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CURRAGH RESOURCES INCORPORATED

ANALYSIS OF PRODUCTION SHORTFALLS AT THE FARO OPEN PIT

MARCH 1990

William Hill Mining Consultants Limited
704 - 80 Richmond Street West
Toronto, Ontario, CANADA
M5H 2C7

TEL: (416) 362-1474
FAX: (416) 362-1933

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1.0 INTRODUCTION

In early February, 1990, William Hill Mining Consultants Limited (HMC) was requested by Mr. Kurt Forgaard, President of Curragh Resources Incorporated (Curragh) to look into the recent shortfalls in metal production at the Faro operations.

With both ore tonnage and grade below expected since early 1989, HMC's terms of reference were to define the problem, investigate the possible source(s), and if possible suggest how to correct the situation. HMC has proceded with the evaluation on the assumption that the mill-calculated head and tail grades are the true benchmarks from which to evaluate the mining operations; HMC has included several comments on this aspect in the body of the report.

During the week of February 12th, 1990, HMC visited the Faro Operations near Faro, Yukon, and the regional exploration office in Whitehorse, Yukon. The open pit mining operations were observed at Faro and discussions were held with key personnel. At Whitehorse, the emphasis was placed on reviewing the block model interpretation of the orebody.

HMC found all Curragh staff to be most helpful and willing to assist HMC with its work despite their busy work schedules.

The report which follows begins with a summary of the problem and its likely causes, followed by HMC's preliminary recommendations. Details of HMC's work will be included in the chapters, exhibits, and appendices of the final report.

2.0 SUMMARY

Recent shortfalls in recovered metal from the Faro operation compared to the expected schedule can be traced to several factors; the central issue is the ability of the existing geologic block model to adequately predict tonnage and grade in both the short-and long-term, and under varying geological situations.

HMC's understanding is that the problem is a relatively recent one having arisen only in early-1989 although it has now persisted through to this year. The fact that this coincided with a change in ore type from predominantly massive in the core of the deposit to stringer ore on the fringes, points to the capability of the block model in relation to the sample pattern as being the principal source of the problem. Therefore despite the adoption (in mid-1989) of a refined block model that has apparently improved the actual versus expected comparison, this could only be a temporary solution and in any case may not be applicable to other ore reserve situations that Curragh has in mind.

In an attempt to more precisely define the problem, HMC has spent some time examining several benches mined in 1989. This has shown the general tendency of the block model to over-estimate the proportion of high-grade ore (+6% combined lead-zinc), and in particular the difficulty the model has in predicting the distribution of ore from bench to bench; in other words, the model by itself does not provide reliable estimates for monthly or quarterly scheduling.

For this short-term estimation, the problem is essentially a lack of sufficient sample data in relation to a search pattern which is large compared to the local geological features, principally vertical faulting of horizontal stringer ore. This situation leads to "smearing" or smoothing of the higher grade composites into areas where the ore is actually medium, of medium into low grade, and of low grade into actual waste rock. This pattern has shown up clearly in comparisons made between actual blasthole grades and the original block model. For example, in the higher grade set of blocks, 63% were found to have been over-estimated by a factor of between 1.65 and 2.41 (the ratio of expected block model over actual blasthole). In the medium and low grade category, the reverse is the case with about 65% of the blocks being underestimated.

For longer term reserves (currently based solely on diamond drill sampling), adoption of the latest version of the model (8908) has significantly improved the precision of the estimates. HMC does believe however that the model's search pattern is governed more by the sample pattern than by the deposit's characteristics since the variograms show very little structure, and in any case are based on limited statistical data. If the inverse-distance method is to be used elsewhere, HMC would suggest that some time is taken to ensure that the search pattern is suitable for the deposit.

In addition to the fundamental problem of the block model, HMC believes that several other factors may also be indirectly affecting the planning estimates. For example in the mining operation, current practice is to blast without separating the in situ ore from waste which inevitably leads to excess dilution. In the mill, HMC has made the assumption that the benchmark estimate of head and tail grades are true despite doubtful weightometer readings from the belts leading to the rod mills, and questionable results obtained from the shark fin samplers on the slurry feed (although the operators are now in the process of studying solutions to this problem).

One final observation is that the existing laboratory appears overwhelmed by the quantity of samples submitted for analysis, so much so that results from blasthole drilling are often not available until after blasting takes place. There may also be some prioritizing of the mill samples ahead of the mine's requirements which is slowing down the turnaround time.

