

ASX ANNOUNCEMENT

29 August 2018

PRE-FEASIBILITY STUDY (PFS) RESULTS FOR THE JAURDI GOLD PROJECT

HIGHLIGHTS

- Pre-Feasibility study results confirm Jaurdi Gold Project to be a technically and economically viable project at a 500,000tpa processing capacity;
- Processing an estimated 2.5Mt @ 1.9g/t (148koz)² will deliver 126koz of recovered gold.
- Pre-production capital cost of \$21.4M;
- Pre-production payback period 11 months;
- Ore Reserves provide a mine life of 5 years which is likely to increase with the inclusion of the Black Cat Resources and exploration upside;
- Forecast Life of Mine (LOM) revenue \$208.5M and surplus operating cash flow of \$98.4M at \$1,650/oz. gold price¹;
- Development of one shallow, low strip ratio open pit provides a low mining cost with an extremely low pre-stripping cost;
- LOM operating cash cost (C1) of A\$830/oz³;
- LOM All-In-Sustaining Cost (AISC) of A\$870/oz⁴;
- NPV A\$70.49M (before tax);
- IRR of 75% (before tax);
- Carried forward Australian tax losses of approximately \$17M up to and including 30 June

¹ Based on production of 148,000oz at \$US1,200 gold price, A\$/US\$ exchange rate of \$0.73. All amounts in A\$ unless otherwise stated.

² 100% of the material in the mine plan is classified as an Ore Reserve.

³ C1 operating costs include all mining, processing costs and royalties.

⁴ AISC includes C1 costs plus refining and sustaining capital.

^{*}Differences may occur due to rounding.



Beacon Minerals Limited ("Beacon" or the "Company") is pleased to announce the completion of its PFS for the Jaurdi Gold Project. The PFS demonstrates an economically and technically viable project with considerable upside. The key financial parameters are tabled below:

Key Financial Parameter*	A\$1,650 oz		
NPV (A\$M)	\$70.49m		
Revenue (A\$M) ¹	\$208.5m		
Operating Costs (A\$M)	\$89.5m		
Royalties	\$15.5m		
Cashflow (A\$M)	\$98.4m		
Initial Capital Costs (A\$M)	\$21.4m		
Sustaining Capital Costs (A\$M)	\$5.0m		
EBITD (A\$M)	\$103.5m		
C1 Cash Cost (A\$oz)	\$830		
AISC (A\$oz)	\$870		
IRR (%)	75.04%		
Payback (months)	11 months		

^{1.} Includes year 5 Lost Dog mill feed will be supplemented wiith approx. 30,000 tonnes from the Black Cat North open pit to fill the mill to its 500,0000 tpa capacity.

*Pre-Feasiblity Parameters – Cautionary Statement

The PFS is based on Proved and Probable Ore Reserves derived from Measured and Indicated Mineral Resources respectively. No Inferred Mineral Resource was included in the estimation of Ore Reserves. The PFS was prepared to an overall level of accuracy of +10%/-5%. It is based on material assumptions outlined elsewhere in this announcement and in Appendix 1 Material Assumptions PFS. The Company has concluded it has a reasonable basis for providing the forward-looking statements included in this announcement.

Managing Director Graham McGarry said:

"The release of the PFS for the Jaurdi Gold Project is an important milestone for the Company. The results from the PFS demonstrates Jaurdi is a valuable gold resource and an economically viable mining operation can be developed.

"The PFS has concluded that the Jaurdi Gold Project will enjoy low pre-production and operating costs which underpin a low risk, high margin gold operation with a short payback period and strong free cash flow.

"Beacon will work diligently on producing gold in the first half of 2019 from this new and exciting gold environment and and we will continue to explore the several "high priority" exploration targets we have identified from previous high level exploration activities.

"On behalf of the Board, we thank our staff, contractors, consultants and advisors for their hard work in completing the PFS."

BEACON MINERALS LIMITED ACN 119 611 559



Overview

The Company engaged Minecomp Pty Ltd ("Minecomp"), a Kalgoorlie based company, to carry out the PFS at the Jaurdi Gold Project ("Jaurdi Project"), producing a high level mining and processing schedule.

The Jaurdi Project is located 35km north west of Coolgardie and approximately 75km west of Kalgoorlie. The area is well serviced by infrastructure including a network of high quality roads, Kalgoorlie airport with regular services to Perth and an established mining supply network.

The PFS investigates the potential economic viability of the Jaurdi Project on the mining and on-site treatment of the Lost Dog Resource

Independent JORC 2012 estimates of the Mineral Resource at the Jaurdi Project by BM Geological Services (BMGS) total 2.88Mt @ 1.8g/t for 163.1koz of contained gold (refer ASX Announcement 12 July 2017).

The PFS envisages an open pit mine that will deliver material to a new, 500,000tpa capacity carbon-in-pulp (CIP) gold treatment facility at the Jaurdi Project.

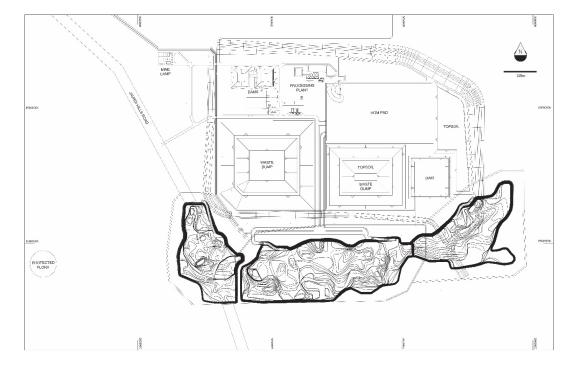


Figure 1: Site Plan - Jaurdi Gold Project

The open pit will be mined utilizing conventional open pit methods with 90t hydraulic excavator, a fleet of articulated dump trucks and ancillary mining equipment.



The mining strategy is focused on delivering an appropriate blend of ore to the process plant so as to optimise plant recoveries and throughput.

The TSF strategy is based upon depositing tailings into the voids left by open pit mining.

First gold production, based upon the PFS production forecast, is expected in the first half of 2019.

The following results relates to the work carried out by Mr Gary McCrae from Minecomp. All outputs relating to these works are dated May 2018.

Mineral Resource

The Jaurdi Project overlies a portion of the Bali Monzo granite immediately adjacent to the Jaurdi Hills-Dunnsville greenstone sequence. The gold mineralisation is hosted in either a bleached, siliceous siltstone or an interbedded clay and siltstone unit.

The Mineral Resource was estimated by BMGS and announced by the Company in June 2017, see Table 1.

Tonnes Au g/t Au Classification (Kt) (g/t)(kOz) Measured 30 1.6 1.5 Indicated 2,752 1.8 158.4 Inferred 101 3.2 1.0 **Total** 2,883 1.8 163.1

Table 1 – Lost Dog Mineral Resource

Calculations have been rounded to the nearest 1,000t, 0.1g/t grade and 100 ounces

For further details see JORC Code 2012 Edition – Table Report Template Sections 1, 2, 3 starting on page 194 of the report attached to the Appendix of this announcement.

Mining and Metallurgical Factors and Assumptions

Mining at the Jaurdi Project is to be performed using conventional open pit mining techniques. Mining equipment will comprise articulated dump trucks, matching 90t hydraulic excavator with additional ancillary equipment rounding out the fleet.

Beacon envisages that load and haul activities will be undertaken by owner miner operators using a mixture of owned and dry hired equipment. All drill and blast and grade control drilling will be undertaken by contractors. All technical and managerial direction will be governed by Beacon.

The Lost Dog, June 2017 Resource was imported into Whittle pit optimisation software. The optimisation analysis included inputs from Beacon's Executive Directors and external consultants. These input parameters comprised contractor estimates based upon experience and were inclusive



of all on-site operating costs. Where applicable these costs, were reflective of the use of articulated trucks and matching equipment. Milling costs were reflective of treatment at an on-site milling facility.

The metallurgical recovery used in this study is based upon testwork conducted by ALS Metallurgy Perth, Bureau Veritas Kalgoorlie and the results of a 4,625t trial parcel of Lost Dog ore processed at a nearby custom milling facility. The 85% recovery used is at the lower end of the range of recoveries established from the testwork.

Geotechnical parameters utilised were based upon the recommendations of Tim Green of Green Geotechnical.

The orebody geometries (shallow, flat lying and nominally 1,200m long, 180m wide and 12m thick) resulted in the application of a mining dilution factor of 2% at 0.00g/t and a mining recovery of 98%. Given these orebody dimensions no allowances were made for minimum mining widths.

Optimisation analysis was conducted for a gold price range of A\$1,000/oz to A\$2,000/oz in A\$50/oz increments, with \$1,650/oz considered to be the "Base Case" gold price.

Inferred Resources were assigned a grade of 0.00g/t and hence categorized as waste material throughout the course of this study.

A state royalty of 2.5% is payable on the average monthly price as advised by the DMIRS Royalties Branch. No allowance has been made for the exemption of this royalty on the first 2,500 ounces produced in each financial year.

A third-party royalty of \$80/oz recovered is also payable.

Mine Design and Ore Reserve

Open pit mining methods are well known and widely used in the local mining industry. The design was focused on maximizing profitability from the optimised Whittle shells. The optimum and most profitable outcome was to design the pit ramp to single lane at a 1 in 6 gradient which suited the 40t articulated dump truck fleet. This ramp configuration being one which Beacon management has had significant historical exposure to.



The detailed open pit mine design produces a Maiden Ore Reserve of:

Table 2 – Jaurdi Project Ore Reserve

Ore Reserve Category	Tonnes	Au (g/t)	Au (oz)
Proved	27,000	1.6	1,400
Probable	2,443,000	1.9	147,100
Total	2,470,000	1.9	148,500

Notes:

- Calculations have been rounded to the nearest 1,000t, 0.1g/t grade and 100 ounces
- For further details see JORC Code 2012 Edition Table Report Template Sections 1, 2, 3 and 4 starting on page 194 of the report attached to the Appendix of this announcement.

For the purpose of the Ore Reserve Estimate, a marginal cut-off grade of 0.6g/t was calculated based upon an assumed gold price of Au\$1,650/oz and the applicable Western Australian State Government and 3rd Party Royalties, ore/waste cost differentials, processing and haulage costs and metallurgical recovery.

Ore Processing

Beacon have acquired many of the key processing components for the construction of a 500,000tpa processing plant. Major equipment acquired to date is as follows:

- SAG mill 1500kW, 4m x 6m;
- Ball mill 450kW, 3m x 4m;
- Adsorption tanks 6 x 200m³;
- Leach tank agitators and superstructure to suit 3 x 630m³ tanks;
- Radial stacker 35m;
- Coarse ore bin and feeder;
- MCC switch rooms; and
- Process slurry pumps.

The process plant general arrangement is shown in Figure 2 and the process flow diagram for the 0.5Mtpa processing plant is illustrated in Figure 3. All main elements that comprise the processing plant are typical of conventional CIP plants operating throughout the WA Goldfields. The treatment circuit has been designed to produce a grind of P100 106 μ m and a leach retention time of 15 hours. The Company has made a financial provision for an additional Leach Tank if required.



Figure 2: Process Plant Arrangement

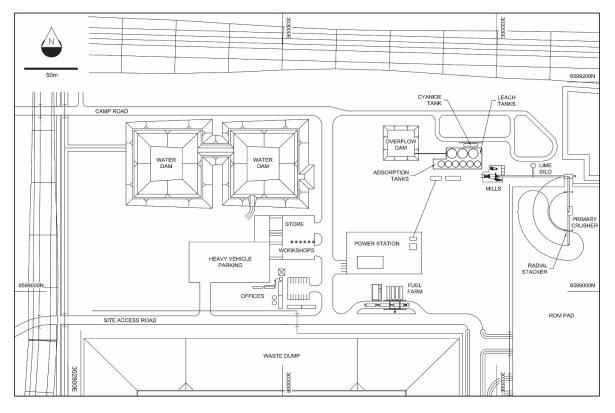
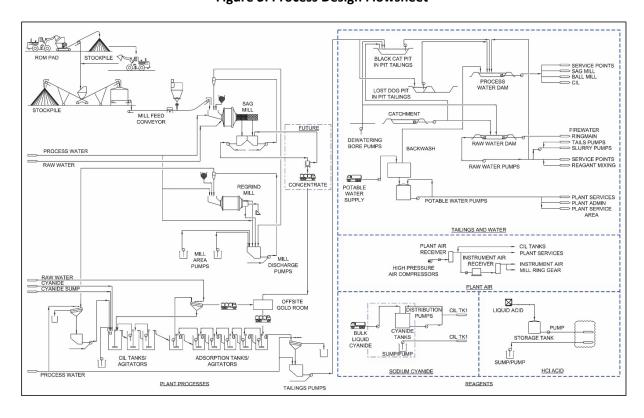


Figure 3: Process Design Flowsheet



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Tailings Storage Facility (TSF)

The TSF strategy is based on backfilling the void left by open pit mining. The open pit will be mined in panels and engineered retaining walls will be constructed to provide tailings disposal cells. Initially the Black Cat Pit will be utilised as a tailings facility until Panel 1 of the Lost Dog open pit has been prepared. The estimated tailings capacity of the Black Cat and Lost Dog open pits is 5,000,000 tonnes.

Production Target

The detailed open pit mine design has been used to schedule a potential production profile for the Jaurdi Project.

Table 3 - Jaurdi Project Design Physicals

Mining Reserve	Total	Stripping	Ounces	C1 – Cash Cost	
	Volume	Ratio	Recovered	per Ounce	
2.47 Mt @1.9g/t	4.44Mbcm	1.9:1	126,300	\$830	

A simplified, high level global scoping level production schedule based upon the open pit mine design physicals has been completed for the Jaurdi Project. The main constraint applied to the production schedule is the 500,000tpa capacity of the processing plant. The maximum pit depth (32.5 metres) and the low strip ratio (average 1.9:1) enables the mining and processing schedules to be run in parallel which minimises the working capital expense.

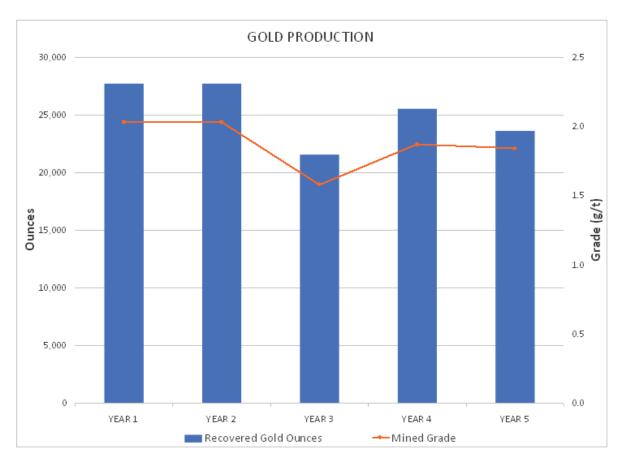
Table 4 – Production Target Schedule

Key Parametrers	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Waste Tonnes	620,000	620,000	700,000	1,290,000	1,800,000	5,030,000
Mined Ore Tonnes	500,000	500,000	500,000	500,000	500,000	2,500,000
Mined Gold Grade	2.0	2.0	1.6	1.9	1.7	1.9
Processing Input Tonnes	500,000	500,000	500,000	500,000	500,000	2,500,000
Recovered Gold Ounces	27,790	27,790	21,580	25,500	23,830	126,490

Note: Year 5 Lost Dog mill feed will be supplemented with approx. 30,000 tonnes from the Black Cat North open pit to fill the mill to its 500,000 tpa capacity.



Figure 4: Gold Production (Annual)



Capital Costs

Capital costs have been estimated as follows:

Table 5 – Estimated Capital Costs

Estimated Capital Costs	(A\$M)
Expenditure to Date	5.6
Processing Plant Construction	14.0
Waste Pre-Strip	1.8
Sustaining Capital	5.0
Total	26.4



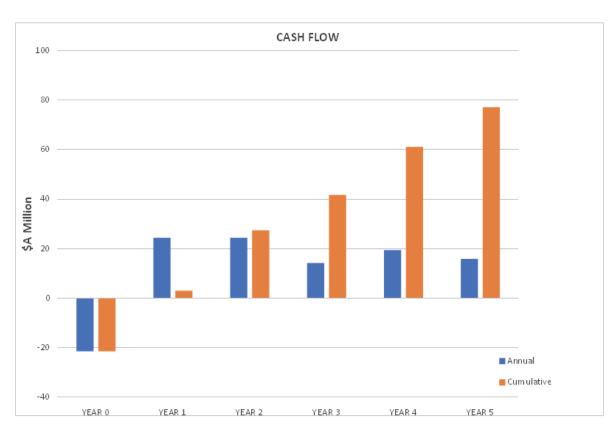
Financial Analysis

A high level financial analysis was undertaked on the Jaurdi Project using cost inputs provided by Beacon and work undertaken for this mining study. Upfront capital of AU\$21.4M (\$5.6m actual expenditure to date and \$15.8m estimated pre-production construction) was included in the financial analysis to account for the acquisition, relocation and refurbishment of a second hand processing plant with a nominal 0.5Mtpa throughput and other Project start-up costs. Sustaining capital of AU\$1.0m/year was also included.

Table 6: Production Target Cashflow

Annual Production Financials (\$M)	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mining Cost	-	3.3	3.3	3.5	5.3	6.7	22.1
Grade Control Cost	-	0.3	0.3	0.3	0.3	0.3	1.5
Variable Processing Cost	-	12.5	12.5	12.5	12.5	12.5	62.5
General and Adminsitration Cost	-	0.7	0.7	0.7	1.0	1.3	4.4
Royalties	0.9	2.6	3.4	2.6	3.1	2.9	15.5
Capital Cost	21.4	1.0	1.0	1.0	1.0	1.0	26.4
Revenue	-	45.8	45.8	35.6	42.1	39.2	208.5
Cashflow	-	25.4	24.6	15.0	18.9	14.5	98.4
Discounted Cashflow	(22.30)	24.85	23.55	14.05	17.32	13.01	70.48

Figure 5: Production Target Cashflow





Pre-production Activities

Pre-production activities at Jaurdi would include the following;

- Identification and development of a process water borefield (completed);
- Mining fleet mobilisation (completed);
- Construction of offices, workshops/store and camp (completed);
- Clearing, grubbing and stockpiling of top-soil and wood mulch (partially completed);
- The construction of a processing plant (on-going);
- Preparation of the ROM pad (partially completed); and
- Construction of the TSF tailings line and decant water line to the Black Cat open pit (on-going).

Sensitivity on Material Assumptions

A series of optimisation analyses, testing Ore Reserve sensitivity to Gold Price were performed by Minecomp using a financial model developed for owner operated mining and ore processing.

Further sensitivity testing was performed on the open pit Ore Reserve. The parameters tested for sensitivity were:

- Revenue Stream (Gold Price, Metallurgical Recovery or Both)
- Total Operating Costs
- Processing Costs
- Mining Costs

Figure 6: Ore Reserve Sensitivity (Exclusive of Capital)

The Board considers that the range of sensitivities are a reasonable basis for a PFS level study.



Project Finance

The financing required to acquire, explore, construct and commission the Jaurdi Gold Project is as follows:

- Actual total expenditure to date \$5.6M;
- Estimated pre-production construction \$15.8M.

Preparation for final project financing is advanced and the Company will provide further detail in the near future.

Risks and Opportunities

Key risks identified during the PFS work include, but are not limited to:

- Access to project funding;
- Adverse movements in the United States gold price;
- Adverse movements in the USD:AUD exchange rates; and
- Not achieving the processing production rates and metallurgical recovery rates.

Key opportunities identified during the PFS work include, but are not limited to:

- Achieving higher mill throughput rates. The installed SAG Mill power is in excess of the power requirements for a 500ktpa plant installation;
- Improved metallurgical recovery; and
- Expansion of the resource base via exploration success and/or acquisitions.

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Competent Persons Statement

The information in this report that relates to the Ore Reserves is based on information compiled by Mr Gary McCrae, a Competent Person who is a Member of the Australasian Institute of Mining and Metallurgy. The estimated ore reserves and/or mineral resources underpinning the production target have been prepared by Mr McCrae in accordance with the requirements in Appendix 5A (JORC Code). Mr McCrae is a full-time employee of Minecomp Pty Ltd. Mr McCrae has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr McCrae consents to the inclusion in the report of the matters based on his information in the form and context in which is appears.



Disclaimer

This ASX announcement (Announcement) has been prepared by Beacon Minerals Limited ("Beacon" or "the Company"). It should not be considered as an offer or invitation to subscribe for or purchase any securities in the Company or as an inducement to make an offer or invitation with respect to those securities. No agreement to subscribe for securities in the Company will be entered into on the basis of this Announcement.

This Announcement contains summary information about Beacon, its subsidiaries and their activities which is current as at the date of this Announcement. The information in this Announcement is of a general nature and does not purport to be complete nor does it contain all the information which a prospective investor may require in evaluating a possible investment in Beacon.

By its very nature exploration for minerals is a high risk business and is not suitable for certain investors. Beacon's securities are speculative. Potential investors should consult their stockbroker or financial advisor. There are a number of risks, both specific to Beacon and of a general nature which may affect the future operating and financial performance of Beacon and the value of an investment in Beacon including but not limited to economic conditions, stock market fluctuations, gold price movements, regional infrastructure constraints, timing of approvals from relevant authorities, regulatory risks, operational risks and reliance on key personnel.

Certain statements contained in this announcement, including information as to the future financial or operating performance of Beacon and its projects, are forward-looking statements that:

- may include, among other things, statements regarding targets, estimates and assumptions in respect of mineral reserves and mineral resources and anticipated grades and recovery rates, production and prices, recovery costs and results, capital expenditures, and are or may be based on assumptions and estimates related to future technical, economic, market, political, social and other conditions;
- are necessarily based upon a number of estimates and assumptions that, while considered reasonable by Beacon, are inherently subject to significant technical, business, economic, competitive, political and social uncertainties and contingencies; and,
- involve known and unknown risks and uncertainties that could cause actual events or results to differ materially from estimated or anticipated events or results reflected in such forward-looking statements.

Beacon disclaims any intent or obligation to update publicly any forward-looking statements, whether as a result of new information, future events or results or otherwise. The words 'believe', 'expect', 'anticipate', 'indicate', 'contemplate', 'target', 'plan', 'intends', 'continue', 'budget', 'estimate', 'may', 'will', 'schedule' and similar expressions identify forward-looking statements.

All forward looking statements made in this announcement are qualified by the foregoing cautionary statements. Investors are cautioned that forward-looking statements are not guarantees of future performance and accordingly investors are cautioned not to put undue reliance on forward-looking statements due to the inherent uncertainty therein.

No verification: Although all reasonable care has been undertaken to ensure that the facts and opinions given in this Announcement are accurate, the information provided in this Announcement has not been independently verified.



APPENDIX -JAURDI GOLD REPORT LOST DOG



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Minecomp Pty Ltd as trustee for the Minecomp Unit Trust

JAURDI GOLD PROJECT LOST DOG

OPTIMISATION ANALYSIS AND PIT DESIGN STUDY

AUGUST 2018

SUMMARY REPORT

PREPARED BY:

Minecomp Pty Ltd

PREPARED FOR:

Beacon Minerals Limited

DATE: 24 August 2018

REPORT:

Lost Dog

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Executive Summary

Based upon the June 2017 ore mineral resource model, slope parameters, and the cost structure applied, the optimum mining reserves, recovered ounces, and operating profit for Lost Dog were calculated to be:-

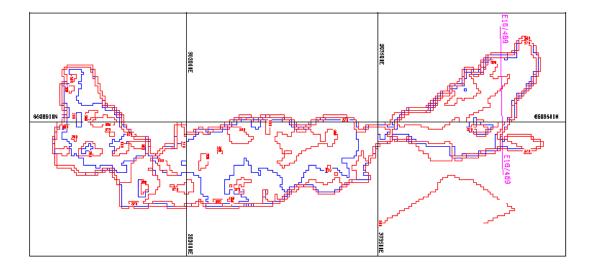
GOLD	MININGR	ESERVES	TOTAL	STRIPPING	OUNCES	CASH COST	PROFIT
PRICE	TONNAGE	GRADE	VOLUME	RATIO	RECOVERED	per OUNCE	@ \$1,650/oz
(\$)	(t)	(g/t)	(b cm)	(bcm:bcm)	(oz)	(\$)	(\$)
1,000	1,389,907	2.45	2,420,508	1.9	93,101	648	91,749,634
1,050	1,503,921	2.37	2,559,538	1.8	97,223	665	94,264,127
1,100	1,624,655	2.28	2,688,761	1.8	101,254	683	96,538,393
1,150	1,764,381	2.21	2,945,803	1.8	106,384	705	99,181,821
1,200	1,851,311	2.16	3,044,117	1.7	109,038	718	100,397,816
1,250	1,932,004	2.11	3,148,680	1.7	111,492	730	101,424,360
1,300	2,017,341	2.07	3,298,292	1.7	114,177	744	102,396,084
1,350	2,082,089	2.04	3,385,239	1.7	116,009	755	102,984,490
1,400	2,143,297	2.01	3,479,910	1.7	117,743		103,452,196
1,450	2,207,288	1.98	3,594,806	1.7	119,554	776	103,858,125
1,500	2,261,588	1.96	3,700,405	1.7	121,095	786	104,136,161
1,550	2,299,679	1.94	3,759,114	1.7	122,076	793	104,269,254
1,600	2,339,502	1.93	3,840,391	1.7	123,128	801	104,337,447
1,650	2,380,877	1.91	3,932,051	1.7	124,226	810	104,368,179
1,700	2,410,692	1.90	3,985,941	1.7	124,934	816	104,333,998
1,750	2,440,609	1.88	4,040,390	1.7	125,642	822	104,296,130
1,800	2,458,510	1.88	4,075,959	1.7	126,068	827	104,254,544
1,850	2,481,435	1.87	4,141,413	1.7	126,665	833	104,138,652
1,900	2,495,092	1.86	4,161,055	1.7	126,938	836	104,072,508
1,950	2,511,109	1.86	4,200,578	1.7	127,318	841	103,973,796
2,000	2,543,911	1.84	4,273,293	1.8	128,017	848	103,755,072

The above results are summarised in Appendix 5 and presented tabularly in Appendix 6.

As summarised above the Au\$1,650/oz or "Base Case" optimum shell was evaluated to produce 2,380,877t of processed ore at a grade of 1.91g/t. These milled ore tonnes are mined in conjunction with 2,489,609BCM of sub-grade ore and waste, thus representing a stripping ratio of 1.7:1 (waste and sub-grade ore volume to processed ore volume). When financially evaluated the Au\$1,650/oz or "Base Case" optimum shell produces 124,226 ounces of recovered gold at an all in operating cash cost per ounce of Au\$10. At the "Base Case" gold price of Au\$1,650/oz this optimum shell produces an operating profit of Au\$104,368,179 at a return of 103.7%.

Executive Summary - continued

• The optimisation analysis resulted in the delineation of one continuous optimum shell. This optimum shell measured approximately 1300m long, 250m wide and 27.5m deep.



- The portion of the "Base Case" mining reserve within the exploration license E16/469 was estimated to be 105,200t @ 1.62g/t.
- E16/469 is under an option to purchase agreement between BCN and the current tenement holders.
- No issues, legal or otherwise are foreseen for the mining application over E16/469. This application has been submitted to the Western Australian Department of Minerals and Energy and is currently pending. This mining application has been assigned mining lease number M16/561.

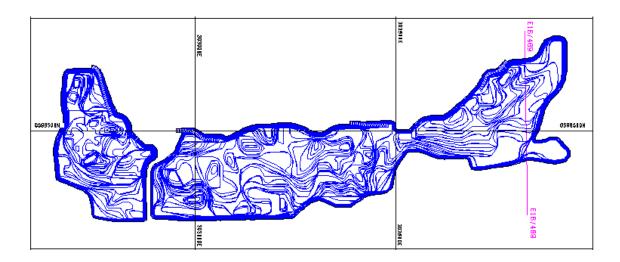
The subsequent Lost Dog open pit mine design was evaluated to produce 2,469,847t of milled ore at a grade of 1.87g/t. These milled ore tonnes are mined in conjunction with 2,933,292BCM of sub-grade ore and waste, thus representing a stripping ratio of 1.9:1 (waste and sub-grade ore volume to milled ore volume). When financially evaluated the Lost Dog open pit design produces 126,259 ounces of gold at an all-in operating cash cost per ounce of Au\$843. At the "Base Case" gold price of Au\$1,650/oz the open pit mine design produces an operating profit of \$101,917,012 at a return of 95.7%.

Comparison with the optimum shell upon which the pit design was based indicates excellent design efficiencies have been achieved.

	MINING RESERVES		RVES TOTAL STRIPPING OUNCES CASH COS		CASH COST	OPERATING PROFIT			
SCENARIO	TONNAGE	GRADE	VOLUME	RATIO	RECOVERED	per OUNCE	@ \$1,600/oz	@ \$1,650/oz	@ \$1,700/oz
	(t)	(g/t)	(bcm)	(bcm:bcm)	(oz)	(\$)	(\$)	(\$)	(\$)
DESIGN	2,469,847	1.87	4,439,293	1.9	126,259	843	95,761,862	101,917,012	108,072,162
SHELL	2,380,877	1.91	3,932,051	1.7	124,226	810	98,312,150	104,368,179	110,424,208

The above results are presented in Appendix 8 whilst the open pit mine design plan is presented in Appendix 9

Executive Summary - continued



As requested by Alex McCulloch, Project Manager BCN, a "bridge" area at approximately 302880E was to remain unmined and intact. It is proposed to utilise this "bridge" as the western wall of an inpit tailings storage facility. Staged mining sequencing will then enable the eastern wall to be constructed, using suitably categorized waste material.

This bridge/tailings dam wall area was calculated to effectively sterilize 14,600t of ore grading 1.25g/t (approximately 587 ounces of gold).

Also of note is:-

- This study is considered to be of a pre-feasibility level standard.
- All costs and revenues are expressed in Australian dollars.
- That no allowances have been made for capital and/or start-up costs in the above calculations.
- Only material classified as Measured and Indicated has been used to calculate the Ore Reserve. Any material classified as Inferred in the June 2017 Resource estimate has been categorised as waste.
- The shallow orebody depth (<32.5m) and weathering profile result in Lost Dog being mined by conventional open pit methods.
- The application of a 2% mining dilution factor and 98% mining recovery are appropriate to the style and nature of the Lost Dog given the orebody widths (averaging 200m) and thickness (averaging > 10m) and weathered nature of the resource.
- The Lost Dog ore is best suited to CIP processing.
- CIP processing is a tried and tested method for extracting gold from ore of this nature.
- The metallurgical recoveries used throughout this study have been substantiated by metallurgical testwork and treatment of a 4,625t bulk sample.
- The cut-off grade used for the calculation the Ore Reserve was 0.6g/t and was purely financially based (Sum of Costs)/[(Processing Recovery)*(Gold Price-Royalties)].
- No Native Title Claimants on DIA over the mining leases.

Executive Summary - continued

- A Project Management Plan and Mining Proposal have been approved by the Western Australian DMIRS.
- A miscellaneous license L16/122 application has been lodged with the DMIRS. This has been applied to facilitate pipeline access between a portion of the bore field and the proposed ore processing plant. This application is pending.
- Environmental permitting has been submitted to the Western Australian DMIRS and DWER. All approvals have subsequently been received except to construct and commission the processing plant. Draft conditions for this approval have been received and are under consideration.
- The statements and opinion in this Report are given in good faith and this Report is based upon information provided by BCN, along with technical reports prepared by consultants and other relevant published and unpublished data for the area.

Introduction and Objectives

In May 2018, Minecomp Pty Ltd was retained by Alex McCulloch, of Beacon Minerals Limited (BCN) to carry out a pre-feasibility level optimisation analysis and subsequent open pit mine design on Lost Dog which forms part of BCN's Jaurdi Gold Project.

The optimisation analysis was to: -

- Produce results which would assist BCN in progressing towards the commencement of open pit mining at the Jaurdi Gold Project.
- Utilise the gold grade estimation attribute "au" as supplied in the June 2017 mineral resource estimate model generated by Richard Finch of BM Geological Services (BMGS). This mineral resource estimate model was titled "Jaurdi_BMGS_1706.mdl" and was supplied in a SURPAC block model format. It should be noted that no high-grade top cut was applied.
- Assign zero grade and value to any material delineated by the mineral resource estimate model which was **NOT** JORC2012 classified as either measured or indicated.
- Incorporate the existing Lost Dog surface topography as supplied by Minecomp Pty Ltd.
- Utilise a "Base Case" mining dilution factor of 2% @ 0.00g/t and a mining recovery factor of 98%.
- Be performed at a gold price range of between \$1,000/oz to \$2,000/oz in \$50/oz increments, with \$1,650/oz to be considered the "Base Case" gold price.
- Incorporate the 2.5% Western Australian State Gold Royalty and an additional 3rd Party Royalty of \$80/oz.
- Utilise an overall slope angle of 50 degrees. This angle was derived from the geotechnical parameters recommended by Green Geotechnical Pty Ltd in April 2018 and perceived pit and ramp geometries and configurations.
- Utilise optimisation and evaluation parameters either supplied by Alex McCulloch, Geoff Greenhill (BCN) or Gary McCrae of Minecomp Pty Ltd. These cost parameters comprised contractor estimates based upon experience and are inclusive of all on-site operating costs.
- Make no allowances for capital costs.

The subsequent open pit mine design was to:-

- Be based upon the Au\$1,650/oz optimum shell generated in the optimisation analysis.
- Leave a 10m wide area at approximately 302,880E intact with the intention of utilizing this area as the western wall of an in-pit tailings storage facility.
- Utilise open pit mine batter and berm parameters in line with those recommended by Green Geotechnical.
- Utilise ramp design parameters suited to mining by an articulated dump truck fleet and associated ancillary equipment.
- Produce an open pit mine design which was practical, workable and safe.
- Make no allowances for capital costs in the open pit mine design evaluation.

Optimisation and Evaluation Parameters

Natural Surface Topography

The natural surface topography used for the optimisation analysis and subsequent optimum shell and open mine design evaluations was as surveyed and supplied by Minecomp Pty Ltd. The files were in SURPAC string (STR) and digital terrain model (DTM) formats. File names were "jaurdi_ns999.str" and "jaurdi_ns999.dtm" and comprised of comprehensive topographical and drill hole collar information.

Ore Resource Model

The June 2017 mineral estimate model was generated by Richard Finch and supplied to Minecomp Pty Ltd in a SURPAC block model format. This resource model was titled "Jaurdi BMGS 1706.mdl"..

Information contained within this model and utilised for this optimisation analysis included:-

Block Centroid Positions
Block Uncut Gold Grade (au)
Block Density
Block Resource Category
Block Material (i.e. Ore, Waste, Air)

The densities as supplied in this resource estimate model and as utilised throughout the course of this study were as follows:-

MATERIAL TYPE	SG
CLAY	1.10
CLAY/SILT	1.20
SILT/CLAY	1.40
SILTSTONE	2.40

The Mineral Resource update pertaining to this resource estimate, and subsequently announced by Beacon on the 12th July 2018 is presented in Appendix 1

Gold Price

The sensitivity of the Lost Dog Project to gold price was determined for a gold price range of \$1,000/oz through to \$2,000/oz in \$50/oz increments with \$1,650/oz being considered the "Base Case" gold price.

Optimisation and Evaluation Parameters - continued

Royalties

The optimisation analysis and subsequent optimum shell and open pit mine design evaluations incorporated the 2.5% Western Australian State Gold Royalty and an additional 3rd Party Royalty \$80/oz of gold produced.

Capital and Start-up Cost Information

No allowances have been made for capital and/or start-up costs in the optimisation analysis or the subsequent optimum shell and pit design evaluations.

Operating Cost Information

The operating costs used for the optimisation analysis and subsequent optimum shell and open pit mine design evaluations were either supplied by Alex McCulloch, Geoff Greenhill or Gary McCrae. These parameters were contractor estimates based upon experience and are inclusive of all site costs. Note that the "mining extras" cost category incorporates mine dewatering, surveying and on-site camp costs.

A summary of these costs is presented in Appendix 2.

Mining Recovery and Dilution

A mining dilution factor of 2% at 0.00g/t and a mining recovery factor of 98% were incorporated into the optimisation process and the subsequent optimum shell and open pit mine design evaluations. These factors were considered appropriate for an optimisation analysis of a resource of this nature.

Metallurgical Recovery

The metallurgical recovery used for the analysis and subsequent optimum shell and open pit mine design evaluations was 85% regardless of ore type. This figure was supplied by Geoff Greenhill and comprised of information from three separate sources.

Firstly, metallurgical testing by ALS on a composite sample from 6 diamond drill holes drilled into the Lost Dog orebody indicated a recovery of 84% could be achieved on the Lost Dog composite. The ALS Metallurgy Report, A18169 is presented in Appendix 3A.

Secondly, numerous in-house, bottle roll tests performed by BCN on various ore types from selected drill samples resulted in metallurgical recoveries in the range of 82-96%, but typically averaging 88%. These results are presented in Appendix 3B.

Finally trial milling of a 4,625t parcel of West Pit ore at a custom milling facility, by Fenton and Martin Mining Development achieved a metallurgical recovery of 91.8%. These results are presented in Appendix 3C.

Optimisation and Evaluation Parameters - continued

Optimisation Slope Parameters

The overall slope angle used for the optimisation process was 50°. This angle was derived from the geotechnical guidelines generated by Green Geotechnical in April 2018. The Green Geotechnical guidelines are presented in Appendix 4.

Open Pit Design Parameters

The design parameters used for the Lost Dog open pit mine designs were:-

BATTERS

N/S -> 380m RL - 45 Degrees 380 -> 355m RL - 60 Degrees

BERMS

Not Required

RAMPS

Suitable for Articulated Dump Truck Fleet and Ancillary Equipment All at 1 in 6 – 8m Wide (nominal)

Processed Ore/Waste Cut-off Grades

Break-even grades are those grades at which ore material can be processed profitably.

For Lost Dog the costs and factors used to calculate the processed ore/waste cut-off grades are:-

- Gold Price
- Mining Dilution
- State Gold and 3rd Party Royalties
- Costs comprising of Ore/Waste Differential, Grade Control, and Ore Processing
- Processing Recovery

The formula used to calculate the cut-off grade was:-

(Sum of Costs)/[(Processing Recovery)*(Gold Price-Royalties)]

At \$1,650/oz the processed ore/waste cut-off grade was calculated to be 0.60g/t. This figure represents a diluted grade.

APPENDICES

APPENDIX 1 LOST DOG

 ${\bf JAURDI~GOLD~PROJECT~-~LOST~DOG~MINERAL~RESOURCE~UPDATE}$

12th JULY 2017



ASX ANNOUNCEMENT

12 July 2017

JAURDI GOLD PROJECT - LOST DOG MINERAL RESOURCE UPDATE

Highlights

- June Mineral Resource update totalling 2.9M tonnes @ 1.76 for 163.1 K Oz
- Mineral Resource tonnage has increased by 12% with a 9% increase in ounces

Majority of Mineral Resource is classified as Measured and Indicated (96% of the tonnes and 98% of the ounces)

Classification	('000) Tonnes	Au g/t	('000) Ounces
Measured	30	1.56	1.5
Indicated	2,752	1.79	158.4
Inferred	101	0.96	3.2
Total	2,883	1.76	163.1

Beacon Minerals Limited ("Beacon" or the "Company") is pleased to inform shareholders they have an update of the Lost Dog Mineral Resource following the completion of the May-June 2017 infill and extensional drilling campaigns. The updated June 2017 Mineral Resource increases the gold inventory at the Jaurdi Gold Project to 163,100 ounces (see Tables 1 and 2). The resource is based on 348 reverse circulation (RC), air core (AC) and diamond core drill holes completed by Beacon and historical explorers for a total of 9,847 metres.

The Lost Dog orebody is now defined to have an overall strike length of 1,450 metres in an East-West orientation, has an average thickness of 8 metres and an average width of 180 metres. It remains open to the North East. The deposit consists of three main areas; the Western Arm, the Eastern Arm and the North East Extension. The latter has narrowed to 120 metres wide; the average width remains at 180 metres; however, it attains a maximum width of 260 metres on the "elbow" as its orientation rotates to the North-East at a bearing of 040 degrees after striking predominantly East-West.

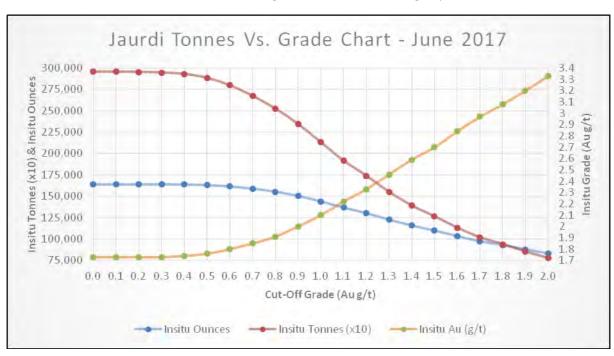
The Jaurdi Gold Project overlies a portion of the Bali Monzogranite immediately adjacent to the Jaurdi Hills-Dunnsville greenstone sequence. The Lost Dog gold mineralisation is hosted in either a bleached, siliceous siltstone or an interbedded clay and siltstone unit.



Table 1: Lost Dog June 2017 Mineral Resource estimate by classification and lithology (Au> 0.5 g/t)

Res Cat	Rock Unit	('000) Volume	('000) Tonnes	Au	('000)Ounces	Density
	Siltstone	1	3	1.11	0.1	2.40
NATAC	Silt/Clay	2	3	1.35	0.1	1.40
MEAS	Clay/Silt	10	12	1.29	0.5	1.20
	Clay	10	11	2.08	0.7	1.10
Sub	Total	24	30	1.56	1.5	1.25
	Siltstone	496	1,191	1.50	57.6	2.40
INDI	Silt/Clay	637	891	1.95	55.7	1.40
INDI	Clay/Silt	473	567	2.14	39.1	1.20
	Clay	94	103	1.81	6.0	1.10
Sub	Total	1,699	2,752	1.79	158.4	1.62
	Siltstone	30	72	0.88	2.0	2.40
INFE	Silt/Clay	19	27	1.18	1.0	1.40
IIVE	Clay/Silt	2	3	1.20	0.1	1.20
	Clay	-	-	-	-	-
Sub	Total	51	101	0.96	3.2	1.98
To	otal	1,774	2,883	1.76	163.1	1.66

Table 2: Grade tonnage curve for the Lost Dog deposit





The Mineral Resource has been drilled out using reverse circulation, air core drilling and diamond core techniques. Sampling was collected through a cyclone and split through a rig mounted riffle splitter. A cone splitter was utilised for the latter of the recent Stage 5 program. All sample components were taken as a 12.5% split of the original. One metre samples were collected to obtain a 3 to 4 Kg sample. All samples were pulverised to typically 95% passing -75µm to produce a 50g charge for Fire Assay with an AAS finish.

The drilling has been completed on a 25m x 50m pattern for the majority of the deposit (see Figures 2 to 5). A significant region of the Eastern Arm was in-filled to a 25m x 25m spacing during the Stage 4 program. In addition, a close space drilling programme was completed in the Western Arm on a 10m x 10m pattern during Stage 2. The purpose of this drilling was to understand the short scale continuity of the mineralisation with the aim to use this in the resource modelling process. The density of drilling for this style of deposit has given sufficient confidence to categorise the Mineral Resource predominantly as Measured and Indicated (96% of the tonnes and 98% of the ounces).

Grade estimation was completed using ordinary Kriging. A nested spherical variogram with two structures was derived for each domain using Snowden Supervisor software. The variogram was created as normal scores and was back transformed for use with 3DS Surpac. Inverse Distance Squared was utilised to estimate a small low-grade domain. A 0.5g/t Au was selected as the optimal cut-off grade from both a statistical and an economical stand-point. The mining method considered at this early stage is conventional drill and blast and load and haul with an excavator and articulated dump trucks. The current defined Mineral Resource estimate is situated on both the granted Mining Lease M16/529 (92%) and the adjoining E16/469 (8%); the latter being under an option agreement whereby Beacon can acquire a 100% interest.

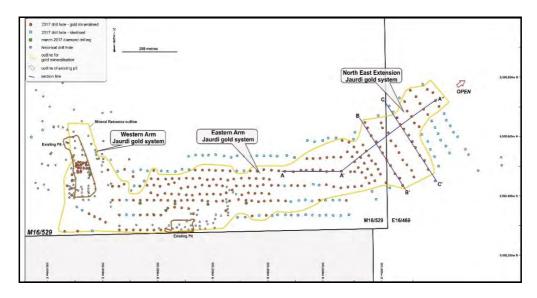


Figure 1: Plan of the Jaurdi palaeochannel showing the drilling which has defined the Western Arm, the Eastern Arm and the North East Extension.



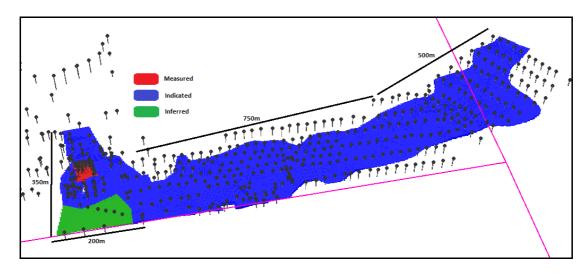


Figure 2: 3D image of the Lost Dog deposit showing confidence categories of June 2017 Mineral Resource and overall strike length of 1,450 metres

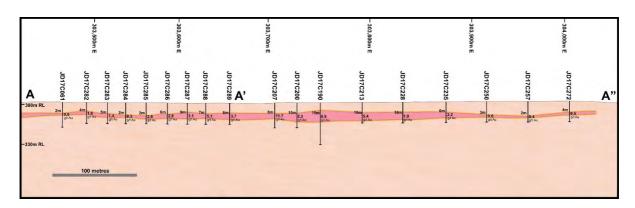


Figure 3: Long section A - A' - A'' of the Eastern Arm and North East Extension of the Lost Dog deposit



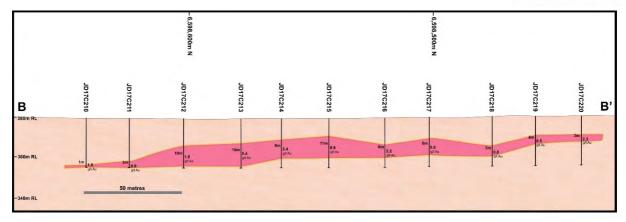


Figure 4: Cross section B – B' through the North East Extension of the Lost Dog deposit

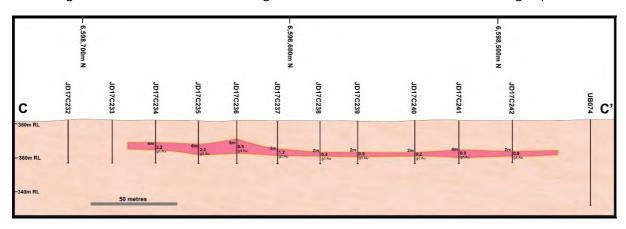


Figure 5: Cross section C – C' through the North East Extension of the Lost Dog deposit



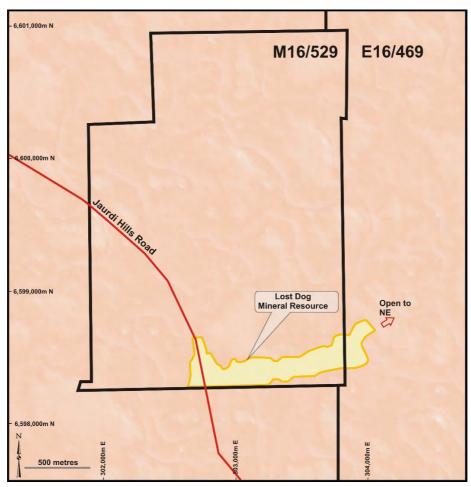


Figure 6: Location of the Lost Dog Mineral Resource North East Extension on E16/469



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Competent Persons Statement

The information in this report that relates to the Jaurdi Gold Project Mineral Resource estimation is based on information compiled by Mr Richard Finch and Mr Darryl Mapleson, both full time employees of BM Geological Services. Mr. Finch is a Member of the Australian Institute of Mining and Metallurgy, while Mr Mapleson is a Fellow of the Australian Institute of Mining and Metallurgy. Mr Finch and Mr Mapleson have been engaged as consultants by Beacon Minerals Limited. Mr Finch and Mr Mapleson have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Finch and Mr Mapleson consent to the inclusion in the report of the matters based on his information in the form and context in which it appears.

Disclaimer

This ASX announcement (Announcement) has been prepared by Beacon Minerals Limited ("Beacon" or "the Company"). It should not be considered as an offer or invitation to subscribe for or purchase any securities in the Company or as an inducement to make an offer or invitation with respect to those securities. No agreement to subscribe for securities in the Company will be entered into on the basis of this Announcement.

This Announcement contains summary information about Beacon, its subsidiaries and their activities which is current as at the date of this Announcement. The information in this Announcement is of a general nature and does not purport to be complete nor does it contain all the information which a prospective investor may require in evaluating a possible investment in Beacon.

By its very nature exploration for minerals is a high risk business and is not suitable for certain investors. Beacon's securities are speculative. Potential investors should consult their stockbroker or financial advisor. There are a number of risks, both specific to Beacon and of a general nature which may affect the future operating and financial performance of Beacon and the value of an investment in Beacon including but not limited to economic conditions, stock market fluctuations, gold price movements, regional infrastructure constraints, timing of approvals from relevant authorities, regulatory risks, operational risks and reliance on key personnel.

