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CURRAGH INC FARO DIVISION
A GENERAL GUIDE FOR ORE GRADE CONTROL IN THE GRUM and VANGORDA ORE BODIES
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NOTED BY D. TENNEY.

INTRODUCTION

With the completion of mineable reserves in the Faro pit in the spring of 1993, and the introduction of ore from the Grum pit at the same time, the practices and procedures of maintaining a consistent (and acceptable) blend of feed to the concentrator had to undergo significant changes. Unlike the ore from the Faro deposit the Vangorda/Grum ore is finer grained due to a lower metamorphic grade (greenschist as opposed to amphibolite facies in Faro pit). As a result the ore requires more grinding than the Faro ore to increase liberation of the contained base metal minerals. The Plateau ores also have inherant differences that create flotation problems. They have a significant copper content which in soluable form can render the ore as refractory (particularly in the upper horizons of the deposits). The ore remaining in the Vangorda pit however does not display refractory characteristics and is currently the best suitable ore for metallurgical treatment. In the Grum pit the initial ore benches contain a majority of carbonaceous quartzites with a largely varying degree of carbon content and poses to be the most problematic in maintaining consistent and qualifying mill feed due to the affect carbon has on the lead flotation circuit.

CARBON IN GRUM ORE DOES NOT APPEAR TO CAUSE PROBLEMS. D.T.

CC. DICK

ROCK DESCRIPTIONS

A detailed description of the ore and waste types for the Grum and Vangorda pits can be found in the Geology department files, also there is detailed structural description by Gilson and Jennings. A numeric lithstratigraphic coding system (see appendix A) was recently developed to integrate the rock types into the Gemcom modelling software and the blasthole database. It is important to know this code. The following is a general description of rock types encountered in the two pits, and some of there visual characteristics.

Rock Type; Baritic Massive Sulphide – This is the main high grade ore type with grades ranging from 6% to 20% Pb+Zn. This ore has a high specific gravity (about 4.0) and is generally easy to identify. It has an iron content ranging from 15% to 25%. This rock type can be found as distinct localized layers between a barren pyritic massive sulphide and can be separated if they are thick enough with the proper equipment, however it is difficult to identify these layers in the blasthole cuttings due to mixing and as a result the whole column is assayed as one minable bench, with the barren sulphide included as dilution. (old code = 60 or 4G, new code = 7)

Carbonaceous Quartzites – This ore type generally overlies the massive sulphides in the hanging wall. It can vary in grade from 0% to 10% combined Pb+Zn. It is easy to identify in the blasthole cuttings because it produces a dark grey smudge on the fingers when handled. In the face it will appear light grey to black and is highly foliated along the S2 plane (prominant foliation plane). Pyrite and chalcopyrite are visible in the quartzite matrix. The ← MINOR CHALCOPYRITE D.T. carbonaceous material is found mainly in the foliations. It has been observed that the Carbon content can be correlated with the grade, lower grades indicate more highly foliated quartzite resulting in higher concentrations of carbon. Therefore flotation test results from a +10% Pb+Zn sample will be better than from a 5% Pb+Zn sample. (old code = 20 or 4A, new code = 2)

RECOVERIES FROM HIGHER GRADE ORES ARE GENERALLY HIGHER THAN FOR LOWER GRADE ORES.

Semi Massive Pyritic Quartzites – This is an important waste rock located in the footwall. It contains varying degrees of pyrite and chalcopyrite and is extremely hard. Some lead and zinc mineralization does occur near the contact with the baritic ore in the form of thin bands and is gradational towards the ore. The grade can range between 0% and 6% Pb+Zn. Waste grade footwall quartzites are highly acid generating and must be contained in the designated "sulphide waste cells". This rock type requires more drilling time and can be difficult to break. (old code = 40 or 4EC, new code = 4)

ESPECIALLY WELL DEVELOPED AT VANGORDA. CHALCOPYRITE MINOR D.T.

Massive Pyritic Sulphide – This rock type contains +60% pyrite and can grade as high as 40% iron. Pb+Zn grades can range from about 1% to 10%. It is easy to identify in blasthole cuttings but can be mistaken for footwall pyritic quartzites, especially in the winter months with long periods of darkness (as a rule all sulphides should be sampled for assay). It was observed that this rock type was especially refractory in the upper benches of Vangorda pit due to the remobilization of zinc and high soluable copper content. It was not uncommon to have a very high lead to zinc ratio in this rock type and the ratio could be used as an indicator for possible refractory ore. (old code = 50/55 or 4EG/4E, new code = 5

YES! D.T.

