13.2.3 2005 Drill Program

Core boxes were collected from the drill platform by Crystallex's exploration geologists in the early morning and late afternoon each day. From there, the core was transported to the exploration camp by the exploration geologists or geological technicians.

Sampling of split core was done at the exploration camp by Crystallex's geological technicians and geologists under the supervision of Dr. Richard Spencer (Vice President Exploration, Crystallex) and Eng. Freddy Quijano (former Chief Geologist, Las Cristinas Project). Samples from the 14 drill holes were delivered to SGS's preparation facility in Tumeremo by Crystallex personnel.

Sample preparation followed the same procedure as was used in the 2003 and 2004 programs described above. All samples were analyzed for gold by fire assay of a 30g aliquot with an AA finish. Samples with grades over 10g Au/t were fire assayed with a gravimetric finish. All samples were analyzed for copper by ICP, and samples from 13 of the 14 holes were analyzed for a suite of 34 elements by ICP after *aqua regia* digestion. Samples from the sulfide saprolite and mixed sulfide-oxide saprolite were submitted for CNSCu analysis by the procedure described for the 2003 program.

Sample QA/QC was undertaken by Spencer (January 2006) for the 2005 drill program.

13.2.4 2006 and 2007 Drill Program

Similar sampling procedures were used in 2006 and 2007 as in previous drill programs and hence will not be repeated here. Differences included:

- Material used for blanks that were inserted at a rate of about 1 in 30 samples was fresh diorite taken from a quarry located some 100km south of the Las Cristinas property.
- In previous drill programs undertaken by Crystallex, pulps were sent directly to the analytical facility by the laboratory that undertook the preparation. In 2006-2007, following the recommendation of Mr. Trevor Nicholson, the independent consultant responsible for the QA/QC program, pulps were returned to Las Cristinas after preparation by the SGS laboratory situated near El Dorado, about 100km to the north of the Las Cristinas property.
- Certified standards were purchased from CDN Resource Laboratories of Burnaby, British Columbia. The standards included:
    - CDN-GS-P5B
    - CDN-GS-1C
    - CDN-GS-1P5
    - CDN-GS-1P5A
    - CDN-GS-15
- Bar-coded and numbered empty pulp bags were registered by the preparation laboratory so that the standards could be inserted in their correct position in the sample sequence before being shipped to SGS Lima, Peru, for analysis.
- Check assays were done by ALS-Chemex in Lima, Peru.
- The sample QA/QC was undertaken by independent consultant, Mr. Trevor Nicholson of Nicholson Analytical Consulting, of Comox, British Columbia. Mr. Nicholson was on site for about 40% of the drill program.

14.0 DATA VERIFICATION

As most of the Las Cristinas database is derived from Placer's work, it is important to note that based on Placer's descriptions of their procedures, their data collection and exploration procedures conform to or exceed industry standards. If conducted as reported, Placer's QA/QC program was high quality. In general, MDA found that, again based on reported methodology, Placer's exploration data were collected in a technically sound manner. According to Placer documentation, quality assurance checks were in place for most of the project, and validation of data was ongoing. Nevertheless, it was clear that additional verification was necessary because one company had completed all development work, there were no independent checks or studies of the work, and most of the original hardcopy data were unavailable for detailed study or auditing.

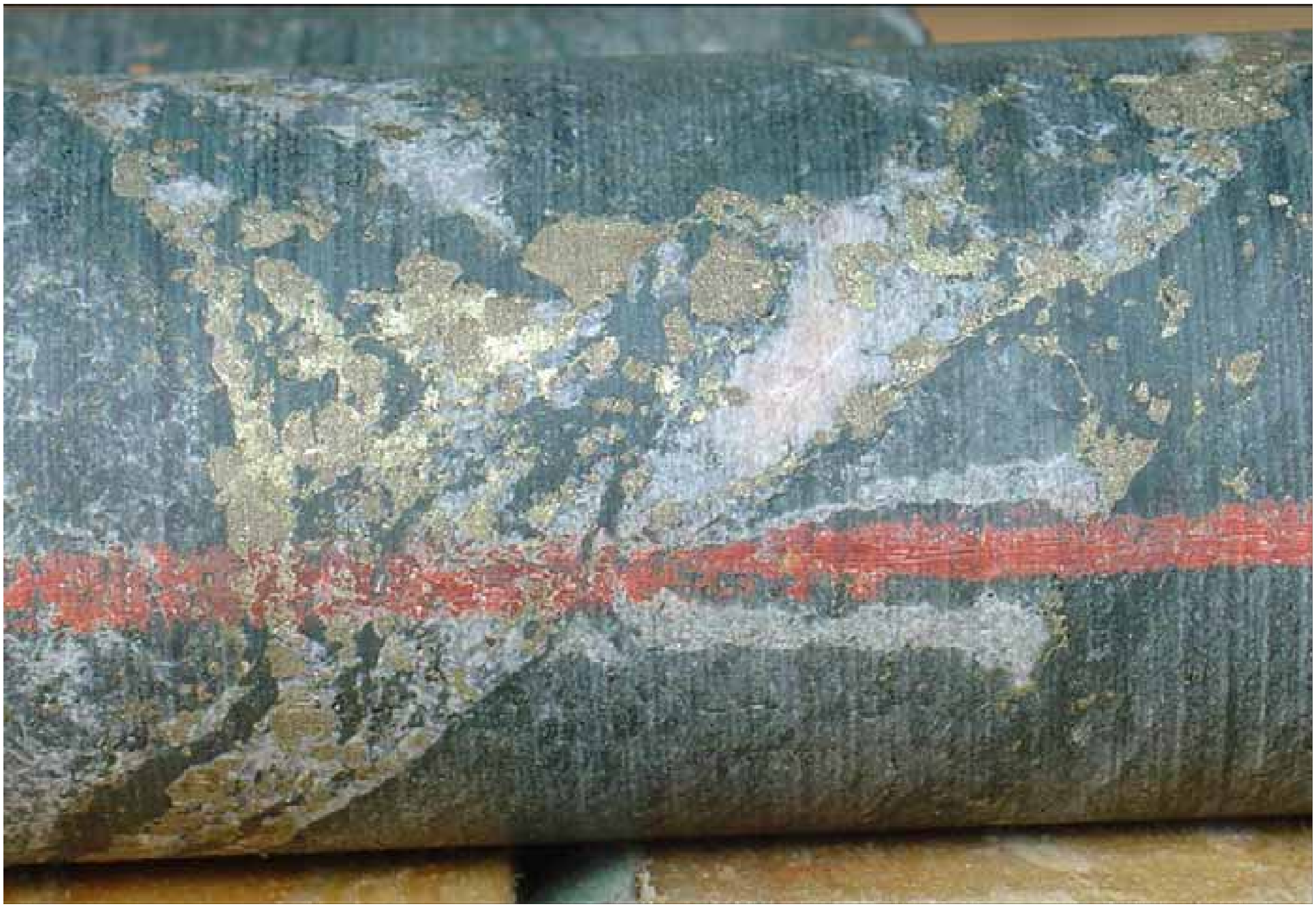
Under the terms of the September 2002 agreement between Crystallex and CVG, Crystallex obtained an electronic database from CVG, which included Placer's drill, topographic, geological, and engineering data. At that time, data from 1,174 drill holes and 108 trenches were included in the Las Cristinas database. Although about 99% of the drill data were obtained, hard copies of the assay and geological data were not available, leaving a gap in the ability to validate the database.

When MDA visited the Las Cristinas site in October 2002, they found drill pads, drill collars, drill core and samples, core photographs, and other supporting data demonstrating that exploration was done in a manner not incompatible with what was described in the documentation of Placer's work. To conduct independent corroboration, Crystallex drilled 2,198m in 12 diamond drill holes, for a total of 1,079 core samples, to verify the presence and tenor of mineralization. These 12 holes twinned previously drilled Placer holes. In addition, 275 QA/QC samples from this drill program were analyzed. The Crystallex drill results and check samples corroborate the general tenor of gold mineralization reported by the previous operator. For additional confirmation, Crystallex re-assayed 262 pre-existing pulps, 200 pre-existing coarse rejects, and 342 quarter-core samples of pre-existing core. Although mean grades are similar for both datasets, there is a large variance in grade between individual pairs of Placer's core assays and Crystallex's core check samples. The variance is lower in the pulp and coarse reject checks. However, as a result of some of these just-mentioned discrepancies, several additional studies were completed to aid in the understanding of grade variability.

Natural grade variability (heterogeneity) is an issue at Las Cristinas. Although it has become better understood through the efforts of Crystallex, it is an issue that should continue to be addressed prior to and during production. The issue can be a problem if left unchecked during production possibly resulting in massive misclassifications of ore and waste. The effect of material heterogeneity on the resource estimate will be dominated by local variance and may have instilled a minor low bias to the sample database. The issue is introduced by the distribution of metals originally in primary ore as shown in Figure 14.1.

For this reason, Pitard (2005) rhetorically questioned: "*Can the existing gold grade database, created with diamond drilling and conventional 30-g fire assays, lead to an accurate block model?*" To which he responded: "*The answer is no. But, with good geology of the various quartz and sulfide events, it can make a world of a difference.*" The problem he is referring to is the ability to estimate accurately locally and with precision. MDA believes that this is difficult to do, but the consequence is not so great that it would negatively impact a mine and deposit of this scale in an open-pit scenario; essentially higher grades will be generally where higher grades are estimated to be, and the same with the mid- and low grades. While the gold occurs in the free state, it is generally not coarse grained nor visible but does appear to occur in clots of sulfides (Figure 14.1). It is not possible to compensate for the issue of a potential low bias instilled in the sample assay results.

Figure 14.1 Photograph of Well-Mineralized Core



(photograph courtesy of Richard Spencer, January 2006 showing clots of pyrite ±chalcopyrite)

14.1 Data Verification by Placer

In addition to the internal check assaying, systematic QA/QC program, and external, independent check assaying program described in Section 13.1, Placer conducted a PQ/HQ drilling comparison and a closely spaced drilling program.

Grade versus core recovery was reviewed by Placer. The results indicate that the influence of core recovery is negligible on total grade and virtually non-existent on the ore grade. The differences, though negligible, show higher core recovery drill intervals being slightly lower grade than the grade of drill intervals with lower core recovery in saprolite. [MDA believes that this may indicate a bias in sampling due to selective recovery of mineralized material in the saprolite material. While Placer had eliminated many samples due to poor recovery or "contamination", a procedure that MDA continued, MDA reduced the resource classification for those blocks estimated from samples with low core recovery.]

Placer drilled four 12m-by-18m areas in a star pattern with 13 HQ diamond drill holes in each pattern. Drill hole spacing was 3m in an east-west direction and 3m in a north-south direction, with two holes drilled at 6m spacing on the north and south ends of the pattern. The long axis of the pattern was oriented approximately parallel to foliation, *i.e.*, 000° azimuth in the Conductora area and 020° azimuth in the Cuatro Muertos area. Depth of the holes was dictated by the oxide/sulfide saprolite contact, with a minimum of 5m being drilled into the sulfide saprolite. The average depth of the holes was 40m. Both splits of the drill core were sent to the Placer Research Center for analysis to test the variability in the sample collection, preparation, and analysis procedures.

The results of this drill program show that correlation coefficients typically fall within a range of 0.4 to 0.6 for gold samples 3m apart and quickly falls to less than 0.1 for samples up to 9m apart. Generally sample pairs show stronger correlation for drill-hole comparisons along the NNE strike direction than across or down the dip. Copper typically shows higher correlation coefficients than gold for holes the same distance apart. Copper also shows the same general trend correlation, with better correlation in the north-south direction and poor correlation in the east-west direction.

A comparison of gold fire assays with an AA finish was made for 2,489 drill core sample splits, with both halves of the core assayed. The mean grades of the two halves of the core were the same at 1.39 g Au/t with similar variability. The correlation coefficient was 0.95. The Placer-generated quantile-quantile ("QQ") plots showed similar distributions, while the relative difference plots did not show any conditional bias.

If done as reported above, the QA/QC program demonstrated that Placer's exploration work was high quality.

14.2 Placer Data Verification by Crystallex

Crystallex completed a 12-hole drill verification program and duplicate sampling/check assaying program for which MDA's involvement was to ensure some independence. The verification program collected:

- 1,086 split core samples from 11 holes and one re-drilled hole, all completed by Crystallex,
- 342 splits of Placer core (quarter cores) from Placer drilling (1 sample was lost),
- 262 Placer pulps (3 samples lost), and
- 200 splits of Placer coarse-reject samples (2 samples lost).

MDA supervised drill sampling, sample collection, and sample packaging for the first half of the program, with the goal of maintaining sample integrity and chain of custody. Sample preparation and assaying were done by independent laboratories. The program inserted standards, blanks, and coarse rejects at irregular intervals in the sample stream with an overall frequency of two standards, two coarse rejects, and one blank per 25 samples submitted for analysis.

A QA/QC program for the Crystallex core drilling program was outlined by Dr. Luca Riccio, former Vice President of Exploration for Crystallex, and Mr. Ristorcelli of MDA. Dr. Riccio worked with Mr. Maynard during the initiation of the project while Mr. Maynard carried out the project for the first three weeks. Dr. Riccio was responsible for the program and was on site after Mr. Maynard's departure. Mr. Maynard was on the Las Cristinas property from January 15, 2003 until February 7, 2003, living and working at the camp.

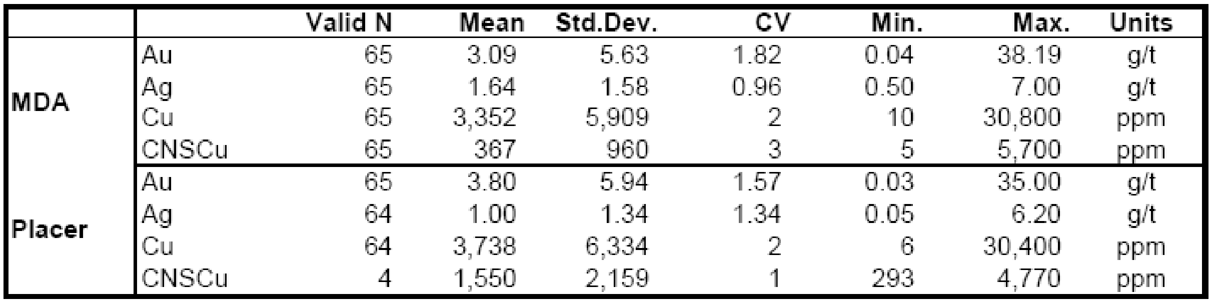
14.3 First Preliminary Independent Corroboration of Project

At the initiation of this project, MDA compared topographic data with drill-hole collar elevations and found they agreed. MDA also plotted drill-hole maps with traces of drill holes and found that the database compiled by MDA from CVG data corresponded well with electronic drawing files presumed to have been compiled by Placer. In addition, MDA requested that Crystallex contract an independent surveyor to check drill-hole locations. Crystallex had 25 drill holes surveyed and, aside from an equal shift with all surveys (~34m in the east and ~3m in the north), the surveys showed that these original survey data stand up to verification relative to each other. Correcting for this shift, all holes were within 1.7m of the surveyed coordinates and generally off by less than one meter. The coordinate shift is an issue that has recently been resolved by Crystallex in that independent surveyor Mr. David Rogerson (Surco CA) has resurveyed the surface in the planned pit area and in the process has corrected this shift.

In late 2002, MDA took 65 independent samples of core, pulps, and coarse rejects. After choosing and receiving the samples, MDA renumbered the samples to names known only to MDA. At most, though not all, times MDA had the samples in their direct control. But at no time when out of MDA's control (except during cutting the core) did anyone know which samples were which. Due to the preliminary nature of this program, which was only the initial part of the larger program, the check samples, though lower grade, corroborate the general tenor of historic data.

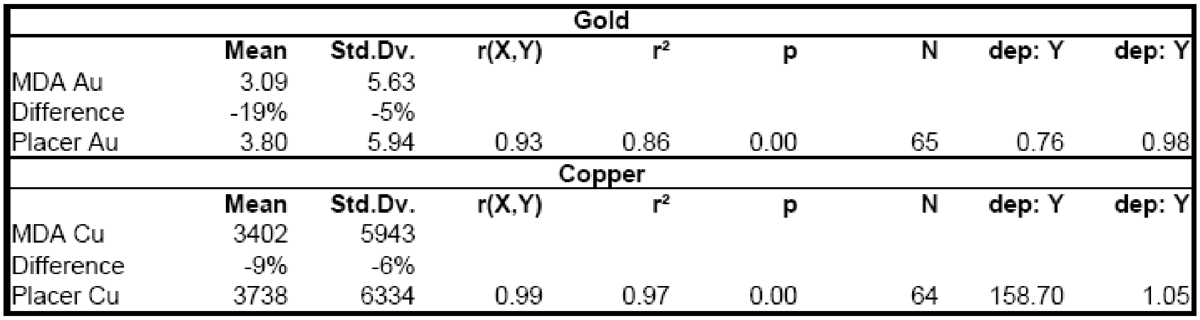
MDA's samples were taken from various resource areas and were from varied grade ranges. Samples were taken from split core, sawn core, coarse rejects and pulps. MDA compared the results of MDA's and Placer's samples for copper and gold only. Table 14.1 presents descriptive statistics of MDA's check-sampling program, and the table shows MDA's samples are lower grade for both copper and gold. Table 14.2 shows the correlations of gold and copper between MDA and Placer samples. On closer review (Figure 14.2 and Figure 14.3), it is clear that the differences occur mostly in the check samples of split core. Since this is such a small dataset, no global, definitive statements can be made concerning the Placer database based on these samples alone; however, it does suggest that initial sample preparation may be very important, as the comparisons are better further down the sample preparation process.

Table 14.1 Descriptive Statistics of MDA 2002 Check Samples



		Valid N	Mean	Std.Dev.	CV	Min.	Max.	Units
MDA	Au	65	3.09	5.63	1.82	0.04	38.19	g/t
	Ag	65	1.64	1.58	0.96	0.50	7.00	g/t
	Cu	65	3,352	5,909	2	10	30,800	ppm
	CNSCu	65	367	960	3	5	5,700	ppm
Placer	Au	65	3.80	5.94	1.57	0.03	35.00	g/t
	Ag	64	1.00	1.34	1.34	0.05	6.20	g/t
	Cu	64	3,738	6,334	2	6	30,400	ppm
	CNSCu	4	1,550	2,159	1	293	4,770	ppm

Table 14.2 Correlation of 2002 MDA Check Samples



	Gold							
	Mean	**Std.Dv.**	**r(X,Y)**	**r^2**	**p**	**N**	**dep: Y**	**dep: Y**
MDA Au	3.09	5.63						
Difference	-19%	-5%						
Placer Au	3.80	5.94	0.93	0.86	0.00	65	0.76	0.98
	Copper							
	Mean	**Std.Dv.**	**r(X,Y)**	**r^2**	**p**	**N**	**dep: Y**	**dep: Y**
MDA Cu	3402	5943						
Difference	-9%	-6%						
Placer Cu	3738	6334	0.99	0.97	0.00	64	158.70	1.05

Figure 14.2 Gold Check Assay Correlations

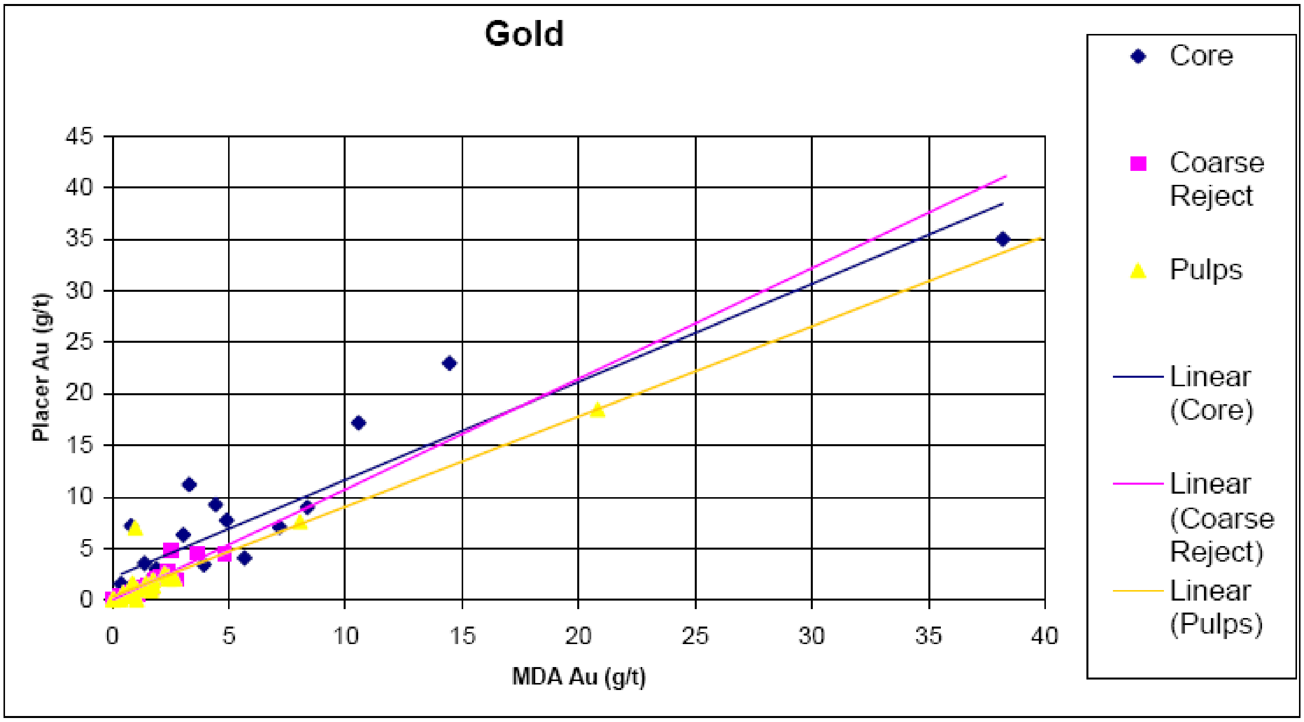
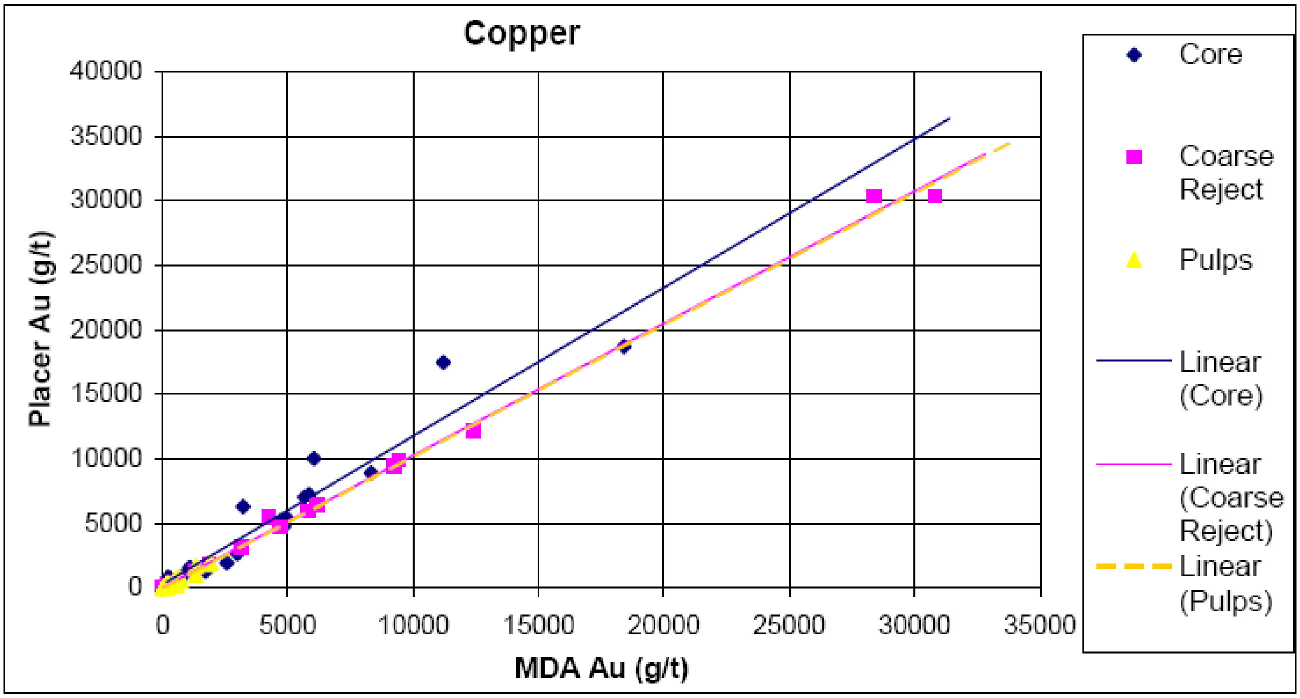


Figure 14.3 Copper Check Assay Correlations



14.4 MDA/Crystallex Joint Check Program on Previous Samples

Crystallex took samples of existing Placer quarter-core splits, coarse rejects, and pulps for gold grade re-assay.

Quarter-core splits of Placer core samples (341) were sawn, prepared, and analyzed to corroborate assay data. Sample selection was determined by location and grade. Mr. Maynard chose the intervals to be re-split and did the sawing personally with a tile cutter rock saw. Each quarter core sample had the original hole number and sample number recorded and was bagged in a white cloth bag identified with only a four-digit number on the outside and a slip of paper with the four-digit number in the bag. These samples were submitted to Triad for sample preparation and sent to Chemex for analysis.

The 341 quarter-core check samples of Placer core showed poor reproducibility, poor correlation, but a modest comparison of mean grades. The Crystallex check samples are 8% lower grade (Table 14.3 and Figure 14.4). Note that most of the difference is caused by four of the highest-grade samples. Eliminating these four samples increases the slope of the line from 0.42 to 0.83 (Figure 14.5), though does not materially affect the r^2, which remains a low 0.4. By eliminating the four highest mean-grade samples, Crystallex mean grades become higher grade than Placer by 6%.

Table 14.3 Descriptive Statistics on Quarter-Core

	All samples						
	Placer	**Diff.**	**KRY**	**Avg.**	**Diff.**	**Var.**	**Abs. Var.**
N	341		341	341	341	341	341
Mean	1.96	8%	1.82	1.89	25%	10%	63%
Std	3.72	56%	2.39	2.79	128%	149%	135%
Mn	0.02	122%	0.01	0.02	-89%	-833%	0%
Max	40.35	93%	20.90	30.63	1501%	1501%	1501%
	Greater than 0.4 g Au/t Average						
	Placer	**Diff.**	**KRY**	**Avg.**	**Diff.**	**Var.**	**Abs. Var.**
N	305		305	305	305	305	305
Mean	2.16	8%	2.01	2.09	24%	9%	64%
Std	3.89	58%	2.46	2.89	133%	155%	141%
Mn	0.27	366%	0.06	0.40	-89%	-833%	0%
Max	40.35	93%	20.90	30.63	1501%	1501%	1501%

Figure 14.4 Scatterplot of All Crystallex Checks on Quarter Core

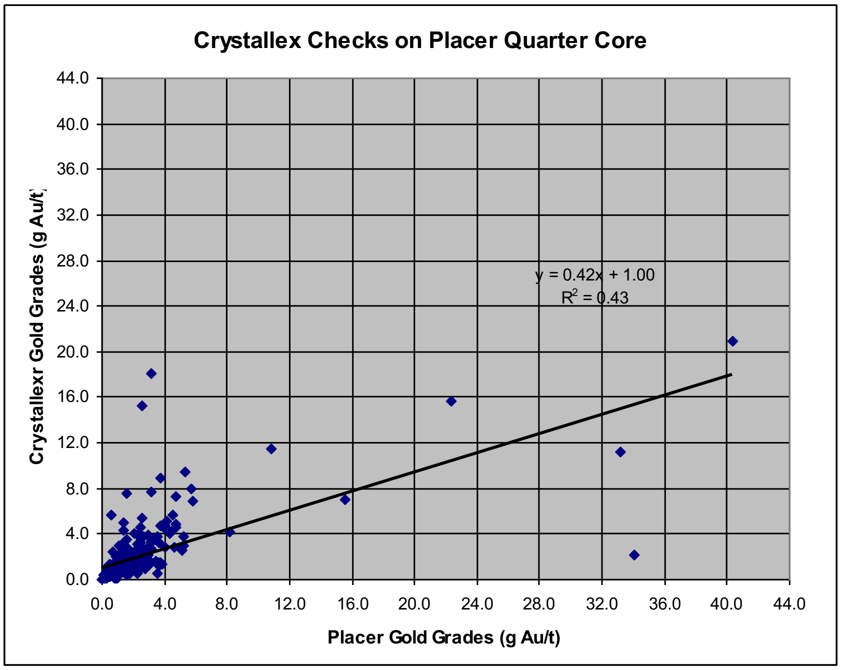
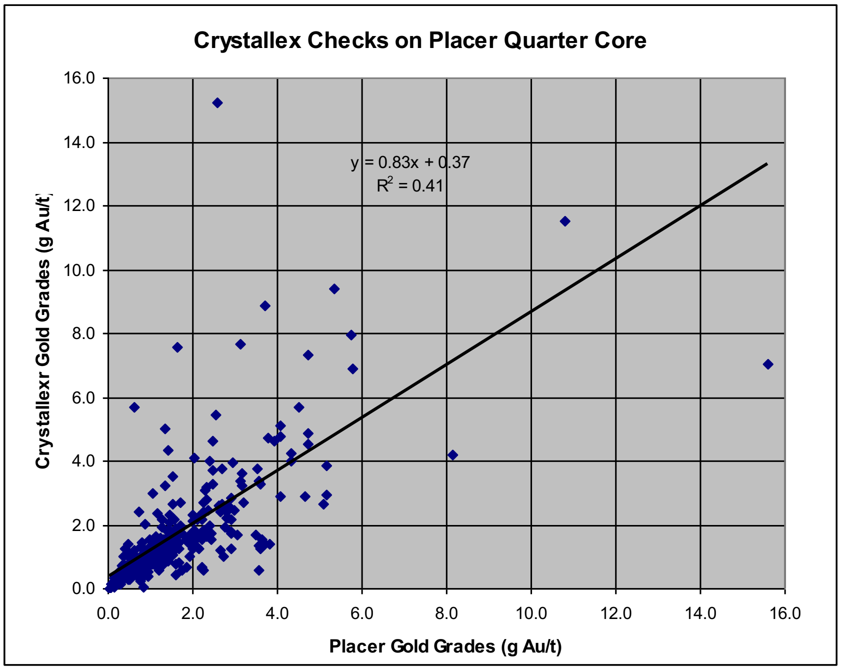


Figure 14.5 Scatterplot of Crystallex Checks on Quarter Core

(excluding four highest-grade samples)

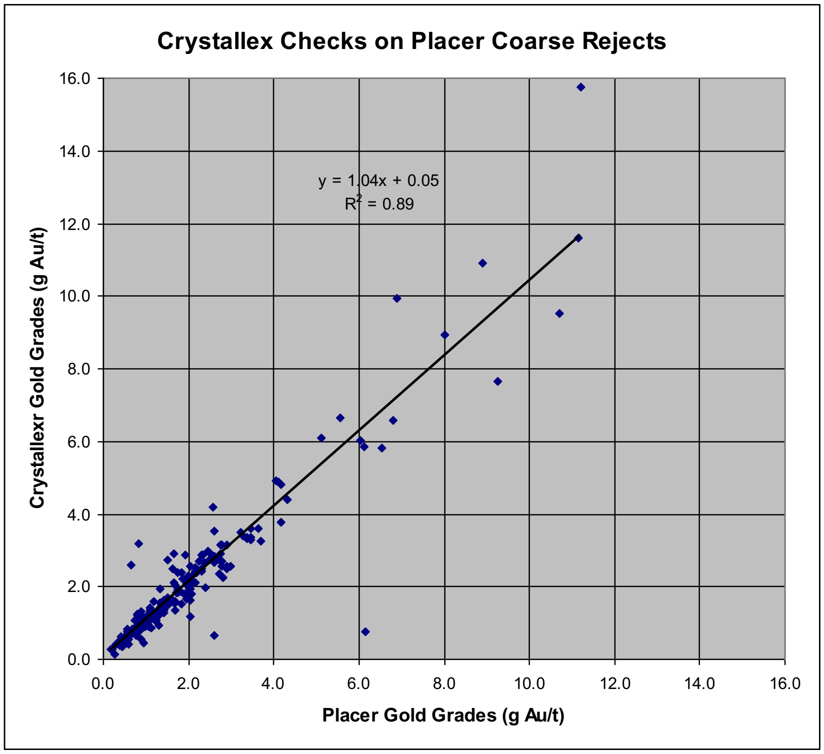


Coarse reject checks on the Placer drilling (198) were submitted for analysis. Coarse rejects, selected by location and grade, were placed in a four-digit numbered cloth bag while the drill hole and depth were blind to the laboratory. These were assayed to corroborate original assays and to check for reproducibility. Results from 198 check assays on coarse rejects showed good correlation, although mean grades were 6% higher for the Crystallex samples (Table 14.4 and Figure 14.6).

Table 14.4 Descriptive Statistics on Coarse Rejects

	All samples						
	Placer	**Diff.**	**KRY**	**Avg.**	**Diff.**	**Var.**	**Abs. Var.**
Count	198		198	198	198	198	198
Mean	2.01	-6%	2.14	2.08	1%	-4%	25%
Std. Dev.	1.91	-9%	2.11	1.98	58%	66%	61%
Min.	0.19	48%	0.13	0.20	-75%	-302%	0%
Max.	11.20	-29%	15.75	13.48	701%	701%	701%
	Greater than 0.4 g Au/t Average						
	Placer	**Diff.**	**KRY**	**Avg.**	**Diff.**	**Var.**	**Abs. Var.**
Count	192		192	192	192	192	192
Mean	2.07	-6%	2.20	2.13	0%	-5%	24%
Std. Dev.	1.91	-9%	2.11	1.98	58%	66%	62%
Min.	0.37	-12%	0.42	0.43	-75%	-302%	0%
Max.	11.20	-29%	15.75	13.48	701%	701%	701%

Figure 14.6 Scatterplot of Crystallex Checks on Coarse Rejects



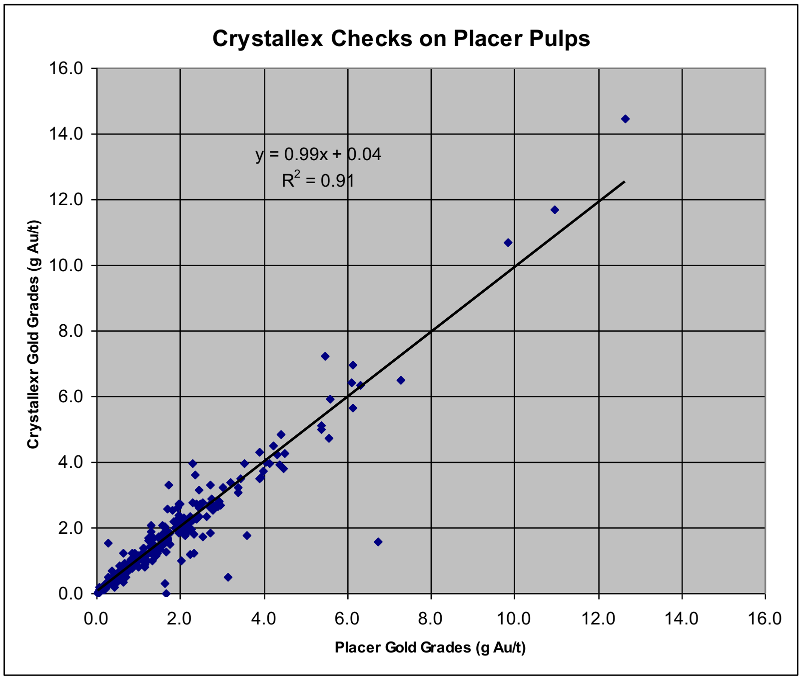
<u>*Pulps*</u> from Placer drilling (259) were submitted for analysis. Selection of pulps was based on location and grade. Pulps are stored in paper envelopes in plastic bags in woven rice bags in open sided sheds at site. MDA obliterated the sample numbers on the paper envelopes with black markers, and the envelope was inserted in a new paper envelope with a four-digit sample number; the original hole and sample number were kept in MDA records only. The newly numbered pulps were sent directly by courier to Chemex for analysis.

