American Bullion Minerals Ltd.

886916

Memo

To: File

From: Don Barker

Date: July 23, 1998

Re: Pre-Feasibility Study, 1998

The pre-feasibility is essentially complete. It is very important that we understand the strengths and weaknesses of the pre-feasibility such that we can market it appropriately and, correspondingly, improve it.

In completing this pre-feasibility over the last 6 months, we had to agree to certain criteria and standards such that we could go forward. Now at the end, we have the ability through hindsight to assess the individual technical aspects as a whole and in economical terms. A hindsight evaluation of the project indicates that the project can be vastly improved from that outlined in the pre-feasibility. Many aspects and terms are emerging that are unique to this project. Some of these terms are campaign mining styles, new paradigm with respect to the chosen style and, most important, the interrelationship of the various components and how optimization of those various components can contribute to a much improved project. A further complication is that some technical aspects and improvements emerged as the project was unfolding and it was impossible to go back and revamp the entire process.

The pre-feasibility is a practical document that defines a mining strategy that is defendable in terms of style, grades, capital costs and operating costs. However, many of the components of the pre-feasibility in terms of understanding the mining style are outside the experience level of those doing the pre-feasibility. There is no apology here, as we have defined a unique plan and design that as a whole has not been done or engineered anywhere in the world. Traditional open pit experience has been brought to play in this study but this has proven to be limited as a new and different paradigm for mining has emerged.

The contributing engineers have done a very good job of penetrating unknown territory and defining an innovative practical plan. It is much easier to improve a situation from a position of review than it is to charge to the final solution. However, given that much unknown territory and innovation is involved, it is incumbent on us to constructively criticize and improve the project. That is what this memo is all about.

The following items are potential areas for improvement.

Grade and Mine Size Capacity

The grades, particularly within the various cut-off regimes, are much improved and much better understood than in previous work. The combination of using higher cut-off grades, with an improved understanding of the vertical continuity as a function of the sub-vertical structural elements, is far superior to previous work. However, more work is required in this area to understand the intensity of grade distribution with respect to tonnages throughout the zone.

Gary Giroux has chosen to cut copper and gold grades to 2.0% copper and 2.0 g/tonne gold. In my evaluation of the deposit, I don't think there is a need to cut either gold or copper. This conclusion is based on the consistency of high grade copper mineralization within the core, in both the vertical and horizontal sense, and the direct relationship between gold and copper within the deposit.

However, the cutting is close to 5 times average and, consequently, is not a great factor.

The reserve grades, in general, fairly represent the deposit.

The most important contributor to the economics of this project is the schedule that the grades follow in the early years of the project. These grades are a function of the bench by bench grades within the deposit. More importantly, they are a function of the mining technique and in the ability go down on the resource using dozers at a very rapid rate. The grades shown in the schedule are the best we can ever do as far as pulling grade forward. If anything, we could experience a slight reduction in grade in the early years.

The productive capability of the mill has been chosen as 30,000 tpd. This choice was based on a desire to define a smaller mine than that which FDW engineered and 30,000 was an easy number with respect to 90,000 tpd. The 30,000 tpd figure was also viewed as the upper limit for the lower cost power line distribution system.

Based on the choice of 30,000 tpd, a life of mine of 20 years was determined. This meant that 219 x 10^6 tonnes were required for the ore reserves.

After this decision was made it was determined that, based on an approximate cut-off of 0.4% copper, a tonnage existed of 120 x 10^6 tonnes grading approximately 0.58% copper and 0.47 g/tonne gold. To date it is not clearly understood, based on structural conditions, how the intensity of mineralization is distributed. However, it is felt that the understanding is closer and that the 120 x 10^6 tonne figure is within the neighborhood of discrete tonnage.

The ideal tonnage capacity for the mill should be, more than any other factor, fashioned around the highest grade ore and the ability to extract that ore on a reasonable scheduled basis.

The high grade zone of 120 x 10^6 tonnes has good vertical dimension but is tube-like and narrow within two zones. It will be difficult to extract as a unit in any type of mining style and, consequently, to try to extract this tonnage and grade will require a reduced mining output to stay within the limits of the narrow zone. In order to keep the life of mine about 20 years, implies that a mill tonnage rate of about 17,000 tpd would be ideal. A daily tonnage rate in the 17,000 tpd region would be more adaptable to the geometrics of the high grade portion of the ore zone and to the ability to extract the ore at the highest grade within the dozer push concept.

