Tom Schroeter
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AMERICAN BULLION MINERALS LTD.

RED CHRIS PROJECT

SCOPING STUDY

D.J. Barker and Associates Inc.
MINING AND GEOLOGICAL SERVICES

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For:

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INTRODUCTION

This document outlines a new directional plan and strategy for the Red Chris Project. This new plan and strategy is based on a review of the geological database, a review of the mine plan as outlined in the Fluor Daniel Wright (FDW) Prefeasibility, and some new technical input into the project.

This work has been done over the past few months and has, to a large extent, been conceptual in nature. However, the new direction is superior technically, economically and environmentally.

The objectives of this report are as follows:

i. Finalize on a scoping study for American Bullion Minerals Ltd.;
ii. Establish a new and improved mining scenario along with identifying the benefits and problems with the new mining direction; and,
iii. Draw conclusions and make recommendations for future work.

Many individuals have contributed to the thought process in this document, including Mr. Norm Anderson, P.Eng. and Mr. Roger Taylor, P.Eng. Mr. Scott Broughton, P.Eng. has made a significant contribution in terms of the plant site location, underground access network and geotechnical considerations.

SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

1. The Red Chris ore zone is made up of a high-grade core, which can be extracted selectively through a modified dozer push mining plan. The dozer push plan will allow grades to be mined at approximately 35% better grade than the average resource grade. This is accomplished by mining at reduced ore capacities (i.e. 30,000 tpd vs. 90,000 tpd) and by improved mining geometrics around the ore zone.

2. A dozer push open pit mining style that allows ore and waste to be drawn through ore passes and transported to a lower elevation mill site and dump site will reduce unit mining costs by about 40%.

3. The dozer push mine plan will allow pit walls to be steepened by reducing water pressures on the walls, by the elimination of wide haulage ramps and by the dozer mining style which eliminates shovels and trucks.

4. The new mine plan will show significant reductions in development cost, capital cost of mining equipment, and result in improved simplicity to the operation.

5. The new mine plan capitalizes on a high-grade portion of the ore zone to produce 210×10^6 tonnes of ore over 20 years, at a rate of 30,000 tonnes/day. This scenario is developed as a first pass. This new plan has the best chance to reduce capital and operating costs and also capitalize on the best grades of the ore zone. The plan should not be treated as a stand-alone plan. It should be treated as the best plan to reduce the debt and provide the cash flow to expand the mine in a systematic manner, such that higher milling capacities can be realized at a later date. At present the 210×10^6 tonnes of high-grade represents 40% of the defined resource.

   It is important to ensure that the overall resource is accounted for in short-term design and that development undertaken for the mine plan can be readily utilized for an expanded plan.

6. In more detailed engineering work, the entire resource should be considered in determining the plant site layout, waste dump layout and the tailings area.

7. The new mining direction is a plan that conforms to the basic principles of adapting the movement of materials to the laws of gravity. The proposed scheme is somewhat innovative with respect to present day mining practices. However, each component is part of proven mining practices and the associated difficulties are simply problems to be overcome. Some of these problems relate to dozer push distances, ore pass diameter, ore pass production capacity, crushing requirements and on level transportation styles. These problems are important and will require additional specialized expertise to address.

BACKGROUND AND HISTORY

The Red Chris Project has been well described in the various geological documents and within the Prefeasibility Study Report by FDW. A summary is as follows.

The copper-gold resource is defined by a 0.2% copper cutoff to contain about 500 x 10^6 tonnes at 0.32% copper and 0.254 g/tonne gold. This resource is defined by 244 drill holes and is capable of being mined by traditional open pit techniques at a stripping ratio at 1.39:1. The copper gold resource is prismatic in nature and, both copper and gold grades increase in depth. There appears to be a strong relationship between the gold and copper grades and, consequently, a high degree of confidence can be given to the resource.

A prefeasibility report was completed by FDW in April 1996. This study represents the initial engineering analysis of the project. The plan was for traditional truck-shovel open pit mining style in which production mill capacities were planned at 90,000 tonnes/day, such that a seventeen-year mine life was realized to give the best unit costs and best capital costs/tonne of reserves. The prefeasibility gave a 10% IRR (after tax) at gold prices of $400.00/oz and copper prices at $1.00/lb.