3.0 RECOMMENDATIONS

Many of the factors influencing the grade estimates at Faro seem to be well understood by the operators and technical staff on site and action is already underway in some areas for example refinement of the 8908 model. Based on discussions with the operators and its work to date, HMC believes that the precision of production estimates in the short and longterm can be further improved by adopting a few key procedures:

1. For short-term scheduling, the immediate need is for additional information to supplement existing diamond drill data. HMC believes that this information can be adequately obtained from extending every 4th or 5th blasthole to the maximum capacity of the drill (52 feet) which would provide just over one-and-a-half benches of additional grade and geological information below the bench being readied for blasting. At a cost of $2.92 per foot, the extra expense (±1.6¢ per tonne drilled) can easily be justified especially if compared to diamond drilling. Some variation in extending every 4th or 5th hole may be possible after some study, for example confining the program only to the suspected higher grade areas.

2. For yearly and longterm planning, the precision of the block model could be improved by making more use of the "Mixup" program which applies dilution more realistically than the current practice of applying a global 5% ore loss factor (but no dilution) to blocks no matter the location or grade. Possibly some refinement of the software code could be considered to speed up the program which currently takes several days to complete.

3. It will be difficult to enhance the basic accuracy of the block model with such widely spaced drill data however the existing blasthole data should be used wherever this is possible. HMC would also suggest that variograms be run with the blasthole data to provide a more reliable guide to localized structure.

4. The capacity of the laboratory is seriously delaying timely availability of blasthole samples to the mine operators. Apart from adopting alternative instrumentation (now under study), HMC also suggests that the lab be expanded and/or a solution is found to improve existing performance. Perhaps an incentive system could be considered in combination with a system to monitor the accuracy and repeatability of assays on a daily basis (for example, using deviation from a benchmark standard deviation).

5. In the mining operation, selective mining of ore and waste should be possible once the more detailed blasthole information is available ahead of time; HMC therefore suggests that this mining technique be considered.

6. On the milling side, HMC has two suggestions referred to earlier. Both concern the reliability of the calculated metallurgical balance since both the calibration of belt weightometers and the accuracy of the shark fin samplers are in question. HMC therefore suggests that the accuracy of the head and tail samples be investigated to ensure an accurate benchmark for comparison with the mine.

4.0 BACKGROUND

Exhibit 1 sets out the 1989 budget statistics compared to actual production achieved. The total shortfall of metal in combined lead and zinc amounted to approximately 125 million lbs from a scheduled production of 692 million lbs; this budgeted production was based on reserves calculated by the 8805 block model (included in Exhibit 1 for comparison). Some improvement would have been possible with the latest 8908 version of the model, but there would still have been a significant shortfall in metal produced and particularly in mined grade compared to scheduled. This can be seen in Exhibit 2 which compares the budget against a hypothetical 8908 projection that assumes actual amounts of low grade were stockpiled; in this case the expected grade would have been 7.9% combined Pb and Zn against the original (8805) budget of 8.1% but still well above the actual 7.6%. Exhibits 3 and 4 provide a breakdown of percent metal loss due to tonnage, grade and recovery, compared to the 1989 budget.

During the year, the bulk of the mining took place in the lower portions of "B" phase and the upper portions of "S" phase, in the predominantly stringer zone. Since little massive, core material was taken during the year, this change in ore type has obviously had a significant impact on the reliability of the existing block model capabilities.

At the time of HMC's visit, the major open pit equipment at Faro consisted of 27 haul trucks, including 2 new 170 ton units that were being assembled, 4 shovels, 3 rotary drills and 7 front-end loaders. A new P&H 2800 XPA shovel had been ordered and is due to be commissioned next September. Another Euclid 170 ton haul truck is due to be commissioned in the second quarter of 1990.

Mining takes place on 20 ft high benches. Both ore and waste have recently been blasted together in 40 ft benches in order to maximize production. In general, the mining operations are tight with the large blasts causing constrictions and limited choices if problems occur with one muckpile. Ore and waste are blasted using the same explosives density, giving uniform breakage in massive sulphides, but lots of craters and a wide throw in the disseminated material.

5.0 DESCRIPTION OF THE BLOCK MODELS

5.1 Outline

The Faro block models utilize a block size of 35.35 feet along the deposit, 25 feet across the deposit and 20 in height. Rows of blocks are parallel to the geological cross-sections and normal to the structural grain of the deposit. The models have been numbered according to the date of origin, hence the last three Faro models are F8805, F8908 and F8910.