In the recognition that the mining scheme allows about 30% more drilling and blasting costs, has a dozer grinding element to the ore pass and an autogenous grinding component in the ore pass, it is likely that the mill should be designed for about 15,000 tpd and scheduled to mill about 40% more tonnage (i.e. 21,000 tpd).

It might be desirable to plan the above schedule for about 4 years and then plan for an expansion in year 5 to about 23,000 tpd within the ability to run the mill at about 32,500 tpd

This type of strategy allows the mine to stay on the highest grade possible in the early years, cut the capital risk to a minimum, plan for expansions on a systematic basis and be within the power constraints, but be close to maximizing the benefits on the capital cost of the powerline.

A more compelling reason to decrease the capacity of the mill could lie in the pure economics of the project. The following analysis is a bit of a good news/bad news story.

Of the ore reserves in the mine plan, 80 x 10^6 tonnes grades 0.584% copper and 0.47 g/tonne gold. This implies that the remainder is about 0.33% copper and 0.25 g/tonne gold. The total cost of this material is US$5.47 (on and off property). The value of this material at US$0.85 copper and US$310.00 gold/oz is US$6.96 g/tonne. A general rule of thumb is that the NSR should be at least double the property operating costs. In this case the NSR is US$5.12 and the operating cost is US$3.63. Consequently, there is US$1.49 to support capital and profitability. The conclusion must be that 65% of the defined mineralized zoned within our mine plan will not support capital in an adequate manner and, consequently, is not ore. Of course, this conclusion must be tempered with incremental economics with respect to the low grade and the economy of scale with respect to operating costs and capital costs. However, in general the conclusion is correct. Now if this is the bad news what is the good news? The good news is that it is always preferable to have an ore zone made up of discretely different grades than a homogenous grade. In this case, an ore zone that looks like:

Tonnes	Cu %	Au g/tonne
80 x 10^6	0.584	0.47
144 x 10^6	0.330	0.25
224 x 10^6	0.419	0.33

is more valuable than an ore zone that is homogeneously:

224 x 10^6 grading 0.419% copper and 0.33 g/tonne gold

Now we have never had any illusion that a high grade core did not exist within the ore zone. However, a number of important events have magnified the importance:

i. Commodity prices for gold and copper have dropped from about $360.00 region to $290.00 and copper has dropped from about $1.10/lb to $0.75/lb.

ii. A review of the ore reserves had indicated that the core is close to 20% higher grade than originally anticipated.

iii. The mining style of short push dozing and drawing out the center and bottom of the ore zone is much more compatible to high grading than large scale open pit mining with trucks and shovels, which require excessive operating room and are limited in annual depth development.

The conclusion to this is that we must maximize our understanding of the high grade inner zone and the distribution of grade over various tonnages. We have the confidence that the grade and tonnage has good vertical and horizontal continuity. However, at this time it appears that a plant designed for 16,000 tpd to be run at about 22,000 tpd would be ideal initially.

In consideration of the overall resource it is clear that out of about 550×10^{6} tonnes a larger proportion of the metal is contained in about 20% of the tonnage, and perhaps only that 20% is truly economical on a stand-alone capital supporting basis. Consequently, the following can be observed and concluded:

i. In order to attack the highest grade possible, the initial mill should be designed for about 16,000 tpd to expand to about 22,000 tpd.

ii. An expansion should be built into the pre-feasibility thinking such that as the benefit of the better grade is realized and the majority of the debt is returned, the property can address the lower grade ores from the perspective of least risk associated with capital. The planned expansion could be two or three phased but should be planned on at the outset.

iii. It is clear that some 40% of the copper value is devoted to downstream costs (i.e. transportation to Stewart, ocean freight, smelting and refining). Given that some 80% of the ore tonnage cannot fully support the capital costs of the project on a stand-alone basis, it is clear that the hydrometallurgical process should be strongly considered.

