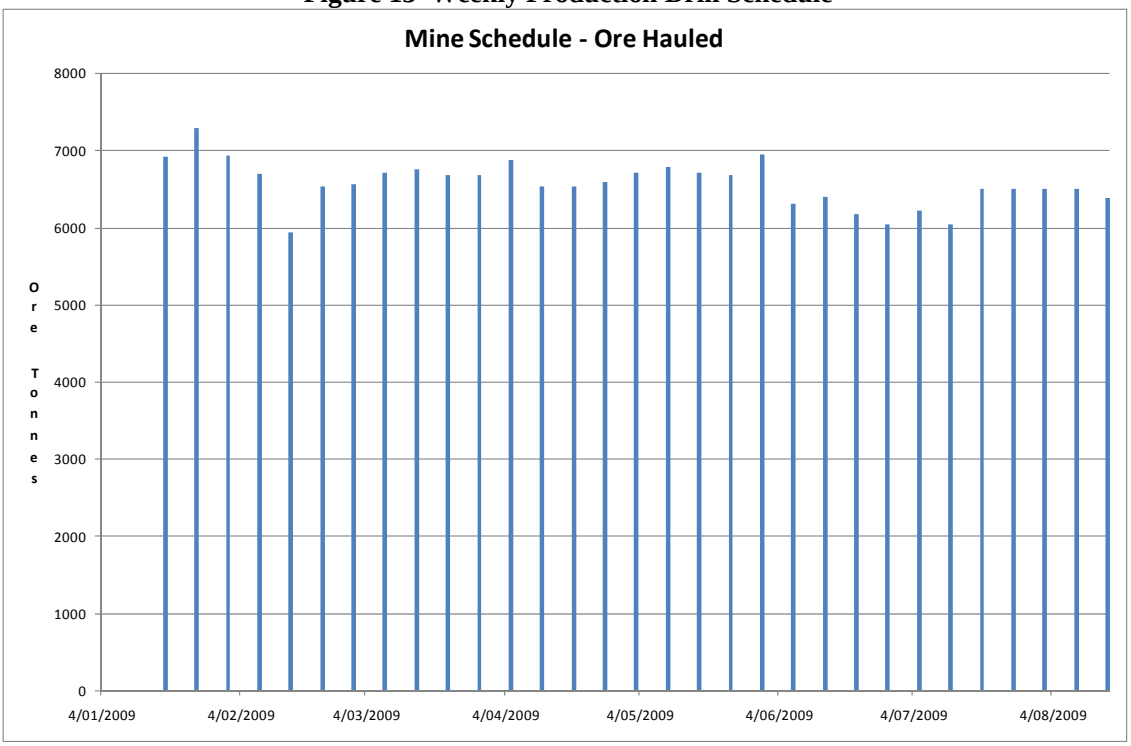


Figure 14 Weekly Ore Hauled Schedule

Extraction Sequence

The proposed extraction sequence for the mine is top-down. The majority of ounces in the Ore Reserve are to be sourced from the Upper Zone stopes. The lower tonnage high-grade stopes of the Apollo, Lower Zone and Lower Zone West are likely to be unable to provide the required mill feed if extracted by themselves, so they are scheduled to be taken concurrently

with the Upper Zone. The Soyuz and Voyager orebodies are likewise to be extracted before the Upper Zone stopes are exhausted. Gemini orebody is the last region scheduled to be mined due to its proximity to the main decline, its location close to surface in more transitional ground and likely production interruptions with haulage from this location.

Remnant Mining

There are high-grade sill pillars yet to be mined in the Lower Zone from the 1060 to the 1028 level. These are on hold to maintain level access whilst extraction of the Lower Zone West stopes continues. A sill pillar also exists along the length of the 1045-Apollo access that will be retreat mined after the Apollo orebody has been depleted.

Mine Design

All mine design was prepared by Intrepid site engineers with subsequent review by Per Scrimshaw.

Geotechnical

No fill has been incorporated in the costing. Fault offsets, dykes, and in-ore pillars will be used for regional ground support.

Stable Stope Dimensions

A 6.3% pillar factor was applied to all stope recoveries in Upper Zone and 11.3% for all other areas. This pillar recovery factor will allow for stable stope dimensions.

Ground Support

Where shotcrete has been recommended by the Q system, Intrepid has generally used mesh in preference. To date there has been no major bagging of the mesh requiring reinstallation.

For the decline (5m wide x 5.5m high) the ground support has incorporated pattern bolting using 7 x2.4m fully grouted galvanised split sets at 1.2m spacing, pinning mesh across the backs and shoulder of the excavation. Spot bolting of the drive sidewalls is specified every second row. Additionally, random spot bolting and meshing has been used in areas of slabbing. At decline intersections 6m fully grouted cable bolts at 2.5m spacing have been used.

Ore drive dimensions are typically 4.1m wide x 3.8m high. Again, pattern bolting has been incorporated into the Intrepid ground support guidelines, using rings of 5 x 1.8m split sets at a spacing of 1.2m, pinning mesh to the backs and shoulders. Side walls are spot bolted, and cable bolting occurs at intersections on 2.5m spacing.

Bolting of 5m x 5m cross cuts and stockpiles is based on a pattern bolting standard of 7 x 2.4m galvanised split sets on 1.2m ring spacing. Galvanised mesh is also pinned across the back and shoulder of the excavation. Spot bolting has been used where required. A standard cable bolting pattern at a spacing of 2.5m has been used at intersections.

These development standards were deemed adequate, and for the purpose of Reserve estimation, the above patterns were used for a variable ground support costing. The additional bolts and mesh used above design for 2008 were factored into the variable ground support cost.

Pillar Design

Rib pillars are used as a means of reducing the maximum allowable stope span. In 2004 Snowdens completed a pre-mining assessment of stope stability based on the stability graph

method. The stability graphs for the Upper and Lower Zones are provided in Appendix B. The resulting stope design criteria are provided in Table 7.

Recommended maximum unsupported stope span dimensions				
Stope wall	Unsupported stope void		Supported stope geometry	
	Hydraulic Radius (m)	Equivalent stope void span (dip x strike)	Radius Factor (m)	Equivalent pillar spacing (dip x strike)
Upper Zone 30°	3.5	10 x 20	8	20 x 20
Upper Zone 45°	4	15 x 17	9	27 x 17
Lower Zone 30°	7	10 x ∞	12	20 x 30
Lower Zone 45°	8	15 x ∞	12.5	27 x 17

Table 7 Snowden 2004 recommended maximum unsupported stope span dimensions

These initial recommendations have proven conservative. The Upper Zone 1040B3 was mined to 55m x 15m resulting in a hydraulic radius of 6m, this stope has experienced minor post-mining slabbing. In addition the 1025B7 has been mined and broke through to the 1040B7, the void resulting in a hydraulic radius of 7.5m. Snowden's current recommended hydraulic radius for the Upper Zone is 6.5m

Snowden also performed an assessment of extraction ratio. This assessment was based on a regular pillar interval and an average mining depth of 150m. Dip and strike spans used in the assessment were at the maximum recommended dimensions. Pillar loading was based on tributary area theory. Pillar strength, based on quartz rock and insitu stress at 150m below surface, at 30 degrees and 45 degrees, was estimated at 6MPa and 8MPa respectively. Pillar geometry is expressed in terms of minimum width, where for rib pillars the recommended length is its dimension along strike of the orebody. Its height is defined by the stope width, its length on dip is typically defined by ore drive spacing, and for the purposes of the assessment was kept at a constant 20m. A criterion of 25% to 50% probability of failure was selected as the design level. Table 8 illustrates the recommended rib pillar dimensions and extraction ratios based on this study.

Recommended minimum rib pillar dimensions										
Stope width	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Upper Zone 30° (20m x 20m stope span (stope void + ore drives), 10m pillar dip length)										
Pillar width (m)	3	3.75	4.75	5.25	5.75	6.25	7.0	7.25	7.5	7.75
Extraction ratio (%)^	91	89	87	86	85	84	83	82	82	81
Upper Zone 45° (27m x 17m stope span (stope void + ore drives), 10m pillar dip length)										
Pillar width (m)	3.75	5.25	6.5	7.75	8.75	9.75	10.5	11.25	11.75	12.25
Extraction ratio (%)	91	89	87	85	84	83	82	81	81	80
Lower Zone 30° (20m x 30m stope span (stope void + ore drives), 10m pillar dip length)										
Pillar width (m)	3.5	4.75	5.75	6.5	7.25	7.75	8.25	8.75	9.25	9.75
Extraction ratio (%)	93	91	89	88	87	86	86	85	84	84
Lower Zone 45° (27m x 24m stope span (stope void + ore drives), 10m pillar dip length)										
Pillar width (m)	4.25	6.5	8	9.25	10.5	11.5	12.25	13	13.5	14
Extraction ratio (%)	93	90	88	87	86	85	84	83	83	82
^ Extraction ratio is the overall stope void and ore drive extraction expressed as a percentage of the total stope unit area (void plus pillar areas)										

Table 8 Snowden 2004 recommended minimum rib pillar dimensions and extraction ratio.

Historically, Paulsens stoping has exceeded these design criteria and the placement of pillars has been chosen in low grade/waste or dykes, resulting in an extraction ratio greater than estimated for Upper Zone, and at the higher end of the estimated range for Lower Zone.

Mine Design

Mine design for this Ore Reserve estimate has been performed by Intrepid site engineers and subsequently reviewed by Per Scrimshaw prior to conducting economic evaluation and Ore Reserve estimation. All mine design has been revised based upon the H&S Resource model grade and geological reinterpretation / re-estimation. The major revisions in this Resource Model from a mine planning perspective are the reclassification of Resource material into lower confidence categories and the inclusion of geological interpretation based upon strike drive development and sludge drilling.

All major mine infrastructure to date has been located in the footwall sediments. The remaining decline and access development for Reserve extraction is now limited to the lower levels of the Upper Zone and Voyager orebodies. The decline is oriented to maintain minimum cross cut access to the Upper Zone, as the Upper Zone contains the majority of ounces. A turning radius of 18m was used to ensure a suitable horizontal offset due to the shallow dipping nature of the Upper Zone orebody. The decline maintains a minimum 25m pillar between it and current or future stoping activities. A single cross cut has been located at the western turn of the main decline. A 16m stockpile has been designed for each major level, with access from the decline turnout.

Access drives from the main decline have been designed for strike drive and extraction levels for the Gemini satellite orebody with minimal supporting infrastructure due to the small tonnages and small production timeline associated with these stopes.

Escapeways have been designed at intermediate level intervals.

Ventilation is currently exhausted through the Upper Zone stopes, and then up a vent rise from the 1080 level. The same means of ventilation has been chosen for the mine design for this Reserve estimation.

Safety

All mine design has been undertaken in such a way as to ensure safe extraction of ore. Stoping and development has been designed for non-entry methods (except for the air-leg room and pillar mining areas). Geotechnical recommendations have been incorporated into all proposed designs, and in consideration of required primary ventilation quantities.

Ventilation

A 200Kw axial surface fan with variable speed drive is utilised for primary ventilation. The full capacity of the fan is 120m^3 at 750Pa. Air is drawn down the main decline, and exhausted through stopes linked to a 143m long 3.5m diameter raise bore. Table 9 shows that there is sufficient capacity within the primary ventilation circuit to concurrently operate all underground diesel equipment. For the purpose of Reserve estimation the same primary ventilation circuit was assumed.

Secondary ventilation is drawn from the main decline with the use of twin stage 50kw and 55kw fans.

Diesel Equipment	***Power (KW)***	***Requirement (m^3)***
2 x Elphinstone R1700G Loaders	2 x 231	2 x 11.6
Toro 151D Loader	72	3.6
Cat AD45 dump truck	362	18.1
Cat AD55 dump truck	485	24.3
Normet charge up	97	4.9
Cat IT 28G	150	7.5
Volvo L50 IT	150	7.5
Cat 12G	120	6
Total	***1898***	***95.1***

Table 9 Ventilation Requirements

Minimum Mining Width

Stopes have previously been designed and successfully extracted with widths as low as 1m in the Lower Zone. Revised mine design parameters have used a minimum mining width of 2m and this minimum mining width is used for this Reserve Estimate.

Reconciliation of all stopes mined to date was performed using survey asbuilts and sectional analysis, to determine actual average mining width. These are outlined in Table 10 for all stopes currently incorporated in the database. This work validates the minimum mining width assumptions used for Reserve stope design, utilising current mining practises and equipment.

Sublevel Interval and Stope Wall Angle

Table 10 also provides the average stope footwall angle mined to date in the Lower Zone. It had been common practice for drill holes to be 20m long in both the Upper and Lower Zones, and for longhole stopes to have footwall angles as low as 26 degrees. New design guidelines now specify a minimum footwall angle of 45 degrees and there is some evidence to suggest this may be contributing to increased stope mass recoveries in recent Upper Zone stoping. There are no minimum dip criteria for Airleg stopes as broken material is typically scraped to the level for extraction. Longhole rings are typically drilled with 1.5m burden, and fired three at a time leading from the hanging wall holes, with the foot wall holes the last fired.

For the purpose of Reserve Estimation the sublevel interval varied between each orebody and was chosen based on the accuracy of the long hole drilling equipment and the dip and width

of the orebody. A maximum drill hole length of 15m is typically used when designing levels for Reserve estimation and a minimum footwall angle of 45 degrees used. Typical sublevel intervals (floor to floor) for the respective zones are summarised below

- Apollo – 15m
- Upper Zone – 20m
- Lower Zone – 10m
- Soyuz – 12m
- Gemini – 15m
- Voyager – 15m

In all cases sublevel intervals may be varied at the peripheries of economic grade in order to maximise economic return from stope material by excluding sub grade diluting material and minimising access development expenditure.

Stope_Id	Level	Zone	Method	Avg Width (m)	Avg Dip (°)
1028B2	1028	LZ	Longhole	2.0	25.1
1028B4	1028	LZ	Longhole	3.0	46.1
1028B6	1028	LZ	Longhole	4.1	31.0
1037B14	1037	LZ	Longhole	1.4	25.6
1037B12	1037	LZ	Longhole	3.0	25.6
1037B6	1037	LZ	Longhole	2.8	30.4
1052B10A	1052	LZ	Longhole	3.0	31.3
1052B6	1052	LZ	Longhole	2.1	26.2
1052B8	1052	LZ	Longhole	3.3	38.3
1060B2	1060	LZ	Longhole	1.5	65.5
1067B4	1067	LZ	Longhole	3.1	31.0
1067B6	1067	LZ	Longhole	1.1	47.0
1067B7	1067	LZ	Longhole	1.4	27.5
1067B8	1067	LZ	Longhole	1.7	33.0
1075B2	1075	LZ	Longhole	1.7	23.0
1075B4	1075	LZ	Longhole	3.5	43.0
1075B5	1075	LZ	Longhole	1.2	33.0
1075B6	1075	LZ	Longhole	3.3	47.0
1075B7	1075	LZ	Longhole	1.0	33.0
1060B6a	1060	LZ	Jumbo	2.0	29.0
1060B8	1060	LZ	Jumbo	1.7	23.0
1060B10	1060	LZ	Jumbo	2.6	24.0
1019B2	1019	LZ	Longhole	4.1	32.0
1019B4	1019	LZ	Longhole	2.4	50.0
1025B7	1023	UZ	Longhole	6.2	45.0
1037B8	1037	LZ	Longhole	2.4	21.0
1037B10	1037	LZ	Longhole	2.1	22.0
1040B1a	1040	UZ	Longhole	5.5	38.0
1040B1aa	1040	UZ	Longhole	4.2	40.0
1040B1b	1040	UZ	Longhole	10.3	47.0
1040B1c	1040	UZ	Longhole	6.4	37.0
1040B1d	1040	UZ	Longhole	9.7	52.0
1040B5	1040	UZ	Longhole	5.5	47.0
1045B8	1045	LZ	Jumbo	4.1	29.0
1060B7_ext	1060	UZ	Longhole	11.9	24.0
1060B41	1060	UZ	Longhole	11.3	58.0
1060B42	1060	UZ	Longhole	19.5	38.0
1080B5	1080	UZ	Longhole	5.3	20.0
1010B2	1010	LZ	Longhole	2.9	36.0
1010B4	1010	LZ	Longhole	2.5	48.0
1012B2	1010	LZ	Longhole	1.6	71.0
0999_B2	0997	LZ	Longhole	4.6	85.0
0999_B4	0997	LZ	Jumbo	3.3	53.0
0989_200_2	0989	LZ	Longhole	3.2	40.0
0971_200_2	0971	UZ	Longhole	8.7	58.0
0983_100_3	0983	UZ	Longhole	13.0	70.0
0983_110_1	0983	UZ	Longhole	10.3	61.0
0983_110_3	0983	UZ	Longhole	6.7	62.0
0995_200_4	0995	UZ	Longhole	7.8	58.0
0995_310_1	0995	UZ	Longhole	22.6	90.0
1000_700_3	1000	UZ	Longhole	28.7	35.0
1046_200_1	1046	AP	Longhole	3.5	64.0
1068_200_1	1068	AP	Longhole	4.4	59.0
1081_200_1	1081	AP	Longhole	3.7	55.0
1081_200_2	1081	AP	Longhole	2.3	53.0
1081_200_3	1081	AP	Longhole	2.0	55.5
1081_200_4	1081	AP	Longhole	2.6	63.5
0895_100_1	895	UZ	Longhole	10.3	37.5
0910_200_1	910	UZ	Longhole	7.5	61.0
0910_200_2	910	UZ	Longhole	8.2	57.5
0919_200_1	919	UZ	Longhole	9.3	62.0
0919_200_2	919	UZ	Longhole	9.7	53.5
0919_300_1	919	UZ	Longhole	11.5	50.5
0919_300_2	919	UZ	Longhole	7.6	34.0
0943_100_1	943	UZ	Longhole	13.8	50.5
0943_200_1	943	UZ	Longhole	7.9	59.0
0943_200_2	943	UZ	Longhole	10.6	44.0
0963_100_1	963	UZ	Longhole	8.9	35.0
0963_200_1	963	UZ	Longhole	8.1	50.5
0971_100_1	971	UZ	Longhole	5.9	34.5

Table 10 Stope widths and dip angles mined to date.

Stope Transitions

Stope transitions were made as smooth as possible. During the digitizing phases of creating the stopes, all sections ensured a free face was available for blasting practices. Tie lines were used where required to ensure the correct triangulation face was being created, therefore ensuring correct volume reporting.

Opportunity Material

Opportunity material is referred to as gold mineralization which on its own would not be able to justify capital or level development. Essentially there was minimal opportunity material, as the cost of development could be spread evenly amongst stopes on each level. In the case where access development would be shared by multiple stopes, but a high value stope could carry the entire (or bulk) of the access costs, it may be the case that all development costs have been assigned to that high value stope. This process ensures optimization of material in Reserve, by not penalizing more marginal stopes, when access development costs would be incurred regardless. In all cases an individual stope had to carry its own individual cost requirements (i.e. Ore drive and production costs).

Naming Convention

The naming convention used in Reserve design is provided below (Figure 15). This reflects the current site nomenclature introduced in 2007 and ensures consistency between the Reserve Estimation process and actual site production terminology.

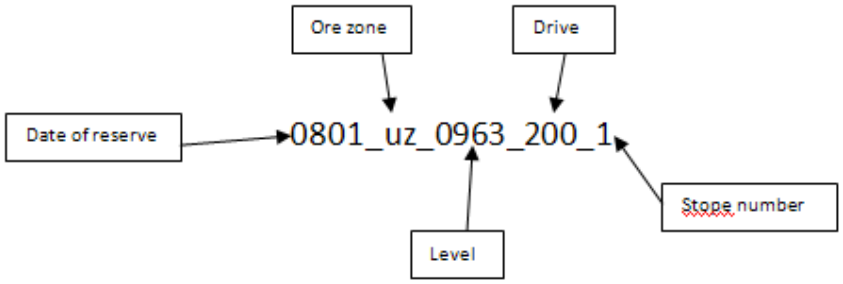


Figure 15 Naming Convention

Ore Zone codes are:

- uz = Upper Zone
- lz = Lower Zone
- lw = Lower Zone West
- ap = Apollo
- cs = Cassini
- gm = Gemini
- sy = Soyuz
- vg = Voyager

The level is the RL of the access drive for that stope.

