Appendix 1

Cresson Project **Hydrogeochemical** Evaluation

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Mining and Safety

Prepared for:

Cripple Creek & Victor Gold Mining Company
Victor, Colorado 80860

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Project No. 1385L

AdrianBrown

Innovative Environmental Solutions

130 West Fourth Avenue
Denver, Colorado 80223
303.698-9080 Fax 303.698-9241
www.abch2o.com

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1. INTRODUCTION

The Cripple Creek Mining District ("District") is located in a seven square mile volcanic diatreme structure and contiguous rocks that have been altered by the diatremal volcanic activity in the central Colorado Rocky Mountains near the towns of Cripple Creek and Victor. Gold has been produced from this District for more than a century, primarily from underground mines operated between 1890 and 1940. To facilitate underground mining, the entire District was dewatered by a series of tunnels constructed between the late 1800s and 1942. In 1993 large-scale mining was restarted in the District using surface mining methods, with overburden storage near the mines, and ore processing in a valley leach facility ("VLF") at the south end of the District (the "Cresson Project").

Cripple Creek & Victor Gold Mining Company ("CC&V") is proposing a mine life extension, by mining an additional 360 million tons of rock in the District, of which approximately 110 million tons would be processed as ore in a phased extension of the valley leach system, and approximately 250 million tons of overburden would be used for mine backfill or placed in constructed overburden storage facilities.

The proposed extension will take place within the District. The geohydrology and geochemistry of the District have been evaluated in detail in submissions to the State of Colorado, Division of Reclamation Mining and Safety, most recently in support of Amendment No. 8 of the Cripple Creek and Victor Mining Company's Mine Permit (CC&V, 2000).

This report presents a detailed update of the system-wide understanding of the regional hydrologic and geochemical processes that control the quantity and quality of water that infiltrates into the diatreme and contiguous rocks and migrates vertically downward to the regional ground water system, and flows out of the Carlton Tunnel. Based on this update, an evaluation is made of the potential for current and proposed surface mining activities to affect the quality of water in the District.

The evaluation of the impact of mining on the hydrology system of the District was conducted in two parts:

1. Updated Regional Hydrologic Evaluation (Section 2); and
2. Hydrogeochemical Evaluation (Section 3).

2. UPDATED REGIONAL HYDROLOGIC EVALUATION

2.1 Hydrogeology

The principal geologic unit that contains the regional groundwater system associated with the District is the Cripple Creek Diatreme, a volcanic caldera approximately 7 square miles in area (Figure 1).

The diatreme is located within the host Precambrian-age Pikes Peak granite. Since the construction of the drainage tunnels, groundwater flow from the diatreme includes water drawn from the immediately adjacent granite. The diatremal and adjacent rocks contain the orebodies in the District, and therefore contain most of the past and proposed mining activities.

The diatremal rocks are moderately permeable, while the surrounding Pikes Peak granite is of low permeability. As a result, the diatreme is an enclosed regional groundwater regime, with limited hydrologic interaction with the surrounding rockmass. This has been observed from the earliest times of evaluation, in a study performed by Lindgren and Ransome (1906):

> "The records of the various drainage tunnels all suggest the same conclusion, namely that the volcanic neck, filled with breccia and eruptive rocks, contains a body of water which is prevented from escaping into the streams of the region by the relatively impervious character of the older crystalline rocks of the prevolcanic plateau. The relation is suggestive of a water-soaked sponge set in a glass cup.
>
> "For all practical purposes, therefore, the underground water of the central part of the Cripple Creek district is to be regarded … as an underground lake or pond bounded by the steep walls of the old volcanic throat."

2.2 Groundwater System

The source of groundwater in the vicinity of the Cripple Creek Diatreme is precipitation. The precipitation impinges on the surface of the District, and then runs off, evaporates, transpires, or infiltrates. Percolating water within and immediately contiguous to the diatreme ultimately reaches the deep saturated groundwater system, which in turn flows to the Carlton Tunnel. The groundwater flow system that results from this behavior is presented schematically in Figure 2.

2.3 Climate

The hydrologic system in the District is controlled by precipitation. The precipitation in the District has been measured since the current surface mining has been undertaken, and is presented by month in Table 1. In the period 1992-2007 area received an average of 19.58 inches per year of total precipitation, one quarter fell in the winter and spring (mostly as snow), and three quarters fell in spring and summer.

2.4 Surface Runoff

Little water runs off from the Diatreme footprint, except in peak precipitation and snowmelt events.

The principal surface water outflow that exits from the diatreme occurs to the east from the footprint of Grassy Valley. This flow is highly seasonal, and appears to be primarily due to surface runoff during high precipitation events. The flow has been monitored approximately monthly at three stream locations in the valley by CC&V since 1997. The monthly average flow is presented in Table 2, and plotted by month in Figure 3. The annual average surface flow from the lowest monitor point in the valley is 40 gpm. In the overall catchment area of Grassy Valley of 2 square miles (above GV-3) the basin surface water yield averages 0.60 inches per year, or 3.1% of precipitation. This is a small surface water yield for Rocky Mountain terrains at this altitude.

2.5 Groundwater Flow

2.5.1 Collection and measurement

Three principal tunnels were driven into the diatreme as follows:

1. Moffat Tunnel. The Moffat Tunnel was driven in 1903 from a portal elevation of 8,800 feet from the south of the town of Cripple Creek to the north-east, and is approximately one mile long (Lindgren and Ransome, 1906). Flow from the tunnel was initially as much as 11,000 gpm, reducing to less than 2,000 gpm by 1906. Flow from the tunnel ceased when tunnels at lower elevations reduced the water level in the diatreme below the elevation of the tunnel.
2. Roosevelt Tunnel. The Roosevelt Tunnel was driven between 1910 and 1917 from a portal located southwest of the town of Cripple Creek, at an elevation of 8,100 feet (Sheldon, 1915). The tunnel was ultimately driven 4.6 miles, reaching into the heart of the District. Flow from the portal rose to nearly 9,000 gpm by 1917, dropping back by the end of that year to about 4,000 gpm (Henderson, 1926). Flow from this tunnel largely ceased when the Carlton Tunnel was installed at a lower elevation.
3. Carlton Tunnel. The Carlton Tunnel was driven in 1939-1942 from a portal elevation of 6,890 feet, with the portal located immediately downstream of the confluence of Cripple Creek with Four Mile Creek (Vivian, 1940). The tunnel was ultimately driven approximately 6 miles and enters the southern portion of the diatreme. Flow from the tunnel was estimated to be 125,000 gpm after breakthrough into the Portland No. 2 Shaft, but reduced rapidly to its then-normal flow rate of 7,000 gpm (Vivian, 1941). The flow in recent decades has been approximately 1,660 gpm, and appears to be at approximate steady state.

These tunnels (today the Carlton Tunnel) capture essentially all the groundwater in the District. The water from the Carlton Tunnel is discharged under permit to Four Mile Creek, immediately downstream of its confluence with Cripple Creek.

The flow from the Carlton Tunnel represents the collection of groundwater infiltration to the diatremal materials, as described above. The flow from the Carlton Tunnel has been measured (and monitored) continuously since 1989. The monthly flows are presented in Table 3, and presented together with the precipitation in Figure 4.

In 1996 major modifications were made at the Carlton Tunnel portal. The four settling ponds were taken out of service and cleaned of sediment, a fifth pond was installed, and the flow measurement flume and sampling location was moved to immediately downstream of the settling ponds. The reported flow readings in 1996 show a sharp reduction, and flows in subsequent years appears to be consistently lower than flows prior to these changes (Table 3; Figure 4).

For the purposes of evaluating the regional groundwater system, the portal flow is required. A correction has been applied to the measured flow since these changes were made, based on comparison of years of similar total precipitation before and after the measurement change. The measured and corrected flows are shown by month in Figure 4. The evaluation concludes that:

Average Carlton Tunnel portal flow 1992-2007 = 1662 gpm.

2.5.2 Infiltration

Taken as an infiltration rate over the footprint of the diatreme, the average Carlton Tunnel Portal flow represents an infiltration rate of 6.99 inches per year, or 37% of the total precipitation of 19.58 inches. This is a high infiltration rate; the expected value would be no more than 25%, based on infiltration evaluations of mountain basins in Nevada (Maxey and Eakin, 1949).

Evaluation of the data indicates that this infiltration rate is not in fact representative of the groundwater system in the District. On an annual basis, the infiltration rate has been computed in the same way, and the results presented on Figure 5. This figure shows that the groundwater yield decreases with precipitation, indicating that processes other than precipitation and infiltration are influencing the flow rates at the Carlton Tunnel. In most natural groundwater systems, the infiltration is rate increases with precipitation: the higher the precipitation the higher the infiltration. The data of Maxey and Eakin (1949) are plotted on Figure 5 to show the more normal relationship.

The reason for this behavior can be seen by inspection of the flow from the tunnel (Figure 4). A large component of the flow from the tunnel is constant, and is derived water that spends many years in transit from the point of infiltration to the Carlton Tunnel[1]. This flow rate is constant, and appears to comprise approximately 75% of the total flow (1229 gpm). The remaining 25% of the flow from the tunnel (433 gpm) responds rapidly to precipitation, and must infiltrate into the diatreme relatively close to the location of the Carlton Tunnel, and either directly enter the tunnel, or rapidly raise the head in the diatreme that is directly hydraulically connected to the tunnel such as to rapidly increase the flow to the tunnel.

This behavior is a result of the conformation of the diatreme. The subsurface shape of the diatreme has been evaluated by CC&V from drilling, geophysical, and underground mapping. The results are shown on Figure 9. The diatreme is made up of three main vertical volcanic feeder tubes, one to the south (actually made up of two feeders that coalesce below the Carlton Tunnel level), one to the northwest, and one to the northeast beneath Grassy Valley. The Carlton Tunnel penetrates and directly drains the relatively high permeability southern feeder. It indirectly drains the other feeders, by drawing flow to the south feeder through the relatively low permeability granitic zones that separate them.

Analysis of the regional groundwater system drained by the Carlton Tunnel has been conducted using this geometry, and is presented in Attachment 1. The analysis uses the following conceptual model:

1. Infiltration through the ground surface above the south feeder directly supplies the Carlton Tunnel, with little time lag. This flow varies with time, proportional to the precipitation, which builds up in the "reservoir" represented by the south feeder.

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[1] It is noted that tritium sampling of water from the Carlton Tunnel water in 1997-8 indicates an average of approximately 6 tritium units. Surface water and shallow groundwater in the district had a corresponding average of 20 tritium units (ABC, 1998). This indicates that approximately 75% of the water flowing from the tunnel entered the ground before 1950 (i.e. before significant atmospheric testing of hydrogen bombs that introduced high concentrations of tritium into the atmosphere) and the rest entered the ground after 1950. This is consistent with the slow-flow concept presented here.

2. The south feeder also acts as a "well", drawing water through the surrounding granitic materials from the other diatreme feeders and from the granite surrounding the diatreme. This flow is essentially invariant over time, as the head gradients driving the water to the south feeder from the remainder of the diatreme and the surrounding granite are relatively constant.

The results of the analysis are as follows:

1. Infiltration into the diatreme and the surrounding granitic rockmass is estimated to be constant at approximately 25% ± 5% of total precipitation, or 5 ± 1 inches per year. This is an average annual flow of 1,662 gpm. The diatremal volcanic rocks exhibit hydraulic conductivity values that would accept up to 200 inches per year of infiltration, so this infiltration can readily enter the subsurface system.
2. Carlton Tunnel drains water from an area that is approximately 1.7 times the area of the entire diatreme, extending out into the surrounding granite approximately 2,000 feet from the perimeter of the diatreme (Figure 6). This infiltration collection area includes most of the Grassy Valley catchment. The amount of water that infiltrates in this area (5 inches per year) can be largely accepted by the granite, which has exhibited hydraulic conductivity values that would accept approximately 4-12 inches per year of infiltration (3×10^{-7} to 1×10^{-6} cm/sec).
3. Infiltration behavior is modified by mining, causing a reduction in the rate of infiltration to disturbed areas, due mainly to sequestration of infiltration in overburden, placed in storage areas or used as mine backfill, and reductions of direct flow paths (shafts, adits) for surface water to enter the diatreme. Mining activities associated with the Cresson Project have occurred in the south feeder area of the diatreme since 1993, and now attenuate a substantial fraction of the water that previously reported rapidly to the Carlton Tunnel. This causes a temporary reduction in flow to the Carlton Tunnel, and a permanent reduction in peak flows.

2.6 Impact of Mining

The infiltration to the regional groundwater system is driven by precipitation. The quantity of water that infiltrates to the groundwater system is affected by the surface condition; the more porous and less vegetated the surface, the higher the infiltration that would be expected. Surface mining in the diatreme since 1993 has modified the surface conditions, creating relatively large areas of surface mines, overburden storage, and backfilled mines, as shown in Figure 7. Based on these photographs, the surface disturbance in 1990 was 13% of the diatremal area, and in 2005 it was 33% of the diatremal area.

The impact that the 15 years of mining to date has had on infiltration can be determined by comparing the regional groundwater flow (as observed at the Carlton Tunnel portal) with precipitation. The results of this comparison are shown in a double-mass curve, presented in Figure 8. These results are interpreted as follows.

The slope of the flow-precipitation double mass curve does not change during the entire period of record, except for some minor short-term excursions during extremely high or extremely low precipitation years. This constant slope shows that on average the annual volume of water that infiltrates remains the same proportion of the annual precipitation. This shows that, despite the increase in disturbed area caused by surface mining, no net change to basin yield rates has occurred[2].

The empirical evidence shows that the evapotranspirative losses from disturbed land in this District are in fact approximately the same as those from naturally vegetated surfaces. This may be a result of the following:

1. There is little precipitation in winter, and no infiltration due to the surface being frozen at this altitude.
2. There is little runoff in the area at any time: before, during or after disturbance, so runoff is not a significant water balance item.
3. In summer and fall potential evapotranspiration greatly exceeds precipitation. As a result, there is apparently little difference between losses due to evaporation (from bare surfaces) and evapotranspiration (from vegetated surfaces); both evaporate most (>75%) of the available moisture. Thus infiltration to the diatreme is not significantly changed by disturbance.
4. Overburden in storage areas and used for mine backfill is placed essentially dry. Initially it has high permeability, and the infiltration rate is expected to be high. The water infiltrating to the pile will first be sequestered until the moisture capacity increases to the field capacity of 7.5% by volume, based on column draindown tests performed on coarse and fine mined rock (CC&V, 2000). Thus the high infiltration is approximately balanced by the uptake of water to reach field capacity.

The behavior exhibited since the inception of surface mining in the Cripple Creek District is not expected to change in the future. Accordingly, no net increase in flow from the Carlton Tunnel is expected as a result of changing surface disturbance patterns in the future.

2.7 Conclusion

No increase in the average regional groundwater flow rate is occurring due to the current mining (1993-2007), and no increase will occur as a result of the proposed MLE mining (2008-2016).

3. HYDROGEOCHEMICAL EVALUATION

3.1 Hydrogeochemical System

Mined overburden materials from the proposed MLE will be placed in completed mines as backfill and will be placed in overburden storage facilities in Squaw Gulch and Grassy Valley. Infiltration through these materials and through the remaining surface mines will mobilize any soluble products of oxidation of sulfides in the overburden and transport them into the subsurface diatremal rock. The flow passes

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[2] This was not the expected outcome. In earlier evaluations of the diatremal flow system presented in support of Amendments 6, 7, and 8, it was expected that an increase in the disturbed area would increase infiltration to the diatreme, thus increasing the flow from the diatreme until the mining area was reclaimed, and the evapotranspiration returned to approximately the pre-mining condition.

vertically through approximately 3000 feet of unsaturated rock and will contact neutralizing materials in the rock. The flow will then pass laterally to the Carlton Tunnel through the saturated portion of the diatreme, contacting further neutralizing materials. Finally the water flows through the Carlton Tunnel into Four Mile Creek. This hydrogeochemical system is shown schematically in Figure 10.

This section of the report describes and quantifies the components of the hydrogeochemical system, and the impact that it has on groundwater quality.

3.2 Mining Operations

3.2.1 Mining

The proposed Mine Life Extension ("MLE") is illustrated in Figure 11. The MLE will increase the production of the mine by extraction of a total of 360 million tons of additional material. The increased production will be mined from four locations: Wildhorse Extension ("WHEX"), Globe Hill, Schist Island, and South Cresson. A total of approximately 110 million tons of the material mined is ore, and will be crushed and placed on Phases I, II, and IV the new Phase 5 of the VLF for processing. The remaining approximately 250 million tons is overburden.

The proposed MLE mining will expand the amount of material that will be mined in the project. From the present (2008) to the anticipated end of the MLE (2016) a total of approximately 610 million tons of additional material are to be mined from six mine locations: Wildhorse, WHEX, Globe Hill, Schist Island, Main Cresson, and South Cresson. A total of approximately 251 million tons of the material mined is ore, and will be crushed and placed on the existing and extended VLF for processing. The remaining approximately 359 million tons is overburden.

Table 4 summarizes the proposed mining, by location, elevation, rock type, and material type (ore or overburden).