Early indications were that the data collection from drilling and reserve calculations was of high quality. Independent reserve calculations by FDW and Montgomery Consultants proved to be reasonable. The conclusion is that the resource definition and geometrics are good.

A review of the FDW study indicated that the study was a traditional approach, with little or no optimization as far as the style of mining. Trial runs of various tonnage capacities indicated that the best rate of return would be realized at the highest capacities. This led to the conclusion that the best case was for a 90,000 tpd mill indicating a 10% ROR on a capital cost of \$550 x 10^6. Although the IRR was marginal, the greater concern was for the high risk associated with the capital cost.

It was clear that the pre-feasibility could be improved. A large contributor to the capital cost was the powerline (\$100 x 10^6). This could be reduced by reducing the plant capacity. However, it was also decided to reduce the plant capacity such that the capital investment for the project could be seen as reasonable. It was also clear that the FDW study had not been optimized in terms of infrastructure locations, dump locations, ramping locations, access and general planning and capital costs. It was decided to review the FDW study with the view to change ramping locations, primary crusher location and damp locations. The intent was to minimize the capital and operating costs for a 30,000 tonnes/day operation. At this time, commodity prices dropped by about 15%.

It became clear that a topographic advantage was available from the south side of the pit area which would allow a level adit access from the 1050 elevation, in the vicinity of the northern part of Kluea Lake to the bottom of the defined pit. Given that the grade of the ore zone increases in depth it was decided to determine if a significant tonnage of economical underground ore existed, such that an underground operation could be run in conjunction with an open pit and better grades could be realized in the early years of the project. This was tested, and it was determined that about 20 x 10^6 tonnes grading 0.7% Cu and 0.65g/tonne Au existed in the vicinity of the pit bottom. Although it was determined that this material might have marginal economics, the real advantage was emerging as to how to open pit mine the ore zone by dropping the ore down an ore pass and tramming it to a plant site in the vicinity at the lake elevation (i.e. 1041 elevation).

It was decided to pursue this strategy because preliminary evaluation of costs indicated that operating costs would be significantly reduced. The pursuit of this strategy indicated many advantages on the development side of the economic equation as well as on the capital and operating cost side. The developed strategy is economically more sound, as well as developing a better environmental plan and stability of materials plan. In its basic simple form the new plan conforms to the basic principles of gravity, as opposed to a plan that resisted gravity for all material movement.

The following outlines this strategy:

Geology

The most important aspect of this ore zone is the basic geometrics of the zone with respect to potential mining opportunities.

The ore zone parallels existing structures, running about N 60° E. The ore zone is prismatic in nature and is continuous. However, two strongly mineralized zones forming the Main Zone and the East Zone allows a figure eight shaped pit to be developed around the high grade zones as well as capturing the low grade between the two zones.

In both the East Zone and the Main Zone, the copper grade increases in depth. The gold grade also increases in depth, although more dominantly in the East Zone.

The important features at the East and Main ore zones are as follows:

i. The combined ore zones form a very large ore zone, presently outlined by an open pit at 550 x 10^6 tonnes. However, the ore zone could be increased significantly by increasing the depth and utilizing improved mining economic parameters. The cutoff for the resource calculation is about 0.2% Copper.

ii. The ore zone is enveloped by lower grade zones. A review of the raw data indicates that higher-grade continuity occurs in both the vertical and within the higher-grade core of each ore zone. This indicates that substantial tonnages exist which are at elevated grade levels, and the ore zone can be defined by discrete tonnages as a function of raising the cutoff grade. This conclusion is arrived at by an evaluation of the raw data, rather than evaluations of geostatistical data.

iii. The grade increases substantially with depth. This poses a cash flow problem for traditional open pit mining techniques.

These features of the ore zone are the more important aspects and must be addressed properly in any mine plan. The following mine plan addresses these characteristics by:

i. Capitalizing on the highest grade possible for a 20-year mine life.

ii. Utilizing a mining technique that centers on the highest-grade ore in both the vertical and horizontal sense.

iii. The plan develops the ore such that the high grade carries the capital debt and potentially allows the lower grade to become economical.

The ore reserves are dominated by the copper mineralization and gold is proportional to the copper content. A high degree of confidence can be given to the resource classification of proven and probable ore, and approximately 92% of the resource are within this category. Both Montgomery and FDW checked and confirmed geologic resources.