Certain statements contained in this announcement, including information as to the future financial or operating performance of Beacon and its projects, are forward-looking statements that:

may include, among other things, statements regarding targets, estimates and assumptions in



respect of mineral reserves and mineral resources and anticipated grades and recovery rates, production and prices, recovery costs and results, capital expenditures, and are or may be based on assumptions and estimates related to future technical, economic, market, political, social and other conditions:

- are necessarily based upon a number of estimates and assumptions that, while considered reasonable by Beacon, are inherently subject to significant technical, business, economic, competitive, political and social uncertainties and contingencies; and,
- involve known and unknown risks and uncertainties that could cause actual events or results to differ materially from estimated or anticipated events or results reflected in such forward-looking statements.

Beacon disclaims any intent or obligation to update publicly any forward-looking statements, whether as a result of new information, future events or results or otherwise. The words 'believe', 'expect', 'anticipate', 'indicate', 'contemplate', 'target', 'plan', 'intends', 'continue', 'budget', 'estimate', 'may', 'will', 'schedule' and similar expressions identify forward-looking statements.

All forward looking statements made in this announcement are qualified by the foregoing cautionary statements. Investors are cautioned that forward-looking statements are not guarantees of future performance and accordingly investors are cautioned not to put undue reliance on forward-looking statements due to the inherent uncertainty therein.

No verification: Although all reasonable care has been undertaken to ensure that the facts and opinions given in this Announcement are accurate, the information provided in this Announcement has not been independently verified.



Appendix 1

JORC Code, 2012 Edition – Table 1 report – Jaurdi Gold Project: June 2017 Mineral Resource Update

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.	The sampling of drill cuttings has been carried out on Reverse Circulation (RC) drilling for the Stage 5 infill and extensional program. A total of 101 holes were completed for 2,520m. The Stage 5 program was conducted on both the M16/529 & E16/469 tenements.
	Include reference to measures taken to ensure sample representation and the appropriate calibration of any measurement tools or systems used.	The drill hole collar locations were surveyed by DGPS using Kalgoorlie based registered surveyors of Minecomp Pty Ltd. Sampling was carried out under Beacon's protocols and QAQC procedures as per industry best practice. See further details below.
	Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases, more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	The RC holes were drilled using a 138mm face-sampling bit. One metre samples were collected through a cyclone and split through a rig mounted riffle splitter. An increased Clay content became evident early in Stage 5 and as a result, a cone splitter was utilised for the remainder of the program. A 25% split was used to produce a sample size of approximately 3-4kg per metre for both splitters. All samples were pulverised at the lab to -75um, to produce a 50g charge for Fire Assay with an AAS finish.
Drilling techniques	Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	Ausdrill Ltd completed 66 vertical RC holes for 1,647m using a DRA GC600 rig. Raglan Drilling Pty Ltd completed the remaining 35 RC holes for 873m using a Schramm T685W. Both rigs utilised a 138mm diameter face sampling bit.
Drill sample recovery	Method of recording and assessing core and chip sample recoveries and results assessed.	Ground water ingress occurred in some holes at rod change, but overall the holes were kept dry. Typically, drilling operators ensured water was lifted from the face of the hole at each rod change to ensure water did not interfere with drilling and to make sure samples were collected dry. RC recoveries were visually estimated, and recoveries recorded in the log as a percentage. Recovery of the samples was good, generally estimated to be full, except for some sample loss at the collar of the hole.
	Measures taken to maximise sample recovery and ensure representative nature of the samples.	RC face-sample bits and dust suppression were used to minimise sample loss. Drilling airlifted the water column above the bottom of the hole to ensure dry sampling. RC samples are collected through a cyclone and then split to capture a 3 to 4 Kg sample.
	Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	No relationship between recovery and grade has been identified.
Logging	Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.	All chips and drill core were geologically logged by experienced industry geologists, using the Beacon Minerals geological logging legend and protocol.
	Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	Logging of RC chips and drill core records lithology, mineralogy, mineralisation, weathering, colour and other features of the samples. All samples are wet-sieved and stored in a chip tray.

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Criteria	JORC Code explanation	Commentary
	The total length and percentage of the relevant intersections logged	All holes were logged in full.
Sub-sampling techniques and sample preparation	If core, whether cut or sawn and whether quarter, half or all core taken.	N/A
	If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	Samples of the 66 holes completed by Ausdrill Ltd were split through a rig mounted riffle splitter. The remaining 35 holes completed by Raglan Drilling Pty Ltd were split through a rig mounted cone splitter. Results of the two splitting techniques were analysed, with no disparities between the two evident. The majority of samples were kept dry, with some wet samples produced at rod change.
	For all sample types, the nature, quality and appropriateness of the sample preparation technique.	Samples were prepared at the SGS Laboratory in Kalgoorlie. Samples were dried, and the whole sample pulverised to 90% passing -75um, and a sub-sample of approx. 200g retained. A nominal 50g was used for the fire assay analysis. The procedure is industry standard for this type of sample.
	Quality control procedures adopted for all sub-sampling stages to maximise representation of samples.	A CRM standard, fine blank and field duplicate was submitted at a rate of approximately 1 in 27 samples. At the laboratory, regular Repeats and Lab Check samples are assayed.
	Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	The technique to collect the one metre samples was via a rig mounted riffle or cone splitter. Both splitters were routinely inspected by the field geologist. Field duplicates were collected and results were satisfactory, suggesting the duplicate field samples replicated the original samples.
	Whether sample sizes are appropriate to the grain size of the material being sampled.	Sample sizes are considered appropriate to give an indication of mineralisation given the particle size and the preference to keep the sample weight at a targeted 3 to 4kg mass.
Quality of assay data and laboratory tests	The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	Samples for the 5 th campaign of drilling completed by Beacon were analysed at the SGS Laboratory in Kalgoorlie. The analytical method used was a 50g Fire Assay with AAS finish for gold. The techniques is considered to be appropriate for the material and style of mineralization.
	For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	Not applicable.
	Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	Beacon Minerals protocol for the 2017 RC/AC/DD drilling programs was for a single CRM (Certified Reference Material), fine blank and field duplicate to be inserted in every 90 samples. A total of 2,517 samples were submitted during the Stage 5 program, along with 32 CRM standards, 33 fine blanks and 31 field duplicates. This at a rate of approximately 1 QA/QC sample per 27 regular samples.
		At the SGS Laboratory, regular assay Repeats, Lab Standards and Blanks are analysed. Results of the Field and Lab QAQC were analysed on assay receipt. On analysis, all assays passed QAQC protocols, showing no levels of contamination or sample bias. Analysis of field duplicate assay data suggests appropriate levels of sampling precision have been achieved for the sampling technique employed.
Verification of sampling and assaying	The verification of significant intersections by either independent or alternative company personnel.	Significant results were checked by Beacon Minerals executives and BMGS senior geologists.
	The use of twinned holes.	Nil twinned holes were completed as part of the Stage 5 program.
	Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	All field logging is carried out using a customised logging form on a Tough Book and transferred into an Access database. Assay files are received electronically from the Laboratory. All data is stored in the Jaurdi Gold Project Access database and managed by BMGS in Perth.

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Criteria	JORC Code explanation	Commentary
	Discuss any adjustment to assay data.	No assay data was adjusted.
Location of data points	Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	RC hole collar locations were surveyed by a registered Surveyor. The group used was the Kalgoorlie based Minecomp Pty Ltd. All Stage 5 drill holes were vertical – previous down-hole surveys observed minimal deviation with vertical holes and it was therefore deemed to be not necessary to continue completing downhole surveys of shallow, vertical holes.
	Specification of the grid system used. Quality and adequacy of topographic control.	Grid projection is MGA94, Zone 51. Minecomp Pty Ltd has completed a topographic survey over the lease picking up the two shallow pits on the Mining Lease and a suite of historical holes.
Data spacing and distribution	Data spacing for reporting of Exploration Results.	Stage 5 infill and extensional drilling was completed at a regular spacing of 25m x 50m; in line with previous exploration campaigns at Lost Dog.
	Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	This spacing is sufficient to test the continuity of mineralisation for this style of mineralisation.
	Whether sample compositing has been applied.	All RC samples collected were 1 metre composites.
Orientation of data in relation to geological structure	Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.	It is considered the orientation of the drilling and sampling suitably captures the "structure" of the palaeochannel style of mineralisation.
Structure	If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	This is not considered material.
Sample security	The measures taken to ensure sample security.	Samples were transported by company transport to the SGS laboratory in Kalgoorlie.
Audits or reviews	The results of any audits or reviews of sampling techniques and data.	Sampling and assaying techniques are industry-standard. Beacon have had the Jaurdi database reviewed by a second geological consultant (Kaldera Pty Ltd) who concluded the geological, survey and QAQC data collected during the Beacon drill campaigns 1 to 4 meets industry standard.

Section 2 Reporting of Exploration Results

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

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.	The RC drilling occurred within tenements M16/529 and E16/469. Beacon holds a 100% controlling interest of M16/529 and have an option agreement on E16/529.
	The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	The tenements are in good standing with the WA DMP.
Exploration done by other parties	Acknowledgment and appraisal of exploration by other parties.	There have been three campaigns of drilling undertaken on this M16/529 by third parties; previously a suite of Prospecting Licenses. The early phase was completed by a private firm called Coronet Resources in 2007. A second phase of drilling was completed by a group of "prospectors", the program being supervised by BM Geological Services in 2009. A report was produced outlining an unclassified resource. The third phase of

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Criteria	JORC Code explanation	Commentary
		drilling was commissioned by Fenton and Martin Mining Developments in 2015 (the current owners of the Jaurdi Gold Project). BCN has since completed five exploration and grade control campaigns on E16/529. In addition, there has been one drilling programme completed on E16/469 which the data and information pertaining to the drilling has been appraised by BMGS to meet industry standard.
Geology	Deposit type, geological setting and style of mineralisation.	pertaining to the drilling has been appraised by BMGS to meet industry standard. The Jaurdi Gold Project overlies a portion of the Bali Monzogranite immediately adjacent to the Jaurdi Hills-Dunnsville greenstone sequence. The Bali Monzogranite and Dunnsville Granodiorite to the north, together occupy the core of the gently north plunging anticline. The tenement making up the project is located to the west of the anticlinal axis and immediately adjacent to the granite-greenstone contact. The Bali Monzogranite is poorly exposed. The greenstone-granite contact is foliated where exposed. Shear zones developed locally within the adjacent greenstones, may continue within the granite. Gold mineralised palaeochannels are known in the Jaurdi area Regional magnetic data suggest that the western portion of the project lies within a broad demagnetised corridor following the western contact of the Bali Monzogranite, and which may continue in a north northwest direction through the greenstone sequence to Dunnsville. A magnetic dyke, akin to the Parkeston dyke in the Kalgoorlie area, has intruded this corridor. Another paired east northeast magnetic dyke set is located immediately to the south of the project area. This dyke set is part of the regionally extensive Widgiemooltha Dyke Suite, and passes to the north of Kalgoorlie-Boulder. The Jaurdi Gold Project is located close to the western margin of the Bali Monzogranite immediately to the south east of the exposed Jaurdi Hills greenstone sequence. The tenement is entirely soil covered, with well-developed nodular carbonate increasing in intensity southwards towards an active contemporary drainage. Recent drilling programs have revealed the known soil anomaly overlies an extensive system of Au-bearing sand channels indicating that a major long-lived palaeoalluvial system was present in the area. A typical profile consists of transported lateritic gravels overlying plastic clay zones, which in turn overly thick, water saturated silt and clay sequences with minor cobble la
		distance of at least 1,450 metres. Two horizons of mineralisation have been identified in the Western Arm with the shallower lode situated between 12 to 16 metres vertical depth, and the second horizon between 18 to 25 metres. The Eastern Arm has been identified by a system which is at least 1,450 metres strike (East – West orientated), 180 metres wide and 8 metres deep; and appears open to the North-East and connects with the Western Arm.



Criteria	JORC Code explanation	Commentary
Drill hole Information	A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: - easting and northing of the drill hole collar - elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar - dip and azimuth of the hole - down hole length and interception depth - hole length. If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	Refer to Appendix 2 in the body of the text.
Data aggregation methods	In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.	Grades are reported as down-hole length-weighted averages of grades above approximately 0.5 ppm Au. No top cuts have been applied to the reporting of the assay results. Intercepts averaging values significantly less than 0.2 g/t Au were assigned the text "NSI" (No Significant Intercept). Intercepts with minimal mineralisation that are located within the delineated ore body (internal dilution) were reported with intercept metres and grade.
	Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.	Higher grade intervals are included in the reported grade intervals.
	The assumptions used for any reporting of metal equivalent values should be clearly stated.	No metal equivalent values are used.
Relationship between mineralisation widths and intercept lengths	These relationships are particularly important in the reporting of Exploration Results. If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	The geometry of the mineralisation has been well established by the recent drilling. There is no ambiguity with the geometry of this relatively simple alluvial system.
Diagrams	Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	Refer to Figures 1 to 9 in the body of text.
Balanced reporting	Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	No misleading results have been presented in this announcement.
Other substantive exploration data	Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	Not applicable.
Further work	The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	Further exploration work is currently under consideration, the details of which will be released in due course.

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Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	 Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. Data validation procedures used. 	 Database inputs were logged electronically at the drill site and at the BCN Kalgoorlie yard for the diamond core. The collar metrics, assay, lithology and down-hole survey interval tables were checked and validated by numerous staff of BMGS and Beacon Minerals.
Site visits	 Comment on any site visits undertaken by the Competent Person and the outcome of those visits. If no site visits have been undertaken indicate why this is the case. 	 Mr. Finch was on-site throughout Stage 1 & 2, as well as the conclusion of the diamond program. A BMGS Senior Geologist provided daily supervision of the diamond drill program. An Independent Geologist was on-site throughout the Stage 4 and Stage 5 RC program's. Mr. Mapleson is based out of the BMGS Kalgoorlie office and oversaw the various drilling campaigns.
Geological interpretation	 Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. Nature of the data used and of any assumptions made. The effect, if any, of alternative interpretations on Mineral Resource estimation. The use of geology in guiding and controlling Mineral Resource estimation. The factors affecting continuity both of grade and geology. 	 Consistent logging of the lithology has correlated well with resultant assay values. A distinct correlation was identified between gold mineralisation and the presence of a heavily silicified siltstone and clay units. RC, AC and diamond drilling data has been used in the estimation. Aerial photography and geological logging were used to aid the interpretation. Fundamental palaeochannel characteristics were identified, confirming the style of mineralisation. No known factors have been identified to influence grade and/ or geological continuity of the deposit.
Dimensions	The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	 The Eastern Arm of mineralisation extends 1,450m along strike, 180m in width, is an average of 8m thick and is at average of 10m below the natural surface. The Western arm of mineralisation extends 250m along strike, 140m in width, is an average of 7m thick and is at an average of 10m below the natural surface. A third domain exists as a low-grade repetition of mineralisation, below the central regions of the main ore horizon. Dimensions of the third domain are 230m in length, 80m in width and 2m thick.
Estimation and modelling techniques	The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and	Grade estimation was completed via ordinary kriging (OK) for the two main ore domains and Inverse-distance-squared (ID²) techniques for the smaller low-grade domain. A nested spherical variogram with two structures was derived for each OK domain using Snowden Supervisor software. The variogram was created as normal

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Criteria	JC	ORC Code explanation	С	ommentary
		parameters used.		scores and was back transformed for use with 3DS Surpac modelling software.
	•	The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	•	Nil assumptions were made.
	•	The assumptions made regarding recovery of by-products.	•	Three domains were created, based on variable grade distribution and orientation of mineralisation.
	•	Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	•	A statistical analysis was undertaken, with nil extreme or outlier Gold grades identified. A low coefficient of variation value exists with all domains.
	•	In the case of block model interpolation, the block size in relation to the average	•	Nil by-products have been identified.
		sample spacing and the search employed.	•	Nil deleterious elements have been identified.
	•	Any assumptions behind modelling of selective mining units. Any assumptions about correlation between variables.	•	Block size was determined via a kriging neighbourhood analysis (KNA), using Snowder Supervisor software. A series of checks are used to confirm the block size to be being geologically suitable.
	•	Description of how the geological interpretation was used to control the resource estimates. Discussion of basis for using or not using grade cutting or capping.	•	The selective mining unit (SMU) was developed based on open-pit mining using a 120t backhoe excavator.
	•		•	Nil assumptions were made regarding correlation between variables
	•	 The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available. 		A statistical analysis was undertaken for determination of a Gold top-cut for each domain. Grade distribution was determined to be homogenous; as a result, a top-cut was not required.
			•	A previous 2009 resource estimate by BMGS was used as a check, as well visual checks and a series of swath validation plots that spatially compare block grades to rav composite data.
			•	Nil reconciliation data was available.
Moisture	•	Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	•	Tonnage has been estimation on a dry basis. Moisture values were obtained from diamond core analysis. The Diamond core samples were weighed prior to a wax immersion SG analysis. After the analysis, the samples were dried and re-weighed to obtain a moisture value.
Cut-off parameters	•	The basis of the adopted cut-off grade(s) or quality parameters applied.	•	A suite of cut-off grades was presented for a scoping study. 0.5g/t Au was selected as the optimal cut-off grade.
Mining factors or assumptions	• S	Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	•	The assumption of open-pit mining, using a 120t backhoe excavator was used. Minimal mining dilution is expected due to the simplicity and orientation of mineralisation.
Metallurgical	•	The basis for assumptions or predictions regarding metallurgical amenability. It is	•	Detailed metallurgical analysis is underway and will be factored into the economics of



Criteria	J(ORC Code explanation	C	ommentary					
factors or assumptions		always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	•	the deposit when complete. • Further work will be undertaken to identify any potential deleterious elements.					
Environmen-tal factors or assumptions	•	Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	 Waste material is expected to be back-filled into completed sectors of the open-p The location of ore treatment is yet to be determined. A detailed environmental study will be undertaken before any mining activity take place. 						
Bulk density	•	Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.	•		core represer		orlie via a wax immersion SG from a variety of locations		
	•	The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.					culated moisture values were Lost Dog model as a bulk		
				Rock Unit	Wet SG	Avg Moisture %	Dry SG		
				Siltstone	2.45	2%	2.4		
				Siltstone/Claystone	1.80	25%	1.4		
				Claystone/Siltstone	1.69	31%	1.2		
				Claystone	1.65	35%	1.1		
			•	A down-hole density analysis of the Diamo		orovided additional corr	relation with wet SG data from		
Classification	•	The basis for the classification of the Mineral Resources into varying confidence categories.	•	Resource classification grade continuity between			d on drill-hole density and		
	•	Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in	 Data integrity has been analysed and a high level of confidence has been placed on the dataset and resultant resource estimation. 						
		continuity of geology and metal values, quality, quantity and distribution of the data).	•			resource estimation pro methodology reported	ocess was undertaken during as suitable and		



Criteria	JORC Code explanation	Commentary
_	Whether the result appropriately reflects the Competent Person's view of the deposit.	representative of the deposit.
		 Mr. Finch and Mr. Mapleson retain a high degree of confidence in the result of the resource estimation.
Audits or reviews	The results of any audits or reviews of Mineral Resource estimates.	 An independent audit of the entire resource estimation process was undertaken during May 2017, with all parameters and methodology reported as suitable and representative of the deposit.
Discussion of relative accuracy/ confidence	 Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. 	mining observations on the lease has resulted in a high level of confidence of the estimation on a global scale.
	 These statements of relative accuracy and confidence of the estimate should be compared with production data, where available. 	

Appendix 2

Drill Results used for the June 2017 Mineral Resource

All Drilling - June Mineral Resource

Hole II	MGA Northing (mN)	MGA Easting (mE)	Elevation (mRL)	Hole Depth (m)	Azimuth (🛭)	<i>Dip (</i> 2)	Intercept Grade (g/t Au)	Intercept (m)	Intercept From (m)	Intercept To (m)
JD17C1	o 6,598,433	303,362	382	30	000	-90	10.2	11	9	20
JD17C1	6,598,456	303,198	383	60	000	-90	6.0	14	8	22
JD17C1	6,598,460	303,150	383	25	000	-90	4.1	18	7	25

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JD17C160	6,598,459	303,126	383	25	000	-90	4.0	18	7	25
JD17C207	6,598,515	303,708	380	30	000	-90	11.7	6	14	20
JD17C165	6,598,454	303,251	382	25	000	-90	4.9	13	10	23
JD17C67	6,598,427	303,085	384	30	000	-90	3.8	16	8	24
JD17C170	6,598,459	303,379	381	55	000	-90	3.5	17	7	24
JD17C162	6,598,455	303,178	383	25	000	-90	5.0	12	10	22
JD17C159	6,598,461	303,101	383	25	000	-90	4.2	14	7	21
JD09-044	6,598,479	302,747	383	30	000	-90	5.4	10	12	22
JD17C213	6,598,584	303,790	381	25	000	-90	5.4	10	13	23
JD17C182	6,598,406	303,225	383	25	000	-90	3.5	14	8	22
JD17C208	6,598,533	303,698	380	30	000	-90	4.4	11	10	21
JD17C19	6,598,382	303,182	384	30	000	-90	3.3	14	7	21
JD09-031	6,598,432	302,722	383	30	000	-90	4.0	11	11	22
JD17C171	6,598,456	303,401	381	25	000	-90	4.9	9	10	19
GC02	6,598,482	302,720	381	30	182	-60	5.2	8	16	24
JD17C15	6,598,437	303,213	383	30	000	-90	2.8	15	6	21
JD17C69	6,598,440	303,039	383	30	000	-90	3.0	13	8	21
JD17C288	6,598,488	303,632	380	25	000	-90	5.1	7	13	20
JD17C11	6,598,436	303,336	382	30	000	-90	2.7	13	8	21
JDD002	6,598,383	303,183	384	29	000	-90	3.1	10.9	7.9	18.8
JD17C21	6,598,388	303,294	383	30	000	-90	2.4	14	6	20
JD17A07	6,598,471	302,691	385	30	000	-90	2.5	13	8	21
CC0042	6,598,379	303,250	383	20	000	-90	2.2	15	5	20
JD17C68	6,598,445	303,060	383	30	000	-90	3.1	10	10	20
JD17C17	6,598,437	303,162	383	30	000	-90	2.0	16	6	22
JD17C176	6,598,407	303,076	384	50	000	-90	2.6	12	9	21
JD17C214	6,598,567	303,802	381	25	000	-90	3.4	9	11	20
CC0048	6,598,431	303,251	383	30	000	-90	2.1	15	8	23



JD17C119	6,598,435	303,437	382	30	000	-90	2.4	13	9	22
JD09-021	6,598,499	302,702	383	30	000	-90	2.8	11	8	19
JD17C52	6,598,437	303,383	382	30	000	-90	2.4	13	8	21
CC0028	6,598,300	303,120	383	24	000	-90	2.2	14	10	24
JD17C03	6,598,486	303,210	382	30	000	-90	6.1	5	17	22
CC0046	6,598,412	303,272	382	31	000	-90	1.9	16	8	24
JD17C13	6,598,434	303,285	382	30	000	-90	1.9	16	8	24
JD17C191	6,598,537	303,760	381	42	000	-90	2.7	11	11	22
JD09-035	6,598,575	302,722	383	30	000	-90	3.7	8	8	16
JD17C167	6,598,460	303,300	382	25	000	-90	4.2	7	18	25
JD17C289	6,598,487	303,661	380	25	000	-90	3.7	8	12	20
JD17C181	6,598,408	303,200	383	25	000	-90	1.9	15	6	21
JD17C178	6,598,404	303,123	384	25	000	-90	2.2	13	7	20
JD09-030	6,598,423	302,725	383	30	000	-90	1.4	20	10	30
JD17C183	6,598,412	303,247	383	25	000	-90	2.2	13	8	21
JD17C44	6,598,487	303,413	381	30	000	-90	2.8	10	10	20
JD17C287	6,598,486	303,610	380	25	000	-90	3.1	9	10	19
JD17C49	6,598,429	303,136	383	30	000	-90	2.1	13	7	20
JD17C177	6,598,405	303,101	384	25	000	-90	1.7	16	7	23
JD17C83	6,598,386	302,938	384	30	000	-90	1.4	20	6	26
JD17C14	6,598,420	303,271	382	30	000	-90	2.1	13	8	21
JD17C168	6,598,460	303,328	382	23	000	-90	1.8	15	8	23
JD17C175	6,598,408	303,049	384	25	000	-90	2.4	11	14	25
JD17C48	6,598,430	303,109	384	30	000	-90	2.2	12	7	19
JD17C229	6,598,631	303,812	382	23	000	-90	3.7	7	12	19
JD09-040	6,598,415	302,747	383	30	000	-90	3.7	7	15	22
JD17C180	6,598,408	303,177	383	25	000	-90	1.8	14	6	20
JD17C56	6,598,383	303,261	383	30	000	-90	1.5	16	6	22



GC17	6,598,503	302,740	380	24	000	-90	4.1	6	13	19
JD17C50	6,598,434	303,235	383	30	000	-90	1.6	15	6	21
JD17C66	6,598,384	303,063	384	30	000	-90	1.6	15	8	23
JD17D070	6,598,854	302,664	385	56	000	-90	1.6	15	8	23
JD17C201	6,598,548	303,717	381	30	000	-90	2.0	12	12	24
JD17C186	6,598,410	303,349	382	25	000	-90	1.5	16	6	22
JD09-029	6,598,411	302,724	383	28	000	-90	2.6	9	12	21
JD17D072A	6,598,874	302,694	385	57	000	-90	2.6	9	12	21
CC0008	6,598,295	303,099	383	24	000	-90	2.0	12	10	22
JD17C172	6,598,411	302,974	384	30	000	-90	2.4	10	12	22
JD17C90	6,598,361	303,063	384	30	000	-90	1.4	17	7	24
JD17C179	6,598,408	303,150	383	25	000	-90	1.9	12	6	18
JDD003	6,598,386	303,064	384	28	000	-90	1.8	12.6	9.4	22
CC0033	6,598,423	302,774	383	30	000	-90	2.1	11	13	24
JD17C164	6,598,458	303,227	383	25	000	-90	1.8	13	9	22
CC0039	6,598,343	303,255	383	20	000	-90	2.2	10	9	19
JD17C200	6,598,530	303,731	381	30	000	-90	2.2	10	13	23
JD17C85	6,598,383	302,984	384	30	000	-90	1.6	14	8	22
GC03	6,598,495	302,730	380	30	181	-60	1.4	16	8	24
JD09-041	6,598,425	302,748	383	30	000	-90	2.7	8	18	26
JRC012	6,598,420	302,706	383	57	000	-90	2.3	9	11	20
CC0002	6,598,293	303,069	383	21	000	-90	2.9	7	12	19
JD17C54	6,598,384	303,410	383	30	000	-90	1.2	16	5	21
CC0047	6,598,419	303,260	382	31	000	-90	1.5	13	9	22
JA24	6,598,526	302,786	384	30	000	-90	6.6	3	18	21
JD17C87	6,598,379	303,033	384	30	000	-90	1.2	17	7	24
JD17C16	6,598,434	303,187	383	30	000	-90	1.2	16	6	22
JD17C185	6,598,412	303,327	382	25	000	-90	1.4	14	6	20



GC22	6,598,510	302,703	381	24	000	-90	1.4	14	7	21
JD17C57	6,598,383	303,236	383	30	000	-90	1.7	11	9	20
JD17C228	6,598,613	303,830	381	25	000	-90	1.9	10	12	22
JD17C84	6,598,380	302,963	384	30	000	-90	1.3	14	7	21
JD17C158	6,598,460	303,076	383	25	000	-90	1.2	15	6	21
CC0045	6,598,406	303,283	382	25	000	-90	1.1	16	7	23
JD17C212	6,598,606	303,770	381	25	000	-90	1.8	10	14	24
JD17C46	6,598,488	303,136	383	30	000	-90	3.6	5	17	22
JD17C18	6,598,385	303,161	383	30	000	-90	1.4	13	7	20
JD09-060	6,598,503	302,797	383	30	000	-90	3.0	6	20	26
CC0035	6,598,496	302,730	383	30	000	-90	1.0	17	10	27
JD17C65	6,598,385	303,085	384	30	000	-90	1.5	12	9	21
JD09-042	6,598,435	302,748	383	30	000	-90	1.4	12	18	30
JD09-032	6,598,508	302,728	383	30	000	-90	1.0	18	12	30
CC0040	6,598,355	303,254	383	20	000	-90	1.4	12	7	19
JD17C187	6,598,410	303,373	382	25	000	-90	1.4	12	7	19
JD17C286	6,598,488	303,586	380	25	000	-90	2.8	6	13	19
GC05	6,598,471	302,753	380	24	000	-90	2.4	7	13	20
JDD004	6,598,360	303,040	384	30	000	-90	1.3	12.7	7.7	20.4
JD17C280	6,598,435	303,712	380	25	000	-90	2.1	8	7	15
CC0004	6,598,300	303,138	383	24	000	-90	2.0	8	12	20
JD17C77	6,598,431	302,835	383	30	000	-90	1.5	11	13	24
JDD005	6,598,387	302,940	384	30	000	-90	1.6	10	7.1	17.1
JAC008	6,598,586	302,700	383	30	000	-90	2.3	7	17	24
JD17C89	6,598,362	303,088	384	30	000	-90	1.0	16	7	23
JD09-056	6,598,548	302,774	384	30	000	-90	1.8	9	13	22
GC08	6,598,497	302,730	380	24	000	-90	1.1	14	7	21
JD17C82	6,598,383	302,913	384	30	000	-90	2.2	7	6	13



JD17C59	6,598,385	303,108	384	30	000	-90	1.2	13	8	21
JD17C88	6,598,361	303,110	384	30	000	-90	1.2	13	7	20
JD17C202	6,598,565	303,705	381	30	000	-90	2.6	6	12	18
JDD001	6,598,390	303,295	383	28	000	-90	1.1	14	8.9	22.9
JD17C09A	6,598,482	303,362	381	30	000	-90	1.5	10	10	20
JD17C58	6,598,385	303,129	384	30	000	-90	1.1	14	7	21
JD09-026	6,598,617	302,695	383	30	000	-90	0.8	18	8	26
JD17C230	6,598,655	303,805	382	25	000	-90	1.3	11	13	24
JD17C23	6,598,384	303,341	383	30	000	-90	1.5	10	7	17
JD17C76	6,598,431	302,859	383	30	000	-90	1.4	10	12	22
CC0036	6,598,493	302,686	383	30	000	-90	1.1	13	10	23
GC15	6,598,504	302,720	381	24	000	-90	1.1	13	7	20
JD17C86	6,598,385	303,007	384	30	000	-90	1.1	13	8	21
JD17C169	6,598,460	303,354	382	25	000	-90	1.1	13	10	23
JD17C190	6,598,552	303,750	381	51	000	-90	0.9	15	9	24
JD17C70	6,598,431	303,008	383	30	000	-90	2.2	6	16	22
CC0019	6,598,305	303,181	383	24	000	-90	1.3	10	9	19
JD17C235	6,598,642	303,874	382	25	000	-90	2.2	6	14	20
JD17C216	6,598,522	303,828	382	25	000	-90	2.2	6	14	20
JD17C29	6,598,350	303,247	383	30	000	-90	2.7	5	11	16
JD17C45	6,598,488	303,385	381	30	000	-90	1.5	9	11	20
GC01	6,598,483	302,709	382	32	183	-60	1.3	10	14	24
JD17C12	6,598,434	303,313	382	30	000	-90	1.1	12	10	22
JD17C234	6,598,665	303,864	382	25	000	-90	3.2	4	13	17
JD09-045	6,598,505	302,747	383	30	000	-90	3.2	4	18	22
JD17C95	6,598,338	302,937	384	30	000	-90	3.2	4	10	14
GC09	6,598,484	302,720	381	24	000	-90	1.0	12	8	20
JD17C290	6,598,481	303,695	380	25	000	-90	2.5	5	13	18



JD17C22	6,598,386	303,321	383	30	000	-90	1.1	11	6	17
CC0041	6,598,365	303,253	383	20	000	-90	0.9	14	6	20
GC13	6,598,500	302,703	382	24	000	-90	1.2	10	8	18
JD17C20	6,598,382	303,213	383	30	000	-90	1.1	11	7	18
CC0012	6,598,304	303,078	383	22	000	-90	1.7	7	11	18
CC0044	6,598,388	303,248	383	25	000	-90	1.1	11	7	18
JD17C184	6,598,406	303,301	382	25	000	-90	0.8	14	6	20
JD17C79	6,598,381	302,837	384	30	000	-90	1.0	12	8	20
JD17C91	6,598,358	303,038	384	30	000	-90	1.0	12	7	19
JD17C75	6,598,440	302,877	383	30	000	-90	3.8	3	21	24
CC0016	6,598,298	303,152	383	24	000	-90	1.4	8	11	19
GC11	6,598,494	302,710	381	24	000	-90	1.1	10	10	20
JD17C24	6,598,381	303,366	383	30	000	-90	0.8	13	6	19
JD17C166	6,598,457	303,272	382	25	000	-90	1.0	10	12	22
CC0032	6,598,400	302,770	383	30	000	-90	1.3	8	12	20
JD17C80	6,598,381	302,865	384	30	000	-90	5.1	2	9	11
JD17C53	6,598,435	303,413	382	30	000	-90	1.5	7	9	16
JD17C188	6,598,408	303,399	382	25	000	-90	1.1	9	9	18
JD17C281	6,598,435	303,737	380	25	000	-90	2.0	5	11	16
GC10	6,598,484	302,709	382	27	000	-90	1.4	7	13	20
JD17C08A	6,598,482	303,336	381	30	000	-90	1.4	7	11	18
CC0020	6,598,315	303,178	384	24	000	-90	1.3	7	9	16
JD09-048	6,598,537	302,737	383	30	000	-90	1.3	7	12	19
JD17C130	6,598,384	303,438	383	30	000	-90	0.9	10	7	17
CC0043	6,598,334	303,256	383	24	000	-90	1.0	9	11	20
JD17C209	6,598,549	303,681	381	30	000	-90	1.2	7	14	21
JD17C196	6,598,453	303,781	382	30	000	-90	1.7	5	12	17
JD17C297	6,598,532	303,662	380	25	000	-90	1.2	7	16	23



JAC007	6,598,501	302,713	383	30	000	-90	0.8	10	13	23
JD17C215	6,598,545	303,812	381	25	000	-90	0.8	11	9	20
JD09-046	6,598,515	302,741	383	30	000	-90	1.2	7	19	26
GC20	6,598,513	302,719	381	24	000	-90	0.7	12	8	20
JD17C203	6,598,444	303,761	381	30	000	-90	0.8	10	9	19
JD17C94	6,598,335	302,964	384	30	000	-90	0.6	13	9	22
JD17C98	6,598,335	302,860	384	30	000	-90	1.6	5	8	13
JD17C204	6,598,462	303,749	381	30	000	-90	1.1	7	12	19
GC21	6,598,515	302,711	381	24	000	-90	1.0	8	13	21
JD17C100	6,598,289	302,860	383	30	000	-90	0.8	9	5	14
JD17A11	6,598,432	302,671	385	30	000	-90	1.5	5	9	14
JD17C282	6,598,483	303,490	381	25	000	-90	1.8	4	10	14
JD17C227	6,598,590	303,843	381	25	000	-90	1.5	5	14	19
JD17C291	6,598,480	303,712	380	25	000	-90	1.2	6	13	19
JD17C55	6,598,382	303,387	383	30	000	-90	0.7	11	6	17
JD17C236	6,598,625	303,889	382	25	000	-90	0.9	8	11	19
JA13	6,598,576	302,758	384	41	000	-90	1.8	4	11	15
JD17C226	6,598,569	303,860	381	25	000	-90	0.7	10	11	21
JD17C174	6,598,409	303,025	384	30	000	-90	1.2	6	16	22
JD17C283	6,598,487	303,515	380	25	000	-90	1.4	5	13	18
CC0056	6,598,340	303,120	383	24	000	-90	0.6	12	10	22
JD17C220	6,598,439	303,883	383	25	000	-90	2.3	3	10	13
JD17C114	6,598,484	302,984	383	30	000	-90	2.3	3	10	13
JD09-055	6,598,539	302,776	384	29	000	-90	0.9	8	14	22
CC0021	6,598,302	302,980	383	24	000	-90	2.3	3	12	15
CC0013	6,598,314	303,105	383	24	000	-90	0.8	8	11	19
JD17A01	6,598,311	302,810	383	30	000	-90	1.7	4	7	11
JD17C195	6,598,475	303,801	382	44	000	-90	1.3	5	12	17



GC04	6,598,494	302,741	381	24	000	-90	0.7	9	10	19
JRC011	6,598,453	302,689	383	34	000	-90	0.6	10	10	20
CC0049	6,598,337	302,983	384	24	000	-90	1.2	5	10	15
CC0010	6,598,291	303,004	383	22	000	-90	1.0	6	10	16
JD17A02	6,598,322	302,791	383	30	000	-90	1.0	6	7	13
JD17C173	6,598,404	303,000	384	30	000	-90	0.5	13	8	21
JD17C251	6,598,689	303,901	382	25	000	-90	1.5	4	15	19
JD17C285	6,598,491	303,561	380	25	000	-90	2.9	2	18	20
JD17A08	6,598,450	302,690	385	30	000	-90	0.8	7	13	20
JD17C04	6,598,485	303,235	382	30	000	-90	1.4	4	17	21
JD17C296	6,598,532	303,634	380	25	000	-90	1.9	3	21	24
JD09-043	6,598,460	302,749	383	30	000	-90	1.0	6	12	18
JD17A10	6,598,403	302,685	385	30	000	-90	0.8	7	10	17
JD17C02A	6,598,487	303,186	382	30	000	-90	0.8	7	8	15
CC0005	6,598,316	303,132	383	24	000	-90	0.6	10	10	20
JD09-015	6,598,400	302,695	383	15	000	-90	1.4	4	11	15
CC0024	6,598,300	303,040	383	24	000	-90	0.8	7	10	17
CC0018	6,598,302	302,921	384	24	000	-90	0.9	6	10	16
JD17C206	6,598,497	303,721	380	30	000	-90	1.1	5	15	20
GC12	6,598,494	302,720	381	24	000	-90	0.5	11	7	18
CC0001	6,598,290	303,054	383	20	000	-90	0.7	8	10	18
JD17C252	6,598,717	303,890	383	25	000	-90	1.4	4	16	20
JD17C141	6,598,336	303,397	383	30	000	-90	1.8	3	8	11
JD09-033	6,598,538	302,725	383	30	000	-90	1.7	3	19	22
JD17C01A	6,598,489	303,162	382	30	000	-90	0.7	7	12	19
CC0027	6,598,323	303,208	384	20	000	-90	0.6	9	11	20
JD09-062	6,598,525	302,802	383	30	000	-90	1.7	3	18	21
JD17C273	6,598,750	303,991	383	25	000	-90	1.3	4	12	16



JD17C121	6,598,434	303,485	382	30	000	-90	1.0	5	11	16
JD09-024	6,598,562	302,699	383	30	000	-90	2.5	2	19	21
JD09-025	6,598,599	302,694	383	30	000	-90	1.2	4	18	22
JD17C197	6,598,471	303,769	381	30	000	-90	0.5	9	10	19
JD17C30	6,598,340	303,129	385	30	000	-90	0.9	5	8	13
JB1	6,598,603	302,748	384	52	000	-90	1.2	4	10	14
JD09-061	6,598,515	302,801	384	30	000	-90	0.6	8	14	22
JD17C217	6,598,504	303,842	382	25	000	-90	0.6	8	11	19
JD17C293	6,598,529	303,560	380	25	000	-90	2.2	2	21	23
CC0029	6,598,336	303,210	383	20	000	-90	1.1	4	9	13
JAC011	6,598,459	302,702	383	30	000	-90	1.4	3	20	23
JA26	6,598,437	302,820	383	30	000	-90	2.1	2	15	17
JD17C218	6,598,476	303,856	382	25	000	-90	0.8	5	15	20
JD17C192	6,598,520	303,770	381	48	000	-90	1.0	4	14	18
JD17C224	6,598,526	303,884	382	25	000	-90	0.7	6	13	19
JD17C93	6,598,334	302,983	384	30	000	-90	0.7	6	8	14
JD17C101	6,598,284	302,886	383	30	000	-90	1.0	4	9	13
JD17C303	6,598,587	303,665	381	25	000	-90	1.3	3	22	25
JD17C156	6,598,457	303,028	383	30	000	-90	1.0	4	14	18
JD17A12	6,598,281	302,662	384	30	000	-90	1.2	3	6	9
JD17C205	6,598,480	303,734	381	30	000	-90	1.8	2	17	19
JD17C274	6,598,762	303,973	383	25	000	-90	1.2	3	15	18
GC07	6,598,493	302,748	380	24	000	-90	0.7	5	15	20
JD17C193	6,598,508	303,782	381	48	000	-90	1.8	2	16	18
GC19	6,598,511	302,729	380	24	000	-90	1.2	3	14	17
JDD006	6,598,481	302,749	380	23	000	-90	3.5	1	18.4	19.4
JD17C131	6,598,382	303,463	383	30	000	-90	0.9	4	7	11
JD17C237	6,598,607	303,906	382	25	000	-90	1.2	3	17	20



CC0017	6,598,313	303,152	383	23	000	-90	0.6	6	10	16
CC0030	6,598,333	303,229	383	20	000	-90	1.1	3	10	13
JD17C241	6,598,520	303,964	382	25	000	-90	0.8	4	18	22
JD17C154	6,598,458	302,976	383	30	000	-90	1.6	2	14	16
JD17C71	6,598,436	302,983	383	30	000	-90	1.1	3	16	19
CC0025	6,598,293	303,206	383	20	000	-90	1.0	3	8	11
JD17C295	6,598,531	303,611	380	25	000	-90	1.5	2	21	23
JD17A04	6,598,340	302,750	384	30	000	-90	1.0	3	8	11
JD17C255	6,598,736	303,929	383	25	000	-90	1.5	2	23	25
JD17C135	6,598,384	303,561	382	30	000	-90	1.0	3	9	12
CC0007	6,598,352	303,146	383	21	000	-90	1.5	2	10	12
JD17C122	6,598,437	303,510	382	30	000	-90	0.7	4	12	16
CC0031	6,598,290	303,152	383	20	000	-90	0.6	5	8	13
JD17C05	6,598,484	303,263	382	30	000	-90	0.7	4	16	20
JD17C211	6,598,631	303,758	381	25	000	-90	0.9	3	22	25
JD17C189	6,598,571	303,738	381	50	000	-90	1.4	2	13	15
GC14	6,598,504	302,711	381	24	000	-90	0.3	11	9	20
JD17C194	6,598,490	303,793	382	48	000	-90	0.7	4	13	17
GC18	6,598,511	302,739	380	24	000	-90	0.6	4	15	19
JD17C157	6,598,459	303,051	383	25	000	-90	0.5	5	14	19
JD17C307	6,598,446	303,824	382	25	000	-90	0.8	3	7	10
JD17A09	6,598,426	302,689	385	30	000	-90	0.7	3	8	11
JD17C258	6,598,675	303,974	382	25	000	-90	0.5	4	16	20
CC0022	6,598,289	303,172	383	24	000	-90	0.5	4	12	16
JD17C219	6,598,459	303,871	382	25	000	-90	0.5	4	10	14
JD17C272	6,598,724	304,001	383	25	000	-90	0.5	4	13	17
CC0015	6,598,300	302,860	384	24	000	-90	1.0	2	10	12
JD17C155	6,598,457	303,000	383	30	000	-90	1.0	2	14	16



CC0009	6,598,290	303,031	383	20	000	-90	0.6	3	11	14
JD17C120	6,598,437	303,461	382	30	000	-90	0.5	4	8	12
JD17C250	6,598,668	303,916	382	25	000	-90	0.6	3	16	19
JD09-057	6,598,555	302,773	383	30	000	-90	0.9	2	12	14
JD09-023	6,598,537	302,695	383	16	000	-90	1.8	1	15	16
JD17C136	6,598,384	303,587	382	30	000	-90	0.8	2	8	10
JD17C25	6,598,335	303,369	383	30	000	-90	0.8	2	10	12
JD17C210	6,598,650	303,747	381	25	000	-90	1.6	1	24	25
JD17C294	6,598,530	303,586	380	25	000	-90	0.8	2	21	23
JD17C242	6,598,492	303,977	382	25	000	-90	0.8	2	19	21
JD17C60	6,598,484	303,439	381	30	000	-90	0.8	2	14	16
JD17C271	6,598,702	304,015	382	25	000	-90	0.7	2	10	12
JD17A13	6,598,281	302,711	383	30	000	-90	0.7	2	6	8
JD17C72	6,598,434	302,962	383	30	000	-90	0.5	3	17	20
JD09-014	6,598,392	302,695	383	31	000	-90	0.7	2	11	13
JD17C92	6,598,335	303,008	384	30	000	-90	0.7	2	8	10
CC0034	6,598,441	302,770	383	30	000	-90	0.7	2	17	19
CC0026	6,598,308	303,201	384	20	000	-90	0.7	2	10	12
JA25	6,598,482	302,797	384	30	000	-90	0.7	2	17	19
JD17C304	6,598,581	303,690	381	25	000	-90	0.7	2	22	24
JD17A03	6,598,331	302,770	383	30	000	-90	0.7	2	8	10
JD17C254	6,598,754	303,914	383	25	000	-90	0.6	2	23	25
JD17C26	6,598,332	303,343	383	30	000	-90	0.6	2	9	11
JD17C222	6,598,487	303,915	382	25	000	-90	0.6	2	11	13
JD17C248	6,598,636	303,942	382	25	000	-90	0.6	2	18	20
JD17C221	6,598,464	303,925	383	25	000	-90	0.6	2	12	14
JD17C249	6,598,649	303,928	382	25	000	-90	0.6	2	17	19
JD17C123	6,598,436	303,535	382	30	000	-90	0.6	2	8	10



JD17C292	6,598,528	303,539	380	25	000	-90	0.5	2	20	22
JD09-027	6,598,387	302,728	383	29	000	-90	0.5	2	15	17
JD17C61	6,598,483	303,461	381	30	000	-90	0.5	2	14	16
JD17C223	6,598,506	303,900	382	25	000	-90	0.5	2	17	19
JD17C107	6,598,487	302,820	383	30	000	-90	0.5	2	19	21
JD09-054	6,598,526	302,779	384	30	000	-90	0.5	2	19	21
JD17C239	6,598,567	303,928	382	25	000	-90	0.5	2	19	21
JD17C256	6,598,716	303,947	383	25	000	-90	0.5	2	18	20
JD17A06	6,598,363	302,706	384	30	000	-90	0.5	2	10	12
JD17C137	6,598,386	303,611	381	30	000	-90	0.5	2	8	10
JD17C278	6,598,431	303,659	380	25	000	-90	0.5	2	12	14
JD17C277	6,598,433	303,634	380	25	000	-90	0.5	2	12	14
JD17C199	6,598,511	303,744	381	30	000	-90	0.5	2	17	19
JD17C96	6,598,334	302,912	384	30	000	-90	0.4	2	7	9
JD17C257	6,598,695	303,960	382	25	000	-90	0.4	2	18	20
JD17A05	6,598,352	302,726	383	30	000	-90	0.4	2	7	9
JD17C125	6,598,436	303,588	381	30	000	-90	0.4	2	10	12
JD17A14	6,598,282	302,762	383	30	000	-90	0.4	2	6	8
JD17C198	6,598,491	303,755	381	30	000	-90	0.4	2	15	17
JD17C225	6,598,548	303,871	382	25	000	-90	0.4	2	11	13
JD17C238	6,598,587	303,919	382	25	000	-90	0.4	2	19	21
JD17C97	6,598,329	302,888	384	30	000	-90	0.4	2	8	10
JD09-028	6,598,400	302,726	383	30	000	-90	0.4	2	12	14
JD17C102	6,598,285	302,910	383	30	000	-90	0.7	1	9	10
GC06	6,598,482	302,750	380	24	000	-90	0.1	7	11	18
JD17C124	6,598,438	303,564	381	30	000	-90	0.3	2	7	9
JD17C284	6,598,487	303,536	380	25	000	-90	0.3	2	16	18
JD09-034	6,598,555	302,725	383	29	000	-90	0.2	4	8	12



JD17C132	6,598,381	303,484	383	30	000	-90	0.3	2	8	10
JD17C134	6,598,386	303,536	382	30	000	-90	0.3	2	8	10
CC0006	6,598,336	303,142	383	22	000	-90	0.3	2	10	12
JD17C81	6,598,384	302,887	384	30	000	-90	0.3	2	8	10
JD17C133	6,598,383	303,512	383	30	000	-90	0.2	2	7	9
JD17C240	6,598,539	303,947	382	25	000	-90	0.2	2	19	21
JD17C276	6,598,439	303,609	380	25	000	-90	0.1	2	11	13
JAC012	6,598,418	302,710	383	27	000	-90	0.1	2	14	16
JD17C279	6,598,432	303,685	380	25	000	-90	0.1	1	13	14

APPENDIX 2 LOST DOG COST INPUT TABLE

LOST DOG
COST INPUT PARAMETERS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50	2.00	1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

APPENDIX 3 LOST DOG METALLURGY TEST RESULTS

APPENDIX 3A LOST DOG

ALS METALLURGY REPORT - A18169



Metallurgical Testwork

conducted upon

Jaurdi (Lost Dog) Gold Ore Composites

for

Beacon Minerals Limited



Report No. A18169

October 2017



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FIGURES

Figure 1	Comminution Test Program Flowsheet: Beacon Minerals Ltd - Lost Dog Project
Figure 2	Metallurgical Testwork Program Flowsheet: Beacon Minerals Ltd - Lost Dog Project

APPENDICES

Appendix I	Comminution Testwork - Details and Results
Appendix II	Head Assays and Site Water Analysis
Appendix III	Cyanidation Testwork - Details and Results
Appendix IV	Short Diagnostic Gold Leach Testwork - Details and Results



SUMMARY

In June 2017, ALS was requested by Mr Geoff Greenhill, representing Beacon Minerals Limited, to conduct a defined program of metallurgical testwork on two gold ore composites from the Jaurdi (Lost Dog) Gold Project, Western Australia.