3.2.2 Overburden Storage

The proposed MLE mining will increase the amount of overburden material that will be produced from the project by a total of approximately 250 million tons, and will increase the total overburden to be produced in the balance of the project to approximately 359 million tons. The overburden will be stored on the site, either in mines as backfill or on the extended Squaw Gulch Overburden Storage Area ("SGOSA") and the new East Cresson Overburden Storage Area ("ECOSA"). Figure 11 shows the SGOSA and ECOSA locations.

Table 4 summarizes the proposed mining, by location, rock type, and classification (ore and overburden).

The disposition of the overburden produced from the mining project is summarized in Table 5. Approximately 77% of the overburden will be used as backfill for surface mines, while the remaining 23% will be placed in the SGOSA and ECOSA locations.

3.3 Impact of Sulfide Oxidation

The mining activities associated with the Mine Life Extension have the potential to impact groundwater quality as a result of oxidation of sulfide minerals in the material mined, and of materials exposed in the mine walls that remain after closure. Oxidation of sulfides generates acid and mobilizes metals, which have the potential to impact water quality. The impacts that will result from the mining activities associated with the MLE are evaluated in this section.

3.3.1 Method

Acid base accounting was conducted for the entire rockmass that will be mined from now until the completion of the proposed MLE. The methodology used was to identify for each 50' x 50' x35' block of material that will be mined the expected total sulfur and total carbon concentrations, based on a total of 30,758 LECO furnace analyses. The sulfur and carbon concentrations were converted to Acid Generating Potential ("AGP") and Acid Neutralizing Potential ("ANP") respectively, and Acid Base Accounting ("ABA") computations performed on the resulting totals by mine and by material type (ore or overburden) to determine the release of constituents from the ore and overburden during and after mining. The impact of release is evaluated after passage through the District to the Carlton Tunnel portal.

3.3.2 Acid Generating Potential

AGP is the potential for a material to generate acid by oxidation of sulfides. The AGP is related to the total sulfur content of a rock material; the total sulfur measurement includes the sulfides that oxidize. The AGP for the rock is computed from the total sulfur concentration by a three step process as follows:

1. Conversion of Total Sulfide to Maximum AGP. The maximum AGP is computed by assuming that the entire sulfur content of the sample is in pyrite form (as the pyrite content is not generally measured in LECO analyses), and that the pyrite form oxidizes as follows:

   $$FeS_2 + 15/4\ O_2 + 7/2\ H_20 = Fe(OH)_3 + 2\ SO4^{2-} + 4\ H^+$$

   Stoichiometric evaluation shows that:

   $$AGP_{MAX} = 31.2 * S_{TOTAL}$$

   where: AGP_{MAX} = maximum acid generating potential (t $CaCO_3$/kt)
   S_{TOTAL} = total sulfur content (%)

2. Correction of Total Sulfur to Pyrite. Not all the sulfur in the samples is present in the form of pyrite. A correction for the amount of pyrite in a sample was performed (Attachment 2). A total of 644 LECO analyses were evaluated for which total sulfur speciation (into sulfate sulfur, pyritic sulfur, and non-extractable sulfur) were available. The relationship between total sulfur and pyritic sulfur was found to be a function of total sulfur content, as follows, for each rock type:

   Tertiary (Volcanic) Rock: $S_{PYRITE}/S_{TOTAL} = 0.1074 * \ln(S_{TOTAL}) + 0.7711$
   Precambrian (Granitic) Rock: $S_{PYRITE}/S_{TOTAL} = 0.1123 * \ln(S_{TOTAL}) + 0.6187$

where: S_{PYRITE} = pyritic sulfur content (%)
S_{TOTAL} = total sulfur content (%)

3. Correction of Pyrite to Reactive Pyrite. A total of 14 samples of graded sulfide content were subjected to detailed evaluation to determine the extent to which the pyrite in the sample was actually oxidizable (Attachment 3). The method used was to analyze the sample for pyritic sulfur, and then convert the pyritic sulfur content to AGP as above. The amount of the pyrite that is capable of oxidation in the same samples was then determined by performance of the Net Acid Generation ("NAG") test. In this test a finely ground sample is aggressively oxidized with concentrated hydrogen peroxide, and the amount of calcium carbonate required to neutralize the net produced acid is determined by titration.

   The reactivity of the pyrite is[3]:

   $$S_{REACTIVE}/S_{PYRITE} = AGP_{NAG} / AGP_{ABA} = (NAG_{NAG} + ANP_{ABA}) / AGP_{ABA} = 76.8\%$$

   where: $S_{REACTIVE}$ = reactive pyritic sulfur content (%)
   S_{PYRITE} = pyritic sulfur content (%)
   AGP_{NAG} = Acid generation potential from NAG test (t $CaCO_3$/kt)
   NAG_{NAG} = Net acid generation potential from NAG test (t $CaCO_3$/kt)
   AGP_{ABA} = Acid generation potential from ABA test (t $CaCO_3$/kt)
   ANP_{ABA} = Acid neutralization potential from ABA test (t $CaCO_3$/kt)

The AGP for any sample is computed as the product of these three factors:

$$AGP = AGP_{MAX} * S_{PYRITE}/S_{TOTAL} * S_{REACTIVE}/S_{PYRITE}$$

Thus:

Tertiary (Volcanic) Rock:

$$AGP = 31.2 * S_{TOTAL} * [0.1074 * \ln(S_{TOTAL}) + 0.7711] * 0.768$$
$$AGP = S_{TOTAL} * [2.573 * \ln(S_{TOTAL}) + 18.477]$$

Precambrian (Granitic) Rock:

$$AGP = 31.2 * S_{TOTAL} * [0.1123 * \ln(S_{TOTAL}) + 0.6187] * 0.768$$
$$AGP = S_{TOTAL} * [2.691 * \ln(S_{TOTAL}) + 14.825]$$

For the typical total sulfur content of mined overburden of 1.17% the AGP computes to:

$$AGP = 18.9 * S_{TOTAL}$$

3.3.3 Acid Neutralizing Potential

ANP is computed from the carbon content of the sample. All carbon in the samples from the District is inorganic, and derives from calcite and dolomite, both of which are reactive carbonates. Accordingly,

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[3] The ANP is added to the NAG to adjust the NAG value to the AGP value for the NAG test, as any neutralization capacity in the NAG test is consumed by the produced acid.

the ANP is determined from the molecular weight of calcium carbonate and unit conversion from carbon percent, as follows:

$$ANP = [MW_{CaCO3} / MW_{C}] * [C_{TOTAL} (\%)] * [10 (ton/kt) / (\%)]$$

where: ANP = Acid neutralizing potential (t $CaCO_3$/kt)
MW_{CaCO3} = Molecular weight of $CaCO_3$ (g)
MW_{C} = Molecular weight of carbon (g)
C_{TOTAL} = Total carbon in sample (%)

Thus:

$$ANP = 83.42 * [C_{TOTAL} (\%)]$$

3.3.4 Acid-Base Accounting

Acid-Base Accounting (ABA) analysis has been performed on the materials that will be produced during the MLE, divided by mine and material (overburden and ore). The results are shown in Table 6.

3.3.5 Overburden

Acid-Base Accounting has been performed for the overburden materials produced during the MLE. The results of this evaluation are presented in Table 6.

The overburden material to be stored from 2008 to the end of the proposed MLE will have the following ABA characteristics:

Total amount of overburden	359,000,000 tons
Carbonate	1.43% by weight[4]
Pyrite (reactive)	1.29% by weight[5]
Acid Neutralizing Potential	14.3 tons $CaCO_3$/kiloton rock
Acid Generating Potential	21.6 tons $CaCO_3$/kiloton rock
Net Neutralizing Potential	-7.2 tons $CaCO_3$/kiloton rock
Net Neutralizing Requirement	2,619,000 tons $CaCO_3$

Approximately 359 million tons of overburden will be produced in the duration of the Mine Life Extension. The weighted average net neutralizing capacity of the exposed wall material is -7 tons $CaCO_3$ per 1000 tons of material, indicating a slight net acid generating propensity. It will require a maximum of 2.6 million tons of calcium carbonate equivalent in the diatreme to neutralize the products of oxidation originating in the exposed mine walls.

The effect of the mining process is to move forward in time the oxidation of the sulfides in this material. If the material were to remain un-mined, the sulfide would over time oxidize naturally, and the products

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[4] To convert from carbon (in table) to calcium carbonate multiply by 8.34.

[5] To convert from sulfur (in table) to pyrite multiply by 1.87.

would require the same mass of natural carbonate for neutralization. In this respect, the mining process has no net impact.

The strategy for storage of overburden in the project considers the ABA status of the overburden materials as follows:

1. Precambrian (Granitic) Material: Precambrian material contains the lowest concentration of carbonate, and some of the higher average pyrite contents. To minimize impact of this material to the environment, it will be stored in two locations:
    a. Below ground surface in the Main Cresson Mine. The Main Cresson Mine provides high carbonate, net acid neutralizing material immediately underlying the placed overburden. Placement in an existing mine also reduces the likelihood that air will circulate through the overburden material after placement, which will limit oxygen availability for sulfide oxidation. Finally, subsurface placement eliminates the possibility that infiltrating water will emerge at the ground surface, rather than flowing through the diatreme to the Carlton Tunnel.
    b. Above ground in the ECOSA. The ECOSA is an engineered facility with a high-clay compacted cover that will minimize infiltration to the overburden material, and will minimize entry of air into the stored rockmass. Overburden stored in this facility will be subject to minimal sulfide oxidation, and minimal leaching.
2. Low NNP Tertiary (Volcanic) Material: The volcanic overburden to be mined from the South Cresson mine exhibits relatively high pyrite content, and correspondingly low NNP. All of this material will be placed below ground surface, to limit access to air, and to ensure that any water from the overburden is directed to the diatremal rock. Current plans propose location of the overburden in the Main Cresson Mine, which is underlain by net neutralizing material, is below ground surface, and constrains any infiltrating water to exit to the diatreme.
3. Balance of Overburden. The balance of the overburden is stored with the following preference:
    a. Below ground storage in surface mines, to act as mine backfill, and to also minimize air circulation through the material.
    b. Above-ground storage in the SGOSA, where all water reports to the diatreme, and rapidly contacts the overwhelmingly large carbonate content present in the diatreme.
    c. Above-ground storage in the ECOSA, where contact with water and air is minimized, in turn minimizing sulfide oxidation and potentially impacted seepage. The ECOSA is also located above high carbonate volcanic materials in the diatremal rocks beneath, providing further protection against potential discharge of acidic seepage. No significant sulfide oxidation is expected in overburden placed in this engineered facility.

3.3.6 Ore

Ore materials are mixed with lime, agglomerated, and the resulting prepared ore placed on the Valley Leach Facility for cyanide leach recovery of gold and silver values. In this process, approximately 11-12 lb of lime per ton is added to the ore material. This addition has the effect of modifying the net neutralizing potential of the ore materials (Table 6) as follows:

Total amount of ore	251,000,000 tons
Carbonate	2.2 % by weight
Lime added	0.6% by weight
Pyrite (reactive)	1.5 % by weight
Acid Neutralizing Potential (ROM)	21.9 tons $CaCO_3$/kiloton rock
Lime ANP (added)	5.8 tons $CaCO_3$/kiloton rock
Total ANP (ROM + Lime)	27.7 tons $CaCO_3$/kiloton rock
Acid Generating Potential (ROM)	25.7 tons $CaCO_3$/kiloton rock
Net Neutralizing Potential	+2.0 tons $CaCO_3$/kiloton rock
Net Neutralizing Requirement	0 tons $CaCO_3$

Accordingly the ore is expected to remain net neutralizing, at relatively elevated pH, for the duration of the processing and after closure.

Closure comprises cyanide destruction, followed by breaching of the liners of the pond, and reclamation of the surface of the facility. After closure, water infiltrating through the surface cover will pass through the ore material and enter the subsurface materials. This seepage water is expected to reach the diatreme, either through direct seepage to the diatremal rockmass, or by collection by the drawdown cone in the granitic rockmass around the diatreme caused by flow from the Carlton Tunnel.

The effect of mining the ore material is to reduce the total natural carbonate that would otherwise be required to neutralize the products of natural oxidation of the pyrite contained within the ore. If the material were to remain un-mined, the sulfide would over time oxidize naturally, and the products would require a total of 969,000 tons of carbonate for neutralization. In this respect, the mining process has a positive impact.

3.3.7 Exposed Mine Walls

At the completion of mining, the East Cresson, North Cresson, Wildhorse, Wildhorse Extension, and Schist Island mines will be substantially backfilled with overburden materials. The South Cresson, and Globe Hill mines will be un-backfilled, and the Main Cresson Mine will be partially backfilled.

Accordingly, there will remain some areas of mine wall which will remain exposed to the atmosphere, and will be subject to sulfide oxidation. The sulfide oxidation products from these walls have been evaluated by the following process:

1. The areas of the mine walls that will be exposed at the end of mining and mine reclamation were determined by measurement from the mine closure plan (Figure 16).
2. The depth within the wall of the walls was assumed to be equal to the depth of a single bench excavation (35 feet), as this is the maximum wall penetration that could be affected by blasting, unloading, and mining activities. The oxidation of pyrite in materials deeper in the wall will not be changed from its pre-mining condition as a result of mining.

3. The tonnage of material that may be exposed to additional pyrite oxidation due to mining is computed from the affected area, the depth of additional weathering, and the specific weight of the in-place material (14 cu.ft./ton).
4. Acid-Base Accounting is performed for the material in each mine wall, assuming that the material is the same as the material to be excavated as overburden from the mine.

The acid-base accounting for exposed mine walls is presented in Table 7, and are summarized as follows:

Affected rockmass	49,000,000 tons
Carbonate	1.79% by weight
Pyrite (reactive)	1.33% by weight
Acid Neutralizing Potential	18.0 tons $CaCO_3$/kiloton rock
Acid Generating Potential	22.2 tons $CaCO_3$/kiloton rock
Net Neutralizing Potential	-4.2 tons $CaCO_3$/kiloton rock
Net Neutralizing Requirement	207,000 tons $CaCO_3$

The mine walls contain approximately 49 million tons of material that may be exposed to accelerated weathering due to mining. The weighted average net neutralizing capacity of the exposed wall material is -4 tons $CaCO_3$ per 1000 tons of material, indicating a slight net acid generating propensity. It will require a maximum of 207,000 tons of calcium carbonate equivalent in the diatreme to neutralize the products of oxidation originating in the exposed mine walls.

The effect of the mining process is to move forward in time the oxidation of the sulfides in this material. If the material were to remain un-mined, the sulfide would over time oxidize naturally, and the products would require the same mass of natural carbonate for neutralization. In this respect, the mining process has no net impact.

3.4 Weathering of Overburden Material

3.4.1 Accelerated Weathering Tests of Overburden Material

Overburden material mined in the future at the Cresson Project will be either stored on the surface in overburden storage areas, or used as backfill for the surface mines. After placement the overburden will be subject to weathering to varying degrees, as a result of contact with atmospheric oxygen and infiltrating water from precipitation.

The environmental effects of weathering of the overburden has been evaluated in detail, by the performance of a total of 42 long-term accelerated weathering tests of samples of the rockmass in and adjacent to the diatreme. The testing protocol used was ASTM Test Method D5744-96, Standard Test Method for Accelerated Weathering of Solid Materials Using a Modified Humidity Cell (ASTM, 2001). The results of these tests are presented in Attachment 4.

3.4.2 Pyrite Oxidation

All samples tested exhibited pyrite oxidation as the principal chemical reaction affecting the material when exposed to the atmosphere and to water. In general, the material produced progressively more acidic effluent, with the pH of the effluent generally decreasing with increasing sulfide content of the samples tested (Figure 13). When the samples contained measurable acid neutralizing material (almost entirely carbonate) the decline in pH of the effluent was either prevented or delayed due to neutralization of the acid products of pyrite oxidation.

The rate of oxidation of pyrite is a function of pyrite concentration, particle size, and rock type. Pyrite oxidation in rock materials takes place by the following process (Davis and Ritchie, 1986). Oxygen diffuses into each particle of the overburden, and oxidizes sulfides within the particle using water present in the particle. The products of reaction diffuse back out from the location of the reaction to the particle surface, where the concentration of the reaction constituents is lower than at the reaction site. The rate limiting processes are the diffusion of atmospheric oxygen into the particles of the overburden, and the diffusion of the reaction products out of the particles. Rate of diffusion is controlled primarily by the diffusion path length[6]. Accordingly, the average pyrite oxidation rates measured in the humidity cell tests ("HCTs") were converted to pyrite oxidation rates for the Run of Mine ("ROM") overburden material by scaling using the mean particle diameter of the HCT test material (~4 mm) and the mean particle diameter of the ROM overburden (~100 mm):

$$R_{ROM} = R_{HCT} * (D_{HCT} / D_{ROM})$$

where: R_{ROM} = Pyrite oxidation rate in run of mine material (mg/kg/yr)
R_{HCT} = Pyrite oxidation rate in humidity cell test material (mg/kg/yr)
D_{HCT} = mean diameter of humidity cell test material (mm)
D_{ROM} = mean diameter of run of mine material (mm)

The pyrite oxidation rate in the ROM overburden is presented in Figure 14. These are maximal rates of oxidation, as they assume that the overburden will have excess oxygen for the reaction, and sufficient water flow to remove the products of reaction from the reaction sites. For the overburden material placed in surface mines and in the ECOSA neither of these conditions is likely to be met.