The drive naming convention is derived from the direction of the drive. A drive in an easterly direction (ie turning right off the access) will have an even number for the first digit. A drive heading to the west will have an odd number. The drives are numbered in ascending order from the decline (so the first drive on the left will be 100, the second 300 and so on). Sub-drives will be identified using the second and third digits, so the first drive turning left

off the 200 will be 210, and the first drive turning right off the 210 will be named 212. This naming convention is illustrated below (Figure 16).

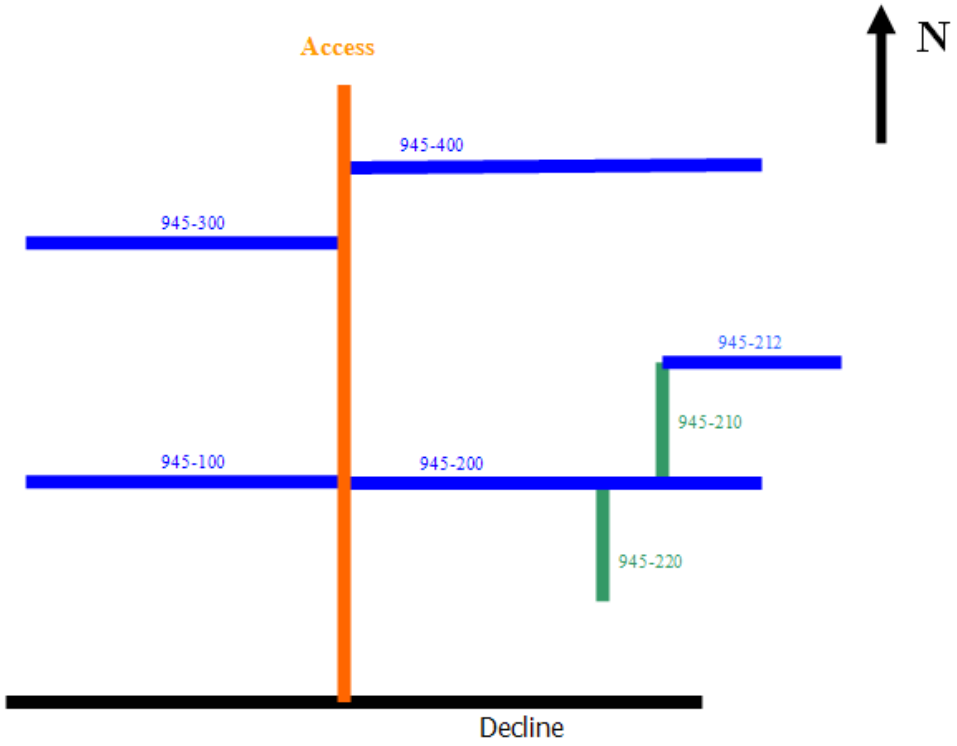


Figure 16 Graphical Illustration of Naming Convention

The stope number is derived from the sequence of stoping in a particular drive, with the furthest stope from the take-off (i.e. the first to be extracted) numbered as one, the second as two, and so on.

Stope Design

Polygons for individual stoping blocks were created on 4m section intervals, digitised perpendicular to the strike of the orebody. This section interval was chosen as being approximately twice the operational ring spacing. At the margins of blocks the interval spacing was reduced to 2m, to ensure maximum recovery of economic material. Material that lay on the stope edge beyond the 2m interval was deemed opportunity material and this material was not included in Reserves. VULCAN trisolations were created from the polygons and cut with the development drives. The resultant trisolation was imported into Gemcom Surpac, and validated for closure and consistency, before reporting against the Resource Block Model for design tonnes and grade.

Development Design

Development design considered the practical capabilities of the equipment used, with allowances within the economic evaluation for stripping turnouts. Total development stripping for 2008 per development meter was calculated at 1.9m^3/m. This stripping cost was applied to all development. Declines and inclines had a minimum turning radius of 18m and maximum and minimum gradients of 1:6.

Drive outlines were based on the current mine practices being:

- Main Decline - 5m wide x 5.5 High
- Access from Decline & Stockpiles - 5m wide x 5 high
- Ore drives and ore drive access – 4.2m wide x 4.2 high
- Airleg rising – 1.5m wide x 1.8m high

Modifying Factors

Mining performance has been assessed for both development and stoping, by comparison of planned development centrelines and stope shapes, to actual survey asbuilt shapes. This procedure has currently been adopted for all stoping areas and a stope is not deemed complete until the comparison has been made. Additions to this dataset are expected to improve both mine performance and Reserve estimation accuracy. Figure 17 below graphically illustrates the sectional analysis undertaken in quantifying historical mining performance.

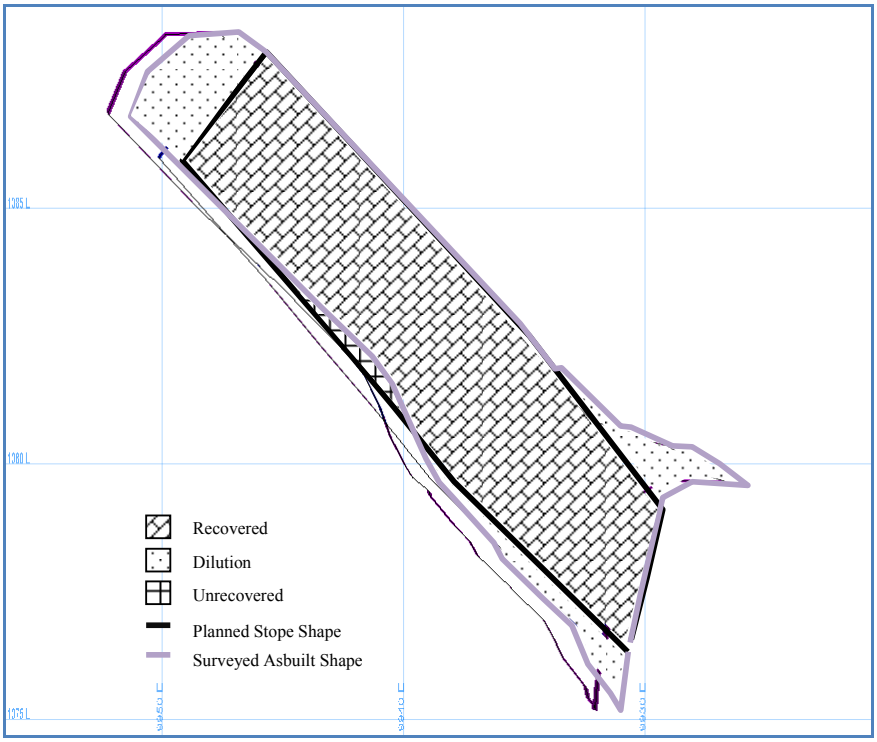


Figure 17 XS Illustrating Dilution and Recovery

Recovered, Dilution and Unrecovered volumes for each stope are calculated by volumetrically assigning fill factors into a regularised block model using both survey asbuilt (CMS) and final drill and blast void shapes. Resource model densities are also stored within this model and used for tonnage determination based on the spatial location of the voids in respect to the density distribution within the model. This approach eliminates triangulation and volume reporting issues associated with small triangle validation errors when cutting or merging (Boolean) almost coincident object boundaries.

Mining Recovery

Historical Mass Recovery

For this Reserve estimate historical stope mass recovery is calculated using the following relationship.

Mining Recovery% = (Mass recovered stope shape)*100/(Mass planned stope shape)

During the 2008 calendar year there was significant addition to the dataset of stopes mined in Upper Zone, the introduction of Apollo stopes into the analysis and no changes to the Lower Zone stopes used.

Based on the stopes contained within the analysis dataset, the average stope mass mining recovery for the Lower Zone to the 31st Dec 2008 remained unchanged at 84%, while the average recovery for the Upper Zone to the 31st Dec 2008 increased to 91% (89% in previous Reserve Estimate). Mass recovery for Apollo stopes has been calculated at 86% however the limited number of stopes within this dataset (six) did not provide sufficient confidence in using this recovery for application in the Reserve Estimation process.

In general stope recovery increases with increasing stope width (Figure 18), however not to the extent that a meaningful relationship can be described.

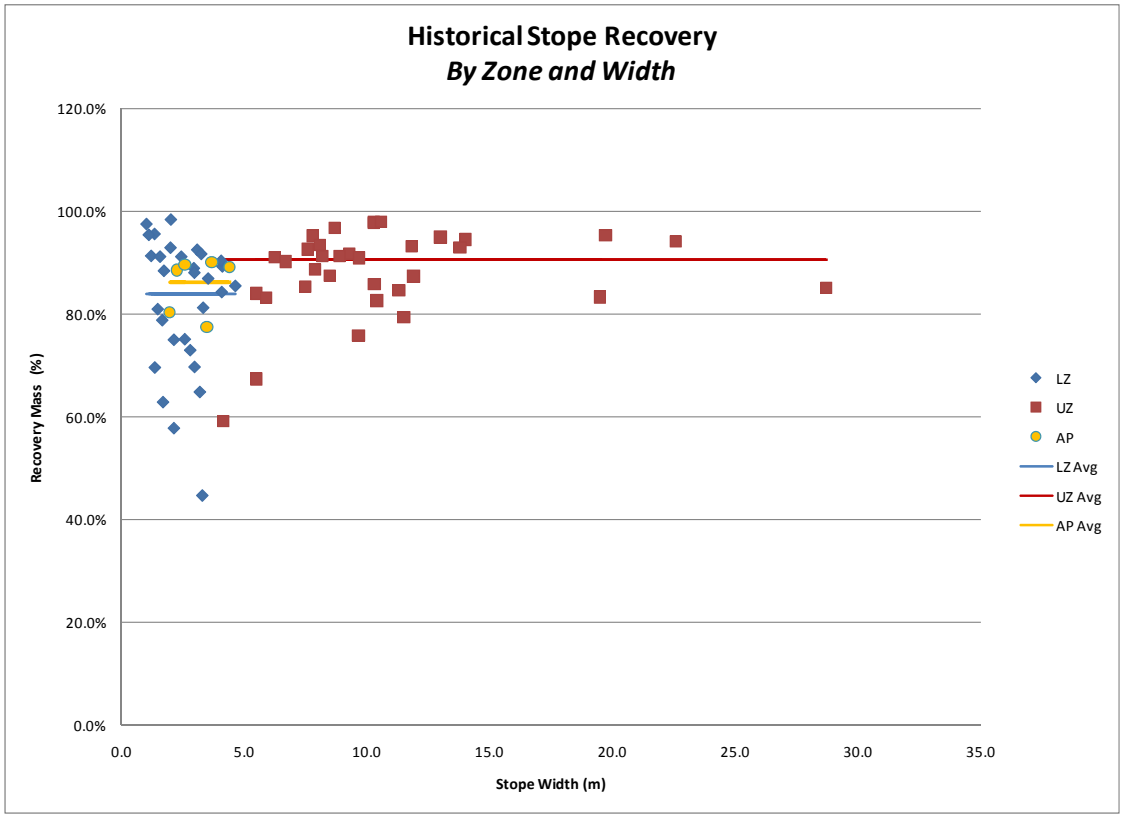


Figure 18 Historic Stope Mass Recovery - All Zones

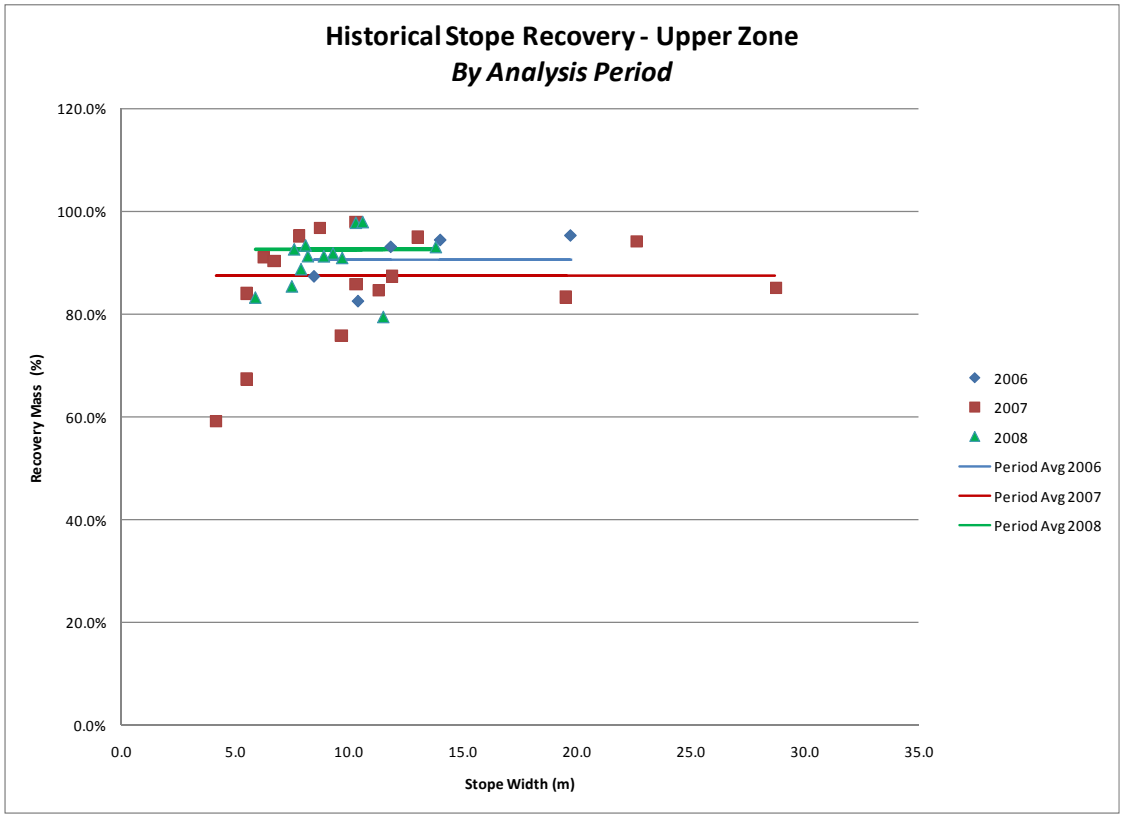


Figure 19 Historic Stope Mass Recovery by Period - Upper Zone

Consideration of the stope mass recovery on a period basis suggests that for Upper Zone stopes, recovery has increased from the previous year's analysis (Figure 19). This could be due to a number of factors including

- Introduction of CMS survey instrumentation and better quality void models from such
- Better drill and blast performance
- Increased stope design footwall angles
- Reducing strike lengths of Upper Zone mineralized domains with increasing depth

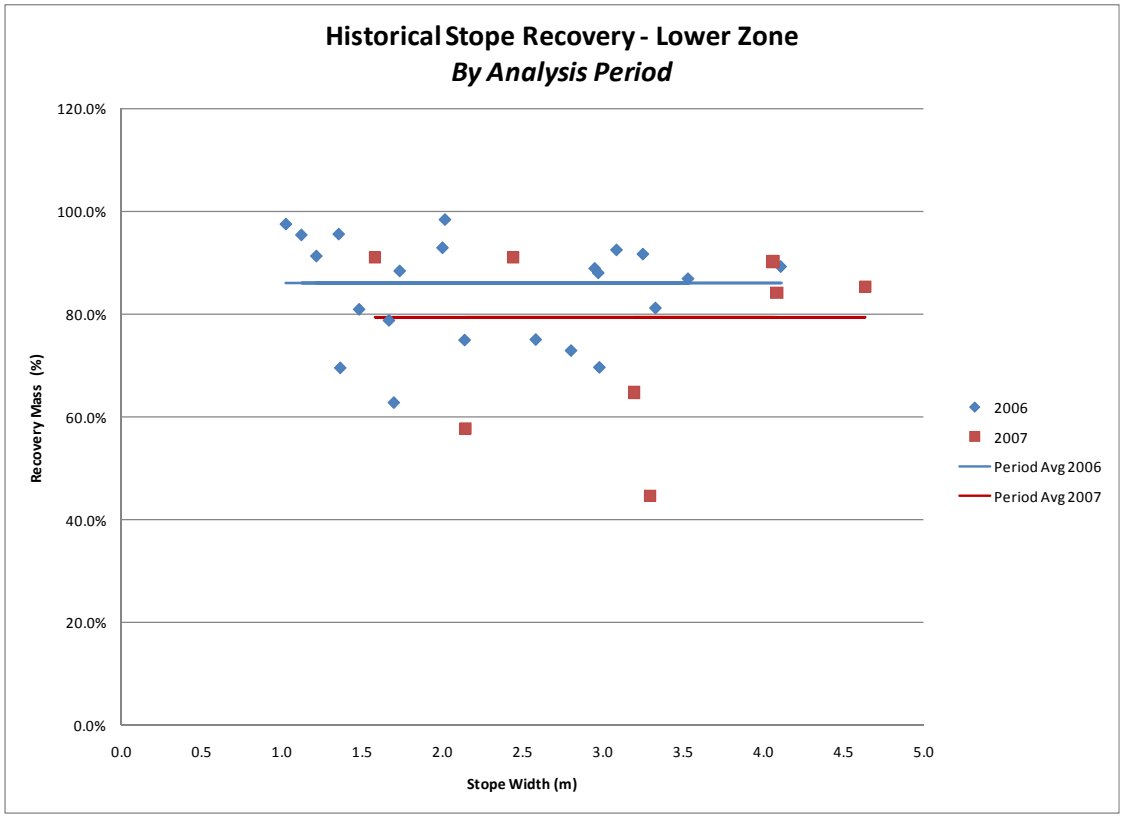


Figure 20 Historic Stope Mass Recovery by Period - Lower Zone

Model Recovery

Based on the increased stope performance data available and using a stope mass based recovery calculation method, actual mine stope recovery has increased from the previous estimation date for Upper Zone stopes only.

Global average stope mass recoveries are applied based on geological zone, as these are directly related to the average stope geometries.

For the purpose of Reserve estimation a stope mining recovery of 91% was used for the Upper Zone, and a stope mining recovery of 84% for the Lower Zone, Apollo and other satellite zones.

Dilution

Historical Dilution

The historic stope dilution is calculated using the following relationship

Mining Dilution = (Mass dilution skin)*100/(Mass planned stope shape)

A plot of stope dilution vs. stope average width is provided in Figure 21. This plot illustrates that, generally, as stope width increases dilution decreases. This can be justified through over break. If the over break on the perimeter of a stope remains constant (i.e. an extremely wide stope experiences the same amount of overbreak on side wall holes as a narrow stope) then the proportion of dilution experienced in wider stopes will be less than the proportion experienced in narrower stopes. Anecdotal information provided by site engineers would

suggest that overbreak is relatively consistent over the section profile of a stope, although insufficient stope sectional reconciliation work exists to confirm this.

As dilution estimation is ideally based on actual stope performance, a line of best fit through the data has been suggested based on the sample population. The previous Reserve Estimate used two dilution overbreak factors, depending upon which geological zone a stope was within. For Upper Zone stoping a 0.75m dilution skin had been applied and for all remaining stopes a 0.5m dilution skin was applied. The additional stopes included in the global dataset (Figure 22) add further support for continued application of these modifying factors in the Reserve estimate.