Mine Plan

The mine plan is defined by the following criteria:

1. Open pit surface ore is to be drilled and blasted by large diameter drills (i.e. 10" boreholes). Blasting would be done exclusively with dry Anfo with a minimum of slurry. This will achieve the lowest cost, best fragmentation and maximum swell in the muckpile. Anticipated drilling and blasting costs are $0.21/tonne including secondary breakage.

2. The ore would be moved to ore passes located at approximately 300 meter centers. This is based on 150 meter optimum push distances for dozers. The ore passes would be no less than 3 meters in diameter and possibly up to 5 meters.

3. The material would be moved to the ore passes by very large dozers (i.e. D-11-CD). The estimated average cost is $0.09/tonne for a maximum 150 meter push.

4. Underground haulage is estimated to be 0.08/tonne. This is based on train haulage or similar bulk material transport system.

The plan would be to do minimal pre-production development such that an ore pass could be developed on both the East Zone and the Main Zone. In the vicinity of the ore pass, preliminary blasting would be done with an air track using 2¼" holes, such that small muck could be guaranteed in the first blast. Plugging of an ore pass is a major concern. A large diameter drill would be utilized for production blasting at some distance from the ore pass. A production blast would have a 30% higher powder factor than normal open pit mining to ensure a high degree of fragmentation.

Production blasts will always have about 1 – 2 % oversized muck, and this will be moved to selected areas and fragmented by secondary blasting. Heavy reliance will be on dozer operators to ensure that no oversized materials are allowed in the ore pass area. It is not anticipated that a grizzly will be required. A grizzly could cause other problems and reduce productivity.

It is anticipated that conventional hydraulic chutes will be utilized to load large Granby cars pulled by an electric locomotive system. Underground haulage costs are anticipated to be about $0.08/tonne. Alternative material transport schemes exist and will be evaluated, but are likely to have similar capital and operating costs.

It is also anticipated that the ore haulage system will be located at the 1050 elevation, with a portal located at the north end of Kluea Lake. A surface tram will continue to the area north of the lake. Total unit operating costs for this mining plan are anticipated to be about $0.40 – 0.45/tonne ore.

Waste mining can be achieved in a similar way to the above, however haulage routes are to the large valley to the north east of the pit. It is anticipated that this area to the east of the pit will carry the volume of waste, not only for the initial mining phase but also for future phases. The following contractor costs are anticipated.

Underground Ore Haulage Access	=	\$7.5 x 10^6
Underground Ore Passes	=	\$2.7 x 10^6
Waste Passes	=	\$2.0 x 10^6
Main Waste Haulage	=	\$6.25 x 10^6
Underground Waste Access	=	\$2.5 x 10^6
		\$19.95 x 10^6

Table 1 indicates the available tonnage and grades defined by a pit outlining approximately 200 x 10^6 tonnes of the best grade ore. The pit has been extracted from tables 5.2 and 5.3 and are approximate formulations that the average slope angle is 55°. The average slope in the FDW design was about 40°. The resulting average stripping ratio is 1.45:1. Table 2 is a production schedule for a 30,000 tpd mine and mill based on Table 1.