The expected pyrite oxidation rate is weakly proportional to the pyrite content of the rock, and the rate of oxidation can be summarized as follows:

Pyrite oxidation rate – volcanic overburden..............0.45% of total pyrite content per year
Pyrite oxidation rate – granitic overburden ....0.10% of total pyrite content per year

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[6] This is true for a planar diffusion pathway, as for example into a plate of material. For approximately spherical particles, as in this case, the diffusion rate is a function of the diffusion pathway length and the particle surface area. The rate relationship assumed here is conservative, as it ignores the surface area effect, and produces the highest rate estimate available from the data.

This rate has been applied to the average rock conditions in all rock affected by mining: ore, overburden, and affected wall rock (Table 8). The maximum rate of sulfate generation by pyrite oxidation in the 659 million tons of affected rock is:

Maximum Sulfate Generation Rate = 81,000 tons per year

Total mass flux of sulfate currently transported in the Carlton Tunnel flow is presently approximately 5,000 tons per year. The sulfate generated by the entire inventory of rock affected by the MLE could, if it were to reach the Carlton Tunnel in the same time period, increase the concentration in the flow by up to 22,000 mg/L. This will not occur, because the transport of the generated sulfate takes place over a very long time period due to water sequestration in the mined materials and transport time from the mined materials to the Carlton Tunnel portal, and because of geochemical processes occurring in the diatreme along the transport pathway (addressed below).

3.4.3 Zinc liberation rate

The oxidation of sulfide minerals in the overburden has been observed in the HCTs to liberate zinc into the water associated with the tests. The rate of liberation is complex, and is not strongly dependent on the pyrite or zinc content of the overburden materials.

The average rate of release of zinc from the HCTs was:

Rate of zinc liberation = 22.1 mg/kg/yr

The rate of liberation is also expected to be a function of the surface area of the fragments of material that are oxidizing. The zinc liberation rate of ROM material can also be computed from the HCT rate by adjusting for the surface area:

$$R_{ROM} = R_{HCT} * (D_{HCT} / D_{ROM})$$

where: R_{ROM} = Zinc liberation rate in run of mine material (mg/kg/yr)
R_{HCT} = Zinc liberation rate in humidity cell test material (mg/kg/yr)
D_{HCT} = mean diameter of humidity cell test material (mm)
D_{ROM} = mean diameter of run of mine material (mm)

Samples of ROM materials have been tested by optical methods, and the average mean diameter determined to be ~100 mm. Mean diameter of humidity cell test materials is ~4 mm. Based on these results:

$$R_{ROM} = 22.1 \text{mg/kg/yr} * (4 \text{ mm} / 100 \text{ mm}) = 0.88 \text{ mg/kg/yr}$$

This rate has been applied to the entire inventory of 659 million tons of rock (251 million tons ore, 359 million tons overburden, and 49 million tons affected wall rock) affected by the proposed mining, and it is computed that zinc will be liberated at the following rate:

Zinc liberation = 580 tons per year

The zinc generated by the entire inventory of rock affected by the proposed mine extension could, if it were to reach the Carlton Tunnel, increase the concentration of zinc in the discharge water by up to 159 mg/L. This will not occur, because the transport of the liberated zinc takes place over a very long time period due to water sequestration in the mined materials and transport time from the mined materials to the Carlton Tunnel portal, and because of geochemical processes occurring in the diatreme along the transport pathway (addressed below).

3.5 Neutralization

3.5.1 Effects of neutralization

The groundwater that seeps through the overburden storage facilities and the mine backfill has the potential to dissolve acid and metallic constituents of sulfide oxidation in the overburden. An analog of the quality of the resulting water was obtained from the HCT. The chemical analysis of this water is presented in Table 9.

The effect of neutralization on this water was determined by neutralization using four alkaline materials:

- Natural calcite blend from the Main Cresson mine and the Globe Hill Mine
- Reagent grade calcium carbonate
- Reagent grade calcium oxide
- Reagent grade sodium hydroxide

The results are presented in Attachment 3, and are presented in Table 9.

Contact and neutralization with natural and reagent calcite causes the following:

- Increases the pH of the water from ~3 units to ~8 units
- Increases alkalinity from essentially zero to >100 mg/L
- Eliminates acidity
- Increases total dissolved solids concentration to ~3000 mg/L
- Decreases Al, As, Cd, Cr, Fe, Mn, Ni, and Zn, most to close to detection
- Leaves Cu, Pb, Hg, and Se essentially unchanged, but generally at low levels
- Increases Mo, Sb, Sr and U due to the presence of these constituents in the natural calcite

The water quality resulting from neutralization of the products of oxidation of overburden material is compared with the concentration of water emerging from the Carlton Tunnel in Table 9. The quality of the neutralized water is substantially indistinguishable from that of the neutralized HCT water, indicating that the genesis of both is the same: neutralized sulfide oxidation products.

3.5.2 Neutralization of products of sulfide oxidation from mine extension overburden

The amount of calcium carbonate required to neutralize the acid produced by complete oxidation of the reactive pyrite in the overburden rock to be stored between 2008 and the end of the MLE has been computed using the following neutralization equation:

$$CaCO_3 + 2H^+ = Ca^{2+} + H_2O + CO_2$$

Based on these results presented in Table 6, if all of the reactive pyrite in all of the 659 million tons of rock that will be affected by mining were to oxidize, then:

Calcium carbonate required to neutralize oxidation products ≤ 3.79 million tons.

This computation takes credit for the net neutralizing capacity of the overburden to be produced from the Cresson Mine. The computation does not take credit for any neutralization by lime added to the ore.

3.5.3 Carbonate Abundance in the Diatreme

The protection of regional groundwater quality by neutralization in the diatremal rocks requires an adequate mass of calcite to react with the sulfide inventory in the mined overburden.

The calcite inventory of the diatreme has been evaluated from testing of drill samples obtained from the drilling of the diatremal materials. Three deep holes have been drilled and sampled to provide neutralization data from depths down to and below the Carlton Tunnel elevation:

- CC-2272 - Located in the south center of the diatreme (5030 ft deep)
- GHC-474 - Located in the west of the diatreme (2915 ft deep)
- GC-07-704 - Located in the north of the diatreme (3465 ft deep)

These data have been combined with data for neutralization capacity from a total of 520 other shallower holes that sampled diatremal material. The average carbonate concentration in all holes has been determined by sample elevation, and is presented in Table 10, and presented graphically in Figure 17.

The total mass of available carbonate that is present in the diatremal rocks is computed from the carbonate abundance and the diatremal rock mass, in 500 foot thick elevation slices (Table 11).

Infiltrating groundwater infiltrates vertically downward through the unsaturated diatreme to the regional groundwater system, at an elevation between 7,000 ft msl (at the Carlton Tunnel) and 7,200 ft msl (remote from the Carlton Tunnel). The regional groundwater then flows essentially horizontally in the saturated diatremal materials; the thickness of the rock that hosts this flow is estimated to be a further 500 feet depth, down to elevation 6,500 ft msl. Accordingly, infiltrating groundwater contacts calcium carbonate in the entire diatremal rockmass down to an elevation of approximately 6,500 ft msl. From Table 11:

Mass of carbonate available for neutralization of infiltrating groundwater = 1,883 million tons

3.5.4 Adequacy of Neutralization Capacity

The adequacy of the identified calcite in the diatremal rock to neutralize the maximum amount of acidic seepage that can be generated by oxidation of sulfide contained in the overburden produced from the present (2008) to the anticipated end of the MLE based on current production rates is determined as follows:

- Carbonate to neutralize affected rock oxidation products ≤ 3.79 million tons (§3.5.2 above)
- Carbonate available for neutralization = 1,883 million tons (§3.5.3 above)
- Proportion of carbonate to neutralization overburden acid products = 0.20%

The calcium carbonate available in the diatreme is approximately 500 times the mass required to neutralize any products of sulfide oxidation in the overburden mined in the proposed extension. To express the available neutralization capacity in a different way, the uppermost 25 feet of the diatreme has sufficient carbonate to neutralize the products of oxidation of all sulfides in the MLE-affected rock (i.e. the top 25 feet contains 3.79 million tons of calcium carbonate equivalent).

Neutralization of water infiltrating from the surface of the diatreme requires contact with the neutralizing material. This contact could be avoided by flow down preferential pathways in the diatreme, in particular fractures, faults, or mined underground openings (such as stopes). In the Cripple Creek Diatreme this is not expected to be a significant factor, for the following reasons:

1. There is a great excess of calcite over requirements, so contact with calcite is inevitable in a flowpath length of between 3,000 feet and 12,000 feet.
2. A significant portion of the flowpath for all groundwater movement to Carlton Tunnel will take place in the saturated portion of the diatreme, where there is relatively high calcite concentration, very slow flow, and flow generally not along major natural or mined conduits. All of these factors encourage contact with neutralizing material within the rockmass.
3. Neutralization of the products of sulfide oxidation by natural calcite creates gypsum, as shown in the neutralization equation below (for high concentrations):

   $$CaCO_3 + H_2SO_4 = CaSO_4 + H_2O + CO_2$$

   Accordingly, locally neutralization causes the replacement of the dissolved calcite by gypsum, resulting in little if any decrease in solid phase volume. Thus formation of preferential pathways by dissolution of carbonate is not expected to be significant in this system[7].

Accordingly it has been determined that there will be no detrimental change in the water quality in the diatreme, or in the Carlton Tunnel discharge as a result of the proposed Mine Life Extension.

3.5.5 Carlton Tunnel Water Quality Verification

Verification that the protective geochemical process of neutralization is occurring and will continue to occur in the Cripple Creek Diatreme is available by reference to the quality of the regional groundwater flow from the Carlton Tunnel. This tunnel collects essentially all of the groundwater entering the diatreme and the immediately contiguous rocks, and forms a single location for monitoring and evaluating the geochemical processes occurring in the diatreme and contiguous areas.

The water quality data from the Carlton Tunnel provides three verification opportunities, as follows.

3.5.5.1 Average Water Quality

The average water quality in the Carlton Tunnel in the period 1988 to the present is functionally indistinguishable from the quality of the water that is produced by neutralizing effluent from the humidity cell tests with natural calcite, as shown in Table 9. This result verifies that the neutralization process in the diatreme is protective of the quality of the regional groundwater exiting the diatreme to the Carlton Tunnel.

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[7] In the natural system, massive gypsum and/or selenite are present as joint and fault fillings in considerable quantity at depth in the diatreme, apparently as a result of neutralization of naturally-produced products of sulfide oxidation.

3.5.5.2 Alkalinity

The alkalinity of Carlton Tunnel water provides a method of monitoring the effectiveness of the neutralization process within the diatremal rocks. The alkalinity data is plotted in Figure 18.

The alkalinity history can be broken into three periods:

1. Pre-1993. Up to 1993 the alkalinity appears to have been approximately 200 mg/L (as $CaCO_3$).
2. 1993-1999. Alkalinity rose from 200 mg/L to approximately 400 mg/L steadily over this seven year period, which was the first seven years of surface mining operations by CC&V.
3. 2000-Present. Alkalinity has remained steady on an annual basis, with a seasonal peak of approximately 400 mg/L in summer, and a low of approximately 360 mg/L in winter.

The increase in alkalinity appears to be the result of the current surface mining, as there has been no corresponding change in precipitation. Since the initiation of the current surface mining activity, there must have been reduced acid generation in the pre-existing subsurface workings. This is probably the result of reduction of air and water ingress into the underground workings due to reclamation of old surface and underground mines, and due to covering of prior air and water access locations to the underground workings with overburden materials.

The continued and increasing availability of alkalinity in the Carlton Tunnel flow is a clear indication that there is a great excess of available neutralizing capacity in the diatremal rocks, supporting the same finding arrived at by direct measurement of calcium carbonate at depth in the diatreme (Sec. 3.5.3 above).

3.5.5.3 Gypsum

The current surface mining operations started in 1993, and with increasing production rates over the succeeding decade. The evaluation above predicts that this mining should not have negatively impacted the quality of groundwater exiting the diatreme. This can be verified by reference to the quality of water in the Carlton Tunnel. Figure 19 shows the concentration of calcium and sulfate, which are the major constituents which would occur in the Carlton Tunnel water as a result of sulfide oxidation in the mined overburden. There has been no significant change in the concentration of calcium and sulfate since the inception of surface mining in 1993.

These parameters are likely controlled by saturation considerations; saturation at some point in the flow system guarantees that the major ions will not significantly change unless the neutralization process that controls pH in the water ceases. As identified above, cessation of this process is not foreseeable.

3.6 Conclusion

No deleterious change in the average regional groundwater quality has occurred due to recent mining (1993-2007). No deleterious change in the average regional groundwater quality will occur due to the proposed future MLE (2008-2016).

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TABLES

Table 1 - Cresson Mine Precipitation

YEAR	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	OCT	NOV	DEC	TOT
1992	0.04	0.12	1.67	0.31	3.28	3.84	2.10	3.13	0.17	0.31	0.80	0.41	**16.18**
1993	0.24	0.47	0.97	0.48	2.11	1.07	1.38	2.21	2.78	2.11	1.23	0.14	**15.19**
1994	0.71	0.00	1.20	2.49	5.17	1.63	1.88	6.32	2.18	2.15	0.78	3.06	**27.57**
1995	1.36	0.85	2.69	2.62	4.03	3.72	2.92	4.36	2.72	0.23	0.30	0.16	**25.96**
1996	1.98	0.15	0.60	1.30	1.99	1.85	3.23	2.97	1.43	0.70	0.30	0.28	**16.78**
1997	0.17	0.80	0.50	1.03	2.01	3.78	2.45	3.60	1.59	0.21	0.71	0.31	**17.16**
1998	0.31	0.79	0.85	0.16	0.09	0.06	10.47	5.40	0.88	0.12	0.00	0.00	**19.13**
1999	0.00	0.86	0.15	5.44	2.81	1.97	5.95	4.10	0.91	1.39	0.28	0.12	**23.98**
2000	0.74	0.53	2.25	1.02	1.83	2.04	2.92	5.26	0.50	0.91	0.40	0.47	**18.87**
2001	0.41	0.64	1.50	1.21	2.53	1.68	4.06	6.68	0.52	0.07	0.98	0.20	**20.48**
2002	0.45	0.80	0.74	0.23	1.50	0.73	3.76	1.20	1.48	1.65	0.28	0.05	**12.87**
2003	0.20	1.49	2.43	1.01	1.83	3.18	2.71	3.60	1.25	0.64	0.36	0.26	**18.96**
2004	0.78	0.62	0.75	3.03	0.49	4.02	4.08	3.40	0.91	0.70	0.43	0.17	**19.34**
2005	0.80	0.73	1.19	1.52	0.71	1.53	2.29	4.50	1.48	0.41	0.58	0.52	**16.26**
2006	0.46	0.33	1.57	1.19	1.16	1.17	5.40	5.11	1.35	2.21	0.38	0.67	**21.00**
2007	1.94	0.90	2.33	2.08	4.06	1.58	4.01	3.91	1.54	0.46	0.23	*0.45*	**23.49**
Avg	**0.66**	**0.63**	**1.34**	**1.57**	**2.22**	**2.12**	**3.73**	**4.11**	**1.36**	**0.89**	**0.50**	**0.45**	**19.58**
% Tot	**3%**	**3%**	**7%**	**8%**	**11%**	**11%**	**19%**	**21%**	**7%**	**5%**	**3%**	**2%**	**100%**

Notes:

1. Data taken from Bateman Station at the mine office unless noted below.
2. DMR data from Guffey, CO station used for 1/92 through 6/94; Florissant Fossil Beds used for 2/92.
3. Hunter's Data used for 5/98 and during power outage at Bateman in 4/99 and 5/99.
4. 11/00 data from Carlton Security, 12/00 through 5/01 data from Security Office ("Rigi")
5. Guffey station data used for 10/95, 11/95, 12/95, 2/96, 3/96, 4/96, 5/96
6. NOAA data used for 3/97
7. Belfort rain gauge at Bateman Stations used for 6/98.
8. Storm water sampler (Sigma 900 Max) gauge used for 7/98 and 8/98.
9. 2000 data are average of Rigi and Bateman, except Nov and Dec 2000 are only Rigi
10. 2001 data are average of Rigi and Bateman (Pad), except June-September based on Bateman
11. 2002 and 2003 data are from Bateman
12. 2004 data are average of Bateman and Rigi
13. 2005 and later data are from Rigi
14. Data in italics are fill for the year based on monthly average.