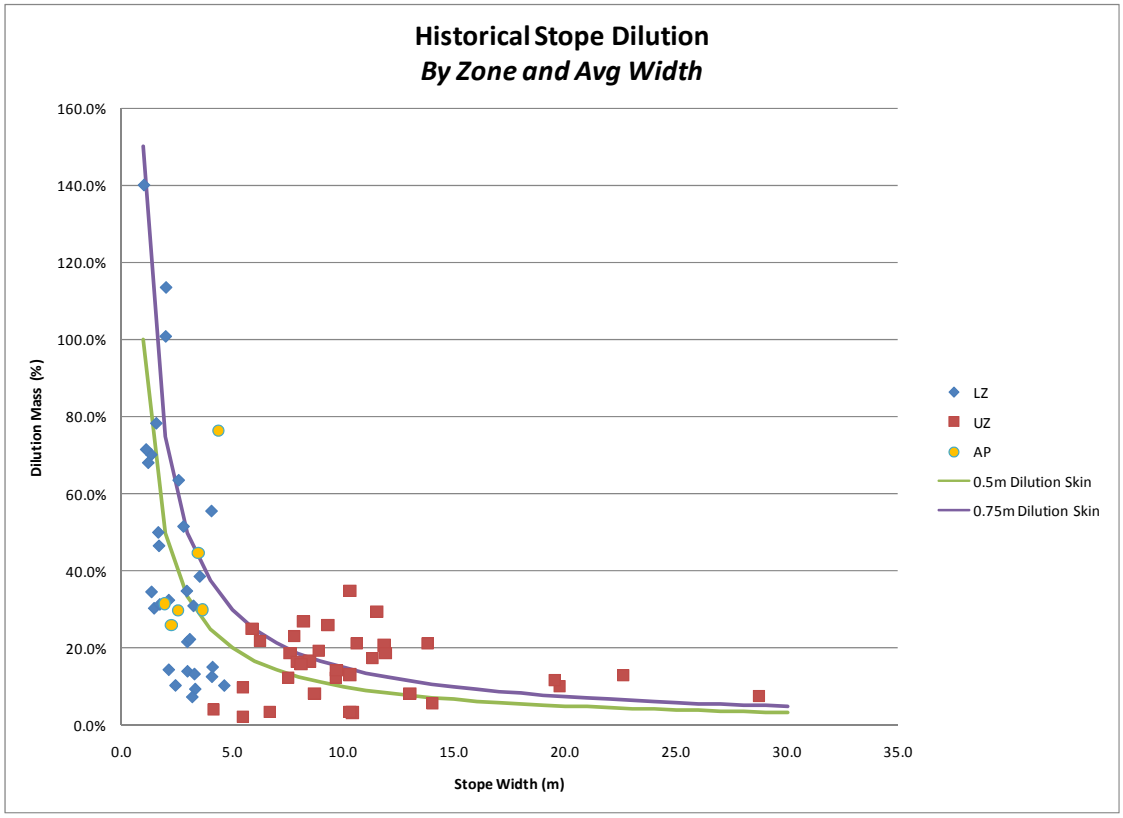


Figure 21 Historic Stope Dilution vs Avg Width

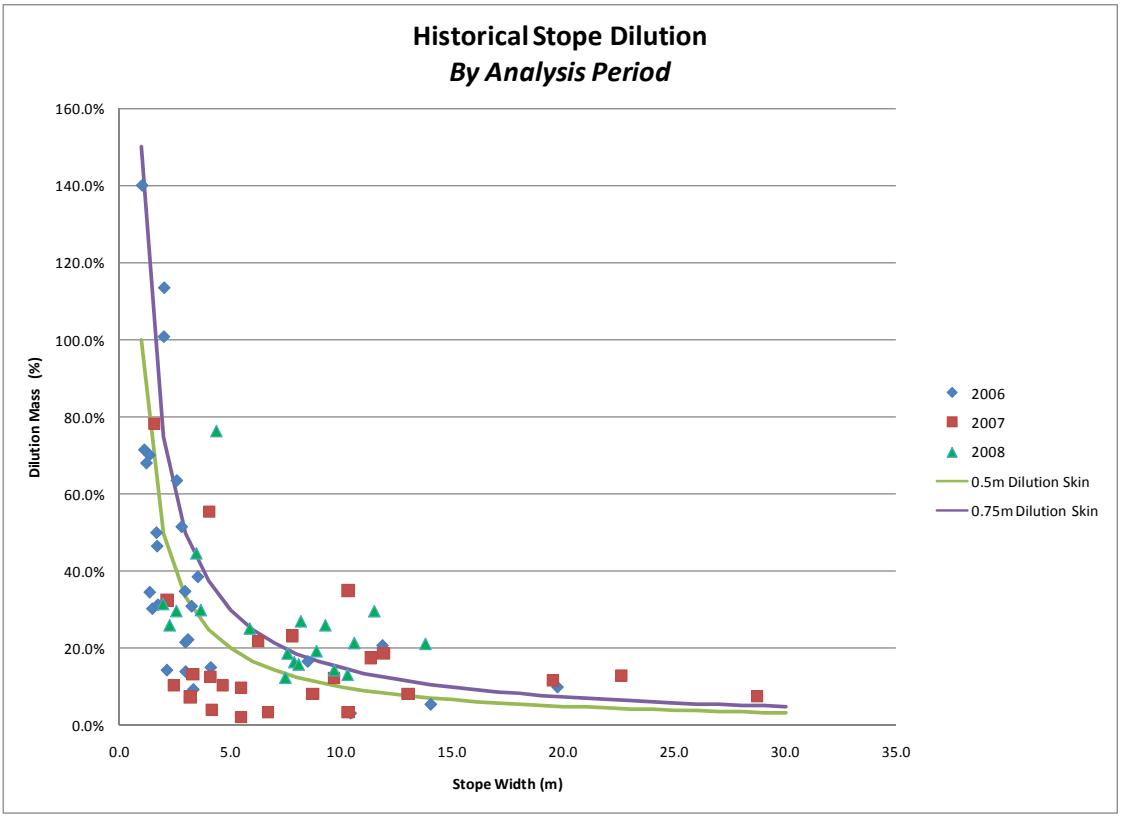


Figure 22 Historic Stope Dilution by Period

Model Dilution

For the purpose of Reserve estimation all mining stopes had a dilution factor applied based on the previously described dilution skin and average stope width, dependent upon their geological zone. This factor is validated through the reconciliation work to date, however should be reviewed as more production data becomes available. This factor justifiably penalises narrow stopes with a higher dilution. Airleg stopes have no unplanned dilution factor applied but do incorporate planned dilution to the minimum mining width should the stope design width be lower than this.

As the block model resolution was not considered fine enough for use in quantifying dilution grade at a 0.5m or 0.75m expansion, a default dilution grade of 0.5 g/t was assigned based on advice from Intrepid Mines Senior Mine Geologist.

Geotechnical recovery

Historical Geotechnical Recovery

Historical geotechnical recovery estimates have been updated for the advancing production stoping blocks in Upper Zone only as this region constitutes the majority of the material mined in the 12 months to December 31st 2008. This entailed plan delineation of completed mining blocks and pillars based on survey asbuilt data. Plan area analysis was then conducted to determine pillar factor percentages for both the Upper and Lower Zones separately. As pillars are preferentially left in barren or dyke domains, care was taken to exclude these areas in the analysis as this would unfavourably skew the factor estimates. This method of analysis assumes relatively uniform dip and width for each of the regions analysed.

Figure 23 and Figure 24 below graphically illustrate the pillars for each zone (pink) and the total delineated mining block (grey).

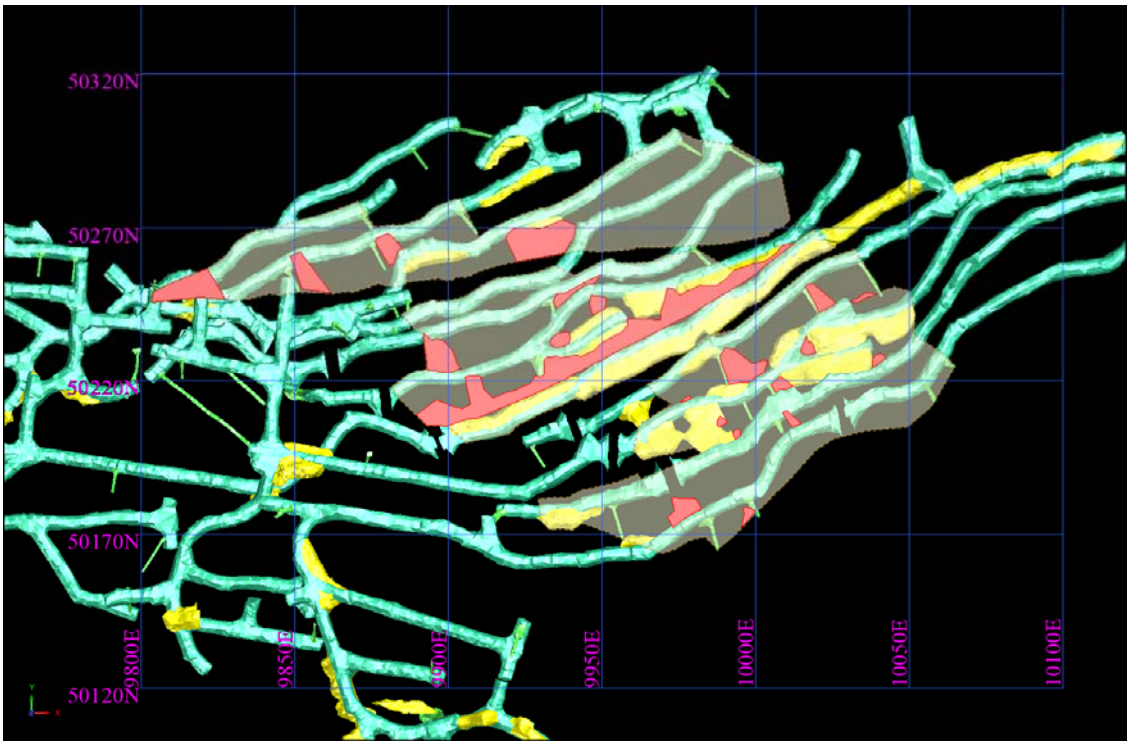


Figure 23 Lower Zone Pillars

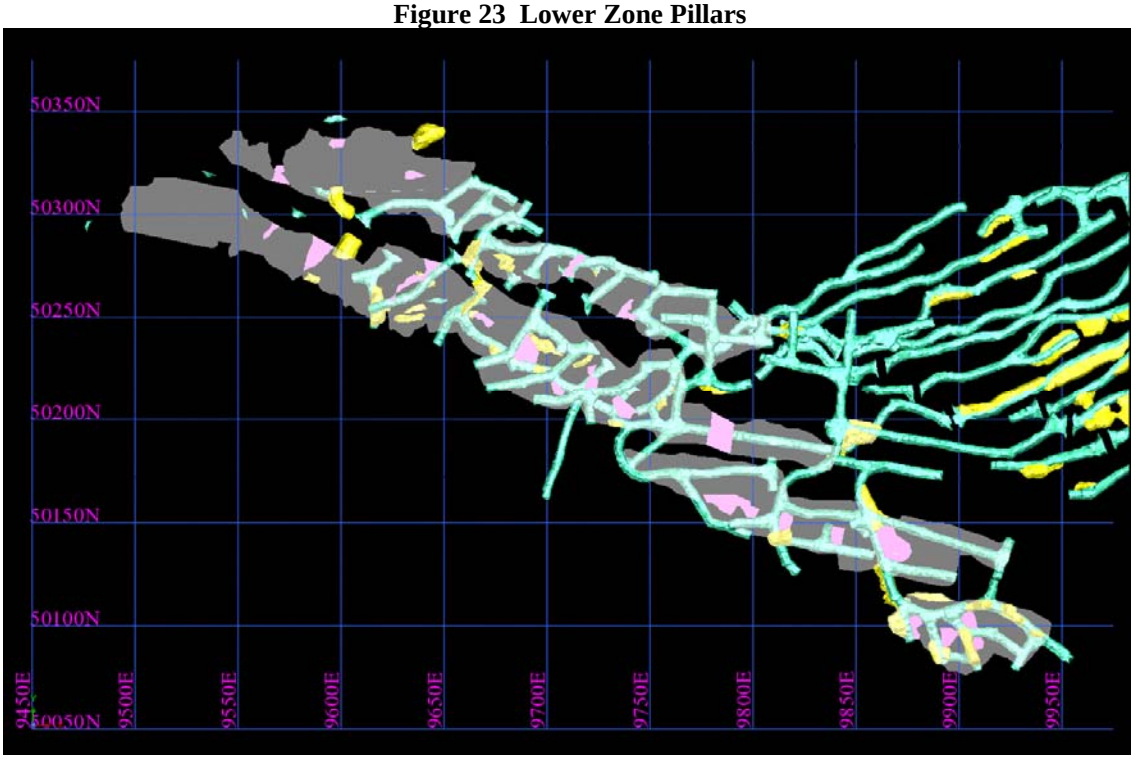


Figure 24 Upper Zone Pillars

Model Geotechnical Recovery

The area analysis approach described above indicates a historical pillar factor of 6.3% for Upper Zone (reduced from 8.7% in previous estimate) and 11.3% for Lower Zone (unchanged from previous estimate). It is expected that current mining practises will continue to reflect what has occurred in the past and as such that these factors are considered

appropriate for Reserve Estimation purposes. The reduction in material remnant in pillars for the Upper Zone may be explained by reduced strike lengths of the mineralised domain with increasing depth.

Economic Evaluation

All mine design was undertaken using the Maptek Vulcan3D mine planning software product. Gemcom Surpac software was used to report tonnes and grade for each stope shape and access ore drive.

As the native resource model format has been subcelled for appropriate resolution within the mineralised domains, not necessarily consistent with stope shapes, partial cell reporting was used within each of the defined shapes for suitable volume reconcile. As a validation check object volumes were compared to partial block cell volumes. The result of this analysis indicated a 0.1% variance in object 3dm vs. object partial cell volumes for stopes and 0.7% variance for development shapes. This was considered suitable resolution.

Each stope shape was subjected to an individual economic evaluation based on physical mining requirements. The cost of capital development including decline, stockpiles, accesses, ore passes and escapeways was distributed throughout stopes on level by level basis. The capital development required to access a lower level was distributed throughout stopes on that particular level. Stopes subjected to the economic evaluation that did not generate positive cash flow have not been reported as Reserves. Figure 25 provides an isometric view of the resultant mine design with stopes classified by their economic status. Only stopes considered to be economic would subsequently be reported as Ore Reserve.

A summary of the stope tonnes and grades subjected to modifying factors used for the generation of revenue is provided in Appendix D. A summary of physicals associated with each stope and the economics is provided in Appendix E and Appendix F.

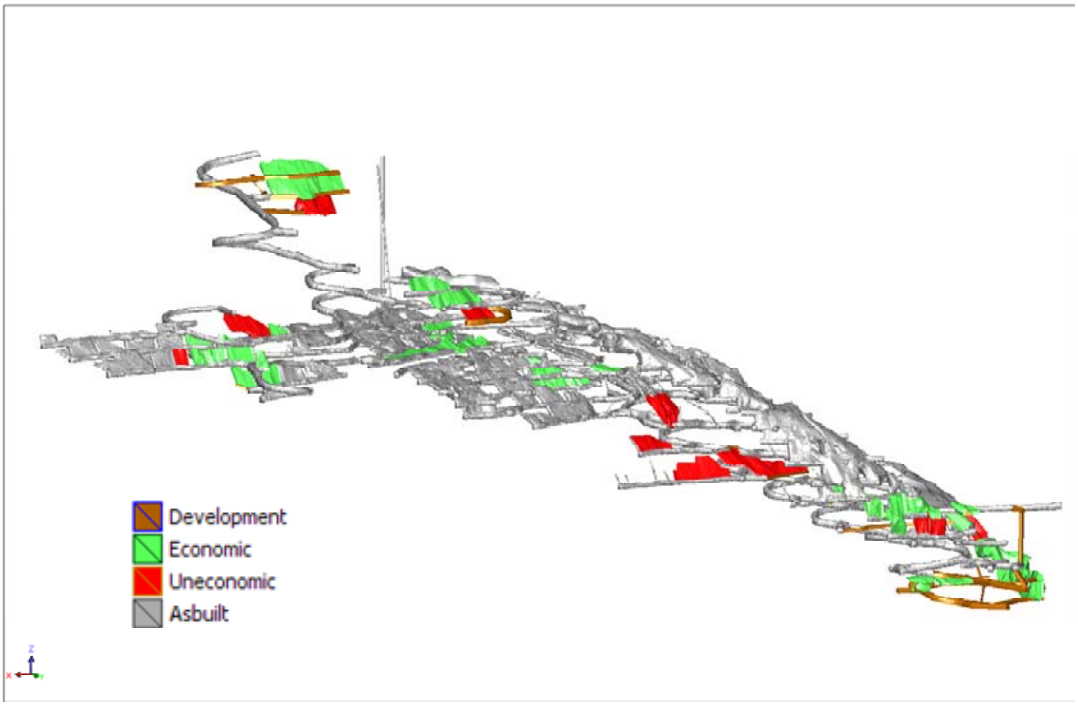


Figure 25 Isometric View of Reserves, 31st December 2008

Statement of Reserves

All Reserves stated are inclusive of Resources.

As of the 31st December 2008 Paulsens Ore Reserves are:

219,500 tonnes at 7.0g/t Au for 49,300 ounces of gold.

Table 11 below provides a breakdown of the stated Ore Reserves by zone and category.

	Proven			Probable			Total		
Zone	Tonnes t	Au g/t	Au Ounces	Tonnes t	Au g/t	Au Ounces	Tonnes t	Au g/t	Au Ounces
Apollo				18,000	5.2	3,000	18,000	5.2	3,000
Gemini				24,300	5.5	4,300	24,300	5.5	4,300
Lower West				3,900	5.1	600	3,900	5.1	600
Lower Zone				24,100	9.3	7,200	24,100	9.3	7,200
Soyuz				11,900	5.2	2,000	11,900	5.2	2,000
Upper Zone				123,500	7.4	29,400	123,500	7.4	29,400
Voyager				6,900	5.6	1,200	6,900	5.6	1,200
Stockpile High Grade	6,900	6.9	1,500				6,900	6.9	1,500
Total	**6,900**	**6.9**	**1,500**	**212,700**	**7.0**	**47,800**	**219,500**	**7.0**	**49,300**

Table 11 Paulsens Ore Reserves, 31st December 2008

Proportion of Proved Reserves

Proven Ore Reserves for Paulsens mine site as of the 31st December 2008 are 6,900 tonnes at 6.9 g/t Au for 1,500oz of gold, and Probable Ore Reserves for Paulsens are 212,700 tonnes at 7.0 g/t Au for 47,800oz of gold. On an ounce gold basis 3% of total ounces are in the Proven category (consisting of Stockpile Material only) and the remaining 97% in the Probable category. Figure 26 below graphically represents this relationship.

This represents a reduction in Proven Reserves from the previous estimate due to the current Resource Model reclassifying material into only Indicated and Inferred categories. There is no material currently classified as Measured in the current Resource Model.