Table 1 – Dozer Pit

Bench	Ore (x 10^6)	Copper (%)	Gold (gm/tonne)	Waste (x 10^6)	Strip Ratio	Remarks
1575				0.08		
1560				0.44		1. Pit volumes and grades for highest grade 200 x 10^6 tonnes.
1545	0.117	0.31	0.23	3.27	27.95	
1530	1.8	0.39	0.22	10.32	5.73	
1515	3.52	0.37	0.23	15.20	4.32	2. Average pit slope is 55°
1500	4.58	0.37	0.28	18.15	3.96	
1485	5.48	0.37	0.29	20.13	3.67	3. Ore proportioned from Table 5.3 i.e. 210/536 = .3918
1470	6.34	0.39	0.31	21.62	3.41	
1455	6.89	0.39	0.32	22.26	3.23	
1440	7.55	0.39	0.31	21.50	2.85	4. Tonnes / Cu / Au 536x10^6 0.318 0.251 207 x 10^6 0.462 0.38
1425	7.71	0.41	0.31	20.33	2.64	
1410	8.42	0.40	0.29	18.39	2.18	
1395	8.49	0.39	0.29	16.98	2.0	
1380	8.53	0.40	0.28	15.65	1.83	5. Copper better by 45%
1365	8.57	0.41	0.29	14.28	1.67	
1350	8.77	0.42	0.30	12.75	1.45	6. Gold better by 51%
1335	9.27	0.42	0.31	10.97	1.18	7. Above based on COG $8.00 (Table 5.2)
1320	9.51	0.44	0.34	9.52	1.00	
1305	9.55	0.46	0.37	8.31	0.87	
1290	9.47	0.48	0.39	7.22	0.76	8. Based on volume of cone calculation $V = {}^1/_3 \Pi R^2 H$
1275	9.27	0.51	0.42	6.21	0.67	
1260	9.16	0.52	0.45	5.20	0.57	
1245	8.81	0.53	0.46	4.48	0.51	
1230	8.30	0.52	0.46	3.91	0.47	9. Volume of new pit = volumes of FDW pit/2.87
1215	7.87	0.51	0.45	3.27	0.42	
1200	7.24	0.51	0.46	2.82	0.39	
1185	6.81	0.51	0.46	2.34	0.34	10. Actual waste is 246 x 10^6 raised to 300 x 10^6
1170	6.14	0.51	0.47	1.61	0.26	
1155	5.28	0.52	0.50	1.05	0.20	
1140	4.54	0.53	0.52	0.73	0.16	
1125	3.52	0.57	0.58	0.44	0.13	

Table 1 – Dozer Pit (cont'd)

Bench	Ore ($x\ 10^6$)	Copper (%)	Gold (gm/tonne)	Waste ($x\ 10^6$)	Strip Ratio	Remarks
1110	2.62	0.60	0.65	0.28	0.11	
1095	1.92	0.63	0.67	0.16	0.08	
1080	1.53	0.67	0.70	0.08	0.05	
1065	1.13	0.73	0.79	0.04	0.04	
1050	0.82	0.78	0.86			
1035	0.39	0.84	0.94			
	207.2	**0.46**	**0.38**	**301.20**	**1.45:1**	

Table 2 – Production Schedule

Year	Stockpile $x\ 10^6$	Ore Milled $x\ 10^6$	Copper Grade %	Gold Grams/tonne	Waste $X\ 10^6$	Stripping Ratio	Remarks
-1	0.117		0.31	0.23	3.79	32.4	
1		10.5	0.44	0.35	17.325	1.65	
2		10.5	0.44	0.35	17.325	1.65	
3		10.5	0.44	0.35	17.325	1.65	
4		10.5	0.44	0.35	17.325	1.65	
5		10.5	0.44	0.35	17.325	1.65	
6		10.5	0.44	0.35	17.325	1.65	
7		10.5	0.44	0.35	17.325	1.65	
8		10.5	0.44	0.35	17.325	1.65	
9		10.5	0.44	0.35	17.325	1.65	
10		10.5	0.44	0.35	17.325	1.65	
11		10.5	0.44	0.35	17.325	1.65	
12		10.5	0.44	0.35	17.325	1.65	
13		10.5	0.44	0.35	17.325	1.65	
14		10.5	0.44	0.35	17.325	1.65	
15		10.5	0.44	0.35	17.325	1.65	
16		10.5	0.44	0.35	17.325	1.65	
17		10.5	0.44	0.35	17.325	1.65	
18		10.5	0.51	0.46	2.24	0.21	
19		10.5	0.52	0.51	2.24	0.21	
20		10.5	0.64	0.68	2.24	0.21	
		210 x 10^6	**0.46**	**0.38**	**301.25**	**1.43**	

The following are the key points with regards to this schedule:

1. The pit captures approximately 200 x 10^6 of the best grade ore from benches 1545 to 1035. Although the better grade ore is within the central core of the ore zone, the stripping ratio does not climb significantly because of the ramp elimination and wall steepening to 55°.

2. The copper and gold grades assigned to years 1 – 17 are an average of the grades available on the 1545 - 1200 meter benches. Based on the geometrics of the dozer push situation with the ability to concentrate on the higher grade cores at the east and main zone, it is likely that these grades can be attained.

3. It is assumed that the ultimate pit limits can be attained by beginning an initial pit and completing two further pushbacks to ultimate pit limits. Sustaining a stripping ratio of about 1.65:1 over the first 17 years should be sufficient to ensure a continuous ore flow.