Results from the 259 sample pulps showed good correlation and similar mean grades (Table 14.5 and Figure 14.7). This is true for all samples as well as those samples greater than 0.4 g Au/t, a value that is approximately the economic cutoff. A cluster of five samples was noticeably higher grade in the Placer set than in the Crystallex set. Variance of check sample grades is considered high for pulps.

Table 14.5 Descriptive Statistics on Pulps

	All samples						
	Placer	**Diff.**	**KRY**	**Avg.**	**Diff.**	**Var.**	**Abs. Var.**
Count	259		259	259	258	258	258
Mean	1.71	-1%	1.73	1.72	5%	0%	27%
Std. Dev.	1.74	-4%	1.81	1.75	53%	64%	57%
Min.	0.02	NA	0.00	0.01	-82%	-443%	0%
Max.	12.65	-12%	14.45	13.55	513%	513%	513%
	Greater than 0.4 g Au/t Average						
	Placer	**Diff.**	**KRY**	**Avg.**	**Diff.**	**Var.**	**Abs. Var.**
Count	216		216	216	215	215	215
Mean	2.02	-1%	2.04	2.03	4%	0%	25%
Std. Dev.	1.75	-4%	1.82	1.76	55%	64%	59%
Min.	0.28	NA	0.00	0.40	-82%	-443%	0%
Max.	12.65	-12%	14.45	13.55	513%	513%	513%

Figure 14.7 Scatterplot of Crystallex Checks on Pulps



14.5 Twin Hole Analysis

MDA tabulated the Placer drill holes with corresponding Crystallex twin holes so that the same intervals were represented. Analyses were made on a hole-by-hole basis, which yielded highly variable results, and on all data. Table 14.6 shows that overall Crystallex drilling yields average gold grades for those true twins are 15% lower than the corresponding Placer intervals (not all the drilling were true twins).

Table 14.6 Twin Hole Comparison

Crystallex	Diff.	Placer	Comments
1,669	-1%	1,683	Total meters
1.28	-15%	1.49	Mean Grade (g Au/t)
0.00	-70%	0.01	Minimum grade (g Au/t)
50.50	-38%	80.83	Maximum grade (g Au/t)

The comparison of location of gold grades was found to be reasonable in that the higher-grade intervals were found to be in the same locations for the most part. Not unexpectedly, the twin-hole sample assays were more similar in Conductora than in Mesones-Sofia. One apparent difference was that the Crystallex drilling did not duplicate the higher-grade single assays, *i.e.*, >~ 7 g Au/t. For example, there were 54 (3%) samples over 7 g Au/t in the Placer data averaging 14.79 g Au/t, but only 14 (2%) above 7 g Au/t in the Crystallex data, though with a similar mean grade of 14.88 g Au/t. At least some of this can be attributed to sample lengths, as Placer sampled 0.82m intervals on average compared to Crystallex's average sample length of 1.94m. Using composited sample lengths, Placer had 2.7% of the samples greater than 7 g Au/t while Crystallex had 1.9% greater than 7 g Au/t. Placer's mean composite grade of composites over 7 g Au/t was 12.5 g Au/t, while Crystallex's was 13.39 g Au/t.

A comprehensive evaluation was done by Ristorcelli and Hardy (2004b). In that study, MDA suggested:

> *"In 2003 after the 12 twin hole program was completed, a difference in mean grades was noted when a comparison was made between Placer Dome's (Placer) data and Crystallex's initial verification drilling. During this most recent estimation process, a similar difference was noted with the latest drilling being approximately 6% lower in grade than the nearest Placer drill data. MDA has not attempted to compensate for this apparent sample bias in the estimation nor is any adjustment warranted.*
>
> *Taken in context with geologic information, the results of Crystallex sample verification programs present information on the behavior of gold distribution of the Las Cristinas deposit. Briefly, the Crystallex/MDA check assays verified Placer's pulps and coarse rejects. The checking program did show differences in quarter core (compared to Placer's one half core) checks but it is important to note that Crystallex only had quarter core to check and mean differences are dominated by outlier sample grades. Statistical analysis by Dr. Peter Knudsen, Dean of the School of Mines & Engineering, University of Montana, concluded there was no significant difference between the means for the pulp and quarter core and the T test for the coarse rejects was inconclusive. The Crystallex 2003 twin hole assays yielded a global mean difference of 15%, with the Crystallex drill assays coming in lower than the Placer drill core samples. A principal issue regarding this difference is the fact that Crystallex drilled smaller*

> *diameter core than Placer. Each step up in core size represents a difference of 80% in volume. Placer tested for a potential bias (five holes and 277 paired samples) and found that there was a 4% difference in mean grades with HQ being lower than PQ, though they deemed the difference not statistically significant. The visual heterogeneity of the deposit along with the just-mentioned check results suggest that the difference in grades could be caused by this sample volume difference.*
>
> *These mean grade differences, though not statistically significant, could indicate a sampling and subsampling issue related to heterogeneity of Las Cristinas mineralization raising the possibility of a difference in mean grade of the deposit, possibly even higher grade than is presently reported. A heterogeneity study has been initiated to better understand the phenomenon and to obtain better parameters for subsampling protocol and grade control during mining operations."*

Interestingly, the completed heterogeneity study (Section 14.11) suggested that sample size does affect the mean global grade of sample assays returned, and that this represents an incalculable upside to Las Cristinas.

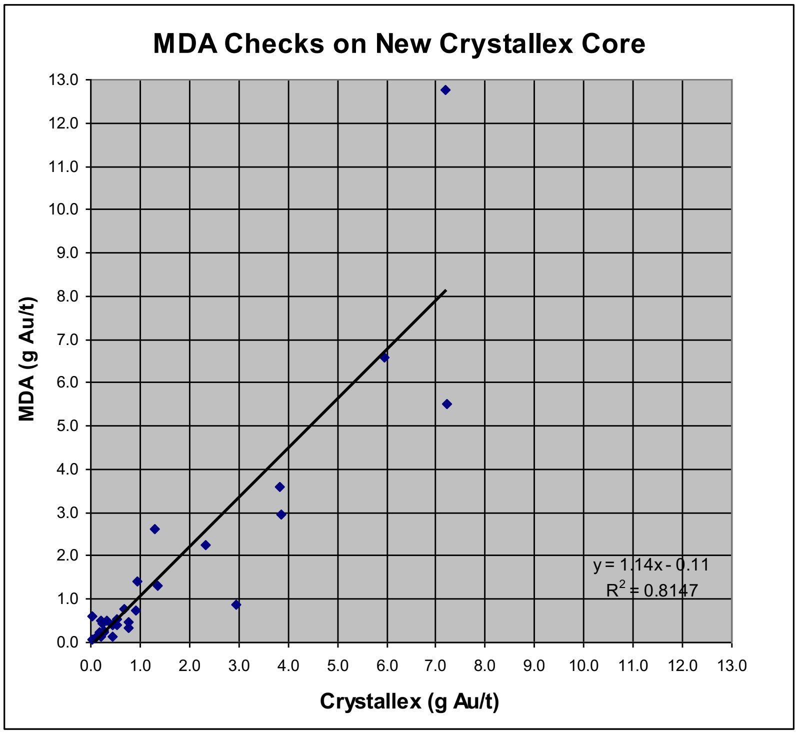
14.6 MDA Checks on 2003 Crystallex Sampling

MDA took independent core samples from Crystallex's 2003 verification drill program. The samples were always in the custody of MDA representative, Mr. Maynard. These samples were taken by Mr. Maynard and relabeled so as to avoid any possibility of tampering with the samples. As shown in Table 14.7 and Figure 14.8, Crystallex's data are corroborated. It should also be noted that the difference in grades between Crystallex core samples and MDA's core check samples is high.

Table 14.7 MDA Checks on Crystallex 2003 Drilling

		Difference		
	MDA	**Diff.**	**Crystallex**	
Count		29		
Mean	1.62	7%	1.52	g Au/t
Std. Dev.	2.68	27%	2.11	g Au/t
CV	1.66	19%	1.39	
Min.	0.06	77%	0.03	g Au/t
Max.	12.75	77%	7.22	g Au/t

Figure 14.8 MDA Checks on Crystallex 2003 Drilling



14.7 2004 Drill Program

Much of the 2004 drill program verification was based on the 2003 drilling, which post-dated the modeling done for the first resource estimate. The analysis of this work was first done by Ristorcelli and Hardy (2004c). MDA also took independent samples from the 2004 drill program (Table 14.8), which verified the general tenor of mineralization. No formal reports were completed for the QA/QC work of Crystallex's 2004 drill program. Some preliminary assessment was done that showed that the analytical work and standards used were not particularly clean; however, nothing was noted that would have negated the use of these 18 holes considering they represented less that 2% of all drilling at Las Cristinas.

Table 14.8 2004 MDA Independent Samples

MDA Sample No.	MDA (g Au/t)	Crystallex (g Au/t)	Crystallex Sample No.	Hole	From (m)	To (m)	Length (m)
LC04-01	2.440	2.400	23742	K-4CO1130	261.5	262.5	1.00
LC04-03	1.260	2.400	22419	K-4CO1135	238.5	239.5	1.00
LC04-04	0.190	0.477	22418	K-4CO1135	237.5	238.5	1.00
LC04-05	3.370	3.910	22459	K-4CO1135	276.5	277.5	1.00
LC04-06	6.460	27.100	22484	K-4CO1135	299.5	300.5	1.00
LC04-07	2.380	2.540	22456	K-4CO1135	273.5	274.5	1.00
LC04-08	1.270	5.000	22527	K-4CO1135	340.5	341.5	1.00
LC04-09	2.600	2.500	22452	K-4CO1135	269.5	270.5	1.00
LC04-10	4.230	5.100	23775	K-4CO1130	292.5	293.5	1.00
LC04-11	2.470	2.370	23714	K-4CO1130	235.5	236.5	1.00
LC04-12	0.750	0.616	23712	K-4CO1130	233.5	234.5	1.00
LC04-13	1.660	6.000	22481	K-4CO1135	296.5	297.5	1.00
LC04-14	2.005	2.230	23709	K-4CO1130	230.5	231.5	1.00
LC04-15	6.365	7.500	22426	K-4CO1135	245.5	246.5	1.00
LC04-16	2.580	2.910	22468	K-4CO1135	284.5	285.5	1.00
LC04-17	18.750	4.900	22431	K-4CO1135	249.5	250.5	1.00
LC04-18	3.050	1.536	54593	K-4CO1144	304.5	305.5	1.00
LC04-19	0.770	0.488	54594	K-4CO1144	305.5	306.5	1.00
LC04-20	1.220	0.673	54559	K-4CO1144	273.5	274.5	1.00
LC04-21	5.110	5.700	54525	K-4CO1144	235.5	236.5	1.00
LC04-22	2.010	0.310	54526	K-4CO1144	236.5	237.5	1.00
LC04-23	2.750	2.400	54433	K-4CO1144	145.5	147.5	2.00
Number of Samples		22					
Mean	3.350	4.048	-17% Difference of the means (MDA/Crystallex)				
Std. Deviation	3.824	5.528					
Minimum	0.190	0.310					
Maximum	18.750	27.100					

14.8 2005 Drill Program

Ristorcelli (July 2005) reported that "*Overall there is nothing in this data set to preclude using the assay data in the resource estimate. There is a high failure rate on the standards, although most of these could be sample-handling issues. The inserted blanks show that two submittals are suspect. This QA/QC program has not had external check assays by second laboratories and there are no duplicate lab checks. There is no split core or checking on coarse rejects. Overall, the 2005 drill program QA/QC is limited and leaves some doubts. However, given that all but three failures could be explained by mishandling standards, the 2005 data are accepted but with some hesitation. There is high confidence that the drill data did hit the intended targets and there is nothing suggesting that the data are in fact in error. Rather, the hesitation is caused by the lack of a comprehensive QA/QC program. The reader must understand that many ounces in this latest estimate are based on this relatively small drill database with this limited QA/QC.*"

As a result of the preceding observations, Spencer (2006) evaluated the QA/QC of the 2005 drill program. Spencer took pains to evaluate the data in light of some sub-standard standard assay material and verified all samples that failed in light of the checks. His work resulted in Crystallex obtaining new standards for future programs, re-assaying failed batches, and explaining discrepancies. Spencer's conclusions (2006) were that "*The high percentage of repeatable values in the reassay programme demonstrates the integrity of the assay data from the 2005 drill sampling programme at Las Cristinas.*" and "*Poor assay repeatability in high-grade spikes, which are quite common at Las Cristinas, has the potential to significantly affect the calculated average grade of a mineralized interval, although less so* *on a global basis when considering the effect on the grade of the entire deposit."* MDA concurs and found the 2005 drill-sample assay data suitable for use in resource estimation to classification of up to and including Measured.

14.9 2006-2007 Drill Program

The following section concerning QA/QC for the 2006-2007 drill program was taken from Nicholson (2007).

14.9.1 Introduction

Nicholson Analytical Consulting ("NAC") was contracted to aid in the design of and to oversee the QA/QC program for Crystallex's 2006-2007 drill program. NAC's involvement in the program included:

- inspection and recommendations of lab facilities to be used for the program;
- recommendations on design and implementation of the program prior to the start of drilling;
- active monitoring of Crystallex's QA/QC data for the primary element of Au; and
- analysis of any internal and external duplicate assaying.

NAC was on-site for approximately 40% of the time at various points throughout the drill program. NAC's primary focus was the quality control of the analytical data. However, at Crystallex's invitation, NAC also examined the procedures being used in all parts of the drill program. Although only involved in the design of the QA/QC program, NAC noted no irregularities in any areas of the drill program. NAC was impressed by the thoroughness and professionalism displayed by all of the Crystallex personnel.

14.9.2 Lab Inspection and Recommendations

Prior to the start of the drilling program, NAC and Crystallex carried out an investigation of the possible labs that could be used during the program. As a result, the samples were shipped from the property to the SGS lab at El Dorado for sample preparation only. NAC was on-site for approximately 40% of the drill program and accompanied the samples to the lab during this time. NAC made routine visits to the lab while delivering these samples and gave guidance to the lab staff concerning procedures they were using to prepare the core samples.

The prepared samples and reject material were picked up and transported back to the Las Cristinas compound. The prepared pulp samples were secured in a locked room until they were shipped to SGS-Lima for analysis. NAC conducted sieve tests on several of the pulp and reject samples from the SGS El Dorado lab. All of the samples passed the sample preparation criteria set out by the lab.

Overall, NAC is satisfied with the work done at SGS-El Dorado and is confident that the samples were adequately prepared.

All samples from this program were shipped to SGS-Lima Peru for analysis on recommendation from NAC. NAC has dealt with SGS-Lima on other projects and felt that they would be the best lab within

South America to assay the samples. The lab is very modern and is ISO 9001:2000 and ISO 17025 certified.

For convenience, the check lab selected was ALS-Chemex, also in Lima. NAC has also dealt with this lab on previous projects and found the quality of the work to be excellent. This lab is also a very large modern facility and is ISO 9001:2000 certified.

14.9.3 Program Design and Implementation

The QAQC program was designed in consultation with Richard Spencer, VP Exploration, of Crystallex.

Active QA/QC Monitoring

Standard Insertion

The active monitoring portion of the program utilized certified reference materials ("CRM") inserted into the sample stream to verify the accuracy of the data being received from the primary assay lab as the data were returned to Crystallex. The program employed the use of five CRM's inserted into the sample stream on a rotating basis. The CRM's used in this program were obtained commercially from CDN Resources labs in Burnaby, BC, Canada. Each of the standards has undergone extensive homogenization testing and has been round-robin assayed by several labs both in Canada and abroad. These standards come with a certification which includes a recommended value and confidence interval (Table 14.9) as well as outlining the procedures used for determining these values.

Table 14.9 CRM Gold Grades and Confidence Intervals

Standard Name	Gold (recommended value and 95% confidence interval)
GS-P5B	0.44 ± 0.04 g/t
GS-1C	0.99 ± 0.08 g/t
GS-1P5A	1.37 ± 0.12 g/t
GS-1P5	1.58 ± 0.16 g/t
GS-15	15.31 ± 0.58 g/t

The analysis protocol called for all samples to be assayed using 30g fire assay fusion followed by determination by atomic absorption spectrometry. Any sample with a gold value above 5 g Au/t was to be re-assayed using a 30g fire assay fusion followed by a gravimetric finish. It was important that at least one of the standards used in the program had a value above 5 g Au/t in order to assess the quality of those analyses that were done by this alternate higher-grade method.

The standards were ordered in bulk (several kilograms each) and shipped to the Las Cristinas site prior to the start of drilling. NAC re-labeled these standards with generic names (S1 to S5) and sent them to SGS labs in El Dorado to be re-homogenized, split and bagged in 100g splits. The standards were relabeled to prevent the lab from determining the origin and values of the standards. The standards' bags were then returned to Las Cristinas and held in a locked room until the start of the drilling program. The decision of which standard to be inserted at any given location in the sample stream was made by NAC or one of the Crystallex geologists after examining the core samples surrounding the standard insertion position. The intent was to have the gold concentration in the standard be as close as practical to gold concentration in the surrounding samples. Standards with higher grades were inserted into areas that had visible mineralization, and those with lower grades were inserted where little or no mineralization was seen.

The program called for one standard to be inserted approximately every 25th sample. The position of the first standard in each batch was randomized within the first 25 samples. This way the standard did not appear in the same ordinal sample position within each batch of samples. Core samples were shipped to the lab with an empty core bag containing a core tag placed in the position that the standard was to occupy. The instructions accompanying the sample batch told the lab that this was a standard position and to leave an empty labeled pulp bag in that position with a number matching the accompanying core tag.

After preparation of the core samples was complete and the pulp samples were returned to the Las Cristinas site, either NAC or one of the Crystallex geologists added one of the five standards to the empty pre-labeled pulp bag. Inserting the standards on-site prevented the lab from knowing which of the standards had been inserted in any given position, and it allowed NAC/Crystallex to check that the sample numbering and positions were correct prior to submitting the samples for analysis.

Although one can never completely disguise the presence of standards in a sample stream, this is as close as one can possibly get. The standards appeared in bags identical to those of the samples. The bags and labels did not have any identifying characteristics to distinguish them from regular samples in the stream.

Blank Insertion

Barren rock material was inserted into the sample stream at the rate of one every 30th sample position. This rock was from a barren diorite quarry located off-site and was cut with a diamond saw into 5-10cm fragments that were not conspicuously dissimilar to core fragments. Blank material was bagged as a sample and not identified to the laboratory.

Data Treatment

The active monitoring portion of the QA/QC program was carried out for gold only. Analysis data were obtained directly from SGS-Lima via e-mail. Shewhart and Cumulative Sum ("CuSum") control charts were constructed as the data came in and were used to determine quality.

Crystallex and SGS-Lima were notified by NAC when any group of data failed QA/QC tests. A standard determination that falls outside the control limits indicated a control failure. The control limits used were ± 2 S.D. for warning limits and ± 3 S.D. for control limits. When a control failure occurred, NAC directed SGS-Lima to have the affected range of samples re-analyzed. The protocol for selecting affected samples is that for any sequence that a QA/QC standard fails:

1) Re-analysis starts earlier in the sequence at the position of the last valid QA/QC standard and finishes later in the sequence at the position of the next valid QA/QC standard. This range includes all samples, standards, blanks and duplicates that fall between these valid QA/QC standards and also includes the both-valid QA/QC standards on each end of the sequence.

2) In the event that there are no QA/QC standards in the sequence prior to the failed QA/QC standard, the range includes all samples prior to the failed QA/QC to the beginning of the batch.
3) In the event that there are no QA/QC standards in the sequence after the failed QA/QC standard, the range includes all samples after the failed QA/QC to the end of the batch.

NAC also produced range charts along with the Shewhart control charts. The range charts are a good indication of the precision of data. These were not used for active monitoring but for informational purposes only. Since the Crystallex data are only a subset of the data produced by SGS and the data are not contiguous, large shifts in range do not necessarily indicate a failure.

No run rules (for excessive runs above and below the centerline and 2 S.D.) were applied to the Shewhart chart. Bias was measured using CuSum charts as they give a much faster and clearer picture than can be obtained from using Shewhart charts.

External Data Verification

Duplicates

Several different types of sample duplicates were generated during the drilling program. These duplicates were assayed by either the primary lab, SGS-Lima, or by the external check lab, ALS-Chemex-Lima. As well as sample duplicate analyses, each lab produces analysis replicates on a subset of the pulp samples in a batch. These are supplied as a part of the dataset and will be called "Internal Duplicates" for the purposes of this report. Two different types of duplicate samples were generated for analysis by the primary assay lab: 1) duplicate samples split from -10 mesh material and 2) duplicate samples obtained by prepping ¼ core samples.

The duplicates obtained from the -10 mesh splits were generated approximately every 50th sample. The duplicates from the ¼ core were also generated approximately every 50th sample. In both cases, the duplicate appeared immediately following the original sample and was numbered as a normal sample in order to be blind to the primary assay lab.

In the case of the duplicate split from the -10 mesh material, an empty bag with a core tag in it was placed in the position that the duplicate was to occupy. The instructions accompanying the samples told the preparation lab that this was a duplicate position and that a -10 mesh duplicate split of the previous sample was to occupy the empty bag.

The ¼ core duplicates were not identified to the laboratory, as the core was already in its assigned bag. In this case, sample prep proceeded as normal.

Three different types of duplicates were generated for analysis by the check assay lab: 1) analysis of the original pulp analyzed by the primary lab, 2) duplicates created by splitting the -10 mesh material, and 3) duplicates created by splitting the sample pulp.

When duplicate samples were required to be prepared for the external check lab, both types of duplicates were created from the same sample. The preparation lab generated the -10 mesh duplicates according to Crystallex's instructions during the original preparation of the samples. The pulp duplicate was created by the check lab by splitting the -10 mesh duplicate it received from the primary assay lab. All duplicates sent to the external check assay lab were sent directly from the primary assay lab.

Data Treatment

Regression plots, t-statistics and basic descriptive statistics were determined for each duplicate set. Six plots were constructed:

1) Pulp assay (primary lab) vs. pulp assay (check lab) - these analyses were performed on the same sample pulp;
2) Pulp assay (primary lab) vs. -10 mesh duplicate assay (primary lab);
3) Pulp assay (primary lab) vs. ¼ core duplicate assay (primary lab);
4) Pulp assay (primary lab) vs. internal duplicate assay (primary lab); these analyses were performed on the same sample pulp;
5) Pulp assay (primary lab) vs. -10 mesh duplicate assay (check lab); and
6) Pulp assay (check lab) vs. -10 mesh duplicate assay (check lab).

The t-statistic (comparison of means) was used as the primary indicator of fitness of the data. Single regression plots were also constructed. All t-statistics in this report are calculated at the 95% confidence level.

Data Handling

Data were obtained directly from each lab without Crystallex's involvement. NAC performed the necessary quality checks on the data and forwarded QA/QC validated data to Crystallex in Excel format on an ongoing basis as the data became available.

14.9.4 Active QA/QC Monitoring for Gold (Au)

Standard Monitoring

The QA/QC program resulted in the insertion of 543 standards in a total of 12,173 drill core samples. This gives an overall insertion rate of 4.46% or one standard for every 22.4 samples. Of the 543 standards that were submitted, there were four standard failures (Table 14.10). All of the failed standards and the associated samples were re-assayed as per the QA/QC protocol, and all of the re-assayed sequences passed the QA/QC criteria on the second pass (Table 14.11).

Table 14.10 Standard Data Summary for 2006/07 Crystallex Las Cristinas QA/QC Program

Standard Name	No. Of Determinations	No. of Failures	Failure Rate	Analysis Mean (g/t)	Standard Deviation	Recommended Value at 95% C.I.
GS-P5B	58	1	1.72%	0.43	0.002078	0.44 ± 0.04 g/t
GS-1C	190	0	0.00%	0.98	0.002643	0.99 ± 0.08 g/t
GS-1P5A	162	0	0.00%	1.36	0.004319	1.37 ± 0.12 g/t
GS-1P5	27	1	3.73%	1.55	0.005154	1.58 ± 0.16 g/t
GS-15	106	2	1.89%	15.34	0.1693	15.31 ± 0.58 g/t
Overall	**543**	**4**	**0.74%**			

Table 14.11 Standard Failures/Corrections for 2006/07 Crystallex Las Cristinas QA/QC Program

Sample ID	Hole ID	Standard ID	Original Analysis (g/t)	Recommended Value at 99% C.I	Corrected Analysis (g/t)
305768	K6MO1196	GS-15	9.830	15.31 ± 0.87 g/t	15.235
307100	K6MO1165	GS-15	10.850	15.31 ± 0.87 g/t	15.291
302382	K6MO1174	GS-1P5	0.526	1.58 ± 0.24 g/t	1.360
307175	K6MO1165	GS-P5B	0.358	0.44 ± 0.06 g/t	0.410

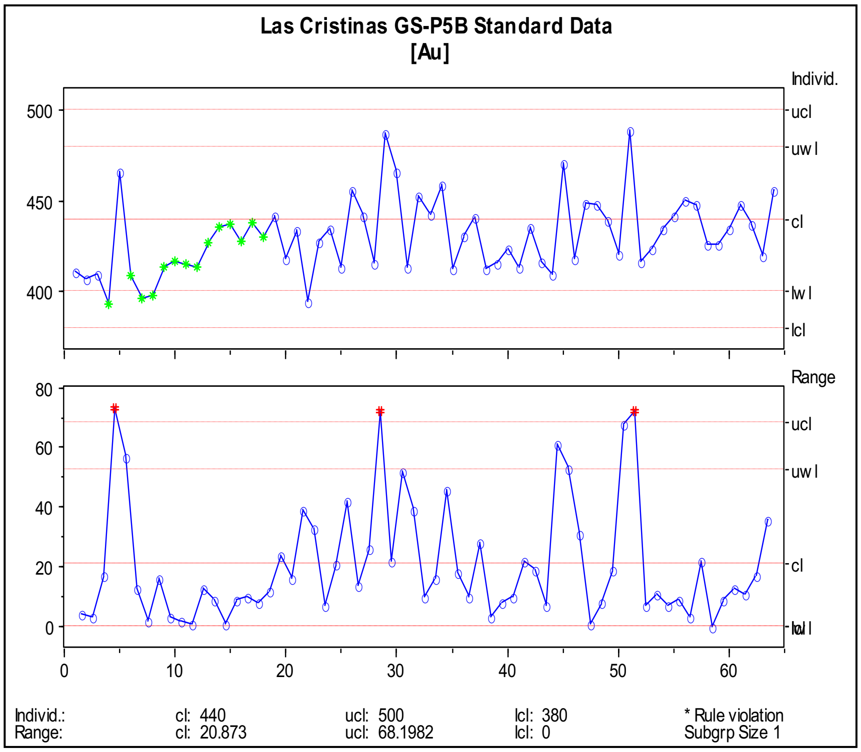
Small biases were detected on all standards used in the program. This is not uncommon as all of the major assay labs use a batch fluxing and fusion procedure for any given project. The optimal fusion procedure is determined for the matrix of the samples within the program. The standards may have a slightly different matrix than those of the samples, which cause small biases to be seen in the final result. It is usually a low negative bias that is seen as the fusion/fluxing process is not optimized for the standard matrix. In this program, four of the five standards employed showed a small negative bias (Table 14.12). All of the biases are well with the 95% confidence interval for the each standard.

Table 14.12 Biases for 2006/07 Crystallex Las Cristinas QA/QC Program Standards

Standard Name	Analysis Bias (g/t)	Recommended Value
GS-P5B	-0.01	0.44 ± 0.04 g/t
GS-1C	-0.01	0.99 ± 0.08 g/t
GS-1P5A	-0.01	1.37 ± 0.12 g/t
GS-1P5	-0.03	1.58 ± 0.16 g/t
GS-15	+0.03	15.31 ± 0.58 g/t

In addition to gold, each sample was analyzed by ICP-AES/*aqua regia* digestion for 38 other elements. There was no active monitoring in this program for any of these elements. The standards used in this QA/QC program have no recommended or certified values for any element other than gold. Since there have been no quality control measures implemented on any of the ICP elements, they should not be used in any ore reserve calculations. Final Shewhart and CuSum charts for gold are given in Figure 14.9 to Figure 14.13.

Figure 14.9 Control and Range Charts for Standard GS-P5B



Cumulative Sum Chart for Standard GS-P5B

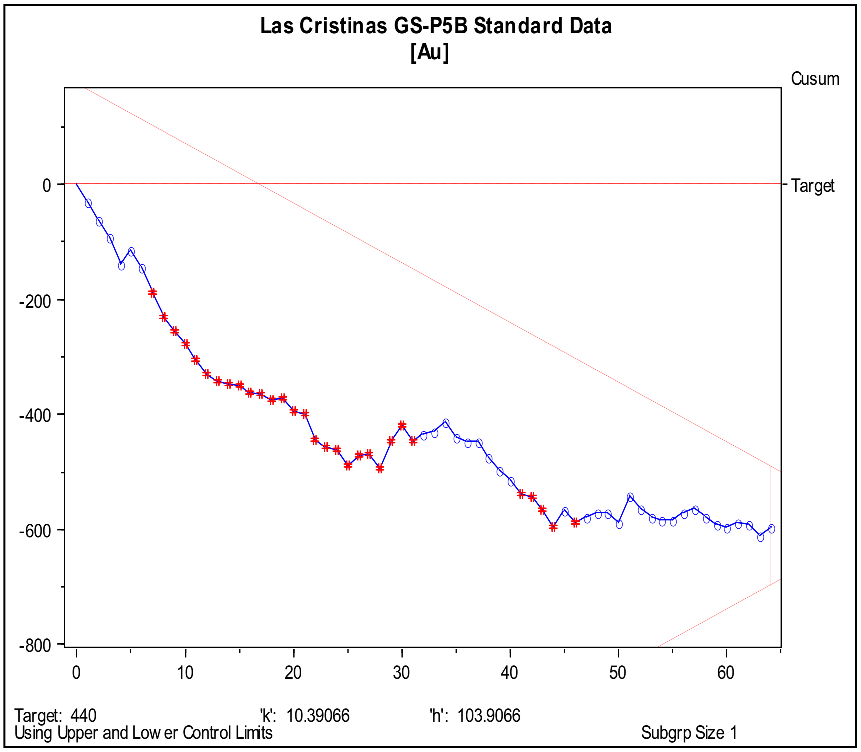
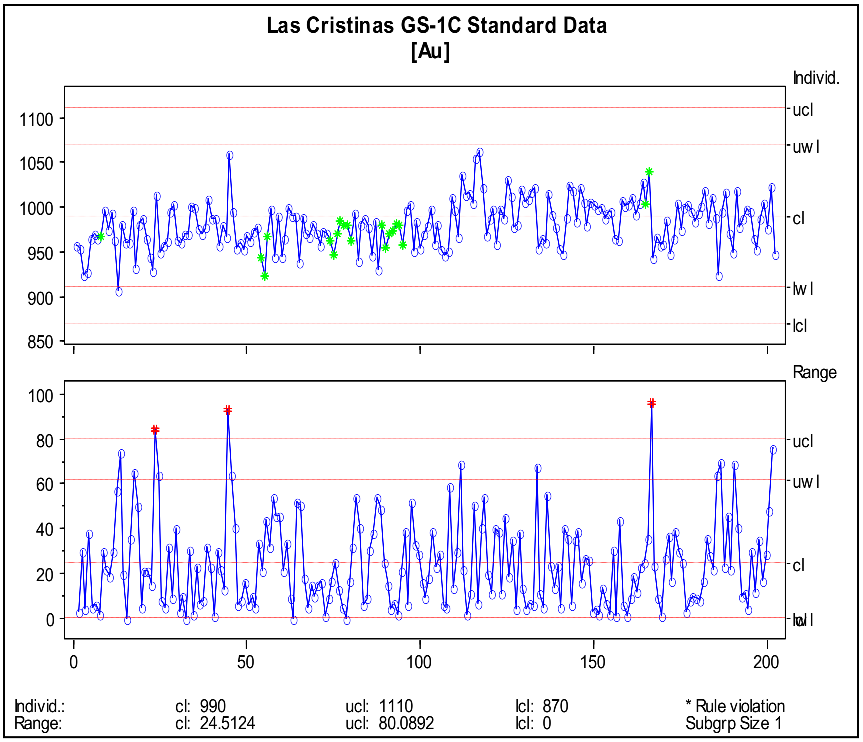


Figure 14.10 Control and Range Charts for Standard GS-1C



Cumulative Sum Chart for Standard GS-1C

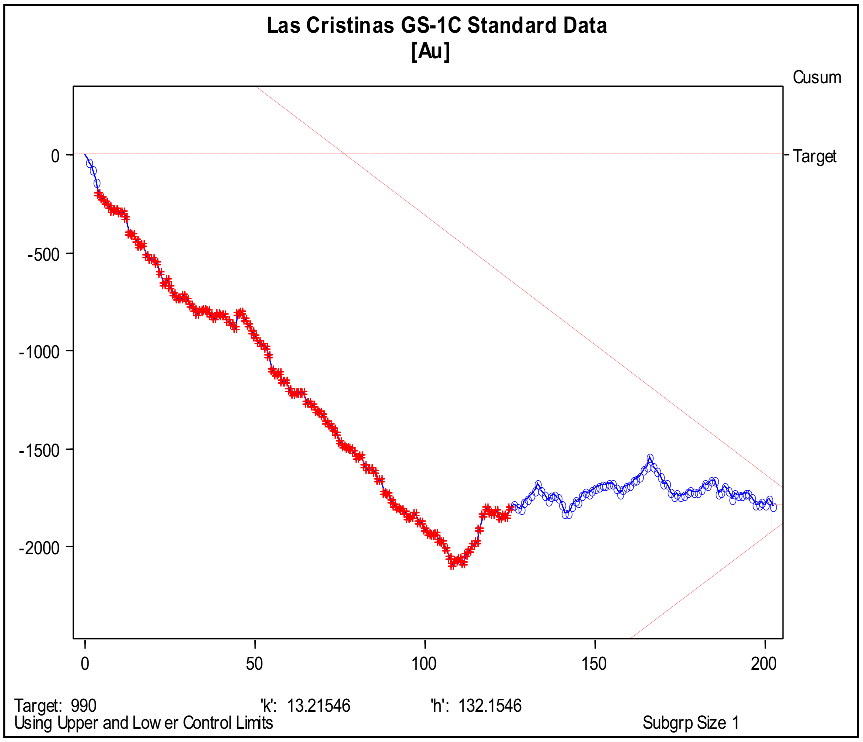
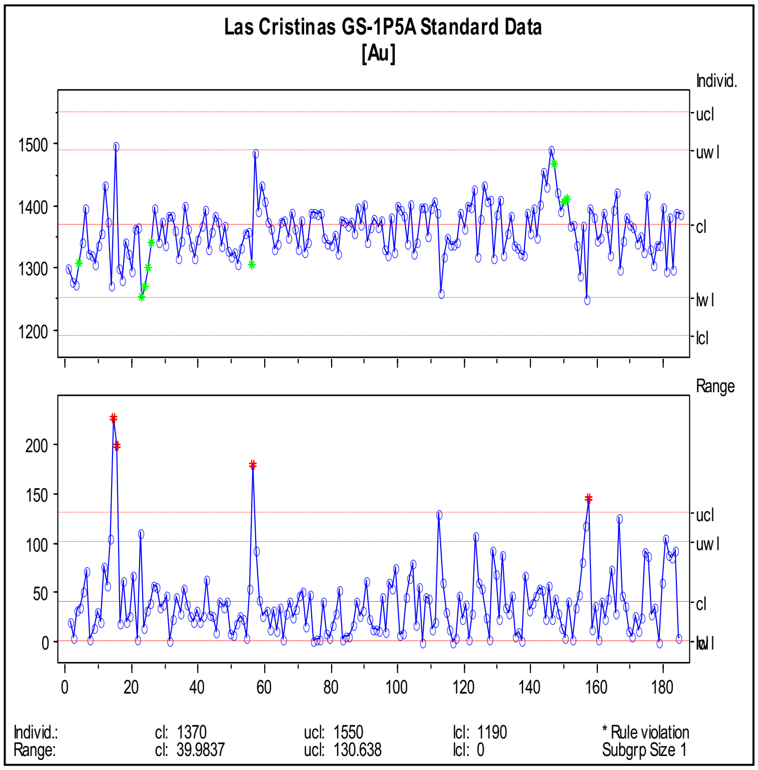