The deposit consists of three roughly equal portions in cross-section, a southwest part that dips 12 degrees to the southwest, a central horizontal section, and a northeast portion that also dips approximately 12 degrees to the southwest. To accommodate the changes in dip, each of the three portions was interpolated in independent computer runs. The parameters interpolated into each block were %Pb, %Zn, grams/tonne Ag and Au, and density; the initial interpolation was completed in three passes.

The search volume required for block estimation has now been increased from 150 ft by 225 ft by 14 ft, to 200 ft by 300 ft by 37.5 ft. The search consists of up to three passes to obtain samples for a block; in each succeeding pass, only blocks with zero grade are interpolated. In the southwest and northeast portions of deposit, the search ellipsoid is tilted 12 degrees.

Fault limits are not entered into the model therefore the search ellipsoid passes through major faults when searching for ore. The model is generated by weighting composites by inverse distance squared.

Variogram analysis has reportedly indicated that a drillhole spacing of 140 ft provides reliable data for the block model however the variograms reviewed by HMC do not appear to be statistically reliable owing to a lack of samples.

The following notes briefly outline the three latest models in use at Faro:

F8805: Composited by geological intervals. Minimum 3 composites required to interpolate a block; maximum allowable is 20 composites. Strict geology matching of rock types between blocks and composites. Model diluted uniformly at 10% with 0% grade and 95% mining recovery applied. Dilution density of 2.5 tonnes/bcy. No cutting of assays or composites.

F8908: Composited by bench intervals of 20 ft. Minimum 3, maximum 20 composites for block estimation. Model more loosely defined to match all massive sulphide rock types (codes 40-70) and quartzite rock types (codes 20-30). Model not diluted since waste in the composite benches provides dilution. 95% mining recovery applied. No cutting of assays or composites.

F8910: Composited by bench intervals of 20 ft. Minimum 2, maximum 20 composites. Massive sulphide and quartzite rock types matched. Model not diluted. 95% mining recovery. Drillhole assays cut to the 95th percentile before compositing.

6.2 Reserves

Mineable reserves have been classified into 3 categories based on the grade of combined lead and zinc:

High grade ore	greater than 6.0%
Medium grade ore	5.0% to 6.0%
Low grade ore	4.0% to 5.0%

Refinement of the model to the F8908 version has resulted in some improvement in narrowing the difference between actual and estimated reserves; variances between the model and blasthole data for 1989 are smaller than the F8805 figures although short-term fluctuations are still marked from month to month. In the past when mining took place predominately in the core of the deposit, the model was found to be generally accurate over several years.

Blasthole data has not been entered into the model.

Recently 28 fill-in diamond drillholes were completed in the S phase ore zone in an attempt to model the lenticular areas more accurately; the results of this drilling are currently being evaluated.

6.0 BLASTHOLE DRILLING AND SAMPLING

6.1 Outline of Sampling Procedure

There are currently 3 rotary drills, capable of drilling 51-52 ft holes; bit diameter is 10-5/8 inches.

The 20 ft benches are drilled with a 5 ft subgrade with the ore holes on a 22 ft x 22 ft pattern and waste on a 24 ft x 24 ft pattern.

The drill chippings are sampled by grade control technicians. The procedure is as follows:

After the drill moves from the hole, the subgrade portion of the chippings is estimated and removed. At a suitable location where the pile is average in shape, a channel is cut half way through the cuttings, from outer radius to the hole. The channel is scraped down the pre-existing surface and the back of the cut is made vertical. A 1-1/2 inch thick cut is scooped out of the back of the channel, placed in a sample bag and tagged. This method of sampling may lead to a biased assay because of the different distributions of heavy and light drill chips ejected from the drillhole: the sample is taken at the high point of the pile where it is likely there is an accumulation of heavier chips and hence higher grade particles.

HMC suggests that the company investigate the use of a wedge-shaped sampler which should improve the sampling procedure and provide a more representative sample. It may also be possible to attach the steel sampler to a winch which would remove it from the chippings, however the added expense would have to be studied.

One other suggestion is to use a marker, such as lime, to delineate the subgrade from the rest of the hole. The lime could be placed automatically by the drill operator operating a lever from his cab once the subgrade is reached.

Finally, the "Ore Control" software package of Gemcon appears to have some worthwhile applications particularly since it would provide an integrated survey and grade control capability.