Phyllites/Schists – There is a wide spectrum of these rock types in the Vangorda and Mount Mye formations (see code). The main difference between the two formations is that the Mount Mye formation contain calcareous metasediments and the Vangorda formation contain noncalcareous. The most important of these are the altered phyllites in the Van–gorda formation. These rocks are situated as an alter–ation envelope around the ore zones. They are identified mainly by color where they are light cream to white. Some appear pale green due to chlorite. Pyrite and chalco–pyrite are visible and there may be some minor base metal mineralization. It is very important to minimize dilution of this rock type in the mill feed.

NON CALCAREOUS
CALCAREOUS

✓

OWING TO CONTENT OF MICACEOUS MINERALS. WE WOULD PREFER NO PHYLLITE IN MILL FEED. D.T.

GRADE CONTROL

Proper grade control is extremely important for ore recovery in the pit and efficency in the concentrator. Under normal operating conditions in the latter months of 1992, with two pits being mined, the Geology Department staffed a total of three geologists and four technicians. Twenty four hour coverage was maintained by the technicians in the field and proved to be most effective. They were responsible for blasthole logging and sampling, ore flagging, data entry, and dozer and shovel supervision at the ore face. The technicians reported to the Grade Control Geologist who was responsible for stockpile management, forecasting, ore recovery and mill feed blending. A Mine Geologist was responsible for pit mapping and geological modelling on a continuous basis. The geologists reported to the Chief Geologist who was responsible for the Department and was the liason between Mine and Mill Operations.

NOTE: INTER DEPARTMENTAL.

The following is a general guide as to how the geology department functioned in the latter part of 1992.

CHIEF GEOLOGIST LIAISES WITH:

V.P. EXPLORATION, *
V.P. MINING,
MANAGER MILLING
MANAGER MINING.
GENERAL MANAGER.

*EXTENSIVELY.
D.T.

Geological Modelling:

Curragh Inc. uses the Gemcom software (PCXPLOR, GEOMODEL, PCMINE, MINESURVEY) to produce geological models of the Anvil deposits. PCXPLOR contains all of the diamond drill hole information used in the modelling process. This program is also used for blasthole data in creating short range models. The X,Y and Z coordinates for all the data are digitized from survey information and are combined with lithologic and assay data in individual bench databases. GEOMODEL is used to create files defining the geometrical parameters of the ore body after the structural interpretation is done. Cross and long section interpretations are digitized and then plotted in plan view by bench. The plan views are then interpreted and inconsistancies are "smoothed out". The new plans are digitized into bench files. The three dimensional information along with the lithology and assay data are manipulated in PCMINE to create a model where reserve calculations can be made. Then the pit is designed and its parameters are put into the system and minable reserves can be calculated by bench, rocktype and grade. As mining progresses and new information is obtained from pit mapping and blastholes, the Gemcom system allows the user to make model updates for more accurate reserve estimates and to make separate short range models that can assist in grade control as well as blast pattern design. The power of the Gemcom system as a mining tool is extensive and great effort should be applied to creating short range models. It does however require rapid information gathering and time consuming digitizing. The role of the technician should be expanded into learning the Gemcom modelling system. The entire mine and mill operations evolve from the model. Its' integrity is vital for success. Reconciliations between blasthole calculated and model reserves should be done at least once a month and after a bench is mined. This will tell the user where the model might be in error and establish targets for pit mapping and possibly a small diamond drill project if neccessary.

Blast Pattern Design:

The Geology Department played an important role in the blast pattern design and worked closely with the Blasting Engineer. Because of the varying densities of ore and waste types blasthole spacing and powder factors changed with the rock drilled throughout each bench. It was particularly important to know precisely the locations of contacts near the highwall so that wall stability could be maintained. The Blasting Engineer initially designed his patterns based on the most recent short range model created by Geology. Modifications would be made based on the subgrade logs of blastholes from the bench above and the most recent pit mapping. As the bench was drilled off and the blastholes logged, a geologist or technician would discuss with the Blast Engineer any potential problems recognized in the current drill production. It was important for him to know if any sulphide holes bottomed out in phyllites or vise versa because it could create a hard toe or dilution problem. Fault gouge was also important to recognize and if possible locate the trend of any fault before further drilling was done. Copies of the blasthole logs with all available information was given to the engineer prior to loading the holes. He would create a "recipe" for powder factors in each hole based on this knowledge. Several months of experimentation in blasting techniques led to better blasts that minimized dilution and added to highwall stability.

Due to the extremely folded and faulted structure of the Grum Pit, new blasting techniques may have to be developed. In any case the Geology Department will have to continue to work closely with the Blasting Engineer to minimize dilution due to blasting.