Salient results of the testwork program are as follows:

Comminution Testwork

The Mill Scats were submitted for SMC testwork and Bond abrasion index determinations, whilst a sub-sample of the Lost Dog Composite was submitted for Bond ball mill work index determination. Results are summarised in the table below.

	9	MC Testworl	k	Bond BWi		
Sample ID	DWi (kWh/m³)	Α	b	(kWh/t) *	Bond Ai	
Lost Dog Composite	,	1	-	17.0	-	
Mill Scats Composite	4.28	74.2	0.78	19.9	0.0278	

Closing screen size 106 µm

Extractive Testwork

Sub-sample of the Lost Dog Composite were submitted for cyanide leach testwork. Results from the various leach tests are summarised in the table below.

CYANIDATION TESTWORK – SUMMARY OF RESULTS									
Grind Size	Test No.	% Au Extraction @ hours				Au Grade (g/t)		Consumption (kg/t)	
(µm)		Gravity	2	4	24	Calc'd Head	Leach Residue	NaCN	Lime
	JS3915*	-	71.0	74.1	78.8	1.98	0.42	0.20	15.9
P ₈₀ : 106	JS3916		70.7	74.1	79.0	1.95	0.41	0.43	15.2
	JS3917	2.15	72.0	79.1	82.6	2.10	0.41	0.46	14.4
P ₁₀₀ : 75	JS3934	-	73.0	79.5	85.2	2.23	0.33	0.84	3.01
P ₁₀₀ : 106	JS3947	-	NA	NA	83.9	2.05	0.33	1.11	4.32

^{*} Direct leach; all others are CIL's



Comments on the above data are as follows:

- Leaching under CIL conditions did not result in increased gold extraction, as evidenced by the results from test JS3915 (direct leach) and test JS3915 (CIL).
- Gravity gold recovery was negligible and had no impact on overall gold extraction.
- For test JS3934, the leach feed was stage-ground to P_{100} 75 µm. The finer grind size appears to have resulted in increased gold extraction.
- For test JS3947, the leach feed was also stage-ground, this time to P_{100} 106 μ m. Despite the coarser screen size, the recovery was in-line with that achieved for test JS3934.
- For tests JS3934 and JS3947, the target slurry pH was reduced to 9.4 (compared to 10.5 for the first three tests). This resulted in a significant reduction in lime consumption, albeit with an increase in sodium cyanide consumption.
- Size-by-size gold assays on the JS3916 leach residue suggest that gold extraction is grind-sensitive, with the gold grades dropping in the finer size fractions.
- Diagnostic analysis of the JS3917 leach residue indicated a very small amount of gold could be recovered under aggressive leaching conditions. Most of the gold was recovered via aqua regia digest. It is difficult to determine exactly the mineral species with which this gold is associated.



1. INTRODUCTION

In June 2017, ALS was requested by Mr Geoff Greenhill, representing Beacon Minerals Limited, to conduct a defined program of metallurgical testwork on two gold ore composites from the Jaurdi (Lost Dog) Gold Project located in Western Australia.

The proposed program comprised:

- Sample preparation
- Comminution (SMC, Bond Ai, and BWi) testwork
- Head assays
- Site water analysis
- Grind establishment determination
- Knelson gravity separation testwork
- Size-by-size gold analysis
- Gold leach extraction testwork
- Short diagnostic gold leach testwork.

The proposed test program is presented as flow diagrams in Figures 1 and 2.

The testwork was controlled by Mr Geoff Greenhill, on behalf of Beacon Minerals Limited, with Mr Jack Smith supervising the program on behalf of ALS. Testwork results were communicated to the client immediately when available, which enabled the program to progress on a fully informed basis.

The purpose of this report is to describe testwork procedures and present results with

some commentary and observations.

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2. THE SAMPLES

In June 2017, ALS received two samples from the Jaurdi (Lost Dog) Gold Project, Western Australia. Details of the samples received are presented in the table below.

Sample ID	Mass (kg)
Lost Dog Met Composite	35
Mill Scats	20

In addition, four 20-litre containers of site water and a water sample identified as 'FMR LT3 6/3' were received for use in the test program.

3. SAMPLE PREPARATION

The sample preparation procedures are illustrated in Figures 1 and 2 and are briefly described below:

- (1) The Mill Scats sample was control-crushed to <22.4 mm, homogenised by passing three times through a rotary sample divider (RSD) and split into the following charges:
 - 1 x 15 kg for SMC testwork
 - 1 x 5 kg for Bond abrasion index (Ai) determination
 - 1 x 10 kg for Bond ball mill work index (BWi) determination.
- (2) The Lost Dog Met sample was control-crushed to 100% passing 3.35 mm, then thoroughly homogenised by passing three times through a 12-segment rotary sample divider (RSD). The crushed material was then split into representative 1.0 kg charges for use in the testwork program.



4. SMC TESTWORK

A sub-sample of the Mill Scats sample was submitted for SMC testwork (Figure 1). The standard (full) JKTech drop-weight testwork provides ore-specific parameters for use in the JK Sim Met Mineral Processing Simulator Software and JK Sim Met Crusher model.

The SMC test was developed by SMC Testing Pty Ltd to provide a cost-effective means of obtaining these parameters from drill core or broken ROM rock samples in situations where limited quantities of material are available.

The SMC Test generates a relationship between specific input energy (kWh/t) and the proportion of fragmented/broken product passing a specified sieve size. The results are used to determine the drop-weight index (DWi), which is a measure of the strength of the ore sample when broken under impact conditions. The DWi is directly related to the JK rock breakage parameters A and b, and can be used to determine the values of these parameters.

4.1 Test Procedure

The test was conducted as follows:

(1) A suitable sub-sample from the composite was utilised for in-situ SG determinations and to measure the mass of material per unit length of the sample. In-situ SG measurements were carried out by weighing each specimen in air and then fully submerged in water; the data were inserted in the following formula:

$$Density (in - situ SG) = \frac{A}{A - B}$$

Where:

A = Weight of quartered core specimen in air

B = Weight of specimen when totally submerged in water

- (2) The composite was control crushed to a nominal 25 mm and screened to generate particles in the size range <22.4+19.0 mm.
- (3) One hundred (100) particles in the nominated size range were selected in accordance with the SMC proprietary test procedure. These were divided into 5 groups of 20 specimens.
- (4) Each group of 20 specimens was broken using the JK drop-weight tester at a different specific energy level. The target-specific breakage energy levels (Eis) are listed below:
 - 3.5 kWh/t
 - 2.5 kWh/t
 - 1.0 kWh/t
 - 0.5 kWh/t
 - 0.25 kWh/t.



(5) The drop-weight device was configured such that a drop-weight head was raised to a known height and allowed to fall onto each specimen resting on the anvil. The key measurement used in the drop-weight test is the input energy. This was determined from the mass of the drop-weight head and the distance through which it falls and also the mass of the specimen being tested.

The difference in distance between the initial starting point and the final rest height of the drop-weight head is used to calculate the energy that is expended in breaking the particle.

Thus:

$$Ei = Mg(h - X_m)$$

Where:

Ei = Energy used for breakage

M = Drop-weight mass

g = Gravitational constant

h = Initial height of the drop-weight head above the anvil

 X_m = Final height of the drop-weight head above the anvil

The assumption is made that all the energy provided is utilised in the breakage of the particles (i.e. no rebounds).

$$E_{cs} = E_{is} = E_i/m$$

Where:

 E_{is} = Specific energy input

 E_{cs} = Specific comminution energy

m = Mean particle mass

(6) For each single drop test, the rest height was recorded. After all 20 particles in each group had been broken, the combined product was screened at a sieve size that is one-tenth of the original test particle size.

4.2 Results

The full JKTech SMC report is included in Appendix I, whilst a summary of results is provided in the following table.

	DWi			Derived Values					
Sample ID	(kWh/m³)	SG	Α	b	A*b	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	t _a
Mill Scats	4.28	2.5	74.2	0.78	57.9	14.8	10.0	5.2	0.61



5. BOND ABRASION INDEX (Ai) DETERMINATION

A sub-sample of the Mill Scats was tested to determine the Bond Ai value using the standard procedure developed by F.C. Bond¹ (Figure 1).

5.1 Test Procedure

The test procedure was conducted as follows:

- (1) The testing machine consists of an impeller rotating at 632 rpm within a contrarotating drum. The impeller incorporates a *Bisalloy*, 500 Brinell hardness steel paddle.
- (2) Four successive 400 g samples of ore (<19+12.7 mm) were processed for 15 minutes each in the testing machine.
- (3) At the completion of the test, the weight lost by the paddle was measured to 0.10 of a milligram. The abrasion index, Ai, is equivalent to the paddle weight loss expressed in grams.
- (4) The combined products from the four 15-minute test periods were combined and then screened.

5.2 Results

A detailed test report sheet is included in Appendix I, whilst a summary of results is presented in the following table.

Sample ID	Bond Abrasion Index (Ai)
Mill Scats	0.0278

¹ Bond, F.C. "Metal Wear in Crushing and Grinding" (1963), 54th Ann. Mtg of Inst. Chem. Engrs. Houston, Texas

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6. BOND BALL MILL WORK INDEX (BWi) DETERMINATION

Sub-samples of the Lost Dog and Mill Scats samples were tested using the standardised procedure detailed by F.C. Bond² to determine the Bond BWi using a closing screen size of $106 \mu m$ (Figure 1).

6.1 Test Procedure

The test procedure was as follows:

- (1) The sub-sample was stage-crushed to 100% passing 3.35 mm and test portions rotary split out for the work index test.
- (2) A known volume of ore (700 mL) was ground in the standard mill for a counted number of revolutions.
- (3) The ground material was screened at a test aperture of 106 μ m to remove the <106 μ m material.
- (4) Fresh feed was added to the >106 μm fraction to make-up to the original test weight.
- (5) The number of mill revolutions was adjusted at each cycle until a stable recirculating load was achieved.
- (6) The work index was calculated from the formula:

$$(Wi)_B = \frac{44.5}{(Pi)^{0.23} \times (Gbp)^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}}\right)} \times 1.102$$

Where:

(Wi)B = Work index value expressed in kWh/tonne

Pi = Grindability test aperture (micrometres)

Gbp = Mean of equilibrium grindability values (g/rev)

 $P_{80} = 80\%$ passing size of the equilibrium product (micrometres)

 F_{80} = 80% passing size of the feed to period 1 (micrometres).

² Bond, F.C. "Crushing and Grinding Calculations" (1961) British Chemical Engineering, Vol 6, No's 6, 8



6.2 Results

Detailed test report sheets are presented in Appendix I, whilst a summary of results is presented in the following table.

Samula ID	Micror	metres	Gbp	Bond BWi* (kWh/t)	
Sample ID	F ₈₀	P ₈₀	(g/rev)		
Lost Dog	1860	76	1.100	17.0	
Mill Scats	2844	80	0.888	19.9	

^{* 106} µm closing screen size

7. TESTWORK WATER

Site water was used throughout the test program.

A site water sub-sample was submitted for complete water analysis and the FMR LT3 6/3 water sample was assayed for gold and silver.

The full site water analysis is included in Appendix II and a summary of both water analyses is presented in the table below.

JAURDI (LOST DOG) GOLD PROJECT: WATER ANALYSIS						
Analyte	FMR LT3 6/3					
Au (mg/L)	<0.005	0.755				
Ag (mg/L)	<0.2	0.40				
Cl (mg/L)	28000	-				
SO₄ (mg/L)	4870	-				
TDS	54900	-				
рН	7.84	-				
Cond (ms/cm)	71.6	-				
SG	1.045	-				

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8. ANALYTICAL PROCEDURES

All assay samples generated during the test program were submitted to the on-site analytical laboratory in Balcatta for analysis.

The following analytical techniques were used:

Gold in ores and leach residues: Fire assay/ICP-OES

Sherritt method/CS2000

Multi-element scan of solids: D3 acid digest/ICP-OES

Gold and silver in solution: Direct ICP-MS/ICP-OES

9. HEAD ASSAYS

A sub-sample of the Lost Dog Composite was submitted for head assays (Figure 2). Complete results are included in Appendix II, whilst a summary is presented in the following table.

Comp. ID	$\begin{array}{ccc} Au_1 & Au_2 \\ (g/t) & (g/t) \end{array}$		Au _{AVE} (g/t)	Ag (g/t)	S _{SULPHIDE} (%)	
Lost Dog Composite	2.15	2.09	2.12	4.2	<0.02	

Comments on the above data are as follows:

- Low variance in the duplicate gold assays suggests the sample is unlikely to contain significant coarse gold content.
- Sulphide content is negligible.



10. GRIND ESTABLISHMENT TESTWORK

A sub-sample of the Lost Dog sample was submitted for grind establishment testwork. The objective of the grind establishment was to determine the time required by a laboratory rod mill to grind a sub-sample of <3.35 mm crushed material to the target grind size in preparation for extractive testwork.

10.1 Test Procedure

The required grind time was established as follows:

- (1) The 1.0 kg (P_{100} : 3.35 mm) sub-samples were ground with stainless steel rods in a closed stainless steel mill, at 50% solids (w/w) (Perth tap water), for various times.
- (2) The ground solids were fully removed from the mill and wet screened at 106 µm.
- (3) The oversize fraction was dried and then re-screened over a series of sieves down to 106 µm.
- (4) The resultant sizing data were utilised to determine the requisite grind time necessary to realise the target grind size.

10.2 Grind Time

The required time to grind the composite to the required P_{80} value is presented in the following table.

Sample ID	P ₈₀ : 106 μm Grind Time (min' sec")					
Lost Dog Composite	8'45"					



11. CYANIDATION TESTWORK

Sub-samples of the Lost Dog sample were submitted for cyanidation testwork. Various tests were conducted to determine the impact of the following variables on gold extraction and reagent consumption:

- Grind size
- Gravity gold recovery ahead of cyanide leaching
- Leaching under carbon-in-leach (CIL) conditions
- Slurry pH.

11.1 Gravity Separation Procedure

A sub-sample was submitted for gravity gold recovery ahead of cyanide leach testwork. The procedure used is summarised as follows:

- (1) The 1.0 kg sample was ground to P_{80} 106 μ m and passed through a laboratory *Knelson* KC-MD3 gravity concentrator, with the following specifications:
 - Feed rate ~750 g/min
 - 1500 rpm (60 G's)
 - 3.5 L/min fluidising water flow rate.
- (2) The *Knelson* gravity concentrate was transferred to a 4-litre bottle and subjected to mercury amalgamation. Five grams of mercury was added to the bottle, which was placed on a roller for a period of at least 2 hours.
- (3) On completion of the amalgamation, the loaded amalgam was recovered and assayed for gold and silver.
- (4) The amalgamation tail (*Knelson* concentrate) was combined with the *Knelson* tail.
- (5) The combined gravity tail was submitted for cyanide leach testwork.



11.2 Cyanidation Procedure

The generic test was conducted as follows:

- (1) The milled sample slurry at the target grind size was transferred into a 4-litre leach bottle with a screw-on lid. Slurry agitation was applied by mechanical rollers.
- (2) Site water was added to establish a % solids comprising 35% (w/w).
- (3) Sufficient hydrated lime (60% CaO) was added to the slurry to establish a pH of approximately 10.5 and the slurry was thoroughly agitated for 5 minutes.
- (4) The pH of the slurry was measured again, and if necessary more lime was added to achieve a pH of 10.5.
- (5) For CIL tests, *Haycarb Yao* activated carbon was added at 15 g/L.
- (6) Sodium cyanide was added to the slurry to establish an initial concentration of 0.035% (w/v).
- (7) The slurry was sparged with oxygen to provide an elevated DO content before the bottle was sealed and leaching commenced.
- (8) At regular intervals (2, 4, 6, 8, and 12 hours), slurry pH, DO, and cyanide concentration were monitored and recorded.
- (9) Lime and cyanide were added as required to maintain target levels (pH >10.0, cyanide >0.025 %).
- (10) For CIL tests, loaded carbon samples were removed at each check. The carbon was thoroughly washed, dried, weighed, and submitted for gold and silver assay.
- (11) A 30 mL solution sample was also collected at each check. A 10 mL aliquot was used to determine sodium cyanide concentration by titration with silver nitrate, whilst the remainder was submitted for gold and silver assay.
- (12) Upon termination of the test (24 hours), the terminal pH, oxygen, and cyanide levels were determined.
- (13) The residual slurry sample was filtered, washed, and dried to provide leach.



11.3 Results

Detailed cyanidation test report sheets are included in Appendix III, whilst a summary of results is presented in the following table.

CYANIDATION TESTWORK – SUMMARY OF RESULTS										
Grind Size	To at No		% Au Ex @ ho				irade /t)	Consumption (kg/t)		
(µm)	Test No.	Gravity	2	4	24	Calc'd Head	Leach Residue	NaCN	Lime	
	JS3915*	-	71.0	74.1	78.8	1.98	0.42	0.20	15.9	
P ₈₀ : 106	JS3916	-	70.7	74.1	79.0	1.95	0.41	0.43	15.2	
	JS3917	2.15	72.0	79.1	82.6	2.10	0.41	0.46	14.4	
P ₁₀₀ : 75	JS3934	-	73.0	79.5	85.2	2.23	0.33	0.84	3.01	
P ₁₀₀ : 106	JS3947	-	NA	NA	83.9	2.05	0.33	1.11	4.32	

^{*} Direct leach: all others are CIL's

Comments on the above data are as follows:

- Leaching under CIL conditions did not result in increased gold extraction, as evidenced by the results from test JS3915 (direct leach) and test JS3915 (CIL).
- Gravity gold recovery was negligible and had no impact on overall gold extraction
- For test JS3934, the leach feed was stage-ground to P_{100} 75 μ m. The finer grind size appears to have resulted in increased gold extraction.
- For test JS3947, the leach feed was also stage-ground, this time to P_{100} 106 μ m. Despite the coarser screen size, the recovery was in-line with that achieved for test JS3934. A sizing on the leach tail from each test is recommended so that the actual P_{80} 's can be compared.
- For tests JS3934 and JS3947, the target slurry pH was reduced to 9.4 (compared to 10.5 for the first three tests). This resulted in a significant reduction in lime consumption, albeit with an increase in sodium cyanide consumption.



12. SIZE-BY-ASSAY ANALYSIS

Size-by-size gold analysis was conducted on a sub-sample of the JS3916 leach residue (Figure 2).

12.1 Test Procedure

The test procedure was as follows:

- (1) The sub-sample was wet screened over a 38 µm aperture sieve.
- (2) The screen oversize material was dried and then re-screened over a deck of screens from 106 μ m to 38 μ m. The screen undersize material was combined with the undersize material from step (1).
- (3) Each size fraction was dried and weighed. The weights were used to determine the particle size distribution.
- (4) Sub-samples of each fraction were assayed for gold.

12.2 Results

The size-by-size assays are summarised below, whilst more detailed results are included in Appendix III.

LEACH RESIDUE SIZE-BY-SIZE ASSAYS: SUMMARY OF RESULTS								
Size (mm)	Mass	Gold						
	(%)	g/t	% Distribution					
0.106	19.7	0.74	34.4					
0.075	11.4	0.58	15.6					
0.053	9.05	0.51	10.9					
0.038	7.16	0.41	6.91					
-0.038	52.6	0.26	32.2					
CALCULATED HEAD	100.0	0.42	100.0					
ASSAY HEAD	-	0.41	-					

The size-by-size assay results suggest that gold extraction is sensitive to grind size, with the gold grades dropping in the finer size fractions.



13. SHORT DIAGNOSTIC GOLD LEACH TESTWORK

A sub-sample of the JS3917 leach residue was screened to generate a -75 μ m fraction. This material was subsequently submitted for diagnostic analysis to determine the deportment of unrecovered gold.

13.1 Test Procedure

The sub-sample was subjected to direct intensive cyanidation in accordance with the following test procedure:

- (1) The sample was transferred into a 4-litre leach bottle with a screw-on lid. Slurry agitation was applied by mechanical rollers.
- (2) Perth tap water was added to establish a % solids comprising 40% (w/w).
- (3) Caustic soda (NaOH) at a dosage of 0.7% was added to the slurry to establish a pH in excess of 12.0.
- (4) LeachWELLTM at a dosage of 2.0% was added to the slurry.
- (5) Sodium cyanide solution was added to the slurry to establish an initial cyanide concentration of 5.0% (w/v).
- (6) The leach slurry was sparged with oxygen to provide an elevated DO content to the slurry.
- (7) At the conclusion of the test (24 hours), the terminal pH, oxygen, and cyanide levels were determined and a solution sample was assayed for gold.
- (8) The residual slurry sample was filtered, washed, dried, and weighed to provide leach residue solids. One representative split was assayed for gold. The second leach residue split was subjected to aqua regia digestion to destroy all remaining sulphide minerals (mostly pyrite) and simultaneously release the contained gold into solution.
- (9) The residue from the aqua regia digestion was fire assay smelted to determine the silicate (gangue) encapsulated gold content.



13.2 Results

A detailed diagnostic test report sheet is included in Appendix IV, whilst a summary is presented in the following table.

LEACH RESIDUE DIAGNOSTIC ANALYSIS - SUMMARY OF RESULTS								
Diagnostic	Gold Content Description	Au Dist	ribution					
Stage	dola Content Description	g/t	%					
1	Direct/Cyanidable	0.04	14.56					
2	Acid Digestible Mineral Locked	0.16	62.14					
3	Silicate (Gangue) Encapsulated	0.06	23.30					
	TOTAL GOLD CONTENT	0.26	100.00					

Comments on the above results are as follows:

- The diagnostic results show that a small amount of cyanide-soluble gold remained in the leach residue, although it should be noted that this gold was recovered under very aggressive leaching conditions.
- Most of the gold was recovered via aqua regia digest. It is unclear as to which
 minerals this gold is associated with, however, when examined under a
 microscope, the presence of a small content of oxidised sulphide minerals was
 detected. It is possible that gold is encapsulated within these minerals.



FIGURES



FIGURE 1: COMMINUTION TEST PROGRAM FLOWSHEET

BEACON MINERALS LTD - LOST DOG PROJECT



RECEIVE SAMPLES OF CORE & CONDUCT INVENTORY (~20kg EACH) GENERATE TWO COMPOSITES FOR TESTWORK LOST DOG MET COMPOSITE MILL SCATS COMPOSITE CONTROL CRUSH TO <22.4mm HOMOGENISE & SPLIT 1 x 10.0 kg 1 x 5kg RESERVE [MILL SCATS COMP ONLY] [MILL SCATS COMP ONLY] TO FIG 2 CONTROL CRUSH SCREEN @ TO < 3.35 mm -19.0+12.5mm CONDUCT

GENERATE 100 x SPECIMENS @ -22.4+19.0mm : [5 x SETS OF 20 PIECES] CONDUCT SMC DROP-WT TESTWORK @ 5 ENERGY LEVELS SIZE ANALYSIS ON PRODUCTS FROM SMC TESTWORK RELAY DATA TO JK TECH

FOR THE EVALUATION OF SMC PARAMETERS [DWi, A, b, Ta, Mia]

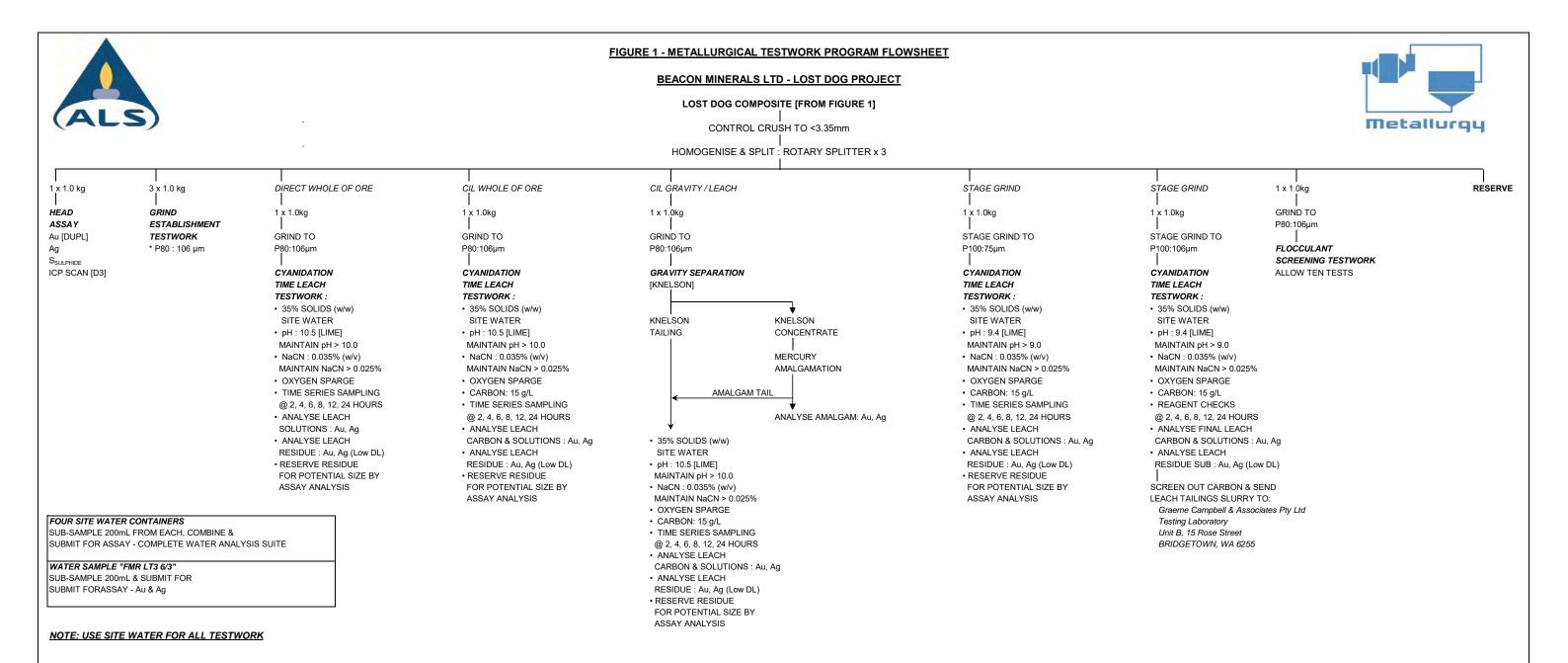
1 x SUB-SAMPLE

FOR SMC

TESTWORK

BOND ABRASION INDEX **DETERMINATION** [Ai]

BOND BALL MILL WORK INDEX DETERMINATION [BWi] Pi : 106µm





APPENDICES



APPENDIX I

Comminution Testwork Details and Results



SMC TEST® REPORT

Beacon Mineral

Tested by: ALS Metallurgy WA

Perth, Western Australia

Prepared by: Matt Weier

JKTech Job No: 17001/P39 Testing Date: June 2017

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1 INTRODUCTION

SMC data for one sample from Lost Dog Project were received from ALS Metallurgy WA on June 28, 2017, by JKTech for SMC test analysis. The sample was identified as Mill Scats Comp. The data were analysed to determine the JKSimMet and SMC Test comminution parameters. SMC Test results were forwarded to SMC Testing Pty Ltd for the analysis of the SMC Test data. Analysis and reporting were completed on June 29, 2017.

2 THE SMC TEST®

2.1 Introduction

The standard JK Drop-Weight test provides ore specific parameters for use in the JKSimMet Mineral Processing Simulator software. In JKSimMet, these parameters are combined with equipment details and operating conditions to analyse and/or predict SAG/autogenous mill performance. The same test procedure also provides ore type characterisation for the JKSimMet crusher model.

The SMC Test was developed by Steve Morrell of SMC Testing Pty Ltd (SMCT). The test provides a cost effective means of obtaining these parameters, in addition to a range of other power-based comminution parameters, from drill core or in situations where limited quantities of material are available. The ore specific parameters have been calculated from the test results and are supplied to Beacon Mineral in this report as part of the standard procedure

2.2 General Description and Test Background

The SMC Test[®] was originally designed for the breakage characterisation of drill core and it generates a relationship between input energy (kWh/t) and the percent of broken product passing a specified sieve size. The results are used to determine the so-called JK Drop-Weight index (DWi), which is a measure of the strength of the rock when broken under impact conditions and has the units kWh/m³. The DWi is directly related to the JK rock breakage parameters A and b and hence can be used to estimate the values of these parameters as well as being correlated with the JK abrasion parameter - t_a . For crusher modelling the t_{10} - E_{cs} matrix can also be derived. This is done by using the size-by-size A*b values that are used in the SMC Test[®] data analysis (see below) to estimate the t_{10} - E_{cs} values for each of the relevant size fractions in the crusher model matrix.

For power-based calculations, (see APPENDIX B), the SMC Test® provides the comminution parameters M_{ia} , M_{ih} and M_{ic} . M_{ia} is the work index for the grinding of coarser particles (> 750 μ m) in tumbling mills such as autogenous (AG), semi-autogenous (SAG), rod and ball mills. M_{ih} is the work index for the grinding in High Pressure Grinding Rolls (HPGR) and M_{ic} for size reduction in conventional crushers.

The SMC Test® is a precision test, which uses particles that are either cut from drill core using a diamond saw to achieve close size replication or else selected from crushed material so that particle mass variation is controlled within a prescribed range. The particles are then broken at a number of prescribed impact energies. The high degree of control imposed on both the size of particles and the breakage energies used, means that the test is largely free of the repeatability problems associated with tumbling-mill based tests. Such tests usually suffer from variations in feed size (which is not closely controlled) and energy input, often assumed to be constant when in reality it can be highly variable (Levin, 1989).

The relationship between the DWi and the JK rock breakage parameters makes use of the size-by-size nature of rock strength that is often apparent from the results of full JK Drop-Weight tests. The effect is illustrated in Figure 1, which plots the normalized values of A*b against particle size. This figure also shows how the gradient of these plots varies across the full range of rock types tested. In the case

of a conventional JK Drop-Weight test, these values are effectively averaged and a mean value of A and b is reported. The SMC Test[®] uses a single size and makes use of relationships such as that shown in Figure 1 to predict the A and b of the particle size that has the same value as the mean for a JK full Drop-Weight test.

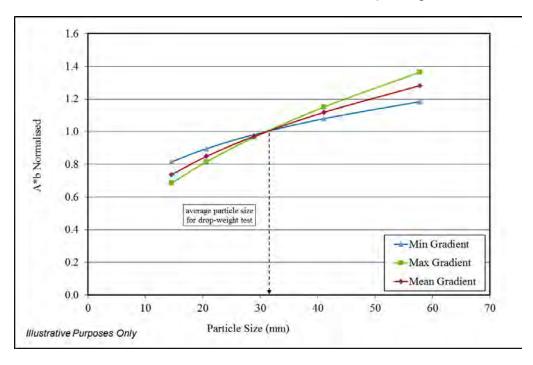


Figure 1 – Relationship between Particle Size and A*b

2.3 The Test Procedure

In the SMC Test[®], five sets of 20 particles are broken, each set at a different specific energy level, using a JK Drop-Weight tester. The breakage products are screened at a sieve size selected to provide a direct measurement of the t₁₀ value.

The test calls for a prescribed target average volume for the particles, with the target being chosen to be equivalent to the mean volume of particles in one of the standard JK Drop-Weight test size fractions.

The rest height of the drop-head (gap) is recorded after breakage of each particle to allow for a correction to the drop energy. After breaking all 20 particles in a set, the broken product is sieved at an aperture size, one tenth of the original particle size. Thus, the percent passing mass gives a direct reading of the t₁₀ value for breakage at that energy level.

There are two alternative methods of preparing the particle sets for breakage testing: the particle selection method and the cut core method. The particle selection method is the most commonly used as it is generally less time consuming. The cut core method requires less material, so tends to be used as a fallback method, only when necessary to cope with restricted sample availability.

2.3.1 Particle Selection Method

For the particle selection method, the test is carried out on material in one of three alternative size fractions: -31.5+26.5, -22.4+19 or -16+13.2 mm. The largest size fraction is preferred but requires more material.

In the particle selection method, particles are chosen so that their individual masses lie within $\pm 30\%$ of the target mass and the mean mass for each set of 20 lies within $\pm 10\%$ of the target mass. A typical set of particles is shown in Figure 2.



Figure 2 – A Typical Set of Particles for Breakage (Particle Selection Method)

Before commencing breakage tests on the particles, the ore density is determined by first weighing a representative sample of particles in air and then in water.

2.3.2 Cut Core Method

The cut core method uses cut pieces of quartered (slivered) drill core. Whole core or half core can be used, but when received in this form it needs to be first quartered as a preliminary step in the procedure. Once quartered, any broken or tapered ends of the quartered lengths are cut, to square them off. Before the lengths of quartered core are cut to produce the pieces for testing, each one is weighed in air and then in water, to obtain a density measurement and a measure of its mass per unit length.

The size fraction targeted when the cut core method is used depends on the original core diameter. The target size fraction is selected to ensure that pieces of the correct volume will have "chunky" rather than "slabby" proportions.

Having measured the density of the core, the target volume can be translated into a target mass and with the average mass per unit length also known, an average cutting interval can be determined for the core.

Sufficient pieces of the quartered core are cut to generate 100 particles. These are then divided into the five sets of 20 and broken in the JK Drop-Weight tester at the five different energy levels. Within each set, the three possible orientations of the particles are equally represented (as far as possible, given that there are 20 particles). The orientations prescribed for testing are shown in Figure 3.



Figure 3 – Orientations of Pieces for Breakage (Cut Core Method)

The cut core method cannot be used for cores with diameters exceeding 70 mm, where the particle masses would be too large to achieve the highest prescribed energy level.

2.4 SMC Test® Results

The SMC Test® results for the Mill Scats Comp sample from Lost Dog Project are given in Table 1. This table includes the average rock density and the DWi (Drop-Weight index) that is the direct result of the test procedure. The values determined for the M_{ia} , M_{ih} and M_{ic} parameters developed by SMCT are also presented in this table. The M_{ia} parameter represents the coarse particle component (down to 750 µm), of the overall comminution energy and can be used together with the M_{ib} (fine particle component) to estimate the total energy requirements of a conventional comminution circuit. The use of these parameters is explained further in APPENDIX B. The derived estimates of parameters A, b and t_a that are required for JKSimMet comminution modelling are given in Table 2.

Also included in the derived results are the SAG Circuit Specific Energy (SCSE) values. The SCSE value is derived from simulations of a "standard" circuit comprising a SAG mill in closed circuit with a pebble crusher. This allows A*b values to be described in a more meaningful form. SCSE is described in detail in APPENDIX A.

In the case of the Mill Scats Comp sample from Lost Dog Project, the $\it A$ and $\it b$ estimates are based on a correlation using the database of all results so far accumulated by SMCT.

Table 1 - SMC Test® Results

Sample	DWi	DWi	<i>Mi</i> Pa			
Designation	(kWh/m³)	(%)	Mia	Mih	Mic	SG
Mill Scats Comp	4.28	21	14.8	10.0	5.2	2.5

For more details on how the M_{ia}, M_{ih} and M_{ic} parameters are derived and used, see APPENDIX B or go to the SMC Testing website at http://www.smctesting.com/about and click on the link to download Steve Morrell's paper on this subject.

Table 2 – Parameters derived from the SMC Test® Results

Sample Designation	А	b	t a
Mill Scats Comp	74.2	0.78	0.61

The influence of particle size on the specific comminution energy needed to achieve a particular t_{10} value can also be inferred from the SMC Test[®] results. The energy requirements for five particle sizes, each crushed to three different t_{10} values, are presented in Table 3.

Table 3 – Crusher Simulation Model Specific Energy Matrix

Sample							Partic	le Size	(mm)						
Designation		14.5		20.6		28.9		41.1		57.8					
			t ₁₀ Values (%) for Given Specific Energies in kWh/t												
	10	20	30	10	20	30	10	20	30	10	20	30	10	20	30
Mill Scats Comp	0.24	0.50	0.80	0.21	0.44	0.70	0.18	0.38	0.61	0.16	0.33	0.53	0.14	0.29	0.46

The SMC Test[®] database now contains over 35,000 test results on samples representing more than 1300 different deposits worldwide.

Around 99% of the DWi values lie in the range 0.5 to14.0 kWh/m³, with soft ores being at the low end of this range and hard ores at the high end.

A cumulative graph of DWi values from the SMC Test® Database is shown in Figure 4 below. This graph can be used to compare the DWi of the material from Lost Dog

Project, with the entire population of ores in the SMCT database. The figures on the y-axis of the graph represent the percentages of all ores tested that are softer than the x-axis (DWi) value selected.

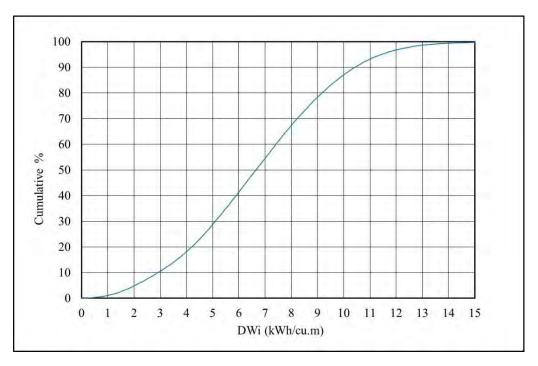


Figure 4 – Cumulative Distribution of DWi Values in SMCT Database

A further cumulative distribution graph is provided in Figure 5 to allow a comparison of the M_{ia}, M_{ih} and M_{ic} values obtained for the Lost Dog Project material, with the entire population of values for these parameters contained in the SMCT database.

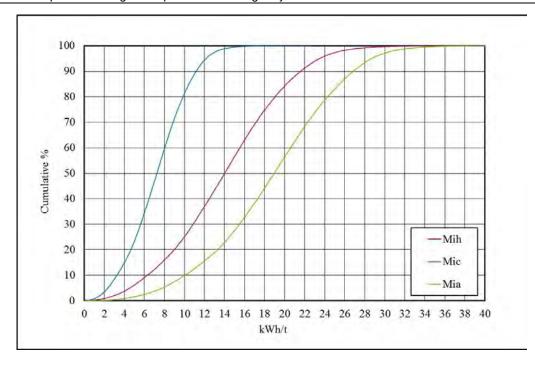


Figure 5 - Cumulative Distribution of M_{ia}, M_{ih} and M_{ic} Values in the SMCT Database

The value of A*b, which is also a measure of resistance to impact breakage, is calculated and presented in Table 4, which also gives a comparison to the population of samples in the JKTech database, with the percent of samples present in the JKTech database that are softer. Note that in contrast to the DWi, a high value of A*b means that an ore is soft whilst a low value means that it is hard.

Table 4 – Derived Values for A*b, t_a and SCSE

Sample Designation	A*b		t a		SCSE (kWh/t)	
	Value	%	Value	%	Value	%
Mill Scats Comp	57.9	33.2	0.61	31.8	8.29	28.8

In Figure 6 and Figure 7 below, histogram style frequency distributions for the A*b values and for the SCSE values in the JKTech DW database are shown respectively.

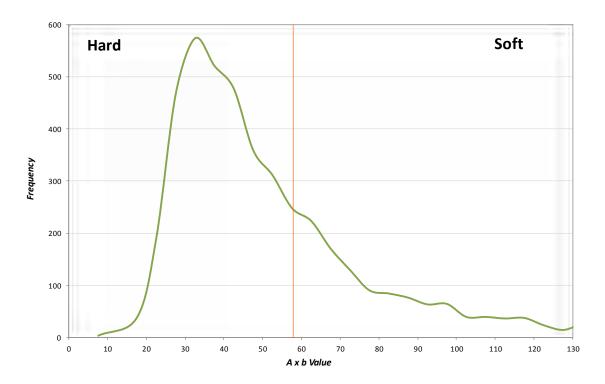


Figure 6 - Frequency Distribution of $A \!\!\!\!\!\!\!\!\!^{*} b$ in the JKTech Database

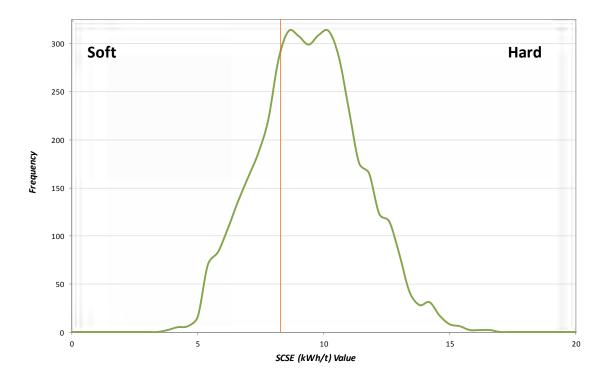


Figure 7 - Frequency Distribution of SCSE in the JKTech Database

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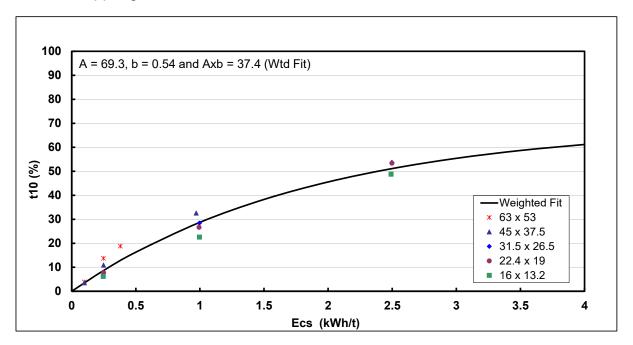
Severability

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APPENDICES

APPENDIX A. SAG CIRCUIT SPECIFIC ENERGY (SCSE)

For a little over 20 years, the results of JK Drop Weight tests and SMC tests have been reported in part as A, b and t_a parameters. A and b are parameters which describe the response of the ore under test to increasing levels of input energy in single impact breakage. A typical t_{10} v Ecs curve resulting from a Drop Weight test is shown in App Figure 1.



App Figure 1 – Typical t₁₀ v Ecs curve

The curve shown in App Figure 1 is represented by an equation which is given in Equation 1.

$$t_{10} = A(1 - e^{-b.Ecs})$$
 Equation 1

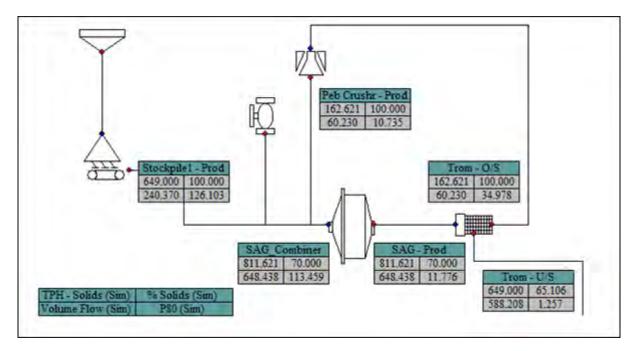
The parameters A and b are generated by least squares fitting Equation 1 to the JK Drop Weight test data. The parameter t_a is generated from a tumbling test.

Both A and b vary with ore type but having two parameters describing a single ore property makes comparison difficult. For that reason the product of A and b, referred to as A*b, which is related to the slope of the $t_{10}-E_{cs}$ curve at the origin, has been universally accepted as the parameter which represents an ore's resistance to impact breakage.

The parameters A, b and t_a have no physical meaning in their own right. They are ore hardness parameters used by the AG/SAG mill model in JKSimMet which permits prediction of the product size distribution and the power draw of the AG/SAG mill for a given feed size distribution and feed rate. In a design situation, the dimensions of the mill are adjusted until the load in the mill reaches 25 % by volume when fed at the required feed rate. The model predicts the power draw under these conditions and from the power draw and throughput the specific energy is determined. The specific energy is mainly a function of the ore hardness (A and b values), the feed size and the dimensions of the mill (specifically the aspect ratio) as well as to a lesser extent

the operating conditions such as ball load, mill speed, grate/pebble port size and pebble crusher activity.

There are two drawbacks to the approach of using A*b as the single parameter to describe the impact resistance of a particular ore. The first is that A*b is inversely related to impact resistance, which adds unnecessary complication. The second is that A*b is related to impact resistance in a non-linear manner. As mentioned earlier this relationship and how it affects comminution machine performance can only be predicted via simulation modelling. Hence to give more meaning to the A and b values and to overcome these shortcomings, JKTech Pty Ltd and SMC Testing Pty Ltd have developed a "standard" simulation methodology to predict the specific energy required for a particular tested ore when treated in a "Standard" circuit comprising a SAG mill in closed circuit with a pebble crusher. The flowsheet is shown in App Figure 2.



App Figure 2 – Flowsheet used for "Standard" AG/SAG circuit simulations

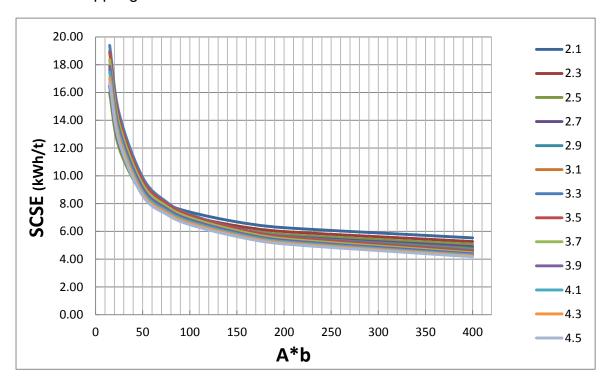
The specifications for the "standard" circuit are:

- SAG Mill
 - o inside shell diameter to length ratio of 2:1 with 15 ° cone angles
 - o ball charge of 15 %, 125 mm in diameter
 - o total charge of 25 %
 - o grate open area of 7 %
 - apertures in the grate are 100 % pebble ports with a nominal aperture of 56 mm
- Trommel
 - Cut Size of 12 mm
- Pebble Crusher
 - Closed Side Setting of 10 mm
- Feed Size Distribution
 - F₈₀ from the t_a relationship given in Equation 2

The feed size distribution is taken from the JKTech library of typical feed size distributions and is adjusted to meet the ore specific 80 % passing size predicted using the Morrell and Morrison (1996) F_{80} – t_a relationship for primary crushers with a closed side setting of 150 mm given in Equation 2.

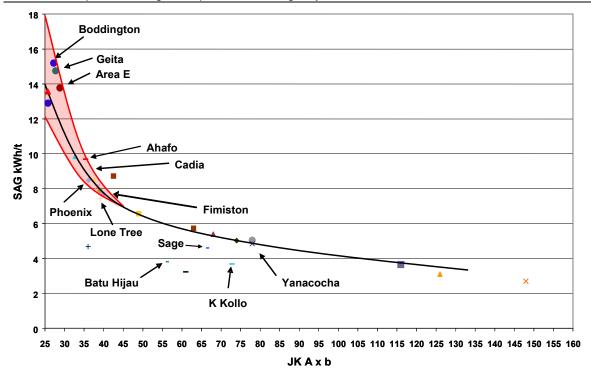
$$F_{80} = 71.3 - 28.4 * \ln(t_a)$$
 Equation 2

Simulations were conducted with A*b values ranging from 15 to 400, ta values ranging from 0.145 to 3.866 and solids SG values ranging from 2.1 to 4.2. For each simulation, the feed rate was adjusted until the total load volume in the SAG mill was 25 %. The predicted mill power draw and crusher power draw were combined and divided by the feed rate to provide the specific energy consumption. The results are shown in App Figure 3.



App Figure 3 – The relationship between A*b and specific energy at varying SG for the "Standard" circuit.

It is of note that the family of curves representing the relationship between Specific energy and A^*b for the "standard" circuit is very similar to the specific energy $-A^*b$ relationship for operating mills published in Veillette and Parker, 2005 and reproduced here in App Figure 4.



App Figure 4 – A*b vs SAG kWh/t for operating AG/SAG mills (after Veillette and Parker, 2005).

Of course, the SCSE quoted value will not necessarily match the specific energy required for an existing or a planned AG/SAG mill due to differences in the many operating and design variables such as feed size distribution, mill dimensions, ball load and size and grate, trommel and pebble crusher configuration. The SCSE is an effective tool to compare in a relative manner the expected behaviour of different ores in AG/SAG milling in exactly the same way as the Bond laboratory ball mill work index can be used to compare the relative grindability of different ores in ball milling (Bond, 1961 and Rowland and Kjos, 1980). However the originally reported A and b parameters which match the SCSE will be still be required in JKSimMet simulations of a proposed circuit to determine the AG/SAG mill specific energy required for that particular grinding task. Guidelines for the use of JKSimMet for such simulations were given in Bailey *et al*, 2009.

APPENDIX B. BACKGROUND AND USE OF THE SMC TEST®

B 1 Introduction

The SMC Test® was developed to provide a range of useful comminution parameters through highly controlled breakage of rock samples. Drill core, even quartered small diameter core is suitable. Only relatively small quantities of sample are required and can be re-used to conduct Bond ball work index tests.

The results from conducting the SMC Test® are used to determine the so-called drop-weight index (DW_i), which is a measure of the strength of the rock, as well as the comminution indices M_{ia} , M_{ih} and M_{ic} . The SMC Test® also estimates the JK rock breakage parameters A, b and t_a as well as the JK crusher model's t10-Ecs matrix, all of which are generated as part of the standard report output from the test.

In conjunction with the Bond ball mill work index the DW_i and the M_i suite of parameters can be used to accurately predict the overall specific energy requirements of circuits containing:

- AG and SAG mills.
- Ball mills
- Rod mills
- Crushers
- High Pressure Grinding Rolls (HPGR)

The JK rock breakage parameters can be used to simulate crushing and grinding circuits using JKTech's simulator – JKSimMet.