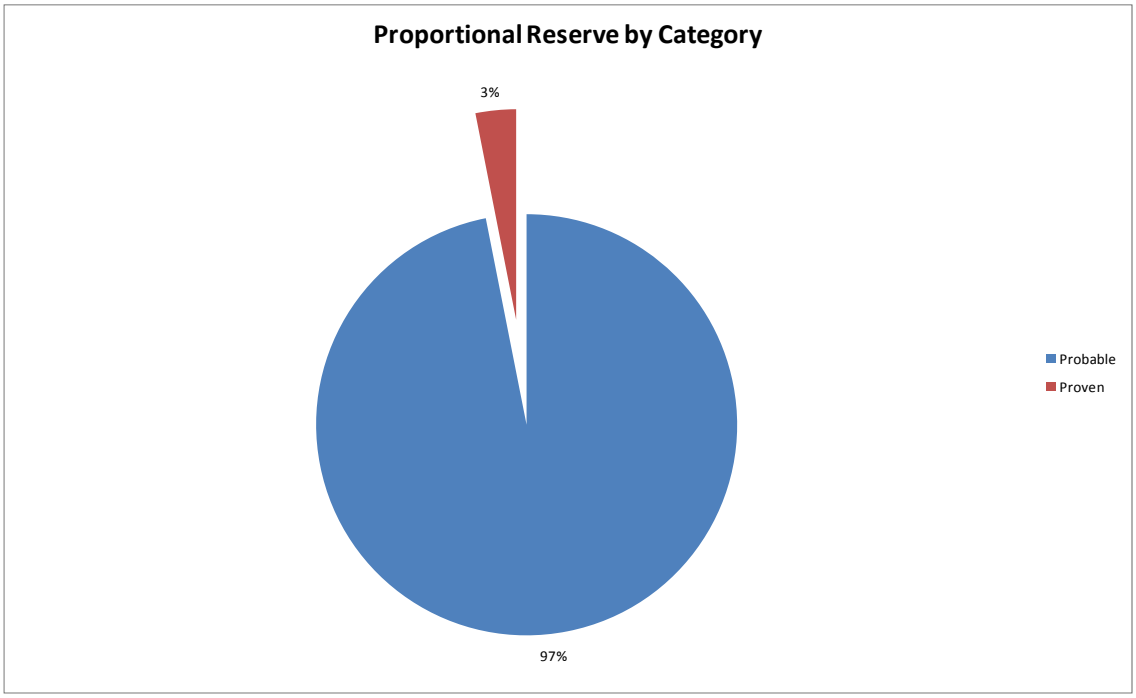


Figure 26 Reserve by Category, 31st December 2008

Comparison to Previous Reserves

The Ore Reserve estimate for 31st December 2007 was

591,000 tonnes at 8.6g/t Au for 163,000 ounces of gold.

The Ore Reserve estimate for 31st December 2008 is

219,500 tonnes at 7.0g/t Au for 49,300 ounces of gold.

Reconciled ore mined over the 12 month period to December 31st 2008 was

323,767 tonnes at 7.80g/t Au for 81,238 ounces of gold.

The Ore Reserve estimate represents a reduction in both tonnage and grade when mining depletion is taken into account from that previously disclosed. As mining factors and economics have not changed markedly it is proposed that the reduction is largely a result of the reinterpretation and reclassification of Resource that has resulted from new resource modelling. However changed economic parameters have resulted in the exclusion of low grade stockpile material from this Ore Reserve estimate.

Distribution of Reserves

The pie chart provided in Figure 27 shows the distribution of Reserves by zone. Upper Zone continues to comprise the bulk of the Reserve base at 60% followed by the Lower Zone containing 15% and Gemini containing 9% of the Reserve ounces. Significantly, Apollo has been displaced as an Ore Reserve source location which can be attributed to depletion by mining activities throughout the 2008 calendar year in the Apollo region.

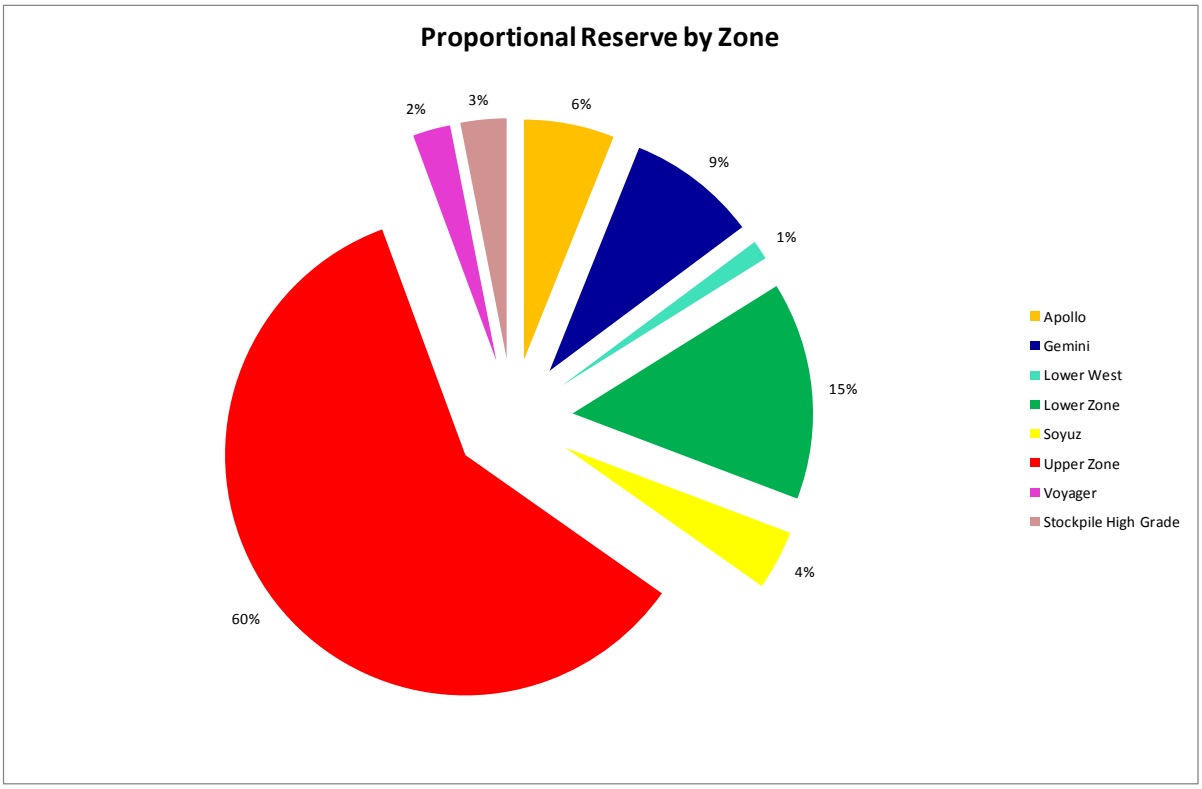


Figure 27 Reserve by Zone, 31st December 2008

Reserves by elevation for each of the geological zones at Paulsens are provided in Figure 28. The majority of the Reserve base consists of the lower levels of the Upper Zone, with pillar recovery and remnant stoping in the Lower Zone the next major contributor. However it should be noted that mining of the Lower Zone Remnants and also the Gemini stopes is unlikely to occur until the end of mine life as there may be access or production issues associated with their extraction.

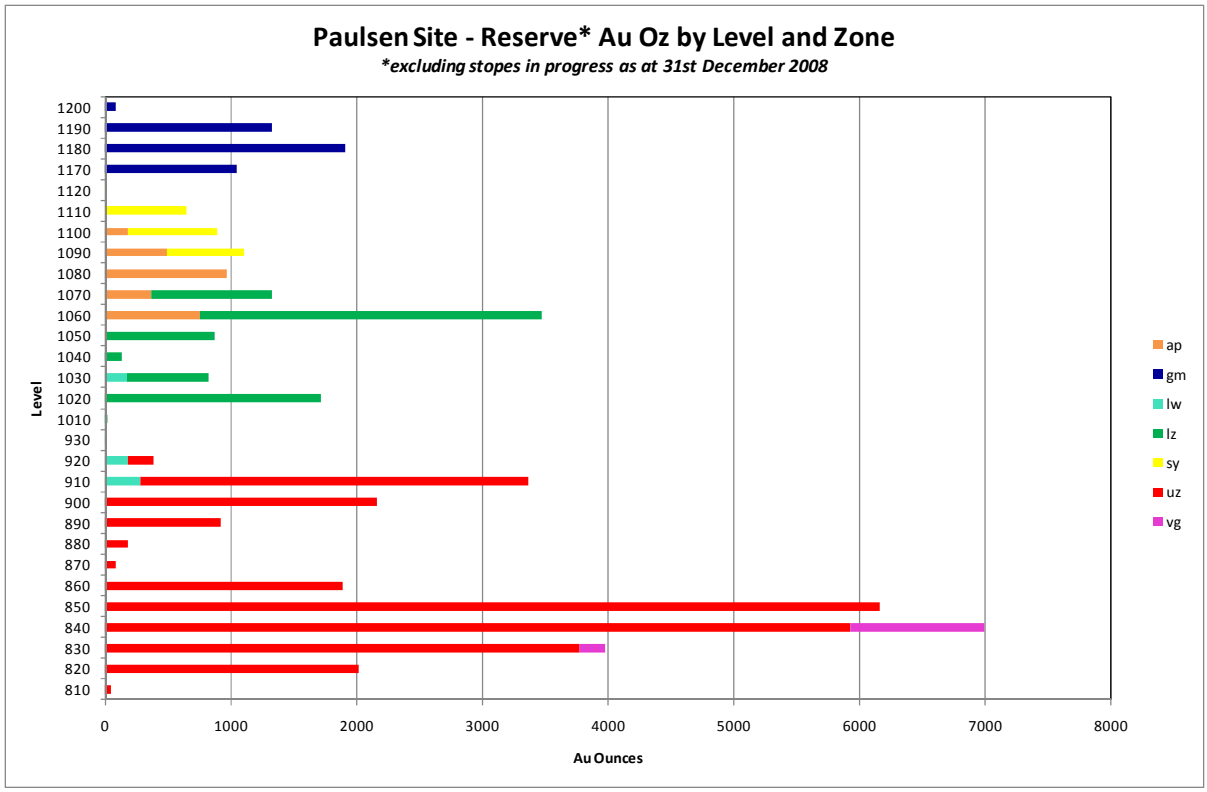


Figure 28 Reserve by Level and Zone, 31st December 2008

Reserves by Type

Reserves by elevation and extraction method are provided in Figure 29 below. As decline and level development is sufficiently ahead of active stoping areas, the majority of mill feed is to be sourced from active and planned stoping areas rather than development. The ore development that is a contributor to the Reserve base is located predominately at the peripheral satellite mining regions and lower levels of the mine.

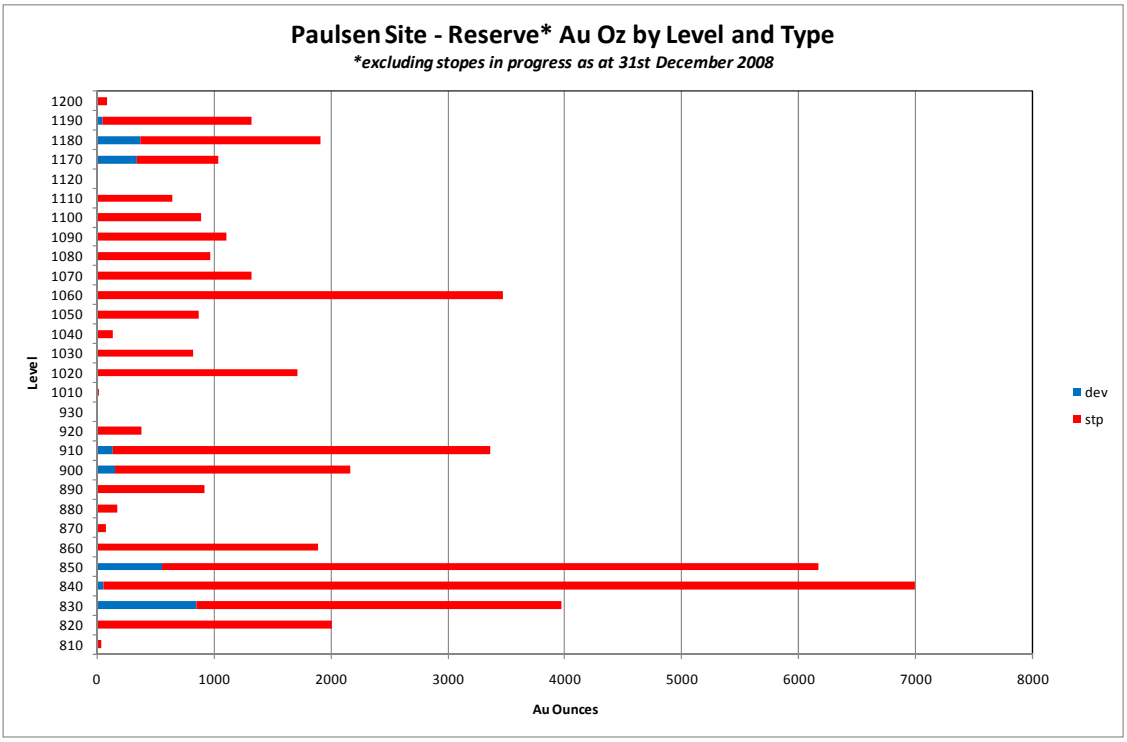


Figure 29 Reserve by Level and Type, 31st December 2008

Reserves as a Proportion of Resources

Total Reserves as a portion of Resource by elevation are provided in Figure 30.

Excluding stopes in progress and stockpile material, conversion of Indicated Resource Category ounces to Reserve is 56%. This conversion factor remains the same as the previous Ore Reserve estimate which is not unexpected as modifying mining factors have not changed significantly and the Resource is not overly sensitive to minor changes in cut off grade due to variations in economic parameters.

There is no Measured classification material in the Resource model used for this Ore Reserve estimation.

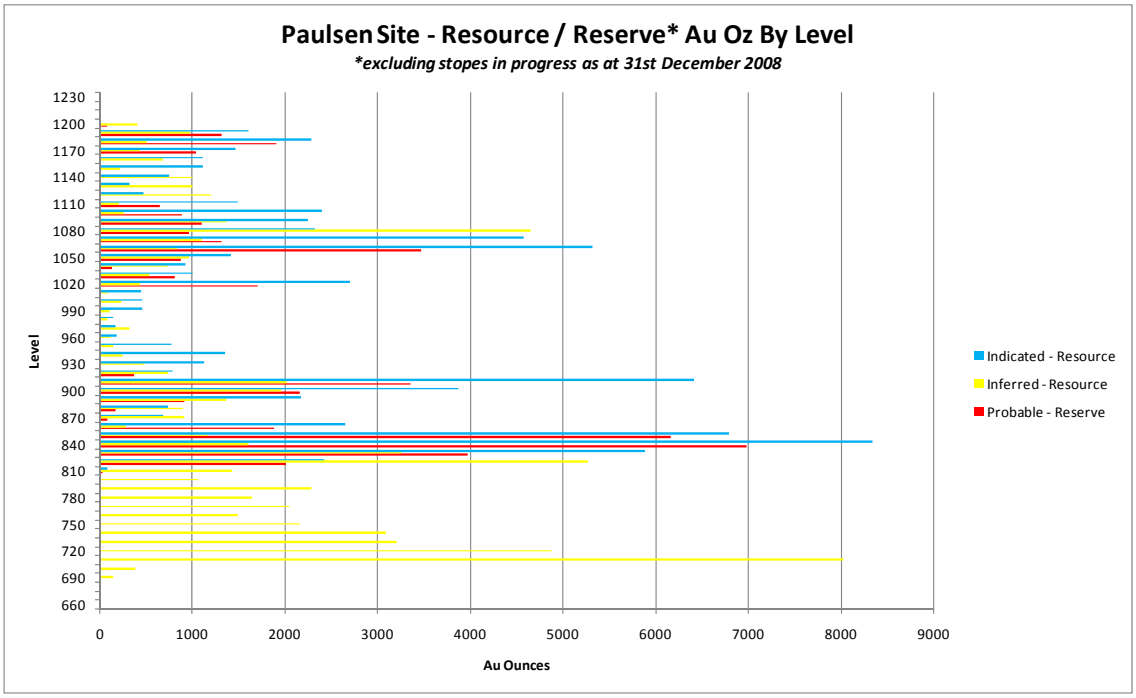


Figure 30 Resource to Reserve Conversion, 31st December 2008

Statement of Uneconomic Material Not in Reserve

Table 12 is an indication of material (contained in either stopes or stockpiles) that is deemed uneconomic when physicals and costs are applied. This material could possibly make future Reserves if the mining cut off grade can be lowered due to reduced cost or increase in commodity price.

	Proven			Probable			Total		
Zone	Tonnes t	Au g/t	Au Ounces	Tonnes t	Au g/t	Au Ounces	Tonnes t	Au g/t	Au Ounces
Apollo				7,400	3.3	800	7,400	3.3	800
Gemini				6,700	4.4	1,000	6,700	4.4	1,000
Lower West				8,300	4.4	1,200	8,300	4.4	1,200
Lower Zone				7,900	2.5	600	7,900	2.5	600
Upper Zone				17,300	1.9	1,000	17,300	1.9	1,000
Stockpile Low Grade	20,200	1.6	1,100				20,200	1.6	1,100
Total	**20,200**	**1.6**	**1,100**	**47,600**	**3.0**	**4,600**	**67,800**	**2.6**	**5,700**

Table 12 Uneconomic Material not in Reserve

The December 2007 Ore Reserve Estimate reported 54,700t at 3.8 g/t for 6700 ounces contained in Uneconomic material. The new estimate has resulted in an increased tonnage at lower grade than the previous estimate and a major contributing factor for this is the inclusion of stockpiled low grade material as marginally uneconomic at current indicated grade and processing cost profile.

Risk Analysis

Natural, infrastructure, environmental, legal, social, governmental, mineral tenure.

Paulsens gold mine has been operational for the past 4 years. During this time the site has managed to maintain constant mill throughput even during the extensive 2005/2006 cyclone season in North-West Western Australia. Bush fires during the dry season have come close to camp however with the fire fighting truck and trained personnel these have been kept beyond the graded fire break.

There is minimal risk associated with infrastructure. From commissioning in May 2005, the mine has maintained full mill feed with successive annual increases in throughput. The mill is expected to maintain similar availability (98%) as the past few years of operation.

Based on information provided by Intrepid personnel, Paulsens Site has taken out a $460,000 environmental bond with the government. To date government audits have been carried out and Paulsens has been in compliance in all material respects. There are no pending private or governmental law suites associated with this project. Socially, Intrepid has come to an agreement with the local indigenous community. The local indigenous community are paid a royalty of $A2/oz produced. There has been no negative feedback from the local community that the author is aware of.

References

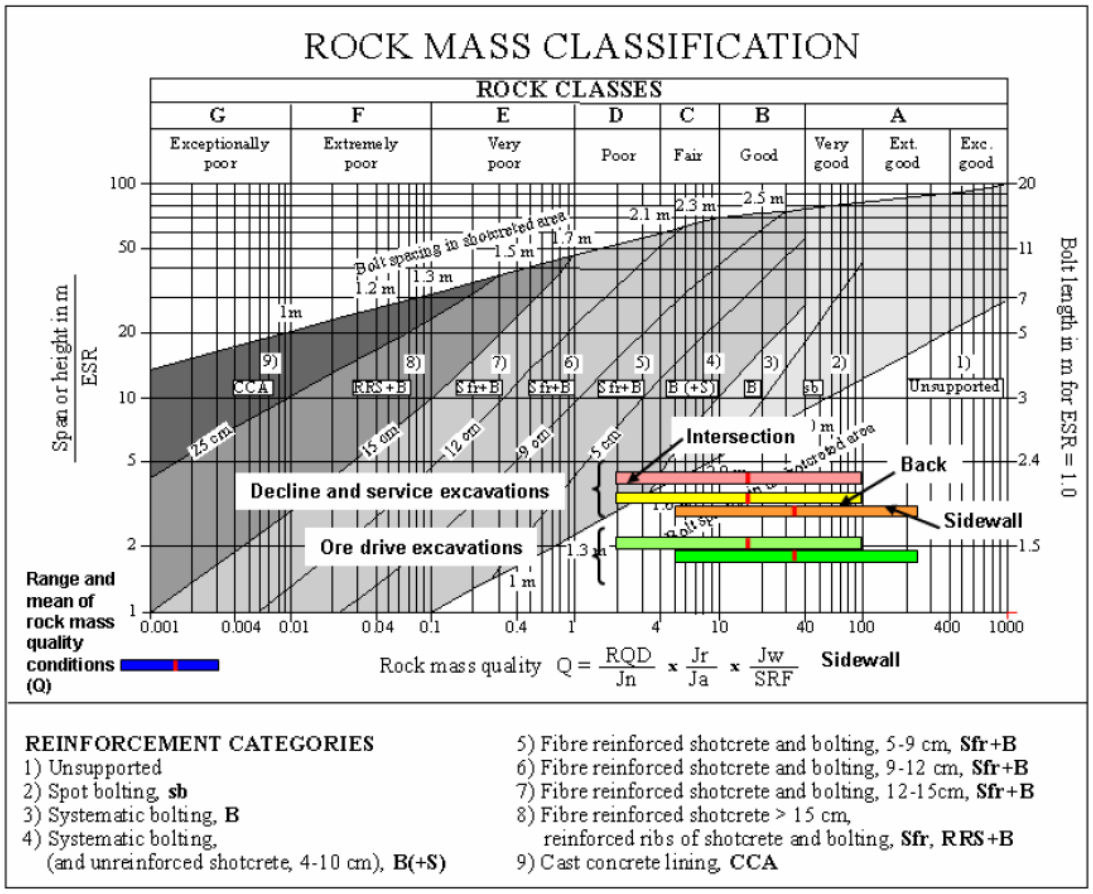
Golder Associates,2004. Paulsens Gold Mine Geotechnical Engineering Assessment 1 November 2004 - Project Number 4667.