THERE ARE FIVE STAGES OF DEFORMATION AT GRUM OF WHICH THE FIRST TWO ARE IMPORTANT (D_1 AND D_2). F_1 FOLDS ARE ISOCLINAL, AND ARE REFOLDED BY F_2 FOLDS. FAULTING AT GRUM IS EXTENSIVE. DT.

Blasthole Logging and Sampling:

The Geological Technicians were responsible for sampling and logging the blastholes as well as many other important tasks. Twenty four hour coverage was maintained by working a rotating four by four, twelve hour shift schedule. This was especially helpful when processing blastholes in the winter months when wet holes would freeze solid within thirty minutes after being drilled. The technicians knew the rock codes and had learned how to recognize the rock types in the cuttings of the blastholes. A mylar copy of the blast pattern was kept in their field binder where they recorded the code of each rock unit encountered and the approximate depth (to the nearest half meter) to the contact from the top of the hole (logging from the bottom of the cuttings pile upwards). It is not an accurate method of estimating horizon depth but it has shown to be satisfactory most of the time.

NOTE !!

A statistical study was made in the late eighties on the various methods a person might use when sampling and logging a blasthole. When a technician is pressed for time and faces several blastholes to process, he/she might be persuaded to do a quick and dirty job of it. The study showed that the best way to cut the pile (using a track shovel) is to make two separate slots on opposite sides, choosing the location where the height of the cuttings are about average. The slots should be dug outward from the very edge of the hole scraping all the material down to the base. This allows the best cross section view and by making two such slots variances in horizon thickness can be averaged. The total drill depth of each hole as well as a hole identifcation was marked on a stake placed in the cuttings. Using a simple visual ratio technique the depths can be estimated. It is important to consider the subgrade depth in the estimates. The study also showed that to obtain a better representative sample it is sufficient to use a minimum of two locations. Independant assay results from different locations in the pile can vary from being sulphide waste to highgrade ore. So it is very important to take portions of the sample from at least two locations. The subgrade portion of the cuttings was removed before sampling (3–5 cm off the top of the pile). The sample was removed by taking a vertical slice (about 3 cm thick) from the side of each slot where the horizon thickness appeared to be average. The two portions were combined in a sample bag and a sample tag was then folded, rolled into the top of the bag and stapled to seal the contents and keep the tag clean and dry. The tags were numerically sequenced but unfortunately the blasthole identifier was alpha–numeric. This resulted in having to write the corresponding blasthole identifier on each duplicate tag and later recording them in a book so that when the assays came back with only the numeric tag value, we were able to match the proper assay with the blasthole. This was a tedious and unneccessary step, and measures were being taken to rectify the problem by having the Blasting Engineer create blast plans with only a numeric identification on each blasthole. New tags would then be made that would match a pre–established sequence for each bench. This way the engineers, assayers, and geologists would have one number for one blasthole, making less room for error and smoother information flow. This system should be implemented in the Grum pit to help simplify and speed up the data gathering process, which would ultimately result in more efficient use of the Gemcom software. ("ORE CONTROL").

THIS PROBLEM ALSO OCCURS WITH "ORE CONTROL". D.T.

YES!

YES!

Normally, all blastholes containing sulphides and/or quartzites were sampled even if they were determined by the model as being in a sulphide waste zone. It was discovered that a large portions of the model sulphide waste actually turned out to be a low grade ore, especially near the contacts of high grade zones. Some sulphide waste zones were sampled on an "every other hole" basis. Ore blastholes that contained distinct waste horizons were sampled with a reasonable mining perspective in mind. The technicians would decide from the logs if it was feasable to clean the horizons with a dozer or backhoe after blasting. They would then either sample just the ore portion or include the waste in the sample, depending on their decision. In certain zones where refractory ore was suspected, a second screened sample was taken and analyzed for cyanide soluable copper. Values greater than 0.1% were considered high and appropriate stockpiling measures were taken.

YES!

NOTE: CLEANING/SCALPING. D.T.

WE MAY HAVE TO CONSIDER A TEST LIKE THIS FOR GRUUM D.T.

Blasthole Bench Masters:

Large sheets of mylar were used to construct a visual plan of the assay results and logs. The appropriate grids and section lines were plotted on with the Gemcom software (there are five individual map sheet files for the Grum pit, scale = 1:500). As blast patterns were drilled off, the holes were surveyed and plots were produced. These plots were then copied on to the mylar by hand with a fine drafting pen. Each blasthole on the mylar was labeled with the appropriate identifier by overlaying it on the blast design. The logs of each hole were also drafted on. As the assay results came in, the hole locations were coloured according to an established colour code listed below.