Dozer Push Concept

Much has been written on the dozer concept and the aspect of treating the mining technique as a value function rather than a cost comparison function. It has been determined, when treating the mining style as a combination contributor to reduced operating costs, reduced capital costs, grade enhancement in the early years, reduced pre-stripping, increased highwall angle, and as a grinding contributor to the ore, that the maximum distance of push is increased from about 150 meters to about 400 meters. Given that we could increase the capacity of the dozer to a D-12 size (on the drawing board) or a Komatsu 575 (90 cubic yard machine) and probably increase the distance of push and reduce costs, there is no doubt that the dozer push style is highly competitive with any other technique in the unique Red Chris situation.

It has been demonstrated that if we can push up to 400 meters, then we create ore passes every 800 meters. Based on the necessity of having an ore pass in the middle of the two high grade zones, this theoretically means that only two ore passes are required for the life of the mine.

The pursuit of understanding the dozer push technology as a value function has two main advantages:

i. The analysis has demonstrated that, without any doubt, the large dozer as applied to the uniqueness of the Red Chris deposit, is the priority mining alternative and is the priority up to about 400 meters.

ii. The ability to increase the dozer push distance has a direct affect on the ability to eliminate underground driveage capital costs and ore and waste handling system costs.

The Mine to Mill Process

Much has been written about the potential of the mine to mill process and the work of the Julius Krushnett Mineral Research Center (JKMR) at Highland Valley Copper (HVC). The concept of drilling and blasting to maximize mill throughput is not new, at least with respect to the crushing component. However, the concept with respect to grinding, also, as part of the operating and economic equation is a recent concept.

It appears that a variation at HVC in drilling and blasting practices can mean up to a 30% improvement in mill throughput, at no corresponding deterioration in metallurgy. In the unique situation at Red Chris with materials moving downwards and out the bottom of the pit, the effect of comminution by blasting, dozer grinding, and ore pass grinding is much more enhanced and we should pursue this area as much as possible.

To a certain extent, this subject is related to the grade and dozer push component of the project. All of these subjects are somewhat related. On one hand, we want to have the highest mill throughput that the geometrics of the ore zone will tolerate, with respect to ensuring the highest grade within the limits of the practicality of the dozer. This highest limit should be after consideration is given to the blasting and grinding components of the dozer and ore pass. However, other factors should be considered:

i. The cost of powerline infrastructure.

ii. The capital and operating costs of the mine and mill complex will be an important component in determining the final capacity. In general, at any mine, the higher the capacity the lower the unit operating costs and the lower the capital cost per tonne of ore milled. This is particularly true, as in our case, when the reserves can support a long life of mine.

    However, at Red Chris there is more benefit in having a large mill capacity. In general, more capital and operating costs are directed towards the milling cost center than the mining side. The direct mining cost per tonne of milled ore is about 33% of the total, whereas the milling cost is 58%.

    When the improvements are included with respect to the underground network, the capital cost associated with the mine will be much less than those associated with milling. This is primarily due to the mining style that accommodates and takes advantage of gravity as opposed to resisting gravity and lifting all materials. The operating cost advantage is obvious, but the capital cost advantage is less obvious.

    There is very little advantage of the proposed mining style over the traditional open pit mining style, with respect to initial capital. The cost of purchasing enough shovel, truck, grader, 824 and other equipment capacity to produce 30,000 tpd of ore is about the same as in the recent pre-feasibility for crawler power, ore passes, underground work, and ore and waste handling systems. This was initially surprising. However, all of the underground work is a result of capitalizing on the gravity and dozer push advantages and must be charged accordingly.

    However, the capital cost associated with the dozer push-underground system is a different type of capital.

    (a) The new system will be put in for a maximum of Canadian dollars vs. U.S. dollars for a traditional system.

    (b) The new system will require much less sustaining capital. The capital associated with the dozer push underground system does not involve a high change out and replacement cost as does mobile equipment.

In summary, the final mill capacity should be chosen as a function of the many variables that affect that decision. The geometrics and mineability of the ore zone should be the prevailing concern, such that the highest grade can be realized. Coupled with this, and realizing that this is very much a mill dominated mine in terms of capital cost and operating cost, it is important to push the capacity to a level whereby the capital cost is approaching an optimum, as is the operating cost. Other variables, such as location and power supply, also play a role.

My gut feeling is that this optimum is to size the mine at about 17,500 tpd and run it at about 24,000 tpd.