Hellman & Schofield Pty Ltd, 2009. Resource Estimation for the Paulsens Gold Mine, Western Australia. March 2009

Creative Mined Enterprises, 2008. Paulsens Gold Mine Ore Reserves, 31st December 2007, Western Australia. June 2008

Appendix A

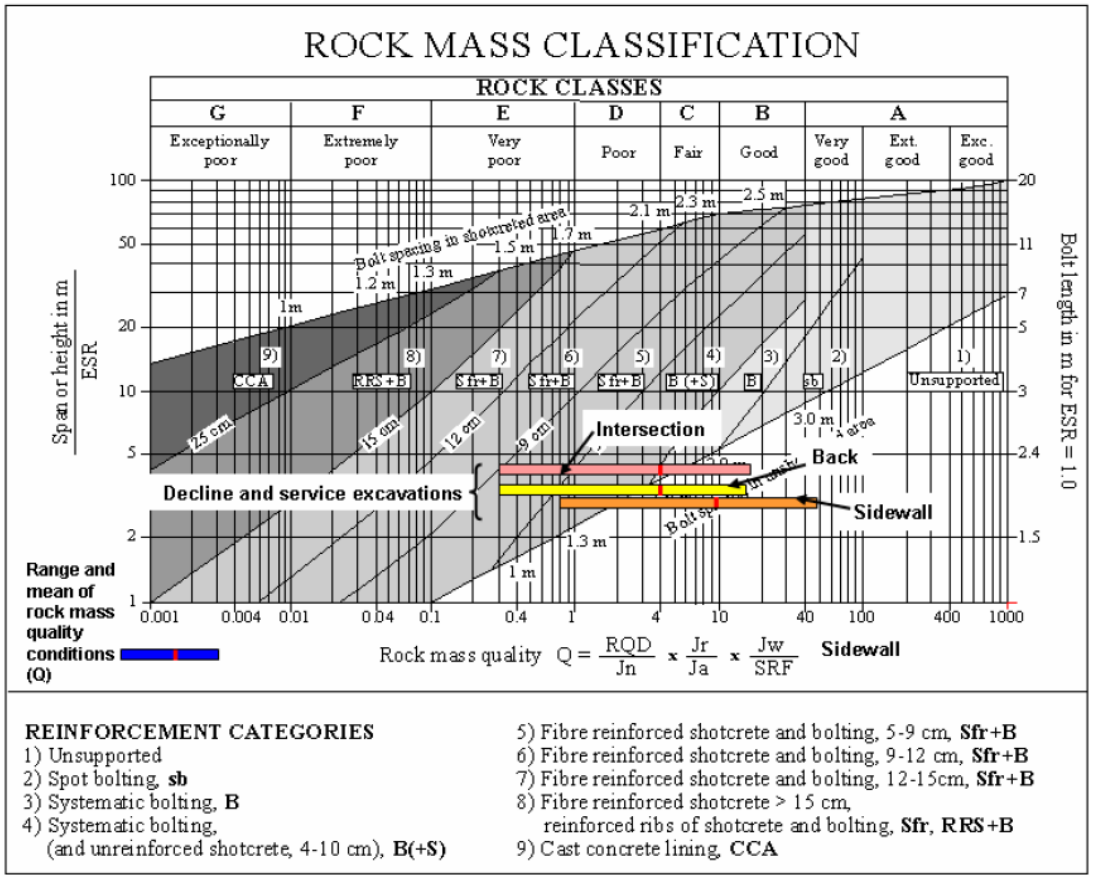
Q System of Support Selection and Q Values for Excavation in Quartz Domain



Quartz Domain

Appendix A

Q System of Support Selection and Q Values for Excavation in Sediment Domain



REINFORCEMENT CATEGORIES

1) Unsupported
2) Spot bolting, **sb**
3) Systematic bolting, **B**
4) Systematic bolting, (and unreinforced shotcrete, 4-10 cm), **B(+S)**
5) Fibre reinforced shotcrete and bolting, 5-9 cm, **Sfr+B**
6) Fibre reinforced shotcrete and bolting, 9-12 cm, **Sfr+B**
7) Fibre reinforced shotcrete and bolting, 12-15cm, **Sfr+B**
8) Fibre reinforced shotcrete > 15 cm, reinforced ribs of shotcrete and bolting, **Sfr**, **RRS+B**
9) Cast concrete lining, **CCA**

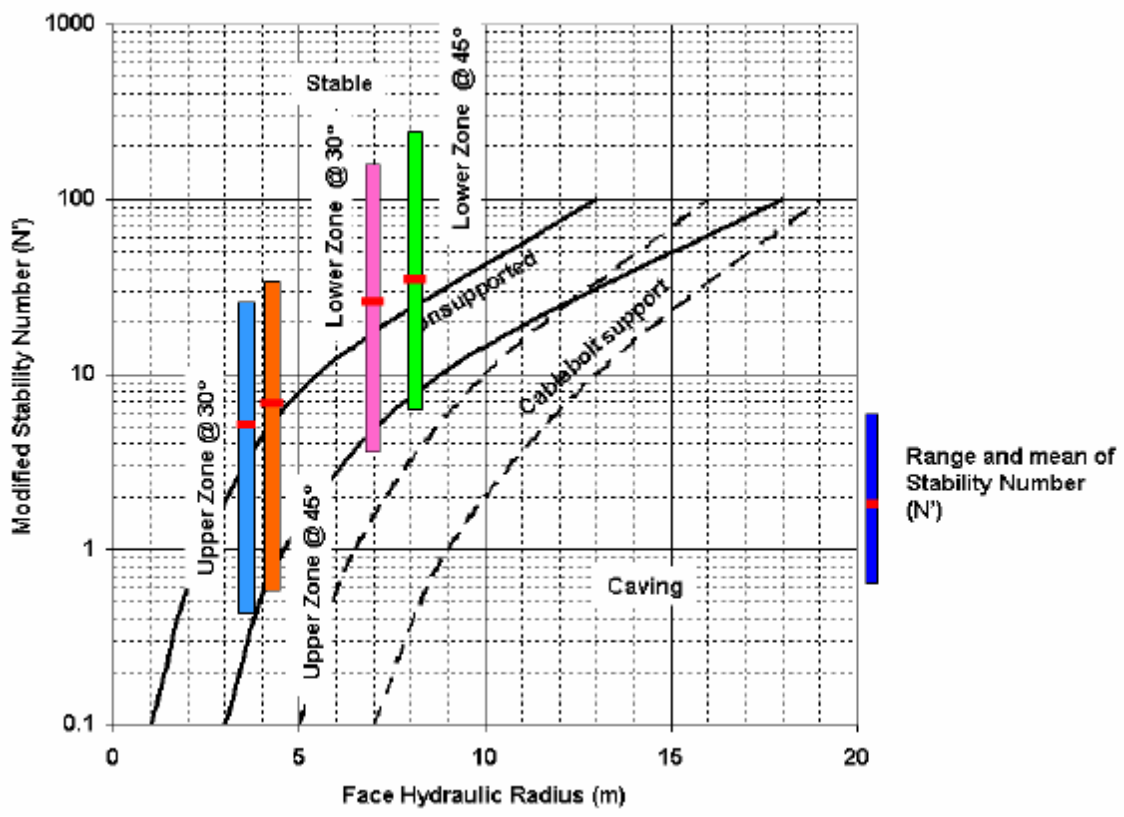
Sediment Domain

Appendix B

Recommended Maximum Stope Span Dimensions

Recommended maximum unsupported stope span dimensions				
Stope wall	Unsupported stope void		Supported stope geometry	
	Hydraulic Radius (m)	Equivalent stope void span (dip x strike)	Radius Factor (m)	Equivalent pillar spacing (dip x strike)
Upper Zone 30°	3.5	10 x 20	8	20 x 20
Upper Zone 45°	4	15 x 17	9	27 x 17
Lower Zone 30°	7	10 x ∞	12	20 x 30
Lower Zone 45°	8	15 x ∞	12.5	27 x 17

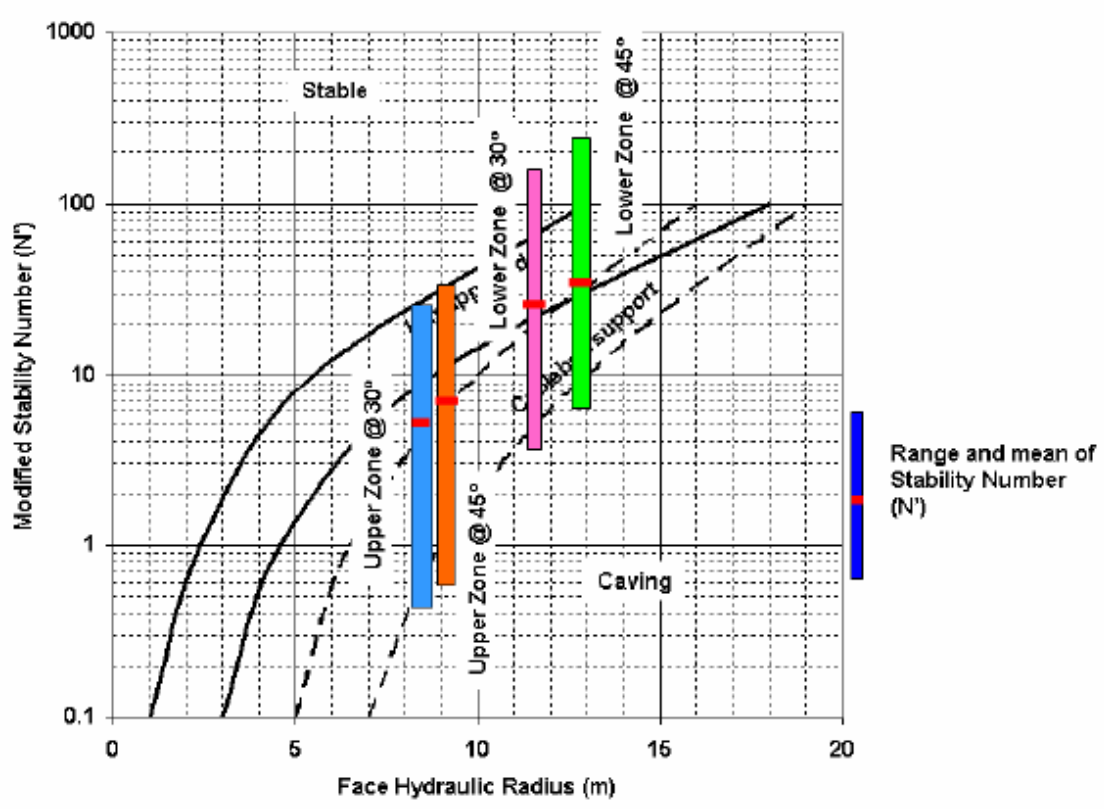
Unsupported Stope Criteria



Appendix B

Recommended Maximum Stope Span Dimensions

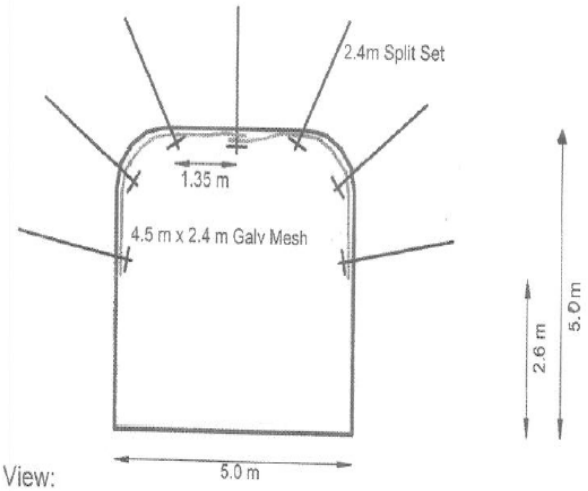
Cable Bolted Stope Criteria



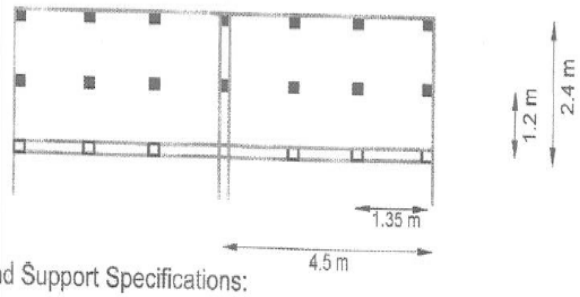
Appendix C
Development Ground Support Regimes

Ground Support Regime GSR 7

Section View:



Plan View:



Ground Support Specifications:

- 2.4 m Split Set
  Galvanised 2.4 m long x 0.46 mm diam
  1.35m spacing between bolts
  Fully cement grouted
  Grouted bolts no more than 30m from face

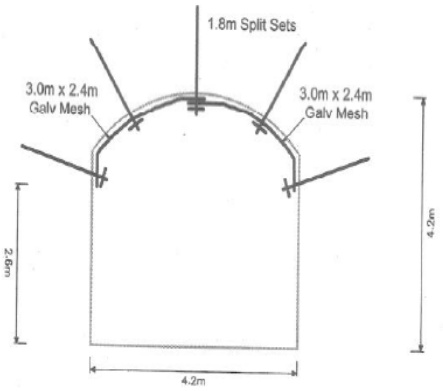
- 0.9 m Split Set
  Galvanised 0.9 m long x 0.38 mm diam
  1.35m spacing between bolts
  Fully cement grouted
  Grouted bolts no more than 30m from face

- 4.5 m x 2.4 m Mesh
  Galvanised
  0.2 - 0.4m overlap
  Long side accross the drive

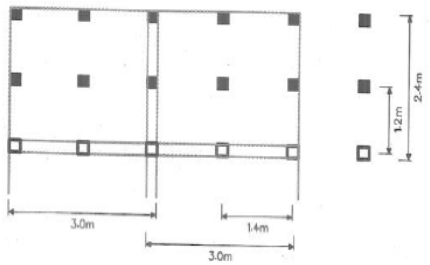
Approved	Signed	Date
Snr Mining Eng		
Snr Geologist		9/8/04
Snr Surveyor		9/8/04
Barminco Manager		9/8/04
Mining Manager		9/8/04

Ground Support Regime GSR 13

Section View:



Plan View:



Ground Support Specifications:

■ 1.8 m Split Set
Galvanised 1.8 m long x 0.46 mm diam
complete with 280 mm x 300 mm butterfly plate
1.4m spacing between bolts
Bolts Ungrouted

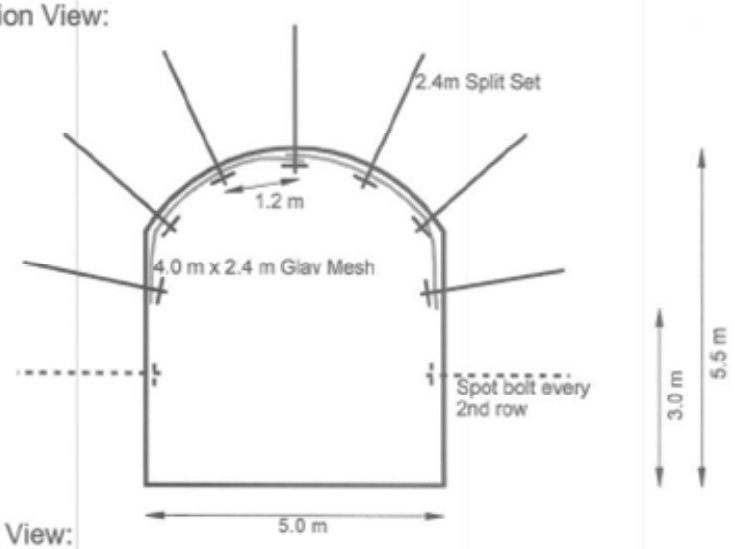
□ 0.9 m Split Set
Galvanised 0.9 m long x 0.38 mm diam
complete with 150 mm x 150 mm butterfly plate
1.4m spacing between bolts
Bolts Ungrouted

┐ 2 x 3.0 m x 2.4 m Mesh
Galvanised
0.2 - 0.4m overlap
Long side across the drive

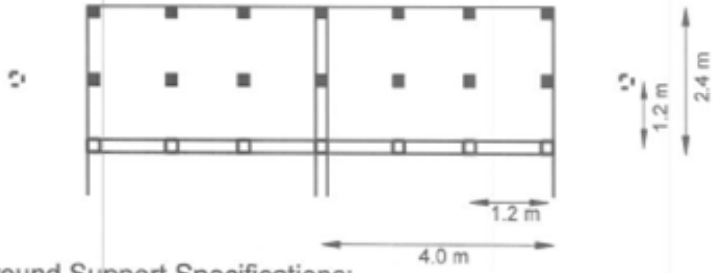
Approved	Signed	Date
Snr Mining Eng		
Snr Geologist		21/7/05
Snr Surveyor		22/7/05
Barminco Manager		22/7/05
Mining Manager		21/7/05

Ground Support Regime GSR 2

Section View:



Plan View:



Ground Support Specifications:

- ■ 2.4 m Split Set
  Galvanised 2.4 m long x 0.46 mm diam
  1.2m spacing between bolts
  Fully cement grouted
  Grouted bolts no more than 30m from face
- □ 0.9 m Split Set
  Galvanised 0.9 m long x 0.38 mm diam
  1.2m spacing between bolts
  Fully cement grouted
  Grouted bolts no more than 30m from face
- ┐ 4.0 m x 2.4 m Mesh
  Galvanised
  0.2 - 0.4m overlap
  Long side accross the drive

Approved	Signed	Date
Snr Mining Eng		
Snr Geologist		22/7/04
Snr Surveyor		12/7/04
Barminco Manager		12-07-04
Mining Manager		12/7/04