AND AUTOCAD.

colour	grade category Pb+Zn
solid blue	<1% – phyllite waste (not assayed)
blue with blue circle	0–3% – sulphide waste
yellow with slash	3–4% – very low grade ore
solid yellow	4–5% – low grade ore
green	5–6% – medium grade ore
solid red	6–9% – high grade ore
red with red circle	>9% – very high grade ore

THESE VALUES MAY CHANGE. D.T.

When adequate assay information was plotted blocks of ore were outlined on the mylar according to grade and rock type. By having the logs and assays in colour code on the same plan and knowing what types of ore went into the designated stockpiles, it was simply an "eye ball" task in producing the ore blocks. The blocks were labeled according to their stockpile destination. These outlined blocks or "dig packets" represented the actual mining blocks as they would be mucked and stockpiled. Numbered "geopoints" were drawn on the mylar at the intercies of the outlined blocks and then the coordinates were measured and delivered to the surveyors. They were responsible for shooting them in on the blast in advance of mining. The technician would then flag the blast to prepare it for mining.

I PREFER TO SEE THE ASSAY ITSELF PLOTTED TOO. D.T.

LAYING OUT ORE PACKETS ON THE BROKEN MUCKPILE.

The entire method described above for producing a blasthole bench master by hand can be done with the Gemcom software, combined with AutoCAD. The survey data is in a useable form without having to make plots for manual transfer (by implementing the previously mentioned numeric blasthole system the identifier of each hole can be recorded as the holes are surveyed). Assay information can be put on disk, formatted to a usable form and imported directly into the Gemcom software. Only the blasthole logs themselves would require manual entry in numeric code but would not be a complicated task. Ore blocks could be produced on the screen with geopoints, and the coordinates transferred back into the survey instrument. Plots could be generated for immediate use. The files created could be added to as new data arrives until a bench was completed and a final bench master could be ~~produced~~. PLOTTED.

Gemcom "ORE CONTROL"

YES!!

An attempt at this idealized streamlined system was made in 1992, however, several problems were encountered that could not be rectified at the time. One problem was in the pit. The drill production could not stay far enough ahead of the shovel operation. Patterns were being drilled, blasted and mucked before all the required data was in. The drills would go down, water problems caused blasthole caving resulting in time consuming redrills. Tight spots slowed down drilling. In addition survey and assay data were sometimes late due to instrument or equipment trouble. Also, not all the Geology staff was adequately trained in the Gemcom and AutoCAD software. A domino affect was created that forced us back to a manual method. But these problems are not insurmountable and can be overcome.

NO LEAD TIME!

THE COMPUTERIZED SYSTEM MUST BE MADE TO WORK. D.T.

This is "MANUAL" SYSTEM

GEMCOM'S "ORE CONTROL" SOFTWARE IS BETTER.

Blasthole Database:

As the blasthole data was obtained it was entered into a Lotus Symphony database called BLASTDAT.WR1 (currently located in the Dell System 325 computer in E:\SYM\CURRENT\). It is very simple to use even with limited Symphony knowledge. The database is set up to record date of entry, phase, bench, blast, blasthole ID, grid dimensions, rocktype, density, tonnes/hole, coordinates, status, tonnes of metal and blasthole logs. Data entry must be made in the form environment (ALT F9). A volumetric calculation is used to calculate tonnage. The density used is a variable dependant on the Pb, Zn and Fe assays calculated by a regression formula (see appendix B). The grid dimensions of each blasthole are entered according to the blast design but should be checked against the survey pickup (highwall holes have different dimensions). A partial grid entry is used when a hole may be part waste. Each blasthole is designated a grade value according to the ore blocks marked out on the bench master. Blasthole logs are entered by code and cumulative depth to contact. Default values can be entered in the definition range. The file contains a "macro" (programmed key strokes) that calculates a weighted average for any selected phase, bench or blast. To invoke this macro type F7, DSUM and follow the prompts. As ore blastholes are being mucked in the pit each day, the status of those holes must be changed from broken to mined, adding the date (ie, BJUL23 changes to MJUL25). The tonnes and grade mined for the period can then be calculated using the DSUM macro. The calculation is then used to update stockpile status and mill feed (separate file).