B 2 Simulation Modelling and Impact Comminution Theory

When a rock fragment is broken, the degree of breakage can be characterised by the " t_{10} " parameter. The t_{10} value is the percentage of the original rock mass that passes a screen aperture one tenth of the original rock fragment size. This parameter allows the degree of breakage to be compared across different starting sizes.

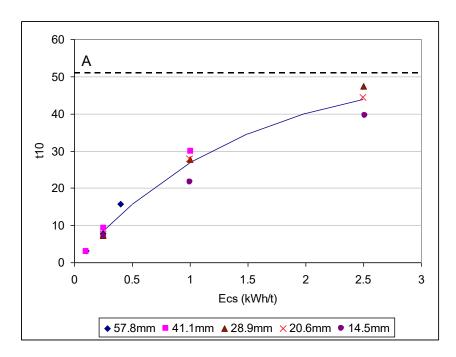
The specific comminution energy (Ecs) has the units kWh/t and is the energy applied during impact breakage. As the impact energy is varied, so does the t_{10} value vary in response. Higher impact energies produce higher values of t_{10} , which of course means products with finer size distributions.

The equation describing the relationship between the t_{10} and E_{CS} is given below.

$$t_{10} = A(1 - e^{-b.Ecs})$$
 Equation 1

As can be seen from this equation, there are two rock breakage parameters A and b that relate the t_{10} (size distribution index) to the applied specific energy (Ecs). These parameters are ore specific and are normally determined from a full JK Drop-Weight test.

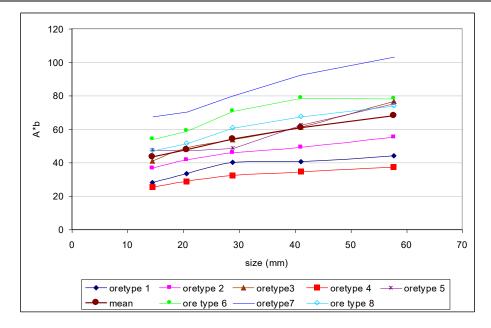
A typical plot of t_{I0} vs Ecs from a JK Drop-Weight test is shown in App Figure 5. The relationship is characterised by the two-parameter equation above, where t_{I0} is the dependent variable.



App Figure 5 - Typical t₁₀ v Ecs Plot

The t_{10} can be thought of as a "fineness index" with larger values of t_{10} indicating a finer product size distribution. The value of parameter A is the limiting value of t_{10} . This limit indicates that at higher energies, little additional size reduction occurs as the Ecs is increased beyond a certain value. A*b is the slope of the curve at 'zero' input energy and is generally regarded as an indication of the strength of the rock, lower values indicating a higher strength.

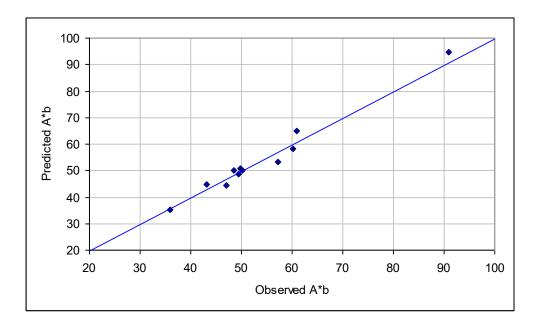
The SMC Test[®] is used to estimate the JK rock breakage parameters A and b by utilizing the fact that there is usually a pronounced (and ore specific) trend to decreasing rock strength with increasing particle size. This trend is illustrated in App Figure 6 which shows a plot of A*b versus particle size for a number of different rock types.



App Figure 6 - Size Dependence of A*b for a Range of Ore Types

In the case of a conventional JK Drop-Weight test these values are effectively averaged and a mean value of A and b is reported. The SMC Test[®] uses a single size and makes use of relationships such as that shown in App Figure 6 to predict the A and b of the particle size that has the same value as the mean for a full JK Drop-Weight test.

An example of this is illustrated in App Figure 7, where the observed values of the product A*b are plotted against those predicted using the DWi. Each of the data points in App Figure 7 is a result from a different ore type within an orebody.



App Figure 7 - Predicted v Observed A*b

The A and b parameters are used with Equation 1 and relationships such as illustrated in App Figure 6 to generate a matrix of Ecs values for a specific range of

 t_{10} values and particle sizes. This matrix is used in crusher modelling to predict the power requirement of the crusher given a feed and a product size specification (Napier-Munn et al (1996)).

The *A* and *b* parameters are also used in AG/SAG mill models, such as those in JKSimMet, for predicting how the rock will break inside the mill. From this description the models can predict what the throughput, power draw and product size distribution will be (Napier-Munn et al (1996)). Modelling also enables a detailed flowsheet to be built up of the comminution circuit response to changes in ore type. It also allows optimisation strategies to be developed to overcome any deleterious changes in circuit performance predicted from differences in ore type. These strategies can include both changes to how mills are operated (eg ball load, speed etc) and changes to feed size distribution through modification of blasting practices and primary crusher operation (mine-to-mill).

B 3 Power-Based Equations

B 3.1 General

The DW_i , M_{ia} , M_{ih} and M_{ic} parameters are used in so-called power-based equations which predict the specific energy of the associated comminution machines. The approach divides comminution equipment into three categories:

- Tumbling mills, eg AG, SAG, rod and ball mills
- · Conventional reciprocating crushers, eg jaw, gyratory and cone
- HPGRs

Tumbling mills are described using 2 indices: M_{ia} and M_{ib}

Crushers have one index: M_{ic} HPGRs have one index: M_{ih}

For tumbling mills the 2 indices relate to "coarse" and "fine" ore properties plus an efficiency factor which represents the influence of a pebble crusher in AG/SAG mill circuits. "Coarse" in this case is defined as spanning the size range from a P80 of 750 microns up to the P80 of the product of the last stage of crushing or HPGR size reduction prior to grinding. "Fine" covers the size range from a P80 of 750 microns down to P80 sizes typically reached by conventional ball milling, ie about 45 microns. The choice of 750 microns as the division between "coarse" and "fine" particle sizes was determined during the development of the technique and was found to give the best overall results across the range of plants in SMCT's data base. Implicit in the approach is that distributions are parallel and linear in log-log space.

The work index covering grinding in tumbling mills of coarse sizes is labelled M_{ia} . The work index covering grinding of fine particles is labelled Mib (Morrell, 2008). M_{ia} values are provided as a standard output from a SMC Test® (Morrell, 2004a) whilst M_{ib} values can be determined using the data generated by a conventional Bond ball mill work index test (M_{ib} is NOT the Bond ball work index). M_{ic} and M_{ih} values are also provided as a standard output from a SMC Test® (Morrell, 2009).

The general size reduction equation is as follows (Morrell, 2004b):

$$W_i = M_i \cdot 4(x_2^{f(x_2)} - x_1^{f(x_1)})$$
 Equation 3

where

 M_i = Work index related to the breakage property of an ore (kWh/tonne); for grinding from the product of the final stage of crushing to a P80 of 750 microns (coarse particles) the index is labelled Mia and for size reduction from 750 microns to the final product P80 normally reached by conventional ball mills (fine particles) it is labelled M_{ib} . For conventional crushing M_{ic} is used and for HPGRs Mih is used.

Wi = Specific comminution (kWh/tonne)

 x_2 = 80% passing size for the product (microns) x_1 = 80% passing size for the feed (microns)

 $f(x_i) = -(0.295 + x_i/1000000)$ (Morrell, 2006) Equation 4

For tumbling mills the specific comminution energy (Wi) relates to the power at the pinion or for gearless drives - the motor output. For HPGRs it is the energy inputted to the rolls, whilst for conventional crushers Wi relates to the specific energy as determined using the motor input power less the no-load power.

B 3.2 Specific Energy Determination for Comminution Circuits

The total specific energy (W_T) to reduce primary crusher product to final product size is given by:

$$W_T = W_a + W_b + W_c + W_h + W_s$$
 Equation 5

where

 W_a = specific energy to grind coarser particles in tumbling mills W_b = specific energy to grind finer particles in tumbling mills

 W_c = specific energy for conventional crushing

 W_h = specific energy for HPGRs

 W_s = specific energy correction for size distribution

Clearly only the W values associated with the relevant equipment in the circuit being studied are included in Equation 5.

B 3.2.1 Tumbling mills

For coarse particle grinding in tumbling mills Equation 3 is written as:

$$W_a = K_1 M_{ia} \cdot 4(x_2^{f(x_2)} - x_1^{f(x_1)})$$
 Equation 6

where

 K_I = 1.0 for all circuits that do not contain a recycle pebble crusher and 0.95 where circuits do have a pebble crusher

 x_1 = P_{80} in microns of the product of the last stage of crushing before grinding

 $x_2 = 750 \text{ microns}$

 M_{ia} = Coarse ore work index and is provided directly by SMC Test[®]

For fine particle grinding Equation 3 is written as:

$$W_b = M_{ib}.4(x_3^{f(x_3)} - x_2^{f(x_2)})$$
 Equation 7

where

 x_2 = 750 microns

 x_3 = P₈₀ of final grind in microns

 M_{ib} = Provided by data from the standard Bond ball work index test using the following equation (Morrell, 2006):

$$M_{ib} = \frac{18.18}{P_1^{0.295}(Gbp)(p_{80}^{f(p_{80})} - f_{80}^{f(f_{80})})}$$
 Equation 8

where

Mib = fine ore work index (kWh/tonne) P_1 = closing screen size in microns

Gbp = net grams of screen undersize per mill revolution

 p_{80} = 80% passing size of the product in microns f_{80} = 80% passing size of the feed in microns

Note that the Bond ball work index test should be carried out with a closing screen size which gives a final product P80 similar to that intended for the full scale circuit.

B 3.2.2 Conventional Crushers and HPGR

Equation 3 for conventional crushers is written as:

$$W_c = S_c K_2 M_{ic}. 4(x_2^{f(x_2)} - x_1^{f(x_1)})$$
 Equation 9

where

 S_c = coarse ore hardness parameter which is used in primary and secondary crushing situations. It is defined by Equation 10 with K_s set to 55.

 K_2 = 1.0 for all crushers operating in closed circuit with a classifying screen. If the crusher is in open circuit, eg pebble crusher in a AG/SAG circuit, K_2 takes the value of 1.19.

 x_1 = P_{80} in microns of the circuit feed

 x_2 = P_{80} in microns of the circuit product

 M_{ic} = Crushing ore work index and is provided directly by SMC Test[®]

The coarse ore hardness parameter (S) makes allowance for the decrease in ore hardness that becomes significant in relatively coarse crushing applications such as primary and secondary cone/gyratory circuits. In tertiary and pebble crushing circuits it is normally not necessary and takes the value of unity. In full scale HPGR circuits where feed sizes tend to be higher than used in laboratory and pilot scale machines the parameter has also been found to improve predictive accuracy. The parameter is defined by Equation 10.

$$S = K_s(x_1.x_2)^{-0.2}$$
 Equation 10

where

 K_s = machine-specific constant that takes the value of 55 for conventional crushers and 35 in the case of HPGRs

 x_1 = P_{80} in microns of the circuit feed x_2 = P_{80} in microns of the circuit product

Equation 3 for HPGR's crushers is written as:

$$W_h = S_h K_3 M_{ih} \cdot 4(x_2^{f(x_2)} - x_1^{f(x_1)})$$
 Equation 11

where

 S_h = coarse ore harness parameter as defined by Equation 10 and with K_s set to 35

 K_3 = 1.0 for all HPGRs operating in closed circuit with a classifying screen. If the HPGR is in open circuit, K3 takes the value of 1.19.

 x_1 = P_{80} in microns of the circuit feed x_2 = P_{80} in microns of the circuit product

 M_{ih} = HPGR ore work index and is provided directly by SMC Test[®]

B 3.2.3 Specific Energy Correction for Size Distribution (Ws)

Implicit in the approach described in this appendix is that the feed and product size distributions are parallel and linear in log-log space. Where they are not, allowances (corrections) need to be made. By and large, such corrections are most likely to be necessary (or are large enough to be warranted) when evaluating circuits in which closed circuit secondary/tertiary crushing is followed by ball milling. This is because such crushing circuits tend to produce a product size distribution which is relatively steep when compared to the ball mill circuit cyclone overflow. This is illustrated in App Figure 8, which shows measured distributions from an open and closed crusher circuit as well as a ball mill cyclone overflow. The closed circuit crusher distribution can be seen to be relatively steep compared with the open circuit crusher distribution and ball mill cyclone overflow. Also the open circuit distribution more closely follows the gradient of the cyclone overflow. If a ball mill circuit were to be fed two distributions, each with same P80 but with the open and closed circuit gradients in App Figure 8, the closed circuit distribution would require more energy to grind to the final P80. How much more energy is required is difficult to determine. However, for the purposes of this approach it has been assumed that the additional specific energy for ball milling is the same as the difference in specific energy between open and closed crushing to reach the nominated ball mill feed size. This assumes that a crusher would provide this energy. However, in this situation the ball mill has to supply this energy and it has a different (higher) work index than the crusher (ie the ball mill is less energy efficient than a crusher and has to input more energy to do the same amount of size reduction). Hence from Equation 9, to crush to the ball mill circuit feed size (x_2) in open circuit requires specific energy equivalent to:

$$W_c = 1.19 * M_{ic} \cdot 4(x_2^{f(x_2)} - x_1^{f(x_1)})$$
 Equation 12

For closed circuit crushing the specific energy is:

$$W_c = 1 * M_{ic} \cdot 4(x_2^{f(x_2)} - x_1^{f(x_1)})$$
 Equation 13

The difference between the two (Equation 12 and Equation 13) has to be provided by the milling circuit with an allowance for the fact that the ball mill, with its lower

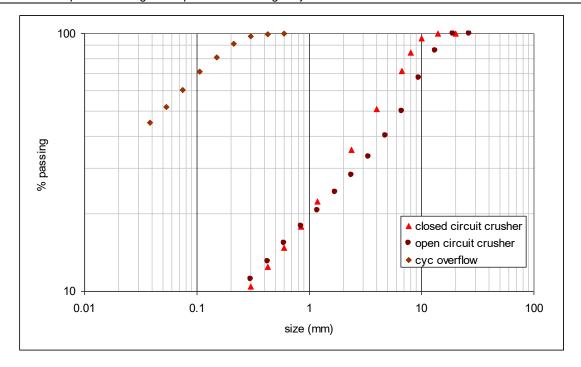
energy efficiency, has to provide it and not the crusher. This is what is referred to in Equation 5 as W_s and for the above example is therefore represented by:

$$W_s = 0.19 * M_{ia}.4(x_2^{f(x_2)} - x_1^{f(x_1)})$$
 Equation 14

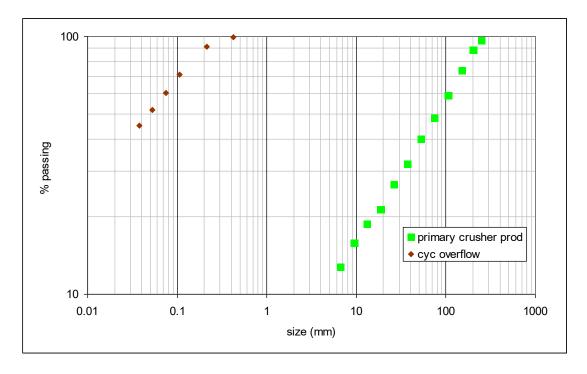
Note that in Equation 14 M_{ic} has been replaced with M_{ia} , the coarse particle tumbling mill grinding work index.

In AG/SAG based circuits the need for W_s appears to be unnecessary as App Figure 9 illustrates. Primary crusher feeds often have the shape shown in App Figure 9and this has a very similar gradient to typical ball mill cyclone overflows. A similar situation appears to apply with HPGR product size distributions, as illustrated in App Figure 10. Interestingly SMCT's data show that for HPGRs, closed circuit operation appears to require a lower specific energy to reach the same P80 as in open circuit, even though the distributions for open and closed circuit look to have almost identical gradients. Closer examination of the distributions in fact shows that in closed circuit the final product tends to have slightly less very fine material, which may account for the different energy requirements between the two modes of operation. It is also possible that recycled material in closed circuit is inherently weaker than new feed, as it has already passed through the HPGR previously and may have sustained micro-cracking. A reduction in the Bond ball mill work index as measured by testing HPGR products compared it to the Bond ball mill work index of HPGR feed has been noticed in many cases in the laboratory (see next section) and hence there is no reason to expect the same phenomenon would not affect the recycled HPGR screen oversize.

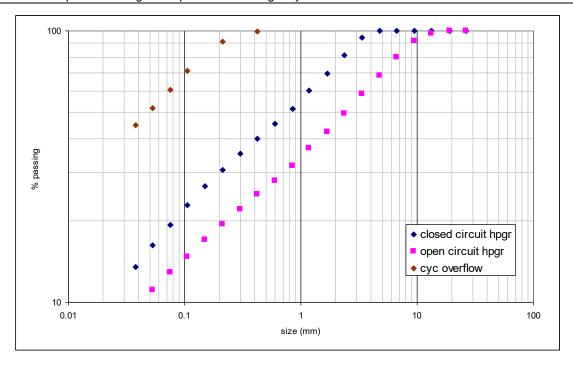
It follows from the above arguments that in HPGR circuits, which are typically fed with material from closed circuit secondary crushers, a similar feed size distribution correction should also be applied. However, as the secondary crushing circuit uses such a relatively small amount of energy compared to the rest of the circuit (as it crushes to a relatively coarse size) the magnitude of size distribution correction is very small indeed – much smaller than the error associated with the technique - and hence may be omitted in calculations.



App Figure 8 – Examples of Open and Closed Circuit Crushing Distributions Compared with a Typical Ball Mill Cyclone Overflow Distribution



App Figure 9 – Example of a Typical Primary Crusher (Open and Circuit)
Product Distribution Compared with a Typical Ball Mill Cyclone Overflow
Distribution



App Figure 10 – Examples of Open and Closed Circuit HPGR Distributions Compared with a Typical Ball Mill Cyclone Overflow Distribution

B 3.2.4 Weakening of HPGR Products

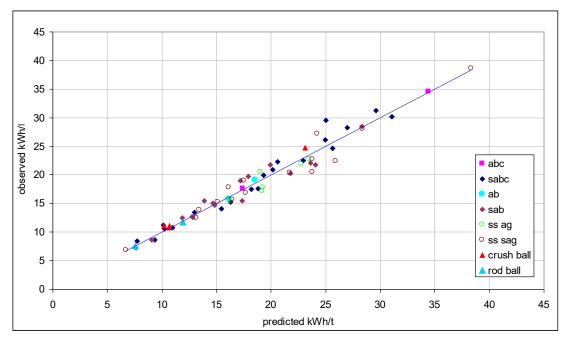
As mentioned in the previous section, laboratory experiments have been reported by various researchers in which the Bond ball work index of HPGR products is less than that of the feed. The amount of this reduction appears to vary with both material type and the pressing force used. Observed reductions in the Bond ball work index have typically been in the range 0-10%. In the approach described in this appendix no allowance has been made for such weakening. However, if HPGR products are available which can be used to conduct Bond ball work index tests on then M_{ib} values obtained from such tests can be used in Equation 7. Alternatively the M_{ib} values from Bond ball mill work index tests on HPGR feed material can be reduced by an amount that the user thinks is appropriate. Until more data become available from full scale HPGR/ball mill circuits it is suggested that, in the absence of Bond ball mill work index data on HPGR products, the M_{ib} results from HPGR feed material are reduced by no more than 5% to allow for the effects of micro-cracking.

B 3.3 Validation

B 3.3.1 Tumbling Mill Circuits

The approach described in the previous section was applied to over 120 industrial data sets. The results are shown in App Figure 11. In all cases, the specific energy relates to the tumbling mills contributing to size reduction from the product of the final stage of crushing to the final grind. Data are presented in terms of equivalent specific energy at the pinion. In determining what these values were on each of the plants in the data base it was assumed that power at the pinion was 93.5% of the measured gross (motor input) power, this figure being typical of what is normally

accepted as being reasonable to represent losses across the motor and gearbox. For gearless drives (so-called wrap-around motors) a figure of 97% was used.



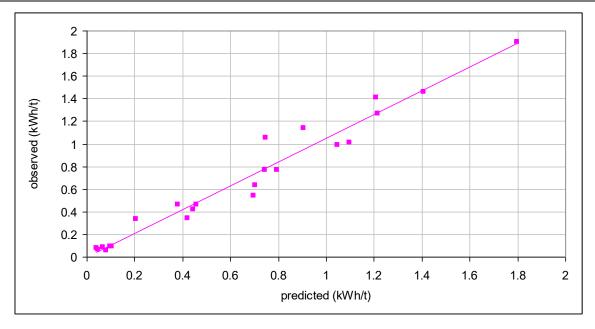
App Figure 11 - Observed vs Predicted Tumbling Mill Specific Energy

B 3.3.2 Conventional Crushers

Validation used 12 different crushing circuits (25 data sets), including secondary, tertiary and pebble crushers in AG/SAG circuits. Observed vs predicted specific energies are given in App Figure 12. The observed specific energies were calculated from the crusher throughput and the net power draw of the crusher as defined by:

Net Power = Motor Input Power – No Load Power Equation 15

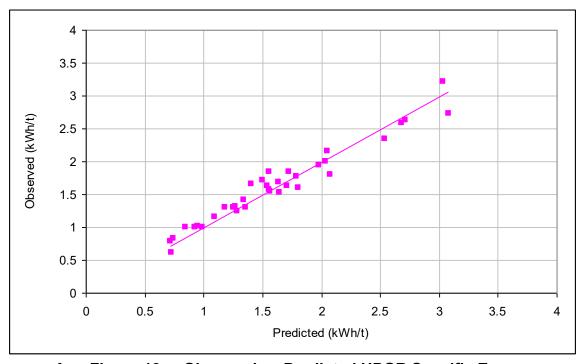
No-load power tends to be relatively high in conventional crushers and hence net power is significantly lower than the motor input power. From examination of the 25 crusher data sets the motor input power was found to be on average 20% higher than the net power.



App Figure 12 - Observed vs Predicted Conventional Crusher Specific Energy

B 3.3.3 HPGRs

Validation for HPGRs used data from 19 different circuits (36 data sets) including laboratory, pilot and industrial scale equipment. Observed vs predicted specific energies are given in App Figure 13. The data relate to HPGRs operating with specific grinding forces typically in the range 2.5-3.5 N/mm². The observed specific energies relate to power delivered by the roll drive shafts. Motor input power for full scale machines is expected to be 8-10% higher.



App Figure 13 - Observed vs Predicted HPGR Specific Energy

B 4 WORKED EXAMPLES

A SMC Test[®] and Bond ball work index test were carried out on a representative ore sample. The following results were obtained:

SMC Test®:

 $M_{ia} = 19.4 \text{ kWh/t}$ $M_{ic} = 7.2 \text{ kWh/t}$ $M_{ih} = 13.9 \text{ kWh/t}$

Bond test carried out with a 150 micron closing screen:

 M_{ib} = 18.8 kWh/t

Three circuits are to be evaluated:

- SABC
- HPGR/ball mill
- Conventional crushing/ball mill

The overall specific grinding energy to reduce a primary crusher product with a P_{80} of 100 mm to a final product P_{80} of 106 μ m needs to be estimated.

B 4.1 SABC Circuit

Coarse particle tumbling mill specific energy:

$$W_a = 0.95 * 19.4 * 4 * \left(750^{-(0.295 + 750/1000000)} - 100000^{-(0.295 + 100000/1000000)}\right)$$

= 9.6 kWh/t

Fine particle tumbling mill specific energy:

$$W_b = 18.8 * 4 * \left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$
= 8.4 kWh/t

Pebble crusher specific energy:

In this circuit, it is assumed that the pebble crusher feed P_{80} is 52.5mm. As a rule of thumb this value can be estimated by assuming that it is 0.75 of the nominal pebble port aperture (in this case the pebble port aperture is 70mm). The pebble crusher is set to give a product P_{80} of 12mm. The pebble crusher feed rate is expected to be 25% of new feed tph.

$$W_c = 1.19 * 7.2 * 4 * \left(12000 - (0.295 + 12000/1000000) - 52500 - (0.295 + 52500/1000000)\right)$$

= 1.12 kWh/t when expressed in terms of the crusher feed rate

= 1.12 * 0.25 kWh/t when expressed in terms of the SABC circuit new

feed rate

= 0.3 kWh/t of SAG mill circuit new feed

Total net comminution specific energy:

$$W_T = 9.6 + 8.4 + 0.3$$
 kWh/t = 18.3 kWh/t

B 4.2 HPGR/Ball Milling Circuit

In this circuit primary crusher product is reduced to a HPGR circuit feed P_{80} of 35 mm by closed circuit secondary crushing. The HPGR is also in closed circuit and reduces the 35 mm feed to a circuit product P_{80} of 4 mm. This is then fed to a closed circuit ball mill which takes the grind down to a P_{80} of 106 μ m.

Secondary crushing specific energy:

$$W_c = 1*55*(35000*100000)^{-0.2}*7.2*4*(35000^{-(0.295+35000/1000000)} - 100000^{-(0.295+100000/1000000)}$$
= 0.4 kWh/t

HPGR specific energy:

$$W_h = 1*35*(4000*35000)^{-2}*13.9*4*\left(4000^{-(0.295+4000/1000000)} - 35000^{-(0.295+35000/1000000)}\right)$$
= 2.4 kWh/t

Coarse particle tumbling mill specific energy:

$$W_a = 1*19.4*4*\left(750^{-(0.295+750/1000000)} - 4000^{-(0.295+4000/1000000)}\right)$$

= 4.5 kWh/t

Fine particle tumbling mill specific energy:

$$W_b = 18.8 * 4 * \left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$

= 8.4 kWh/t

Total net comminution specific energy:

$$W_T = 4.5 + 8.4 + 0.4 + 2.4$$
 kWh/t
= 15.7 kWh/t

B 4.3 Conventional Crushing/Ball Milling Circuit

In this circuit primary crusher product is reduced in size to P_{80} of 6.5 mm via a secondary/tertiary crushing circuit (closed). This is then fed to a closed circuit ball mill which grinds to a P80 of 106 μ m.

Secondary/tertiary crushing specific energy:

$$W_c = 1*7.2*4*\left(6500^{-(0.295+6500/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$
= 1.7 kWh/t

Coarse particle tumbling mill specific energy:

$$W_a = 1*19.4*4*\left(750^{-(0.295+750/1000000)} - 6500^{-(0.295+6500/1000000)}\right)$$

= 5.5 kWh/t

Fine particle tumbling mill specific energy:

$$W_b = 18.8*4*\left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$
 = 8.4 kWh/t

Size distribution correction;

$$W_s = 0.19 * 19.4 * 4 * \left(6500^{-(0.295 + 6500/1000000)} - 100000^{-(0.295 + 100000/1000000)}\right)$$

= 0.9 kWh/t

Total net comminution specific energy:

$$W_T$$
 = 5.5 + 8.4 + 1.7 + 0.9kWh/t
= 16.5 kWh/t



BOND ABRASION INDEX DETERMINATION

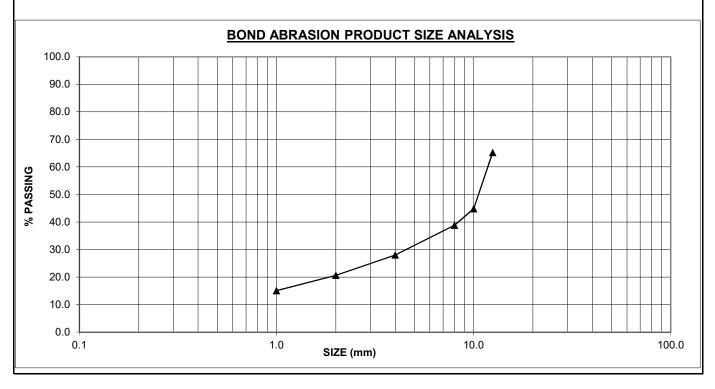


PROJECT	A18169 - LOST DOG GOLD PROJECT		
CLIENT	BEACON MINERALS LTD		
SAMPLE ID	MILL SCATS		
DATE	JUNE 2017		

	SIZE ANALYSIS : ABRASION INDEX PRODUCT				
Operation	Size	Weight	Weight	Weight	
	(mm)	(g)	(%)	% <	
Screening	12.5	555.7	34.8	65.2	
	10.0	325.6	20.4	44.8	
	8.00	95.7	6.0	38.8	
	4.00	172.4	10.8	28.0	
	2.00	118.3	7.4	20.6	
	1.00	88.5	5.5	15.1	
	-1.00	240.3	15.1		
Total		1596.5	100.00		

PADDLE WEIGHT BEFORE TEST (g): 89.0375
PADDLE WEIGHT AFTER TEST (g): 89.0097

BOND ABRASION INDEX (Ai): 0.0278



BOND BALL MILL CLOSED CIRCUIT GRINDABILITY: 106 MICROMETERS

PROJECT No	A18169 - LOST DOG GOLD PROJECT
CLIENT	BEACON MINERALS LTD
SAMPLE IDENTITY	LOST DOG COMPOSITE
DATE	JULY 2017

PERIOD	REVS OF MILL	WT OF 700 mL (g)	WT OF NEW FEED (g)	WT OF O/SIZE (g)*	WT OF U/SIZE (g)*	NET WT OF U/SIZE (g)*	NET WT OF U/SIZE PER REV (g)		WT OF FRESH FEED ADDED TO NEXT CYCLE (g)	CYCLE
1	50	1167.7	1167.7	826.9	340.8	59.1	1.1819	243	340.8	82.2
2	213	1167.7	340.8	836.6	331.1	248.9	1.1685	253	331.1	79.9
3	217	1167.7	331.1	861.6	306.1	226.2	1.0425	281	306.1	73.8
4	249	1167.7	306.1	819.5	348.2	274.4	1.1018	235	348.2	84.0
5	227	1167.7	348.2	834.1	333.6	249.6	1.0996	250	333.6	80.5
6	230	1167.7	333.6	834.1	333.6	253.1	1.1005	250	333.6	80.5

Note: * = Ex grinding mill

PRODUCT IN THE FEED 24.12 (%) BULK DENSITY 1.6681 (t/m^3) IDEAL POTENTIAL PRODUCT 333.6 (g) AVERAGE EQUILIBRIUM CIRC LOAD 250 (%) AVERAGE PRODUCT 1.100 (g/rev) 80 % PASSING FEED SIZE 1860 (μm) 80 % PASSING PRODUCT SIZE 76 (μm)

BOND BALL MILL WORK INDEX (Kilowatt hours / dry tonne): 17.0





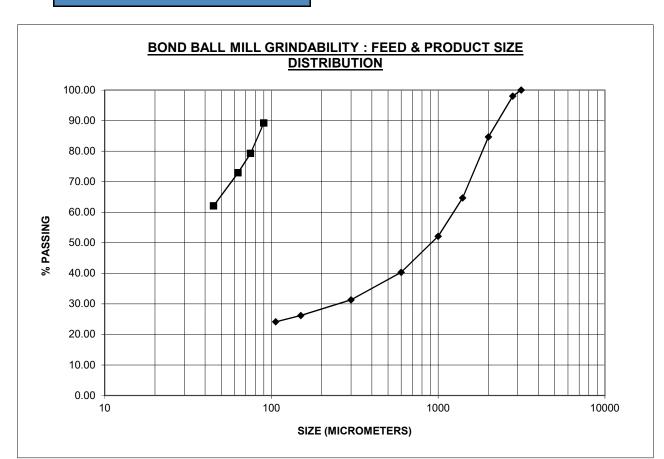
BOND BALL MILL GRINDABILITY TEST FEED AND PRODUCT SIZINGS

A18169 - LOST DOG GOLD PROJECT LOST DOG COMPOSITE

	FEED TO PERIOD No. 1						
Size	Weight	Retained	Passing				
(mm)	(g)	(%)	(%)				
3.150	0.0	0.00	100.00				
2.800	12.7	1.97	98.03				
2.000	86.3	13.37	84.66				
1.400	128.9	19.97	64.69				
1.000	81.3	12.60	52.09				
0.600	75.8	11.74	40.35				
0.300	58.2	9.02	31.33				
0.150	33.3	5.16	26.17				
0.106	13.2	2.05	24.12				
-0.106	155.7	24.12					
TOTAL	645.4	75.88					
F 80 (μm) :	1860					

EQUILIBRIUM PRODUCTS						
Size	Weight	Retained	Passing			
(mm)	(g)	(%)	(%)			
0.090	36.0	10.79	89.21			
0.075	33.2	9.95	79.26			
0.063	21.2	6.35	72.90			
0.045	36.1	10.82	62.08			
-0.045	207.1	62.08				
TOTAL	333.6	100.00				

P 80 (μm) : 76







BOND BALL MILL CLOSED CIRCUIT GRINDABILITY: 106 MICROMETERS

PROJECT No	A18169 - LOST DOG GOLD PROJECT
CLIENT	BEACON MINERALS LTD
SAMPLE IDENTITY	MILL SCATS
DATE	JULY 2017

PERIOD	REVS OF MILL	WT OF 700 mL (g)	WT OF NEW FEED (g)	WT OF O/SIZE (g)*	WT OF U/SIZE (g)*	NET WT OF U/SIZE (g)*	NET WT OF U/SIZE PER REV (g)		WT OF FRESH FEED ADDED TO NEXT CYCLE (g)	CYCLE
1	150	1171.0	1171.0	935.2	235.8	113.0	0.7537	397	235.8	24.7
2	411	1171.0	235.8	791.0	380.0	355.3	0.8644	208	380.0	39.8
3	341	1171.0	380.0	848.1	322.9	283.1	0.8301	263	322.9	33.8
4	362	1171.0	322.9	816.0	355.0	321.2	0.8872	230	355.0	37.2
5	335	1171.0	355.0	836.4	334.6	297.4	0.8877	250	334.6	35.1
6	337	1171.0	334.6	836.4	334.6	299.5	0.8888	250	334.6	35.1

Note: * = Ex grinding mill

PRODUCT IN THE FEED 10.48 (%) BULK DENSITY 1.6729 (t/m³) IDEAL POTENTIAL PRODUCT 334.6 (g) AVERAGE EQUILIBRIUM CIRC LOAD 250 (%) AVERAGE PRODUCT 0.888 (g/rev) 80 % PASSING FEED SIZE 2844 (μ m) 80 % PASSING PRODUCT SIZE 80 (μ m)

BOND BALL MILL WORK INDEX (Kilowatt hours / dry tonne): 19.9





BOND BALL MILL GRINDABILITY TEST FEED AND PRODUCT SIZINGS

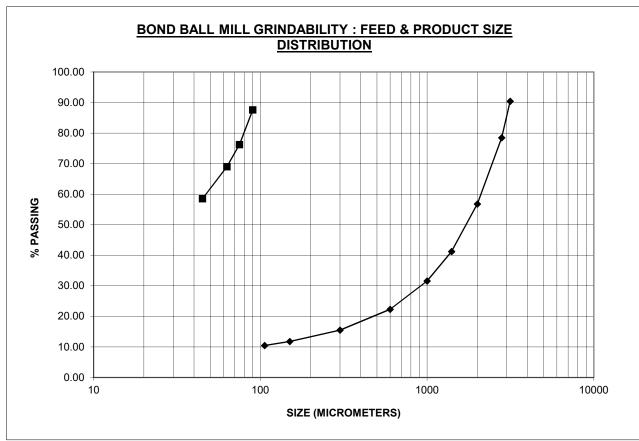
A18169 - LOST DOG GOLD PROJECT MILL SCATS

·	FEED TO PERIOD No. 1						
Size	Weight	Retained	Passing				
(mm)	(g)	(%)	(%)				
3.150	72.9	9.59	90.41				
2.800	90.6	11.92	78.50				
2.000	164.9	21.69	56.81				
1.400	118.6	15.60	41.21				
1.000	73.2	9.63	31.58				
0.600	70.5	9.27	22.31				
0.300	51.8	6.81	15.49				
0.150	28.3	3.72	11.77				
0.106	9.8	1.29	10.48				
-0.106	79.7	10.48					
TOTAL	760.3	89.52					
E 90 /		2044					

EQUILIBRIUM PRODUCTS						
Size	Weight	Retained	Passing			
(mm)	(g)	(%)	(%)			
0.090	41.6	12.43	87.57			
0.075	37.9	11.33	76.24			
0.063	24.3	7.26	68.98			
0.045	35.0	10.46	58.51			
-0.045	195.8	58.51				
TOTAL	334.6	100.00				

P 80 (μm) : 80

F 80 (µm) : 2844









APPENDIX II

Head Assays and Site Water Analysis



A18169 - Beacon Minerals Ltd Lost Dog Project Head Assays



Analyte	Lost Dog Composite
Au (g/t)	2.15
Au (g/t)	2.09
Ag (ppm)	4.2
Al (ppm)	8800
Ba (ppm)	<20
Be (ppm)	<20
Bi (ppm)	<25
Ca (%)	14.9
Cd (ppm)	<20
Co (ppm)	40
Cr (ppm)	200
Cu (ppm)	40
Fe (ppm)	3800
Hg (ppm)	<0.1
K (ppm)	500
Li (ppm)	<20
Mg (%)	8.92
Mn (ppm)	360
Mo (ppm)	<20
Na (ppm)	2800
Ni (ppm)	140
P (ppm)	<250
Pb (ppm)	<20
S-2 (%)	<0.02
SiO2 (%)	25.6
Sr (ppm)	100
Ti (ppm)	600
V (ppm)	30
Y (ppm)	<100
Zn (ppm)	10



A18169 - Beacon Minerals Ltd



Lost Dog Project

Head Assay Results - Details And Results

Analyte	Site Water	FMR LT3 6/3
Au (mg/l)	<0.005	0.755
Ag (mg/l)	<0.2	0.40
Al (mg/l)	2.0	-
Ba (mg/l)	<0.5	-
Bi (mg/l)	<1.0	-
Ca (mg/l)	358	-
Cd (mg/l)	<0.5	-
Co (mg/l)	<0.5	-
Cr (mg/l)	<1.0	-
Cu (mg/l)	0.2	-
Fe (mg/l)	<1.0	-
K (mg/l)	140	-
Li (mg/l)	<0.5	-
Mg (mg/l)	2516	-
Mn (mg/l)	<0.5	-
Mo (mg/l)	<0.5	-
Na (mg/l)	15440	-
Ni (mg/l)	0.5	-
P (mg/l)	<10	-
Pb (mg/l)	<0.5	-
Sr (mg/l)	5.8	-
Ti (mg/l)	<1.0	-
V (mg/l)	<0.2	-
Y (mg/l)	<0.1	-
Zn (mg/l)	<0.2	-
Zr (mg/l)	<0.5	-
*HCO3	400	-
*CO3	<100	-
Cl	28000	-
SO4	4870	-
TDS	54900	-
рН	7.84	-
**Cond	71.6	-
SG	1.0450	-

<u>COMMENTS</u>:

Data is in mg/l unless otherwise stated

^{*}Data is in mg/l CaCO3

^{**} Data is in ms/cm



APPENDIX III

Cyanidation Testwork Details and Results





PROJECT	A18169 : LOST DOG PROJECT
CLIENT	BEACON MINERALS LTD
TEST No	JS3915
SAMPLE	LOST DOG COMPOSITE
GRIND	P80: 106 μm
WATER	LOST DOG COMPOSITE
DATE	JULY 2017

DIRECT CYANIDATION TIME LEACH TESTWORK - OXYGEN SPARGED

			Additions			;	Solution Dat	а		Au	Ag
Time (Hours)	Ore (g)	Water (mL)	NaCN (g)	Lime (g)	Oxygen (mg/L)	рН	NaCN (%)	Au (mg/L)	Ag (mg/L)	Extrn. Total (%)	Extrn. Total (%)
	1000.0	1777.0			9.5	7.7					
0		1777.0	0.65	11.35		10.0	0.037	0.000	0.0	0.00	0.00
2		1747.0	0.00	0.53	32.9	9.9	0.030	0.790	0.15	70.99	47.05
4		1717.0	0.00	0.90	31.9	9.8	0.028	0.825	0.15	74.08	47.05
6		1687.0	0.00	1.03	29.8	9.8	0.028	0.845	0.15	75.81	47.05
8		1657.0	0.00	1.31	33.4	9.8	0.025	0.865	0.15	77.52	47.05
12		1627.0	0.00	0.74	29.2	9.8	0.025	0.870	0.15	77.94	47.05
24		1597.0	0.00	0.00	32.1	9.8	0.025	0.880	0.15	78.76	47.05
TOTAL			0.65	15.9							

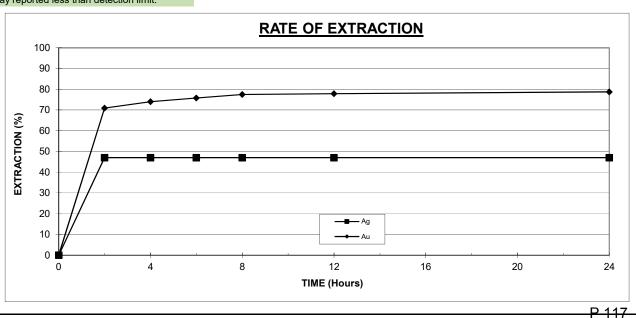
GOLD & SILVER EXTRACTION CALCULATIONS

COMMENTS:

			Gold		Silver 1		1. NaCN addition :	0.65 (kg/t)	
Product	Quantity	Assay	Mass	Dist'n	Assay	Mass	Dist'n	2. NaCN consumption :	0.20 (kg/t)
		(ppm)	(μg)	(%)	(ppm)	(μg)	(%)	3. Lime consumption :	15.86 (kg/t)
Solids (g)	1000.0	0.42	420	21.24	0.3	300	52.95	4. Water SG :	1.045
Solution (mL)	1597.0	0.88	1405	71.06	0.2	240	42.28	5. Water to leach :	1857.0 (g)
Solution Samples *			152	7.70		27	4.77	6. Grind size P80:	106 (µm)
								7. 30 mL solution sample:	s were removed at
								Each sampling period.	
Total Extraction				78.76			47.05		
Total			1978	100.00		567	100.00		
Calculated Head		1.98			0.6				
								1	

Assay Head 2.15 / 2.09
*: Intermediate solution samples removed during the test.

Assay reported less than detection limit.







PROJECT	A18169 : LOST DOG PROJECT
CLIENT	BEACON MINERALS LTD
TEST No	JS3916
SAMPLE	LOST DOG COMPOSITE
GRIND	P80: 106 MICRONS
WATER	LOST DOG SITE WATER
DATE	JULY 2017

CIL CYANIDATION TIME LEACH TESTWORK : OXYGEN SPARGED

		ADDITIONS					SOL	UTION D	ATA		PRI	EG. CARI	BON	EXTRACTION	
TIME (Hours)	Ore (g)	Water (mL)	Carbon Haycarb Yao (g)	NaCN (g)	Lime (g)	Oxygen (mg/L)	рН	NaCN (%)	Au (mg/L)	Ag (mg/L)	Wt (g)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
	1000.0	1777.0				9.8	7.7								
			33.6												
0				0.65	10.70		10.0	0.037	0.000	0.00		0	0	0.00	0.00
2			29.2	0.00	0.69	29.0	9.9	0.028	0.040	0.15	4.47	39	12	70.71	59.95
4			24.6	0.19	0.84	30.0	9.8	0.025	0.010	0.15	4.58	43	12	74.08	59.95
6			20.1	0.00	0.97	30.3	9.7	0.030	0.002	0.15	4.50	45	14	75.90	64.42
8			14.8	0.00	1.47	29.9	9.8	0.028	0.002	0.15	5.29	48	14	78.99	64.42
12			10.2	0.00	0.56	29.8	9.8	0.025	0.002	0.15	4.57	48	16	78.99	67.11
24			10.2	0.00	0.00	30.3	9.8	0.023	0.002	0.15		48	22	78.99	72.69

GOLD & SILVER EXTRACTION CALCULATIONS

			Oold			Olivoi		
Product	Quantity	Assay	Mass	Dist'n	Assay	Total	Dist'n	2
		(ppm)	(μg)	(%)	(ppm)	(μg)	(%)	3
								4
Solids (g)	1000.0	0.41	410	21.01	0.3	300	27.31	5.
Solution (mls)	1700.5	0.002	3	0.17	0.15	255	23.22	6.
Intermed. Carbon *			1047	53.67		319	29.01	7
Final Carbon (g)	10.2	48.0	491	25.14	22	225	20.46	
								8
Total Extraction				78.99			72.69	
T-4-1			4054	400.00		4000	400.00	

COMMENTS:

	1. NaCN Addition :		0.84	(kg/t)
n	2. NaCN Consumption :		0.43	(kg/t)
	3. Lime Consumption :		15.23	(kg/t)
	4. Site Water:		1.045	(SG)
1	5. Grind Size P 80 :		106	(µm)
2	6. Water to leach :		1857	(g)
1	7. Haycarb Yao activated carbon was			
_		11.00		

added to the slurry at the start of cyanidation.8. Evaporation losses made up prior to sampling at each period.

* Carbon samples removed at each sampling interval.

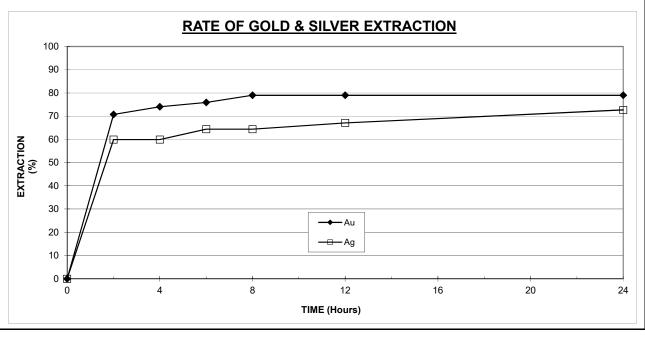
1.95

2.15 / 2.09

Calculated Head

Assay Head

Assay reported less than detection limit.



1.1

4.2





PROJECT	A18169 : LOST DOG PROJECT
CLIENT	BEACON MINERALS LTD
TEST No	JS3917
SAMPLE	LOST DOG COMPOSITE
GRIND	P80: 106 MICRONS
WATER	LOST DOG SITE WATER
DATE	JULY 2017

GRAVITY SEPRATION / CIL CYANIDATION TIME LEACH TESTWORK : OXYGEN SPARGED

		ADDITIONS					SOL	UTION D	ATA		PRI	EG. CAR	BON	EXTRACTION	
TIME (Hours)	Ore (g)	Water (mL)	Carbon Haycarb Yao (g)	NaCN (g)	Lime (g)	Oxygen (mg/L)	рН	NaCN (%)	Au (mg/L)	Ag (mg/L)	Wt (g)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
	1000.0	1777.0				11.9	7.6								
			33.6												
0				0.65	10.64		10.1	0.037	0.000	0.00		0	0	2.15	1.33
2			31.7	0.22	0.40	33.3	9.9	0.023	0.030	0.15	1.90	42	10	72.00	66.83
4			29.9	0.00	0.63	29.7	9.9	0.035	0.005	0.15	1.80	48	12	79.05	73.86
6			28.1	0.00	0.94	26.8	9.8	0.030	0.002	0.15	1.80	48	12	78.81	73.86
8			26.3	0.00	1.16	25.9	9.8	0.028	0.002	0.15	1.80	48	12	78.81	73.86
12			24.5	0.00	0.63	32.6	9.8	0.025	0.002	0.15	1.82	51	12	82.58	73.86
24			24.5	0.00	0.00	30.1	9.9	0.023	0.002	0.15		51	16	82.58	84.72

GOLD & SILVER EXTRACTION CALCULATIONS

			Gold			Olivei		ı
Product	Quantity	Assay	Mass	Dist'n	Assay	Total	Dist'n	2
		(ppm)	(μg)	(%)	(ppm)	(μg)	(%)	3
								4
Solids (g)	1000.0	0.41	410	19.57	0.15	150	16.61	ţ
Solution (mls)	1700.5	0.002	3	0.16	0.15	255	28.25	6
Intermed. Carbon *			432	20.61		106	11.70	7
Final Carbon (g)	24.5	51.0	1250	59.66	16	392	43.43	l
								8
Gravity Gold**			45	2.15		12	1.33	l
								l
Total Extraction				82.58			84.72	l
Total			2005	100.00		003	100.00	1

COMMENTS:

	1. NaCN Addition:	0.87 (kg	/t)
n	2. NaCN Consumption :	0.46 (kg	/t)
	3. Lime Consumption :	14.40 (kg	/t)
	4. Site Water :	1.045 (SC	3)
1	5. Grind Size P 80 :	106 (µr	n)
5	6. Water to leach :	1857 (g)	
0	7. Haycarb Yao activated carbon was		

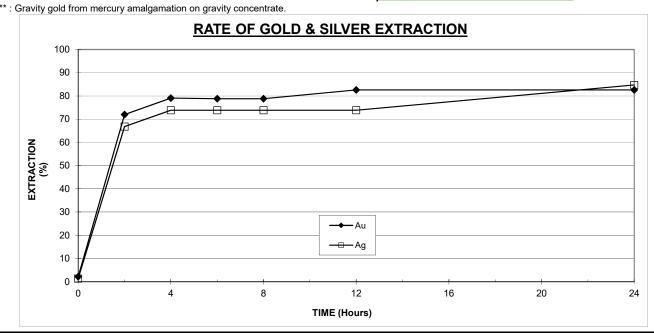
- added to the slurry at the start of cyanidation.

 8. Evaporation losses made up prior to sampling
- at each period.

2.15 / 2.09

Calculated Head

Assay reported less than detection limit.



0.9

^{*} Carbon samples removed at each sampling interval.