Drilling and Blasting

It is likely that the whole subject relating to drilling and blasting is intimately tied to the question of dozer grinding, ore pass grinding and maximum throughput. At this time, we have S0.28/tonne into drilling and blasting costs and plan to use 9-inch sized boreholes. It is anticipated that greater than 95% of the holes will be dry. The overall drilling and blasting cost is about 30% greater than normal and the purpose of this was to ensure good dozer productivity and to cut the risk of blockages in the various ore passes.

However, recent work by HVC has indicated that there might be significant merit in drilling and blasting for the objective of maximizing mill throughput. There could also be additional advantage to Red Chris because of the grinding components.

More work is required in this area and it could be that drill hole diameters should be reduced (i.e. to 8-inch or even 6-inch) and the pattern size be crimped further. However, there is very little risk in this area. A marginal growth in drilling and blasting costs will have benefits on the productivity side that will more than offset the increase.

Mine Plan

The mine plan for the Red Chris attempts to maximize grade by diving down on the high grade ore and at the same time hold the stripping ratio on an even keel.

This is always difficult in prismatic types of pits, particularly those with high grade cores. It was a little easier at Red Chris because, as the pit design moved inwards to capture the high grade ore, the walls steepened at the same time. One result of this is that the overall stripping ratio did not deteriorate from the FDW study (i.e. 1:4:1).

However, to hold the stripping ratio even had a further consequence and that is that the number of mining cuts are too narrow in places.

Although the temporary walls have accommodation for a narrow 10-meter ramp grading –15% built into the wall angle, the fact remains that this is a very development-intensive pit.

The intensity of development lies around the proper scheduling of ore pass development, ore campaign mining, low grade mining and waste campaign mining, as well as the proper layout of the intermediate access ramps. This is pushed to the maximum in the drive to pull as much grade forward as possible.

The mine plan and operational capability could be improved significantly by reducing the number of cuts and widening the remaining cuts. This would play into the economic viability of increasing dozer pushing to the 400-meter region and correspondingly reduce the underground development.

The end result would be a reduction of unit costs due to larger mining areas and reduced development.

The negative side of this is that the stripping ratio will go up in the early years of the mine's life. However, I believe that increased stripping can be justified by the following:

i. Reduction of underground work by virtue of increasing the dozer push distance. It is highly desirable to reduce the underground work. However, this will likely mean wider cuts.

ii. To increase the stripping at any point in time is not as difficult as in traditional open pit truck-shovel mining. To increase stripping is to increase drill and dozer power, both of which are mobile (i.e. can be brought onto the property on low bed, etc.). This is much easier than trying to increase shovel and truck power.

Consequently, leasing arrangements or contractors could be utilized on a short-term basis.

iii. Larger cuts will bring simplicity to planning and operating through less development. This will result in cost benefits.

iv. Overall unit costs should drop because of efficiencies due to large volume mining.

v. The mining cost/tonne is not the significant cost with respect to the overall cost. The significance is less because of the mining style. Consequently, to increase stripping on the front does not affect the economic equation as much as one would normally expect.

vi. Reducing the number of cuts, tends to preserve the integrity of the ore body. The ore zone generally decreases in grade concentrically as we move away from the center. Consequently, most grade boundaries will be concentrically parallel to the defined cuts. When the grade boundaries (i.e. ore-low grade and low grade-waste) are in the vicinity of cut boundaries, the integrity and ability to separate materials cleanly always suffers.

vii. Reducing the cuts and increasing the stripping ratio on the front part of the project guarantees the flexibility to control to the best ability the grade going to the mill. It is assumed that this probably means the highest grade. However, with this kind of flexibility, it might be desirable to have a lower grade strategy in times of low prices and systematically preserve stockpiles of higher grades for times of higher prices.

viii. Probably most important, by accepting larger cuts and increased stripping ratio on the front, will allow a stockpile of lower grade ores and marginal materials to be built up at some <u>rate of development</u>. It is this <u>rate of development</u> of stockpiled material that should define the timing of future expansions of the mill and the magnitude of those expansions.

Again, there must be some balance in this type of analysis between underground development, dozer push, cut size, open pit development, mill capacity and operational practicality.

However, the analysis should be carried to the extreme in attempting to reduce the complexity of operational mining. That extreme should include an analysis of mining to final pit limits on the first pass and on each bench as the pit progresses downward. This might not be the optimum solution, but certainly it should be tested and a reduction to only two cuts might be viable.