6.2 Grade Control

Once the chippings are assayed, the results are passed on to the Grade Control Engineer who plots the results by bench for the Short Term Planning Engineer.

After each blast, the Grade Control Engineer demarcates ore and waste zones in the muckpiles with flags. The shovel operators are instructed to load from one composite until the flags marking the composite fall down the muckpile.

7.0 COMPARISON OF BLOCK MODEL AND BLASTHOLE DATA

7.1 Procedure

The basis of HMC's analysis has been a series of F8908 model ore blocks mined in 1989 on 10 benches selected from ore in both "B" and "S" phases, that is from areas that have contributed to the recent grade variances.

The first step of the analysis was to measure the ore areas (high, medium, and low) on each bench; three benches (3350,3570, and 3650) were digitized by computer then checked and verified for accuracy by comparison against manually estimated numbers for the corresponding benches. The remaining 7 benches were then manually estimated. Information from the complete 10 benches then provided the raw data for comparing modelled versus blasthole quantities in high, medium, and low grade categories.

7.2 Results

Exhibit 5 sets out the results of the three digitized benches and illustrates that while total ore areas are similar, the proportion of high, medium and low grade is not as predicted by the model. In particular, the model interpolates too much high grade ore and not enough low grade. Another point to be noted from this exhibit is that the total perimeter of the digitized blasthole areas is more than that estimated by the model, in other words the potential for dilution is in fact greater than being allowed for by the block model.

The high, medium and low grade ore areas delineated from the blastholes do not always coincide with the blocks estimated by the model, which tends to interpret zones offset and to one side of the actual ore as sampled by the blastholes.

Exhibit 6 summarizes the results of obtained from all 10 benches (details of the exhibit can be found in Appendices I and II). The distributions indicate that in 65% of the cases, the block model is generally overestimating high grade ore compared to the blastholes; in 19% of the cases, the ratio of actual to expected (i.e. blasthole to model) was found to be 2.41. In the medium and low grade categories, the opposite is the case i.e. approximately 65% of the blocks in both categories are underestimated compared to actual blastholes. Exhibit 7 sets out the same information as in Exhibit 6, but by the number of ore blocks.

Given the large amount of high grade ore predicted by the model compared to blasthole data, the problem appears to be that the high grade composites are being smoothed in with lower grade. This evens out the distribution of high grade ore which in fact the high grade which in fact occurs more erratically surrounded by low/medium grade.

7.3 Discussion

Since one of the principal problems over the past year has been the difficulty of predicting ore from month to month, the solution seems to lie in providing more sample information for this short-term planning work.

One possibility is to drill every 4th or 5th hole to the full limit of the blasthole drill's capacity (52 feet) which would provide assay and geological information for the bench below and 12 ft of the following bench (the lime marker method could be used to define each bench in the drillhole chippings pile). Once the hole is drilled to full depth and sampled, the hole could be backfilled to the proper 25 ft depth and loaded with explosives as per normal practice.

At an extra cost of $79 per hole deepened, the cost per tonne drilled would increase by only about $0.016 which is considerably less expensive than diamond drilling. The blasthole drill would not require an extra set up, saving operating time and labour, plus the rig would not interfere with normal pit operations.

HMC has tested the reliability of having only one hole out of every four by plotting a 40 by 40 foot grid over the three digitized benches and assuming that only the one blasthole closest to the centre is used for grade interpretation. The results (set out in Appendix III) were found to give an acceptable estimate of ore on the benches compared to using all the blasthole information. Further delineating of the third bench down will result once 40 ft holes are drilled from the second bench down and from the usual blasthole drilling of the third bench itself. The short term mine planning engineer will know exactly what the third bench contains by the time the bench is ready for blasting.

For the longer term reserves, a refined F8910 model should be an improvement on version F8908 and provide better comparison to blasthole data. The 8910 will involve cutting of the assays which should lessen the amount of smoothing of the high grade composites over large areas.

The existing F8908 model has been found to be an improvement on version 8805 used for the 1989 budget and could be further enhanced by using the "Mixup" program which assigns dilution on a realistic basis. Mixup parameters consisting of a 10 ft slice on the horizontals and 5 ft slice vertically resulted in too much dilution. The 1989 tonnage increased to over 1.0 million tonnes in excess of the blasthole tonnage, and the yearly grade decreased to 7.65% combined Pb-Zn compared to 8.44% blasthole grade. The model is being run again with 5 ft slices on the horizontals and 2.5 ft vertically. Results are not yet known.