PROJECT	A18169 : LOST DOG PROJECT
CLIENT	BEACON MINERALS LTD
TEST No	JS3934
SAMPLE	LOST DOG COMPOSITE
GRIND	P100: 75 MICRONS [STAGE GRIND]
WATER	LOST DOG SITE WATER
DATE	AUGUST 2017

CIL CYANIDATION TIME LEACH TESTWORK : OXYGEN SPARGED

		ΑI	DDITIONS				SOL	UTION D	ATA		PREG. CARBON			EXTRACTION	
TIME (Hours)	Ore (g)	Water (mL)	Carbon Haycarb Yao (g)	NaCN (g)	Lime (g)	Oxygen (mg/L)	рН	NaCN (%)	Au (mg/L)	Ag (mg/L)	Wt (g)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
	955.4	1697.8				8.2	7.3								
			33.6												
0				0.65	2.30		9.4	0.038	0.000	0.00		0	0	0.00	0.00
2			28.2	0.27	0.00	37.8	9.4	0.028	0.025	0.15	5.48	45	6	73.04	41.44
4			21.8	0.00	0.00	22.3	9.4	0.025	0.005	0.15	6.40	51	8	79.45	46.68
6			17.0	0.00	0.00	27.9	9.3	0.030	0.002	0.15	4.75	51	8	79.22	46.68
8			11.6	0.27	0.00	28.1	9.1	0.028	0.002	0.15	5.40	54	8	81.62	46.68
12			6.8	0.00	0.58	31.3	8.9	0.025	0.002	0.15	4.76	57	8	83.25	46.68
24			6.8	0.00	0.00	30.4	8.9	0.023	0.002	0.15		63	8	85.18	46.68

J & SILVER	REXTRACTION	CALCULATIONS

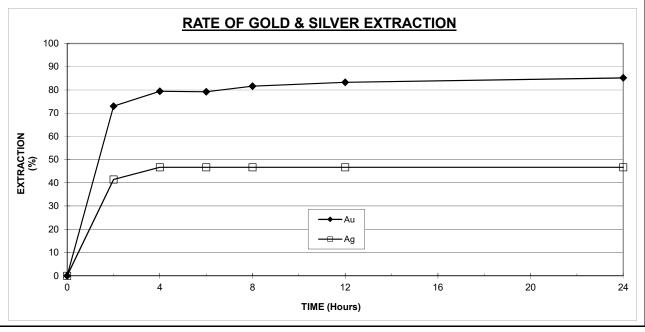
			Gold		Silver			
Product	Quantity	Assay	Mass	Dist'n	Assay	Total	Dist'n	2
		(ppm)	(μg)	(%)	(ppm)	(μg)	(%)	3
								4
Solids (g)	955.4	0.33	315	14.82	0.6	573	53.32	5
Solution (mls)	1624.7	0.002	3	0.15	0.15	244	22.67	6
Intermed. Carbon *			1378	64.78		203	18.92	7
Final Carbon (g)	6.8	63.0	431	20.25	8	55	5.09	
								8
Total Extraction				85.18			46.68	
Total			2128	100.00		1075	100.00	
Calculated Head		2.23			1.1			
Assav Head		2.15 / 2.09)		4.2			

COMMENTS:

	1. NaCN Addition:	1.25 (kg/t)
'n	2. NaCN Consumption:	0.84 (kg/t)
)	3. Lime Consumption :	3.01 (kg/t)
	4. Site Water:	1.045 (SG)
2	5. Grind Size P100 :	75 (µm)
7	6. Water to leach :	1774.2 (g)
2	7. Haycarb Yao activated carbon was	

- added to the slurry at the start of cyanidation. 8. Evaporation losses made up prior to sampling
- at each period.

Assay reported less than detection limit.



Carbon samples removed at each sampling interval.



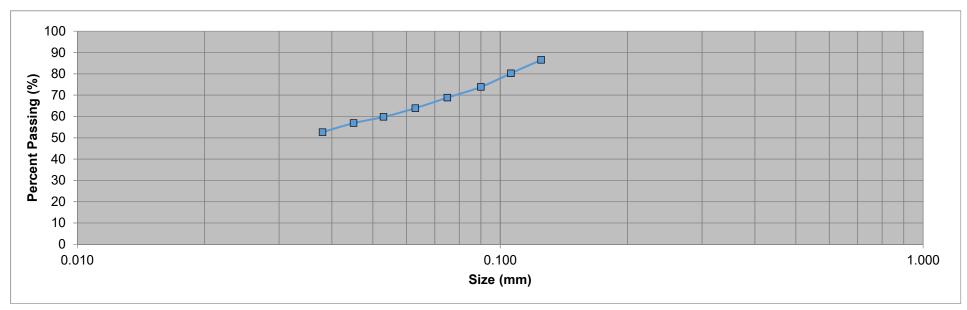


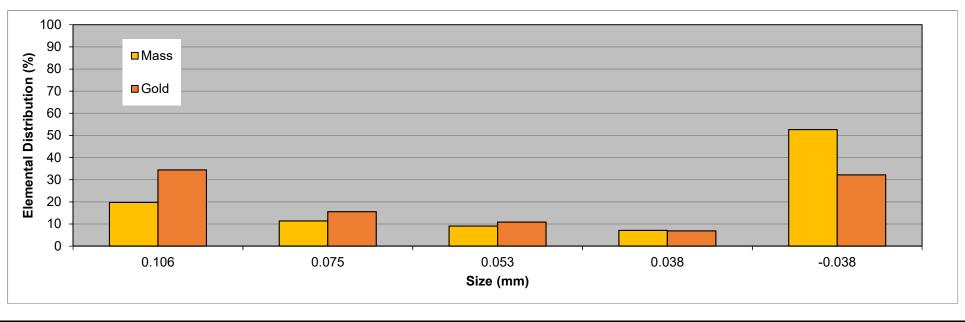
Size by Assay Testwork Details and Results

PROJECT	A18169 - Lost Dog Project
CLIENT	Beacon Minerals Ltd
COMPOSITE	Lost Dog Composite - Leach Residue ex JS3916
DATE	Thursday, 27 July 2017

Size	Mass	Mass	Passing	Go	old
(mm)	(g)	(%)	(%)	(g/t)	(% dist)
0.106	99.9	19.7	80.3	0.74	34.4
0.075	57.8	11.4	68.8	0.58	15.6
0.053	45.8	9.05	59.8	0.51	10.9
0.038	36.2	7.16	52.6	0.41	6.91
-0.038	266.2	52.6		0.26	32.2
Calc'd Head	505.9	100.0		0.42	100.0
Assay Head				0.41	

Calc'd P80: 105 μm







APPENDIX IV

Short Diagnostic Gold Leach Testwork Details and Results





PROJECT	A18169 : LOST DOG PROJECT
CLIENT	BEACON MINERALS LTD
TEST No	JS3935
SAMPLE	LOST DOG COMPOSITE
	LEACH RESIDUE: -75µm SIZE FRACTION [EX JS3917]
WATER	PERTH TAP WATER
DATE	AUGUST 2017

<u>DIRECT CYANIDATION LEACH AS PART OF DIAGNOSTIC TESTWORK</u>

Time											Au
(Hours)	Sample	Water	NaCN	NaOH	LeachWELL	Oxygen	рН	NaCN	Au		Extraction
	(g)	(mL)	(g)	(g)		(ppm)		(%)	(mg/L)		(%)
							7.4				
0	479.4	719.1	35.95	5.03	14.4	9.2	10.2	5.00	0.00		0.00
24			0.00	0.00	0.00	36.3	10.3	4.28	<0.05		14.02

GOLD EXTRACTION CALCULATIONS

			Gold		
Product	Quantity	Assay (ppm)	Mass (μg)	Distrib'n (%)	
Solids (g) Solution (mL)	479.4 719.1	0.23 <0.05	110 18	85.98 14.02	
Total Extraction			18	14.02	
Total			128	100.00	
Calculated Grade		0.27	·		
Assay Grade		N/A	·		

COMMENTS:

•	1.	NaCN addition :	75.0	(kg/t)
2	2.	NaCN consumption :	10.8	(kg/t)
3	3.	NaOH addition :	10.5	(kg/t)
4	4.	Perth tap water used :	1.000	(SG)
Ę	5.	Water weight to leach :	1500.0	(g)
6	3.	Grind size :	#REF!	

Assay reported below detection limit

Gold Diagnostic Summary:

ppm	Distribution (%)
0.04	14.56
0.16	62.14
0.06	23.30
0.26	100.00
	0.04 0.16 0.06

APPENDIX 3B LOST DOG

BUREAU VERITAS MINERAL LABORATORIES METALLURGY REPORT – 7KM4071A



BV MINERAL LABORATORIES

ABN 30 008 127 802

22A Atbara Street, Kalgoorlie, Western Australia, 6430 Phone +61 8 9021 8049 Fax +61 8 9021 8136

Submission No: LD 1700

Date received: Date reported:

Our ref:

3/03/2017 10/03/2017

7KM4071A

				Final Soln					Residu	е			Calculated	%	Calculated	%
Sample	Solid Wt	Soln Wt	t % Solids	Au 1	Au 2	Au Av	Au 1	Au 2	Au 3	Au 4	Au 5	Au Av	Head	Extraction	Head	Extraction
JD17C 130 9-10	500.0	1000.0	33.3	1.05		1.05	0.30	0.26				0.28	11745	200.0000	2.38	88.2
JD17C 119BR 12-13	500.0	1000.0	33.3	1.48		1.48	0.15	0.16				0.16			3.12	95.0
JD17C121BR 14-15	500.0	1000.0	33.3	0.600		0.600	0.05	0.04		-		0.05			1.25	96.4
JD17C 118BR 10-11	500.0	1000.0	33.3	0.276		0.276	0.05	0.04				0.05			0.60	92.5
JD17C 80 9-10	500.0	1000.0	33.3	1.04		1.04	0.26	0.22				0.24			2.32	89.7
JD17C 67 20-21	500.0	1000.0	33.3	0.410		0.410	0.09	0.09				0.09			0.91	90.1
JD17C 67 21-22	500.0	1000.0	33.3	0.936		0.936	0.20	0.20				0.20			2.07	90.3
JD17C 110BR 14-15	500.0	2000.0	20.0	0.018	0.012	0.015	0.20	0.26	0.23	0.04	0.04	0.23	0.10	60.0	0.29	20.7
GC 08BR 8-9	500.0	1000.0	33.3	0.496		0.496	0.11	0.11	0,00	0.01	0.01	0.11	0,10	00.0	1.10	90.0
JD17A 08BR 15-16	500.0	1000.0	33.3	0.668		0.668	0.30	0.31				0.31			1.64	81.4
JD17A 10BR 14-15	500.0	1000.0	33.3	0.464		0.464	0.12	0.10				0.11			1.04	89.4
GC 19BR 14-15	500.0	1000.0	33.3	0.718		0.718	0.28	0.25				0.27			1.70	84.4
GC 17BR 16-17	500.0	2000.0	20.0	2.64	2.92	2.64	0.91	0.91	0.95	0.42	0.42	0.91	11.0	96.2	11.5	92.1
GC 10BR 17-18	500.0	1000.0	33.3	1.98	1.98	1.98	0.16	0.15	7.44	51.14	0.12	0.16	71.0	30,2	4.12	96.2
GC 11BR 11-12	500.0	1000.0	33,3	0.386		0.386	0.12	0.11				0.12			0.89	87.0
Units	g	g		ppm	ppm	ppm	g/t	g/t				g/t			·g/t	%
Det Lim Scheme	Ť	1		0.005 Met4A	0.005 Met4A	0.005 Met4A	0.01 MET5C	0.01 MET5C				0.01 MET5C			0.01	0.1
Sample	Solid	Solution		Solution	Solution	Solution	Solid	Solid				Solid				10000

Considered Normal Filtering Brecciated Siltstone Considered Normal Filtening Siltstone Considered Normal Filtering Siltstone Take Significant Time for Filtering Siltstone/Clay Considered Normal Filtering Clay/Siltstone Considered Normal Filtering Clay/Siltstone Considered Normal Filtering Clay/Siltstone Take Significant Time for Filtering Clay/Siltstone Considered Normal Filtering Siltstone Considered Normal Filtering Siltstone Take Significant Time for Filtering Siltstone Considered Normal Filtering Sittstone Take Significant Time for Filtering Clay Take Significant Time for Filtering Clay Considered Normal Filtering Siltstone

APPENDIX 3C LOST DOG

TRIAL PARCEL METALLURGICAL STATISTICS

Metallurgical Stats

Day	Date	Dry T Milled	Assay LF total Au g/t	Assay Au in LF g	Gravity Au g	Grav Rec Calc %	Calc MF Assay g/t	Tail assay Au g/t	Total Tail Au g	Calc Rec %	CV7 Au g/t
1	2-Mar-17	788	3.17	2495	295	11	3.54	0.15	116	95.8	1.72
2	3-Mar-17	1,281	2.65	3,393	87	2	2.72	0.25	314	91.0	1.66
3	4-Mar-17	1,098	2.48	2,726	42	1	2.52	0.28	307	88.9	2.94
4	5-Mar-17	916	3.16	2,891	0	0	3.16	0.27	252	91.3	2.81
5	6-Mar-17	542	2.90	1,572	0	0	2.90	0.22	117	92.6	1.94
Total		4,625	2.83	13,078	424	3.1	2.92	0.239	1,105	91.8	2.23

Metallurgical Balance	
Opening circuit stock (g Au) =	36,772.00
Closing circuit stock (g Au) =	27,039.14
Change in circuit stock (g Au) =	-9,733
Gold poured (g) =	25,629
Total Au Produced (g) =	15,897
Total Au Produced (oz) =	511.088
Total Au in Tail (g) =	1,105
Total Au to Mill (g) =	17,002
Tonnes Milled (t) =	4,625
Bullion Reconciled Mill Feed grad	le (g/t) = 3.68
Assay calculated grade (g/t) =	2.92
Reconciled Recovery (%) =	93.50
Reconciled : Assayed feed grade	(%) = 125.9

APPENDIX 4 LOST DOG

"GEOTECHNICAL ASSESSMENT OF THE LOST DOG PROJECT" $26^{\rm th}\,{\rm APRIL}\,2018$ GREEN GEOTECHNICAL PTY LTD

26th April 2018

Mr Alex McCulloch Project Manger Beacon Minerals Limited 144 Vivian St Boulder WA 6432

via email: alexm@beaconminerals.com.au

Dear Alex,

Tel (08) 9091 1878

ACN 131 764 817 ABN 20 158 533 644

8 Vitali Crescent

Fax (08) 9091 8095

Kalgoorlie WA 6430

Green Geotechnical Pty Ltd

Geotechnical Assessment of the Lost Dog Project

1. Introduction

This report outlines the results of a geotechnical assessment of the proposed pit at the Lost Dog Prospect which is located approximately 35km northwest of Coolgardie. Work was undertaken by Tim Green at the request of Mr Alex McCulloch, Project Manager, Beacon Minerals Limited. The purpose of the assessment was to review the stability of the proposed slope parameters for the proposed pit at the Lost Dog Prospect.

This assessment was conducted based on data collected from a previous geotechnical study completed by MineGeoTech Pty Ltd^[1] which focused on the stability of the Cammi's Creek and West pits which occur within the project area. Additional assessments have been completed by CMW Geosciences and Land and Marine Geological Services Pty Ltd (L&MGSPL which assessed the proposed in-pit tailings storage facility. Additional strength testing data was provided by Beacon Minerals. Stability analyses conducted are based on the material strength data provided and assessments of the performance of the existing pits within the project area.

2. Scope of Work

The scope of work covered by this assessment includes the following:

- 1. Review the geotechnical assessment completed by MineGeoTech Pty Ltd^[1] on the existing Cammi's Creek and West Pits which occur within the project area.
- 2. Assess the proposed pit design at the Lost Dog Project, based on additional strength testing data collected by Beacon Minerals Limited, CMW Geosciences^[3] and L&MGSPL^[5], and inspection of the existing pits within the project area.

3. Data Sources

- Evans, P., 2017. Geotechnical review of Cammi's Creek Mining. MineGeoTech Pty Ltd. Report dated 21st February 2017.
- Hogg, C., 2018. Stability Assessment Lost Dog Pit TSF. Beacon Mining Project. Report No. PER2017-0581AB let 190118 rev1. Report dated 21st February 2018. CMW Geosciences.
- Lane, C., 2018. Lost Dog and Black Cat In-pit Tailings Storage Facilities Geotechnical Assessment. Report dated 17th March 2018. Land and Marine Geoogical Services Pty Ltd.
- Summary of the Regional and Local Geology completed by Greg Jorgensen.
- Report on intact Rock Properties testing completed by the Western Australian School of Mines

 Discussions with Alex McCulloch regarding the Proposed Mining Plan Limited, and Greg Jorgensen regarding the Local Geology.

4. Mine Grid and Slope Terminology

All data structural and directional data collected in this study is reported relative to MGA94 North.

5. Proposed Mining Plan

The proposed pit at the Lost Dog Prospect will be mined in four panels from east to west (Figure 1). The pit will reach a maximum depth of 27m below the surface.

The maximum wall height on the edges of the pit will be 20m.

The proposed slope parameters are:

Surface (383mRL) to 380mRL: 45°

• 380mRL to Pit Floor: 60°

Access will be via ramps located on the northern wall of each panel.

The pits will be progressively backfilled to the surface with tailings as mining progresses. An engineered embankment will be keyed into the pit walls prior to the placement of tailings within each panel.

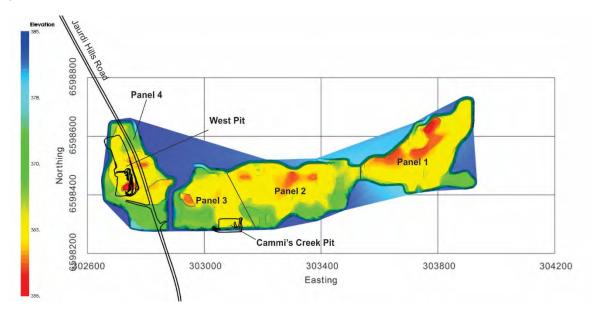


Figure 1: Proposed Pit at the Lost Dog Prospect

6. Previous Mining

Previous mining within the Lost Dog Project area occurred in two pits: Cammi's Creek and West Pit (Figure 1). Detailed surveys of the pits are unavailable, however the pits reached depths of between 10 and 15m (obscured below the current level of water in each pit).

Batter angles were formed at between 60 and 65° from 2m below surface, with the upper part of the slopes mined at 45° within the overlying soils.

The current water level within each pit is around 10m, and despite the formation of pit lakes the walls remain stable (Figures 2). Minor undercutting of the western wall is evident in the West Pit immediately above the current water level (Figure 3).





Figure 2: Walls within the Cammi's Creek Pit remain stable following formation of Pit Lake (looking west)



Figure 3: Minor undercutting of western wall in the West Pit (viewed looking south).



7. Geology

The Lost Dog Prospect occurs within a palaeochannel system up to 50m thick. The palaeochannel system comprises the following sequence^[2]:

- Near-surface mottled clays which vary in thickness from 3 to 10m;
- Puggy, sticky plasticine-like clays up to 20m thick;
- Transported clay, pisolitic gravel and sandy clay up to 30m thick;
- Basal sand to sandy clay horizon 5 to 7m thick.

The entire sequence is covered by 1 to 3m of calcareous soil and/or ferruginous alluvium.

Gold mineralisation at Lost Dog occurs within two flat-lying horizons within the palaeochannel.

The larger lode occurs 8 to 18m below the surface, and the thinner lode occurs 18 to 25m below the surface. The main gold-bearing horizons are hosted by a very hard silcrete, silica dolomite unit and inter-layered clays. Less silcrete is present within lower Au-bearing lode, which is dominated by clays.

Within the existing Cammi's Creek and West Pits the silcrete unit is the dominant rock type (Figure 4) which is inter-layered with sub-horizontal clay horizons.

The Bali Monzogranite underlies the Lost Dog Prospect and is variably weathered 40 to 60m below the surface. It is not expected to be intersected by the proposed pit.



Figure 4: Hard silcrete exposed within the Cammi's Creek Pit.



8. Material Strength Parameters

Intact rock property testing was conducted on four samples of silcrete in the Cammi's Creek (East) Pit and West Pits by the Western Australian School of Mines and are summarised in Table 1. Test Reports are included in Appendix A.

Rocktype	Location	Unit Weight (kN/m³)	UCS (MPa)	Young's Modulus (MPa)	Poisson's Ratio
Pale brecciated silcrete	East Pit	26.28	128		
Pale silcrete	West Pit	24.36	183	56.2	0.278
Dark brecciated silcrete	East Pit	22.71	48		
Greenish silcrete	West Pit	22.58	112	32.6	0.243

Table 1: Results UCS testing on samples of weathered material

Atterberg Limit testing on the clays exposed within the Cammi's Creek (East) Pit were conducted on four samples; two by MineGeoTech^[1] and two samples (17110001 & 17110002) by CMW Geosciences^[3]. Atterberg Limit test results are summarised in Table 2.

Sample No.	1	2	17110001	17110002
Liquid Limit (%)	32	36	66	67
Plastic Limit (%)	22	22	22	27
Plastic Index (%)	10	14	44	40
Linear Shrinkage	5	7	18.0+	13
(%)				
Moisture Content	-	-	7.3	4.1

Table 2: Results of Atterberg Limit Testing^[2]

Based on the method describe by Wesley (2003)^[4], MineGeoTech determined the upper clays have an internal friction angle of 25°.

Soil penetrometer testing conducted by MineGeoTech^[1] indicated the compressive strength of the clay material exceeded 600 and 1000kPa in the west and Cammi's Creek Pits respectively.

A single triaxial test on a remoulded sample of gravelly clay was reported by CMW Geosciences^[3]. The sample was remoulded to a target density of 95% of the standard maximum dry density. The following peak shear strength parameters were determined:

Cohesion: 26.9kPa

Angle of shear resistance (φ): 22.4°

9. Structure

The Lost Dog Prospect occurs within a flat-lying palaeochannel – sequence. No large-scale structures or joints were observed within the existing pits and slope stability is expected to be controlled by the sub-horizontal contacts between the hard silcrete units and interlayered clays.

10. Hydrogeology

The hydrogeology has been described by L&MGSPL^[5], and is summarised below.

The Lost Dog paleo-drainage is the local groundwater aquifer unit which will be the source of process supply during mining operations. During resource drilling, water has been intersected at various depths ranging from 12 to 24 meters. Recent monitoring has identified water levels which range from



7 to 11 m below ground level. The correction of water levels to topography indicate that the groundwater surface along the north edge of Panels 1 to 5 is consistently at an elevation of 177 to 178 mAHD.

L&MGSPL^[5] determined hydraulic conductivities of 10⁻⁸ to 10⁻⁹m/s for composite clay material exposed within the existing pits at the Lost Dog Prospect. The sandy clay units below the existing pits are likely to have much higher hydraulic conductivities.

The water table occurs approximately 10m below the surface based on the current water level within the existing Cammi's Creek and West Pits.

11. Previous Geotechnical Investigations and Results

A previous geotechnical investigation on the Cammi's Creek and West Pit which are located within the Lost Dog Project Area (Figure 1) was completed by MineGeoTech for Fenton & Martin Mining Development^[1]. Results of the investigation are summarised below.

- Observational assessment of both pits indicate the walls mined with batter face angles (BFA) of 60 to 65° and berm widths of 0.5m are stable.
- No deterioration of the crest was observed.
- The length of time the pits remain open will affect wall stability.
- Material within the Cammi's Creek Pit plots on the transition from moderate to high resistance to erosion.
- It was recommended that pit walls are maintained at an angle no steeper than 65° without further assessment
- Material in the West pit appears to be of lower strength than that of Cammi's Creek and further testing of material strength parameters was recommended if the West Pit was to be expanded.

The geotechnical assessments completed by CMW Geosciences^[3] and L&MGSPL^[5] relate to the stability of the containment embankments and mine waste buttressing for the tailings storage facility, which is beyond the scope of this assessment.

12. Numerical Analysis of the proposed wall angles of the Lost Dog Pit

Two-dimensional finite element modelling using the Shear Strength Reduction (SSR) Method described by Hammah *et al.*^[6] was conducted on sections constructed through the eastern and western walls of the proposed trial pit using the finite element analysis package *RS2*^[7].

The strength parameters used in the modelling are based on test results described in Section 8. Material strength parameters assumed in modelling are listed in Table 3.

Material	Density (kN/m ³⁾	Cohesion (kPa)	Friction Angle
Overlying soils	20	10	30°
Upper Clay	20	8	25°
Main silcrete unit*	24.0	1327	21.5°
Sandy Clays	20	26.9	22.4°
Weathered granite (saprolite)	24.0	125	25

Table 3: Strength parameters assumed in SSR analysis of the Lost Dog Pit. *Intact strength reduced assuming a GSI of 20 based on the presence of inter-layed clays.

The geological sequence assumed in modelling is shown in Figure 4.



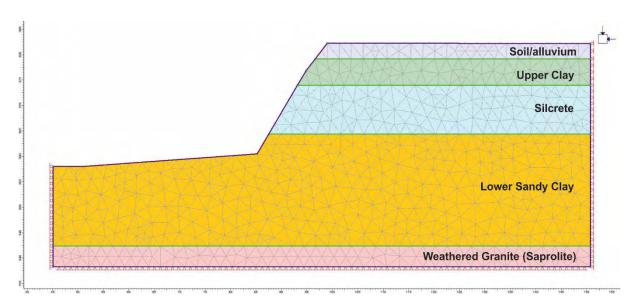


Figure 4: Geological sequence assumed in modelling

Hydraulic conductivities (K) of the various materials are unknown, and a value of 1 x 10⁻⁷m/s has been assumed for the upper clays and silcrete (slightly higher than that determined from Atterberg Limit and Partical Size Distribution testing^[5]). A higher K value of 5.8 x 10⁻⁴ m/s has been assumed for the lower sandy clays (aquifer), based on values is similar material from another site. The water table is assumed to occur at the 372mRL (10m below surface).

Figure 5 shows the potential failure surface modeled for the proposed pit walls. The predicted failure surface involves foundation failure of the underlying clay unit at the toe of the slope. A factor of safety (equivalent to the critical strength reduction factor – SRF) of 1.36 was determined for the proposed wall angles of:

- 45° (Surface to 380mRL)
- 60° (380mRL to Pit floor).

The resultant factor of safety (FOS) determined for the proposed slope parameters are in line with guidelines outlined by the DMIRS^[8]. It should be noted that the material parameters assumed in modelling are considered to be conservative, based on the limited material test work conducted. The proposed mining sequence will allow adjustments to the slope parameters to be made following the completion of Panel 1.

Upon completion of mining of each panel, the excavation will be used as a tailings storage facility, with an engineered embankment to be keyed into the pit wall. Progressive backfilling of the pits will result in an increase of the FOS of the final pit walls, and the construction of an engineered embankment will prevent deterioration of the slope as the water level in the pit rises. Geotechnical analysis of the in-pit tailings storage facility has been conducted by CMW Geosciences^[3] and L&MGSPL^[5] and is beyond the scope of this assessment.



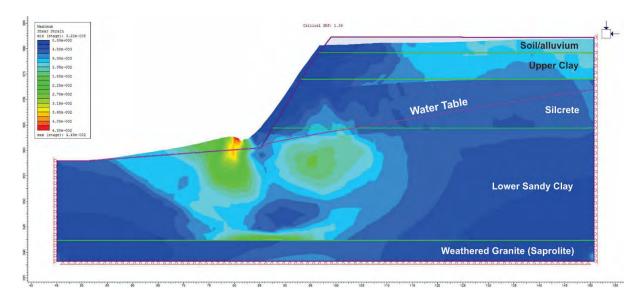


Figure 5: Potential failure surface (exaggerated) determined for the southern wall of the proposed Lost Dog Pit. FOS = 1.36.

13. Mine Closure

The proposed Lost Dog Pit will be used as an in-pit tailings storage facility as mining progresses. Prior to the placement of tailings, an engineered embankment to be keyed into the pit wall. The pits will be backfilled to the surface thereby mitigating the risk of inadvertent access to the excavation following mine closure, and negating the requirement for construction of an abandonment bund around the pit.

14. Impact of Mining on the Jaurdi Hills Road

Initial mining approvals are being sought for Panels 1 to 3, with Panel 4 to be mined at a later stage. Mining of Panel 4 will require the temporary relocation of the Jaurdi Hills Road in the western part of the prospect (Figure 1). The Jaurdi Hills Road is located 41m from the western edge of Panel 3 at its closest point, on the southern edge of the proposed pit

The planned height of the western wall of Panel 3 is 13m. The zone of potential instability as defined by DMIRS guidelines^[9] will extend 18m from the pit crest, with any temporary (or permanent) abandonment bunt to be located 28m from the crest on the western side of Panel 3. As such mining of Panels 1 to 3 is unlikely to impact on the Jaurdi Hills Road.

15. Summary and Recommendations

This geotechnical assessment was conducted based on data collected from previous geotechnical study completed by MineGeoTech Pty Ltd^[1] which focused on the stability of the Cammi's Creek and West pits located within the project area. Additional data was obtained from geotechnical studies on the proposed in-pit tailings storage facility conducted by CMW Geosciences^[3] and L&MGSPL^[5].

Additional material strength testing data was provided by Beacon Minerals. Stability analyses conducted are based on the material strength data provided and assessments of the performance of the existing pits within the project area.

Two-dimensional finite element modelling using the Shear Strength Reduction (SSR) Method was conducted on a section constructed through the proposed (steepest) slope on the southern wall. 20m



high slopes formed in inter-layered clay and silcrete were analysed based on the following slope parameters:

- 45° (Surface to 380mRL);
- 60° (380mRL to Pit floor).

A factor of safety (FOS) of 1.36 was determined for the steepest slopes in the proposed Lost Dog Pit.

The analysis completed is based on limited geotechnical data given the size of the overall pit. Variations in the thicknesses of the units across the deposit may have a significant influence of slope stability.

Increased water inflows are expected to be encountered towards the base of the pit as the main part of the aquifer is exposed. This could have a significant impact on mining operations and uncertainties in the hydrogeological model may have an adverse impact on slope stability.

If additional diamond core is available or further diamond drilling is to be conducted at the Lost Dog Prospect, it is recommended that a series of multi-stage consolidated undrained triaxial tests are conducted on samples of the various materials in which the pit is to be mined. Any additional adta obtained should be used as inputs into the slope stability assessment.

Uncertainties in the geological and hydrogeological models and material strength parameters (in particular the foundation material) are somewhat mitigated by the shallow depth of the pit and wide nature of the pit floor, thereby reducing the risk of exposure slope instability. The proposed mining sequence will allow learnings gained in during mining of Panel 1 to be applied to remaining panels and improve understanding of geological variations and variations in material strengths.

Modification to wall designs should be made based on geotechnical assessment of these additional data. Regular inspections of the pit crests should be conducted in order to identify tension cracks which may develop within the weathered material.

Recommendations made in this report are based upon the information made available to the author and assume mining activities will be carried out to a high standard over the life of the project.

Yours faithfully,

Timothy Green

Green Geotechnical Pty Ltd



References

- [1] Evans, P., 2017. Geotechnical review of Cammi's Creek Mining. MineGeoTech Pty Ltd. Report dated 21st February 2017.
- [2] Jorgensen, G. Summary of regional and Local Geology Descriptions Jaurdi Hills Gold Project Beacon Minerals Ltd. Email dated 15th January 2018.
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- [4] Wesley, L.D., 2003. Residual Strength of Clays and Correlations Using Atterberg Limits. Géotechnique 53, No7, pp 669 672.
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- [8] Department of Industry Resources, 1999. Geotechnical Considerations in Open Pit Mines.
- [9] Department of Industry Resources, 1997. Safety Bund Walls Around Abandoned Open Pit Mines. Guideline. Document No. ZMA048HA.



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Appendix A

UCS Test reports





REPORT ON INTACT ROCK PROPERTIES TESTING

for

Lost Dog

Beacon Minerals Ltd

WESTERN AUSTRALIAN SCHOOL OF MINES GEOMECHANICS LABORATORY

Tested by: Mrs Myriam Sullivan Laboratory Technician

Reported by: Mr Pat Hogan Research Engineer

Reviewed by: Dr Alan Thompson Principal Research Fellow

June 2017





CONTENTS

1.	Sample Description	1
2.	Uniaxial Compressive Strength (UCS) Test Results	2
3.	Elastic Properties for UCS Samples	3
4.	Analysis of Elastic Properties	4
5.	Photographs of Test Samples	ť





 Table 1
 Sample Description

WASM Sample No.	Client Sample No.	Borehole ID	Location	Lithology	Comments	Test Requested
1	1	-	East Pit	Pale brecciated siltstone	-	UCS
2	2	-	West Pit	Pale siltstone	-	UCS, E & v
3	3	-	East Pit	Dark Brecciated siltstone	-	UCS
4	4	-	West pit	Greenish siltstone	-	UCS, E & v

Note:

¹⁾ All sample preparations and testing followed the International Society for Rock Mechanics standards (ISRM, 1981 and 1999).

²⁾ The unit weights of the samples were measured using calliper and scale method, and provided free of charge.





 Table 2
 Uniaxial Compressive Strength (UCS) Test Results

WASM	Client	Client Borehole		T '4h ala ara	Diameter	Length	Mass	Unit	Load	UCS	Failure Feature			
Sample No.	Sample No.	ID	Location	Lithology	(mm)	(mm)	(g)	Weight (kN/m ³)	(kN)	(MPa)	Mode	Angle (°)	Nature	
1	1	-	East Pit	Pale brecciated siltstone	54.61	136.81	858.60	26.28	299.6	128	A, Bi	13, 27	Violent	
2	2	-	West Pit	Pale siltstone	54.52	136.80	793.30	24.36	427.1	183	C	-	Violent	
3	3	-	East Pit	Dark Brecciated siltstone	54.46	136.69	737.40	22.71	111.4	48	Bi	9	Quiet	
4	4	-	West pit	Greenish siltstone	54.49	136.69	733.90	22.58	262.1	112	С	-	Violent	

1) Failure Modes: A: Axial splitting

B: Shear (Bi for shear through intact rock and Bs for shear along structure)

C: Multiple cracking

2) The strong samples were tested using the INSTRON machine and the weaker samples tested using the Avery machine.

3) The failure angles were measured between the loading axis and the failure plane.





Table 3 Elastic Properties for UCS Samples

WASM	Client	Client	Lagation		Young	g's Modulus (C	GPa)	Poisson's Ratio			
Sample No.	Sample No.	Borehole ID	Location	Lithology	$\mathbf{E}_{Tangent}$	$\mathbf{E}_{\mathbf{Secant}}$	$\mathbf{E}_{\mathbf{Average}}$	$\mathbf{v}_{\mathrm{Tangent}}$	$v_{ m Secant}$	$ u_{ m Average}$	
2	2	-	West Pit	Pale siltstone	56.3	59.8	56.2	0.276	0.285	0.278	
4	4	-	West pit	Greenish siltstone	32.6	34.6	32.6	0.243	0.234	0.243	

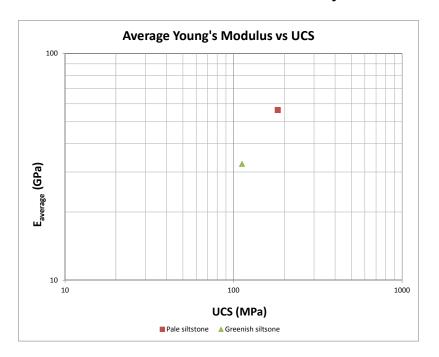
Note:

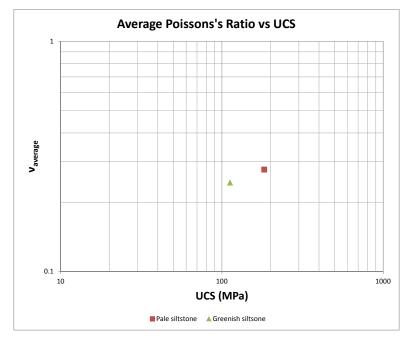
- 1) Young's Modulus and Poisson's ratio were calculated at stress level equal to 50% of the uniaxial compressive strength for $E_{Tangent}$, E_{Secant} and $v_{Tangent}$,
- 2) The average values for E and v were determined from the average slope of the straight-line portion between 35% and 65% of the uniaxial compressive strength.
- 3) All stress and strain data were acquired automatically using the software and hardware of National Instruments.





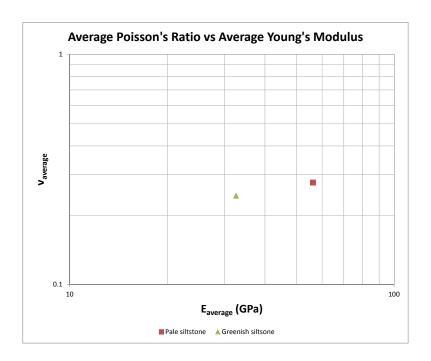
Analysis of Elastic Properties







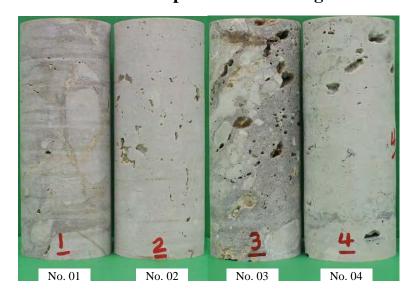








UCS Samples before Testing



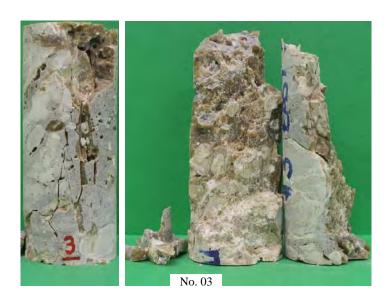




UCS Samples after Testing











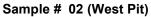


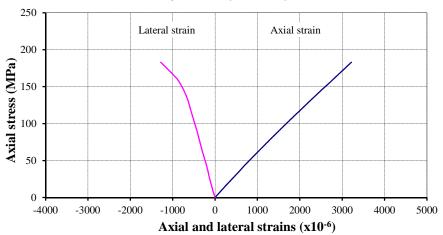




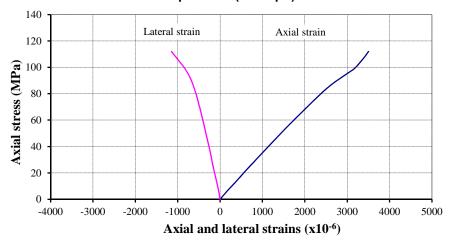
APPENDIX A

Stress- Strain Curves for Elastic Properties





Sample # 04 (West pit)



APPENDIX 5 LOST DOG

OPTIMUM MINING TONNAGE SENSITIVITIES SUMMARY

LOST DOG SENSITIVITY SUMMARY INCORPORATING 2% MINING DILUTION at 0.00g8 AND 98% MINING RECOVERY

GOLD	MIN	ING RESERVE	S WAS	STE	TOTAL S	STRIPPING	MILL	OUNCES	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CASH COST										OPE	RATING PRO	IT.									
PRICE	VOLUME	TONNAGE	GRADE VOL	UME V	OLUME	RATIO	RECOVERY	RECOVERED	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	per OUNCE	@ \$1,000/oz	@ \$1,050/oz	@ \$1,100/oz	@ \$1,150/oz	(2) \$1,200/oz	@ \$1,250/oz	@ \$1,300/oz	@ \$1,350/oz	@ \$1,400/oz	@ \$1,450/oz	@ \$1,500/oz	@ \$1,550/oz	@ \$1,600/oz	@ \$1,650/oz	@ \$1,700/oz	@ \$1,750/oz	@ \$1,800/oz	@ \$1,850/oz	@ \$1,900/oz	@ \$1,950/oz	@ \$2,000/oz
(\$)	(bcm)	(t)	(g/t) (bc	m)	(bcm)	(bam:bcm)	(%)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
1,000	845,548	1,389,907	2.45 1,57	4,960	2,420,508	1.9	85	93,101	2,327,521	7,448,066	9,775,587	50,578,257	60,353,843	648	32,746,984	37,285,650	41,824,315	46,362,980	50,901,646	55,440,311	59,978,976	64,517,642	69,056,307	73,594,972	78,133,638	82,672,303	87,210,968	91,749,634	96,288,299	100,826,964	105,365,630	109,904,295	114,442,961	118,981,626	123,520,291
1,050	910,583	1,503,921	2.37 1,64	8,955	2,559,538	1.8	85	97,223	2,552,092	7,777,804	10,329,896	54,364,844	64,694,739	665	32,649,337	37,388,936	42,128,535	46,868,135	51,607,734	56,347,333	61,086,932	65,826,532	70,566,131	75,305,730	80,045,329	84,784,928	89,524,528	94,264,127	99,003,726	103,743,325	108,482,925	113,222,524	117,962,123	122,701,722	127,441,321
1,100	974,935	1,624,655	2.28 1,71		2,688,761	1.8	85	101,254	2,784,494	8,100,345	10,884,839	58,254,138	69,138,976	683	32,368,472	37,304,620	42,240,768	47,176,915	52,113,063	57,049,211	61,985,359	66,921,506	71,857,654	76,793,802	81,729,950	86,666,097	91,602,245	96,538,393	101,474,540				121,219,131		131,091,427
1,150	1,059,070	1,764,381	2.21 1,88		2,945,803	1.8	85	106,384	3,058,543	8,510,730	11,569,273	63,452,903	75,022,176	705		36,947,110	42,133,336	47,319,562	52,505,788	57,692,014	62,878,240	68,064,466	73,250,691	78,436,917	83,623,143			99,181,821					125,112,950		135,485,402
1,200	1,108,384	1,851,311	2.16 1,93		3,044,117	1.7	85	109,038	3,271,151	8,723,069	11,994,220	66,294,579	78,288,799	718	31,294,754	36,610,375	41,925,995	47,241,615	52,557,235	57,872,855	63,188,475	68,504,095	73,819,716	79,135,336	84,450,956	89,766,576	95,082,196		105,713,436					132,291,537	
1,250	1,154,978	1,932,004	2.11 1,99		3,148,680	1.7	85	111,492	3,484,130	8,919,372	12,403,502	69,019,262	81,422,764	730		36,201,453	41,636,696	47,071,938	52,507,180		63,377,665	68,812,907	74,248,149	79,683,391	85,118,634			101,424,360						134,035,814	
1,300	1,209,652	2,017,341	2.07 2,08		3,298,292	1.7	85	114,177	3,710,747	9,134,146	12,844,893	72,151,745		744	30,036,518	35,602,638	41,168,759	46,734,879	52,301,000	57,867,120	63,433,241	68,999,361	74,565,482	80,131,602	85,697,723	91,263,843		102,396,084						135,792,807	
1,350	1,248,761	2,082,089	2.04 2,13		3,385,239	1.7	85	116,009	3,915,307	9,280,729	13,196,036	74,364,433		755	29,463,719	35,119,163	40,774,607	46,430,051	52,085,494	57,740,938	63,396,382	69,051,826	74,707,270	80,362,714	86,018,158			102,984,490						136,917,154	142,572,598
1,400	1,286,913	2,143,297	2.01 2,19		3,479,910	1.7	85	117,743	4,121,001	9,419,430	13,540,430	76,547,221	90,087,651	765	28,832,650	34,572,615	40,312,580	46,052,546	51,792,511	57,532,476	63,272,441	69,012,406	74,752,371	80,492,336	86,232,301	91,972,266	97,712,231			114,932,126			132,152,021		143,631,951
1,450	1,327,782	2,207,288	1.98 2,26		3,594,806	1.7	85	119,554	4,333,822	9,564,297	13,898,119	78,909,612	92,807,731	776	28,090,960	33,919,204	39,747,447	45,575,691	51,403,934	57,232,177	63,060,421	68,888,664	74,716,908	80,545,151	86,373,395	92,201,638		103,858,125						138,827,586	
1,500	1,363,463	2,261,588	1.96 2,33		3,700,405	1.7	85	121,095	4,541,051	9,687,576	14,228,628	80,987,369	95,215,997	786	27,392,392	33,295,758	39,199,125	45,102,492	51,005,859	56,909,226	62,812,593	68,715,960	74,619,326	80,522,693	86,426,060	92,329,427		104,136,161					133,652,995		145,459,729
1,550	1,389,134	2,299,679	1.94 2,36		3,759,114	1.7	85	122,076	4,730,430	9,766,049	14,496,479	82,353,841	96,850,320		26,903,833	32,855,020	38,806,206	44,757,392	50,708,578	56,659,764	62,610,950	68,562,137	74,513,323	80,464,509	86,415,695	92,366,881		104,269,254					134,025,185		145,927,557
1,600	1,414,783	2,339,502	1.93 2,42		3,840,391	1.7	85	123,128	4,925,102	9,850,203	14,775,305	83,893,785	98,669,090		26,305,366	32,307,834	38,310,302	44,312,770	50,315,237	56,317,705	62,320,173	68,322,640	74,325,108	80,327,576	86,330,044	92,332,511	98,334,979						134,349,785		146,354,721
1,650	1,442,442	2,380,877	1.91 2,48		3,932,051	1.7	85	124,226	5,124,332	9,938,098	15,062,430	85,542,671			25,639,805	31,695,834	37,751,863	43,807,891	49,863,920	55,919,949	61,975,978	68,032,006	74,088,035	80,144,064	86,200,093	92,256,121	98,312,150						134,648,323		146,760,380
1,700	1,462,421	2,410,692	1.90 2,52	3,520	3,985,941	1.7	85	124,934	5,309,711	9,994,751	15,304,462	86,659,445			25,156,831	31,247,382	37,337,934	43,428,485	49,519,036	55,609,588	61,700,139	67,790,690	73,881,242	79,971,793	86,062,344	92,152,896	98,243,447			116,515,101			134,786,755		146,967,858
1,750	1,482,367	2,440,609	1.88 2,55	8,023	4,040,390	1.7	85	125,642	5,496,832	10,051,350	15,548,181	87,778,877			24,670,595	30,795,636	36,920,678	43,045,719	49,170,760	55,295,801	61,420,842	67,545,883	73,670,924	79,795,965	85,921,007	92,046,048	98,171,089							141,046,377	147,171,418
1,800	1,495,120	2,458,510	1.88 2,58		4,075,959	1.7	85	126,068	5,673,054	10,085,430	15,758,484	88,471,720			24,359,028	30,504,837	36,650,646	42,796,455	48,942,264	55,088,073	61,233,881	67,379,690	73,525,499	79,671,308	85,817,117	91,962,926		104,254,544					134,983,588		147,275,206
1,850	1,511,442	2,481,435	1.87 2,62		4,141,413	1.7	85	126,665	5,858,254	10,133,197	15,991,451	89,500,406		833	23,864,733	30,039,650	36,214,567	42,389,484	48,564,400	54,739,317	60,914,234	67,089,151	73,264,068	79,438,984	85,613,901	91,788,818		104,138,652						141,188,152	
1,900	1,520,588	2,495,092	1.86 2,64		4,161,055	1.7	85	126,938	6,029,560	10,155,048	16,184,607	89,984,106		836	23,625,490	29,813,722	36,001,954	42,190,186	48,378,419	54,566,651	60,754,883	66,943,115	73,131,347	79,319,579	85,507,811	91,696,044		104,072,508						141,201,901	
1,950	1,532,024	2,511,109	1.86 2,66		4,200,578	1.7	85	127,318	6,206,739	10,185,418	16,392,157	90,663,170	107,055,326	841	23,286,191	29,492,930	35,699,669	41,906,407	48,113,146	54,319,885	60,526,624	66,733,363	72,940,102	79,146,841	85,353,580	91,560,319	97,767,058	103,973,796	110,180,535	116,387,274				141,214,230	147,420,969
2,000	1,552,187	2,543,911	1.84 2,72	1,106	4,273,293	1.8	85	128,017	6,400,867	10,241,388	16,642,255	91,951,443	108,593,697	848	22,624,080	28,864,926	35,105,771	41,346,617	47,587,462	53,828,308	60,069,154	66,309,999	72,550,845	78,791,690	85,032,536	91,273,381	97,514,227	103,755,072	109,995,918	116,236,763	122,477,609	128,718,454	134,959,300	141,200,145	147,440,991

APPENDIX 6 LOST DOG

OPTIMUM MINING TONNAGE SENSITIVITIES TABULAR

\$1,000/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	383,770	383,770
380.0 -> 377.5	88	211	2.17	366,214	366,302
377.5 -> 375.0	33,002	78,611	1.75	311,025	344,027
375.0 -> 372.5	95,183	222,678	1.79	230,579	325,762
372.5 -> 370.0	148,697	315,972	2.10	148,768	297,465
370.0 -> 367.5	167,168	257,095	2.55	86,287	253,455
367.5 -> 365.0	177,100	248,411	2.74	29,126	206,226
365.0 -> 362.5	144,891	173,614	3.55	12,247	157,138
362.5 -> 360.0	70,223	82,941	2.79	5,718	75,941
360.0 -> 357.5	8,798	9,939	2.00	1,147	9,945
357.5 -> 355.0	397	437	1.84	80	477
TOTAL	845,548	1,389,907	2.45	1,574,960	2,420,508

UNIT COSTS

		MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE &
FLITCH	ł			EXTRAS		CONTROL	TREATMENT
		(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 3	380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 3	377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 3	375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 3	372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 3	370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 3	367.5	4.50	2.00	1.00	0.20	0.60	25.00
367.5 -> 3	365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 3	362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 3	360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 3	357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 3	355.0	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