Figure 14.11 Control and Range Charts for Standard GS-1P5A



Cumulative Sum Chart for Standard GS-1P5A

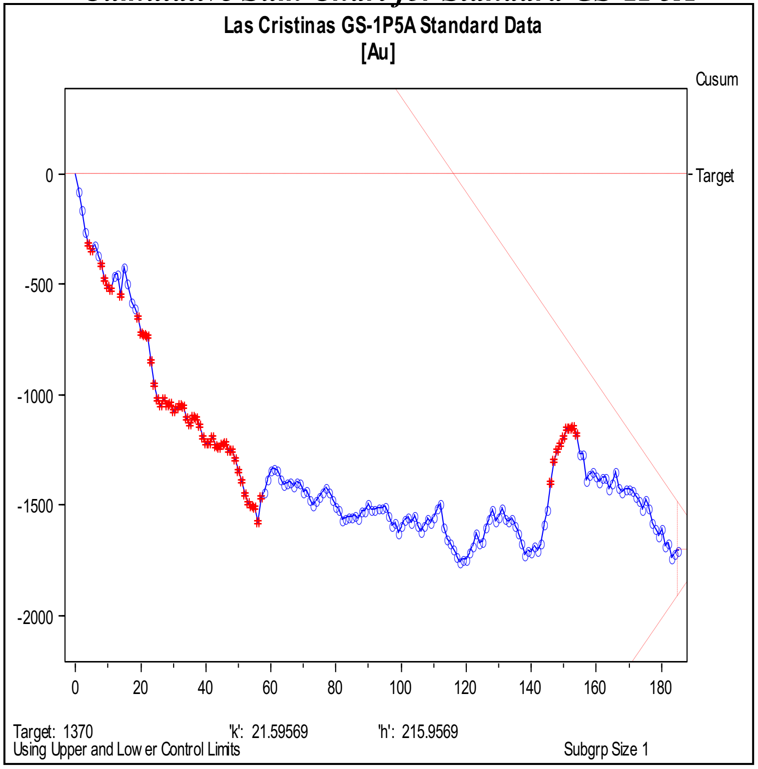
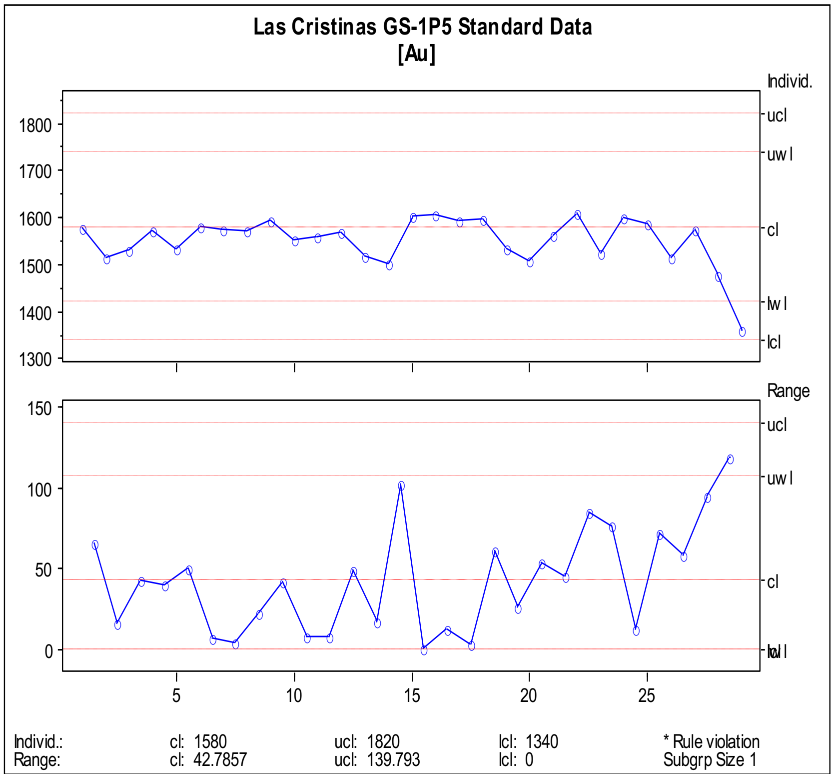


Figure 14.12 Control and Range Charts for Standard GS-IP5



Cumulative Sum Chart for Standard GS-1P5

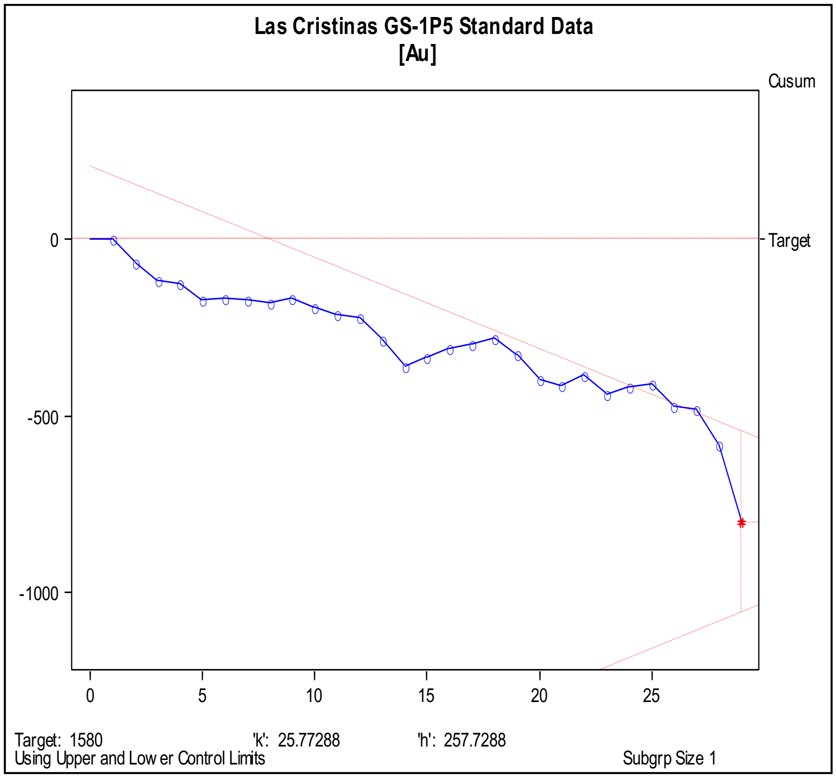
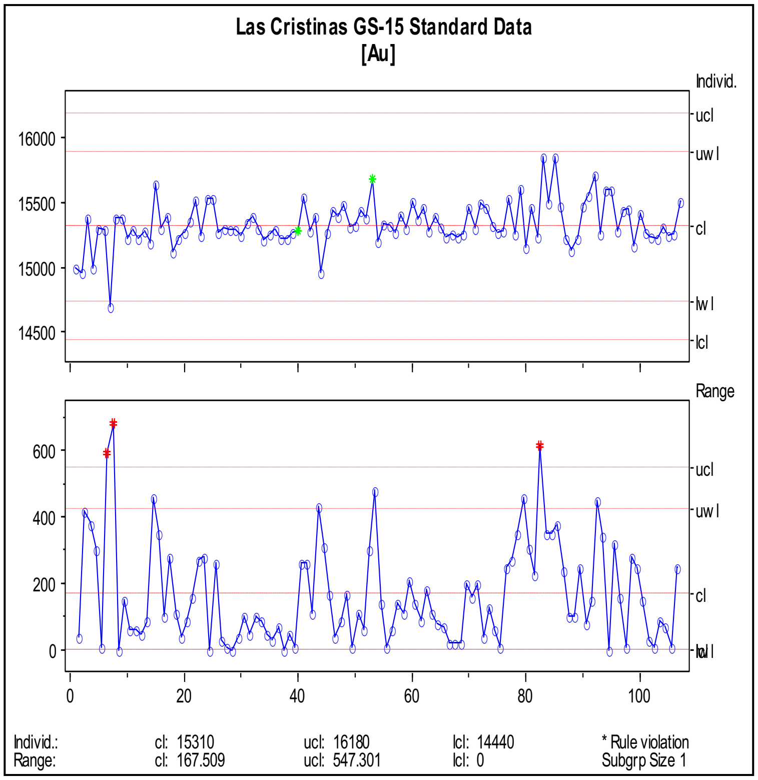
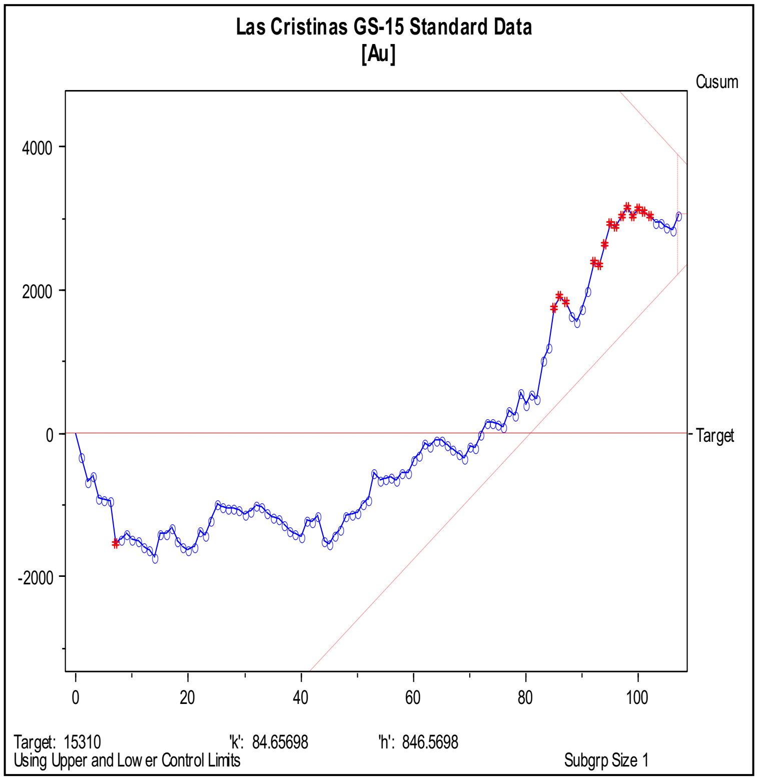


Figure 14.13 Control and Range Charts for Standard GS-15



Cumulative Sum Chart for Standard GS-15



Blank Monitoring

The blank material used in this program was collected from a diorite quarry located some 100km south of the property. Previous assays of the diorite, which is used for construction aggregate, showed that the rock is essentially barren of gold, and it was therefore considered to be useful as blank material. The failure level for the blank material was set at 100 ppb Au by the Crystallex geologists. This is high for testing contamination in a lab setting, but the uncertainty over the baseline levels in the material was taken into consideration.

There were numerous failures of the blank material starting right at the beginning of the program. After multiple analyses, it became apparent that some of the material used as a blank for this program is not completely barren. Several of the blank samples analyzed returned concentrations of gold over the allowable limit of 100 ppb. Since all of these blanks have been analyzed at least twice by the primary lab, it is believed that this gold actually exists in the blank material and is not contamination or analytical error. Several of the blank samples have been re-split from rejects and have returned a comparable result as the initial assay (Table 14.13). There were 436 blanks inserted into 2173 samples (Table 14.14). This gives an overall insertion rate of 3.58% or one standard for every 27.9 samples. Final Shewhart chart is given in Figure 14.14.

Table 14.13 Blank Failures/Corrections for 2006/07 Crystallex Las Cristinas QA/QC Program

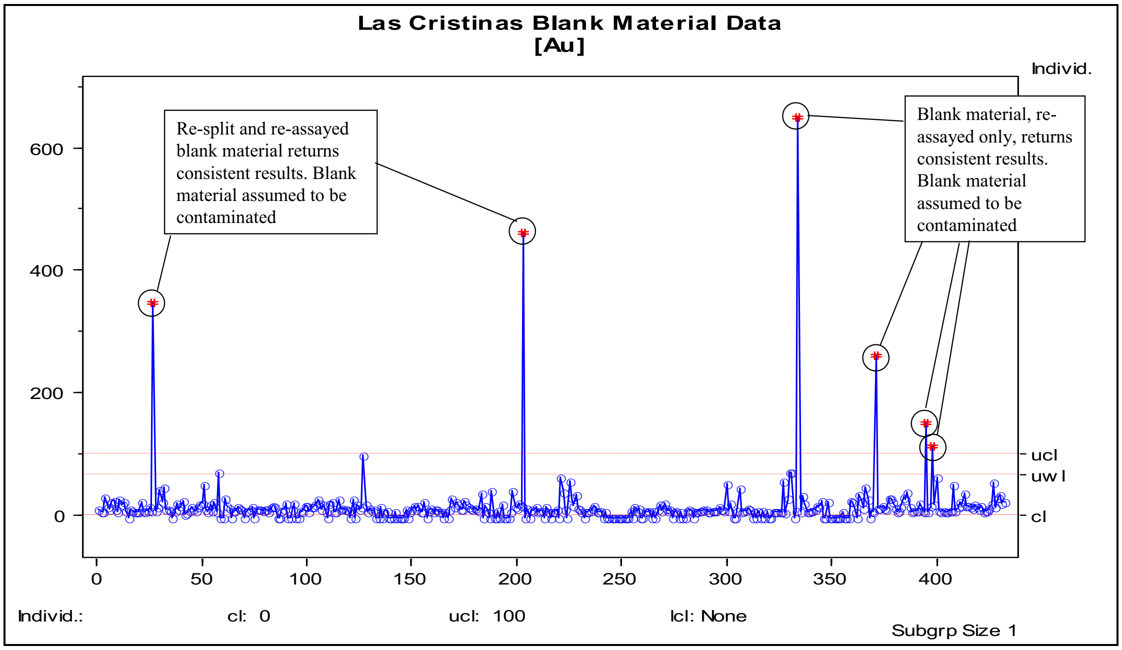
Sample ID	Hole ID	Original Analysis (ppb Au)	Additional Analyses (ppb Au)	
301107	K6MO1166	346	357	323
301567	K6MO1168	188	17	
301624	K6MO1168	421	55	
301717	K6MO1168	461	414	431
302099	K6MO1171	694	15	
304255	K6CO1187	183	70	
306557	K7MO1200	650	650	638
301006	K6MO1166	263	256	
307320	K6MO1170	2052	4	
307812	K6MO1178	112	28	
311825	K7MO1205	150	146	
313476	K7MO1204	260	60	68
302996	K6MO1180	281	112	121

(Red indicates continuing failure as explained in text above)

Table 14.14 Blank Data Summary for 2006/07 Crystallex Las Cristinas QA/QC Program

Standard Name	No. Of Determinations	No. of Failures	Failure Rate	Analysis Mean (ppb)	Standard Deviation	Recommended Value
Blank	436	13	2.98%	14.6	45.35	<100 ppb

Figure 14.14 Control Chart for Blank Material



Re-split and re-assayed blank material returns consistent results. Blank material assumed to be contaminated

Blank material, re-assayed only, returns consistent results. Blank material assumed to be contaminated

14.9.5 External Data Verification

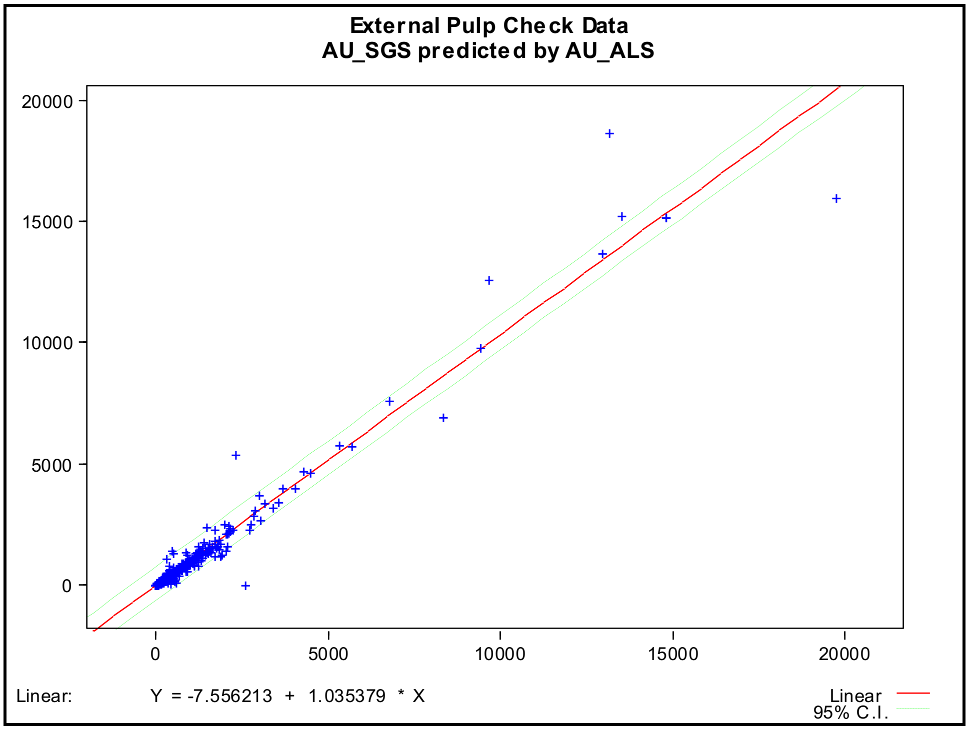
"Original Pulp" Assay (primary lab) vs. "Original Pulp" Assay (external lab)

Of 13,486 original pulps assayed by the primary lab (SGS-Lima), 673 were forwarded to the external lab (ALS Chemex-Lima) for check assaying. This represents a total of 5% or one in 20 samples. The regression plot (Figure 14.15) and statistical analysis (Table 14.15) are presented below.

Table 14.15 Statistical Analysis of External Duplicate Pulp Samples

Mean of primary lab data set	0.707 g Au/t
Standard error of primary lab data set	0.06710
Mean of external lab data set	0.694 g Au/t
Standard error of external lab data set	0.06502
Data items	673
Correlation coefficient	0.979071
Paired t-statistic, 95% C.L.	1.2145
t-critical, 95% C.L., 672 D.F.	1.9635

Figure 14.15 Regression Plot for External Pulp Duplicate Samples



Note: plot is in units of "ppb"

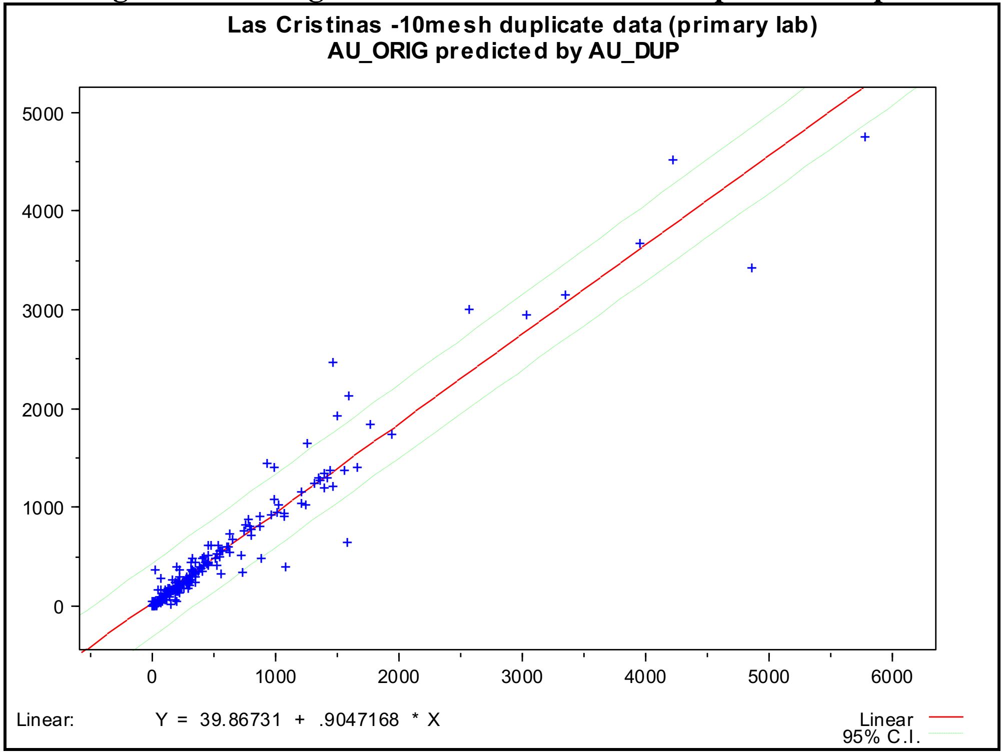
"Original Pulp" Assay (primary lab) vs. -10 Mesh Duplicate Assay (primary lab)

Out of 12,178 drill core samples assayed by the primary lab (SGS-Lima), 203 were accompanied by pulps, which are duplicates obtained by splitting the sample material while still at the -10 mesh stage of sample preparation. This represents a total of 1.66% or one in 60 drill core samples. The regression plot (Figure 14.16) and statistical analysis (Table 14.16) are presented below.

Table 14.16 Statistical Analysis of External Duplicate -10 Mesh Samples

Mean of primary lab data set	0.538 g Au/t
Standard error of primary lab data set	0.05444
Mean of -10 mesh dup. data set	0.540 g Au/t
Standard error of -10 mesh dup data set	0.05648
Data items	203
Correlation coefficient	0.968242
Paired t-statistic, 95% C.L.	0.0838
t-critical, 95% C.L., 202 D.F.	1.9718

Figure 14.16 Regression Plot for -10 Mesh Duplicate Samples



Note: plot is in units of "ppb"

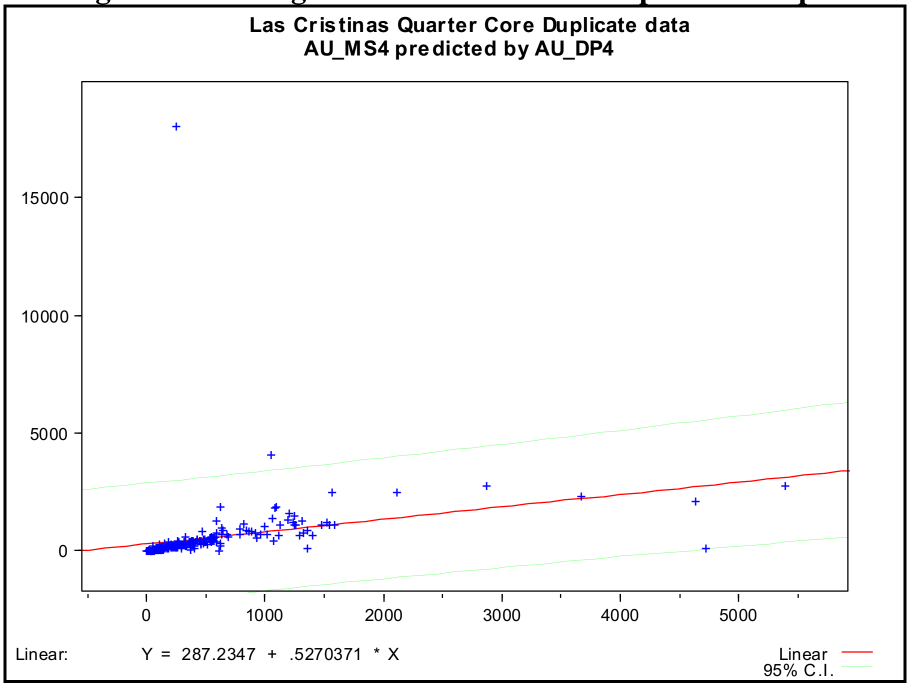
"Original Pulp" Assay (primary lab) vs. ¼ Core Duplicate Assay (primary lab)

A total of 213 out of 12,178 drill core samples assayed by the primary lab (SGS-Lima) were ¼ core samples. These samples were accompanied by pulps, which represented duplicates obtained by splitting the regular half-core samples in half at the core sampling stage and creating two ¼ core samples. This represents a total of 1.74% or one in 57 drill core samples. The regression plot (Figure 14.17) and statistical analysis (Table 14.17) are presented below.

Table 14.17 Statistical Analysis of External Core Duplicate Samples

Mean of primary ¼ core data set	0.556 g Au/t
Standard error of primary ¼ core data set	0.09178
Mean of duplicate ¼ core data set	0.511 g Au/t
Standard error of duplicate ¼ core data set	0.04942
Data items	213
Correlation coefficient	0.28377
Paired t-statistic, 95% C.L.	0.50207
t-critical, 95% C.L., 213 D.F.	1.9712

Figure 14.17 Regression Plot for Core Duplicate Samples

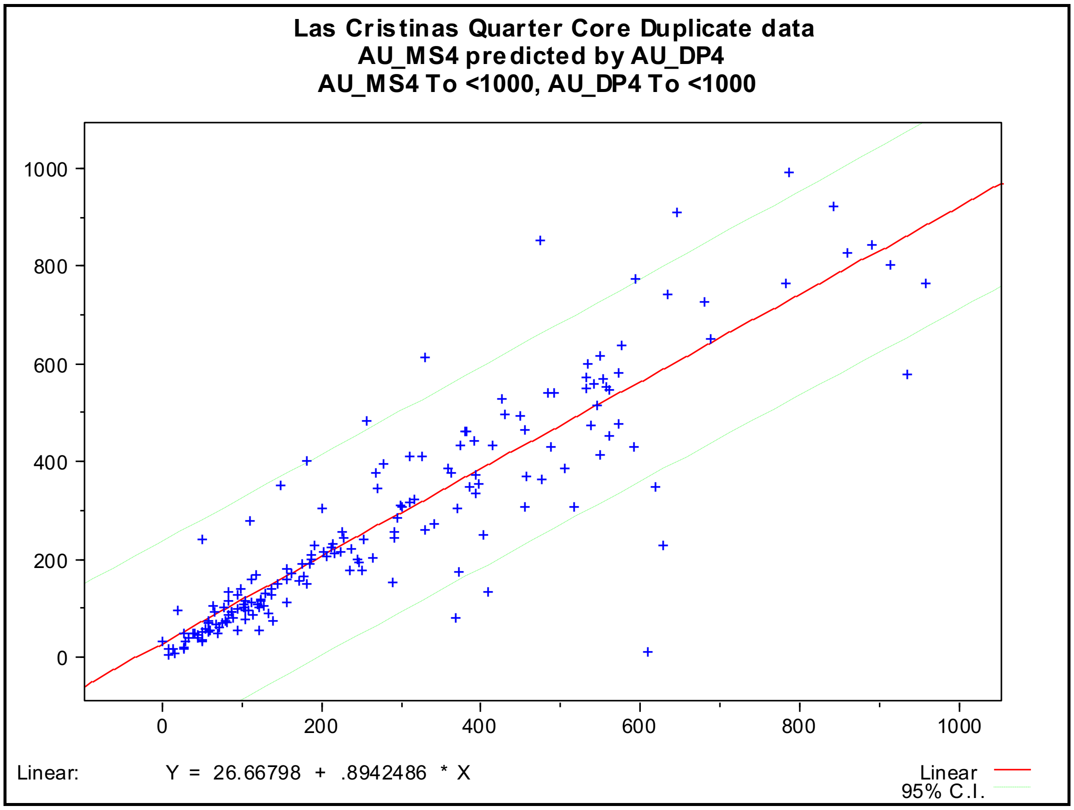


Note: plot is in units of "ppb"

As expected with ¼ core duplicates, the higher the grade of gold in the sample the less reproducible the assay value becomes. This is likely due to the nugget effect. Samples with high grades typically have a metallic gold component to them where the gold occurs within the sample as discrete "nuggets" instead of as an evenly distributed gold ore. This can be seen by constructing regression plots with the high-grade values filtered out. Figure 14.18 is plot for values with Au concentration <1 g Au/t. The correlation gets better as the upper limit of Au concentration is lowered.

Figure 14.18 Quarter-core Duplicate Data

(Au in ppb)



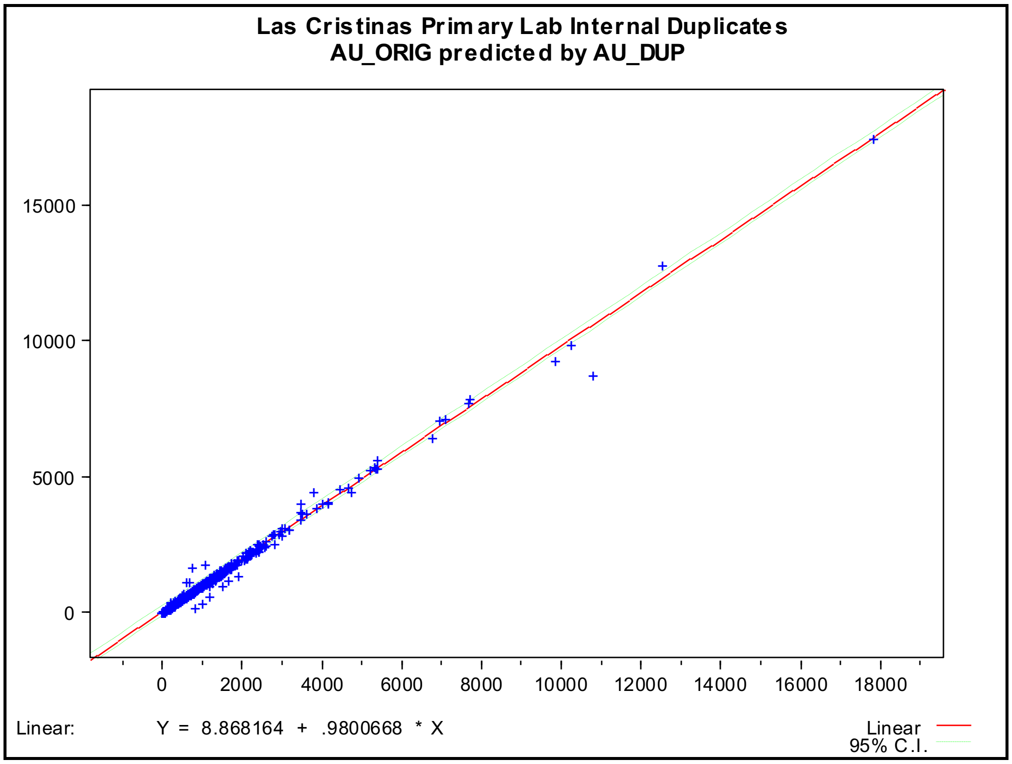
"Original Pulp" Assay (primary lab) vs. Internal Duplicate Assay (primary lab)

Out of 12,178 drill core samples assayed by the primary lab (SGS-Lima), 1,221 duplicate pulp samples were forwarded to the external lab (ALS Chemex-Lima) for check assaying. This represents a total of 10.02% or one in 10 drill core samples. The regression plot (Figure 14.19) and statistical analysis (Table 14.18) are presented below.

Table 14.18 Statistical Analysis of Internal Lab Duplicate Pulp Samples

Mean of primary lab data set	0.610 g Au/t
Standard error of primary lab data set	0.03245
Mean of internal duplicate data set	0.613 g Au/t
Standard error of internal duplicate data set	0.03299
Data items	1221
Correlation coefficient	0.99659
Paired t-statistic, 95% C.L.	1.2145
t-critical, 95% C.L., 1220 D.F.	1.9619

Figure 14.19 Regression Plot for Internal Lab Duplicate Pulp Samples



Note: plot is in units of "ppb"

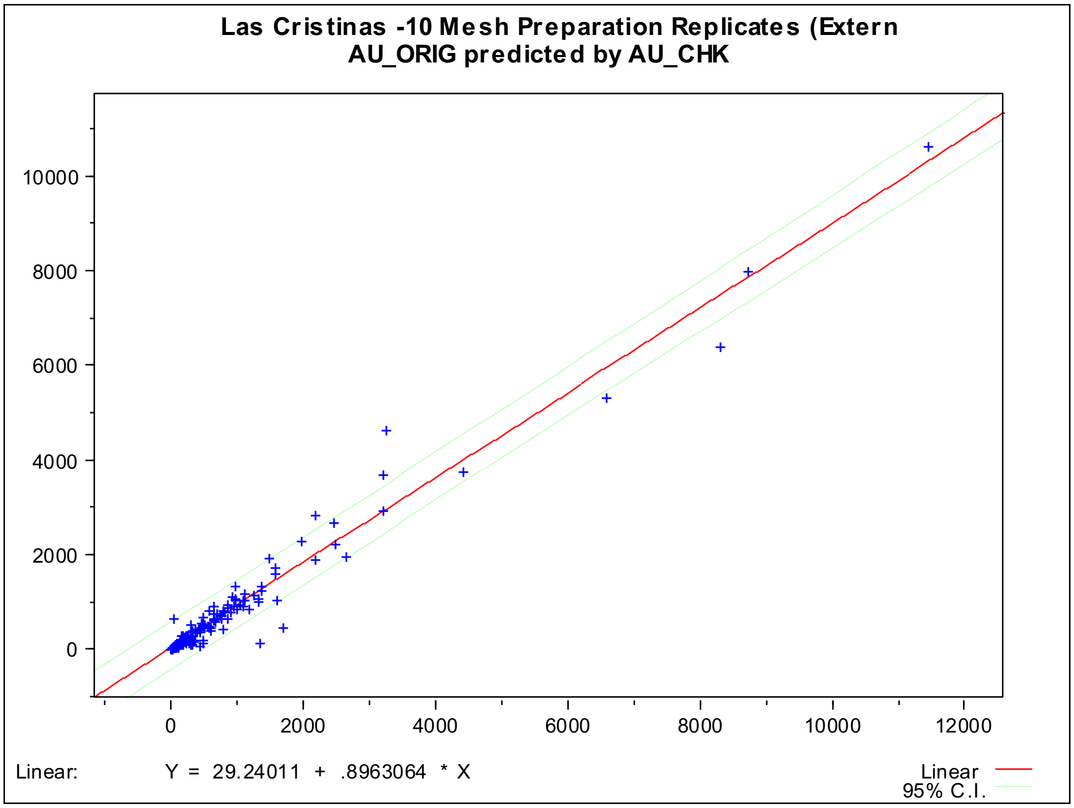
"Original Pulp" Assay (primary lab) vs. -10 mesh Duplicate Assay (external lab)

Out of 12,178 drill-core samples assayed by the primary lab (SGS-Lima), 181 -10 mesh duplicate samples were forwarded to the external lab (ALS Chemex-Lima) for check assaying. This represents a total of 1.5% or one in 66 drill core samples. The regression plot (Figure 14.20) and statistical analysis (Table 14.19) are presented below.