Appendix D

Stope Tonnes and Grade Report

						Dilution T&G		Diluted & Recovered		Development T&G		Total Reserves T&G			
StopeID	Orebody	Level	Design Date	Rescat	Mining Recovery	Average Stope Width	Unplanned Mining Dilution	Tonnes	Au	Tonnes	Au	Tonnes	Volume	SG	Au
0901_ap_1046_400_1	Apollo	1046	0901	Probable	74.4%	3.5	29%	2674	6.18	453	0.28	2674	897	2.98	6.18
0901_ap_1068_200_3	Apollo	1068	0901	Probable	74.4%	3.2	31%	3150	6.16	0	0.00	3150	1052	2.99	6.16
0901_ap_1068_200_4	Apollo	1068	0901	Probable	74.4%	2.1	48%	4781	5.05	0	0.00	4781	1575	3.03	5.05
0901_ap_1079_100_1	Apollo	1079	0901	Probable	74.4%	3.5	0%	6134	3.29	0	0.00	6134	1991	3.08	3.29
0901_ap_1081_100_1	Apollo	1081	0901	Probable	74.4%	3.3	30%	1001	4.40	0	0.00	1001	327	3.06	4.40
0901_ap_1082_200_1	Apollo	1082	0901	Probable	74.4%	1.7	59%	447	3.91	0	0.00	447	150	2.98	3.91
0901_gm_1148_100_1	Gemini	1148	0901	Probable	74.4%	5.9	17%	6732	4.40	3911	1.51	6732	2657	2.53	4.40
0901_gm_1161_100_1	Gemini	1161	0901	Probable	74.4%	4.4	23%	10132	6.53	3795	2.78	13927	5395	2.58	5.51
0901_gm_1177_100_1	Gemini	1177	0901	Probable	74.4%	2	50%	6860	6.39	3559	3.67	10419	4022	2.59	5.46
0901_lw_0904_100_1	Lower West	0904	0901	Probable	74.4%	2	0%	1643	6.41	1226	3.38	2869	1025	2.80	5.12
0901_lw_0928_100_1	Lower West	0928	0901	Probable	74.4%	2	0%	4097	4.62	1071	2.81	5167	1846	2.80	4.24
0901_lz_0928_200_1	Lower Zone	0928	0901	Probable	74.4%	2	50%	4831	3.21	0	0.00	4831	1726	2.80	3.21
0901_lz_0953_200_1	Lower Zone	0953	0901	Probable	74.4%	2	50%	3084	1.52	0	0.00	3084	1102	2.80	1.52
0901_lz_1010_200_1	Lower Zone	1010	0901	Probable	74.4%	2	50%	2300	9.79	0	0.00	2300	822	2.80	9.79
0901_lz_1019_200_1	Lower Zone	1019	0901	Probable	74.4%	2	50%	357	8.64	0	0.00	357	127	2.81	8.64
0901_lz_1019_200_b	Lower Zone	1019	0901	Probable	74.4%	4	25%	1747	9.47	0	0.00	1747	624	2.80	9.47
0901_lz_1028_200_b	Lower Zone	1028	0901	Probable	74.4%	2	50%	378	11.48	0	0.00	378	135	2.80	11.48
0901_lz_1060_100_1	Lower Zone	1060	0901	Probable	74.4%	2	0%	2248	13.00	0	0.00	2248	803	2.80	13.00
0901_sy_1082_200_1	Soyuz	1082	0901	Probable	74.4%	4.3	23%	7073	5.35	0	0.00	7073	2343	3.02	5.35
0901_sy_1097_200_1	Soyuz	1097	0901	Probable	74.4%	3	33%	4858	4.87	0	0.00	4858	1593	3.05	4.87
0901_uz_0806_100_1	Upper Zone	0806	0901	Probable	84.8%	12.1	12%	12524	8.06	3012	0.39	12524	4012	3.12	8.06
0901_uz_0823_100_1	Upper Zone	0823	0901	Probable	84.8%	11.7	13%	10537	10.43	3554	2.25	14092	4597	3.07	8.36
0901_uz_0823_100_2	Upper Zone	0823	0901	Probable	84.8%	16.9	9%	24859	8.84	2705	5.95	27564	8717	3.16	8.56
0901_uz_0842_100_2	Upper Zone	0842	0901	Probable	84.8%	16.5	9%	9924	8.62	0	0.00	9924	3202	3.10	8.62
0901_uz_0859_100_3	Upper Zone	0859	0901	Probable	84.8%	3	50%	178	6.37	0	0.00	178	59	3.04	6.37
0901_uz_0873_100_2	Upper Zone	0873	0901	Probable	84.8%	9.7	15%	5176	5.43	0	0.00	5176	1658	3.12	5.43
0901_uz_0895_200_1	Upper Zone	0895	0901	Probable	84.8%	6.3	24%	5355	14.95	0	0.00	5355	1699	3.15	14.95
0901_uz_0895_200_2	Upper Zone	0895	0901	Probable	84.8%	4.2	36%	3682	5.61	0	0.00	3682	1173	3.14	5.61
0901_uz_0895_200_3	Upper Zone	0895	0901	Probable	84.8%	11	14%	10172	3.68	0	0.00	10172	3204	3.17	3.68
0901_vg_0823_200_2	Voyager	0823	0901	Probable	74.4%	14	7%	4668	6.50	4102	1.28	4668	1667	2.80	6.50
0901_vg_0823_210_1	Voyager	0823	0901	Probable	74.4%	9.7	10%	808	5.99	1395	2.58	2202	787	2.80	3.83
0901_lz_1045_200_1	Lower Zone	1045	0901	Probable	74.4%	3	33%	12289	7.42	0	0.00	12289	4389	2.80	7.42
0901_lz_1052_200_1	Lower Zone	1052	0901	Probable	74.4%	3.5	29%	1276	16.05	0	0.00	1276	456	2.80	16.05
site_hg_surf	Stockpile Hig	surf	site	Proven	100.0%		0%	6857	6.95	0	0.00	6857	3825	1.79	6.95
site_lg_surf	Stockpile Lov	surf	site	Proven	100.0%		0%	20180	1.62	0	0.00	20180	10090	2.00	1.62
site_uz_0895_100	Upper Zone	0895	site	Probable	84.8%		0%	5783	11.85	0	0.00	5783	2065	2.80	11.85
site_uz_0910_100	Upper Zone	0910	site	Probable	84.8%		0%	4359	4.40	0	0.00	4359	1557	2.80	4.40
site_uz_0910_200	Upper Zone	0910	site	Probable	84.8%		0%	421	4.12	0	0.00	421	151	2.80	4.12
site_uz_0963_300	Upper Zone	0963	site	Probable	84.8%		0%	871	5.36	0	0.00	871	311	2.80	5.36
site_lz_1028	Lower Zone	1028	site	Probable	74.4%		0%	525	10.76	0	0.00	525	188	2.80	10.76
site_ap_1046	Apollo	1046	site	Probable	74.4%		0%	776	4.65	0	0.00	776	277	2.80	4.65
site_ap_1068	Apollo	1068	site	Probable	74.4%		0%	633	4.96	0	0.00	633	226	2.80	4.96
0901_ap_1046_200_2	Apollo	1046	0901	Probable	74.4%	4.9	20%	3763	4.31	0	0.00	3763	1270	2.96	4.31
0901_ap_1068_200_2	Apollo	1068	0901	Probable	74.4%	2.2	45%	1229	3.44	0	0.00	1229	407	3.02	3.44
0901_ap_1081_200_5	Apollo	1081	0901	Probable	74.4%	3	0%	808	5.04	0	0.00	808	269	3.00	5.04
0901_lw_0976_100_1	Lower West	0976	0901	Probable	74.4%	2	0%	3138	4.75	0	0.00	3138	1121	2.80	4.75
0901_lw_1019_100_1	Lower West	1019	0901	Probable	74.4%	2	0%	1068	5.11	0	0.00	1068	382	2.80	5.11
0901_lz_1010_200_b	Lower Zone	1010	0901	Probable	74.4%	4.5	22%	2954	10.76	0	0.00	2954	1055	2.80	10.76
0901_uz_0842_100_1	Upper Zone	0842	0901	Probable	84.8%	12.7	12%	12461	4.88	3795	4.61	16257	5393	3.01	4.81
0901_uz_0859_100_1	Upper Zone	0859	0901	Probable	84.8%	11.5	13%	6297	0.71	0	0.00	6297	2163	2.91	0.71
0901_uz_0873_100_1	Upper Zone	0873	0901	Probable	84.8%	11.3	13%	2946	1.76	0	0.00	2946	969	3.04	1.76
0901_uz_0873_200_1	Upper Zone	0873	0901	Probable	84.8%	5.3	28%	2823	5.83	0	0.00	2823	954	2.96	5.83
0901_uz_0873_200_2	Upper Zone	0873	0901	Probable	84.8%	6.9	22%	2440	0.95	0	0.00	2440	796	3.07	0.95
0901_uz_0873_200_3	Upper Zone	0873	0901	Probable	84.8%	6.1	25%	5616	3.60	0	0.00	5616	1810	3.10	3.60
0901_uz_0895_400_1	Upper Zone	0895	0901	Probable	84.8%	21	7%	3294	3.88	995	4.75	4289	1390	3.09	4.09

Appendix E

Stope Physicals Report

			Physicals																																			
			Lateral Development Advance					Vertical Development Advance				Waste Development	Stoping					Geological Definition				Ground Support - Longhole		Ground Support - Airleg		Scraping - Airleg	Production Drilling		Remote Bogging		Conventional Bogging		Long Tram	Haulage - Stope			Haulage - Waste		
StopeID	OrebodyID	Stoping Method	Capital	Operating Waste	Ore	Total (m)	Stripping (m3)	Capital	Operating Waste	Ore	Total	Total T	Design Stope T	Diluted & Recovered Stope T	Diluted & Recovered Stope m3	Total Ore T (Stope + Dev)	Ounces Produced	Diamond Drilling Required	Diamond Drill Meters	Sludge Drilling Required	Sludge Drill Meters	Cable Bolting Required	Cable Meters	Bolts (quantity)	Mesh (m2)	Hours	Drill m	Charge m	Remote %	Remote T	Conventional %	Conventional T	Long Tram T	Level	Distance	TKM	Level	Distance	TKM
0901_ap_1046_400_1	ap	LH	0	0	22.5	22.5	43	0	0	0	0	453	2795	2674	897	2674	497	No		No		No					792	660	62.3%	1666	37.7%	1008	1495	1045L	1.56	4185	1045L	1.69	767
0901_ap_1068_200_3	ap	LH				0	0				0	0	3225	3150	1052	3150	583	No		No		No					914	761	62.3%	1962	37.7%	1188	1761	1068L	1.43	4489	1068L	1.56	0
0901_ap_1068_200_4	ap	LH				0	0				0	0	4352	4781	1575	4781	726	No		No		No					1233	1027	62.3%	2979	37.7%	1802	2673	1068L	1.43	6813	1068L	1.56	0
0901_ap_1079_100_1	ap	AL				0	0				0	0	8242	6134	1991	6134	606	No		No		No		578	38	331			0.0%	0	100.0%	6134	3429	1079L	1.36	8331	1079L	1.49	0
0901_ap_1081_100_1	ap	LH				0	0				0	0	1032	1001	327	1001	132	No		No		No					292	244	62.3%	623	37.7%	377	559	1081L	1.35	1347	1081L	1.48	0
0901_ap_1082_200_1	ap	LH				0	0				0	0	378	447	150	447	53	No		No		No					107	89	62.3%	278	37.7%	168	250	1082L	2.00	891	1082L	2.11	0
0901_gm_1148_100_1	gm	LH	45	0	34	79	150				0	6208	7735	6732	2657	6732	890	No		No		Yes	50				2191	1826	62.3%	4194	37.7%	2538	3763	1148L	0.55	3723	1148L	0.67	4128
0901_gm_1161_100_1	gm	LH	21	0	77	98	186				0	1720	11093	10132	3960	13927	2306	No		No		Yes	72				3142	2619	62.3%	6312	37.7%	3820	7785	1161L	0.46	4663	1161L	0.57	986
0901_gm_1177_100_1	gm	LH	101.5	0	74	175.5	333	14.3	0	0	14.3	5307	6145	6860	2685	10419	1711	No		No		Yes	40				1741	1451	62.3%	4274	37.7%	2586	5824	1177L	0.46	3157	1177L	0.57	3041
0901_lw_0873_200_1	lw	AL				0	0				0	0	840	625	223	625	38	No		No		No		59	4	34			0.0%	0	100.0%	625	349	0873L	2.30	1435	0873L	2.41	0
0901_lw_0880_100_1	lw	AL	0	0	106	106	201				0	0	6678	4970	1767	7108	2211	No		No		No		469	31	268			0.0%	0	100.0%	4970	3974	0880L	2.23	11087	0880L	2.34	0
0901_lw_0904_100_1	lw	AL				0	0				0	0	2208	1643	587	2869	441	No		No		No		155	10	89			0.0%	0	100.0%	1643	1604	0904L	2.08	3410	0904L	2.19	0
0901_lw_0904_200_1	lw	AL	0	0	60	60	114				0	0	3792	2822	1008	2822	379	No		No		No		266	18	152			0.0%	0	100.0%	2822	1577	0904L	2.08	5856	0904L	2.19	0
0901_lw_0928_100_1	lw	AL	0	0	54.6	54.6	104				0	0	5505	4097	1463	5167	659	No		No		No		386	26	221			0.0%	0	100.0%	4097	2889	0928L	1.88	7688	0928L	2.01	0
0901_lz_0928_200_1	lz	LH				0	0				0	0	4328	4831	1726	4831	466	No		No		No					1226	1022	62.3%	3010	37.7%	1821	2701	0928L	1.88	9067	0928L	2.01	0
0901_lz_0953_200_1	lz	LH				0	0				0	0	2763	3084	1102	3084	141	No		No		No					783	652	62.3%	1922	37.7%	1163	1724	0953L	1.75	5401	0953L	1.86	0
0901_lz_1010_200_1	lz	LH				0	0				0	0	2060	2300	822	2300	677	No		No		No					584	486	62.3%	1433	37.7%	867	1285	1010L	1.38	3169	1010L	1.51	0
0901_lz_1019_200_1	lz	LH				0	0				0	0	320	357	127	357	93	No		No		No					91	76	62.3%	223	37.7%	135	200	1019L	1.32	473	1019L	1.45	0
0901_lz_1019_200_b	lz	LH				0	0				0	0	1878	1747	624	1747	497	No		No		No					532	443	62.3%	1088	37.7%	659	977	1019L	1.32	2312	1019L	1.45	0
0901_lz_1028_200_b	lz	LH				0	0				0	0	339	378	135	378	131	No		No		No					96	80	62.3%	236	37.7%	143	212	1028L	1.27	480	1028L	1.40	0
0901_lz_1060_100_1	lz	AL				0	0				0	0	3021	2248	803	2248	879	No		No		No		212	14	121			0.0%	0	100.0%	2248	1257	1060L	1.07	2414	1060L	1.20	0
0901_sy_1070_200_1	sy	LH	55	0	43	98	186				0	2807	4053	3959	1317	6695	872	No		Yes	59	Yes	26				1148	957	62.3%	2466	37.7%	1492	3743	1070L	0.91	3610	1070L	1.02	2877
0901_sy_1082_200_1	sy	LH				0	0				0	0	7711	7073	2343	7073	1139	No		No		No					2184	1820	62.3%	4406	37.7%	2667	3954	1082L	1.60	11281	1082L	1.71	0
0901_sy_1097_200_1	sy	LH				0	0				0	0	4896	4858	1593	4858	711	No		No		No					1387	1156	62.3%	3027	37.7%	1832	2716	1097L	0.91	4431	1097L	1.02	0
0901_uz_0806_100_1	uz	LH	215.8	0	60.2	276	524	19.4	0	0	19.4	20279	13140	12524	4012	12524	3033	Yes	1372	Yes	191	Yes	85				2438	2032	62.3%	7802	37.7%	4721	2112	0806L	2.60	32562	0806L	2.71	55038
0901_uz_0823_100_1	uz	LH	224.39	0	73.2	297.59	565	17.3	0	0	17.3	15125	11014	10537	3375	14092	3543	Yes	1150	Yes	160	Yes	71				2043	1703	62.3%	6565	37.7%	3972	2376	0823L	2.52	26505	0823L	2.65	40011
0901_uz_0823_100_2	uz	LH	0	0	49.3	49.3	94				0	0	26925	24859	7838	27564	7092	Yes	2811	Yes	390	Yes	174				4995	4163	62.3%	15487	37.7%	9372	4647	0823L	2.52	62528	0823L	2.65	0
0901_uz_0842_100_2	uz	LH	0	0	29.7	29.7	56	45	0	0	45	1852	10728	9924	3202	9924	2571	No		Yes	156	Yes	69				1990	1659	62.3%	6183	37.7%	3741	1673	0842L	2.31	22965	0842L	2.43	4494
0901_uz_0859_100_3	uz	LH				0	0				0	0	140	178	59	178	34	No		No		No					26	22	62.3%	111	37.7%	67	30	0859L	2.30	409	0859L	2.43	0
0901_uz_0873_100_2	uz	LH				0	0				0	0	5286	5176	1658	5176	845	No		No		No					981	817	62.3%	3224	37.7%	1951	873	0873L	2.30	11883	0873L	2.41	0
0901_uz_0895_200_1	uz	LH				0	0				0	0	5101	5355	1699	5355	2407	No		No		No					946	789	62.3%	3336	37.7%	2019	903	0895L	2.08	11125	0895L	2.21	0
0901_uz_0895_200_2	uz	LH				0	0				0	0	3199	3682	1173	3682	621	No		No		No					594	495	62.3%	2294	37.7%	1388	621	0895L	2.08	7648	0895L	2.21	0
0901_uz_0895_200_3	uz	LH				0	0				0	0	10556	10172	3204	10172	1125	No		No		No					1958	1632	62.3%	6337	37.7%	3835	1715	0895L	2.08	21131	0895L	2.21	0
0901_vg_0823_200_2	vg	LH	0	0	107.9	107.9	205				0	4102	5855	4668	1667	4668	912	Yes	611	Yes	85	Yes	38				1659	1382	62.3%	2908	37.7%	1760	2610	0823L	2.52	11743	0823L	2.65	10851
0901_vg_0823_210_1	vg	LH				0	0				0	0	984	808	288	2202	254	Yes	103	Yes	14	Yes	6				279	232	62.3%	503	37.7%	305	1231	0823L	2.52	2032	0823L	2.65	0
0901_lz_1045_200_1	lz	LH				0	0				0	0	12385	12289	4389	12289	2741	No		No		No					3508	2924	62.3%	7656	37.7%	4633	6870	1045L	1.16	14317	1045L	1.29	0
0901_lz_1052_200_1	lz	LH				0	0				0	0	1334	1276	456	1276	616	No		No		No					378	315	62.3%	795	37.7%	481	714	1052L	1.12	1432	1052L	1.23	0
site_hg_surf	hg					0	0				0	0	6857	6857	3825	6857	1432																	surfL	0.00	0	surfL	0.00	0
site_lg_surf	lg					0	0				0	0	20180	20180	10090	20180	983																	surfL	0.00	0	surfL	0.00	0
site_uz_0895_100	uz	LH				0	0				0	0	6820	5783	2065	5783	2060												62.3%	3603	37.7%	2180	975	0895L	2.08	12014	0895L	2.21	0
site_uz_0910_100	uz	LH				0	0				0	0	5141	4359	1557	4359	577												62.3%	2716	37.7%	1644	735	0911L	1.93	8420	0911L	2.06	0
site_uz_0910_200	uz	LH				0	0				0	0	497	421	151	421	52												62.3%	263	37.7%	159	71	0911L	1.93	814	0911L	2.06	0
site_uz_0963_300	uz	LH				0	0				0	0	1027	871	311	871	140												62.3%	543	37.7%	328	147	0963L	1.62	1407	0963L	1.75	0
site_lz_1028	lz	LH				0	0				0	0	706	525	188	525	170												62.3%	327	37.7%	198	294	1028L	1.27	666	1028L	1.40	0
site_ap_1046	ap	LH				0	0				0	0	1043	776	277	776	108												62.3%	484	37.7%	293	434	1045L	1.56	1215	1045L	1.69	0
site_ap_1068	ap	LH				0	0				0	0	851	633	226	633	94												62.3%	395	37.7%	239	354	1068L	1.43	903	1068L	1.56	0
0901_ap_1046_200_2	ap	LH				0	0				0	0	4199	3763	1270	3763	488										1190	991	62.3%	2344	37.7%	1418	2103	1045L	1.56	5888	1045L	1.69	0
0901_ap_1068_200_2	ap	LH				0	0				0	0	1135	1229	407	1229	127										322	268	62.3%	765	37.7%	463	687	1068L	1.43	1751	1068L	1.56	0
0901_ap_1081_200_5	ap	AL				0	0				0	0	1086	808	269	808	122							76	5	44			0.0%	0	100.0%	808	452	1081L	1.35	1088	1081L	1.48	0
0901_lw_0953_100_1	lw	AL	0	0	18	18	34				0	0	1887	1404	502	1783	231							132	9	76			0.0%	0	100.0%	1404	997	0953L	1.75	2459	0953L	1.86	0
0901_lw_0976_100_1	lw	AL				0	0				0	0	4216	3138	1121	3138	448							296	20	169			0.0%	0	100.0%	3138	1754	0976L	1.59	4995	0976L	1.70	0
0901_lw_1019_100_1	lw	AL				0	0				0	0	1435	1068	382	1068	164							101	7	58			0.0%	0	100.0%	1068	597	1019L	1.32	1413	1019L	1.45	0
0901_lz_1010_200_b	lz	LH				0	0				0	0	3248	2954	1055	2954	955										920	767	62.3%	1841	37.7%	1114	1651	1010L	1.38	4071	1010L	1.51	0
0901_uz_0842_100_1	uz	LH	0	0	40.7	40.7	77				0	0	13143	12461	4091	16257	2352			Yes	191	Yes	85				2438	2032	62.3%	7763	37.7%	4698	2741	0842L	2.31	28836	0842L	2.43	0
0901_uz_0859_100_1	uz	LH				0	0				0	0	6569	6297	2163	6297	134			Yes	95						1219	1016	62.3%	3923	37.7%	2374	1062	0859L	2.30	14460	0859L	2.43	0
0901_uz_0873_100_1	uz	LH				0	0				0	0	3067	2946	969	2946	156										569	474	62.3%	1835	37.7%	1111	497	0873L	2.30	6764	0873L	2.41	0
0901_uz_0873_200_1	uz	LH				0	0				0	0	2595	2823	954	2823	495										481	401	62.3%	1759	37.7%	1064	476	0873L	2.30	6482	0873L	2.41	0
0901_uz_0873_200_2	uz	LH				0	0				0	0	2364	2440	796	2440	69										439	365	62.3%	1520	37.7%	920	411	0873L	2.30	5603	0873L	2.41	0
0901_uz_0873_200_3	uz	LH				0	0				0	0	5316	5616	1810	5616	607										986	822	62.3%	3499	37.7%	2117	947	0873L	2.30	12895	0873L	2.41	0
0901_uz_0895_400_1	uz	LH	14.4	0	17	31.4	60				0	735	3626	3294	1075	4289	527										673	561	62.3%	2052	37.7%	1242	723	0895L	2.08	6844	0895L	2.21	1622
0901_uz_0963_300_1	uz	LH				0	0				0	0	2726	2589	924	2589	4										506	421	62.3%	1613	37.7%	976	437	0963L	1.62	4182	0963L	1.75	0