Table 2 - Surface Water Flow in Grassy Valley

Month	GV-1	GV-2	GV-3
Jan	0.0	0.0	0.0
Feb	0.0	0.0	0.0
Mar	13.8	13.8	17.0
Apr	53.7	20.3	1.0
May	21.4	64.2	78.9
Jun	45.3	43.1	143.4
Jul	0.5	12.5	120.0(1)
Aug	20.8	37.9	92.9
Sep	1.6	15.8	11.2
Oct	0.0	3.9	11.1
Nov	0.0	1.6	0.0
Dec	0.0	10.0	0.0
Average	**13.1**	**18.6**	**39.6**

Note:

(1) July flow data at GV-3 was interpolated between adjacent months to compensate for sporadic timing of storm flows compared with sampling events.

Table 3 - Average Monthly Flow from Carlton Tunnel (gpm)

Year	**Jan**	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**	**Annual**
1989											1495	1484	
1990	1551	1616	1549	1484	1449	1583	1500	1690	1664	1664	1655	1649	**1588**
1991	1586	1322	1192	1253	1299	1299	1354	1438	1667	1667	1507	1493	**1423**
1992	1465	1465	1465	1491	1639	1733	1741	1708	1733	1687	1667	1667	**1622**
1993	1667	1625	1625	1708	1771	1646	1625	1583	1583	1542	1562	1704	**1637**
1994	1583	1625	1507	1507	1844	2005	1819	1757	1799	1732	1754	1654	**1716**
1995	1604	1625	1625	1667	1778	2095	2021	1941	2001	2063	2021	1931	**1864**
1996	1931	1708	1072	983	1083	1123	1234	1317	1379	1399	1346	1356	**1328**
1997	1421	1372	1365	1333	1338	1432	1519	1449	1474	1458	1271	1387	**1402**
1998	1318	1352	1378	1405	1446	1493	1356	1611	1537	1548	1495	1378	**1443**
1999	1284	1302	1311	1239	1442	1815	1730	1793	1749	1622	1537	1489	**1526**
2000	1438	1421	1421	1414	1560	1554	1506	1480	1404	1446	1429	1387	**1455**
2001	1369	1378	1372	1404	1506	1631	1506	1685	1789	1718	1605	1537	**1542**
2002	1457	1455	1429	1480	1370	1338	1380	1347	1540	1338	1365	1356	**1405**
2003	1378	1299	1221	1257	1286	1311	1320	1235	1179	1239	1230	1235	**1266**
2004	1257	1266	1202	1221	1257	1284	1302	1293	1372	1378	1372	1395	**1300**
2005	1387	1329	1284	1266	1338	1338	1320	1335	1320	1347	1365	1338	**1331**
2006	1365	1356	1329	1365	1329	1293	1311	1299	1257	1257	1257	1257	**1306**
2007	1275	1194	1222	1239	1239	1329	1378	1275	1347	1365	1365	*1365*	**1299**
Total	**1463**	**1428**	**1365**	**1373**	**1443**	**1517**	**1496**	**1513**	**1544**	**1526**	**1489**	**1483**	**1487**

Notes: Flow measured at tunnel adit flume until March 1996
Flow measured in flume at end of settlement ponds after March 1996
Value in *italics* (Dec 2007) is set equal to prior month, based on past record.

Table 4 – 2008-2016 Production

MINE[1]	ORE ton	OVERBURDEN ton	TOTAL ton	Comment
CRESSON	151,000,000	109,000,000	260,000,000	Permitted under Am.8
WHEX-pC	2,000,000	8,000,000	10,000,000	
WHEX-T	31,000,000	111,000,000	142,000,000	
S CRESSON	17,000,000	27,000,000	44,000,000	
GLOBE HILL	21,000,000	37,000,000	58,000,000	
SCHIST-pC	8,000,000	11,000,000	19,000,000	
SCHIST-T	21,000,000	56,000,000	77,000,000	
TOTALS	**251,000,000**	**359,000,000**	**610,000,000**	

Notes: (1) pC = Precambrian (Granitic); T = Tertiary (Volcanic)

Table 5 - Overburden Source and Proposed Storage Location

OVERBURDEN SOURCE	**OVERBURDEN STORAGE LOCATION**							**TOTAL**
	East Cresson Backfill	**Main Cresson Backfill**	**Schist Island Backfill**	**Wild Horse Backfill**	**WHEX Backfill**	**SGOSA**	**ECOSA**	
Main Cresson		51,000,000		25,000,000		33,000,000		**109,000,000**
Globe Hill			11,000,000		26,000,000			**37,000,000**
South Cresson		27,000,000						**27,000,000**
Schist Island		39,000,000	21,000,000			7,000,000		**67,000,000**
WHEX	75,000,000				3,000,000		41,000,000	**119,000,000**
TOTAL	**75,000,000**	**117,000,000**	**32,000,000**	**25,000,000**	**29,000,000**	**40,000,000**	**41,000,000**	**359,000,000**

Notes:
WHEX = Wildhorse Extension
SGOSA = Squaw Gulch Overburden Storage Area
ECOSA = East Cresson Overburden Storage Area

Table 6 - Acid-Base Accounting (2008-2016)

ORE (2008-2016)

MINE(1)	MINED	S_{TOTAL}	S_{PYRITE}	$S_{REACTIVE}$	AGP	C_{TOTAL}	ANP (t/kt)	NNP (t/kt)	$CaCO_3$(3)
	ton	%	%	%	t/Kt(2)	%	t/Kt(2)	t/Kt(2)	tons
CRESSON	151,000,000	1.26	1.00	0.78	24.26	0.36	29.95	5.69	-859,000
WHEX-pC	2,000,000	2.41	1.73	1.34	41.83	0.11	8.81	-33.02	66,000
WHEX-T	31,000,000	1.21	0.96	0.75	23.24	0.09	7.76	-15.48	480,000
S CRESSON	17,000,000	2.48	2.15	1.67	52.06	0.20	16.42	-35.63	606,000
GLOBE HILL	21,000,000	0.56	0.40	0.31	9.68	0.08	6.37	-3.32	70,000
SCHIST-pC	8,000,000	2.17	1.86	1.44	44.87	0.08	7.04	-37.83	303,000
SCHIST-T	21,000,000	1.59	1.06	0.82	25.71	0.14	11.34	-14.37	302,000
TOTALS	**251,000,000**	**1.35**	**1.06**	**0.82**	**25.72**	**0.26**	**21.87**	**-3.85**	**968,000**

OVERBURDEN (2008-2016)

MINE(1)	MINED	S_{TOTAL}	S_{PYRITE}	$S_{REACTIVE}$	AGP	C_{TOTAL}	ANP (t/kt)	NNP (t/kt)	$CaCO_3$(3)
	ton	%	%	%	t/Kt(2)	%	t/Kt(2)	t/Kt(2)	tons
CRESSON	109,000,000	1.05	0.82	0.63	19.71	0.33	27.65	7.94	-865,000
WHEX-pC	8,000,000	1.56	1.04	0.81	25.23	0.13	10.66	-14.57	117,000
WHEX-T	111,000,000	0.86	0.65	0.50	15.65	0.10	8.50	-7.15	794,000
S CRESSON	27,000,000	2.35	2.03	1.57	49.08	0.16	13.74	-35.34	954,000
GLOBE HILL	37,000,000	0.56	0.40	0.31	9.60	0.09	7.10	-2.49	92,000
SCHIST-pC	11,000,000	2.69	2.36	1.83	57.02	0.07	5.96	-51.06	562,000
SCHIST-T	56,000,000	1.66	1.12	0.87	27.06	0.12	9.82	-17.23	965,000
TOTALS	**359,000,000**	**1.19**	**0.91**	**0.71**	**22.03**	**0.18**	**14.74**	**-7.29**	**2,619,000**

Notes:

(1) pC = Precambrian (Granitic); T = Tertiary (Volcanic). Tonnages conformed to Mine Plan totals, and rounded to nearest million tons.

(2) Units for AGP, ANP, and NNP are equivalent tons of $CaCO_3$ per 1000 tons of material

(3) Calcium carbonate required to neutralize the products of the net acid production (negative where material is net neutralizing).

Table 7 - Exposed Wall Rock Acid-Base Accounting

MINE(1)	**AREA**(2)	**DEPTH**	**WALL ROCK**(3)	S_{TOTAL}	S_{PYRITE}	$S_{REACTIVE}$	**AGP**	C_{TOTAL}	**ANP**	**NNP**	**CaCO$_3$**(4)
	sq.ft.	ft	**ton**	**%**	**%**	**%**	**t/kt**(5)	**%**	**t/kt**(5)	**t/kt**(5)	**ton**
CRESSON	9,174,866	35	23,000,000	1.05	0.82	0.63	19.71	0.33	27.65	7.94	-183,000
WHEX-pC	110,265	35	300,000	1.56	1.04	0.81	25.23	0.13	10.66	-14.57	4,000
WHEX-T	1,529,932	35	4,000,000	0.86	0.65	0.50	15.65	0.10	8.50	-7.15	29,000
S CRESSON	2,847,790	35	7,000,000	2.35	2.03	1.57	49.08	0.16	13.74	-35.34	247,000
GLOBE HILL	4,370,561	35	11,000,000	0.56	0.40	0.31	9.60	0.09	7.10	-2.49	27,000
SCHIST-pC	253,115	35	600,000	2.69	2.36	1.83	57.02	0.07	5.96	-51.06	31,000
SCHIST-T	1,288,585	35	3,000,000	1.66	1.12	0.87	27.06	0.12	9.82	-17.23	52,000
TOTALS	**19,575,114**	**35**	**48,900,000**	**1.17**	**0.92**	**0.71**	**22.25**	**0.22**	**18.01**	**-4.24**	**207,000**

Notes:

(1) pC = Precambrian (Granitic); T = Tertiary (Volcanic)

(2) True surface area computed from mine walls not covered by mine backfill

(3) In-place density assumed to be 14 cuft/ton

(4) Calcium carbonate required to neutralize the products of the net acid production (negative where material is net neutralizing).

(5) Units for AGP, ANP, and NNP are equivalent tons of CaCO$_3$ per 1000 tons of material

Table 8 - Pyrite Oxidation

TERTIARY (VOLCANIC) ROCK

Pyrite Range (%)	Median Pyrite (%)	Test Samples (#)	Test Samples (%)	Pyrite Oxidation Rate (mg/kg/y)	Sulfate Generation Rate (mg/kg/y)	Affected Rock Mass(1) (ton)	Sulfate Generation(2) (ton/yr)
0%-1%	0.50	14789	52.2%	22	42	328,470,000	13,675
1%-2%	1.50	6423	22.7%	67	125	142,658,000	17,817
2%-5%	3.50	6886	24.3%	156	291	152,941,000	44,570
5%-10%	7.50	213	0.8%	334	624	4,731,000	2,954
10%-20%	15.00	6	0.0%	668	1,249	133,000	166
20%-50%	35.00	2	0.0%	1559	2,914	44,000	128
>50%	60.00	1	0.0%	2673	4,996	22,000	110
Totals	**2.40**	**28320**	**100.0%**	**68**	**126**	**629,000,000**	**79,421**

PRECAMBRIAN (GRANITIC) ROCK

Pyrite Range (%)	Median Pyrite (%)	Test Samples (#)	Test Samples (%)	Pyrite Oxidation Rate (mg/kg/y)	Sulfate Generation Rate (mg/kg/y)	Affected Rock Mass(1) (ton)	Sulfate Generation(2) (ton/yr)
0%-1%	0.50	685	40.8%	5	9	12,213,000	108
1%-2%	1.50	299	17.8%	14	27	5,331,000	142
2%-5%	3.50	519	30.9%	33	62	9,253,000	574
5%-10%	7.50	164	9.8%	71	133	2,924,000	389
10%-20%	15.00	8	0.5%	142	266	143,000	38
20%-50%	35.00	2	0.1%	332	620	36,000	22
>50%	60.00	0	0.0%	569	1,063	0	0
Grand Total	**3.87**	**1677**	**100.0%**	**23**	**43**	**29,900,000**	**1,273**

TOTAL MINE-AFFECTED ROCK

Pyrite Range (%)	Median Pyrite (%)	Total Samples (#)	Total Samples (%)	Pyrite Oxidation Rate (mg/kg/y)	Sulfate Generation Rate (mg/kg/y)	Affected Rock Mass(1) (ton)	Sulfate Generation(2) (ton/yr)
All	**2.46**	**29997**	**100%**	**66**	**122**	**658,900,000**	**80,694**

Notes:

(1) Affected rock mass represents the total of ore, overburden, and wall rock affected by mining.

(2) Maximum, assuming excess oxygen and water availability to all materials.

Table 9 - Neutralization Test Results

Species	**Unit**	**HCT Test Water (1)**	**Calcite-Neutralized HCT Water(2)**	**Carlton Tunnel Water(3)**
pH		3.10	8.04	7.83
SO4	mg/L	435	1685	1250
Acidity	mg $CaCO_3$/L	507	5.4	<25
Alkalinity	mg $CaCO_3$/L	1.0	136	260
TDS	mg/L	608	2805	2220
Al	mg/L	9.8	0.011	<0.1
Sb	mg/L	0.001	0.014	n/a
As	mg/L	0.172	0.001	<0.005
Cd	mg/L	0.029	0.000	<0.001
Cr	mg/L	0.002	0.001	<0.001
Cu	mg/L	0.123	0.172	<0.005
Fe	mg/L	117	0.050	<0.05
Pb	mg/L	0.002	0.002	<0.001
Mn	mg/L	4.544	1.025	0.51
Hg	mg/L	0.00002	0.00007	<0.0001
Mo	mg/L	0.001	0.156	<0.02
Ni	mg/L	0.073	0.016	<0.01
Se	mg/L	0.001	0.005	<0.005
Sr	mg/L	0.1	14.9	12
U	mg/L	0.054	0.211	n/a
Zn	mg/L	0.854	0.024	0.052

Notes:
(1) Average of leachate samples from four different HCT tests
(2) Calcite from samples taken from depth in Main Cresson and Globe Hill areas
(3) Data represent median value of approximately 200 water samples taken 1988-2007
(4) n/a indicates "not analyzed"

Table 10 - Calcium Carbonate Abundance in Diatreme (%)

Elevation Range (ft)	AC-708	AR-740	CC-2272 (1)	GC-07-704	GHC-474(2)	GR-166	GR-167	GR-256	GR-267	HR-25	HR-31	HR-33	HR-36	UGC97-5	Average(3)
10500-11000	Values from 292 drill holes														1.54
10000-10500	Values from 498 drill holes														0.92
9500-10000	Values from 178 drill holes														1.21
9000-9500	0.4	0.2	4.3	0.4	2.6	1.0	3.4	2.3	0.6	3.0	4.7	3.9	6.0		2.5
8500-9000			5.8	1.7	2.4					5.0	7.7	3.5	3.5	6.8	4.5
8000-8500			7.1	1.6	11.5							4.9			7.3
7500-8000			2.7	7.0	33.8										19.3
7000-7500			3.8	7.4	41.7										17.6
6500-7000			7.8												7.8
6000-6500			8.0												8.0
5500-6000			8.7												8.7

Notes: (1) Values from CC-2272 and CR-2272 averaged by elevation
(2) Values from GHC-474 and GHC-474D2 averaged by elevation
(3) Averages weight each hole equally at each elevation

Table 11 - Carbonate Mass in Diatreme

Elevation (ft msl)	Diatreme Mass (million tons)	$CaCO_3$ (%)	$CaCO_3$ (million tons)	Total CaCO3 (million tons)
10000-10500	2206	0.9	20	20
9500-10000	4511	1.2	55	75
9000-9500	4570	2.5	115	190
8500-9000	3902	4.5	177	368
8000-8500	3441	7.3	251	619
7500-8000	3074	19.3	594	1213
7000-7500	2730	17.6	481	1694
6500-7000	2427	7.8	189	1883
6000-6500	2142	8.0	172	2055
5500-6000	1986	8.7	173	2228

Note: Shaded area represents diatremal rock contacted by groundwater that exits through Carlton Tunnel.