Table 14.19 Statistical Analysis of External Lab -10 Mesh Duplicate Samples

Mean of primary lab data set	0.6774 g Au/t
Standard error of primary lab data set	0.0096
Mean of external lab data set	0.7131 g Au/t
Standard error of external lab data set	0.0105
Data items	181
Correlation coefficient	0.980985
Paired t-statistic, 95% C.L.	0.31917
t-critical, 95% C.L., 180 D.F.	1.9732

Figure 14.20 Regression Plot for External Lab -10 Mesh Pulp Samples



Note: plot is in units of "ppb"

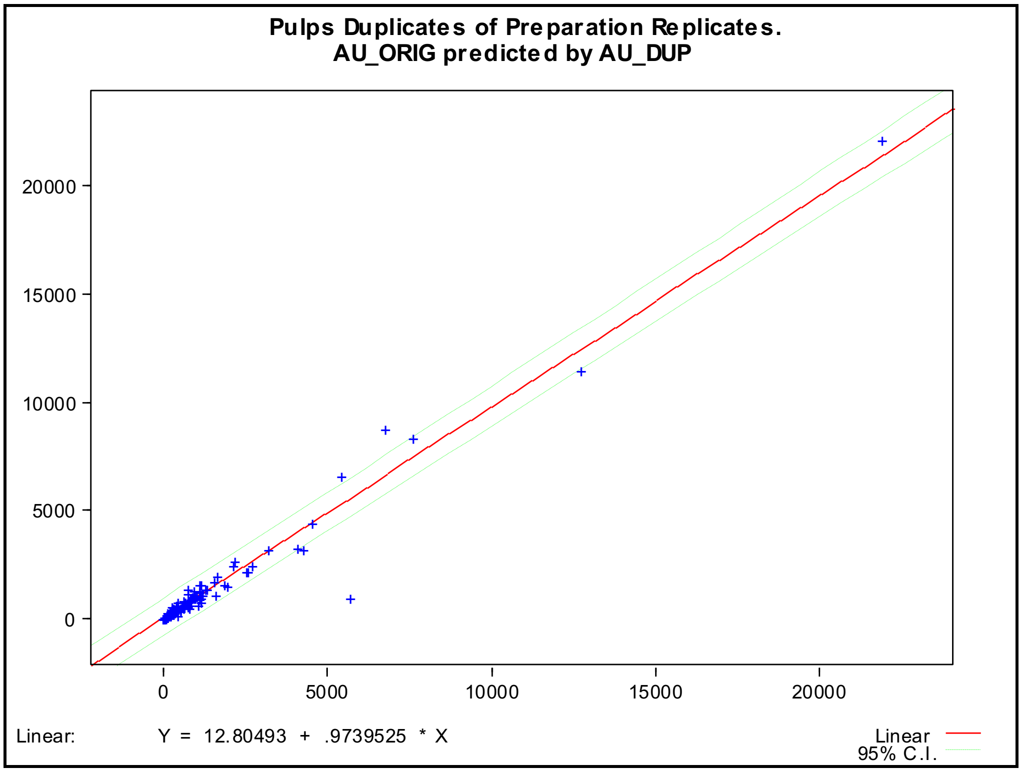
-10 mesh Duplicate Assay (external lab) vs. Pulp Duplicate Assay (external lab)

Out of 12,178 drill core samples assayed by the primary lab (SGS-Lima), 188 external lab -10 mesh samples were forwarded to the external lab (ALS Chemex-Lima) for check assaying. This represents a total of 1.54% or one in 65 drill core samples. The regression plot (Figure 14.21) and statistical analysis (Table 14.20) are presented below.

Table 14.20 Statistical Analysis of External Lab -10 Mesh Duplicate Samples vs. Pulp Duplicate

Mean of -10 mesh dup data set	0.840 g Au/t
Standard error of -10 mesh dup data set	0.1559
Mean of pulp duplicate data set	0.849 g Au/t
Standard error of pulp duplicate data set	0.1566
Data items	188
Correlation coefficient	0.978512
Paired t-statistic, 95% C.L.	0.28678
t-critical, 95% C.L., 187 D.F.	

Figure 14.21 Regression Plot of External Lab -10 Mesh Duplicate Samples vs. Pulp Duplicate



Note: plot is in units of "ppb"

14.9.6 Summary and Conclusions

Crystallex's 2006-2007 Las Cristinas drill program was subject to quality control measures that have ensured that the resulting data are precise and accurate. The program used a combination of standard, blank, and duplicate analyses to achieve this goal.

Primary quality control was achieved through the use of certified reference standards embedded in the sample stream and blind to the laboratory. Through the use of Shewhart and CuSum charts, NAC was able to correct any deficiencies in the data on a continuing basis.

The program identified four instances where the analytical data for the certified standards were of an unacceptable quality and 13 instances where the analytical data for the blank material were of an unacceptable quality. This triggered the re-assay of 645 of the drill core samples in order to correct the failures. All failures that occurred during this phase of the quality control process were corrected to NAC's satisfaction.

Data verification was achieved by use of duplicate analysis by both the primary and an external check laboratory. In all instances, the comparisons of means of the duplicate and original datasets agree at the 95% confidence level.

All duplicate dataset pairs also show good correlation with each other, with the exception of the ¼ core duplicates with grades above 1 g Au/t.

NAC believes that Crystallex has a dataset from this drill program that they can depend on to advance their objectives at Las Cristinas.

14.10 Grade versus Core Recovery Comparison

As grade bias can be introduced into samples while drilling core in rock of variable hardness and because there was a suggestion of such an effect in Placer's work, MDA evaluated the relationship between metal grades and core recovery. A bias was discovered in the saprolite gold data, which was found to be most prevalent in low-grade samples. This bias does not exist in the bedrock, which makes up the majority of the resource and reserve. The bias should not materially affect the global estimated gold and silver grades; however within the saprolite in areas where core recovery is low, grades could be lower than predicted. A summary of the saprolite data at different grade cutoffs is shown in Table 14.21. The number of saprolite samples with recoveries below 90% represents just over 40% of the total saprolite samples in the database, for both cutoffs in the table.

Table 14.21 Gold Grade vs. Core Recovery in Saprolite

Core Recovery	> 0.0 g Au/t Avg Grade g Au/t	> 0.3 g Au/t Avg Grade g Au/t
< 90%	0.89	1.71
> 90%	0.72	1.56
Difference	24%	10%

In bedrock, the copper grade is 3% lower in lower-recovery (<90%) samples, which is not considered significant. However, in the combined saprolite and saprock, copper grades range from 5% to 9% higher for low-recovery samples, which amounts to roughly half the population of saprolite samples. This could result in overstating the copper grade by up to 10% in the saprolite. This is not significant for the oxide saprolite, but could be significant for the mixed and sulfide saprolite. Figure 14.22 and Figure 14.23 illustrate these relationships graphically.

The decreased confidence in the lower-core-recovery samples was considered when classifying material into Measured, Indicated and Inferred resource categories. Lower core-recovery values estimated into the blocks were assigned a lower confidence rating by modifying the distance used for classification.

Figure 14.22 Box and Whisker Plot for Gold Grade versus Core Recovery

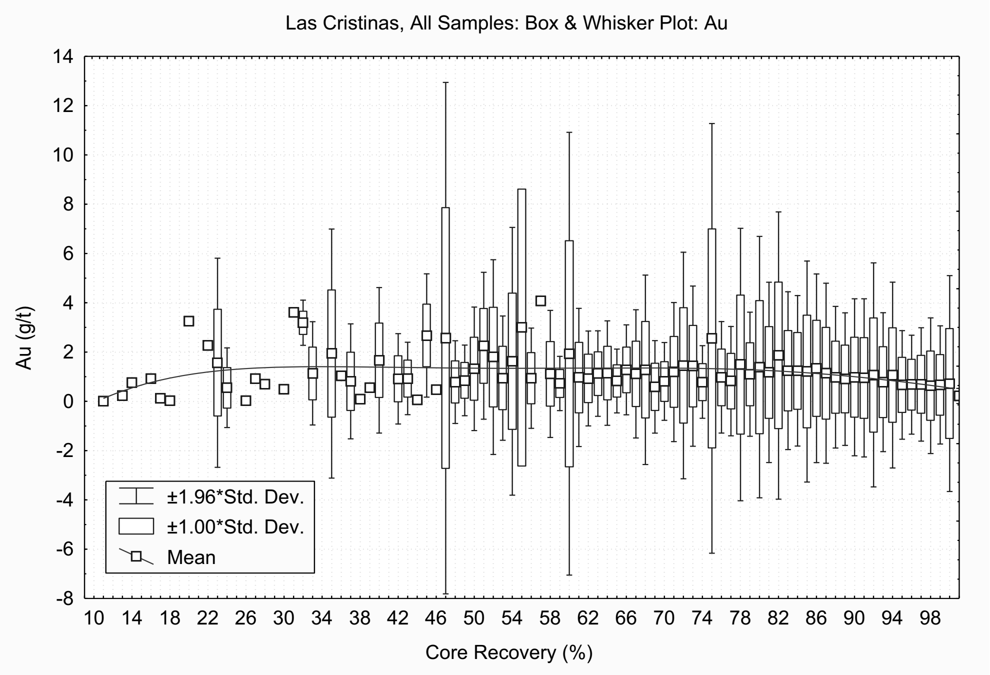
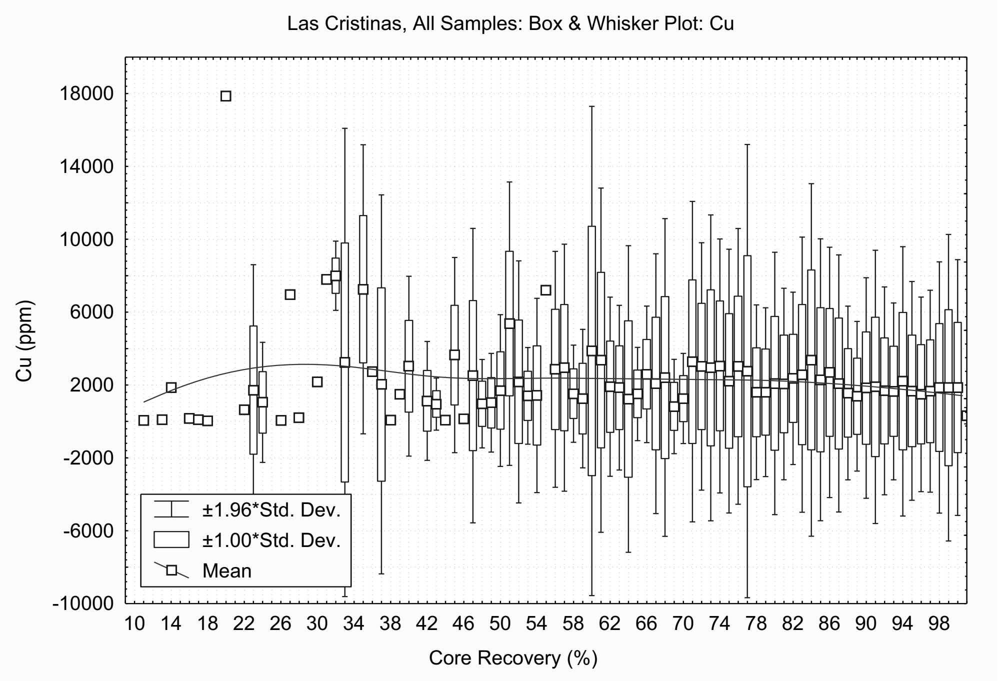


Figure 14.23 Box and Whisker Plot for Copper Grade versus Core Recovery



14.11 Miscellaneous Data Verification and Sampling Studies

Early in the project history, Ristorcelli (2003) reported that the Crystallex verification drilling showed a potential bias, although it was passed on as not material compared to the total amount of drilling done by Placer. Repeating the statement from the 2003 Las Cristinas Technical Report: "*After analysis of the 2003 drill program, MDA believes that the Las Cristinas database can be used for feasibility-level study and resource estimation. Having said this, all future work must be cognizant of the underlying difference in grades between Placer data and the Crystallex verification drilling and the difference must be explained. It cannot be stated which is the more accurate at this time but the data remains sufficiently accurate for further use. Negligible contamination during sample preparation may have occurred during sample preparation of the Crystallex samples. The larger concern is the high variance noted in check assays, which should not affect the global metal estimate but could affect local estimates. This concern can be mitigated by completing a heterogeneity study of gold in the rock.*"

As a consequence of the above statement, Crystallex contracted Mr. Francis Pitard to perform a heterogeneity test to 1) quantify the effect of sample mass on estimated sample grades, which could clarify substantial differences observed between Crystallex sampling and Placer sampling, 2) assess the adequacy of samples based on the material heterogeneity of the deposit, and 3) recommend a sampling protocol. From this, Pitard (2005) concluded that:

> *"When comparing composited grades from 30-g conventional fire assays with average 12859-g total assays, the following observations can be made:*
>
> - *Between 0.40 and 1.10 g/t, original assays can be underestimated by about 25%.*
> - *Between 1.10 g/t and 2.3 g/t, original assays can be underestimated by about 7%.*
> - *Above 2.3 g/t, original assays can be overestimated by about 15%.*
> - *Overall, large 12859-g samples show an average increase of the gold content about 7.7%.*
>
> *However, this test was performed with only 49 samples* [derived from the 266 original individual samples]*, while I originally requested 50 to 100 samples: Biases are real, but they lack accuracy."*

Pitard recommended, *"Do not focus too much on the grade increase. However, you should most certainly focus your attention on the possible increase of the ore reserves. The difference can be quite potent for the project"* and went on to report:

> ***"The In Situ Nugget Effect***
>
> *Results from the Heterogeneity Test help to roughly quantify the effect of diamond drilling diameter on the evaluation of the gold resources. Results are as follows:*
>
> - *A ½ PQ 2m core is 9 times too small to include a gold cluster where it should for a 0.5 g/t grade, 4 times too small for 1.1 g/t, and 2 times too small for 2.5 g/t.*
> - *A ½ HQ 2m core is 18 times too small to include a gold cluster where it should for a 0.5 g/t grade, 8 times too small for 1.1 g/t, and 3 times too small for 2.5 g/t.*
> - *A ½ NQ 2m core is 25 times too small to include a gold cluster where it should for a 0.5 g/t grade, 11 times too small for 1.1 g/t, and 5 times too small for 2.5 g/t.*
> - *These are only approximate numbers and you should look at them with circumspection. It is the general message that counts: Placer Dome had a much better chance to include gold clusters with the systematic use of a larger diameter.*
> - *However, it is very clear the diameter used by Placer Dome was already too small. Furthermore, when you drill, the smaller the diameter the less the recovery, and*

especially for the gold contained in crumbling clusters of sulfides/quartz which may occur at Las Cristinas. So, we are really looking at the tip of the iceberg with the present Heterogeneity Test: It is good news for the Crystallex management.

- *Unfortunately, nobody can make miracles by looking at only 49 samples, therefore it would be ludicrous to make any attempt at quantifying what I called "good news."*

14.12 Data and Sample Verification Conclusions

The original objective of the 12 twin-hole program in 2003 was to have independent verification of the Las Cristinas mineralization, which the program did accomplish. Additional checks on Placer's sample data that included pulps, coarse rejects, and quarter core which were sampled and reassayed further verified the sample grades reported by Placer. Throughout the exploration conducted by Crystallex, Crystallex has maintained a high degree of technical quality, responding and correcting those issues that were deemed weak or improper. Crystallex has also allowed independent consultants free access to the data and samples, such as Mr. Maynard, an associate of MDA, who maintained his own sample custody and Mr. Trevor Nicolson, who was involved with sampling in the 2006-2007 drill campaign.

Issues of variability and biased-low samples were addressed in a heterogeneity study. The high variability must be addressed prior to and during production to avoid massive misclassifications of ore and waste rock during production. This material heterogeneity or grade variability has negatively impacted the ability to make any resource estimate precisely reflect local estimated grades. Importantly, the style of mineralization and its natural variability are the likely causes of the underlying difference in grades between Placer data and Crystallex data, where Crystallex samples are both smaller and slightly lower grade than Placer's grade. It has been demonstrated that this is likely due to sample size. Taking this further, it is possible that the entire sample database might be understating the mean grade of the deposit. While this appears possible and even likely, there is no possible way to quantify this potential underreporting of grade and no way to incorporate this into the database or resource model.

15.0 ADJACENT PROPERTIES

The value of Las Cristinas is not dependent on any adjacent properties. It is a stand-alone property based on its own merits. There are numerous artisanal mining workings scattered in the region, and recent exploration by Gold Reserve Inc. ("GRI") of Spokane, Washington, has resulted in the definition of a reserve of gold and copper in the Brisas del Cuyuni ("Las Brisas") property located immediately to the south of Las Cristinas. The following selected paragraphs describing the Las Brisas property are taken from GRI's website (http://www.goldreserveinc.com/properties.asp):

> *Brisas is a large resource of low-grade disseminated gold and copper mineralization of Precambrian greenstone type. The mineralization is hosted in a fine-grained volcanic rock that was deposited in a water-filled basin as sediment. The copper and gold mineralization was introduced into the rocks during deposition of the host and subsequently modified by metamorphism and tropical weathering.*
>
> ***Surface assessment***
> *Surface exposures of weathered rock are limited to the walls of small flooded pits, rare weathered outcrops and areas cleared by past surface mining activity. Intense weathering produced saprolites to a depth of 60 meters making drilling the primary tool used to define subsurface geology. The rocks encountered in drilling include oxide saprolite, sulfide saprolite and the hard or unweathered bedrock. Andesite tuff units defined in the hardrock include a series of vitric tuffs, lapilli tuffs and crystal tuffs. Based on megascopic and microscopic features, the andesite tuffs are interpreted to have been deposited in shallow water. All the rock units have been metamorphosed to greenschist facies. The andesite tuffs have been intruded by a series of basic dikes and sills and a large monzonite stock. The stock is confined to the east edge of the property.*
>
> ***Geologic structure of the property***
> *Geologic correlation has identified a stratigraphic sequence from top to bottom of: 1) a thick unit of vitric tuff; 2) a two hundred meter thick unit consisting of mixed lapilli tuffs, crystal tuffs and vitric tuffs characterized by rapid vertical and lateral textural changes, 3) a series of thicker, more consistent crystal tuffs, vitric tuffs and lapilli tuffs. The structure of the property is very simple. The tuffs dip shallowly to the west and strike north-south. Very little faulting has been identified and those faults identified are the sites of the basic dikes. Movement along these faults is minimal and often there appears to be no movement at all.*
>
> *Ore grade mineralization is stratabound and strataform within the 200-meter thick unit characterized by rapid vertical and horizontal changes. Mineralization follows this unit down from the surface and is open at depth. In addition, the deposit is open to the southwest. Three basic types of mineralization exist. Oxide mineralization, restricted to the oxide saprolites, is gold only and makes up about four percent of the total mineralization. Massive sulfide mineralization, as both laminated sulfides and quartz-tourmaline-sulfide breccia pipes, has been identified on the surface and from drilling.*
>
> *The Blue Whale is an example of this type of mineralization. It contains relatively high-grade copper and gold mineralization. The Blue Whale makes up only a small percentage of the total deposit. The majority of the mineralization is disseminated sulfide mineralization in discrete pyrite grains within the tuffs and as narrow restricted quartz-carbonate-pyrite veinlets. These veinlets often contain visible gold. The disseminated mineralization can be further subdivided into a copper-gold-pyrite mineralization and a pyrite-gold mineralization. The sulfide saprolite and the underlying weathered rock unit are unoxidized and contain typical disseminated sulfide mineralization. The copper-gold-pyrite mineralization dominates*

the northern portion of the deposit while the gold-pyrite mineralization dominates the southern portion of the orebody. Alteration within the deposit includes massive carbonate often associated with epidote and chlorite. The character of the mineralization and the alteration is consistent with typical gold-in-greenstone type deposits found elsewhere in the world's greenstone-granite terrenes.

Mineral Resource and Reserve Estimates

Pincock, Allen & Holt ("PAH"), of Denver, Colorado, reviewed the methods and procedures utilized by the Company at the Brisas Project to gather geological, geotechnical, and assaying information and found them reasonable and meeting generally accepted industry standards for a bankable feasibility level of study…

Mineral Resource Estimate

Based on work completed by PAH for the Brisas bankable feasibility study, using an off-site smelter process for treating copper concentrates, the Brisas Project is estimated to contain a measured and indicated mineral resource of 12.1 million ounces of gold and approximately 1.6 billion pounds of copper (based on 0.4 gram per tonne gold equivalent cut-off). The October 2006 estimated measured and indicated mineral resource utilizing an off-site smelter process is summarized in the following table:

(kt=1,000 tonnes)	Measured			Indicated			Measured and Indicated		
Au Eg		Au	Cu		Au	Cu		Au	Cu
Cut-off Grade	kt	(gpt)	(%)	kt	(gpt)	(%)	kt	(gpt)	(%)
0.40	250,565	0.686	0.119	323,371	0.637	0.130	573,936	0.658	0.125
(In Millions)	Measured			Indicated			Measured and Indicated		
Au Eg		Au	Cu		Au	Cu		Au	Cu
Cut-off Grade		oz.	lb.		oz.	lb.		oz.	lb.
0.40	-	5.527	657	-	6.621	927	-	12.148	1,584

The inferred mineral resource, based on an off-site smelter process (0.4 gram per tonne gold equivalent cut-off), is estimated at 115 million tonnes containing 0.59 grams gold per tonne and 0.12 percent copper, or 2.18 million ounces of gold and 294 million pounds of copper.

[Some text eliminated for brevity.]

The mineral resource and gold equivalent (AuEq) cut-off is based on $400 per gold ounce and $1.15 per pound copper. The qualified persons involved in the property evaluation and resource and reserve estimates were Susan Poos, P.E. of Marston & Marston, Inc., Richard Addison, P.E., and Rick Lambert, P.E., of PAH.

Mineral Reserve Estimate

Based on the NI 43-101 Technical Report completed by PAH during October 2006, using an off-site smelter process for treating copper concentrates, the Brisas Project is estimated to contain a proven and probable mineral reserve of approximately 10.4 million ounces of gold and 1.3 billion pounds of copper. The October 2006 estimated proven and probable mineral reserve utilizing traditional flotation and off-site smelter processes is summarized in the following table:

	Reserve tonnes	Au Grade	Cu Grade	Au Ounces	Cu Pounds	Strip
Class	(millions)	(gpt)	(%)	(thousands)	(millions)	Ratio
Proven	226.3	0.69	0.12	5,032	601	
Probable	258.4	0.64	0.13	5,357	737	
Total	484.6	0.67	0.13	10,389	1,338	1.96

The mineral reserve (within a pit design) has been estimated in accordance with the SME Reporting Guide and CIMM Standards as adopted by CSA National Instrument 43 - 101, which we believe is substantially the same as SEC Industry Guide 7. The mineral reserve was estimated using metal prices of U.S. $400 per ounce gold and U.S. $1.15 per pound copper with an internal revenue cut-off of $3.04 per tonne. The qualified persons involved in the property evaluation and resource and reserve estimates were Susan Poos, P.E., of Marston & Marston, Inc., Richard Addison, P.E. and Rick Lambert, P.E. of PAH.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Introduction

This section was written by J.R. Goode and Associates ("Goode"); it had been reviewed by SNC-Lavalin in 2005 and has been edited for reporting consistency for this report, but the opinions expressed herein are those of Goode.

The Las Cristinas deposit comprises oxidized saprolite ("SAPO") over sulfide- and copper-enriched saprolite ("SAPS") lying over bedrock; SAPS for most purposes, as well as metallurgy, includes mixed saprolite ("SAPM"). A thin transitional zone of saprock ("SAPK") lies between saprolite and the underlying harder bedrock layers. Leaching has removed calcite from the uppermost bedrock layer forming what is termed the carbonate-leached bedrock ("CLB"). The lowermost bedrock layer is carbonate-stable bedrock ("CSB"). Gold occurs in all layers and at similar grades. Copper is absent from the SAPO, enriched in the SAPS, and present at low levels in the CLB and CSB. The sections on geology (Sections 7 and 9) should be referenced for more details.

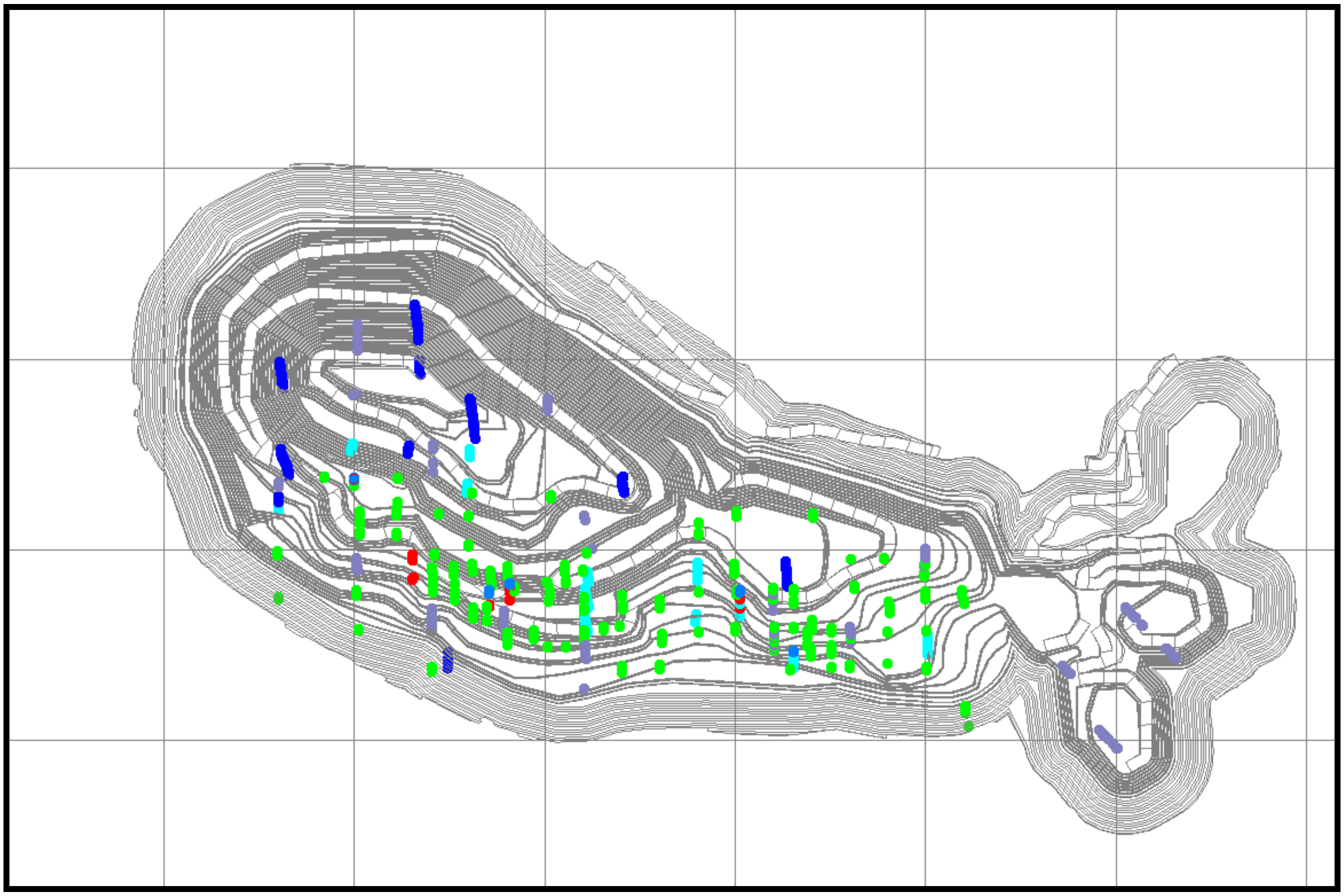
The Las Cristinas property was extensively explored and tested by Placer after the company acquired an interest in the property in the early 1990s. Much of the testwork was performed at the Metallurgical Research Centre in Vancouver which issued 18 reports on the metallurgy of the project between September 1992 and June 1998. Bench testwork covered most aspects of the metallurgy of the deposit. . The location of Placer's metallurgical samples is presented in Figure 16.1, Figure 16.2, and Figure 16.3.

In early 1994, Placer concluded (Placer Dome Inc., Metallurgical Research Centre, 1994) that "*test results so far support a simple gravity concentration/cyanidation circuit for gold recovery from the Las Cristinas ores. Anticipated gold recoveries are:*

Oxidized Saprolite	*92%*
Sulphide Saprolite	*94%*
Carbonate Leached Bedrock	*93%*
Carbonate Stable Bedrock	*89%*"

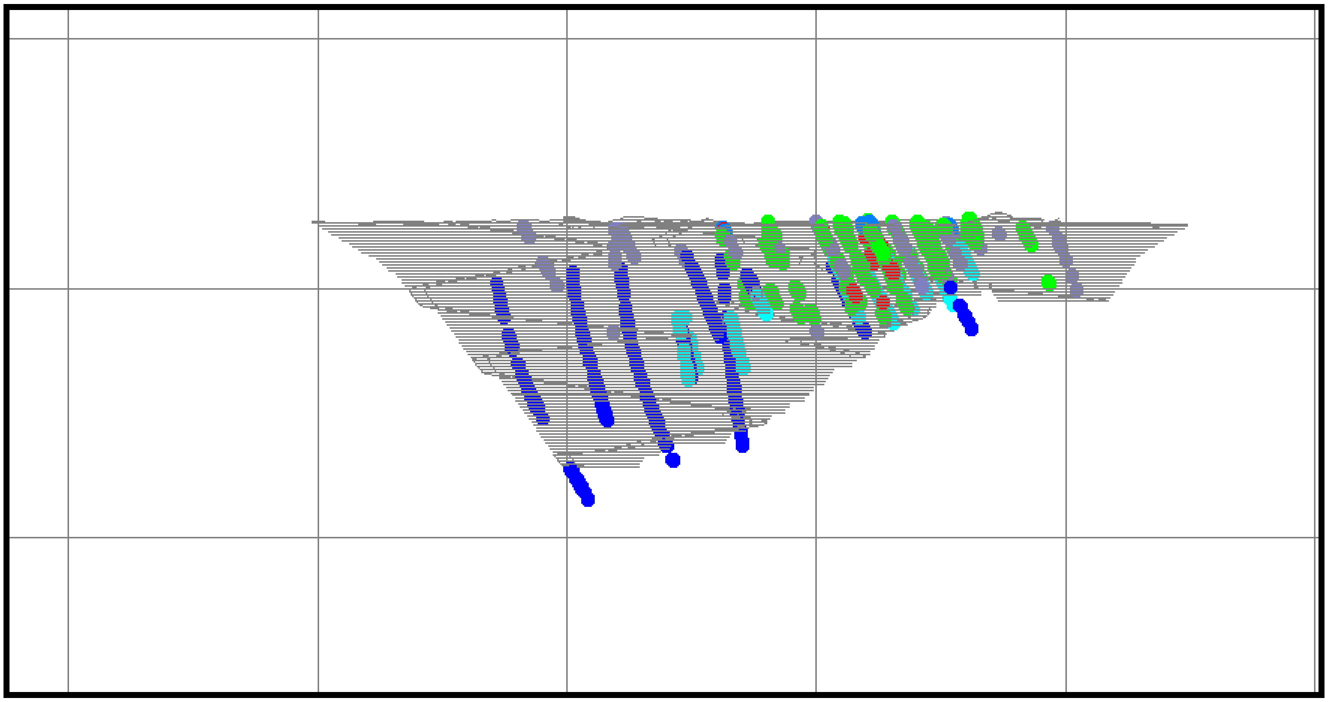
However, Placer decided to additionally recover copper from the deposit and developed a gravity-flotation circuit that would produce a copper-gold flotation concentrate for custom processing in an off-shore smelter. To give adequate overall gold recovery, it was necessary to cyanide leach certain flotation products.

Figure 16.1 Plan Map Showing Metallurgical Samples



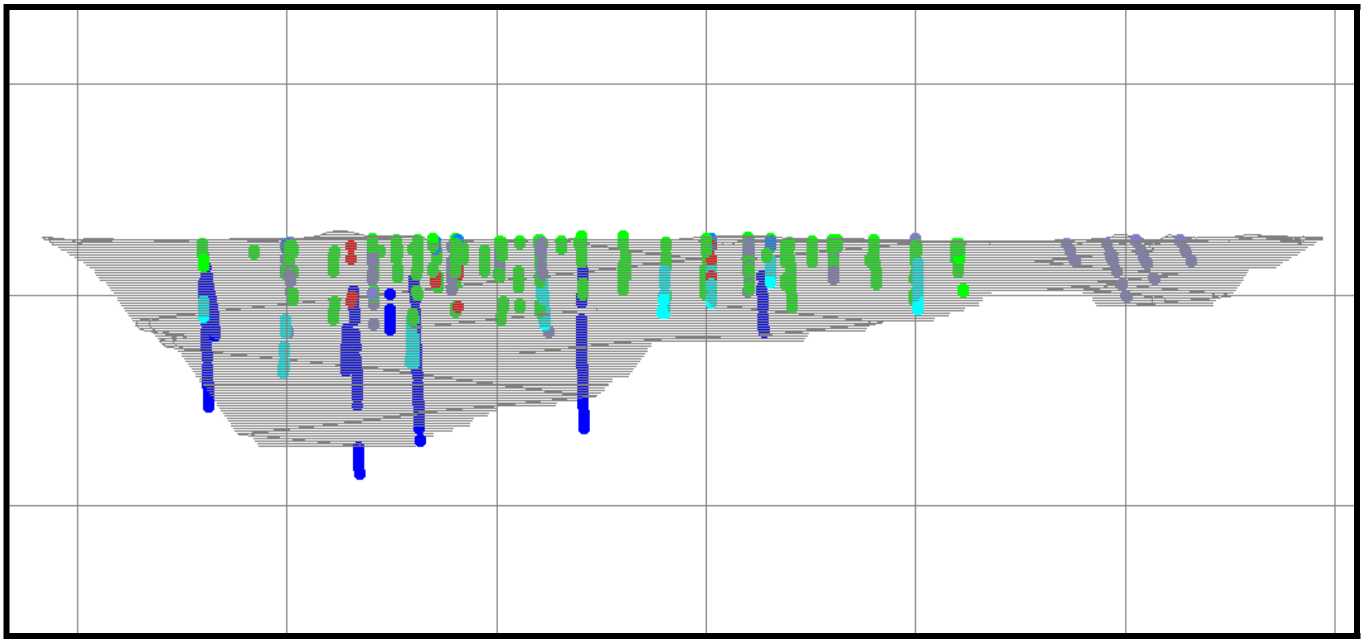
(Placer metallurgical samples: green; acid rock drainage samples: red; Crystallex 2003 metallurgical samples: dark blue; Crystallex 2004 metallurgical samples: cyan; Crystallex 2005 metallurgical samples: medium blue; Crystallex 2006 metallurgical samples: purple-blue)

Figure 16.2 Cross Section Showing Metallurgical Samples – Looking North



(Placer metallurgical samples: green; acid rock drainage samples: red; Crystallex 2003 metallurgical samples: dark blue; Crystallex 2004 metallurgical samples: cyan; Crystallex 2005 metallurgical samples: medium blue; Crystallex 2006 metallurgical samples: purple-blue)

Figure 16.3 Cross Section Showing Metallurgical Samples – Looking West



(Placer metallurgical samples: green; acid rock drainage samples: red; Crystallex 2003 metallurgical samples: dark blue; Crystallex 2004 metallurgical samples: cyan; Crystallex 2005 metallurgical samples: medium blue; Crystallex 2006 metallurgical samples: purple-blue)

The flotation flowsheet was demonstrated in several pilot plant runs operated at solids flowrates of up to 150 kg/h. As Placer continued its metallurgical development work, the flowsheet became more complex. In particular, an acidification-volatilization-recovery ("AVR") plant was added to the flowsheet because cyanide consumption in an earlier version of the flotation flowsheet appeared to be excessive. The AVR process was bench tested and was included in Placer's 1996 Feasibility Study (Placer Dome Technical Services Ltd., 1996c), although by February 1998 Placer had achieved process improvements and decided that the $23 million capital cost of the AVR plant was not justified (Placer Dome Technical Services Ltd., 1998c).

Most of the Placer program of studies and testwork was done internally but some work was performed by contract groups. For example, Laplante at McGill University examined gravity concentration (Laplante, 2003). Pocock Industrial studied thickening and filtration of flotation products (Pocock Industrial Inc., 1995). MacPherson examined grinding (A.R. MacPherson Consultants Ltd., 1994), and other groups studied mineralogy. The Placer metallurgical work is summarized in the MDA study of 2003 (Mine Development Associates and Kappes, Cassiday and Associates, 2003) and is not summarized in total herein.