Once the model is finalized in Whitehorse, it should be delivered to the geology department at Faro. The blasthole data would be entered into the database and the model would be run again. All detailed face mapping of faults should be plotted on the resultant polygons that are currently given to mine planning.

The polygons could be manually reinterpreted to ensure accuracy. At this point, the sections and plans could be handed to mine planning. As each bench is mined, the process of entering in the new blastholes, plotting new faults and reinterpreting the model would be repeated. In order for this process to be successful, good communication must be maintained between all parties involved.

8.0 OTHER FACTORS

Although the central problem dealt with in this report is the block model and ways of improving its performance, there are several points on the operating side that also influence the operation's ability to meet scheduled tonnage and grade as budgeted. HMC's notes on these points are set out in the following paragraphs.

8.1 Milling Practices

The basic assumption in this report is that the material balances calculated in the mill are reliable.

In HMC's opinion, two items could be suspect and warrant investigation; namely, weightometer accuracy and sampling accuracy. A third item, assay lab turnaround time, should be improved.

8.1.1 Weightometer Accuracy

Weightometers are Milltronics Compuscale model IIA. They are currently being tared electronically, but are not being calibrated against a known weight. The standard method of calibrating a weightometer is to place a known weight mounted on rollers on a conveyor, chain the weight to a wall, start the conveyor, measure the weight and adjust the scale as required. Simply taring the scale does not compare weight against known weight. It is therefore HMC's opinion that the accuracy of the weightometers in the mill may be unreliable, leading to questionable results in the metallurgical balances.

8.1.2 Head And Tails Sampling

The in-line samplers on the head and tails streams at the mill are of the shark fin type. A small sampling tube projects from the bottom of the slurry pipeline into the middle of the pipeline, thus cutting the sample as it passes through the pipe. This type of sampler may be reliable if the flow through the pipeline is laminar, but the Curragh mill personnel report that the flow is turbulent. The proper method of sampling turbulent flow is to cut the flow from one side to the other, channeling the sample into a container as the flow is cut.

The mill is not always sure whether to adjust the heads or the tails, but reportedly usually modifies the tails when calculating the metallurgical balance. With the ore weights in question as well, a substantial amount of error within the balance is possible.

8.1.3 Assay Laboratory Capacity

The long turnaround time required for the blasthole assaying makes the production engineers' job often difficult since he has to stake the muckpile by visual estimate if the sample results are not available on time. Obviously this is not a good situation and may be leading to some ore loss and/or unplanned waste reporting to the mill.

Curragh are investigating the application of an x-ray fluorescence instrument that can apparently determine lead and zinc assays quickly. This type of instrument would help grade control, but the accuracy of the results may be questionable.

The main problem is the capacity and/or the efficiency of the existing facilities. Given that it is not desirable to expand the capacity of the assay lab, a means of improving performance seems to be the preferable route to take. One method may be to provide a bonus to the lab technicians for exceeding their usual number of assays performed in a shift. The accuracy of sampling procedures (and results) could be checked by using repeat assay on a random basis to test the divergence from the known standard deviation of sampling under optimum conditions. It may also be possible to enter the assay results directly into a computer for easy integration with the block model.

8.2 Mining Practices

8.2.1 Blasting

One method of selectively mining the lenticular portions of the orebody is to blast and mine ore and waste separately. A row of exploratory holes could be drilled between the last row of ore and the first row of waste. If the material is ore, the first row of waste holes would be filled with tailings and blown out by a rotary drill after the ore is blasted. In this manner, the amount of high grade ore is maximized and dilution is minimized.

8.2.2 Level Bench Control

Explosive load in the disseminated rock must be carefully monitored and adjusted to suit the rock density and so prevent high points on the bench floors. Shovel operators should be advised of the shovel level several times during a shift. One method of taking several level shots in a shift is to construct metal level profiles and survey them into position so they are set to view a given distance above the ground. The profiles consist of two horizontal bars set apart and supported by vertical bars. When viewed from the top of one bar to the top of the other and compared to a marker line on the shovel, the elevation difference would be indicated.

8.2.3 Truck And Shovel Factors

Previous studies have indicated that the trucks were not being loaded to the proper tonnage in accordance with the accepted truck factors that were being used. There are no permanent weigh scales at the pit to monitor truck factors on a continuing basis.