Appendix 1

FIGURES

Figure 1 - Cripple Creek Mining District (2007)

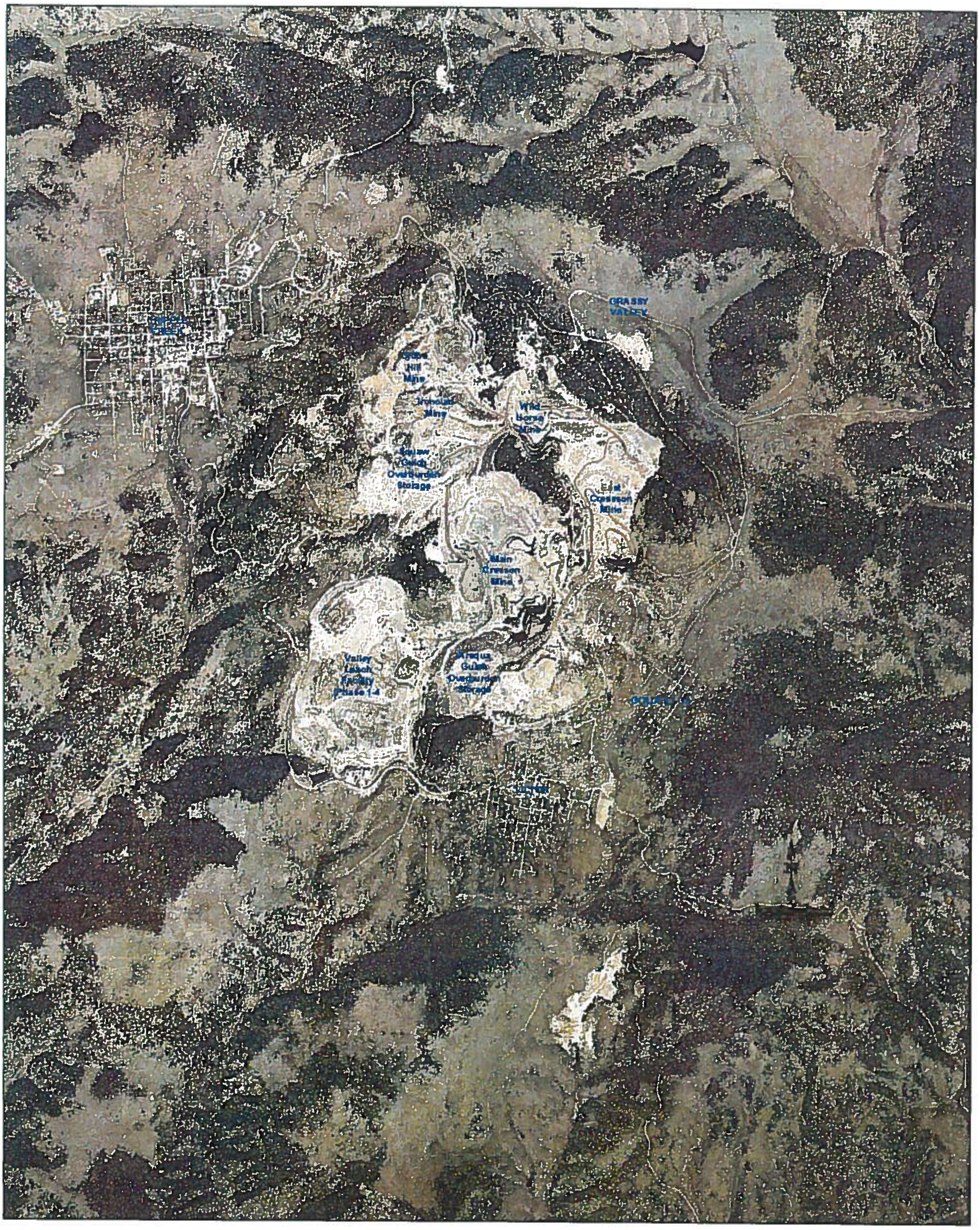


Figure 2 - Schematic Groundwater Flow System

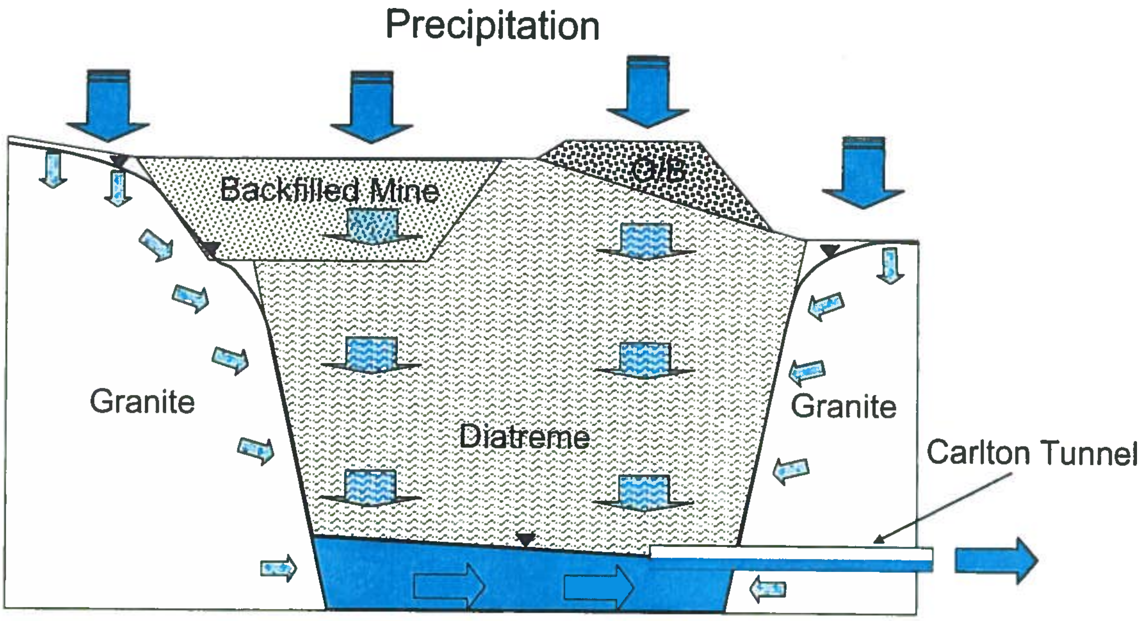


Figure 3 - Surface Flow in Grassy Valley

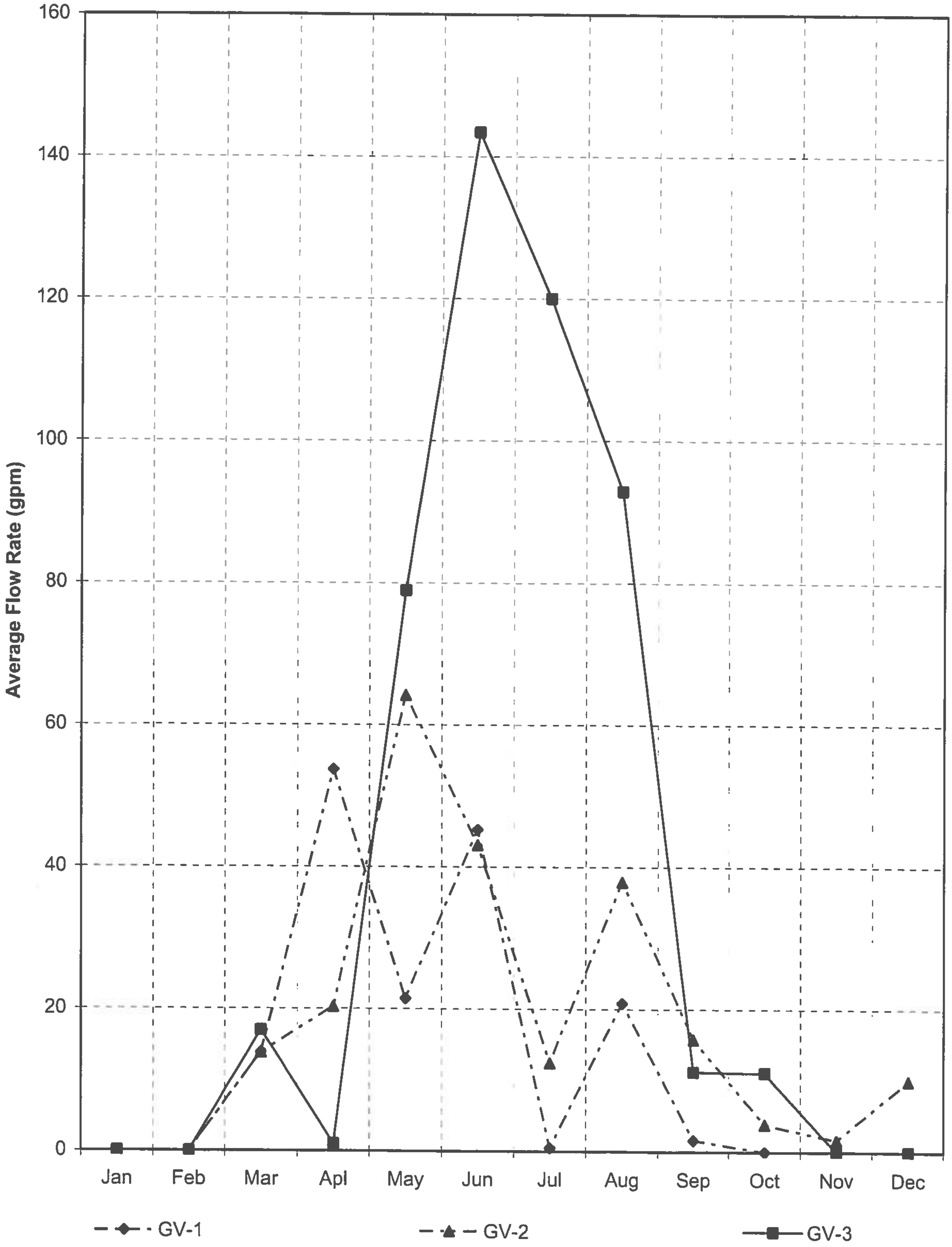


Figure 4 - Precipitation and Carlton Tunnel Flow

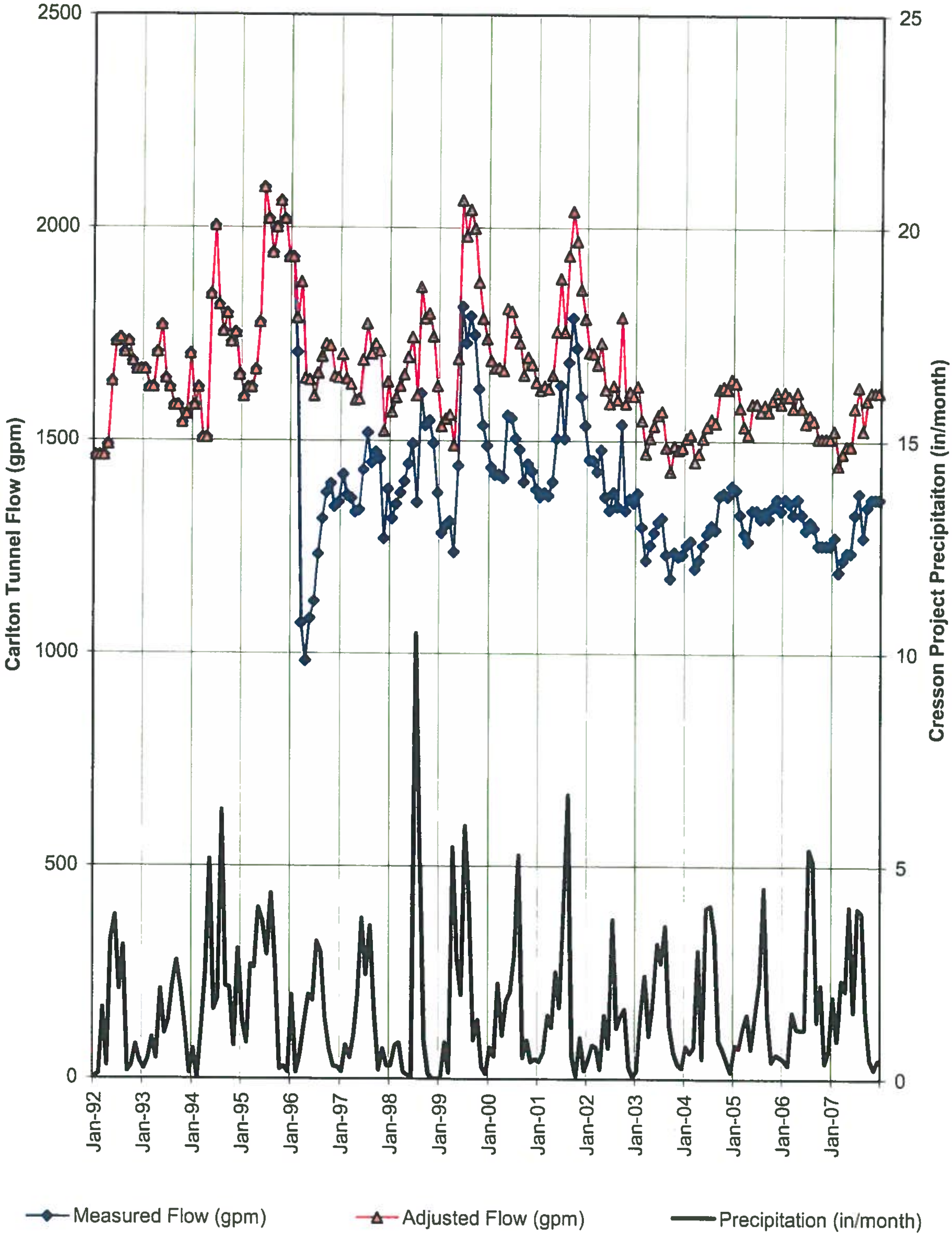


Figure 5 – Annual Groundwater Yield vs. Precipitation

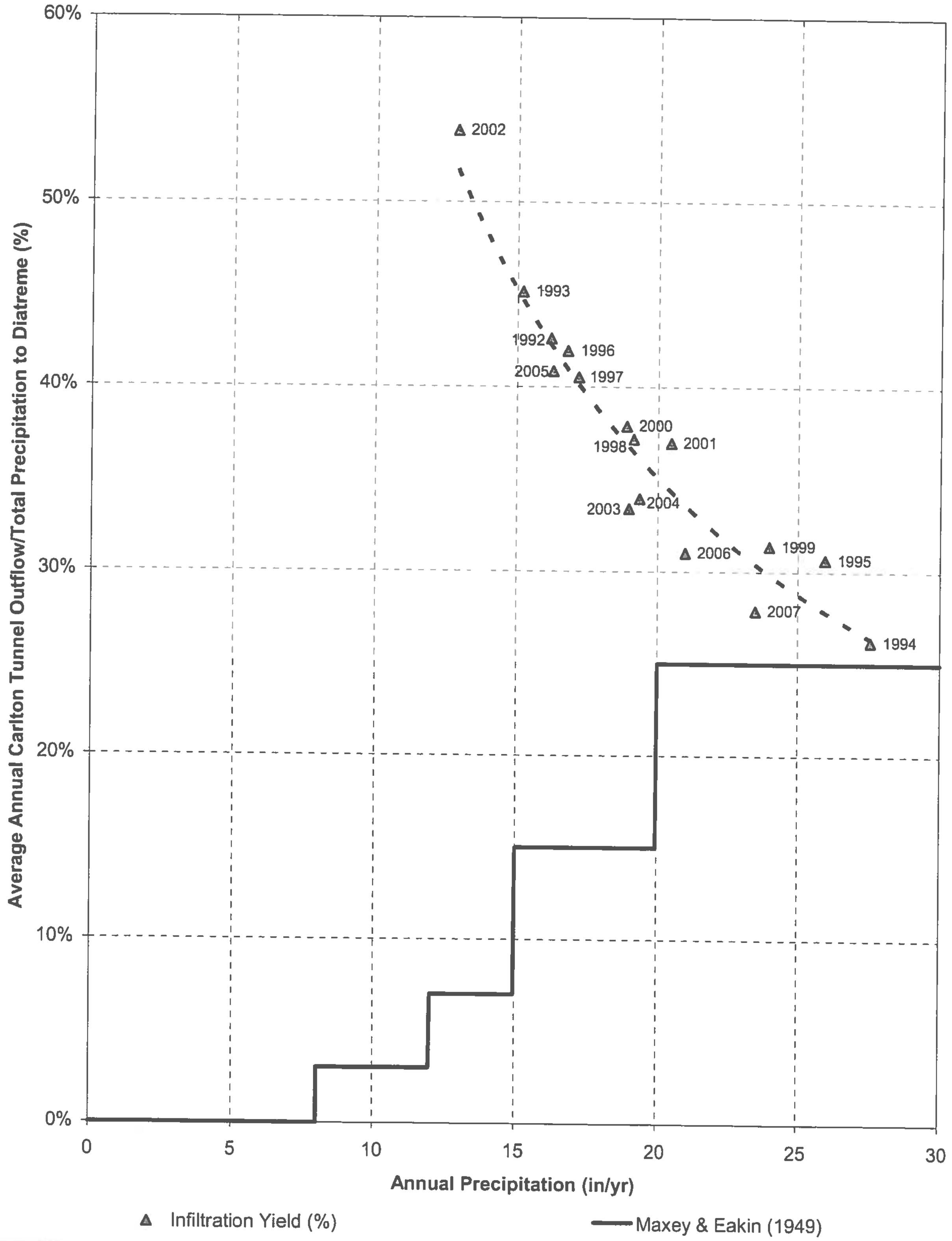