The justification for mining to final limits initially would be based on the above 8 points. It has now been demonstrated that the dozer push distance could be as much as 400 meters. Theoretically this would require only 2 ore passes, one in the East Zone and one in the Main Zone. This would involve a reduction of about $\$20 \times 10^6$ in underground costs if the two ore pass situation could be attained. On the basis that about $\$20 \times 10^6$ could be saved if only 2 ore passes would be required and that the unit costs of waste could be reduced to 0.55/tonne from 0.65/tonne if all waste was removed in the first 10 years of mining, then the effect on the economics would be marginal.

Consequently, further work on reducing the cuts to a minimum is well worthwhile. The benefits are simplicity, reduced development costs, reduced underground costs, reduced unit operating costs and increased flexibility for other options (i.e. low grade stockpiling and milling, planned expansions, etc.). In my view there is good opportunity to improve the economics by substantially reducing the cut numbers. This is an example of this paradigm of open pit mining being different that traditional mining. By utilizing gravity to the maximum and using dozers on a downwards push to their limit, both the capital and operating costs are reduced to a minimum and unit mining costs for mining are not as an important factor as in traditional open pit mining. Consequently, the economic benefit of waste deferral is not as great.

An understanding of all of these factors will ultimately lead to the underground network costs, ore pass spacing and cut widths. In any event, significant improvement is anticipated from the pre-feasibility plan

Mine Scheduling

The scheduling of ore and waste materials will also be different for this type of pit than traditional pits. Some of the considerations are as follows. The ore pass performs the function of materials transportation link as well as sinking cut access to the next bench. However, in traditional open pit mining, sinking cuts to establish an ore flow are normally temporary and do not interfere with the ability to continually produce ore and waste from upper benches.

In the case of Red Chris , with the ore pass used as both a transportation network and as a sinking cut, it will be necessary that all waste, low grade and ore be removed and down the ore pass before airtrack development around the ore pass begins. Once any development starts, the ore pass is out of commission for production for at least 2 weeks. If we assume that only one ore pass is in each ore zone, and that they are centered in the heart of the highest grade ore, then all development muck will be ore. Further production around the ore pass will likely be from 4 or 5 large production blasts which will be almost 100% ore or at least ore and low grade. Consequently, it will be a considerable period of time that a single pit can be a producer of ore and waste on a stand-alone basis to meet daily milling and waste production requirements. During this period of time, the other pit will have to be developed and capable of supplying the ore and waste requirements. The geometrics of the two ore zones plus the location of the ore passes coupled with the necessary development style around the ore passes leads to a number of conclusions about this type of mining.

i. From all points of view, it would be highly desirable to establish a campaign mining style in which large quantities of waste and/or ore could be moved through the system over intermediate or long periods of time.

ii. A large quantity of milling grade ore and low grade ore should be in live inventory at the mill site. In future planning it is likely desirable to include higher rehandling costs for a live inventory stockpile.

iii. The engineering and operational management of this type of pit will be driven by the need to develop a bench within a schedule and move ore and waste materials on medium to long range production terms. It will not be driven on the terms of short term mining production targets or milling targets in the short term. Consequently, the engineering and operational management will have to be very disciplined to stick to a development and campaign mining style for the long term.

This again points to a different paradigm. Although the tools of traditional open pit mining are utilized (i.e. large equipment), the movement of materials to a certain point, the nature of transmitting various materials through common restricted areas, and the erratic availability of ore, low grade and waste materials all point to an adaptation of a new organizational, engineering and operational style. If one does not think this through properly then the risk of being alternatively development bound, ore bound and waste bound could be very high.

Bench Height

To a large extent, bench heights are determined by regulation criteria, ore body geometrics, and blasting criteria. We have the opportunity to increase our bench thickness, mainly because the mine regulations do not apply to this style of mining (i.e. maximum height of muckpile approximately 5 feet above the top of the sheave wheel). We have to demonstrate that the operation will be safe to employees. However, we will be working the muckpile from the top down, rather than from the toe.

There could be some problems with this at the back limits of the blast which might have to be accommodated for by leaving buffers, etc.