The blasthole data were used to create a short range model in the Gemcom system. The data first had to be put in the PCXPLOR format by extracting the desired range into a Lotus 123 file called DATASORT.WK3 located in E:\FORMS\ (same hardware). There are four macros in this file that must be invoked (ALT F3) in the order AUTO, IMPORT, SORTDATA, PRNT−EXP. The first adjusts the record number and clears the spreadsheet for new data. The second imports the selected records from the blasthole database. The third sorts the data into PCXPLOR format and the fourth prints it out and exports it into an ASCII file where it is ready to be brought in to the PCXPLOR database. COMPLICATED

Any assay data delivered on disk from the lab was in ASCII format produced directly from the atomic absorption equipment. It did not yet have correction factors applied. It had to be put in a usable form with the help of a Lotus 123 file called ASSAYOR.WK3 located in E:\FORMS\ (same hardware) and is self explanitory.

Ore Flagging:

After the surveyors shot in the geopoints (usually as soon a the blast was shot and cleared), the geotech would flag the ore blocks for mucking. The flags used for the Vangorda and Grum pits were a firm vinyl plastic cut in triangular shape and in three colors, red – high grade, yellow – low grade and green – refractory. These flags were stapled on to lathe with 1/2" staples in various combinations and singularly. A supply was maintained in the core logging shed at Vangorda. A copy of the benchmaster showing the block outlines to be flagged was carried in the field for reference. The flags were placed a minimum of three meters apart for good resolution using the geopoints as a guide. They were adjusted for the heave of the blast and the direction that the shovel would be mucking. Any dozer work identified in the blasthole logs was done prior to flagging.

YES!

RED, YELLOW, GREEN, BLUE } FLAGS IN HAND

Dig Maps:

Dig maps were made for each blast and updated at the end of each twelve hour shift by the geotech. Copies were distributed to the shovel operators, pit shifters, geologists and shortrange engineer. They were produced on a 11" by 14" mylar by overlaying it on the bench master. As much information as possible was included on the map such as highwalls, section lines and grid, locations of dozer work showing depths to the ore or waste, stockpile destinations, sumps, ramps, geopoints and block outlines colored according to the flagging. An accurate face advance was drawn on at the end of each shift so that the oncoming shovel operator would know exactly what he/she had in front of them when they began there shift. The geotech would discuss the map with the shovel operator and shifter before the shift began and advise them of any potential problem areas and make arrangements for a dozer or backhoe.

Ore Mucking and Scalping:

The most important role of the geotech was the supervision of the ore mucking. This is where "the dollars were either stuffed in the mattress or burned in the fire". Each geotech would spend time with the shovel operator as they were mucking on an ore contact or any other problem area. It was beneficial to get the operators to know how to recognize ore from waste and identify dilution zones on their own because the geotechs weren't always around to watch. A radio call from a concerned shovel operator was all it took to save thousands of dollars in wasted ore. There was an established system of horn signals that the shovel operators would use to communicate with the haulage drivers to identify the stockpile destinations. One horn blast meant waste (sulphide or phyllite waste was established by radio), two horn blasts meant highgrade ore, three horn blasts was refractory ore, and four blasts was lowgrade ore. Because there were several stockpiles in each category the geotech would make a radio announcement to the haulage drivers designating the proper stockpile. The horn system worked alright but was far from perfect. At times the drivers did not hear all of the signal and the load went to the wrong location. Several times each shift the geotech would check all the dump locations for stray loads and take corrective action when neccessary. A computerized dispatch system would be ideal to ensure loads were not misplaced.

I LIKE TO PAINT ORE CONTACTS IN PIT IF THEY ARE VISIBLE AND ACCESSIBLE D.T.

YES! WE NEED A CONTROL ON TRUCKS.

The best way to prevent dilution was to expose the ore as much as possible with dozers and backhoes (scalping). The Grade Control Geologist would discuss the scalping operations with the shifters and geotechs. Timing was important and the shifters needed to know in advance so that the equipment would be available. Usually only short sections could be prepared at a time until the shovel advanced further ahead because the material generally had to be pushed over the face and long pushes with a dozer is not practical. Low angle contacts are best handled with a dozer whereas high angle contacts were exposed cleanly with a backhoe. The geotechs would discuss the geology with the operators and sometimes get ideas on how best to handle the project. Good communications and relationships with the equipment and shovel operators made a significant impact on ore recovery.

VERY IMPORTANT AT GRUM.

✓

Stockpiles:

As many as seventeen separate ore stockpiles exsisted on the property in the latter part of 1992 containing in excess of three million tonnes. Ore was being mined in three pits simultaneously and was piling up fast. Areas for practical location were becoming a rare commodity. The requirement for temporary transfer stockpiles at Grum and Vangorda meant rehandling the ore as it was hauled to the concentrator. Not all of the transferred ore could be dumped directly into the crusher and therefore mill stockpiles for each plateau category had to be built to control the crusher feed blending. Maintaining an accurate account of the many various stockpiles was very important and required complicated spreadsheets and databases. As the needs for new and different stockpiles grew the system was rewritten again and again.