Figure 6 - Diatreme Drainage Area and MLE Elements

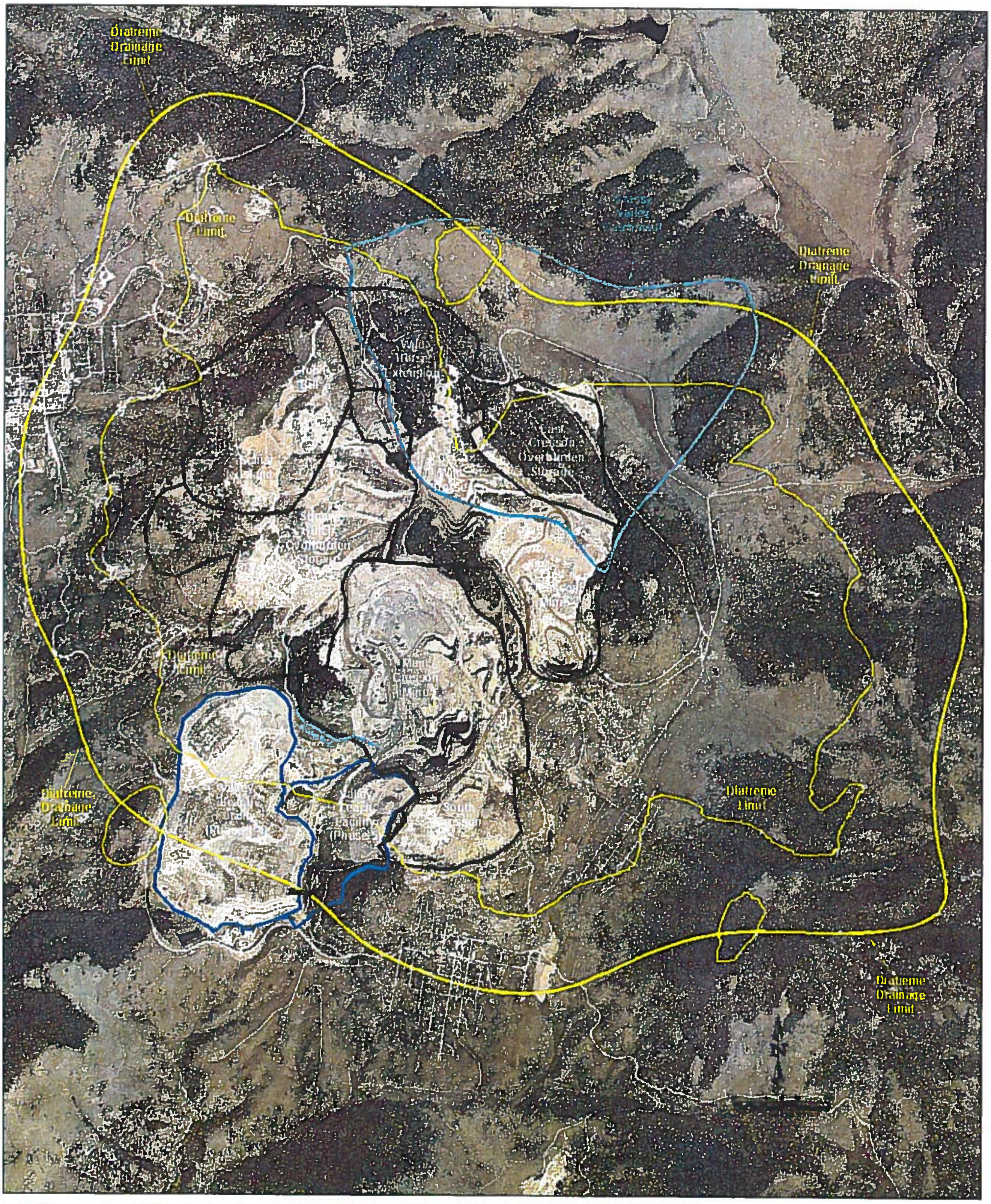


Figure 7 - Disturbance Areas - Cripple Creek Diatreme

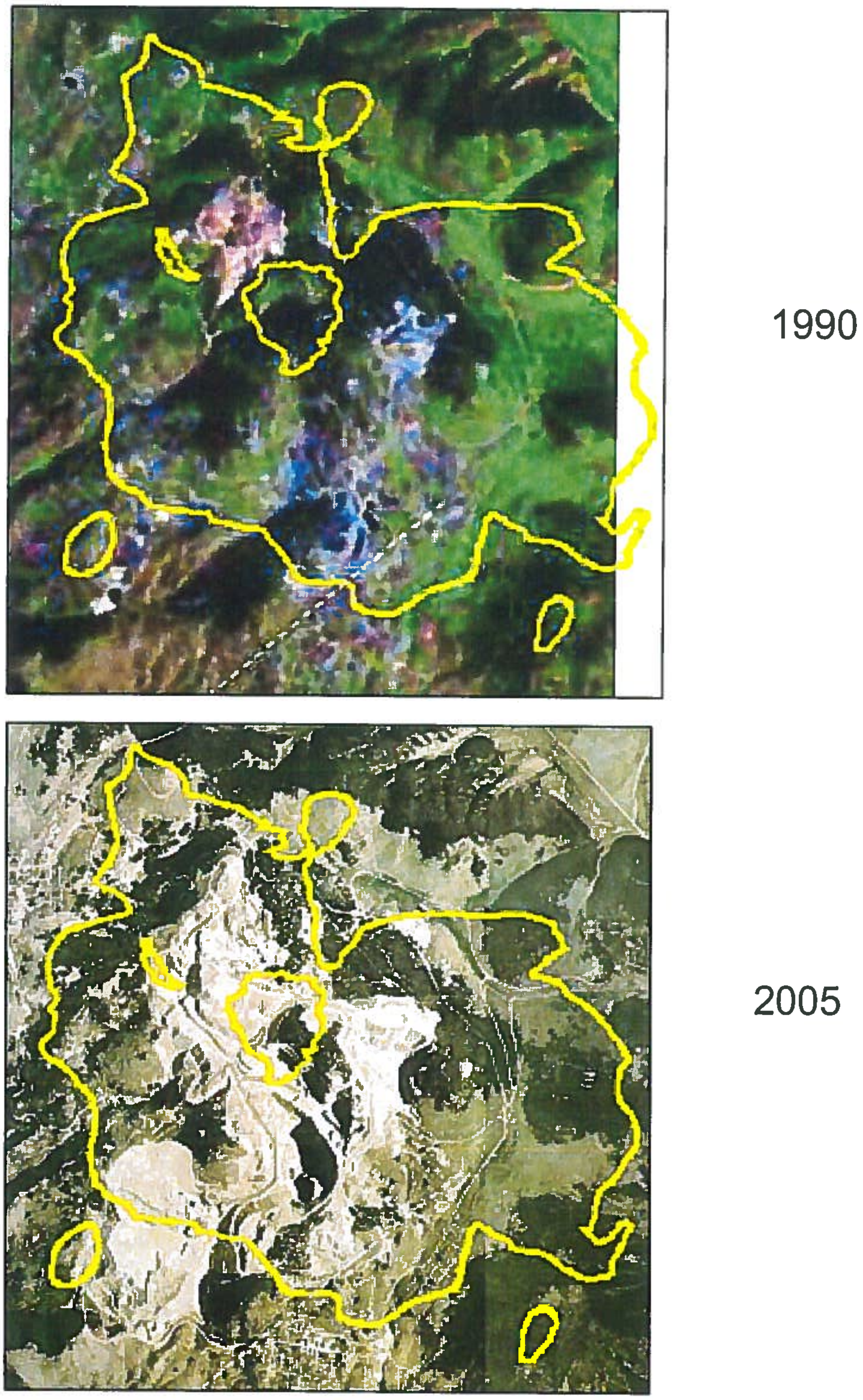


Figure 8 - Double Mass Curve - Flow vs. Precipitation

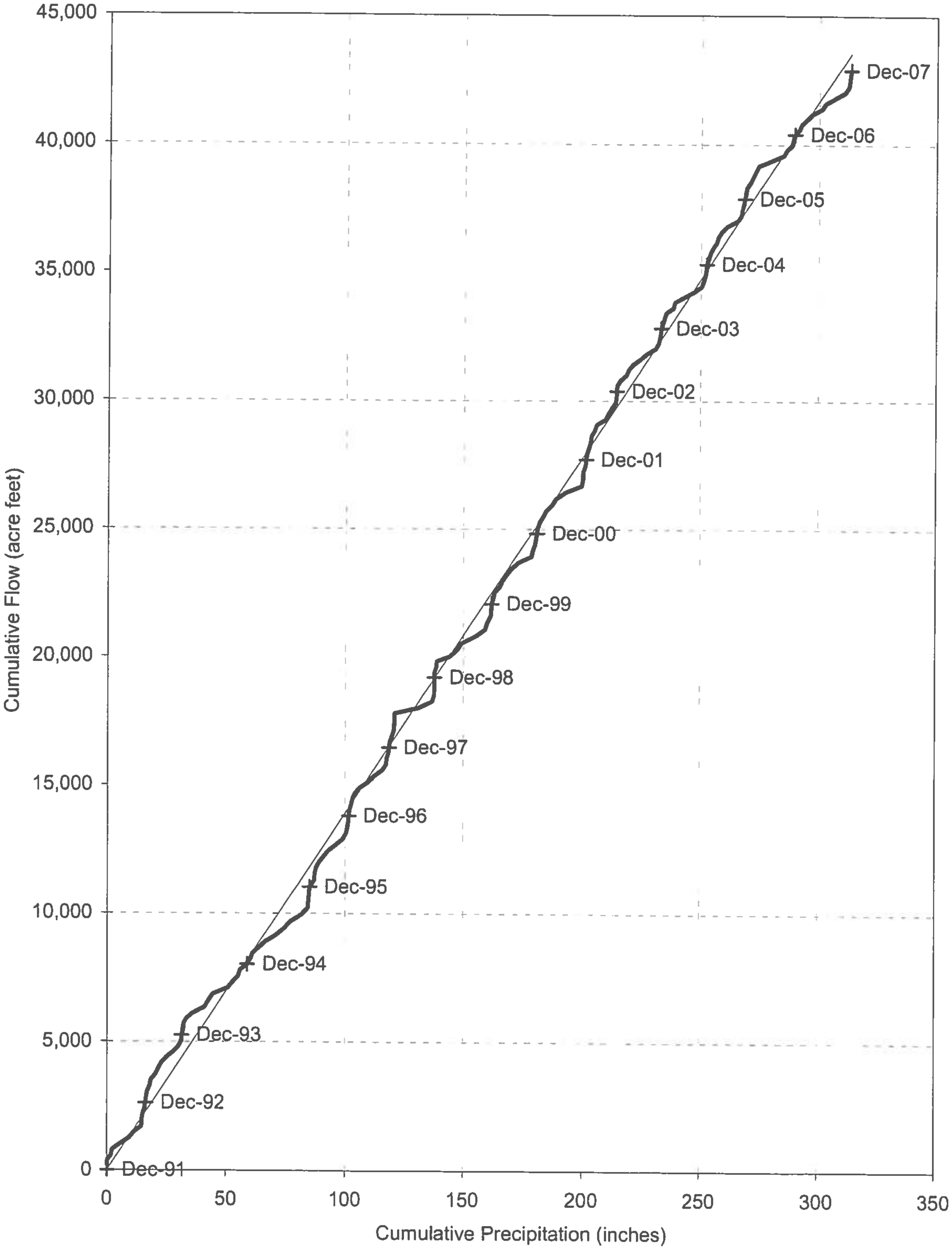
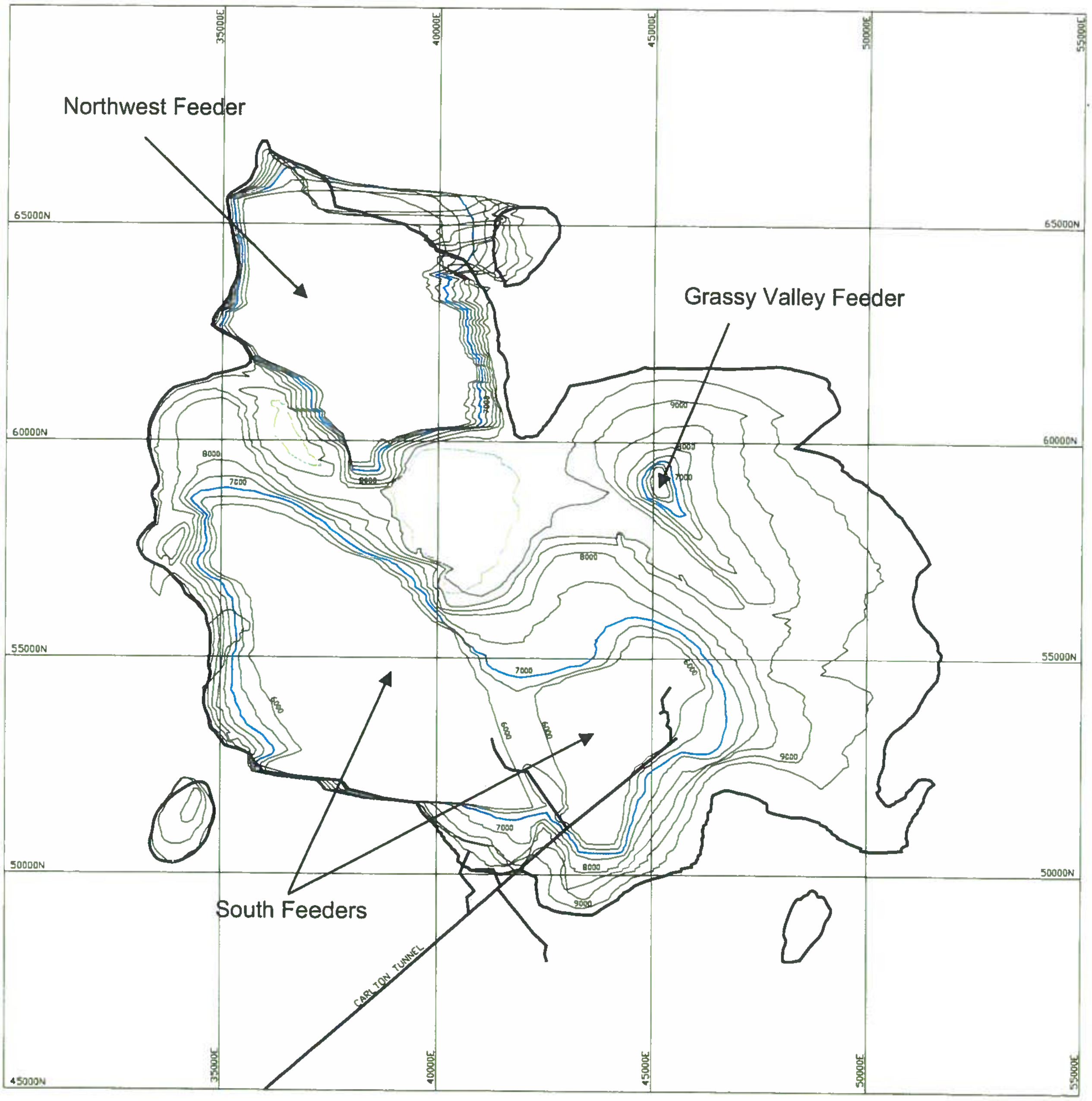


Figure 9 - Cripple Creek Diatreme



Notes: Contour Interval = 500 ft
Overlying 10,000 and 10,500 contours omitted for clarity
Geology provided by CC&V Exploration Department