8.2.4 Dump Location Instructions

The current practice of advising the truck operator of the dump location by horn signals from the shovel operator leaves some room for misinterpretation and error. Another method of instructing the truck operator would be for the shovel operator to post a sign which can be easily read on each side of the shovel. The sign is switched when the shovel moves into new material. Shovelling two types of material is not a problem as each side of the shovel would have its own sign.

8.2.5 Mining At The Bottom Of The Pit

Once the economic pit floor is reached, 40 ft drillholes from the bench above would indicate pockets of high grade ore. The ore near the bottom of the pit is prone to folding and is difficult to pinpoint its position. With the extra drilling, short stab ramps could be blasted, and excavators and front-end loaders used to mine several pockets of high grade ore.

EXHIBITS

EXHIBIT 1

1989 BUDGET, MODEL AND PRODUCTION STATISTICS

	1989 BUDGET	BLOCK MODEL 8805	BLOCK MODEL 8908	ACTUAL
Tonnes (Thous.)	5,266	5,373	5,159	4,944
Grade (Combined % Pb/Zn)	7.85	7.89	7.49	7.20
Stockpile				
Tonnes (Thous.)	378[1]	378[1]	370[1]	565[2]
Grade (Estimated % Pb/Zn)	4.5	4.5	4.5	4.5
Millfeed				
Tonnes (Thous.)	4,888	4,995	4,789	4,379
Grade (% Pb/Zn)	8.10	8.15	7.72	7.60
Metal Lbs x 10^6	872.6	897.2	814.84	733.5
Metal Recovery Lbs x 10^6	692	711.5	646	567
Metallurgical Recovery (%)	79.3	79.3	79.3	77.3
Tails (% Pb/Zn)	1.67	1.68	1.60	1.72

Notes:
1. 7.2% of total tonnes.
2. 11.4% of total tonnes.

EXHIBIT 2

COMPARISON OF ACTUAL VS MODEL TONNES AND GRADES

	ACTUAL		MODEL 8805		MODEL 8908	
	Tonnes	% Tonnes	Tonnes	% Tonnes	Tonnes	% Tonnes
Mined - Tonnes	4,944	100.0	5,266	100.0	5,159	100.0
Grade	7.20		7.85		7.49	
Stockpile - Tonnes	565	11.4	378	7.2	588	11.4
Grade	4.50		4.50		4.50	
Milled - Tonnes	4,379	88.6	4,888	92.8	4,571	88.6
Grade	7.60		8.10		7.90	

EXHIBIT 3

ANALYSIS OF METAL SHORTFALL

Model 8805 vs Budget

	METAL LOSS COMPARED TO BUDGET		
	%	LBS x 10^6	DISTRIBUTION %
Loss of Metal - Total	18.1%	125	
Due to Tonnage	9.8% (90.2%)	67.5	54
Due to Grade	6.0% (94.0%)	42	33
Due to Metallurgical Recovery	2.3% (97.7%)	16	13

Notes: 1. Figures in brackets are compared to 100% total production.

EXHIBIT 4

ANALYSIS OF METAL SHORTFALL

Model 8908 vs Budget

	METAL LOSS COMPARED TO BUDGET		
	%	LBS x 10^6	DISTRIBUTION %
Loss of Metal - Total	12.2%	79	100.0
Due to Tonnage	8.4% (91.6%)	54	68.3
Due to Grade	1.4% (98.6%)	9	11.4
Due to Metallurgical Recovery	2.5% (97.5%)	16	20.3

Notes: 1. Figures in brackets are compared to 100% total production.

EXHIBIT 5

COMPARISON OF 8908 BLOCK MODEL AND BLASTHOLE ORE BLOCKS
BENCHES 3350, 3570 AND 3650

Bench and Grade	BLASTHOLES Digitized Area (ft^2)	BLASTHOLES Perimeter (ft)	BLASTHOLES Manual Interpretation of Area (ft^2)	MODEL Digitized Area (ft^2)	MODEL Perimeter (ft)	MODEL Manual Interpretation of Area (ft^2)	DIFFERENCE IN DIGITIZED AREAS (Model-Blasthole) (ft^2)
3350							
High Grade	45,620	2,815	57,440	100,220	1,910	107,870	54,600
Med Grade	41,010	3,900	31,800	12,850	1,302	15,020	-28,160
Low Grade	20,615	2,080	22,980	6,340	624	4,420	-14,275
	107,245	8,795	112,220	119,410	3,836	127,260	12,165
3650							
High Grade	57,510	4,070	61,860	44,465	2,860	42,420	-13,095
Med Grade	24,880	2,670	23,000	15,215	1,150	19,442	- 9,665
Low Grade	7,250	1,110	11,500	19,820	1,465	21,210	12,570
	89,640	7,850	96,360	79,500	5,475	83,072	-10,140
3570							
High Grade	159,510	6,650	169,680	192,645	6,080	173,220	33,135
Med Grade	31,500	4,040	15,908	62,150	4,230	60,980	30,650
Low Grade	75,230	6,720	81,305	41,310	2,850	47,720	-33,920
	266,240	17,410	266,893	296,105	13,160	281,920	29,865
TOTAL	463,125	34,055	475,473	495,015	22,471	492,252	31,890