The Vangorda and Grum ores were stockpiled separately on transfer pads where a contractor would load a fleet of eighty ton trucks and haul it the fourteen kilometers to the mill. There were five ore stockpile categories for each pit, which are listed below.

	category	description
COMBINE	G1	high grade massive sulphides — MUCH
	G2	high grade carbonaceous quartzites — LITTLE
COMBINE	LgG1	low grade massive sulphides
	LgG2	low grade carbonaceous quartzites
	R	refractory high & low grade sulphides

(NO)

SUGGEST

G HG – HIGH GRADE MASSIVE (MAINLY BARITIC) SULPHIDES

G MG – MEDIUM GRADE (MAINLY CARBONACEOUS QUARTZITE + PYRITIC QTZ)

G LG – LOW GRADE (— " —)

R – REFRACTORY ORE. (IF ANY – ? NEAR SURFACE)

Three large low grade stockpiles containing Faro pit ore exist near the edge of the pit;

category	description
LgA	4–5% Pb+Zn massive sulphides
LgL	3–4% Pb+Zn massive sulphides
LgC	4–5% Pb+Zn carbonaceous quartzites

Oxidation in these stockpiles causes flotation problems and would possibly have to be screened before feeding to the mill.

Blending restrictions, head grade requirements, and ore availability governed the stockpile situation near the primary crusher. At one time we experimented with "blended stockpiles" where a two pile system was used. Each of the two stockpiles was constructed with layers of different ore in varying thickness (depending on the metallurgical characteristics). While one pile was built a loader would feed the crusher from the other. This system worked very well until some unidentified refractory ore was put in one pile and feed could no longer be taken from it. It caused conjestion and as a result each ore type was piled and blended separately until the contaminated pile was slowly fed through.

Stockpile Management:

As previously mentioned, several computer spreadsheets and databases were made to manage the ore flow to and from transfer and mill stockpiles. The latest version used was a Lotus 123 spreadsheet located in the CVR AST 486/33E computer in F:\MILLFEED\DAILY.WK3. A blank form, WKFORM.WK3 is found at the same address. This file combines the daily and weekly forecasts with the stockpile inventories and cannot be described in detail here. A new user with 123 knowledge would need to study the spreadsheet and figure out how it works.

Information from several sources must be gathered each day. The previous days' mined reserves are found using the blasthole database (5% dilution applied at this point). The ore haul from the plateau to the crusher and crusher stockpiles is obtained from the contractor (scaled tonnes). The crusher feed from each stockpile is found on the mine operations shift report (it is very important that the loader operator keeps an individual count from each feed stockpile, using the ratio the tonnes from each are found by reconciling to the rod mill feed). Grade calculations are weighted averages based on a "last in, first out" scenario, due to the way the stockpiles are usually built. Stockpile inventories and grades must be updated daily in order to accurately predict the headgrade to the mill and to make shortrange forecasts. Active stockpiles are surveyed at the end of each month and the tonnes are calculated using a calculated density based on the average Pb, Zn and Fe grades of the pile (regression formula). Large discrepancies are then reconciled.

The stockpile categories and requirements for the Grum pit will undoubtedly continue to change as the pit is mined. The methods for keeping track of ore flow will change with it but will actually become much simpler once Faro and Vangorda stockpiles are gone.

Forecasting:

Predicting headgrades to the mill was done for one, two, four and six week periods to check if budget targets could be attained. If rod mill throughput and/or mining slowed for what ever reason, it directly affected the headgrade required to maintain the budget target. We had to find out how things would be affected down the road.

The Engineering Department would redo the mining sequences and the ore release (tonnes and grade) were calculated from the model for each sequenced block. The updated stockpile inventories were entered into the equation and by knowing what ore would be released, and when, a forecast was made. Weekly forecasts were the most important as they were the first indicators when problems arose. By monitoring actual vs predicted headgrade and ore release on a daily basis according to the weekly forecast, action could be taken as soon a problem was identified.

Ore Blending:

The most challenging aspect of grade control was maintaining a consistant blend of rod mill feed. The flotation circuit demanded a steady flow in order to achieve the maximum concentrate grades and recoveries. Satisfactory concentrate grades were 50% Zn in the zinc con and 60% Pb in the lead con with recoveries of 80% or better.