Figure 10 - Hydrogeochemical System Schematic

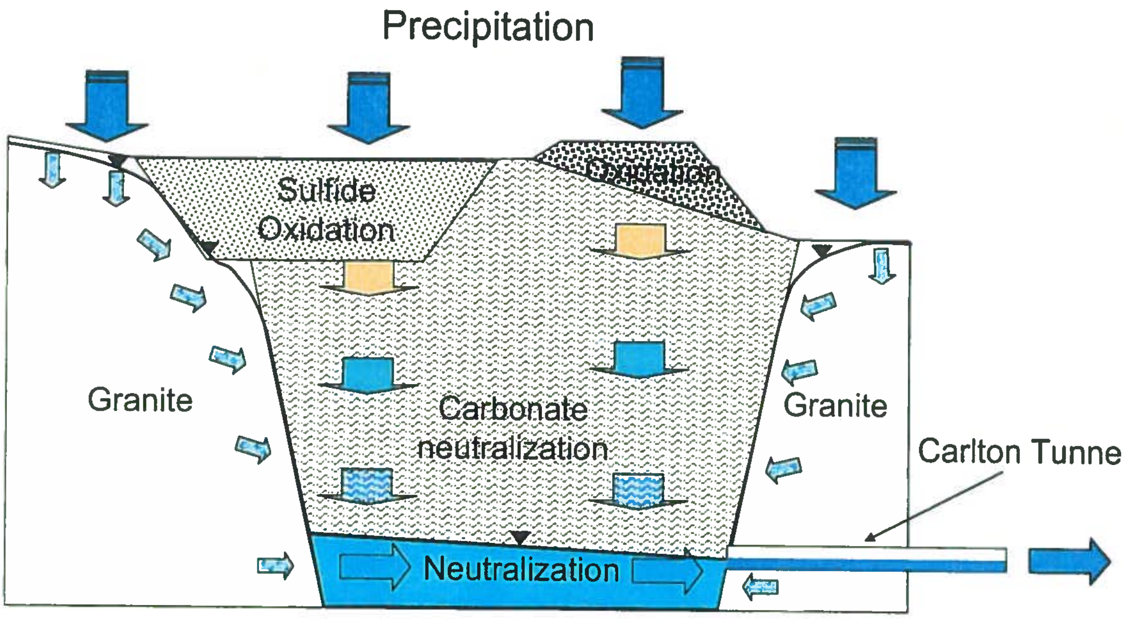


Figure 11 - Proposed Project Development Plan

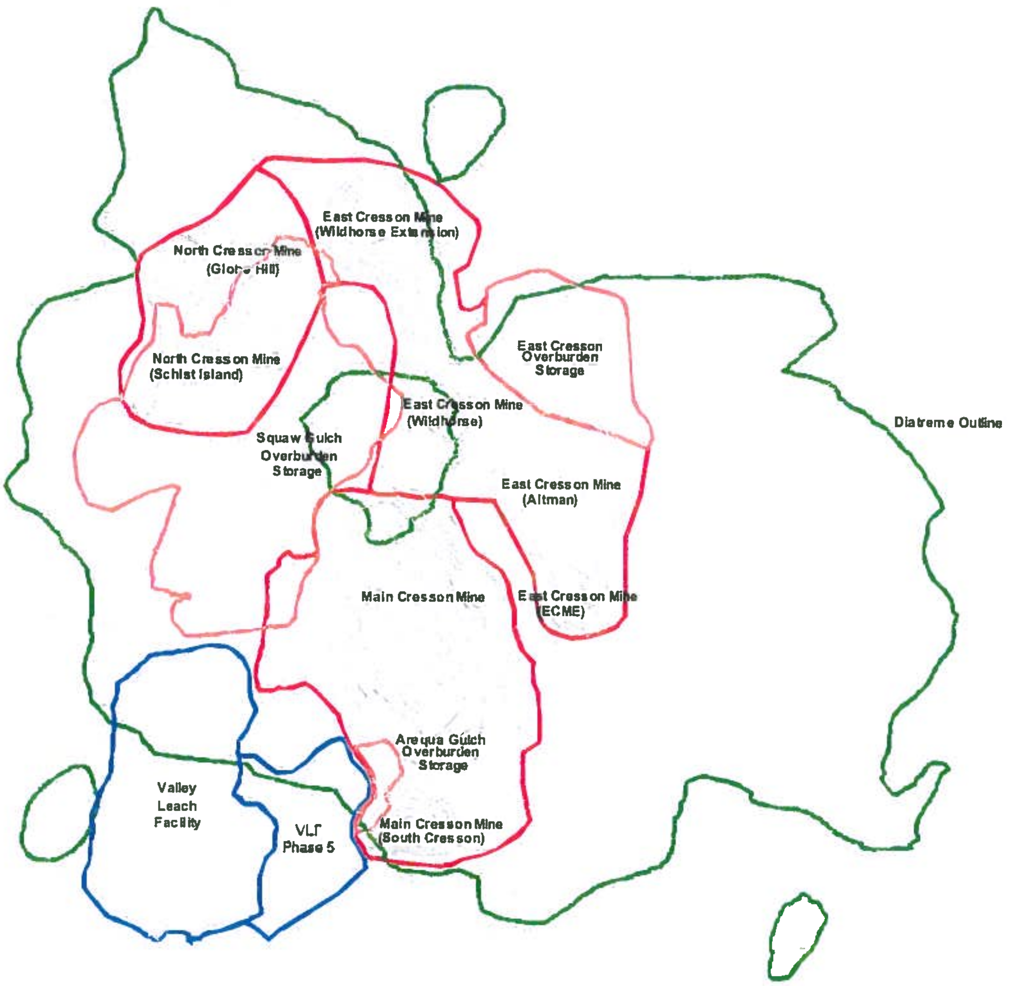


Figure 12 - Net Acid Generation

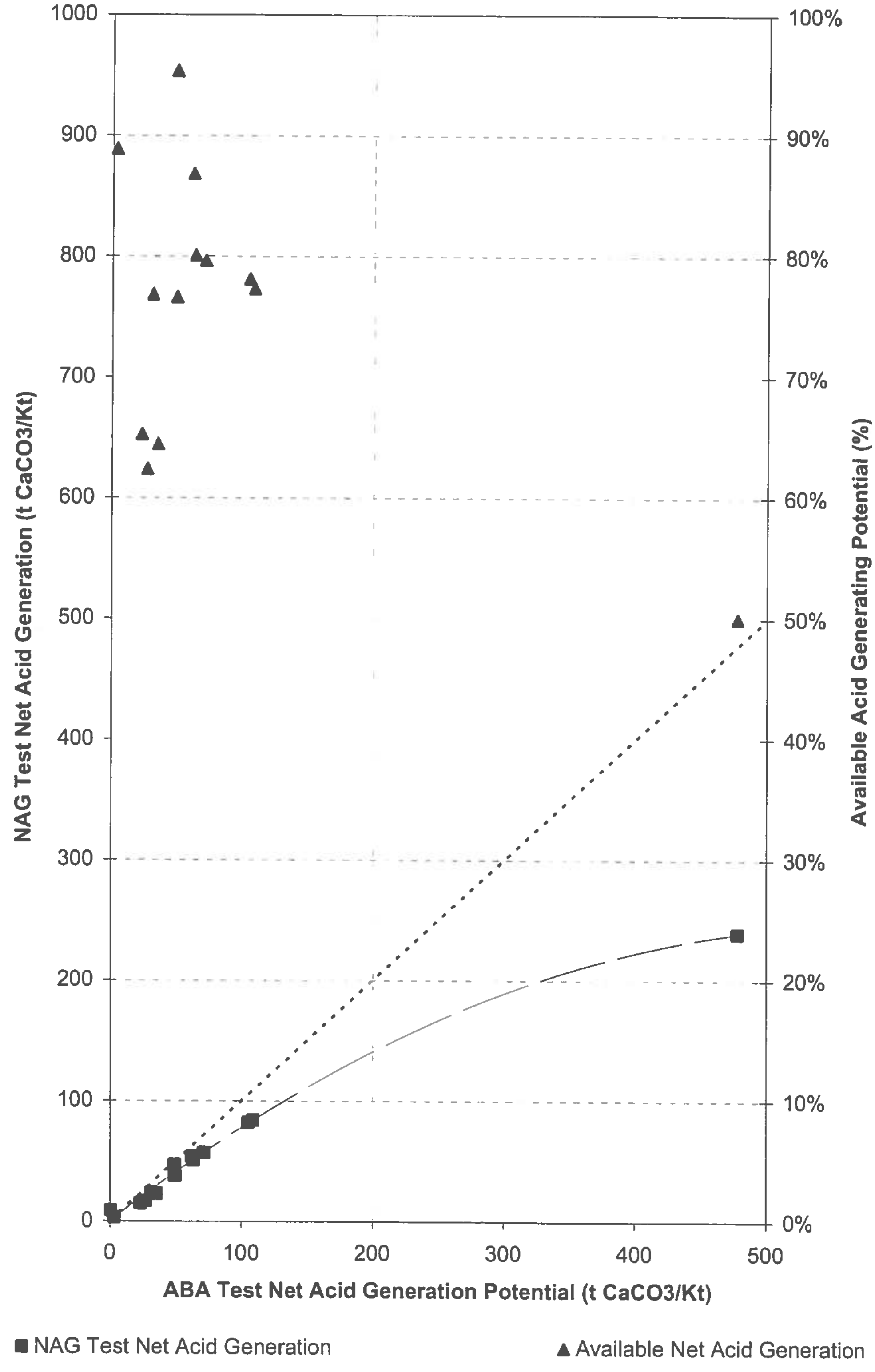


Figure 13 - Average pH of Humidity Cell Test Effluent

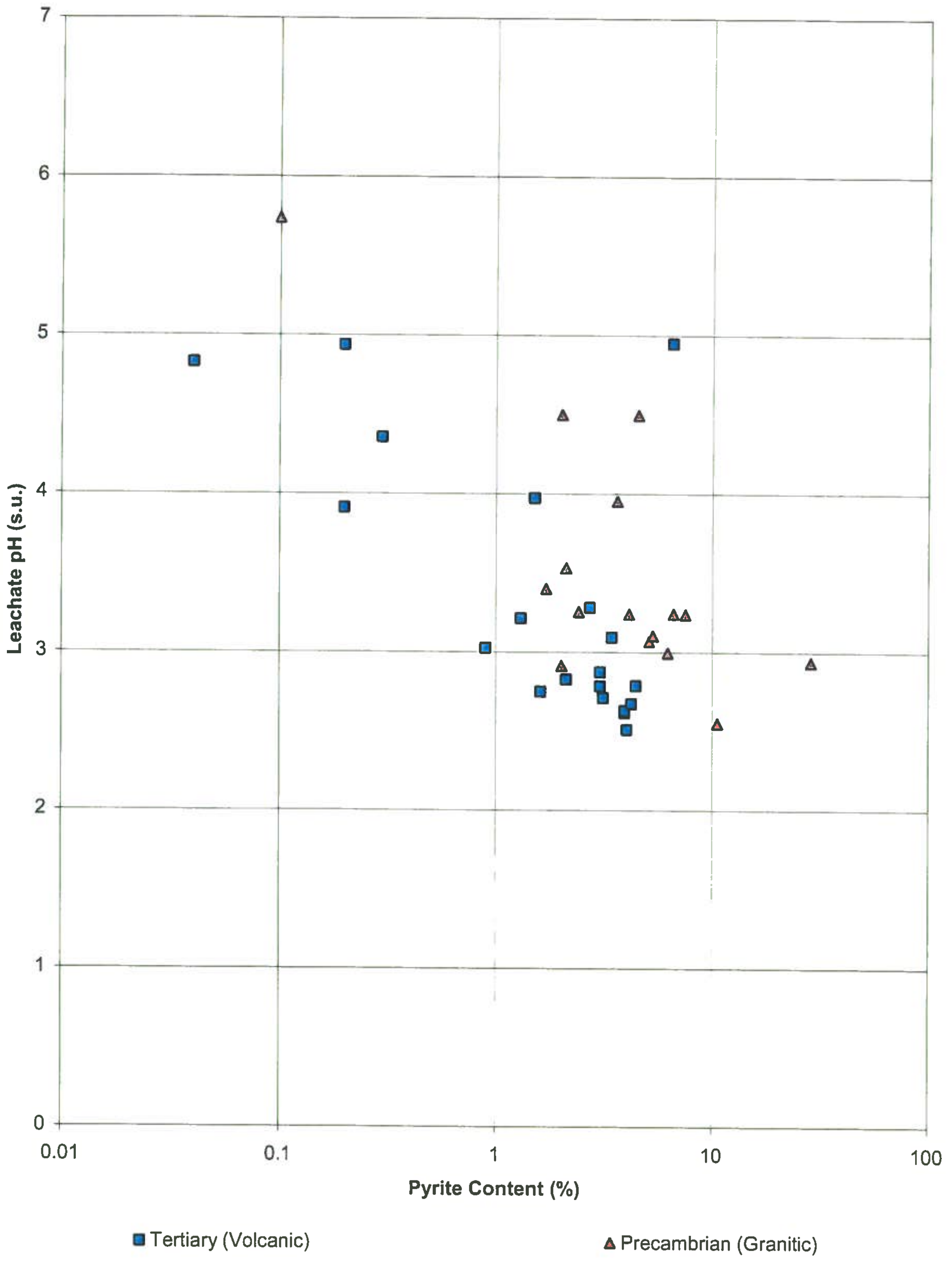


Figure 14 - Rate of Pyrite Oxidation in Run-of-Mine Overburden

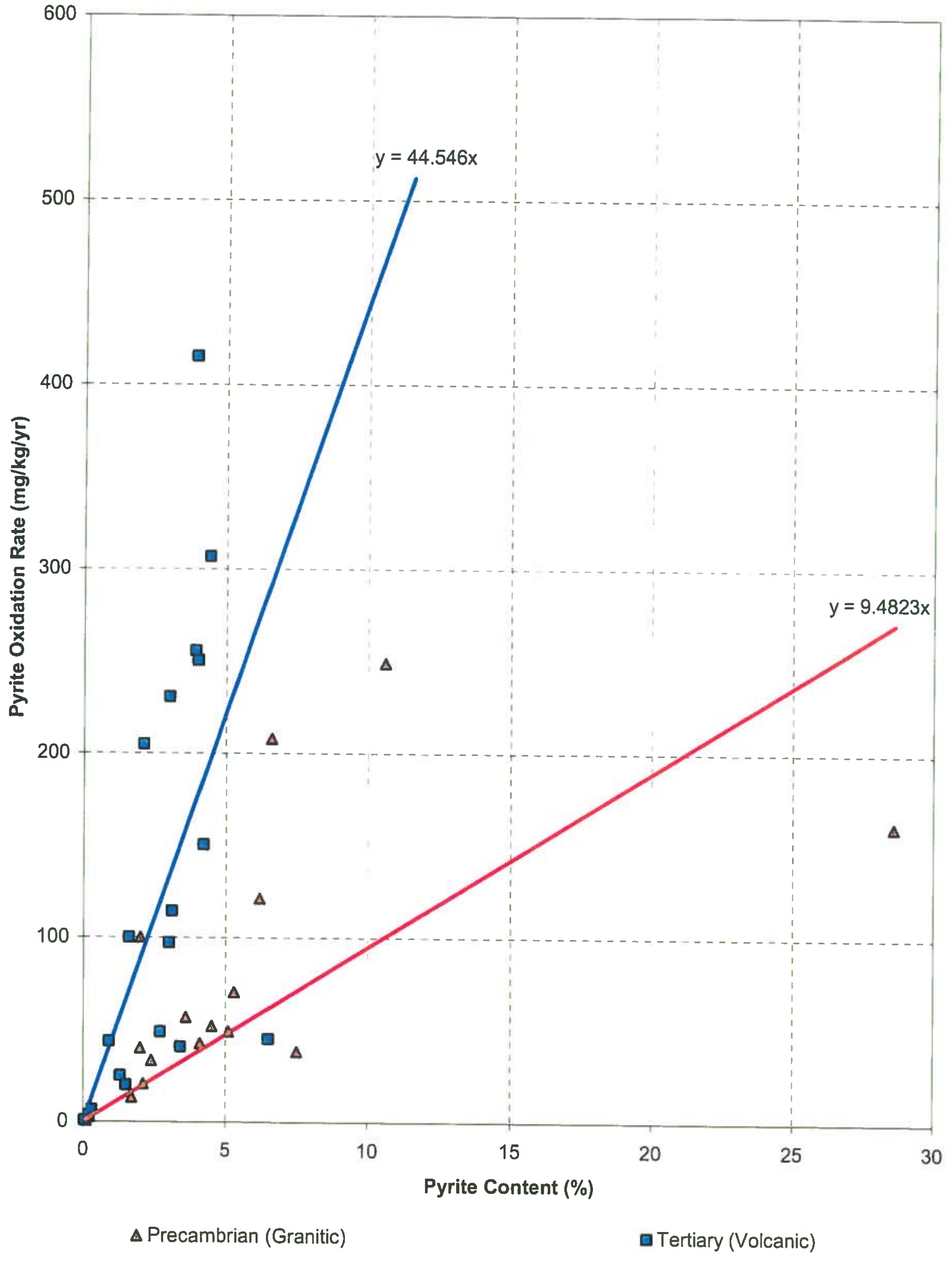