EXHIBIT 6

% DISTRIBUTION OF BLOCKS BY GRADE CATEGORY

8908 MODEL VERSUS BLASTHOLE

Performance[1]	HIGH GRADE % Distribution	HIGH GRADE Ratio[2]	MEDIUM GRADE % Distribution	MEDIUM GRADE Ratio[2]	LOW GRADE % Distribution	LOW GRADE Ratio[2]	TOTAL % Distribution	TOTAL Ratio[2]
Low	5	0.74	48	0.62	46	0.45	16	0.59
Good	32	1.05	17	1.05	28	1.05	30	1.05
High	44	1.65	13	1.85	8	1.85	35	1.65
Extra High	19	2.41	22	3.83	18	4.33	19	2.86

Notes:
1. Performance of 1.0 indicates a perfect estimator.
2. Expected (Model) over Actual (Blasthole).

EXHIBIT 7

NUMBER OF ORE BLOCKS BY GRADE CATEGORY

8908 MODEL VERSUS BLASTHOLE

Performance	HIGH GRADE Number Benches	HIGH GRADE Number Blocks	MEDIUM GRADE Number Benches	MEDIUM GRADE Number Blocks	LOW GRADE Number Benches	LOW GRADE Number Blocks	TOTAL Number Benches	TOTAL Number Blocks
Low	2	90	7	146	6	130	15	366
Good	3	543	1	51	2	79	6	673
High	4	738	1	41	1	24	6	803
Extra High	1	318	1	69	1	52	3	439
TOTAL		1,689		307		285		2,281

APPENDICES

APPENDIX I

COMPARISON OF ORE BLOCKS, 8908 MODEL VERSUS BLASTHOLE

Summary of High, Medium and Low Grade Ore Blocks

Number of Ore Blocks

Bench #	Blasthole	Model	Ratio of Blocks Model/Blasthole
3350	127	144	1.13
3370	226	189	0.84
3390	284	342	1.20
3550	244	311	1.27
3570	302	319	1.06
3590	297	258	0.87
3610	233	283	1.21
3630	158	224	1.42
3650	109	94	0.86
3670	93	117	1.26
TOTAL	2073	2281	1.10

APPENDIX I (Continued)

COMPARISON OF ORE BLOCKS, 8908 MODEL VERSUS BLASTHOLE

High Grade

Number of Ore Blocks

Bench #	Blasthole	Model	Ratio of Blocks Model/Blasthole
3350	65	122	1.88
3370	116	117	1.01
3390	132	318	2.41
3550	132	227	1.72
3570	192	196	1.02
3590	197	230	1.17
3610	157	245	1.56
3630	100	144	1.44
3650	70	48	0.69
3670	50	42	0.84
TOTAL	1211	1689	1.52

APPENDIX I (Continued)

COMPARISON OF ORE BLOCKS, 8908 MODEL VERSUS BLASTHOLE

Medium Grade

Number of Ore Blocks

Bench #	Blasthole	Model	Ratio of Blocks Model/Blasthole
3350	36	17	0.47
3370	73	32	0.44
3390	109	11	0.10
3550	49	51	1.04
3570	18	69	3.83
3590	26	19	0.73
3610	28	22	0.79
3630	22	41	1.86
3650	26	22	0.85
3670	31	23	0.74
TOTAL	418	307	0.73

APPENDIX I (Continued)

COMPARISON OF ORE BLOCKS, 8908 MODEL VERSUS BLASTHOLE

Low Grade

Number of Ore Blocks

Bench #	Blasthole	Model	Ratio of Blocks Model/Blasthole
3350	26	5	0.19
3370	37	40	1.08
3390	43	13	0.30
3550	63	33	0.52
3570	92	54	0.59
3590	74	9	0.12
3610	48	16	0.33
3630	36	39	1.08
3650	13	24	1.85
3670	12	52	4.33
TOTAL	444	285	0.64