In early 2003, Crystallex, SNC-Lavalin, and Goode reviewed available metallurgical test data and performed various trade-off studies. These analyses indicated that the production, transportation off-shore, and smelting of a copper-gold flotation concentrate, as proposed by Placer, was a less attractive alternative and that direct leaching of most or all of the ore and on-site production of bullion would give better gold recovery. The trade-off studies also showed that the direct leach process, which is the flowsheet originally selected by Placer, would simplify the process, improve plant operability, and give lower capital and operating costs.

Crystallex organized new samples to be shipped from Venezuela and arrangements were made with SGS Lakefield Research Limited ("Lakefield") to test the direct leach process. The program ran from the time that samples arrived at Lakefield in early April 2003 until mid-2004. The work was supervised by Goode, Crystallex, and SNC-Lavalin, and Goode visited Lakefield on several occasions to observe tests and discuss and monitor the progress of the program of work.

16.2 Summary

Several samples of SAPO, SAPS, carbonate CLB and CSB ore from within the limits of the planned the Conductora pit were examined in bench tests and pilot plant operations by Lakefield during the months of April through December 2003 (SGS Lakefield Research Limited, 2003a-c, 2004a-b). Samples of waste from the Conductora pit and four samples of Mesones ore were also studied. Sub-samples of Conductora ore were sent to McGill University for gravity recovery testwork (Laplante, 2003). Outokumpu conducted pilot plant settling tests on several samples (Outokumpu Mintec Canada Ltd., 2003a-b). The various test programs were designed to confirm relevant data generated by Placer, determine the gold recovery and reagent requirements for the proposed gravity-leach flowsheet, and generate plant design data.

Grinding data are generally in accordance with data generated by Placer. Pilot-scale gravity concentration tests at Lakefield on Conductora ore show about 30% gold recovery from both a SAPO-CSB blend and a SAPO-SAPS-CLB-CSB blend at mass concentration ratios of about 4000:1 (SGS Lakefield Research Limited, 2003a). Preliminary data for Mesones (SGS Lakefield Research Limited, 2004a) show an even better response. Intensive cyanidation of the concentrates from Conductora gave >99% leach recovery. Tests at McGill (Laplante, 2003) to determine the gravity recoverable gold ("GRG") content of Conductora SAPO and CSB samples showed 39% and 46% GRG, respectively, which would translate into practical recoveries of about 25%.

Thirty-six hour bottle-roll leach tests on Conductora gravity tailings confirm that SAPO leaches very well to give about 99% overall (gravity+leaching) extraction and a 0.02 g Au/t tailing. With a 24h leach time, tailings were 0.03 g Au/t corresponding to 98% extraction. CSB gives about 85% overall extraction (0.17 g Au/t tailing). Cyanide additions for SAPO and CSB have been less than 1 kg/t ore. Pure SAPS samples with cyanide soluble copper ("CNSCu") levels of 370 ppm or less have been tested and gave 85 to 88% extraction, albeit with cyanide additions of 1.7 to 1.9 kg/t. Mixtures containing SAPO, SAPS and CSB gave 85 to 90% overall extraction provided that sufficient NaCN was present. The NaCN addition varied with the CNSCu level in the ore.

An initial gravity-leach test on each of the four Mesones samples showed an average 85% overall gold extraction and modest reagent consumption. It is believed that higher extraction could be obtained with optimization of the leach conditions.

Duplicate bench scale tests on a series of samples containing 20%CLB and 80% CSB and between 1 and 2 g Au/t yielded an average of 88.7% overall gold recovery (gravity and leaching) with no measurable dependency on head grade.

A 2 kg/h pilot plant was operated for three weeks in which batch-ground/gravity concentrated Conductora ore was subjected to carbon-in-leach ("CIL") processing. During the first 13 days (PP1), a blend of 20% SAPO and 80% CSB was leached with 0.7 kg/t of cyanide to give a final overall gold extraction of 89.6% (tailings average of 0.15 g Au/t). A SAPO-SAPS-CLB-CSB blend was processed for the last week (PP2). The plant tailing was 0.15 g Au/t for an extraction of 89.3% with a cyanide addition of 0.8 kg/t.

Viscosity measurements by Lakefield (SGS Lakefield Research Limited, 2003b) indicated nothing problematical in the mixtures that will be handled in the Las Cristinas plant.

Outokumpu conducted high-rate thickening tests on nine sample blends, ranging from pure SAPO to pure bedrock, using its pilot-scale thickener (Outokumpu Mintec Canada Ltd., 2003a-b). At 50% solids in the underflow, all blends containing 50% SAPO or less could be processed at 0.46 t/m^2/h or greater. Allowing for a 15% scale-up, the data showed that a 50m diameter thickener would give at least 47% solids in the underflow when processing up to 20,000 t/d of a 50% SAPO, 50% CSB mixture. Acid-base-accounting ("ABA") tests and various geotechnical studies were performed by Lakefield on several samples to determine the potential for acid generation.

Natural-degradation tests and continuous INCO Air/SO_2 cyanide-destruction tests have been performed on pilot plant tailings (SGS Lakefield Research Limited, 2004a). Natural degradation under Lakefield climatic conditions reduced weak-acid dissociable cyanide ("CNWAD") to below 20 ppm in about 40 d for pilot plant tailings from PP1 and 100 d for PP2 tailings. The INCO process then reduced CNWAD to <0.3 ppm and Cu to about 1 ppm under industry-typical operating conditions. INCO tests on naturally degraded PP2 tailings solution gave <0.1 ppm CNWAD and <0.5 ppm Cu.

16.3 Samples

Composite samples of the different rock types from the Las Cristinas deposit were prepared from drill core stored at the mine site in Venezuela under the direction of Dr. Luca Riccio, Crystallex's prior Vice President of Exploration. Each sample was composited from individual drill-core intervals, each with a mass of between 0.5 and about 7kg and probably averaging about 2kg across all samples. The location of the samples within the orebody is presented in Figure 16.1, Figure 16.2, and Figure 16.3. The samples summarized in Table 16.1 were shipped to Lakefield.

Table 16.1 Summary of Main Sample Shipments

Sample	Mass kg	Mine est. Au – g/t	Major Assays – Lakefield				
			Au – g/t		Ag – g/t	Cu – %	CNSCu – %
			Assay (2 assays)	Calculated (testwork)			
SAPO 1	313	1.59	1.63	1.47	1.2	0.038	0.004
SAPO 2	101	1.38	Not used – no data				
SAPS1	39	1.55	1.32	1.55	2.2	0.14	0.018
SAPS2	31	2.29	2.16	2.20	1.9	0.15	0.033
SAPS(2)	36	1.29	1.33	-	1.4	0.11	0.037
SAPS3	31	1.64	2.15	-	6.1	0.21	0.12
SAPS4	32	1.53	1.84	-	1.7	0.43	0.31
CSB1	1001	1.38	1.28	1.24	0.9	0.15	0.006
CLB-CSB	1002	1.31	Not composited – see text				

As well as the foregoing, four waste samples from the Conductora deposit were received by Lakefield (the M samples) and used for ABA testwork (SGS Lakefield Research Limited, 2004a). Two samples of Mesones CSB (samples E2 and E4) and two samples (E1 and E3) of a mixture of CSB and bedrock (CLB) were also received and tested. The gold, silver, copper and cyanide soluble copper assays for the CSB were about 1.4 g Au/t, 2 g Ag/t, 0.35%, and 0.016%, respectively. The equivalent data for the Mesones CSB-CLB mixture are 1.2 g Au/t, 2.3 g Ag/t, 0.6%, and 0.035%.

The CLB-CSB sample comprised bags C1 to C21. The CLB was confined to five bags in the shipment and these were used to make high-CLB composites to allow investigation of this material. Lakefield also prepared a series of six composites containing a nominal 20% CLB and covering gold grades from about 1 to 2 g Au/t using material from the individual bags of the CLB-CSB shipment. These composites were used to determine the head grade – gold recovery effect in a series of tests performed in December 2003.

Graphitic carbon assays were obtained as the difference between CTotal and CO_2 on all samples and were found to be in the range of 0.01 to 0.08%. Preg-robbing tests were done on the earlier samples and samples CSB, SAPS2, and SAP(2) were found to be mildly preg-robbing with 11, 9, and 16% of a 10 ppm spike adsorbed after 24h. SAPO and the other SAPS samples returned values of 4% or less. Mercury assays in the various samples were either 0.3 g/t or <0.3 g/t except for SAPS1 which was reported as 0.4 g/t.

The as-received screen analyses of SAPO, SAPS2 and SAPS3 were 63µm, 182µm, and 69µm, respectively. It is presumed that this is the in-situ screen analysis for these materials. All other samples were provided as fragments of drill core.

Individual samples within the first CSB sample were used to make eight depth samples before the CSB composite was formed. The analytical data for the depth samples are presented in the graph presented as Figure 16.4. Some fluctuations are evident although there may be little statistical significance to the observed effects. CSB samples from 151m, 259m, and 437m were leached to see if there was a significant depth effect. Various composites have been produced for metallurgical testwork as presented in Table 16.2.

Figure 16.4 Variation of Head Assay with Depth in CSB

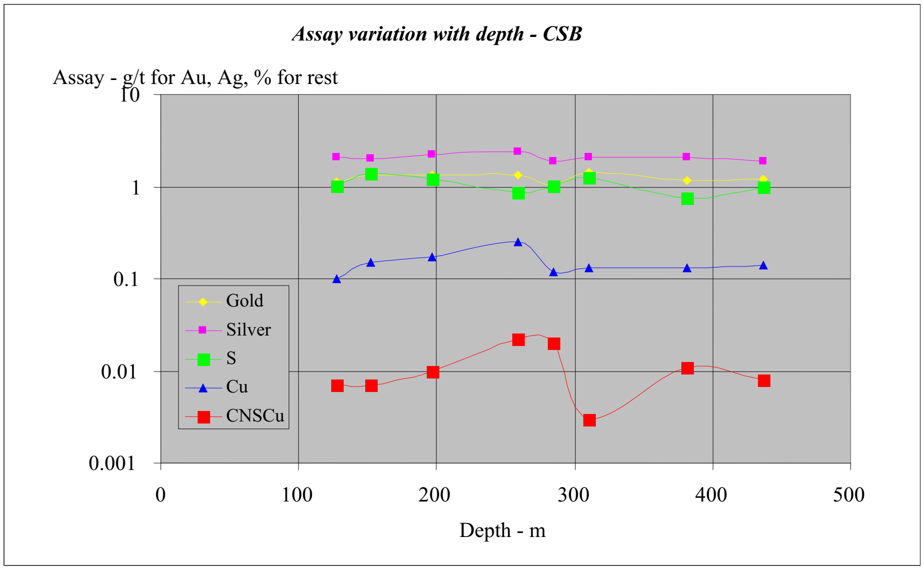


Table 16.2 Composites Used in Testwork

Composite name	Composition
SAPO-CSB	20% SAPO1, 80% CSB
Comp S1	10% SAPO1, 10% SAPS2, 80% CSB1 (target 85 ppm CNSCu)
Comp S2	10% SAPO1, 20% SAPS2, 70% CSB1 (target 112 ppm CNSCu)
Comp S3	10% SAPO1, 10% SAPS3, 80% CSB1 (target 172 ppm CNSCu)
Comp S4	10% SAPO1, 20% SAPS3, 70% CSB1 (target 286 ppm CNSCu)
Comp S5	10% SAPO1, 10% SAPS4, 80% CSB1 (target 362 ppm CNSCu)
SAPS350	40% SAPS1, 50% SAPS2, 10% SAPS3 (target 350ppm CNSCu)
CLB-CSB Comp. 1	68% CLB, 32% CSB (selected from CLB-CSB sample)
CSB2	100% CSB (balance of CLB-CSB sample)
CLB-CSB Comp. 2	40% CLB-CSB Comp.1, 60% CSB1 (to give 27% CLB, 73% CSB)
Mine blend	15% SAPO1, 5% SAPS350, 15% CLB-CSB Comp1, 65% CSB1 Equals 15% SAPO, 5% SAPS(350ppm CNSCu), 10% CLB, 70% CSB
CLB-CSB G1	Part bags C2, 3, 7, 12, 17, and 20 from CLB-CSB shipment, estimated 20% CLB
CLB-CSB G2	Part bags C2, 3, 7, 10, 12, 17, 20, and 21 from CLB-CSB shipment, estimated 20% CLB
CLB-CSB G3	Part bags C3, 4, 6, 10, 11, 17, 20, and 21 from CLB-CSB shipment, estimated 20% CLB
CLB-CSB G4	Part bags C1, 4, 6, 8, 9, 10, 11, 13, 14, 17, 18, 19, and 21 from CLB-CSB shipment, estimated 20% CLB
CLB-CSB G5	Part bags C1, 5, 8, 9, 13, 14, 15, 16, 18, and 19 from CLB-CSB shipment, estimated 20% CLB
CLB-CSB G6	Part bags C5, 15, 16, and 19 from CLB-CSB shipment, estimated 20% CLB

Full compositing and analytical data are provided in the Lakefield documents (see SGS Lakefield references in Reference Section 22.0).

16.4 Grinding Tests

Standard Bond rod mill (14 mesh screen) and ball mill work indices (150 mesh screen) and abrasion index data were obtained on selected samples and sample composites. Metric data are presented in Table 16.3.

Table 16.3 Grinding Parameters from Standard Tests

Sample	Rod mill index	Ball mill index	Abrasion index – g
CSB1	17.1	15.0	0.27
80%CSB – 20% SAPO	-	14.2	-
Mine Blend	15.9	14.4	0.24
CLB-CSB Comp.2	-	14.7	-

The data are similar to data obtained by MacPherson (A.R. MacPherson Consultants Ltd., 1994) on samples provided by Placer.

Bond work indices ("BWi") were also estimated from grinding data obtained in a small mill used to prepare feed for leach tests. These data, which are indicative of Bond work indices, are not as reliable as full Bond indices, are presented in Table 16.4.

Table 16.4 Selected Bond Ball Mill Work Indices from Leach Grinds

Material	Identity	Comparative BWi (metric)
Comp S1	SAPO-SAPS-CSB blend	13.2
Comp S2	SAPO-SAPS-CSB blend	12.6
Comp S3	SAPO-SAPS-CSB blend	13.6
Comp S4	SAPO-SAPS-CSB blend	12.3
Comp S5	SAPO-SAPS-CSB blend	13.7
Average of Comp S samples	-	**13.1**
Comp 2	CSB depth sample – 151 m	14.7
Comp 4	CSB depth sample – 259 m	19.0, 17.4*
Comp 8	CSB depth sample – 437 m	16.1, 15.5*
Average of CSB depth samples	**-**	**16.5**

* Second BWi data obtained from pebble mill and probably less accurate.

The Bond ball mill work index data (metric) obtained by MacPherson (A.R. MacPherson Consultants Ltd., 1994) for Placer were 15.3 for CSB and 10.5 for CLB. These data, which are similar to the values tabulated above, have been used by SNC-Lavalin in the design criteria. MacPherson also reported on semi-autogenous grind ("SAG") work indices and rod mill work indices.

SAPO and SAPS were not subject to Bond work index tests in the Lakefield work or by MacPherson because the material was too fine to test. However, an apparent work index for saprolite can be calculated from the work index measurements for blends containing this material. The formal Bond test noted in Table 16.3 suggests a work index of 11 but this value is suspect. The saprolite work index calculated from the data of Table 16.4 ranges from 6 to 8.5.

The abrasion index measurements obtained by Lakefield (SGS Lakefield Research Limited, 2003a) are a little higher than those obtained by MacPherson (A.R. MacPherson Consultants Ltd., 1994) who obtained 0.06 and 0.10 g for CLB and 0.15 to 0.23 g for CSB.

16.5 Gravity Recovery of Gold

The feed for bottle-roll leach tests was prepared by grinding 2kg batches of ore to the desired grind then removing coarse gold using a 3in. Knelson concentrator with upgrading of the concentrate on a Mozley table. The leach feed was then made by mixing all of the Knelson tailings and the Mozley tailings. Average data for the different ore types from twenty small-scale gravity recovery tests are provided in Table 16.5.

Table 16.5 Average Data from Gravity Tests Ahead of Bottle Roll Leach Tests

Sample	Grind	Gravity Concentrate			Tail	Head, g/t Au	
	K_{80}, µm	Wt %	Au, g/t	% Rec'y	Au, g/t	calc.	direct
SAPO	35	0.031	252	5.3	1.35	1.47	1.63
SAPS	50	0.06	900	18.4	1.39	1.71	-
COMP S	63	0.097	356	22.9	1.14	1.48	1.43
SAPO/CSB1 20/80	77	0.082	278	15.7	1.00	1.19	1.38
CSB	67	0.091	328	22.5	0.96	1.24	1.24
CSB depth	94	0.086	254	17.2	1.03	1.25	1.29
CLB/CSB2	99	0.026	1198	22.2	1.07	1.38	1.46
CLB-CSB G1 to G6	54	0.078	436	23.8	1.06	1.38	1.12

There was no discernable relationship between the head grade and percentage gravity recovery of gold in the six CLB-CSB samples.

The four samples of Mesones CSB and CLB-CSB mixtures were also processed by gravity concentration and responded well. From an average feed grade of 1.1 g Au/t, 37% of the gold was recovered to a 711 g Au/t concentrate.

A pilot plant was operated to process about 1 tonne of Las Cristinas material over a 20d period. The first part of the pilot plant run used a feed comprising 20% SAPO and 80% CSB. The second part of the pilot plant run used a feed comprising 15% SAPO, 5% SAPS, 10% CLB, and 70% CSB. The feed for the pilot plant was prepared in 30kg batches which were processed by the same Knelson-Mozley flowsheet as described above. Gravity recovery data from the pilot plant are provided below in Table 16.6.

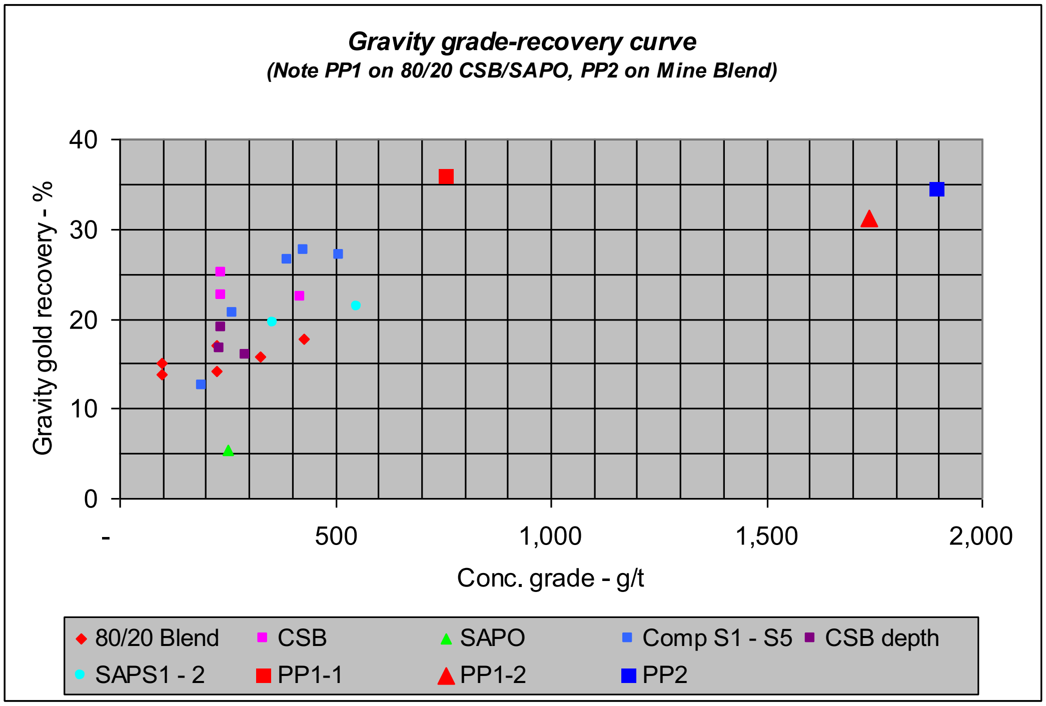
Table 16.6 Gravity Concentration Data from Pilot Plant Feed Preparation Work

Feed	Phase	Gold assays – g/t			% recovery to conc.	
		Head	Tail	Conc.	Mass	Gold
SAPO-CSB	PP1-1	1.50	0.95	775	0.071	36
SAPO-CSB	PP1-2	1.39	0.95	1760	0.025	32
SAPO-CSB	PP2	1.38	0.90	1920	0.025	34

In the first part of the pilot plant (Phase PP1-1), the mass of Mozley concentrate was set at 15 to 25g per 30kg batch grind or about 0.07% mass pull. In the later operation (PP1-2 and PP2), the mass pull was reduced to 5 to 10g of concentrate or about 0.025% mass. The tabulated concentrate assays are based on the assay head and the gravity tail assay estimated from the cyanidation data.

Gravity recovery in the pilot plant was far higher than in the small-scale tests as indicated in Figure 16.5. This is as expected and reinforces the importance of processing large samples to determine gravity recovery potential.

Figure 16.5 Gravity Recovery Data



Samples of CSB and SAPO were processed by Professor André Laplante at McGill University (Laplante, 2003). About 40kg of SAPO containing 1.34 g Au/t and 100kg of CSB containing 1.5 g/t were dispatched and used in the McGill work. Using the standard Laplante GRG-test protocol, it was established that SAPO contained 39% GRG while CSB contained 46%. It was noted that about 10% of the total gold in each sample was –20µm in size. Based on an analysis of the data, Laplante concluded that about 25% gold recovery would be obtained by gravity processing. Using its circuit modeling system and the same data, Knelson projected 18 to 20% gold recovery from SAPO and 24 to 27% recovery from CSB. SNC-Lavalin has used a conservative 20% gravity gold recovery from blended ore.

Samples of the concentrates produced during the three different segments of the gravity recovery portion of the pilot plant were leached under intensive conditions to simulate processing in an Acacia or Gekko concentrate leach system. The intensive leach procedure used 2% NaCN solution, H_2O_2 as an oxidant, and a leach time of 48h. Results are summarized in Table 16.7.

Table 16.7 Intensive Cyanidation of Gravity Concentrate

Feed	**Pilot run**	**NaCN**		**Data**	**Extraction**					**Tail**	**Calc head**
		Add	**Cons**		**2 h**	**6 h**	**12 h**	**24 h**	**48 h**	**g/t**	**g/t**
SAPO-CSB	PP1-1	233	74	Au	90	84	90	95	98.6	6.6	484
				Ag	95	90	96	96	95.7	2.3	54
	PP1-2	240	78	Au	91	93	101	99	99.5	6.3	1,378
				Ag	102	95	103	100	98.3	2.3	138
Mine blend	PP2	260	100	Au	101	96	108	99	99.3	8.5	1,246
				Ag	104	95	101	98	98.3	2.3	136

Available data indicate that gravity recovery should be very effective at Las Cristinas and gravity gold recovery should comfortably exceed 20% of the gold in the feed. The concentrates are very amenable to intensive leaching.

16.6 Cyanide Leaching

16.6.1 Bottle roll tests

All bottle-roll tests were preceded by the removal of coarse gold in a gravity concentration step. The gravity concentration effect is typified by the data presented earlier in Table 16.5. The results discussed in this section are overall gold recovery, i.e., gravity recovery plus leach extraction.

An initial series of 9 CIL tests investigated the effects of grind (P80 of 110, 75, and 50µm) and time (12, 24, 48h) on leaching of the SAPO-CSB blend with cyanide strength of 0.5 g/L. This work showed very little difference in overall recovery at 50 and 75 µm at the longer leach times, and a grind of 75 µm and CIL time of 36 h was selected for most additional leach tests.

A second series of tests looked at cyanide addition strategy and showed that an initial 0.5 g/L held for 4h gave reasonable gold extraction (87%), low tailings gold grade (0.15 g Au/t) and lower cyanide addition (0.9 kg/t).

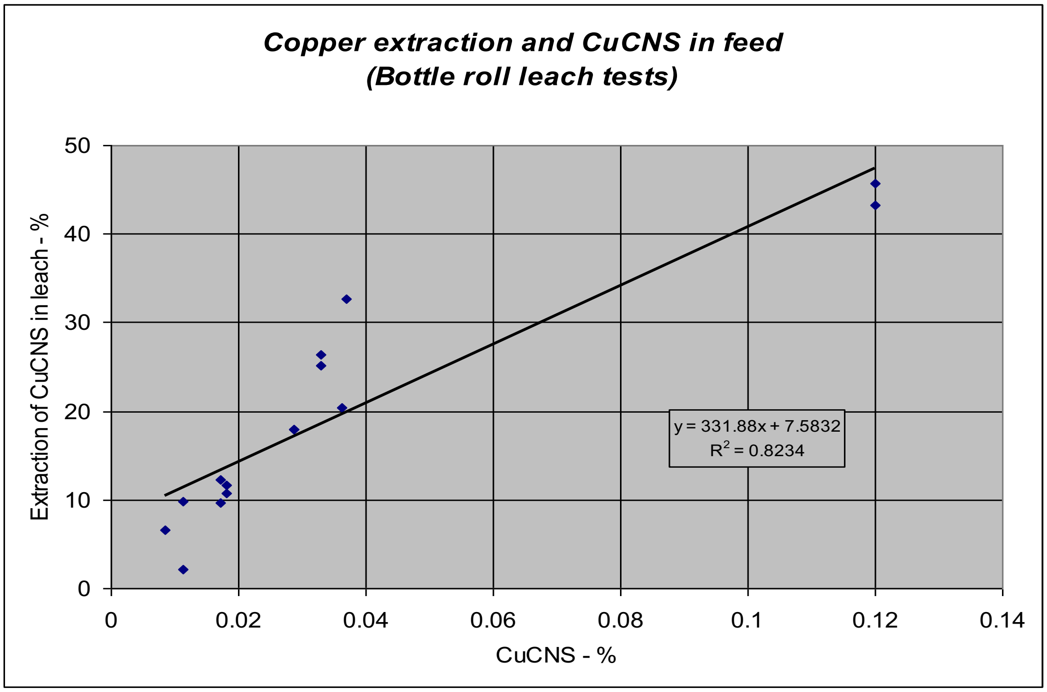
Tests on pure SAPO showed that 99% extraction (tailings of 0.02 g Au/t) was possible after 36h of CIL with 0.9 kg/t NaCN addition. Other tests showed that overall extraction from SAPO was 98% (0.03 g Au/t tailings) across a range of leach times between 24 and 36h. A 36h leach of CSB gave 85%

recovery (0.17 g Au/t tails) following 0.8 kg/t NaCN addition. Leach tests on SAPO-CSB and SAPO-SAPS-CSB blends containing 75% saprolitic material (tests number CN68 to 76) showed very little sensitivity of the overall gold recovery to leach time in the 24 to 36h range. Lime and cyanide additions were also relatively unaffected across this time range.

Copper leaching from the SAPO, CSB, and blends was generally less than 5% from heads of about 0.05% for SAPO and 0.15% Cu for CSB.

Leaching of samples of pure SAPS, which contain CNSCu, gave higher tailings grade and higher cyanide consumption. SAPS1 (180 ppm CNSCu) gave 88 and 91% overall extraction (tails of 0.18 and 0.12 g Au/t) after 36h with the addition of 2 and 1.2 kg/t of NaCN, respectively. SAPS2 (330 ppm CNSCu) gave 85 and 89% extraction (tails of 0.34 and 0.26 g Au/t) following the addition of 1.9 and 1.4 kg/t of NaCN, respectively. SAP(2), containing 370 ppm CNSCu, gave 94% overall extraction (tails of 0.07 g Au/t) with the addition of 1.5 kg/t of NaCN. A sample of SAPS3 (1200 ppm CNSCu) gave 88% overall extraction (0.18 g Au/t tails) but required a cyanide addition of 2.45 kg/t. CNSCu extraction from the SAPS-bearing material was in the 2 to 45% range as is illustrated in Figure 16.6.

Figure 16.6 Copper Leached From SAPS-Bearing Ore



Mixtures of SAPO, CSB, and different amounts of various SAPS composites were combined to form Comp S1 to S5 with total CNSCu values between 85 and 362 ppm. Initial tests used an NaCN addition of about 1 kg/t. Gold recovery was 84% for the low CNSCu composite (tails 0.22 g Au/t) with recoveries of about 76% (tails of 0.37 g Au/t) for the high CNSCu sample. Other tests on the higher Cu composites using higher NaCN additions (1.2 to 2.1 kg/t) gave recoveries of about 87 to 90% (tails of about 0.15 g Au/t).

Reagent additions during the bottle-roll leach tests are summarized in graphs presented in Figure 16.7 and Figure 16.8. There is an obvious relationship between the CNSCu level of the ore and the amount of cyanide consumed during the leach process. Lime consumption for SAPO and SAPS is higher than for the bedrock material as indicated in Figure 16.8.

Figure 16.7 Cyanide Consumed in Bottle Roll Tests

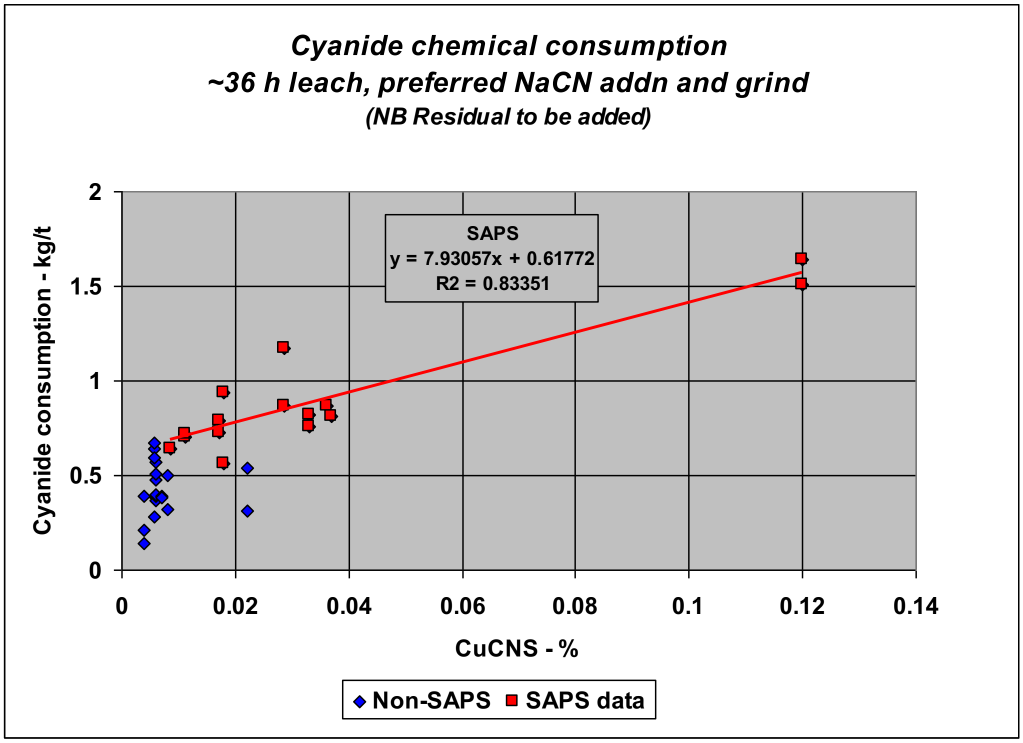
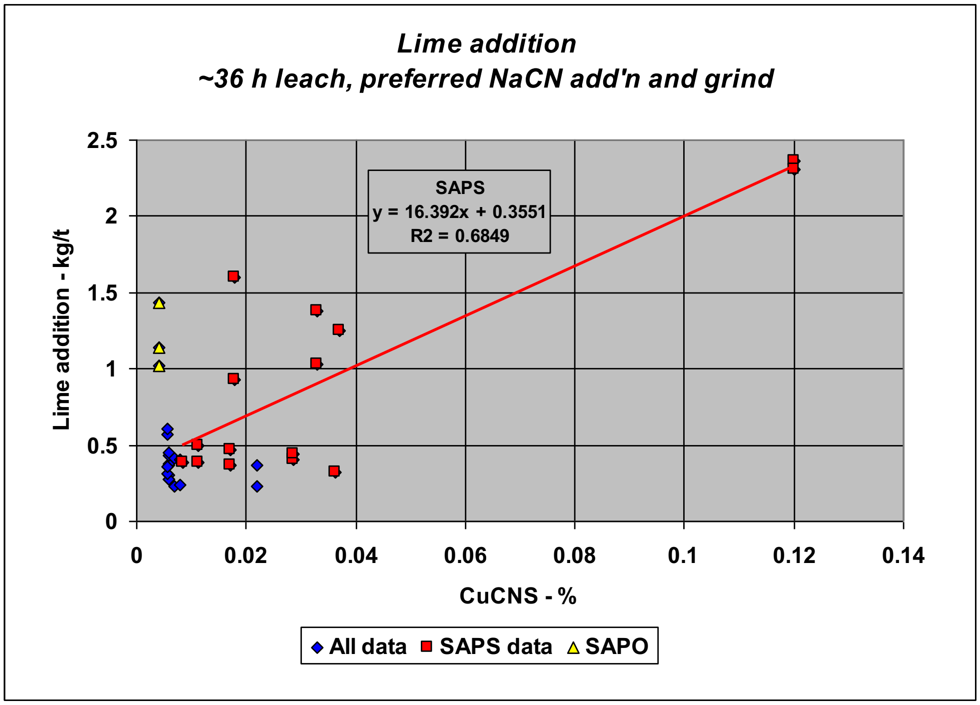


Figure 16.8 Lime Addition in Bottle Roll Tests



The regression equations indicated on the above graphs have been used to develop the operating costs for different ore types and blends. In the case of the relationship between CNSCu level and cyanide consumption, the equation indicated above has been tempered by data obtained by Placer in its work on SAPS-containing material. This has had the effect of increasing the cyanide consumption over the levels indicated above.

Lakefield also performed four leach tests on samples of Mesones gravity tailings at grinds in the 71 to 103µm range (SGS Lakefield Research Limited, 2004a). Overall gold extraction (gravity and cyanidation) varied from 84% to 88%. The average calculated head grade was 1.09 g Au/t and the average tailings grade was 0.16 g Au/t for an average overall extraction of 85%. Cyanide additions were in the 0.9 to 1.6kg/t range (average of 0.77kg/t) and obviously related to the high CNSCu content of the samples. Lime addition was modest at an average of 0.4kg/t. It is likely that, with optimization of reagent addition strategies, the gold recovery from Mesones samples could be improved.

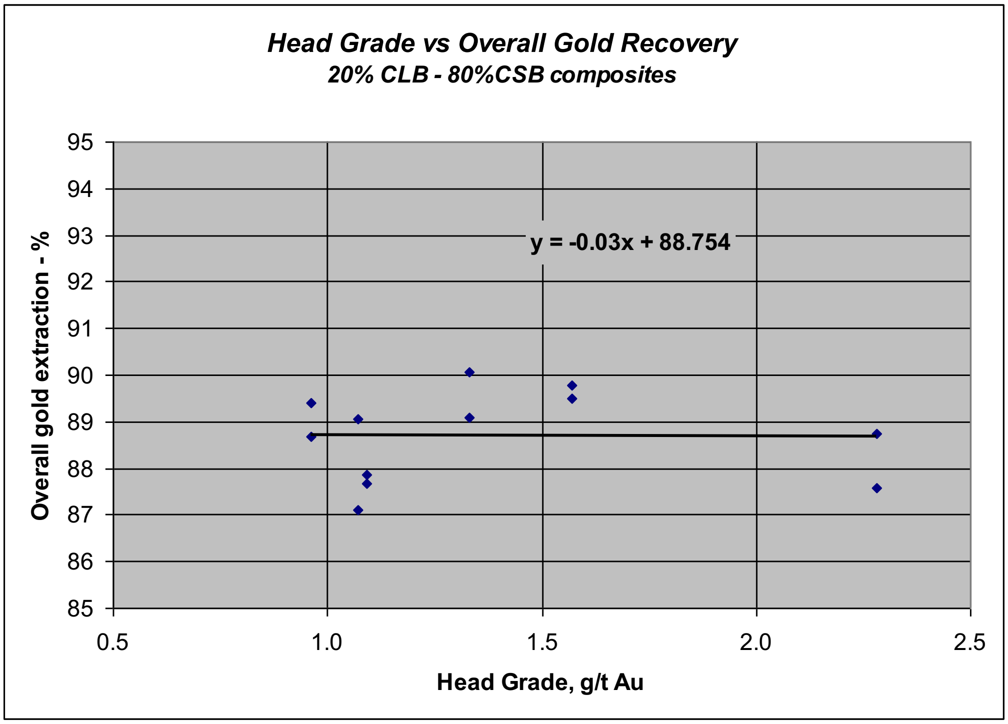
A series of gravity recovery – bottle-roll leach tests was performed on CLB–CSB mixtures composited to contain 20% CLB, 80% CSB and gold grades between a nominal 1 g Au/t and 2 g Au/t. In each case, 3kg of composite was prepared, mixed, ground to a nominal 70 µm, subjected to gravity recovery in the 3in. Knelson with the concentrate upgraded on the Mozley table and tailings combined. The gravity tailings were then leached for 36 h in duplicate under the standard leach conditions.