Each ore type has its own metallurgical characteristics. Massive sulphides will float much better than carbonaceous quartzites (type 2). As a general rule, no more than 20% of the rod mill feed can be type 2, Refractory ore (screened), no more than 5%. An ore blend was established each day based on the weekly forecast. Although the blend may have fluctuated slightly day by day, at the end of the week the quantities of each category fed to the mill usually matched closely with the weekly forecast. Close communications were maintained with the mill metallurgist. When a flotation problem occured the Grade Control Geologist was notified and a sample from the rod mill belts was taken and observations made to see if it was indeed the ore causing the problem. On many occations a flotation problem was the result of an internal problem in the mill. But if it did appear to be ore related a change to the blend was made. Only the Grade Control Geologist or the Chief Geologist were authorized to order a blend change (instructions on alternative blends were given to the geotechs on night shifts and weekends).

PB+ZN SULPHIDES IN QUARTZITE ARE COARSER GRAINED THAN PBS+ZNS IN MASSIVE (BARYTIC) SULPHIDES, SHOULD FLOAT BETTER! CARBONACEOUS MATERIAL NOT A PROBLEM. ACCORDING TO TEST WORK

Experience gained from observing the results of various blend combinations resulted in a "feel" for the ore. One would get to know what combination would perform satisfactorily and by watching and checking the mill stockpiles regularly, potential problems could be averted before any serious consequences happened.

Wet ore put through the primary crusher proved disastrous. Screens and shutes would plug solid in the secondary crushing circuit and the ore would either freeze or plug in the fine ore bins. This was a bad problem in the spring when the ore comming from the pit was usually wet from melting snow and a rising water table. Direct dumping of the 80 tonne trucks had to stop completely and an alternative blend using one or two loaders tramming a bucket at a time in the crusher had to be set up with what ever dry ore was available (usually low grade massive sulphides from the Faro pit ore stockpiles). The wet ore literally had to be "chased" through the crushing circuit with dryer material. This did however affect the head grade and put a kink in the plan, but at least ore was able to be fed.

The long haul from the Plateau to the mill was organized by the Grade Control Geologist. Each morning the contractor was notified as to what ore they would be hauling and how many trucks out of the fleet could dump directly to the crusher (it was desirable to have as many as possible dump direct but the flotation circuit did not always permit it). Blends could sometimes be set up on the plateau side with the ore haul, but to maintain control every single load had to be dumped direct, otherwise the sequence went out of whack. Because of the "highball it" attitude of the long haul contractor, slower trucks would cause the fleet to bunch up and as one was dumping in the crusher another came in right behind him and would dump to the pile, instead of waiting for the crusher pocket to clear. Unless the loader at the crusher could somehow compensate for this, the consistancy of the blend was lost.

N.B.

When the Vangorda pit is completed and the last of its ore fed, the Grum pit will be on its own to supply a satisfactory feed. There will definately be problems at this stage due to the large quantities of type 2 ore. Stockpiling and blending techniques will have to be modified as well as having a new regrind circuit put in the concentrator. The type 2 ore could possibly be segregated into two or more categories based on texture alone. The transitional behavior of the rock from quartz and pyrite rich to quartz and pyrite poor (grading towards the carbonaceous phyllite contact) may allow a stockpile separation of the highgrade G2 into separate categories. Innovative grade control ideas will none the less come about and satisfactory flotation of the Grum ores will be successful.

MOST OF TOP BENCHES ARE TYPE "2".

PROBLEMS WITH TYPE "2" SHOULD NOT BE AS GREAT AS BRAD SUGGESTS!

Month End Reconciliation:

Ore mined, stockpiled, and milled are reconciled at the end of each month. An inventory balance must be maintained by making neccessary positive or negative adjustments at some point in the ore flow process. The mill feed was the benchmark in the process and considered untouchable. Stockpile surveys had a margin of error as well as the volumetric calculations in the blasthole database. By the end of a twelve month period however the positive and negative adjustments, for the most part, cancelled each other out with only minor shuffling of tonnes and grade from one stockpile to another.

CONCLUSION

The procedures and systems for grade control in the Faro and Vangorda pits worked well. The Geology Department operated as a team and positive results were achieved. The department had one of the lowest rates of employee turnover within the Faro Operations. There was always room for improvement (as were pointed out) and there will always be room for improvement.

The future of grade control in the Grum pit depends largely on the streamlining of data processing for faster and less labour intensive office activity. The "hands on" approach in the field will continue to be a key factor in maximizing grades and recoveries both in the pit and concentrator.

YES! "ORE CONTROL" SOFTWARE

With proper initiative, training and innovation the success of the Grum operation is inevitable.