Figure 15 - Grainsize for Run-of-Mine and HCT Material

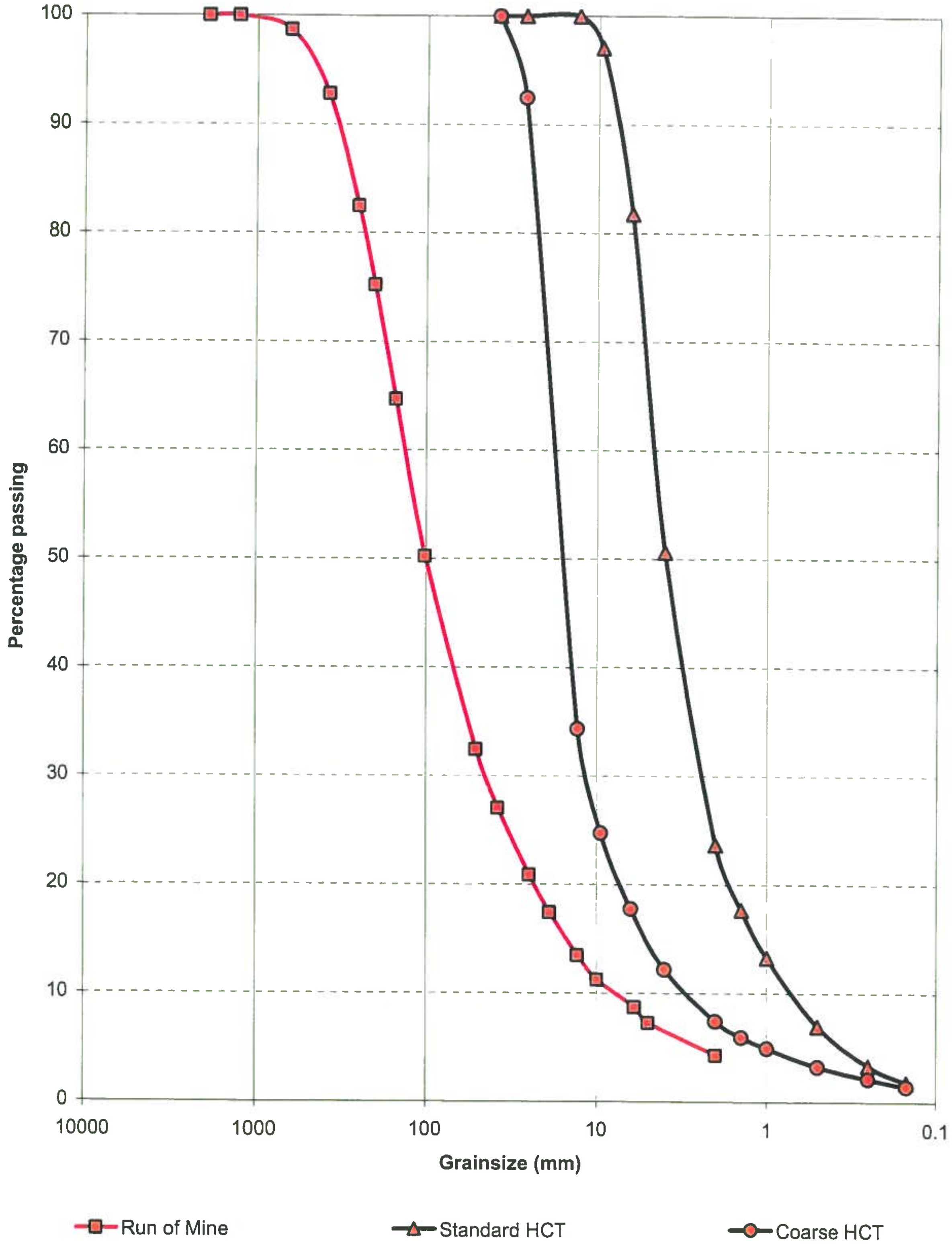


Figure 16 - Post-Closure Mine Wall Exposure

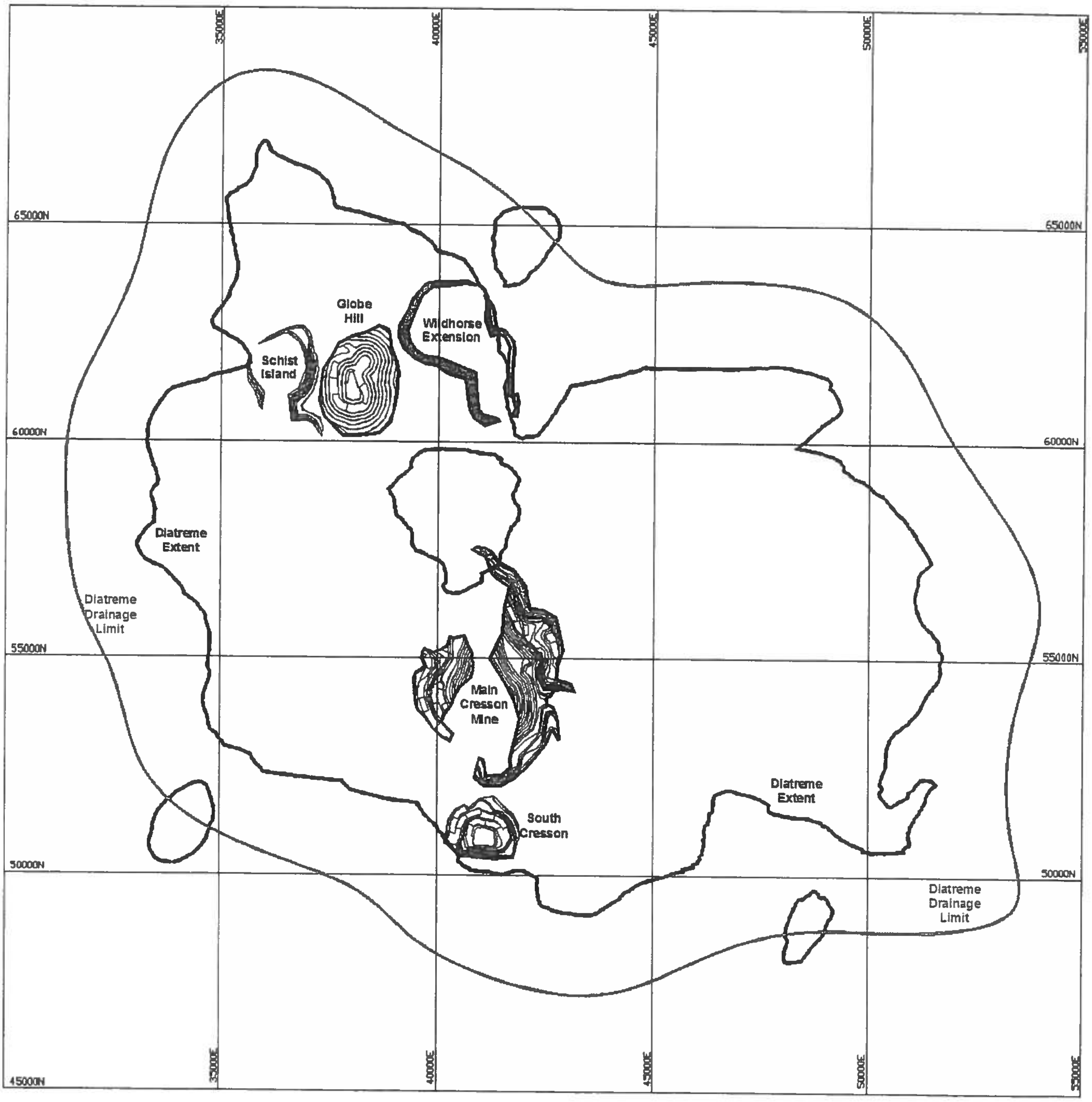


Figure 17 - Calcium Carbonate Abundance in the Diatreme

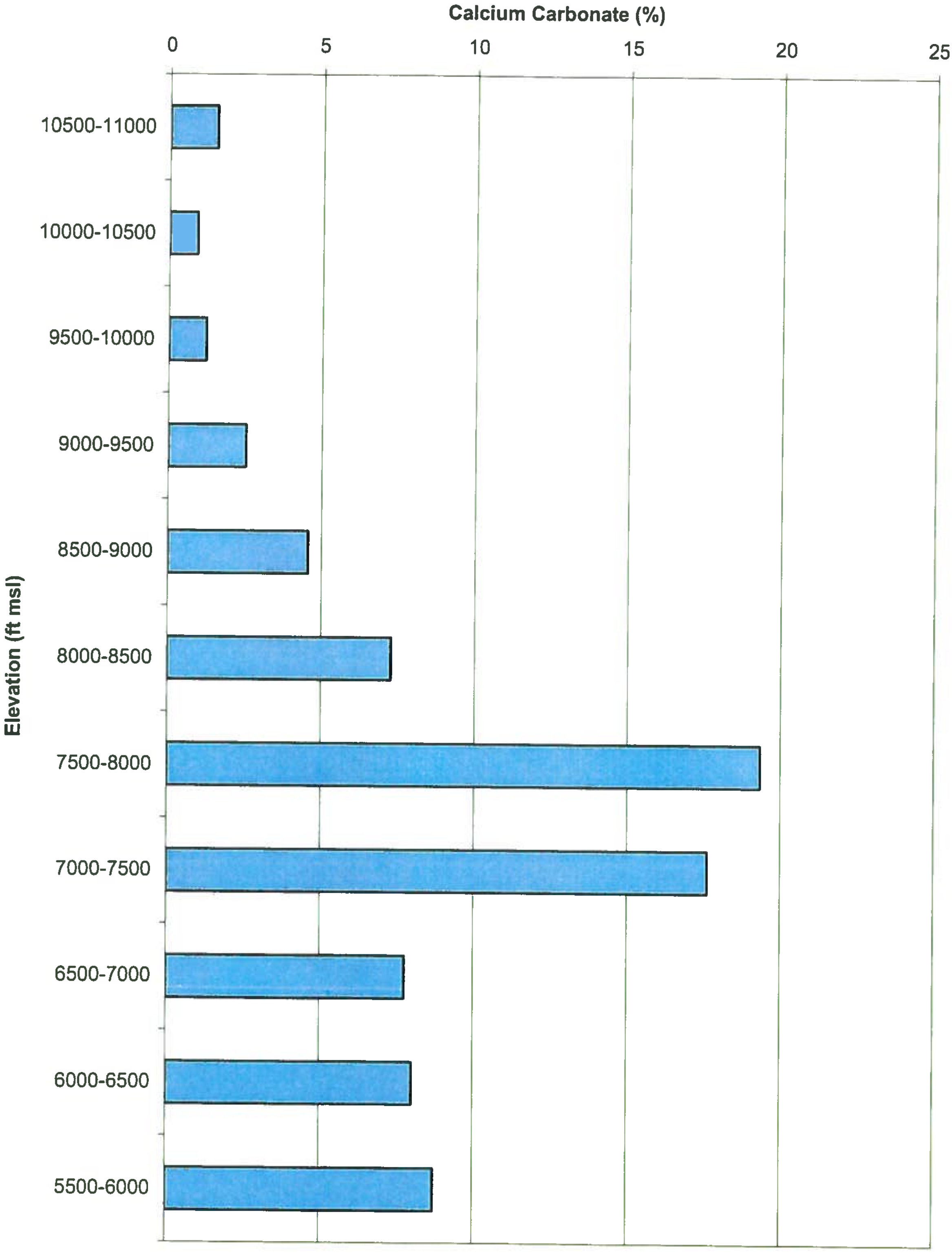


Figure 18 - Carlton Tunnel Alkalinity

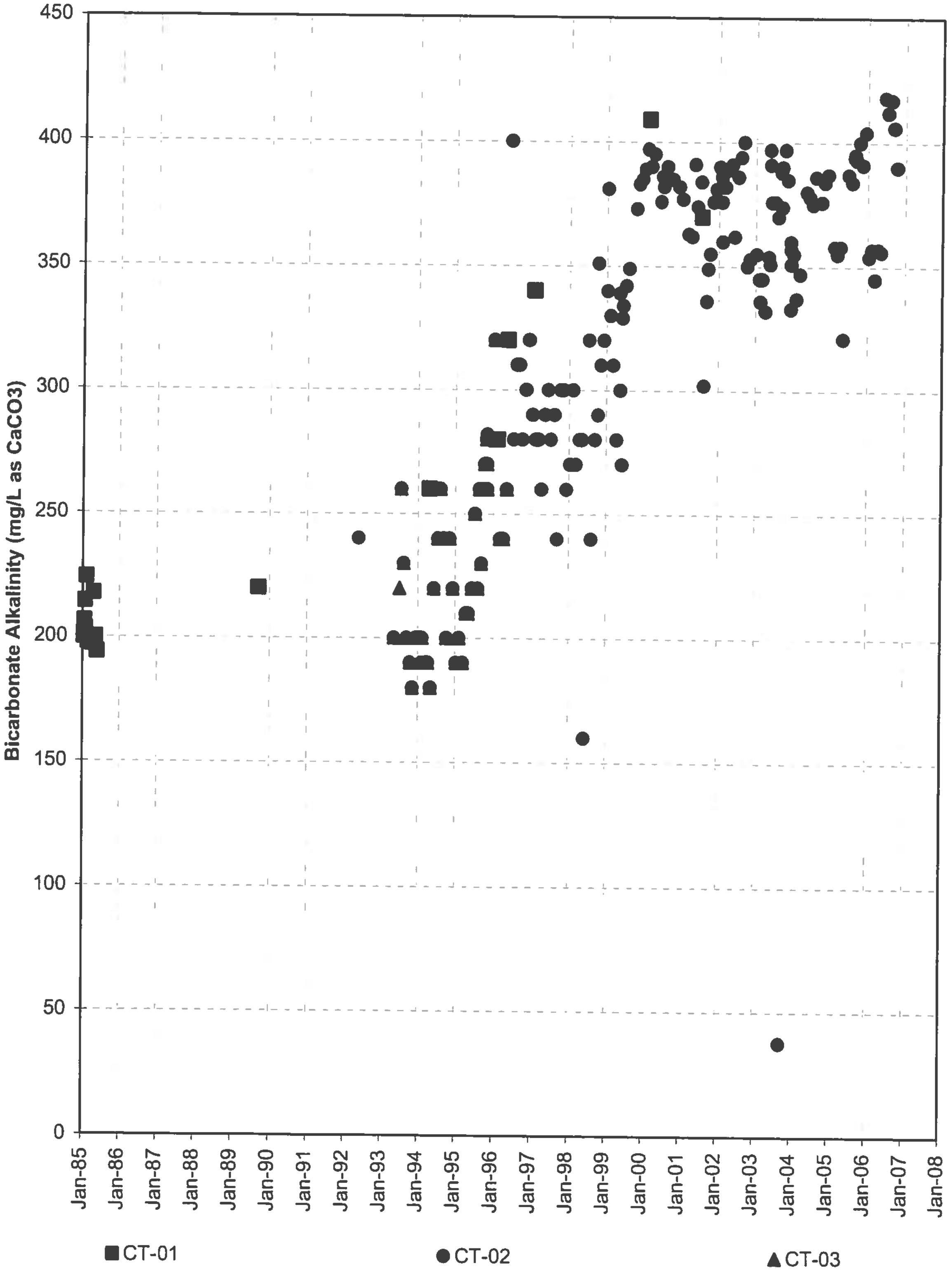
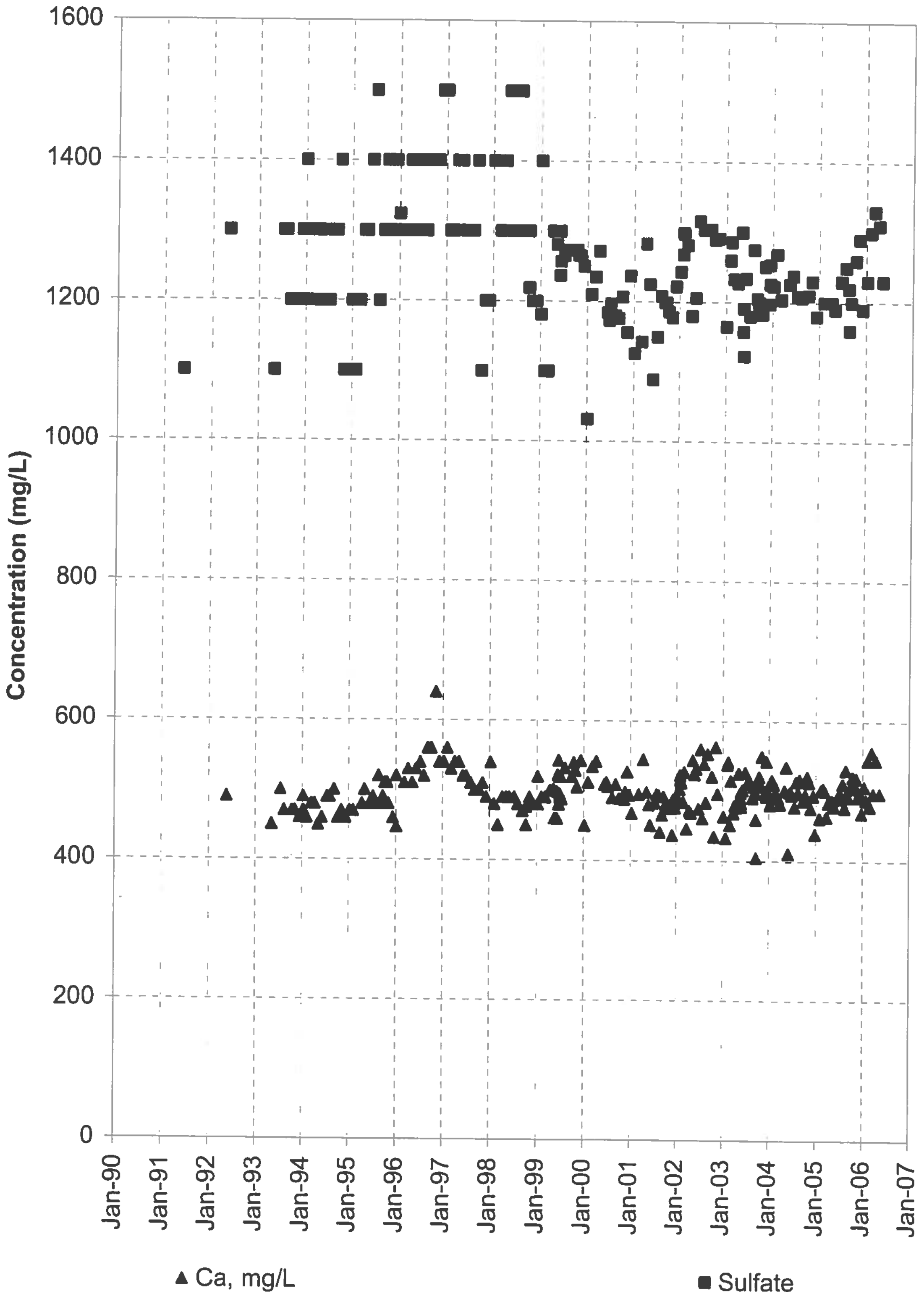


Figure 19 - Carlton Tunnel - Calcium and Sulfate



Appendix 1

ATTACHMENT ONE

INFILTRATION MODEL