The actual grind was finer than intended with a range of 53 to 55 µm and averaged 54 µm. As mentioned earlier, testwork showed that there is little sensitivity to grind in the 50 to 70µm range with a 36h leach time so the data are valid despite the finer grind.

The head grade calculated from the gravity-leach tests for the six samples ranged from 0.96 to 2.28 g Au/t and averaged 1.38 g Au/t. The expected head grade, based on the assays of the individual components of each sub-sample, ranged from 1.0 to 1.9 g Au/t and averaged 1.34 g Au/t. The direct head assays for the samples ranged from 0.96 to 1.56 g Au/t and averaged 1.1 g Au/t.

Overall recovery (gravity plus leaching) for the six duplicate tests was 88.7% with a range of from 87.8 to 89.6% for the averages of the pairs. The range in the individual tests was from 87.1 to 90.1%. There is no discernable (or statistical) relationship between the overall gold recovery and calculated head grade. The data are plotted in Figure 16.9.

Figure 16.9 Grade - recovery relationship



Lakefield made a fixed cyanide addition of 0.91 kg/t in the grade-recovery tests (SGS Lakefield Research Limited, 2003c). The actual consumption averaged 0.51 kg/t with a slight trend to higher consumption (0.6 kg/t) with the 2 g Au/t head grade material.

Lime additions in the subject series of tests averaged 0.62 kg/t with no relationship between head grade and reagent addition.

The twelve gravity-leach tests in the grade-recovery series on the CLB-CSB mixtures show that there is very little, if any, change in overall gold recovery across the range of 1 to 2 g Au/t. Other parameters, including gravity gold recovery, lime and cyanide consumption, were not significantly affected across the range of samples examined.

16.7 Pilot plant

16.7.1 Pilot plant configuration

The CIL pilot-plant operation included the batch ball milling of 30kg aliquots of feed material, removal of a gravity concentrate using a 3in. Knelson concentrator, and upgrading of the composite on a Mozley table with table tails combined with Knelson tails. Initially the table was operated to give a 0.07% mass pull, but this was changed early in the operation to a 0.025% mass pull.

Gravity tailings were transferred to a holding tank ahead of the CIL pilot plant where the density was adjusted and trash removed on a 28-mesh screen. Feed slurry was then pumped at a rate corresponding to 1.9 kg/h of solids to the first of six CIL tanks providing a total of 36h residence time. Lime was added to adjust the pH to the desired level and, for the SAPO-CSB blend, a total of 0.7 kg/t of NaCN was added as a solution – 67% to the first tank with the balance to the second tank. During processing of the Mine Blend (comprising 15% SAPO, 5% SAPS, 10% CLB, and 70% CSB), which contains CNSCu-containing SAPS, the NaCN addition was increased to 0.8 kg/t.

Each CIL tank contained 4 g/L of activated carbon during the initial operation. This was changed to 8 g/L part way through PP1, which processed a SAPO-CSB blend because it was initially not clear that it was sufficient. Carbon was retained in each tank with a 20-mesh screen located on the tank outlet and was manually advanced every 12h. Based on modeling studies, a carbon loading of 1500 g Au/t was selected in the design of the pilot plant operation. The carbon loaded into the pilot plant was pre-loaded to ensure rapid attainment of equilibrium.

A 28-mesh safety screen was fitted to the final CIL tank discharge late in the first pilot plant campaign.

A feed sample was taken from each batch of feed to the Knelson concentrator and every 8h from the feed to the CIL plant. Tailings were sampled every hour, filtered, and combined to form 4h composites. A full profile through the CIL circuit (solids, solution, carbon) was taken every day, and screen analyses were periodically checked.

16.7.2 Gravity concentration data

As noted in Table 16.6, after adjustment of the Knelson procedure to give a high concentration ratio, the pilot plant achieved better than 30% gold recovery to a concentrate assaying more than 1700 g Au/t.

16.7.3 CIL pilot plant data

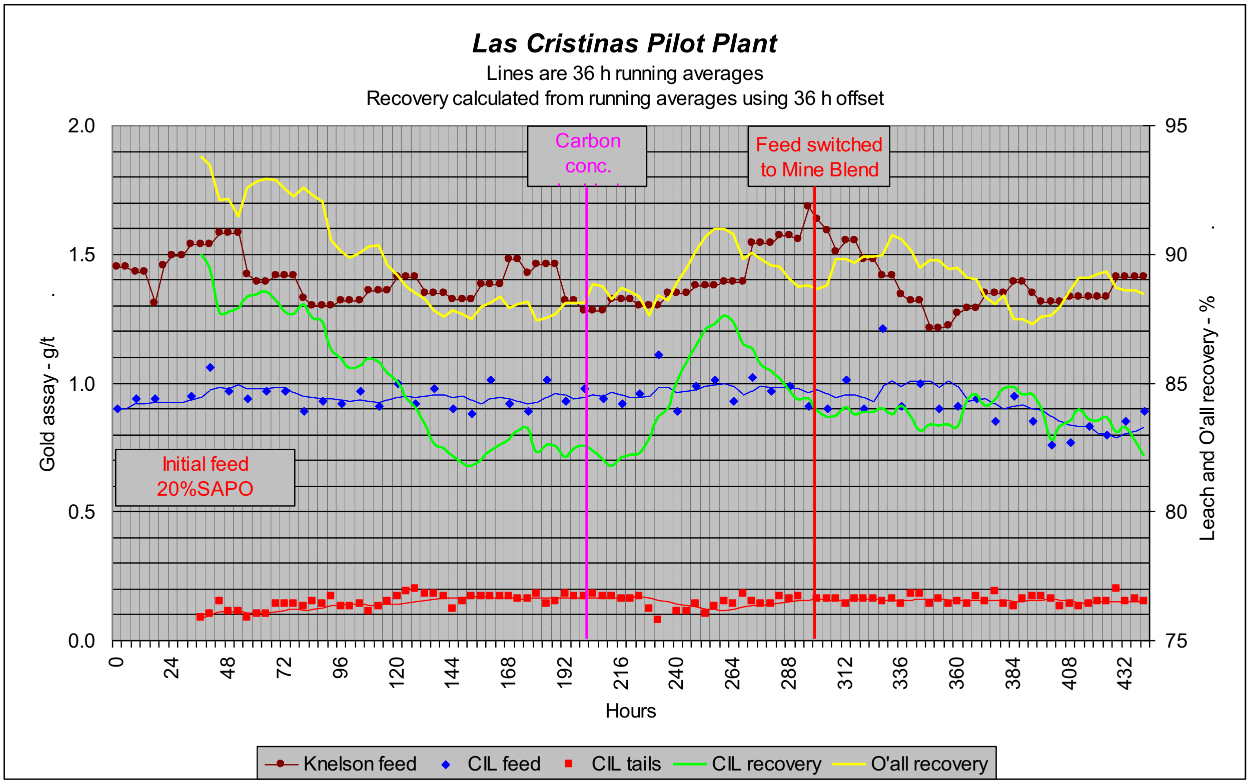
Lakefield data (SGS Lakefield Research Limited, 2003a) show that the average tailing assay for gold when the plant was at equilibrium was 0.15 g Au/t during processing of the SAPO-CSB blend and the Mine Blend. Corresponding overall gold extraction levels are about 89%.

The cyanide addition during the pilot plant operation was set at 0.7 kg/t for the SAPO-CSB blend and 0.8 kg/t for the Mine Blend. The cyanide consumption was 0.3 kg/t during the last four days of PP1B and 0.34 kg/t during the last four days of PP2. Residual cyanide concentration must be added to the chemical consumption to arrive at expected total cyanide addition. Lime addition was 0.78 kg/t and 0.85 kg/t, respectively. Results are summarized in Table 16.8 and in the graph presented as Figure 16.10.

Table 16.8 Summary of Pilot Plant Data

Parameter	PP1A (day 7-9)	PP1B (day 10-14)	PP1Carbon (day 7-14)	PP2 (day 16-20)
Feed	20/80 Blend	20/80 Blend	20/80 Blend	Mine Blend
Carbon Concentration, g/L	4	8	-	8
NaCN Addition, kg/t	0.71	0.70	0.70	0.77
NaCN Consumption, kg/t	0.27	0.30	0.28	0.34
CaO Addition, kg/t	0.87	0.78	0.84	0.85
Average CIL Feed Assay, g/t Au	0.95	0.95	0.95	0.90
Average CIL Tail Assay, g/t Au	0.17	0.15	0.16	0.15
% Gravity recovery	36.3	31.4	33.0	34.5
% Extraction in CIL	82.2	84.8	83.6	83.6
% Overall recovery	88.6	89.6	89.0	89.3

Figure 16.10 Summary Pilot Plant Data



Certain issues arose during the pilot-plant operation and are discussed below:

1) Possible contamination of tailings samples with partially loaded active carbon: Carbon was observed in some tailings samples and some high-grade tailings assays were suspicious. To investigate and correct this problem, all tailings samples with a grade of 0.18 g Au/t or higher were screened at 28-mesh and re-assayed. Several high-grade assays were thereby eliminated.

   Selected tailings samples were microscopically investigated and others assayed for graphitic carbon. Three pilot-plant head samples from PP1 and three from PP2 were bottle-roll leached and returned tailings of 0.13 and 0.15 g Au/t, respectively – similar to the pilot plant tails. It transpired that carbon contamination was an occasional problem but it had been eliminated.

   Some tailings samples were re-leached to determine if carbon loss was a problem or if longer leach times would be warranted. The re-leach data suggested that carbon losses were minimal. An additional 48h of leaching (more than double the 36h-leach time of the CIL pilot plant) gave a reduction in tailings assay of between 0.01 and 0.04 g Au/t suggesting that longer leach times would not be justified.

2) Coarse particle hold-up: The transit times for fine particles through the CIL pilot plant were obviously shorter than the transit times for coarse particles. This was directly evidenced by the fact that the P80 of the pilot plant tailings started at 38µm and did not reach 60µm until 100h after the plant operation was started. Coarse particle hold-up was also suggested by the graph of tailings grade against time. The latter showed very low initial tails assays of 0.09 g Au/t followed by an increase to 0.17 g Au/t over 100h which is similar to the final running average of about 0.15 g Au/t for the SAPO-CSB blend.

   Screen analyses of the CIL tank contents showed a coarse P80 of about 71µm compared to a feed of less than 70µm. Additionally, the slurry percentage solids in CIL tanks 3, 4, 5, and 6 (where there are no solution additions) are higher than the tailings according to the daily surveys. From 2003-05-09 to 2003-05-13 inclusive, the percentage solids in the last four CIL tanks was 45.9% but the average tailings percentage solids was 44.4% in the survey suite of samples. According to data from the routine tailings samples, the average was 46.6% solids so the accumulation of solids may not be serious.

In summary, the pilot plant eventually reached equilibrium with respect to particle transit time. The data reported in the summary tables of this report are for periods where equilibrium had been reached.

16.8 Carbon elution

Two samples composited from loaded carbon from the Las Cristinas pilot plant were eluted using the high-pressure Zadra approach. Data are summarized in Table 16.9. The data indicate no problems with eluting gold from carbon.

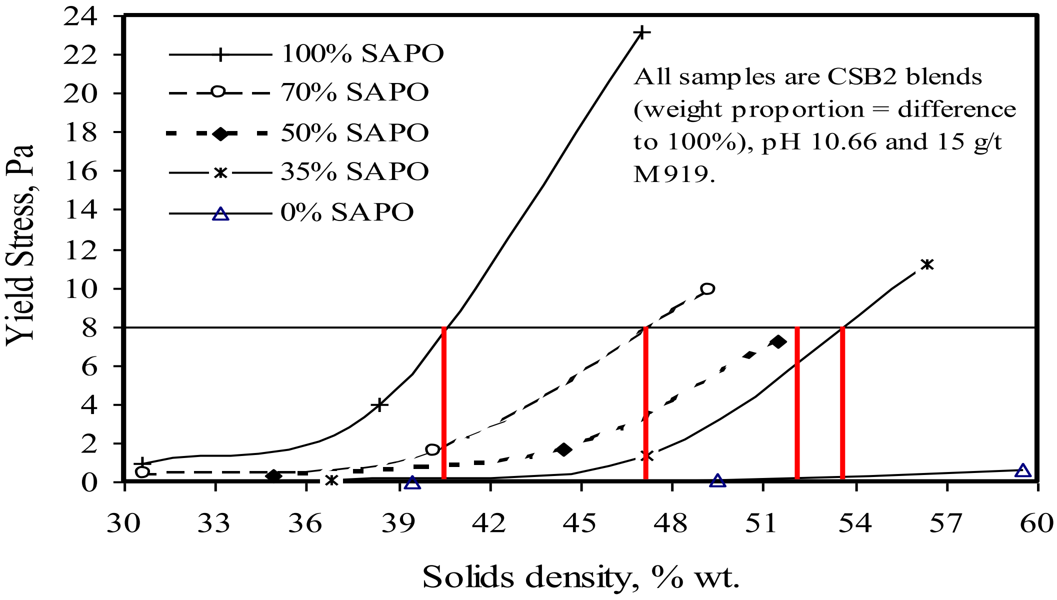
Table 16.9 Carbon Stripping Results

	Unit	Au	Ag	Cu	Au	Ag	Cu
Loaded carbon assay	g/t	1552	185	334	1534	287	555
Acid washed assay	g/t	1598	306	366	1615	319	364
Eluted carbon assay	g/t	32	1.2	<20	38	40	20
Recovery	%	98.0	99.6	94.6	97.6	87.8	96.4

16.9 Viscosity Tests

Lakefield measured viscosity using a Haake rheometer (SGS Lakefield Research Limited, 2003b). The data show that 100% SAPO has a critical solids density ("CSD" – defined as the percentage solids where the Yield Stress ("YS") exceeds 8 Pa) of about 40% solids, while 70% SAPO-30% CSB has a CSD of about 47%. At 50% SAPO, the CSD is about 52% solids. Figure 16.11 indicates the basic viscosity data. Data show that the CSD for SAPS is about 56% solids, which is far higher than the 40% indicated for SAPO. Lakefield concluded that all samples indicated good flowability.

Figure 16.11 Basic Viscosity Data



16.10 Thickening Tests

16.10.1 Flocculant Scoping Tests

Using a limed sample of pilot plant feed (SAPO-CSB slurry at pH 10.7), Lakefield investigated the following flocculants at dosages of ~30 g/t and higher (SGS Lakefield Research Limited, 2003b).

Anionic	Magnafloc 10, Magnafloc 919
Non-ionic	Magnafloc 333
Cationic	Magnafloc 455, 368

Lakefield selected Magnafloc 368 and 919 for further work since it gave preferred clarity and underflow density.

Outokumpu (2003a) also performed flocculant scoping tests but on non-limed SAPO-CSB and concluded that Magnafloc 919 was a superior flocculant. Outokumpu went on to use this material in most of its thickener tests.

In its work for Placer, Pocock (Pocock Industrial Inc., 1995) recommended the use of Percol E10, which is now known as Magnafloc 10, and is a low charge density, anionic, flocculant.

16.10.2 Laboratory Thickening Tests

Lakefield undertook laboratory thickening tests in measuring cylinders without rakes on various Las Cristinas ore types and blends (SGS Lakefield Research Limited, 2003b). The better results for each ore type/blend are tabulated below in Table 16.10.

Table 16.10 Lakefield Static Thickening Tests

Ore /blend	Floc. dose	U/F solids	Unit area	Conventional Thickener diameter for 20,000 t/d - m
	g/t	%	$t/m^2/h$	
CSB2	15	44.9	0.83	45
CLB/CSB	15	45.8	0.83	45
SAPO-CSB (20:80 – PP1 feed)	10	51.7	0.07	156
SAPO-CSB (35:65)	27	39.3	0.23	86
SAPO-CSB (50:50)	18	42.6	0.15	107
SAPO-CSB (70:30)	15	37.4	0.10	128
SAPO	23	37.3	0.22	95
SAPS	33	42.0	1.04	41

Note that thickener diameters use a 1.5 rate scale-up factor

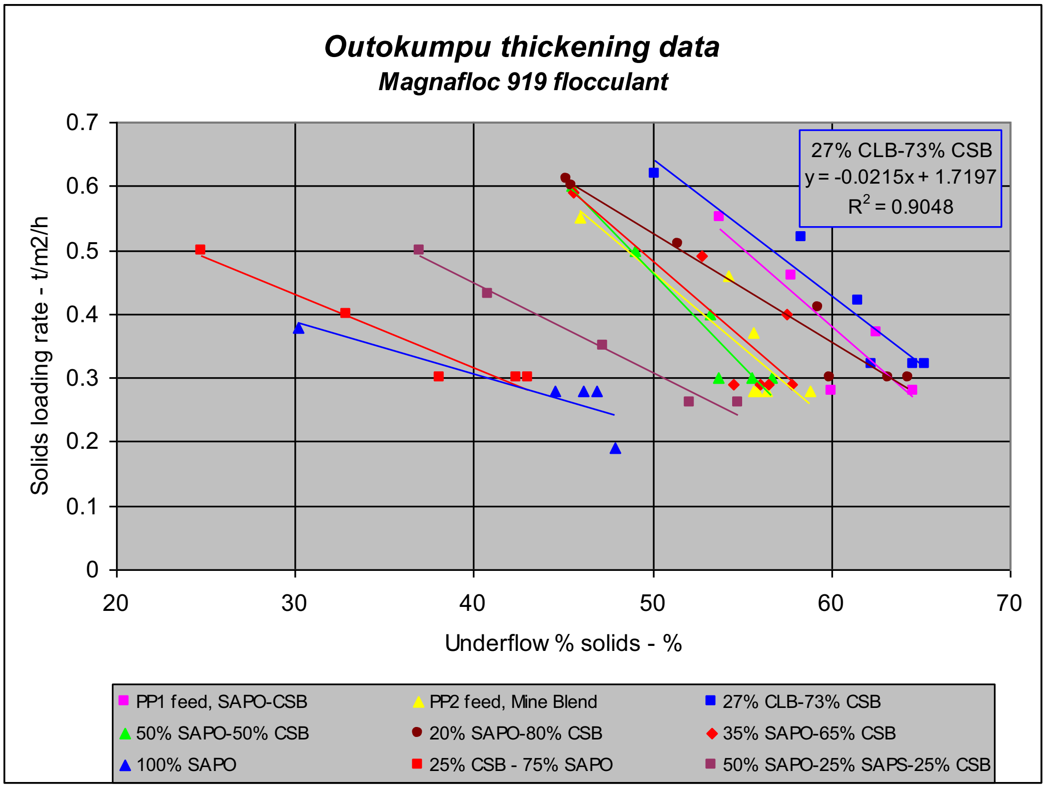
It will be realized that the data in Table 16.10 apply to conventional thickener designs and that settling rates in high-rate thickeners are typically 10 times greater than in conventional thickeners leading to about 1/3 the thickener diameter.

16.10.3 Outokumpu Thickening Tests

Outokumpu operated its continuous $0.1m^2$ pilot-scale thickener at Lakefield in three campaigns (Outokumpu Mintec Canada Ltd., 2003a-b). In all, Outokumpu conducted 58 tests on nine ore blends ranging from pure SAPO through various SAPO-SAPS-CLB-CSB blends, to a simple mixture of CSB and CLB. The Outokumpu pilot unit has been widely used and its scale-up characteristics are well established.

The results of the Outokumpu tests are summarized in Figure 16.12. The data show that, with the correct flocculant, thickener underflow solids concentrations of 50% or greater can be obtained at a loading rate of 0.47 t/m2/h or lower with all of the ore types/mixtures that were tested, provided that the saprolite content of the thickener feed does not exceed 50%. At the design feed rate of 20,000 t/d (900 t/h) and using the Outokumpu scale-up factor of 1.15, the Las Cristinas thickener would need to be 53m in diameter to give 50% solids in the underflow. If a thickener underflow percentage solids of 47% is satisfactory, then a 50m diameter thickener would be sufficient for the 50% saprolite mixture.

Figure 16.12 Outokumpu Thickening Test Data



The Outokumpu (2003a) and Lakefield (SGS Lakefield Research Limited, 2003b) testwork showed that Magnafloc 919, an anionic flocculant, was suitable. The average flocculant dose in all of the tests that were performed was 27 ppm and a dosage of 30 to 40 ppm will probably be needed in the plant. Overflow clarity was generally good and well under 500 ppm of suspended solids.

16.11 Environment-Related Testing

Lakefield completed fifteen modified EPA ABA tests on SAPO, CSB, 20%SAPO:80%CSB blend (PP1 pilot plant feed), pilot plant tailings, SAPS2 (about 330 ppm CNSCu, 0.7% S), 50% SAPS3:50% SAPS4 (about 2100 ppm CNSCu, 1.2%S), samples of Mesones ore, and waste rock from Conductora. The data are reported in two Lakefield reports (SGS Lakefield Research Limited, 2003a and 2004b). Lakefield concluded that Conductora SAPO ore and waste would be classified as non-acid generating and that the SAPS blend with very high cyanide soluble copper and a SAPS waste sample may be acid generating. The acid generating potential of the other samples was deemed uncertain.

Standard settling tests, without rakes, were performed on flocculated but degraded tailings from pilot plant operations with SAPO-CSB blend and Mine Blend. After seven days, settled solids reached 60 to 61% solids. Consolidation tests up to 5 bar were performed in a consolidation (Rowe) cell. Results are presented in Lakefield's 2003 report (SGS Lakefield Research Limited, 2003b) and are discussed in the tailings and environmental section of the SNC-Lavalin feasibility reports.

Hydrometer tests on flocculated but degraded tailings from pilot plant operation with SAPO-CSB blend and Mine Blend showed that both samples contained about 33% passing 10µm and 5% passing 1.3µm.

Natural degradation tests on tailings from pilot plant operation with SAPO-CSB blend and Mine Blend were performed in a 57L aquarium located outside at Lakefield (SGS Lakefield Research Limited, 2003a). Results are summarized in Figure 16.13 presented below. It will be noted that PP1 was terminated after 55d when CNWAD had dropped to less than 15ppm. PP2 tailings took 100d to reach 20 ppm – probably because the initial sample contained more cyanide and copper than the PP1 tailings sample.

Figure 16.13 Natural Degradation of PP1 and PP2 Tailings

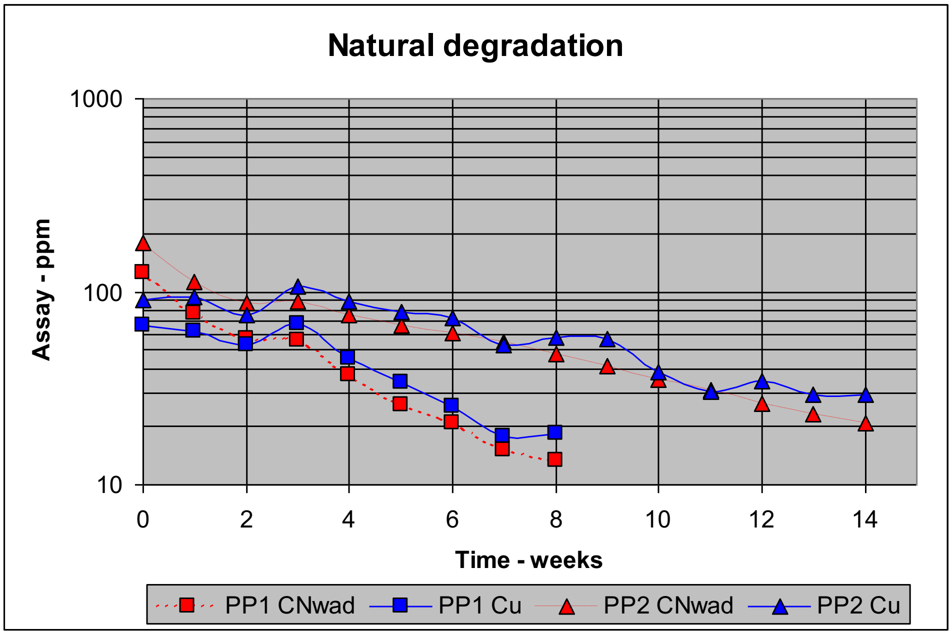


Figure 16.13 shows that cyanide and copper are rapidly removed in the Lakefield environment. Data from Crystallex's operations in Venezuela show even faster cyanide degradation rates under the more favorable temperature and insolation conditions in that location.

Four continuous cyanide destruction tests were performed using the INCO process and standard test procedures on degraded tailings solution from PP1. The tests showed that with an SO_2 to CNWAD ratio of about 6, a starting CNWAD of 13 ppm, a residence time of 20 to 30 minutes, and at a pH of 8, CNWAD values (measured by the Scaler distillation method) of <0.3 ppm could be attained with or without copper addition. When the SO_2 to CNWAD ratio was dropped to 3, without using copper addition, the CNWAD level was reduced to 0.7 ppm. Residual copper levels were 1.1 to 1.7 ppm at the higher SO_2 addition levels and 4 ppm at the lower level. Lime additions in these tests were in the range of 2 to 3 kg/kg of CNWAD destroyed.

Five continuous tests were done on degraded PP2 tailings solution containing 21 ppm CNWAD and 22 ppm Cu. At SO_2 to CNWAD ratios of about 6, CNWAD was reduced to <0.1 ppm with or without copper addition. Copper in treated solution was <0.5 ppm. Lime consumption was 3 to 4 kg/kg of CNWAD destroyed.

The treatment conditions noted above are similar to industrial experience elsewhere with the INCO air/SO2 process and the results are very acceptable.

17.0 MINERAL RESOURCE ESTIMATES

17.1 Database

MDA received a copy of the Placer digital database for the Las Cristinas project from Crystallex on four compact discs in 2002. This database was later augmented by ASCII and spreadsheet files. Original data available included:

- Drill data in GEOLOG format with:
    - Assay data (Au, Cu, CNSCu, Ag, and some trace elements),
    - Geological descriptions,
    - Structural data,
    - Geotechnical data, and
    - Check sample data;
- PCXPLOR databases (incomplete);
- Survey information;
- Geological code definitions;
- Cross sections with drill data and some geological interpretations;
- Geological maps and drill hole maps;
- Site maps;
- Trench geological maps with assays;
- Point load test results;
- Surface geochemical data;
- Topographic data; and
- Photographs of core.

The initial database was missing data from about 85 holes, but Crystallex was able to obtain data from 75 of them, bringing the final database to within 10 holes of being complete. The missing holes are located at Mesones-Sofia, outside of the main resource areas. Most of the 75 holes obtained by Crystallex from CVG were missing copper data, and all were missing geological data. A description of the original database is given in Table 17.1.

Table 17.1 Descriptive Statistics of Database Used

Data	Number
Drill holes	1,174
Meters of drilling*	160,600
Gold assays	162,806
Copper assays	145,547
Copper CN Soluble assays	40,655
Silver assays	145,221
Trenches	108

*Includes trenches

The database received by MDA had up to three check assays for gold and sometimes one check assay for silver and copper. There is no record of what each check assay represents, such as core split, coarse reject, or pulp. Correlation between these duplicate samples is very good to excellent.

Since constructing the original database, all of which was data derived from Placer's work, Crystallex has drilled 90 core holes in four drill campaigns from 2003 through 2007. All Crystallex drilling was NQ size and was completed by the same drilling contractor as was used by Placer Dome, Majortec Drilling. Crystallex's data in the database are described in Table 17.2.

Table 17.2 Descriptive Statistics of Crystallex Data

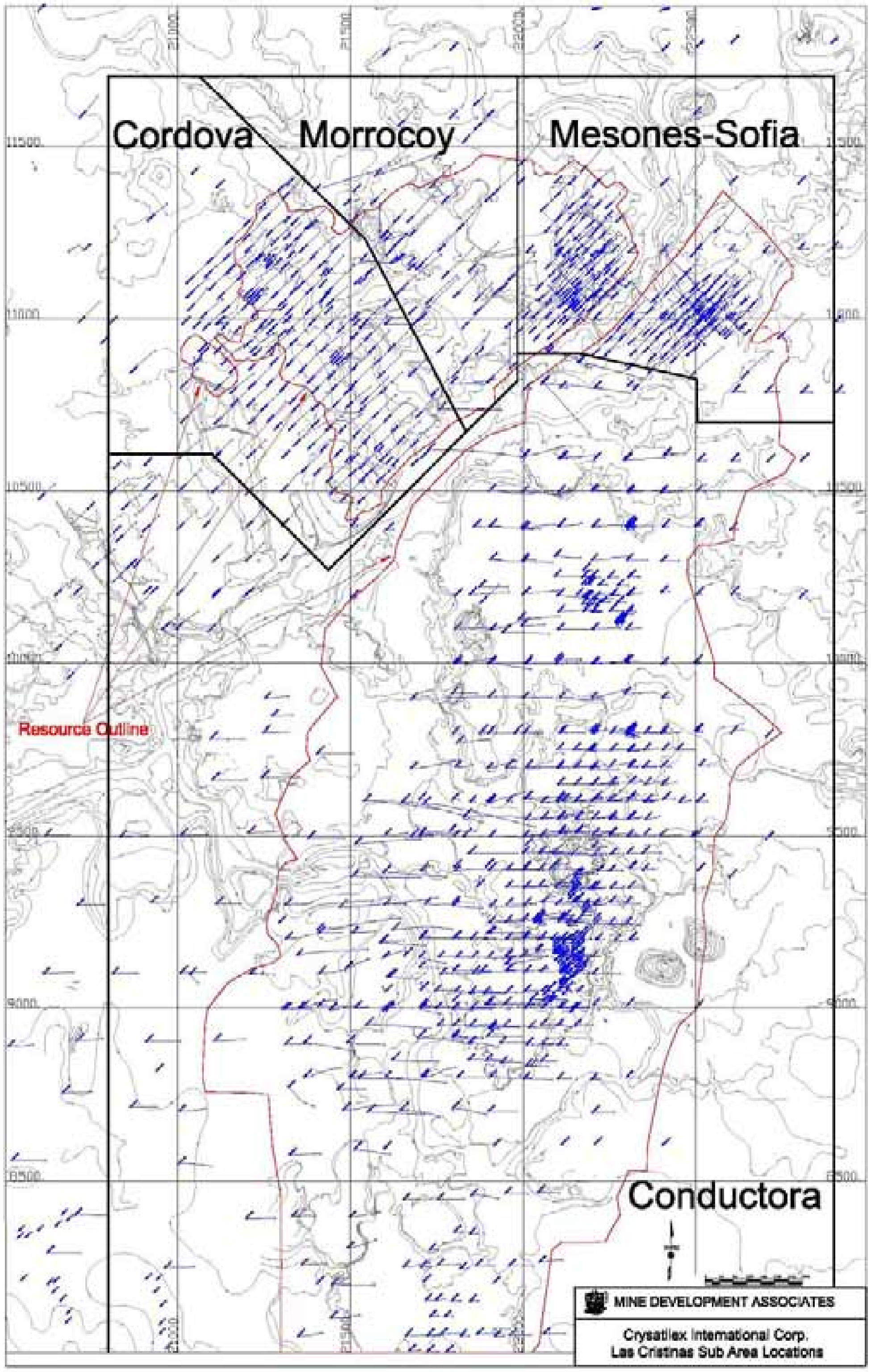
Data	Number
Drill holes	90
Meters of drilling	28,427
Gold assays	24,669
Copper assays	22,661
Copper CN Soluble assays	3,250
Silver assays	None
Trenches	None

17.2 Model Areas

Boundary files were made around four areas of the concession, which were used for modeling, estimation, and tabulations. These boundaries differentiated areas of varying amounts of exploration data, degree of geological understanding, geological contacts, and level of confidence. Four areas were defined: Conductora, which includes Cuatro Muertos and Potaso, Mesones-Sofia, Cordova, and Morrocoy (Figure 17.1).

For grade modeling, the different areas were defined on the basis of geographic location and drill pattern. Most of the drilling in Conductora was at azimuth 090°, while in Mesones-Sofia, Morrocoy and Cordova it was at azimuth 045°. All areas lie within the same block model, but each was estimated from its respective drill-hole files, and all work was limited by bounding files which were contiguous but did not overlap (Figure 17.1).

Figure 17.1 Locations of the Four Las Cristinas Sub-Areas



17.3 Las Cristinas Resources – General

MDA classified the resource in order of increasing geological and quantitative confidence, into Inferred, Indicated and Measured categories to be in compliance with Canadian National Instrument 43-101 and the "CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines," issued in 2000 and modified with adoption of the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in 2005. CIM mineral resource definitions are given below:

Mineral Resource

> *Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.*
>
> A 'Mineral Resource' is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
>
> *The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.*

Inferred Mineral Resource

> An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.
>
> *Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to*

> *enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.*
>
> **Indicated Mineral Resource**
>
> An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
>
> *Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.*
>
> **Measured Mineral Resource**
>
> A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.
>
> *Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit."*

Because of the requirement that the resource exists "in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction," MDA is reporting the resources at cutoffs that are reasonable for deposits of this nature and mining conditions of this type. MDA has considered geological understanding and use it in modeling, sample integrity, verifiability of data, and estimation parameters in the classification. For example, there are no Measured or Indicated resources at Cordova because at this time the geology is poorly understood and the sample data have not been validated. For the entire deposit the relative amount of Measured material has increased as compared to previous estimates because of the extensive work in geology and data verification that Crystallex has done.

17.4 Gold

17.4.1 Conductora

Quantile-Quantile ("QQ") plots were made for the Conductora gold grades. Grade populations of 0.15 g Au/t, 1 g Au/t, 7 g Au/t, and 18 g Au/t were analyzed. Cross sections were made using color coding of the drill samples to match these cutoffs. After constructing and plotting these sections, and working with the analytical and geological data (lithological and structural information), certain modeling criteria and gold distributions became apparent. While these zones were defined in 2002 and have proven to be quite predictable and robust, it was not until well into Crystallex's exploration and geological studies that the reason for the gold distribution was understood. The controls on mineralization are lithological with the more favorable units having more primary porosity and permeability. As such, the highest-grade zones lie within what is considered to be a dominantly volcaniclastic unit. While the lithology correlates with the mineralization, the amounts of sulfides and alteration do as well. As such, the higher-grade zones can be visually identified by lithology, alteration, and sulfides. The sulfides are dominated by pyrite and lesser chalcopyrite.

The following zones are described in terms of geology and distribution of mineralization:

- Traces of gold (~0.1 g Au/t or 100 ppb) are found throughout the entire sequence of rocks including outside the defined mineralized zones. These rocks are weakly altered and have low sulfide contents.
- The low-grade zone is defined by low-sulfide content, moderate alteration, and grade ranges from approximately 0.2 g Au/t to 1 g Au/t. These zones strike for the entire length of the concession from south to north broken only by an 80m-wide, vertical, unmineralized dike.
- The high-grade zone is defined by high sulfide content, strong alteration within a dominantly volcaniclastic lithology, and grade ranges above approximately 1 g Au/t. This main body of +1 g Au/t material, the central core, occurs in tabular deposits up to 100m thick along a strike length of over 1,000m beginning about 600m north of the southern property boundary.