APPENDIX II

DISTRIBUTION OF ORE BLOCKS
8908 MODEL VS BLASTHOLE

	HIGH GRADE		MEDIUM GRADE		LOW GRADE		TOTAL	
Ratio of Blocks Model/Blastholes	Number Benches	Number Blocks	Number Benches	Number Blocks	Number Benches	Number Blocks	Number Benches	Number Blocks
0.00 - 0.10	-	-	-	-	-	-	-	-
0.10 - 0.20	-	-	1	11	2	14	3	25
0.20 - 0.30	-	-	-	-	-	-	-	-
0.30 - 0.40	-	-	-	-	2	29	2	29
0.40 - 0.50	-	-	2	49	-	-	2	49
0.50 - 0.60	-	-	-	-	2	87	2	87
0.60 - 0.70	1	48	-	-	-	-	1	48
0.70 - 0.80	-	-	3	64	-	-	3	64
0.80 - 0.90	1	42	1	22	-	-	2	64
0.90 - 1.00	-	-	-	-	-	-	-	-
1.00 - 1.10	3	543	1	51	2	79	6	673
1.10 - 1.20	-	-	-	-	-	-	-	-
1.20 - 1.30	-	-	-	-	-	-	-	-
1.30 - 1.40	-	-	-	-	-	-	-	-
1.40 - 1.50	1	144	-	-	-	-	1	144
1.50 - 1.60	1	245	-	-	-	-	1	245
1.60 - 1.70	-	-	-	-	-	-	-	-
1.70 - 1.80	1	227	-	-	-	-	1	227
1.80 - 1.90	1	122	1	41	1	24	1	187
1.90 - 2.00	-	-	-	-	-	-	-	-
+.200	1	318	1	69	1	52	3	439

APPENDIX III

COMPARISON OF ORE BLOCKS, 8908 MODEL AND 40 FT GRID VERSUS BLASTHOLE

Summary of High, Medium and Low Grade Ore Blocks

	--Number of Ore Blocks--			---------Ratio of Blocks--------	
Bench #	Grid*	Blasthole	Model	Model/Blasthole	Grid/Blasthole
3550	291	244	311	1.27	1.19
3570	315	302	319	1.06	1.04
3650	132	109	94	0.86	1.21
TOTAL	738	655	724	1.11	1.13

* 40 ft x 40 ft grid blocks stated in equivalent 25 ft x 35.35 ft ore model blocks.

APPENDIX III (Continued)

COMPARISON OF ORE BLOCKS, 8908 MODEL AND 40 FT GRID VERSUS BLASTHOLE

High Grade

	--Number of Ore Blocks--			---------Ratio of Blocks--------	
Bench #	Grid*	Blasthole	Model	Model/Blasthole	Grid/Blasthole
3550	156	132	227	1.72	1.18
3570	183	192	196	1.02	0.95
3650	78	70	48	0.69	1.11
TOTAL	417	394	471	1.20	1.06

* 40 ft x 40 ft grid blocks stated in equivalent 25 ft x 35.35 ft ore model blocks.

APPENDIX III (Continued)

COMPARISON OF ORE BLOCKS, 8908 MODEL AND 40 FT GRID VERSUS BLASTHOLE

Medium Grade

	--Number of Ore Blocks--			---------Ratio of Blocks--------	
Bench #	Grid*	Blasthole	Model	Model/Blasthole	Grid/Blasthole
3550	40	49	51	1.04	0.82
3570	47	18	69	3.83	2.60
3650	34	26	22	0.85	1.31
TOTAL	121	93	142	1.53	1.30

* 40 ft x 40 ft grid blocks stated in equivalent 25 ft x 35.35 ft ore model blocks.

APPENDIX III (Continued)

COMPARISON OF ORE BLOCKS, 8908 MODEL AND 40 FT GRID VERSUS BLASTHOLE

Low Grade

	--Number of Ore Blocks--			---------Ratio of Blocks--------	
Bench #	Grid*	Blasthole	Model	Model/Blasthole	Grid/Blasthole
3550	96	63	33	0.52	1.52
3570	85	92	54	0.59	0.92
3650	20	13	24	1.85	1.50
TOTAL	201	168	111	0.66	1.20

* 40 ft x 40 ft grid blocks stated in equivalent 25 ft x 35.35 ft ore model blocks.