APPENDIX A

CURRAGH RESOURCES INC. – NUMERIC ANVIL LITHOSTRATIGRAPHIC CODE

ROCK CODES (OLD CODES INCLUDED FOR COMPARISON)

DISSEMINATED QUARTZITES

2	4A	Ribbon banded carbonaceous quartzite
3	4C/4D	Pyritic quartzite (<30% pyrite)

SEMI MASSIVE SULPHIDE (Generally low grade)

4	4EC/4E1/ 4C3	Siliceous pyritic sulphides (30–60% pyrite)

MASSIVE SULPHIDES

5	4E/4F	Massive pyritic sulphides (60–100% pyrite)
6	4K	Massive pyritic sulphide with clasts of dolomite/ankerite
7	4G	Baritic pyrite sulphides (>10% barite)
8	4H	Pyrrhotitic sulphides
9	4J	Nonpyritic sulphides & oxides – pyrite poor

METASEDIMENTS

20	3G	Noncalcareous, muscovite–chlorite, medium grey phyllite
22	1C/1CD/1D	Noncalcareous, bio–musc–qtz staurolite+andalusite+garnet+fibrolite schist
30	5A/5G/3E/1E	Carbonaceous phyllite/schist
32	5E/3F/1G	Marble + calc–silicate bands
33	1B	Skarn and 'silicated' marble
38	3D	Calc–silicate
40	5B	Calcareous, silvery grey, muscovite chlorite phyllite
44	5C/3C/1F	Metabasite, poorly foliated greenstone (relict igneous texture)
45	5C/3C/1F	Pyroxinite – commonly serpentinized (relict bastites)
46	5C/3C/1F	Amphibolite – blue–green hornblende + plagioclase + quartz
47	5D/3B/1H	Chloritic phyllite/schist – pale green, homogenous

ALTERED ROCKS

52	4L0	Muscovite>chlorite quartz phyllite/schist – light cream to white
54	4L5	Chlorite>muscovite quartz phyllite/schist – pale green

CRETACEOUS INTRUSIVES

60	10Q	Quartz vein – white bull quartz
61	10AB	Anvil Batholith – Mt Mye phase of Anvil plutonic suite. Musc–bio granite
65	10C	Pegmatite
66	–	Aplite
68	10E	Hornblende–biotite quartz diorite – massive and unfoliated
69	10F	Smokey quartz–feldspar porphyry – massive and unfoliated

FAULT ROCK (use only if parent not recognized)

72	Gouge
74	Tectonic breccia
76	Mylonite

OVERBURDEN

82	Unclassified – general
84	Triconed – no recovery
86	Till – silt – sand
88	Ferricrete
99	Air

OTHER

0	No Recovery
&	+/–

GRADE MODIFIERS

N	no visible grade
W	1–3% Pb+Zn
L	3–5% Pb+Zn
H	5–10% Pb+Zn
V	>10% Pb+Zn

MINERAL IDENTIFIERS

Carbonates

- c calcite
- k ankerite
- v carbonate–non specific
- w dolomite

Micas

- b biotite
- j 'fuchsite'
- l chlorite
- m muscovite
- s sericite
- t talc

Felspars – Quartz

- f feldspar
- q quartz (fine grained)
- y kaolinite (clay minerals)
- p potash feldspar
- Q quartz (vein)

Calc Silicates

- a actinolite
- e epidote
- h hornblende
- i diopside

Alumino–Silicates/Pelites

- d andalusite
- n garnet
- r fibrolite
- u staurolite
- z chloritoid

Oxide/Sulphide/Sulphates

- A Arsenopyrite
- B Barite
- C Chalcopyrite
- G Galena
- L Limonite (iron oxides)
- M Magnetite
- P Pyrite
- R Pyrrhotite
- Z Sphalerite
- F Marcasite

Other

- g carbon
- x noncalcareous

ROCK TEXTURES

+	equigranular
I	foliated
=	laminated/ribbon banded
>	coarse–grained
^	medium grained
<	fine grained
\	clotted
:	porphyroblastic
%	porphyritic
'	interstitial
@	porous
*	weathered
~	fault gouge
X	fault breccia (tectonic)
?	mylonite
#	altered
$	'stringered'
o	spotted

Appendex B

The regression formula used to calculate Specific Gravity was:

A "BETTER" REGRESSION FORMULA USING A POWER WAVE MAY BE CALCULATED

ALMOST S.G. OF QUARTZ

((2.430+(0.05*Pb assay)+(0.02*Zn assay)+(0.054672*Fe assay))*0.98

Where 0.98 is the porosity factor

The formula is entered in the default range in the blasthole database, BLASTDAT.WR1