The highest-grade zone, which could not be modeled separately, is defined statistically as grading over 7 g Au/t. MDA took considerable time attempting to model this zone. In the central core of the deposit in the well-mineralized volcaniclastic unit, there is a thin zone about 5m thick of +7 g Au/t (on a sample-interval scale) that extends for close to 1000m down dip. But this zone strikes only a few hundred meters, and the intercepts for the most part cannot certainly be correlated. Outside this central zone, the +7 g Au/t material is spotty and does not correlate.

Grade boundaries are gradational except at the very highest cutoffs of over 10 to 20 g Au/t. Most of the gold mineralization behaves more like that of a disseminated deposit, as would be expected of a lithologically controlled gold deposit. MDA derived a color-coded cutoff of 0.2 g Au/t from grade-distribution plots. This figure of 0.2 g Au/t is well below economic cutoff but well within the tenor of the gradational lower-grade boundary. In order to better define the next natural, gradational mineralization boundary, MDA used a majority-in/majority-out rule-of-thumb that enclosed a coherent cluster of higher grades. This higher-grade zone was determined to be over ~1 g Au/t. The next higher-grade coherent mineralized zone was defined as having a grade of ~1.7 g Au/t. The highest-grade zones are over ~7 g Au/t and are found to have moderate continuity from hole to hole and section to section only in the central part of the deposit around northing 9,000N in local grid coordinates. In the core of the deposit, such a zone of high grade but only a few meters thick can extend with moderate continuity from the surface to the deepest holes, a distance of up to 600m down dip.

Unmineralized dikes cut the mineralized zones described above. These mafic dikes are easily identified and distinguished from the mineralized country rock by their weak alteration and negligible pyrite content. These units were segregated during modeling. Quaternary alluvial material overlies all saprolite (which in turn overly bedrock) units. Tailings and workings from recent small-scale *artisinal* mining, which has been particularly intense since the 1980s, are included with the alluvial material unit for classification and modeling purposes.

Table 17.3 lists the zones being modeled. A typical cross section through the core of the deposit is shown in Figure 9.1.

Table 17.3 Modeled Gold Zones at Conductora

Zone/Material	Description
8	Overburden
9	Dikes – Not modeled (given 0 grade; all below overburden)
21	Footwall low-grade zone (all below overburden)
31	Main low-grade zone (all below overburden)
41	Hanging wall low-grade zone (all below overburden)
22	Footwall high-grade zone (all below overburden)
32	Main high-grade zone (all below overburden)
42	Hanging wall high-grade zone (all below overburden)
99	Background outside mineralized units (all below overburden)

Gold grades typically do not change significantly across contacts between bedrock and saprolite material types. As such, the same gold zones described above were carried across the saprolite-bedrock contact. These gold zones were modeled as semi-soft boundaries, in that down-hole compositing was first done to six-meter intervals and then coded from the cross-sectional zones. Later in the modeling process, the grades of those blocks that straddle contacts between material types were weight-averaged with the percentage of each zone represented in those blocks, thereby maintaining the integrity of the zone and honoring gradational boundaries. The only hard boundaries used in the geological model for gold were the contacts of the unmineralized dikes and the contact between the saprolite and overburden.

17.4.2 Mesones-Sofia

Separate QQ plots were made for drill sample gold grades from the Mesones-Sofia and Conductora areas. Data from these underwent separate statistical analysis in order to honor the distinct differences in geology, alteration and mineralization style evident between the Mesones – Sofia and Conductora areas. Both Mesones and Sofia are interpreted as being breccia pipes characterized by proximal, relatively high-temperature alteration and mineralization assemblages. Mineralization is largely confined to the breccias. In contrast, mineralization at Conductora is concentrated in an extensive, inclined sheet and is characterized by lower-temperature alteration assemblages in comparison to Mesones-Sofia. Grade populations of ~0.2 g Au/t, ~1.0 g Au/t, ~2.7 g Au/t, and ~24 g Au/t were reviewed, but only the low-grade ~0.2 g Au/t and the higher-grade zone, ~1.0 g Au/t, were used in the final model. Otherwise, the same procedures and geological parameters were used to model Mesones-Sofia as were used to model Conductora, including the segregation of the unmineralized dikes and the overburden alluvial material from the mineralized zones.

There is less confidence in the Mesones-Sofia model due to the complexity of the shape of the breccia body in comparison to the relatively simple, sheet-like geometry of Conductora. Hard boundaries were required in this deposit because of the extremely high grades in the central zones. Local high-grade zones are found with sulfide clots and quartz-flooded breccia zones. These pockets are presumed to have very short continuity, on the order of meters or less. A list of the modeled zones is given in Table 17.4, and a typical cross section through Mesones-Sofia is given in Figure 9.3.

Table 17.4 Modeled Gold Zones at Mesones-Sofia

Zone/Material	Description
8	Overburden
9	Dikes – Not modeled (given 0 grade; all below overburden)
31	Main low-grade zone (all below overburden)
32	Main high-grade zone (all below overburden)
99	Background outside mineralized units (all below overburden)

17.4.3 Morrocoy and Cordova

Similar modeling procedures were used at Morrocoy and Cordova as were used at Conductora. The resulting model contains relatively small high-grade zones within more extensive low-grade boundaries. Another difference between Morrocoy and Cordova with respect to Conductora is steeper dips of the mineralized zones (~60^{o} to the southwest) and their overall northwest strike. This change in attitude of the mineralized zones is due to folding in which the Conductora and Sofia areas are located in the north-striking (southern) limb of a regional synform, while Cordova, Morrocoy and Mesones are located in the northwest-striking limb. Coding and procedures are otherwise the same as for Conductora, described in Section 17.4.1. A typical cross section through Morrocoy and Cordova (and Mesones) is given in Figure 9.3.

17.5 Copper

17.5.1 Conductora

The copper model is dominated by material types that resulted from surface weathering processes. Primary copper mineralization at Conductora is disseminated in low to moderate grades (~1,000 to ~2,000 ppm) with no well-defined grade boundaries or geological controls, although there is a weak relationship to gold, with higher copper grades in some of the volcaniclastic units that typically host the higher gold grades. The copper occurs as disseminations and also, but rarely, in clots up to 10s of centimeters across.

Weathering has modified the original copper distribution. Much of the copper has been leached from the oxide saprolite, although some pods of high copper and high soluble copper do exist where primary copper sulfides are encapsulated by quartz, typically in breccia zones. The copper that was leached was re-deposited in the mixed and sulfide saprolite. This re-deposited copper occurs as secondary copper sulfides including chalcocite, covellite, and bornite, and is cyanide soluble. As with the gold, copper in the overburden was modeled as a distinct unit. Consequently, the following material types and copper zones were modeled:

- The alluvial material was modeled separately;
- The oxide saprolite was modeled separately;
- The sulfide and mixed saprolite were modeled together; and
- All material from the saprock and below was modeled.

Table 17.5 gives a list of the material types that were modeled, and Figure 9.2 presents a typical cross section with the material types related to copper mineralization at Conductora.

Table 17.5 Modeled Copper Zones at Conductora

Zone/Material	Description
8	Overburden
9	Dikes – Not modeled (assigned 0 grade)
6	Oxide saprolite
4, 5	Mixed and sulfide saprolite high-grade zone
1, 2, 3	Background outside mineralized units

17.5.2 Mesones-Sofia

Copper mineralization at Mesones-Sofia has a similar distribution to that of gold in that it is largely confined to the breccia zones that are clustered to form breccia-dominated areas that have an overall pipe-like geometry. Differences between mineralization at Mesones-Sofia as compared with Conductora can be summarized as follows:

- The majority of the mineralization at Mesones-Sofia cross-cuts stratigraphy with a relatively small component oriented parallel to bedding. Conductora-style mineralization is overwhelmingly bedding parallel.
- Mesones and Sofia are composed of two distinct pipe-like bodies in which vein-breccias are concentrated. Two dominant breccia-vein orientations are evident in core: one is steeply dipping and the other, subordinate set, is bedding sub-parallel. Both of the breccia pipes are oval-shaped in cross section with the long axis orientated northeast, probably due to a preferential northeast strike of the sub-vertical vein-breccias. The Sofia breccia is located on the south side of a fold axis, in the same limb as Conductora. This limb strikes north and dips moderately (30-40°) to the west. Mesones lies on the north side of the fold axis, with Morrocoy and Cordova, where strata strike northwest and dip more steeply (average 60°) to the southwest. Mineralization decreases in intensity and grade abruptly at the margins of both breccia pipes.
- Most sulfide grains and aggregates in the Mesones and Sofia breccia pipes are encapsulated in quartz, whereas silicification is rare in Conductora. Sulfide grains and aggregates are, on average, significantly coarser in Mesones-Sofia than in Conductora.

Sofia is separated from Mesones by a steeply dipping diorite dike that varies from about 80 to 100m wide. As is the case with the shallow-dipping *en echelon* diorite sills that are ubiquitous in the northern part of the Las Cristinas deposit, the wide dike in the Mesones-Sofia area is post-mineralization in age and cuts mineralization. These barren dikes are assigned a zero gold and copper value in the model. A list of modeled zones is given in Table 17.6, and a typical cross section showing the copper mineralization is given in Figure 9.4.

Table 17.6 Modeled Copper Zones at Mesones-Sofia

Zone/Material	Description
8	Overburden
9	Dikes – Not modeled (assigned grade of 25 ppm Cu; all below overburden)
6	Oxide saprolite
4, 5	Sulfide and mixed saprolite
61	Low-grade zone in carbonate-stable bedrock, carbonate-leached bedrock and saprock
62	High-grade zone in carbonate-stable bedrock, carbonate-leached bedrock and saprock
99	Outside the mineralized zones in carbonate-stable bedrock, carbonate-leached bedrock and saprock

17.5.3 Morrocoy and Cordova

Similar procedures for modeling the copper were used at Morrocoy and Cordova as were used at Conductora, but overall there is substantially much less copper at Morrocoy and Cordova in comparison with Mesones-Sofia and Conductora. Like the gold, the copper is modeled in the bedrock units along northwest strikes and ~60^o dips. Coding and procedures are otherwise the same as for Conductora, described in Section 17.5.1. Figure 9.4 is a cross section through Morrocoy and Cordova (and Mesones-Sofia).

17.6 Silver

Silver occurs in low concentrations at Conductora, Cordova and Morrocoy and partly because of these low grades, and partially because it has received little study, the silver mineralization is poorly understood. It seems that the silver is finely disseminated throughout the deposit in very low concentrations of less than ~0.5 g Ag/t. Relatively high-grade silver zones have an erratic, discontinuous distribution throughout the Las Cristinas deposit. Table 17.7 shows the list of units used in modeling silver grades at Conductora. Some of the same material types that were used in the modeling of copper were also used in order to model silver.

Table 17.7 Modeled Silver Zones at Conductora

Zone/Material	Description
8	Overburden
9	Dikes – Not modeled (given 0 grade; all below overburden)
1, 2, 3, 4, 5, 6	Background (all material below overburden)

The distribution of silver at Mesones-Sofia is similar to that in Conductora. Table 17.8 shows the list of units used in the modeling of silver. The same material types that were used in silver modeling were used for modeling copper grades.

Table 17.8 Modeled Silver Zones at Mesones-Sofia

Zone/Material	Description
8	Overburden
9	Dikes – Not modeled (given 0 grade; all below overburden)
1, 2, 3, 4, 5, 6	Background (all below overburden)

17.7 Specific Gravity

Specific gravity at Las Cristinas is dominantly controlled by weathering and is incorporated into the model with material type. The specific gravity values used are given in Table 17.9 The bedrock essentially has similar density throughout, except for the dikes, which have a lower density. Weathering processes have saprolitized the rocks and decreased their densities. Generally, density of material types at Las Cristinas increases with depth. The leaching of carbonate from the bedrock unit located beneath the saprolite, the carbonate-leached bedrock unit, resulted in the development of voids and vugs, resulting in a decrease in the specific gravity relative to primary bedrock.

Table 17.9 Material Types used to Define Specific Gravity

All Areas

Material Type	SG
CBS Bedrock	2.79
CLB Bedrock	2.35
Saprock	1.92
Sulfide Saprolite	1.69
Mixed Saprolite	1.69
Oxide Saprolite	1.56
Overburden	1.63
Dike	1.93

Mesones-Sofia Only

Material Type	SG
CBS Bedrock	2.79
CLB Bedrock	2.39
Saprock	2.13
Sulfide Saprolite	1.89
Mixed Saprolite	1.64
Oxide Saprolite	1.68
Overburden	1.64
Dike	1.89

17.8 Metallurgical Model

The metallurgical model is the same as the material-type model. The principal differences in these rock types that would affect metallurgical characteristics are: a) the hardness, manifested in the amounts of clay, b) the amount of copper (in both primary and remobilized secondary chalcocite), and c) the specific gravity. The specific gravity was described earlier and is a direct consequence of surface weathering. Generally the specific gravity increases while the relative clay content decreases with depth. Approximate copper grades for the main material types are given in Table 17.10. No hardness model of material for bedrock has been determined or made.

Table 17.10 Copper and Soluble Copper Grades by Material Type and Area

(All grades in ppm)

	CBS, CLB, SAPR	CBS, CLB, SAPR	SAPS, SAPM	SAPS, SAPM	SAPO	SAPO
Area	**ppmCu**	**ppmCNSCu**	**ppmCu**	**ppmCNSCu**	**ppmCu**	**ppmCNSCu**
Conductora	756	82	1634	900	552	33
Mesones-Sofia	3104	419	2755	1733	298	41
Morrocoy/Cordova	332	123	348	277	143	14
Note: Numbers of samples are necessarily the same for Cu and CNSCU mean grades						

17.9 Conductora Grade Models

17.9.1 Conductora - Assays

The assay database from which Conductora was modeled is described in Table 17.11. In the defined Conductora area, there are 78,253 samples with gold grades and 77,117 samples with copper grades. There are 1,853 samples that were eliminated from the modeling database because they were from trench samples and 873 eliminated because they were deemed "contaminated." "Contaminated" samples were not used for obvious reasons, which were described more thoroughly earlier in the report in Section 13.0. Trench samples were eliminated for three reasons: there is a positive bias compared to drill samples, surface hand sampling commonly introduces biases, and the variography results were distinctly different and difficult to model when trench data were combined with drill sample data.

Table 17.11 Descriptive Statistics of the Conductora Assay Database
(including trench data and those samples deemed "contaminated")

AREA 1 Conductora-Cuatro Muertos

	Valid N	Mean	Median	Std. Dev.	CV	Min	Max.	Units
East	103,927					20,800	22,805	m
North	103,927					8,003	10,903	m
Elevation	103,923					(454)	145	m
From	103,927	118.59				0.0	599.0	m
To	103,927	119.59				0.1	600.0	m
Length	103,927	1.00				0.0	450.0	m
Au	102,648	0.84	0.37	5.14	6.10	0.00	1296.5	ppm
AuCap	102,648	0.80	0.37	1.52	1.90	0.00	40.0	ppm
Cu	97,780	831	366	1533	2	0	90,800	ppm
CuCap	97,780	827	366	1438	2	0	25,000	ppm
CuCNA	32,574	262	32	985	4	1	48,250	ppm
CuCnCap	32,574	261	32	984	4	1	48,250	ppm
CuRatio	32,484	17	6.00	24	1	0.00	100.0	%
Ag	78,376	0.90	0.30	7.57	8.40	0.00	680.0	ppm
AgCap	78,376	0.79	0.30	2.57	3.28	0.00	130.0	ppm
CREC	95,065	91	98	17	0.2	0	232	%
RQD	59,833	77	85	23	0.3	0	253	%
MaterialCode	5,941					1	9	%
Zone	103,927					0	42	%
Code	103,927					0	0	ppm
Area	103,927					1	1	
Type	103,927					0	9	
Use	893					2	2	
DHorTR	102,872					1	2	

Capping limits were determined iteratively considering:

- the context of modeled zones, material types,
- grade distribution plot profiles of each metal,
- the affected "contained" metal content,
- the geology, and
- the resulting coefficient of variation (CV).

The final capping limits are given in Table 17.12. The total-metal-content reduction caused by capping ranged from 0.2% for copper to 43% for silver. The extreme effect in the reduction of contained silver is justified by the fact that the silver high-grades have little to no continuity and are poorly understood.

Gold-metal-content reductions ranged from 2% for the overburden to 10% for mineralization located outside the mineralized zones. The apparently extreme 10% reduction in metal content caused by capping in the areas outside defined mineralized zones is justified by the fact that there is no continuity of the higher grade. It is believed to be "pockets" or blebs, outside the mineralized zones. Gold-zone metal reduction caused by capping was 6% for low-grade and 5% for high-grade zones. Although continuity of the higher grades is, in most cases, good, the rather loose, broad mineral zones at Conductora necessitated that capping be done.

Table 17.12 Capping Limits and Assay Statistics Conductora Samples

Zones	**21, 31, 41**	**Low-sulfide zone**			**Capped at**	**7**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	46,482	0.39	0.61	3.38	5.51	0.00	808	g/t
AuCap	46,482	0.39	0.58	0.70	1.20	0.00	7	g/t
Difference in grade		0%	-5%		Difference in metal		-5%	

Zones	**22, 32, 42**	**High-sulfide zone**			**Capped at**	**40**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	22,955	1.26	1.96	9.40	4.81	0.00	1,297	g/t
AuCap	22,955	1.26	1.87	2.50	1.34	0.00	40	g/t
Difference in grade		0%	-5%		Difference in metal		-5%	

Zone	**8**	**Overburden**			**Capped at**	**7**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	1,676	0.88	1.22	1.41	1.16	0.00	14	g/t
AuCap	1,676	0.88	1.20	1.27	1.06	0.00	7	g/t
Difference in grade		0%	-2%		Difference in metal		-2%	

Zone	**0**	**Outside Zones**			**Capped at**	**6**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	26,815	0.09	0.21	2.04	9.58	0.00	282	g/t
AuCap	26,815	0.09	0.18	0.42	2.28	0.00	6	g/t
Difference in grade		0%	-14%		Difference in metal		-14%	

Type	**1, 2, and 3**	**Bedrock and Saprock**			**Capped at**	**ppm**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	69,192	359	756	1246	1.65	0	68400	ppm
CuCap	69,192	359	755	1207	1.60	0	22000	ppm
Difference in grade		0%	0%		Difference in metal		0%	
CuCNA	16,249	32	82	220	2.67	1	6700	ppm
CuCnCap	16,249	32	82	207	2.53	1	3750	ppm
CuRatio	16,237	6	10	13	1.34	0	100	ppm

Type	**4, 5**	**Sulfide and Mixed Saprolite**			**Capped at**	**ppm**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	12,587	800	1634	2781	1.70	0	90800	ppm
CuCap	12,587	800	1615	2474	1.53	0	25000	ppm
Difference in grade		0%	-1%		Difference in metal		-1%	
CuCNA	7,280	244	900	1870	2.08	1	48250	ppm
CuCnCap	7,280	244	900	1870	2.08	1	48250	ppm
CuRatio	7,256	40	42	31	0.75	0	100	ppm

Type	**6**	**Oxide Saprolite**			**Capped at**	**ppm**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	11,444	275	552	832	1.51	0	12300	ppm
CuCap	11,444	275	550	814	1.48	0	7300	ppm
Difference in grade		0%	0%		Difference in metal		0%	
CuCNA	6,887	7	33	220	6.58	1	9700	ppm
CuCnCap	6,887	7	33	220	6.58	1	9700	ppm
CuRatio	6,840	2	7	13	1.89	0	100	ppm

Type	**8**	**Overburden**			**Capped at**	**ppm**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	1,653	154	255	346	1.35	0	4640	ppm
CuCap	1,653	154	249	295	1.19	0	1900	ppm
Difference in grade		0%	-3%		Difference in metal		-3%	
CuCNA	981	18	37	78	2.14	1	1455	ppm
CuCnCap	981	18	37	78	2.14	1	1455	ppm
CuRatio	978	13	19	19	0.99	0	100	ppm

17.9.2 Conductora - Composites

Gold: After capping, the gold-assay sample intervals were composited to six-meter lengths. The gold grades were down-hole composited using geological restrictions for some material types. Honored material types were overburden (material type 8) and dikes (material type 9), because there is a discontinuity between both of these and the gold mineralization and because both post-date mineralization; only the dike is barren. After compositing and excluding the overburden and dike, the six-meter composites were coded from the cross-sectional gold-zone interpretations. This set of composites was used for modeling gold and core recovery. Gold-composite statistics are given in Table 17.13.

Copper, copper solubility and silver by material type: After capping, the copper-assay sample intervals were composited to six-meter lengths. All the material types were honored during compositing. These composites were used for estimating copper, copper solubility, and silver. After compositing, the composites were back-coded from the model with the relative elevation from the top of the mixed or sulfide-saprolite unit. This relative elevation was used as a reference for a sample's distance above or below the oxide-sulfide contact in modeling copper and copper solubility ratios in the saprolite units. Descriptive statistics of the composite-samples files all showed reasonably well-behaved data, and for those data sets that were not, estimation routines were changed to account for high-variance data sets, which did not occur in the gold data. Copper composite statistics are given in Table 17.13.

Table 17.13 Statistics by Zone (Au) and Type (Cu) of Conductora Composites

All composites

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	17474	0.0	5.3	0.0	0.0	0.0	6.0	m
Au	16274	0.48	0.82	2.24	2.73	0.00	226.54	g Au/t
Aucap	16274	0.48	0.78	0.93	1.20	0.00	15.23	g Au/t

ZONE 21,31,41 Low-sulfide mineralization

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	7871	0.0	5.6	0.0	0.0	0.0	6.0	m
Au	7609	0.50	0.63	1.17	1.85	0.01	88.40	g Au/t
Aucap	7609	0.50	0.60	0.44	0.74	0.01	9.17	g Au/t

ZONE 22,32,42 High-sulfide mineralization

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	4008	0.0	5.3	0.0	0.0	0.0	6.0	m
Au	3687	1.49	1.93	4.05	2.10	0.01	226.54	g Au/t
Aucap	3687	1.49	1.83	1.27	0.69	0.01	15.23	g Au/t

ZONE 8 Overburden

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	378	0.0	3.4	0.0	0.0	0.0	6.0	m
Au	255	0.92	1.12	0.88	0.79	0.04	4.93	g Au/t
Aucap	255	0.92	1.10	0.83	0.76	0.04	4.37	g Au/t

ZONE 99 Outside mineralized zones

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	4719	0.0	5.5	0.0	0.0	0.0	6.0	m
Au	4539	0.12	0.21	0.81	3.78	0.00	34.33	g Au/t
Aucap	4539	0.12	0.19	0.25	1.34	0.00	6.00	g Au/t

TYPE 1,2,3 CBS, CLB, SAPR

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	11155		5.2			0.0	6.0	m
Cu	10334	505	772	900	1.16	1	15522	ppm
Cucap	10334	505.00	770.70	884.42	1.15	1.00	13887	ppm
CuCN	2770	41	82	170	2.07	1	3035	ppm
CuCNC	2770	41	81	163	2.01	1	3035	ppm

TYPE 4,5 SAPS, SAPM

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	2586		4.4			0.0	6.0	m
Cu	2344	940	1668	2161	1.29	0	46779	ppm
Cucap	2344	940.00	1648.66	1935.45	1.17	0.00	22196	ppm
CuCN	1368	291	927	1512	1.63	1	23136	ppm
CuCNC	1368	291	927	1512	1.63	1	23136	ppm

TYPE 6 SAPO

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	2200		4.6	0.0		0.0	6.0	m
Cu	1984	307	560	717	1.28	0	8168	ppm
Cucap	1984	307.00	558.62	708.42	1.27	0.00	7300	ppm
CuCN	1201	9	33	173	5.22	1	6823	ppm
CuCNC	1201	9	33	173	5.22	1	6823	ppm

TYPE 8 OVB

	Valid N	Median	Mean	Std.Dev.	CV	Minimum	Maximum	Units
Length	657		2.2			0.0	6.0	m
Cu	559	171	259	314	1.21	0	3582	ppm
Cucap	559	171.00	252.08	263.14	1.04	0.00	1900	ppm
CuCN	334	23	38	61	1.63	1	540	ppm
CuCNC	334	23	38	61	1.63	1	540	ppm

17.9.3 Conductora - Geostatistics and Estimation

MDA calculated numerous variograms and correlograms at varying lags, cutoffs, azimuths and dips, and with separate and combined zones. In the end, variogram models were chosen that were parallel to the mineralization-controlling geological fabric, namely 15° strike azimuth, 285° dip azimuth and dip of -35°. The variograms were calculated on composites of gold, copper, CNSCu-to-total-Cu ratio, and silver. Silver has a small (~1%) population of very high-grade samples with no apparent continuity, and consequently, the silver variograms portray the low-grade "disseminated" style of mineralization, which was modeled differently from the high-grade spikes.

All metal grades were estimated by ordinary Kriging. The estimation parameters are given in Appendix B. Multiple overwriting estimation passes were done for gold to compensate for over-smoothing in a single pass. This was not necessary for either copper or silver; however, the cyanide-soluble ratio yielded an over-smoothed model compared to the actual grades. The ratio of CNSCu to total copper in each block was estimated from the ratio in the drill composites. This method was chosen because CNSCu data were incomplete and estimating the ratio gave the ability to use estimation parameters pertinent to cyanide solubility, since the ratio is rock and grade dependent.

Gold distribution has not been materially affected by the weathering process. However, gold distribution has been significantly modified in the overburden due to alluvial concentrations and rudimentary mining. Included in the overburden are tailings and reworked material. As this unit is a catch-all term for all surficial material, no Measured resources were defined in this unit. As a result, gold was modeled in domains that crossed the bedrock and saprolite contacts, but never the overburden contact.

Copper distributions were materially affected by the weathering process. The overburden was treated as its own unit for the same reasons as it was for gold estimation. Copper has been leached from the oxide saprolite, although some local areas of high copper are preserved in areas of intense silicification that have protected copper minerals from leaching above the water table. These remnant pods of cyanide-soluble copper required that a locally accurate estimate of the cyanide-soluble copper be done. The copper leached from the oxide saprolite was deposited at and below the mixed/sulfide and oxide-saprolite contact. Little redistribution of copper occurred in the saprock, CLB and CSB bedrock. Consequently, both CNSCu and total-copper estimation were restricted to:

- oxide saprolite,
- combined mixed and sulfide saprolite, and
- combined saprock, CLB and CSB bedrock.

Very few differences were noted between the material types for silver grades. Throughout the sequence of material types, the silver is low-grade but with erratic spotty high-grades, often occurring as single assay spikes. The silver was modeled in the overburden as one unit, and all the other units combined as the second.

17.9.4 Conductora - Resources

MDA classified the resource by a combination of distance to the nearest sample, the number of samples used to estimate a block, the geological understanding and predictability of the resources, and the quality of the drill samples used, *i.e.*, core recovery. As gold is the dominant metal from a value standpoint and Crystallex has no mining rights to the copper, all blocks were classified based on gold (Table 17.14). The ranges used for resource classification were chosen based on an average of the directional gold variogram ranges.

Table 17.14 Criteria for Classification of Conductora Resources

Class	Distance*	Min. No. of Samples	Min. No. Drill Holes
Measured*	0 to 20 m	2	1
Indicated	1 to 20 m	1	1
Indicated	20 to 60 m	2	1
Inferred	60 to 110 m	1	1

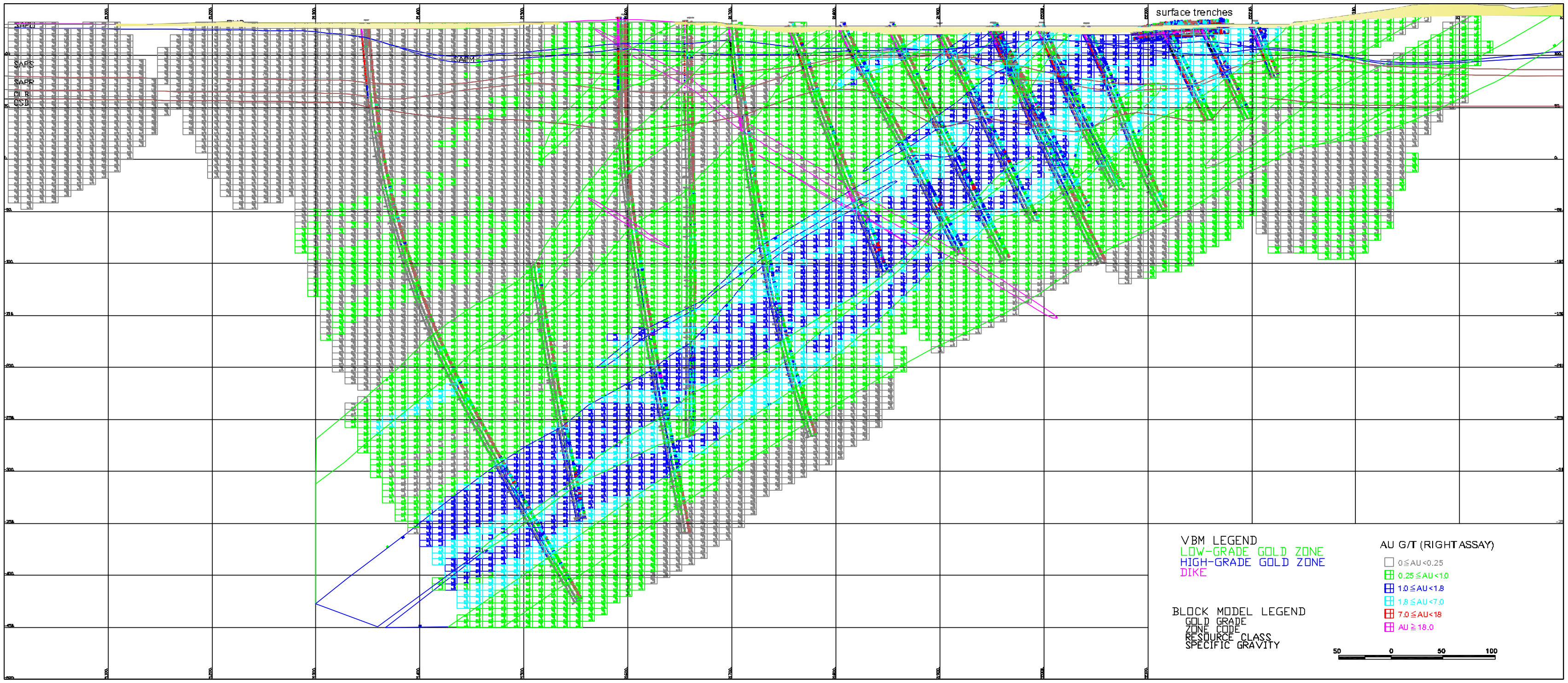
* See next paragraph for explanation of modified distances; all overburden is classified as Inferred.

MDA modified the distances used for classification by the percent core recovery. It was shown in an earlier section of this report that core recovery affects gold and copper grades and introduces a bias in the saprolite. The lower core recovery decreases confidence in the results and therefore is introduced into the definition of Measured, Indicated, and Inferred. MDA modified the distance between the closest sample and the model block by the following relationship:

- Estimated core recovery between 80% and 100%, no factor;
- Estimated core recovery between 60% and 80%, distance multiplied by 1.1; and
- Estimated core recovery below 60%, distance multiplied by 1.2.

The modified distance was used for the classification scheme given in Table 17.14. Essentially, those blocks with estimated lower core recovery were downgraded in classification.

The classification and the estimation described above resulted in a Measured, Indicated and Inferred resource at Conductora. Measured and Indicated resources are broken out in Table 17.15 and Table 17.16 and combined in Table 17.17, while the total Inferred resources are given in Table 17.18. This does not represent the entire body of mineralization at Conductora, as additional drilling will likely better define additional mineralization. The deposit is open ended at depth but is bounded at the south by a property boundary and at the north where it trends into Mesones-Sofia, Cordova and Morrocoy. A typical section of the Conductora gold model is given in Figure 17.2, and the copper model is in Figure 17.3.