4. Pre-production requirements with this plan are approximately 4 million tonnes. This tonnage is significantly less than traditional mining because of the reduced geometric requirements of the dozer-push style.

5. It is anticipated that approximately 200 x 10^6 tonnes of the waste will, in fact, be low grade ore (i.e. Cu =0.22%, Au = 0.16 g/t). Of this, approximately 100 x 10^6 will grade 0.28% copper and 0.21 g/tonne gold. It will be important to stockpile material close to the mill for possible future processing. However, from a milling point of view, low grade will always be available for inventory purposes. Consequently, it is worthwhile to minimize pre-production stripping and hold the operating stripping to a minimum.

6. In general, the practicality of Table 2 is considered reasonable. However, the end conclusion is that approximately 40% of the tonnage of the mineralized zone can be extracted at about 30% higher grade without compromising the stripping ratio, and doing so at a much reduced pre-production cost, capital cost and at an approximate mine operating cost of one half the original cost. For this to happen, the interplay of three or four variables must occur.

i. Slope Angle

The overall slope angle can be brought to about 55° by elimination of the wide access ramp and by reducing piezometric water pressures on the pit slopes by doing the preliminary underground work. This slope would allow for a 15 meter berm on a triple bench (i.e. 15 meter berm every 45 meters). This will satisfy mine safety regulations.

Normal safety criteria call for bench thicknesses to be tied to equipment dimensions working at the toe of the working face (i.e. shovels, FEL's). In this case, dozers will be pushing from the crest. Consequently, it might be possible to further increase the bench thickness (i.e. to 20 meters).

In this pit design, the longest exposure of the final wall will be about 8 – 10 years. Weather conditions are generally dry. Consequently, the maximum slope angle will likely be a function of structures within the rock mass. Considerable work should be done on this, particularly on the temporary walls. It could well be that these walls could be steepened to ± 60 °. This should be based on mapping of temporary walls while mining is in progress.

ii. Although dozers have increased in size in the last few decades, they are small compared to truck-shovel systems. In this particular case, a dozer takes the place of a shovel, trucks, rubber-tired dozers and graders. The operating mode is a simple cyclical rhythm that can move large volumes over short distances. The mode of operation will always be a downhill push to flat and, perhaps, downhill loaded. A large dozer can push 30,000 – 40,000 tonnes/day. The capital cost at this dozer is about \$2.1 x 10^6 and the operating cost about \$0.09/tonne. This system replaces a truck-shovel system costing about \$20 x 10^6 that has an operating cost of \$0.42/tonne.

It is likely that, aside from a dragline, the dozer push mining style is the most economical style in mining as long as the application is practical.

iii. The dozer push mining style allows confined mining around the core of the high grade without comprising on mining geometrics or cost. This ability to be confined around the ore zone allows a minimum of pre-production development.

It is the inter-relationship of these variables that has allowed the schedule of Table 2 to be reasonably practical.

The following estimated costs indicate the advantages of this plan:

Mining Component	**Conventional 30,000 tpd Plan**	**1997 Plan 30,000 tpd**
Pre-production Cost	$22,0000,000	$6,000,000
Capital Cost	$27,000,000	$9,000,000
Underground Development		$20,000,000
Operating Cost/Tonne Ore	$1.96/tonne	$1.14/tonne

The initial capital for pre-production and underground work is almost equal to the pre-production requirement plan. The capital and operating cost savings of the new plan reflects the basic simplicity and the ability to work with gravity as opposed to working against gravity. The NSR of the ore at $325.00 gold price and $0.90/lb copper price is $9.93/tonne. If mining costs of 1.14/tonne of ore can be attained, it is likely that a rate of return of plus 15% can be attained at current commodity prices.

GENERAL COMMENTS

1. The driving of raises ore has significant cost. The cost translates into about $30,000/bench. However, pioneering ramp development in any open pit is very expensive and for each pioneered ramp the additional cost for the tonnage moved will be about $65,000. The present value of the extra development cost will go a long way in paying for the ore passes, in terms of development room and blasting costs. As ore passes are driven in ore, some revenue will be generated.