Appendix F
Stope Economics Report

			Costs																																Financials Summary					
			Lateral Development									Vertical Development																												
StopeID	OrebodyID	Stoping Method	Capital - Advance	Operating Waste - Advance	Ore - Advance	Capital - Support	Operating Waste - Support	Ore - Support	Wet Development	Stripping	Total Lateral	Capital	Operating Waste	Ore	Total Vertical	Diamond Drilling	Sludge Drilling	Cable Bolting	Longhole Drilling	Longhole Charging	Jumbo Stoping	Airleg Stoping	Airleg Ground Support	Airleg Scraping	Remote Bogging	Remote Cuddies & Slot Rises	Conventional Bogging	Long Tram	Haulage Ore	Haulage Waste	Barminco Fixed	Barminco Rise and Fall	Mining Services Fixed	Geology Services Fixed	Processing	Administration	Total Cost	Revenue	State Gov Royalty	PKKP Royalty	Profit	ROI	Stope Economic
0901_ap_1046_400_1	ap	LH	$0	$0	$24,323	$0	$0	$0	$0	$1,359	$25,682	$0	$0		$0				$16,786	$12,207					$13,245	$8,095	$4,809	$792	$11,970	$2,195	$39,906	$24,271	$26,288	$18,613	$122,216	**$24,657**	$351,732	$532,389	$13,310	$994	$166,353	45%	Yes
0901_ap_1068_200_3	ap	LH							$0	$0	$0				$0				$19,368	$14,085					$15,602	$9,535	$5,665	$933	$12,839	$0	$47,004	$19,772	$30,965	$21,924	$143,956	**$29,043**	$370,690	$624,760	$15,619	$1,166	$237,285	61%	Yes
0901_ap_1068_200_4	ap	LH							$0	$0	$0				$0				$26,137	$19,007					$23,679	$14,471	$8,598	$1,416	$19,486	$0	$71,341	$28,582	$46,997	$33,276	$218,491	**$44,081**	$555,561	$777,562	$19,439	$1,451	$201,111	35%	Yes
0901_ap_1079_100_1	ap	AL							$0	$0	$0				$0							$233,001	$12,785	$36,359	$0	$0	$29,257	$1,817	$23,825	$0	$91,525	$85,407	$60,294	$42,690	$280,307	**$56,552**	$953,820	$649,555	$16,239	$1,213	-$321,717	-33%	No
0901_ap_1081_100_1	ap	LH							$0	$0	$0				$0				$6,198	$4,507					$4,956	$3,029	$1,800	$296	$3,852	$0	$14,933	$6,244	$9,837	$6,965	$45,734	**$9,227**	$117,578	$141,800	$3,545	$265	$20,412	17%	Yes
0901_ap_1082_200_1	ap	LH							$0	$0	$0				$0				$2,270	$1,651					$2,213	$1,352	$803	$132	$2,549	$0	$6,667	$2,780	$4,392	$3,110	$20,418	**$4,119**	$52,456	$56,329	$1,408	$105	$2,359	4%	Yes
0901_gm_1148_100_1	gm	LH	$60,795	$0	$45,934	$19,222	$0	$14,523	$1,544	$4,773	$146,792				$0			$2,543	$46,454	$33,781					$33,342	$20,376	$12,106	$1,994	$19,253	$17,755	$100,453	$84,737	$66,175	$46,855	$307,652	**$62,069**	$1,002,339	$953,655	$23,841	$1,780	-$74,305	-7%	No
0901_gm_1161_100_1	gm	LH	$34,272	$0	$104,027	$15,750	$0	$32,891	$871	$5,921	$193,732				$0			$3,648	$66,621	$48,447					$50,180	$30,666	$18,219	$4,126	$28,976	$4,919	$207,816	$113,912	$136,903	$96,932	$636,465	**$128,407**	$1,769,969	$2,470,613	$61,765	$4,612	$634,267	35%	Yes
0901_gm_1177_100_1	gm	LH	$137,127	$0	$99,974	$43,357	$0	$31,610	$3,483	$10,604	$326,154	$15,458	$0		$15,458			$2,021	$36,905	$26,837					$33,975	$20,763	$12,336	$3,087	$19,618	$15,178	$155,466	$129,825	$102,416	$72,514	$476,136	**$96,061**	$1,544,749	$1,832,882	$45,822	$3,421	$238,890	15%	Yes
0901_lw_0873_200_1	lw	AL							$0	$0	$0				$0							$26,121	$1,303	$3,706	$0	$0	$2,982	$185	$4,105	$0	$9,328	$9,731	$6,145	$4,351	$28,568	**$5,764**	$102,288	$40,630	$1,016	$76	-$62,749	-61%	No
0901_lw_0880_100_1	lw	AL	$0	$0	$111,300	$0	$0	$19,080	$0	$6,405	$136,785				$0							$206,793	$10,359	$29,459	$0	$0	$23,706	$2,106	$31,710	$0	$106,071	$111,728	$69,876	$49,475	$324,858	**$65,540**	$1,168,466	$2,369,268	$59,232	$4,423	$1,137,147	92%	Yes
0901_lw_0904_100_1	lw	AL							$0	$0	$0				$0							$68,699	$3,425	$9,740	$0	$0	$7,838	$850	$9,751	$0	$42,816	$25,417	$28,206	$19,971	$131,129	**$26,455**	$374,297	$472,773	$11,819	$883	$85,774	22%	Yes
0901_lw_0904_200_1	lw	AL	$0	$0	$63,000	$0	$0	$10,800	$0	$3,625	$77,425				$0							$117,894	$5,882	$16,728	$0	$0	$13,461	$836	$16,747	$0	$42,109	$63,090	$27,740	$19,641	$128,964	**$26,019**	$556,536	$406,282	$10,157	$758	-$161,170	-28%	No
0901_lw_0928_100_1	lw	AL	$0	$0	$57,330	$0	$0	$9,828	$0	$3,299	$70,457				$0							$171,181	$8,539	$24,285	$0	$0	$19,542	$1,531	$21,989	$0	$77,106	$80,460	$50,795	$35,965	$236,147	**$47,643**	$845,639	$705,965	$17,649	$1,318	-$158,641	-18%	No
0901_lz_0928_200_1	lz	LH							$0	$0	$0				$0				$25,993	$18,902					$23,929	$14,623	$8,688	$1,431	$25,931	$0	$72,092	$30,281	$47,492	$33,626	$220,790	**$44,545**	$568,322	$498,981	$12,475	$931	-$82,747	-14%	No
0901_lz_0953_200_1	lz	LH							$0	$0	$0				$0				$16,594	$12,067					$15,276	$9,336	$5,546	$914	$15,446	$0	$46,023	$19,050	$30,319	$21,467	$140,953	**$28,437**	$361,428	$151,063	$3,777	$282	-$214,423	-59%	No
0901_lz_1010_200_1	lz	LH							$0	$0	$0				$0				$12,372	$8,997					$11,389	$6,960	$4,135	$681	$9,062	$0	$34,314	$13,581	$22,605	$16,005	$105,090	**$21,202**	$266,393	$724,943	$18,124	$1,353	$439,074	154%	Yes
0901_lz_1019_200_1	lz	LH							$0	$0	$0				$0				$1,922	$1,398					$1,769	$1,081	$642	$106	$1,352	$0	$5,330	$2,096	$3,511	$2,486	$16,325	**$3,294**	$41,311	$99,451	$2,486	$186	$55,467	126%	Yes
0901_lz_1019_200_b	lz	LH							$0	$0	$0				$0				$11,279	$8,202					$8,653	$5,288	$3,142	$518	$6,611	$0	$26,068	$11,071	$17,173	$12,159	$79,838	**$16,107**	$206,107	$532,920	$13,323	$995	$312,495	142%	Yes
0901_lz_1028_200_b	lz	LH							$0	$0	$0				$0				$2,036	$1,481					$1,874	$1,145	$681	$112	$1,373	$0	$5,647	$2,205	$3,720	$2,634	$17,294	**$3,489**	$43,690	$139,954	$3,499	$261	$92,504	195%	Yes
0901_lz_1060_100_1	lz	AL							$0	$0	$0				$0							$93,949	$4,686	$13,327	$0	$0	$10,724	$666	$6,904	$0	$33,547	$33,007	$22,100	$15,648	$102,743	**$20,728**	$358,030	$941,483	$23,537	$1,757	$558,159	146%	Yes
0901_sy_1070_200_1	sy	LH	$74,305	$0	$58,093	$23,494	$0	$18,368	$1,887	$5,921	$182,068				$0		$1,434	$1,333	$24,341	$17,701					$19,607	$11,982	$7,119	$1,984	$11,322	$8,229	$99,908	$72,756	$65,816	$46,600	$305,980	**$61,732**	$939,911	$933,948	$23,349	$1,743	-$31,055	-3%	No
0901_sy_1082_200_1	sy	LH							$0	$0	$0				$0				$46,310	$33,676					$35,032	$21,409	$12,719	$2,096	$32,265	$0	$105,542	$46,500	$69,528	$49,228	$323,236	**$65,213**	$842,753	$1,219,901	$30,498	$2,277	$344,373	39%	Yes
0901_sy_1097_200_1	sy	LH							$0	$0	$0				$0				$29,404	$21,382					$24,061	$14,704	$8,736	$1,439	$13,894	$0	$72,492	$28,792	$47,755	$33,812	$222,015	**$44,792**	$563,279	$762,049	$19,051	$1,422	$178,296	31%	Yes
0901_uz_0806_100_1	uz	LH	$337,714	$0	$81,330	$152,326	$0	$25,715	$8,578	$16,676	$622,339	$20,971	$0		$20,971	$153,918	$4,649	$4,321	$51,682	$37,583					$62,029	$37,907	$22,521	$1,119	$93,127	$157,409	$186,878	$282,708	$123,109	$87,166	$572,339	**$115,470**	$2,637,247	$3,249,654	$81,241	$6,066	$525,100	19%	Yes
0901_uz_0823_100_1	uz	LH	$345,411	$0	$98,893	$123,796	$0	$31,268	$8,773	$17,980	$626,122	$18,701	$0		$18,701	$129,014	$3,897	$3,622	$43,320	$31,503					$52,189	$31,894	$18,949	$1,259	$75,803	$114,431	$210,272	$258,896	$138,520	$98,077	$643,985	**$129,924**	$2,630,377	$3,796,541	$94,914	$7,087	$1,064,164	39%	Yes
0901_uz_0823_100_2	uz	LH	$0	$0	$66,604	$0	$0	$21,059	$0	$2,979	$90,642				$0	$315,391	$9,526	$8,853	$105,902	$77,012					$123,120	$75,242	$44,703	$2,463	$178,831	$0	$411,298	$181,509	$270,950	$191,842	$1,259,654	**$254,136**	$3,601,073	$7,598,911	$189,973	$14,185	$3,793,681	100%	Yes
0901_uz_0842_100_2	uz	LH	$0	$0	$40,125	$0	$0	$12,687	$0	$1,794	$54,606	$180,045	$0		$180,045		$3,796	$3,528	$42,195	$30,685					$49,153	$30,039	$17,847	$887	$65,679	$12,852	$148,087	$124,498	$97,555	$69,072	$453,536	**$91,501**	$1,475,559	$2,754,659	$68,866	$5,142	$1,205,091	78%	Yes
0901_uz_0859_100_3	uz	LH							$0	$0	$0				$0				$551	$400					$882	$539	$320	$16	$1,170	$0	$2,657	$983	$1,750	$1,239	$8,138	**$1,642**	$20,288	$36,562	$914	$68	$15,292	72%	Yes
0901_uz_0873_100_2	uz	LH							$0	$0	$0				$0				$20,791	$15,119					$25,634	$15,666	$9,307	$462	$33,986	$0	$77,229	$30,653	$50,876	$36,022	$236,525	**$47,719**	$599,991	$905,832	$22,646	$1,691	$281,505	45%	Yes
0901_uz_0895_200_1	uz	LH							$0	$0	$0				$0				$20,063	$14,590					$26,525	$16,210	$9,631	$479	$31,819	$0	$79,913	$30,235	$52,644	$37,274	$244,745	**$49,377**	$613,505	$2,579,353	$64,484	$4,815	$1,896,549	278%	Yes
0901_uz_0895_200_2	uz	LH							$0	$0	$0				$0				$12,582	$9,150					$18,234	$11,143	$6,620	$329	$21,873	$0	$54,935	$20,255	$36,189	$25,623	$168,246	**$33,944**	$419,124	$665,032	$16,626	$1,241	$228,040	52%	Yes
0901_uz_0895_200_3	uz	LH							$0	$0	$0				$0				$41,519	$30,193					$50,380	$30,789	$18,292	$909	$60,436	$0	$151,784	$58,920	$99,990	$70,797	$464,859	**$93,786**	$1,172,653	$1,205,747	$30,144	$2,251	$701	0%	Yes
0901_vg_0823_200_2	vg	LH	$0	$0	$145,773	$0	$0	$46,090	$0	$6,519	$198,383				$0	$68,584	$2,071	$1,925	$35,163	$25,571					$23,122	$14,131	$8,395	$1,383	$33,585	$31,035	$69,662	$94,965	$45,891	$32,493	$213,350	**$43,043**	$942,752	$977,421	$24,436	$1,825	$8,409	1%	Yes
0901_vg_0823_210_1	vg	LH							$0	$0	$0				$0	$11,526	$348	$324	$5,910	$4,297					$4,001	$2,445	$1,453	$653	$5,811	$0	$32,864	$6,396	$21,650	$15,329	$100,651	**$20,306**	$233,963	$271,808	$6,795	$507	$30,543	13%	Yes
0901_lz_1045_200_1	lz	LH							$0	$0	$0				$0				$74,380	$54,089					$60,866	$37,197	$22,099	$3,641	$40,946	$0	$183,376	$74,302	$120,802	$85,532	$561,612	**$113,306**	$1,432,147	$2,936,412	$73,410	$5,481	$1,425,373	94%	Yes
0901_lz_1052_200_1	lz	LH							$0	$0	$0				$0				$8,012	$5,826					$6,322	$3,863	$2,295	$378	$4,096	$0	$19,046	$7,803	$12,547	$8,884	$58,331	**$11,768**	$149,171	$659,763	$16,494	$1,232	$492,866	295%	Yes
site_hg_surf	hg								$0	$0	$0				$0																				$313,365	**$63,222**	$376,586	$1,534,062	$38,352	$2,864	$1,116,261	267%	Yes
site_lg_surf	lg								$0	$0	$0				$0																				$922,226	**$186,060**	$1,108,286	$1,052,936	$26,323	$1,965	-$83,638	-7%	No
site_uz_0895_100	uz	LH							$0	$0	$0				$0				$0	$0					$28,644	$17,505	$10,400	$517	$34,361	$0	$86,297	$23,167	$56,849	$40,251	$264,295	**$53,322**	$615,607	$2,207,277	$55,182	$4,120	$1,532,368	227%	Yes
site_uz_0910_100	uz	LH							$0	$0	$0				$0				$0	$0					$21,592	$13,195	$7,840	$390	$24,081	$0	$65,051	$17,003	$42,854	$30,342	$199,229	**$40,195**	$461,771	$617,809	$15,445	$1,153	$139,440	29%	Yes
site_uz_0910_200	uz	LH							$0	$0	$0				$0				$0	$0					$2,087	$1,276	$758	$38	$2,328	$0	$6,289	$1,644	$4,143	$2,933	$19,260	**$3,886**	$44,641	$55,925	$1,398	$104	$9,782	21%	Yes
site_uz_0963_300	uz	LH							$0	$0	$0				$0				$0	$0					$4,313	$2,636	$1,566	$78	$4,023	$0	$12,995	$3,197	$8,561	$6,061	$39,799	**$8,030**	$91,259	$150,345	$3,759	$281	$55,047	58%	Yes
site_lz_1028	lz	LH							$0	$0	$0				$0				$0	$0					$2,602	$1,590	$945	$156	$1,906	$0	$7,840	$1,824	$5,165	$3,657	$24,011	**$4,844**	$54,540	$182,083	$4,552	$340	$122,651	206%	Yes
site_ap_1046	ap	LH							$0	$0	$0				$0				$0	$0					$3,844	$2,349	$1,396	$230	$3,474	$0	$11,582	$2,862	$7,630	$5,402	$35,472	**$7,157**	$81,399	$116,249	$2,906	$217	$31,727	38%	Yes
site_ap_1068	ap	LH							$0	$0	$0				$0				$0	$0					$3,137	$1,917	$1,139	$188	$2,581	$0	$9,450	$2,271	$6,225	$4,408	$28,942	**$5,839**	$66,097	$101,173	$2,529	$189	$32,358	47%	Yes
0901_ap_1046_200_2	ap	LH							$0	$0	$0				$0				$25,218	$18,338					$18,636	$11,389	$6,766	$1,115	$16,841	$0	$56,145	$24,910	$36,986	$26,188	$171,951	**$34,691**	$449,172	$522,760	$13,069	$976	$59,543	13%	Yes
0901_ap_1068_200_2	ap	LH							$0	$0	$0				$0				$6,816	$4,957					$6,085	$3,719	$2,209	$364	$5,007	$0	$18,333	$7,389	$12,077	$8,551	$56,147	**$11,328**	$142,982	$136,304	$3,408	$254	-$10,340	-7%	No
0901_ap_1081_200_5	ap	AL							$0	$0	$0				$0							$31,520	$1,685	$4,791	$0	$0	$3,855	$239	$3,111	$0	$12,060	$11,454	$7,945	$5,625	$36,934	**$7,452**	$126,670	$131,089	$3,277	$245	$897	1%	Yes
0901_lw_0953_100_1	lw	AL	$0	$0	$18,900	$0	$0	$3,240	$0	$1,088	$23,228				$0							$58,686	$2,927	$8,324	$0	$0	$6,698	$528	$7,032	$0	$26,611	$27,221	$17,531	$12,412	$81,500	**$16,443**	$289,142	$247,166	$6,179	$461	-$48,616	-16%	No
0901_lw_0976_100_1	lw	AL							$0	$0	$0				$0							$131,128	$6,540	$18,599	$0	$0	$14,966	$930	$14,285	$0	$46,817	$47,246	$30,842	$21,837	$143,385	**$28,928**	$505,502	$479,803	$11,995	$896	-$38,590	-7%	No
0901_lw_1019_100_1	lw	AL							$0	$0	$0				$0							$44,667	$2,226	$6,330	$0	$0	$5,094	$316	$4,041	$0	$15,935	$15,882	$10,498	$7,433	$48,804	**$9,846**	$171,073	$175,899	$4,397	$328	$101	0%	Yes
0901_lz_1010_200_b	lz	LH							$0	$0	$0				$0				$19,506	$14,185					$14,632	$8,942	$5,313	$875	$11,642	$0	$44,083	$19,029	$29,041	$20,562	$135,011	**$27,238**	$350,060	$1,023,679	$25,592	$1,911	$646,117	171%	Yes
0901_uz_0842_100_1	uz	LH	$0	$0	$54,986	$0	$0	$17,385	$0	$2,459	$74,830				$0		$4,650	$4,322	$51,694	$37,592					$61,720	$37,718	$22,409	$1,453	$82,470	$0	$242,582	$96,003	$159,805	$113,148	$742,940	**$149,889**	$1,883,224	$2,520,416	$63,010	$4,705	$569,477	29%	Yes
0901_uz_0859_100_1	uz	LH							$0	$0	$0				$0		$2,324		$25,837	$18,789					$31,188	$19,060	$11,324	$563	$41,357	$0	$93,962	$38,122	$61,899	$43,827	$287,772	**$58,058**	$734,082	$143,569	$3,589	$268	-$594,370	-81%	No
0901_uz_0873_100_1	uz	LH							$0	$0	$0				$0				$12,063	$8,772					$14,591	$8,917	$5,298	$263	$19,345	$0	$43,960	$17,548	$28,959	$20,504	$134,632	**$27,162**	$342,015	$167,228	$4,181	$312	-$179,280	-52%	No
0901_uz_0873_200_1	uz	LH							$0	$0	$0				$0				$10,207	$7,422					$13,983	$8,546	$5,077	$252	$18,539	$0	$42,129	$16,224	$27,753	$19,650	$129,025	**$26,031**	$324,840	$529,966	$13,249	$989	$190,888	56%	Yes
0901_uz_0873_200_2	uz	LH							$0	$0	$0				$0				$9,298	$6,762					$12,087	$7,387	$4,389	$218	$16,025	$0	$36,416	$14,232	$23,989	$16,985	$111,528	**$22,501**	$281,816	$74,360	$1,859	$139	-$209,454	-74%	No
0901_uz_0873_200_3	uz	LH							$0	$0	$0				$0				$20,909	$15,205					$27,817	$17,000	$10,100	$502	$36,880	$0	$83,807	$32,540	$55,209	$39,090	$256,669	**$51,783**	$647,510	$650,738	$16,268	$1,215	-$14,255	-2%	No
0901_uz_0895_400_1	uz	LH	$19,454	$0	$22,967	$6,151	$0	$7,262	$494	$1,897	$58,226				$0				$14,262	$10,371					$16,317	$9,972	$5,924	$383	$19,573	$4,640	$63,998	$35,392	$42,160	$29,851	$196,004	**$39,544**	$546,617	$564,365	$14,109	$1,053	$2,585	0%	Yes
0901_uz_0963_300_1	uz	LH							$0	$0	$0				$0				$10,722	$7,797					$12,823	$7,836	$4,656	$231	$11,959	$0	$38,633	$14,197	$25,450	$18,019	$118,317	**$23,871**	$294,511	$4,467	$112	$8	-$290,164	-98%	No