DATE	Sep 12, 2007
DRAWN BY	S Ristorcelli
CHECKED BY	MDA
SCALE	as shown

MINE DEVELOPMENT
ASSOCIATES
Reno
Nevada

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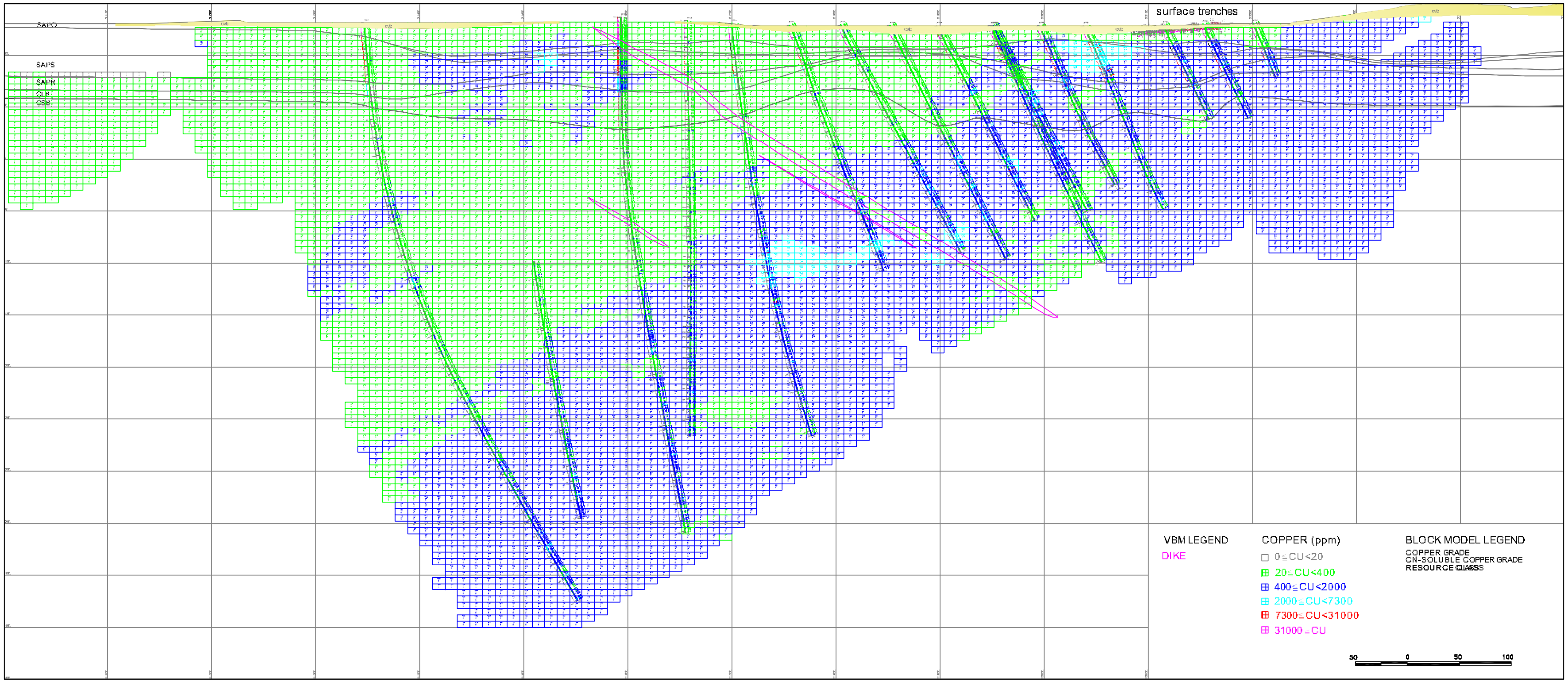
Crystallex International Corp.
Las Cristinas
Conductora Block Model
Section 9150 Au
Bolivar State
Venezuela

FIGURE NO.
17.2

CAD CONTROL BLOCK R&K\Steve\Projects\Las Cristinas_2007\Reports\43-101_2007\Figures\Fig17-2_n9150_2007.dwg
DATE:4/2/03



Crystallex International Corp.
Las Cristinas
Conductora Block Model
Section 9150 Cu

Bolivar State
Venezuela

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MINE DEVELOPMENT ASSOCIATES
Reno
Nevada

DATE	Sep 12, 2007
DRAWN BY	S Ristorcelli
CHECKED BY	MDA
SCALE	as shown

FIGURE NO.
17.3

CAD CONTROL BLOCK REF: R:\Las Cristinas_2007\Reports\43-101_2007\Figures\Fig17-3_bc9150_2007.dwg
DATE: 1/2/03

Table 17.15 Conductora Measured Resources
(Including Reserves*)

Conductora Measured (inclusive of Cuatro Muertos and Potaso) (rounded)

Cutoff	**Tonnes**	**Gold**	**Gold**	**Silver**	**Copper**	**CNSolCu**	**Silver**	**Copper**
(g Au/t)		(g/t)	Ounces	(g/t)	(ppm)	(ppm)	Ounces	Kilograms
0.2	194,937,000	0.91	5,691,000	0.50	960	132	3,159,000	187,081,000
0.4	162,676,000	1.03	5,361,000	0.53	1,019	143	2,751,000	165,783,000
0.5	**135,221,000**	**1.14**	**4,969,000**	0.55	1,067	153	2,369,000	144,227,000
0.6	109,954,000	1.28	4,528,000	0.56	1,121	166	1,994,000	123,237,000
0.7	91,264,000	1.41	4,143,000	0.58	1,166	177	1,705,000	106,405,000
0.8	77,528,000	1.53	3,814,000	0.59	1,198	187	1,481,000	92,848,000
0.9	68,219,000	1.62	3,562,000	0.60	1,221	196	1,320,000	83,281,000
1.0	61,403,000	1.70	3,356,000	0.61	1,244	204	1,202,000	76,361,000
1.5	35,751,000	2.03	2,328,700	0.64	1,354	245	737,900	48,400,000
2.0	14,551,000	2.47	1,153,700	0.68	1,483	316	319,100	21,584,000
2.5	4,927,000	2.98	472,500	0.72	1,535	366	113,700	7,563,000
3.0	1,769,000	3.48	198,000	0.77	1,578	403	44,000	2,790,000
3.5	596,000	4.04	77,000	0.79	1,646	471	15,000	981,000
4.0	211,000	4.64	32,000	0.75	1,621	476	5,000	343,000
5.0	38,000	6.04	7,000	0.76	1,465	380	1,000	56,000

Note: inconsistencies between grade, tonnes, and ounces are due to rounding

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Table 17.16 Conductora Indicated Resources
(Including Reserves*)

Conductora Indicated (inclusive of Cuatro Muertos and Potaso) (rounded)

Cutoff	**Tonnes**	**Gold**	**Gold**	**Silver**	**Copper**	**CNSolCu**	**Silver**	**Copper**
(g Au/t)		(g/t)	Ounces	(g/t)	(ppm)	(ppm)	Ounces	Kilograms
0.2	835,798,000	0.68	18,165,000	0.44	861	92	11,689,000	719,538,000
0.4	544,465,000	0.89	15,509,000	0.45	959	100	7,930,000	522,251,000
0.5	**428,293,000**	**1.01**	**13,853,000**	0.47	1,006	105	6,417,000	430,734,000
0.6	333,008,000	1.14	12,184,000	0.48	1,047	109	5,086,000	348,793,000
0.7	257,856,000	1.28	10,628,000	0.48	1,089	114	4,004,000	280,831,000
0.8	206,666,000	1.42	9,409,000	0.49	1,114	117	3,236,000	230,308,000
0.9	171,985,000	1.53	8,471,000	0.49	1,131	120	2,709,000	194,463,000
1.0	148,712,000	1.62	7,765,000	0.49	1,144	122	2,357,000	170,141,000
1.5	78,206,000	1.98	4,976,000	0.51	1,215	138	1,269,800	94,989,000
2.0	29,290,000	2.41	2,267,600	0.51	1,254	159	481,200	36,715,000
2.5	8,997,000	2.88	834,300	0.52	1,262	204	150,700	11,358,000
3.0	2,475,000	3.34	266,000	0.53	1,267	233	42,000	3,136,000
3.5	545,000	3.92	69,000	0.57	1,261	293	10,000	687,000
4.0	143,000	4.64	21,000	0.53	1,279	254	2,000	182,000
5.0	36,000	5.77	7,000	0.43	1,325	155	1,000	48,000

Note: inconsistencies between grade, tonnes, and ounces are due to rounding

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Table 17.17 Conductora Measured and Indicated Resources
(Including Reserves*)

Conductora Measured and Indicated (inclusive of Cuatro Muertos and Potaso) (rounded)

Cutoff (g Au/t)	Tonnes	Gold (g/t)	Gold Ounces	Silver (g/t)	Copper (ppm)	CNSolCu (ppm)	Silver Ounces	Copper Kilograms
0.2	1,030,735,000	0.72	23,856,000	0.45	929	99	14,848,000	906,620,000
0.4	707,141,000	0.92	20,870,000	0.47	973	110	10,681,000	688,034,000
0.5	**563,514,000**	**1.04**	**18,822,000**	0.48	1,020	117	8,786,000	574,961,000
0.6	442,963,000	1.17	16,712,000	0.50	1,066	123	7,079,000	472,030,000
0.7	349,120,000	1.32	14,771,000	0.51	1,109	130	5,709,000	387,236,000
0.8	284,194,000	1.45	13,222,000	0.52	1,137	136	4,716,000	323,156,000
0.9	240,203,000	1.56	12,033,000	0.52	1,156	141	4,030,000	277,744,000
1.0	210,115,000	1.65	11,121,000	0.53	1,173	146	3,559,000	246,502,000
1.5	113,958,000	1.99	7,304,700	0.55	1,258	171	2,007,700	143,390,000
2.0	43,841,000	2.43	3,421,300	0.57	1,330	211	800,300	58,299,000
2.5	13,924,000	2.92	1,306,800	0.59	1,359	261	264,400	18,921,000
3.0	4,244,000	3.40	464,000	0.63	1,396	304	86,000	5,926,000
3.5	1,141,000	3.98	146,000	0.68	1,462	386	25,000	1,668,000
4.0	354,000	4.64	53,000	0.66	1,483	386	8,000	525,000
5.0	75,000	5.91	14,000	0.60	1,397	271	1,000	104,000

Note: inconsistencies between grade, tonnes, and ounces are due to rounding

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Table 17.18 Conductora Inferred Resources
(Including Reserves*)

Conductora Inferred (inclusive of Cuatro Muertos and Potaso) (rounded)

Cutoff (g Au/t)	Tonnes	Gold (g/t)	Gold Ounces	Silver (g/t)	Copper (ppm)	CNSolCu (ppm)	Silver Ounces	Copper Kilograms
0.2	408,902,000	0.52	6,770,000	0.41	761	59	5,351,000	310,970,000
0.4	203,094,000	0.76	4,930,000	0.42	813	57	2,749,000	165,095,000
0.5	**143,346,000**	**0.89**	**4,083,000**	0.44	850	55	2,028,000	121,830,000
0.6	105,917,000	1.01	3,426,000	0.45	876	53	1,536,000	92,825,000
0.7	76,295,000	1.15	2,816,000	0.47	900	51	1,143,000	68,650,000
0.8	58,262,000	1.27	2,385,000	0.48	915	48	899,000	53,298,000
0.9	44,898,000	1.40	2,022,000	0.50	927	46	719,000	41,616,000
1.0	36,094,000	1.51	1,757,000	0.51	942	45	588,000	33,997,000
1.5	15,232,000	1.93	943,700	0.53	981	45	258,600	14,937,000
2.0	4,814,000	2.34	362,600	0.49	987	47	76,000	4,749,000
2.5	1,124,000	2.85	103,100	0.41	992	53	14,800	1,115,000
3.0	228,000	3.51	26,000	0.44	947	57	3,000	216,000
3.5	62,000	4.30	9,000	0.54	997	90	1,000	62,000
4.0	22,000	5.38	4,000	0.39	1,111	11	-	24,000
5.0	14,000	5.74	3,000	0.39	1,102	12	-	16,000

Note: inconsistencies between grade, tonnes, and ounces are due to rounding

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

17.10 Mesones-Sofia Grade Model

17.10.1 Mesones-Sofia - Assays

The Mesones-Sofia assay database used for modeling is described in Table 17.19. There are 38,275 samples with gold grades, and 30,253 samples with copper grades in the defined Mesones-Sofia area. There are some “contaminated” samples but no trench samples in the Mesones-Sofia database. Capping limits were assessed considering:

- the context of modeled zones, material types,
- grade distribution plot profiles of each metal,
- the affected “contained” metal content,
- the geology, and
- the resulting coefficient of variation (CV).

The final capping limits are given in Table 17.20.

Table 17.19 Descriptive Statistics of the Mesones-Sofia Assay Database

	Valid N	Mean	Std.Dev.	CV	Min	Max	Units
Hole	230						
From	39,528				0	381	m
To	39,528				0.1	382	m
Length	39,528	0.86			0.01	198	m
AuA	38,275	0.81	2.75	3.39	0.00	370.00	g/t
AuC	38,275	0.78	1.73	2.23	0.00	33.00	g/t
CuA	30,253	2,363	4,370	2	0.00	174,300	ppm
CuC	30,253	2,355	4,252	2	0.00	55,000	ppm
CuCNA	10,869	808	2,863	4	0.00	79,500	ppm
CuCNC	10,869	808	2,863	4	0	79500	ppm
Cu-Ratio	10,867	32	33	1.04	0	100	ppm
AgA	29,770	0.90	8.22	9.15	0.00	620.00	g/t
AgC	29,770	0.76	2.12	2.80	0.00	50.00	g/t
Material	31,099				0	9	
Zone	39,375				8	99	
Code	39,375				61	99	
Area	39,528				2	2	
Type	39,375				1	9	
Core Recover	30,622	94	94	1.00	0.00	102	%
Core RQD	16,933	72	73	1.01	0.00	102	%
Use	39,138				1	2	
Drill hole/Trench	39,528				1	1	

The “contained” metal was reduced between 3% and 5% for the gold by capping, depending on the zone. Copper capping levels were negligible on the “contained” amount of metal, but the mixed/sulfide saprolite still has a high CV; however, this is a manifestation of the style of mineralization, which is a combination of primary and supergene mineralization and has some locally enriched areas. Silver CVs are very high, just as in Conductora; this is a manifestation of the style of mineralization which is typified by small isolated “blebs” or “spikes” of very high-grade silver.

Table 17.20 Capping Limits and Assay Statistics at Mesones-Sofia

Zones	**21, 31, 41**	**Low-sulfide zone**			**Capped at**	**13**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	20,527	0.35	0.53	1.25	2.36	0.00	73	g/t
AuCap	20,527	0.35	0.51	0.75	1.46	0.00	13	g/t
Difference in grade		0%	-3%		Difference in metal		-3%	

Zones	**22, 32, 42**	**High-sulfide zone**			**Capped at**	**33**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	7,534	1.29	2.29	5.50	2.40	0.01	370	g/t
AuCap	7,534	1.29	2.17	3.32	1.53	0.01	33	g/t
Difference in grade		0%	-5%		Difference in metal		-5%	

Zone	**8**	**Overburden**			**Capped at**	**4**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	4,051	0.12	0.19	0.69	3.55	0.01	43	g/t
AuCap	4,051	0.12	0.18	0.30	1.61	0.01	4	g/t
Difference in grade		0%	-5%		Difference in metal		-5%	

Zone	**0**	**Outside Zones**			**Capped at**	**5**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	788	0.36	0.84	1.35	1.61	0.01	13	g/t
AuCap	788	0.36	0.78	1.08	1.38	0.01	5	g/t
Difference in grade		0%	-6%		Difference in metal		-6%	

Type	**1, 2, and 3**	**Bedrock and Saprock**			**Capped at**	**55000**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	16,517	1360	3104	4622	1.49	1	82700	ppm
CuCap	16,517	1360	3095	4552	1.47	1	55000	ppm
Difference in grade		0%	0%		Difference in metal		0%	
CuCNA	3,122	106	419	1142	2.73	0	22250	ppm
CuCnCap	3,122	106	419	1142	2.73	0	22250	ppm
CuRatio	3,118	10	18	20	1.08	0	100	ppm

Type	**4, 5**	**Sulfide and Mixed Saprolite**			**Capped at**	**55000**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	4,728	950	2755	5411	1.96	4	174300	ppm
CuCap	4,728	950	2729	4991	1.83	4	55000	ppm
Difference in grade		0%	-1%		Difference in metal		-1%	
CuCNA	3,694	420	1733	4344	2.51	1	79500	ppm
CuCnCap	3,694	420	1733	4344	2.51	1	79500	ppm
CuRatio	3,694	64	57	31	0.55	0	100	ppm

Type	**6**	**Oxide Saprolite**			**Capped at**	**5000**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	2,918	192	298	548	1.84	0	16010	ppm
CuCap	2,918	192	290	393	1.35	0	5000	ppm
Difference in grade		0%	-3%		Difference in metal		-3%	
CuCNA	2,039	10	41	382	9.33	1	10910	ppm
CuCnCap	2,039	10	41	382	9.33	1	10910	ppm
CuRatio	2,039	6	12	15	1.32	0	100	ppm

Type	**8**	**Overburden**			**Capped at**	**1400**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	548	115	191	249	1.31	9	3980	ppm
CuCap	548	115	187	217	1.16	9	1400	ppm
Difference in grade		0%	-2%		Difference in metal		-2%	
CuCNA	430	26	55	163	2.95	1	2229	ppm
CuCnCap	430	26	55	163	2.95	1	2229	ppm
CuRatio	430	26	29	21	0.71	0	97	ppm

17.10.2 Mesones-Sofia - Composites

The same logic and methodology were used in compositing at Mesones-Sofia as at Conductora.

Gold by zone: After capping, the gold assay sample intervals were composited to six-meter lengths. The gold grades were down-hole composited using geological restrictions for some material types. Pertinent material types were overburden (material type 8) and dikes (material type 9) because there is a discontinuity between both of these and the primary gold mineralization. Both post-date mineralization, and while the dike is barren, the overburden has remobilized, dispersed, and/or re-concentrated gold. Sample data from dikes and overburden were used as hard boundaries for compositing. After compositing, the six-meter composites were coded from the cross-sectional gold zone interpretations. This effectively smoothed out, or “softened,” the hard boundaries. The impact of the hard boundary is further reduced later by weight-averaging the grades of the different zones into a “diluted average gold grade” within each block that straddled these boundaries. This set of composites was used for modeling gold and core recovery and for calculating distances, number of samples, and number of drill holes. Gold composite statistics are given in Table 17.21.

Copper, copper solubility and silver by material type: After capping, the copper assay sample intervals were composited to six-meter lengths. Material types were used to control down-hole compositing. These composites were not re-coded on section as the gold composites were. After compositing, the composites were back coded from the model with the relative elevation of the top of the mixed or sulfide saprolite unit. This relative elevation was used in modeling copper and copper solubility ratios in the saprolite units. Gold composite statistics are given in Table 17.21.

Table 17.21 Statistics by Zone (Au) and Type (Cu) of Conductora Composites

Zones	**21, 31, 41**	**Low-sulfide zone**			**Capped at**	**13**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	20,527	0.35	0.53	1.25	2.36	0.00	73	g/t
AuCap	20,527	0.35	0.51	0.75	1.46	0.00	13	g/t
Difference in grade		0%	-3%		Difference in metal		-3%	

Zones	**22, 32, 42**	**High-sulfide zone**			**Capped at**	**33**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	7,534	1.29	2.29	5.50	2.40	0.01	370	g/t
AuCap	7,534	1.29	2.17	3.32	1.53	0.01	33	g/t
Difference in grade		0%	-5%		Difference in metal		-5%	

Zone	**8**	**Overburden**			**Capped at**	**4**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	4,051	0.12	0.19	0.69	3.55	0.01	43	g/t
AuCap	4,051	0.12	0.18	0.30	1.61	0.01	4	g/t
Difference in grade		0%	-5%		Difference in metal		-5%	

Zone	**0**	**Outside Zones**			**Capped at**	**5**	**g/t**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Au	788	0.36	0.84	1.35	1.61	0.01	13	g/t
AuCap	788	0.36	0.78	1.08	1.38	0.01	5	g/t
Difference in grade		0%	-6%		Difference in metal		-6%	

Type	**1, 2, and 3**	**Bedrock and Saprock**			**Capped at**	**55000**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	16,517	1360	3104	4622	1.49	1	82700	ppm
CuCap	16,517	1360	3095	4552	1.47	1	55000	ppm
Difference in grade		0%	0%		Difference in metal		0%	
CuCNA	3,122	106	419	1142	2.73	0	22250	ppm
CuCnCap	3,122	106	419	1142	2.73	0	22250	ppm
CuRatio	3,118	10	18	20	1.08	0	100	ppm

Type	**4, 5**	**Sulfide and Mixed Saprolite**			**Capped at**	**55000**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	4,728	950	2755	5411	1.96	4	174300	ppm
CuCap	4,728	950	2729	4991	1.83	4	55000	ppm
Difference in grade		0%	-1%		Difference in metal		-1%	
CuCNA	3,694	420	1733	4344	2.51	1	79500	ppm
CuCnCap	3,694	420	1733	4344	2.51	1	79500	ppm
CuRatio	3,694	64	57	31	0.55	0	100	ppm

Type	**6**	**Oxide Saprolite**			**Capped at**	**5000**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	2,918	192	298	548	1.84	0	16010	ppm
CuCap	2,918	192	290	393	1.35	0	5000	ppm
Difference in grade		0%	-3%		Difference in metal		-3%	
CuCNA	2,039	10	41	382	9.33	1	10910	ppm
CuCnCap	2,039	10	41	382	9.33	1	10910	ppm
CuRatio	2,039	6	12	15	1.32	0	100	ppm

Type	**8**	**Overburden**			**Capped at**	**1400**	**ppm**	
	Valid N	**Median**	**Mean**	**Std. Dev.**	**CV**	**Min**	**Max.**	**Units**
Cu	548	115	191	249	1.31	9	3980	ppm
CuCap	548	115	187	217	1.16	9	1400	ppm
Difference in grade		0%	-2%		Difference in metal		-2%	
CuCNA	430	26	55	163	2.95	1	2229	ppm
CuCnCap	430	26	55	163	2.95	1	2229	ppm
CuRatio	430	26	29	21	0.71	0	97	ppm

17.10.3 Mesones-Sofia – Geostatistics and Estimation

MDA calculated variograms and correlograms at varying lags, cutoffs, azimuths and dips, and with separate and combined zones for Mesones-Sofia. Variograms parallel to the dominant mineralization-controlling geological fabric were used, namely 315° azimuth with a dip azimuth of 225° and dip of -65°. The variograms were calculated on composites of gold, copper, CN-soluble ratio, and silver. No grade restrictions were used in variogram calculations for any of the metals except silver. Silver has a small (~1%) population of very high-grade samples with no continuity. Consequently, the silver variograms portray the low-grade "disseminated" style of mineralization, which was modeled differently from the high-grade spikes.

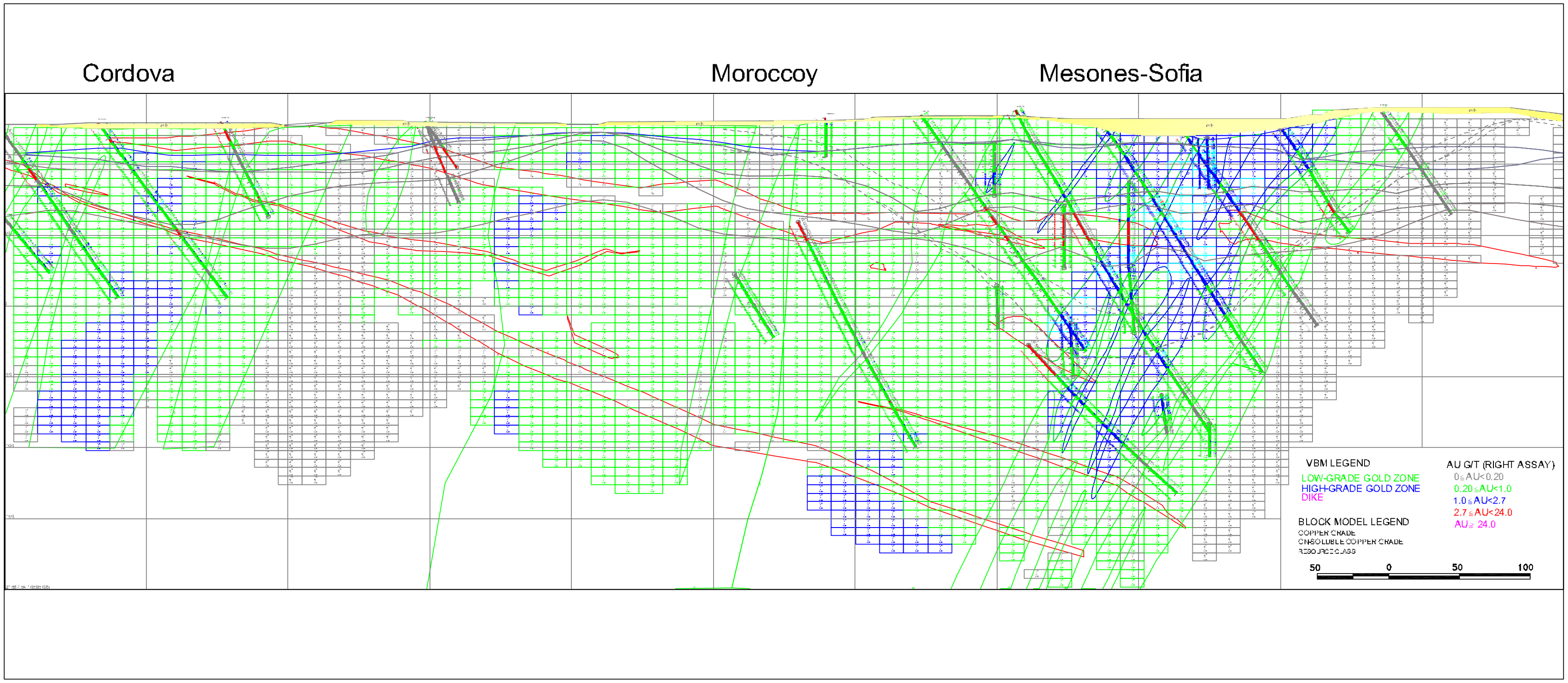
Ordinary Kriging was used for all estimates. The estimation parameters are given in Appendix B. Multiple passes were done for gold and the CN-soluble ratio to compensate for over-smoothing in the single pass. Estimation of CNSCu was done using the CNSCu-to-total-Cu ratios. As at Conductora, gold distribution has not been materially affected by weathering processes, except in the alluvium. Consequently, gold was modeled in gold domains that crossed the bedrock and saprolite contacts, but stopped at the overburden contact. A cross section of the gold model is given in Figure 17.4 and for the copper model in Figure 17.5.

MDA classified the resource by a combination of distance to the nearest sample, the number of samples used to estimate a block, the geological understanding and predictability of the resources, and the quality of the drill samples used, *i.e.*, core recovery. As gold is the dominant metal from a value standpoint and Crystallex has no mining rights to the copper, all blocks were classified based on gold (Table 17.22). The ranges used for resource classification were chosen based on an average of the directional gold variogram ranges.

Table 17.22 Criteria for Classification of Mesones-Sofia Resources

Class	Distance*	Min. No. of Samples	Min. No. Drill Holes
Measured*	0 to 10 m	2	1
Indicated	1 to 10 m	1	1
Indicated	20 to 40 m	2	1
Inferred	40 to 80 m	1	1

* See text in 17.10.3 for explanation; all overburden is classified as Inferred.



Crystallex International Corp.
Las Cristinas
Mesones-Sofia, Moroccoy and Cordova
Block Model Section 950 Au

Bolivar State Venezuela

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SCALE	as shown

FIGURE NO.
17.4

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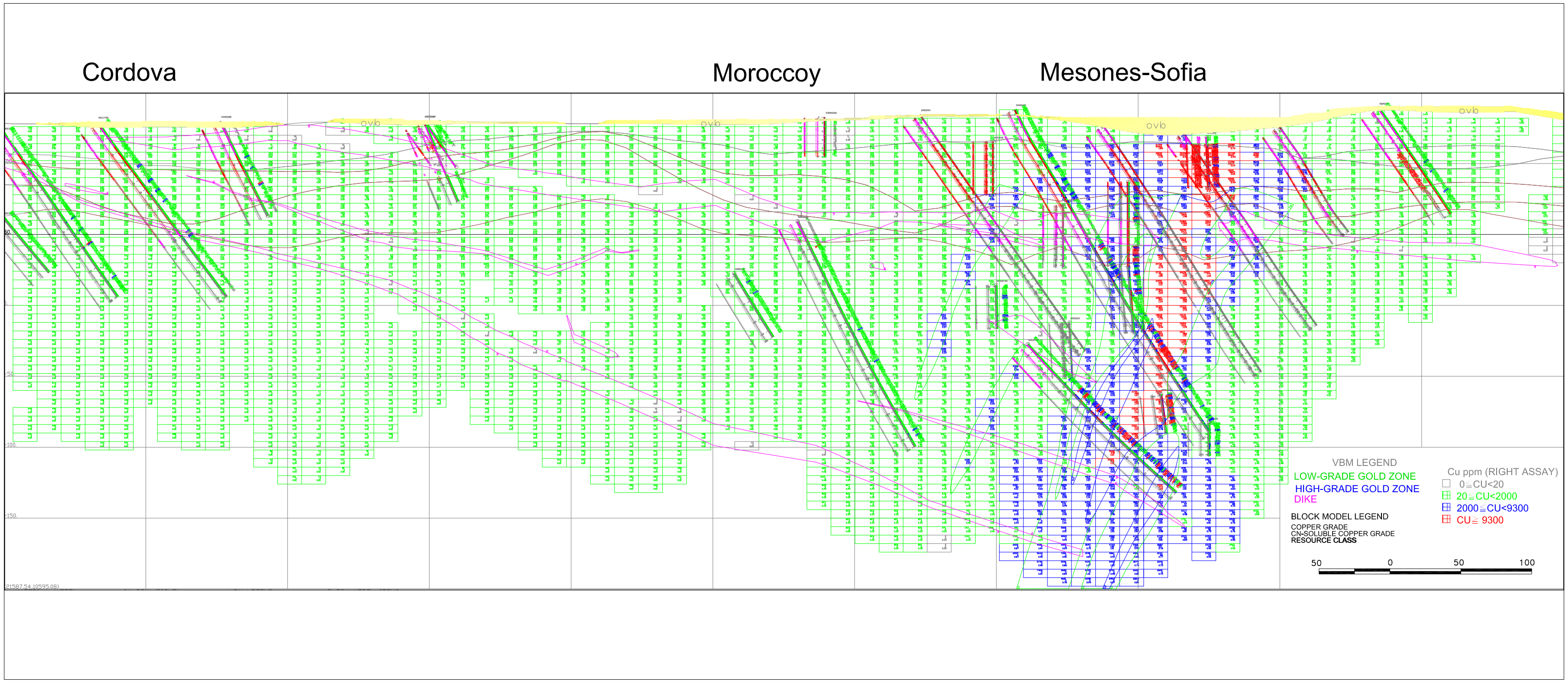

Crystallex International Corp.
Las Cristinas
Mesones-Sofia, Moroccoy and Cordova
Block Model Section 950 Cu
Bolivar State Venezuela

MINE DEVELOPMENT
ASSOCIATES
Reno Nevada

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FIGURE NO.
17.5

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DATE:9/13/2007

MDA modified the distances used for classification by the percent core recovery. It was shown in an earlier section of this report that core recovery affects gold and copper grades and introduces a bias in the saprolite. The lower core recovery decreases confidence in the results and therefore is introduced into the definition of Measured, Indicated, and Inferred. MDA modified the distance between the closest sample and the model block by the following relationship:

- Estimated core recovery between 80% and 100%, no factor;
- Estimated core recovery between 60% and 80%, distance multiplied by 1.1; and
- Estimated core recovery below 60%, distance multiplied by 1.2.

The modified distance was used for the classification scheme given in Table 17.22. Essentially, those blocks with estimated lower core recovery were downgraded in classification.

The classification and the estimation described above resulted in a Measured, Indicated and Inferred resource at Mesones-Sofia. Measured and Indicated resources are broken out in Table 17.23 and Table 17.24 and are combined in Table 17.25, while the total Inferred resources are given in Table 17.26.

Table 17.23 Mesones-Sofia Measured Resources
(Including Reserves*)

Mesones Sofia Measured								(rounded)
Cutoff (g Au/t)	**Tonnes**	**Gold** (g/t)	**Gold** Ounces	**Silver** (g/t)	**Copper** (ppm)	**CNSolCu** (ppm)	**Silver** Ounces	**Copper** Kilograms
0.2	16,285,000	0.85	445,000	0.72	2,778	546	375,000	45,241,000
0.4	11,910,000	1.05	402,000	0.78	3,240	677	299,000	38,590,000
0.5	**9,361,000**	**1.21**	**365,000**	0.82	3,612	776	246,000	33,813,000
0.6	7,566,000	1.37	334,000	0.85	3,938	873	207,000	29,797,000
0.7	6,283,000	1.52	307,000	0.89	4,235	967	179,000	26,608,000
0.8	5,397,000	1.65	286,000	0.92	4,471	1,046	159,000	24,129,000
0.9	4,772,000	1.76	269,000	0.94	4,661	1,109	144,000	22,238,000
1.0	4,240,000	1.86	253,000	0.96	4,801	1,150	131,000	20,357,000
1.5	2,264,000	2.42	175,800	1.01	4,991	1,409	73,400	11,299,000
2.0	1,234,000	3.00	118,900	1.05	5,405	1,733	41,800	6,670,000
2.5	741,000	3.52	84,000	1.06	5,752	1,928	25,100	4,263,000
3.0	462,000	4.01	60,000	1.10	5,972	2,056	16,000	2,761,000
3.5	289,000	4.47	41,000	1.13	5,849	1,683	10,000	1,689,000
4.0	178,000	4.91	28,000	1.14	6,131	1,688	7,000	1,091,000
5.0	53,000	6.08	10,000	0.93	5,908	813	2,000	316,000
Note: inconsistencies between tonnes, grade, and ounces are caused by rounding								

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Table 17.24 Mesones-Sofia Indicated Resources
(Including Reserves*)

Mesones Sofia Indicated								(rounded)
Cutoff (g Au/t)	**Tonnes**	**Gold** (g/t)	**Gold** Ounces	**Silver** (g/t)	**Copper** (ppm)	**CNSolCu** (ppm)	**Silver** Ounces	**Copper** Kilograms
0.2	73,955,000	0.59	1,412,000	0.58	2,025	341	1,374,000	149,744,000
0.4	45,125,000	0.78	1,136,000	0.64	2,453	448	934,000	110,706,000
0.5	**32,248,000**	**0.92**	**953,000**	0.69	2,742	513	711,000	88,432,000
0.6	23,349,000	1.06	798,000	0.73	3,066	595	545,000	71,586,000
0.7	17,289,000	1.21	673,000	0.77	3,419	687	427,000	59,101,000
0.8	13,002,000	1.37	571,000	0.81	3,772	779	339,000	49,045,000
0.9	10,256,000	1.51	497,000	0.85	4,064	873	280,000	41,678,000
1.0	8,322,000	1.64	438,000	0.88	4,281	948	235,000	35,625,000
1.5	3,656,000	2.20	258,700	0.93	4,634	1,200	109,400	16,939,000
2.0	1,812,000	2.69	156,700	0.96	4,976	1,438	55,700	9,014,000
2.5	885,000	3.20	91,000	1.03	5,482	1,760	29,200	4,853,000
3.0	433,000	3.72	52,000	1.07	5,232	1,647	15,000	2,266,000
3.5	211,000	4.24	29,000	1.10	5,326	1,369	7,000	1,122,000
4.0	101,000	4.80	16,000	1.18	5,499	1,439	4,000	555,000
5.0	30,000	5.87	6,000	0.74	5,211	549	1,000	154,000
Note: inconsistencies between tonnes, grade, and ounces are caused by rounding								

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Table 17.25 Mesones-Sofia Measured and Indicated Resources

(Including Reserves*)

Mesones/Sofia Measured and Indicated (rounded)

Cutoff	**Tonnes**	**Gold**	**Gold**	**Silver**	**Copper**	**CNSolCu**	**Silver**	**Copper**
(g Au/t)		(g/t)	Ounces	(g/t)	(ppm)	(ppm)	Ounces	Kilograms
0.2	90,240,000	0.64	1,857,000	0.60	2,355	378	1,749,000	194,985,000
0.4	57,036,000	0.84	1,538,000	0.67	2,618	496	1,233,000	149,297,000
0.5	**41,609,000**	**0.99**	**1,318,000**	0.72	2,938	573	957,000	122,245,000
0.6	30,916,000	1.14	1,132,000	0.76	3,279	663	752,000	101,384,000
0.7	23,572,000	1.29	981,000	0.80	3,636	762	606,000	85,709,000
0.8	18,399,000	1.45	857,000	0.84	3,977	857	498,000	73,174,000
0.9	15,027,000	1.59	766,000	0.88	4,253	948	424,000	63,915,000
1.0	12,562,000	1.71	691,000	0.91	4,456	1,016	366,000	55,982,000
1.5	5,919,000	2.28	434,400	0.96	4,770	1,280	182,900	28,238,000
2.0	3,046,000	2.81	275,600	1.00	5,150	1,557	97,500	15,684,000
2.5	1,626,000	3.35	175,000	1.04	5,605	1,837	54,300	9,116,000
3.0	896,000	3.87	111,000	1.09	5,614	1,858	31,000	5,028,000
3.5	499,000	4.37	70,000	1.11	5,628	1,551	18,000	2,811,000
4.0	279,000	4.87	44,000	1.15	5,903	1,598	10,000	1,646,000
5.0	83,000	6.00	16,000	0.86	5,659	719	2,000	470,000

Note: inconsistencies between tonnes, grade, and ounces are caused by rounding

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

Table 17.26 Mesones-Sofia Inferred Total Resources

(Including Reserves*)

Mesones Sofia Inferred (rounded)

Cutoff	**Tonnes**	**Gold**	**Gold**	**Silver**	**Copper**	**CNSolCu**	**Silver**	**Copper**
(g Au/t)		(g/t)	Ounces	(g/t)	(ppm)	(ppm)	Ounces	Kilograms
0.2	33,617,000	0.47	505,000	0.46	1,149	129	493,000	38,632,000
0.4	18,140,000	0.61	355,000	0.49	1,183	131	286,000	21,465,000
0.5	**10,245,000**	**0.74**	**242,000**	0.50	1,081	110	164,000	11,073,000
0.6	5,605,000	0.90	162,000	0.53	1,002	88	96,000	5,615,000
0.7	3,765,000	1.03	124,000	0.56	891	75	68,000	3,354,000
0.8	2,552,000	1.16	95,000	0.59	803	60	48,000	2,050,000
0.9	1,782,000	1.30	74,000	0.62	743	50	36,000	1,324,000
1.0	1,326,000	1.42	61,000	0.62	700	41	26,000	928,000
1.5	654,000	1.65	34,700	0.57	576	19	11,900	377,000
2.0	57,000	2.47	4,500	1.13	323	68	2,100	18,000
2.5	18,000	3.07	1,800	1.43	364	58	800	7,000
3.0	9,000	3.46	1,000	1.57	336	73	-	3,000
3.5	4,000	3.59	-	2.06	213	72	-	1,000
4.0	-	-	-	-	-	-	-	-
5.0	-	-	-	-	-	-	-	-

Note: inconsistencies between tonnes, grade, and ounces are caused by rounding

*Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.