2. One of the big irritants in terms of cost in the FDW plan was the cost of transporting 143 x 10^6 tonnes of AMD material an extra 7 km to be submerged in the tailings pond. This cost would have played a major role in the high cost of loading and hauling (i.e. $0.42/tonne). Under the new plan this material will have a much shorter haulage distance and can, in fact, be incorporated into the general waste plan with similar costs.

3. Costing with respect to underground raises and haulage routes are based on contractor cost. They do not assume raise-boring or tunnel-boring. It might be worthwhile to consider an ownership situation for underground work.

4. The suggested overall approach to mining is new to British Columbia. However, each component is fairly straight-forward and simple. In the future, the Company will have to identify various operations with similar mining problems and visit those operations. Also, it will be important to engage various consultants with expertise in specialized areas.

5. Although the above plan is for 200 x 10^6 tonnes of ore, final infrastructure development including tailings pond location, waste dump location and plant site location should be done with the entire resource in mind.

6. The outlined strategy and direction could prove to be economical at today's commodity prices. This is important, given that commodity prices are 10 – 20 % lower than the prefeasibility and that the production rate is reduced. It is anticipated that the capital cost would be on the order of $200 x 10^6 or less.

7. Most of the recent work has been geological and mining in nature. The biggest improvements will probably be realized on the mining and geological side. However, little or no work has been done on the milling and infrastructure side. There are potentially some improvements relating to gold recovery, evaluation of concentrate shipment alternatives, and capital costing possibilities.

D.J. Barker, P.Eng.

TABLE 5.2

RED CHRIS PROJECT DEPOSITS

MINEABLE RESERVES IN MAIN PIT (420) BY CUTOFF GRADE ($)

COG($)	Proven / Probable / Possible					Proven / Probable					Possible		
	Ore (Mt)	Copper (%)	Gold (gpt)	Waste (Mt)	Strip Ratio	Ore (Mt)	Copper (%)	Gold (gpt)	Waste (Mt)	Strip Ratio	Tonnes (Million	Copper (%)	Gold (gpt)
0.50	775	0.241	0.194	505	0.65	687	0.253	0.201	593	0.86	87	0.147	0.139
1.00	735	0.254	0.202	545	0.74	664	0.261	0.206	616	0.93	71	0.189	0.165
1.50	686	0.269	0.213	594	0.87	625	0.275	0.216	654	1.05	61	0.207	0.182
2.00	642	0.283	0.223	638	0.99	588	0.289	0.226	692	1.18	54	0.218	0.191
2.50	609	0.294	0.232	671	1.10	558	0.299	0.235	721	1.29	51	0.239	0.199
3.00	572	0.306	0.241	707	1.24	526	0.311	0.244	754	1.43	46	0.249	0.207
3.50	536	0.318	0.251	743	1.39	494	0.323	0.254	786	1.59	43	0.260	0.216
4.00	501	0.330	0.261	778	1.55	462	0.335	0.264	818	1.77	39	0.271	0.226
4.50	462	0.344	0.273	817	1.77	428	0.349	0.276	852	1.99	35	0.282	0.236
5.00	424	0.358	0.286	856	2.02	393	0.363	0.288	887	2.26	31	0.295	0.261
5.50	380	0.376	0.302	900	2.37	356	0.379	0.303	924	2.60	24	0.332	0.287
6.00	345	0.391	0.316	935	2.71	324	0.394	0.317	956	2.95	21	0.346	0.301
6.50	314	0.405	0.330	966	3.08	294	0.410	0.332	986	3.36	20	0.333	0.301
7.00	282	0.421	0.346	998	3.54	264	0.426	0.348	1,016	3.85	18	0.347	0.316
7.50	246	0.442	0.365	1,034	4.20	231	0.447	0.368	1,049	4.55	15	0.367	0.320
8.00	216	0.462	0.385	1,064	4.92	203	0.466	0.388	1,077	5.31	13	0.401	0.339
8.50	188	0.482	0.407	1,091	5.79	177	0.488	0.410	1,103	6.24	12	0.391	0.362
9.00	167	0.501	0.427	1,113	6.68	156	0.508	0.431	1,124	7.19	10	0.397	0.367
9.50	149	0.519	0.445	1,131	7.61	140	0.525	0.449	1,140	8.14	9	0.422	0.380
10.00	131	0.538	0.465	1,148	8.74	124	0.545	0.468	1,156	9.31	7	0.418	0.414
10.50	115	0.560	0.485	1,165	10.12	110	0.566	0.488	1,170	10.65	5	0.436	0.423
11.00	101	0.582	0.506	1,179	11.67	97	0.587	0.509	1,183	12.22	4	0.466	0.436
11.50	90	0.601	0.524	1,189	13.17	87	0.606	0.526	1,193	13.68	3	0.462	0.469
12.00	79	0.624	0.544	1,200	15.12	77	0.627	0.545	1,202	15.54	2	0.506	0.505
12.50	70	0.647	0.565	1,210	17.27	69	0.649	0.566	1,211	17.66	1	0.554	0.518
13.00	63	0.665	0.586	1,217	19.41	61	0.668	0.587	1,218	19.85	1	0.524	0.539
13.50	56	0.685	0.609	1,224	21.89	55	0.688	0.610	1,225	22.36	1	0.541	0.561
14.00	50	0.702	0.633	1,230	24.54	49	0.705	0.634	1,231	25.04	1	0.547	0.581
14.50	45	0.721	0.656	1,235	27.46	44	0.724	0.657	1,236	27.98	1	0.556	0.601
15.00	40	0.739	0.680	1,239	30.69	40	0.742	0.682	1,240	31.30	1	0.581	0.575