FLITCH	MINING	BLASTING	MINING EXTRAS	REHAB	GRADE CONTROL	ORE HAULAGE & TREATMENT	OPERATING COSTS
FLITCH	(6)	(6)		(e)		(\$/t ore)	
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t tire)	(\$)
N/S -> 380.0	1,343,195	0	383,770	76,754	0	0	1,803,719
380.0 -> 377.5	1,282,057	0	366,302	73,243	127	5,273	1,727,001
377.5 -> 375.0	1,204,095	0	344,027	62,205	47,166	1,965,264	3,622,756
375.0 -> 372.5	1,303,048	651,524	325,762	46,116	133,607	5,566,947	8,027,004
372.5 -> 370.0	1,189,860	594,930	297,465	29,754	189,583	7,899,289	10,200,880
370.0 -> 367.5	1,140,548	506,910	253,455	17,257	154,257	6,427,378	8,499,805
367.5 -> 365.0	928,017	412,452	206,226	5,825	149,046	6,210,265	7,911,831
365.0 -> 362.5	785,690	314,276	157,138	2,449	104,168	4,340,338	5,704,060
362.5 -> 360.0	379,705	151,882	75,941	1,144	49,764	2,073,520	2,731,956
360.0 -> 357.5	49,725	19,890	9,945	229	5,963	248,476	334,228
357.5 -> 355.0	2,385	954	477	16	262	10,921	15,015
TOTAL	9,608,324	2,652,818	2,420,508	314,992	833,944	34,747,670	50,578,257

		MILLED ODE		OUNCE	DEVENUE	OTATE COLD	ADD DADTY	TOTAL	ODEDATING	TOTAL	OUMUL ATIME	OACHOOOT
l L		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,000/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	1,803,719	1,803,719	-1,803,719	-
380.0 -> 377.5	88	211	2.17	13	12,534	313	1,003	1,316	1,727,001	1,728,317	-3,519,503	137,893
377.5 -> 375.0	33,002	78,611	1.75	3,749	3,748,958	93,724	299,917	393,641	3,622,756	4,016,397	-3,786,941	1,071
375.0 -> 372.5	95,183	222,678	1.79	10,900	10,899,974	272,499	871,998	1,144,497	8,027,004	9,171,501	-2,058,468	841
372.5 -> 370.0	148,697	315,972	2.10	18,167	18,167,180	454,180	1,453,374	1,907,554	10,200,880	12,108,434	4,000,277	667
370.0 -> 367.5	167,168	257,095	2.55	17,909	17,909,233	447,731	1,432,739	1,880,470	8,499,805	10,380,274	11,529,236	580
367.5 -> 365.0	177,100	248,411	2.74	18,602	18,602,090	465,052	1,488,167	1,953,219	7,911,831	9,865,051	20,266,276	
365.0 -> 362.5	144,891	173,614	3.55	16,866	16,866,352	421,659	1,349,308	1,770,967	5,704,060	7,475,027	29,657,601	443
362.5 -> 360.0	70,223	82,941	2.79	6,329	6,328,752	158,219	506,300	664,519	2,731,956	3,396,475	32,589,878	
360.0 -> 357.5	8,798	9,939	2.00	544	543,762	13,594	43,501	57,095	334,228	391,323	32,742,317	720
357.5 -> 355.0	397	437	1.84	22	21,991	550	1,759	2,309	15,015	17,324	32,746,984	788
TOTAL	845.548	1.389.907	2.45	93,101	93,100,828	2.327.521	7,448,066	9,775,587	50.578.257	60.353.843	32,746,984	648

\$1,050/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	403,665	403,665
380.0 -> 377.5	88	211	2.28	387,138	387,226
377.5 -> 375.0	36,847	87,834	1.68	325,546	362,393
375.0 -> 372.5	104,600	244,854	1.73	238,982	343,582
372.5 -> 370.0	162,688	346,277	2.03	154,814	317,502
370.0 -> 367.5	180,260	277,104	2.46	88,492	268,752
367.5 -> 365.0	189,760	266,326	2.65	29,600	219,360
365.0 -> 362.5	152,719	183,033	3.46	13,183	165,902
362.5 -> 360.0	73,578	86,937	2.75	6,188	79,766
360.0 -> 357.5	9,627	10,887	1.95	1,267	10,894
357.5 -> 355.0	397	437	1.84	80	477
355.0 -> 352.5	19	21	1.30	0	19
TOTAL	910,583	1,503,921	2.37	1,648,955	2,559,538

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,412,828	0	403,665	80,733	0	0	1,897,226
380.0 -> 377.5	1,355,291	0	387,226	77,428	127	5,273	1,825,344
377.5 -> 375.0	1,268,376	0	362,393	65,109	52,700	2,195,846	3,944,424
375.0 -> 372.5	1,374,328	687,164	343,582	47,796	146,912	6,121,350	8,721,133
372.5 -> 370.0	1,270,008	635,004	317,502	30,963	207,766	8,656,936	11,118,179
370.0 -> 367.5	1,209,384	537,504	268,752	17,698	166,262	6,927,603	9,127,204
367.5 -> 365.0	987,120	438,720	219,360	5,920	159,796	6,658,161	8,469,077
365.0 -> 362.5	829,510	331,804	165,902	2,637	109,820	4,575,819	6,015,491
362.5 -> 360.0	398,830	159,532	79,766	1,238	52,162	2,173,430	2,864,958
360.0 -> 357.5	54,470	21,788	10,894	253	6,532	272,166	366,103
357.5 -> 355.0	2,385	954	477	16	262	10,921	15,015
355.0 -> 352.5	95	38	19	0	13	525	689
TOTAL	10,162,624	2,812,508	2,559,538	329,791	902,353	37,598,030	54,364,844

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,050/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	1,897,226	1,897,226	-1,897,226	-
380.0 -> 377.5	88	211	2.28	13	13,777	344	1,050	1,394	1,825,344	1,826,738	-3,710,186	139,218
377.5 -> 375.0	36,847	87,834	1.68	4,024	4,225,296	105,632	321,927	427,560	3,944,424	4,371,984	-3,856,874	1,086
375.0 -> 372.5	104,600	244,854	1.73	11,572	12,150,802	303,770	925,775	1,229,545	8,721,133	9,950,679	-1,656,751	860
372.5 -> 370.0	162,688	346,277	2.03	19,242	20,203,753	505,094	1,539,334	2,044,427	11,118,179	13,162,607	5,384,396	684
370.0 -> 367.5	180,260	277,104	2.46	18,635	19,566,620	489,165	1,490,790	1,979,956	9,127,204	11,107,159	13,843,856	596
367.5 -> 365.0	189,760	266,326	2.65	19,287	20,251,604	506,290	1,542,979	2,049,269	8,469,077	10,518,346	23,577,114	545
365.0 -> 362.5	152,719	183,033	3.46	17,316	18,181,330	454,533	1,385,244	1,839,777	6,015,491	7,855,269	33,903,175	454
362.5 -> 360.0	73,578	86,937	2.75	6,531	6,857,769	171,444	522,497	693,941	2,864,958	3,558,899	37,202,045	545
360.0 -> 357.5	9,627	10,887	1.95	580	608,849	15,221	46,388	61,610	366,103	427,713	37,383,180	738
357.5 -> 355.0	397	437	1.84	22	23,091	577	1,759	2,337	15,015	17,351	37,388,920	789
355.0 -> 352.5	19	21	1.30	1	785	20	60	79	689	769	37,388,936	1,028
TOTAL	910,583	1,503,921	2.37	97,223	102,083,676	2,552,092	7,777,804	10,329,896	54,364,844	64,694,739	37,388,936	665

\$1,100/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	426,607	426,607
380.0 -> 377.5	88	211	2.17	404,870	404,958
377.5 -> 375.0	40,432	96,437	1.62	339,083	379,515
375.0 -> 372.5	119,482	280,124	1.65	242,759	362,241
372.5 -> 370.0	177,369	378,480	1.96	155,286	332,655
370.0 -> 367.5	189,063	292,379	2.40	91,856	280,919
367.5 -> 365.0	200,725	282,039	2.57	30,765	231,490
365.0 -> 362.5	160,195	192,064	3.38	14,280	174,475
362.5 -> 360.0	76,509	90,432	2.70	6,706	83,215
360.0 -> 357.5	10,603	11,973	1.89	1,510	12,113
357.5 -> 355.0	436	480	1.81	99	535
355.0 -> 352.5	33	36	1.30	5	38
TOTAL	974,935	1,624,655	2.28	1,713,826	2,688,761

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,493,125	0	426,607	85,321	0	0	2,005,053
380.0 -> 377.5	1,417,353	0	404,958	80,974	127	5,273	1,908,684
377.5 -> 375.0	1,328,303	0	379,515	67,817	57,862	2,410,935	4,244,432
375.0 -> 372.5	1,448,964	724,482	362,241	48,552	168,074	7,003,098	9,755,411
372.5 -> 370.0	1,330,620	665,310	332,655	31,057	227,088	9,461,989	12,048,719
370.0 -> 367.5	1,264,136	561,838	280,919	18,371	175,427	7,309,475	9,610,166
367.5 -> 365.0	1,041,705	462,980	231,490	6,153	169,223	7,050,978	8,962,530
365.0 -> 362.5	872,375	348,950	174,475	2,856	115,238	4,801,604	6,315,498
362.5 -> 360.0	416,075	166,430	83,215	1,341	54,259	2,260,795	2,982,116
360.0 -> 357.5	60,565	24,226	12,113	302	7,184	299,330	403,720
357.5 -> 355.0	2,675	1,070	535	20	288	11,995	16,583
355.0 -> 352.5	190	76	38	1	22	900	1,226
TOTAL	10,676,085	2,955,362	2,688,761	342,765	974,793	40,616,372	58,254,138

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,100/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,005,053	2,005,053	-2,005,053	-
380.0 -> 377.5	88	211	2.17	13	13,787	345	1,003	1,347	1,908,684	1,910,032	-3,901,298	152,391
377.5 -> 375.0	40,432	96,437	1.62	4,274	4,700,927	117,523	341,886	459,409	4,244,432	4,703,841	-3,904,211	1,101
375.0 -> 372.5	119,482	280,124	1.65	12,631	13,894,304	347,358	1,010,495	1,357,852	9,755,411	11,113,263	-1,123,170	880
372.5 -> 370.0	177,369	378,480	1.96	20,260	22,286,424	557,161	1,620,831	2,177,991	12,048,719	14,226,710	6,936,544	702
370.0 -> 367.5	189,063	292,379	2.40	19,161	21,076,822	526,921	1,532,860	2,059,780	9,610,166	11,669,946	16,343,420	609
367.5 -> 365.0	200,725	282,039	2.57	19,843	21,827,631	545,691	1,587,464	2,133,155	8,962,530	11,095,685	27,075,366	559
365.0 -> 362.5	160,195	192,064	3.38	17,748	19,522,795	488,070	1,419,840	1,907,909	6,315,498	8,223,408	38,374,753	463
362.5 -> 360.0	76,509	90,432	2.70	6,680	7,347,854	183,696	534,389	718,086	2,982,116	3,700,201	42,022,405	554
360.0 -> 357.5	10,603	11,973	1.89	620	681,742	17,044	49,581	66,625	403,720	470,345	42,233,802	759
357.5 -> 355.0	436	480	1.81	24	26,047	651	1,894	2,546	16,583	19,128	42,240,721	808
355.0 -> 352.5	33	36	1.30	1	1,411	35	103	138	1,226	1,364	42,240,768	1,064
TOTAL	974,935	1,624,655	2.28	101,254	111,379,744	2,784,494	8,100,345	10,884,839	58,254,138	69,138,976	42,240,768	683

\$1,150/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	466,487	466,487
380.0 -> 377.5	88	211	2.17	439,046	439,134
377.5 -> 375.0	44,543	105,988	1.57	368,777	413,320
375.0 -> 372.5	131,867	308,630	1.59	264,386	396,253
372.5 -> 370.0	191,201	408,131	1.90	173,531	364,732
370.0 -> 367.5	201,640	313,579	2.32	106,263	307,903
367.5 -> 365.0	215,146	301,795	2.50	40,434	255,580
365.0 -> 362.5	175,992	210,585	3.25	17,950	193,942
362.5 -> 360.0	86,084	101,351	2.63	8,107	94,191
360.0 -> 357.5	11,801	13,333	1.84	1,639	13,440
357.5 -> 355.0	647	712	1.60	99	746
355.0 -> 352.5	61	67	1.28	14	75
TOTAL	1,059,070	1,764,381	2.21	1,886,733	2,945,803

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,632,705	0	466,487	93,297	0	0	2,192,489
380.0 -> 377.5	1,536,969	0	439,134	87,809	127	5,273	2,069,312
377.5 -> 375.0	1,446,620	0	413,320	73,755	63,593	2,649,690	4,646,978
375.0 -> 372.5	1,585,012	792,506	396,253	52,877	185,178	7,715,762	10,727,589
372.5 -> 370.0	1,458,928	729,464	364,732	34,706	244,878	10,203,267	13,035,976
370.0 -> 367.5	1,385,564	615,806	307,903	21,253	188,147	7,839,463	10,358,135
367.5 -> 365.0	1,150,110	511,160	255,580	8,087	181,077	7,544,881	9,650,895
365.0 -> 362.5	969,710	387,884	193,942	3,590	126,351	5,264,618	6,946,095
362.5 -> 360.0	470,955	188,382	94,191	1,621	60,811	2,533,786	3,349,746
360.0 -> 357.5	67,200	26,880	13,440	328	8,000	333,317	449,164
357.5 -> 355.0	3,730	1,492	746	20	427	17,793	24,208
355.0 -> 352.5	375	150	75	3	40	1,674	2,317
TOTAL	11,707,877	3,253,724	2,945,803	377,347	1,058,629	44,109,524	63,452,903

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,150/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,192,489	2,192,489	-2,192,489	-
380.0 -> 377.5	88	211	2.17	13	14,414	360	1,003	1,363	2,069,312	2,070,675	-4,248,750	165,208
377.5 -> 375.0	44,543	105,988	1.57	4,541	5,221,690	130,542	363,248	493,790	4,646,978	5,140,768	-4,167,827	1,132
375.0 -> 372.5	131,867	308,630	1.59	13,445	15,462,049	386,551	1,075,621	1,462,172	10,727,589	12,189,761	-895,539	907
372.5 -> 370.0	191,201	408,131	1.90	21,224	24,408,013	610,200	1,697,949	2,308,149	13,035,976	15,344,125	8,168,349	723
370.0 -> 367.5	201,640	313,579	2.32	19,886	22,869,265	571,732	1,590,905	2,162,637	10,358,135	12,520,772	18,516,843	630
367.5 -> 365.0	215,146	301,795	2.50	20,611	23,702,266	592,557	1,648,853	2,241,410	9,650,895	11,892,305	30,326,804	577
365.0 -> 362.5	175,992	210,585	3.25	18,686	21,489,428	537,236	1,494,917	2,032,152	6,946,095	8,978,248	42,837,984	480
362.5 -> 360.0	86,084	101,351	2.63	7,275	8,365,854	209,146	581,972	791,119	3,349,746	4,140,865	47,062,974	569
360.0 -> 357.5	11,801	13,333	1.84	670	770,240	19,256	53,582	72,838	449,164	522,002	47,311,211	779
357.5 -> 355.0	647	712	1.60	31	35,832	896	2,493	3,388	24,208	27,596	47,319,447	886
355.0 -> 352.5	61	67	1.28	2	2,687	67	187	254	2,317	2,571	47,319,562	1,101
TOTAL	1,059,070	1,764,381	2.21	106,384	122,341,738	3,058,543	8,510,730	11,569,273	63,452,903	75,022,176	47,319,562	705

\$1,200/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	482,449	482,449
380.0 -> 377.5	88	211	2.17	451,642	451,730
377.5 -> 375.0	47,447	112,744	1.53	379,090	426,537
375.0 -> 372.5	142,899	334,090	1.55	266,899	409,798
372.5 -> 370.0	199,094	425,003	1.87	176,277	375,371
370.0 -> 367.5	208,120	324,261	2.28	108,945	317,065
367.5 -> 365.0	225,291	315,994	2.44	40,433	265,724
365.0 -> 362.5	182,627	218,607	3.18	19,174	201,801
362.5 -> 360.0	89,436	105,315	2.58	8,823	98,259
360.0 -> 357.5	12,670	14,303	1.80	1,873	14,543
357.5 -> 355.0	652	717	1.60	113	765
355.0 -> 352.5	61	67	1.28	14	75
TOTAL	1,108,384	1,851,311	2.16	1,935,733	3,044,117

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50		1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,688,572	0	482,449	96,490	0	0	2,267,510
380.0 -> 377.5	1,581,055	0	451,730	90,328	127	5,273	2,128,513
377.5 -> 375.0	1,492,880	0	426,537	75,818	67,646	2,818,597	4,881,478
375.0 -> 372.5	1,639,192	819,596	409,798	53,380	200,454	8,352,258	11,474,678
372.5 -> 370.0	1,501,484	750,742	375,371	35,255	255,002	10,625,073	13,542,927
370.0 -> 367.5	1,426,793	634,130	317,065	21,789	194,557	8,106,531	10,700,864
367.5 -> 365.0	1,195,758	531,448	265,724	8,087	189,596	7,899,839	10,090,452
365.0 -> 362.5	1,009,005	403,602	201,801	3,835	131,164	5,465,163	7,214,570
362.5 -> 360.0	491,295	196,518	98,259	1,765	63,189	2,632,871	3,483,897
360.0 -> 357.5	72,715	29,086	14,543	375	8,582	357,582	482,882
357.5 -> 355.0	3,825	1,530	765	23	430	17,918	24,491
355.0 -> 352.5	375	150	75	3	40	1,674	2,317
TOTAL	12,102,948	3,366,802	3,044,117	387,147	1,110,787	46,282,779	66,294,579

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,200/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,267,510	2,267,510	-2,267,510	
380.0 -> 377.5	88	211	2.17	13	15,040	376	1,003	1,379	2,128,513	2,129,892	-4,382,361	169,933
377.5 -> 375.0	47,447	112,744	1.53	4,712	5,654,688	141,367	376,979	518,346	4,881,478	5,399,824	-4,127,498	1,146
375.0 -> 372.5	142,899	334,090	1.55	14,152	16,981,921	424,548	1,132,128	1,556,676	11,474,678	13,031,354	-176,931	921
372.5 -> 370.0	199,094	425,003	1.87	21,703	26,043,892	651,097	1,736,259	2,387,357	13,542,927	15,930,284	9,936,677	734
370.0 -> 367.5	208,120	324,261	2.28	20,208	24,249,111	606,228	1,616,607	2,222,835	10,700,864	12,923,700	21,262,088	640
367.5 -> 365.0	225,291	315,994	2.44	21,072	25,286,818	632,170	1,685,788	2,317,958	10,090,452	12,408,410	34,140,496	589
365.0 -> 362.5	182,627	218,607	3.18	19,012	22,814,066	570,352	1,520,938	2,091,289	7,214,570	9,305,859	47,648,702	489
362.5 -> 360.0	89,436	105,315	2.58	7,429	8,915,210	222,880	594,347	817,228	3,483,897	4,301,124	52,262,787	579
360.0 -> 357.5	12,670	14,303	1.80	704	844,765	21,119	56,318	77,437	482,882	560,319	52,547,233	796
357.5 -> 355.0	652	717	1.60	31	37,721	943	2,515	3,458	24,491	27,948	52,557,006	889
355.0 -> 352.5	61	67	1.28	2	2,804	70	187	257	2,317	2,574	52,557,235	1,102
TOTAL	1,108,384	1,851,311	2.16	109,038	130,846,034	3,271,151	8,723,069	11,994,220	66,294,579	78,288,799	52,557,235	718

\$1,250/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	498,504	498,504
380.0 -> 377.5	88	211	2.17	464,963	465,051
377.5 -> 375.0	50,134	119,097	1.50	390,559	440,693
375.0 -> 372.5	151,532	353,577	1.52	272,795	424,327
372.5 -> 370.0	207,412	443,030	1.84	180,210	387,622
370.0 -> 367.5	214,803	335,460	2.24	112,240	327,043
367.5 -> 365.0	233,556	327,366	2.40	42,203	275,759
365.0 -> 362.5	189,393	226,708	3.12	20,194	209,587
362.5 -> 360.0	93,094	109,669	2.53	9,649	102,743
360.0 -> 357.5	14,145	15,984	1.74	2,173	16,318
357.5 -> 355.0	741	815	1.57	179	920
355.0 -> 352.5	80	88	1.28	33	113
TOTAL	1,154,978	1,932,004	2.11	1,993,702	3,148,680

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,744,764	0	498,504	99,701	0	0	2,342,969
380.0 -> 377.5	1,627,679	0	465,051	92,993	127	5,273	2,191,122
377.5 -> 375.0	1,542,426	0	440,693	78,112	71,458	2,977,434	5,110,122
375.0 -> 372.5	1,697,308	848,654	424,327	54,559	212,146	8,839,413	12,076,407
372.5 -> 370.0	1,550,488	775,244	387,622	36,042	265,818	11,075,743	14,090,957
370.0 -> 367.5	1,471,694	654,086	327,043	22,448	201,276	8,386,494	11,063,040
367.5 -> 365.0	1,240,916	551,518	275,759	8,441	196,420	8,184,150	10,457,203
365.0 -> 362.5	1,047,935	419,174	209,587	4,039	136,025	5,667,707	7,484,467
362.5 -> 360.0	513,715	205,486	102,743	1,930	65,801	2,741,728	3,631,403
360.0 -> 357.5	81,590	32,636	16,318	435	9,590	399,590	540,159
357.5 -> 355.0	4,600	1,840	920	36	489	20,367	28,252
355.0 -> 352.5	565	226	113	7	53	2,199	3,163
TOTAL	12,523,678	3,488,864	3,148,680	398,740	1,159,202	48,300,097	69,019,262

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,250/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,342,969	2,342,969	-2,342,969	-
380.0 -> 377.5	88	211	2.17	13	15,667	392	1,003	1,394	2,191,122	2,192,516	-4,519,818	174,929
377.5 -> 375.0	50,134	119,097	1.50	4,872	6,090,612	152,265	389,799	542,064	5,110,122	5,652,187	-4,081,392	1,160
375.0 -> 372.5	151,532	353,577	1.52	14,693	18,366,019	459,150	1,175,425	1,634,576	12,076,407	13,710,982	573,644	933
372.5 -> 370.0	207,412	443,030	1.84	22,220	27,775,285	694,382	1,777,618	2,472,000	14,090,957	16,562,957	11,785,973	745
370.0 -> 367.5	214,803	335,460	2.24	20,537	25,671,217	641,780	1,642,958	2,284,738	11,063,040	13,347,779	24,109,411	650
367.5 -> 365.0	233,556	327,366	2.40	21,462	26,827,943	670,699	1,716,988	2,387,687	10,457,203	12,844,890	38,092,464	598
365.0 -> 362.5	189,393	226,708	3.12	19,303	24,129,099	603,227	1,544,262	2,147,490	7,484,467	9,631,957	52,589,607	499
362.5 -> 360.0	93,094	109,669	2.53	7,593	9,490,666	237,267	607,403	844,669	3,631,403	4,476,072	57,604,200	590
360.0 -> 357.5	14,145	15,984	1.74	761	951,222	23,781	60,878	84,659	540,159	624,818	57,930,604	821
357.5 -> 355.0	741	815	1.57	35	43,599	1,090	2,790	3,880	28,252	32,132	57,942,072	921
355.0 -> 352.5	80	88	1.28	3	3,856	96	247	343	3,163	3,506	57,942,422	1,136
TOTAL	1,154,978	1,932,004	2.11	111,492	139,365,186	3,484,130	8,919,372	12,403,502	69,019,262	81,422,764	57,942,422	730

\$1,300/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	518,748	518,748
380.0 -> 377.5	88	211	2.17	485,174	485,262
377.5 -> 375.0	51,937	123,353	1.48	409,207	461,144
375.0 -> 372.5	158,532	368,904	1.50	286,522	445,054
372.5 -> 370.0	215,688	460,380	1.81	191,245	406,933
370.0 -> 367.5	224,918	350,238	2.20	118,171	343,089
367.5 -> 365.0	244,625	341,799	2.35	44,718	289,343
365.0 -> 362.5	200,091	239,269	3.03	21,250	221,341
362.5 -> 360.0	97,445	114,770	2.48	10,861	108,306
360.0 -> 357.5	15,503	17,509	1.71	2,517	18,020
357.5 -> 355.0	746	820	1.57	193	939
355.0 -> 352.5	80	88	1.28	33	113
TOTAL	1,209,652	2,017,341	2.07	2,088,640	3,298,292

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50	2.00	1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,815,618	0	518,748	103,750	0	0	2,438,116
380.0 -> 377.5	1,698,417	0	485,262	97,035	127	5,273	2,286,113
377.5 -> 375.0	1,614,004	0	461,144	81,841	74,012	3,083,816	5,314,817
375.0 -> 372.5	1,780,216	890,108	445,054	57,304	221,343	9,222,609	12,616,635
372.5 -> 370.0	1,627,732	813,866	406,933	38,249	276,228	11,509,494	14,672,502
370.0 -> 367.5	1,543,901	686,178	343,089	23,634	210,143	8,755,946	11,562,891
367.5 -> 365.0	1,302,044	578,686	289,343	8,944	205,080	8,544,981	10,929,076
365.0 -> 362.5	1,106,705	442,682	221,341	4,250	143,562	5,981,731	7,900,271
362.5 -> 360.0	541,530	216,612	108,306	2,172	68,862	2,869,252	3,806,734
360.0 -> 357.5	90,100	36,040	18,020	503	10,505	437,725	592,894
357.5 -> 355.0	4,695	1,878	939	39	492	20,492	28,534
355.0 -> 352.5	565	226	113	7	53	2,199	3,163
TOTAL	13,125,526	3,666,276	3,298,292	417,728	1,210,404	50,433,519	72,151,745

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,300/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,438,116	2,438,116	-2,438,116	
380.0 -> 377.5	88	211	2.17	13	16,294	407	1,003	1,410	2,286,113	2,287,523	-4,709,345	182,509
377.5 -> 375.0	51,937	123,353	1.48	4,977	6,470,332	161,758	398,174	559,933	5,314,817	5,874,750	-4,113,762	1,180
375.0 -> 372.5	158,532	368,904	1.50	15,102	19,633,166	490,829	1,208,195	1,699,024	12,616,635	14,315,659	1,203,745	948
372.5 -> 370.0	215,688	460,380	1.81	22,720	29,536,500	738,412	1,817,631	2,556,043	14,672,502	17,228,546	13,511,699	758
370.0 -> 367.5	224,918	350,238	2.20	21,010	27,313,059	682,826	1,680,804	2,363,630	11,562,891	13,926,521	26,898,237	663
367.5 -> 365.0	244,625	341,799	2.35	21,932	28,512,136	712,803	1,754,593	2,467,396	10,929,076	13,396,473	42,013,900	611
365.0 -> 362.5	200,091	239,269	3.03	19,789	25,726,262	643,157	1,583,155	2,226,311	7,900,271	10,126,582	57,613,581	512
362.5 -> 360.0	97,445	114,770	2.48	7,777	10,109,523	252,738	622,124	874,863	3,806,734	4,681,597	63,041,507	602
360.0 -> 357.5	15,503	17,509	1.71	818	1,062,947	26,574	65,412	91,986	592,894	684,880	63,419,574	838
357.5 -> 355.0	746	820	1.57	35	45,650	1,141	2,809	3,950	28,534	32,485	63,432,740	925
355.0 -> 352.5	80	88	1.28	3	4,011	100	247	347	3,163	3,510	63,433,241	1,138
TOTAL	1,209,652	2,017,341	2.07	114,177	148,429,879	3,710,747	9,134,146	12,844,893	72,151,745	84,996,638	63,433,241	744

\$1,350/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	529,818	529,818
380.0 -> 377.5	88	211	2.17	496,292	496,380
377.5 -> 375.0	52,432	124,540	1.47	420,549	472,981
375.0 -> 372.5	164,150	381,817	1.48	292,056	456,206
372.5 -> 370.0	223,061	476,286	1.78	194,530	417,591
370.0 -> 367.5	233,440	363,604	2.15	119,000	352,440
367.5 -> 365.0	250,982	350,403	2.32	47,229	298,211
365.0 -> 362.5	206,461	246,896	2.97	22,482	228,943
362.5 -> 360.0	100,767	118,726	2.44	11,438	112,205
360.0 -> 357.5	16,488	18,624	1.68	2,790	19,278
357.5 -> 355.0	794	873	1.56	241	1,035
355.0 -> 352.5	99	109	1.27	52	151
TOTAL	1,248,761	2,082,089	2.04	2,136,478	3,385,239

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,854,363	0	529,818	105,964	0	0	2,490,145
380.0 -> 377.5	1,737,330	0	496,380	99,258	127	5,273	2,338,368
377.5 -> 375.0	1,655,434	0	472,981	84,110	74,724	3,113,504	5,400,752
375.0 -> 372.5	1,824,824	912,412	456,206	58,411	229,090	9,545,430	13,026,374
372.5 -> 370.0	1,670,364	835,182	417,591	38,906	285,772	11,907,160	15,154,975
370.0 -> 367.5	1,585,980	704,880	352,440	23,800	218,162	9,090,088	11,975,350
367.5 -> 365.0	1,341,950	596,422	298,211	9,446	210,242	8,760,070	11,216,340
365.0 -> 362.5	1,144,715	457,886	228,943	4,496	148,138	6,172,405	8,156,583
362.5 -> 360.0	561,025	224,410	112,205	2,288	71,236	2,968,162	3,939,326
360.0 -> 357.5	96,390	38,556	19,278	558	11,174	465,589	631,545
357.5 -> 355.0	5,175	2,070	1,035	48	524	21,816	30,668
355.0 -> 352.5	755	302	151	10	65	2,724	4,008
TOTAL	13,478,304	3,772,120	3,385,239	427,296	1,249,253	52,052,221	74,364,433

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,350/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,490,145	2,490,145	-2,490,145	
380.0 -> 377.5	88	211	2.17	13	16,921	423	1,003	1,426	2,338,368	2,339,794	-4,813,018	186,680
377.5 -> 375.0	52,432	124,540	1.47	5,005	6,756,850	168,921	400,406	569,327	5,400,752	5,970,080	-4,026,247	1,193
375.0 -> 372.5	164,150	381,817	1.48	15,447	20,853,360	521,334	1,235,755	1,757,089	13,026,374	14,783,462	2,043,650	957
372.5 -> 370.0	223,061	476,286	1.78	23,135	31,232,703	780,818	1,850,827	2,631,644	15,154,975	17,786,620	15,489,733	769
370.0 -> 367.5	233,440	363,604	2.15	21,344	28,814,694	720,367	1,707,537	2,427,905	11,975,350	14,403,254	29,901,173	675
367.5 -> 365.0	250,982	350,403	2.32	22,203	29,973,832	749,346	1,776,227	2,525,573	11,216,340	13,741,913	46,133,093	619
365.0 -> 362.5	206,461	246,896	2.97	20,056	27,076,167	676,904	1,604,514	2,281,418	8,156,583	10,438,001	62,771,259	520
362.5 -> 360.0	100,767	118,726	2.44	7,911	10,679,895	266,997	632,883	899,880	3,939,326	4,839,206	68,611,949	612
360.0 -> 357.5	16,488	18,624	1.68	854	1,152,540	28,813	68,299	97,112	631,545	728,657	69,035,832	853
357.5 -> 355.0	794	873	1.56	37	50,217	1,255	2,976	4,231	30,668	34,899	69,051,150	938
355.0 -> 352.5	99	109	1.27	4	5,115	128	303	431	4,008	4,439	69,051,826	1,171
TOTAL	1,248,761	2,082,089	2.04	116,009	156,612,295	3,915,307	9,280,729	13,196,036	74,364,433	87,560,469	69,051,826	755

\$1,400/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	539,622	539,622
380.0 -> 377.5	102	246	1.99	508,132	508,234
377.5 -> 375.0	52,642	125,044	1.47	432,096	484,738
375.0 -> 372.5	168,904	392,895	1.46	299,084	467,988
372.5 -> 370.0	230,177	492,315	1.75	201,030	431,207
370.0 -> 367.5	240,144	373,425	2.12	123,399	363,543
367.5 -> 365.0	257,616	359,284	2.29	50,434	308,050
365.0 -> 362.5	213,723	255,427	2.92	23,294	237,017
362.5 -> 360.0	104,460	123,067	2.39	12,605	117,065
360.0 -> 357.5	18,193	20,548	1.63	2,970	21,163
357.5 -> 355.0	853	939	1.54	279	1,132
355.0 -> 352.5	99	109	1.27	52	151
TOTAL	1,286,913	2,143,297	2.01	2,192,997	3,479,910

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

FLITOLI	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,888,677	0	539,622	107,924	0	0	2,536,223
380.0 -> 377.5	1,778,819	0	508,234	101,626	148	6,148	2,394,974
377.5 -> 375.0	1,696,583	0	484,738	86,419	75,026	3,126,099	5,468,866
375.0 -> 372.5	1,871,952	935,976	467,988	59,817	235,737	9,822,369	13,393,839
372.5 -> 370.0	1,724,828	862,414	431,207	40,206	295,389	12,307,875	15,661,919
370.0 -> 367.5	1,635,944	727,086	363,543	24,680	224,055	9,335,614	12,310,921
367.5 -> 365.0	1,386,225	616,100	308,050	10,087	215,571	8,982,106	11,518,138
365.0 -> 362.5	1,185,085	474,034	237,017	4,659	153,256	6,385,670	8,439,720
362.5 -> 360.0	585,325	234,130	117,065	2,521	73,840	3,076,669	4,089,550
360.0 -> 357.5	105,815	42,326	21,163	594	12,329	513,694	695,921
357.5 -> 355.0	5,660	2,264	1,132	56	563	23,466	33,141
355.0 -> 352.5	755	302	151	10	65	2,724	4,008
TOTAL	13,865,668	3,894,632	3,479,910	438,599	1,285,978	53,582,433	76,547,221

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,400/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,536,223	2,536,223	-2,536,223	-
380.0 -> 377.5	102	246	1.99	13	18,724	468	1,070	1,538	2,394,974	2,396,513	-4,914,012	179,189
377.5 -> 375.0	52,642	125,044	1.47	5,015	7,021,378	175,534	401,222	576,756	5,468,866	6,045,622	-3,938,255	1,205
375.0 -> 372.5	168,904	392,895	1.46	15,716	22,002,595	550,065	1,257,291	1,807,356	13,393,839	15,201,195	2,863,145	967
372.5 -> 370.0	230,177	492,315	1.75	23,531	32,943,957	823,599	1,882,512	2,706,111	15,661,919	18,368,030	17,439,072	781
370.0 -> 367.5	240,144	373,425	2.12	21,601	30,240,807	756,020	1,728,046	2,484,066	12,310,921	14,794,988	32,884,892	685
367.5 -> 365.0	257,616	359,284	2.29	22,496	31,494,500	787,362	1,799,686	2,587,048	11,518,138	14,105,186	50,274,206	627
365.0 -> 362.5	213,723	255,427	2.92	20,359	28,503,045	712,576	1,628,745	2,341,322	8,439,720	10,781,042	67,996,209	530
362.5 -> 360.0	104,460	123,067	2.39	8,052	11,272,604	281,815	644,149	925,964	4,089,550	5,015,514	74,253,299	623
360.0 -> 357.5	18,193	20,548	1.63	916	1,281,725	32,043	73,241	105,285	695,921	801,206	74,733,819	875
357.5 -> 355.0	853	939	1.54	40	55,381	1,385	3,165	4,549	33,141	37,690	74,751,509	953
355.0 -> 352.5	99	109	1.27	4	5,305	133	303	436	4,008	4,443	74,752,371	1,173
TOTAL	1,286,913	2,143,297	2.01	117,743	164,840,022	4,121,001	9,419,430	13,540,430	76,547,221	90,087,651	74,752,371	765

\$1,450/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	556,083	556,083
380.0 -> 377.5	220	527	1.39	523,387	523,607
377.5 -> 375.0	53,783	127,754	1.45	446,944	500,727
375.0 -> 372.5	172,588	401,213	1.45	311,300	483,888
372.5 -> 370.0	237,034	506,878	1.73	208,536	445,570
370.0 -> 367.5	248,015	384,864	2.08	127,852	375,867
367.5 -> 365.0	266,823	371,919	2.25	53,089	319,912
365.0 -> 362.5	221,216	264,269	2.86	23,134	244,350
362.5 -> 360.0	107,477	126,600	2.36	13,075	120,552
360.0 -> 357.5	19,577	22,110	1.59	3,219	22,796
357.5 -> 355.0	949	1,044	1.52	354	1,303
355.0 -> 352.5	99	109	1.27	52	151
TOTAL	1,327,782	2,207,288	1.98	2,267,024	3,594,806

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50	2.00	1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

FLITCH	MINING	BLASTING	MINING EXTRAS	REHAB	GRADE CONTROL	ORE HAULAGE & TREATMENT	OPERATING COSTS
FEITCH	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,946,291	0	556,083	111,217	0	0	2,613,590
380.0 -> 377.5	1.832.625	0	523,607	104,677	316	13.170	2,474,395
377.5 -> 375.0	1,752,545	0	500,727	89,389	76,652	3,193,847	5,613,159
375.0 -> 372.5	1,935,552	967,776	483,888	62,260	240,728	10,030,336	13,720,540
372.5 -> 370.0	1,782,280	891,140	445,570	41,707	304,127	12,671,954	16,136,778
370.0 -> 367.5	1,691,402	751,734	375,867	25,570	230,918	9,621,600	12,697,091
367.5 -> 365.0	1,439,604	639,824	319,912	10,618	223,152	9,297,979	11,931,089
365.0 -> 362.5	1,221,750	488,700	244,350	4,627	158,562	6,606,731	8,724,720
362.5 -> 360.0	602,760	241,104	120,552	2,615	75,960	3,165,008	4,208,000
360.0 -> 357.5	113,980	45,592	22,796	644	13,266	552,754	749,032
357.5 -> 355.0	6,515	2,606	1,303	71	626	26,090	37,211
355.0 -> 352.5	755	302	151	10	65	2,724	4,008
TOTAL	14,326,057	4,028,778	3,594,806	453,405	1,324,373	55,182,193	78,909,612

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,450/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,613,590	2,613,590	-2,613,590	
380.0 -> 377.5	220	527	1.39	20	29,020	725	1,601	2,327	2,474,395	2,476,721	-5,061,292	123,753
377.5 -> 375.0	53,783	127,754	1.45	5,079	7,365,221	184,131	406,357	590,488	5,613,159	6,203,647	-3,899,717	1,221
375.0 -> 372.5	172,588	401,213	1.45	15,920	23,083,856	577,096	1,273,592	1,850,688	13,720,540	15,571,229	3,612,910	978
372.5 -> 370.0	237,034	506,878	1.73	23,902	34,657,248	866,431	1,912,124	2,778,555	16,136,778	18,915,333	19,354,824	791
370.0 -> 367.5	248,015	384,864	2.08	21,891	31,742,059	793,551	1,751,286	2,544,837	12,697,091	15,241,929	35,854,954	696
367.5 -> 365.0	266,823	371,919	2.25	22,919	33,231,827	830,796	1,833,480	2,664,276	11,931,089	14,595,364	54,491,417	637
365.0 -> 362.5	221,216	264,269	2.86	20,661	29,957,787	748,945	1,652,843	2,401,788	8,724,720	11,126,508	73,322,697	539
362.5 -> 360.0	107,477	126,600	2.36	8,154	11,823,530	295,588	652,333	947,921	4,208,000	5,155,921	79,990,306	632
360.0 -> 357.5	19,577	22,110	1.59	961	1,394,081	34,852	76,915	111,767	749,032	860,799	80,523,588	895
357.5 -> 355.0	949	1,044	1.52	43	62,759	1,569	3,463	5,032	37,211	42,242	80,544,105	976
355.0 -> 352.5	99	109	1.27	4	5,494	137	303	440	4,008	4,448	80,545,151	1,174
TOTAL	1,327,782	2,207,288	1.98	119,554	173,352,882	4,333,822	9,564,297	13,898,119	78,909,612	92,807,731	80,545,151	776

\$1,500/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL	
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME	
	(BCM)	(t)	(g/t)	(BCM)	(BCM)	
N/S -> 380.0	0	0	0.00	566,322	566,322	
380.0 -> 377.5	229	550	1.37	536,743	536,972	
377.5 -> 375.0	54,278	128,941	1.45	459,932	514,210	
375.0 -> 372.5	174,897	406,552	1.45	323,360	498,257	
372.5 -> 370.0	242,206	518,222	1.71	217,311	459,517	
370.0 -> 367.5	256,386	396,863	2.05	132,850	389,236	
367.5 -> 365.0	275,482	383,707	2.22	57,030	332,512	
365.0 -> 362.5	227,181	271,423	2.82	25,433	252,614	
362.5 -> 360.0	110,965	130,695	2.33	14,054	125,019	
360.0 -> 357.5	20,739	23,424	1.57	3,438	24,177	
357.5 -> 355.0	1,002	1,102	1.50	416	1,418	
355.0 -> 352.5	99	109	1.27	52	151	
TOTAL	1,363,463	2,261,588	1.96	2,336,942	3,700,405	

UNIT COSTS

MINING		BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50	2.00	1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH	FLITCH		EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)		(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,982,127	0	566,322	113,264	0	0	2,661,713
380.0 -> 377.5	1,879,402	0	536,972	107,349	330	13,745	2,537,797
377.5 -> 375.0	1,799,735	0	514,210	91,986	77,365	3,223,535	5,706,831
375.0 -> 372.5	1,993,028	996,514	498,257	64,672	243,931	10,163,808	13,960,210
372.5 -> 370.0	1,838,068	919,034	459,517	43,462	310,933	12,955,541	16,526,555
370.0 -> 367.5	1,751,562	778,472	389,236	26,570	238,118	9,921,580	13,105,538
367.5 -> 365.0	1,496,304	665,024	332,512	11,406	230,224	9,592,686	12,328,157
365.0 -> 362.5	1,263,070	505,228	252,614	5,087	162,854	6,785,585	8,974,437
362.5 -> 360.0	625,095	250,038	125,019	2,811	78,417	3,267,368	4,348,747
360.0 -> 357.5	120,885	48,354	24,177	688	14,054	585,591	793,749
357.5 -> 355.0	7,090	2,836	1,418	83	661	27,539	39,627
355.0 -> 352.5	755	302	151	10	65	2,724	4,008
TOTAL	14,757,121	4,165,802	3,700,405	467,388	1,356,953	56,539,700	80,987,369

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,500/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,661,713	2,661,713	-2,661,713	
380.0 -> 377.5	229	550	1.37	21	30,822	771	1,644	2,414	2,537,797	2,540,211	-5,171,103	123,622
377.5 -> 375.0	54,278	128,941	1.45	5,106	7,658,926	191,473	408,476	599,949	5,706,831	6,306,780	-3,818,957	1,235
375.0 -> 372.5	174,897	406,552	1.45	16,055	24,083,244	602,081	1,284,440	1,886,521	13,960,210	15,846,731	4,417,557	987
372.5 -> 370.0	242,206	518,222	1.71	24,173	36,258,963	906,474	1,933,811	2,840,285	16,526,555	19,366,840	21,309,679	801
370.0 -> 367.5	256,386	396,863	2.05	22,180	33,270,260	831,757	1,774,414	2,606,170	13,105,538	15,711,708	38,868,231	708
367.5 -> 365.0	275,482	383,707	2.22	23,275	34,912,241	872,806	1,861,986	2,734,792	12,328,157	15,062,949	58,717,524	647
365.0 -> 362.5	227,181	271,423	2.82	20,914	31,371,640	784,291	1,673,154	2,457,445	8,974,437	11,431,882	78,657,281	547
362.5 -> 360.0	110,965	130,695	2.33	8,316	12,474,486	311,862	665,306	977,168	4,348,747	5,325,915	85,805,852	640
360.0 -> 357.5	20,739	23,424	1.57	1,005	1,508,057	37,701	80,430	118,131	793,749	911,880	86,402,029	907
357.5 -> 355.0	1,002	1,102	1.50	45	67,733	1,693	3,612	5,306	39,627	44,933	86,424,829	995
355.0 -> 352.5	99	109	1.27	4	5,684	142	303	445	4,008	4,453	86,426,060	1,175
TOTAL	1,363,463	2,261,588	1.96	121,095	181,642,057	4,541,051	9,687,576	14,228,628	80,987,369	95,215,997	86,426,060	786

\$1,550/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL	
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME	
	(BCM)			(BCM)	(BCM)	
N/S -> 380.0	0	0	0.00	571,020	571,020	
380.0 -> 377.5	229	550	1.37	543,249	543,478	
377.5 -> 375.0	54,367	129,153	1.45	466,586	520,953	
375.0 -> 372.5	176,342	409,993	1.44	328,363	504,705	
372.5 -> 370.0	245,353	525,190	1.70	222,037	467,390	
370.0 -> 367.5	263,581	407,052	2.01	134,119	397,700	
367.5 -> 365.0	280,928	391,178	2.20	58,711	339,639	
365.0 -> 362.5	230,649	275,570	2.80	26,912	257,561	
362.5 -> 360.0	113,849	134,120	2.29	14,626	128,475	
360.0 -> 357.5	22,506	25,409	1.55	3,812	26,318	
357.5 -> 355.0	1,232	1,354	1.44	492	1,724	
355.0 -> 352.5	99	109	1.27	52	151	
TOTAL	1,389,134	2,299,679	1.94	2,369,980	3,759,114	

UNIT COSTS

MINING		BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50	2.00	1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH	FLITCH		EXTRAS		CONTROL	& TREATMENT	COSTS
(\$)		(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	1,998,570	0	571,020	114,204	0	0	2,683,794
380.0 -> 377.5	1,902,173	0	543,478	108,650	330	13,745	2,568,375
377.5 -> 375.0	1,823,336	0	520,953	93,317	77,492	3,228,833	5,743,931
375.0 -> 372.5	2,018,820	1,009,410	504,705	65,673	245,996	10,249,823	14,094,427
372.5 -> 370.0	1,869,560	934,780	467,390	44,407	315,114	13,129,746	16,760,997
370.0 -> 367.5	1,789,650	795,400	397,700	26,824	244,231	10,176,303	13,430,108
367.5 -> 365.0	1,528,376	679,278	339,639	11,742	234,707	9,779,462	12,573,204
365.0 -> 362.5	1,287,805	515,122	257,561	5,382	165,342	6,889,243	9,120,455
362.5 -> 360.0	642,375	256,950	128,475	2,925	80,472	3,353,008	4,464,206
360.0 -> 357.5	131,590	52,636	26,318	762	15,245	635,221	861,773
357.5 -> 355.0	8,620	3,448	1,724	98	813	33,861	48,565
355.0 -> 352.5	755	302	151	10	65	2,724	4,008
TOTAL	15,001,629	4,247,326	3,759,114	473,996	1,379,807	57,491,969	82,353,841

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,550/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,683,794	2,683,794	-2,683,794	-
380.0 -> 377.5	229	550	1.37	21	31,850	796	1,644	2,440	2,568,375	2,570,815	-5,222,760	125,112
377.5 -> 375.0	54,367	129,153	1.45	5,111	7,921,868	198,047	408,871	606,917	5,743,931	6,350,848	-3,651,740	1,243
375.0 -> 372.5	176,342	409,993	1.44	16,125	24,994,471	624,862	1,290,037	1,914,899	14,094,427	16,009,326	5,333,405	993
372.5 -> 370.0	245,353	525,190	1.70	24,343	37,731,488	943,287	1,947,432	2,890,719	16,760,997	19,651,716	23,413,177	807
370.0 -> 367.5	263,581	407,052	2.01	22,412	34,737,883	868,447	1,792,923	2,661,370	13,430,108	16,091,478	42,059,582	718
367.5 -> 365.0	280,928	391,178	2.20	23,476	36,388,526	909,713	1,878,117	2,787,831	12,573,204	15,361,034	63,087,074	654
365.0 -> 362.5	230,649	275,570	2.80	21,049	32,626,481	815,662	1,683,947	2,499,609	9,120,455	11,620,065	84,093,490	552
362.5 -> 360.0	113,849	134,120	2.29	8,409	13,033,228	325,831	672,683	998,513	4,464,206	5,462,719	91,663,999	650
360.0 -> 357.5	22,506	25,409	1.55	1,073	1,662,962	41,574	85,830	127,404	861,773	989,177	92,337,785	922
357.5 -> 355.0	1,232	1,354	1.44	53	82,572	2,064	4,262	6,326	48,565	54,891	92,365,466	1,030
355.0 -> 352.5	99	109	1.27	4	5,873	147	303	450	4,008	4,458	92,366,881	1,176
TOTAL	1,389,134	2,299,679	1.94	122,076	189,217,202	4,730,430	9,766,049	14,496,479	82,353,841	96,850,320	92,366,881	793

\$1,600/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	580,850	580,850
380.0 -> 377.5	239	574	1.35	554,118	554,357
377.5 -> 375.0	54,660	129,859	1.45	477,150	531,810
375.0 -> 372.5	179,844	418,216	1.43	335,756	515,600
372.5 -> 370.0	248,694	531,787	1.69	229,434	478,128
370.0 -> 367.5	269,298	415,205	1.99	137,462	406,760
367.5 -> 365.0	285,951	398,149	2.18	62,137	348,088
365.0 -> 362.5	235,347	281,178	2.76	28,659	264,006
362.5 -> 360.0	115,972	136,609	2.28	15,385	131,357
360.0 -> 357.5	23,367	26,372	1.53	4,060	27,427
357.5 -> 355.0	1,312	1,443	1.44	545	1,857
355.0 -> 352.5	99	109	1.27	52	151
TOTAL	1,414,783	2,339,502	1.93	2,425,608	3,840,391