The expansive nature of the ore zone in the horizontal sense and the good vertical continuity should allow the ore zone to not be adversely affected by very thick benches.

There has been some work lately that points to an optimization of burden, spacing and collar lengths with thicker benches that can lead to superior fragmentation.

The possibility of increasing bench height should be pursued.

Diameter of Ore Passes

The diameter of the ore passes is currently 3.1 meters. There are varying opinions on this, but in my view the diameter should be increased to 4.0 - 5.0 meters. In general, the 3.1 meter diameter ore pass is adequate for some 99% of the muck. However, we are averaging 75,000 – 80,000 tpd through the ore pass network and we have to be absolutely certain that they do not hang up.

As a safety measure in the pre-feasibility we included an extra ore pass in each ore zone (i.e. 80 meters apart) in case an ore pass hung up. It might be better to increase the diameter rather than simply adding an equal diameter ore pass that has the same risk. As the above has indicated, there is opportunity to significantly reduce the number of ore passes, perhaps to only two. If this is the case, then we could effectively put some quality funding into the remaining ore passes.

Underground Transportation Options

In order to move the pre-feasibility along, it was decided to incorporate a conveyor as the underground transportation system. This was based mainly on the experience of the majority of the members of the Senior Review Board and, to a certain degree, on some concern with respect to ore pass blockages.

The conveyor system was not compared to other underground transportation systems. However, it was clear that in order to incorporate an underground conveyor system we would have to crush the material prior to it going on the belt. A significant amount of time was devoted to studying options with regard to in pit or underground crushing systems. It was finally decided to incorporate an underground crushing system by utilizing jaw crushers at the bottom of each ore pass. Unfortunately, the decision to use a conveyor resulted in the necessity to also crush waste.

The end result is an expensive underground system, the necessity to crush waste, and a crushing system that is not compatible with the productive capacities of the mine or the mill. With the benefit of hindsight, it is clear that the entire underground cost is very high and 50% of that cast is devoted to the ore and waste handling system. The decision to choose a conveyor system backed us into a corner as far as crushing waste, and we have accepted the capital and operating cost of crushing waste. However, it makes no sense to be forced into crushing waste as a result of choosing a transportation network which requires crushing prior to loading on the belt. In fact, 60% of the crushing capital and operating cost is devoted to a function that has no economic benefit to the operation.

Again, with the benefit of hindsight, it is clear that the weak link in our mining system is the bottle neck situation in between the open pit and the mill and/or dump at the surface. That weak link is the numerous ore passes, jaw crusher stations and transfer points prior to getting the material on the main line belt.

The mine plan allows for close to an average of 80,000 tpd. This is an extremely large mine. In terms of the size of Similkameen in the 1970's, this represents 340 M-85 sized loads per eight hour shift. This is a lot of open pit material to be "thread through a needle" in ore passes and crushed by numerous jaw crushers, which in sum total are not meant for this size of operation.

Crushing underground is only preferable to crushing in the pit. We chose crushing underground mainly because we did not want to have a restricted area at the end of the dozer push and prior to the ore pass. However, crushing underground is not the best situation particularly with numerous crushers that are each sized for significantly smaller production

This situation exists because we chose the conveyor system. Alternatives must be examined. The most obvious alternative is a very large train system transporting materials to the waste dump, stockpile and mill. The train must also be sized to transport 80,000 tpd and must consequently be very large. It is envisaged that car size would be about 90 – 100 tonnes, similar to that used at the Carol Mine at Iron Ore of Canada. The train would have to be of this sized magnitude to transport 80,000 tpd. If these were 20 cars/train, each trip would have to be done in 20-25 minutes to meet production requirements.

The advantages of the train option are as follows:

i. No crushing would be required underground. It is envisaged that hydraulic operated chutes would be utilized to load the train, directly from the ore passes.

ii. Waste and low grade would not be crushed at all.

iii. Crushing of mill feed would be done on the surface, utilizing either a single or double gyratory crusher. This is the type of crusher that is compatible with the size of operation (i.e. 30,000 tpd) and for expansion possibilities up to 40,000 – 50,000 tpd. If the initial mill size were chosen to be 10,000 – 15,000 tpd the ideal crusher would also be a gyratory crusher.

iv. The train system would be simpler, with overall fewer moving parts.

v. By not crushing waste, we reduce the impact of exposing sulfides within the dumping network and, consequently, reduce the acid rock drainage potential.