20. OTHER RELEVANT DATA AND INFORMATION

No other information or explanation necessary.

21. INTERPRETATION AND CONCLUSIONS

While robust, this resource has seen a marked downgrade in overall ounces from previous resource reports. This has been caused by evolving interpretations, resource classification and mining parameters. Incremental increases in ounces have been added, particularly in Upper Zone but not enough to offset the drop in ounces from last resource.

The resource drilling focus for 2008 was primarily on the newly named Voyager area of the orebody which is primarily below where the Apollo fault intersects the orebody. This area is structurally very complex and has had a significant impact on the ability to delineate the orebody at depths 300 metres below surface.

The Paulsens project currently has less than one years mine life in reserves; however increased understanding of ore body complexities are revealing opportunities down plunge as well as 260,000 tonnes at 9g/t of inferred category that can be upgraded.

22. RECOMMENDATIONS

A significant amount of drilling is still required to bring the rest of the remaining resource up from an inferred to an indicated status to be included into Reserves. This will occur in the first part of 2009 with a budgeted allocation of $1,000,000 with the intent to extend the reserves to the 750 Rl or 50 metres below the current reserve limit at 800 Rl.

While still open at depth, the ore body is restricted to an area bounded by the Gabbro offsets and reliant on favourable fold positions. This will be tested in 2009 by various programs separate to the programs mentioned in the above paragraph budgeted at a further A$1,000,000. It is anticipated that this drilling will test the orebody to depths 100 to 150 metres below the current resource and potentially extended inferred resources to 600 Rl.

A detailed structural review has commenced with the aim of increasing confidence in the down dip resource potential. Drilling in 2008 indicated the structural complexities of the mineralisation was increasing and this structural review will assist in targeting mineralisation at depth and ultimately extending the resources and mine life.

23. REFERENCES

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Abbott, 2009. Resource Estimation for the Paulsens Gold Mine, Western Australia. March 2009

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24. DATE AND SIGNATURES

The effective date of this Technical report, titled **"**TECHNICAL REPORT MINERAL RESOURCES AND RESERVES FOR THE PAULSENS GOLD MINE" is March 2009.

Signed,

"signed" or "signed and sealed"

Jonathon Abbott (Hellman and Schofield)
Per Scrimshaw (Creative Mined)
Craig Jones (Kadgie Mining)
Brook Ekers (Intrepid Mines).
Don Russell (Intrepid Mines)

Dated March 30, 2009

25. ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

Mining Method

The current mining method at Paulsens utilises long hole retreat bench stoping methods and air leg mining techniques. Structurally controlled boundaries are generally used and empirical design data used to determine hydraulic radius for stope geometry controls.

Metalurgical Process

The metallurgical process selected involves multi-stage crushing, ball milling, thickening of grinding circuit product, cyanide leaching using the carbon-in-leach ("CIL") process, disposal of the leach residue to the tailings storage facility, cold cyanide and then hot acid washing of loaded carbon to remove copper and then nickel, elution of the washed carbon to remove gold, electrowinning of gold from loaded eluate and smelting of cathodes and slimes to produce gold bullion. The 2009 production forecast is to continue mining at current rates for an annual production of approximately 338,000 tonnes for 76,000 ounces produced.
The ore processed on site is recovered using industry standard practices. There are no unfavourable elements of the ore which would constitute any other form of recovery processes to be utilised. Mill recovery has been in excess of 96% and dore purity has been at times above 92%.

Commodity Markets

As a result of the operation of the Paulsens Gold Mine, the Company produces gold through its operations. There is a world wide gold market into which the Company can sell its gold. Gold is a metal that is traded on world markets, with benchmark prices generally based on the London market (London fix). Given the nature of the market for gold, the Company is not dependent on any particular purchaser.

Contracts

Paulsens uses the services of contractors for mining, crushing, power generation, flights and camp accommodation to operate the Paulsens Gold mine. All major contracts are secure and are reviewed in relation to current industry standards on a regular basis.

Environmental Considerations

Permitting approvals for the establishment of the Paulsens Gold Mine were received from the State Mining Engineer and the Department of Environment in November 2004. Other permits or licences received to allow mining to proceed included 5C Groundwater Well Licences and modifications to the Prescribed Premises Licence.

Rehabilitation bonds are lodged with the DMP as shown below.

SUMMARY OF REHABILITATION BONDS LODGED WITH DMP AND CLOSURE DEMOLITION ESTIMATES FOR THE PAULSEN'S GOLD MINE AT DECEMBER 2008 (MORATORIUM)						
Lease/Tenure	Area of Disturbance (ha)		Closure Liability $AUD		Current Bonds $AUD	Comments
	Approved	Actual	Demo	Rehab		
M08/99 - Mine and Plant	20	20	796,000	95,000	95,000	Developments have been bonded using the new DMP (2009) Moratorium Bond Rates. These rates are the same as current bonding rates imposed in 2007. Demo costs relate to steel framed buildings, vent fans, Gold Plant, foundations, WWTP.
M08/196 – TSF – Village	42	42	77,000	348,000	348,000	
L08/14 - Access Roads	5.6	5.6	0.0	17,000	17,000	Main access road will remain at closure as Pastoral Road.
TOTAL	**67.6**	**67.6**	**873,000**	**460,000**	**460,000**	Moratorium to December 2009 means Bond Rates will not change following AER review early in 2009.

Taxes

Royalties are paid to the state government at a rate of 2.5% of the value of ounces produced expressed in Australian Dollars .In addition the local indigenous community are paid a royalty $A2.00 / ounces produced.

The Australian Federal tax rate applicable to the project is 30%. Tax losses may be carried forward indefinitely subject to satisfying proscribed tests.

Suppliers of goods and services include GST at 10% on the invoice value of taxable supplies made to Paulsens. The GST incurred on business inputs can be recovered from the Federal Government by filing monthly returns.

Capital and Operating Costs

The Paulsens Gold Mine has established cost models that have been used to estimate both operating and capital costs for the 2009 calendar year and are expressed in Australian Dollars. The operation has established service contracts and supply agreements in place. The following table outlines the operating expenditure estimates for 2009 by quarter.

Expenditure	Quarter 1	Quarter 2	Quarter 3	Quarter 4	Full Year 2009
Mining costs	5,851	5,546	5,666	5,545	22,609
Geology costs	541	480	583	701	2,305
Processing costs	3,829	3,841	3,873	3,895	15,438
Royalties and refining costs	524	451	480	453	1,908
Administration Paulsens	988	1,006	985	974	3,953
Stock movement	(1,086)	(218)	(308)	545	(1,068)
Total operating costs	10,646	11,106	11,279	12,113	45,145

Table 13 Operating cost estimates for 2009

The following table outlines the capital expenditure estimates for 2009 by quarter.

Capital Expendicture	Quarter 1	Quarter 2	Quarter 3	Quarter 4	Full Year 2009
Paulsens-PP&E	(951)	(634)	(265)	(18)	(1,867)
Paulsens - Development	(2,636)	(3,673)	(3,295)	(1,167)	(10,771)
Paulsens - Resource Extension	(748)	(758)	(360)	(360)	(2,226)
Total Capital costs	(4,335)	(5,065)	(3,919)	(1,544)	(14,864)

Table 14 Capital cost estimates for 2009

2009 PAULSENS BUDGET ASSUMPTIONS		Quarter 1	Quarter 2	Quarter 3	Quarter 4	Full Year 2009
Exchange Rate						
Exchange Rate	US$/A$	0.70	0.70	0.70	0.70	0.70
Commodity Prices						
Gold Price	US$/oz	750.00	750.00	750.00	750.00	750.00
Gold Price	A$/oz	1,071.4	1,071.4	1,071.4	1,071.4	1,071.43
Westpac crude oil (USD/bbl)		133	133	133	133	133
Diesel Fuel	A$/l	1.20	1.20	1.20	1.20	1.20

Table 15 Budget Assumption 2009

Economic Analysis

The Paulsens Gold Mine current economic analysis expressed in Australian Dollars indicates that the 2009 calendar year will deliver both robust cash flow and profit by treating the remaining mining reserves (8 months) and some of the high confidence resource (4 months). The mine life is expected to continue beyond 2009 with further resource drilling to be completed in 2009 and extend the economic viability of the operation well into 2010 and beyond.

Revenue	Quarter 1	Quarter 2	Quarter 3	Quarter 4	Full Year 2009
Gold revenue	21,427	19,384	20,730	19,446	80,986
Hedging income	-	-	-	-	-
Silver revenue	15	15	15	15	60
Interest income	9	9	9	9	36
Other income	-	-	-	-	-
Total Income	21,451	19,408	20,754	19,470	81,082
Total operating costs	10,646	11,106	11,279	12,113	45,145
Total Capital costs	(4,335)	(5,065)	(3,919)	(1,544)	(14,864)
Depreciation and Amortisation Paulsen	4,795	4,168	4,442	3,870	17,275
Net (Loss)/Profit before Tax	6,009	4,133	5,033	3,487	18,663
Cash Flow from Paulsens Activities	6,469	3,236	5,555	5,813	21,074

Table 16 Economic Analysis Paulsens Gold Mine 2009

As the table indicates the current project is extremely robust. It is sensitive to gold price and exchange rate fluctuations with a 10% reduction in Australian Dollar gold price having an impact to reduce revenue by AUD 8 Million and therefore reduce cash flow by 40% to AUD 13 Million. It is important to note that at the timing of writing this report Australian Dollar Gold Price is presently 25% above the assumed $750 US and Australian Dollar USD Dollar exchange rate of 0.70 and therefore potentially delivering AUD 20 Million in extra revenue and cash flow.

Jonathon Abbott
Hellman & Schofield Pty Ltd

102 Colin Street, West Perth, Western Australia, AUSTRALIA
Telephone: 618 9485 0403, Fax: 612 9485 0406
Email: jon@hellscho.com.au

CERTIFICATE of AUTHOR

I, Jonathon Abbott, BASc, MAusIMM, do hereby certify that:

a. I am a Consulting Geologist with:
Hellman & Schofield Pty Ltd
102 Colin Street
West Perth, Western Australia
AUSTRALIA

b. This Certificate applies to the technical report titled "Technical Report Mineral Resources and Reserves for the Paulsens Gold Mine Pilbara, Western Australia" prepared for Intrepid Mines Ltd dated March 2009 (the "Technical Report") relating to Paulsens property.

c. I graduated with a Bachelor of Applied Science in Applied Geology from the University of South Australia in 1990. I am a member of the Australasian Institute of Mining and Metallurgy. I have worked as a geologist for a total of 17 years since my graduation from university. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

d. I have been involved with the Paulsens Project since August 2007, and visited the project site from the 11th to 13th of November 2008.

e. I am responsible for item 19 of the Technical Report.

f. I am an independent issuer as defined in Section 1.4 of the Instrument.

g. I have not had prior involvement with the property that is the subject of the Technical Report.

h. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

i. As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 26th day of March, 2009.

Jonathon Abbott, BASc Appl. Geol MAusIMM

CERTIFICATE

To Accompany the Report Entitled
"Technical Report, Mineral Resources and Reserves for the Paulsens Gold Mine,
Pilbara, Western Australia"
dated March 2009

I, Per Scrimshaw, do hereby certify that:

(a) I am a Professional Mining Engineer, employed by Mintech Services Pty Ltd T/A Creative Mined Enterprises and residing in Fremantle, Western Australia. I have been practicing in the mining industry since 1998;

(b) This certificate applies to the report entitled "Technical Report, Mineral Resources and Reserves for the Paulsens Gold Mine, Pilbara, Western Australia", March 2009;

(c) I hold a Bachelor of Engineering degree in Mining from the University of Queensland, Brisbane awarded 1998 and have been practicing as a professional since graduation. I am in good standing as a registered member (number 204357) of the Australasian Institute of Mining and Metallurgy since 2003. By reason of experience and education, I fulfill the requirements of a Qualified Person as set out in National Instrument 43-101.

(d) I have visited the property several times, the most recent being the 12th and 13th March 2009;

(e) I am the sole author of the Ore Reserves section of this report;

(f) I am an independent of the issuer;

(g) I have visited the property several times and authored previously disclosed Ore Reserve estimates related to the property;

(h) I have read Instrument 43-101 and the technical report has been prepared in compliance with this Instrument; and

(i) that, as of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Fremantle, Western Australia this 26th day of March 2009

Per Scrimshaw

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CERTIFICATE

To Accompany the Report Entitled
"Technical Report Mineral Resources and Reserves for the Paulsens Gold Mine, Pilbara, Western Australia"
dated March 2009

I, Craig Jones, do hereby certify that:

(a) Craig Jones, 19 Lady Eliot Drive, Agnes Water, QLD and Strategic Planning Manager of Paulsens Gold Mine;

(b) Technical Report Mineral Resources and Reserves for the Paulsens Gold Mine, Pilbara, Western Australia, March 2009;

(c) Have over 13 years experience in the mining industry, I have a Bachelor of Engineering (Mining Engineering) from the University of Ballarat and am a "qualified person" for the purposes of Instrument 43-101;

(d) Personal inspection of the property on a regular fly in fly out roster since January 2007;

(e) I am responsible for sections 3 and 25 within the report;

(f) I am an employee of Kadgie Mining Pty Ltd and offer consulting services to Intrepid Mines the issuer as described in section 1.4 of Instrument 43-101;

(g) Registered Underground Manager of Paulsens Gold Mine between January 2007 and November 2008, Strategic Planning Manager since November 2008;

(h) I have read Instrument 43-101 and the technical report has been prepared in compliance with this Instrument; and

(i) that, as of the date of the certificate, to the best of the qualified person's knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Paulsens Gold Mine this 30th day of March 2009

Craig Jones

CERTIFICATE

To Accompany the Report Entitled
"Technical Report Mineral Resources and Reserves for the Paulsens Gold Mine, Pilbara Western Australia" as at December 31, 2008 dated March 2009

I, Brook Ekers, do hereby certify that:

(a) Brook Ekers, of 90 Barridale Drive Kingsley, Perth, Paulsens Geology Manger with Intrepid;

(b) Technical Report Mineral Resources and Reserves for the Paulsens Gold Mine, Pilbara Western Australia"as at December 31, 2008 dated March 2009

(c) I have 15 years relevant experience in gold mineralisation, in the Yandal Belt and Pilbara of Western Australia and am a member of the Australian Institute of Geoscientists and am a "qualified person" for purposes of Instrument 43-101;

(d) Personnel inspection of the property has been as rostered on (8 days on 6 days off) from May 2004.

(e) I am responsible for all items bar 19 and 25.

(f) I am an employee of Intrepid mines, the issuer as described, 1.4 of Instrument 43-101;

(g) Prior involvement in the property has been as a a siute employee for 4 years (since project commencement);

(h) I have read Instrument 43-101 and the technical report has been prepared in compliance with this Instrument; and

(i) that, as of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Kingsley, Perth this 27th day of March 2009

[Brook Ekers]

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CERTIFICATE

To Accompany the Report Entitled
"Technical Report
Mineral Resources and Reserves for the Paulsens Gold Mine, Pilbara, Western Australia"
Dated MARCH, 2009

I, Donald Russell, do hereby certify that:

(a) Donald Russell, 12 Acheron Road, San Remo, WA and General Manager of Paulsens Gold Mine;

(b) Technical Report Mineral Resources and Reserves for the Paulsens Gold Mine, Pilbara, Western Australia, March 2009

(c) Have had 20 years experience in the mining industry with over 10 years in management positions, I have a Bachelor of Engineering (Mining Engineering) from the Western Australia School of mines and am a "qualified person" for the purposes of Instrument 43-101;

(d) Personal inspection of the property on a regular fly in fly out roster since May 2008;

(e) I am responsible for items in section 25;

(f) I am a employee of Intrepid Mines the issuer as described in section 1.4 of Instrument 43-101;

(g) General Manager since May 2008 of the Paulsens Gold Mine

(h) I have read Instrument 43-101 and the technical report has been prepared in compliance with this Instrument; and

(i) that, as of the date of the certificate, to the best of the qualified person's knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Paulsens Gold Mine this 27th day of March 2009

[Donald Russell]

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