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,032,975	0	580,850	116,170	0	0	2,729,995
380.0 -> 377.5	1,940,250	0	554,357	110,824	344	14,344	2,620,119
377.5 -> 375.0	1,861,335	0	531,810	95,430	77,915	3,246,476	5,812,966
375.0 -> 372.5	2,062,400	1,031,200	515,600	67,151	250,929	10,455,391	14,382,672
372.5 -> 370.0	1,912,512	956,256	478,128	45,887	319,072	13,294,680	17,006,535
370.0 -> 367.5	1,830,420	813,520	406,760	27,492	249,123	10,380,121	13,707,437
367.5 -> 365.0	1,566,396	696,176	348,088	12,427	238,889	9,953,717	12,815,694
365.0 -> 362.5	1,320,030	528,012	264,006	5,732	168,707	7,029,462	9,315,949
362.5 -> 360.0	656,785	262,714	131,357	3,077	81,966	3,415,233	4,551,132
360.0 -> 357.5	137,135	54,854	27,427	812	15,823	659,311	895,363
357.5 -> 355.0	9,285	3,714	1,857	109	866	36,086	51,917
355.0 -> 352.5	755	302	151	10	65	2,724	4,008
TOTAL	15,330,278	4,346,748	3,840,391	485,122	1,403,701	58,487,546	83,893,785

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,600/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,729,995	2,729,995	-2,729,995	-
380.0 -> 377.5	239	574	1.35	21	33,795		1,690	2,535	2,620,119	2,622,653	-5,318,853	124,167
377.5 -> 375.0	54,660	129,859	1.45	5,128	8,205,394	205,135	410,270	615,405	5,812,966	6,428,371	-3,541,830	1,253
375.0 -> 372.5	179,844	418,216	1.43	16,303	26,085,134	652,128	1,304,257	1,956,385	14,382,672	16,339,057	6,204,247	1,002
372.5 -> 370.0	248,694	531,787	1.69	24,492	39,187,139	979,678	1,959,357	2,939,035	17,006,535	19,945,570	25,445,815	814
370.0 -> 367.5	269,298	415,205	1.99	22,593	36,149,487	903,737	1,807,474	2,711,212	13,707,437	16,418,648	45,176,654	727
367.5 -> 365.0	285,951	398,149	2.18	23,692	37,907,368	947,684	1,895,368	2,843,053	12,815,694	15,658,746	67,425,276	661
365.0 -> 362.5	235,347	281,178	2.76	21,237	33,978,715	849,468	1,698,936	2,548,404	9,315,949	11,864,353	89,539,639	559
362.5 -> 360.0	115,972	136,609	2.28	8,495	13,592,061	339,802	679,603	1,019,405	4,551,132	5,570,537	97,561,163	656
360.0 -> 357.5	23,367	26,372	1.53	1,105	1,768,141	44,204	88,407	132,611	895,363	1,027,973	98,301,331	930
357.5 -> 355.0	1,312	1,443	1.44	57	90,772	2,269	4,539	6,808	51,917	58,724	98,333,379	1,035
355.0 -> 352.5	99	109	1.27	4	6,063	152	303	455	4,008	4,462	98,334,979	1,178
TOTAL	1,414,783	2,339,502	1.93	123,128	197,004,069	4,925,102	9,850,203	14,775,305	83,893,785	98,669,090	98,334,979	801

\$1,650/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	589,933	589,933
380.0 -> 377.5	239	574	1.35	566,001	566,240
377.5 -> 375.0	55,081	130,869	1.44	490,101	545,182
375.0 -> 372.5	181,995	423,130	1.42	346,622	528,617
372.5 -> 370.0	251,260	536,949	1.68	239,002	490,262
370.0 -> 367.5	277,342	427,015	1.96	141,827	419,169
367.5 -> 365.0	292,990	407,844	2.15	64,831	357,821
365.0 -> 362.5	239,847	286,541	2.73	30,454	270,301
362.5 -> 360.0	118,106	139,133	2.26	15,824	133,930
360.0 -> 357.5	24,045	27,132	1.52	4,390	28,435
357.5 -> 355.0	1,437	1,581	1.43	573	2,010
355.0 -> 352.5	99	109	1.27	52	151
TOTAL	1,442,442	2,380,877	1.91	2,489,609	3,932,051

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50	2.00	1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,064,766	0	589,933	117,987	0	0	2,772,685
380.0 -> 377.5	1,981,840	0	566,240	113,200	344	14,344	2,675,969
377.5 -> 375.0	1,908,137	0	545,182	98,020	78,521	3,271,716	5,901,576
375.0 -> 372.5	2,114,468	1,057,234	528,617	69,324	253,878	10,578,242	14,601,763
372.5 -> 370.0	1,961,048	980,524	490,262	47,800	322,169	13,423,728	17,225,532
370.0 -> 367.5	1,886,261	838,338	419,169	28,365	256,209	10,675,378	14,103,720
367.5 -> 365.0	1,610,195	715,642	357,821	12,966	244,706	10,196,095	13,137,425
365.0 -> 362.5	1,351,505	540,602	270,301	6,091	171,925	7,163,533	9,503,957
362.5 -> 360.0	669,650	267,860	133,930	3,165	83,480	3,478,333	4,636,418
360.0 -> 357.5	142,175	56,870	28,435	878	16,279	678,304	922,941
357.5 -> 355.0	10,050	4,020	2,010	115	949	39,534	56,678
355.0 -> 352.5	755	302	151	10	65	2,724	4,008
TOTAL	15,700,849	4,461,392	3,932,051	497,922	1,428,526	59,521,932	85,542,671

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,650/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,772,685	2,772,685	-2,772,685	-
380.0 -> 377.5	239	574	1.35	21	34,851	871	1,690	2,561	2,675,969	2,678,530	-5,416,364	126,812
377.5 -> 375.0	55,081	130,869	1.44	5,147	8,492,888	212,322	411,776	624,099	5,901,576	6,525,675	-3,449,151	1,268
375.0 -> 372.5	181,995	423,130	1.42	16,404	27,066,729	676,668	1,312,326	1,988,994	14,601,763	16,590,758	7,026,821	1,011
372.5 -> 370.0	251,260	536,949	1.68	24,629	40,637,845	1,015,946	1,970,320	2,986,266	17,225,532	20,211,798	27,452,867	821
370.0 -> 367.5	277,342	427,015	1.96	22,881	37,754,351	943,859	1,830,514	2,774,373	14,103,720	16,878,093	48,329,126	738
367.5 -> 365.0	292,990	407,844	2.15	23,963	39,539,052	988,476	1,917,045	2,905,521	13,137,425	16,042,946	71,825,231	669
365.0 -> 362.5	239,847	286,541	2.73	21,404	35,316,187	882,905	1,712,300	2,595,205	9,503,957	12,099,162	95,042,257	565
362.5 -> 360.0	118,106	139,133	2.26	8,581	14,158,924	353,973	686,493	1,040,466	4,636,418	5,676,884	103,524,296	662
360.0 -> 357.5	24,045	27,132	1.52	1,130	1,863,927	46,598	90,372	136,970	922,941	1,059,911	104,328,312	938
357.5 -> 355.0	1,437	1,581	1.43	62	102,275	2,557	4,959	7,516	56,678	64,193	104,366,394	1,036
355.0 -> 352.5	99	109	1.27	4	6,252	156	303	459	4,008	4,467	104,368,179	1,179
TOTAL	1,442,442	2,380,877	1.91	124,226	204,973,281	5,124,332	9,938,098	15,062,430	85,542,671	100,605,102	104,368,179	810

\$1,700/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	595,202	595,202
380.0 -> 377.5	239	574	1.35	572,313	572,552
377.5 -> 375.0	55,121	130,963	1.44	496,615	551,736
375.0 -> 372.5	183,856	427,184	1.41	351,592	535,448
372.5 -> 370.0	253,851	542,412	1.67	243,738	497,589
370.0 -> 367.5	281,863	433,550	1.94	144,687	426,550
367.5 -> 365.0	297,770	414,317	2.13	66,198	363,968
365.0 -> 362.5	242,581	289,824	2.71	31,316	273,897
362.5 -> 360.0	119,719	141,001	2.24	16,346	136,065
360.0 -> 357.5	25,282	28,517	1.51	4,699	29,981
357.5 -> 355.0	2,005	2,205	1.39	664	2,669
355.0 -> 352.5	133	147	1.28	151	284
TOTAL	1,462,421	2,410,692	1.90	2,523,520	3,985,941

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367.5	4.50		1.00	0.20	0.60	25.00
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352.5	5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,083,207	0	595,202	119,040	0	0	2,797,449
380.0 -> 377.5	2,003,932	0	572,552	114,463	344	14,344	2,705,635
377.5 -> 375.0	1,931,076	0	551,736	99,323	78,578	3,274,065	5,934,777
375.0 -> 372.5	2,141,792	1,070,896	535,448	70,318	256,310	10,679,601	14,754,366
372.5 -> 370.0	1,990,356	995,178	497,589	48,748	325,447	13,560,299	17,417,616
370.0 -> 367.5	1,919,475	853,100	426,550	28,937	260,130	10,838,738	14,326,930
367.5 -> 365.0	1,637,856	727,936	363,968	13,240	248,590	10,357,930	13,349,520
365.0 -> 362.5	1,369,485	547,794	273,897	6,263	173,894	7,245,601	9,616,934
362.5 -> 360.0	680,325	272,130	136,065	3,269	84,600	3,525,014	4,701,404
360.0 -> 357.5	149,905	59,962	29,981	940	17,110	712,915	970,812
357.5 -> 355.0	13,345	5,338	2,669	133	1,323	55,128	77,936
355.0 -> 352.5	1,420	568	284	30	88	3,674	6,064
TOTAL	15,922,174	4,532,902	3,985,941	504,704	1,446,415	60,267,308	86,659,445

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1.700/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
1 211 011	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	(Ψ)	2,797,449	2,797,449	-2,797,449	
380.0 -> 377.5	239	574	1.35	21	35,907	898	1,690	2,587	2,705,635	2,708,223	-5.469.765	128,218
377.5 -> 375.0	55,121	130,963	1.44	5,147	8,750,565	218,764	411,791	630,555	5,934,777	6,565,333	-3,284,532	1,275
375.0 -> 372.5	183,856	427,184	1.41	16,481	28,017,944	700,449	1,318,491	2,018,940	14,754,366	16,773,306	7,960,106	1,018
372.5 -> 370.0	253,851	542,412	1.67	24,734	42,048,212	1,051,205	1,978,739	3,029,945	17,417,616	20,447,561	29,560,757	827
370.0 -> 367.5	281,863	433,550	1.94	23,022	39,138,221	978,456	1,841,799	2,820,254	14,326,930	17,147,184	51,551,794	745
367.5 -> 365.0	297,770	414,317	2.13	24,133	41,025,251	1,025,631	1,930,600	2,956,231	13,349,520	16,305,751	76,271,294	676
365.0 -> 362.5	242,581	289,824	2.71	21,501	36,552,415	913,810	1,720,114	2,633,924	9,616,934	12,250,858	100,572,850	570
362.5 -> 360.0	119,719	141,001	2.24	8,628	14,668,163	366,704	690,266	1,056,971	4,701,404	5,758,375	109,482,639	667
360.0 -> 357.5	25,282	28,517	1.51	1,177	2,000,217	50,005	94,128	144,133	970,812	1,114,946	110,367,909	948
357.5 -> 355.0	2,005	2,205	1.39	84	142,820	3,571	6,721	10,291	77,936	88,227	110,422,503	1,050
355.0 -> 352.5	133	147	1.28	5	8,741	219	411	630	6,064	6,694	110,424,550	1,302
TOTAL	1,462,421	2,410,692	1.90	124,934	212,388,457	5,309,711	9,994,751	15,304,462	86,659,445	101,963,907	110,424,550	816

\$1,750/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	601,535	601,535
380.0 -> 377.5	239	574	1.35	578,919	579,158
377.5 -> 375.0	55,145	131,022	1.44	503,253	558,398
375.0 -> 372.5	185,156	430,303	1.41	356,642	541,798
372.5 -> 370.0	255,553	545,986	1.66	249,128	504,681
370.0 -> 367.5	286,820	441,111	1.92	146,884	433,704
367.5 -> 365.0	303,575	422,653	2.11	67,020	370,595
365.0 -> 362.5	247,088	295,212	2.68	31,803	278,891
362.5 -> 360.0	120,844	142,286	2.23	16,931	137,775
360.0 -> 357.5	25,710	29,001	1.50	4,965	30,675
357.5 -> 355.0	2,085	2,293	1.38	734	2,819
355.0 -> 352.5	153	169	1.28	208	361
TOTAL	1,482,367	2,440,609	1.88	2,558,023	4,040,390

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,105,373	0	601,535	120,307	0	0	2,827,215
380.0 -> 377.5	2,027,053	0	579,158	115,784	344	14,344	2,736,683
377.5 -> 375.0	1,954,393	0	558,398	100,651	78,613	3,275,539	5,967,594
375.0 -> 372.5	2,167,192	1,083,596	541,798	71,328	258,182	10,757,570	14,879,666
372.5 -> 370.0	2,018,724	1,009,362	504,681	49,826	327,591	13,649,638	17,559,822
370.0 -> 367.5	1,951,668	867,408	433,704	29,377	264,667	11,027,787	14,574,611
367.5 -> 365.0	1,667,678	741,190	370,595	13,404	253,592	10,566,322	13,612,780
365.0 -> 362.5	1,394,455	557,782	278,891	6,361	177,127	7,380,297	9,794,912
362.5 -> 360.0	688,875	275,550	137,775	3,386	85,372	3,557,152	4,748,109
360.0 -> 357.5	153,375	61,350	30,675	993	17,401	725,035	988,829
357.5 -> 355.0	14,095	5,638	2,819	147	1,376	57,327	81,402
355.0 -> 352.5	1,805	722	361	42	101	4,223	7,254
TOTAL	16,144,685	4,602,598	4,040,390	511,605	1,464,366	61,015,234	87,778,877

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,750/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,827,215	2,827,215	-2,827,215	
380.0 -> 377.5	239	574	1.35	21	36,963	924	1,690	2,614	2,736,683	2,739,297	-5,529,548	129,689
377.5 -> 375.0	55,145	131,022	1.44	5,150	9,011,992	225,300	411,977	637,277	5,967,594	6,604,870	-3,122,427	1,283
375.0 -> 372.5	185,156	430,303	1.41	16,532	28,931,516	723,288	1,322,584	2,045,872	14,879,666	16,925,538	8,883,551	1,024
372.5 -> 370.0	255,553	545,986	1.66	24,824	43,442,102	1,086,053	1,985,925	3,071,977	17,559,822	20,631,799	31,693,854	831
370.0 -> 367.5	286,820	441,111	1.92	23,176	40,557,747	1,013,944	1,854,068	2,868,012	14,574,611	17,442,623	54,808,978	753
367.5 -> 365.0	303,575	422,653	2.11	24,335	42,586,123	1,064,653	1,946,794	3,011,447	13,612,780	16,624,227	80,770,873	683
365.0 -> 362.5	247,088	295,212	2.68	21,656	37,897,898	947,447	1,732,475	2,679,923	9,794,912	12,474,835	106,193,936	576
362.5 -> 360.0	120,844	142,286	2.23	8,665	15,163,856	379,096	693,205	1,072,301	4,748,109	5,820,411	115,537,381	672
360.0 -> 357.5	25,710	29,001	1.50	1,190	2,083,174	52,079	95,231	147,310	988,829	1,136,139	116,484,416	954
357.5 -> 355.0	2,085	2,293	1.38	87	151,596	3,790	6,930	10,720	81,402	92,122	116,543,890	1,063
355.0 -> 352.5	153	169	1.28	6	10,305	258	471	729	7,254	7,983	116,546,212	1,356
TOTAL	1,482,367	2,440,609	1.88	125,642	219,873,271	5,496,832	10,051,350	15,548,181	87,778,877	103,327,059	116,546,212	822

\$1,800/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	605,329	605,329
380.0 -> 377.5	239	574	1.35	583,133	583,372
377.5 -> 375.0	55,145	131,022	1.44	507,067	562,212
375.0 -> 372.5	185,412	430,919	1.40	361,121	546,533
372.5 -> 370.0	257,061	549,075	1.66	252,335	509,396
370.0 -> 367.5	289,060	444,578	1.91	148,801	437,861
367.5 -> 365.0	306,251	426,235	2.10	67,601	373,852
365.0 -> 362.5	250,634	299,413	2.66	32,203	282,837
362.5 -> 360.0	122,919	144,719	2.21	17,077	139,996
360.0 -> 357.5	26,126	29,475	1.49	5,169	31,295
357.5 -> 355.0	2,105	2,315	1.38	753	2,858
355.0 -> 352.5	168	185	1.27	250	418
TOTAL	1,495,120	2,458,510	1.88	2,580,839	4,075,959

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,118,652	0	605,329	121,066	0	0	2,845,046
380.0 -> 377.5	2,041,802	0	583,372	116,627	344	14,344	2,756,489
377.5 -> 375.0	1,967,742	0	562,212	101,413	78,613	3,275,539	5,985,520
375.0 -> 372.5	2,186,132	1,093,066	546,533	72,224	258,551	10,772,964	14,929,470
372.5 -> 370.0	2,037,584	1,018,792	509,396	50,467	329,445	13,726,882	17,672,566
370.0 -> 367.5	1,970,375	875,722	437,861	29,760	266,747	11,114,452	14,694,917
367.5 -> 365.0	1,682,334	747,704	373,852	13,520	255,741	10,655,886	13,729,037
365.0 -> 362.5	1,414,185	565,674	282,837	6,441	179,648	7,485,330	9,934,114
362.5 -> 360.0	699,980	279,992	139,996	3,415	86,831	3,617,977	4,828,192
360.0 -> 357.5	156,475	62,590	31,295	1,034	17,685	736,880	1,005,959
357.5 -> 355.0	14,290	5,716	2,858	151	1,389	57,877	82,280
355.0 -> 352.5	2,090	836	418	50	111	4,623	8,128
TOTAL	16,291,640	4,650,092	4,075,959	516,168	1,475,106	61,462,755	88,471,720

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,800/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,845,046	2,845,046	-2,845,046	-
380.0 -> 377.5	239	574	1.35	21	38,020	950	1,690	2,640	2,756,489	2,759,129	-5,566,156	130,628
377.5 -> 375.0	55,145	131,022	1.44	5,150	9,269,477	231,737	411,977	643,714	5,985,520	6,629,233	-2,925,912	1,287
375.0 -> 372.5	185,412	430,919	1.40	16,544	29,779,933	744,498	1,323,553	2,068,051	14,929,470	16,997,521	9,856,499	1,027
372.5 -> 370.0	257,061	549,075	1.66	24,906	44,830,251	1,120,756	1,992,456	3,113,212	17,672,566	20,785,778	33,900,972	835
370.0 -> 367.5	289,060	444,578	1.91	23,239	41,829,979	1,045,749	1,859,110	2,904,860	14,694,917	17,599,777	58,131,175	757
367.5 -> 365.0	306,251	426,235	2.10	24,416	43,948,046	1,098,701	1,953,247	3,051,948	13,729,037	16,780,985	85,298,236	687
365.0 -> 362.5	250,634	299,413	2.66	21,764	39,174,462	979,362	1,741,087	2,720,449	9,934,114	12,654,563	111,818,135	581
362.5 -> 360.0	122,919	144,719	2.21	8,732	15,717,249	392,931	698,544	1,091,476	4,828,192	5,919,668	121,615,716	678
360.0 -> 357.5	26,126	29,475	1.49	1,204	2,166,328	54,158	96,281	150,439	1,005,959	1,156,399	122,625,645	961
357.5 -> 355.0	2,105	2,315	1.38	87	156,864	3,922	6,972	10,893	82,280	93,174	122,689,335	1,069
355.0 -> 352.5	168	185	1.27	6	11,567	289	514	803	8,128	8,931	122,691,970	1,390
TOTAL	1,495,120	2,458,510	1.88	126,068	226,922,175	5,673,054	10,085,430	15,758,484	88,471,720	104,230,204	122,691,970	827

\$1,850/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	610,058	610,058
380.0 -> 377.5	239	574	1.35	589,753	589,992
377.5 -> 375.0	55,155	131,046	1.44	514,843	569,998
375.0 -> 372.5	185,901	432,082	1.40	368,584	554,485
372.5 -> 370.0	258,992	553,131	1.65	258,908	517,900
370.0 -> 367.5	291,297	448,270	1.90	155,097	446,394
367.5 -> 365.0	308,878	429,731	2.09	72,880	381,758
365.0 -> 362.5	254,536	304,026	2.64	35,120	289,656
362.5 -> 360.0	125,955	148,266	2.19	18,293	144,248
360.0 -> 357.5	27,660	31,199	1.48	5,251	32,911
357.5 -> 355.0	2,628	2,891	1.38	910	3,538
355.0 -> 352.5	201	221	1.28	274	475
TOTAL	1,511,442	2,481,435	1.87	2,629,971	4,141,413

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

FLITCH	MINING	BLASTING	MINING EXTRAS	REHAB	GRADE CONTROL	ORE HAULAGE & TREATMENT	OPERATING COSTS
TEITOIT	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,135,203	0	610,058	122,012	0	0	2,867,273
380.0 -> 377.5	2,064,972	0	589,992	117,951	344	14,344	2,787,603
377.5 -> 375.0	1,994,993	0	569,998	102,969	78,627	3,276,139	6,022,726
375.0 -> 372.5	2,217,940	1,108,970	554,485	73,717	259,249	10,802,052	15,016,414
372.5 -> 370.0	2,071,600	1,035,800	517,900	51,782	331,878	13,828,266	17,837,226
370.0 -> 367.5	2,008,773	892,788	446,394	31,019	268,962	11,206,741	14,854,677
367.5 -> 365.0	1,717,911	763,516	381,758	14,576	257,839	10,743,276	13,878,876
365.0 -> 362.5	1,448,280	579,312	289,656	7,024	182,416	7,600,659	10,107,346
362.5 -> 360.0	721,240	288,496	144,248	3,659	88,959	3,706,642	4,953,244
360.0 -> 357.5	164,555	65,822	32,911	1,050	18,719	779,963	1,063,020
357.5 -> 355.0	17,690	7,076	3,538	182	1,735	72,271	102,492
355.0 -> 352.5	2,375	950	475	55	133	5,523	9,510
TOTAL	16,565,532	4,742,730	4,141,413	525,994	1,488,861	62,035,876	89,500,406

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1.850/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,867,273	2,867,273	-2,867,273	-
380.0 -> 377.5	239	574	1.35	21	39,076		1,690	2,667	2,787,603	2,790,270	-5,618,467	132,103
377.5 -> 375.0	55,155	131,046	1.44	5,151	9,528,707	238,218	412,052	650,270	6,022,726	6,672,996	-2,762,756	1,296
375.0 -> 372.5	185,901	432,082	1.40	16,566	30,646,963	766,174	1,325,274	2,091,448	15,016,414	17,107,862	10,776,346	1,033
372.5 -> 370.0	258,992	553,131	1.65	24,971	46,196,511	1,154,913	1,997,687	3,152,600	17,837,226	20,989,826	35,983,031	841
370.0 -> 367.5	291,297	448,270	1.90	23,312	43,126,716	1,078,168	1,864,939	2,943,107	14,854,677	17,797,784	61,311,963	763
367.5 -> 365.0	308,878	429,731	2.09	24,512	45,347,560	1,133,689	1,960,976	3,094,665	13,878,876	16,973,540	89,685,983	692
365.0 -> 362.5	254,536	304,026	2.64	21,895	40,506,248	1,012,656	1,751,622	2,764,278	10,107,346	12,871,624	117,320,607	588
362.5 -> 360.0	125,955	148,266	2.19	8,854	16,380,691	409,517	708,354	1,117,871	4,953,244	6,071,115	127,630,183	686
360.0 -> 357.5	27,660	31,199	1.48	1,266	2,341,215	58,530	101,242	159,772	1,063,020	1,222,792	128,748,606	966
357.5 -> 355.0	2,628	2,891	1.38	109	202,178	5,054	8,743	13,797	102,492	116,289	128,834,494	1,064
355.0 -> 352.5	201	221	1.28	8	14,311	358	619	977	9,510	10,487	128,838,319	1,356
TOTAL	1,511,442	2,481,435	1.87	126,665	234,330,176	5,858,254	10,133,197	15,991,451	89,500,406	105,491,857	128,838,319	833

\$1,900/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	611,277	611,277
380.0 -> 377.5	239	574	1.35	591,509	591,748
377.5 -> 375.0	55,175	131,094	1.44	517,050	572,225
375.0 -> 372.5	186,823	434,296	1.40	369,987	556,810
372.5 -> 370.0	259,334	553,877	1.65	260,052	519,386
370.0 -> 367.5	293,377	451,461	1.89	155,799	449,176
367.5 -> 365.0	311,616	433,533	2.07	73,610	385,226
365.0 -> 362.5	256,236	306,108	2.62	35,813	292,049
362.5 -> 360.0	126,998	149,501	2.18	18,713	145,711
360.0 -> 357.5	27,861	31,427	1.48	5,360	33,221
357.5 -> 355.0	2,727	3,000	1.37	1,024	3,751
355.0 -> 352.5	201	221	1.28	274	475
TOTAL	1,520,588	2,495,092	1.86	2,640,467	4,161,055

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,139,470	0	611,277	122,255	0	0	2,873,002
380.0 -> 377.5	2,071,118	0	591,748	118,302	344	14,344	2,795,856
377.5 -> 375.0	2,002,788	0	572,225	103,410	78,656	3,277,339	6,034,417
375.0 -> 372.5	2,227,240	1,113,620	556,810	73,997	260,578	10,857,405	15,089,650
372.5 -> 370.0	2,077,544	1,038,772	519,386	52,010	332,326	13,846,934	17,866,973
370.0 -> 367.5	2,021,292	898,352	449,176	31,160	270,877	11,286,534	14,957,390
367.5 -> 365.0	1,733,517	770,452	385,226	14,722	260,120	10,838,313	14,002,349
365.0 -> 362.5	1,460,245	584,098	292,049	7,163	183,665	7,652,688	10,179,907
362.5 -> 360.0	728,555	291,422	145,711	3,743	89,701	3,737,529	4,996,661
360.0 -> 357.5	166,105	66,442	33,221	1,072	18,856	785,686	1,071,382
357.5 -> 355.0	18,755	7,502	3,751	205	1,800	74,995	107,008
355.0 -> 352.5	2,375	950	475	55	133	5,523	9,510
TOTAL	16,649,003	4,771,610	4,161,055	528,093	1,497,055	62,377,289	89,984,106

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,900/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,873,002	2,873,002	-2,873,002	-
380.0 -> 377.5	239	574	1.35	21	40,132	1,003	1,690	2,693	2,795,856	2,798,549	-5,631,420	132,495
377.5 -> 375.0	55,175	131,094	1.44	5,153	9,789,823	244,746	412,203	656,949	6,034,417	6,691,366	-2,532,963	1,299
375.0 -> 372.5	186,823	434,296	1.40	16,604	31,548,116	788,703	1,328,342	2,117,045	15,089,650	17,206,695	11,808,458	1,036
372.5 -> 370.0	259,334	553,877	1.65	24,990	47,480,919	1,187,023	1,999,197	3,186,220	17,866,973	21,053,192	38,236,185	842
370.0 -> 367.5	293,377	451,461	1.89	23,369	44,400,832	1,110,021	1,869,509	2,979,529	14,957,390	17,936,920	64,700,097	768
367.5 -> 365.0	311,616	433,533	2.07	24,578	46,698,266	1,167,457	1,966,243	3,133,699	14,002,349	17,136,049	94,262,314	697
365.0 -> 362.5	256,236	306,108	2.62	21,939	41,683,212	1,042,080	1,755,083	2,797,163	10,179,907	12,977,070	122,968,457	592
362.5 -> 360.0	126,998	149,501	2.18	8,892	16,895,109	422,378	711,373	1,133,751	4,996,661	6,130,411	133,733,154	689
360.0 -> 357.5	27,861	31,427	1.48	1,272	2,417,334	60,433	101,782	162,216	1,071,382	1,233,598	134,916,891	970
357.5 -> 355.0	2,727	3,000	1.37	113	213,941	5,349	9,008	14,357	107,008	121,364	135,009,467	1,078
355.0 -> 352.5	201	221	1.28	8	14,698	367	619	986	9,510	10,496	135,013,669	1,357
TOTAL	1,520,588	2,495,092	1.86	126,938	241,182,381	6,029,560	10,155,048	16,184,607	89,984,106	106,168,713	135,013,669	836

\$1,950/OZ OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	614,321	614,321
380.0 -> 377.5	239	574	1.35	596,191	596,430
377.5 -> 375.0	55,180	131,101	1.44	520,855	576,035
375.0 -> 372.5	187,453	435,778	1.40	373,282	560,735
372.5 -> 370.0	259,876	555,045	1.65	263,161	523,037
370.0 -> 367.5	295,356	454,337	1.89	158,566	453,922
367.5 -> 365.0	313,451	436,337	2.07	76,700	390,151
365.0 -> 362.5	259,113	309,703	2.60	38,479	297,592
362.5 -> 360.0	128,410	151,190	2.16	20,029	148,439
360.0 -> 357.5	29,189	32,912	1.47	5,533	34,722
357.5 -> 355.0	3,490	3,838	1.38	1,115	4,605
355.0 -> 352.5	268	295	1.29	321	589
TOTAL	1,532,024	2,511,109	1.86	2,668,554	4,200,578

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,150,124	0	614,321	122,864	0	0	2,887,309
380.0 -> 377.5	2,087,505	0	596,430	119,238	344	14,344	2,817,862
377.5 -> 375.0	2,016,123	0	576,035	104,171	78,660	3,277,513	6,052,502
375.0 -> 372.5	2,242,940	1,121,470	560,735	74,656	261,467	10,894,440	15,155,708
372.5 -> 370.0	2,092,148	1,046,074	523,037	52,632	333,027	13,876,122	17,923,040
370.0 -> 367.5	2,042,649	907,844	453,922	31,713	272,602	11,358,430	15,067,160
367.5 -> 365.0	1,755,680	780,302	390,151	15,340	261,802	10,908,435	14,111,710
365.0 -> 362.5	1,487,960	595,184	297,592	7,696	185,822	7,742,577	10,316,830
362.5 -> 360.0	742,195	296,878	148,439	4,006	90,714	3,779,738	5,061,969
360.0 -> 357.5	173,610	69,444	34,722	1,107	19,747	822,796	1,121,425
357.5 -> 355.0	23,025	9,210	4,605	223	2,303	95,962	135,328
355.0 -> 352.5	2,945	1,178	589	64	177	7,372	12,325
TOTAL	16,816,903	4,827,584	4,200,578	533,711	1,506,665	62,777,729	90,663,170

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,950/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,887,309	2,887,309	-2,887,309	
380.0 -> 377.5	239	574	1.35	21	41,188	1,030	1,690	2,719	2,817,862	2,820,581	-5,666,702	133,538
377.5 -> 375.0	55,180	131,101	1.44	5,153	10,047,986	251,200	412,225	663,425	6,052,502	6,715,927	-2,334,643	1,303
375.0 -> 372.5	187,453	435,778	1.40	16,626	32,420,472	810,512	1,330,071	2,140,582	15,155,708	17,296,291	12,789,538	1,040
372.5 -> 370.0	259,876	555,045	1.65	25,013	48,775,140	1,219,379	2,001,031	3,220,410	17,923,040	21,143,450	40,421,228	845
370.0 -> 367.5	295,356	454,337	1.89	23,420	45,669,660	1,141,742	1,873,627	3,015,369	15,067,160	18,082,529	68,008,359	772
367.5 -> 365.0	313,451	436,337	2.07	24,644	48,054,877	1,201,372	1,971,482	3,172,854	14,111,710	17,284,564	98,778,672	701
365.0 -> 362.5	259,113	309,703	2.60	22,022	42,942,849	1,073,571	1,761,758	2,835,329	10,316,830	13,152,159	128,569,361	597
362.5 -> 360.0	128,410	151,190	2.16	8,940	17,432,850	435,821	715,194	1,151,015	5,061,969	6,212,984	139,789,227	695
360.0 -> 357.5	29,189	32,912	1.47	1,324	2,582,655	64,566	105,955	170,521	1,121,425	1,291,947	141,079,935	975
357.5 -> 355.0	3,490	3,838	1.38	144	281,559	7,039	11,551	18,590	135,328	153,918	141,207,576	1,066
355.0 -> 352.5	268	295	1.29	10	20,321	508	834	1,342	12,325	13,667	141,214,230	1,311
TOTAL	1,532,024	2,511,109	1.86	127,318	248,269,556	6,206,739	10,185,418	16,392,157	90,663,170	107,055,326	141,214,230	841

\$2,000/oz OPTIMUM SHELL EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

PRODUCTION SCHEDULE VOLUMES - DILUTED AND RECOVERED

		ORE		TOTAL	TOTAL
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME
	(BCM)	(t)	(g/t)	(BCM)	(BCM)
N/S -> 380.0	0	0	0.00	625,194	625,194
380.0 -> 377.5	239	574	1.35	607,881	608,120
377.5 -> 375.0	55,224	131,205	1.44	531,720	586,944
375.0 -> 372.5	189,610	440,954	1.39	381,661	571,271
372.5 -> 370.0	263,812	563,973	1.63	268,615	532,427
370.0 -> 367.5	300,417	461,475	1.87	160,895	461,312
367.5 -> 365.0	317,654	442,104	2.05	78,322	395,976
365.0 -> 362.5	262,587	313,891	2.58	39,349	301,936
362.5 -> 360.0	129,422	152,382	2.16	20,395	149,817
360.0 -> 357.5	29,425	33,176	1.47	5,619	35,044
357.5 -> 355.0	3,529	3,881	1.37	1,134	4,663
355.0 -> 352.5	268	295	1.29	321	589
TOTAL	1,552,187	2,543,911	1.84	2,721,106	4,273,293

UNIT COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE
FLITCH			EXTRAS		CONTROL	& TREATMENT
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)
N/S -> 380	0 3.50	0.00	1.00	0.20	0.60	25.00
380.0 -> 377	5 3.50	0.00	1.00	0.20	0.60	25.00
377.5 -> 375	0 3.50	0.00	1.00	0.20	0.60	25.00
375.0 -> 372	5 4.00	2.00	1.00	0.20	0.60	25.00
372.5 -> 370	0 4.00	2.00	1.00	0.20	0.60	25.00
370.0 -> 367			1.00	0.20	0.60	25.00
367.5 -> 365	0 4.50	2.00	1.00	0.20	0.60	25.00
365.0 -> 362	5 5.00	2.00	1.00	0.20	0.60	25.00
362.5 -> 360	0 5.00	2.00	1.00	0.20	0.60	25.00
360.0 -> 357	5 5.00	2.00	1.00	0.20	0.60	25.00
357.5 -> 355	0 5.00	2.00	1.00	0.20	0.60	25.00
355.0 -> 352	5 5.00	2.00	1.00	0.20	0.60	25.00

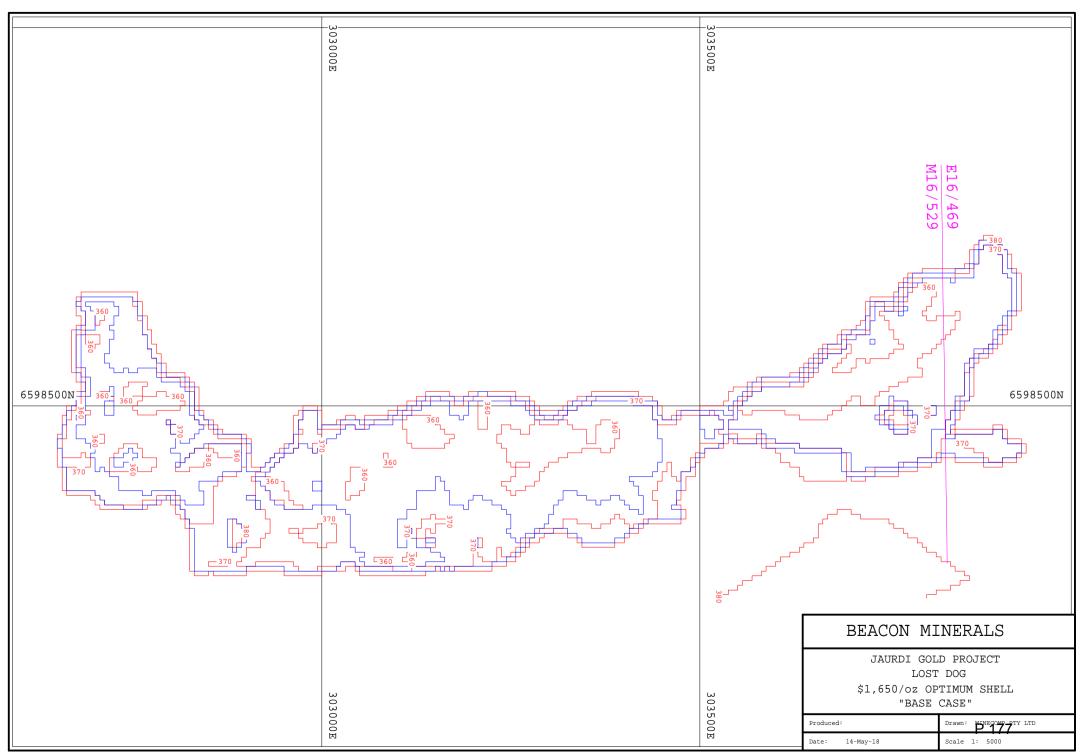
PRODUCTION COSTS

	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	OPERATING
FLITCH			EXTRAS		CONTROL	& TREATMENT	COSTS
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)
N/S -> 380.0	2,188,179	0	625,194	125,039	0	0	2,938,412
380.0 -> 377.5	2,128,420	0	608,120	121,576	344	14,344	2,872,805
377.5 -> 375.0	2,054,304	0	586,944	106,344	78,723	3,280,137	6,106,453
375.0 -> 372.5	2,285,084	1,142,542	571,271	76,332	264,572	11,023,839	15,363,640
372.5 -> 370.0	2,129,708	1,064,854	532,427	53,723	338,384	14,099,333	18,218,429
370.0 -> 367.5	2,075,904	922,624	461,312	32,179	276,885	11,536,883	15,305,788
367.5 -> 365.0	1,781,892	791,952	395,976	15,664	265,262	11,052,602	14,303,349
365.0 -> 362.5	1,509,680	603,872	301,936	7,870	188,335	7,847,285	10,458,977
362.5 -> 360.0	749,085	299,634	149,817	4,079	91,429	3,809,551	5,103,595
360.0 -> 357.5	175,220	70,088	35,044	1,124	19,905	829,393	1,130,774
357.5 -> 355.0	23,315	9,326	4,663	227	2,329	97,036	136,896
355.0 -> 352.5	2,945	1,178	589	64	177	7,372	12,325
TOTAL	17,103,736	4,906,070	4,273,293	544,221	1,526,347	63,597,776	91,951,443

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$2,000/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	2,938,412	2,938,412	-2,938,412	-
380.0 -> 377.5	239	574	1.35	21	42,244	1,056	1,690	2,746	2,872,805	2,875,551	-5,771,718	136,140
377.5 -> 375.0	55,224	131,205	1.44	5,153	10,306,847	257,671	412,274	669,945	6,106,453	6,776,398	-2,241,270	1,315
375.0 -> 372.5	189,610	440,954	1.39	16,729	33,457,686	836,442	1,338,307	2,174,750	15,363,640	17,538,390	13,678,027	1,048
372.5 -> 370.0	263,812	563,973	1.63	25,189	50,377,193	1,259,430	2,015,088	3,274,518	18,218,429	21,492,946	42,562,274	853
370.0 -> 367.5	300,417	461,475	1.87	23,566	47,131,493	1,178,287	1,885,260	3,063,547	15,305,788	18,369,335	71,324,432	779
367.5 -> 365.0	317,654	442,104	2.05	24,768	49,535,708	1,238,393	1,981,428	3,219,821	14,303,349	17,523,170	103,336,970	707
365.0 -> 362.5	262,587	313,891	2.58	22,126	44,252,730	1,106,318	1,770,109	2,876,427	10,458,977	13,335,405	134,254,295	603
362.5 -> 360.0	129,422	152,382	2.16	8,978	17,955,552	448,889	718,222	1,167,111	5,103,595	6,270,706	145,939,142	698
360.0 -> 357.5	29,425	33,176	1.47	1,332	2,663,005	66,575	106,520	173,095	1,130,774	1,303,870	147,298,277	979
357.5 -> 355.0	3,529	3,881	1.37	146	291,388	7,285	11,656	18,940	136,896	155,836	147,433,829	1,070
355.0 -> 352.5	268	295	1.29	10	20,842	521	834	1,355	12,325	13,680	147,440,991	1,313
TOTAL	1,552,187	2,543,911	1.84	128,017	256,034,688	6,400,867	10,241,388	16,642,255	91,951,443	108,593,697	147,440,991	848

APPENDIX 7 LOST DOG

\$1,650/oz "BASE CASE" OPTIMUM SHELL PLAN



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APPENDIX 8 LOST DOG PIT DESIGN EVALUATION SUMMARY

LOST DOG
PIT DESIGN SUMMARY
INCORPORATING 2% MINING DILUTION at 0.00g/t AND 98% MINING RECOVERY

	MINING RESE		WASTE	TOTAL	STRIPPING	MILL	OUNCES	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CASH COST		OPI	ERATING PRO	FIT	
VOLU	ME TONNAGI	GRADE	VOLUME	VOLUME	RATIO	RECOVERY	RECOVERED	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	per OUNCE	@ \$1,550/oz	@ \$1,600/oz	@ \$1,650/oz	@ \$1,700/oz	@ \$1,750/oz
(bcm) (t)	(g/t)	(bcm)	(bcm)	(bcm:bcm)	(%)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
1,506	001 2,469,84	7 1.8	7 2,933,292	4,439,293	1.9	85	126,259	5,208,204	10,100,759	15,308,963	91,102,186	106,411,150	843	89,606,711	95,761,862	101,917,012	108,072,162	114,227,312

PIT DESIGN EVALUATION INCORPORATING 2% MINING DILUTION AT 0.00g/t AND 98% MINING RECOVERY

		ORE		TOTAL	TOTAL	
FLITCH	VOLUME	TONNES	GRADE	WASTE	VOLUME	
	(BCM) (t)		(g/t)	(BCM)	(BCM)	
N/S -> 380.0	0	0	0.00	650,871	650,871	
380.0 -> 377.5	299	718	1.34	628,724	629,023	
377.5 -> 375.0	51,706	122,839	1.44	562,376	614,082	
375.0 -> 372.5	177,370	414,408	1.41	416,775	594,145	
372.5 -> 370.0	257,923	554,130	1.65	298,800	556,723	
370.0 -> 367.5	289,935	448,997	1.91	190,849	480,784	
367.5 -> 365.0	312,351	435,048	2.06	101,073	413,424	
365.0 -> 362.5	255,891	306,067	2.62	54,257	310,148	
362.5 -> 360.0	128,769	151,751	2.18	24,180	152,950	
360.0 -> 357.5	29,133	33,003	1.49	4,868	34,002	
357.5 -> 355.0	2,624	2,886	1.36	518	3,141	
TOTAL	1,506,001	2,469,847	1.87	2,933,292	4,439,293	

UNIT COSTS

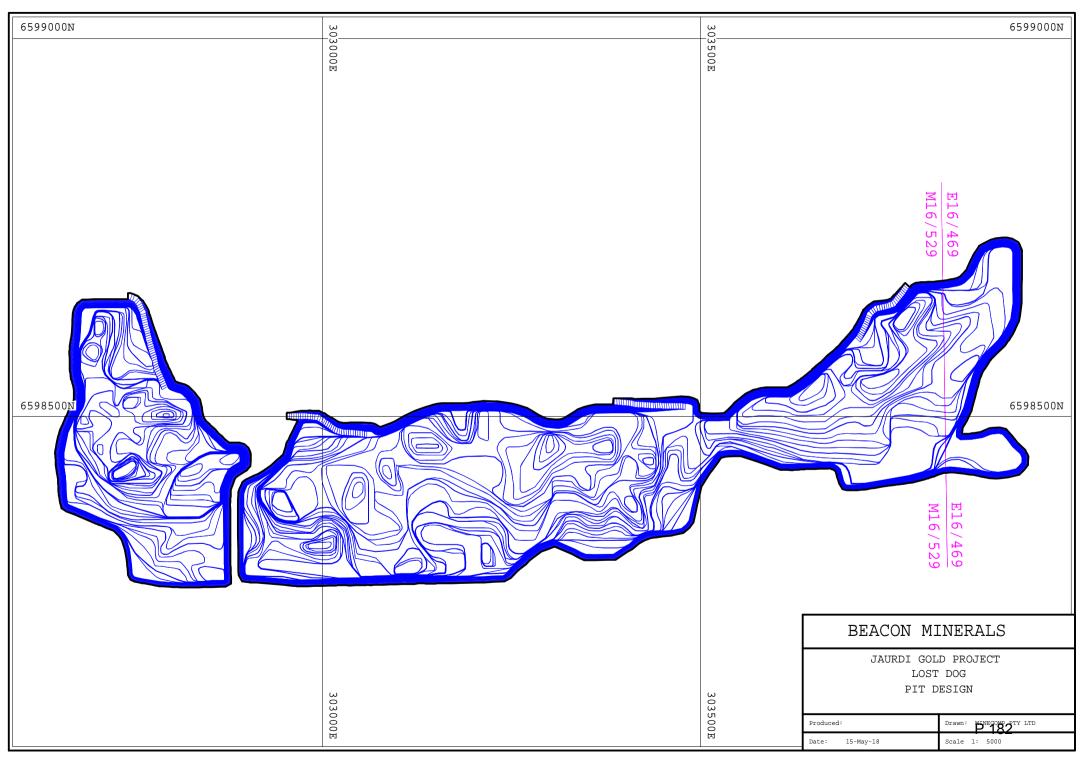
	MINING	BLASTING	MINING	REHAB	GRADE	ORE HAULAGE	
FLITCH			EXTRAS		CONTROL	& TREATMENT	
	(\$/BCM)	(\$/BCM)	(\$/BCM)	(\$/BCM Waste)	(\$/t ore)	(\$/t ore)	
N/S -> 380.0	3.50	0.00	1.00	0.20	0.60	25.00	
380.0 -> 377.5	3.50	0.00	1.00	0.20	0.60	25.00	
377.5 -> 375.0	3.50	0.00	1.00	0.20	0.60	25.00	
375.0 -> 372.5	4.00	2.00	1.00	0.20	0.60	25.00	
372.5 -> 370.0	4.00	2.00	1.00	0.20	0.60	25.00	
370.0 -> 367.5	4.50	2.00	1.00	0.20	0.60	25.00	
367.5 -> 365.0	4.50	2.00	1.00	0.20	0.60	25.00	
365.0 -> 362.5	5.00	2.00	1.00	0.20	0.60	25.00	
362.5 -> 360.0	5.00	2.00	1.00	0.20	0.60	25.00	
360.0 -> 357.5	5.00	2.00	1.00	0.20	0.60	25.00	
357.5 -> 355.0	5.00	2.00	1.00	0.20	0.60	25.00	

PRODUCTION COSTS

FLITCH	MINING	BLASTING	MINING EXTRAS	REHAB	GRADE CONTROL	ORE HAULAGE & TREATMENT	OPERATING COSTS	
	(\$)	(\$)	(\$)	(\$)	(\$)	(\$/t ore)	(\$)	
N/S -> 380.0	2,278,049	0	650,871	130,174	0	0	3,059,094	
380.0 -> 377.5	2,201,582	0	629,023	125,745	431	17,943	2,974,723	
377.5 -> 375.0	2,149,288	0	614,082	112,475	73,703	3,070,971	6,020,520	
375.0 -> 372.5	2,376,580	1,188,290	594,145	83,355	248,645	10,360,204	14,851,219	
372.5 -> 370.0	2,226,892	1,113,446	556,723	59,760	332,478	13,853,256	18,142,555	
370.0 -> 367.5	2,163,527	961,568	480,784	38,170	269,398	11,224,933	15,138,380	
367.5 -> 365.0	1,860,406	826,847	413,424	20,215	261,029	10,876,198	14,258,118	
365.0 -> 362.5	1,550,740	620,296	310,148	10,851	183,640	7,651,663	10,327,338	
362.5 -> 360.0	764,750	305,900	152,950	4,836	91,051	3,793,782	5,113,268	
360.0 -> 357.5	170,008	68,003	34,002	974	19,802	825,070	1,117,858	
357.5 -> 355.0	15,707	6,283	3,141	104	1,732	72,146	99,113	
TOTAL	17,757,528	5,090,632	4,439,293	586,658	1,481,908	61,746,167	91,102,186	

		MILLED ORE		OUNCES	REVENUE	STATE GOLD	3RD PARTY	TOTAL	OPERATING	TOTAL	CUMULATIVE	CASH COST
FLITCH	VOLUME	TONNES	GRADE	PRODUCED	@ \$1,650/oz	ROYALTY @ 2.5%	ROYALTY @ \$80/oz	ROYALTIES	COSTS	COSTS	CASH	PER OUNCE
	(BCM)	(t)	(g/t)	(oz)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)	(\$)
N/S -> 380.0	0	0	0.00	0	0	0	0	0	3,059,094	3,059,094	-3,059,094	-
380.0 -> 377.5	299	718	1.34	26	43,468	1,087	2,108	3,194	2,974,723	2,977,918	-5,993,544	-
377.5 -> 375.0	51,706	122,839	1.44	4,838	7,982,645		387,037	586,603	6,020,520	6,607,123	-4,618,022	1,366
375.0 -> 372.5	177,370	414,408	1.41	15,968	26,346,752		1,277,418	1,936,087	14,851,219	16,787,306	4,941,424	1,051
372.5 -> 370.0	257,923	554,130	1.65	25,049	41,330,942	1,033,274	2,003,924	3,037,198	18,142,555	21,179,753	25,092,612	846
370.0 -> 367.5	289,935	448,997	1.91	23,391	38,594,349		1,871,241	2,836,100	15,138,380	17,974,480	45,712,481	768
367.5 -> 365.0	312,351	435,048	2.06	24,544	40,498,152	1,012,454	1,963,547	2,976,001	14,258,118	17,234,118	68,976,515	702
365.0 -> 362.5	255,891	306,067	2.62	21,950	36,217,644	905,441	1,756,007	2,661,448	10,327,338	12,988,786	92,205,372	592
362.5 -> 360.0	128,769	151,751	2.18	9,043	14,920,233	373,006	723,405	1,096,411	5,113,268	6,209,679	100,915,926	687
360.0 -> 357.5	29,133	33,003	1.49	1,343	2,216,648	55,416	107,474	162,890	1,117,858	1,280,748	101,851,826	953
357.5 -> 355.0	2,624	2,886	1.36	107	177,330	4,433	8,598	13,031	99,113	112,144	101,917,012	1,043
TOTAL	1,506,001	2,469,847	1.87	126,259	208,328,162	5,208,204	10,100,759	15,308,963	91,102,186	106,411,150	101,917,012	843

APPENDIX 9 LOST DOG PIT DESIGN PLAN



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APPENDIX 10 JAURDI GOLD PROJECT LOST DOG ORE RESERVE STATEMENT



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Email: administrator@minecomp.com.au
Minecomp Pty Ltd as trustee for the Minecomp Unit Trust

www.minecomp.com.au

24th August 2018 The Directors Beacon Minerals Limited PO Box 1305 West Leederville Western Australia 6901

RE: ORE RESERVE STATEMENT LOST DOG PROSPECT - JAURDI GOLD PROJECT

Minecomp Pty Ltd (Minecomp) has been commissioned by Beacon Minerals Limited (BCN) to produce a 2018 Reserve Statement for Lost Dog which forms part of the Jaurdi Gold Project in Western Australia.