TABLE 5.3

RED CHRIS PROJECT DEPOSIT

MINEABLE RESERVES IN MAIN PIT (420) BY LEVEL AT COG = $3.50

		Proven / Probable / Possible					Proven / Probable					Possible		
Bench	Toe Elev'n.	Ore (Mt)	Copper (%)	Gold (gpt)	Waste (Mt)	Strip Ratio	Ore (Mt)	Copper (%)	Gold (gpt)	Waste (Mt)	Strip Ratio	Tonnes (Million	Copper (%)	Gold (gpt)
10	1,575	0.0	0.000	0.000	0.2	0.00	0.0	0.000	0.000	0.2	0.00	0.0	0.000	0.000
11	1,560	0.0	0.000	0.000	1.1	0.00	0.0	0.000	0.000	1.1	0.00	0.0	0.000	0.000
12	1,545	0.3	0.215	0.151	8.1	30.25	0.3	0.215	0.151	8.1	30.25	0.0	0.000	0.000
13	1,530	4.6	0.268	0.141	25.6	5.59	4.6	0.268	0.141	25.6	5.59	0.0	0.000	0.000
14	1,515	9.0	0.255	0.151	37.7	4.21	8.7	0.258	0.148	38.0	4.37	0.3	0.158	0.248
15	1,500	11.7	0.258	0.182	45.0	3.85	11.1	0.263	0.177	45.6	4.11	0.6	0.166	0.274
16	1,485	14.0	0.256	0.191	49.9	3.56	13.1	0.263	0.188	50.8	3.88	0.9	0.157	0.234
17	1,470	16.2	0.266	0.205	53.6	3.32	15.3	0.271	0.203	54.5	3.58	0.9	0.182	0.239
18	1,455	17.6	0.271	0.210	55.2	3.13	16.8	0.276	0.209	56.1	3.33	0.8	0.169	0.230
19	1,440	19.3	0.271	0.203	53.3	2.76	18.3	0.277	0.205	54.3	2.97	1.0	0.166	0.168
20	1,425	19.7	0.280	0.203	50.4	2.56	18.7	0.285	0.206	51.4	2.75	1.0	0.187	0.147
21	1,410	21.5	0.273	0.193	45.6	2.13	20.2	0.278	0.196	46.9	2.32	1.3	0.193	0.145
22	1,395	21.7	0.272	0.187	42.1	1.94	20.6	0.277	0.190	43.1	2.09	1.1	0.174	0.128
23	1,380	21.8	0.275	0.186	38.8	1.78	20.8	0.278	0.188	39.8	1.91	1.0	0.210	0.142
24	1,365	21.9	0.282	0.191	35.4	1.62	20.8	0.286	0.194	36.4	1.75	1.0	0.202	0.131
25	1,350	22.4	0.288	0.197	31.6	1.41	21.2	0.291	0.201	32.8	1.55	1.2	0.235	0.127
26	1,335	23.7	0.293	0.204	27.2	1.15	21.5	0.299	0.212	29.5	1.37	2.3	0.236	0.128
27	1,320	24.3	0.303	0.221	23.6	0.97	21.6	0.312	0.229	26.3	1.21	2.7	0.231	0.157
28	1,305	24.4	0.317	0.239	20.6	0.85	21.6	0.324	0.245	23.3	1.08	2.7	0.261	0.191
29	1,290	24.2	0.330	0.253	17.9	0.74	21.7	0.337	0.260	20.4	0.94	2.5	0.269	0.192
30	1,275	23.7	0.350	0.275	15.4	0.65	21.4	0.357	0.282	17.7	0.83	2.3	0.286	0.211
31	1,260	23.4	0.361	0.293	12.9	0.55	21.1	0.366	0.299	15.2	0.72	2.2	0.314	0.236
32	1,245	22.5	0.364	0.299	11.1	0.49	20.7	0.369	0.305	12.9	0.62	1.8	0.307	0.230
33	1,230	21.2	0.361	0.301	9.7	0.46	19.8	0.366	0.305	11.1	0.56	1.4	0.292	0.246
34	1,215	20.1	0.353	0.298	8.1	0.40	19.1	0.357	0.302	9.1	0.48	1.0	0.277	0.222
35	1,200	18.5	0.351	0.299	7.0	0.38	17.6	0.355	0.305	7.9	0.45	1.0	0.279	0.190
36	1,185	17.4	0.350	0.301	5.8	0.33	16.4	0.355	0.307	6.8	0.41	1.0	0.269	0.203