A cost comparison between underground haulage alternatives has not been done. The train option would require a larger drift and considerable work would have to be done to determine the waste dump system. However, the ability to add some simplicity, reduce the impact of restricted areas, eliminate ail waste crushing, and use proper gyratory crushing for ore at surface has considerable merit.

A disadvantage of the train option could be the ability to expand the ore and waste tonnage. This would have to be considered as the ore body and the mine present an excellent opportunity to expand.

Further review of the pre-feasibility will be addressed in additional memos. However, the conclusions to date are:

1. The ore zone is definitely zoned in grade. An understanding of this zoning will better allow a choice of mill size. It is likely that the initial mill size should be in the 15,000 to 20,000 tpd territory.

2. There is tremendous flexibility with the ore zone in terms of both grade and tonnage. We have always known that there was flexibility with tonnage, but the magnitude of the versatility of grade has only recently been discovered while preparing this pre-feasibility. Flexibility is very important. Neither Huckleberry nor Mt. Polly has it. Red Chris has the opportunity to start at very high grade and low tonnage (i.e. 15,000 tpd) and potentially to have a series of expansions up to about 50,000 tpd, with minimal risk associated with capital.

3. The dozer push concept has many advantages in terms of operating costs, capital costs, final highwall angle, reduced pre-stripping cost, contribution to grinding, the realization of high copper grades in the early life of the project, and reduced development costs underground. The concept should be studied further with larger crawlers.

4 The work that Highland Valley Copper is doing with the mine to mill process, and particularly with respect to the Julius Krushnett Mineral Research Center (JKMRC), is very important. The mining style that is being incorporated at Red Chris stands to benefit more from this work than any other mine in the world.

5 It is likely that it is desirable to increase both the drilling and blasting cost functions as a result of JKMRC mine to mill process work.

6. Much more work should be done in mine planning with respect to cut width, dozer push distance, access practicality, mine scheduling, campaign mining and adoption of practical and simple planning (i.e. operational) procedures. This work should be manual, based on the existing computer plan.

7. Technical criteria such as bench height, ore pass diameter, and alternative underground transportation options should be evaluated as they inter-relate with each other.

The present study is a vast improvement over the original FDW Study. However, the project can be improved utilizing the above criteria and, more importantly, it can be further improved in areas that we have not discovered. This will simply take additional work.

July 24/98

D.J. Barker, P.Eng.

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American Bullion Minerals Ltd.

Memo

To: File

From: Don Barker

Date: July 30, 1998

Re: Mine and Mill Capacity – Red Chris

In the memo dated July 23, 1998 entitled "Pre-Feasibility Study 1998" it was implied that there is a good argument for reducing the mill capacity to the 15,000 – 20,000 tpd range. This was based on the view that the high grade core was about 120 x 10^6 grading 0.58% copper and 0.47 g/tonne gold.

A number of other variables are contributing factors to a mill size decision:

i. The economy of scale for capital and operating costs for the mill. If it is important from an ore body point of view to have a reduced mill capacity, one has to consider the lower end of the economy of scale curve.

ii. The rate of development of low grade is important because economy of scale and incremental economics of the developed low grade can contribute to the decision on mill capacity. In the previous memo it was suggested that the rate of low grade development could be a contributor to the timing and capacity of mill expansions.

iii. From a mining point of view, it might be very desirable to increase the dozer push distance, decrease the amount of underground work, decrease the number of mining cuts and thus increase the amount of low grade being developed annually.

Consequently, the engineered pit design and the mining style to attain that design also play a role in the mill capacity decision. This means that not only should the rate of development of low grade play a role as to future expansions of the mill but that rate should also play a role with respect to the initial design capacity.

An ore body which has a discrete boundary around 120 x 10^6 tonnes grading 0.58% copper and 0.47 g/tonne and is solid waste outside that boundary might demand a mill about 15,000 tpd. However, the same ore body that has peripheral low grade material which must be removed to release the ore and the material can economically contribute to the mill economy of scale, might demand a higher capacity initially.

This is the type of analysis that must be done in the future to determine the final capacity. Consequently, it is clear that more work is required on the ore body with respect to grade distribution and geology.

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