The statements and opinions in this Report are given in good faith and this Report is based upon information provided by BCN, along with technical reports prepared by consultants and other relevant published and unpublished data for the area.

The Ore Reserves for Lost Dog are estimated, using a gold price of Au\$1,650/oz, to be:-

Ore Reserve Category	Tonnes	Au (g/t)	Au (oz)
Proved	27 ,000	1.6	1,400
Probable	2,443,000	1.9	147,100
Total	2,470,000	1.9	148,500

Note: - Rounding errors may occur

The classification of Lost Dog Ore Reserve has been carried out in accordance with the recommendations of the JORC Code 2012. It is based on the density of drilling, estimation methodology and the mining method to be employed.

All Proved and Probable Ore Reserves have been derived from Measured and Indicated Mineral Resources.

The Information in this Report that relates to Ore Reserves is based on information compiled by Mr Gary McCrae, a Competent Person who is a Member of the Australasian Institute of Mining and Metallurgy. Mr McCrae is a full-time employee of Minecomp Pty Ltd. Mr McCrae has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr McCrae consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

BEACON MINERALS LIMITED ACN 119 611 559

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Key physical parameters from the PFS include:-

PHYSICALS	UNIT	QUANTITY
Life of Mine	years	5
Ore Mined	(kt)	2,470
Ore Grade	(g/t)	1.9
Contained Gold	(koz)	148.5
Metallurgical Recovery	(%)	85
Gold Recovered	(koz)	126.3
Stripping Ratio	(bcm:bcm)	1.9
Total Volume	(Mbcm)	4.4
Maximum Pit Depth	(m)	32.5

Note: - Rounding errors may occur

Key financial parameters from the PFS include:-

PROJECT ECONOMICS	UNIT	QUANTITY
Gold Price	(A\$/oz)	1,650
Revenue	(A\$M)	208.3
Operating Costs	(A\$M)	89.5
Royalties	(A\$M)	15.3
Free Cashflow	(A\$M)	103.5
Capital Costs - Start-up	(A\$M)	21.4
Capital Costs - Sustaining	(A\$M)	5.0
C1 Cash Cost	(\$/oz)	830
AISC	(\$/oz)	870
Payback	(months)	11

Note: - Rounding errors may occur

SUMMARY OF KEY TECHNICAL ELEMENTS IN PFS

PFS Overview

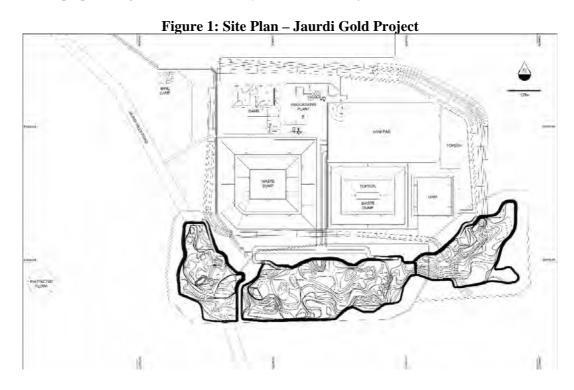
The Company engaged Minecomp Pty Ltd ("Minecomp"), a Kalgoorlie based company, to carry out the PFS at the Jaurdi Gold Project ("Jaurdi Project"), producing a high level mining and processing schedule.

The Jaurdi Project is located 35km north west of Coolgardie and approximately 75km west of Kalgoorlie. The area is well serviced by infrastructure including a network of high quality roads, Kalgoorlie airport with regular services to Perth and an established mining supply network.

The PFS investigates the potential economic viability of the Jaurdi Project on the mining and onsite treatment of the Lost Dog Resource

Independent JORC 2012 estimates of the Mineral Resource at the Jaurdi Project by BM Geological Services (BMGS) total 2.88Mt @ 1.8g/t for 163.1koz of contained gold (refer ASX Announcement 12th July 2017).

The PFS envisages an open pit mine that will deliver material to a new, 500,000tpa capacity carbon-in-pulp (CIP) gold treatment facility at the Jaurdi Project.



The open pit will be mined utilizing conventional open pit methods with hydraulic excavator, a fleet of off-road dump trucks and ancillary mining equipment.

The mining strategy is focused on delivering an appropriate blend of ore to the process plant so as to optimise plant recoveries and throughput.

The TSF strategy is based upon depositing tailings into the voids left by open pit mining.

A 32 person accommodation camp has been constructed on site at the Jaurdi Project.

First gold production, based upon the PFS production forecast, is expected in the first half of 2019.

The following results constitute work carried out by Gary McCrae from Minecomp. All outputs relating to these works are dated May 2018.

Mineral Resource

The Jaurdi Project overlies a portion of the Mali Monzo granite immediately adjacent to the Jaurdi Hills-Dunnsville greenstone sequence. The gold mineralisation is hosted in either a bleached, siliceous siltstone or an interbedded clay and siltstone unit.

The Mineral Resource was estimated by BMGS to be:-

Table 1 – Lost Dog Mineral Resource

Classification	Tonnes	Au (g/t)	Au	
Classification	(Kt)	(g/t)	(kOz)	
Measured	30	1.6	1.5	
Indicated	2,752	1.8	158.4	
Inferred	101	1.0	3.2	
Total	2,883	1.8	163.1	

Calculations have been rounded to the nearest 1,000t, 0.1g/t grade and 100 ounces
For further details see JORC Code 2012 Edition – Table Report Template Sections 1, 2, 3 at conclusion of this report.

Mining and Metallurgical Factors and Assumptions

Mining at the Jaurdi Project is to be performed using conventional open pit mining techniques. Mining equipment will comprise articulated dump trucks, matching 90t hydraulic excavator and additional ancillary equipment rounding out the fleet.

Beacon envisages that load and haul activities will be undertaken by owner miner operators using a mixture of owned and dry hired equipment. All drill and blast and grade control drilling will be undertaken by contractors. All technical and managerial direction will be governed by Beacon.

The Lost Dog, June 2017 Resource was imported into Whittle pit optimisation software. The optimisation analysis included inputs from Beacon's Executive Directors and external consultants. These input parameters comprised contractor estimates based upon experience and were inclusive of all on-site operating costs. Where applicable these costs, were reflective of the use of articulated trucks and matching equipment. Milling costs were reflective of treatment at an on-site milling facility.

The metallurgical recovery used in this study is based upon testwork conducted by ALS Metallurgy Perth, Bureau Veritas Kalgoorlie and the results of a 4,625t trial parcel of Lost Dog ore processed at a nearby custom milling facility. The 85% recovery used is at the lower end of the range of recoveries established from the testwork.

Geotechnical parameters utilised were based upon the recommendations of Tim Green of Green Geotechnical.

The orebody geometries (shallow, flat lying and nominally 1,200m long, 150m wide and 12m thick) resulted in the application of a mining dilution factor of 2% at 0.00g/t and a mining recovery of 98%. Given these orebody dimensions no allowances were made for minimum mining widths.

Optimisation analysis was conducted for a gold price range of A\$1,000/oz to A\$2,000/oz in A\$50/oz increments, with \$1,650/oz considered to be the "Base Case" gold price.

Inferred Resources were assigned a grade of 0.00g/t and hence categorized as waste material throughout the course of this study.

A state royalty of 2.5% is payable on the average monthly price as advised by the DMIRS Royalties Branch. No allowance has been made for the exemption of this royalty on the first 2,500 ounces produced in each financial year.

A third party royalty of \$80/oz recovered is also payable.

Mine Design and Ore Reserve

Open pit mining methods are well known and widely used in the local mining industry. The design was focused on maximizing profitability from the optimised Whittle shells. The optimum and most profitable outcome were to design the pit ramp to single lane at a 1 in 6 gradient which suited the 40t articulated dump truck fleet. This ramp configuration being one which Beacon management has significant historical exposure to.

The detailed open pit mine design produces a Maiden Ore Reserve of:-

Table 2 – Jaurdi Project Ore Reserve

Ore Reserve Category	Tonnes	Au (g/t)	Au (oz)
Proved	27,000	1.6	1,400
Probable	2,443,000	1.9	147,100
Total	2,470,000	1.9	148 500

Notes

Calculations have been rounded to the nearest 1,000t, 0.1g/t grade and 100 ounces
For further details see JORC Code 2012 Edition – Table Report Template Sections 1, 2, 3 and 4 at conclusion of this report.

For the purpose of the Ore Reserve Estimate, a marginal cut-off grade of 0.6g/t was calculated based upon an assumed gold price of Au\$1,650/oz and the applicable Western Australian State Government and 3^{rd} Party Royalties, ore/waste cost differentials, processing and haulage costs and metallurgical recovery.

Ore Processing

Beacon have acquired many of the key processing components for the construction of a 500,000tpa processing plant. Major equipment acquired to date is as follows:-1

- SAG mill 1500kW, 4m x 6m;
- Ball mill 450kW, 3m x 4m;
- Adsorption tanks 6 x 200m³
- Leach tank agitators and superstructure to suit 3 x 630m³ tanks;
- Radial stacker 35m:
- Coarse ore bin and feeder:
- MCC switch rooms and
- Process slurry pumps.

The process plant general arrangement is shown in Figure 2 and the process flow diagram for the 0.5Mtpa processing plant is illustrated in Figure 3. All main elements that comprise the processing plant are typical of conventional CIP plants operating throughout the WA Goldfields. The treatment circuit has been designed to produce a grind P100 106µm and a leach retention time of 15 hours. The Company has made a financial provision for additional Leach Tanks if required.

SOM

CYANGE

CYANGE

TANK

CHARGE

CHARGE

STEACHER

STEACHER

CHARGE

WASTE DUMP

WASTE DUMP

WASTE DUMP

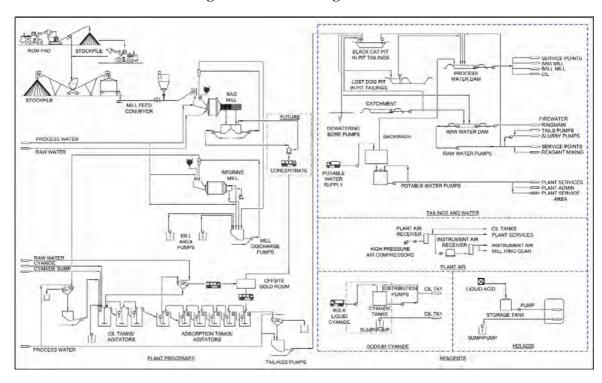
WASTE DUMP

WASTE DUMP

WASTE DUMP

Figure 2: Process Plant Arrangement

Figure 3: Process Design Flowsheet



Tailings Storage Facility (TSF)

The TSF strategy is based on backfilling the void left by open pit mining. The open pit will be mined in panels and engineered retaining walls will be constructed to provide tailings disposal cells. Initially the Black Cat Pit will be utilised as a tailings facility until Panel 1 of the Lost Dog open pit has been prepared. The estimated tailings capacity of the Black Cat and Lost Dog open pits is 5,000,000 tonnes.

Production Target

The detailed open pit mine design has been used to schedule a potential production profile for the Jaurdi Project.

Table 3 – Jaurdi Project Design Physicals

MINING	TOTAL	STRIPPING	OUNCES	C1 - CASH COST
RESERVE	VOLUME	RATIO	RECOVERED	per OUNCE
2.47 Mt @ 1.9g/t	4.44 Mbcm	1.9:1	126,300	830

A simplified, high level global scoping level production schedule based upon the open pit mine design physicals has been completed for the Jaurdi Project. The main constraint applied to the production schedule is the 500,000tpa capacity of the processing plant. The maximum pit depth (32 metres) and the low strip ratio (average 1:1.9) enables the mining and processing schedules to be run in parallel which minimises the working capital expense.

Table 4 – Production Target Schedule

	8										
PHYSICALS	UNIT	YEAR 0	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	TOTALS			
Mined Volume - Total	(bcm)	400,000	640,000	640,000	760,000	900,000	1 099 293	4,439,293			
Mined Tonnage - Ore	(t)	0	500,000	500,000	500,000	500,000	469,847	2,469,847			
Mined Grade	(g/t)	0.00	2.03	2.03	1.58	1.87	1.84	1.87			
Processing Input Tonnage	(t)	0	500,000	500,000	500,000	500,000	470,000	2,470,000			
Recovered Gold Ounces	(o z)	0	27,738	27,738	21,589	25,552	23,642	126 259			

Figure 4: Gold Production

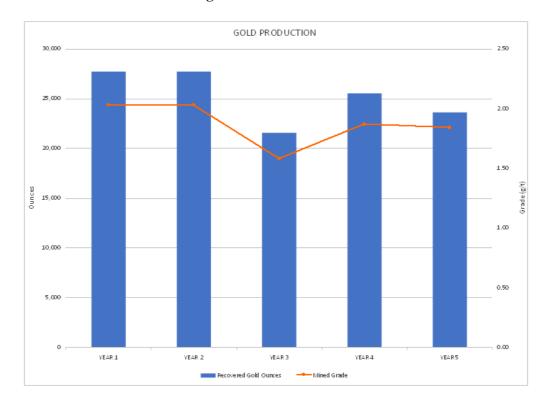
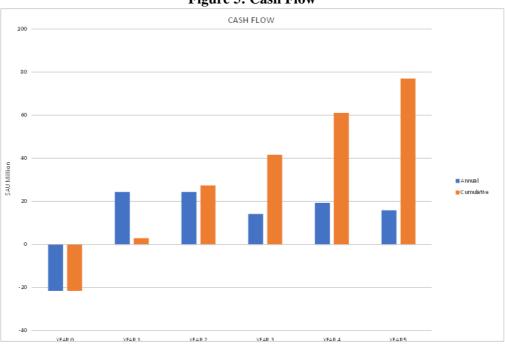


Figure 5: Cash Flow



Capital Costs

Capital costs have been estimated as follows:-

Table 5 – Capital Costs

CAPITAL COSTS	UNIT	QUANTITY
Expenditure to Date	(A\$M)	5.6
Processing Plant Construction	(A\$M)	14.0
Waste Pre-strip	(A\$M)	1.8
Sustaining Capital	(A\$M)	5.0
TOTAL	(A\$M)	26.4

Financial Analysis

A high level financial analysis was undertaken on the Jaurdi Project using cost inputs provided by Beacon and work undertaken for this mining study. Upfront capital of AU\$21.4M (\$5.6m actual expenditure to date and \$15.8m estimated pre-production construction) was included in the financial analysis to account for the acquisition, relocation and refurbishment of a second hand processing plant with a nominal 0.5Mtpa throughput and other Project start-up costs. Sustaining capital of AU\$1.0/year was also included.

Table 6 – Financial Analysis Summary

FINANCIALS	UNIT	YEAR 0	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	TOTALS
Capital Costs	(A\$M)	21.4	1.0	1.0	1.0	1.0	1.0	26.4
Mining Cost	(A\$M)	0.0	3.5	3.5	4.2	5.0	6.1	22.2
Grade Control Cost	(A\$M)		0.3	0.3	0.3	0.3	0.3	1.5
Variable Processing Cost	(A\$M)	0.0	12.5	12.5	12.5	12.5	11.7	61.7
General and Administration Cost	(A\$M)	0.0	0.6	0.6	0.8	0.9	1.1	4.0
Revenue	(A\$M)	0.0	45.8	45.8	35.6	42.2	39.0	208.3
Royalties	(A\$M)	0.0	3.4	3.4	2.6	3.1	2.9	15.3
Cashflow	(A\$M)	-21.4	24.4	24.4	14.3	19.4	15.9	77.1

Pre-production Activities

Pre-production activities at Jaurdi would include the following;

- Identification and development of a process water borefield (completed);
- Mining fleet mobilisation (completed);
- Construction of offices, workshops/store and camp (completed);
- Clearing, grubbing and stockpiling of top-soil and wood mulch (patially completed);
- The construction of a processing plant (on-going);
- Preparation of the ROM pad (patially completed); and
- Construction of the TSF tailings line and decant water line to the Black Cat open pit (on-going).

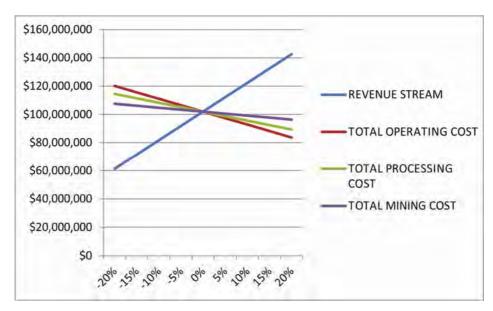
Sensitivity on Material Assumptions

A series of optimisation analyses, testing Ore Reserve sensitivity to Gold Price were performed by Minecomp using a financial model developed for owner operated mining and ore processing.

Further sensitivity testing was performed on the open pit Ore Reserve. The parameters tested for sensitivity were:-

- Revenue Stream (Gold Price, Metallurgical Recovery or Both)
- Total Operating Costs
- Processing Costs
- Mining Costs

Figure 6: Ore Reserve Sensitivty (Exclusive of Capital)



Project Finance

The financing required to acquire, explore, construct and commission the Jaurdi Gold Project is as follows:

- Actual total expenditure to date \$5.6M;
- Estimated pre-production construction \$15.8M.

Preparation for final project financing is advanced and the Company will provide further detail in the near future.

Risks and Opportunities

Key risks identified during the PFS work include, but are not limited to:

- Access to project funding;
- Adverse movements in the United States gold price;
- Adverse movements in the USD:AUD exchange rates; and
- Not achieving the processing production rates and metallurgical recovery rates.

Key opportunities identified during the PFS work include, but are not limited to:

- Achieving higher mill throughput rates. The installed SAG Mill power is in excess of the power requirements for a 500ktpa plant installation;
- Improved metallurgical recovery and
- Expansion of the resource base via exploration success and/or acquisitions.



JORC Code, 2012 Edition – Table 1 report – Jaurdi Gold Project: June 2017 Mineral Resource Update

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	specialised industry standard measurement tools appropriate to the minerals	The sampling of drill cuttings has been carried out on Reverse Circulation (RC), Aircore (AC), Rotary Air Blast (RAB) and Diamond Core (DD) drilling. The database to build the June 2017 Mineral Resource has a total 211 AC drill holes, 6 DD drill holes, 6 RAB drill holes and 306 RC drill holes. These holes are drilled on M16/529 and E16/469.
	Include reference to measures taken to ensure sample representation and the appropriate calibration of any measurement tools or systems used.	The drill hole collar locations were surveyed by DGPS using Kalgoorlie based registered surveyors of Minecomp Pty Ltd. Sampling was carried out under Beacon's protocols and QAQC procedures as per industry best practice. See further details below.
	Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases, more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	The RC holes were drilled using a 138mm face-sampling bit. One metre samples were collected through a cyclone and split through a rig mounted riffle splitter. An increased clay content was encountered in latter programs and as a result, a cone splitter was utilised. A 25% split was used to produce a sample size of approximately 3-4kg per metre for both splitters (riffle and cone). AC holes were drilled using an 89mm face-sampling bit and a cone splitter was used to collect a 3 to 4 Kilogram sample. Diamond core was quarter cut to geological intervals using an Almonte core saw. All samples were dried at 110 degrees Celsius and pulverised at the lab to -75um, to produce a 50g charge for Fire Assay with an AAS finish.
Drilling techniques	Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	Multiple rigs have been used including Ausdrill Ltd's DRA GC600 rig and Raglan Drilling's Schramm T685W. Both rigs utilised a 138mm diameter face sampling bit. The diamond core drilling was completed by Westralian Diamond Drillers, using PQ Triple Tube equipment.

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Drill sample recovery	Method of recording and assessing core and chip sample recoveries and results assessed.	Ground water ingress occurred in some holes at rod change, but overall the holes were kept dry. Typically, drilling operators ensured water was lifted from the face of the hole at each rod change to ensure water did not interfere with drilling and to make sure samples were collected dry. RC recoveries were visually estimated, and recoveries recorded in the log as a percentage. Recovery of the samples was good, generally estimated to be full, except for some sample loss at the collar of the hole.
	Measures taken to maximise sample recovery and ensure representative nature of the samples.	RC face-sample bits and dust suppression were used to minimise sample loss. Drilling airlifted the water column above the bottom of the hole to ensure dry sampling. RC samples are collected through a cyclone and then split to capture a 3 to 4 Kg sample. PQ diamond core resulted in exceptional recovery (96%), given the state of weathering and material type. Core was immediately placed in plastic wrap and transported to the BCN yard in Kalgoorlie.
	Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	No relationship between recovery and grade has been identified.
Logging	Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.	All chips and drill core were geologically logged by experienced industry geologists, using the Beacon Minerals geological logging legend and protocol.
	Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	Logging of RC chips and drill core records lithology, mineralogy, mineralisation, weathering, colour and other features of the samples. All RC and AC samples are wet-sieved and stored in a chip tray. Diamond core which was not used for resource and metallurgical sampling is stored in Beacon's yard in Kalgoorlie, WA.
	The total length and percentage of the relevant intersections logged	All holes were logged in full.
Sub-sampling techniques and	If core, whether cut or sawn and whether quarter, half or all core taken.	Not relevant, as reporting non-core.
sample preparation	If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	Samples have been collected through either a rig mounted riffle or cone splitter. Results of the two splitting techniques were analysed, with no disparities between the two evident. The majority of samples were kept dry, with some wet samples produced at rod change.



	For all sample types, the nature, quality and appropriateness of the sample preparation technique.	Samples were prepared at either the ALS or SGS Laboratory in Kalgoorlie. Samples were dried, and the whole sample pulverised to 90% passing -75um, and a sub-sample of approx. 200g retained. A nominal 50g was used for the fire assay analysis. The procedure is industry standard for this type of sample.
	Quality control procedures adopted for all sub-sampling stages to maximise representation of samples.	A CRM standard, fine blank and field duplicate was submitted at a rate of approximately 1 in 27 samples. At the laboratory, regular Repeats and Lab Check samples are assayed.
	Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	The technique to collect the one metre samples was via a rig mounted riffle or cone splitter. Both splitters were routinely inspected by the field geologist. Field duplicates were collected and results were satisfactory, suggesting the duplicate field samples replicated the original samples.
	Whether sample sizes are appropriate to the grain size of the material being sampled.	Sample sizes are considered appropriate to give an indication of mineralisation given the particle size and the preference to keep the sample weight at a targeted 3 to 4kg mass.
Quality of assay data and laboratory tests	The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	The analytical method used was a 50g Fire Assay with AAS finish for gold. The technique is considered to be appropriate for the material and style of mineralization.
	For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	The assaying and laboratory procedures used are industry standard and are appropriate for the material tested.
	Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	Beacon Minerals protocol for the 2017 RC/AC/DD drilling programs was for a single CRM (Certified Reference Material), fine blank and field duplicate to be inserted in every 90 samples. This is at a rate of approximately 1 QA/QC sample per 30 regular samples. At the ALS and SGS Laboratories, regular assay Repeats, Lab Standards and Blanks are analysed.
		Results of the Field and Lab QAQC were analysed on assay receipt. On analysis, all assays passed QAQC protocols, showing no levels of contamination or sample bias. Analysis of field duplicate assay data suggests appropriate levels of sampling precision have been achieved for the sampling technique employed.



Verification of sampling and assaying	The verification of significant intersections by either independent or alternative company personnel.	Significant results were checked by Beacon Minerals executives and BMGS senior geologists.
ussuymy	The use of twinned holes.	Beacon have twinned historical holes in the early 2017 drilling programs and were satisfied the historical holes repeated within reason and are representative of the mineralisation.
	Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	All field logging is carried out using a customised logging form on a Tough Book and transferred into an Access database. Assay files are received electronically from the Laboratory. All data is stored in the Jaurdi Gold Project Access database and managed by BMGS in Perth.
	Discuss any adjustment to assay data.	No assay data was adjusted.
Location of data points	Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	RC hole collar locations were surveyed by a registered Surveyor. The group used was the Kalgoorlie based Minecomp Pty Ltd. BMGS believe the accuracy and quality of the surveys meet industry standard. Beacon employed ABIM Solutions to undertake down hole surveying on the first drilling program in January 2017. The results of the north seeking gyro surveys demonstrated no significant deviation of the shallow holes (25 to 30 metres). Down hole surveying of subsequent holes was not considered necessary.
	Specification of the grid system used.	Grid projection is MGA94, Zone 51.
	Quality and adequacy of topographic control.	Minecomp Pty Ltd has completed a topographic survey over the lease picking up the two shallow pits on the Mining Lease and a suite of historical holes.
Data spacing and distribution	Data spacing for reporting of Exploration Results.	Resource definition drilling was completed at a regular spacing of 25m x 50m; and grade control drilling was completed on either a 10m x 10m or 12.5m x 12.5 m spacing.
	Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	This spacing is sufficient to test the continuity of mineralisation for this style of mineralisation.
	Whether sample compositing has been applied.	All RC samples collected were 1 metre composites.
	I .	I .



Orientation of	Whether the orientation of sampling achieves unbiased sampling of possible	It is considered the orientation of the drilling and sampling suitably captures the "structure" of
data in relation	structures and the extent to which this is known, considering the deposit type.	the palaeochannel style of mineralisation.
to geological		
structure	If the relationship between the drilling orientation and the orientation of key	This is not considered material.
	mineralised structures is considered to have introduced a sampling bias, this	
	should be assessed and reported if material.	
Sample security	The measures taken to ensure sample security.	Samples were transported by company transport to the ALS and SGS laboratories in Kalgoorlie.
Audits or	The results of any guidite or regions of compline techniques and data	Compling and account to building are industry standard. Descen have had the layed database
	The results of any audits or reviews of sampling techniques and data.	Sampling and assaying techniques are industry-standard. Beacon have had the Jaurdi database
reviews		reviewed by a second geological consultant (Kaldera Pty Ltd) who concluded the geological,
		survey and QAQC data collected during the Beacon drill campaigns meets industry standard.

Section 2 Reporting of Exploration Results

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

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.	The RC and Diamond drilling has been within tenements M16/529 and E16/469, of which BCN holds a 100% controlling interest.
	The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	The tenements are in good standing with the WA DMIRS.
Exploration done by other parties	Acknowledgment and appraisal of exploration by other parties.	There have been three campaigns of drilling undertaken on this lease by third parties; previously a suite of Prospecting Licenses. The early phase was completed by a private firm called Coronet Resources in 2007. A second phase of drilling was completed by a group of "prospectors", the program being supervised by BM Geological Services in 2009. A report was produced outlining an unclassified resource. The third phase of drilling was commissioned by Fenton and Martin Mining Developments in 2015. BCN has since completed four exploration

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		and two grade control campaigns on the tenements.
Geology	Deposit type, geological setting and style of mineralisation.	The Jaurdi Gold Project overlies a portion of the Bali Monzogranite immediately adjacent to the Jaurdi Hills-Dunnsville greenstone sequence. The Bali Monzogranite and Dunnsville Granodiorite to the north, together occupy the core of the gently north plunging anticline. The tenement making up the project is located to the west of the anticlinal axis and immediately adjacent to the granite-greenstone contact.
		The Bali Monzogranite is poorly exposed. The greenstone-granite contact is foliated where exposed. Shear zones developed locally within the adjacent greenstones, may continue within the granite. Gold mineralised palaeochannels are known in the Jaurdi area.
		Regional magnetic data suggest that the western portion of the project lies within a broad demagnetised corridor following the western contact of the Bali Monzogranite, and which may continue in a north northwest direction through the greenstone sequence to Dunnsville. A magnetic dyke, akin to the Parkeston dyke in the Kalgoorlie area, has intruded this corridor. Another paired east northeast magnetic dyke set is located immediately to the south of the project area. This dyke set is part of the regionally extensive Widgiemooltha Dyke Suite, and passes to the north of Kalgoorlie-Boulder.
		The Jaurdi Gold Project is located close to the western margin of the Bali Monzogranite immediately to the south east of the exposed Jaurdi Hills greenstone sequence. The tenement is entirely soil covered, with well-developed nodular carbonate increasing in intensity southwards towards an active contemporary drainage.
		Recent drilling programs have revealed the known soil anomaly overlies an extensive system of Au-bearing sand channels indicating that a major long-lived palaeoalluvial system was present in the area. A typical profile consists of transported lateritic gravels overlying plastic clay zones, which in turn overly thick, water saturated silt and clay sequences with minor cobble layers. Drilling evidence suggests that younger, perched channels over older channels, indicating that an anastomosing series of palaeochannels are present over an east-west distance of at least 1,450 metres. Two horizons of mineralisation have been identified in the Western Arm with
		the shallower lode situated between 12 to 16 metres vertical depth, and the second horizon between 18 to 25 metres. The Eastern Arm has been identified by a system which is at least



		1,450 metres strike (East – West orientated), 180 metres wide and 8 metres deep; and appears open to the North-East and connects with the Western Arm.
Drill hole Information	 A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: easting and northing of the drill hole collar elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar dip and azimuth of the hole down hole length and interception depth hole length. If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case. 	The drill results used to build the June 2017 Mineral Resource have been reported in ASX announcement dated the 12 July 2017 Jaurdi Gold Project — Lost Dog Mineral Resource Update.
Data aggregation methods	In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.	Grades are reported as down-hole length-weighted averages of grades above approximately 0.5 ppm Au. No top cuts have been applied to the reporting of the assay results. Intercepts averaging values significantly less than 0.2 g/t Au were assigned the text "NSI" (No Significant Intercept). Intercepts with minimal mineralisation that are located within the delineated ore body (internal dilution) were reported with intercept metres and grade.
	Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.	Higher grade intervals are included in the reported grade intervals.
	The assumptions used for any reporting of metal equivalent values should be clearly stated.	No metal equivalent values are used.
Relationship between mineralisation widths and	These relationships are particularly important in the reporting of Exploration Results. If the geometry of the mineralisation with respect to the drill hole angle is known, its	The geometry of the mineralisation has been well established by the recent drilling. There is no ambiguity with the geometry of this relatively simple alluvial system.



intercept lengths	nature should be reported.	
g	If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	
Diagrams	Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	Sections and maps representing the Lost Dog orebody have been reported in ASX announcement dated the 12 July 2017 Jaurdi Gold Project – Lost Dog Mineral Resource Update.
Balanced reporting	Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	No misleading results have been presented in this announcement.
Other substantive exploration data	Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	There is no other meaningful and/or materials exploration data.
Further work	The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	 Pre-production activities at Jaurdi would include the following; Identification and development of a process water borefield (completed); Mining fleet mobilisation (completed); Construction of offices, workshops/store and camp (completed); Clearing, grubbing and stockpiling of top-soil and wood mulch (partially completed); The construction of a processing plant (on-going); Preparation of the ROM pad (partially completed); and Construction of the TSF tailings line and decant water line to the Black Cat open pit (on-going).



Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	Database inputs were logged electronically at the drill site and at the BCN Kalgoorlie yard for the diamond core. The collar metrics, assay, lithology and down-hole survey interval tables were checked and validated by numerous staff of BMGS and Beacon Minerals.
	Data validation procedures used.	Validation occurs when the Geologist uses updated Access extracts to both plot and visually inspect.
Site visits	Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	Mr. Finch was on-site throughout Stage 1 & 2, as well as the conclusion of the diamond program. A BMGS Senior Geologist provided daily supervision of the diamond drill program. An Independent Geologist was on-site throughout the Stage 4 and Stage 5 RC program's. Mr. Mapleson is based out of the BMGS Kalgoorlie office and oversaw the various drilling campaigns.
	If no site visits have been undertaken indicate why this is the case.	Not Applicable, as above a site visit has been undertaken.
	Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.	Consistent logging of the lithology has correlated well with resultant assay values. A distinct correlation was identified between gold mineralisation and the presence of a heavily silicified siltstone and clay units.
Geological	Nature of the data used and of any assumptions made.	The geological model developed for the Lost Dog deposit is based predominantly on percussion holes. It is well supported by the diamond core holes drilled into the deposit and the mapping of the two trial pits.
interpretation	The effect, if any, of alternative interpretations on Mineral Resource estimation.	RC, AC and diamond drilling data has been used in the estimation. Aerial photography and geological logging were used to aid the interpretation.
	The use of geology in guiding and controlling Mineral Resource estimation	Fundamental palaeochannel characteristics were identified, confirming the style of mineralisation.
	The factors affecting continuity both of grade and geology.	No known factors have been identified to influence grade and/ or geological continuity of the deposit.

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Criteria	JORC Code explanation	Commentary
Dimensions	The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	The Eastern Arm of mineralisation extends 1,450m along strike, 180m in width, is an average of 8m thick and is at average of 10m below the natural surface. The Western arm of mineralisation extends 250m along strike, 140m in width, is an average of 7m thick and is at an average of 10m below the natural surface. A third domain exists as a low-grade repetition of mineralisation, below the central regions of the main ore horizon. Dimensions of the third domain are 230m in length, 80m in width and 2m thick.
	The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	Grade estimation was completed via ordinary kriging (OK) for the two main ore domains and Inverse-distance-squared (ID²) techniques for the smaller low-grade domain. A nested spherical variogram with two structures was derived for each OK domain using Snowden Supervisor software. The variogram was created as normal scores and was back transformed for use with 3DS Surpac modelling software. Nil assumptions were made. Three domains were created, based on variable grade distribution and orientation of mineralisation.
Estimation and modelling techniques	The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data	A statistical analysis was undertaken, with nil extreme or outlier gold grades identified. A low coefficient of variation value exists with all domains. Nil by-products have been identified.
	Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	Nil deleterious elements have been identified.
	In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	Block size was determined via a kriging neighbourhood analysis (KNA), using Snowden Supervisor software. A series of checks are used to confirm the block size to be being geologically suitable.
	Any assumptions behind modelling of selective mining units.	The selective mining unit (SMU) was developed based on open-pit mining using a 90 tonne backhoe excavator.



Criteria	JORC Code explanation	Commentary
	Any assumptions about correlation between variables.	Nil assumptions were made regarding correlation between variables
	Description of how the geological interpretation was used to control the resource estimates.	The palaeochannel is orientated essentially in an east-west orientation. Variograms and search ellipses were orientated to reflect this geometry
	Discussion of basis for using or not using grade cutting or capping.	A statistical analysis was undertaken for determination of a Gold top-cut for each domain. Grade distribution was determined to be homogenous; as a result, a top-cut was not required.
	The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	A previous 2009 resource estimate by BMGS was used as a check, as well visual checks and a series of swath validation plots that spatially compare block grades to raw composite data. Nil reconciliation data was available.
Moisture	Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	Tonnage has been estimated on a dry basis. Moisture values were obtained from diamond core analysis. The diamond core samples were weighed prior to a wax immersion SG analysis. After the analysis, the samples were dried and re-weighed to obtain a moisture value.
Cut-off parameters	The basis of the adopted cut-off grade(s) or quality parameters applied.	A suite of cut-off grades was presented for a scoping study. 0.5g/t Au was selected as the optimal cut-off grade.
Mining factors or assumptions	Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	The assumption of open-pit mining, using a 90 tonne backhoe excavator was used. Minimal mining dilution is expected due to the simplicity and orientation of mineralisation.
Metallurgical factors or	The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for	Detailed metallurgical analysis has been completed by Beacon and has been factored into the economics of the deposit.
assumptions	eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	No deleterious elements have been identified by Beacon in their metallurgical studies.



Criteria	JORC Code explanation	Commentary
Environmental factors or assumptions	Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	Waste material will be dumped on a surface dump and also back-filled into completed sectors of the open-pit. The location of an ore treatment facility has been included in the Mining Proposal and the site has been approved by DMIRS. A detailed environmental study has been undertaken by Beacon and has been approved by DMIRS.
Bulk density	Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.	Dry bulk density was determined by Bureau Veritas Kalgoorlie via a wax immersion SG analysis of diamond core representing different rock units from a variety of locations within the zone of mineralisation. A wet SG was determined by the analysis, before the calculated moisture values were applied to obtain a dry SG, which has been applied to the Lost Dog model as a bulk density. Rock Unit Wet SG Avg Moisture % Dry SG Siltstone 2.45 2% 2.4 Siltstone/Claystone 1.80 25% 1.4 Claystone/Siltstone 1.69 31% 1.2 Claystone 1.65 35% 1.1
	Discuss assumptions for bulk density estimates used in the evaluation process of the different materials. The basis for the classification of the Mineral Resources into varying confidence cottogories.	A down-hole density analysis has provided additional correlation with wet SG data from analysis of the Diamond core. Resource classification as Measured, Indicated or Inferred was based on drill-hole density and
Classification	whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the	grade continuity between drill holes. Data integrity has been analysed and a high level of confidence has been placed on the dataset and resultant resource estimation.



Criteria	JORC Code explanation	Commentary
	data).	
	Whether the result appropriately reflects the Competent Person's view of the deposit.	Mr. Finch and Mr. Mapleson retain a high degree of confidence in the result of the resource estimation.
Audits or reviews	The results of any audits or reviews of Mineral Resource estimates.	An independent audit of the entire resource estimation process was undertaken during May 2017, with all parameters and methodology reported as suitable and representative of the deposit.
Discussion of relative accuracy/ confidence	Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.	Excellent correlation between the resource estimate, the statistical analysis of composite data, metrics of a 2009 resource estimation and third-party small scale mining observations on the lease has resulted in a high level of confidence of the estimation on a global scale.
	The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	
	These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	

JORC Section 4 - Estimation and Reporting of Ore Reserves

Criteria	JORC Code explanation	Commentary
Miner Resour	Ure Reserve.	The Mineral Resource for the Lost Dog Prospect was estimated by BMGS in June 2017. The Ore Reserve has been determined using this model.

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estimate for		
conversion to		
Ore Reserves	Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	The Mineral Resource is inclusive of the Ore Reserves.
Site visits	Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	A site visit by the Competent Person was undertaken prior to the commencement of the pre- feasibility study.
	If no site visits have been undertaken indicate why this is the case.	A site visit was completed.
Study status	The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.	A pre-feasibility study has been carried out appropriate to the deposit type, mining method and scale. The study was carried out internally and externally using consultants where appropriate.
	The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	This Ore Reserve report leverages on the work of both external consultants and in-house BCN knowledge to optimise and determine a mine plan which is technically achievable and economically viable. This Ore Reserve is inclusive of material modifying factors and comprises material classified as Measured or Indicated in the Lost Dog mineral resource estimate. Material classified as Inferred has been credited zero positive value (i.e. classed as waste) throughout this pre-feasibility study
Cut-off parameters	The basis of the cut-off grade(s) or quality parameters applied.	The cut-off grade is calculated as part of the mine optimisation analysis. For Ore Reserve calculations the cut-off grade was 0.6 g/t gold (diluted).
	The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).	The Mineral Resource model was factored to generate diluted Ore Reserve during optimisation and evaluation processes.
Mining factors or assumptions	The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.	Mining method is conventional open-pit with drill and blast, excavate, load and haul. The ore zone geometry coupled with the low stripping ratio (<1.9 (waste) to 1 (ore) and maximum pit depth (<32.5m) indicate that Lost Dog is most suited to mining by conventional open pit mining methods.
	The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.	An external geotechnical report provided pit slopes and recommended inputs for optimisation. Grade Control to be RC drilled from surface on a 12.5x12.5m staggered grid.
	The major assumptions made and Mineral Resource model used for pit and stope	The Mineral resource model used for the pit optimisation was estimated by BMGS in June 2017. The Ore Reserve has been determined using this model.



	optimisation (if appropriate).	
	The mining dilution factors used.	Additional mining dilution of 2% was applied.
	The mining recovery factors used.	Mining recovery of 98% was applied.
	Any minimum mining widths used.	No minimum widths were utilised with resource lode interpretation being in excess 120m exclusive of mining dilution.
	The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.	Inferred Resources were not used or included in the mining study.
	The infrastructure requirements of the selected mining methods.	Infrastructure required includes mine process bore field, mineral processing plant, tails storage facility, workshop, offices, fuel tank, generator, magazine and process water dam.
	The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.	Processing will be using conventional CIP methods and is a tried and tested means of gold extraction from material of this nature.
	Whether the metallurgical process is well-tested technology or novel in nature.	Well-tested existing metallurgical technology.
	The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.	Previous tenement holder mined and custom milled a 4,625t of ore which has a gold recovery determined by the custom milling facility to be 91.8%. This mill parcel was representative of the ore zones.
Metallurgical factors or		ALS Metallurgical testwork performed on a representative composite sample achieved 84% gold recovery.
assumptions		Bureau Veritas bottle roll testwork on various ore types has resulted in gold recoveries of between 82-96%.
		Based upon these results a gold recovery of 85% has been utilised for this pre-feasibility study.
	Any assumptions or allowances made for deleterious elements.	No deleterious elements are present.
	The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.	Bulk sample processing (i.e. the 4,300t parcel previously mined) has been carried out.
	For minerals that are defined by a specification, has the ore reserve estimation been	There are no minerals that are defined by a specification.



	based on the appropriate mineralogy to meet the specifications?	
Environmental	The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.	All environmental permitting has been submitted to the Western Australian DMIRS and DWER. All approvals have subsequently been received except for the approval to construct and commission the processing plant. Draft conditions for this approval have been received and are under consideration.
		Waste rock is typically non-acid forming.
		Waste material will be stored in conventional above surface waste dumps. As stated above approval is pending.
		Tailings will be stored on site in excavated open pit workings. As stated above approval is pending.
Infrastructure	The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.	Site access is via a public road (Jaurdi Hills Road) which passes along the western edge of the main tenement boundary. The tenements comprising the project area are granted mining leases with a combined area of
		approximately 1,000 hectares. Accommodation will be a mixture of residential and on-site
Costs	The derivation of, or assumptions made, regarding projected capital costs in the study.	Capital costs are based upon BCN in-house knowledge and experience in the establishment of similar mining operations. These costs estimates are considered to be within (+10%/-5%).
	The methodology used to estimate operating costs.	Operating costs are based upon contemporary in-house knowledge and experience for similar mining operations. These costs estimates are considered to be within (+10%/-5%).
	Allowances made for the content of deleterious elements.	No deleterious elements present.
	The source of exchange rates used in the study.	Cost models use Australian dollars.
	Derivation of transportation charges.	There are no transport costs.
	The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.	Treatment costs are based on known current milling costs.



	The allowances made for royalties payable, both Government and private.	State royalty of 2.5% and 3 rd Party Royalty of \$80/oz have been incorporated.
Revenue factors	The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.	Using a gold price of A\$1,650/oz.
juctors	The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	Perth Mint combined April/May2018 monthly average gold price > A\$1,720/oz
	The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.	Gold doré will be sold at the Perth Mint as it is produced.
Market assessment	A customer and competitor analysis along with the identification of likely market windows for the product.	Market window unlikely to change.
ussessment	Price and volume forecasts and the basis for these forecasts.	Price is likely to go up, down or remain same.
	For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	Not industrial mineral.
Economic	The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.	No NPV applied.
	NPV ranges and sensitivity to variations in the significant assumptions and inputs.	Sensitivity analyses have been completed.
ocial	The status of agreements with key stakeholders and matters leading to social license to operate.	No Native Title Claimants on DIA over the mining leases.
Other	To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:	
	Any identified material naturally occurring risks.	A risk review has been completed. No material risks are identified.



	The status of material legal agreements and marketing arrangements.	Government approvals have been received required for the Jaurdi Gold Project to begin mining activities in Q3 2018.
		The Works Approval and Licence Application is with the Western Australian Department of Water and Environmental Regulation for approval.
	The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals	95.7% of Mining Reserves and 96.3% of gold ounces are contained within granted mining tenements.
	will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	4.3% of Mining Reserves and 3.7% of gold ounces are contained within the 3 rd Party owned Exploration License E16/469. An option to purchase agreement between BCN the 3 rd Party owners of E16/469 has been exercised by BCN.
		A Project Management Plan and Mining Proposal have been approved by the Western Australian DMIRS.
	The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility	A miscellaneous license L16/120 application has been lodged and subsequently approved by the DMIRS. L16/120 is to be utilised as a tailing/water pipeline corridor to Black Cat.
	study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	A miscellaneous license L16/122 application has been lodged with the DMIRS. This has been applied to facilitate pipeline access between a portion of the bore field and the proposed ore processing plant.
	The basis for the classification of the Ore Reserves into varying confidence categories.	Reserves are classified according to Resource classification.
Classification	Whether the result appropriately reflects the Competent Person's view of the deposit.	They reflect the Competent Person's view.
	The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	<1.0%
Audits or reviews	The results of any audits or reviews of Ore Reserve estimates.	No audits carried out.
Discussion of relative	Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical	Confidence is in line with gold industry standards and the companies aim to provide effective prediction for current and future mining projects.



accuracy/ confidence	procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.	No statistical quantification of confidence limits has been applied.
	The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	Estimates are global.
	Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.	Reserve confidence is reflected by the Probable category applied, which in turn reflects the confidence of the Mineral Resource. The Reserve is most sensitive to; a) resource grade accuracy, b) gold price c) metallurgical recovery d) ore milling costs.
	It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	Current production data were available has been used.

APPENDIX 11 JAURDI GOLD PROJECT

LOST DOG ORE RESERVE STATEMENT – CONSENT FORM



Competent Person's Consent Form

Pursuant to the requirements of Chapter 5 of the ASX Listing Rules and Clause 15 of the JORC Code 2012 Edition (Written Consent Statement)

Report name

Jaurdi Gold Project – PFS Results for the Jaurdi Gold Project-May 2018
(Insert name or heading of Report to be publicly released) ('Report')
Beacon Minerals Limited
(Insert name of company releasing the Report)
Lost-Dog
(Insert name of the deposit to which the Report refers)
If there is insufficient space, complete the following sheet and sign it in the same manner as this original sheet.
August 2018
(Date of Report)

Statement

I/We.

Gary McCrae

(Insert full name(s))

confirm that I am the Competent Person for the Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code, 2012 Edition, having five years experience
 that is relevant to the style of mineralisation and type of deposit described in the Report, and to the
 activity for which I am accepting responsibility.
- I am a Member or Fellow of The Australasian Institute of Mining and Metallurgy or the Australian Institute of Geoscientists or a 'Recognised Professional Organisation' (RPO) included in a list promulgated by ASX from time to time.
- I have reviewed the Report to which this Consent Statement applies.

I am a full time employee of

Minecomp Pty Ltd

(Insert company name)

Or

I/We am a consultant working for

(Insert company name)

and have been engaged by

Beacon Minerals Limited

(Insert company name)

to prepare the documentation for

Jaurdi Gold Project

(Insert deposit name)

on which the Report is based, for the period ended

August 2018

(Insert date of Resource/Reserve statement)

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the information in the Mineral Resources and Ore Reserve Statement as a whole, in the form and context in which it appears, fairly represents, information and supporting documentation prepare by myself.

Consent

I consent to the release of the Mineral Resources and Ore Reserves Statement and this Consent Statement by the directors of:

Beacon Minerals Limited	
(Insert reporting company name)	
2274 /	
Signature of Competent Person:	Date: 29/08/2018
AUSIMM	315953
Professional Membership: (insert organisation name)	Membership Number:
aboan.	CELESTE BOASE, KALGOOBLIE
Signature of Witness:	Print Witness Name and Residence: (eg town/suburb)

responsibility:	
Nil	
manage and the Hills of	the Competent Person signing this form is accepting
and the second s	
Nil	745 7 1177 1245 105
	(860)
	and the second s
Signature of Competent Person:	Date:29/08/2018
AUSIMM	315953
Professional Membership: (insert organisation name)	Membership Number:
Oboan.	CELESTE BOASE, KALGOORLIE
Signature of Witness:	Print Witness Name and Residence: (eg town/suburb)

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