37	1,170	15.7	0.352	0.307	4.0	0.26	14.3	0.355	0.313	5.4	0.38	1.3	0.320	0.243
38	1,155	13.5	0.361	0.324	2.6	0.19	11.5	0.368	0.330	4.5	0.39	1.9	0.319	0.288
39	1,140	11.6	0.369	0.340	1.8	0.16	9.0	0.387	0.357	4.4	0.49	2.6	0.306	0.281
40	1,125	9.0	0.390	0.381	1.1	0.12	6.8	0.413	0.398	3.2	0.47	2.1	0.316	0.326
41	1,110	6.7	0.415	0.422	0.7	0.11	5.0	0.445	0.445	2.4	0.48	1.7	0.326	0.354
42	1,095	4.9	0.437	0.436	0.4	0.07	4.4	0.452	0.451	0.9	0.21	0.6	0.322	0.321
43	1,080	3.9	0.461	0.460	0.2	0.06	3.7	0.463	0.462	0.5	0.13	0.2	0.430	0.429
44	1,065	2.9	0.504	0.514	0.1	0.03	2.7	0.509	0.523	0.3	0.09	0.2	0.422	0.367
45	1,050	2.1	0.535	0.562	0.0	0.01	2.1	0.539	0.566	0.1	0.02	0.0	0.285	0.312
46	1,035	1.0	0.581	0.615	0.0	0.00	1.0	0.581	0.615	0.0	0.00	0.0	0.000	0.000
47	1,020	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
48	1,005	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
49	990	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
50	975	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
51	960	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
52	945	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
53	930	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
54	915	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
55	900	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000	0.0	0.00	0.0	0.000	0.000
Total		536.4	0.318	0.251	743.8	1.39	493.7	0.323	0.254	786.5	1.59	42.7	0.260	0.216

AMERICAN BULLION MINERALS LTD.

To: Tom Schroeter
BC Ministry of Employment & Investment
Fax: 775-0313

YOU ARE INVITED

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THE "NEW" RED CHRIS PROJECT

Wednesday, November 19th

1:45 p.m. (special preview time for Brokers)

4:30 p.m. (for all interested parties)

(Maximum 40 attendees at each session)

at the

Metropolitan Hotel

645 Howe Street, Vancouver

Vancouver Room, 2nd Floor

Presented to you by representatives of

American Bullion Minerals Ltd.:

(TSE Trading Symbol: "ABP")

Charles Wills, *President and Chairman*
Scott Broughton, *Vice President, Mining*
Richard Whittall, *Vice-President, Finance*
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MEI - Smithers

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FAX: 604-775.0313

RE: RED CHRIS Update

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Number of pages including cover sheet: 19

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Message

Paul - FYI.

Have a good trip to Whitehorse - see you next week.

Tom